# Cactus Mine Project
# NI 43-101 Technical Report and Pre-feasibility Study
## Arizona, United States of America

Effective Date: February 21, 2024

Report Date: March 28, 2024

Prepared for:


Arizona Sonoran Copper Company
950 W Elliot Rd. Suite 122
Tempe, Arizona, 85284


Prepared by:


Ausenco Engineering South USA Inc.
595 S Meyer Avenue
Tucson, Arizona, USA, 85701


List of Qualified Persons:


Erin L. Patterson, P.E., Ausenco Engineering USA South Inc.
Scott C. Elfen, P.Eng., Ausenco Sustainability ULC.
R. Douglas Bartlett, PG, CPG, Clear Creek Associates,
a subsidiary of Geo-Logic Associates.
Gordon Zurowski, P.Eng., AGP Mining Consultants Inc.
Nat Burgio, FAusIMM (CP), AGP Mining Consultants Inc.
Todd Carstensen, RM-SME, AGP Mining Consultants Inc.
Allan L. Schappert, CPG, RM-SME, ALS Geo Resources, LLC.
James L. Sorensen, FAusIMM, Samuel Engineering, Inc.
Matthew Bolling, P.E., PMP, Samuel Engineering, Inc.
Paul Cicchini, P.E., North Star Geotech, LLC.


# CERTIFICATE OF QUALIFIED PERSON
## Ms Erin Lynn Patterson, P. E.

I, Erin Lynn Patterson, P.E., certify that:

1. I am employed as Director of Technical Services within Ausenco with Ausenco Engineering USA South Inc. (“Ausenco”), with an office address of 595 S. Meyer Avenue, Tucson, Az, USA.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the “Technical Report”), prepared for Arizona Sonoran Copper Company (the “Company”) with an effective date of February 21, 2024 (the “Effective Date”).
3. I graduated from the University of Arizona with a Bachelor of Science degree in Chemical Engineering.
4. I am a registered professional engineer in the state of Arizona, USA, license #54243.
5. I have practiced my profession for a total of 18 years since my graduation from university. My relevant experience includes involvement in all levels of engineering studies from conceptual studies to feasibility as well as mineral projects in the construction and operation stages. The works that I have been directly involved in include the mineral commodities copper, nickel, gold, and silver. I have been directly involved with process design, including testwork interpretation and flowsheet development, design specifications, cost estimating, and execution of mineral projects.
6. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
7. I visited the AZ Cactus Mine Project on September 21, 2023. for a visit duration of one day.
8. I am responsible for 1.1, 1.2, 1.19.1, 1.20, 1.22, 1.23, 1.25, 1.26, 2.1, 2.2, 2.3, 2.4.8, 2.6.2, 2.7, 3.1, 3.3, 18.1, 18.2, 18.5, 18.6, 18.7, 18.8, 18.9, 19, 21.1, 21.2, 21.3.1, 21.3.2, 21.4.1, 21.4.4, 21.4.5, 22, 24, 25.1, 25.10.1, 25.12, 25.13, 25.14, 25.15.1, 25.15.1.8.1, 25.15.1.8.2, 25.15.1.8.3, 25.15.2.6.1, 26.1, 26.12.1 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Arizona Cactus Mine.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and Sealed"

Erin Lynn Patterson, P.E.

Ausenco

# CERTIFICATE OF QUALIFIED PERSON
## Mr Scott C. Elfen, P.E.

I, Scott C. Elfen, P.E., certify that:

1. I am employed as a Global Lead Geotechnical and Civil Services within Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC V6E 3S7, Canada.
2. This certificate applies to the technical report titled *Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA,* (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024 (the "Effective Date").
3. I graduated from the University of California, Davis, California, in 1991 with Bachelor of Science degree in Civil Engineering (Geotechnical).
4. I am a Registered Civil Engineer in the State of California (license no. C056527) by exam since 1996 and I am also a member in good standing of the American Society of Civil Engineers (ASCE), and the Society for Mining, Metallurgy & Exploration (SME).
5. I have practiced my profession for 28 years, with experience in the development, design, construction and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities ranging from slurry to dry stack facilities, focusing on precious and base metals, both domestic and international. In addition, I have developed geotechnical design parameters for pit slope design, plant foundation design, and other supporting infrastructure. Examples of detail engineering heap leach projects I have worked on include: Minera Escondida's Escondida Norte Mine, Barrick Gold's Pierina Mine, Barrick Gold's Lagunas Norte Mine.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project on September 21, 2023, for visit duration of one day.
8. I am responsible for 1.19.2, 2.2, 2.4.10, 18.10, 18.11, 18.12, 18.13, 25.10.2, 25.15.1.8.4, 25.15.2.6.2, 26.12.2 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined by Section 1.5 of NI 43-101.
10. I have had no previous involvement with Cactus Mine Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March28, 2024

"Signed and sealed"

Scott C. Elfen, P.E

# CERTIFICATE OF QUALIFIED PERSON
# R. Douglas Bartlett, PG, CPG

I, R. Douglas Bartlett, PG, CPG, certify that:

1. I am employed as a Principal Hydrogeologist with Geologic Associates, Inc., with an office address of [6155 E. Indian School Rd., Suite 200, Scottsdale, AZ 85251.
2. This certificate applies to the technical report titled *Cactus Mine Project, NI 43-101 Technical Report and Pre-feasibility Study, Arizona, United States of America.* (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024 (the "Effective Date").
3. I graduated from Colorado State University with a Bachelor of Science degree in Geology in 1977 and a Master of Science degree in Geology in 1984.
4. I am a Professional Geologist and Certified Hydrogeologist in good standing in California (CA PG 8809, CA CHG 965). I am also a Certified Professional Geologist with the American Institute of Professional Geologists (CPG No. 8433).
5. I have practiced my profession as a geologist/hydrogeologist for a total of 47 years. My experience includes assessing groundwater supplies for mining properties in the southwestern U.S. I have been directly involved in hydrogeologic studies at numerous mines in the Southwest U.S. including Freeport McMoRan mines in Morenci, Safford, Sierrita, Bisbee, Bagdad, Arizona, and Henderson, Colorado as well as Equinox Gold at Castle Mountain in California; Mountain Pass mine, California; and others throughout the western U.S.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Arizona Sonoran Copper Co. Cactus Mine on May 12, 2020, for the purpose of conducting a due diligence review of the property for Tembo Capital. I have subsequently visited the site on several occasions for site tours, review of site data, and oversight of field work for ASCU. I last visited the site in January 2024.
8. I am responsible for Sections 1.21, 2.2, 2.4.2, 3.2, 4.7, 4.8, 4.9, 5, 16.3, 20, 21.3.5, 25.11, 25.15.1.9, 25.15.2.7, 26.13 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I participated in conducting an environmental due diligence assessment of the Cactus Mine for Tembo Capital in 2020.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and sealed"

R. Douglas Bartlett, PG, CPG

# CERTIFICATE OF QUALIFIED PERSON
## Mr Gordon Zurowski, P. Eng.

I, Gordon Zurowski, P. Eng., certify that:

1. I am employed as Principal Mine Engineer with AGP Mining Consultants Inc., with an office address of #246-132 Unit K, Commerce Park Drive, Barrie, Ontario L4N 0Z7, Canada.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024 (the "Effective Date").
3. I graduated from the University of Saskatchewan with a B.Sc. in Geological Engineering in 1988.
4. I am a member in good standing of the Professional Engineers of Ontario (#100077750).
5. I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 30 years in mineral resource and reserve estimations and feasibility studies and mine operations around the world. As a result of my experience and qualifications, I am Qualified Person as defined in NI 43-101.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project on January 24, 2023, for a visit duration of one day.
8. I am responsible for Sections 1.15, 1.16, 1.17.3, 2.2, 2.4.5, 15.1, 15.2, 15.4, 15.5, 16.1, 16.5.1, 16.5.2, 16.5.4, 16.5.5, 16.5.6, 16.5.7, 16.5.8, 16.5.9, 16.5.10, 16.5.11, 16.5.13, 16.5.14, 16.5.15, 16.5.16, 16.6, 16.7, 16.8, 21.3.3, 21.4.2, 25.7, 25.15.1.3, 25.15.1.5, 25.15.2.3, 26.6, 26.7, 26.8, 26.9 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Cactus Mine Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and Sealed"

Gordon Zurowski, P. Eng.

# CERTIFICATE OF QUALIFIED PERSON
## Mr Natale Burgio, BSc, MBA FAusIMM(CP)

I, Natale Burgio, BSc MBA, FAusIMM (CP) certify that:

1. I am employed as Principal Geologist with AGP Mining Consultants Inc., with an office address of 32 Commerce Park Dr #620, Barrie, ON L4N 9P6, Canada.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024(the "Effective Date").
4. I graduated from Ballarat CAE with a degree in Applied Geology in 1982 and an MBA from Southern Cross University in 1999.
5. I am a Fellow Member of the Australian Institute of Mining and Metallurgy (No 112067)
6. I have practiced my profession for 30 years since graduation. I have been directly involved in mine planning, ore reserves and mine reconciliations for block cave and sublevel caving operations since 1995. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine project onJune 12 for a visit duration of 1 day.
8. I am responsible for sections 1.17.2, 2.2, 2.4.3, 15.3, 16.5.3, 16.5.12 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with the Cactus Mine project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 3/28/2024

"Signed and Sealed"

Natale Burgio, BSc, MBA, FAusIMM (CP)

# CERTIFICATE OF QUALIFIED PERSON
## Todd Carstensen, RM-SME

I, Todd Carstensen, RM-SME, certify that:

1. I am employed as a Principal Mine Engineer with AGP Mining Consultants Inc., with an office address 246-132 Unit K, Commerce Park Drive, Barrie, Ontario L4N 0Z7, Canada.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024(the "Effective Date").
3. I graduated from University of Wisconsin-Platteville in 1984 with a Bachelor of Science, Mining Engineering.
4. I am a Registered Member of Society of Mining, Metallurgy & Exploration SME, RM #04063866.
5. I have practiced my profession continuously since 1988. I have extensive experience in resource estimation, mine planning, scheduling, and financial evaluation for precious and base metal deposits including mine operations and project evaluations.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project on January 24, 2023, and June 12, 2023, for a visit duration of 1 day, respectively.
8. I am responsible for sections 1.17.1, 2.2, 2.4.4, 16.4, 18.3, 18.4, 25.8 and 27 the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Cactus Mine Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and Sealed"

Todd Carstensen RM-SME

# CERTIFICATE OF QUALIFIED PERSON
## Allan L Schappert, CPG, RM-SME

I, Allan L. Schappert, CPG, RM-SME, certify that:

1. I am employed as a Principal Resource Geologist with ALS Geo Resources LLC, with an office address of 711 S. Sean Dr., Chandler, AZ 85224.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024(the "Effective Date").
3. I graduated from Lakehead University with a BSc in Geology in May 1979.
4. I am a Certified Professional Geologist registered with the American Institute of Professional Geologist (CPG# 11758), and a Registered Member of the Society of Mining, Metallurgy, and Exploration (SME# 041640710).
5. I have practiced my profession for 44 years. I have experience in drill planning, interpretation, resource evaluation, and production mine geology at many exploration projects and operating mines, both domestic and abroad. I have been directly involved in the review and validation of the exploration data analysis (EDA), geologic modeling, and grade estimation for McEwen Mining's Los Azules project PEA in Argentina. I was Underground Chief Geologist at Freeport, Indonesia responsible for generation and updating of resource models for a large copper Porphyry deposit and associated Skarns.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project on numerous occasions between August 2019 and 2022.
8. I am responsible for 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.14, 1.24, 2.2, 2.4.1, 2.5, 2.6.1, 4.1, 4.2, 4.3, 4.4, 4.5, 4.6, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.4, 25.5, 25.15.1.1, 25.15.1.4, 25.15.2.1, 26.2, 26.5 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Cactus Mine Project from 2019 when then Elim Mining, now Arizona Sonoran Copper Company first purchased the property. With multiple site visits to check drilling, logging, sampling, and QA/QC procedures. I have also visited the associated assay lab to check on their sample security, assay methodologies, and internal QA/QC procedures.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and Sealed"

Allan L. Schappert, CPG, RM-SME

# CERTIFICATE OF QUALIFIED PERSON
## Mr James L. Sorensen, FAusIMM

I, James L. Sorensen, FAusIMM, certify that:

1. I am employed as a Director, Metals & Minerals with Samuel Engineering, Inc., with an office address of 8450 E. Crescent Parkway, Greenwood Village, CO 80111.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024(the "Effective Date").
3. I graduated from the University of Arizona, located in Tucson, Arizona in 1981with a Bachelor of Science degree in Metallurgical Engineering.
4. I am a Fellow Member of The Australasian Institute of Mining & Metallurgy ("FAusIMM") Registration No. 221286.
5. I have practiced my profession continuously for 43 years. I have been directly involved in Sections 1, 2, 13, 17, 25, and 26 in the areas related to metallurgy and recovery methods/processing facilities.
   I have extensive relevant experience (copper leaching, SXEW and support infrastructure) in the operation, metallurgical development, all levels of study, process/project design, construction, commissioning, and start-up of facilities like those described in the subject Technical Report. My roles have included metallurgical consultant, lead process engineer, study manager, engineering manager, project engineer and project manager. This experience extends to over 25 sites and projects located in the United States (Arizona, Nevada), Canada, Chile, Mexico, Peru, Australia, and Argentina.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project on numerous occasions since December 2019 (at least one annually through 2023) and most recently on August 31, 2023, for a visit duration of one (1) day.
8. I am responsible as the Qualified Person specifically for sections 1.13, 1.18, 2.2, 2.4.6, 13, 17, 25.6, 25.9, 25.15.1.2, 25.15.1.7, 25.15.2.2, 25.15.2.5, 26.3, 26.4, 26.11 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Cactus Mine Project since December 2019 including consulting and Technical Report Qualified Person responsibilities. Prior Technical Reports for the Cactus Mine and associated properties I have been involved with are:
    I. Preliminary Economic Assessment, Samuel Engineering, Effective Date: March 1, 2020, Prepared for Elim Mining Inc.
    II. Mineral Resource Estimate and Technical Report, Stantec, Effective Date: 10 November 2022, Prepared for Arizona Sonora Copper Company, Inc.
    III. Preliminary Economic Assessment, Stantec, Effective Date: 31 August 2021, Prepared for Arizona Sonora Copper Company, Inc.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and Sealed"

James L. Sorensen, FAusIMM

Registration No. 221286

# CERTIFICATE OF QUALIFIED PERSON
## Mr Matthew Bolling, P.E., PMP

I, Matthew Bolling, P.E., PMP, certify that:

1. I am employed as a Project Manager with Samuel Engineering Inc., with an office address of 8450 E Crescent Pkwy Suite 200, Greenwood Village, CO 80111.
2. This certificate applies to the technical report titled *Cactus Mine Project, NI 43-101 Technical Report and Pre-feasibility Study, Arizona, United States of America* (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024 (the "Effective Date").
3. I graduated from Colorado State University in Fort Collins, Colorado in 2005 with a Bachelor of Science in Civil Engineering.
4. I am a Professional Engineer in the State of Colorado and the State of Arizona. My Professional Engineering license number in Colorado is 46442 and my Professional Engineering number in Arizona is 77357.
5. I have practiced my profession for over 18 years. I have experience with solvent extraction electrowinning process facilities, material handling and crushing systems, and the development of the capital estimate to engineer, procure, and construct the facilities.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I have visited the Cactus Mine Project at least once a year since being involved in the project starting in early 2021. The most recent site visit occurred on January 17, 2023, for a visit duration of one (1) day.
8. I am responsible as the Qualified Person specifically for sections 2.2, 2.4.7, 3.4, 21.3.4, 21.4.3,& 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Cactus Mine Project since March 2021; During this time, I have been working in a consulting capacity on the design of the solvent extraction electrowinning process facility, the material handling and crushing systems, and also the development of the capital estimate to engineer, procure, and construct the facilities. In addition, I contributed content, but was not a QP on the below previously issued report:
    a. Mineral Resource Estimate and Technical Report, Stantec, Effective Date: 10 November 2022, Prepared for Arizona Sonora Copper Company, Inc.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and sealed"

Matthew Bolling, P.E., PMP

Colorado Professional Engineer No. 46442

# CERTIFICATE OF QUALIFIED PERSON
## Mr Paul F. Cicchini, P.E.

I, Paul F. Cicchini, P.E., certify that:

1. I am the President of North Star Geotech, LLC. (NSG), with an office address of 3609 W. Ironwood Meadows Place. Tucson, Arizona, 85742, USA.
2. This certificate applies to the technical report titled Cactus Mine Project NI 43-101 Technical Report and Pre-feasibility Study, Arizona, USA, (the "Technical Report"), prepared for Arizona Sonoran Copper Company (the "Company") with an effective date of February 21, 2024(the "Effective Date").
3. I graduated from The University of Arizona with a Bachelor of Science in geological engineering in May 1979.
4. I am a registered Professional Engineer, with the State of Arizona (No. 19629 - Geological), the State of Alaska (No. 108950 - Civil), the State of Utah (No. 9590636-22022 – Mining) and the State of Washington (No. 53170 – Mining).
5. I have practiced my profession for 44 years with experience in geotechnical mine design for open-pit and underground mining, serving as engineer and principal of the consulting firm Call & Nicholas, Inc. for 39 years. For the last 5 years I have served as an independent reviewer on geotechnical review boards for Rio Tinto, Freeport, Newmont, Glencore and Silver Standard.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Cactus Mine Project between November 6, 2023, for a visit duration of 5 hours.
8. I am responsible for subsections 2.2, 2.4.9, 16.2, 25.15.1.6, 25.15.2.4, 26.10 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Cactus Mine Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: March 28, 2024

"Signed and sealed"

Paul F. Cicchini, P.E.

## Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Arizona Sonoran Copper Company (ASCU) by Ausenco Engineering USA South, Inc. Ausenco Sustainability ULC. (collectively, Ausenco), Clear Creek Associates LLC., AGP Mining Consultants Inc., ALS Geo Resources LLC., Samuel Engineering Inc., Call & Nicholas Inc (CNI) and North Star Geotech LLC (NSG), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by ASCU subject to the terms and conditions of its contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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# 1 SUMMARY

## 1.1 Introduction

Ausenco Engineering USA South Inc. and Ausenco Sustainability ULC (Ausenco) have compiled this prefeasibility study (PFS) and associated technical report for Arizona Sonoran Copper Company (ASCU) for the Cactus Mine Project (the Project) located in Casa Grande, Arizona. This report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by ASCU to manage and coordinate the work related to the PFS and the technical report. Ausenco was also retained to complete the infrastructure design, leach pad design, and to compile the overall cost estimate and financial model.
- AGP and Call & Nicholas (CNI) were commissioned to provide the mining methods for the underground and open pit. AGP provided designs for view berms, waste piles, and the stockpile relocation. Capital and operating costs were included in their scope.
- Samuel Engineering (Samuel) was commissioned to provide the mineral processing and metallurgical testing basis and plant design. Samuel's scope included the metallurgical testwork supervision and analysis, SX/EW plant, leaching process, conveyor systems, crushing and stockpile designs. Capital and operating costs were included as part of their scope.
- Clear Creek managed the drilling programs, hydrogeologic evaluations and environmental fieldwork for the study.
- ALS Geo Resources (ALS) was retained to provide background data for the Project. The local history, mineralization, exploration, quality assurance/quality control (QA/QC), general geology, creation of the resource model and estimation of final resource numbers were provided by ALS Geo Resources.

## 1.2 Terms of Reference

This report supports disclosures by ASCU in a news release dated February 21, 2024, entitled "Arizona Sonoran Announces a Positive Pre-Feasibility Study for the Cactus Mine Project with a US$509M Post-Tax NPV and 55 kstpa Copper Cathode over 21 Years".

All measurements presented in this report are in imperial units unless otherwise noted. Currency is expressed in US dollars (US$ or USD) unless otherwise noted. Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The ASCU property contains one past-producing mine that operated from 1972 until 1984. The Sacaton Mine, as it was originally named, collectively produced 400 M lbs of copper (Cu), 27,455 oz of gold (Au) and 759,000 oz of silver (Ag).

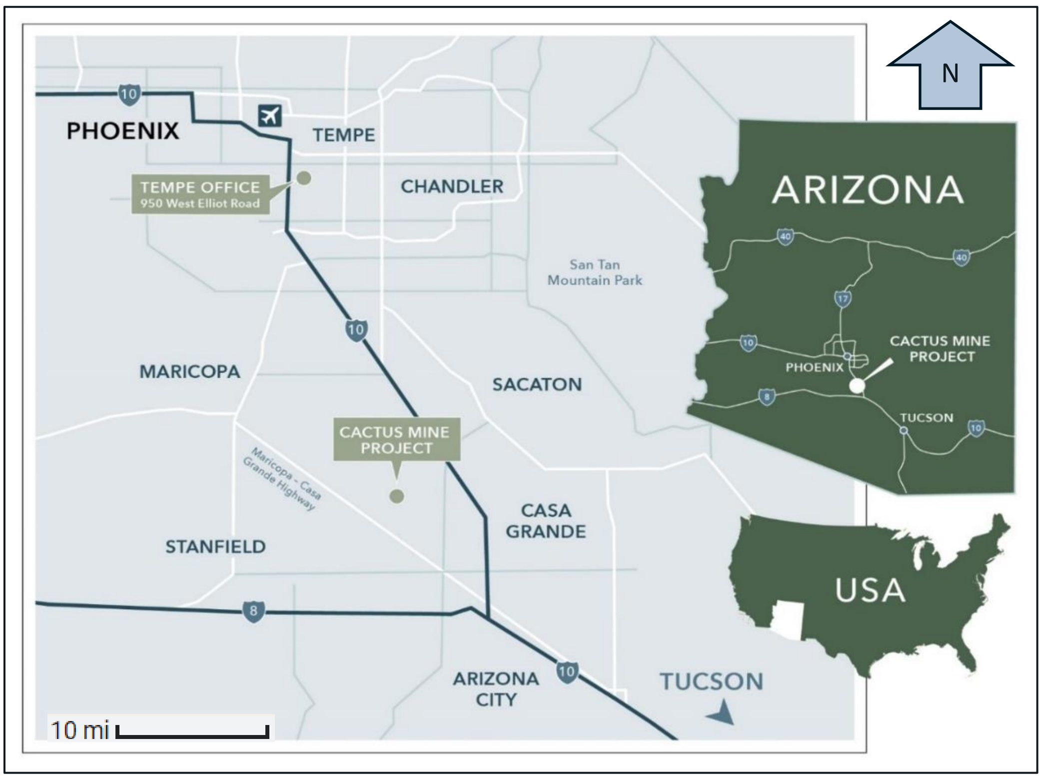
The existing open pit represents the source for the original Sacaton Mine (see cover photo). The Cactus East and the Parks Salyer underground deposits are included in the current project, which is now referred to as the Cactus Mine or Cactus Project.

## 1.3 Property Description and Location

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

The current project is located at the historic Sacaton Mine, which is 10 mi due west of the Interstate 10 (I-10) freeway. Total site area is approximately 5,380 acres. Figure 1-1 shows the Cactus Project location.

**Figure 1-1: Cactus Project Location**




Source: ASCU, 2024.

In August 2019, Cactus 110 LLC, a wholly-owned subsidiary of ASCU, executed a purchase agreement and prospective purchaser's agreement 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, ASCU 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, ASCU acquired a prospecting permit for adjacent land owned by the Arizona State Lands Department.

In February 2021, 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, 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, ASCU 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 ASCU. This MEP consists of 157.50 acres of State-owned surface and minerals.

In February 2023, Cactus 110 LLC executed an agreement with MainSpring Casa Grande LLC to purchase 522.78 acres of land adjacent to the Project, increasing its total landholding to 5,370 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 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The project is located in Pinal County approximately 6 mi (10 km) northwest of the city of Casa Grande and 40 road miles south southwest of the Greater Phoenix metropolitan area. Access to the Project is 4.6 mi (7.4 km) west of AZ-387 on North Bianco Road off West Maricopa-Casa Grande Highway. The coordinates for the centre of the Project are -111.828129° longitude and 32.948166° latitude, with a variable elevation between 1,330 to 1,510 ft (405 to 460 m) above sea level (asl).

## 1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The project is located approximately 3 mi northwest of the City of Casa Grande, Pinal County, Arizona. It is 40 road miles south-southeast of the Greater Phoenix metropolitan area and 70 road miles northwest of Tucson. It is easily accessible from the Interstate 10 (I-10) freeway, which is approximately 10 mi east of the historic Sacaton Mine. The Greater Phoenix area is a major population centre (approximately 4.5 million people) with a major airport and transportation hub and well-developed infrastructure and services that support the mining industry.

Climate at the mine is also typical of the Arizona Sonoran Desert, with temperatures ranging from 19°F to 117°F and with average annual precipitation of 8.6 inches (in), falling primarily in high-intensity, short-duration events. The mine site contains no surface water resources.

Electric power is available from Arizona Public Service's (APS) 69-kilovolt (kV) transmission line which passes on the South side of the site and connects to an existing substation owned by ASCU.

Water rights are discussed in Section 4.4. It is expected that credits will be obtained for de-watering of the pit and underground shaft. A maximum of 2,516 gpm make-up requirement is required from offsite sources.

Casa Grande and Maricopa, in conjunction with Phoenix, are in proximity and can collectively offer an ample pool of skilled labour for the Project.

## 1.6 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 (now known as Cactus East) 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 on March 31, 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) in depth. The pit also has a visible internal lake with the surface positioned at a depth of approximately 980 ft (299 m) from the rim of the pit. During operation, the Sacaton mine consisted of the pit, crushing facilities and coarse ore stockpile, a 9,000 ton/d 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 ton/d. 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 (Mton) of ore were mined and processed, recovering 400 million lbs. of copper (Cu), 27,455 oz of gold (Au), 759,000 oz of silver (Ag).

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

## 1.7 Geology and Mineralization

The Cactus and Parks/Salyer Projects occur in the desert region of the Basin and Range province of Arizona (AZ). 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 sulphides 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 are predominated in the lower portions.

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

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

## 1.8 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 sulphides 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.

## 1.9 Exploration

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. Eighty-two additional holes were drilled from 1962 through the first half of 1963. These eighty-eight holes outlined a northeasterly trending alteration zone approximately 4 mi (6.4 km) long and 1.5 mi (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. Sacaton East and West were re-named as Cactus East and West when the property was purchased by then Elim Mining in 2019. Elim Mining was later renamed Arizona Sonoran Copper Company (ASCU).

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 were conducted for the open pit (Cactus West deposit).

Eight additional holes were drilled from 1974 through 1976, in the Cactus East deposit for definition purposes.

## 1.10 Drilling

In 2019, ASCU drilled two vertical PQ (4.95 in or 12.57 cm) 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. Drilling activities conducted at Cactus East and Cactus West in 2021, 2022, and early 2023 upgraded most of the Inferred material in the resource to Indicated and some Measured.

In 2019, 55 surface sonic drill holes totaling 5,120 ft (1,560 m) of 6-in 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. Sonic drilling continued on the Project to ultimately reduce the spacing to 200 ft (61 m).

## 1.11 Sampling Preparation and Security

Arizona Sonoran has been exclusively using Skyline Assayers and Laboratories (Skyline Labs), in Tucson, Arizona, for their sample preparation and analysis. Bagged samples with identification tags are placed in large 3-ft (1-m) square plastic totes, which are stored at a core shed and 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 ASCU upon dispatch, signed by Skyline Labs upon arrival, and returned to ASCU to show secure delivery.

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 (93°C) for 24 hours to dry before moving into the lab for processing.

As a first pass, each sample was assayed for total copper (CuT) value. 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 (CuAS) value 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.

In late 2020, ASCU 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 ASCU'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, ASCU 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, ASCU began an exploration diamond drilling (DD) 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 Infill drilling process that continued through early 2023 and involved 47 DD holes totaling 105,810 ft (32,251 m), brought the defined Parks/Salyer resource to a mostly Indicated and Inferred confidence level. The total Parks/Salyer program covered 74 DDs for 166,685 ft (50,806 m).

### 1.12 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, ASCU has drilled 73 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 74 new holes drilled by ASCU between 2021 and 2022. There were only four historical holes supporting the Parks/Salyer resource estimate.

Specific data verification work undertaken by ASCU 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.

For the 73 new Cactus drill holes, 74 new Parks/Salyer drillholes, and 206 new Stockpile Project drill holes undertaken by ASCU since 2019, physical checks on collar, downhole survey, and logging have been completed by the qualified person (QP).

Observation and checks completed by the QP included:

- independent GPS check of collar location,
- observation and check of downhole survey results,
- observation of core logging and recording activities,
- drill hole database checks, and
- review of lab internal and ASCU's random external checks using assay duplicates, prepared standards, and blanks (QA/QC) program.

### 1.13 Mineral Processing and Metallurgical Testwork

The metallurgical studies and testing for the Cactus Project have been ongoing since late 2019 and has been conducted in four phases of testing though 2023. The information developed since the prior PEA is disclosed in this report.

Arizona Sonoran geologists are working with metallurgical engineers to quantify the recovery of copper from samples obtained in a series of large drilling campaign. The drill core samples were studied by geologists and subsequently shipped to a well-established mineral processing research and development firm in Reno, Nevada (McClelland Analytical Service Laboratory (McClelland), an ISO 9000, ISO 17025 accredited facility). Additional testing work was completed on-site by ASCU staff and at HydroGeoSense Inc. (HGS) laboratories in Tucson, Arizona. The metallurgical test program completed at McClelland has been developed by and supervised by Mr. James L. Sorensen. Mr. Sorensen has also reviewed and inspected the ongoing metallurgical testing at site and information developed by HGS.

Resources considered for beneficial processing in this Report are related to four sources:

- An existing mine stockpile built during the development and operation of a copper open pit and milling facility from 1974 to 1984. The stockpile includes oxide and lower grade sulphide material containing primarily copper mineralization.
- Further development of the existing Cactus West open pit containing oxide and lower grade sulphide material.
- The underground resource called Cactus East located northeast immediately adjacent to the existing Cactus open pit and at a depth of 1,200 ft. This resource contains mostly lower-grade sulphide material.
- The underground resource called Park Salyer located about 1 mile to the southwest of the Cactus West open pit at a depth of 1,500 ft. This resource contains mostly higher-grade secondary and primary copper sulphide material.

Approximately 45 column tests have been completed (Stockpile - 25, Cactus – 14, Parks/Salyer - 6) covering the resources identified in the current study effort. In addition, over 150 bottle roll tests, mineralogical analyses and other metallurgical and materials property testing has been completed.

The QP believes the metallurgical testing and data collected to date is sufficient to establish the required supporting metallurgical performance expectations used in estimating the project Reserves and economics for the Stockpile, Cactus East, Cactus West and Parks/Salyer deposits included in the Cactus Project. However, only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit and additional confirmatory work is required to better understand the deposit variability.

The materials are believed to be suitable for treatment in a heap leach, solvent extraction, and electrowinning (SX/EW) process facility to produce copper cathodes at LME Grade A quality standards ASTM B115-10 - Cathode Grade 1.

In consideration of a potential copper heap leaching and SX/EW processing facility at Cactus and Parks/Salyer, a hydrometallurgical approach is contemplated to process the oxide and enriched sulphides (chalcocite/minor covellite dominant) material identified in the mineralized Cactus East and Cactus West extensions to the existing open pit and underground and the Parks/Salyer underground mined materials reported in this Mineral Resource Estimate.

The Cactus heap leaching process design includes crushing of all material types for leaching to a minus ¾" $P_{80}$ size. All material types, oxides and enriched are to be leached in a single pad with an initial leaching cycle of 180 days. A maximum 3-year leaching cycle has been assumed (3 lifts) as the practical limit for effective recovery based on experience and hydrodynamic analysis of the materials by HGS. The copper leaching metallurgical test data has been extrapolated from the testing data at one year based on the rates prevailing after one year using a logarithmic curve fit projection that considers the decaying rate of copper extraction.

Scalability has been considered by 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.

The recommended copper recovery projections include this efficiency factor applied to the extraction obtained from the column testing.

Based on the above, the recommended copper extraction estimates for use in evaluating the Cactus Project resources is presented in Table 1-1.

**Table 1-1: Copper Recovery by Sequential Assay Fraction**

| Resource Area | Units | Value |
|---|---|---|
| Stockpile Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 87.7 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Oxide Heap (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 93.1 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Enriched Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 91.2 |
| Cyanide Soluble Copper Recovery | % | 84.5 |

Applying these extraction criteria, the calculated overall soluble copper (Tsol) recovery to cathodes is 86.3% and the corresponding total copper recovery is 76.1% for the resources contained in the mine plan.

A production timing has been assigned for each material type corresponding to the material mined in one year and the expected delays in achieving the two- or three-year final recovery values. This factor is intended to account for material placement timing over the course of a year and leach cycle delays in subsequent new lift placements.

Sulfuric acid consumption per ton of material leached is dependent on several factors. Gross acid consumption varies by materail type in each deposit. Net acid consumption acounts for acid regenerated in the electrowinning process when copper is plated to product. Net acid consumption per ton of material is dependent on recoverable copper content with a stochiometric converion of 1.54 tons of acid generated per ton of copper plated in electrowinning.

The stockpile material is more complex given that there are no geologic constraints to apply and waste materials have been mixed in. Calcium was determined to provide a measurable indicator for acid consumption and a calcium distribution model was developed and applied to estimate gross acid consumption. For the materails included in the mine plan, the average gross acid consumption averages 22 lbs of acid per ton of material and ranges from 16.7 lbs/ton to 25.7 lbs/ton depending on the average calcium content annually.

For Cactus East and West materials, the gross acid consumption for the oxide dominant material is 22 lbs/ton from the column testing. Enriched material gross acid consumption is slightly lower at 21 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals.

For the Parks Salyer enriched material, gross acid consumption is lower at 16 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals, lower clay, biotite and calcite mineralogy when compared to Cactus samples tested.

Applying the specific gross acid consumption for each material the overall LOM gross acid consumption is calcualted to be 19.3 lbs per ton and varies from 27.0 lbs/ton to 15.7 lbs/ton in a given year. The LOM Net acid consumption is calculated to be 6.5 lbs per ton and varies from 15.7 lbs/ton to net acid generating in a given year. Years where acid regenrated exceeds the acid required to be consumed will need to be attenuated with low grade/high calcium content material from the stockpile or tailings.

Similar to copper recovery, acid consumption is distributed over a two-year period with 75% of the estimated acid requirement consumed in the year of placement and 25% of the requirement in the following year.

## 1.14 Mineral Resource Estimate

The Cactus Project resource estimate, including the Cactus East, West, Parks/Salyer, and Stockpile 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. It includes the results of drilling programs undertaken by ASCU between 2019 and 2023. The material mined in the Sacaton open pit, operational from 1974 through 1984, has depleted the resource. The estimate of the Mineral Resources supports Measured, Indicated and Inferred Resources for Cactus, Indicated, and Inferred Resources from Parks/Salyer, and Inferred Resources for the Stockpile Project. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

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 percentages.

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

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

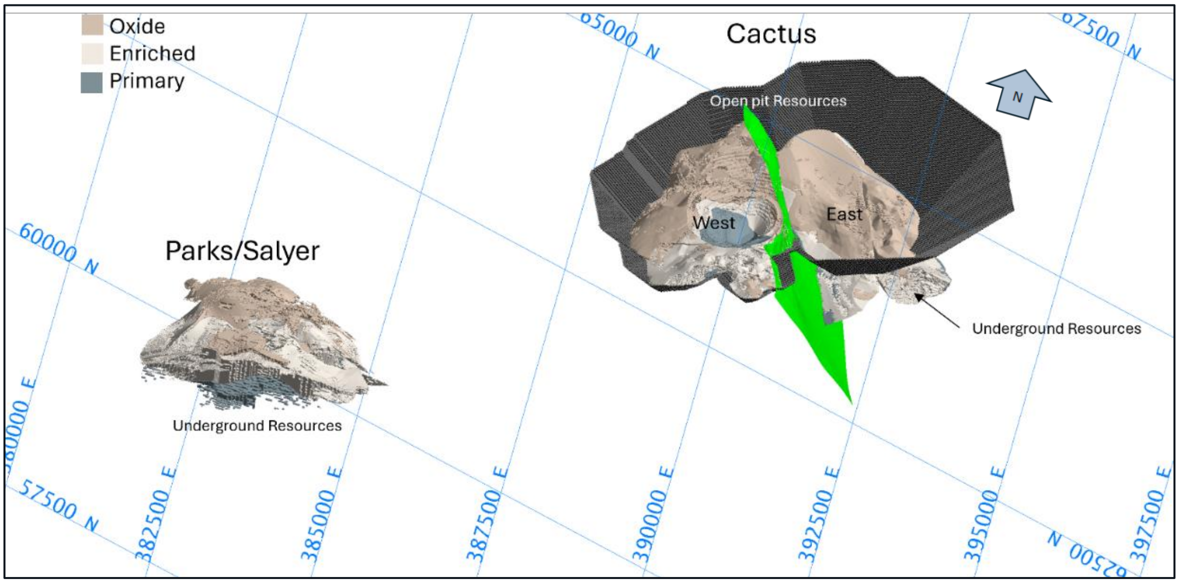
| Material Type | ktons (kt) | CuT (%) | Tsol (%) | Contained Cu (k lbs) |
|---|---|---|---|---|
| **Total Resources** | | | | |
| **MEASURED** | | | | |
| Total Leachable | 9,100 | | 0.230 | 41,900 |
| Total Primary | 1,300 | 0.315 | | 8,000 |
| **Total Measured** | **10,400** | **0.241** | | **49,800** |
| **INDICATED** | | | | |
| Total Leachable | 348,500 | | 0.629 | 4,387,200 |
| Total Primary | 86,800 | 0.425 | | 737,000 |
| **Total Indicated** | **435,300** | **0.588** | | **5,124,200** |
| **M&I** | | | | |
| Total Leachable | 357,600 | | 0.619 | 4,429,000 |
| Total Primary | 88,000 | 0.423 | | 745,000 |
| **Total M&I** | **445,700** | **0.580** | | **5,174,000** |
| **INFERRED** | | | | |
| Total Leachable | 107,700 | | 0.607 | 1,307,900 |
| Total Primary | 126,200 | 0.357 | | 900,000 |
| **Total Inferred** | **233,800** | **0.472** | | **2,207,900** |

Notes:

1. Leachable copper grades are reported using sequential assaying to calculate the soluble copper grade. Primary copper grades are reported as total copper, Total category grades reported as weighted average copper grades of soluble copper grades for leachable material and total copper grades for primary material. Tons are reported as short tons.
2. Stockpile resource estimates have an effective date of 1 March 2022, Cactus resource estimates have an effective date of 29th April 2022, Parks/Salyer resource estimates have an effective date of 19th May 2023. All resources use a copper price of US$3.75/lb.
3. Technical and economic parameters defining resource pit shell: mining cost US$2.43/t; G&A US$0.55/t, 10% dilution, and 44°-46° pit slope angle.
4. Technical and economic parameters defining underground resource: mining cost US$27.62/t, G&A US$0.55/t, and 5% dilution,
5. Technical and economic parameters defining processing: Oxide heap leach (HL) processing cost of US$2.24/t assuming 86.3% recoveries, enriched HL processing cost of US$2.13/t assuming 90.5% recoveries, Primary mill processing cost of US$8.50/t assuming 92% recoveries. HL selling cost of US$0.27/lb; Mill selling cost of US$0.62/lb.
6. Royalties of 3.18% and 2.5% apply to the ASCU properties and state land respectively. No royalties apply to the MainSpring (Parks/Salyer South) property.
7. For Cactus: Variable cutoff grades were reported depending on material type, potential mining method, and potential processing method. Oxide material within resource pit shell = 0.099% Tsol; enriched material within resource pit shell = 0.092% Tsol; primary material within resource pit shell = 0.226% CuT; oxide underground material outside resource pit shell = 0.549% Tsol; enriched underground material outside resource pit shell = 0.522% Tsol; primary underground material outside resource pit shell = 0.691% CuT.
8. For Parks/Salyer: Variable cut-off grades were reported depending on material type, associated potential processing method, and applicable royalties. For ASCU properties - Oxide underground material = 0.549% Tsol; enriched underground material = 0.522% Tsol; primary underground material = 0.691% CuT. For state land property - Oxide underground material = 0.545% Tsol; enriched underground material = 0.518% Tsol; primary underground material = 0.686% CuT. For MainSpring (Parks/Salyer South) properties - Oxide underground material = 0.532% Tsol; enriched underground material = 0.505% Tsol; primary underground material = 0.669% CuT.
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. Totals 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: Orthogonal View of Cactus East and West with Parks/Salyer**




Source: ASCU, 1992.

### 1.14.1 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. The data were analyzed according to material type, potential mining, and potential processing methods. Changing royalty rates on sections of the Project had a variable effect on cutoff grades as well. Histogram and log normal cumulative probability plots were reviewed for CuT assays and Tsol results in each of the mineral zones in the Cactus Project resource. The results of these analysis for both the Cactus and Parks/Salyer are presented in Table 1-3.

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

| Capping Grades | | | | |
|---|---|---|---|---|
| | Leached | Oxide | Enriched | Primary |
| TCu | 0.20 | 2.50 | 3.80 | 1.20 |
| ASCu | 0.04 | 1.80 | 0.50 | 0.03 |
| CuCN | 0.05 | 0.50 | 3.30 | 0.15 |
| Tsol | 0.09 | 2.30 | 3.80 | 0.08 |
| Mo | 0.20 | 0.05 | 0.08 | 0.07 |

For the Stockpile Project, histogram and log normal cumulative probability plots were reviewed for CuT, CuAS, CuCN, and Tsol assays. Cutoffs were defined within individual Stockpile Project lifts and ranged between 0.43% to 0.65% for CuT, 0.33% to 0.50% for CuAS, 0.10% to 0.29% for CuCN, 0.40% to 0.59 for Tsol, and 0.40 to 1.68 for Ca.

### 1.14.2 Resource Cutoff Grades (CoGs)

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

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 sulphides in the primary material was based on CuT assays. High- level cost analysis for the open pit suggested CoGs of 0.099% Tsol for the oxides, and 0.092% Tsol for the enriched material. A cutoff of 0.226% CuT was applied to primary material mined and therefore stockpiled for potential recovery in the future using a sulphide recovery process. Whittle open pit optimization software was applied 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 0.549% Tsol for the oxides and 0.522% Tsol for the enriched. The primary had a 0.691% cutoff applied to the CuT grade for potential recovery in a future sulphide recovery process.

Sections of the Parks/Salyer deposit are subject to variable royalty charges, this leads to slightly variant cutoff grades. Mineral resources for Parks/Salyer were also determined based on its amenability to underground mining. ASCU used a US$3.75/lb Cu price to determine the cutoff grades for the 2024 resource statement. High-level analysis of the material on ASCU property yielded Tsol cutoffs of 0.549% Tsol for the oxides and 0.522% Tsol for the enriched. The primary had a 0.691% cutoff applied to the CuT grade for potential recovery in a flotation mill. For the State Land property, the Tsol cutoffs were 0.545% and 0.518 for oxide and enriched material, respectively. Primary material had a cutoff of 0.686% CuT. Parks/Salyer south (Mainspring) property had Tsol cutoffs of 0.532% and 0.505% for the oxide and enriched material, respectively. The primary material had a cutoff of 0.669% CuT.

The Stockpile Project resources were defined using a CoG of 0.095% Tsol.

## 1.15 Mineral Reserve Estimate

The Mineral Reserve estimates for ASCU's Cactus Mine Project are based on the conversion of the Measured and Indicated Mineral Resources within the current mine plan. Inferred Mineral Resources were treated as waste. The mineral reserves are estimated from two surface mining sources (Cactus West and Historic Stockpile) and two underground sublevel caving mines (Cactus East and Parks/Salyer).

Inputs to the Open Pit Estimate include:

- Open pit slope recommendations for the Cactus West Pit provided by Call & Nicholas Inc. (CNI).
- Ultimate pit designs are based on pit shells generated using the Lerchs-Grossman method in Datamine's Studio NPVS software. The Cactus West ultimate pit design is based on the $2.90/lb Cu price shell with $3.70/lb Cu used for the base price economics.
- CoG decisions for Cactus West and Historic Stockpile are based on a block value calculation in the mine schedule, which is effectively a net-smelter return with expected processing, general and administrative (G&A), and royalty

cost removed. The cutoff block value employed was a marginal CoG of $0/t, meaning that any block which would generate a net positive value was either processed on the heap leach or placed into stockpiles.

- No dilution or ore loss is applied to the Cactus West and the Stockpile Mining.
- The Cactus West pit is mined in two phases while the stockpile is mined in a sequence from east to south to west in order to facilitate construction of the heap leach facility while mining the higher value portions.

The estimates assume conventional open pit mining and equipment.

Inputs to the underground sublevel caving (SLC) estimate include:

- Geotechnical design parameters for the SLC mining of the Cactus East and Parks/Salyer underground deposits has been provided by CNI.
- The underground reserves for the SLC mines account for the mixing of Indicated resources with dilution from low-grade and barren material originating from within the sublevel cave outline and from overlying material.
- The block value calculation (CFTC1) in the mine schedule, which is effectively a net smelter return (NSR) with expected processing, general arrangements (G/A), and royalty cost removed was based on a US$3.70/lb Cu price.

SLC level footprints were designed to a shut-off dollar value (CFTC1) of US$27.62; however, a minor quantity of subeconomic material was incorporated into the upper levels in Cactus East to establish mineable shapes for SLC mining and to accommodate the shallow plunge of the orebody. Drawpoints were shut-off when the grade value fell below a CFTC1 of US$27.62 following the necessary removal of swell material within the footprint regardless of grade.

Sub-level caving is a non-selective bulk mining method where subeconomic material (dilution) is mixed into the flow modelling process and accepted to recover the blasted ore rings. The Power Geotechnical Cellular Automata (PGCA) software program was used to simulate cave flow behaviour. Dilution from production activities is quantified through cave flow modelling and is included in the reported Ore Reserves The overall total dilution (internal and external) for SLC rings valued > US$27.62 is estimated to be 11.1% for Parks/Salyer and 11.2% for Cactus East. Inferred resources included in the mixing process have been assigned zero grade.

## 1.16 Mineral Reserve Statement

The mineral reserve estimates for the ASCU Cactus Mine Project were prepared in accordance with the guidelines of NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum definition Standards for Mineral Resources and Mineral Reserves ("CIM Standards").

The mineral reserve estimates are based on the conversion of Measured and Indicated resources to Proven and Probable reserves from two surface mining sources (Cactus West and Historic Stockpile) and two underground sublevel caving mines (Cactus East and Parks/Salyer). Mineral reserves are reported from engineered mine designs and the life of mine (LOM) plan.

The total reserves for the ASCU Cactus Mine Project are shown in Table 1-4.

**Table 1-4: Cactus Mine Project Reserves Statement**

| Metal | Unit | Cactus West Open Pit | Stockpile Open Pit | Cactus East Underground | Parks/Salyer Underground | Total |
|---|---|---|---|---|---|---|
| Proven | Tons | 3,600,000 | - | - | - | 3,600,000 |
| | CuT (%) | 0.249 | - | - | - | 0.249 |
| | CuAS (%) | 0.052 | - | - | - | 0.052 |
| | CuCN (%) | 0.173 | - | - | - | 0.173 |
| | Cu (M lbs) | 17.9 | - | - | - | 17.9 |
| Probable | Tons | 71,921,000 | 76,777,000 | 27,739,000 | 96,248,000 | 272,686,000 |
| | CuT (%) | 0.310 | 0.163 | 0.950 | 0.930 | 0.552 |
| | CuAS (%) | 0.138 | 0.112 | 0.333 | 0.110 | 0.141 |
| | CuCN (%) | 0.122 | 0.024 | 0.552 | 0.710 | 0.346 |
| | Cu (M lbs) | 445.4 | 251.0 | 527.0 | 1,789.7 | 3,013.0 |
| Proven + Probable | Tons | 75,521,000 | 76,777,000 | 27,739,000 | 96,248,000 | 276,286,000 |
| | CuT (%) | 0.307 | 0.163 | 0.950 | 0.930 | 0.549 |
| | CuAS (%) | 0.134 | 0.112 | 0.333 | 0.110 | 0.140 |
| | CuCN (%) | 0.125 | 0.024 | 0.552 | 0.710 | 0.344 |
| | Cu (M lbs) | 463.3 | 251.0 | 527.0 | 1,789.7 | 3,031.0 |

Note:

1. Mineral Reserves have an effective date of November 10, 2023. The Qualified Person for the underground estimates of Cactus East and Parks/Salyer is Nat Burgio of AGP Mining Consultants Inc. The Qualified Person for the open pit estimates of Cactus West and Stockpile is Gordon Zurowski of AGP Mining Consultants Inc.
2. The Mineral Reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves.
3. The Mineral Reserves are supported by a combined open pit and underground mine plan, based on open pit and underground designs and schedules, guided by relevant optimization procedures.
   Inputs to that process are:
   - Metal prices of Cu $3.70/lb.
   - Processing costs which are variable and based upon material type, processing destination, copper grade, and copper recovery., Processing costs include a fixed unit cost component, a net acid consumption cost, and a cost for refining and selling copper cathode.
   - General and administration cost of $0.47/ton processed.
   - Royalty cost of 2.5% for Parks/Salyer and 2.54% for Cactus/Stockpile Ores.
   - Process recoveries which are variable depending upon mineralization type, sequential copper grades, and comminution size.
   - Open pit geotechnical design criteria from Call and Nicholas, Underground geotechnical design criteria from Call and Nicholas, Open pit mining costs including an escalation factor with pit depth.
   - Underground mining cost of $27.62.
4. The footprint delineations for the Cactus East and Park Slayer mines were based on a resource model block cash flow dollar value (CFTC1) of $27.62 (net of process, G/A and royalties). Drawpoints were shut-off when the grade value fell below a CFTC1 of $27.62 following the necessary removal of swell material within the footprint.
5. Dilution and mining loss adjustments are incorporated into the underground mining inventories by way of cave flow modelling software. Inferred resources included in the mixing process have been assigned zero grade. No allowance for mining dilution or ore loss has been provided in the open pit mining inventories.

## 1.17 Mining Methods

### 1.17.1 Open Pit Mining Methods

The Cactus West orebodies lie adjacent to and beneath the historically mined Cactus (Sacaton) Pit while the Historic Stockpile is located to the South of the existing pit and proposed Cactus West pit expansion. The Stockpile mining area is a historical waste dump which contains significant quantities of oxide copper mineralization. This material was considered waste in the historical operation because the sole processing method on site was a flotation mill which could not recover oxide copper mineralization.

Ore processing in the mine schedules involves all ore material types from Cactus West and the historic Stockpile being processed on a heap leach after multi-stage crushing. Waste from Cactus West and the Stockpile area will be placed into multiple locations, but primarily to the north-east of Cactus West where a view shed berm and waste dump are planned.

Open pit designs were completed in Hexagon's MinePlan software according to geotechnical design parameters provided by Call and Nicholas, with design assumptions for road and minimum mining widths provided by AGP. Cactus West consists of two phases that will be mined on 20 ft (6 m) benches. Triple benches 60 ft (18 m) high are planned with a catch bench of 27 ft (8 m) in all pit areas. The slope design assumes that controlled blasting will be implemented, and horizontal depressurization drains installed to achieve the recommended slope parameters.

The historic stockpile was divided into three phases for mining: the east phase, south phase, and west phase. Mining starts in the east phase followed by south phase with the west phase mined last. This stockpile mining sequence was chosen due to higher average grades mined upfront and to make room for construction of the heap leach pad. Mining the stockpile in this order is required to ensure space is available for leach pad construction.

Waste materials generated from mining Cactus West and the Stockpile areas will be composed of predominantly Gila Conglomerate and Alluvium overburden (80%) with the remainder being granite and porphyry rock with lower copper grades, or unfavourable metallurgical characteristics (including 4 Mton of waste bearing primary copper mineralization).

No waste segregation is required in the mine schedule aside from the stockpiling of primary copper mineralization, and as such different waste types can be placed into any of the available waste facilities as required by scheduling and fleet optimization constraints.

Primary production drilling will be completed with six down the hole (DTH) hammer drills using 6 ¾ in bits. This will provide the capability to drill patterns for either 20 ft (6 m) or 40 ft (12m) bench heights. Two smaller drills using 5 ½ in bits will be utilized to perform wall control drilling in the form of buffer patterns.

Production mining will be completed with two 30-$yd^3$ hydraulic shovel, five 15-$yd^3$ loaders, and twenty-four 150-ton rigid body trucks. It is expected that the larger hydraulic shovels will be utilized in the Cactus West Pit, while the front-end loaders will support mining in the Stockpile area and supplement in Cactus West. Grade control assaying will be performed using cuttings from production blastholes.

### 1.17.2 Underground Mining Methods

As part of the initial phase of the Pre-Feasibility Study, AGP undertook a high-level review of underground mining options which included sublevel open stoping, room and pillar, inclined caving, block caving and the sublevel caving (SLC) method.

The small size of the Cactus East deposit, low angle plunge of the mineralisation and sharp hangingwall and footwall contacts restricted the economic potential for the block caving option. While the mineralization at Parks/Salyer is much larger, the geotechnical conditions were not considered favourable for the complex development geometries required for the development of an extraction level for the block cave option. CNI were of the view that draw point spacings required to be marginally stable would result in relatively poor recovery and high dilution due to the expected fine fragmentation. SLC was, therefore selected as the preferred underground mining method for both the Cactus East and Parks/Salyer deposits based on geotechnical conditions and orebody geometries.

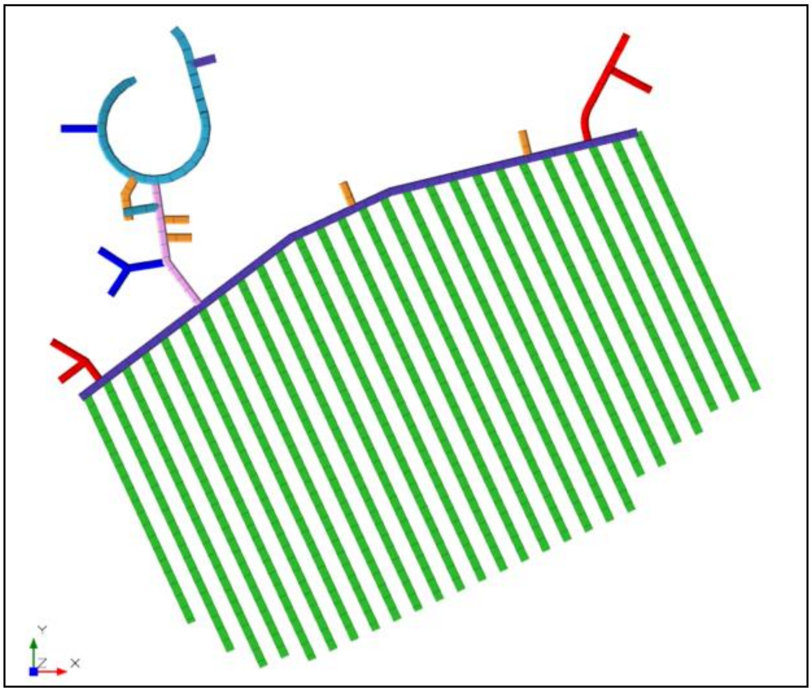
The initial Cactus East SLC level will commence 1,325 ft (404 m) below the surface and will be comprised of 7 sublevels to a final depth 1,845 ft (562 m) below surface. Access will be via a single decline with a portal located within the existing Cactus West pit. Ore haulage to surface will be via a vertical conveyor which can be supplemented with truck haulage to surface via the open pit if necessary. Production will continue for 11 years and will peak at 3.9 Mt/y.

Each level has been designed for the SLC cave front to retreat to the decline and the intra-level infrastructure (Figure 1-3). Locating infrastructure in this position is designed to minimise cave induced damage as the cave propagates and stresses redistribute into the surrounding rock mass.

Parks/Salyer has a similar configuration as Cactus East Mine which also utilises sub-level caving mining method to extract the ore from the underground. The initial Parks/Salyer SLC level will commence 1,120 ft (341 m) below surface and includes 11 sublevels to a final depth of 1,930 ft (588 m) below surface. Access to the Parks/Salyer deposit will be via a surface portal and twin declines. One of the declines will be dedicated to ore haulage using an inclined conveyor while the other providing access for personnel and equipment. Production will continue for 19 years and will peak at 6.9 Mt/y.

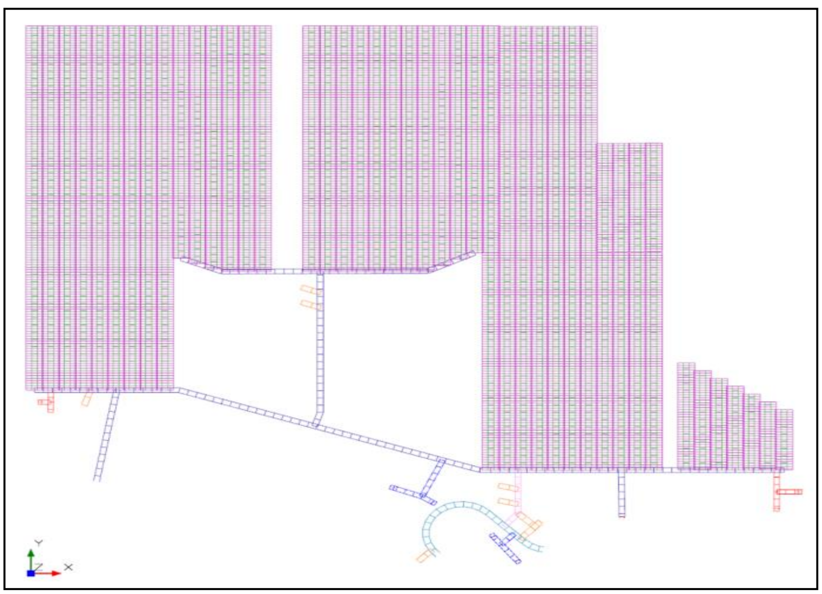
The Parks/Salyer orebody will be one of the largest footprints to be developed using the SLC mining method. Provisions have been made in the mining schedule to limit the maximum mining spans to 800 feet separated by “transition zones” where no sublevel drifts are developed. Figure 1-4 presents a lay-out of the transition zones and SLC mining panels. More detailed geotechnical analysis is required to assess the rock mass response due to the signficant step-out distances required between sublevels.

**Figure 1-3: Cactus East SLC Layout**



Source: AGP, 2023.

**Figure 1-4: Parks/Salyer SLC Layout**



Source: AGP, 2023.

SLC production crosscuts have primarily been designed so that each level is horizontally offset from the level above and below. The design parameters for the SLC production drives at Cactus East and Parks/Salyer are in line with other SLC operations.

The amount of ore to be extracted will be limited in the upper three production levels to the following proportions.

- First Level ~40% (swell only)
- Second Level ~60%
- Third level ~100%
- Lower levels >100% to shutoff grades or dollar values.

The production strategy will help control cave ability, minimise the formation of air gaps and create a blasted ore blanket above the production levels to minimise early dilution entry from the overburden rocks. These restricted draw rates also apply to areas where large step-outs distances are required from one sublevel to the next.

The Cactus East Ore/Waste Handling System consists of a crusher station and a 1,600 ft (488 m) vertical conveyor with a capacity of 630 tons/h that will convey ore from the top of the orebody to surface via a vertical raise feeding an overland conveyor. Ore will be hauled by 55-ton diesel trucks to a sizer located adjacent to the bottom of the vertical conveyor. Ore will be crushed to a maximum 6-in dimension. A short conveyor from the sizer will feed the vertical conveyor. Waste will be trucked to the portal for disposal within the Cactus West open pit.

The mine plan for Parks/Salyer consists of two ramps with one dedicated for material handling. The ore/waste handling system consists of a series of initially four, extending to five switchback conveyors and two crushing sizers on -270 L, one of which will subsequently be relocated at the -470 L. These will deliver material from the mine working levels to the surface portal, from where materials will then be transported on surface via an overland conveyor.

Ventilation is driven by a fresh air drive developed from the access drive, in which the fresh air will be splitting right and left to connect to the return air drives at the extremities of the footprint. This allows natural flow of ventilation through the entire footprint.

### 1.17.3 Mine Plan

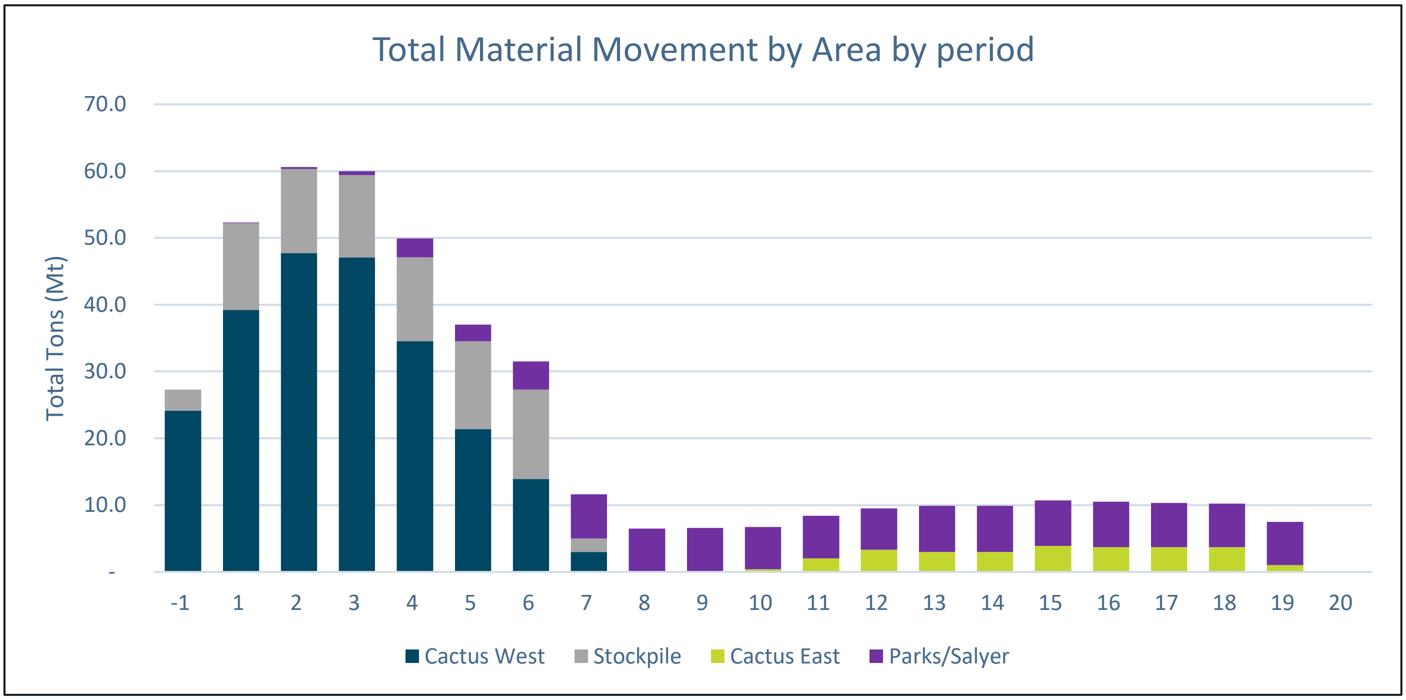
The Cactus Mine plan includes production from four separate mining areas: Cactus West Open Pit, Historical Stockpile, Cactus East Underground, and Parks/Salyer Underground. The mine production schedule is initially focused on the surface sources of ore along with Parks/Salyer underground that starts development in Year 1. The Cactus East deposit is developed later in the mine life, starting in Year 9. The Cactus West and Historic Stockpile ore sources are depleted in Year 7 after which the ore stream becomes exclusively underground. Scheduled material movement by period from each mining area is shown in Figure 1-5.

The Cactus West mine life includes one year of pre-stripping and seven years of mining. Phase 1 starts with 24 Mton of pre-production stripping and is completed in Year 4. Phase 2 mining begins in Year 2 and is mined out in Year 6. Target ore production is 12 Mton per annum with a peak mining rate of 47 Mton in Years 2 and 3. A total of 75.5 Mt of leach ore grading 0.307% total copper is mined at a strip ratio of 1.9 to 1. Bench elevations at Cactus West range from the 1,440-ft level to the 380-ft level.

Over the course of the open pit mine schedule, approximately 13.1 Mton of low-grade ore is stockpiled and reclaimed in order to smooth the ore release from the open pits. This amount includes approximately 2.4 Mton of material stockpiled in the first three years of mining, and then processed in Year 2 and 3, and another 10 Mton stockpiled later in the mine schedule before being reclaimed in Years 6 and 7.

Historic Stockpile mining begins near the end of the pre-production year with approximately 3.0 Mton of ore sent to the leach pad. Mining continues concurrently with the Cactus West pit into Year 7 at an annual ore production rate of 12 Mton. A total of 76.8 Mton of leach ore at 0.163% total copper is mined. A small amount, 5.5 Mton of waste is mined from the historic stockpile and sent to the waste storage areas.

**Figure 1-5: Life of Mine Material Movement by Mining Area**




Source: AGP, 2023.

## 1.18 Recovery Methods

Material mined from the existing stockpile will be placed in 20-ft lifts and material from all other sources will be stacked in 30-ft lifts. Material will be reclaimed and transferred by haul truck to the crushing circuit where it will be crushed down to $P_{80}$ minus ¾-in. From the crushing circuit, the material will transfer by overland conveyor to the agglomeration drums, mobile transfer conveyors, and mobile radial stacker to be placed on the lined heap leach pad facility.

Leaching solutions, containing dilute sulfuric acid will be pumped and applied to the top of each lift and allowed to percolate though the copper leach material. Copper is dissolved into the solution while acid is consumed at approximately 6.5 lbs/ton of material leached. Acid consumption is net of regenerated acid in the SX/EW process and varies year over year from 220,000 tons of 94.5% sulphuric acid to net acid generating after year 7 when higher grades are mined from Cactus and Parks Salyer. The height of the leach material on the pad will eventually reach approximately 180 ft (55 m) in overall height.

The pregnant leach solution from the heap leach ponds will be pumped for processing in a copper SX/EW plant capable of producing initially up to 30,000 ton/y of copper cathodes with a design PLS flow of up to 12,000 gpm and grade at approximately 3.0 g/L Cu based on an overall 71% CuT recovery from the heap leaching methods for the resources considered. The solvent extraction plant is designed to be operated in a series, parallel, or series-parallel configurations with a single stage of stripping. The optionality of the solvent extraction plant will allow the plant to operate at 4,000 gpm, 8,000 gpm, or 12,000 gpm PLS flowrates based on the variability in copper grades and tonnages in the mine plan.

The electrowinning circuit capacity will be expanded in Year 3, doubling in size to the overall plant capacity required to a nominal 60,000 ton/y of copper cathodes.

## 1.19 Infrastructure

### 1.19.1 Project Infrastructure

The property is accessed from west Maricopa Casa Grande Highway that links Casa Grande and Maricopa, Arizona. Bianco road (the primary access road) currently extends north to an existing building known as Truestone facility. The primary access road is a paved road which will be repaired and upgraded with an additional asphalt layer to ensure suitability for daily operational traffic.

Existing unpaved maintenance roads originating from the primary access road will be repaired to ensure suitable light vehicle traffic and additional maintenance roads to connect explosive storage and water wells to existing unpaved roads will be constructed.

The project consists of the necessary infrastructure to support the mining and processing operations. All infrastructure buildings and structures will be built and constructed to all applicable codes and regulations. The existing Truestone facility will be used for maintenance and a warehouse. The project site will also include an administration building, plant maintenance shop and warehouse, truck shop, fuel storage, explosive storage, and other buildings.

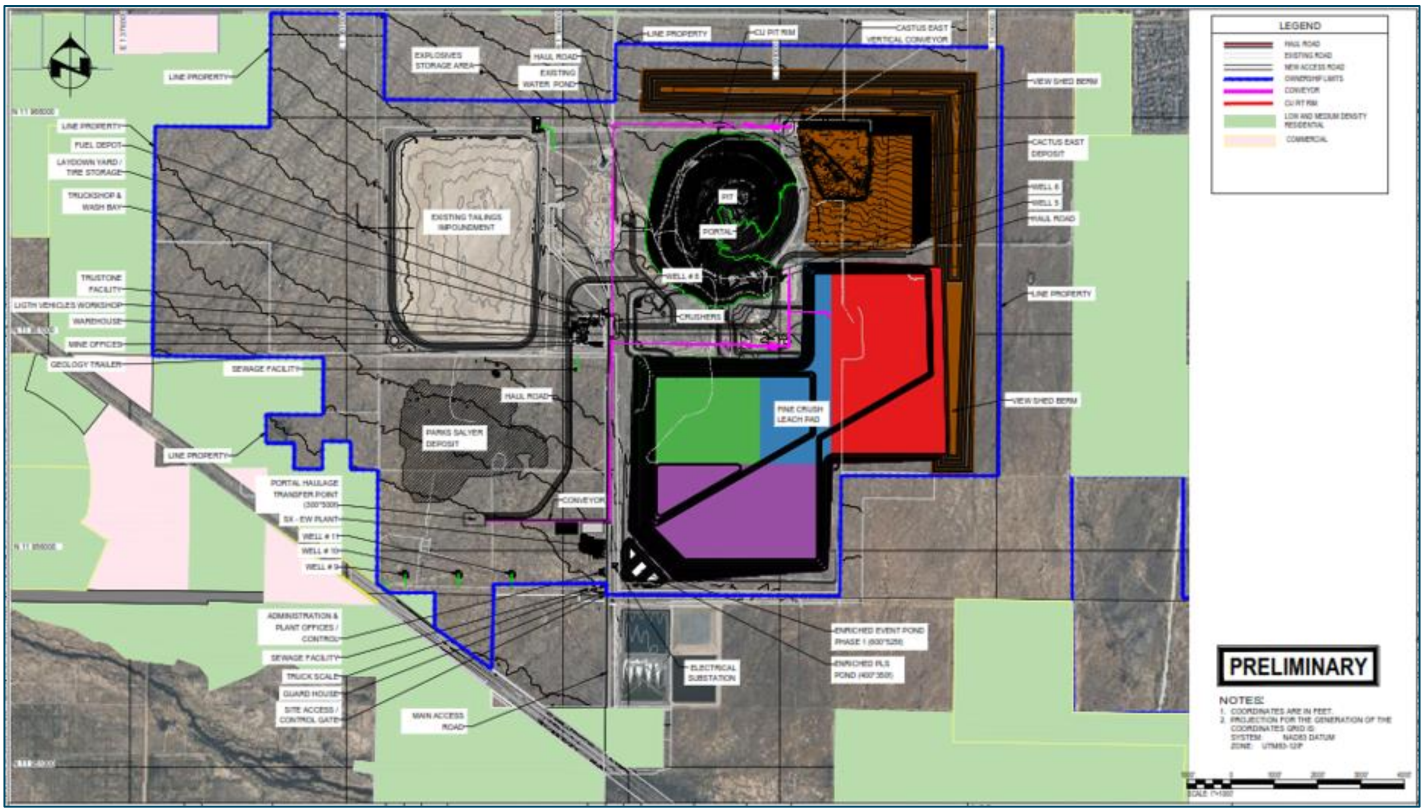
The Project will require the following facilities:

- Mining facilities including administration offices, change house, truck shop, explosives storage, fuel storage and distribution, ore stockpiles, waste stockpiles, and truck wash.
- Process facilities including the SX/EW process plant, crushing facilities, process plant workshop, change house, assay laboratory, and freshwater infrastructure.
- Heap leach pads and ponds.

- Power supply and distribution.
- Water supply for make-up
- General facilities including a gatehouse, administration building, and weighing scale.
- Catchments, ponds, water wells, drainage, and other site water management infrastructure.

An overall site layout is provided in Figure 1-6.

**Figure 1-6: Overall Site Layout**




Source: Ausenco, 2024.

### 1.19.2 Heap Leach Facility (HLF)

The principal objective of the HLF design is to efficiently extract copper by leaching metals within a geotechnically stable facility. The anticipated ore production will be approximately 55,000 ton/d for the first seven years and reduced to 17,500 ton/d after that for the 26-year life-of-mine (LOM). In Phase 1, the pad will be loaded with agglomerated ore on conveyor belts coming in from the west along the northern side of the pad to discharge to the eastern area of the pad. This area provides a relatively flat area that facilitates the construction of the first phase of the pad and allows for mining of the existing stockpile to liberate space for the consecutive phases of construction.

## 1.20 Market Studies and Contracts

Project economics were estimated based on long-term flat metal prices of US$3.90/lb Cu. This copper price is in accordance with consensus market forecasts from various financial institutions and is consistent with historic prices for this commodity.

No market studies or product valuations were completed as part of the 2024 PFS. Market price assumptions were based on a review of public information, industry consensus, standard practices, and specific information from comparable operations in the region.

## 1.21 Environmental, Permitting and Social Considerations

The Project includes legacy environmental issues related to the former ASARCO Sacaton operations that have been addressed by Arizona Department of Environmental Quality (ADEQ) as part of the ASARCO bankruptcy settlement with the state. ADEQ, through a prospective Purchaser Agreement, has released ASCU from any potential liability associated with the legacy environmental issues at the site. Permitting is limited to State of Arizona-required permits including the Aquifer Protection Permit and the Mined Land Reclamation Permit which ASCU has received from state regulators. Modifications of each will be required to address changes in the mine plan presented in this PFS.

ASCU has a well-developed community engagement plan that it has implemented through numerous public meetings and outreach. With the presence of legacy mining in the Casa Grande area, the local community is supportive of this project. There is no significant opposition to the Cactus and Parks/Salyer Project.

### 1.21.1 Environmental Considerations

In 2009, approximately 15 years after the Cactus Mine ceased operation, the mine was conveyed to the ASARCO Multi-State Environmental Custodial Trust (the Trust) as part of ASARCO bankruptcy proceedings. The Trust entered the property into the Voluntary Remediation Program (VRP) with Arizona Department of Environmental Quality in 2010. In the following years, structures were demolished and reclaimed, and characterization studies were conducted. Based on the results of the characterization studies and reclamation work, in August 2019, Elim entered into a Prospective Purchaser Agreement (PPA) with ADEQ. The PPA, which ADEQ issued because of the substantial public benefit to the remedial work conducted at the site, released Elim from potential liabilities related to existing, known contamination under CERCLA, WQARF, and RCRA. The PPA does not cover unidentified environmental conditions or contamination.

### 1.21.2 Closure and Reclamation Considerations

A Mined Land Reclamation Plan was completed and submitted to the Arizona State Mine Inspector's (ASMI) office in January 2023 (Parsons, 2022). The MLRP was approved by the ASMI on March 27, 2023, and the surety bond has been posted.

The Project includes exploration and mining on private land and on two Arizona State Land Department (ASLD) leases. There is no federal nexus for permitting the project.

The primary permit with the longest permitting timeframe is anticipated to be the Aquifer Protection Permit Amendment (APP). ASCU currently has an APP (no. P-513324) for the following facilities: oxide leach pad, enriched leach pad, oxide PLS pond, enriched PLS pond, raffinate pond, oxide events pond, enriched events pond, site runoff pond 1, site runoff pond 2, and the waste rock stockpile runoff pond. ASCU will apply for amendments to the APP for additional discharging facilities, as needed. An APP Significant Amendment (without a public hearing) has a licensing timeframe of 221 business days.

#### 1.21.3 Social Considerations

In keeping with ASCU's community engagement and partnership standards, the Project will be developed with a plan to establish and maintain the support of our host communities. ASCU commenced community outreach at the earliest stages of the Project and is currently evaluating and building partnerships within the community. As the Project's permits will involve a public process and are based on the permit submission and review schedule, ASCU understands the importance of outreach during the permitting process and throughout the life of the mine. ASCU is encouraged by the positive response to the project from the community. Its status as a "brownfields" project makes it potentially more appealing than a new mine might be.

### 1.22 Capital and Operating Cost Estimates

#### 1.22.1 Capital Cost Estimate

The capital cost estimates for this PFS were developed with a -20% to +30% accuracy and an estimated contingency of approximately 15%according to the Association of the Advancement of Cost Engineering International (AACE) Class 4 estimate requirements. The estimates include the cost to complete the design, engineering, procurement, construction, and commissioning of all process plant facilities.

The facilities at the mine site will consist of an open pit, underground mining operation, SX/EW process plant, conveying, crushing, and screening equipment, site sub-station, site power distribution, access roads, heap leach facilities and associated infrastructure.

ASCU has engaged third-party consultants to contribute to the total project scope of work and overall capital cost estimate. On behalf of ASCU, Ausenco incorporated the third-party contributions into an overall Prefeasibility study cost estimate.

All third-party contributors are accountable for the development and quality of their cost estimates, which will be inclusive of all direct costs, growth allowances, project indirect costs, and associated contingency within their scope of work, but separately identified. Each align with the overall project WBS numbering system.

The total initial capital cost for the Cactus Project is US$515M and the LOM sustaining cost including financing is US$1,221M.

Table 1-5 provides a summary of the capital costs for the Project.

**Table 1-5: Total Project Costs Summary**

| Capitalized Costs | Initial ($M) | Sustaining ($M) |
|---|---|---|
| **Mining and Processing** | **174** | **905** |
| Processing | 4 | 0 |
| Mining (Pre-Stripping) | 78 | 0 |
| Mining – Open Pit – Cactus West | 24 | 20 |
| Mining – Underground – Cactus East | 0 | 341 |
| Mining – Underground – Parks/Salyer | 57 | 544 |
| Mining – Underground – Combined/Shared | 11 | 0 |
| **Other** | **342** | **315** |
| Infrastructure | 56 | 0.3 |
| Crushing And Conveying | 29 | 6 |
| Leaching & Waste Rock Storage | 66 | 126 |
| Solvent Extraction (SX) | 30 | 0 |
| Electrowinning (EW) | 26 | 14 |
| Reagents | 1 | 0 |
| Process Plant Services and Utilities | 4 | 0 |
| Project Execution | 54 | 8 |
| Provisions | 75 | 160 |
| **PROJECT TOTAL** | **515** | **1,221** |

Estimated closure requirements inclusive of all necessary demolition, rehabilitation, revegetation, earth grading/contouring, scrap metal disposal/tipping fees, as well as post-closure monitoring. The total closure cost was calculated to be US$23M, with salvage credits of US$97M.

### 1.22.2 Operating Cost Estimate

The project OPEX estimate encompasses mine operating costs, process plant operating costs, and general and administrative (G&A) costs. Cash costs are expressed in dollars per short ton ($/t) of heap feed or dollars per pound of ($/lb) cathode produced. Total cash costs encompass royalties, refining charges, and transportation charges. Additionally, the All-In Sustaining Costs (AISC) and the All-In Costs (AIC) incorporate non-sustaining Capex, closure, and reclamation CAPEX, respectively. A summary of these costs is presented in Table 1-6, with further details provided in Section 21.

**Table 1-6: Operating Cost, AISC and AIC Summary**

| Total Production Cost Item | LOM | | |
|---|---|---|---|
| | ($/t Placed) | ($/lb Cathode Produced) | ($M) |
| Mining | 11.51 | 1.38 | 3,180 |
| Processing | 2.93 | 0.35 | 809 |
| Infrastructure | 0.03 | 0.00 | 8 |
| G&A | 0.12 | 0.01 | 32 |
| **Cash Cost** | **14.58** | **1.75** | **4,029** |
| Royalties | 0.79 | 0.09 | 218 |
| Refining and Transportation | 0.00 | 0.00 | |
| **Total Cash Cost** | **15.37** | **1.84** | |
| Sustaining CAPEX | 4.42 | 0.53 | 1,221 |
| Reclamation and Closure | 0.08 | 0.01 | 23 |
| Salvage | 0.35 | 0.04 | 97 |
| **All-In Sustaining Costs** | **19.52** | **2.34** | |
| Property Taxes | 0.69 | 0.08 | 191 |
| Initial (non-sustaining) CAPEX | 1.86 | 0.22 | 515 |
| **All-In Costs** | **22.08** | **2.64** | |

## 1.23 Economic Analysis

### 1.23.1 Economic Summary

The economic analysis was performed assuming an 8% discount rate. On a post-tax basis, the $NPV_{8\%}$ is US$508.7M, the internal rate of return (IRR) is 15.3%, and the payback period is 6.8 years. A summary of project economics is tabulated in Table 1-7.

**Table 1-7: Economic Analysis Table Summary**

| General | Units | LOM Total / Avg. | |
|---|---|---|---|
| Copper Price | US$/lb | 3.90 | |
| Mine Life | Years | 21.0 | |
| Total Mineralized Material Processed | Kt | 276,286 | |
| Total Waste | Kt | 147,841 | |
| Avg. CuAS Head Grade | % | 0.14 | |
| Avg. CuCN Head Grade | % | 0.34 | |
| Avg. Acid Consumption | lb/t | 18.99 | |
| **Production** | **Units** | **LOM Total / Avg.** | |
| Avg. Head Grade – CuAS | % | 0.14 | |
| Avg. Head Grade – CuCN | % | 0.34 | |
| Avg. Acid Consumption | lb/t | 18.99 | |
| Avg. Recovery Rate – CuAS | % | 90.8 | |
| Avg. Recovery Rate – CuCN | % | 84.5 | |
| Total Payable Copper | M lb | 2,306 | |
| Annual Payable Copper | M lb/y | 110 | |
| **Operating Costs** | **Units** | **LOM Total / Avg.** | |
| Mining Cost | US$/t mined | 7.50 | |
| Mining Cost | US$/t processed | 11.51 | |
| Processing Cost | US$/t processed | 2.96 | |
| G&A Cost | US$/t processed | 0.12 | |
| Operating Cash Costs* | US$/lb Cu | 1.75 | |
| C1 Cash Costs** | US$/lb Cu | 1.84 | |
| C3 Cash Costs (AISC)*** | US$/lb Cu | 2.34 | |
| **Capital Costs** | **Units** | **LOM Total / Avg.** | |
| Initial Capital (Incl. Capitalized Opex) | US$M | 515 | |
| Sustaining Capital | US$M | 1,221 | |
| Closure Costs | US$M | 23 | |
| Salvage Value | US$M | 97 | |
| **Financials** | **Units** | **Pre-Tax** | **Post-Tax** |
| NPV (8%) | US$M | 733.3 | 508.7 |
| IRR | % | 17.7 | 15.3 |
| Payback | Years | 6.3 | 6.8 |

*Operating cash costs consist of mining costs, processing costs, and G&A.
**Total cash coasts consist of operating cash costs plus transportation cost, royalties, treatment, and refinancing.
***AISC consist of total cash costs plus sustaining capital, closure cost, and salvage value.

### 1.23.2 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV8% and IRR of the Project using the following variables: metal price, discount rate, total operating cost, and initial capital cost.

The sensitivity analysis revealed that the Project is most sensitive to commodity price, head grade, and operating cost and less sensitive to initial capital cost.

## 1.24 Adjacent Properties

The nearest adjacent mineral property is the Santa Cruz copper porphyry deposit, located just over 2 mi (3 km) southeast of the Cactus site and 7 mi (11 km) west of Casa Grande, Arizona. Deposit information obtained from an abstract of the Geology of the Santa Cruz Porphyry Copper Deposit by Henry G. Keis (2020), ASARCO, Incorporated, Tucson, Arizona, reports that the associated alteration and mineralization in the Santa Cruz copper porphyry, including that of fault-displaced portions like the Cactus Project, spans about 7 mi (11 km) in length and about a mile (1.6 km) in width. Ivanhoe Electric is currently developing the Santa Cruz property and has published NI 43-101 and S-K 1300 Technical Reports within the past year. A review of those documents has revealed that Santa Cruz had mineralogical properties and structural aspects very similar to Cactus and Parks/Salyer.

There are currently two operating copper mines in Pinal County. These mines are the Florence Copper Mine, owned and operated by Taseko Mines Ltd., situated approximately 25 mi (40 km) ENE of Cactus Mine. Additionally, the Ray Mine, owned and operated by ASARCO LLC, a subsidiary to Grupo Mexico, is located approximately 50 mi ENE of the mine site.

## 1.25 Conclusions and Interpretations

The total measured and indicated mineral resource estimate for the Cactus Mine Project is 445.7 Mton of combined leachable and primary mineralogies, averaging 0.58% copper for a total of 5.2 billion lbs of copper.

The total proven and probable mineral reserve estimate for the Cactus Mine Project is 276.3 Mton, grading 0.48% copper for a total of 3.0 billion lbs of copper.

Based on the assumptions and parameters in this report, the pre-feasibility study shows positive economics (i.e. post tax NPV of $509M and 15.3% post-tax IRR). This PFS supports a decision to carry out additional detailed studies.

## 1.26 Recommendations

Table 1-8 provides a summary of all major recommended works proposed to be completed in support of further engineering studies.

The recommended budget totals $26.32M and the scope for all work listed below is summarized in Section 26.

**Table 1-8: Summary of Budget for Recommendations**

| Budget Item | ($M) |
|---|---|
| Exploration and Drilling | 20.0 |
| Metallurgy and Process Design | 0.1 |
| Metallurgical Testwork | 0.9 |
| Mineral Resource Estimates | 0.05 |
| Mineral Reserve Estimates | 0.1 |
| Open Pit Mine Design and Scheduling | 0.3 |
| Underground Mine Design and Scheduling | 0.8 |
| Mine Capital and Operating Cost Estimation | 0.1 |
| Geotechnical | 1.5 |
| Recovery Methods | 1.0 |
| Roads and Logistics | 0.07 |
| Heap Leach Facility | 0.4 |
| Environmental, Permitting, and Social Recommendations | 1.0 |
| **Total** | **26.32** |

Note: Numbers may not add due to rounding.

# 2 INTRODUCTION

## 2.1 Introduction

Ausenco Engineering USA South Inc. and Ausenco Sustainability ULC (collectively Ausenco) has compiled this Prefeasibility study (PFS) and associated technical report for Arizona Sonoran Copper Company (ASCU) for the Cactus Mine Project (the "Project") located in Casa Grande, Arizona. This report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by ASCU to manage and coordinate the work related to the PFS and the technical report. Ausenco was also retained to complete the infrastructure design, leach pad design, and to compile the overall cost estimate and financial model.
- AGP and Call & Nicholas (CNI) were commissioned to provide the mining methods for the underground and open pit. AGP provided designs for view berms, waste piles, and the stockpile relocation. Capital and operating costs were included in their scope.
- Samuel Engineering was commissioned to provide the mineral processing and metallurgical testing basis and plant design. Samuel's scope included the metallurgical testwork supervision and analysis, SX/EW plant, leaching process, conveyor systems, crushing and stacking system designs. Capital and operating costs for these areas were included as part of their scope.
- Clear Creek managed the drilling programs, hydrogeologic evaluations and environmental fieldwork for the study.
- ALS Geo Resources LLC was retained to provide drilling and resource modelling components of the project.

## 2.2 Qualified Persons (QP)

The QPs for the report are listed in Table 2-1. By virtue of their education, experience, and professional association membership, they are considered Qualified Person as defined by NI 43-101.

**Table 2-1: Report Contributors**

| Qualified Person | Professional Designation | Position | Employer | Independent of ASCU | Report Section |
|---|---|---|---|---|---|
| Erin L. Patterson | P.E. | Director of Technical Services | Ausenco Engineering USA South, Inc. | Yes | 1.1, 1.2, 1.19.1, 1.20, 1.22, 1.23, 1.25, 1.26, 2.1, 2.2, 2.3, 2.4.8, 2.6.2, 2.7, 3.1, 3.3, 18.1, 18.2, 18.5, 18.6, 18.7, 18.8, 18.9, 19, 21.1, 21.2, 21.3.1, 21.3.2, 21.4.1, 21.4.4, 21.4.5, 22, 24, 25.1, 25.10.1, 25.12, 25.13, 25.14, 25.15.1, 25.15.1.8.1, 25.15.1.8.2, 25.15.1.8.3, 25.15.2.6.1, 26.1, 26.12.1 and 27 |
| Scott C. Elfen | P.Eng. | Global Lead Geotechnical Services | Ausenco Sustainability ULC | Yes | 1.19.2, 2.2, 2.4.10, 18.10, 18.11, 18.12, 18.13, 25.10.2, 25.15.1.8.4, 25.15.2.6.2, 26.12.2 and 27 |
| R. Douglas Bartlett | CPG, PG | Principal | Clear Creek Associates, a subsidiary of Geo-Logic Associates | Yes | 1.21, 2.2, 2.4.2, 3.2, 4.7, 4.8, 4.9, 5, 16.3, 20, 21.3.5, 25.11, 25.15.1.9, 25.15.2.7, 26.13 and 27 |
| Gordon Zurowski | P.Eng. | Principal Mine Engineer | AGP Mining Consultants Inc. | Yes | 1.15, 1.16, 1.17.3, 2.2, 2.4.5, 15.1, 15.2, 15.4, 15.5, 16.1, 16.5.1, 16.5.2, 16.5.4, 16.5.5, 16.5.6, 16.5.7, 16.5.8, 16.5.9, 16.5.10, 16.5.11, 16.5.13, 16.5.14, 16.5.15, 16.5.16, 16.6, 16.7, 16.8, 21.3.3, 21.4.2, 25.7, 25.15.1.3, 25.15.1.5, 25.15.2.3, 26.6, 26.7, 26.8, 26.9 and 27 |
| Nat Burgio | FAusIMM (CP) | Principal Geologist | AGP Mining Consultants Inc. | Yes | 1.17.2, 2.2, 2.4.3, 15.3, 16.5.3, 16.5.12 and 27 |
| Todd Carstensen | RM-SME | Principal Mine Engineer | AGP Mining Consultants Inc. | Yes | 1.17.1, 2.2, 2.4.4, 16.4, 18.3, 18.4, 25.8 and 27 |
| Allan L. Schappert | CPG, SME-RM | Principal | ALS Geo Resources LLC | Yes | 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.14, 1.24, 2.2, 2.4.1, 2.5, 2.6.1, 4.1, 4.2, 4.3, 4.4, 4.5, 4.6, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.4, 25.5, 25.15.1.1, 25.15.1.4, 25.15.2.1, 26.2, 26.5 and 27 |
| James L. Sorensen | FAusIMM | Director | Samuel Engineering, Inc. | Yes | 1.13, 1.18, 2.2, 2.4.6, 13, 17, 25.6, 25.9, 25.15.1.2, 25.15.1.7, 25.15.2.2, 25.15.2.5, 26.3, 26.4, 26.11 and 27 |
| Matthew Bolling | P.E., PMP | Project Manager | Samuel Engineering, Inc. | Yes | 2.2, 2.4.7, 3.4, 21.3.4, 21.4.3 and 27 |
| Paul F. Cicchini | P.E. | President | North Star Geotech LLC. | Yes | 2.2, 2.4.9, 16.2, 25.15.1.6, 25.15.2.4, 26.10 and 27 |

## 2.3 Terms of Reference

This report supports disclosures by ASCU in a news release dated February 21, 2024, entitled "Arizona Sonoran Announces a Positive Pre-Feasibility Study for the Cactus Mine Project with a US$509M Post-Tax NPV and 55 kstpa Copper Cathode over 21 Years".

All measurements presented in this report are in imperial units unless otherwise noted. Currency is expressed in US dollars (US$ or USD) unless otherwise noted. Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The ASCU property contains one past-producing mine that operated from 1972 until 1984. The Sacaton Mine, which sourced is ore from the Cactus West deposit, collectively produced 400 M lbs of copper (Cu), 27,455 oz of gold (Au) and 759,000 oz of silver (Ag).

## 2.4 Site Visits and Scope of Personal Inspection

### 2.4.1 Site Inspection by Allan L. Schappert

Mr. Allan L. Schappert, previously with Stantec and now ALS Geoservices, first visited the site in August of 2019. He has made numerous visits to the site each year since; three in 2019, three in 2020, two in 2021, and four in 2022. During these visits, Allan visited active drill sites, checked collar locations against database records, watched core recovery procedures and secure transport to the core shed. He also made trips to the stockpile to watch ongoing sonic drilling and sampling/logging of the material stored there.

Mr. Schappert also visited the core shed where he observed the logging, splitting, sampling, and storage of drill core. Targeted visits were made during the oriented core program to provide direction and training with tools to assist in the logging of fracture data. On each of these visits, Allan had meetings and discussions with on-site personnel and management to provide guidance to maintain compliance with current CIM best practices.

Mr. Schappert also visited Skyline Laboratories in Tucson, Arizona which is Cactus' sole assay lab. One visit each in 2019, 2020, and 2022. During these visits Allan observed sample storage and security protocols, sample prep procedures, assay methodologies, and internal QA/QC programs and reporting. Visits to both site and the lab are ongoing as drilling continues.

### 2.4.2 Site Inspection by R. Douglas Bartlett

Mr. R. Douglas Bartlett conducted a site visit on May 12, 2020, to conduct a due diligence review for Tembo Capital. Additionally, he visited on various dates in 2022 and 2023 to meet with ASCU staff, review data, visit drill rig during well installation for a total of 10 days over two years. During the site inspection, Mr. Bartlett reviewed hydrogeologic and geologic data, maps, reports, lithologic logs, core, and personally interviewed onsite personnel.

### 2.4.3 Site Inspection by Nat Burgio

Mr. Nat Burgio, representing AGP, conducted a site visit to the Cactus project on June 12, 2023, to inspect the surface layout, review drill core samples and have technical discussion with the site geologists. Geotechnical discussions were also held with CNI at their Tucson office on June 13, 2023. The purpose of the site visit was to acquire an initial understanding of the nature of the rock mass, assess its suitability towards the various cave mining options under consideration, and to fulfill the QP mandate as per NI 43-101.

### 2.4.4 Site Inspection by Todd Carstensen

Mr. Todd Carstensen conducted a site visit to the Cactus Project property on January 24, 2023, and a second site visit on June 12, 2023. While on site, Mr. Carstensen viewed the general topography, property boundaries, existing pit, historic stockpile, existing tailings facility, existing site infrastructure, and spent time reviewing drill core in the core shed. Additionally, the underground mining areas and proposed infrastructure locations including the waste storage areas, mine facilities, access roads, and the plant and heap leach pad locations were reviewed.

Meetings were held on site with the various team members including Arizona Sonoran personnel responsible for geology, mining, environmental activities and other team members for processing and infrastructure.

### 2.4.5 Site Inspection by Gordon Zurowski

Mr. Gordon Zurowski conducted a site visit to the Cactus Project property for one day on January 24, 2023.

While on site, Mr. Zurowski reviewed recent drill core, viewed the existing pit from a distance permitted by the existing site protocol, visited the potential waste dump locations and the historic stockpile. As well proposed infrastructure locations including the existing brick works which will be repurposed for mining in the PFS, proposed plant and heap leach locations and nearby railway sidings.

Meetings were held on site with the various team members including ASCU personnel responsible for geology, environmental activities and other team members for processing and infrastructure.

### 2.4.6 Site Inspection by James L. Sorensen

Mr. James L. Sorensen has conducted several site visits between 2019 and 2023 (at least one annually) with respect to metallurgical sampling, geo-metallurgical coordination and project reviews. The most recent site visit was completed on August 31, 2023.

While on site Mr. Sorensen has witnessed and directed metallurgical sample collection activities for the testing completed at the McLelland facilities over the course of the various metallurgical programs, reviewed and inspected the ASCU metallurgical on-site testing facility, inspected the potential process plant facility locations and associated infrastructure.

#### 2.4.7 Site Inspection by Matthew Bolling

Mr. Matthew Bolling has conducted multiple site visits between 2021 and 2023 (at least one annually) for project planning purposes to manage the development and design of the process equipment and project reviews. The most recent site visit was completed on January 17, 2023.

While on site Mr. Bolling has participated in project kick off meetings and reviews, toured the site to review the overall site layout, which included inspecting the potential process plant facility locations and associated infrastructure.

#### 2.4.8 Site Inspection by Erin L. Patterson

Mrs. Erin Patterson participated in a site visit to the Cactus Mine property on September 21, 2023. While on site, Mrs. Patterson visited the open pit overlook, shaft, and the well sites adjacent to the pit, toured the perimeter of the existing stockpile, tailings facility, substation and mine offices. Discussions regarding site drainage, leach pad layout and design, underground facilities, access roads, and other infrastructure were conducted while on site.

#### 2.4.9 Site Inspection by Paul F. Cicchini

Mr. Paul Cicchini conducted a site visit to the Cactus mine on November 6, 2023. During the visit Mr. Cicchini visited the Cactus pit to review bench performance and slope conditions in the various units that comprise the walls. A visit to the core shed was conducted to verify that logging characterization of the rock matched actual core condition. In addition, several locations were visited to examine Gila Conglomerate core, unboxed, and laid out on the surface. Prior to and following the site visit, several meetings were conducted at the CNI offices on October 5, November 3, and November 16, 2023, to review the characterization of geotechnical conditions, geotechnical design parameters, mining method selection and support requirements for the various mining units and to discuss conclusions from the site visit.

#### 2.4.10 Site Inspection by Scott C. Elfen

Mr. Scott Elfen participated in a site visit to the Cactus Mine property on September 21, 2023. While on site, Mr. Elfen visited the open pit overlook, shaft and well sites adjacent to the pit, toured the perimeter of the existing stockpile, tailings facility, substation and mine offices. Discussions regarding site drainage, leach pad layout and design, underground facilities, access roads, and other infrastructure were conducted while on site.

### 2.5 Effective Dates

The Stockpile Resource Estimates has an effective date of March 1, 2022, the Cactus Mineral Resource Estimate has an effective date of April 29, 2022, and the Parks/Salyer Resource Estimate has an effective date of May 19, 2023. All three used a copper price of US$3.75/lb. The effective date of this report is February 21, 2024.

## 2.6 Information Sources and References

### 2.6.1 Introduction

The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. Information related to these matters has been provided directly by ASCU and include, without limitation, validity, status of environmental and other liabilities, and permitting to allow completion of environmental assessment work. Allan L/ Schappert (QP) visited the Arizona State Lands and Department, and Pinal Country Recorder and Assessors Office website to review publicly available data on ASLD leases and property ownership for the project. These matters were not otherwise independently verified by the QPs but appear to be reasonable representations that are suitable for inclusion in Section 4 of this report.

The primary sources of geology and drilling information in this report was ASCU and was collected and validated by Allan L. Schappert (QP) during multiple site visits and communications from 2019 through 2023. Additional information and assay data referenced for Skyline Assayers and Laboratories Tucson, Arizona.

### 2.6.2 Previous Technical Reports

The Cactus Project has been the subject of previous technical reports as follows:

- Mineral Resource Estimate and Technical Report, Stantec, Effective Date: 10 November 2022, Prepared for Arizona Sonora Copper Company, Inc.
- Preliminary Economic Assessment, Stantec, Effective Date: 31 August 2021, Prepared for Arizona Sonora Copper Company, Inc.

## 2.7 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are imperial and all currencies are expressed in US dollars (symbol: US$, or currency: USD) unless otherwise stated. Copper metal is expressed in tons. All material tons are expressed as dry tons unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-2, and units of measurement are listed in Table 2-3.

**Table 2-2: Abbreviations and Acronyms**

| Abbreviation | Description |
|---|---|
| AA | atomic absorption spectroscopy |
| APP | Aquifer Protection Permit Amendment |
| APS | Arizona Public Service |
| ADEQ | Arizona Department of Environmental Quality |
| ADWR | Arizona Department of Water Resources |
| Ag | silver |
| AGP | AGP Mining Consultants Inc. |

| Abbreviation | Description |
|---|---|
| ALS | ALS Geo Resources, LLC |
| AMA | Active Management Area |
| APS | Arizona Public Service |
| ASARCO | American Smelting and Refining Company |
| ASCU | Arizona Sonoran Copper Company |
| ASLD | Arizona State Land Department |
| Au | gold |
| AZ | Arizona |
| AZPDES | Arizona Pollutant Discharge Elimination System |
| BCE | Bronco Creek Exploration |
| BLM | Bureau of Land Management |
| CE | Cactus East |
| CERCLA | Comprehensive Environmental Response, Compensation, and Liability Act |
| CIM | Canadian Institute of Mining, Metallurgy and Petroleum |
| CIM Definition Standards | CIM Definition Standards for Mineral Resources and Mineral Reserves 2014 |
| CNI | Call & Nicholas Inc. |
| CoGs | cut-off grades |
| Cu | Copper |
| CuT | total copper |
| CuAS | acid soluble copper |
| CuCN | cyanide soluble copper |
| CW | Cactus West |
| DDH | diamond drill hole |
| REEEE | reasonable expectation of eventual economic extraction |
| EM | electromagnetic |
| EPA | Environmental Protection Agency |
| FA | fire assay |
| FAA | Federal Aviation Administration |
| FAR | Federal Aviation Regulation |
| FET | federal excise tax |
| FS | feasibility study |
| G&A | general and administration |
| GPS | Global positioning system |
| GRWS | Gila River Water Storage, LLC |
| HLF | Heap Leach Facility |
| HLP | Heap Leach Pad |
| HQ | H-size and Q-group wireline diamond drilling system |

| Abbreviation | Description |
|---|---|
| HV | high voltage |
| IP | induced polarization |
| IPC | International Plumbing Code |
| ISO | International Organization for Standardization |
| IRR | internal rate of return |
| LCU | lower conglomerate unit |
| LHD | load, haul, dump |
| LIDAR | light detection and ranging |
| LLC | limited liability company |
| LOM | life of mine |
| LV | low voltage |
| MCC | motor control centre |
| MEP | Mineral Exploration Permits |
| MOU | Memorandum of understanding |
| MRE | mineral resource estimate |
| MSCU | middle silt and clay unit |
| MV | medium voltage |
| NI 43-101 | National Instrument 43-101 (Regulation 43-101 in Quebec) |
| NN | nearest neighbour |
| NSR | net smelter return |
| PA | purchase agreement |
| PEA | preliminary economic assessment |
| PFS | prefeasibility study |
| PLS | pregnant leach solution |
| PPA | Prospective Purchaser Agreement |
| PQ | cores designed specifically for switched-mode power supplies. |
| PS | Parks/Salyer |
| QA/QC | quality assurance/quality control |
| QP | qualified person (as defined in National Instrument 43-101) |
| RCRA | Resource Conservation and Recovery Act |
| RMSE | root mean square error |
| ROGR | Registry of Groundwater Rights |
| ROM | run of mine |
| RQD | rock quality designation |
| SG | specific gravity |
| SCIP | San Carlos Irrigation Project |
| SIP | site improvement plan |

| Abbreviation | Description |
|---|---|
| SLUP | state land use permit |
| SOP | standard operating procedure |
| Std. dev. | Standard deviation |
| SX/EW | Solvent Extraction / Electrowinning |
| TSF | tailings storage facility |
| Tsol | total soluble copper |
| UAU | upper alluvial unit |
| UG | underground |
| ULC | unlimited liability corporation |
| USA | United States of America |
| UTM | Universal Transverse Mercator coordinate system |
| VMS | volcanogenic massive sulphide |
| VRP | Voluntary Remediation Program |
| WOTUS | waters of the US |
| WQARF | Water Quality Assurance Revolving Fund |
| WG | Water gauge |
| WRD | waste rock dump |

**Table 2-3: Units of Measurement**

| Abbreviation | Description |
|---|---|
| $/t | dollars per ton |
| % | percent |
| % solids | percent solids by weight |
| ° | angular degree |
| °C | degrees Celsius |
| °F | degrees Fahrenheit |
| µm | micron (micrometer) |
| af | acre-foot |
| afy | acre feet per year |
| asl | above sea level |
| ft | foot (12 inches) |
| gpm | gallons per minute |
| gpL | grams per litre |
| h | hour (60 minutes) |
| in | inch |
| K | thousands |
| km | kilometer |

| Abbreviation | Description |
|---|---|
| kt/h | kilometers per hour |
| kt | kiloton |
| kV | kilovolt |
| kW | kilowatt |
| kWh/t | kilowatt-hour per ton |
| lb | pound |
| m | metre |
| $m^2$ | metre squared |
| M | million |
| Ma | million years (annum) |
| mi | mile |
| ml | milliliter |
| Moz | million (troy) ounces |
| mph | miles per hour |
| Mton | million short tons |
| Mt/y | Million short tons per annum |
| MVA | megavolt-amperes |
| MW | megawatt |
| oz | troy ounce |
| oz/ton | ounce (troy) per short ton (2,000 lbs) |
| ppb | parts per billion |
| ppm | parts per million |
| t | metric tonne |
| ton | short ton (2,000 lbs) |
| ton/d | short tons per day |
| US$ | US dollar (as symbol) |
| USD | US dollars (currency) |
| y | year |
| $yd^3$ | yards cubed |

### 2.7.1 Work Breakdown Structure (WBS)

The physical facilities and utilities for the project include, but are not limited to, the following areas defined in Table 2-4. Area descriptions are noted in the WBS Level 2 Descriptions.

Table 2-4: Work Breakdown Structure

| WBS | Level 1 Description | Level 2 Description |
|---|---|---|
| **1000** | **INFRASTRUCTURE** | |
| 1100 | | Site Preparation |
| 1200 | | Sewage and Waste Management |
| 1300 | | Environmental Management Facilities |
| 1400 | | Not used |
| 1500 | | Administration Buildings and Offices |
| 1600 | | Maintenance and Supporting Facilities |
| 1700 | | Power Supply |
| 1800 | | Rail |
| 1900 | | Water Management |
| **2000** | **MINING** | |
| 2100 | | Mining Existing Stockpiles |
| 2200 | | Open Pit Development |
| 2300 | | Open Pit Equipment |
| 2400 | | Open Pit Infrastructure |
| 2500 | | Underground – Cactus East |
| 2600 | | Underground – Parks/Salyer |
| 2700 | | Underground – Combined |
| 2800 | | TBD |
| 2900 | | Mine Maintenance and Support Facilities |
| **3000** | **CRUSHING AND CONVEYING** | |
| 3100 | | Primary Crushing |
| 3200 | | Coarse Ore Storage and Reclaim |
| 3300 | | Secondary Crusher |
| 3400 | | Crushed Ore Stockpile |
| **4000** | **LEACHING & WASTE ROCK STORAGE** | |
| 4100 | | Heap Leach Pads (HLP) |
| 4200 | | HLP Ore Handling |
| 4300 | | Pregnant Leach Solution Management |
| 4400 | | Raffinate Management |
| 4500 | | Event Ponds |
| 4600 | | Waste Rock |
| **5000** | **SOLVENT EXTRACTION (SX)** | |
| 5100 | | Solvent Extraction |
| 5200 | | Tank Farm |
| **6000** | **ELECTROWINNING (EW)** | |
| 6100 | | Electrowinning (EW) |
| 6200 | | Cathode Storage |
| 6300 | | Laboratory |
| 6400 | | Electrowinning Building |
| 6500 | | Control Room |
| **7000** | **REAGENTS** | |

| WBS | Level 1 Description | Level 2 Description |
|---|---|---|
| 7100 | | Reagents |
| **8000** | **PROCESS PLANT SERVICES AND UTILITIES** | |
| 8500 | | Plant Services |
| **9000** | **PROJECT EXECUTION** | |
| 9100 | | Construction Indirect |
| 9200 | | Execution – EPCM |
| 9300 | | Commissioning |
| 9400 | | Spare Parts |
| 9500 | | First Fills |
| 9600 | | Mobile Equipment |
| 9700 | | Owner's Project Costs |
| 9800 | | Other Costs |
| 9900 | | Contingency |

# 3 RELIANCE ON OTHER EXPERTS

## 3.1 Introduction

While the authors have carefully reviewed within the scope of their technical expertise, all the available information presented to them, they cannot guarantee its accuracy and completeness. The authors reserve the right, but will not be obligated to, revise the technical report and its conclusions if additional information becomes known to them subsequent to the effective date of this report.

The QPs have relied upon other experts for data as indicated in the following sections.

## 3.2 Environmental, Permitting, Closure, and Social and Community Impacts

The land tenure and title were validated by visits to the Arizona State Lands Department and Pinal County Recorder and Assessors Offices websites to review publicly available data on ASLD leases and property ownership for the project. Clear Creek relied on information provided by ASCU and on their experience in permitting mining projects in Arizona to prepare Sections 1.21, 20 and 25.11. ASCU and the previous owner, Elim Mining, performed much of the permitting activities to date. Independent verification has not been pursued at this time with the agencies cited to confirm status and potential timing.

Documents relied upon included:

ADEQ, 2020. Letter to ASARCO Multi-State Custodial Trust dated February 28, 2020, granting covenant not to use. Signed by Laura L. Malone, Director of Waste Programs Division, ADEQ.

ADWR, 2020, https://new.azwater.gov/sites/default/files/media/20200305_PAMA4MP_Draft.pdf

City of Casa Grande, 2009. General Plan. https://drive.google.com/file/d/0B4vKG2urQq2OMDd5X0dSSWZBRjA/view

Errol Montgomery and Associates (M&A), (1986): *Hydrogeologic Conditions, ASARCO Sacaton Open-Pit Mine, Pinal County, Arizona*. Document prepared as part of Groundwater Quality Protection Permit Application, November 21, 1986.

Samuel Engineering, 2020. NI 43-101 Technical Report; Preliminary Economic Assessment (PEA), prepared for Elim Mining Incorporated Cactus Mine Stockpile Processing Project, Pinal County, Arizona, USA. March 12, 2020, Revision 1

Tetra Tech, Inc., 2017a. Sacaton Site Characterization Work Plan, prepared for ASARCO Multi-State Environmental Custodial Trust. May 1, 2017.

Tetra Tech, Inc., 2017b. Technical Memorandum Re: Initial Hydrogeologic Characterization Study submitted to John Patricki and Tina LePage, Arizona Department of Environmental Quality. December 21.

Tetra Tech, Inc., 2018a. Technical Memorandum Re: 201 Sacaton –Comprehensive Facility Inspection submitted to John Patricki, Arizona Department of Environmental Quality. July 15.

Tetra Tech, Inc., 2018b. Technical Memorandum Re: TruStone Comprehensive Facility Inspection, submitted to John Patricki, Arizona Department of Environmental Quality. July 15.

Tetra Tech, Inc., 2019a. Demolition Completion Report – Sacaton Mine Site, prepared for ASARCO Multi-State Environmental Custodial Trust. March 11.

Tetra Tech, Inc., 2019b. Site Improvement Plan – Sacaton Mine Site, prepared for ASARCO Multi-State Environmental Custodial Trust. March 11.

Tetra Tech, Inc., 2019c. Site Improvement Plan – Sacaton Mine Site Amendment 1, prepared for ASARCO Multi-State Environmental Custodial Trust. November 26, 2019. United States Environmental Protection Agency (EPA): *Lean & Water Toolkit: Appendix C – Water Unit Conversions and Calculations*. https://www.epa.gov/sustainability/lean-water-toolkit-appendix-c

## 3.3 Taxation

The QPs have not independently reviewed the taxation information. The QPs have fully relied upon, and disclaim responsibility for, taxation information derived from experts retained by ASCU as contained in the following document:

A letter authored by Mining Tax Plan LLC titled: Arizona Sonoran CC_PFS Study_Memo_Client.pdf dated February 20, 2024.

This information is used in Sections 1.23, 22 and 25.14 of the Report.

Mining Tax Plan LLC (MTP) specializes in U.S. federal and state income taxation including foreign income taxation of precious metal, non-metallic ores, coal, and quarry mining companies. MTP has experience with extractive and natural resource industries and specialize in state mineral property and severance taxes in Alaska, Arizona, California, Colorado, Idaho, Montana, Nevada, and Utah.

## 3.4 Agreements

The QP's have fully relied upon, and disclaim responsibility for, information derived from ASCU for information related to the following agreement for the purchase of used equipment:

A letter authored by Arizona Sonoran Copper Company titled: ASCU Letter of Intent to Purchase (Rev 1.0 25Jan2024) dated January 26, 2024.

This information is used in Sections 17.4, 21.3.4, 21.3.4.9, 25.9 and 25.15.1.7 of the Report.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Description of Location

The entire Cactus Mine Project is located on private land approximately 6 mi (10 km) northwest of the city of Casa Grande and 40 road miles south southwest of the Greater Phoenix metropolitan area. Access to the Project is approximately 4.6 mi (7.4 km) west of AZ-387 on North Bianco Road off West Maricopa-Casa Grande Highway. The coordinates for the centre of the Project are 111.82° W longitude and 32.94° N latitude, with a variable elevation between 1,330 to 1,510 ft (405 to 460 m) above sea level (asl).

## 4.2 Project Ownership

In 2019, Cactus 110 LLC, a wholly owned subsidiary of ASCU, 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, ASCU 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, ASCU acquired a prospecting permit for adjacent land owned by the Arizona State Lands Department.

In February 2021, 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.

In May 2021, 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.

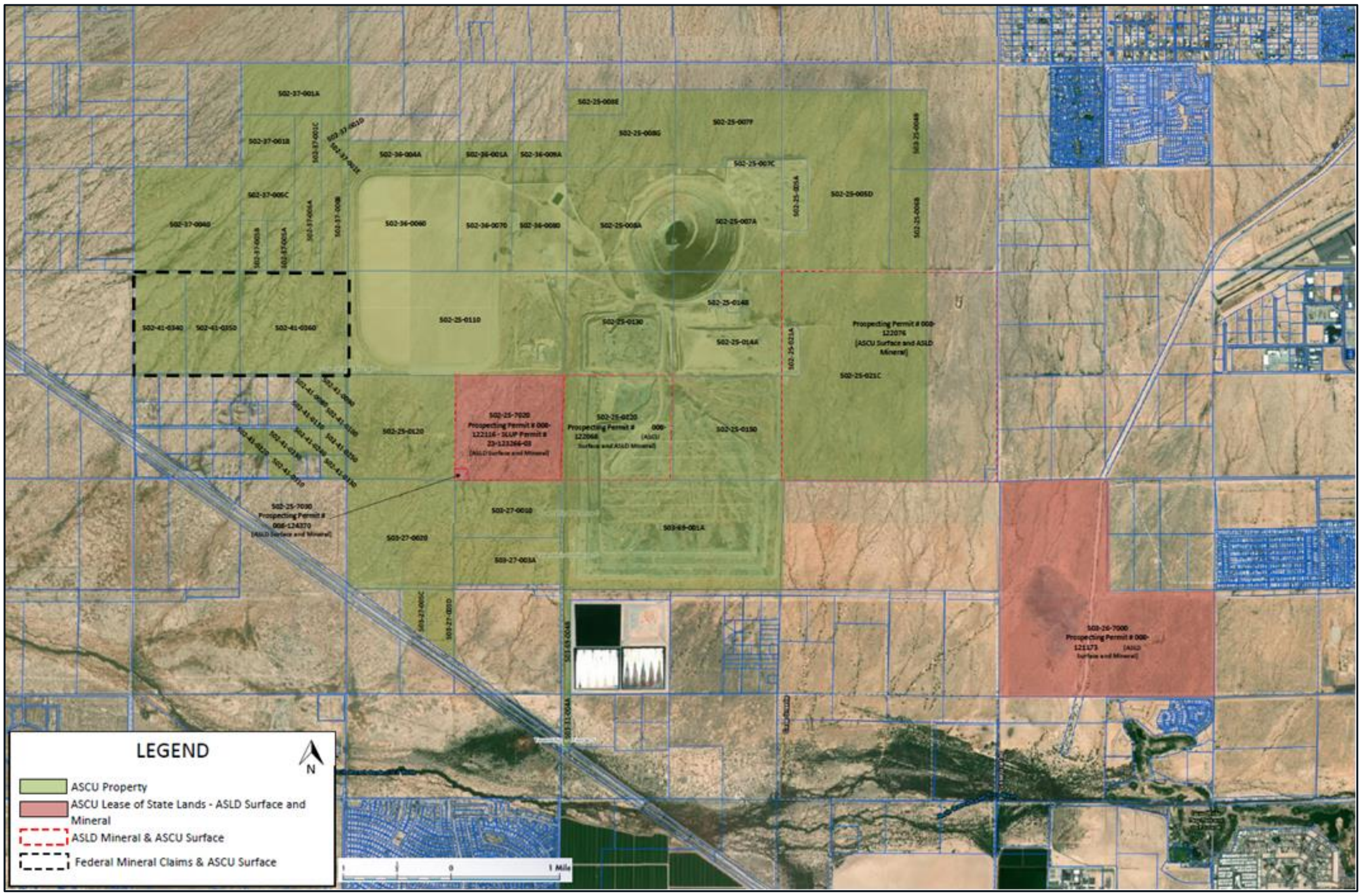
In February 2022, ASCU entered into an agreement to transfer Bronco Creek Explorations Mineral Exploration Lease (MEP) with the Arizona State Lands Department to ASCU. This MEP consists of 157.50 acres of State-owned surface and mineral rights and is held under Cactus 110. This land contained a portion of the Parks/Salyer project.

In February 2023, Cactus 110 LLC executed an agreement with MainSpring Casa Grande LLC to purchase 522.78 acres of land adjacent to the Project.

Cactus 110 LLC also has three ASLD Mineral Prospecting Permits (MEP's) that the State has surface and mineral rights (649.12 acres), two ASLD Mineral Prospecting Permits (MEP's) that the State has mineral rights only with ASCU owning the surface rights(797.5 acres).

The Project comprises total landholdings of approximately 5,370 acres. A summary of the current landholdings is as provided in Section 4.4. Figure 4- is a plan map showing these holdings.

**Figure 4-1:** **Location of** Mineral Tenure and Surface Rights




Source: ASCU, 2023.

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

## 4.3 Property Mineral Tenure Location and Surface Rights

The Project is 100% controlled by ASCU through its wholly owned subsidiary Cactus 110 LLC, which encompasses an area of approximately 5,381 acres. Of that, 4,731.92 acres is fee simple land. This includes:

- 3 ASLD prospecting permits that the State has surface and minerals (649.12 acres).
- 2 ASLD prospecting permits that the State has mineral rights only with ASCU owning the surface rights (797.5 acres).

- 18 BLM unpatented mining lode claims for mineral rights only as ASCU owns the surface rights (320 acres). The BLM unpatented mining claims are outside of the known mineralization and there are no plans for mining in these areas.

Table 4-1 identifies ASCU's Fee Simple Lands it owns or has options for. Property mineral tenure is shown in Figure 4-1.

Table 4-2 Lode Claims and Mineral Exploration Permits (MEP'S) with Arizona State Lands Department (ASLD).

Table 4-3 list all the Arizona State Lands Department Mineral Exploration Permits.

Table 4-4 Mineral Tenure Plan shows the requirements for maintaining the ASCU land package.

**Table 4-1: Fee Simple Lands Table**

| 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 |
| 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<br>(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<br>(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 |
| 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 | 5 South | 5 East | 33 | 160 |

| Owner | Parcel No. | Property Description | Township | Range | Section | Acres |
|---|---|---|---|---|---|---|
| | | (surface only) | | | | |
| 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 |
| 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 |

| Owner | Parcel No. | Property Description | Township | Range | Section | Acres |
|---|---|---|---|---|---|---|
| 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 BEGINNING, 4,873,050.52 SQUARE FEET, 111.87 ACRES | 5 South | 5 East | 26 | 111.87 |
| 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<br>(surface only) | 5 South | 5 East | 36 | 417.67 |
| **TOTAL FOR CACTUS 110 LLC** | | | | | **4,209.14** | |
| **MAINSPRING CASA GRANDE LLC (OPTION)** | | | | | | |

| Owner | Parcel No. | Property Description | Township | Range | Section | Acres |
|---|---|---|---|---|---|---|
| MAINSPRING CASA GRANDE LLC | 503-27-0020 | LOTS 3 4 & S1/2 NW OF SEC 4-6S-5E 170.71 AC | 6 South | 5 East | 4 | 170.71 |
| MAINSPRING CASA GRANDE LLC | 503-27-0010 | LOT 1 & 2 OF SEC 4-6S-5E 90.56 AC | 6 South | 5 East | 4 | 90.56 |
| MAINSPRING CASA GRANDE LLC | 503-27-003A | S1/2 NE EXC S-140' OF SEC 4-6S-5E 72.00 AC | 6 South | 5 East | 4 | 72 |
| MAINSPRING CASA GRANDE LLC | 503-27-005C | THAT PRT OF E1/2 OF SW 6S-5E: COM AT THE CTR QUARTER CR OF SAID SEC 4; TH W-658.75' TO POB; TH S-1278.13' TO N ROW LINE OF CG-MAR HYWY; TH N-53 DEG W-818.53' ALNG SAID ROW; TH N-796.85' TH E-658.75' TO POB SEC 4-6S-5E 15.69 AC | 6 South | 5 East | 4 | 15.69 |
| MAINSPRING CASA GRANDE LLC | 503-27-005D | E1/2 E1/2 SW OF SEC 4-6S-5E N OF R/R SEC 4-6S-5E 28.01 AC | 6 South | 5 East | 4 | 28.01 |
| MAINSPRING CASA GRANDE LLC | 503-27-004A | SE LYNG N OF HWY R/W SEC 4-6S-5E EXC N-150' OF E-660' 142.72 AC | 6 South | 5 East | 4 | 142.72 |
| MAINSPRING CASA GRANDE LLC | 503-27-004C | SW SW SE LYNG S OF SPRR R/W IN SEC 4-6S-5E 3.09 AC | 6 South | 5 East | 4 | 3.09 |
| **TOTAL FOR MAINSPRING CASA GRANDE LLC (OPTION)** | | | | | **522.78** | |

**Table 4-2: BLM Unpatented Mining Lode Claims Table**

| Claim Name | Claim Number | Holder | Type of Claim | Issue Date | Expiration Date | Area (AC) |
|---|---|---|---|---|---|---|
| S1 | AMC459838 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S2 | AMC459839 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S3 | AMC459840 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S4 | AMC459841 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S5 | AMC459842 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S6 | AMC459843 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S7 | AMC459844 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S8 | AMC459845 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S9 | AMC459846 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S10 | AMC459847 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S11 | AMC459848 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |

| Claim Name | Claim Number | Holder | Type of Claim | Issue Date | Expiration Date | Area (AC) |
|---|---|---|---|---|---|---|
| S12 | AMC459849 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S13 | AMC459850 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S14 | AMC459851 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S15 | AMC459852 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S16 | AMC459853 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S17 | AMC459854 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |
| S18 | AMC459855 | Cactus 110 LLC | Lode Claim | 1-17-2020 | 9-3-2024 | 20.66 |

**Table 4-3: Arizona State Lands Department Mineral Exploration Permits Table**

| Owner | Parcel No. | Property Description | Township | Range | Section | Acres |
|---|---|---|---|---|---|---|
| **Arizona State Lands Department (Leased Lands)** | | | | | | |
| Arizona State Lands Department (Mineral Exploration Permit#008-121173) | 503-26-7000 | Lots 3 4 S2NW S2 | 6 South | 5 East | 1 | 489.12 |
| Arizona State Lands Department (Mineral Exploration Permit#008-122116 & SLUP#23-123266-03) | 502-25-7020 | SE EX SWSWSWSE | 5 South | 5 East | 34 | 157.5 |
| Arizona State Lands Department (Mineral Exploration Permit# 008-124370) | 502-25-7030 | SWSWSWSE | 5 South | 5 East | 34 | 2.5 |
| Arizona State Lands Department (Mineral Exploration Permit#008-122068) | 502-25-0220 | SW SEC 35-5S-5E 160.00 AC<br>(mineral only) | 5 South | 5 East | 35 | 160 |
| Arizona State Lands Department (Mineral Exploration Permit#008-122076) | 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<br>(mineral only) | 5 South | 5 East | 36 | 13.5 |

| Owner | Parcel No. | Property Description | Township | Range | Section | Acres |
|---|---|---|---|---|---|---|
| Arizona State Lands Department (Mineral Exploration Permit#008-122076) | 502-25-021C | The entire west half of section 36, township 05 south, range 05 east and the west 894.69 ft 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 ft to the point of beginning, thence south 88 degrees east 227.57 ft to a tangent curve to the right, having a radius 217.19 ft, thence southeasterly along the curve with a central angle of 85 degrees 47 minutes 34 seconds, an arc distance 325.21 ft, thence south 02 degrees east 980.73 ft 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 ft, thence southwesterly along said curve with a central angle of 89 degrees 33 minutes 32 seconds, an arc distance of 192.70 ft, thence west 360.55 ft, thence north 1313.81 ft to the point of beginning, also known as parcel 2 of survey 2022-016495, 18,193,685.88 square ft, 417.67 acres<br>(mineral only) | 5 South | 5 East | 36 | 417.67 |
| Arizona State Lands Department (Mineral Exploration Permit#008-122076) | 502-25-021D | The entire east half of section 36, township 05 south, range 05 east; except the west 894.69 ft of the east half of said section 36; also, except the southeast quarter of the southeast quarter of the southeast quarter of the southeast quarter of said section 36, 206.33 acres<br>(mineral only) | 5 South | 5 East | 36 | 206.33 |
| Arizona State Lands Department (Mineral Exploration Permit#008-122076) | NA | SESESESE<br>(mineral only) | 5 South | 5 East | 36 | 2.5 |
| **Total For Arizona State Lands Department (Leased Lands)** | | | | | | **649.12** |

The majority of the Project is fee simple with additional portions that are held under Mineral Exploration Permits (MEP's) with the Arizona State Lands Department (ASLD) and 18 Federal Lode Claims.

Table 4-4: **Mineral Tenure Plan**

| Land Package Category | Payee | Payment/Renewal Date | Fee USD |
|---|---|---|---|
| Private land parcels | Pinal County Treasurer | 3-Sep-24 | Varies based on location and size of property |
| BLM unpatented claims | BLM | 31-Aug-24 | $165/claim |
| ASLD prospecting permit | ASLD | 008-122116-00 - 01-Aug-26 | $157.50 |
| | ASLD | 008-124370-00 - 23-Aug-28 | $3,000.00 |
| | ASLD | 008-122068-00 - 18-Oct-28 | $320.00 |
| | ASLD | 008-122076-00 - 18-Oct-28 | $1,280.00 |
| | ASLD | 008-121173-00 - 29-Jan-25 | $489.12 |
| | ASLD | 023-123266-03 - 30-Sep-24 | $21,000.00 |

## 4.4 Surface Rights

The Project is 100% controlled by ASCU through its wholly owned subsidiary Cactus 110 LLC, encompasses an area of approximately 5,381 acres of that 4,731.92 acres is fee simple land, three ASLD prospecting permits that the State has surface and minerals (649.12 acres), two ASLD prospecting permits that the State has minerals only with ASCU owning the surface (797.5 acres) and 18 BLM unpatented mining claims, this is for mineral only as ASCU owns the surface rights (320 acres). The BLM unpatented mining claims are outside of the known mineralization and there are no plans for mining in these areas see Figure 4-2.

## 4.5 Water Rights

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, has groundwater rights associated with mining activities.

- Type 2 Non-Irrigation Grandfathered Right No. 58-100706.0004. This right includes 136-acre foot per year (afy).
- Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing Permit No. 59-233782.0000. This permit allows ASCU 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 April 14, 2021, and the Expiration Date of permit is April 14, 2070.
    - This permit was modified and approved on August 4, 2021, to reflect the corporate name change from Elim Mining (USA) inc. to Arizona Sonoran Copper Company (USA) Inc. The revised Permit No. is 59-233782.0001.
    - This permit was modified and approved on May 25, 2023, to add newly acquired lands to the permit. No changes were made to the volume of water per year or how long the permit is valid for. They remain the same at 3,600 afy good until April 14, 2070. The revised Permit No. is 59-233782.0002.

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.

A memorandum of understanding (MOU) is also in place with the City of Casa Grande to purchase Grade A+ Effluent Water from the city at $100/af.

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.

## 4.6 Royalties and Encumbrances

The Project is subject to three royalties based on potential mining production, as detailed in this section. Figure 4-2 shows the claims applicable to royalties.

### 4.6.1 Tembo/Elements

A 3.18% net smelter return (NSR) royalty is payable to Tembo/Elements on a portion of production from the mineral inventory in the PFS based on the current area of the MRE. ASCU can buy back 0.64% of this royalty. This will take the royalty down to 2.54%.

### 4.6.2 Bronco Creek Exploration (BCE)

A 1.50% NSR royalty is payable to BCE on a portion of production from the mineral inventory in the PFS based on the current area of the MRE for the Parks/Salyer Deposit. ASCU can buy back 1.00% of this royalty. This will take the royalty down to 0.50%.

### 4.6.3 Arizona State Lands Department (ASLD)

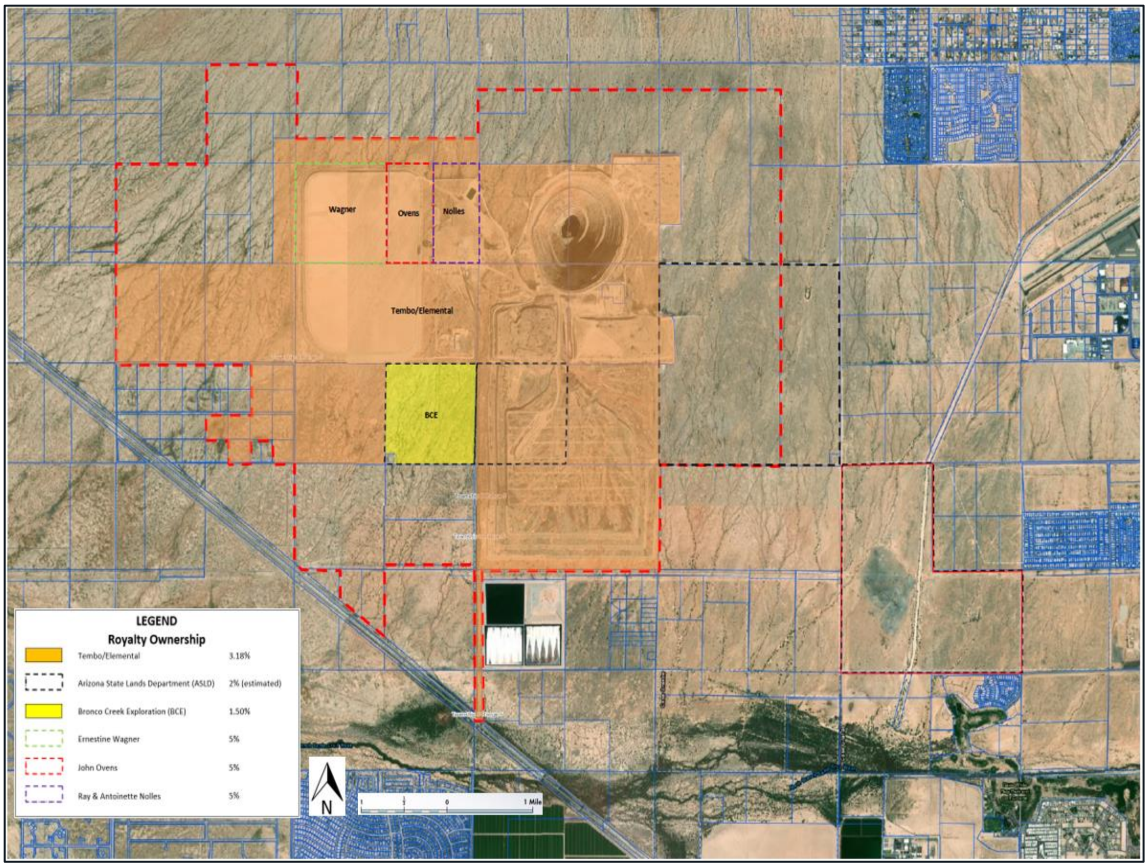
A sliding net returns royalty (2.00% to 8.00%) is payable ASLD and the State Trust on a portion of production from the mineral inventory in the PFS based on the current area of the MRE for the Parks/Salyer Deposit. ASCU still needs to formalize the royalty percentages. This will be done once ACSU submits a Mineral Development Report to ASLD to convert the existing MEP to a Mineral Lease.

For the purposes of this report, it is assumed a 2.00% NSR is payable to ASLD. Figure 4-2, Cactus Property Royalty Ownership Map shows these locations.

### 4.6.4 Additional Royalties

There are also three additional 5.00% net smelter return royalties that are payable to three individuals that ASARCO originally had in place. Based on this current PFS and MRE there is no anticipated production from these areas. Figure 4-2, Cactus Property Royalty Ownership Map shows these locations.

**Figure 4-2: Cactus Property Royalty Ownership Map**




Source: ASCU, November 15, 2023.

## 4.7 Environmental Considerations

The Cactus Mine has an environmental legacy that relates to the former ASARCO Sacaton mining operation. Please refer to Section 20.1 for a complete description of the environmental history of the site.

## 4.8 Permitting Considerations

Mining activities will be on private land and on Arizona State Land. The Army Corps of Engineers has determined that there are no Waters of the US at the project. Therefore, there is no federal nexus to the permitting, which is expected to reduce permitting timeframes. A list of permitting requirements is provided in Section 20.2. Compliance with environmental permits will be required both during and after mine closure as described in Section 26. All permits required to conduct the work described in Section 26 including drilling permits will be obtained prior to initiation of the work.

## 4.9 Social License Considerations

The community near the Project has been well exposed to and is familiar with similar types of mining operations. The historical economic benefits of mining in the area are acknowledged. There is no known organized opposition to the project and announcements regarding project status have been favorably received thus far.

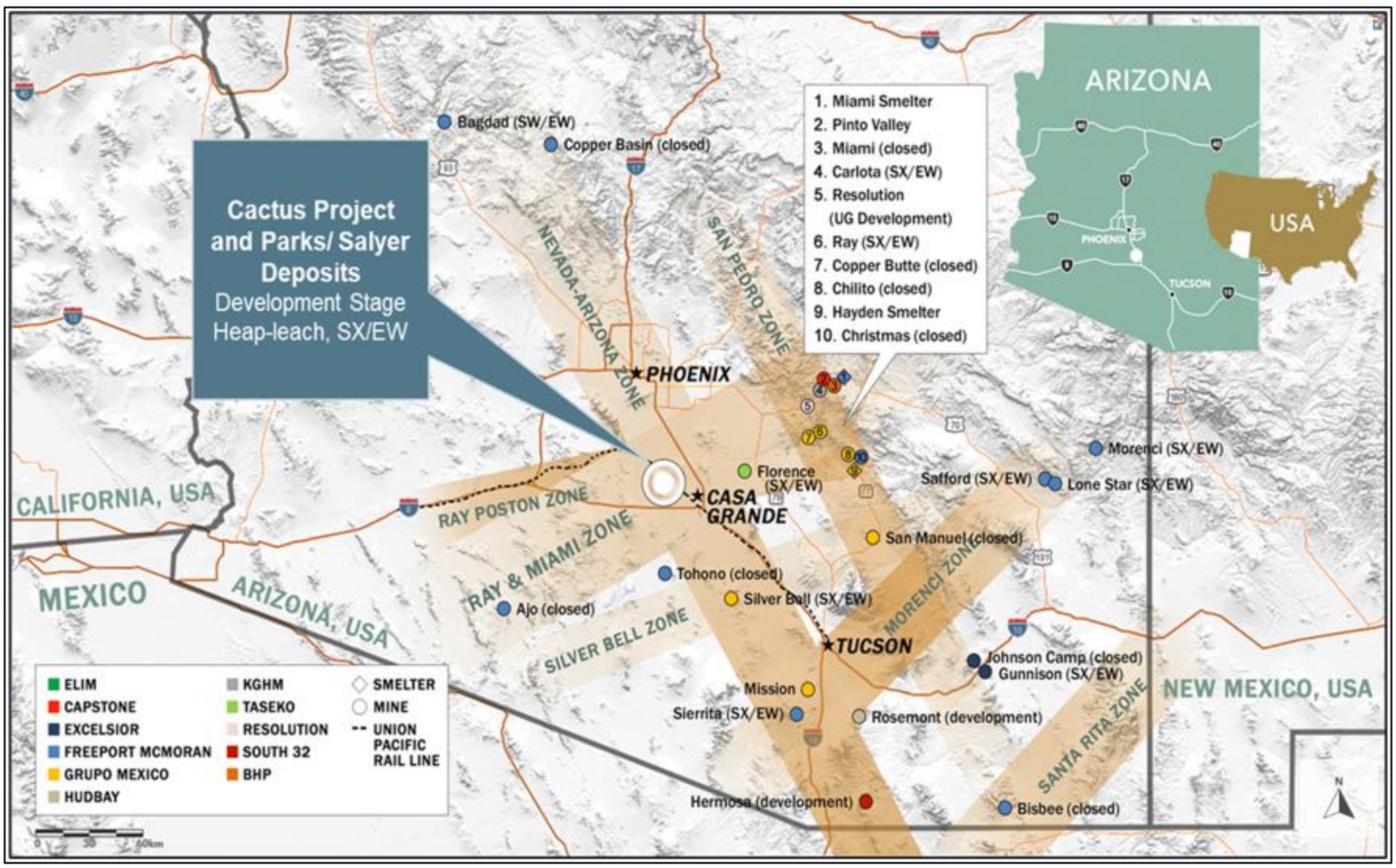
Environmental remediation measures conducted under ADEQ's Voluntary Remediation Program have resulted in ADEQ issuing a prospective purchaser agreement (PPA). The PPA, which releases ASCU from potential liabilities related to existing, known contamination under CERCLA, WQARF, and RCRA, is based on ADEQ's recognition of the substantial public benefit to the remedial work conducted at the site. No other significant risk factors that could affect access to the site are known at this time.

# 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

## 5.1 Accessibility

The Project is located approximately 3 mi northwest of the City of Casa Grande, Pinal County, Arizona. It is 40 road miles south-southeast of the Greater Phoenix metropolitan area and approximately 70 road miles northwest of Tucson. It is easily accessible from the Interstate 10 (I-10) freeway, which is approximately 10 mi east of the historic Sacaton Mine (see Figure 5-1). The Greater Phoenix area is a major population centre (approximately 4.5 million persons) with a major airport and transportation hub and well-developed infrastructure and services that support the mining industry.

**Figure 5-1: Regional Copper Mines and Processing Facilities**



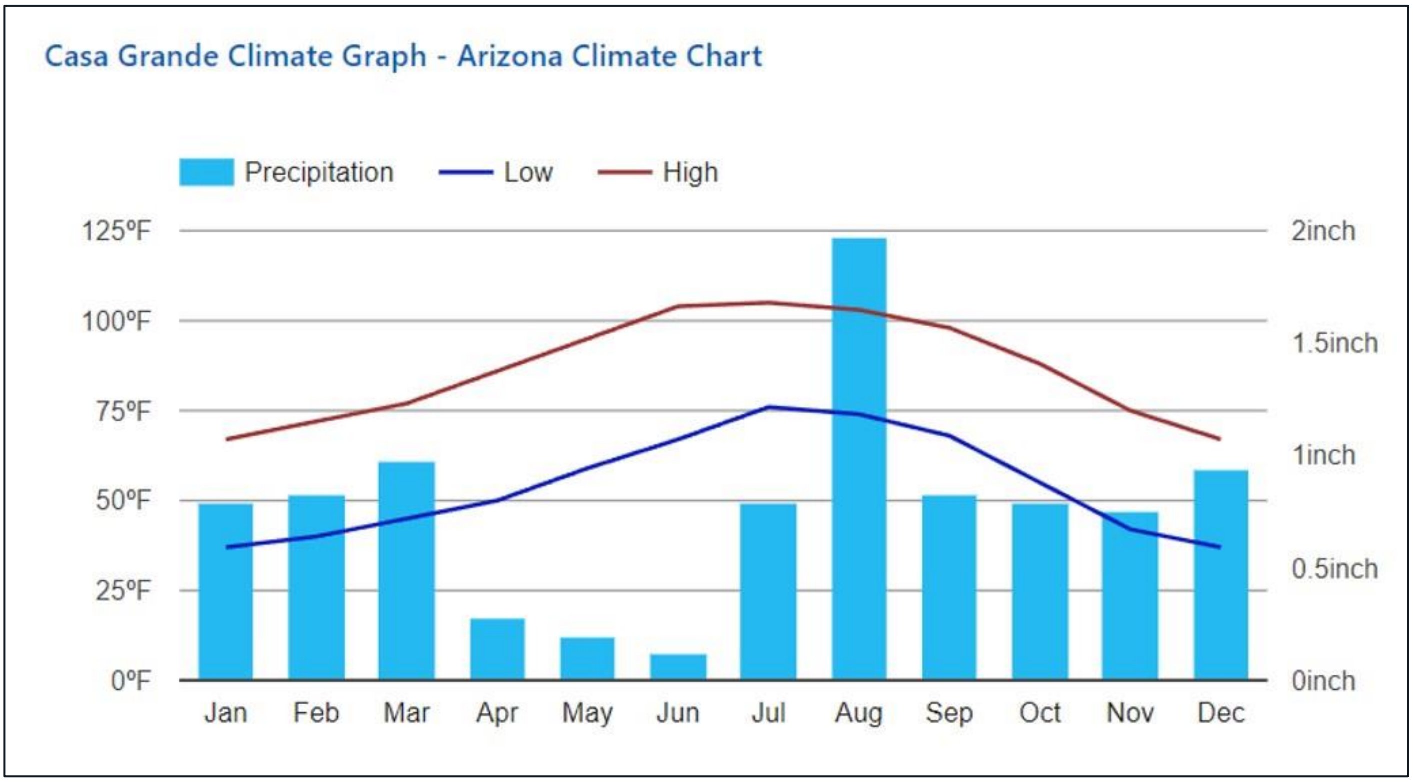

Source: ASCU, 2021.

## 5.2 Climate

The climate at the mine is also typical of the Arizona Sonoran Desert, with temperatures ranging from 19°F to 117°F, and with average annual precipitation of 8.6 in, falling primarily in high-intensity, short-duration events. See climate data in Figure 5-2. The mine site contains no surface water resources. Storm run-off 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. Storm and groundwater that enters the pit lake evaporates before migrating into the surrounding groundwater. The mild climate of Arizona affords year-round operations for mining.

The 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: Climate (High/Low)**

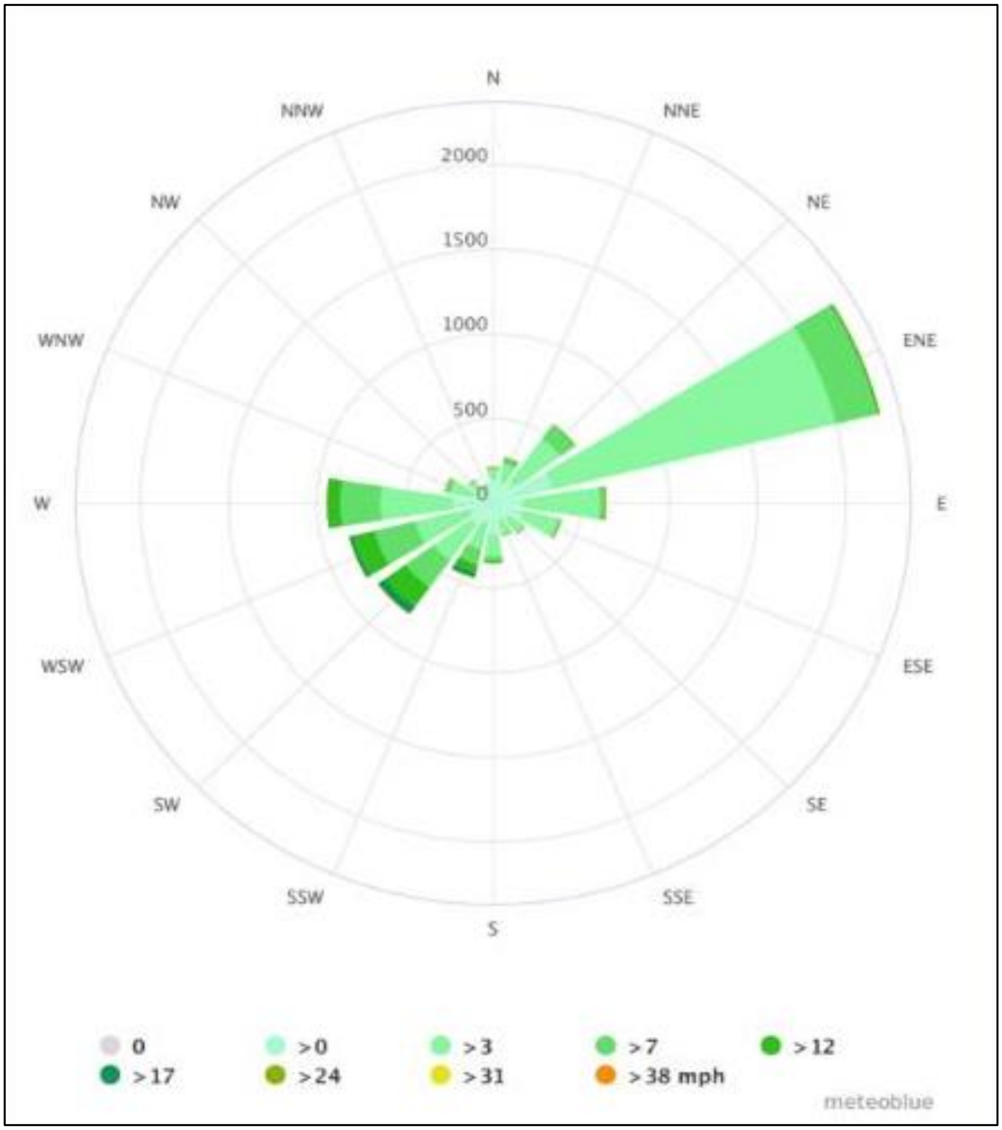



Source: https://www.usclimatedata.com/climate/casa-grande/arizona/united-states/usaz0028, 2021.

Wind is usually calm. The windiest month is May, followed by April and July. May's average wind speed of around 6.4 mph or 10.3 kt/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 19.9 mph, which is considered a fresh breeze. The wind rose for Casa Grande (Figure 5-3) shows how many hours per year the wind blows from the indicated direction. Example SW:

Wind is blowing from South-West (SW) to North-East (NE), refer to Figure 5-3.

**Figure 5-3: Wind and Speed Direction**




Source: Stantec, 2021.

### 5.3 Local Resources and Infrastructure

Electric power is available from Arizona Public Service's (APS) 69 kV transmission line which passes on the South side of the site and connects to an existing substation owned by ASCU.

Paved road and easy access to the interstate networks for transport and two major Interstates Highways (I-10 and I-8) less than 10 miles away from the project.

Well established road network existing from either ADOT, Pinal County or the City of Casa Grande surrounding the property.

Union Pacific Railway line and rail spur adjacent to the property.

Five miles distance to Casa Grande and allowing the ability of the town to supply materials/consumables in addition to just labour.

Kinder Morgan/El Paso Natural Gas two high pressure natural gas pipelines adjacent to the property should natural gas be needed.

The City of Casa Grande Water Treatment Facility located within 3 miles of the project that can supply effluent water for the operation and possibly treat waste.

An existing Arizona Water Company potable water line is adjacent to the property.

Water rights are discussed in Section 4.6. It is expected that credits will be obtained for de-watering of the pit and underground shaft. A maximum of 2,516 gpm make-up requirement is required from offsite sources.

The cities of Casa Grande and Maricopa are nearby and, combined with Phoenix, can supply sufficient skilled labour for the Project. In addition, the State of Arizona has a significant presence of copper mining in the state that can specifically provide skilled labour to the Project.

### 5.4 Physiography

The Project is situated 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 asl to 1,460 ft asl. 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.

### 5.5 Seismicity

The Project is located in one of the areas of lowest seismic activity in the state based on mapping by the Arizona Geological Survey. Most seismic activity in Arizona is north of the Mogollon Rim. The nearest recorded earthquake to the site was 29 mi to the southeast: the 2007 event at less than 1.3 on the Richter scale. The next nearest quake was recorded in Mesa 31 mi to the north in 1915; it had a magnitude of 3.9 (Arizona Geological Survey, n.d.) . The Arizona Geological Survey map does not identify any active faults mapped in the area.

## 5.6 Project Risks and Uncertainties

The risks and uncertainties associated with the Project are related to litigation, economics, regulatory developments, and financing.

ASCU is currently in litigation with RAMM Power Group, which seeks to acquire the project site through imminent domain. This risk is considered low, as the cost to acquire the property, considering the value of the mineral resource is prohibitive.

Economic risks include copper prices, stock market volatility, and interest and currency rates. These factors are not controllable by ASCU. However, the outlook for copper demand is generally positive. Higher interest rates will affect financing costs; ASCU has factored this into the economic model.

Legislative and regulatory developments are a potential risk. However, ASCU knows of no planned or pending legislation that will adversely affect the Project.

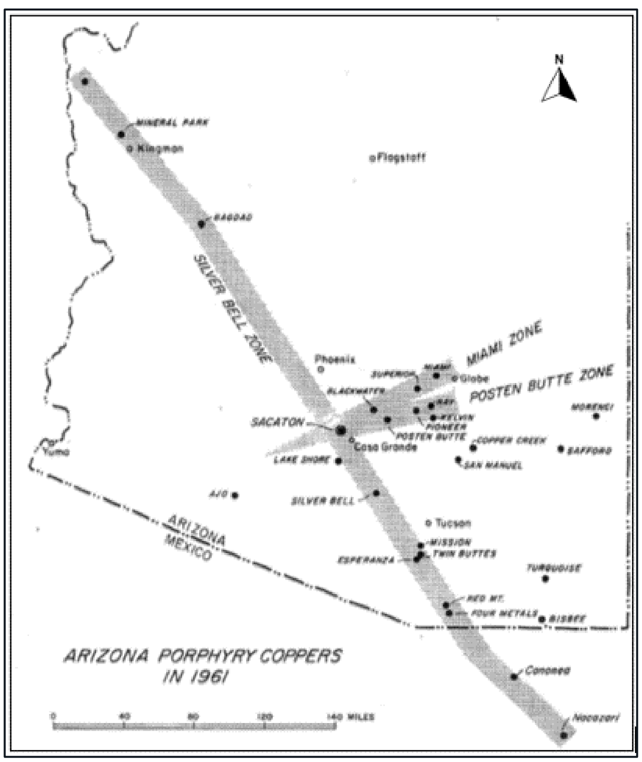
No cash flow has been generated from operations, and there is no assurance that it will generate positive cash flow in the future. Additional financing will be required to continue project development. There is no assurance that this will be obtained.

# 6 HISTORY

ASARCO geologists first discovered the Sacaton (now renamed Cactus West and Cactus East) 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 sulphide mineralization (leached capping interpretation) on the surface to evaluate the sulphide 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. Figure 6-1 is a map showing known porphyry deposits of the day and recognized trends in their relative locations. 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: Arizona Porphyry Coppers in 1961**




Source: ASARCO, 1981.

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 February 10, 1961, Kinnison, along with ASARCO geologist Art Bloucher, noticed an inconspicuous outcrop (Discovery Outcrop) west of Casa Grande. The exposure was about 300 ft (90 m) 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 taken from the Discovery Outcrop are in Figure 6-2 and Figure 6-3.

**Figure 6-2: View from Discovery Outcrop from Historic ASARCO Exploration Site**

Source: ASARCO, 1960's.

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

Source: ASCU, 2019.

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 (68,045.5 m) of both Core and Rotary exploration drilling. Elim Mining, now ASCU, completed a rigorous review and validation of this data before it was included in MRE calculations. Further details are provided in Section 10 of this report.

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 March 31, 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 (299 m) in depth from the pit rim. During operation, the Sacaton mine consisted of the pit, crushing facilities and coarse ore stockpile, a 9,000 ton/d 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 ton/d. Copper flotation mill concentrate was sent by rail to the ASARCO smelter in El Paso, Texas.

During mining of the open pit, a waste dump was created through dumping of defined waste material. All oxide copper mineralization, and sulphide 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 report due to the level of mineralized material discarded.

During the operating period, ASARCO sank a 2,000 ft (610 m) 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 6-4: Historic Overview of Prior Sacaton Mine Site**

Source: ASARCO, 1980s.

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

Source: ASARCO, 1980s.

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 now known as MainSpring 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 ASCU acquired the property in 2020.

In 2005, ASARCO filed for reorganization under Section 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 June 5, 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 $20M 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 (ASCU), 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). With associated royalties, these private land assets represent, among other things, the mineral rights to the old Sacaton East, Sacaton West, Parks/Salyer deposits, and the MainSpring Project. Further landholdings acquired by Arizona Sonoran or leased are also referred to above (as shown in Section 4). 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. ASCU is benefiting from the high quality of work and historical records remaining from the past operators.

# 7 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: Major Intrusions in The Cactus Project Area**

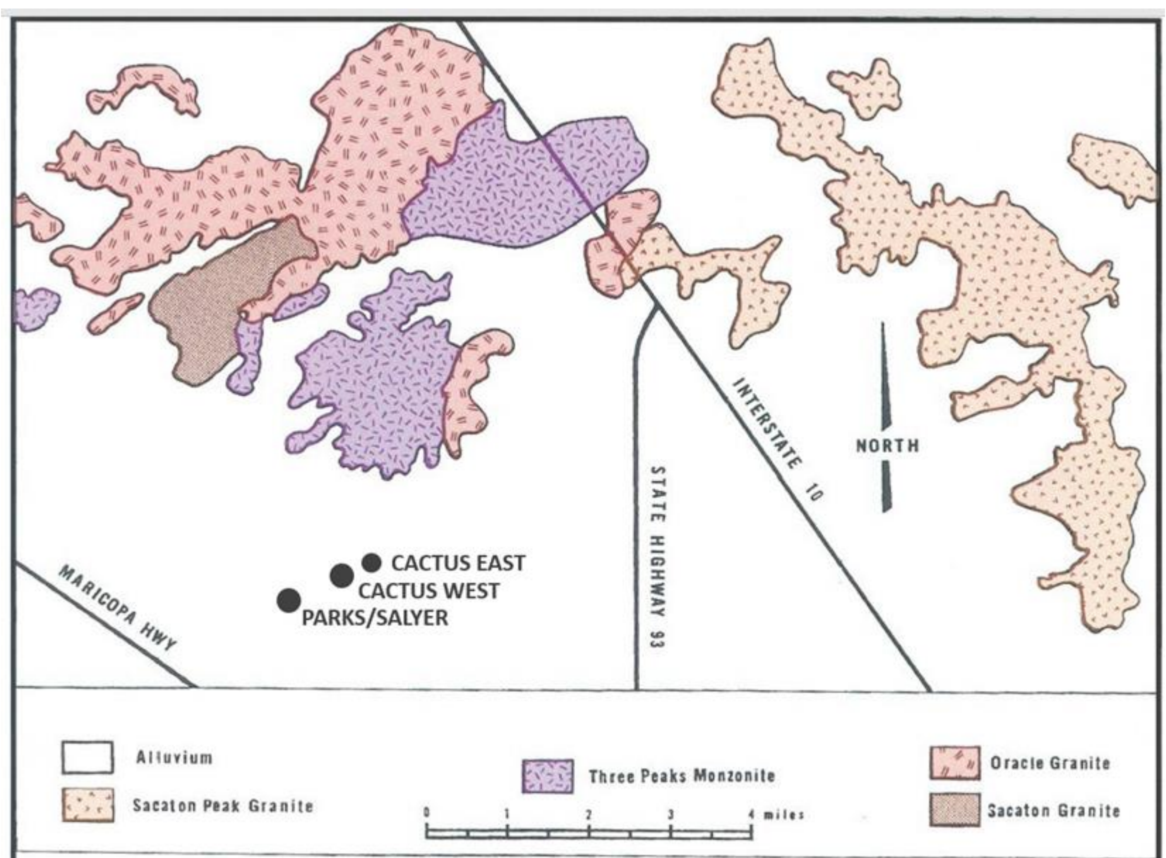



Figure 25.1. Generalized geologic map of the Sacaton Mountains. Modified after Balla (1972).

Source: Balloa, 1972.

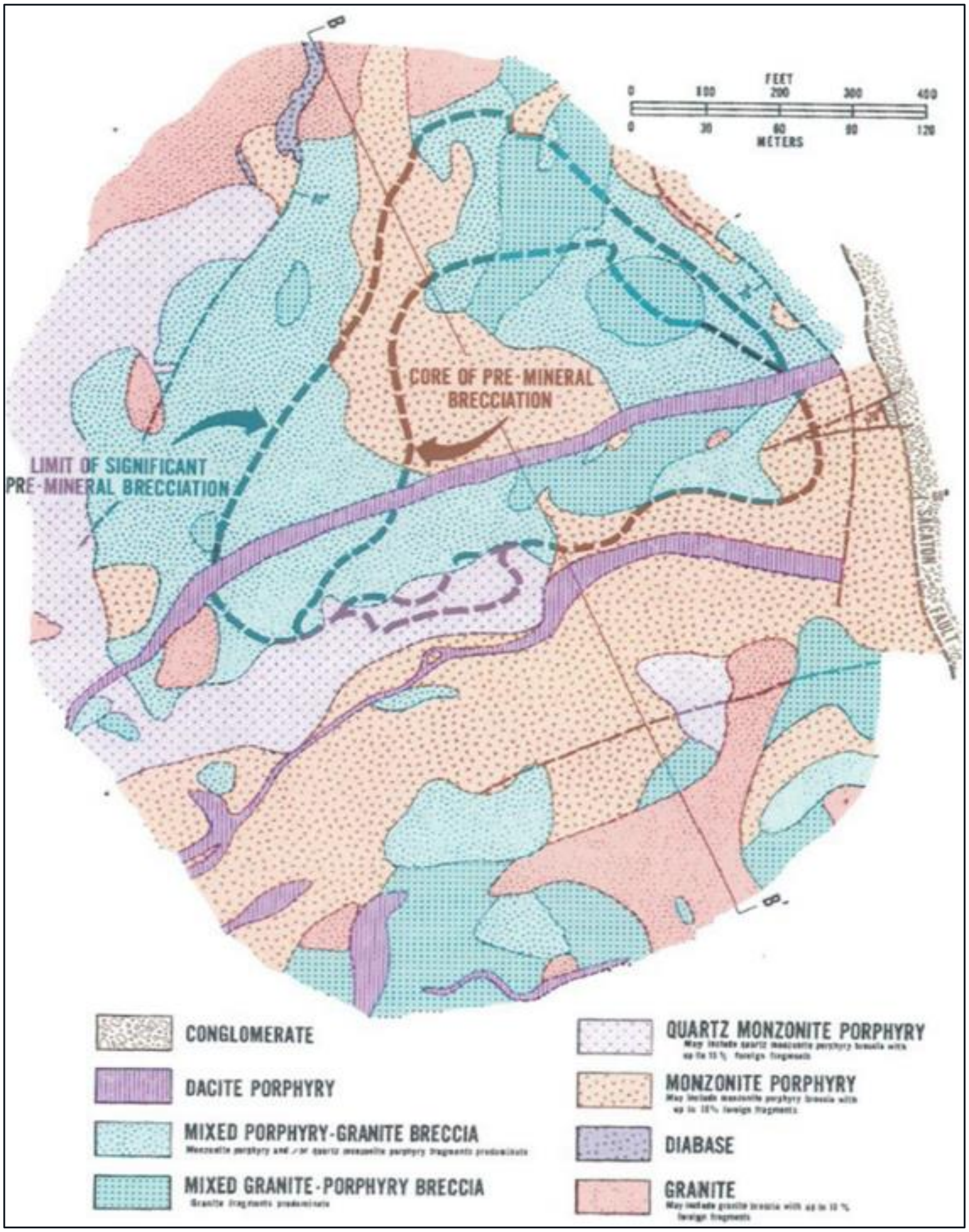
At a location removed from the current mine, Laramide porphyries of a similar composition intruded the Oracle Granite and introduced hydrothermal solutions which altered and mineralized a large area of the surrounding rocks. Subsequent Tertiary extension rotated and dismembered the mineralized rocks. A low angle listric fault (the Basement fault) moved the Cactus 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 ASCU, is located 1.3 mi (2.1 km) 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.

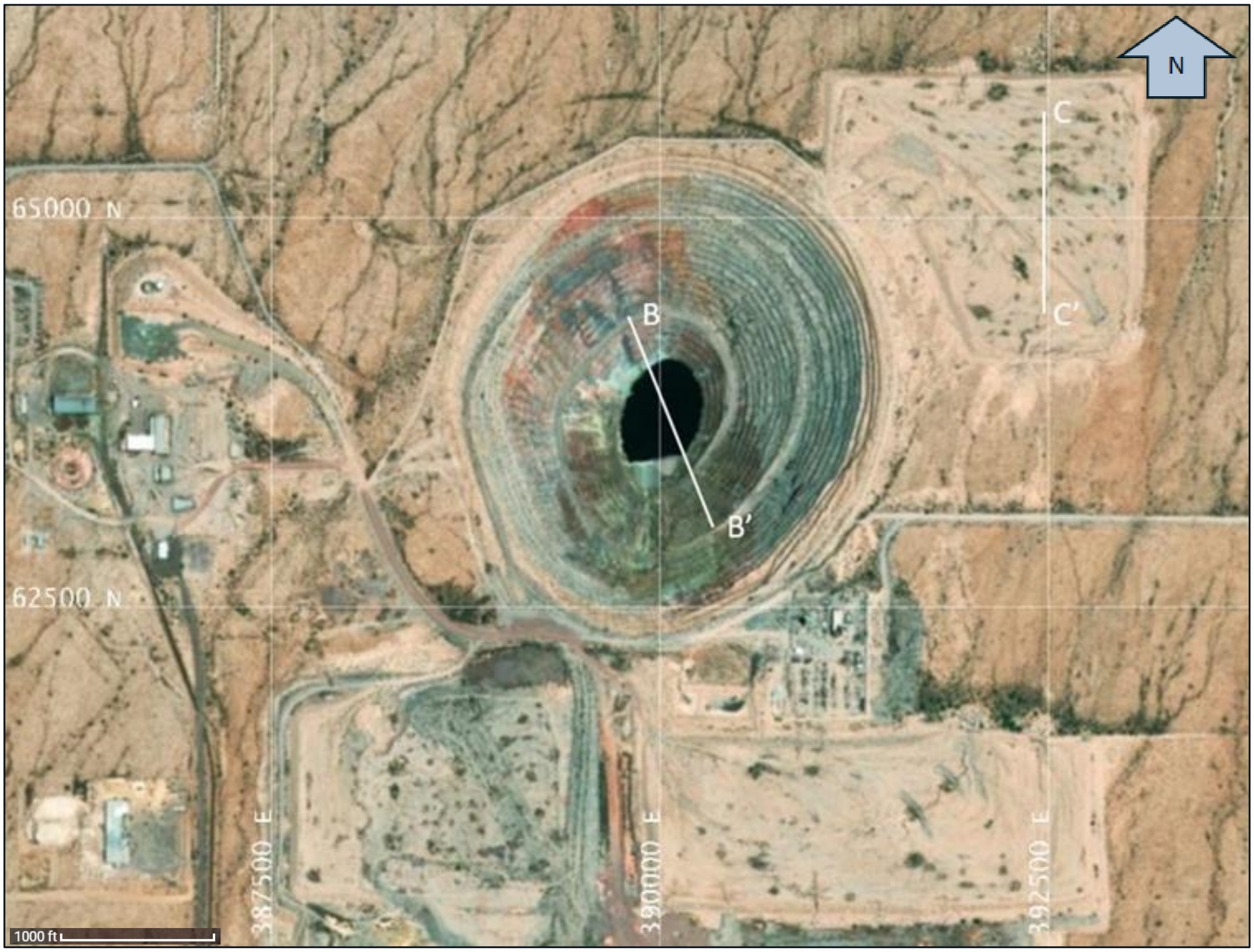
They also form monolithic breccias and mixed breccias containing varying percentages of granite. Discontinuous pre-mineral diabase and post-mineral dacite porphyry dikes intrude the older rocks in both deposits. Figure 7-2 through Figure 7-5 show the rock type distribution of the geological units in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

**Figure 7-2: Plan View through the Cactus West Deposit on the 1,040 ft (317 m) Elevation**



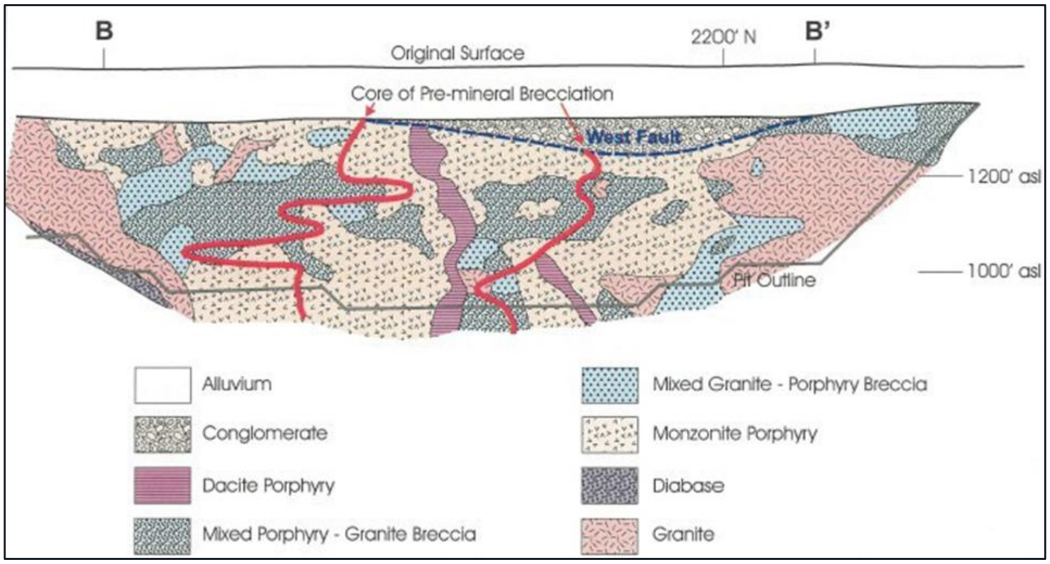

Source: ASCU, 2020.

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



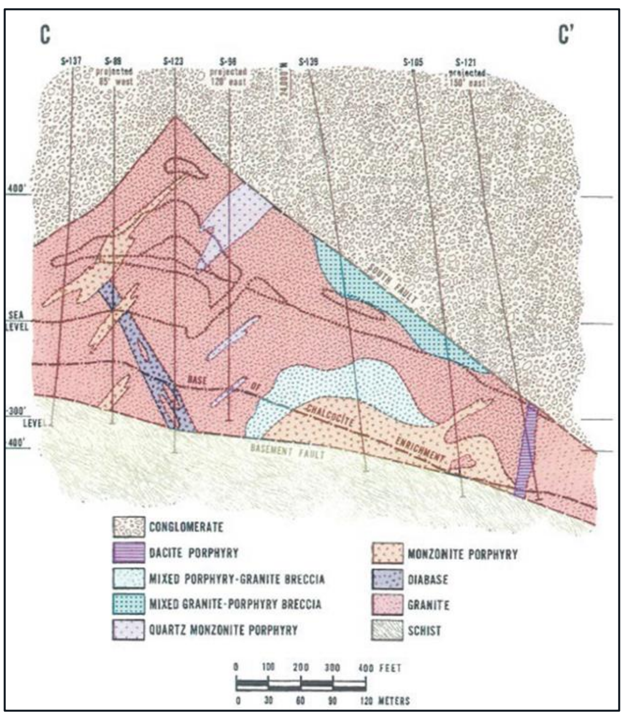

Source: ASCU, 2020.

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




Source: ASCU, 2020.

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




Source: ASCU, 2020.

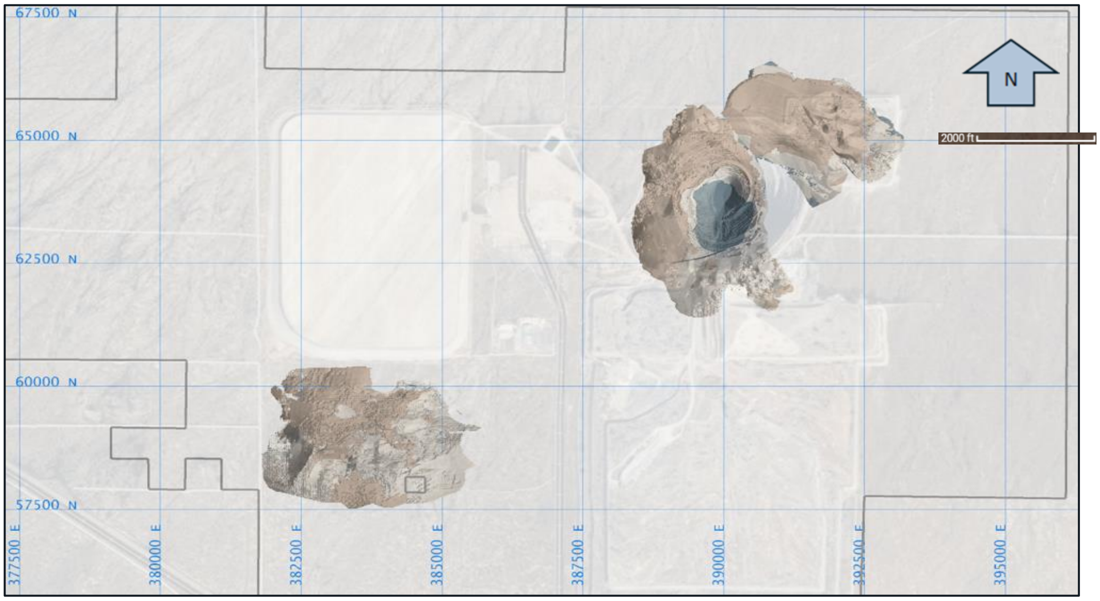
Structurally both deposits are complex with intense fracturing, faulting, and brecciation. Pre- mineral brecciation is related to the intrusion of the Laramide porphyries and occurs primarily in the west deposit which had a central core of pre-mineral brecciation that was a control for hypogene mineralization. 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 in (0.5 cm) to 2.0 in (5.0cm). 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 sulphides. 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 (30 m). 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 ASCU, is located 1.3 mi (2.1 km) 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 insitu 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.**




Source: ASCU, 2022.

## 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- sulphide 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 sulphides 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 sulphide 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 potassic 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 sulphide 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 sulphide minerals at Cactus are pyrite, chalcopyrite, and molybdenite. Traces of bornite and sphalerite have been observed in concentrate samples. Hypogene sulphides occur as disseminated grains, veins, and vug fillings. Disseminated sulphides 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 sulphides, but in the East mineralized zone, where granite breccia is the main rock type, disseminated grains account for over 50% of the sulphides.

The major hypogene sulphide 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 sulphides occur as disseminated grains, veins/veinlets, and patchy blebs. Disseminated sulphides are abundant in the brecciated rocks, Monzonite porphyry, and in the granite.

Sulphides are also present within quartz veins and veinlets throughout the deposit. Disseminated sulphides account for roughly 50% of the hypogene sulphides within the site, but in zones of intensely brecciated porphyries, disseminated grains appear comprise of less than 50% of the sulphides, instead favoring veinlets and patches.

The total sulphide 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 sulphide content. North and south of the mineralized zones the total sulphide 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% Cu as chalcopyrite. Outside the zone the copper grade gradually drops off to less than 0.10% Cu. 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 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 sulphide 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 sulphides averaging at least 0.40% Cu.

The major supergene sulphide 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 which is controlled by a NE trending structural zone containing a higher density of quartz/sulphide veining.

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 DEPOSIT TYPES

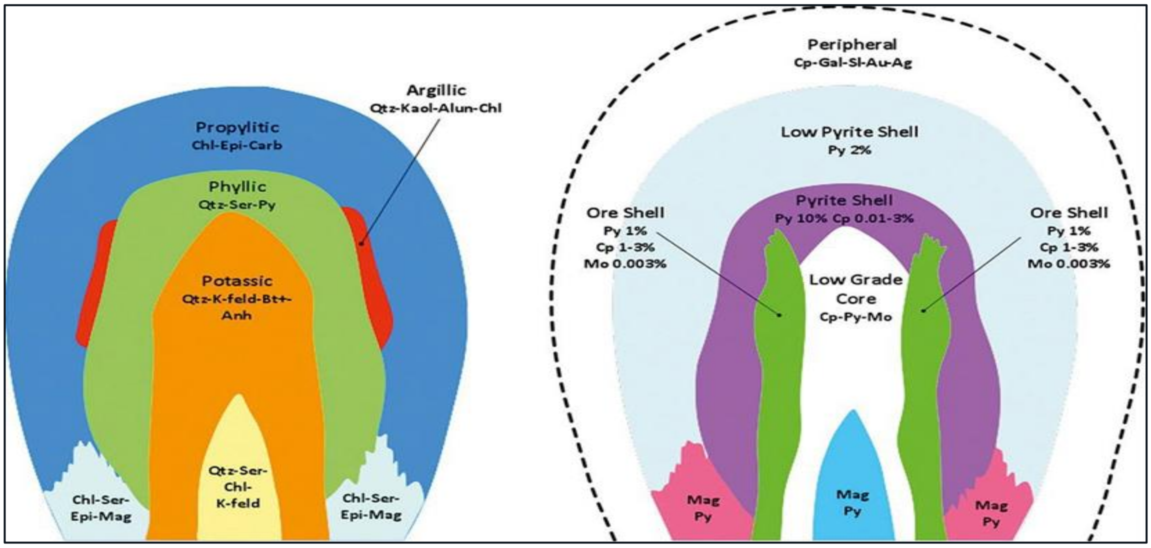
The Cactus and Parks/Salyer deposits are a portion of a large porphyry copper system that has been dismembered and displaced by Tertiary extensional faulting. Porphyry copper deposits form in areas of shallow magmatism within subduction-related tectonic environments (Berger et al., 2008). Both Cactus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

- A deposit wherein copper-bearing sulphides are localized in a network of fracture- controlled stockwork veinlets and as disseminated grains in the adjacent altered rock matrix.
- Alteration and mineralization at 0.6 mi (1 km) to 2.5 mi (4 km) depth are genetically related to magma reservoirs emplaced into the shallow crust 3.5 mi (6 km) to over 5 mi (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). Sulphides 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 sulphide minerals and typical percentages.

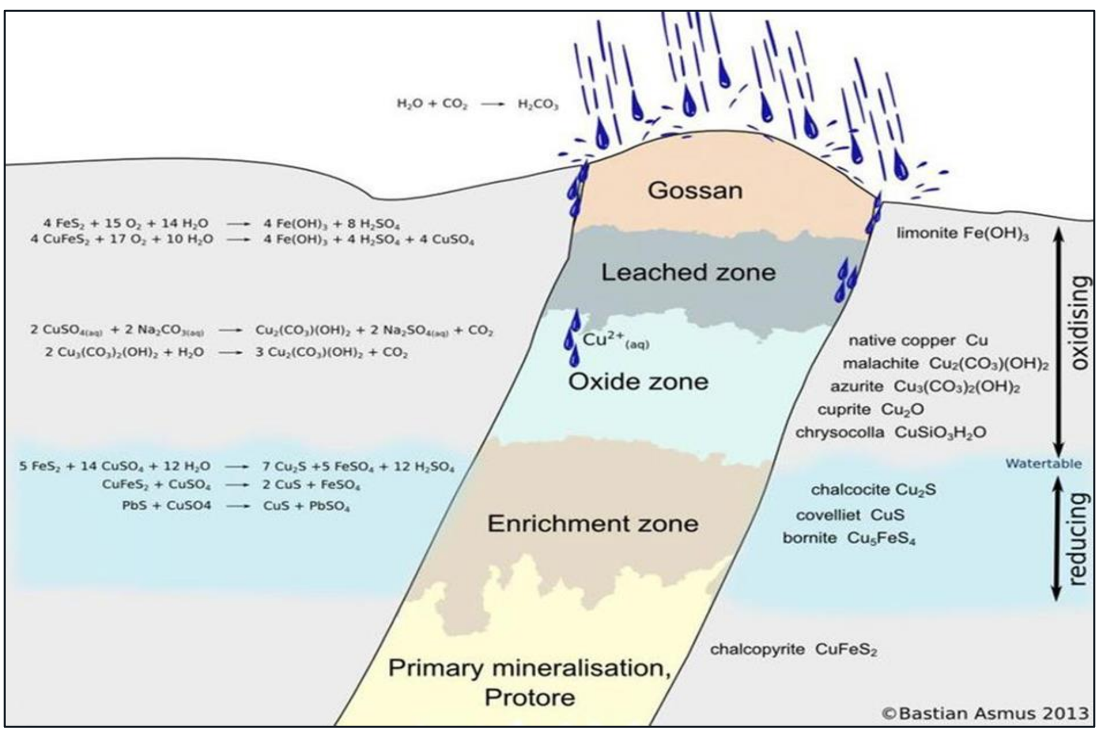
Uplift of the porphyry system to shallow depths can result in secondary enrichment processes where copper is leached from the weathering of hypogene mineralization and redeposited below the water table as supergene copper sulphides such as chalcocite and covellite. Above the water table, copper oxide 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 8-1: Deposit Model of a Porphyry Copper Deposit**




Source: ASCU, 2020. (Modified from Lowell and Guilbert,1970)

**Figure 8-2: Schematic Cross Section of a Porphyry Copper Deposit and Typical Copper Minerals Present**




Source: Asmus, 2013.

# 9 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 sulphide 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 mi (6.4 km) long and 1.5 mi (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.

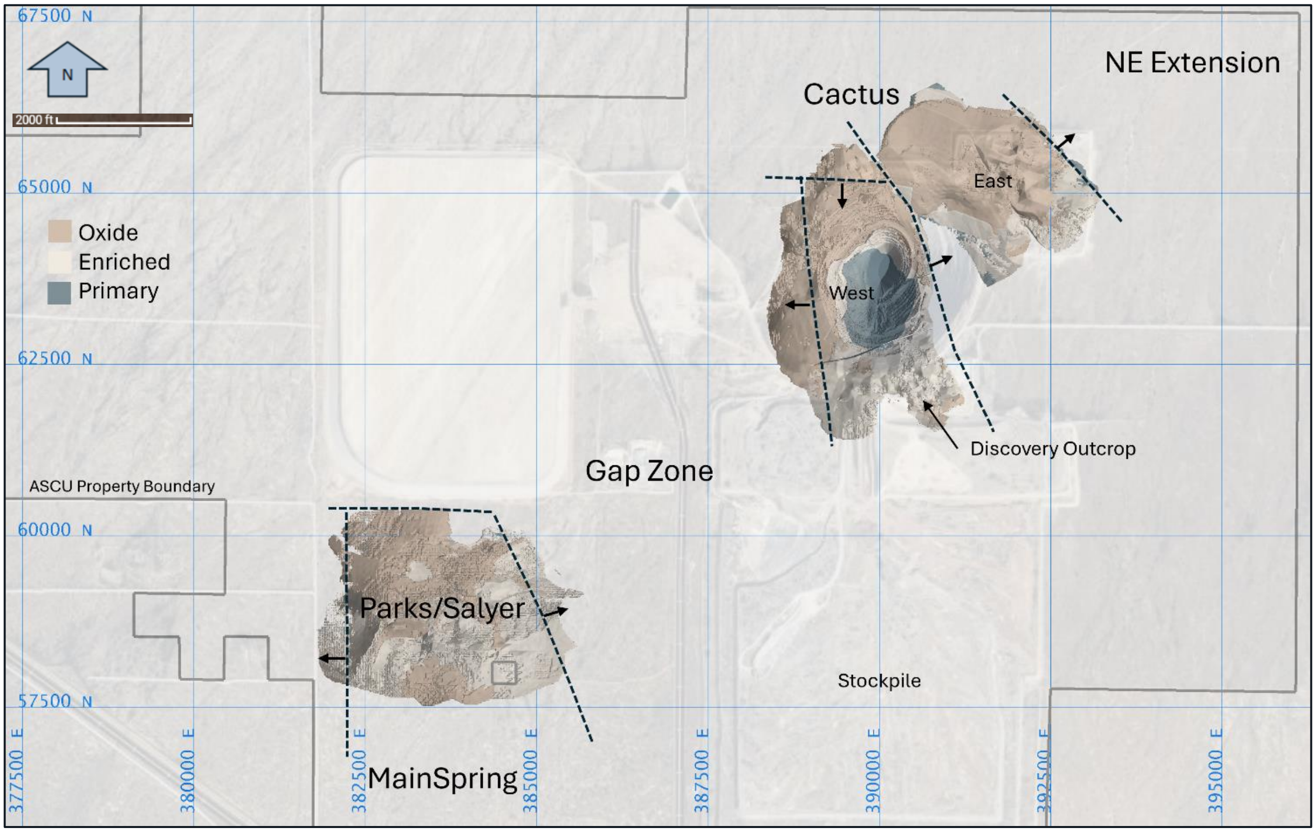
Improving market conditions prompted ASARCO to continue exploration drilling in 1968 and 1969 leading to thirty-seven more holes being drilled. The additional information led to the decision to plan and develop the mine. An additional 10 holes were drilled (1970 and 1971) to sterilize areas under planned facilities. After mining was initiated in 1972, development and definition drilling were 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.

ASCU conducted an ionic leach soil geochemistry program over the Parks/Salyer property in 2019 on 325 ft 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. ASCU 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 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-144, resulting in a total 0f 75 holes totaling 166,658.8 ft of HQ core.

Figure 9-1 plots the location and scale of the potential Parks/Salyer deposit with respect to the Cactus Mine deposits.

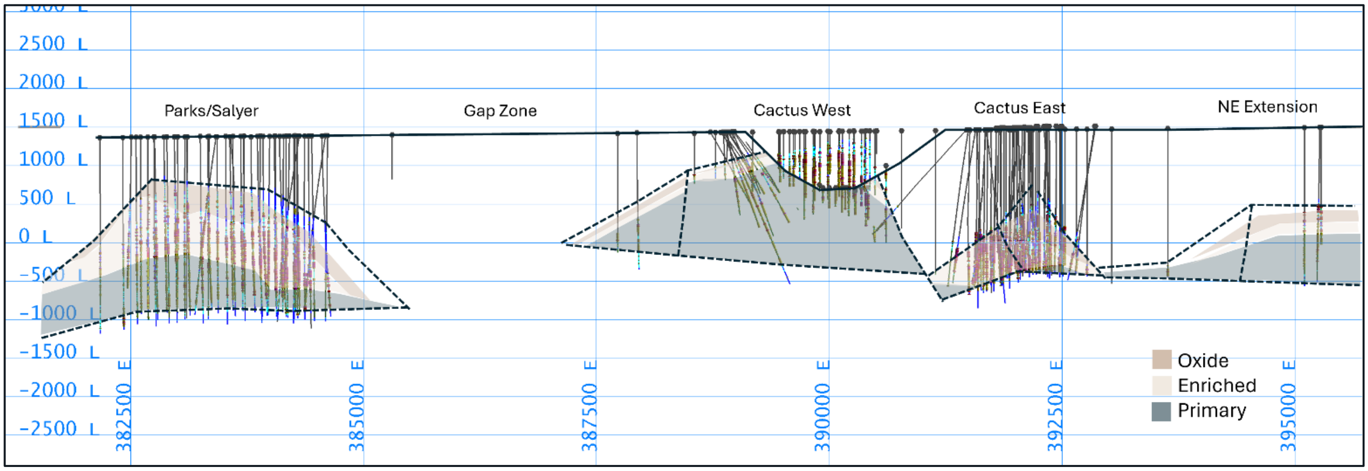
**Figure 9-1: Location and Scale of the Potential Parks/Salyer Deposit with Respect to the Cactus Mine Deposits**




Source: ASCU, 2022.

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 existing Cactus West pit is displayed on the long section.

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




Source: ASCU, 2022.

The NE Extension is located 3,000 ft to the NE of Cactus East. ASARCO defined the mineralized zone with wide spaced exploration drilling (> 1,000 ft) 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. ASCU drilled one exploratory drill hole into the NE Extention in January 2023, intersecting mineralization consistent with previous drilling.

ASCU has continued their exploration by way of definition and expansion core drilling around the Cactus East and West deposits. 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. In 2020, a drilling program was initiated on both Cactus West and Cactus East to expand the deposit extents to define an initial resource that would support a PEA. This consisted of five angled HQ core holes totalling 9,252 ft around the northern and western edges of Cactus East and 11 angled HQ core holes totalling 15,377 ft around the perimeter of the West Pit.

With the successful completion of the PEA incorporating Cactus East and Cactus West, a significant infill drilling program was undertaken throughout 2021 and 2022 to upgrade previous inferred resources to Indicated. Whilst not targeting primary mineralization, this program also drilled into the upper parts of the primary material in Cactus West for exploration purposes that may support future initiatives for mining and processing of primary mineralization. In 2024, ASCU plans to target exploration of the primary mineralization associated with the Cactus West deposit. In 2019, 55 surface sonic drill holes totaling 5,120 ft of 6-in diameter holes were drilled across the Cactus Stockpile Project to support an initial resource based on approximately 750 ft spaced drilling. Through late 2020, 2021 and 2022, an infill surface sonic drill program was undertaken to reduce the spacing to 200 ft. The resource database for the Stockpile Project contains 511 holes for a total of 44,348.2 ft. of drilling.

At present there three defined mineralized deposits on the Cactus Project. Cactus East and Cactus West Combined are approxiametly 4,300 ft (1,300 m) long, average about 3,000 ft (914 m) wide and 1,300 ft (395 m) thick. Parks/Salyer averageses 4,000 ft (1,220 m) long, 3,300 ft (1,005 m) wide, and 2,100 ft (640 m) thick. The stockpile is 4,000 ft (1,220 m) long, 4,500 ft (1,370 m) wide, and 120 ft (36 m) high. The size of the stockpile if fixed however, there is potential to increal the size of both CactusEast, Cactus West, and Parks Salyer.

**Table 9-1: Significant Intercepts for the Three Holes Drilled into the NE Extension Mineralization**

| Hole id | MinZone | From (ft) | To (ft) | Length (ft) | CuT (%) | Tsol (%) |
|---|---|---|---|---|---|---|
| ECN-128 | oxide | 996.7 | 1,114.8 | 118.1 | 0.97 | 0.94 |
| | enriched | 1,182.6 | 1,334.0 | 151.4 | 0.46 | 0.38 |
| | including | 1,334.0 | 1,206.4 | 23.8 | 1.35 | 1.34 |
| | primary | 1,334.0 | 1,987.4 | 653.4 | 0.40 | 0.03 |
| | including | 1,419.0 | 1,469.0 | 50.0 | 0.55 | 0.04 |
| | and | 1,510.0 | 1,629.0 | 119.0 | 0.58 | 0.04 |
| | and | 1,733.3 | 1,752.3 | 19.0 | 1.60 | 0.10 |
| S-68 | oxide | 1,016.50 | 1,044.50 | 28 | 1.27 | n/a |
| | oxide | 1,078.50 | 1,125.80 | 47.3 | 0.95 | n/a |
| | oxide | 1,161.00 | 1,208.80 | 47.8 | 3.05 | n/a |
| | enriched | 1,275.00 | 1,290.10 | 15.1 | 1.96 | n/a |
| | enriched | 1,322.40 | 1,354.10 | 31.7 | 0.97 | n/a |
| | primary | 1,354.10 | 1,526.00 | 171.9 | 0.38 | n/a |
| S-64 | oxide | 1,093.90 | 1,104.20 | 10.3 | 1.01 | n/a |
| | enriched | 1,163.00 | 1,227.30 | 64.3 | 1.37 | n/a |
| | enriched | 1,333.70 | 1,350.90 | 17.2 | 0.89 | n/a |
| | primary | 1,350.90 | 1,776.00 | 425.1 | 0.34 | n/a |

# 10 DRILLING

## 10.1 Introduction

The Cactus (Sacaton) deposits are covered with post mineral alluvium and conglomerate, which may be up to 1,500 ft 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. ASCU 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%.

When Elim Mining (now ASCU) acquired the Sacaton Mine property in 2019 they found the offices and warehouses containing desks and file cabinets filled with disorganized files and data sheets. There were 2 coresheds full of boxed core, samples, and sample pulps. The data was organized and paired with the physical core and samples in the coresheds to build a database of historical drilling from 1961 to the early 1980s.

Each drill hole was reviewed in turn and the associated data and samples validated to ensure that in total, the hole met CIM Best Practices Guidelines for inclusion on a NI 43-101 Technical Report. In total 175 RC and Diamond drill holes were validated and used for subsequent MREs. Drilling completed by ASCU since has been consistent with these original data.

As detailed in Table 10-1, ASCU completed a total of 184 core holes in the Cactus resource area in 2019 through 2023 for a total of 57,262.2 ft of drilling. Table 10-2 details the 74 drillholes undertaken by ASCU in the Parks/Salyer resource area in 2021 through 2023 for a total of 166,685 ft 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 ASCU's recent holes, and white circles locating the collars of ASARCO's historical holes.

Of the 184 diamond drill holes completed in the Cactus area, 182 were used for the Cactus Mineral Resource estimates. All 74 holes completed in the Parks/Salyer areas were used for the Parks/Salyer Mineral Resource estimates.

**Table 10-1: 2019–2023 Cactus Drilling Completed by Arizona Sonoran**

| Drill Hole | Core | Total Depth (ft) | Total Depth (m) | Azimuth | Dip | Deposit |
|---|---|---|---|---|---|---|
| ECE-001 | HQ | 1896.0 | 577.9 | 220 | -80 | CE |
| ECE-002 | HQ | 2013.0 | 613.6 | 230 | -80 | CE |
| ECE-015 | HQ | 1722.5 | 525.0 | 0 | -90 | CE |
| ECE-016 | NQ | 1782.6 | 543.3 | 330 | -80 | CE |
| ECE-017 | HQ | 1837.0 | 559.9 | 260 | -80 | CE |
| ECE-020 | HQ | 1770.2 | 539.6 | 0 | -90 | CE |

| Drill Hole | Core | Total Depth (ft) | Total Depth (m) | Azimuth | Dip | Deposit |
|---|---|---|---|---|---|---|
| ECE-021 | HQ | 1948.7 | 594.0 | 0 | -90 | CE |
| ECE-043 | HQ | 2054.8 | 626.3 | 235 | -80 | CE |
| ECE-044 | HQ | 1917.0 | 584.3 | 0 | -90 | CE |
| ECE-051 | HQ | 1956.0 | 596.2 | 0 | -90 | CE |
| ECE-052 | HQ | 1871.4 | 570.4 | 0 | -90 | CE |
| ECE-053 | HQ | 2035.2 | 620.3 | 0 | -90 | CE |
| ECE-058 | HQ | 1903.0 | 580.0 | 0 | -90 | CE |
| ECE-059 | HQ | 1368.6 | 417.1 | 0 | -90 | CE |
| ECE-059A | HQ | 1906.3 | 581.0 | 0 | -90 | CE |
| ECE-060 | HQ | 1936.5 | 590.2 | 0 | -90 | CE |
| ECE-062 | HQ | 1888.0 | 575.5 | 0 | -90 | CE |
| ECE-063 | HQ | 1976.3 | 602.4 | 0 | -90 | CE |
| ECE-064 | HQ | 1924.3 | 586.5 | 0 | -90 | CE |
| ECE-066 | HQ | 1947.0 | 593.4 | 0 | -90 | CE |
| ECE-067 | HQ | 1897.8 | 578.4 | 0 | -90 | CE |
| ECE-069 | HQ | 1878.8 | 572.7 | 0 | -90 | CE |
| ECE-070 | HQ | 1948.0 | 593.8 | 0 | -90 | CE |
| ECE-072 | HQ | 2055.0 | 626.4 | 0 | -80 | CE |
| ECE-073 | HQ | 2103.0 | 641.0 | 0 | -90 | CE |
| ECE-076 | HQ | 1930.0 | 588.3 | 360 | -80 | CE |
| ECE-078 | HQ | 2093.0 | 637.9 | 360 | -80 | CE |
| ECE-082 | HQ | 2314.1 | 705.3 | 0 | -90 | CE |
| ECE-085 | HQ | 2117.0 | 645.3 | 0 | -90 | CE |
| ECE-143 | HQ | 2274.0 | 693.1 | 90 | -80 | CE |
| ECN-128 | HQ | 2013.6 | 613.7 | 0 | -90 | NE |
| ECW-003 | HQ | 1936.0 | 590.1 | 180 | -60 | CW |
| ECW-004 | HQ | 500.0 | 152.4 | 0 | -60 | CW |
| ECW-005 | HQ | 664.2 | 202.4 | 130 | -60 | CW |
| ECW-006 | HQ | 1000.2 | 304.9 | 10 | -60 | CW |
| ECW-007 | HQ | 1810.5 | 551.8 | 125 | -55 | CW |
| ECW-008 | HQ | 1000.0 | 304.8 | 15 | -65 | CW |
| ECW-009 | HQ | 906.0 | 276.1 | 30 | -60 | CW |
| ECW-010 | HQ | 1469.2 | 447.8 | 110 | -65 | CW |
| ECW-011 | HQ | 1329.0 | 405.1 | 60 | -65 | CW |
| ECW-012 | HQ | 1459.6 | 444.9 | 70 | -65 | CW |
| ECW-013 | HQ | 1616.0 | 492.6 | 205 | -60 | CW |

| Drill Hole | Core | Total Depth (ft) | Total Depth (m) | Azimuth | Dip | Deposit |
|---|---|---|---|---|---|---|
| ECW-014 | HQ | 1687.4 | 514.3 | 160 | -50 | CW |
| ECW-022 | HQ | 1304.6 | 397.6 | 90 | -45 | CW |
| ECW-023 | HQ | 1396.0 | 425.5 | 90 | -55 | CW |
| ECW-024 | HQ | 1011.0 | 308.2 | 80 | -50 | CW |
| ECW-025 | HQ | 1049.3 | 319.8 | 70 | -60 | CW |
| ECW-026 | HQ | 944.0 | 287.7 | 70 | -55 | CW |
| ECW-027 | HQ | 1540.0 | 469.4 | 90 | -60 | CW |
| ECW-028 | HQ | 1300.0 | 396.2 | 94 | -55 | CW |
| ECW-029 | HQ | 1094.0 | 333.5 | 70 | -80 | CW |
| ECW-030 | HQ | 458.0 | 139.6 | 190 | -60 | CW |
| ECW-031 | HQ | 1828.6 | 557.4 | 240 | -45 | CW |
| ECW-032 | HQ | 1367.7 | 416.9 | 140 | -50 | CW |
| ECW-033 | HQ | 1418.0 | 432.2 | 140 | -45 | CW |
| ECW-034 | HQ | 1347.0 | 410.6 | 140 | -45 | CW |
| ECW-035 | HQ | 1008.0 | 307.2 | 135 | -45 | CW |
| ECW-036 | HQ | 1443.0 | 439.8 | 135 | -55 | CW |
| ECW-037 | HQ | 938.3 | 286.0 | 130 | -45 | CW |
| ECW-038 | HQ | 1449.7 | 441.9 | 110 | -65 | CW |
| ECW-039 | HQ | 450.6 | 137.3 | 0 | -90 | CW |
| ECW-040 | HQ | 1287.0 | 392.3 | 110 | -50 | CW |
| ECW-041 | HQ | 1948.0 | 593.8 | 235 | -45 | CW |
| ECW-046 | HQ | 607.0 | 185.0 | 0 | -90 | CW |
| ECW-047 | HQ | 537.0 | 163.7 | 0 | -90 | CW |
| ECW-048 | HQ | 500.0 | 152.4 | 0 | -90 | CW |
| ECW-049 | HQ | 400.0 | 121.9 | 0 | -90 | CW |
| ECW-050 | HQ | 400.0 | 121.9 | 0 | -90 | CW |
| ECW-054 | HQ | 1350.0 | 411.5 | 10 | -55 | CW |
| ECW-055 | HQ | 1600.0 | 487.7 | 100 | -45 | CW |
| ECW-056 | HQ | 1490.4 | 454.3 | 150 | -50 | CW |
| ECW-150 | HQ | 2156.0 | 657.1 | 45 | -65 | CW |
| ECW-153 | HQ | 1874.9 | 571.5 | 175 | -65 | CW |
| SE-01 | PQ | 2058.3 | 627.4 | 0 | -90 | CE |
| SE-02 | PQ | 2013.0 | 613.6 | 0 | -90 | CE |
| **Totals** | | **115,226.2** | **35,120.9** | | | |

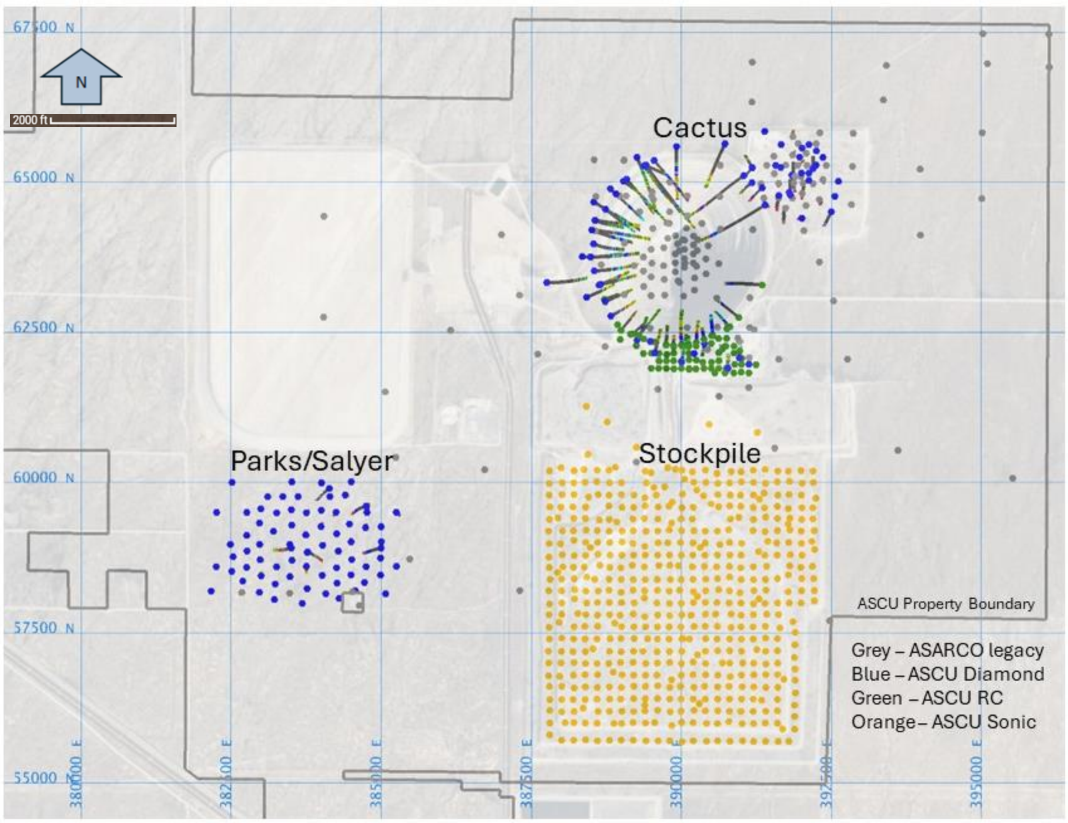
**Table 10-2: 2021–2023 Parks/Salyer Drilling Completed by Arizona Sonoran**

| 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 |
| ECP-096 | HQ | 2,652.1 | 808.4 | 0 | -90 | PS |
| ECP-097 | HQ | 2,344.5 | 714.6 | 0 | -90 | PS |
| ECP-098 | HQ | 2,332.4 | 710.9 | 0 | -90 | PS |
| ECP-099 | HQ | 2,244.0 | 684.0 | 0 | -90 | PS |
| ECP-100 | HQ | 2,157.0 | 657.5 | 0 | -90 | PS |
| ECP-101 | HQ | 2,266.5 | 690.8 | 0 | -90 | PS |
| ECP-102 | HQ | 2,252.4 | 686.5 | 0 | -90 | PS |
| ECP-103 | HQ | 2,060.3 | 628.0 | 0 | -90 | PS |

| Drill Hole | Core | Total Depth (ft) | Total Depth (m) | Azimuth | Dip | Deposit |
|---|---|---|---|---|---|---|
| ECP-104 | HQ | 1,948.0 | 593.8 | 0 | -90 | PS |
| ECP-105 | HQ | 2,067.0 | 630.0 | 0 | -90 | PS |
| ECP-106 | HQ | 1,979.6 | 603.4 | 0 | -90 | PS |
| ECP-107 | HQ | 2,207.0 | 672.7 | 0 | -90 | PS |
| ECP-108 | HQ | 1,957.5 | 596.6 | 0 | -90 | PS |
| ECP-109 | HQ | 2,233.0 | 680.6 | 0 | -90 | PS |
| ECP-110 | HQ | 1,910.5 | 582.3 | 0 | -90 | PS |
| ECP-111 | HQ | 2,335.5 | 711.9 | 0 | -90 | PS |
| ECP-112 | HQ | 2,076.5 | 632.9 | 0 | -90 | PS |
| ECP-113 | HQ | 2,397.0 | 730.6 | 0 | -90 | PS |
| ECP-114 | HQ | 2,252.4 | 686.5 | 0 | -90 | PS |
| ECP-115 | HQ | 2,424.5 | 739.0 | 0 | -90 | PS |
| ECP-116 | HQ | 2,213.7 | 674.7 | 0 | -90 | PS |
| ECP-117 | HQ | 2,444.4 | 745.1 | 0 | -90 | PS |
| ECP-118 | HQ | 2,138.0 | 651.7 | 0 | -90 | PS |
| ECP-119 | HQ | 2,406.0 | 733.3 | 0 | -90 | PS |
| ECP-120 | HQ | 1,949.2 | 594.1 | 0 | -90 | PS |
| ECP-121 | HQ | 2,477.0 | 755.0 | 0 | -90 | PS |
| ECP-122 | HQ | 1,857.6 | 566.2 | 0 | -90 | PS |
| ECP-123 | HQ | 2,377.0 | 724.5 | 0 | -90 | PS |
| ECP-124 | HQ | 2,434.2 | 741.9 | 0 | -90 | PS |
| ECP-125 | HQ | 2,039.4 | 621.6 | 0 | -90 | PS |
| ECP-126 | HQ | 2,151.6 | 655.8 | 0 | -90 | PS |
| ECP-127 | HQ | 2,427.0 | 739.7 | 0 | -90 | PS |
| ECP-129 | HQ | 2,316.0 | 705.9 | 255 | -80 | PS |
| ECP-130 | HQ | 2,367.7 | 721.7 | 0 | -90 | PS |
| ECP-131 | HQ | 2,268.2 | 691.3 | 0 | -90 | PS |
| ECP-132 | HQ | 2,430.0 | 740.7 | 235 | -80 | PS |
| ECP-133 | HQ | 2,417.0 | 736.7 | 0 | -90 | PS |
| ECP-134 | HQ | 2,248.0 | 685.2 | 0 | -90 | PS |
| ECP-135 | HQ | 2,086.0 | 635.8 | 0 | -90 | PS |
| ECP-136 | HQ | 2,420.2 | 737.7 | 0 | -90 | PS |
| ECP-137 | HQ | 2,278.5 | 694.5 | 0 | -90 | PS |
| ECP-138 | HQ | 2,248.0 | 685.2 | 115 | -80 | PS |
| ECP-139 | HQ | 2,285.7 | 696.7 | 0 | -90 | PS |
| ECP-140 | HQ | 2,333.5 | 711.3 | 260 | -80 | PS |

| Drill Hole | Core | Total Depth (ft) | Total Depth (m) | Azimuth | Dip | Deposit |
|---|---|---|---|---|---|---|
| ECP-141 | HQ | 2,290.5 | 698.1 | 0 | -90 | PS |
| ECP-142 | HQ | 2,378.2 | 724.9 | 0 | -90 | PS |
| ECP-144 | HQ | 2,429.2 | 740.4 | 0 | -90 | PS |
| **Totals** | | **166,685.3** | **50,805.7** | | | |

**Figure 10-1: Map Showing Collar Locations of Historical and Recent Drilling Campaigns**




Source: ASCU, 2022.

The Stockpile Project has been infilled drilled by ASCU to 200 ft spacing by sonic surface drilling since the initial 750 ft spacing completed in 2019. This accounts for 511 holes in addition to four historical sterilization holes drilled into the barren alluvium dumps to the immediate north of the Stockpile Project.

## 10.2 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.

## 10.3 Downhole Surveying

All ASCU'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 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; because the hole depth averaged approximately 80 ft, downhole surveys were not deemed necessary.

## 10.4 Core Logging and Photography

Core logging was performed in ASCU'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**

Source: ALS Georesources, 2022.

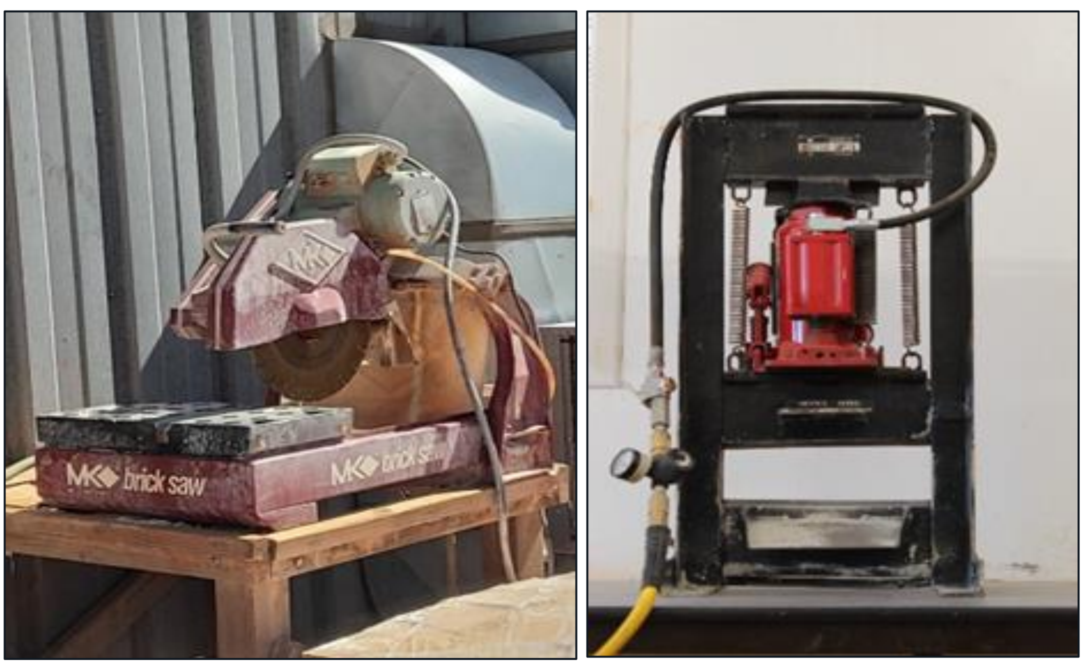
Core sample intervals are determined by the logging geologist based on logging characteristics. Sample interval breaks are determined by geological parameters, but within core containing the same geological characteristics, samples are undertaken on a regular 10 ft sample length.

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 ASCU'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**




Source: ASCU, 2022.

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

Source: ASCU, 2022.

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 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.

## 10.5 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 PFS report.

# 11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

## 11.1 Sample Preparation

ASCU has been exclusively using Skyline Assayers and Laboratories (Skyline Labs), in Tucson, Arizona, for their sample preparation and analysis. This lab is independent of ASCU and any of its subsidiaries. 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/quality control (QA/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 220ºF (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. If the sample was multi-colored the sample is re-mix and split until the color in homogeneous. Three-quarters of the sample was then bagged, labelled, and returned to ASCU as coarse reject. The remaining material was returned to the roll crushers and crushed to 95% passing -10 mesh. A 280-g sample of this material was put in a glass jar sealed, labelled, and returned to ASCU. A 50-gram sample from the same trays as the jarred sample is put in 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.

At each step after crushing and pulverizing, every 20$^{th}$ sample is tested with a sieve to ensure that it meets requirements. The results of these tests are entered int a log and initialed by the operator. This log is kept up to date and is available for review by senior staff and the project QP.

## 11.2 Sample Security

Bagged samples with identification tags are placed in large 3 ft square plastic totes which are stored at the core shed which is within the secured mine site away from any point of access. ASCU 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 ASCU upon dispatch, signed by Skyline Labs upon arrival, and returned to ASCU 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 ($H_2SO_4$) 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 a 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 ASCU together with the coarse reject.

## 11.4 Lab Quality Assurance/Quality Control

Skyline Labs 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 ASCU 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 ASCU and the QP.

## 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 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 PFS report.

# 12 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, ASCU has drilled 73 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 74 new holes drilled by ASCU between 2021 and 2022. There were only four historical holes supporting the Parks/Salyer resource estimate.

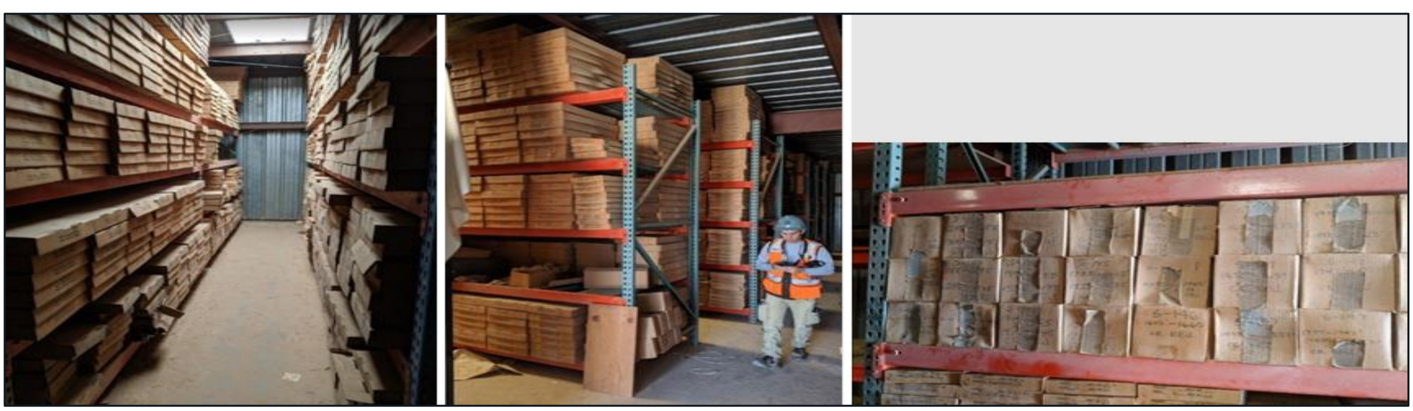
## 12.1 Historical Asarco Exploration Data

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 ASCU 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**



Source: ASCU, 2022.

### 12.2 Historical Collar Locations

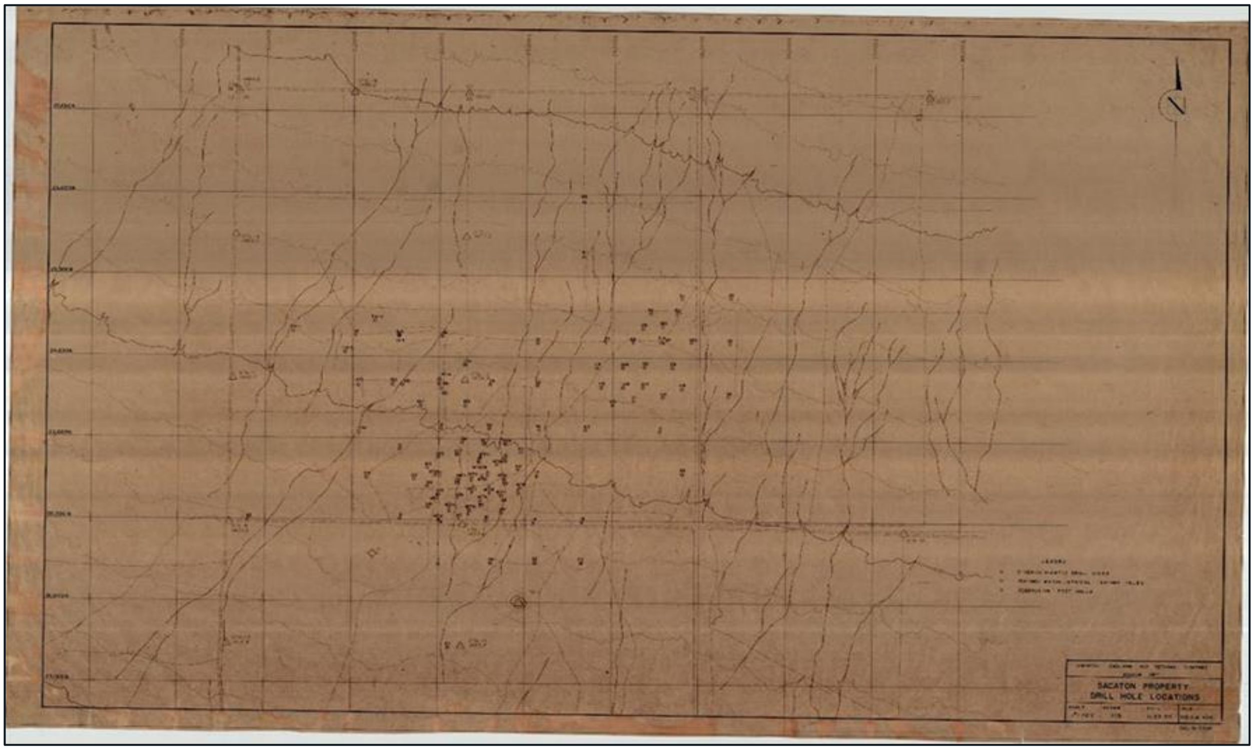
Historical collar locations were verified through the identification of historical survey control and field survey pickup. A final ASARCO control document entitled Sacaton – Drill Hole Files and Information produced in 1998 by Bret S. Canale was located. A page from this document detailed the final collar 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 Coordinates | | Elevation |
| | 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 | - |

Source: ASARCO, 1970.

**Figure 12-3: 1970 Survey Control Map**



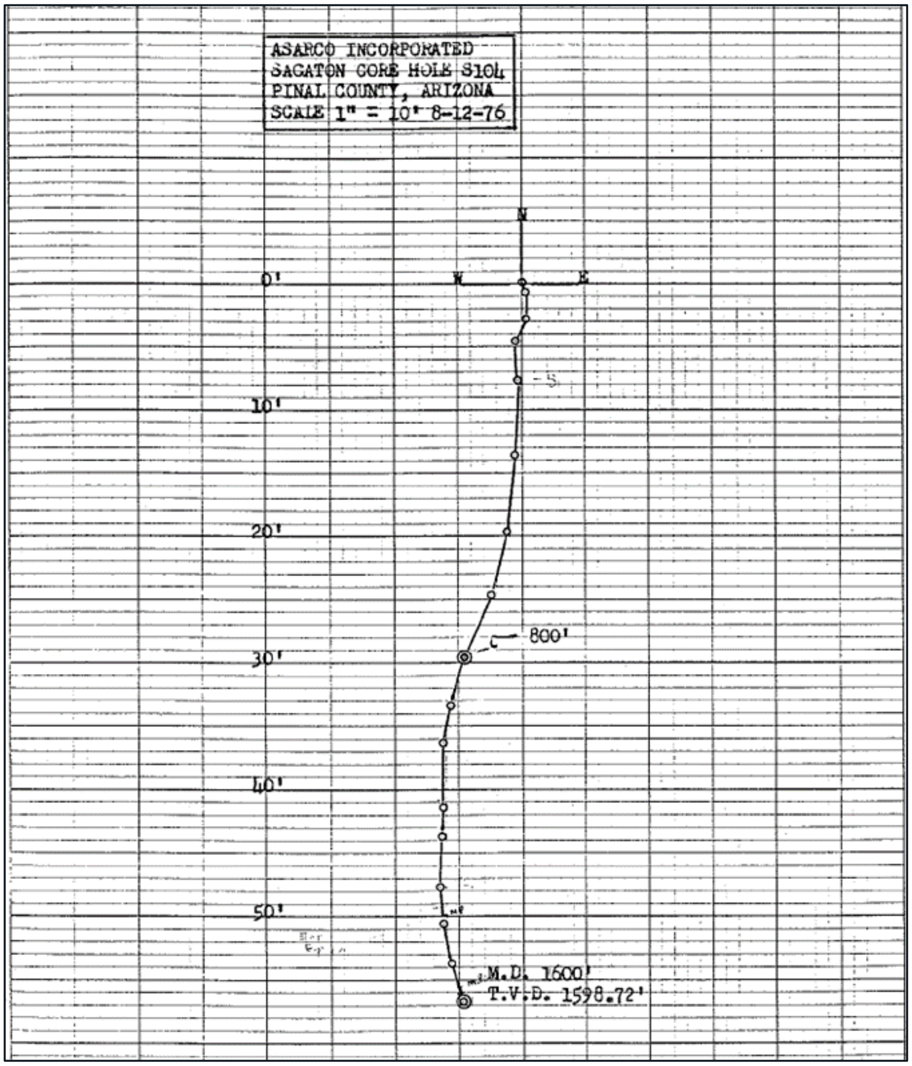

Source: ASARCO, 1970.

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**

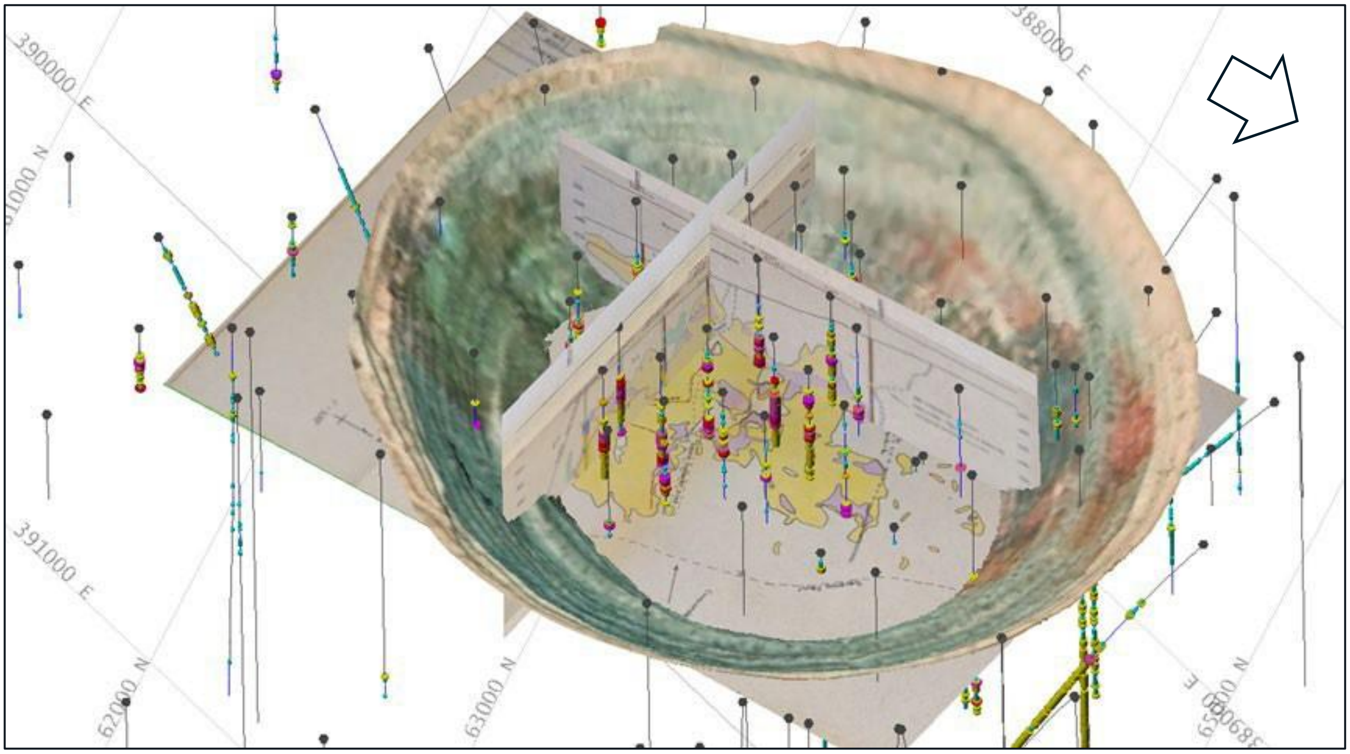



Source: ASCU, 2020.

### 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 ASCU'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**




Source: ASCU, 2020.

### 12.2.3 Relogging of Historical Core

ASCU 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 ASCU. 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.

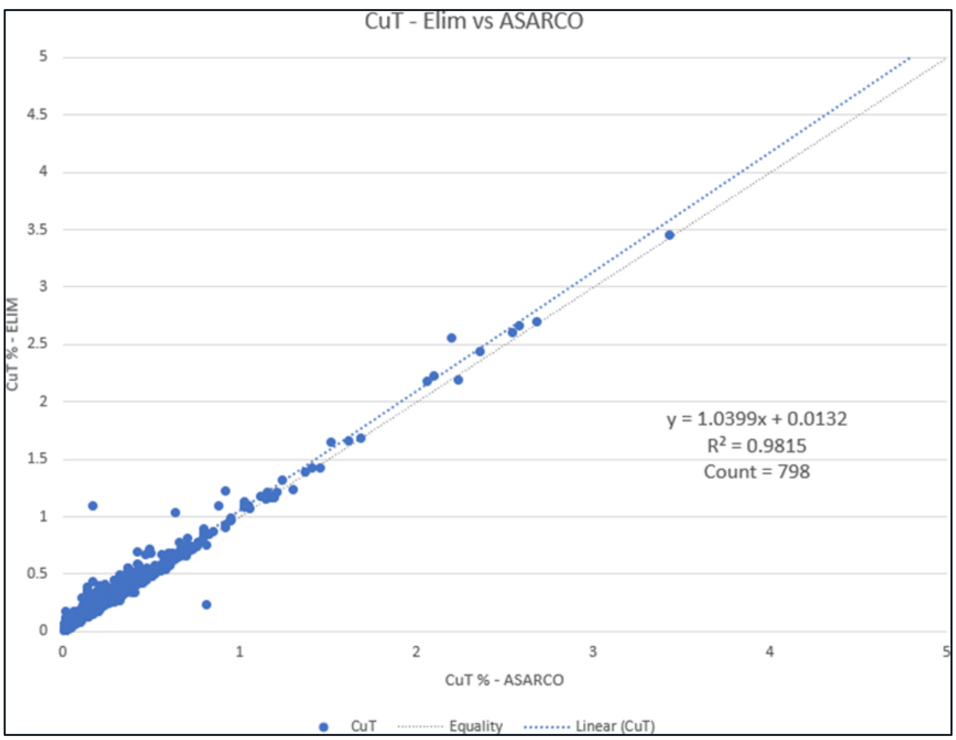
## 12.3 Re-Assaying of Historical Pulps

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 sulphides. 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 sulphide 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**



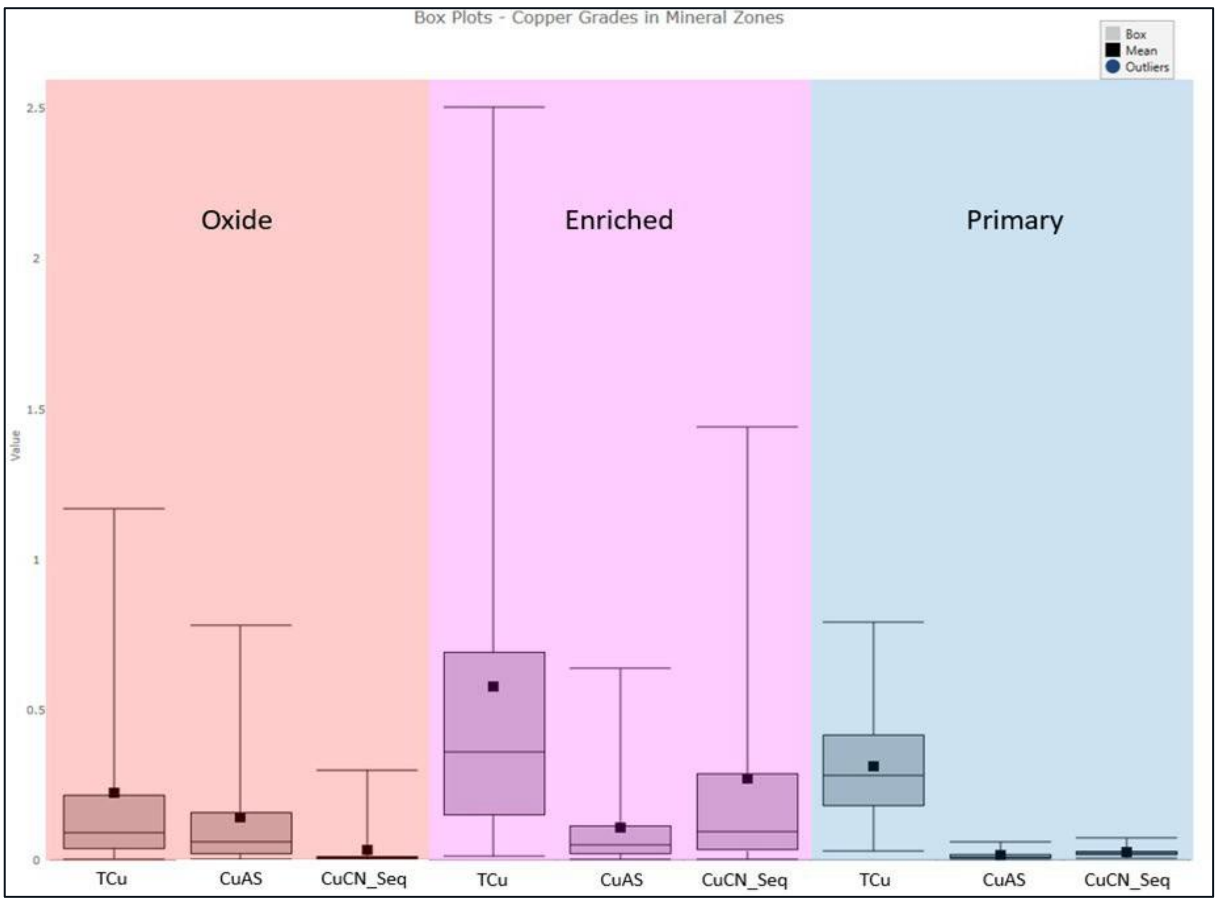

Note: Three-Dimensional View of the Cactus West Pit, Facing Southwest. Source: ASCU, 2022.

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**




Source: ASCU, 2022.

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 sulphides. 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 184 new Cactus drill holes, 74 new Parks/Salyer drillholes, and 511 new Stockpile Project drill holes undertaken by ASCU 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 site visits at the end of the drilling programs 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 184 new Cactus drill holes, 74 new Parks/Salyer drillholes, and 511 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 and Parks/Salyer deposits. Table 12-1 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.

**Table 12-1: Arizona Sonoran Drilling Program Standards and Certified Values**

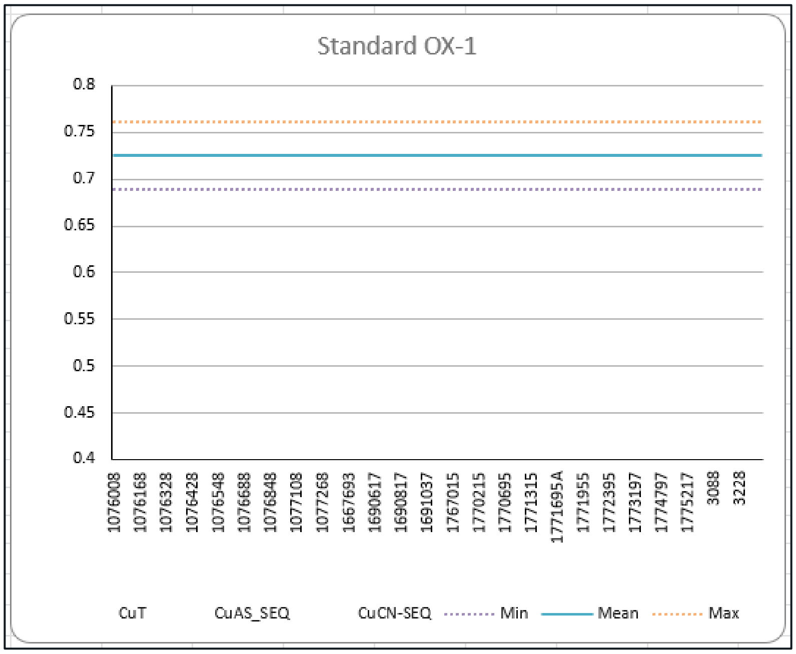
| CRM Code | Sample Decomposition | Analytical Method | Element | Unit | Certified Values | Standard Deviation | 95% Confidence | Minimum Value | Maximum Value |
|---|---|---|---|---|---|---|---|---|---|
| R-Blank | AD | ICP | CuT | % | | | | 0 | 0.015 |
| OX-1 | AD | ICP | CuT | % | 0.725 | 0.043 | 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 | 0.9613 | 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 |

Source: ASCU, 2022.

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%.

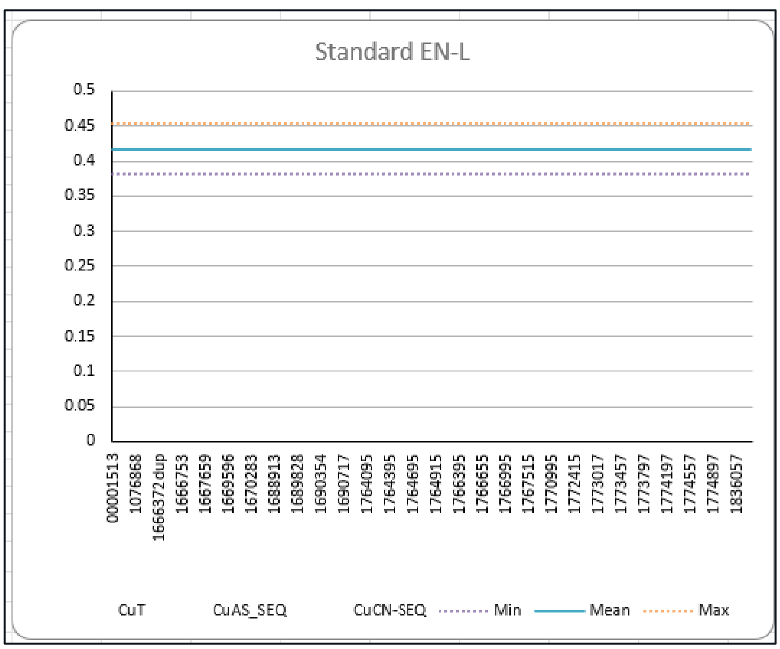
Figure 12-8 through Figure 12-14 show the performance of the standards across both the drilling and re-assay programs supporting the new mineral resource estimates at Cactus West and Cactus East, Parks/Salyer, and the Stockpile Project. All figures plot the expected mean grade (black line), and dashed 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 12-8: Oxide Standard (OX-1)**



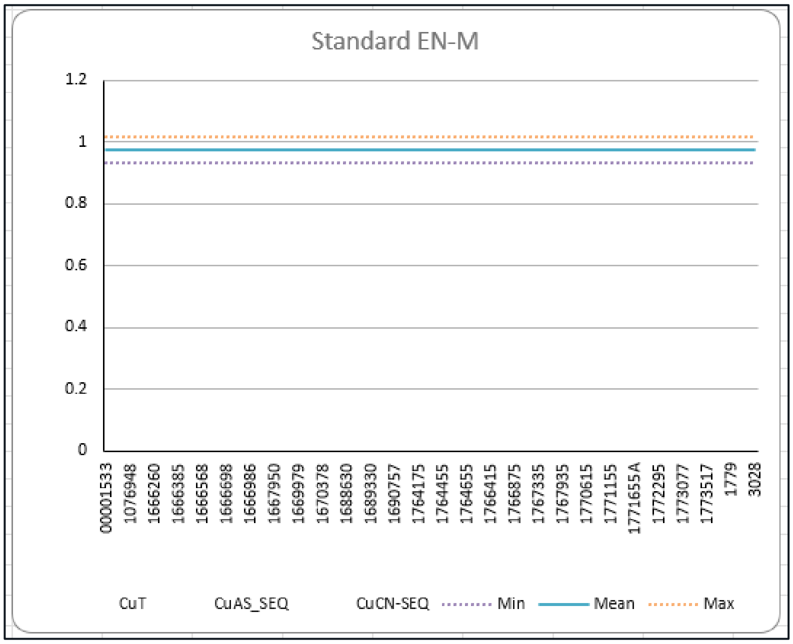

Source: ASCU, 2022.

**Figure 12-9: Enriched Low-Grade Standard (EN-L)**



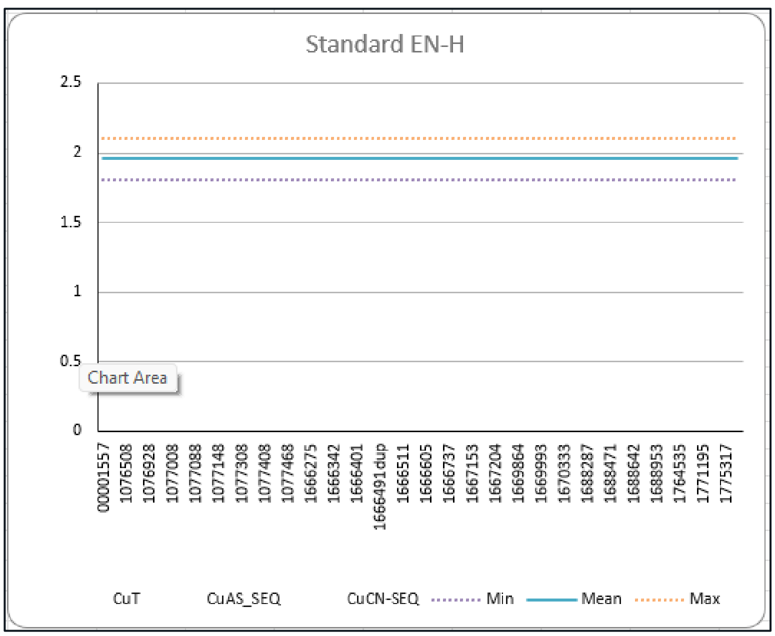

Source: ASCU, 2022.

**Figure 12-10: Enriched Medium Grade Standard (EN-M)**



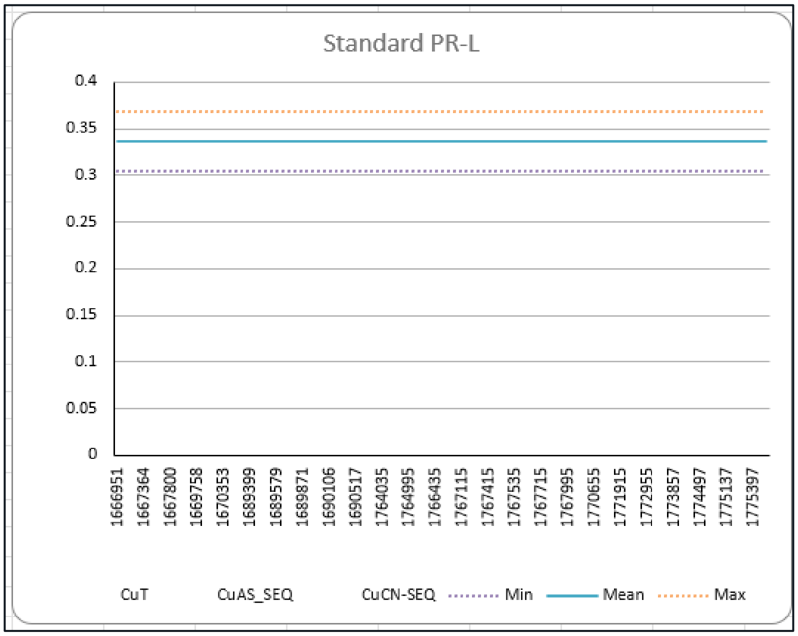

Source: ASCU, 2022.

**Figure 12-11: Enriched High-Grade Standard (EN-H)**



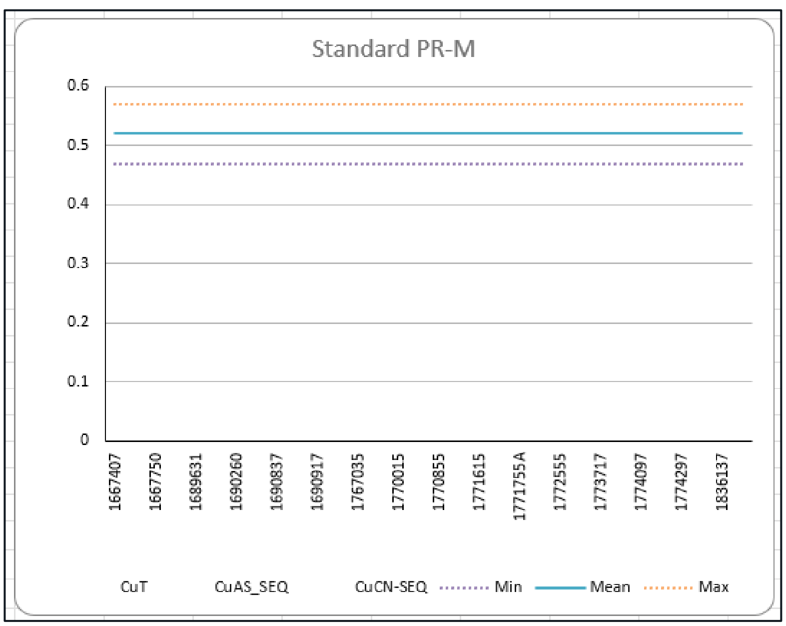

Source: ASCU, 2022.

**Figure 12-12: Primary Low-grade Standard (PR-L)**



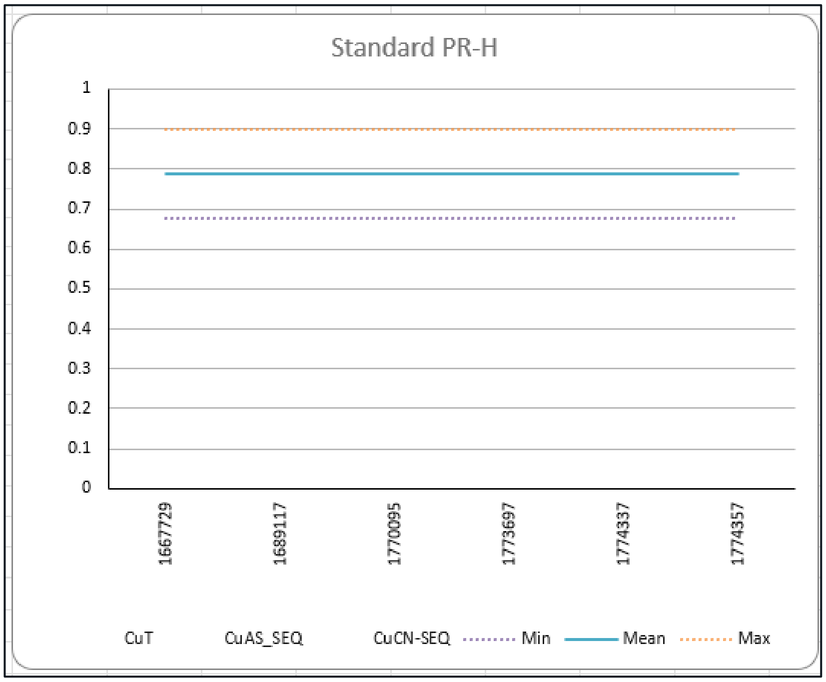

Source: ASCU, 2022.

**Figure 12-13 : Primary Medium Grade Standard (PR-M)**




Source: ASCU, 2022.

**Figure 12-14: Primary High-grade Standard (PR-H)**




Source: ASCU, 2022.

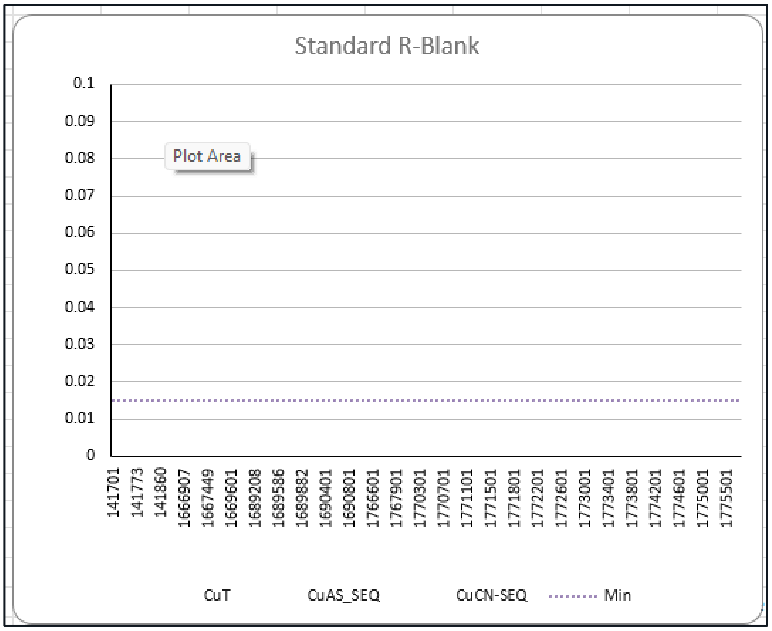
### 12.5.2 Blanks

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.

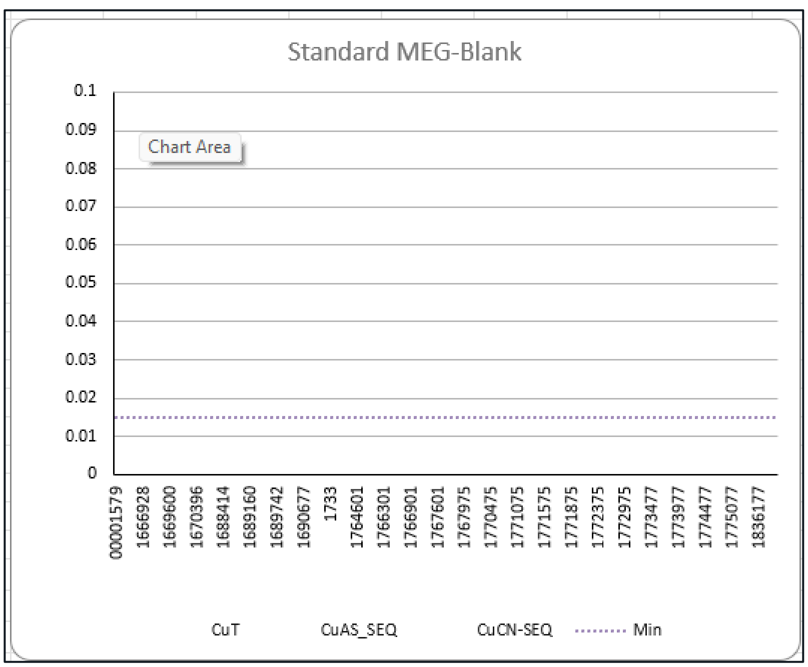
Figure 12-15 and Figure 12-16 show the performance of the blanks across both the drilling and re-assay programs supporting the new mineral resource estimate. All figures plot the maximum expected total copper grade as a dashed line (0.015% CuT). Orange squares represent the total copper grade, blue squares 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 12-15: R Blank**




Source: ASCU, 2022.

**Figure 12-16: MEG Blank**




Source: ASCU, 2022.

## 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 ASCU.

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 meet the level of accuracy expected for this PFS report.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical studies and testing for the Cactus Project have been ongoing since late 2019 and has been conducted in four phases of testing through the current information completed in 2023 disclosed in this Report.

Arizona Sonoran geologists are working with metallurgical engineers to quantify the recovery of copper from samples obtained in a series of large drilling campaigns. The drill core samples were studied by geologists and subsequently shipped to a well-established mineral processing research and development firm in Reno, Nevada (McClelland Analytical Service Laboratory (McClelland), an ISO 9000, ISO 17025 accredited facility). Additional testing work was completed on-site by ASCU staff and at HydroGeoSense Inc. (HGS) laboratories in Tucson, Arizona. The metallurgical test program completed at McClelland has been developed by and supervised by Mr. James L. Sorensen. Mr. Sorensen has also reviewed and inspected the ongoing metallurgical testing at site and information developed by HGS. A summary of the various testing programs completed for the Cactus project is shown in Table 13-1.

**Table 13-1: Historical Testing Programs**

| Year | Source Material | Laboratory | Testwork Performed |
|---|---|---|---|
| 2020/2021 | Stockpile - Oxide | McClelland Laboratories, Inc. | Column Testing, Column Screen Size Analysis, Recovery by Size Fraction |
| 2021 | Cactus-Sulfide | McClelland Laboratories, Inc. | Preliminary flotation, comminution and work indices/abraison index |
| 2021 | Cactus-Sulfide | JK Tech | Preliminary SMC mineral comminution testing |
| 2021 | Stockpile/Cactus | Western Environmental Testing Laboratory & ALS Global | TCLP analysis of McLelland test column PLS and Raffiante, residues analysis |
| 2022 | Stockpile – Oxide | McClelland Laboratories, Inc. | Bottle Roll |
| 2022 | Cactus – Oxide and Sulfide | McClelland Laboratories, Inc. | Column Testing, Head assay, Bottle Roll |
| 2022 | Cactus - Sulfide | Process Mineralogical Testing Ltd. | Mineral Characterization, Copper Deportment by Size Fraction, SEM-BSE Imaging |
| 2022-2023 | Cactus - Oxide and Sulfide | McClelland Laboratories, Inc. | Bottle Roll, Column Testing, Column Screen Size Analysis |
| 2022-2023 | Stockpile, Cactus oxide and sulfide | HydrgeoSense Inc. | Leach pad hydrodynamic and hydrological column testing |
| 2023 | Parks Salyer – Oxide and Sulfide | ASCU Tru-Stone, HydroGeoSense | Column Testing, Head Assay |

| Year | Source Material | Laboratory | Testwork Performed |
|---|---|---|---|
| 2023-2024 | Parks Salyer – Oxide and Sulfide | Process Mineralogical Testing Ltd. | Rapid Ore Characterization, SEM-BSE Imaging |

Resources considered for beneficial processing in this Report are related to four sources:

- An existing mine stockpile built during the development and operation of a copper open pit and milling facility from 1974 to 1984. The stockpile includes oxide and lower grade sulphide material containing primarily copper mineralization.
- Further development of the existing Cactus West open pit containing oxide and lower grade sulphide material.
- The underground resource called Cactus East located northeast immediately adjacent to the existing Cactus open pit and at a depth of 1,200 ft. This resource contains mostly lower-grade sulphide material.
- The underground resource called Park Salyer located about 1 mile to the southwest of the Cactus West open pit at a depth of 1,500 ft. This resource contains mostly higher-grade secondary and primary copper sulphide material.

The QP believes the metallurgical testing and data collected to date is sufficient to establish the required supporting metallurgical performance expectations used in estimating the project Reserves and economics for the Stockpile, Cactus East, Cactus West and Parks/Salyer deposits included in the Cactus Project. However, only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit and additional confirmatory work is required to better understand the deposit variability.

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 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 sulphide concentrates for processing at local smelters.

In consideration of a copper heap leaching and SX/EW processing facility at Cactus based on processing existing Stockpile Project oxidized copper resources, a hydrometallurgical approach utilized 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.

Approximately 45 column tests have been completed (Stockpile - 25, Cactus – 14, Parks/Salyer - 6) covering the resources identified in the current study effort. In addition, over 150 bottle roll tests, mineralogical analyses and other metallurgical and materials property testing has been completed.

Park Salyer 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 samples from newly drilled core were selected to reflect copper grades close to the presumed average of the economically processable material in the underground resource for column testing. Assay data and bottle roll testing were completed on head samples from the three test samples. The three Park Salyer samples had head grades ranging from 0.76% Tsol to 1.34% Tsol.

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 from the completed column testing programs.

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.

For the current mine plans, the distribution of leachable oxide and enriched material types is provided in Table 13-2.

**Table 13-2: Potential Leach Materials Distribution**

| Mining Source | Material Type | Tons of Leach Material (tons) | Grade % Tsol (% Cu) | Leachable Cu (tons) |
|---|---|---|---|---|
| Stockpile | Oxide | 76,777,446 | 0.137% | 105,100 |
| Cactus Open Pit | Oxide & Enriched | 75,520,724 | 0.259% | 195,282 |
| Cactus Underground | Oxide & Enriched | 27,739,290 | 0.886% | 245,661 |
| Park Salyer | Oxide & Enriched | 96,248,457 | 0.821% | 789,750 |
| **Total** | **Oxide & Enriched** | **276,285,916** | **0.483%** | **1,335,794** |

In parallel, copper flotation testing was also 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**

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



Source: D.F. Briggs, October 2004.

### 13.1.1 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.

The following information (Table 13-3) 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 sulphate leach also at pH 2.0. The acid ferric sulphate is necessary because of some chalcocite mineral in the sample. Overall copper extraction was 91.4%. A straight acid leach with no ferric sulphate 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 sulphate at pH 1.5 with, no additional oxidation. Ferric sulphate 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 sulphate to ferrous sulphate 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 sulphides, 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 13-3: 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 13-4. 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 13-4: 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 H2SO4/lb | lb H2SO4/t | lb H2SO4/lb | lb H2SO4/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 31 August 2021, 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. The previous released PEA contained the results of the three initial bulk composite testing.

Arizona Sonoran expanded the initial test work with an addition bulk sample test and 14 variability columns. The 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 (H2SO4 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 H2SO4, 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 13-5.

**Table 13-5: 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 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 13-6.

**Table 13-6: 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**

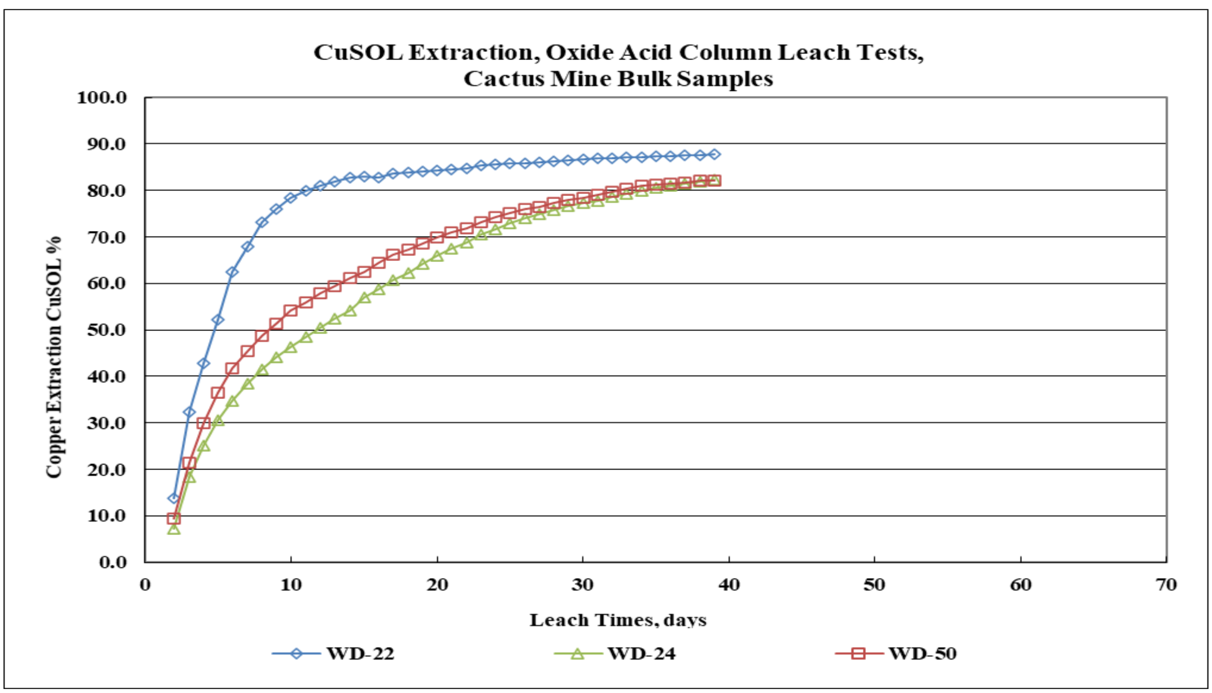
| MLI Test No. | Sample | Leach / Rinse Time (days) | CuT Recovery (%) | Extracted | Tail | Calc/d Head | Avg. Head | Gross (lb/t ore) | Gangue (lb/t ore) | Specific (Gangue) lb/lb ore |
|---|---|---|---|---|---|---|---|---|---|---|
| AC-1 | WD-22 | 39 | 75.9 | 0.082 | 0.026 | 0.108 | 0.119 | 31.0 | 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**




Source: 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 13-7. 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 13-7: 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 13-8. The effective $P_{80}$ size distribution was 1.5 inch for the three columns.

**Table 13-8: Column Screen Size Analysis**

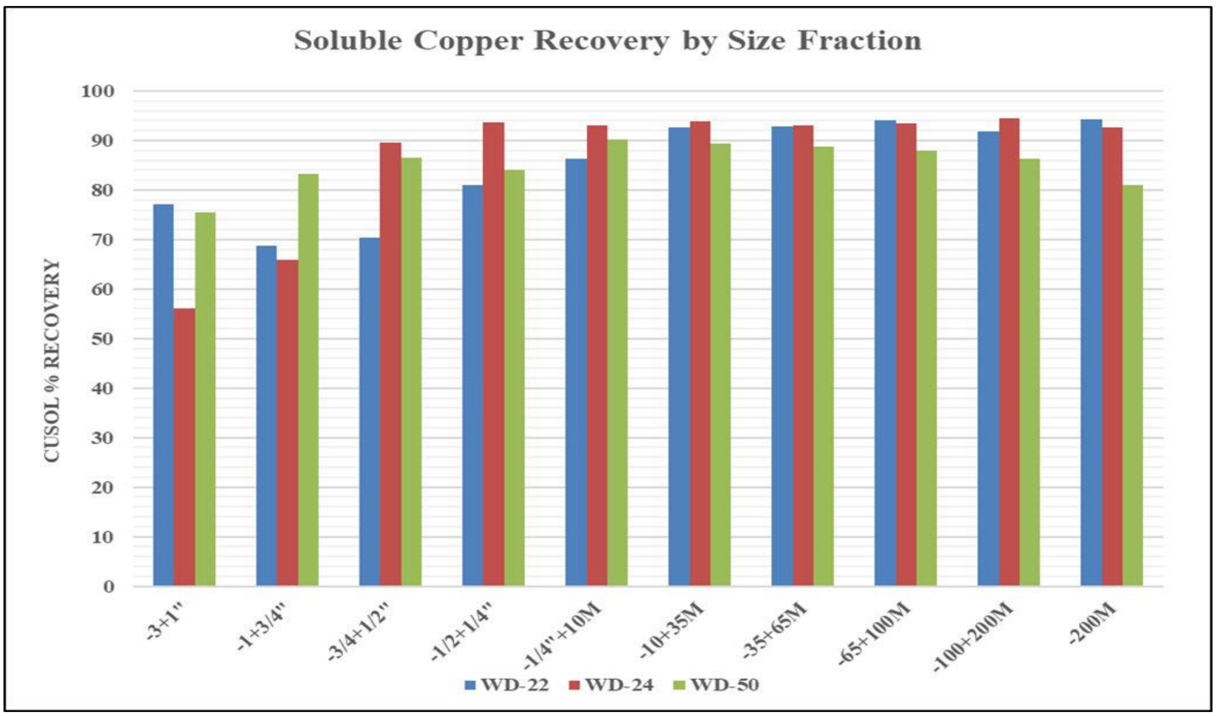
| Size Fraction | WD-22 Weight, % | | WD-24 Weight, % | | WD-50 Weight, % | |
|---|---|---|---|---|---|---|
| | Head | Tail | Head | Tail | Head | Tail |
| -3” + 1” | 28.4 | 21.9 | 29.8 | 27.0 | 31.6 | 22.6 |
| -1 + ¾” | 6.6 | 7.0 | 5.7 | 5.5 | 5.9 | 4.6 |
| -¾ + ½” | 11.8 | 10.0 | 7.9 | 7.2 | 9.4 | 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 |

| Size Fraction | WD-22 Weight, % | | WD-24 Weight, % | | WD-50 Weight, % | |
|---|---|---|---|---|---|---|
| | Head | Tail | Head | Tail | Head | Tail |
| -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 |
| $P_{80}$ | ~1.5" | | ~1.5" | | ~1.5" | |

Source: McClelland, 2020.

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**



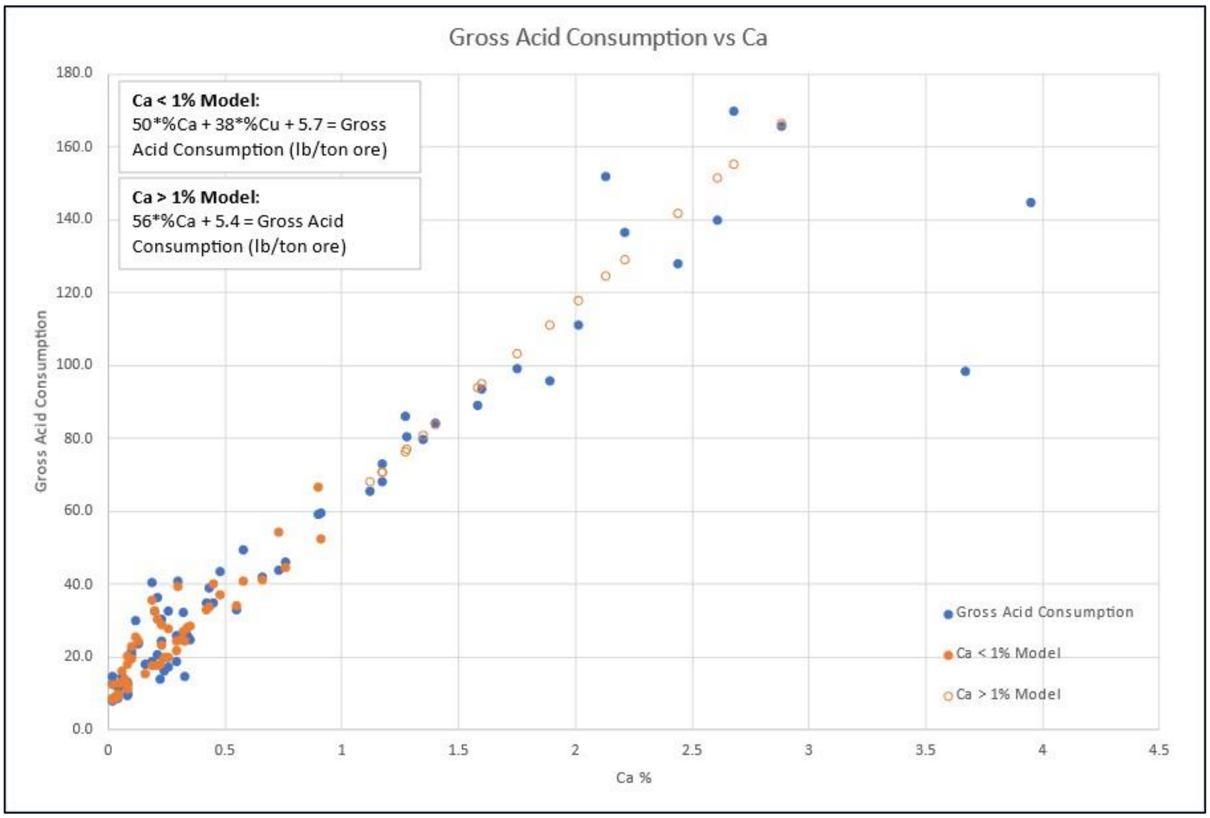

Source: McClelland, 2020.

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 Update

In an effort to improve gross acid consumption modeling of the stockpile project, a large bottle roll test campaign was conducted in 2022 utilizing 2.5-foot interval samples from drill holes WD-005, WD-022, and WD-026 for a total of 89 samples. The bottle roll test procedure utilized aligned with all other testing performed. Full ICP head assays were performed for each interval and analysis indicated a relationship between total copper and calcium to predict gross acid consumption. Figure 13-4 depicts the consumption models established.

**Figure 13-4: Stockpile Pile Gross Acid Consumption Model**




Source: McClelland, 2020.

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. Analysis of the stockpile column and bottle roll test results indicated that the 365-day projected column acid consumptions were 79% that of the bottle roll acid consumption tests. Therefore, a 0.79 factor was applied to the model in Figure 13-4 for predicting stockpile gross acid consumption.

When applied to the mine plan, lower calcium material is mined in the first three years of stockpile mining resulting in an average gross acid consumption of 18 lb/ton. The last four years of stockpile mining contains higher calcium resulting in an average gross acid consumption of 23 lb/ton.

## 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.

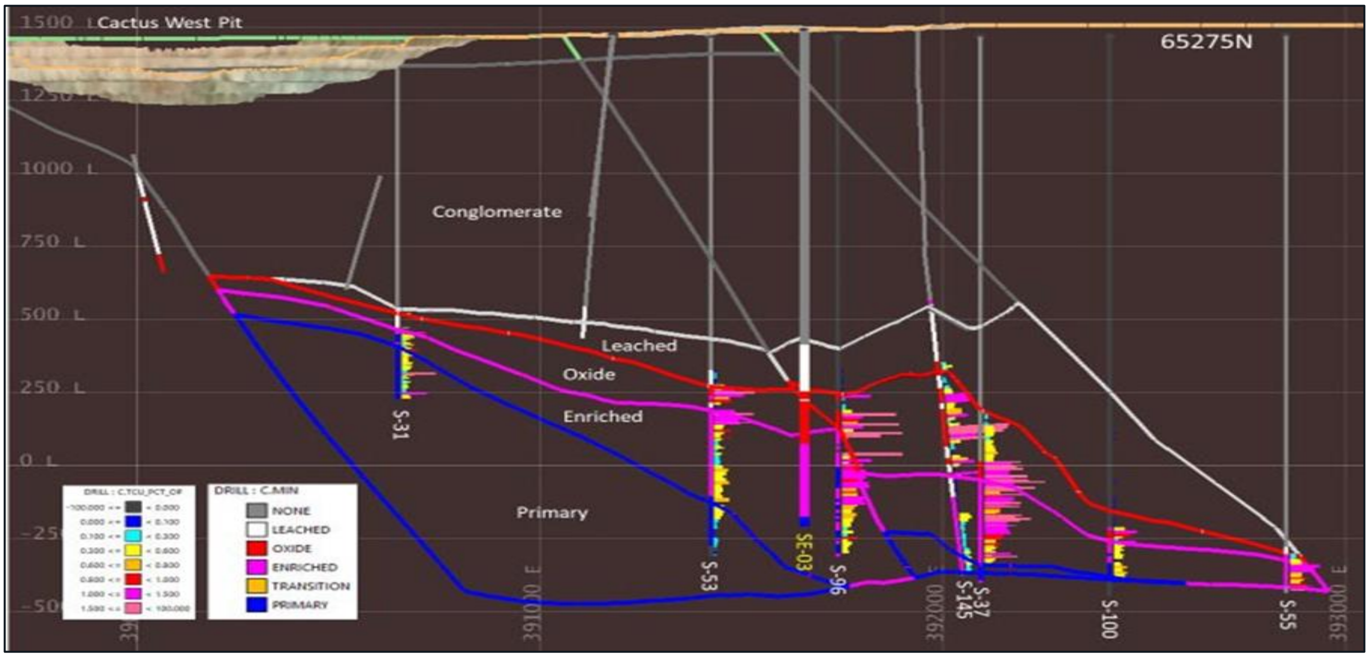
Metallurgical core SE-03 (PQ size) diamond drill core hole was drilled in the Cactus East deposit area, near historic hole S-96 used as a reference for potential copper content and grades. Figure 13-5 describes the location of Drill Hole SE-03.

**Figure 13-5: Metallurgical Sample Drill Hole SE-03 Location**


Source: ASCU, 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 13-6.

**Figure 13-6: Metallurgical Hole SE-03 Section**




Source: 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 | -31/2 | 461.2 (209.2) | oxide |
| BAR 2 | SE-03 | 1,340.0 | 1,415.0 | -31/2 | 519.9 (235.8) | oxide |
| BAR 3 | SE-03 | 1,415.0 | 1.460.1 | -31/2 | 567.5 (257.4) | sulfide |
| BAR 4 | SE-03 | 1,460.1 | 1,521.0 | -31/2 | 546.7 (248.0) | sulfide |
| BAR 5 | SE-03 | 1,521.0 | 1,574.8 | -31/2 | 537.5 (243.8) | sulfide |
| BAR 6 | SE-03 | 1,574.8 | 1,627.5 | -31/2 | 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 PEA Results

Cactus Open Pit data was previously summarised in the PEA and Mineral Resource Estimate referenced in Section 2.6.1. Data included Section 13 of this report is a build upon and inclusive of the previous testing.

### 13.4.2 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**

| Determination | % Cu | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Sample** | 4600-001 | 4600-002 | 4600-003 | 4600-004 | 4600-005 | 4600-006 | 4600-007 | 4600-008* |
| Predicted Head* | | | | | | | | 0.844 |
| Direct Assay, Init. | 0.844 | 2.510 | 0.613 | 0.335 | 0.407 | 0.423 | 0.160 | |
| Direct Assay, Dup. | 0.844 | 2.350 | 0.613 | 0.338 | 0.422 | 0.425 | 0.151 | |
| Direct Assay, Trip. | 0.827 | 2.440 | 0.627 | 0.341 | 0.434 | 0.422 | 0.161 | |
| Direct Assay, (Seq. Assay) | 0.810 | 2.350 | 0.597 | 0.312 | 0.407 | 0.384 | 0.153 | 0.799 |
| Calc'd., Head Screen | 0.835 | 2.310 | 0.687 | 0.370 | 0.427 | 0.410 | 0.161 | 0.843 |
| Calc'd., BRT, -10M | 0.778 | 2.313 | 0.584 | 0.356 | 0.428 | 0.423 | 0.165 | ----- |
| Calculated, Column Test | 0.839 | 2.382 | 0.592 | 0.344 | 0.430 | 0.382 | 0.163 | 0.818 |
| **Average** | **0.825** | **2.379** | **0.616** | **0.342** | **0.422** | **0.410** | **0.159** | **0.820** |
| Std. Deviation | 0.024 | 0.073 | 0.035 | 0.018 | 0.011 | 0.019 | 0.005 | 0.022 |
| Relative Std. Deviation, % | 2.9 | 3.1 | 5.7 | 5.3 | 2.6 | 4.6 | 3.1 | 2.7 |

Source: McClelland, 2023.
*Grade weighted average calculated from screen assays.

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.

### 13.4.3 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. PMC's Rapid Ore Characterization (ROC) Report FEB 2022-05 dated 04 April 2022 and APR 2022-04 dated 19 May 2022 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 sulphides of chalcopyrite / bornite.
- Pyrite (py) is present in moderate amounts (approximately 10%) in samples 4600-002, 005, 007 and present in minor amounts (approximately 1%) in 4600-003, 006. The 4600-003, 006 samples also contain significant amounts of K-feldspar which sets it apart from the other samples.
- 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, 007 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 4600-002, 003.
- Clay minerals are present in all samples in minor amounts.

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

**Table 13-11: Sulfide Composite Mineralogy**

| Mineral Abundance | 4600-002 | 4600-003 | 4600-005 | 4600-006 | 4600-007 |
|---|---|---|---|---|---|
| Mass % | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 |
| Pyrite / Pyrrhotite | 9.77 | 0.97 | 9.64 | 1.41 | 12.00 |
| Chalcocite / Digenite | 4.57 | 1.20 | 0.27 | 0.59 | 0.04 |
| Covellite | 1.22 | 0.17 | 0.02 | 0.00 | 0.01 |
| Chalcopyrite / Bornite | 0.09 | 0.04 | 0.49 | 0.34 | 0.11 |
| Other Sulphides | 0.15 | 0.03 | 0.00 | 0.01 | 0.00 |
| Cuprous FE-Oxy Hydro | 0.90 | 0.54 | 0.09 | 0.10 | 0.00 |
| Alumino-Phospho-Sulphate | 0.57 | 0.01 | 0.50 | 0.20 | 3.09 |
| Quartz | 51.30 | 54.60 | 53.40 | 40.90 | 51.60 |
| Plagioclase | 1.45 | 0.74 | 1.04 | 0.94 | 0.94 |
| K-Feldspar | 7.93 | 26.60 | 4.73 | 37.70 | 10.30 |

| Mineral Abundance | 4600-002 | 4600-003 | 4600-005 | 4600-006 | 4600-007 |
|---|---|---|---|---|---|
| Muscovite / Sericite | 19.10 | 12.90 | 24.40 | 10.10 | 17.70 |
| Biotite | 1.34 | 0.94 | 3.14 | 3.46 | 1.55 |
| Clays | 1.52 | 1.05 | 1.01 | 1.15 | 1.87 |
| Other Minerals | 0.10 | 0.17 | 1.27 | 3.1 | 0.79 |
| **Total** | **100.00** | **100.00** | **100.00** | **100.00** | **100.00** |
| Porosity (Volume %) | 1.93 | 2.44 | | | |
| Copper Deportment | | | | | |
| Chalcocite / Digenite | 74.60 | 82.9 | 63.70 | 77.81 | 19.84 |
| Covellite | 20.60 | 12.9 | 5.00 | 0.35 | 18.39 |
| Chalcopyrite / Bornite | 1.05 | 0.88 | 30.84 | 21.13 | 60.77 |
| Other Sulphides | 0.03 | 0.01 | 0.14 | 0.01 | 0.00 |
| Cuprous Goethite | 3.35 | 2.73 | 0.32 | 0.70 | 1.00 |
| Alumino-Phospho-Sulphate | 0.44 | 0.63 | N/A | N/A | N/A |
| **Total** | **100.00** | **100.00** | **100.00** | **100.00** | **100.00** |

Source: PMC, 2022.

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**

| 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** |

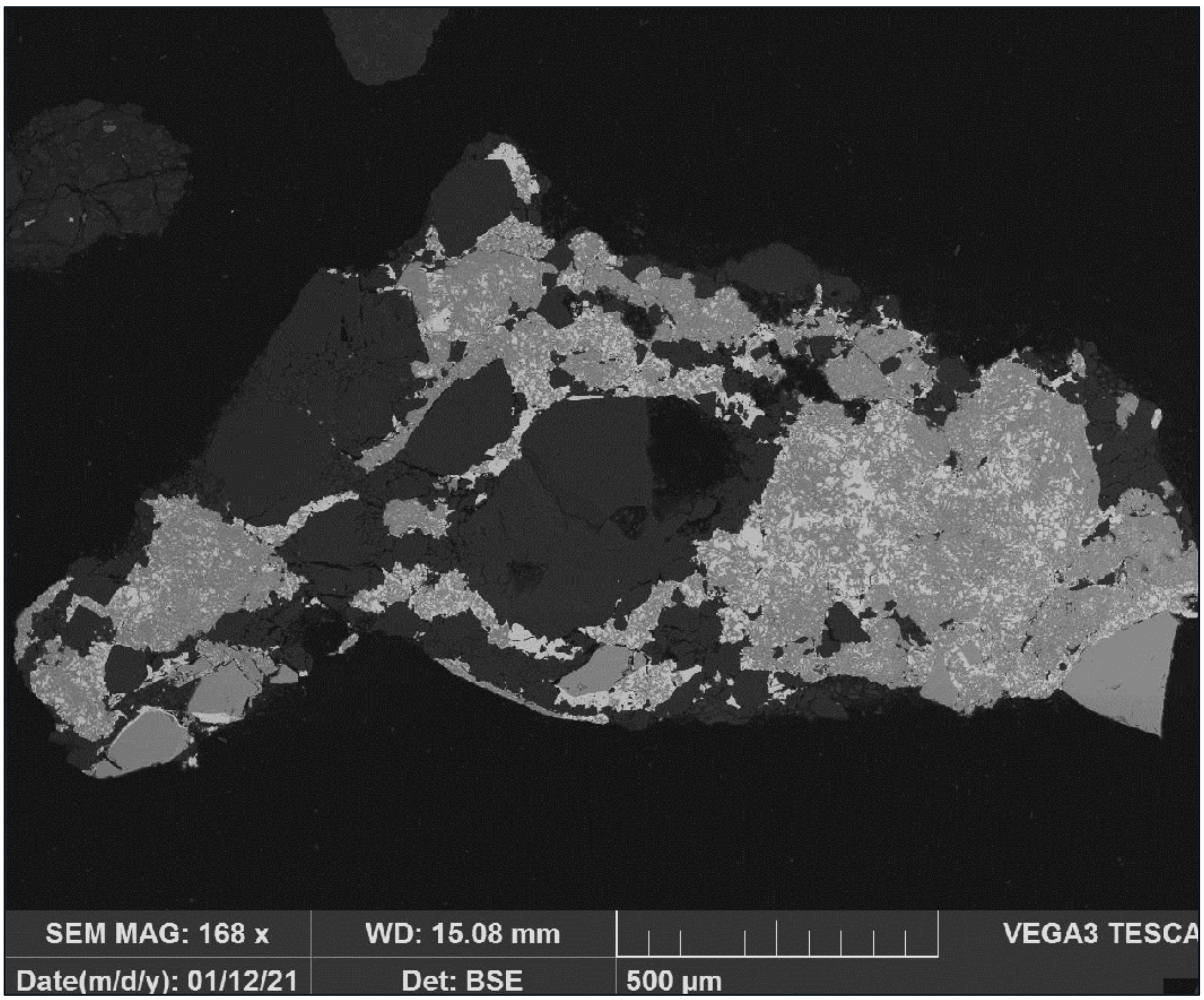
Source: PMC, 2022

Photomicrographs of polished sections were also completed for the samples evaluated. Figure 13-7 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.

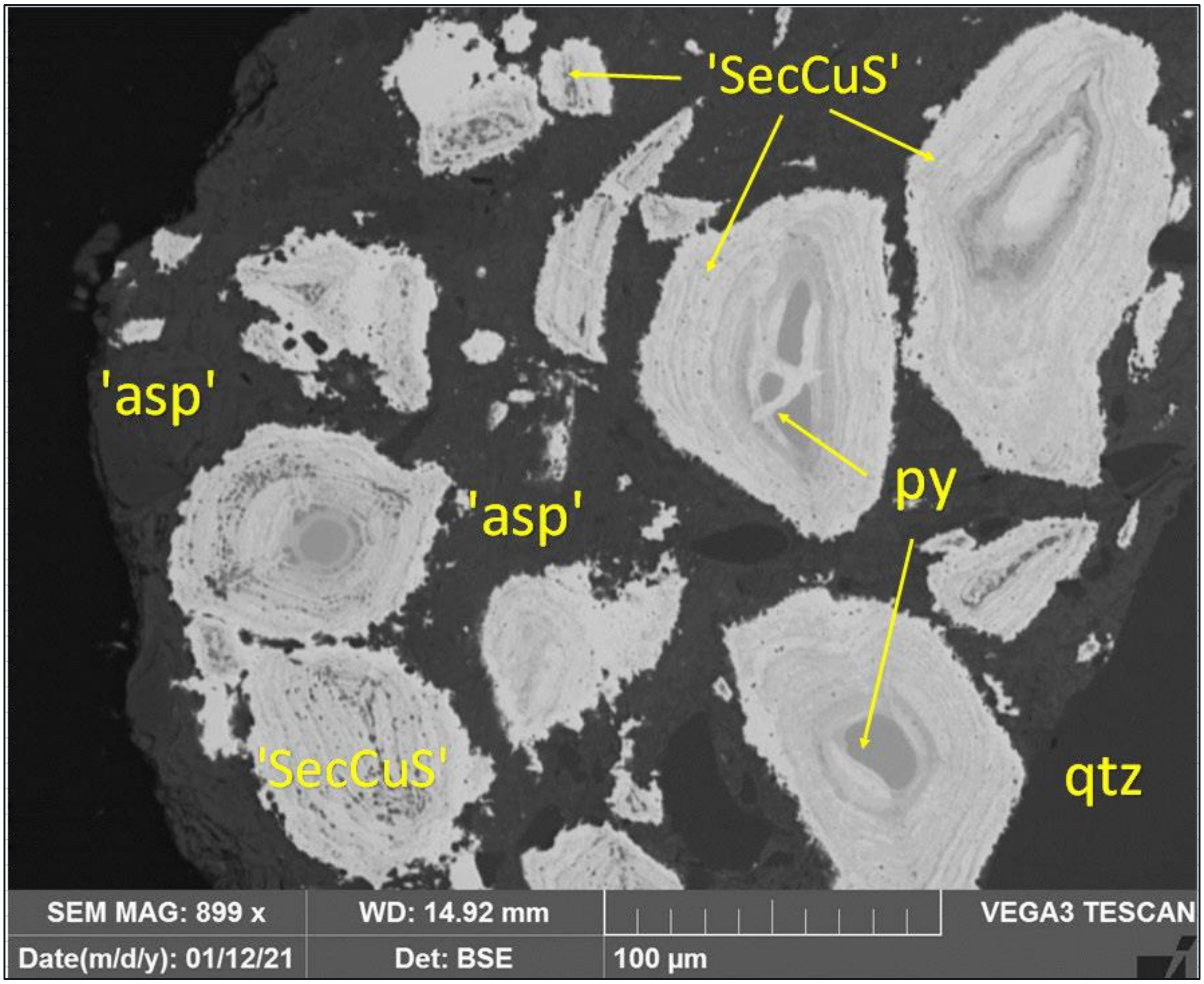
**Figure 13-7: Sample 4600-002 Column Composite Material**




Source: PMC, 2022.

Figure 13-8 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 13-8: Sample 4600-002 Column Composite Material**

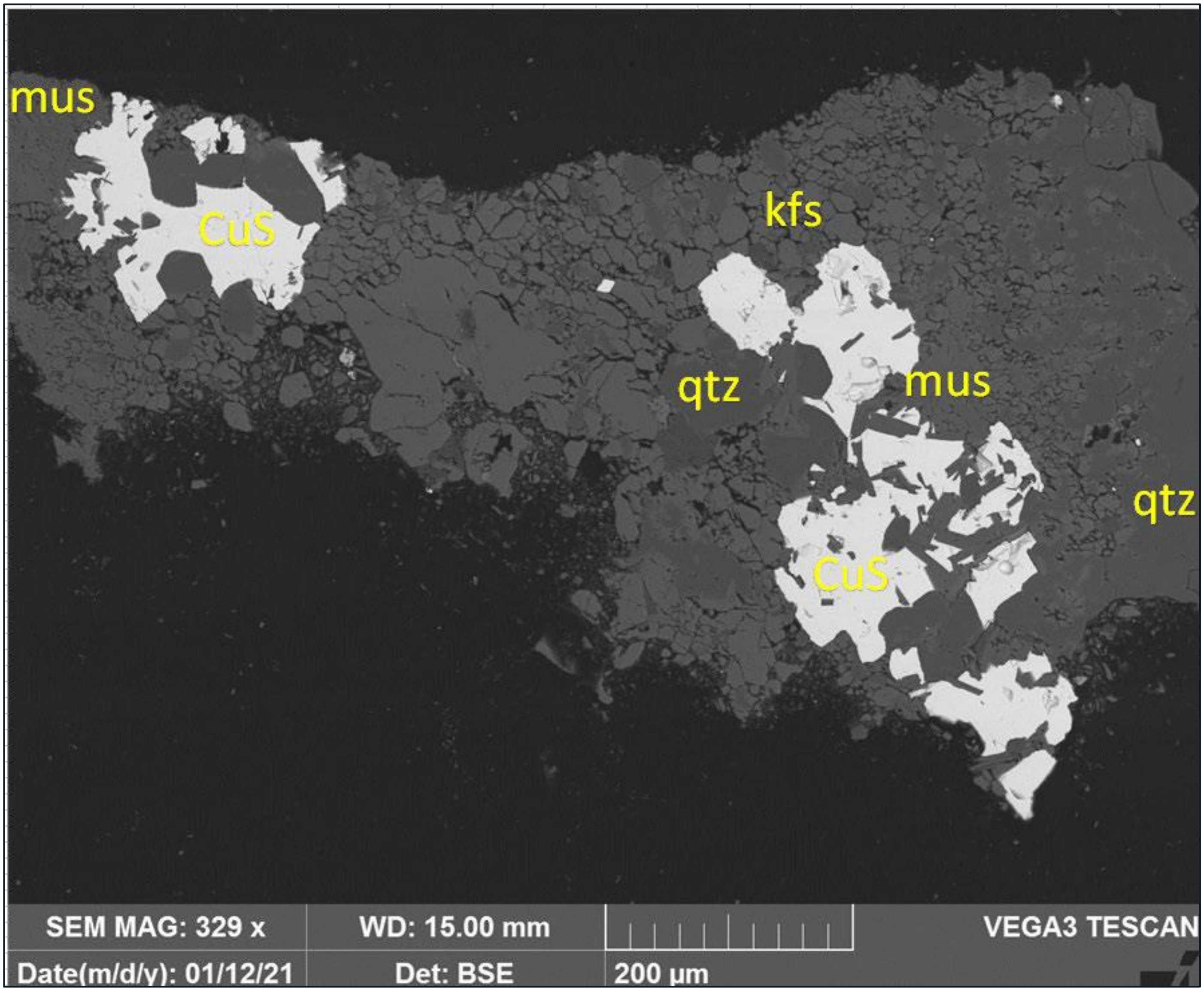



Source: PMC, 2022.

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 13-9 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 13-9: Sample 4600-003 Column Composite Materials**




Source: PMC, 2022.

### 13.4.4 Bottle Roll Testing

A split of each sample was taken, prepped to 100% -10 mesh and subjected to bottle roll acid testing for 96 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**

| Sample | Cu Recovery, % of Total | %Cu | | | | $H_2SO_4$ Added, lb/ton Ore | $H_2SO_4$ 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 | 28.6 | 6.9 | 0.5 |
| 4600-002 (Sulfide) | 8.8 | 0.204 | 2.130 | 2.313 | 2.433 | 32.8 | 13.0 | 6.7 | 1.7 |
| 4600-003 (Sulfide) | 11.3 | 0.066 | 0.523 | 0.584 | 0.618 | 38.7 | 9.5 | 7.5 | 5.6 |
| 4600-004 (Oxide) | 80.9 | 0.288 | 0.069 | 0.356 | 0.338 | 30.3 | 20.6 | 11.7 | 2.0 |
| 4600-005 (Sulfide) | 30.8 | 0.132 | 0.296 | 0.428 | 0.421 | 23.7 | 20.9 | 16.9 | 6.4 |
| 4600-006 (Sulfide) | 34.8 | 0.147 | 0.282 | 0.423 | 0.423 | 24.3 | 20.6 | 16.1 | 5.5 |

Source: McClelland, 2023.

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.

### 13.4.5 Open Pit Copper Recovery

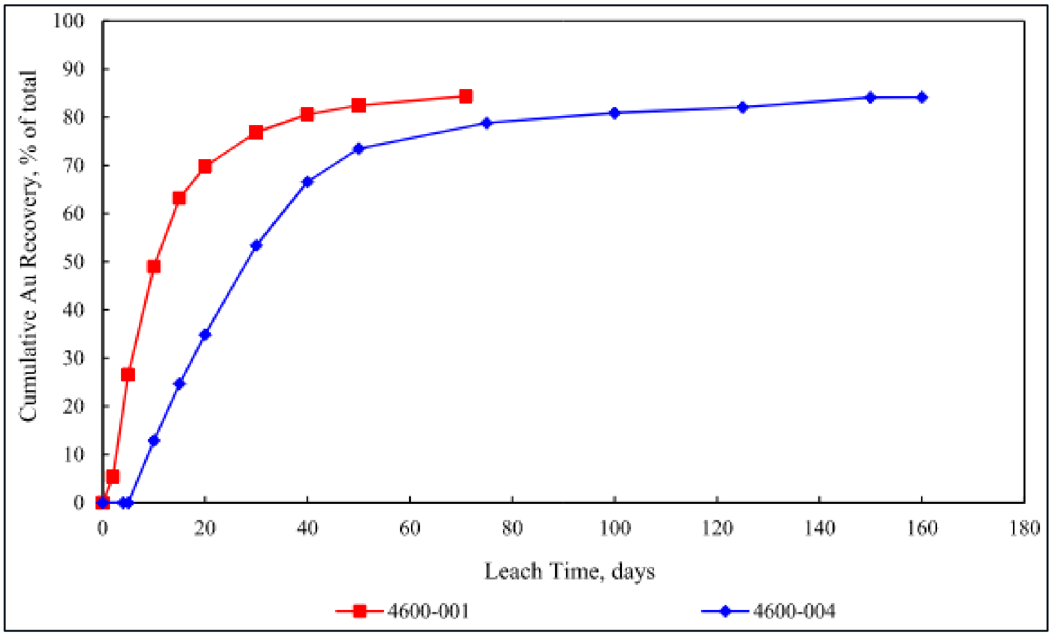
The initial open pit columns 4600-01 to 4600-03 were sampled from drill hole SE-03. Tests are conducted in 12-in I.D. by 10-ft tall columns containing approximately 660 lb (300 kg) of material. Column leach testing in closed circuit with SX was incorporated with saved solution from the Stockpile Project testing as a starting solution.

Additional open pit columns were tested (4600-04 to 4600-08) and were sampled from drill hole SE-04, SE-6A, and SE-08. Tests are conducted in 12-in I.D. by 20 ft tall columns containing approximately 1320 lb (600 kg) of material. Column leach testing in closed circuit with SX was incorporated with saved solution from the 4600-01,02,03 testing as a starting solution.

All copper recoveries are based on calculated head assays from solution samples are column residue characterization.

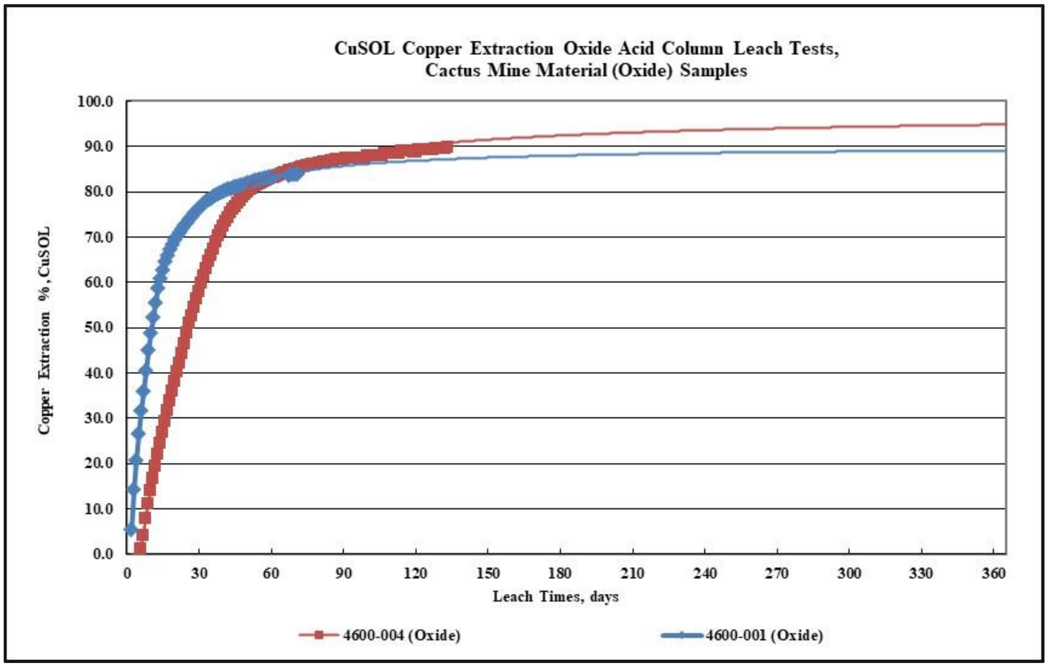
Results for the oxide columns 4600-001, 004 are consolidated in Figure 13-10, Figure 13-11 and Table 13-14.

**Figure 13-10: Oxide Copper Columns, Total Copper Extraction**




Source: McClelland, 2023.

**Figure 13-11: Oxide Copper Columns, Soluble Copper Extraction**




Source: McClelland, 2023.

**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 | 82.6 | 0.689 | 0.146 | 0.839 | 0.832 | 120.7 | 26.8 | 5.7 | 0.4 |
| AC-4 | 4600-004 (Oxide) | 133 | 83.4 | .287 | .057 | .344 | 0.342 | 40.6 | 21.9 | 13.0 | 2.3 |

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 13-12.

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.

CuAS copper content extraction averaged 92% for the two oxide columns completed. CuCN recovery, representing enriched copper mineralization content (chalcocite and covellite) was also significant and averaged 58%. The combined Tsol extraction averaged 90% 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.

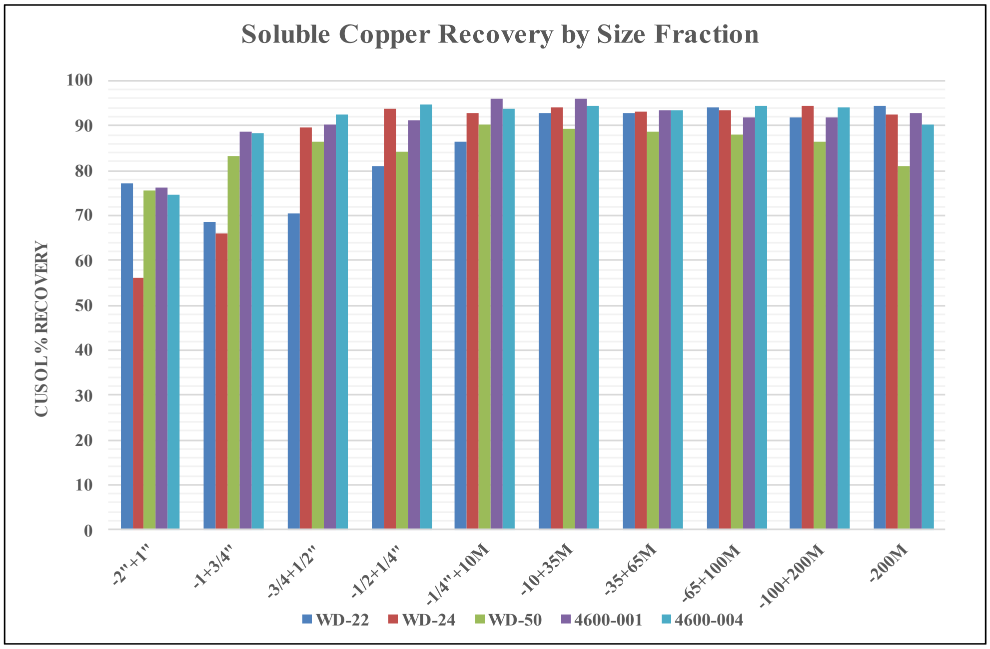
Copper extraction was also considered by size fractions to assess the impact of particle size. Figure 13-13 shows the relative copper extraction by size fraction.

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 13-13. Column tests AC-2 4600-02 and AC-3 4600-03 are in progress and results are indicative only.

**Figure 13-12: Soluble Copper Recovery by Size Fraction**




Source: McClelland, 2023.

**Table 13-15: Copper Extraction by Copper Assay Method – Oxide Columns**

| Name | 4600-01 | | 4600-04 | | AVG (%) | Predicted/Modeled (%) | | |
|---|---|---|---|---|---|---|---|---|
| | Assay % | Recv % | Assay % | Recv % | | 365 days | YR 1 | YR 2 |
| CuAS | 90.9 | 91 | 86.4 | 93 | 92 | - | - | - |
| CuCN | 5.9 | 60 | 6.0 | 54 | 58 | - | - | - |
| Tsol | 96.8 | 89.3 | 92.3 | 90.2 | 90 | - | - | - |
| Tsol Pred | - | 89.1 | - | 94.8 | 92 | 83.3 | 68.2 | 15.1 |
| CuT | - | 82.2 | - | 84.3 | 83 | - | - | - |

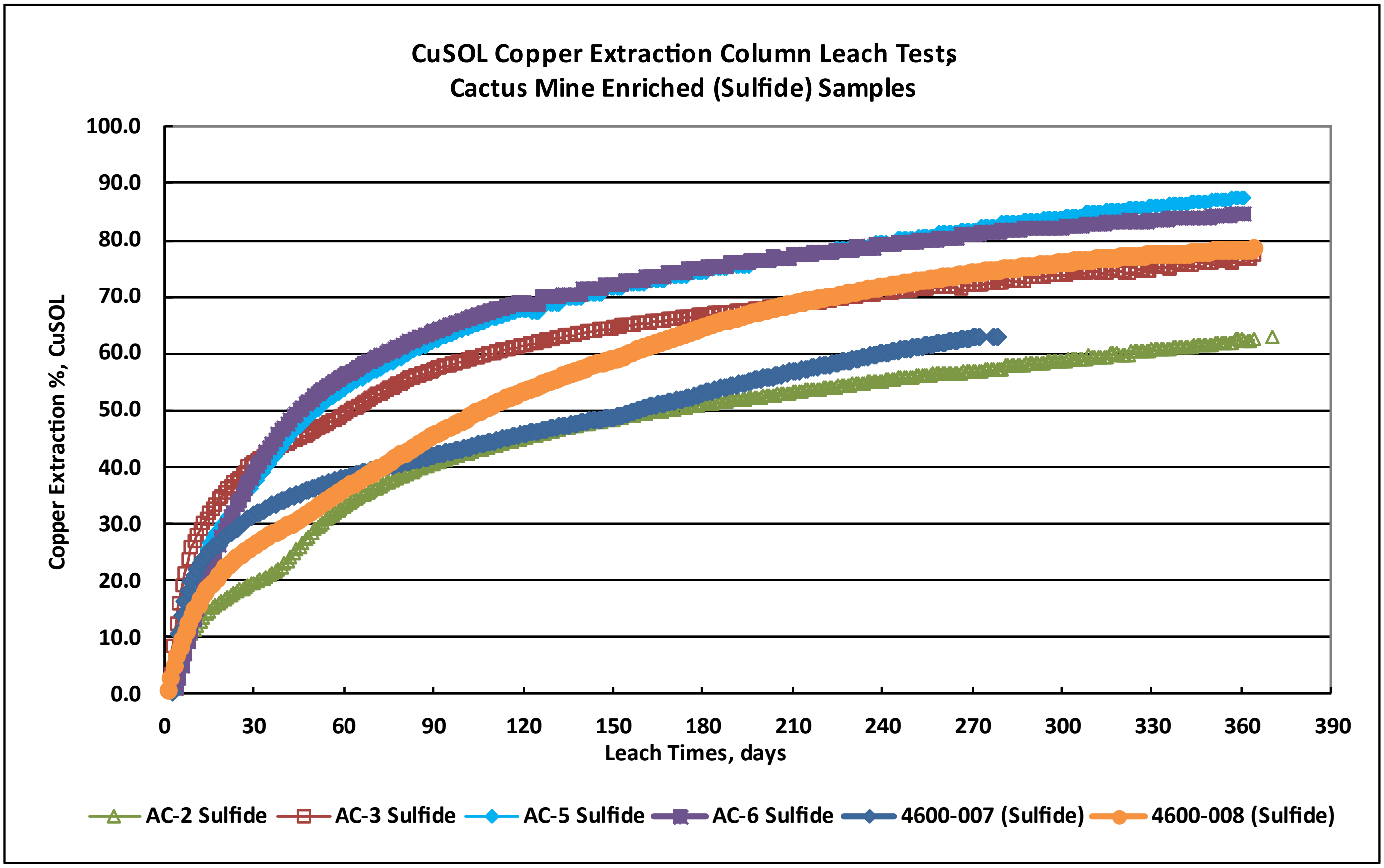
**Table 13-16: Column Screen Size Analysis– Oxide Columns**

| Size Fraction | WD-22 Weight, % | | WD-24 Weight, % | | WD-50 Weight, % | | 4600-01 Weight, % | | 4600-04 Weight, % | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Head | Tail | 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 | 17.1 | 10.6 |
| -1+3/4" | 6.6 | 7 | 5.7 | 5.5 | 5.9 | 4.6 | 22.5 | 10.8 | 22.8 | 11.7 |
| -3/4+1/2" | 11.8 | 10 | 7.9 | 7.2 | 9.4 | 6.8 | 13.7 | 10.1 | 16.7 | 11.4 |
| -1/2+1/4" | 9.7 | 10.3 | 11.0 | 13.0 | 10.1 | 12.0 | 13.8 | 12.3 | 16.4 | 19.4 |
| -1/4"+10 M | 14 | 13.9 | 16.0 | 16.4 | 14.2 | 14.5 | 15.2 | 18.2 | 13.4 | 20.3 |

| Size Fraction | WD-22 Weight, % | | WD-24 Weight, % | | WD-50 Weight, % | | 4600-01 Weight, % | | 4600-04 Weight, % | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Head | Tail | Head | Tail | Head | Tail | Head | Tail | Head | Tail |
| -10+35 M | 12.2 | 14.5 | 12.1 | 12.7 | 11.2 | 14.6 | 7.3 | 12.8 | 6.5 | 11.9 |
| -35+65 M | 4.5 | 6.3 | 4.7 | 4.6 | 4.1 | 6.1 | 2.5 | 4.1 | 1.9 | 3.2 |
| -65+100 M | 2.2 | 3 | 1.7 | 1.6 | 2.2 | 2.6 | 1.0 | 1.4 | 0.6 | 1.5 |
| -100+200 M | 3.2 | 2.8 | 3.0 | 2.8 | 2.8 | 3.7 | 1.3 | 2.5 | 1.2 | 1.9 |
| -200 M | 7.4 | 10.3 | 8.1 | 9.2 | 8.5 | 12.5 | 6.4 | 12.2 | 3.3 | 12.2 |
| Composite | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 |
| $P_{80}$ | ~1.5" | ~1.5" | ~1.5" | ~1" | ~1.5" | ~1" | ~1" | ~1" | ~1" | ~1" |

Source: McClelland, 2021, 2022.

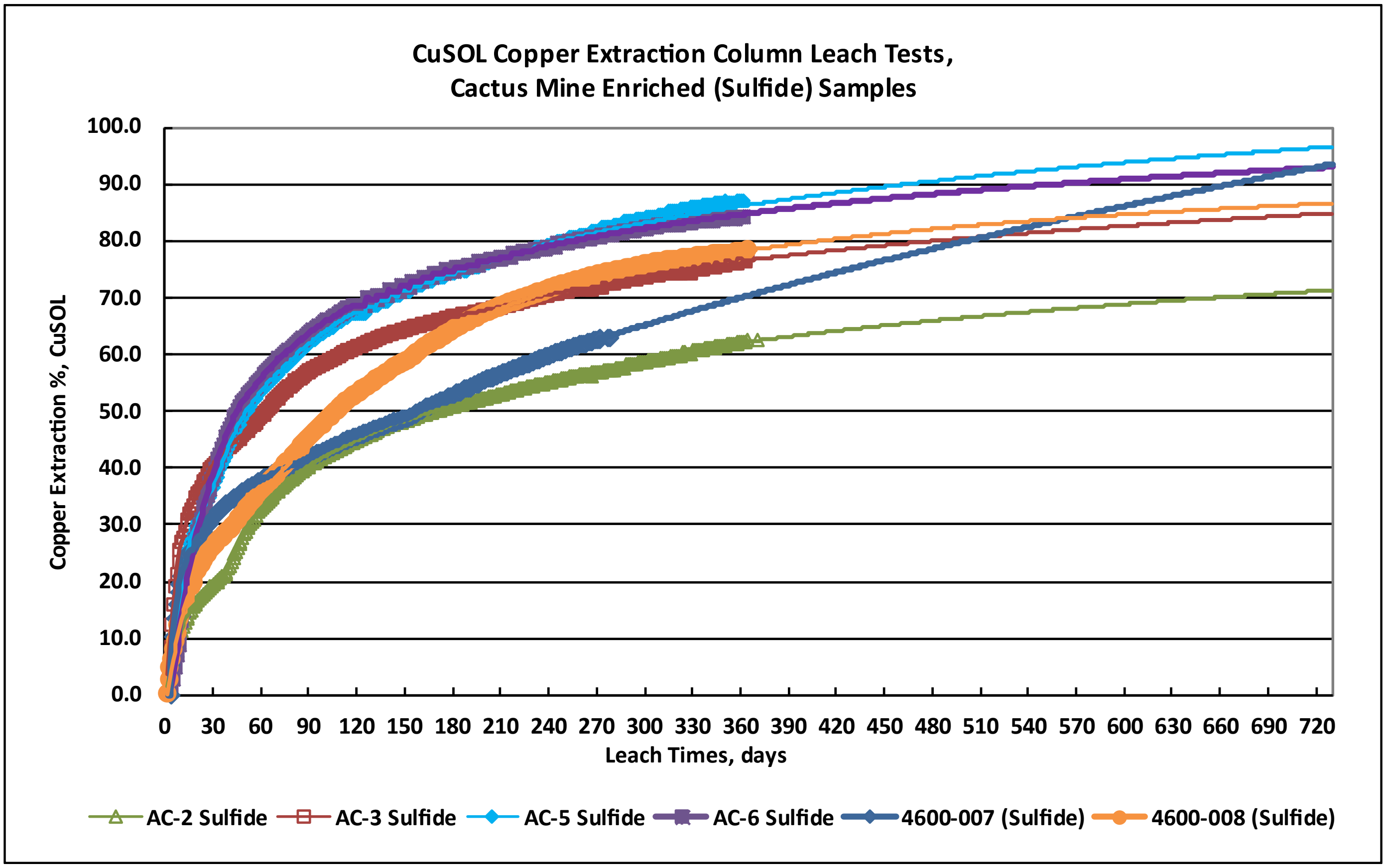
**Figure 13-13: Soluble Copper Extraction for Cactus West/East Sulfide Columns**




Source: McClelland, 2023.

Based on the column results for sulfide materials, a longer leaching time will be required to achieve copper extraction greater than 75% for the soluble copper components as demonstrated in Figure 13-14. Column results indicate a minimum of two years will be required to achieve extraction. Composites 4600-002 and 4600-0007 indicate slower leach kinetics compared to the other four columns. The slower kinetics for both composites can be attributed to covellite abundance identified in the copper sulfide deportment mineralogy.

**Figure 13-14: Extrapolated Two Year Long-Term Copper Extraction for Cactus West/East Sulfide Columns**




Source: McClelland, 2023.

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

**Table 13-17: Copper Extraction by Copper Assay Method – Sulfide Columns**

| Name | AC-2 | | AC-3 | | AC-5 | | AC-6 | | AC7 | | AC-8 | | AVG |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Recv % |
| CuAS | 8.7 | 88 | 11.0 | 93 | 19.5 | 93 | 32.4 | 94 | 15.9 | 80 | 13.1 | 85 | 89 |
| CuCN | 89.0 | 58 | 88.3 | 77 | 64.7 | 86 | 59.6 | 82 | 58.1 | 45 | 80.4 | 67 | 69 |

| Name | AC-2 | | AC-3 | | AC-5 | | AC-6 | | AC7 | | AC-8 | | AVG |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Assay % | Recv % | Recv % |
| Tsol | 97.7 | 61 | 99.3 | 79 | 84.2 | 87 | 92.0 | 86 | 73.9 | 50 | 93.5 | 69 | 72 |
| Tsol Pred | - | 71.3 | - | 84.9 | - | 96.6 | - | 93 | - | 93.5 | - | 86.6 | 87.7 |
| CuT | - | 60 | - | 79 | - | 73 | - | 80 | - | 45 | - | 67 | 67 |

**Table 13-18: Soluble Copper Extraction Model – Sulfide Columns**

| Name | Predicted/Modeled | | | |
|---|---|---|---|---|
| | 730 days | YR 1 | YR 2 | YR 3 |
| CuSOL Pred | 87.7% | 57% | 26.3% | 4.4% |

**Table 13-19: Column Screen Size Analysis– Sulfide Columns**

| Size Fraction | 4600-02 Wt, % | | 4600-03 Wt, % | | 4600-05 Wt, % | | 4600-06 Wt, % | | 4600-07 Wt, % | | 4600-08 Wt, % | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Head | Tail | Head | Tail | Head | Tail | Head | Tail | Head | Tail | Head | Tail |
| -3"+1" | 16.9 | 20.2 | 15.4 | 15.7 | 16.9 | 14.3 | 16.9 | 12.6 | 16.6 | 12.7 | 16.7 | 15.6 |
| -1+3/4" | 27.9 | 17.1 | 26.8 | 13.6 | 13.7 | 11.5 | 21.6 | 13.5 | 16.4 | 12.0 | 19.8 | 13.8 |
| -3/4+1/2" | 16.2 | 14.0 | 15.0 | 12.8 | 14.1 | 12.0 | 16.8 | 14.1 | 11.5 | 10.6 | 15.0 | 13.7 |
| -1/2+1/4" | 13.7 | 15.8 | 14.4 | 20.3 | 16.1 | 15.4 | 14.9 | 20.2 | 17.1 | 15.2 | 15.2 | 13.8 |
| -1/4"+10 M | 11.8 | 14.1 | 13.7 | 16.9 | 19.0 | 18.6 | 16.6 | 17.9 | 16.1 | 18.5 | 16.5 | 18.1 |
| -10+35 M | 5.4 | 6.8 | 5.7 | 8.2 | 8.3 | 10.4 | 6.5 | 9.3 | 10.2 | 12.8 | 7.1 | 9.4 |
| -35+65 M | 1.7 | 2.5 | 1.9 | 3.7 | 2.3 | 3.5 | 1.9 | 3.0 | 3.3 | 4.4 | 2.0 | 3.1 |
| -65+100 M | 0.8 | 1.5 | 0.6 | 1.2 | 1.0 | 1.9 | 0.6 | 1.2 | 1.3 | 1.8 | 0.8 | 1.4 |
| -100+200 M | 1.4 | 1.2 | 3.3 | 1.4 | 1.4 | 2.1 | 0.9 | 1.9 | 1.2 | 2.3 | 1.6 | 2.3 |
| -200 M | 4.2 | 6.8 | 3.2 | 6.2 | 7.0 | 10.3 | 3.2 | 6.3 | 6.2 | 9.7 | 5.3 | 8.8 |
| Composite | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 | 100 |
| $P_{80}$ | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" | ~1" |

Source: McClelland, 2022, 2023.

### 13.4.6 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 13-20 shows the bottle roll acid consumption information for the open pit core composites.

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 13-21.

**Table 13-20: Open Pit Column Material Bottle Roll Results Bottle Roll Tests, Cactus Project, 100%-10 M Feed Size, 24 Hour**

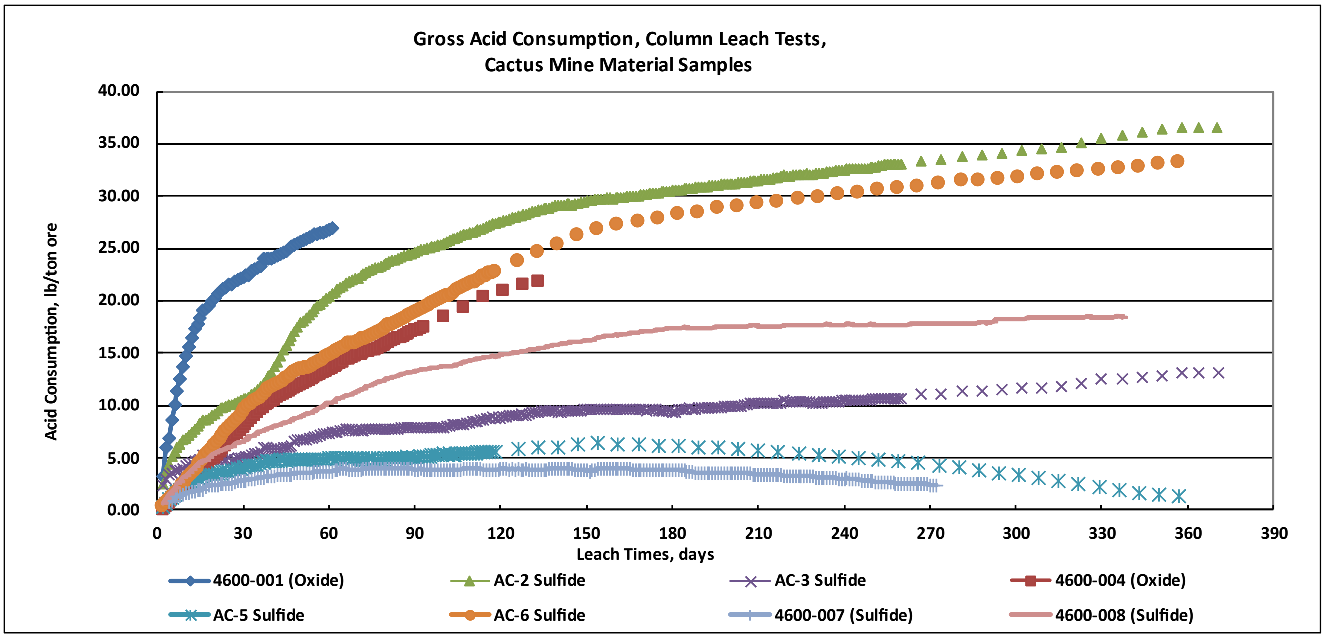
| Sample | $H_2SO_4$ Consumption | | |
|---|---|---|---|
| | Gross lb/t ore | Gangue lb/t ore | Specific (Gangue) lb/lb Cu |
| 4600-001 (Oxide) | 28.6 | 6.9 | 0.5 |
| 4600-002 (Sulfide) | 13.0 | 6.7 | 1.7 |
| 4600-003 (Sulfide) | 9.5 | 7.5 | 5.6 |
| 4600-004 (Oxide) | 20.6 | 11.7 | 2.0 |
| 4600-005 (Sulfide) | 20.9 | 16.9 | 6.4 |
| 4600-006 (Sulfide) | 20.6 | 16.1 | 5.5 |
| 4600-007 (Sulfide) | 23.4 | 22.2 | 27.0 |
| 4600-008 (Sulfide) | Not Tested | Not Tested | Not Tested |

**Table 13-21: Column Testing Acid Consumption Results Net Acid Consumption (lb/t)**

| Name | | 4600-01 | 4600-02 | 4600-03 | 4600-04 | 4600-05 | 4600-06 | 4600-07 | 4600-08 | AVG |
|---|---|---|---|---|---|---|---|---|---|---|
| Bottle Roll | Net | 6.9 | 6.7 | 7.5 | 11.7 | 16.9 | 16.1 | 22.2 | - | 12.6 |
| Column | Net | 5.6 | -9.6 | -1.5 | 12.5 | 1.3 | 33.2 | 0.1 | 1.5 | 5.4 |
| | Gross | 26.6 | 36.1 | 12.6 | 21.3 | 1.3 | 33.2 | 2.4 | 18.6 | 19.0 |
| Column lb acid/lb Cu | Net | 0.4 | -0.3 | -0.2 | 2.2 | 0.2 | 5.5 | 0.0 | 0.1 | 1.0 |

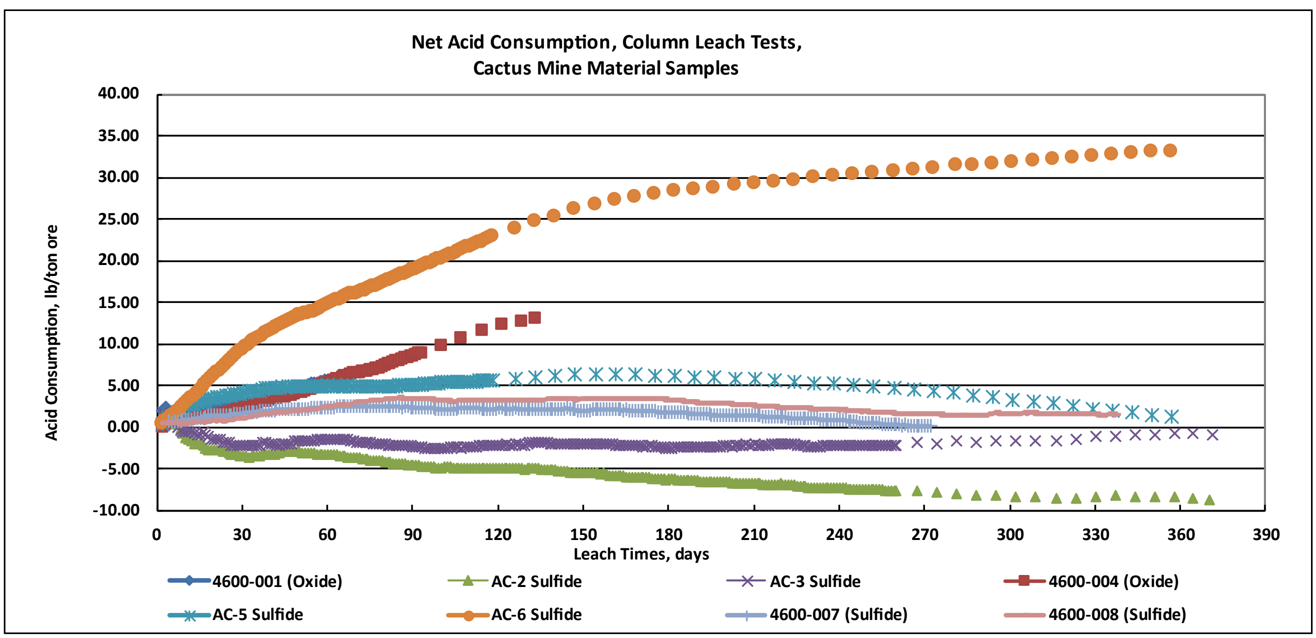
The higher copper grade in 4600-01 contributes to the lower net acid consumption and unit consumption results for the oxide columns. Gross acid consumption for the materials ranged from 1.3 lb/t to 36 lb/t as shown in Figure 13-15. 4600-06 shows that there is the presence of higher acid consuming gangue material such as biotite and chlorite (13.4.3 Composite Mineralogy). 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 13-15: Gross Acid Consumption Column Test Results**




Source: McClelland, 2023.

**Figure 13-16: Gross Acid Consumption Column Test Results**

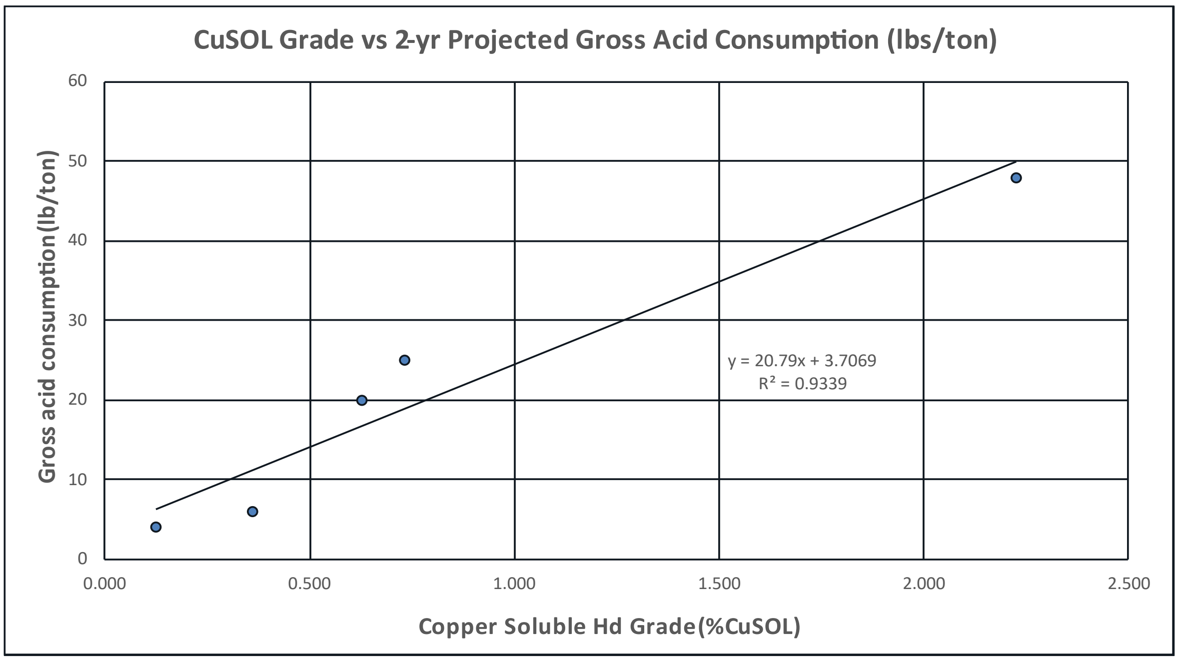



Source: McClelland, 2023.

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 13-16 and Figure 13-17 shows the indicative gross and net results from the column testing, respectively.

Due to the higher copper content and sulfide mineralization oxidation, the sulfide columns are presently net acid producing. Sulfide column 4600-06 is an outlier due to the biotite content (3.5%), chlorite content (0.34%). The net acid producing material may be an advantageous feature once sulfide material is mined. The relationship between copper grade and gross acid consumption is still under development and based on several factors, including recovery of copper and calcium content. However, a preliminary analysis is presented in Figure 13-16 that demonstrates the potential relationship at the Cactus Project.

**Figure 13-17: Copper Grade Versus Gross Acid Consumption**




Source: SE, 2023.

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 Hydro-Metallurgical Testwork – Parks Salyer

The Park Salyer metallurgical column test work was performed on-site at the True-stone facility and composite preparation performed at HydroGeoSense in Tucson, AZ. Three composites consisting of a low grade, mid grade, and high grade began column leach testing in November 2022 and completed leaching in September 2023. Daily and weekly operating logs measuring raffinate addition weight, PLS solution weight, acid titrations, and raffinate sources were incomplete between January 2023 and April 2023. Quality assurance review of the material has invalidated the kinetic results of the original Parks/Salyer tests. A new set of columns were brought online starting in June 2023 with improved data management procedures to ensure quality control and quality assurance of the test results.

The original invalidated columns were designated PS-7, PS-8, and PS-9. The new columns are designated PS-11, PS-12, and PS-13.

PS-7, PS-8, and PS-9 head assays and column residue assays are valid and will be utilized to compare PS-11, PS-12, and PS-13 given that PS-7 to PS-9 leached for 320 days whereas PS-11 to PS-13 leached between 140 and 205 days.

### 13.5.1 Sample Characterization

The drill core samples collected were shipped to HydroGeoSense in Tucson, Arizona, for preparation and analysis. The composites where then shipped back to ASCU to perform closed circuit column leach tests on-site at the Tru-Stone facility. A summary of the samples head assay information is provided in Table 13-22.

**Table 13-22: Parks/Salyer Composite Head Assay**

| Determination | % Cu | | |
|---|---|---|---|
| **Sample** | **PS-11** | **PS-12** | **PS-13** |
| Direct Assay, Init. | 0.74 | 1.17 | 1.41 |
| Direct Assay, (Seq. Assay) | 0.73 | 1.28 | 1.50 |
| *Calc'd., Head Screen | - | - | - |
| Calculated, Column Test | 0.763 | 1.20 | 1.34 |
| **Average** | **0.74** | **1.22** | **1.42** |
| Std. Deviation | 0.014 | 0.046 | 0.065 |
| Relative Std. Deviation, % | 1.9 | 3.8 | 4.6 |

*Head Screen assays only acid soluble and cyanide soluble copper
Source: HGS, 2023.

### 13.5.2 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. PMC's Rapid Ore Characterization (ROC) Report NOV 2023-08 dated January 2024 is summarized as follows.

- The examined samples show that the copper deportment is dominated by secondary copper minerals of chalcocite/digenite in all samples. Moderate to enormous amounts are present as primary sulphides of chalcopyrite / bornite with minor amounts as covellite or other copper silicates.
- Pyrite (py) is present in minor amounts (approximately 2-4%).

- All Parks/Salyer samples had minimal amounts of K-Feldspar and trace amounts of Biotite (~0.1%) both were significantly when compared to the mineralogy of cactus east/west composites.
- Clay minerals are present in all samples in minor amounts.
- Table 13-23 provides the PMC mineralogical determinations for the samples provided.

**Table 13-23: Park Salyer Composite Mineralogy**

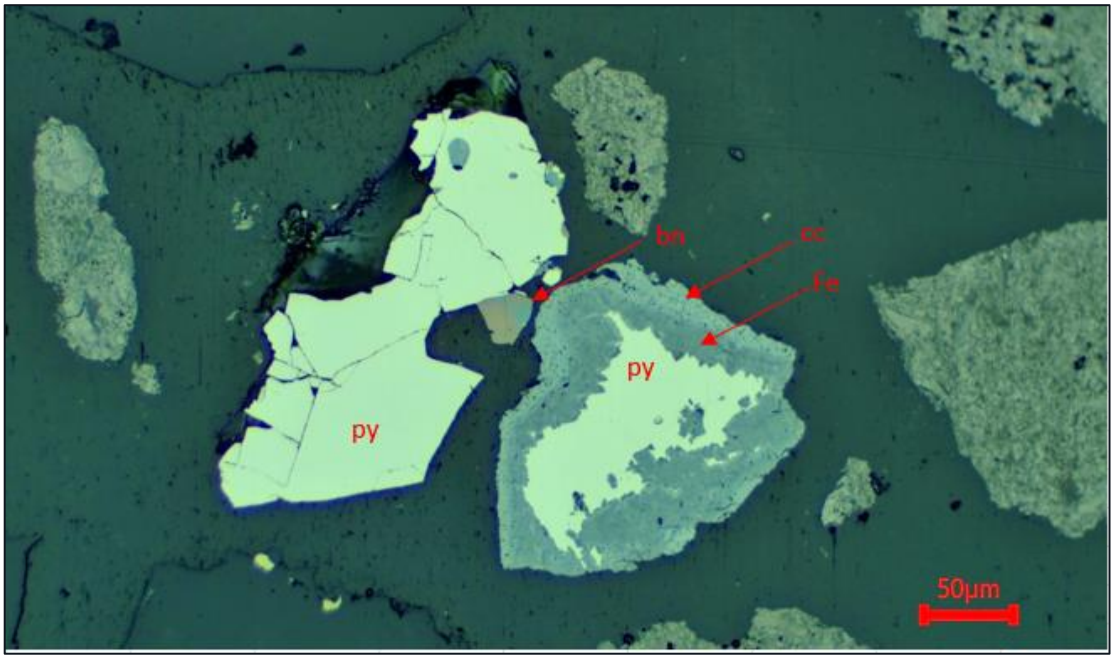
| Mineral Abundance | PS11 – LG | PS12 – MG | PS13 - HG |
|---|---|---|---|
| Mass % | 100.00 | 100.00 | 100.00 |
| Pyrite / Pyrrhotite | 3.71 | 3.21 | 2.73 |
| Chalcocite / Digenite | 1.19 | 1.34 | 3.13 |
| Covellite | - | - | - |
| Chalcopyrite / Bornite | 0.53 | 1.67 | 0.95 |
| Other Sulphides | 0.04 | 0.01 | 0.05 |
| Cuprous FE-Oxy Hydro | 0.02 | 0.03 | 0.00 |
| Alumino-Phospho-Sulphate | 0.03 | 0.00 | 0.01 |
| Quartz | 62.4 | 56.2 | 53.40 |
| Plagioclase | 3.28 | 2.07 | 2.04 |
| K-Feldspar | 0.08 | 0.27 | 0.31 |
| Muscovite / Sericite | 25.3 | 31.2 | 32.4 |
| Biotite | 0.10 | 0.07 | 0.10 |
| Clays | 0.70 | 1.21 | 0.46 |
| Other Minerals | 2.67 | 2.73 | 4.38 |
| **Total** | **100.00** | **100.00** | **100.00** |
| **Copper Deportment** | | | |
| Chalcocite / Digenite | 80 | 51.8 | 80.1 |
| Covellite | - | - | - |
| Chalcopyrite / Bornite | 19.26 | 47.68 | 19.8 |
| Cu gangue (silicates & sulphides) | 0.73 | 0.54 | 0.07 |
| **Total** | **100.00** | **100.00** | **100.00** |

Source: PMC, 2024.

Photomicrographs of polished sections were also completed for the samples evaluated. Figure 13-18 is from the PS11-LG composite and presents an SEM-BSE image showing two pyrite grains, where one pyrite grain is rimmed with iron oxide and secondary Cu sulfides similar to observations.

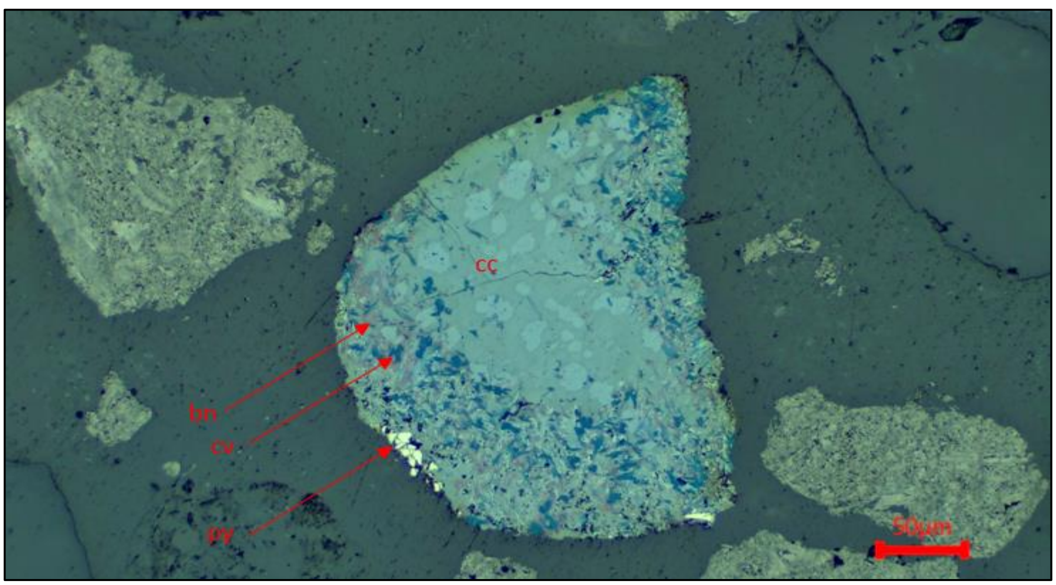
The Fe-oxy / hydroxide contains minor Cu (hence cuprous) and fine-grained inclusions of secondary Cu sulfides (white, SecCuS) such as covellite / chalcocite. Coarser pyrite grains show rims of secondary Cu sulfides similarly observed in sample 4600-02.

**Figure 13-18: Sample PS11-LG Column Composite Material**




Source: PMC, 2024.

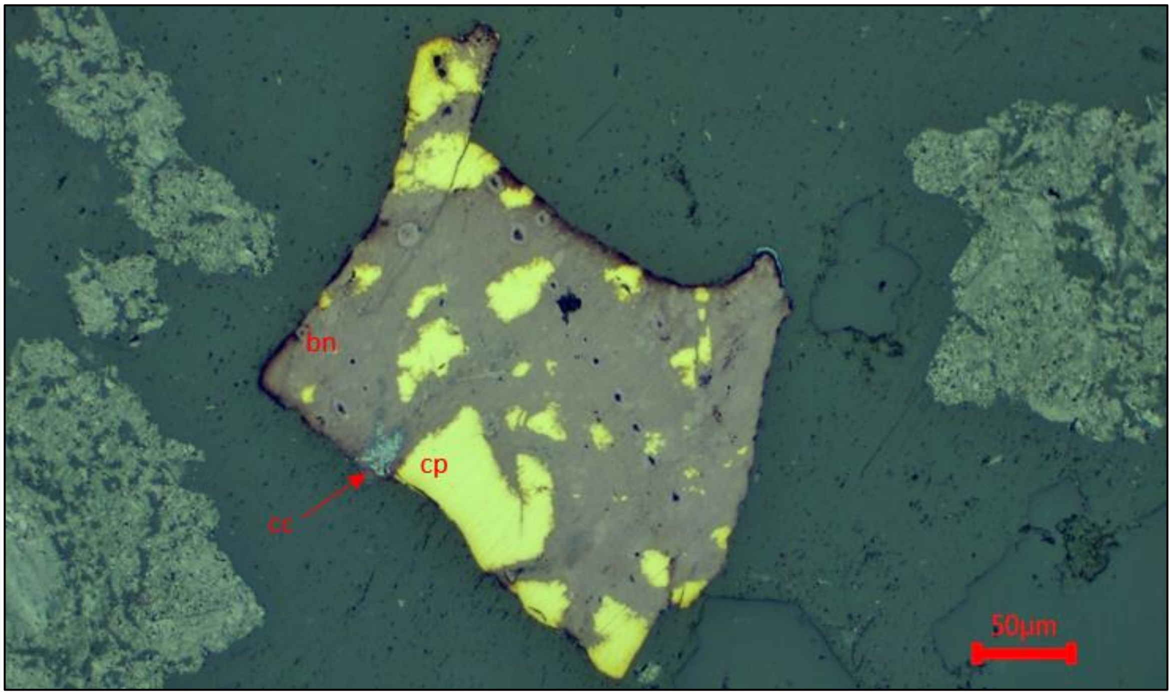
**Figure 13-19: Sample PS13-HG Column Composite Material**




Source: PMC, 2024.

Figure 13-19 shows a secondary copper sulfide mineral that is primarily chalcocite with complex intergrowth of bornite, covellite, and pyrite. Figure 13-20 shows a well liberated pyrite grain that is associated with chalcopyrite and chalcocite.

**Figure 13-20: Sample PS13-HG Column Composite Material**




Source: PMC, 2024.

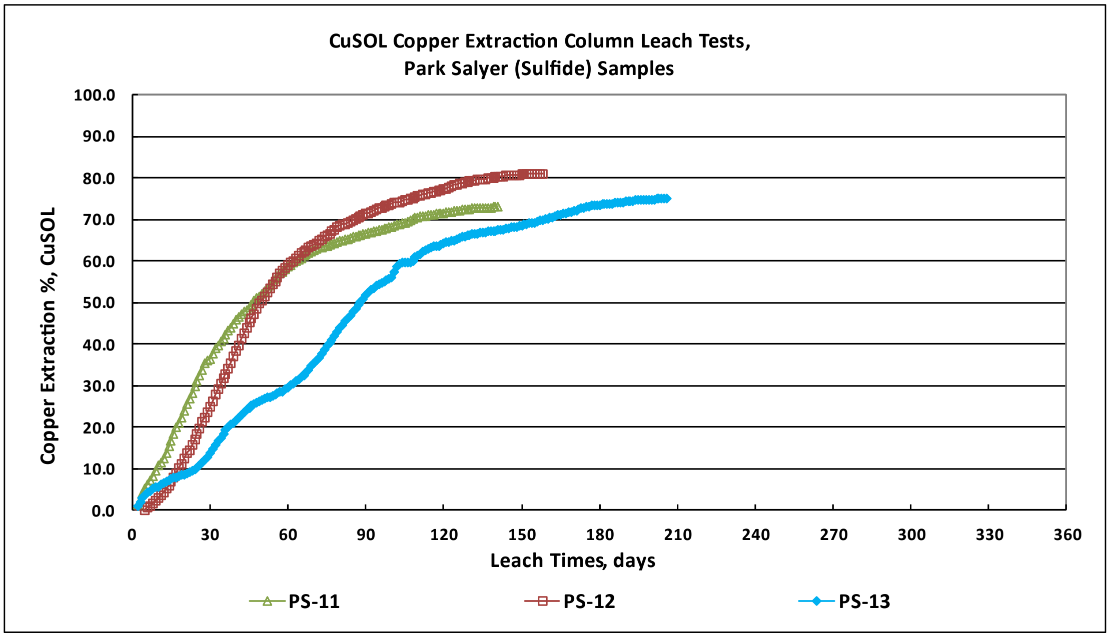
### 13.5.3 Park Salyer Copper Recovery

Tests are conducted in 6-inch I.D. by 20 ft tall columns containing approximately 440 lb (200 kg) of material. Column leach testing in closed circuit with SX was incorporated with saved solution from the previous testing at McClelland as a starting solution.

All copper recoveries are based on calculated head assays from solution samples are column residue characterization.

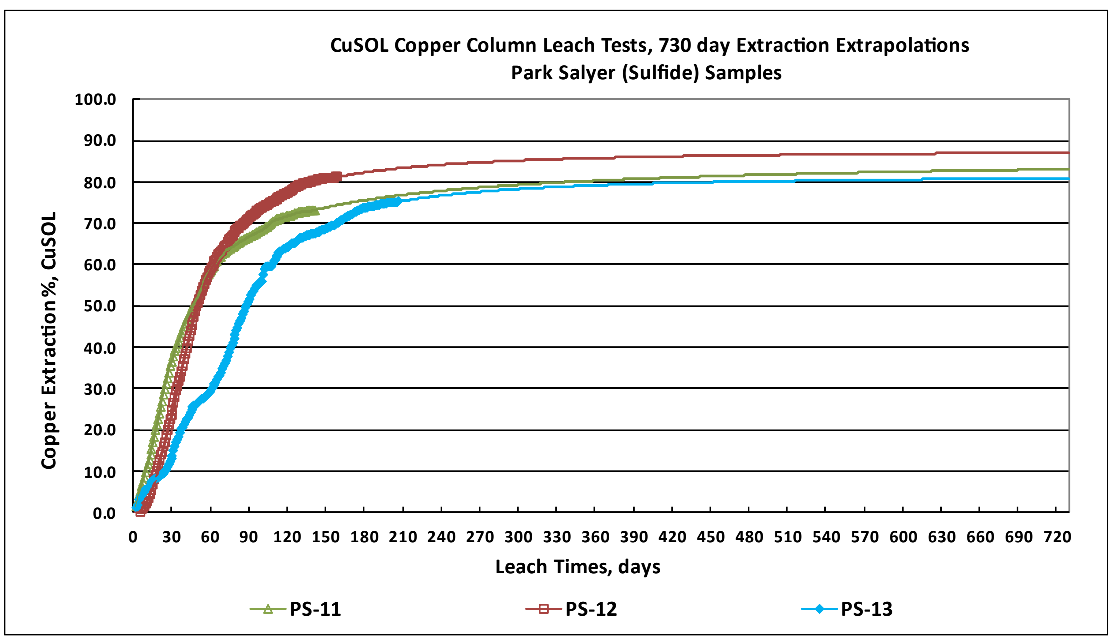
Results for the Park Salyer columns PS-11 to PS-13 are consolidated in Figure 13-21, Figure 13-22, Figure 13-23, Table 13-24 and Table 13-25.

**Figure 13-21: Soluble Copper Extraction for Park Salyer Sulfide Columns**



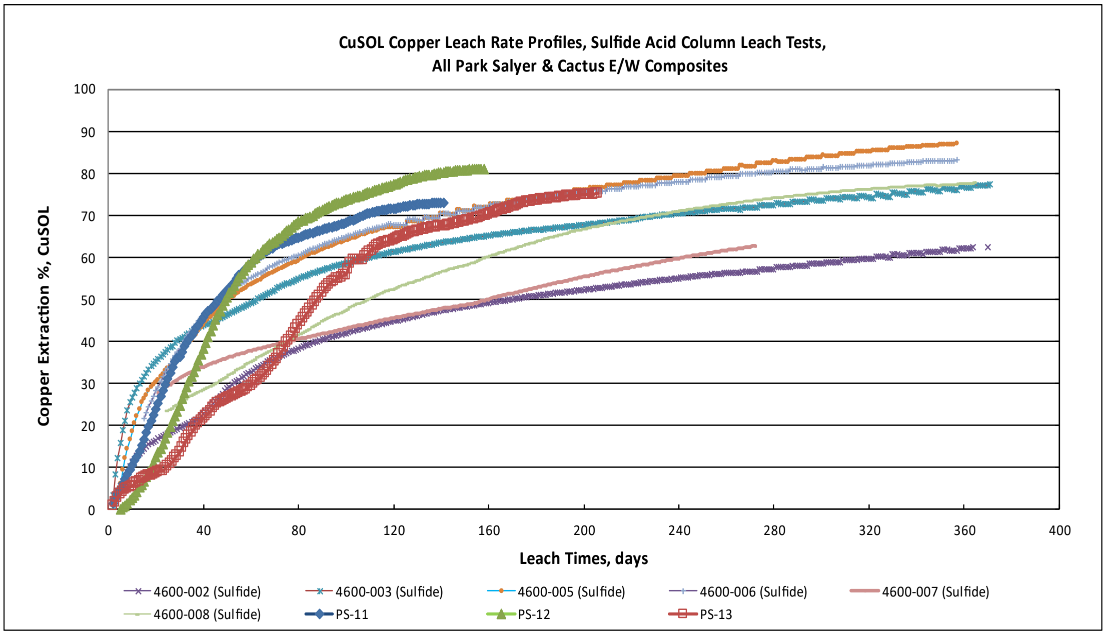

Source: ASCU, 2023

**Figure 13-22: Extrapolated Two Year Long-Term Copper Extraction for Park Salyer Sulfide Columns**




Source: ASCU, 2023

**Figure 13-23: Soluble Copper Extraction for all Park Salyer and Cactus E/W Sulfide Columns**




Source: ASCU, 2023

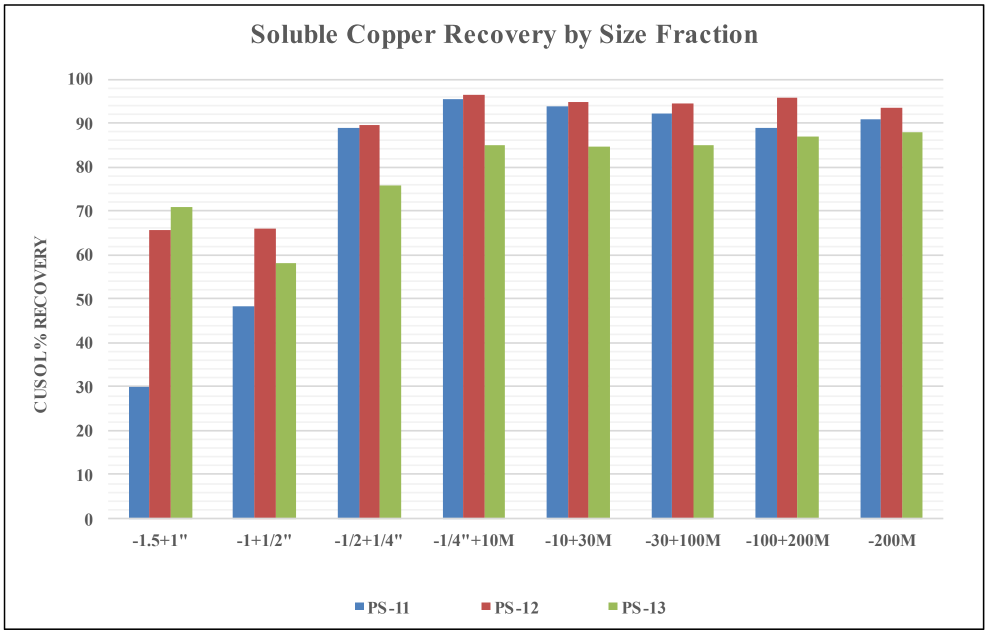
**Table 13-24: Park Salyer Sulphide Column Results**

| Summary Metallurgical Results, Oxide Acid Column Leach Tests, Cactus Mine Bulk Samples, -3 inch Feed Size | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| ASCU Test No. | Sample | Leach/ Rinse Time, days | TCu Recovery % | Extracted | % Cu Calc'd. Tail Head | | Avg. Head | $H_2SO_4$ Added lb/ton ore | $H_2SO_4$ Consumption | | |
| | | | | | | | | | Gross lb/ton ore | Gangue lb/ton ore | Specific (Gauge) lb/lb Cu |
| PS-11 | Low Gr | 141 | 64.5 | 0.492 | 0.271 | 0.763 | 0.74 | 35 | 4.8 | -10.3 | -1.0 |
| PS-12 | Mid Gr | 158 | 72.3 | 0.865 | 0.332 | 1.197 | 1.22 | 40 | 11.6 | -15.1 | -0.9 |
| PS-13 | High Gr | 206 | 73.4 | 0.981 | 0.356 | 1.337 | 1.42 | 43.9 | 10.3 | -19.9 | -1.0 |

Results indicate that Park Salyer material should perform in a similar manner to the Cactus East/West sulfide material.

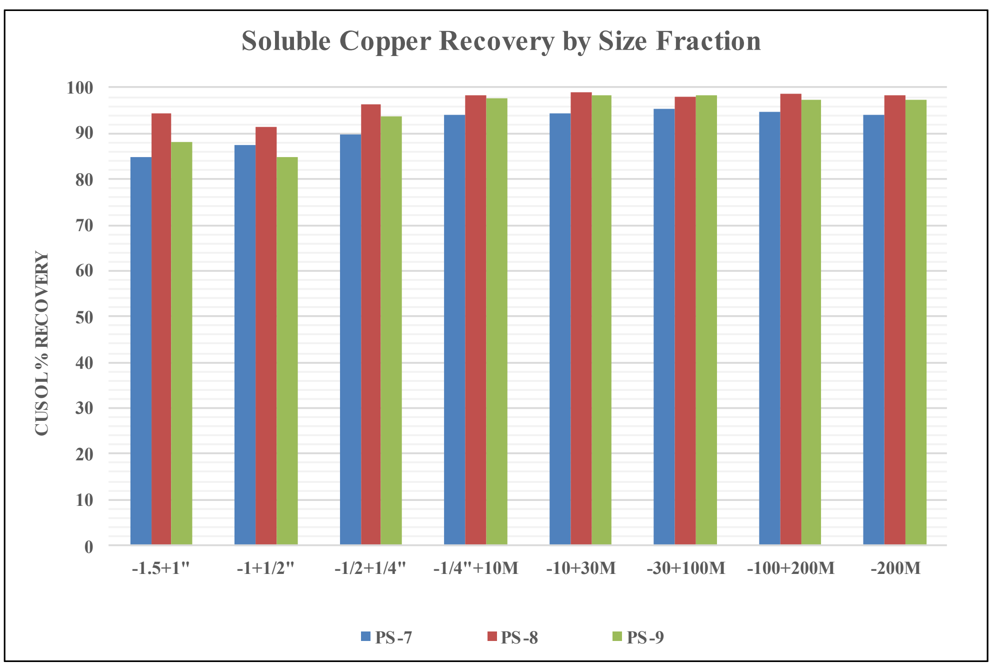
Soluble copper-based extraction for all three Parks/Salyer columns indicate leach kinetics that are similar or better than Cactus East/West as indicated in Figure 13-24.

**Figure 13-24: Soluble Copper Recovery by Size Fraction, PS11 to PS13**




Source: ASCU, 2023.

**Figure 13-25: Soluble Copper Recovery by Size Fraction, PS7 to PS9**




Source: ASCU, 2023.

A breakdown of copper recovery for CuAS (acid soluble), CuCN (cyanide soluble), Tsol (combined CuAS and CuCN) and the CuT is presented in Table 13-25. Extractions are based on head and tail assay data for the column tests.

**Table 13-25: Copper Extraction by Copper Assay Method – Park Salyer Columns**

| Name | PS-11 | | PS-12 | | PS-13 | | AVG (%) | Predicted/Modeled | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Assay % | Ext % | Assay % | Ext % | Assay % | Ext % | | 730 days | YR 1 | YR 2 | YR3 |
| CuAS | 8.4 | 79 | 7.2 | 84 | 8.9 | 86 | 83 | - | - | - | - |
| CuCN | 77.5 | 71 | 92.8 | 84 | 89.2 | 78 | 78 | - | - | - | - |
| Tsol | 85.9 | 73 | 100 | 81.2 | 96.2 | 75.2 | 76 | - | - | - | - |
| Tsol Pred | - | - | - | - | - | - | - | 84% | 50.4% | 29.4% | 4.2% |
| CuT | - | 64.5 | - | 72.3 | - | 73.4 | 70 | - | - | - | - |

CuAS copper content extraction averaged 83% for the three sulfide columns completed. CuCN recovery, representing enriched copper mineralization content (chalcocite and covellite) was also significant and averaged 78%. The combined Tsol extraction averaged 76% for the material tested.

The screen analysis for the samples as loaded into the columns is presented in Table 13-26. 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 13-26: Column Screen Size Analysis– Park Salyer Columns**

| Size Fraction | PS-11, % | | PS-12, % | | PS-13, % | |
|---|---|---|---|---|---|---|
| | Head | Tail | Head | Tail | Head | Tail |
| +3" | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| -3+1.5" | 2.9 | 0.0 | 3.9 | 0.0 | 0.0 | 0.0 |
| -1.5+1" | 8.8 | 13.4 | 10.7 | 11.8 | 2.9 | 2.3 |
| -1+1/2" | 21.9 | 21.0 | 23.4 | 21.6 | 20.6 | 16.6 |
| -1/2+1/4" | 16.1 | 17.6 | 16.4 | 18.7 | 21.5 | 22.3 |
| -1/4"+10M | 18.9 | 18.4 | 18.0 | 21.3 | 27.2 | 31.2 |
| -10+30M | 14.0 | 9.8 | 12.3 | 8.7 | 14.2 | 9.8 |
| -30+100M | 6.8 | 7.6 | 6.3 | 6.8 | 3.3 | 7.2 |
| -100+200M | 2.2 | 2.5 | 2.1 | 2.2 | 2.4 | 2.5 |
| -200M | 8.3 | 9.6 | 6.9 | 8.8 | 7.9 | 8.2 |
| Composite | 100 | 100 | 100 | 100 | 0.0 | 0.0 |
| $P_{80}$ | ~3/4" | ~3/4" | ~3/4" | ~3/4" | ~3/4" | ~3/4" |

Source: HGS, 2023.

Copper extraction was also considered by size fractions to assess the impact of particle size. Figure 13-23 shows the relative copper extraction by size fraction for PS-11 to PS13 and Figure 13-23 shows the relative copper extraction by size fraction for PS-7 to PS-9.

Copper extraction appears to be impacted in particles sizes over ½” inch in the 150–200-day tests, but PS-7 to PS-9 indicate that longer leach times significantly increase copper extraction in coarse particle sizes. 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.

### 13.5.4 Parks/Salyer Sulfuric Acid Consumption

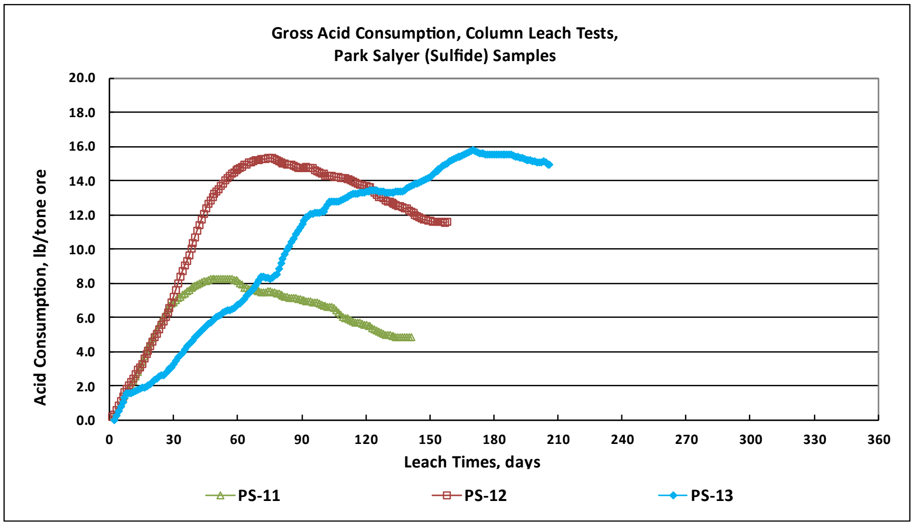
In general, the acid consumption indicated is significantly lower than the open pit and stockpile materials tested so far. The Park Salyer column test acid consumptions are shown in Table 13-27.

**Table 13-27: Park Salyer Column Testing, Acid Consumption Results (lb/ton)**

| Name | | PS-11 | PS-12 | PS-13 | AVG |
|---|---|---|---|---|---|
| Column lb acid/ton ore | Net | -10.3 | -15.1 | -15.3 | -13.6 |
| | Gross | 8.3 | 13.7 | 15.8 | 12.6 |
| Column lb acid/lb Cu | Net | -1.0 | -0.9 | -0.8 | -0.9 |

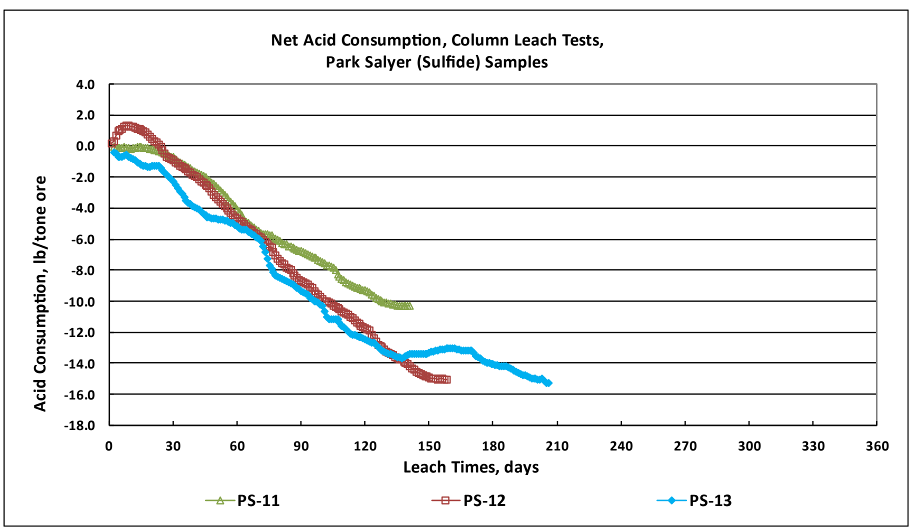
The higher copper grades in PS-12 and PS-13 contribute to the lower net acid consumption and unit consumption results. Gross acid consumption for the materials ranged from 8.3 lb/t to 15.8 lb/t as shown in Figure 13-26.

**Figure 13-26: Gross Acid Consumption Column Test Results**




Source: ASCU, 2023

**Figure 13-27: Gross Acid Consumption Column Test Results**




Source: ASCU, 2023

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 13-26 and Figure 13-27 shows the indicative gross and net results from the column testing, respectively.

The acid consumption model for Park Salyer can be modeled at 16 lb/t with recommendation for additional variability columns to validate a different model for Park Salyer ore compared to Cactus East/West. The column test work shows raffinate and PLS sample free acid titrations begin to converge at approximately 16 lb/t, indicating a plateau of gross acid consumption. The columns test work acid consumption results coupled with the minimal amount of acid consuming gangue Figure 13-27 identified in mineralogy of the head samples point to lower overall acid consumption in Park Salyer compared to Cactus West/East.

## 13.6 Concentrator Opportunity Scoping

### 13.6.1 Introduction

ASARCO mined material for the Sacaton West ore body and milled ore containing primarily primary sulphide 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 sulphide 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.

### 13.6.2 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-ton coarse ore bin from which it passed through a vibrating grizzly feeder. The grizzly oversize reported to a primary 48-in by 60-in Allis- Chalmers jaw crusher. The primary crusher discharge and grizzly undersize (6-in) 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 in) reported to a 200-ton 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-in) reported to one of two fine ore bins, which each had a live capacity of 3,250 ton. 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-in 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 $ft^3$ 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-ft 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 $ft^3$ cells), scavengers (each bank containing five 100 $ft^3$ cells) and second cleaners (each bank containing three 100 $ft^3$ 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:

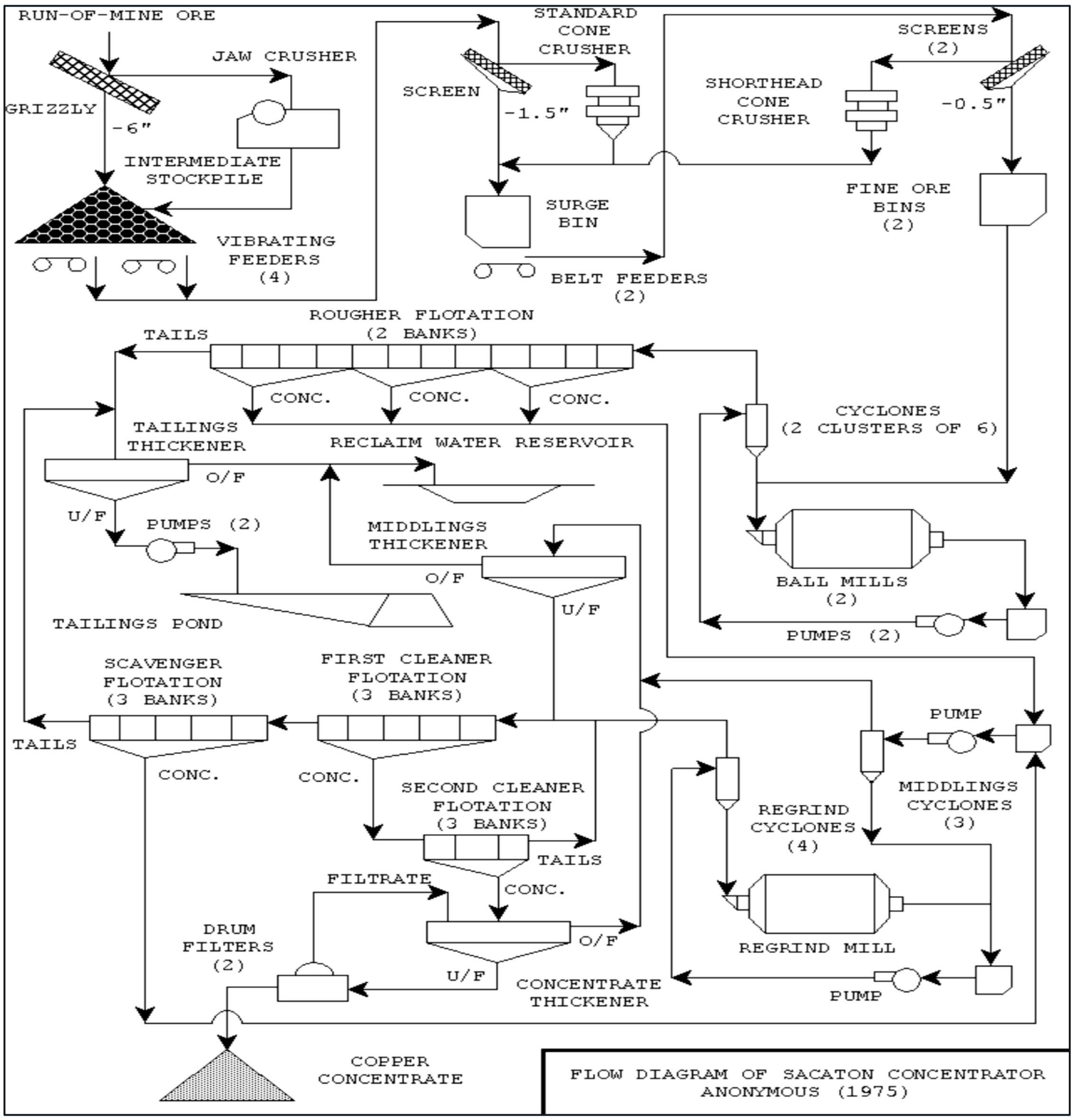
- Sulphide Ore Reagent Scheme:
  - Lime: 2.0 lb/t (1975)
  - A-238: 0.021 lb/t (1975)
  - Z-6: 0.020 lb/t (1975)
  - Frother: 0.06 lb/t (1975)
- Oxide Reagent Scheme:
  - NaHS: 0.9 lb/t (1975)

- 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 13-28 as included in the Briggs 2004 information.

**Figure 13-28: Historic Sacaton Concentrator Flow Diagram**

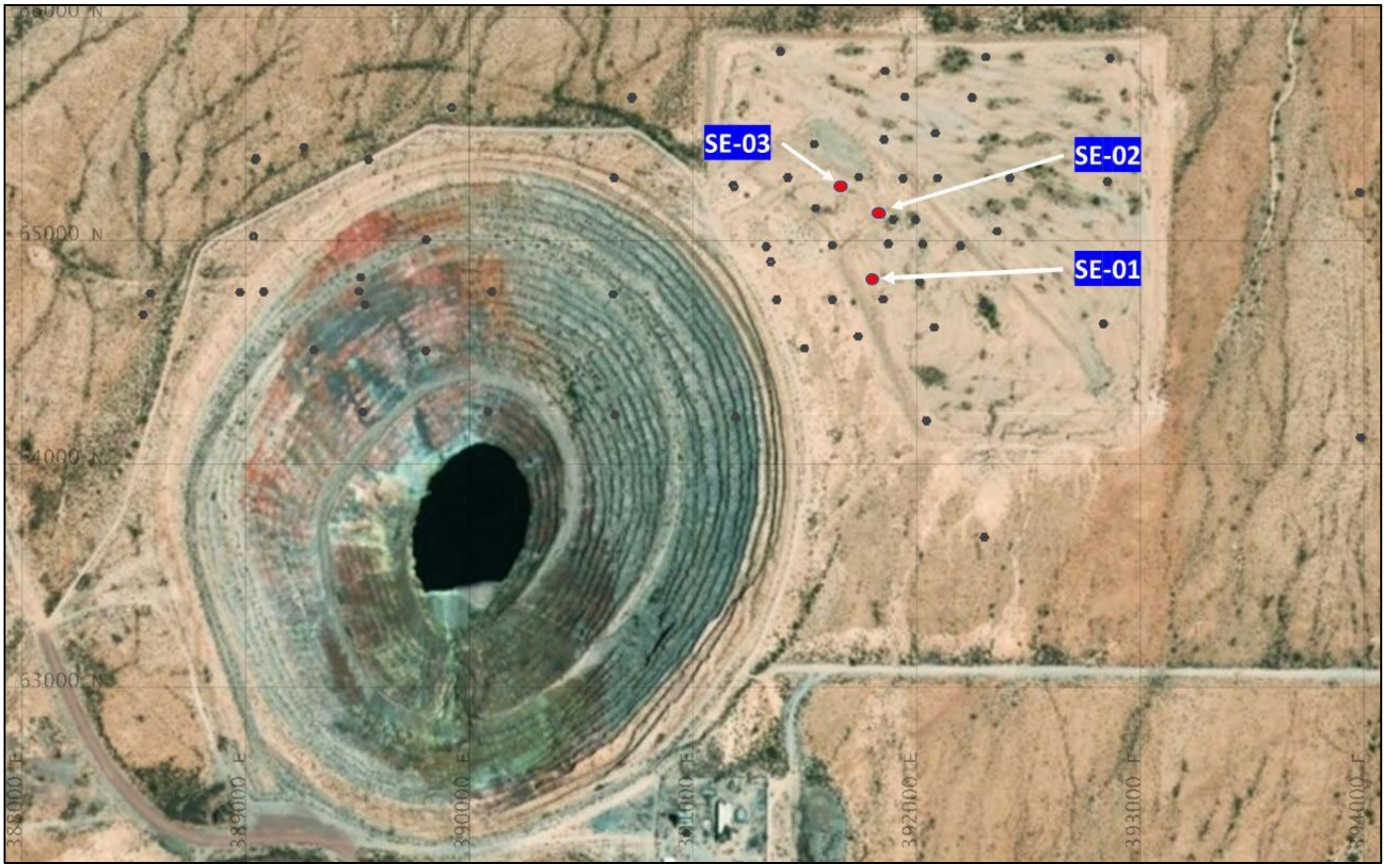



Source: Briggs, 2004.

### 13.6.3 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 13-29.

**Figure 13-29: Metallurgical Sample Drillhole Location**




Source: ASCU, 2021.

- Sample 1666808 Description – Hole SE-01 (1,633 ft to 1,703 ft).
  - 1.38% Cu head grade, secondary copper sulphides (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.

### 13.6.4 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 March 9, 2021.

The SMC Test results for the 4655-001 sample from Cactus Project are given in Table 13-28. This table includes the average rock density and the Dwi (Drop-Weight index) that is the direct result of the test procedure.

**Table 13-28: SMC Test Results**

| 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 |

Source: Hazen/JKTech 2021.

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 13-29.

**Table 13-29: Parameters Derived by JKTech**

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

Source: JKTech 2021.

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.

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 13-30. Results indicate a medium rating for the sample tested.

**Table 13-30: Bond Ball Mill Work Index Testing Results**

| Name | BBWi | Unit |
|---|---|---|
| Ball Mill Work Index | 11.29 | kW-h/ton |
| | 12.45 | kW-h/ton |
| Ball Mill Work Index Classification | Medium | - |

Source: McClelland 2021.

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 13-31.

Table 13-31: Comminution Energy Requirements (JKTech, 2021)

| Sample Designation | Particle Size (mm) | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 14.5 | | | 20.6 | | | 28.9 | | | 41.1 | | | 57.8 | | |
| | t10 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 |

Source: JKTech, 2021.

### 13.6.5 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 $P_{80}$ of approximately 200 mesh for all tests, commensurate with the reported grind used by ASARCO.

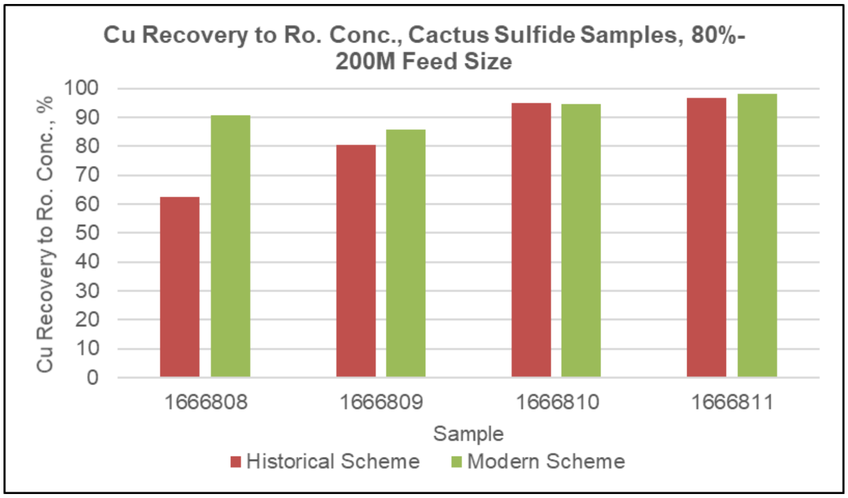
Two regent schemes were employed, the first to simulate the response using the ASARCO reagent scheme and the second in consultation with Solvay as an initial demonstration of what more modern reagent schemes targeting chalcocite/chalcopyrite could do to simplify and improve upon the ASARCO flowsheet.

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 13-30. 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.

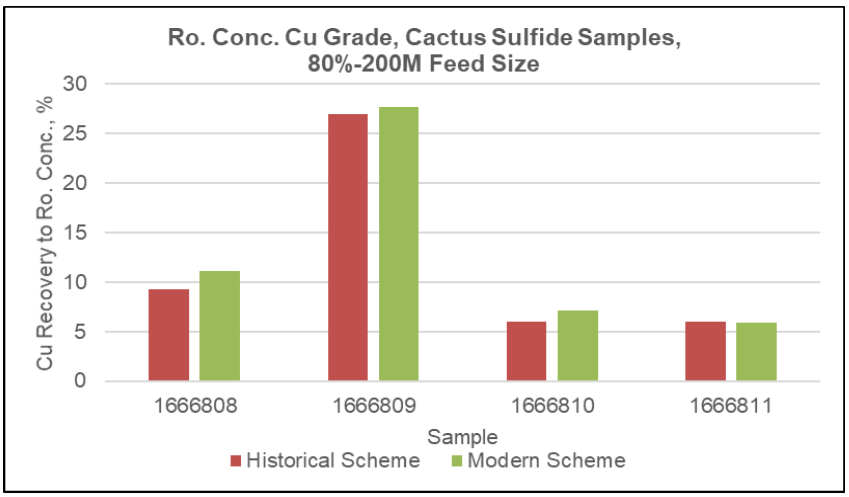
The associated rougher concentrate grade is also presented in Figure 13-31. These results are typical and higher-grade feed resulted in higher grade concentrates. The very high rougher concentrate grade is not typical and may be influenced by the presence of native copper and may also indicate that an 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 13-30: Preliminary Rougher Recovery Results**




Source: McClelland 2021.

**Figure 13-31: Rougher Concentrate Grade**




Source: McClelland 2021.

## 13.7 Results Summary and Conclusions

The QP believes the metallurgical testing and data collected to date is sufficient to establish the required supporting metallurgical performance expectations used in estimating the project Reserves and economics for the Stockpile, Cactus East, Cactus West and Parks/Salyer deposits included in the Cactus Project. However, only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit and additional confirmatory work is required to better understand the deposit variability.

### 13.7.1 Metallurgical Performance Recommendations

The Cactus heap leaching process design includes crushing of all material types for leaching to a minus ¾” $P_{80}$ size. All material types, oxides and enriched are to be leached in a single pad with an initial leaching cycle of 180 days. A maximum 3-year leaching cycle has been assumed (3 lifts) as the practical limit for effective recovery based on experience and hydrodynamic analysis of the materials by HGS. The copper leaching metallurgical test data has been extrapolated from the testing data at one year based on the rates prevailing after one year using a logarithmic curve fit projection that considers the decaying rate of copper extraction.

Based on the above, the recommended copper extraction estimates for use in evaluating the Cactus Project resources is presented in Table 13-32.

**Table 13-32: Copper Recovery by Sequential Assay Fraction**

| Resource Area | Units | Value |
|---|---|---|
| Stockpile Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 87.7 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Oxide Heap (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 93.1 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Enriched Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 91.2 |
| Cyanide Soluble Copper Recovery | % | 84.5 |

Applying these extraction criteria, the calculated overall soluble copper (Tsol) recovery to cathodes is 86.3% and the corresponding total copper recovery is 76.1% for the resources contained in the mine plan.

Scalability has been considered by 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. The recommended copper recovery projections include this efficiency factor applied to the extraction obtained from the column testing.

A production timing has been assigned for each material type corresponding to the material mined in one year and the expected delays in achieving the final recovery values. The oxide copper dominant ore types (stockpile and Cactus) leach more quickly and a 75% Year 1 and 25% Year two distribution factor is recommended for use in the production plan. Enriched ore types that are more sulfide copper dominant ore types (enriched) leach slower due to the bio-oxidation delay and a 65% Year 1, 30% Year 2 and 5% Year 3 distribution factor is recommended for use in the production plan. These factors s intended to account for material placement timing over the course of a year and leach cycle delays in subsequent new lift placements.

Sulfuric acid consumption per ton of materail leached is dependent on several factors. Gross acid consumption varies by materail type in each deposit. Net acid consumption acounts for acid regenerated in the electrowinning process when copper is plated to product. Net acid consumption per ton of material is dependent on recoverable copper content with a stochiometric converion of 1.54 tons of acid generated per ton of copper plated in electrowinning.

The stockpile material is more complex given that there are no geologic constraints to apply and waste materials have been mixed in. Calcium was determined to provide a measurable indicator for acid consumption and a calcium distribution model was developed and applied to estimate gross acid consumption. For the materails included in the mine plan, the average gross acid consumption averages 22 lbs of acid per ton of material and ranges from 16.7 lbs/ton to 25.7 lbs/ton depending on the average calcium content annually.

For Cactus East and West materials, the gross acid consumption for the oxide dominant material is 22 lbs/ton from the column testing. Enriched material gross acid consumption is slightly lower at 21 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals.

For the Parks Salyer enriched material, gross acid consumption is lower at 16 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals, lower clay, biotite and calcite mineralogy when compared to Cactus samples tested.

Applying the specific gross acid consumption for each material the overall LOM gross acid consumption is calcualted to be 19.3 lbs per ton and varies from 27.0 lbs/ton to 15.7 lbs/ton in a given year. The LOM Net acid consumption is calculated to be 6.5 lbs per ton and varies from 15.7 lbs/ton to net acid generating in a given year. Years where acid regenrated exceeds the acid required to be consumed will need to be attenuated with low grade/high calcium content material from the stockpile or tailings.

Similar to copper recovery, acid consumption is recommended to be distributed over a two-year period with 75% of the estimated acid requirement consumed in the year of placement and 25% of the requirement in the following year. This also accounts for periods of time pausedwhen the active solution application is pasued for heap lift construction lifts.

### 13.7.2 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 were 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 (March 11, 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 MINERAL RESOURCE ESTIMATES

The Cactus Project resource was estimated in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by CIM council on November 29, 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. The Cactus West deposit is approximately 4,300 ft (1,300 m) long and 4,300 ft (1,300 m) wide. It averages about 900 ft (275 m) thick and sits near surface. The deposit is drill limited and open to the southwest and northeast. Cactus East is about 2,600 ft (800 m) long, 3,400 ft (1,000 m) wide, and averages 700 ft (210m) thick. The mineralized zone sits about 1,100 ft (335 m) below the surface. The Mineral Resource estimate includes all drilling, geological logging, and historical mapping completed prior to April 29, 2022, and mining depletion of the historical pit mined by ASARCO between 1972 and 1984.
- Cactus Stockpile Project – an historic mineralized stockpile generated as the result of waste dumping from the historic Sacaton pit. The stockpile is about 3,900 ft (1,200 m) long, 4,900 ft (1,500 m) wide and is comprised of 3 lifts, each with a maximum of 40’ (12m) height. Material historically considered as waste included all oxide material, sulphide material considered below the mining CoG of 0.3% CuT, and sulphide material above the mining CoG but where the oxide component was considered too high. The Mineral Resource estimate includes all drilling, geological logging, historic pit dump information, and topographical updates from rehabilitation work to March 1, 2022.
- 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. The defined resource is about 4,000 ft (1,200 m) long, 3,400 ft (1,000 m) wide and averages about 1,280 ft (390 m) thick. The mineralized zone sits about 1,100 ft (335 m) below the surface. This Mineral Resource estimate undertaken for the Parks/Salyer deposit includes all drilling and geological logging completed prior to May 19, 2023.

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 percentages.

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

## 14.1 Cactus Project Deposits

The OK method was used for the estimation of copper grades to the models. Variogram analysis and copper grade 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 neighbour (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_20220129.ddh.isis." The drillhole database used for the Parks/Salyer resource estimation was called "cacdrilling_mx_resource_ps_20230519.ddh.isis."

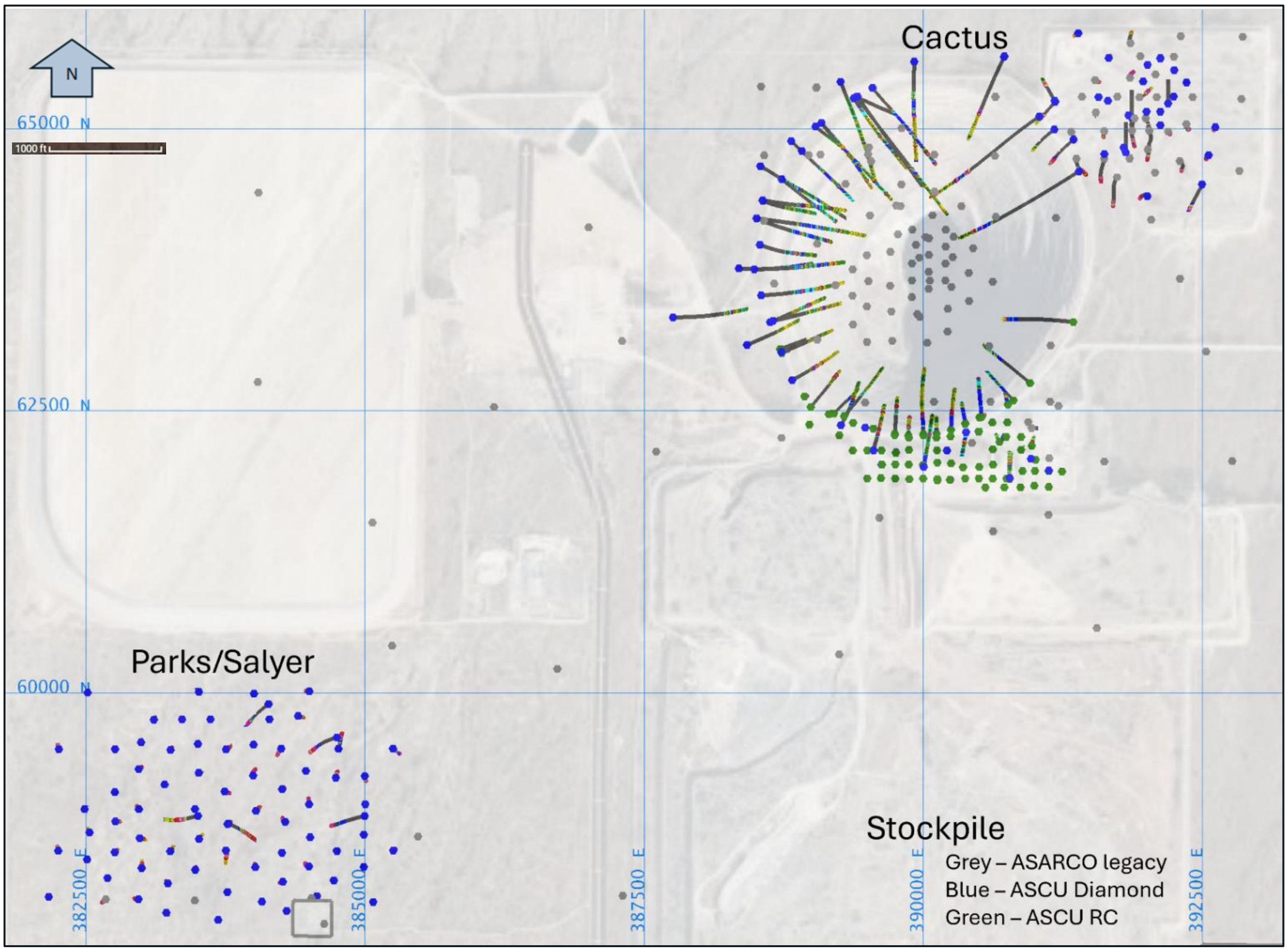
Lithology logging was used to build broad lithological zones that control where potential mineralization could occur and the assignment of specific gravity to the model. Mineralization logging, in addition to sequential copper assaying and historical mapping, was used to determine the main copper mineral 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 267 total holes. This is inclusive of 155 recent drill holes drilled by ASCU since 2019. 149 of these holes were drilled into the Cactus Deposits to support the resource estimate.
- The Parks/Salyer drill hole database contains 77 holes supporting the resource estimate, composed of 74 modern holes drilled by ASCU since 2021 and three 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 mostly drilled vertically. Angled holes were also drilled to support geotechnical analysis and as a check on the interpretation of geology.
- 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**




Source: ASCU, 2022.

#### 14.1.1.1 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 small 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. Calculations were undertaken on the raw drill hole database intercepts prior to compositing. based on Table 14-1. Back-calculated Tsol grades represent only 3.5% of the total Cactus resource database.

**Table 14-1: Values Used to Back calculate missing Tsol Grades**

| Deposit | Minzone | TSol % | CuAS % |
|---|---|---|---|
| CE/PS | Leached | 90 | 99 |
| | Oxide | 97 | 80 |
| | Enriched | 98 | 18 |
| CW | Leached | 90 | 99 |
| | Oxide | 81 | 60 |
| | Enriched | 98 | 18 |

#### 14.1.1.2 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.

Within Cactus, molybdenum (Mo) is present through the deposit but has only been reported on in limited drill hole composites and some recent drill holes. At Parks/Salyer 98% of the copper assay intervals contain Mo assay values. These have been used to estimate Mo in that block model. Future work is planned to re-assay primary material as a potential value addition to the Project.

Gold, silver, and molybdenum are not considered recoverable through planned copper heap leaching applications.

### 14.1.2 Geological Modelling

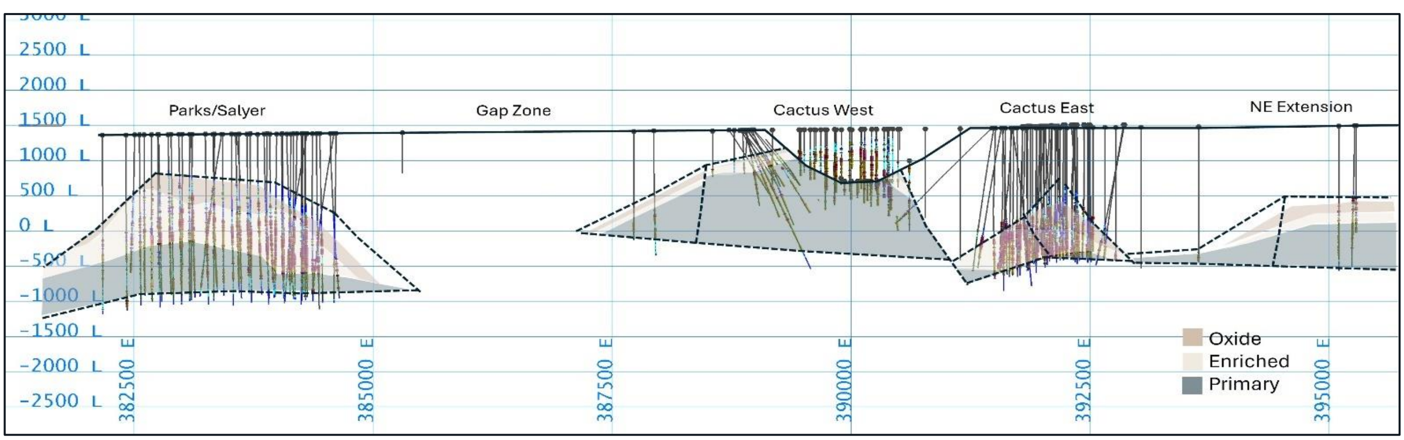
#### 14.1.2.1 Faults

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 4 mi 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 low angle to the north-west. Drilling completed to date at Parks/Salyer has not identified any major vertical or near vertical faults that offset the mineralized package.

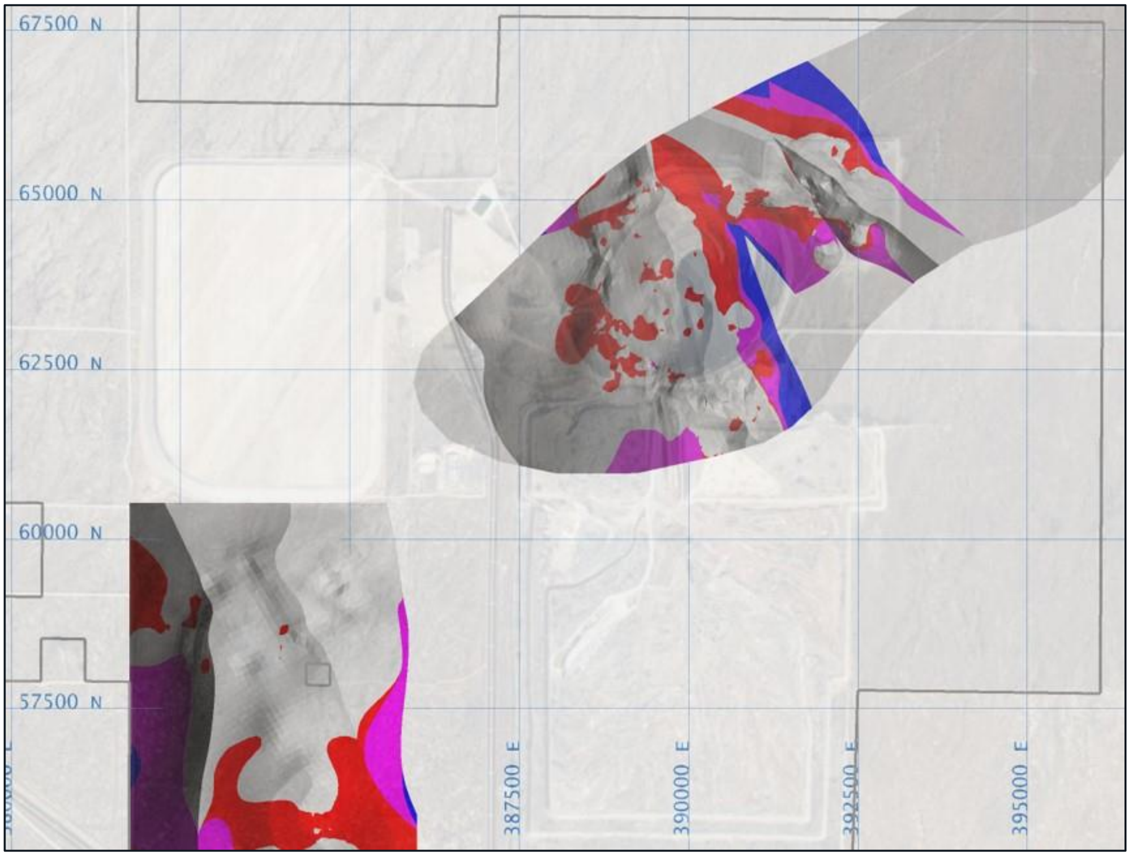
The Sacaton and East faults define the eastern edges of the Cactus West and Cactus East blocks. These represent normal faults that strike approximately 160° and dip between 50°-70° to the east. Blocks were down dropped to the east along these faults. A conjugate set of normal faults, accommodating basement 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 85° 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**




Source: ASCU, 2022.

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




Source: ASCU, 2022.

#### 14.1.2.2 Lithology

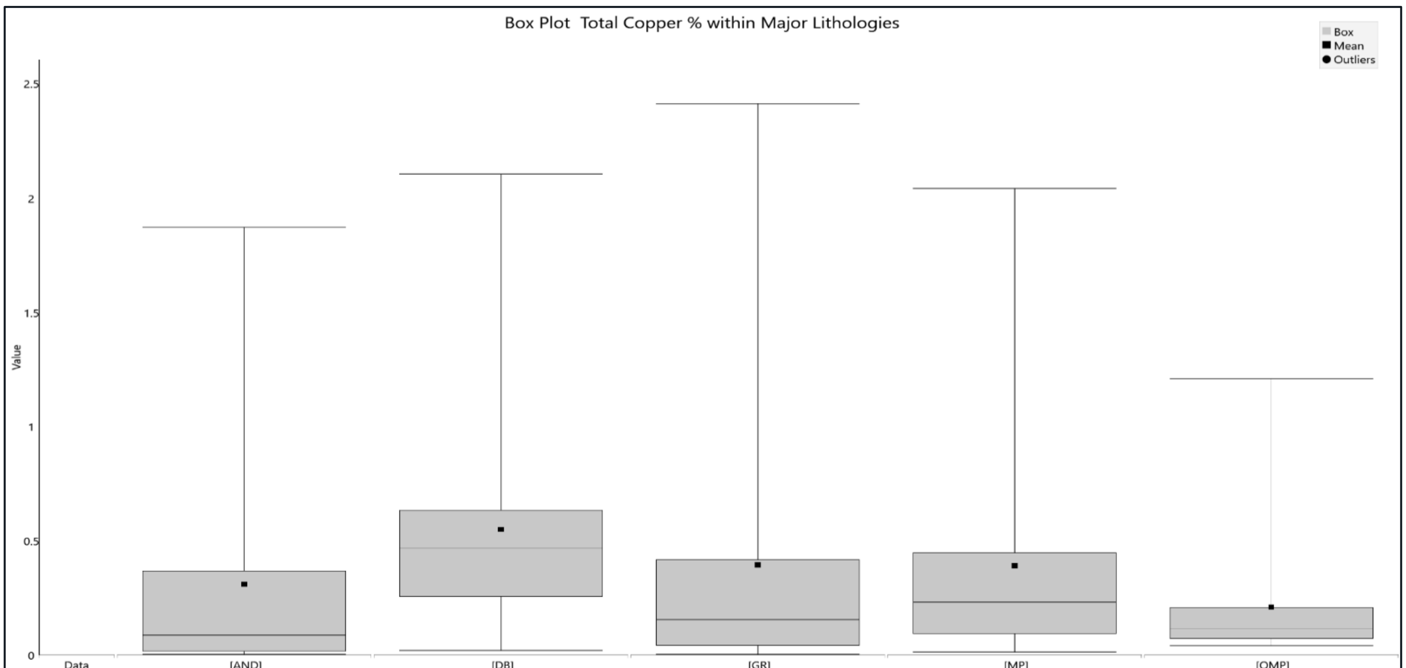
Lithology was grouped into multiple domains within the Cactus Project that relate to the presence or absence of mineralization. The main lithological domains modelled are defined in Table 14-2 along with the expected presence or absence of mineralization in that domain.

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. Dikes are generally a late feature in the system and have been modelled and estimated separately due to their different grade characteristics. Dikes represent only ~1% of the mineralised material. 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 14-2: 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 |

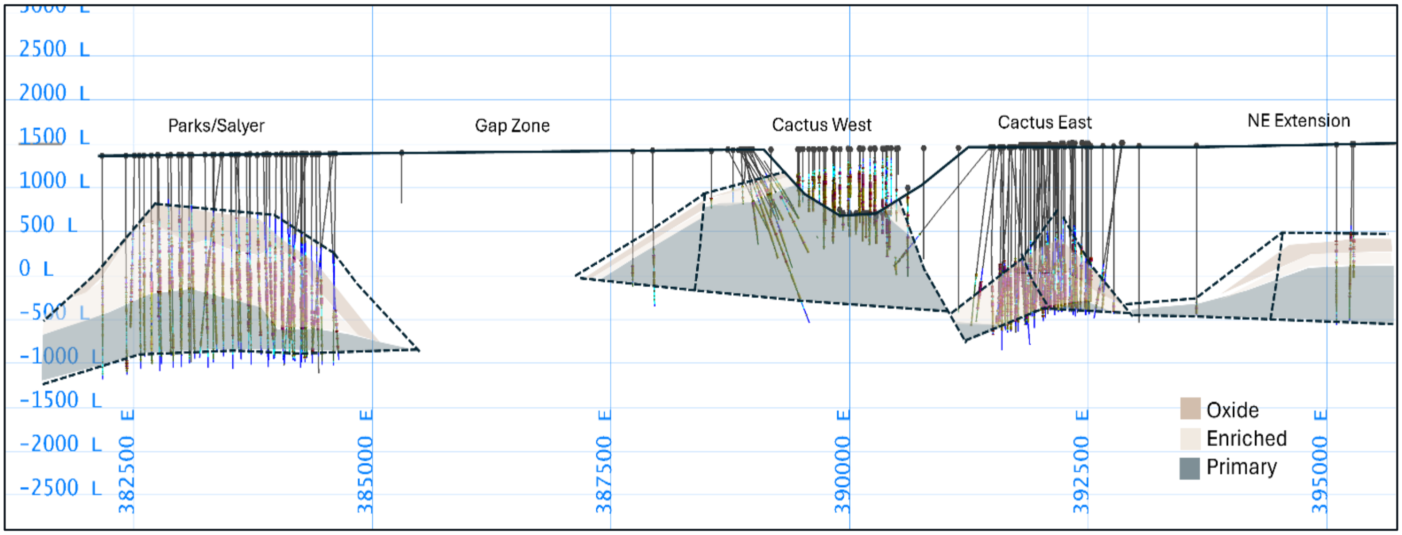
**Figure 14-4: Box Plots of the Main Logged Lithologies Hosting Mineralization**




| | [AND] | [DB] | [GR] | [MP] | [QMP] |
|---|---|---|---|---|---|
| Number of samples | 593 | 208 | 13,452 | 6,685 | 84 |
| Max | 4.807 | 3.004 | 13.100 | 10.000 | 1.323 |
| 97.5% quantile | 1.838 | 2.065 | 2.369 | 2.004 | 1.187 |
| Upper quartile | 0.363 | 0.623 | 0.410 | 0.441 | 0.205 |
| Median | 0.086 | 0.459 | 0.153 | 0.228 | 0.115 |
| Lower quartile | 0.015 | 0.250 | 0.041 | 0.091 | 0.071 |
| 2.5% quantile | 0.003 | 0.020 | 0.003 | 0.012 | 0.040 |
| Min | 0.003 | 0.005 | 0.003 | 0.003 | 0.010 |
| Mean | 0.303 | 0.539 | 0.389 | 0.383 | 0.205 |

Source: ASCU, 2022.

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




Source: ASCU, 2022.

### 14.1.2.3 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 East, Cactus West and Parks/Salyer exhibit typical porphyry copper mineral zonation due to the leaching of copper in sulphides at shallow depths with redeposition below the water table to enriched chalcocite and/or covellite copper sulphides. 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. 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.

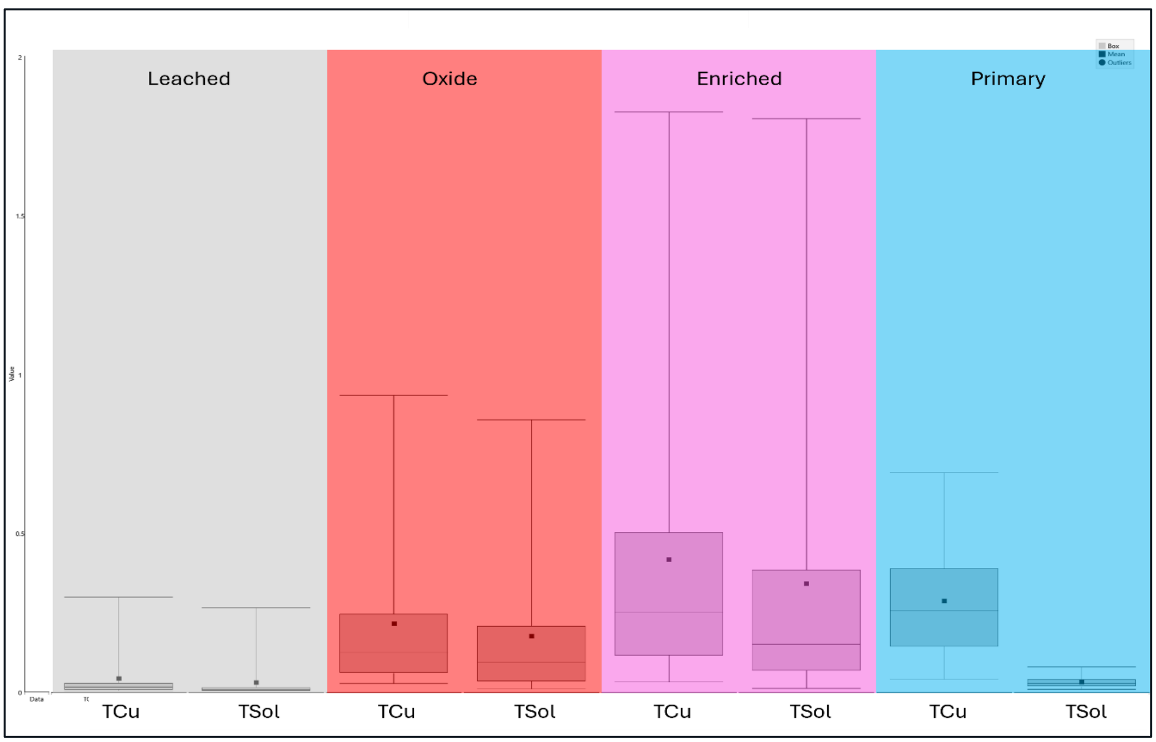
Table 14-3 indicates the main copper mineral domains modelled and their relationship to mineralization.
Figure 14-6 displays box plots for the three copper mineral zones highlighting the different CuT distributions between the zones, the limited transitional material evidenced by high solubilities in 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 14-3: 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 conventional heap leach mineralization. |
| Supergene Enriched | Mineralized with secondary chalcocite and covellite. Represents potential conventional heap leach or mill flotation mineralization or possible leaching with Nuton® technologies. |
| Primary (hypogene) | Mineralized with primary chalcopyrite and pyrite. Represents potential mill flotation mineralization and possible leaching with Nuton® technologies. |

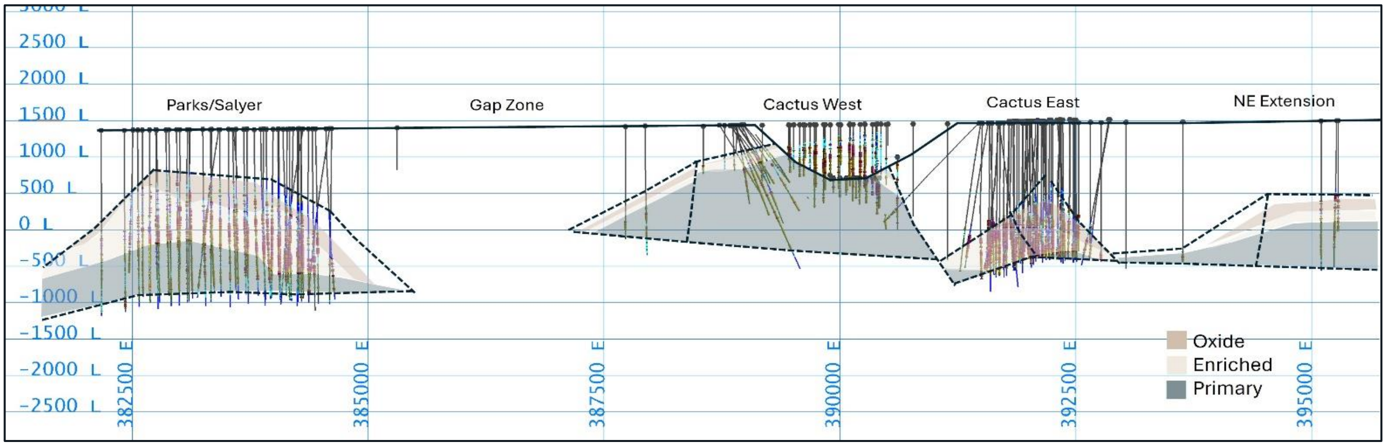
**Figure 14-6: Box Plots of Copper Grades in Mineralized Zones**




Source: ASCU, 2022.

Figure14-6 displays box plots for CuT and the total soluble copper assay components 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. Figure 14-7 shows NE oriented cross-section facing NW.

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

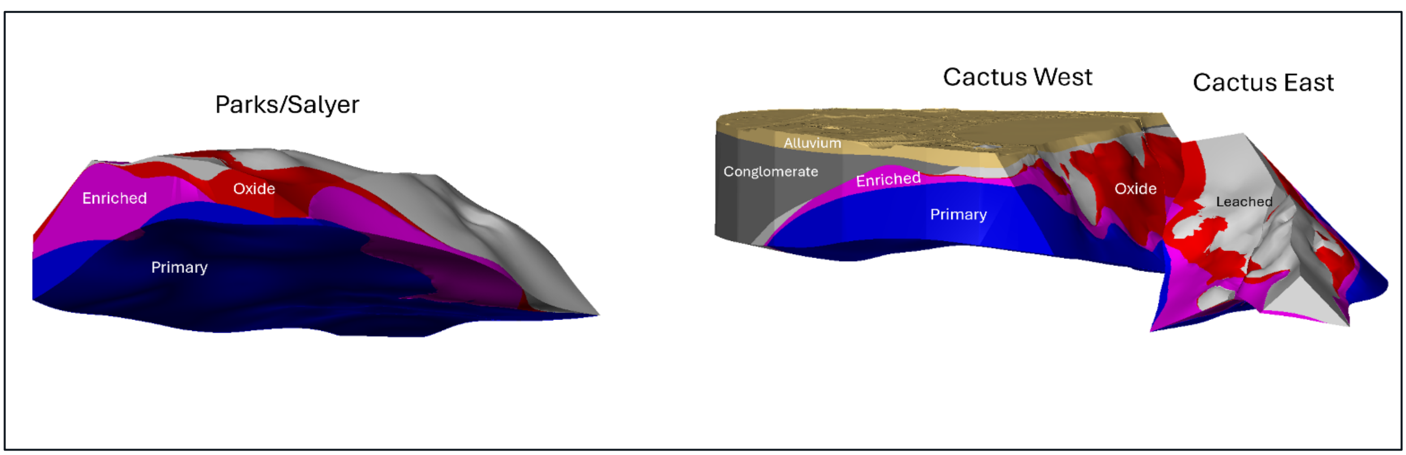



Source: ASCU, 2022.

### 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**




Source: ASCU, 2022.

### 14.1.4 Specific Gravity

As of May 2023, historical drill hole logs for the Cactus Project drilling contained extensive record of specific gravity measurements (3,347 readings). Measurements were undertaken using the wet/dry weight methodology. Values were recorded in metric g/cm$^3$ in the historic logs. To support imperial units and reporting of short tons, the original readings were converted to ft$^3$/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 mineralogizes. Due to this, the larger differences in specific gravity were deemed a result of the level of weathering of the rock or level of brecciation between deposits.

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 domains. Due to the mineralization being disseminated, sulphide content is not highly correlated to specific gravity. Table 14-4 displays the specific gravity values assigned for each domain.

**Table 14-4: Specific Gravity Values Applied per Lithological Domain**

<table>
<tr><th>Area</th><th>Rock type</th><th>Minzone</th><th>Density (st/ft$^3$)</th></tr>
<tr><td rowspan="2"></td><td colspan="2">Alluvium</td><td>0.0468</td></tr>
<tr><td colspan="2">Conglomerate</td><td>0.0780</td></tr>
<tr><td rowspan="9">CW</td><td rowspan="4">Granite or Monzonite porphyry</td><td>Leached</td><td>0.0800</td></tr>
<tr><td>Oxide</td><td>0.0800</td></tr>
<tr><td>Enriched</td><td>0.0810</td></tr>
<tr><td>Primary</td><td>0.0800</td></tr>
<tr><td colspan="2">Andesite Porphyry</td><td>0.0810</td></tr>
<tr><td rowspan="4">Diabase or Dacite</td><td>Leached</td><td>0.0790</td></tr>
<tr><td>Oxide</td><td>0.0790</td></tr>
<tr><td>Enriched</td><td>0.0800</td></tr>
<tr><td>Primary</td><td>0.0810</td></tr>
<tr><td rowspan="8">CE</td><td rowspan="4">Granite or Monzonite porphyry</td><td>Leached</td><td>0.0770</td></tr>
<tr><td>Oxide</td><td>0.7900</td></tr>
<tr><td>Enriched</td><td>0.0800</td></tr>
<tr><td>Primary</td><td>0.0790</td></tr>
<tr><td rowspan="4">Diabase or Dacite</td><td>Leached</td><td>0.0770</td></tr>
<tr><td>Oxide</td><td>0.0770</td></tr>
<tr><td>Enriched</td><td>0.0790</td></tr>
<tr><td>Primary</td><td>0.0790</td></tr>
<tr><td>PS</td><td>Granite or Monzonite porphyry</td><td>Leached</td><td>0.0750</td></tr>
</table>

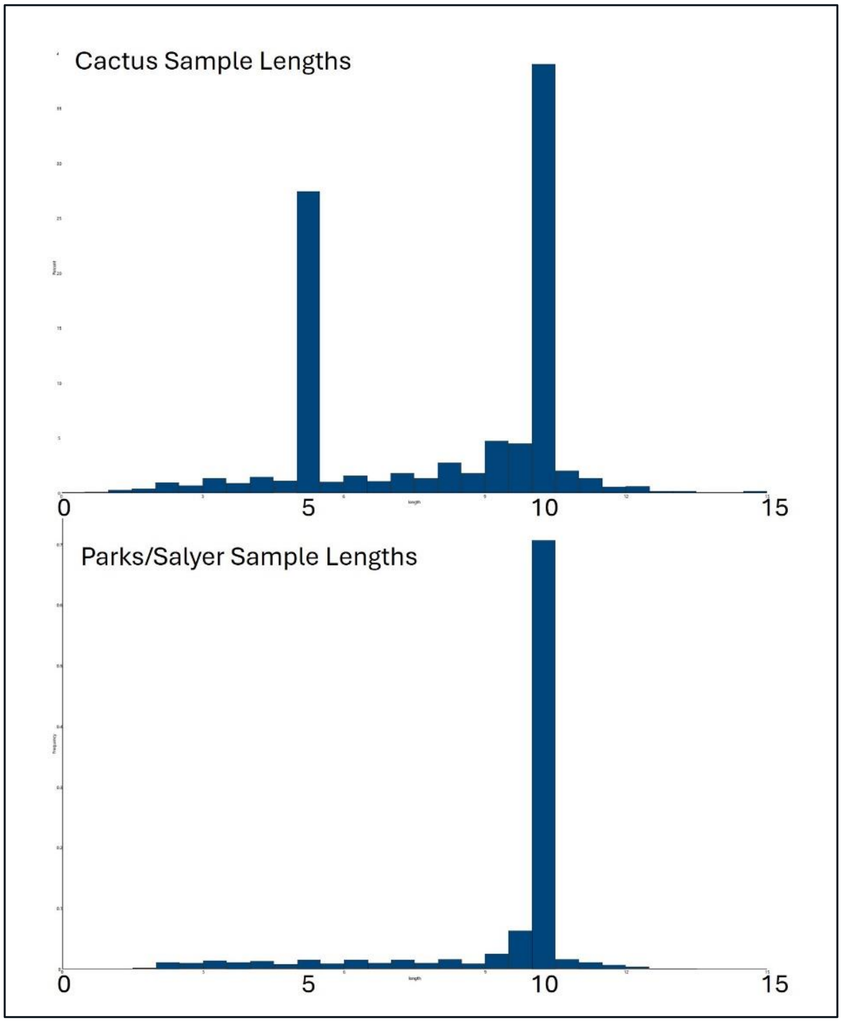
| Area | Rock type | Minzone | Density ($st/ft^3$) |
|---|---|---|---|
| | | Oxide | 0.0720 |
| | | Enriched | 0.0800 |
| | | Primary | 0.0750 |
| | Diabase or Dacite | | 0.0710 |

Note: Standard deviation (std.dev.)

### 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 Project resource drill hole database. At Cactus East and West 95% of sample lengths are 10ft or less in length with 39% sampled at 10ft and 27.5% sampled as 5ft. Most of the 5ft drilling samples were attained from the RC drilling program. At Parks/Salyer9 6% of sample lengths are 10ft or less in length with 70% sampled at 10ft.

**Figure 14-9: Histogram of Drill Hole Sample Lengths**



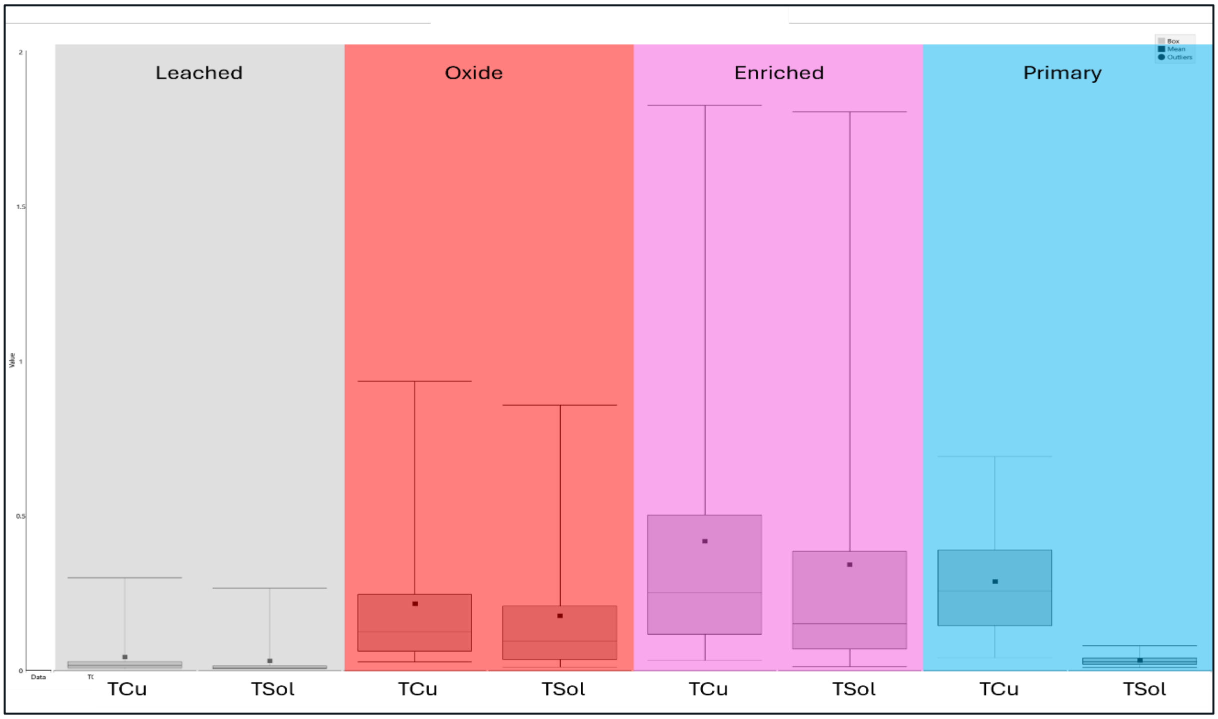

Source: ASCU, 2022.

### 14.1.6 Exploratory Data Analysis

#### 14.1.6.1 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 14-5 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**




Source: ASCU, 2022.

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

| Variable Name | Count | Mean | Std Dev. | Variance | CV | Max | Upper quartile | Median | Lower quartile | Min |
|---|---|---|---|---|---|---|---|---|---|---|
| TCU_FIN_PCT [CW_LEA] <Positive> | 1466 | 0.045 | 0.130 | 0.017 | 2.899 | 2.969 | 0.031 | 0.018 | 0.010 | 0.005 |
| TSOLCU_FIN_PCT [CW_LEA] <POSITIVE> | 1466 | 0.032 | 0.123 | 0.015 | 3.832 | 2.964 | 0.016 | 0.009 | 0.006 | 0.004 |
| TCU_FIN_PCT [CW_OX] <Positive> | 2493 | 0.216 | 0.328 | 0.108 | 1.516 | 7.378 | 0.247 | 0.126 | 0.062 | 0.015 |
| TSOLCU_FIN_PCT [CW_OX] <POSITIVE> | 2493 | 0.178 | 0.280 | 0.078 | 1.575 | 5.379 | 0.209 | 0.095 | 0.035 | 0.006 |
| TCU_FIN_PCT [CW_ENR] <Positive> | 2786 | 0.418 | 0.537 | 0.288 | 1.285 | 9.740 | 0.502 | 0.252 | 0.117 | 0.010 |
| TSOLCU_FIN_PCT [CW_ENR] <POSITIVE> | 2786 | 0.342 | 0.525 | 0.275 | 1.533 | 8.532 | 0.385 | 0.152 | 0.070 | 0.006 |

| Variable Name | Count | Mean | Std Dev. | Variance | CV | Max | Upper quartile | Median | Lower quartile | Min |
|---|---|---|---|---|---|---|---|---|---|---|
| TCU_FIN_PCT [CW_PRI] <Positive> | 2886 | 0.288 | 0.183 | 0.033 | 0.636 | 1.526 | 0.390 | 0.257 | 0.146 | 0.006 |
| TSOLCU_FIN_PCT [CW_PRI] <POSITIVE> | 1827 | 0.033 | 0.020 | 0.000 | 0.598 | 0.242 | 0.041 | 0.029 | 0.020 | 0.006 |

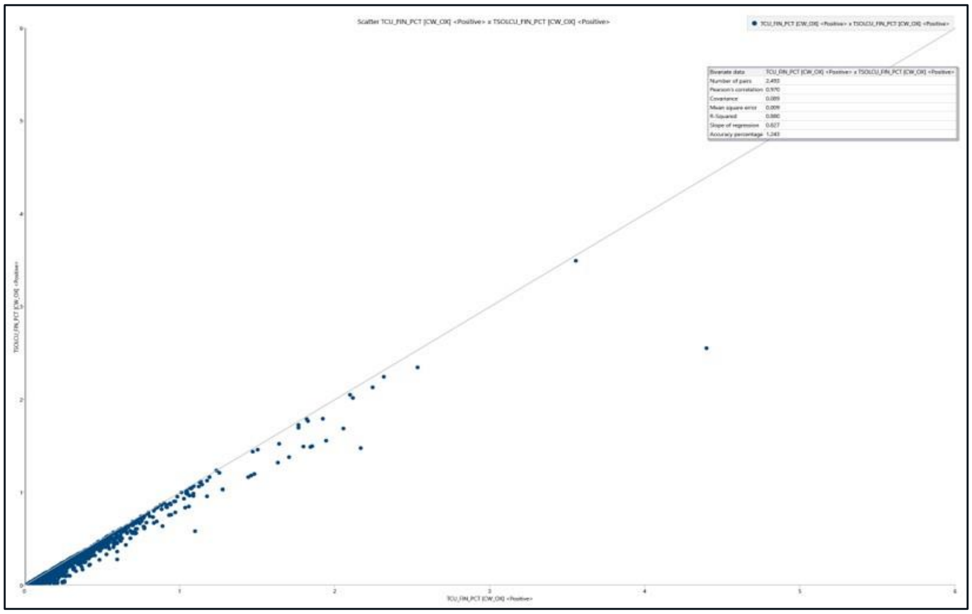
To confirm the relationship between Tsol copper and CuT, scatterplots were plotted for the oxide, enriched, and primary domains (Figure 14-11 through Figure 14-13). For the soluble domains, namely oxide and enriched, the bulk of the Tsol copper is expected to plot towards the 45° line, indicating a 1:1 relationship. Samples plotting well away from this line would indicate significant mixing of populations and the potential for significant transitional zones within the mineralization.

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.

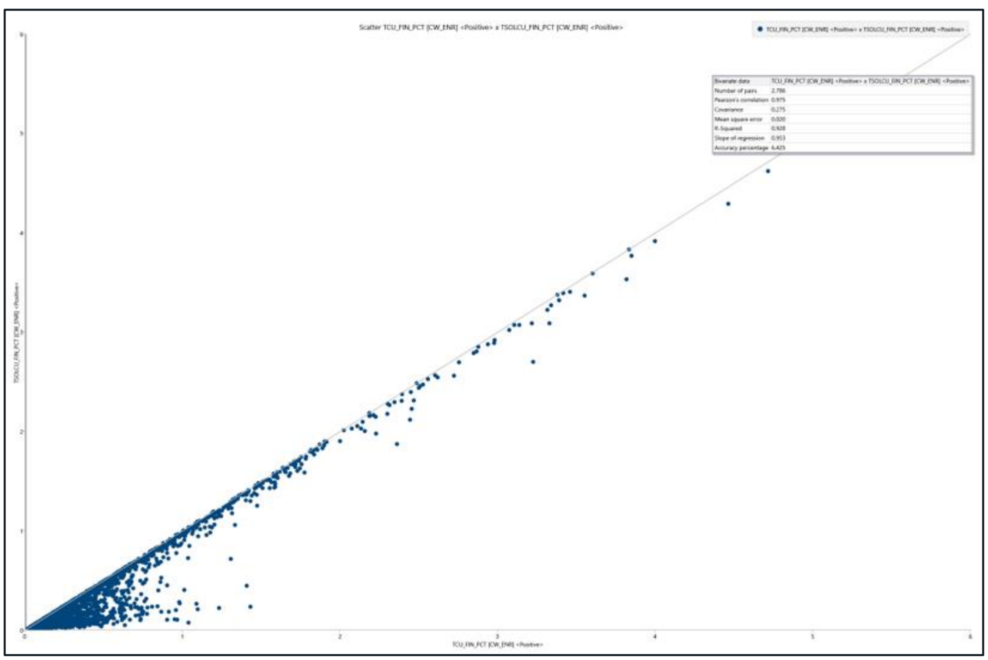
The defined estimation domains show a high degree of control over the copper distributions seen within the Cactus West deposit and are appropriate for use in grade estimation to produce robust estimates of copper grades.

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



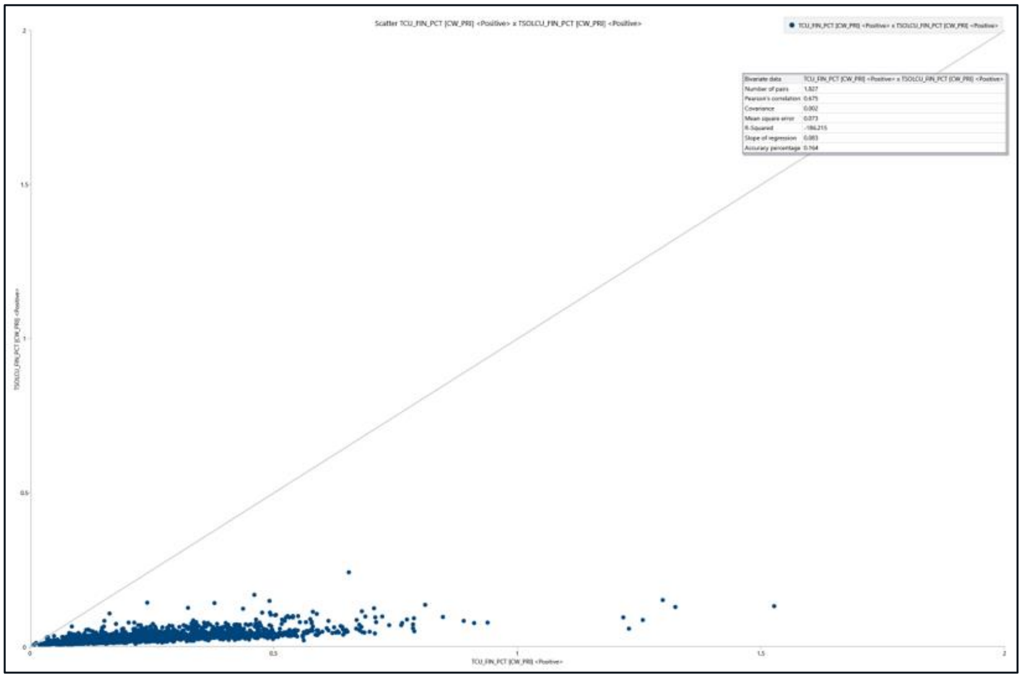

Source: ASCU, 2022.

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




Source: ASCU, 2022.

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



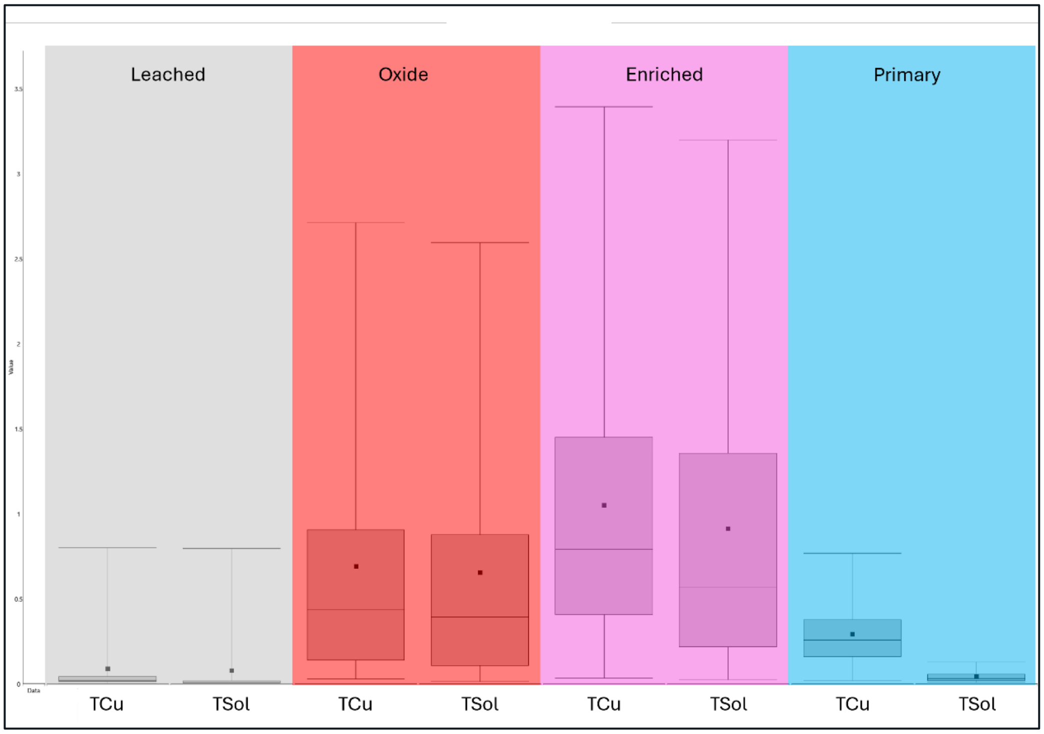
Source: ASCU, 2022.

#### 14.1.6.2 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 14-6 reports the statistics for the main domains in support of the box plot distributions in Figure 14-14.

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.

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

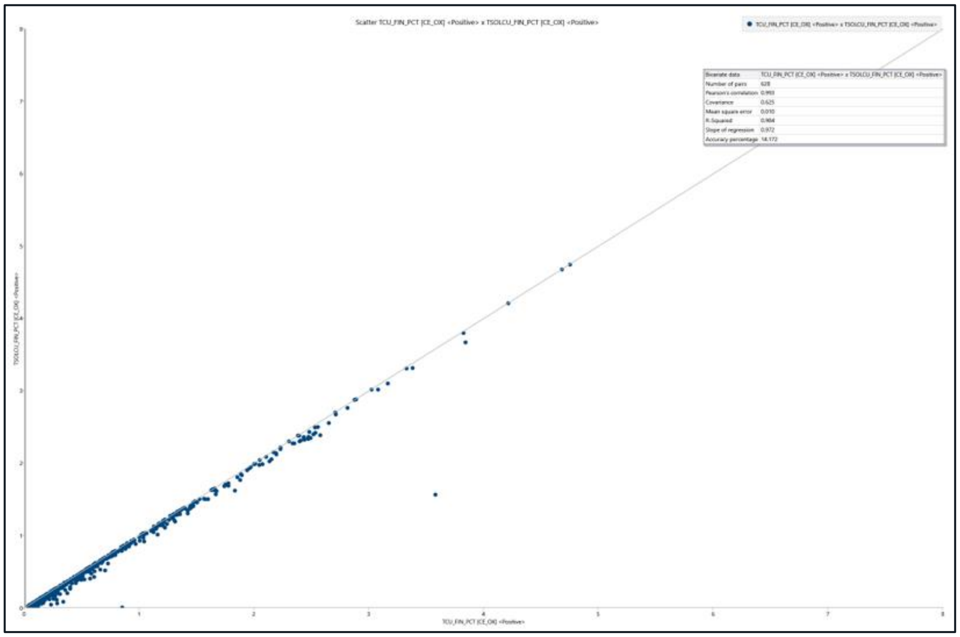



Source: ASCU, 2022.

**Table 14-6: Cactus East Descriptive Statistics of Total Copper and Total Soluble Copper Grades**

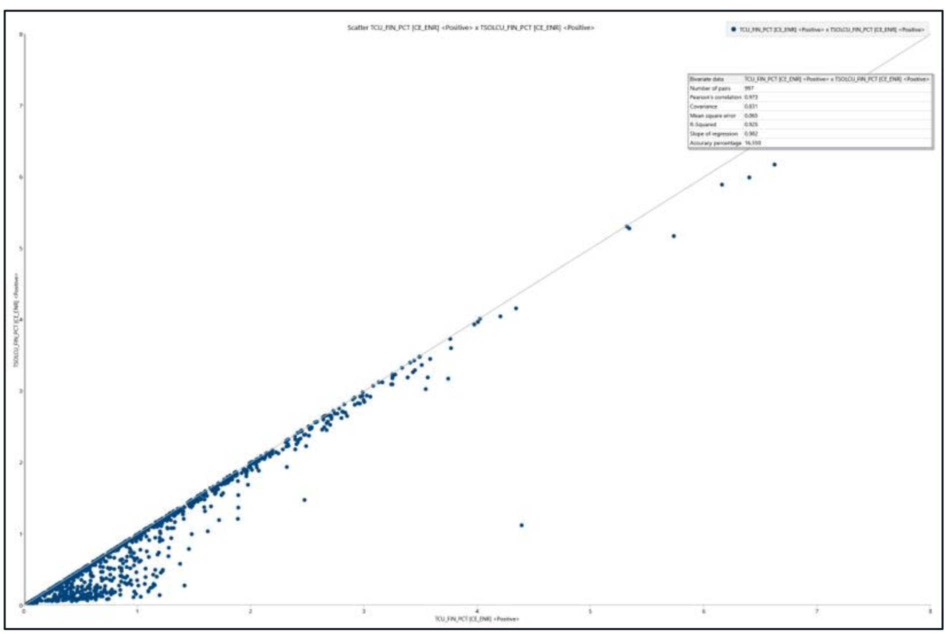
| Variable Name | Count | Mean | Std Dev. | Variance | CV | Max | Upper quartile | Median | Lower quartile | Min |
|---|---|---|---|---|---|---|---|---|---|---|
| TCU_FIN_PCT [CE_LEA] <Positive> | 312 | 0.091 | 0.267 | 0.071 | 2.922 | 2.495 | 0.045 | 0.022 | 0.015 | 0.006 |
| TSOLCU_FIN_PCT [CE_LEA] <POSITIVE> | 312 | 0.078 | 0.266 | 0.071 | 3.405 | 2.491 | 0.020 | 0.010 | 0.006 | 0.006 |
| TCU_FIN_PCT [CE_OX] <Positive> | 628 | 0.691 | 0.802 | 0.644 | 1.161 | 7.340 | 0.906 | 0.436 | 0.139 | 0.026 |
| TSOLCU_FIN_PCT [CE_OX] <POSITIVE> | 628 | 0.653 | 0.785 | 0.616 | 1.201 | 7.016 | 0.879 | 0.393 | 0.108 | 0.006 |
| TCU_FIN_PCT [CE_ENR] <Positive> | 997 | 1.048 | 0.920 | 0.846 | 0.877 | 6.619 | 1.449 | 0.792 | 0.407 | 0.025 |
| TSOLCU_FIN_PCT [CE_ENR] <POSITIVE> | 997 | 0.913 | 0.929 | 0.862 | 1.017 | 6.170 | 1.354 | 0.567 | 0.216 | 0.009 |
| TCU_FIN_PCT [CE_PRI] <Positive> | 705 | 0.292 | 0.189 | 0.036 | 0.646 | 1.582 | 0.378 | 0.258 | 0.159 | 0.003 |
| TSOLCU_FIN_PCT [CE_PRI] <POSITIVE> | 523 | 0.44 | 0.035 | 0.001 | 0.797 | 0.280 | 0.058 | 0.034 | 0.019 | 0.006 |

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



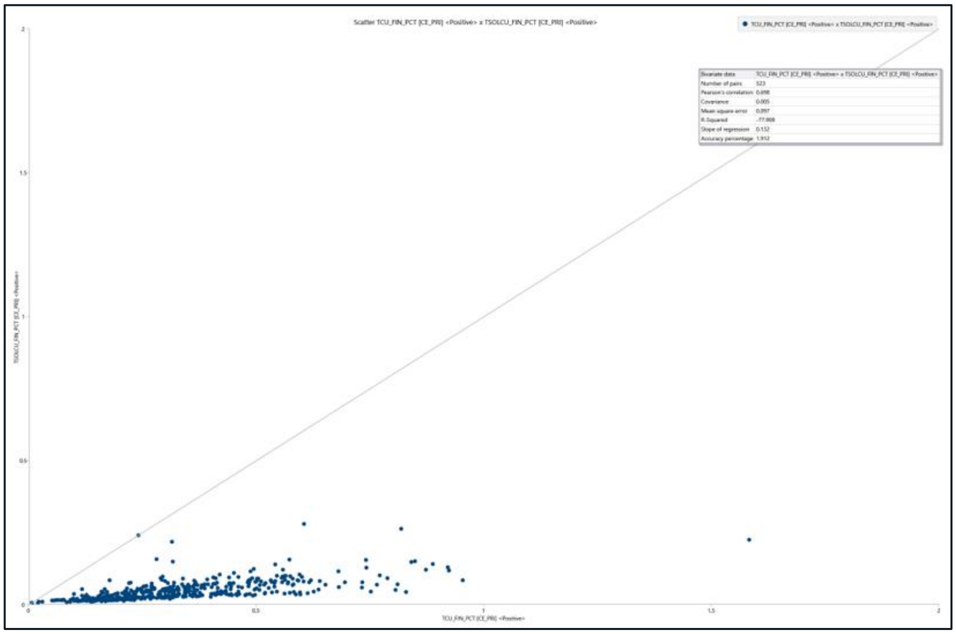

Source: ASCU, 2022.

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




Source: ASCU, 2022.

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




Source: ASCU, 2022.

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.

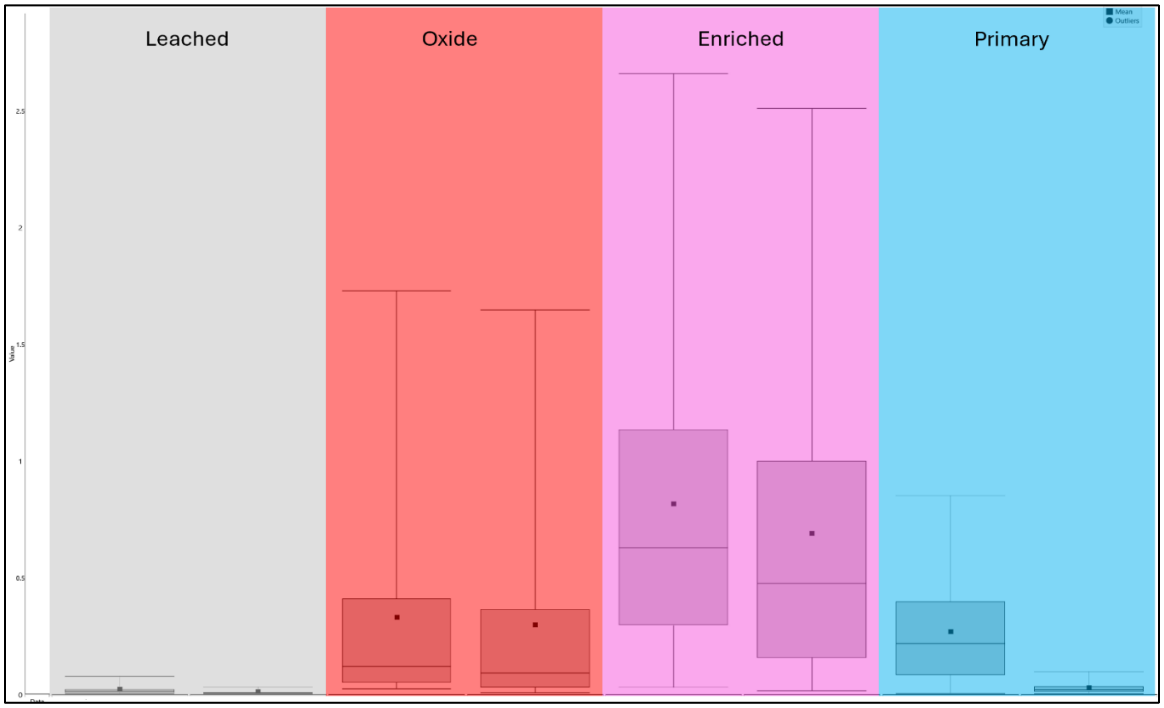
#### 14.1.6.3 Parks/Salyer

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 14-7 reports the statistics for the main domains in support of the box plot distributions in Figure 14-18.

**Table 14-7: Parks/Salyer Descriptive Statistics of Total Copper and Total Soluble Copper Grades**

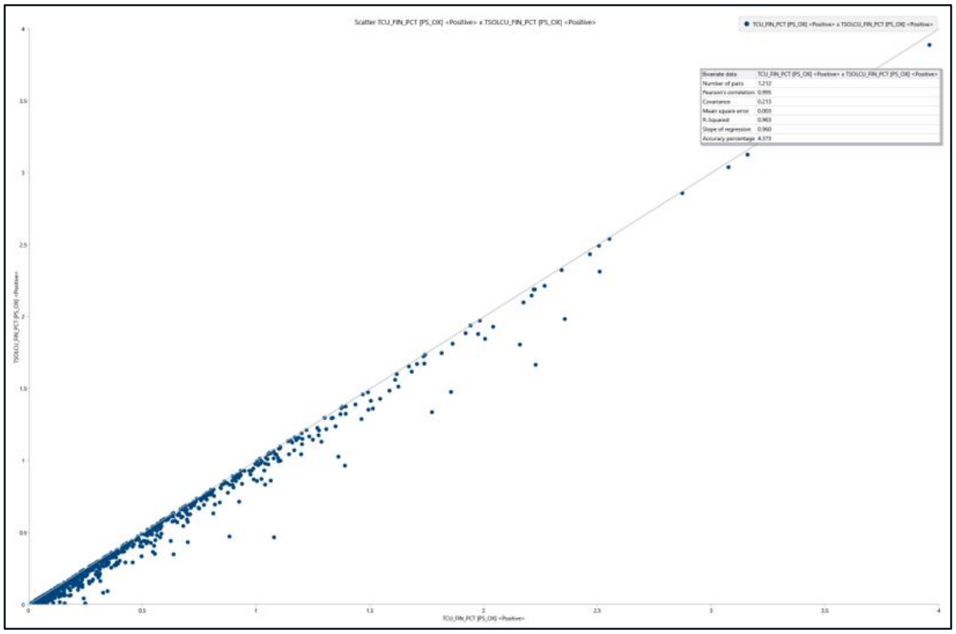
| Variable Name | Count | Mean | Std. Dev. | Variance | CV | Max | Upper Quartile | Med. | Lowe Qurtile | Min |
|---|---|---|---|---|---|---|---|---|---|---|
| TCU_FIN_PCT [PS_LEA] <Positive> | 1962 | 0.023 | 0.061 | 0.004 | 2.599 | 1.442 | 0.023 | 0.015 | 0.010 | 0.006 |
| TSOLCU_FIN_PCT [PS_LEA] <POSITIVE> | 1962 | 0.013 | 0.055 | 0.003 | 4.161 | 1.417 | 0.011 | 0.006 | 0.006 | 0.006 |
| TCU_FIN_PCT [PS_OX] <Positive> | 1212 | 0.330 | 0.471 | 0.222 | 1.427 | 3.956 | 0.411 | 0.120 | 0.051 | 0.009 |
| TSOLCU_FIN_PCT [PS_OX] <POSITIVE> | 1212 | 0.298 | 0.455 | 0.207 | 1.528 | 3.887 | 0.366 | 0.093 | 0.032 | 0.006 |
| TCU_FIN_PCT [PS_ENR] <Positive> | 3823 | 0.817 | 0.747 | 0.558 | 0.915 | 9.396 | 1.133 | 0.628 | 0.298 | 0.008 |
| TSOLCU_FIN_PCT [PS_ENR] <POSITIVE> | 3823 | 0.690 | 0.731 | 0.534 | 1.059 | 9.152 | 1.000 | 0.476 | 0.160 | 0.006 |
| TCU_FIN_PCT [PS_PRI] <Positive> | 3402 | 0.271 | 0.231 | 0.053 | 0.853 | 1.680 | 0.398 | 0.219 | 0.085 | 0.006 |
| TSOLCU_FIN_PCT [PS_PRI] <POSITIVE> | 3338 | 0.029 | 0.030 | 0.001 | 1.056 | 0.642 | 0.035 | 0.022 | 0.013 | 0.006 |

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



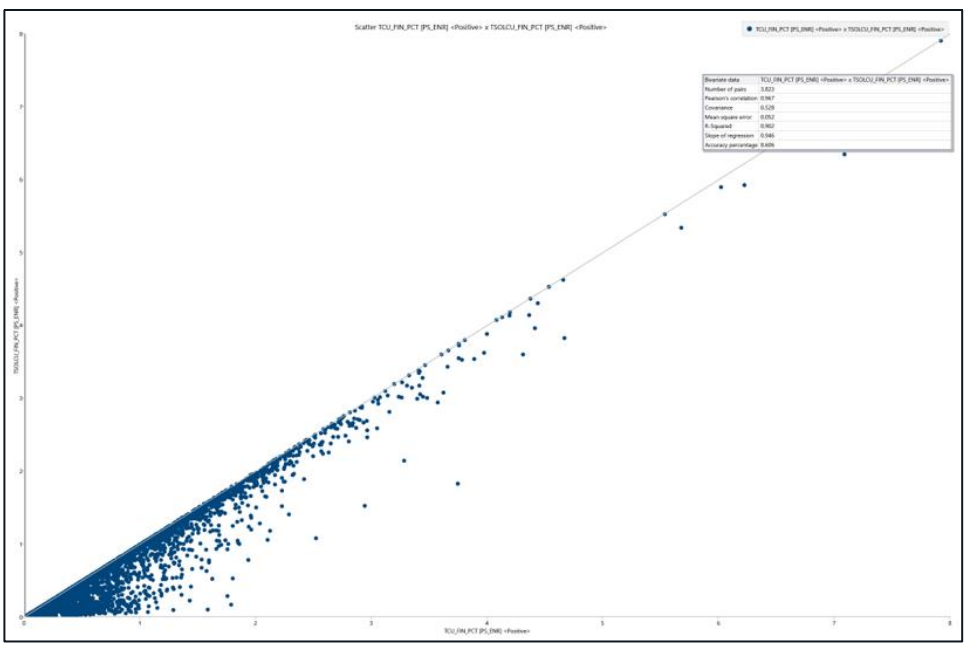

Source: ASCU, 2022.

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



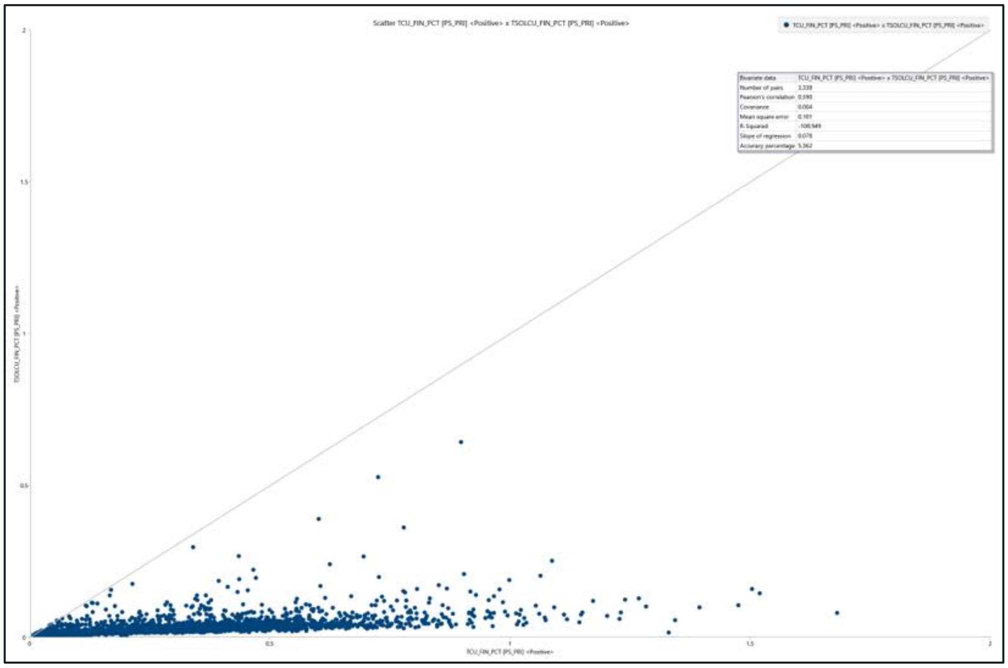
Source: ASCU, 2022.

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



Source: ASCU, 2022.

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



Source: ASCU, 2022.

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.

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 and 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.

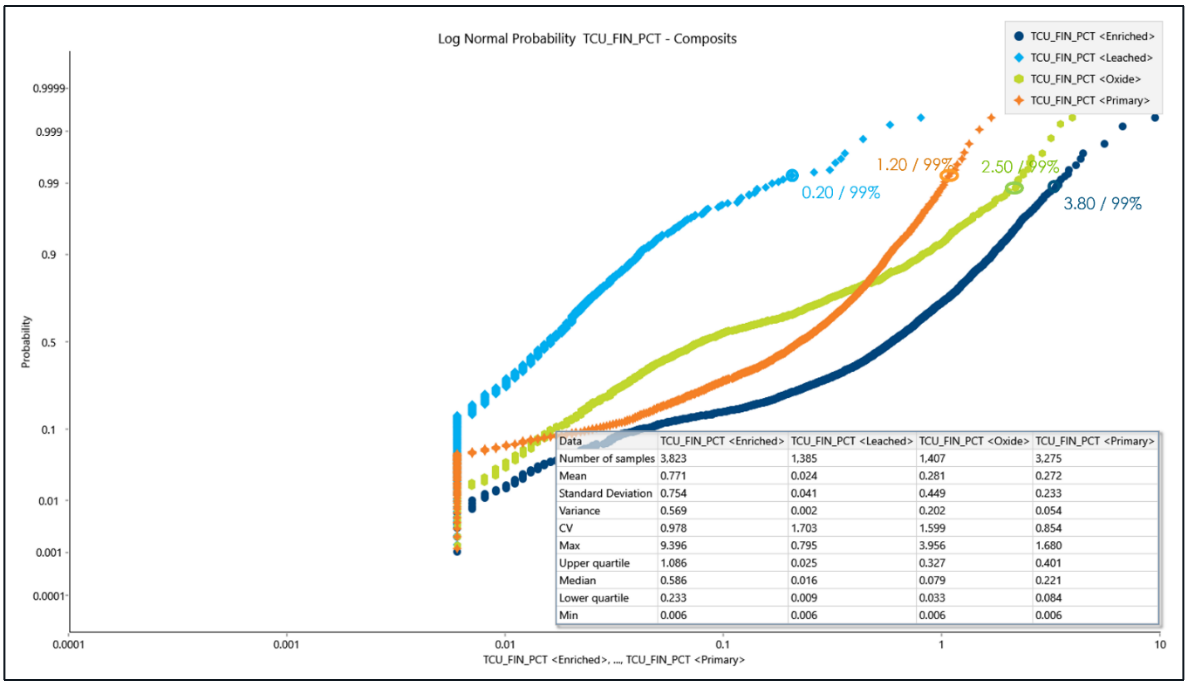
The defined estimation domains show a high degree of control over the copper distributions seen within the Parks/Salyer deposit and are appropriate for use in grade estimation to produce robust estimates of copper grades.

### 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, ASCu, CuCN, Tsol, and Mo in each of the mineral zones in the Cactus Project resource. Figure 14- is an example 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.

The process was repeated for Tsol, CuAS, and CuCN for each of the.

**Figure 14-22: Example Log Normal Probability Plot of Total Copper Assays for Parks/Salyer**




Source: ASCU, 2022.

Capping values per deposit and copper analysis method are posted in Table 14-8.

**Table 14-8: Capping Levels for Parks/Salyer Estimation Domains**

| Area | Rock type | Minzone | TCu (%) | Tsol (%) | CuAS (%) | CuCN (%) | Mo (%) |
|---|---|---|---|---|---|---|---|
| CW | | leached | 0.16 | 0.12 | 0.10 | 0.04 | N/A |
| | | oxide | 1.30 | 1.20 | 1.05 | 0.50 | |

| Area | Rock type | Minzone | TCu (%) | Tsol (%) | CuAS (%) | CuCN (%) | Mo (%) |
|---|---|---|---|---|---|---|---|
| | granite or monzonite porphyry | enriched | 2.75 | 2.70 | 0.55 | 2.30 | |
| | | primary | 1.00 | 0.12 | 0.06 | 0.085 | |
| | diabase nth | | 0.56 | 0.51 | 0.48 | 0.03 | |
| | diabase sth | | 1.83 | 1.77 | 1.51 | 0.24 | |
| | dacite | | 1.00 | 0.93 | 0.91 | 0.03 | |
| | andesite porphyry | | 0.37 | 0.35 | 0.11 | 0.24 | |
| CE | granite or monzonite porphyry | leached | 0.30 | 0.20 | 0.19 | 0.01 | |
| | | oxide | 2.50 | 2.30 | 2.28 | 0.90 | |
| | | enriched | 3.75 | 3.50 | 0.60 | 3.00 | |
| | | primary | 1.00 | 0.15 | 0.05 | 0.11 | |
| | diabase nth | | 0.67 | 0.55 | 0.09 | 0.47 | |
| PS | granite or monzonite porphyry | leached | 0.20 | 0.09 | 0.04 | 0.05 | 0.08 |
| | | oxide | 2.50 | 2.30 | 1.80 | 0.50 | |
| | | enriched | 3.8 | 3.80 | 0.50 | 3.30 | |
| | | primary | 1.20 | 0.18 | 0.03 | 0.15 | |
| | diabase nth | | 0.42 | 0.10 | 0.03 | 0.07 | 0.01 |
| | diabase sth | | 1.54 | 1.48 | 1.42 | 0.71 | 0.01 |

### 14.1.8 Variography

Variogram analysis was undertaken to generated specific variograms for each mineral zone with each deposit. It was found that primary material had the best continuity and therefore the lowest nugget and longest ranges generally. Well developed enrichment blanket having lesser continuity than primary but better than oxide. Oxide, being the most variable weathering process, has the highest variability. Table 14-9 show the variogram models applied for each mineralized zone within the three deposits.

**Table 14-9: Variogram Results Form Mineralized Zones in Each Deposit**

| Variogram | | | | | | Structure 1 | | | | | Structure 2 | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Deposit | Domain | Nugget | Bearing | Azimuth | Dip | Type | Sill Diff | Max | Semi | Minor | Type | Sill Diff | Max | Semi | Minor |
| CW | Oxide/Leached | 0.18 | 45 | 0 | 0 | Spherical | 0.35 | 75 | 130 | 20 | Spherical | 0.47 | 320 | 500 | 90 |
| | Enriched | 0.10 | 45 | 0 | 0 | Spherical | 0.55 | 200 | 190 | 50 | Spherical | 0.35 | 2,500 | 1,500 | 400 |
| | Primary | 0.05 | 45 | 0 | 0 | Spherical | 0.20 | 275 | 375 | 30 | Spherical | 0.75 | 1,300 | 900 | 700 |
| CE | Oxide/Leached | 0.18 | 130 | -25 | 0 | Spherical | 0.52 | 375 | 200 | 55 | Spherical | 0.30 | 1,100 | 600 | 200 |

| Variogram | | | | | | Structure 1 | | | | | Structure 2 | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Deposit | Domain | Nugget | Bearing | Azimuth | Dip | Type | Sill Diff | Max | Semi | Minor | Type | Sill Diff | Max | Semi | Minor |
| | Enriched | 0.18 | 130 | -25 | 0 | Spherical | 0.40 | 260 | 250 | 55 | Spherical | 0.42 | 1,700 | 500 | 550 |
| | Primary | 0.05 | 90 | 0 | 0 | Spherical | 0.35 | 300 | 200 | 30 | Spherical | 0.60 | 1,100 | 600 | 200 |
| PS | Oxide/Leached | 0.05 | 70 | 0 | 0 | Spherical | 0.55 | 300 | 300 | 25 | Spherical | 0.40 | 1,500 | 650 | 160 |
| | Enriched | 0.10 | 70 | 0 | 0 | Spherical | 0.50 | 300 | 700 | 35 | Spherical | 0.40 | 1,000 | 900 | 350 |
| | Primary | 0.05 | 45 | 0 | 0 | Spherical | 0.18 | 350 | 400 | 30 | Spherical | 0.77 | 1,800 | 1,050 | 900 |

### 14.1.9 Block Model

The block model for Cactus was constructed to encompass the full extents of the Cactus East and West 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 ft (1.5 m) by 5 ft (1.5 m) by 2.5 ft (0.8 m).

Table 14-10 outlines the Cactus block model definition parameters.

Table 14-11 outlines the Parks/Salyer block model definition parameters.

**Table 14-10: Cactus Block Model Definition Parameters**

| Block Model Definition | X | Y | Z |
|---|---|---|---|
| Origin | 385,900 | 60,800 | -1,000 |
| Bearing/Plunge/Dip | 90 | 0 | 0 |
| Offset Minimum | 0 | 0 | 0 |
| Offset Maximum | 9,100 | 8,100 | 3,000 |
| Parent Block Size | 20 | 20 | 20 |
| Sub-Block Size | 5 | 5 | 3 |
| **Total Blocks** | | | **13,258,231** |

**Table 14-11: Parks/Salyer Block Model Definition Parameters**

| Block Model Definition | X | Y | Z |
|---|---|---|---|
| Origin | 379,500 | 55,000 | -1,500 |
| Bearing/Plunge/Dip | 90 | 0 | 0 |
| Offset Minimum | 0 | 0 | 0 |
| Offset Maximum | 8,500 | 8,000 | 3,500 |

| Block Model Definition | X | Y | Z |
|---|---|---|---|
| Parent Block Size | 20 | 20 | 20 |
| Sub-Block Size | 5 | 5 | 3 |
| **Total Blocks** | | | **30,567,771** |

### 14.1.10 Estimation Plan

With the completion of infill drilling of the Cactus and Parks/Salyer deposits to 250 ft (76 m) spacing, ordinary kriging (OK) is now a reasonable option for the estimation of copper and other grades into these models. Smoothing checks in the estimation validation support the use of OK as a reasonable approximation of the expected grade tonnage curve supporting open pit and underground CoGs for this level study. For the oxide and enriched domains, 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 overall leached domain.

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 125 ft (38 m) spacing. This drill spacing was planned to target definition of measured resources. Pass 2 was defined to estimate drilling with an approximately 250 ft (76 m) spacing. This drill spacing was planned to target definition of indicated resources. Pass 3 was defined to estimate drilling with an approximately 500 ft (152 m) spacing. This drill spacing was planned to target definition of inferred resources.

The measured (pass 1) and indicated (pass2) were then outlined and smoothed to alleviate small islands and holes in the shape. Measured resource were only outlined south of the existing pit where shallow close spaced RC drilling met the criteria required.

Minor dacite and diabase dikes within the resource area were assigned an average grade the number of samples in the domain was sufficient for estimation to be applied. Table 14-12 details the grade assignment strategy for each dike lithology within the resource area.

**Table 14-12: Dike Grade Assignments by Lithology**

| Lith | Block | Leached | Oxide | Enriched | Primary |
|---|---|---|---|---|---|
| andp | CW | 0.003 applied | 0.003 applied | estimated (andpe) | mean grade applied |
| dac | CW | 0.003 applied | estimated (daco) | mean grade applied | estimated tcu only (dacp) |
| db_nth | CW | 0.003 applied | estimated (dbno) | mean grade applied | mean grade applied |
| db_sth | CW | Leached and oxide estimate as one (dbso) | | mean grade applied | mean grade applied |
| db_nth | CE | 0.003 applied | mean grade applied | estimated (dbne) | mean grade applied |
| db_sth | CE | 0.003 applied | mean grade applied | 0.003 applied | mean grade applied |
| Dac nth | PS | Estimated as one (D=dacn) | | | |
| Dac sth | PS | Estimated as one (D=dacs) | | | |

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 14-13 and Table 14-14, respectively. Block grade estimates were undertaken on the parent cell size.

**Table 14-13: Key Parameters used in Each Search Pass for Cactus**

| Minzone | Domain | Pass | Number of Samples | | | | | Search Distances | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | Min | Max | Max/Octant | Min Octant | Max/Hole | Major | Semi | Minor |
| Enriched and Oxide | mineralized | 1 | 5 | 12 | 3 | 3 | 3 | 160 | 160 | 75 |
| | | 2 | 5 | 10 | | | 3 | 300 | 300 | 100 |
| | | 3 | 3 | 8 | | | 2 | 500 | 500 | 250 |
| | | 4 | 2 | 7 | | | 3 | 750 | 750 | 300 |
| | Leached/Waste | 1 | 5 | 12 | 3 | 3 | 3 | 160 | 160 | 75 |
| | | 2 | 5 | 10 | | | 3 | 300 | 300 | 100 |
| | | 3 | 3 | 8 | | | 2 | 500 | 500 | 250 |
| | | 4 | 2 | 7 | | | 3 | 750 | 750 | 300 |
| Primary | mineralized | 1 | 5 | 12 | 3 | 3 | 3 | 160 | 160 | 75 |
| | | 2 | 5 | 10 | | | 3 | 300 | 300 | 100 |
| | | 3 | 3 | 8 | | | 2 | 500 | 500 | 250 |
| | | 4 | 2 | 7 | | | 3 | 750 | 750 | 300 |

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

| Minzone | Domain | Pass | Number of Samples | | | | Search Distances | | | | Soft Boundaries | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Min | Max | Max/ Octant | Min Octant | Max/ Hole | Major | Semi | Minor | Domain | Major | Semi | Minor |
| Enriched and Oxide | Hg | 1 | 5 | 10 | 3 | 3 | 3 | 320 | 160 | 100 | Lg | 160 | 160 | 35 |
| | | 2 | 3 | 8 | | | 2 | 600 | 300 | 250 | Lg | 300 | 300 | 50 |
| | | 3 | 2 | 7 | | | 3 | 750 | 500 | 300 | Lg | 300 | 300 | 50 |
| | lg | 1 | 5 | 10 | 3 | 3 | 3 | 320 | 750 | 100 | Hg | 130 | 130 | 35 |
| | | 2 | 3 | 8 | | | 2 | 600 | 160 | 250 | Hg | 130 | 130 | 35 |
| | | 3 | 2 | 7 | | | 3 | 750 | 300 | 300 | Hg | 130 | 130 | 35 |
| | Leached/waste | 1 | 5 | 10 | 3 | 3 | 3 | 320 | 500 | 100 | | | | |

| Minzone | Domain | Pass | Number of Samples | | | | Search Distances | | | | Soft Boundaries | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Min | Max | Max/ Octant | Min Octant | Max/ Hole | Major | Semi | Minor | Domain | Major | Semi | Minor |
| | | 2 | 3 | 8 | | | 2 | 600 | 750 | 250 | | | | |
| | | 3 | 2 | 7 | | | 3 | 750 | 160 | 300 | | | | |
| Primary | mineralized | 1 | 5 | 10 | 3 | 3 | 3 | 320 | 300 | 100 | | | | |
| | | 2 | 3 | 8 | | | 2 | 600 | 500 | 250 | | | | |
| | | 3 | 2 | 7 | | | 3 | 750 | 750 | 300 | | | | |

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 estimates for Cactus and Parks/Salyer were undertaken using three passes.
- Un-estimated blocks were assigned a grade of 0.001% CuT.
- Grades were capped using a top-cut method.
- A nearest neighbour 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:

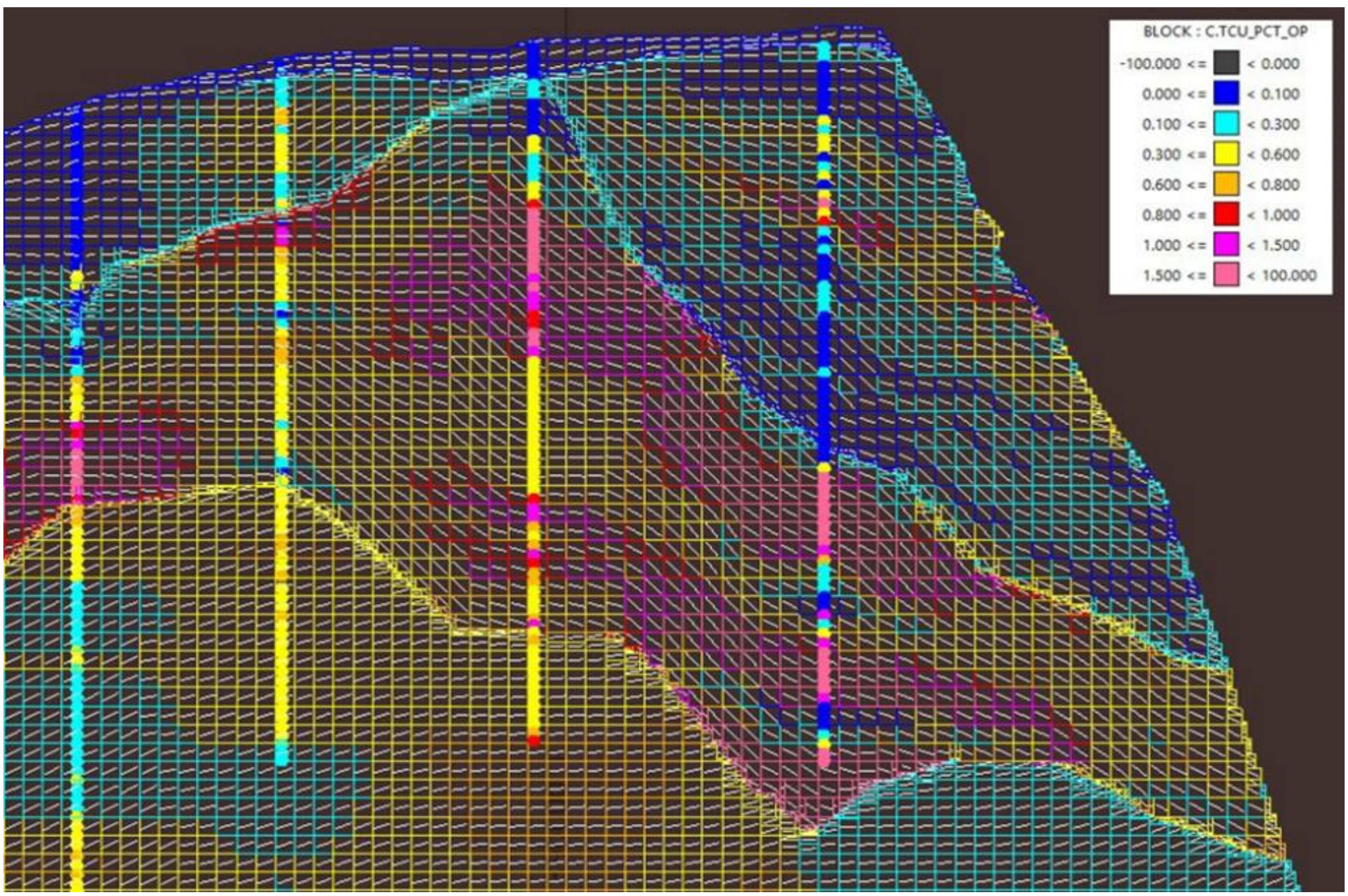
- Structural Controls
- Deeper Leaching Zones
- Historic Water Table Interface

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 14-23) 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 14-15 outlines the surfaces used to define orientation vectors in each copper mineral domain.

Table 14-15: Domain Surfaces

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

Figure 14-23: Representative Cross Section View of the Block Model




Source: ASCU, May 2022.

#### 14.1.11 Mining Depletion

Blocks within the historically mined pit were estimated to aid in validation of the block model estimates and to run a reconciliation of the estimates against reported historical production. 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.

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

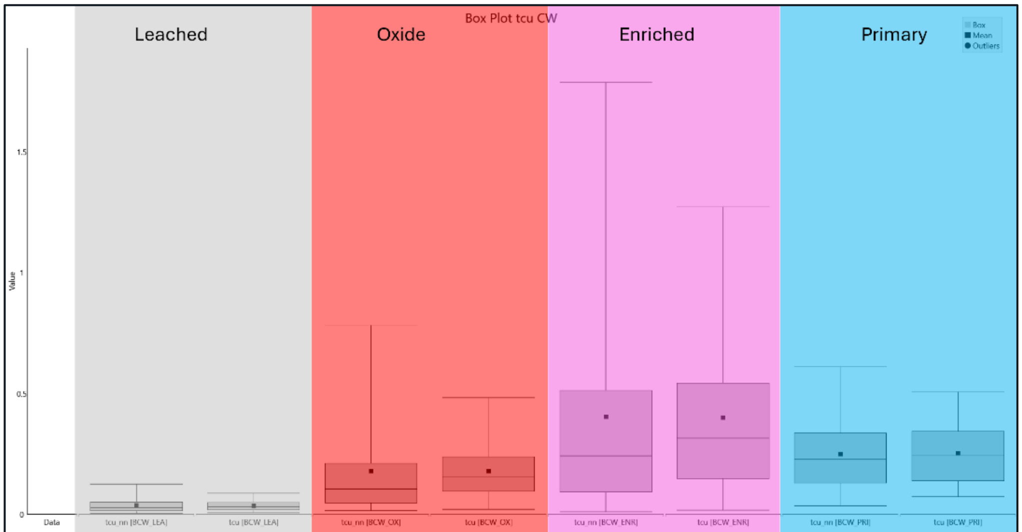
Validations in this Section include the mined material from the historical open pit. Grades reported in this Section for Cactus West include depleted material and therefore reported grades should not be considered as representative of the material that is remaining.

##### 14.1.12.1 Box Plots

Box plots were created for CuT and Tsol copper to compare estimated mean grades and distributions for each domain against the nearest neighbour. Box plots for Cactus West, Cactus East, and Parks/Salyer are presented in Figure 14-24 through Figure 14-29.

Comparisons show similar mean grades between the estimated blocks and the nearest neighbour. The adjustment from a nearest neighbour 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 neighbour plots. No maximum values of the nearest neighbour statistics were reported higher than the planned capping grades, indicating that the top cut was applied to the estimation as planned.

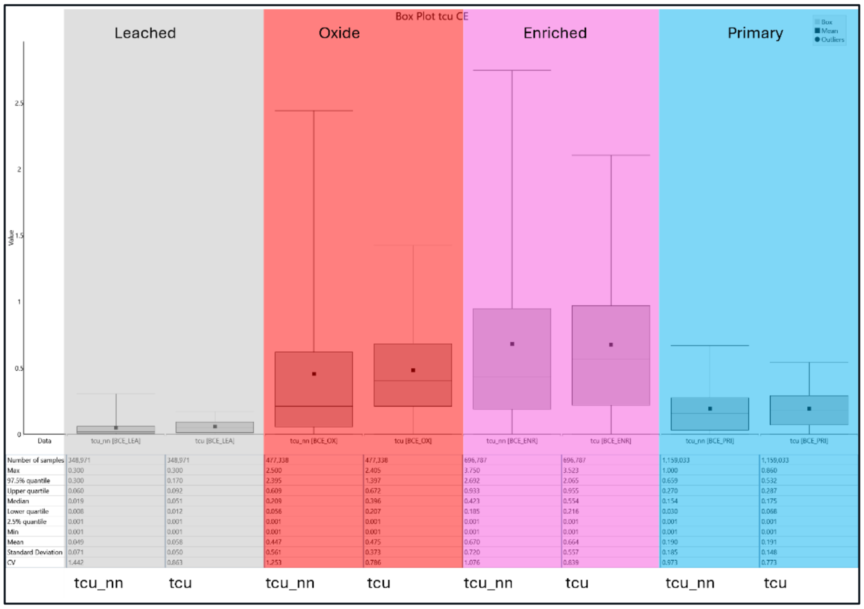
**Figure 14-24: Box Plots Comparing the Total Copper for Cactus West Domains Against the Nearest Neighbour**




| | Leached | | Oxide | | Enriched | | Primary | |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 285,667 | 285,667 | 1,217,338 | 1,217,338 | 1,089,778 | 1,089,778 | 2,022,216 | 2,022,216 |
| Max | 0.160 | 0.143 | 1.300 | 1.250 | 2.750 | 2.608 | 1.000 | 0.935 |
| 97.5% quantile | 0.123 | 0.087 | 0.769 | 0.474 | 1.753 | 1.249 | 0.600 | 0.498 |
| Upper quartile | 0.051 | 0.050 | 0.209 | 0.235 | 0.505 | 0.534 | 0.332 | 0.339 |
| Median | 0.028 | 0.032 | 0.104 | 0.153 | 0.238 | 0.310 | 0.224 | 0.241 |
| Lower quartile | 0.014 | 0.019 | 0.047 | 0.094 | 0.090 | 0.146 | 0.128 | 0.137 |
| 2.5% quantile | 0.006 | 0.007 | 0.017 | 0.021 | 0.011 | 0.017 | 0.035 | 0.072 |
| Min | 0.005 | 0.006 | 0.005 | 0.006 | 0.006 | 0.004 | 0.006 | 0.013 |
| Mean | 0.037 | 0.036 | 0.177 | 0.177 | 0.398 | 0.393 | 0.245 | 0.248 |
| Standard Deviation | 0.030 | 0.021 | 0.213 | 0.120 | 0.478 | 0.329 | 0.152 | 0.122 |
| CV | 0.810 | 0.602 | 1.204 | 0.675 | 1.202 | 0.836 | 0.622 | 0.492 |
| | tcu_nn | tcu | tcu_nn | tcu | tcu_nn | tcu | tcu_nn | tcu |

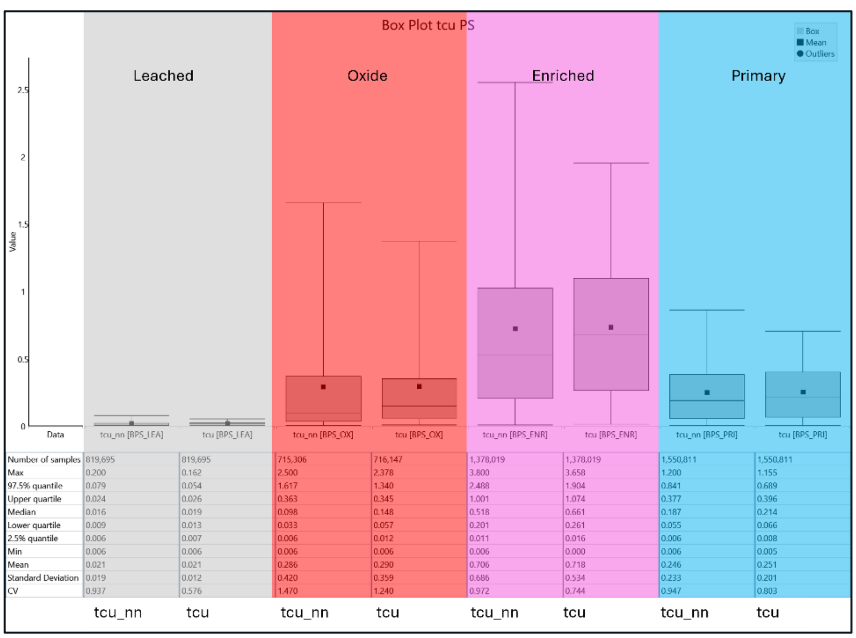
Source: ASCU, 2022.

**Figure 14-25: Box Plots Comparing the CuT for Cactus East Domains Against the Nearest Neighbour**



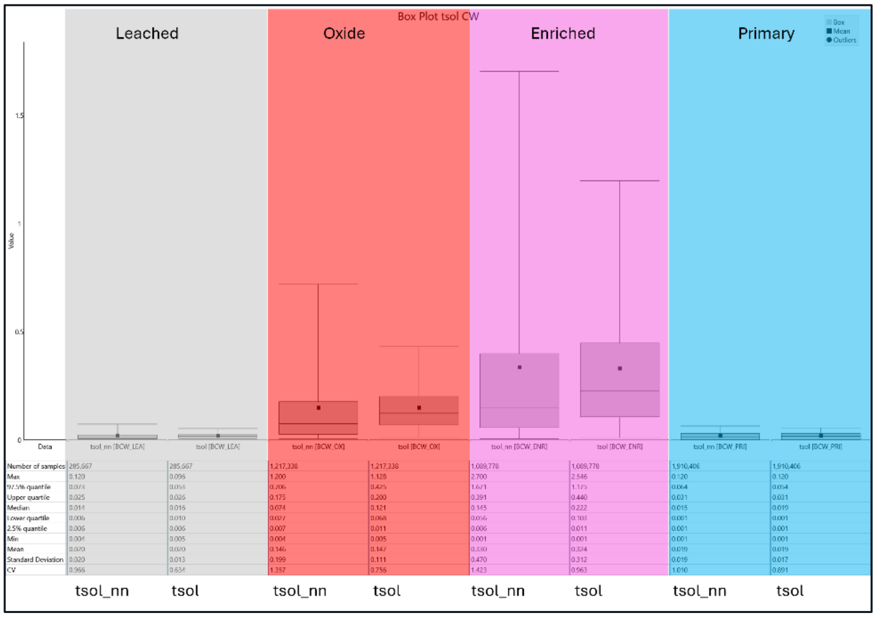

Source: ASCU, 2022.

**Figure 14-26: Box Plots Comparing the CuT for Parks/Salyer Domains Against the Nearest Neighbour**



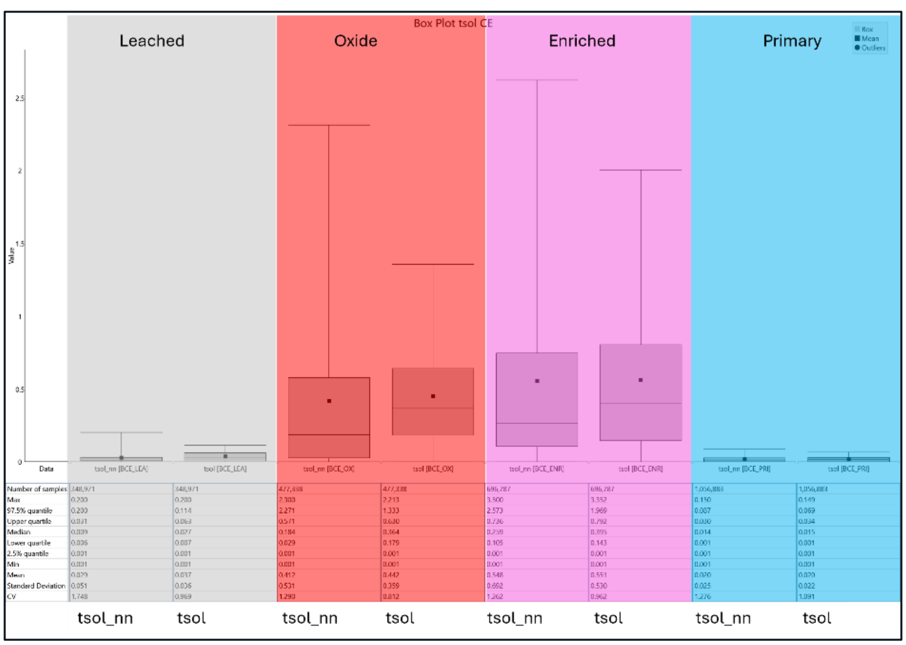

Source: ASCU, 2022.

**Figure 14-27: Box Plots Comparing the Total Soluble Copper for Cactus West Domains Against the Nearest Neighbour**



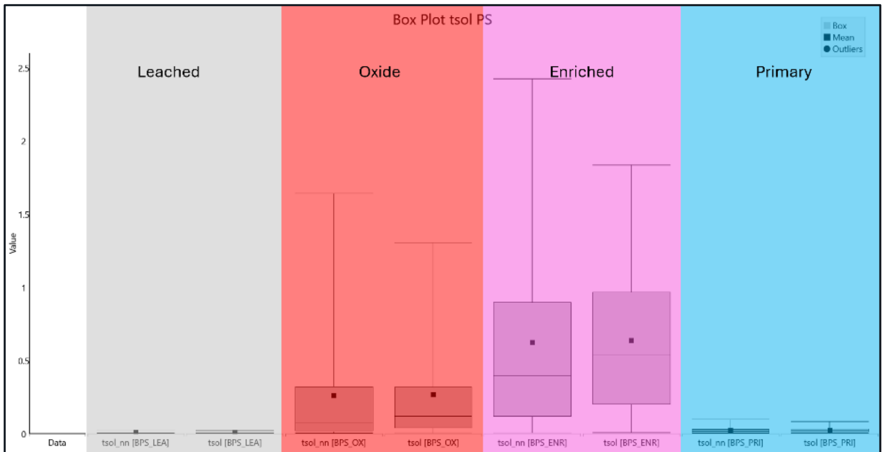

Source: ASCU, 2022.

**Figure 14-28: Box Plots Comparing the Total Soluble Copper for Cactus East Domains Against the Nearest Neighbour**




Source: ASCU, 2022.

**Figure 14-29: Box Plots Comparing the Total Soluble Copper for Parks/Salyer Domains Against the Nearest Neighbour**




| | tsol_nn [BPS_LEA] | tsol [BPS_LEA] | tsol_nn [BPS_OX] | tsol [BPS_OX] | tsol_nn [BPS_ENR] | tsol [BPS_ENR] | tsol_nn [BPS_PRI] | tsol [BPS_PRI] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 819,695 | 819,695 | 715,306 | 716,147 | 1,378,019 | 1,378,019 | 1,511,835 | 1,550,811 |
| Max | 0.090 | 0.074 | 2.300 | 2.235 | 3.800 | 3.550 | 0.180 | 0.161 |
| 97.5% quantile | 0.028 | 0.023 | 1.599 | 1.271 | 2.363 | 1.790 | 0.098 | 0.085 |
| Upper quartile | 0.009 | 0.011 | 0.317 | 0.314 | 0.879 | 0.945 | 0.035 | 0.036 |
| Median | 0.006 | 0.008 | 0.073 | 0.121 | 0.388 | 0.523 | 0.020 | 0.023 |
| Lower quartile | 0.006 | 0.006 | 0.018 | 0.039 | 0.119 | 0.198 | 0.010 | 0.012 |
| 2.5% quantile | 0.006 | 0.006 | 0.006 | 0.008 | 0.008 | 0.012 | 0.006 | 0.006 |
| Min | 0.006 | 0.006 | 0.006 | 0.006 | 0.006 | 0.000 | 0.006 | 0.001 |
| Mean | 0.009 | 0.009 | 0.256 | 0.261 | 0.610 | 0.622 | 0.027 | 0.027 |
| Standard Deviation | 0.007 | 0.005 | 0.403 | 0.341 | 0.664 | 0.504 | 0.025 | 0.020 |
| CV | 0.791 | 0.503 | 1.573 | 1.310 | 1.090 | 0.810 | 0.935 | 0.745 |
| | tsol_nn | tsol | tsol_nn | tsol | tsol_nn | tsol | tsol_nn | tsol |

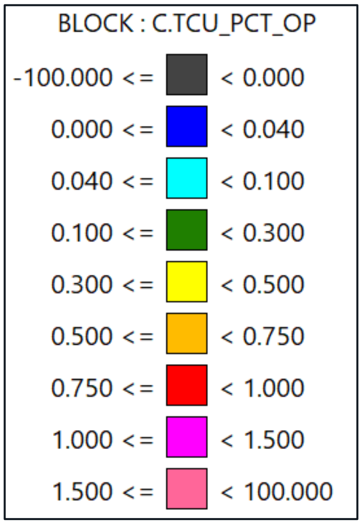
Source: ASCU, 2022.

### 14.1.12.2 Visual Validations

The color legend of Figure 14-30 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.

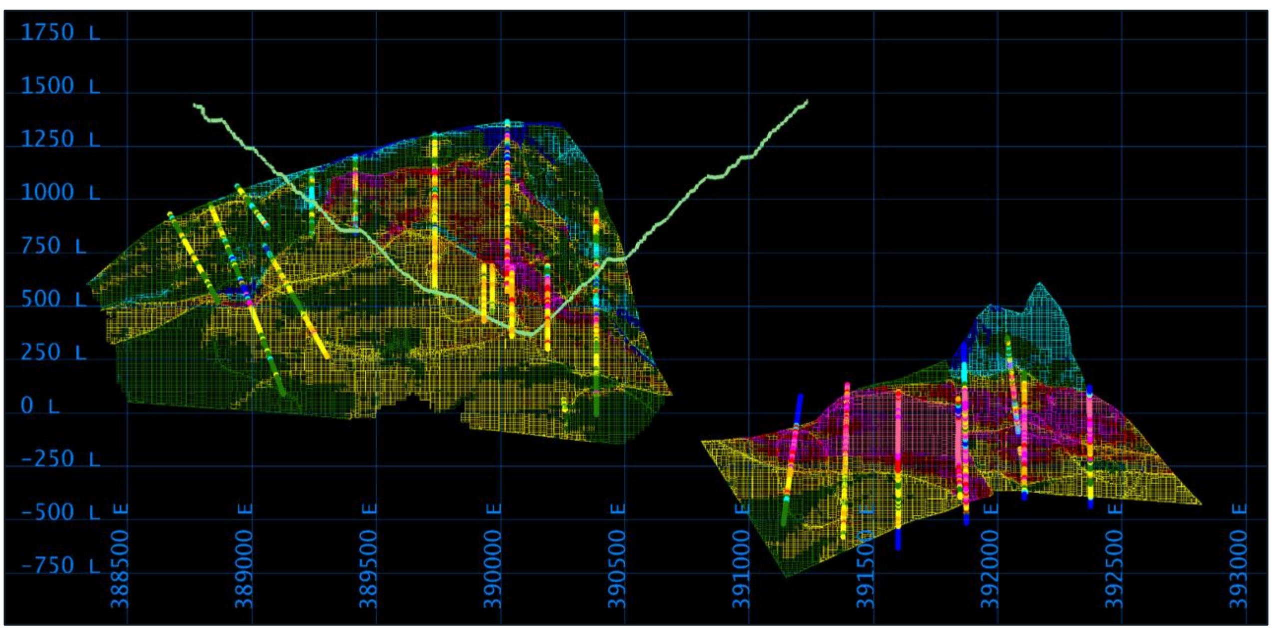
On a local scale, model validation can be confirmed by the visual comparison of block grades to composite grades. A long Section through the Cactus East and Cactus West, plus a cross Section through each of the Cactus East, Cactus West, and Parks/Salyer deposits, show grade trends through the block model. 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 14-31 through Figure 14-38.

**Figure 14-30: Legend for Total Copper and Total Soluble Grades**



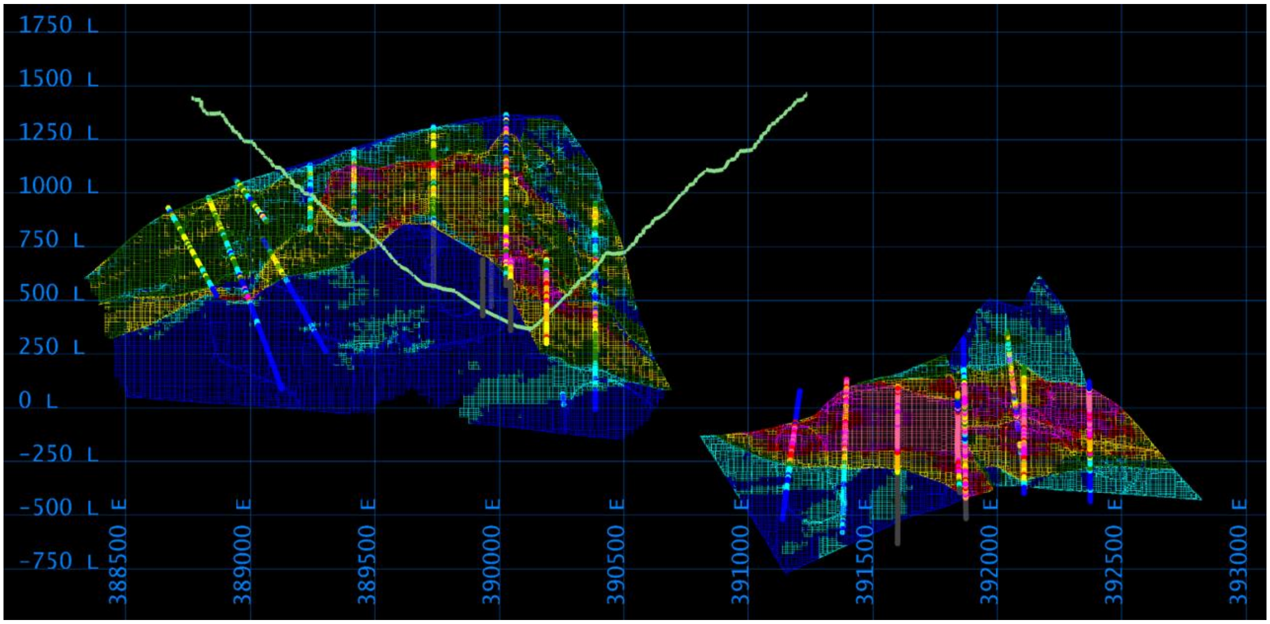

Source: ASCU, 2022

**Figure 14-31: Long Section through Cactus West and Cactus East, Facing Northwest**



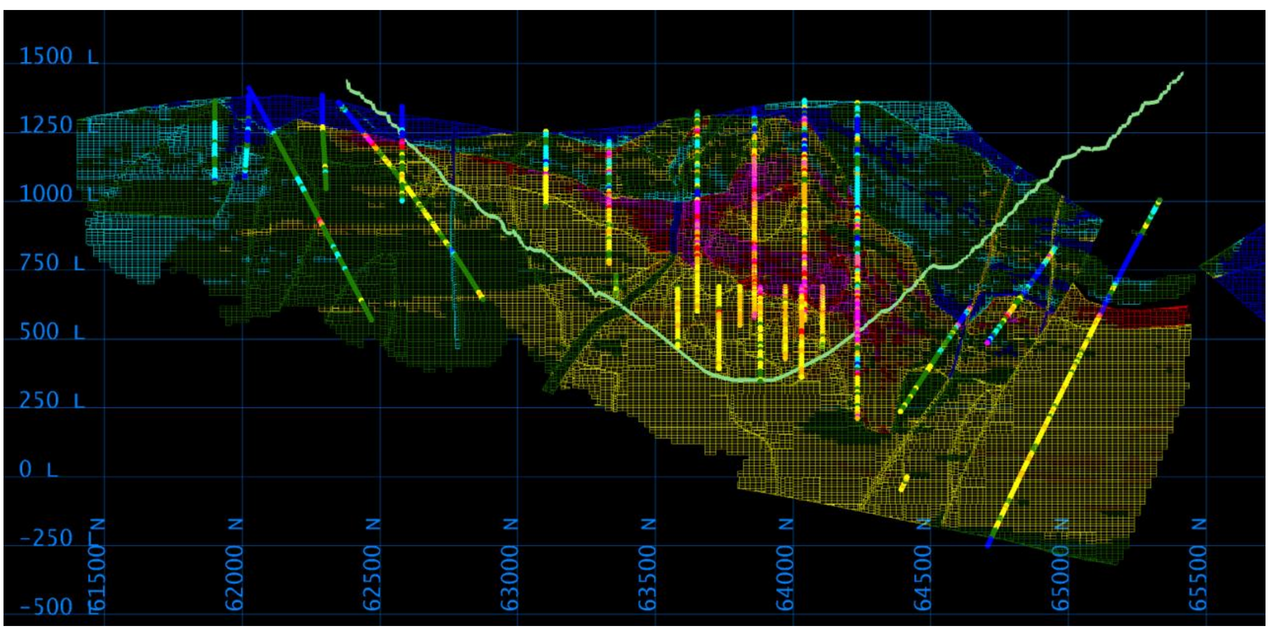

Note: Viewing Total Copper Grades for Both Composites and Block Estimates. Source: ASCU, 2022.

**Figure 14-32: Long Section through Cactus West and Cactus East, Facing Northwest**



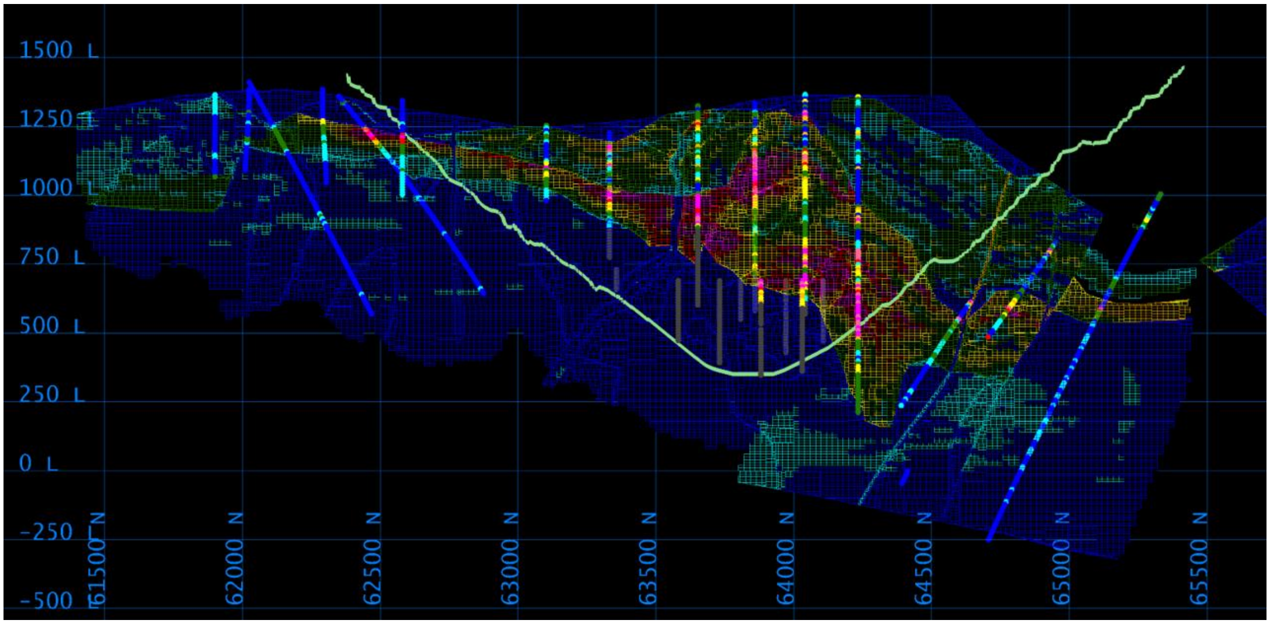

Note: Viewing Tsol grades for both composites and block estimates. Source: ASCU, 2022.

**Figure 14-33: Cross Section (390000E) through Cactus West, Facing West**



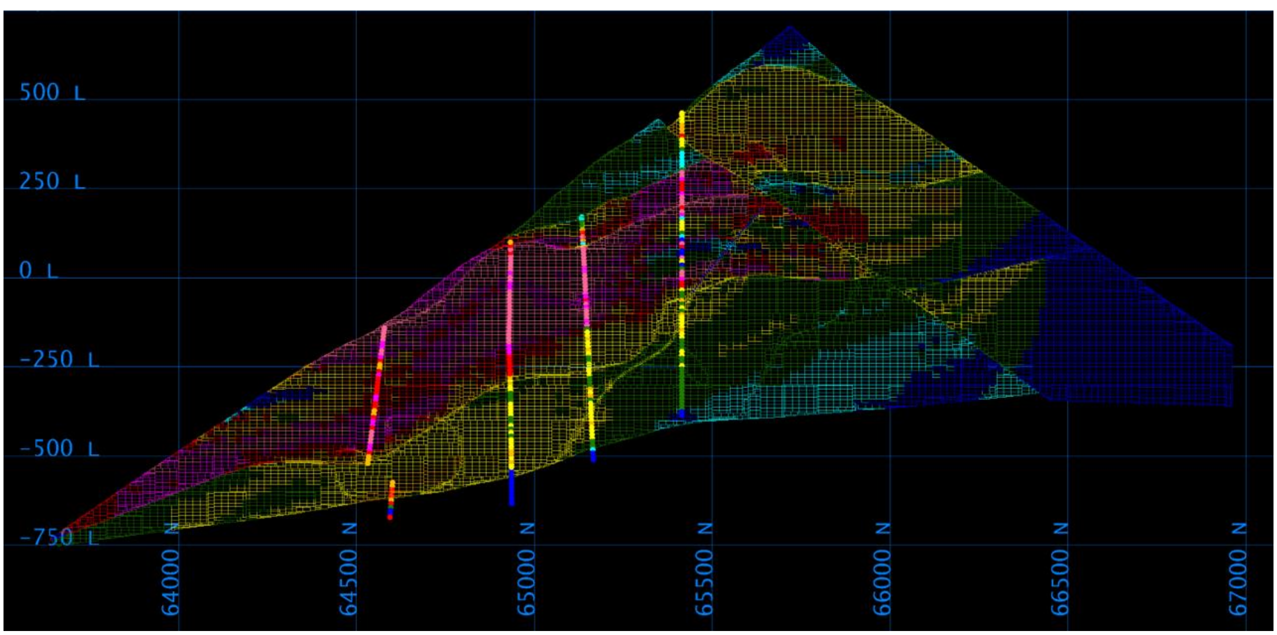

Note: Viewing CuT grades for both composites and block estimates. Source: ASCU, 2022

**Figure 14-34: Cross Section (390000E) through Cactus West, Facing West**



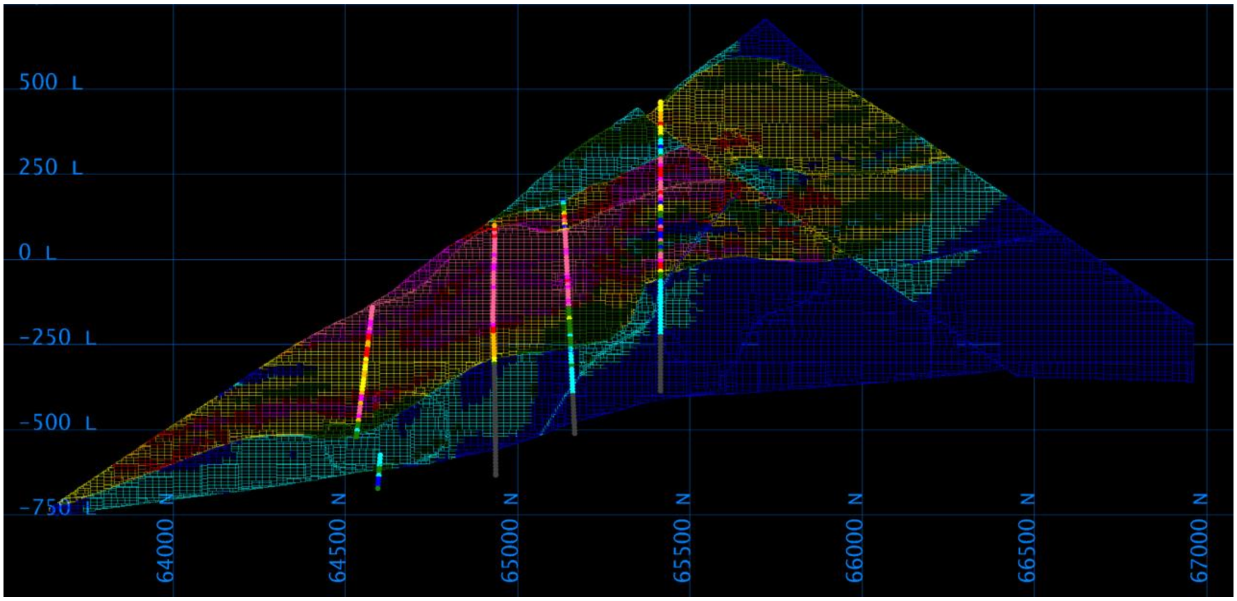

Note: Viewing Tsol grades for both composites and block estimates. Source: ASCU, 2022.

**Figure 14-35: Cross Section (391550E) through Cactus East, Facing West**



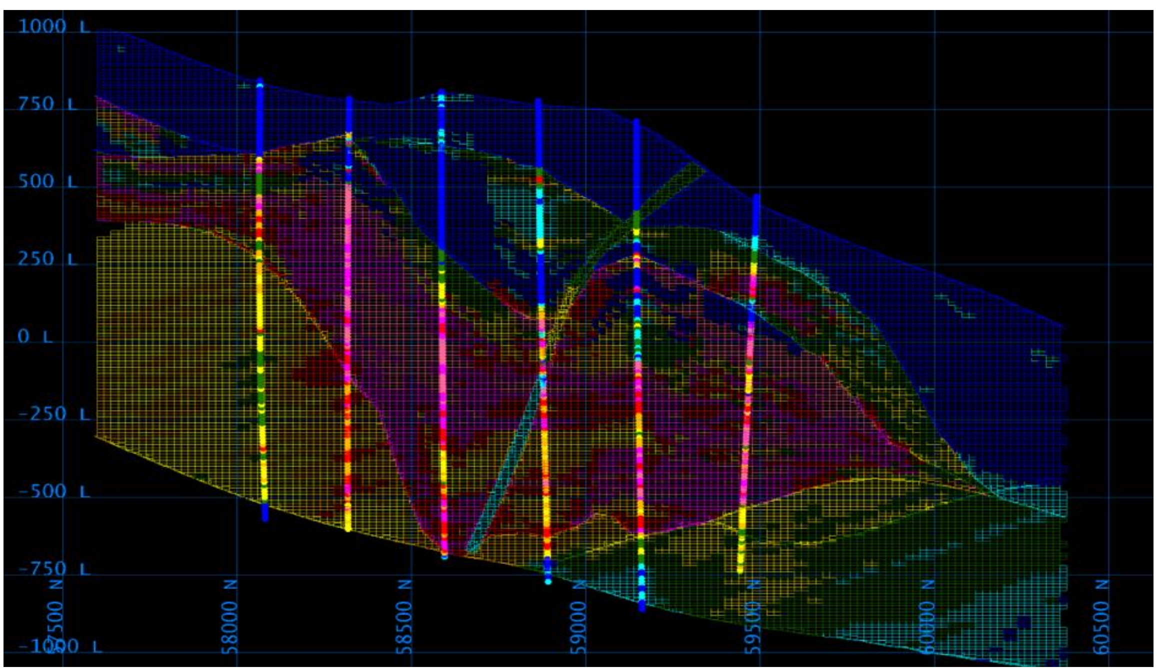

Note: Viewing CuT grades for both composites and block estimates. Source: ASCU, 2022.

**Figure 14-36: Cross Section (391550E) through Cactus East, Facing West**



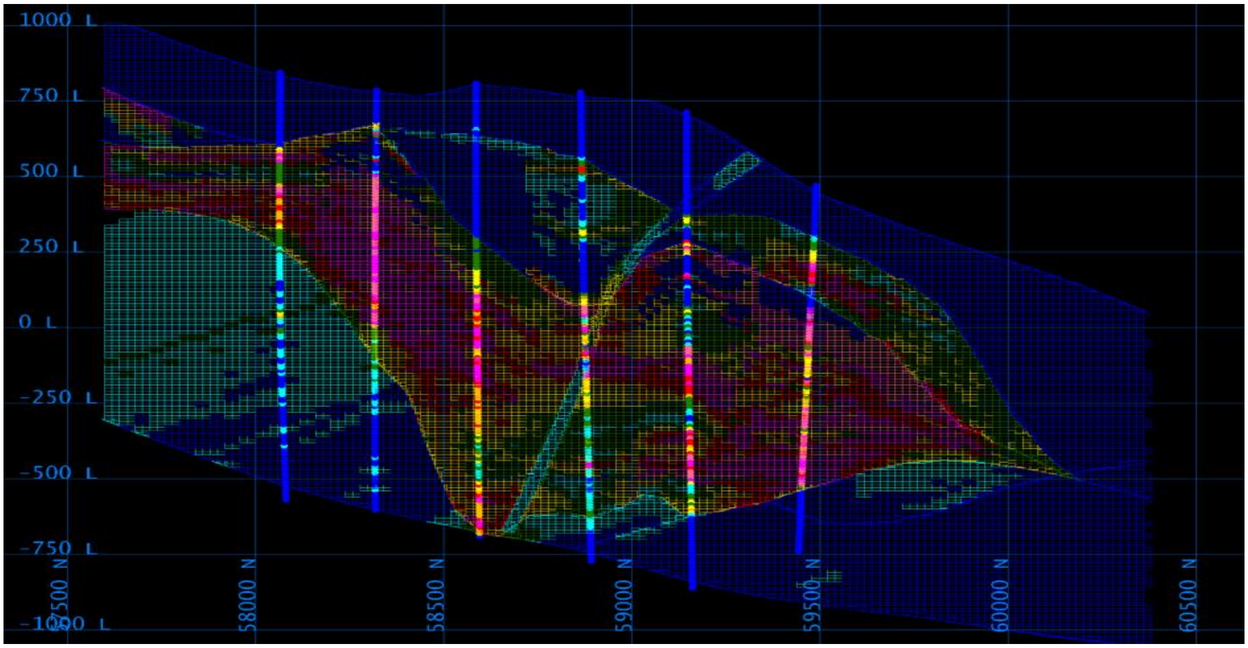

Note: Viewing Tsol grades for both composites and block estimates. Source: ASCU, 2022.

**Figure 14-37: Cross Section (58080E) through Parks/Salyer, Facing West**




Viewing CuT grades for both composites and block estimates. Source: ASCU, 2022.

**Figure 14-38: Cross Section (58080E) through Parks/Salyer, Facing West**




Note: Viewing Tsol grades for both composites and block estimates. Source: ASCU, 2022.

#### 14.1.12.3 Swath Plots

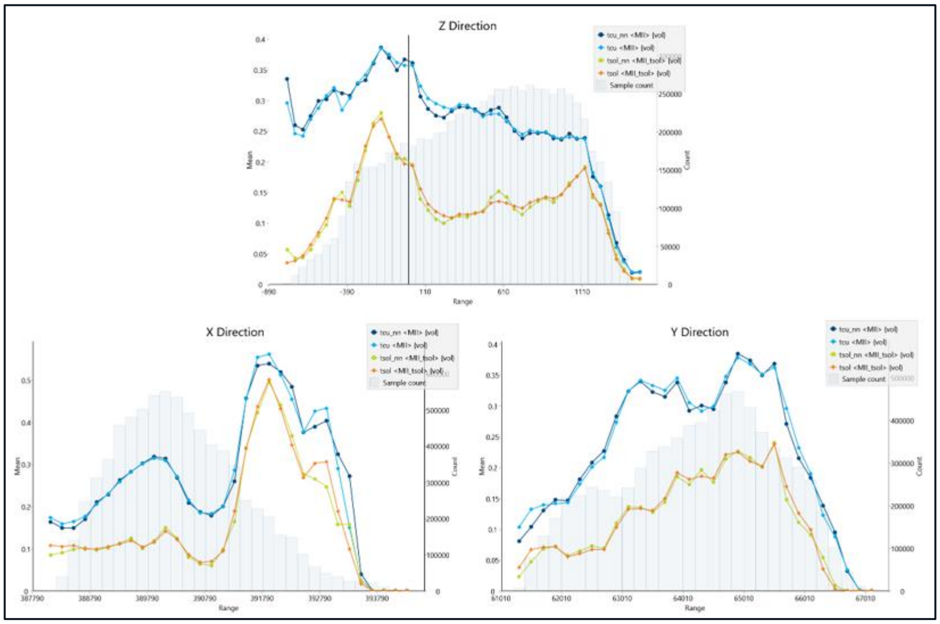
Swath plots were created to compare the grade trends through the Cactus West, Cactus East, and Parks/Salyer deposits between the estimated CuT and Tsol against the nearest neighbour models.

Comparisons for CuT and Tsol in Cactus West and West and Parks/Salyer are shown in Figure 14-39 and Figure 14-40, respectively, for easting (X direction), northing (Y direction), and elevation (Z direction).

There is good consistency in the grade trends defined by both the nearest neighbour values and the estimated block grades for both Cactus West and East, and Parks/Salyer Block Model Regularization

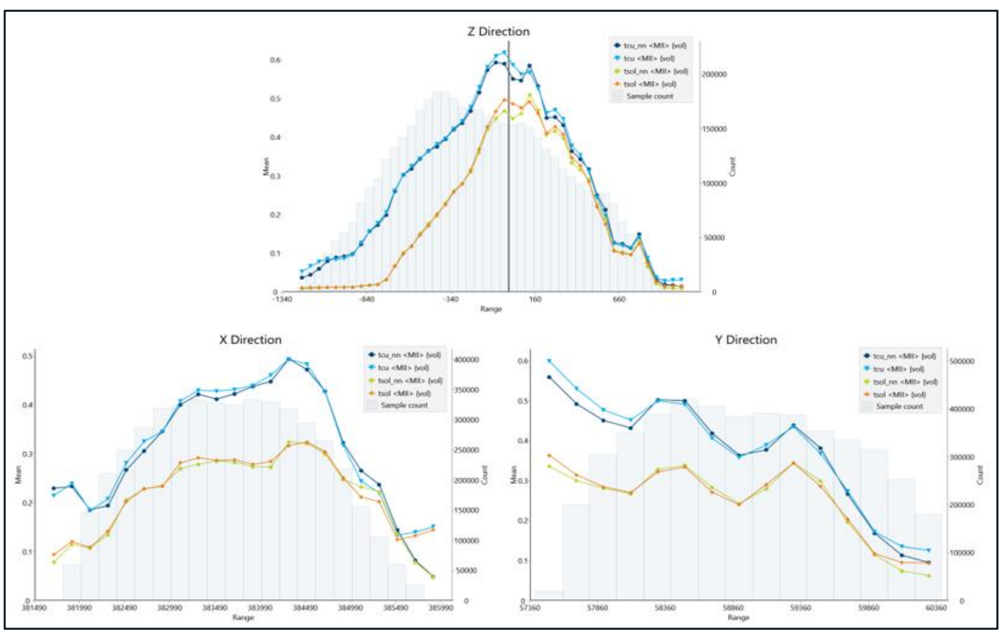
Prior to running the pit optimizer, the Cactus sub-blocked model was regularized to a new block model with regular block dimensions of 20 ft (6 m) by 20 ft (6 m) by 20 ft (6 m). Estimated grades were averaged to the regular blocks using volume weighted averaging of each of the smaller blocks falling 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 14-16 outlines the block model parameters which match the Cactus sub-block model entirely except for the application of sub- blocking.

**Figure 14-39: Swath Plots through Cactus West Comparison with Associated Nearest Neighbour Grade Trends**



Source: ASCU, 2022.

**Figure 14-40: Swath Plots through Cactus East Comparison with Associated Nearest Neighbour Grade Trends**



Source: ASCU, 2022.

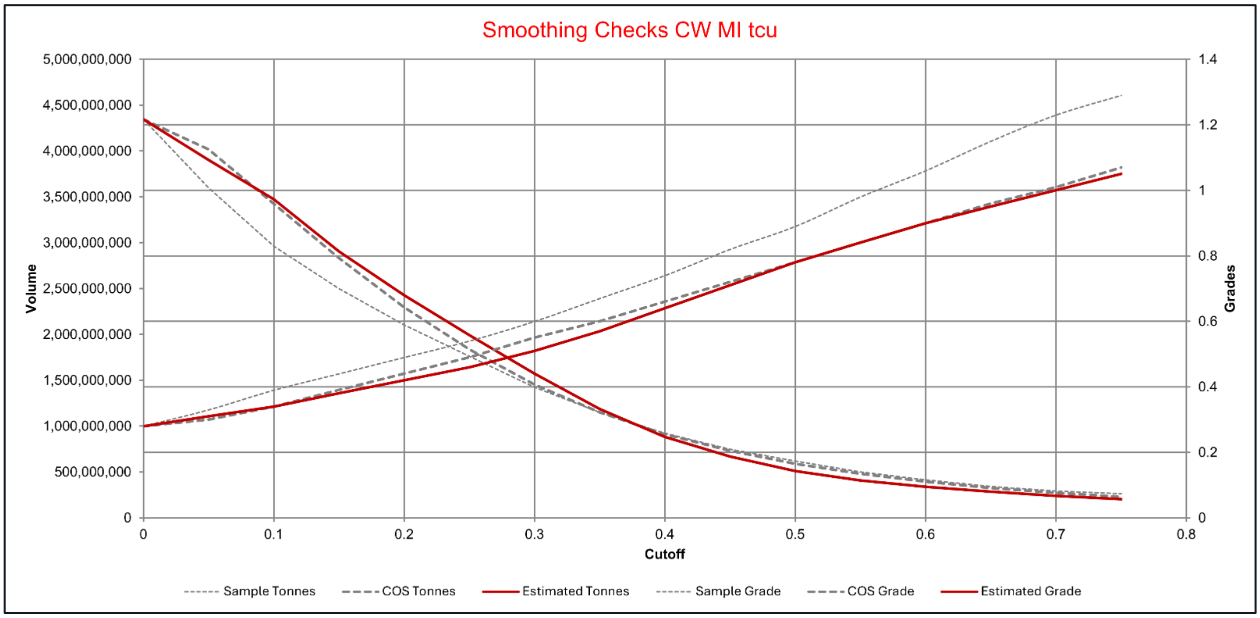
**Table 14-16: Cactus Regularised Block Model Definition Parameters**

| Block Model Definitions | 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** | |

#### 14.1.12.4 Smoothing Checks

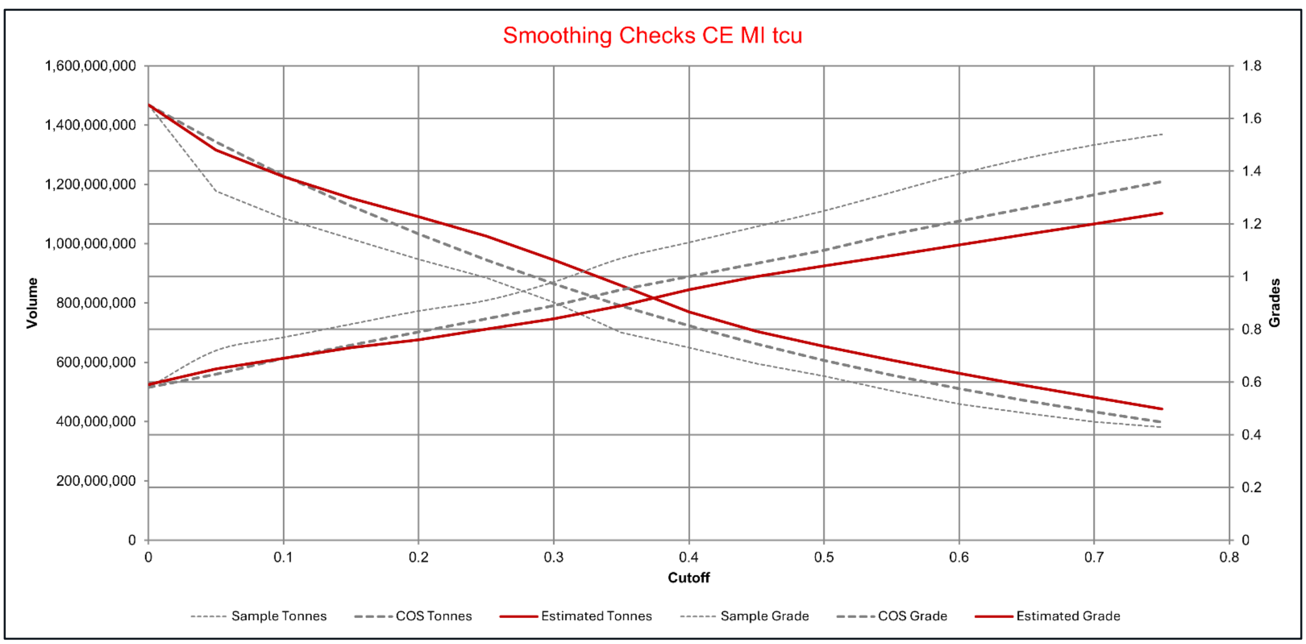
Change of support smoothing checks were undertaken to measure the appropriateness of the estimated grade tonnage curve in generating a recoverable resource appropriate to the potential mining method, associated selective mining unit size, and a range of potential economic CoGs. Change of support smoothing 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 14-41 through Figure 14-43, 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. 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. Figure 14-41 through Figure 14-43 show change of support smoothing check comparisons.

**Figure 14-41: Change of Support Smoothing Check Comparison for Cactus West**



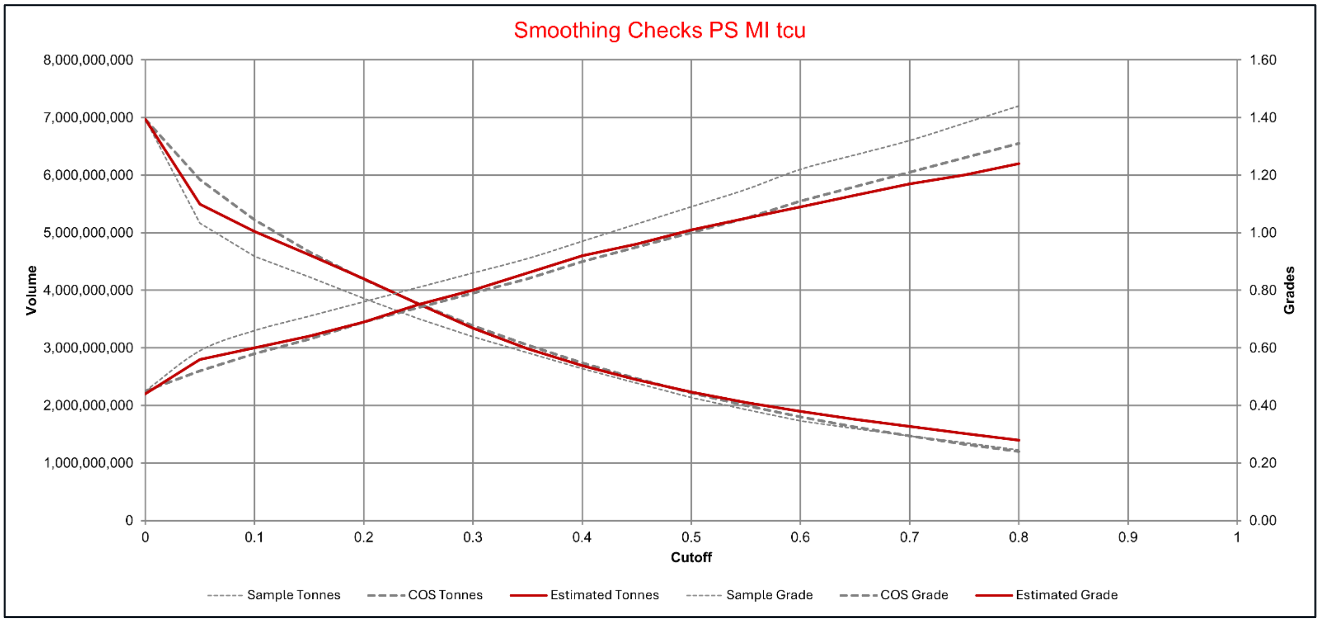

Source: ASCU, 2022.

**Figure 14-42: Change of Support Smoothing Check Comparison for Cactus East**




Source: ASCU, 2022.

**Figure 14-43: Change of Support Smoothing Check Comparison for Parks/Salyer**




Source: ASCU, 2022.

### 14.1.13 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.

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 2023.
- 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 more detailed mine planning. A 250 ft (72 m) drill spacing is seen as an appropriate spacing to determine an Indicated Resource classification.
- Within the south portion of the Cactus West deposit shallow RC drilling was undertaken to reduce spacing down to 125 ft (38 m) This spacing supports Measured Resources.

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 Measured, Indicated, and Inferred classifications was defined by the estimation pass in which the blocks were estimated. Blocks estimated in Pass 1 could be assigned to Measured, blocks estimated in Pass 2 could be assigned to Indicated, and blocks estimated in Pass 3 could be assigned to Inferred. For Measured an additional requirement was applied using and octant reach. This required that blocks had drilling surrounding them before they were flagged as Measured. A subsequent test pass of the Measured and Indicated classification was undertaken using only holes that contained downhole surveys.

For Cactus, interpreted triangulation were created to define the classification of Measured and Indicated encompassing the drillholes drilled to 125 ft and 250 ft spacing respectively and, in the case of CE, ensuring that holes contained downhole surveys. Inferred classification was assigned based on material falling outside these triangulations but having been estimated in any of passes 1 to 3.

For Parks/Salyer, an interpreted triangulation was created to define the classification of Indicated and Inferred encompassing the drillholes drilled to 250 ft and 500 ft spacing respectively.

## 14.2 Cactus Stockpile Project

The inverse distance (ID1) method was used for the estimation of copper grades to the model due to the generally unstructured geological nature of a stockpile. 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 neighbour (polygonal) method for statistical and visual review.

### 14.2.1 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.
- Understand historical stockpile dumping schedules to honour long-lived internal stockpile boundaries.

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 itself, 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 14-44 identifies these areas within the mineralized Stockpile Project that were adjusted.

**Figure 14-44: Plan View of Mineralized Stockpile Project**


Source: ASCU, 2022.

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.

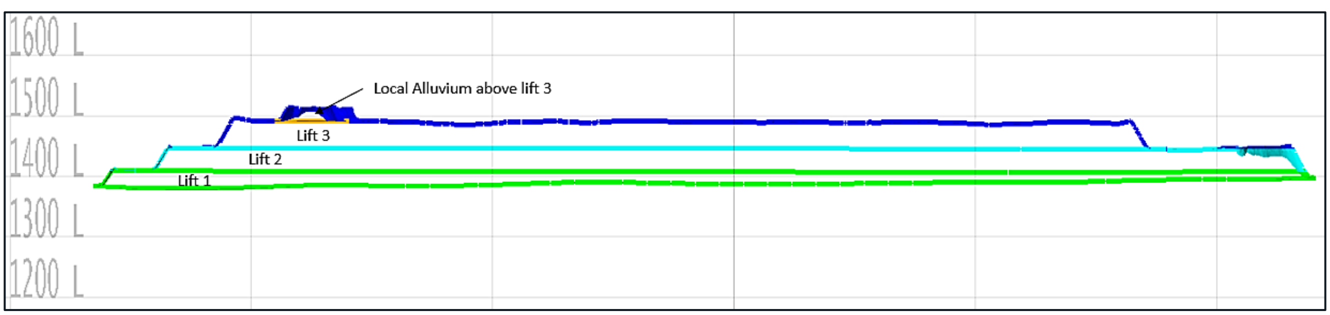
The three lifts of the mineralized Stockpile Project were defined spatially to enable separate treatment of composites and blocks for exploratory data analysis (EDA) and estimation. The lifts were separated by modelling surfaces for the original topography below the Stockpile Project (base of the Stockpile Project), the base of Lift 2, and the base of Lift 3. Lift 4 has been defined as the northern surveyed area in Figure 14-45. It is part of rehabilitation work material from a small historic primary sulphide 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 14-45). 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 14-46 is set to 250 to aid visualization.

**Figure 14-45: Section Through WRD Showing Lifts**

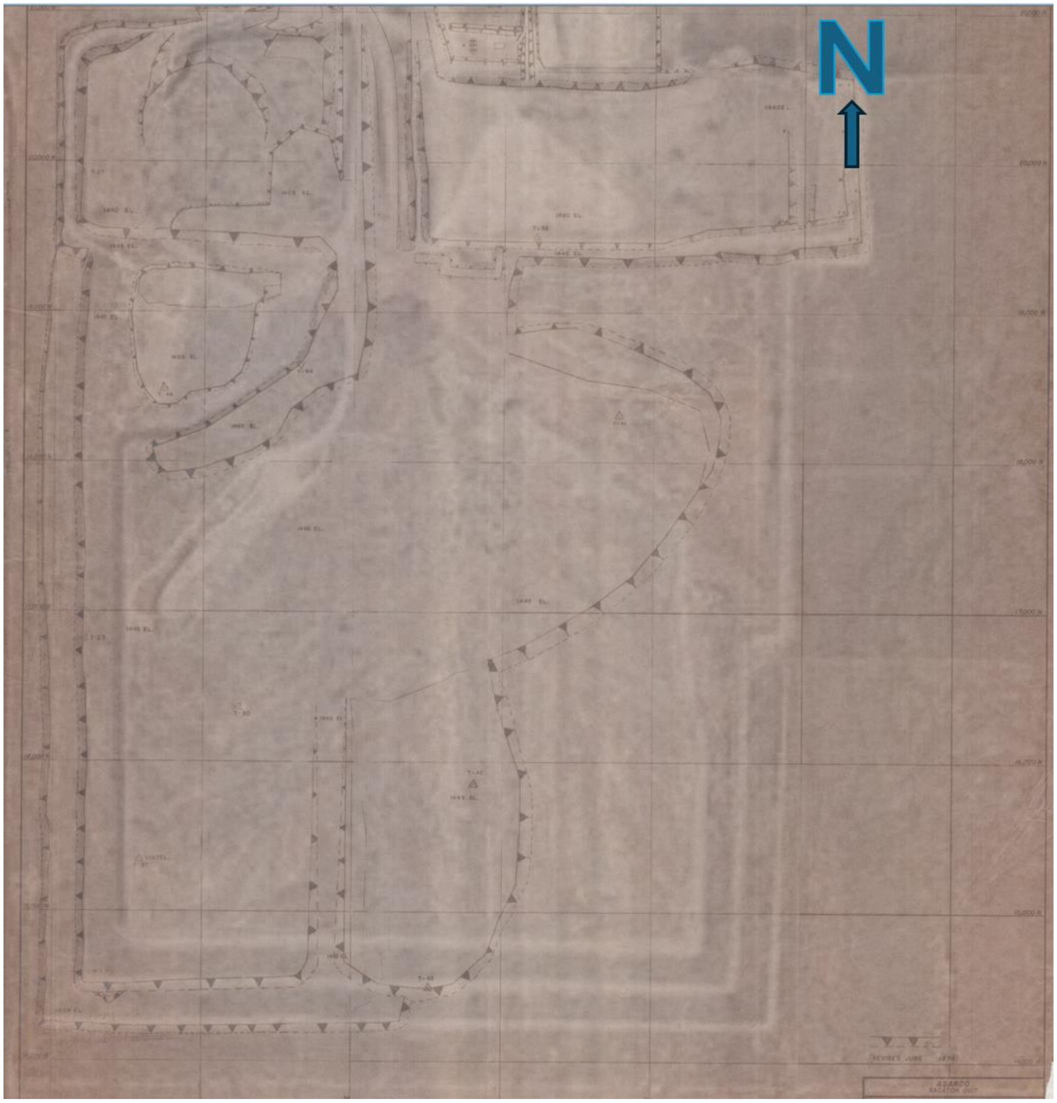



Source: ASCU, 2022.

Historical dump maps were registered into Vulcan and analysed to identify long-lived internal stockpile faces that may separate material removed from the pit at very different time periods but that may be located spatially near each other within the current stockpile. Long-lived faces may separate material that has very different grade characteristics. From this work, it was recognised that large portions of lift 1 and 2 were actually dumped as part of a single face and therefore there are areas of the stockpile that have very little difference between lifts 1 and 2 vertically. This was supported by visual review of the grades down drillholes in these areas. Lift 3 by contrast, was dumped as a single lift and far later in the pit life and therefore displays very different grade characteristics to lifts 1 and 2. Crest and toe contours were created from the historical maps so that the stockpile could be separated into different time periods which were then be honoured in the estimation plan. Figure 14-49 shows a plan view of the stockpile with the dump progress on June 30, 1976. Of note in this map is the presence of a long-lived ramp on the southern side of the stockpile and the singular dump face for lifts 1 and 2 covering the eastern side of the stockpile. Lifts 1 and 2 can therefore be combined on the eastern side of the stockpile as they were created at the same time.

Figure 14-49 – Dump map progress on June 30, 1976, showing a long-lived ramp present on the south side of the stockpile, and the singular dumping of the entire eastern side of the stockpile for lifts 1 and 2.

**Figure 14-49: June 1976 Dump Plan**




Source: ASCU, 2022.

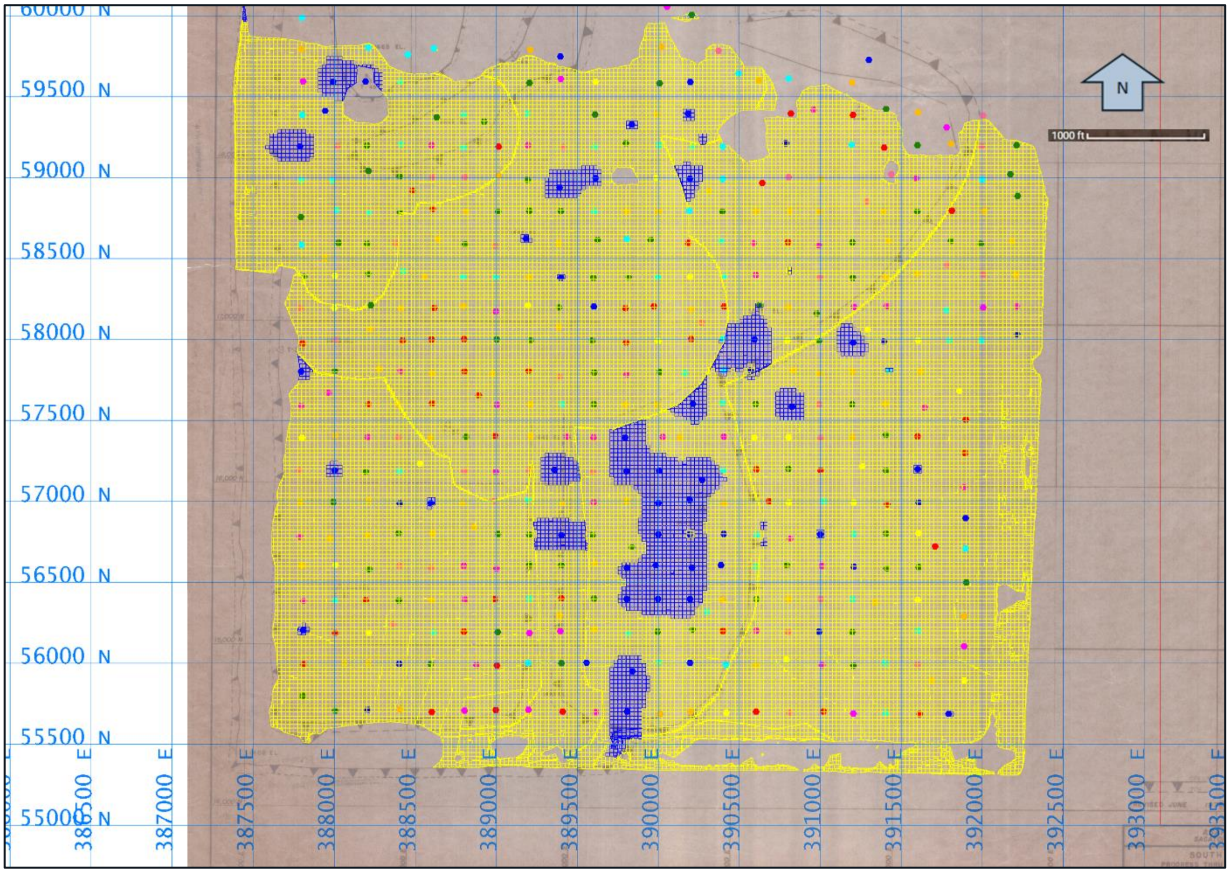
### 14.2.2 Waste Indicator

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 material, composite grades provide a general view to the grade however, individual drill holes may not a good predictor of the grades of the local volume they support. For this reason, the estimate is highly smoothed with the goal to estimate the global grade distribution and identify broad zones that are mineralized with economic grades from those that are not.

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 waste 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 14-48, 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 conventional heap leaching.

**Figure 14-46: Plan View (1405L) Showing the Indicator Defining Zones of Consistent Waste Intercepts**




Source: ASCU, 2022.

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

- For CuAS, a mineralized composite for stockpile 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 blocks using the stockpile time periods 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 domain stockpile time period, 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.

#### 14.2.2.1 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 _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.

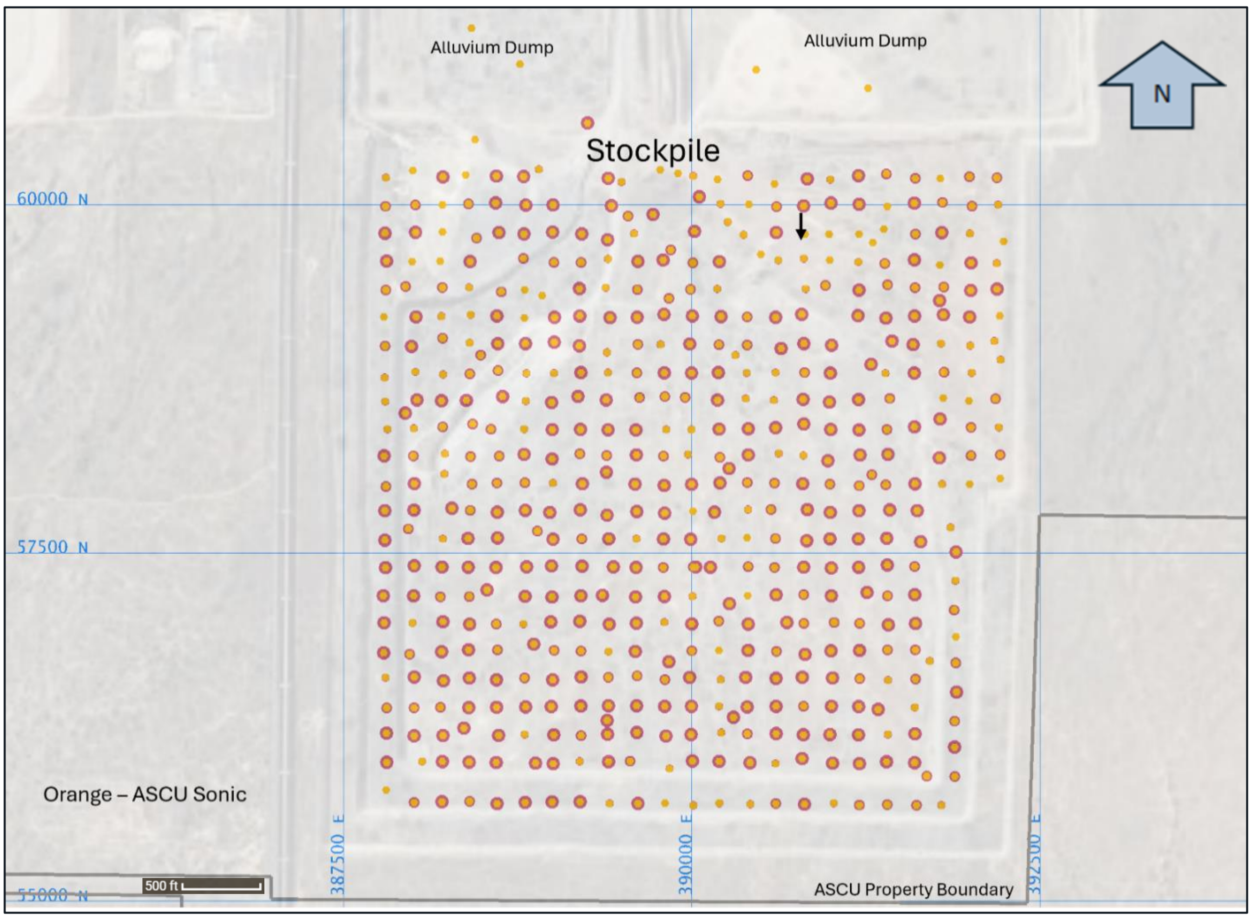
The Cactus Stockpile Project drill hole databases can be summarized by the following points:

- 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 200 ft 60 m) across the stockpile.
- 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 so the sample could be ignored for future use.
- Lithology and colour 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 colours can relate to both mineralized and non-mineralized material within the Stockpile Project.
- Copper mineralization, including copper oxides, were logged.

Figure 14-47 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 14-47: Drill Hole Collars on the Cactus Stockpile Project**




Source: ASCU, 2022.

### 14.2.3 Lithology

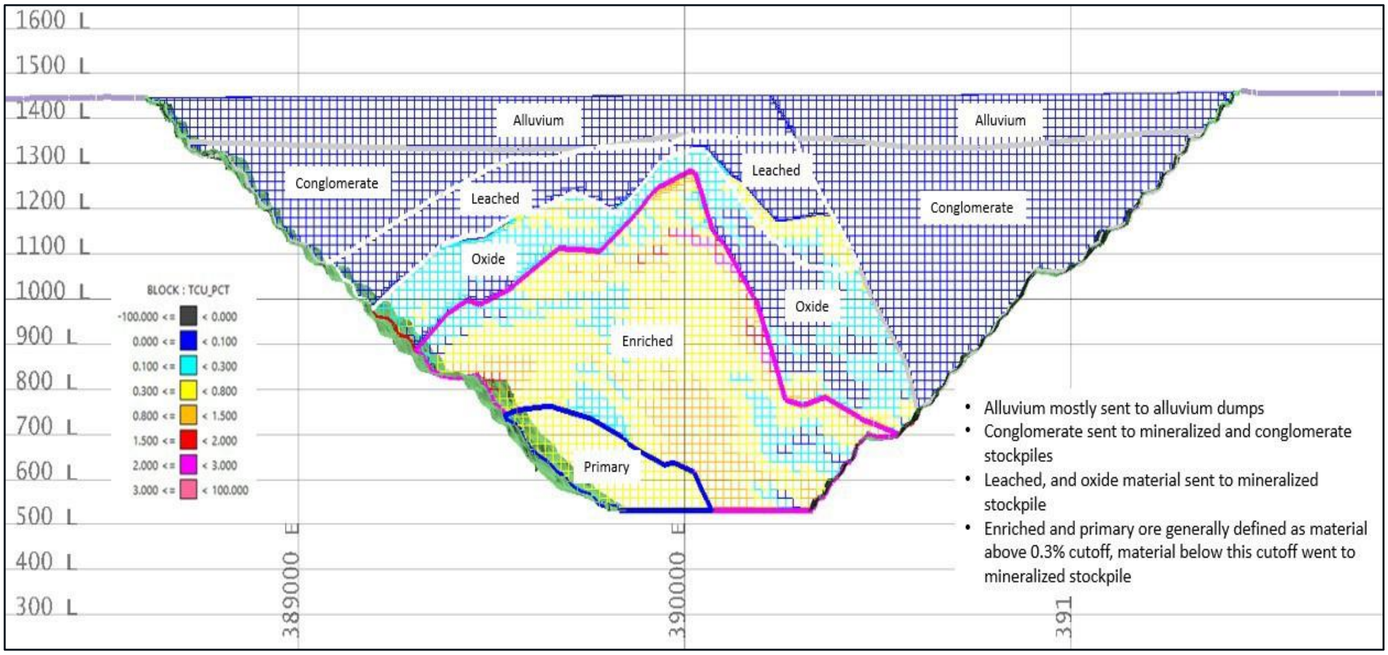
The nature of the mining operations at the historic Sacaton open pit from 1974 through 1984 has led to the dumping of material on the mineralized Stockpile Project where material types are broadly mixed. Lithology within the stockpile has no geological context, and as such is not used as any basis for the stockpile t mineral resource estimate, except to withhold assaying where broad zones of non-mineralized lithologies were present and assigned a grade of 0.002%. Table 14-17 and Figure 14-48 present the major lithological and porphyry copper alteration material types that represent mineralized and/or non-mineralized material within the stockpile . The host units to mineralization are monzonite porphyry and granite.

**Table 14-17: 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 excepting 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. |

Note: *Refer to Figure 14-54 for map of destinations.

**Figure 14-48: Cross Section (64000N) Lithologies and Destinations of Material Mined from the Pit**




Source: ASCU, 2022.

### 14.2.4 Estimation Domains

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

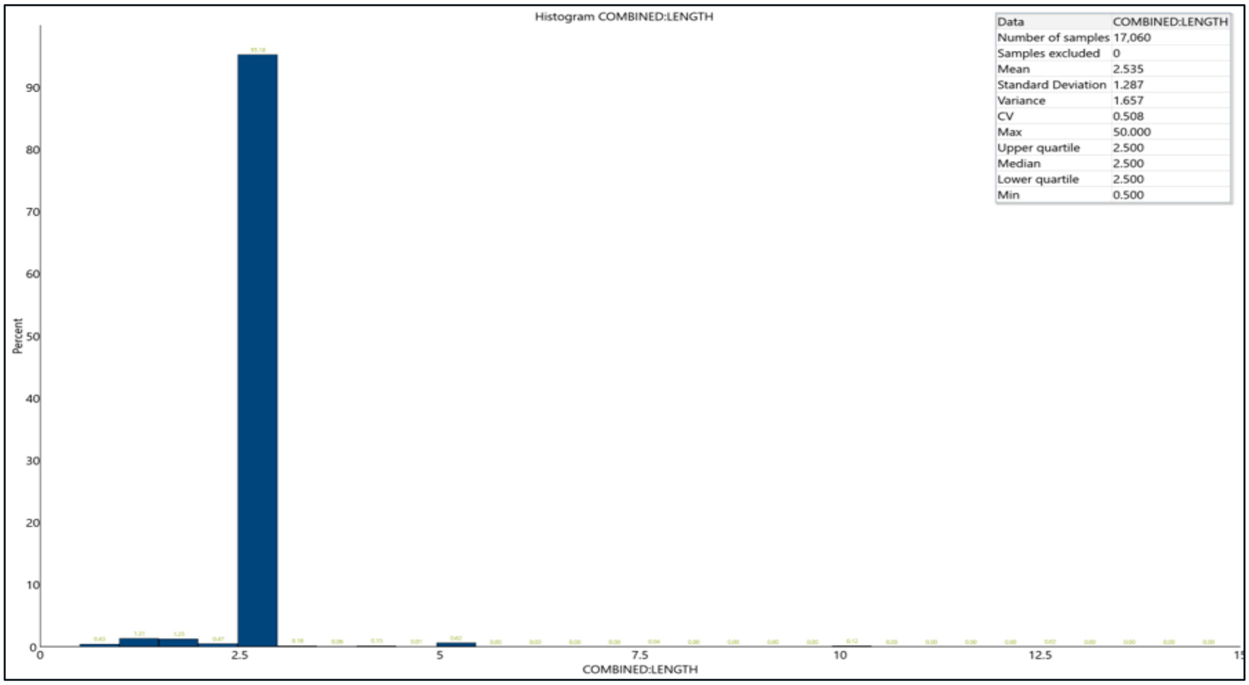
### 14.2.5 Specific Gravity

Due to the unconsolidated nature of the stockpile material, measuring bulk density can be problematic. In September 2021 four test pits were excavated to provide direct measurement of the bulk density of the insitu material. These were undertaken by excavating test pits, surveying an accurate volume of the material removed, drying the material removed, and then accurately weighing the removed material. Bulk densities for the four samples range between 0.0535 st/ft³ to 0.0753 st/ft³ with a mean of 0.0643 st/ft³. The mean bulk density was applied to the stockpile blocks.

### 14.2.6 Compositing

The drillhole intercepts were composited to 10 ft (3 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 14-49. Composites were split at the modelled lift contacts and the lifts and stockpile time periods were flagged to the composites. Where a composite was generated at less than half the composite length of 5 ft (1.5 m), it was combined into the previous 10 ft (3 m) composite to ensure short length composites were not generated. Sample grades with values of -99 were ignored during compositing.

**Figure 14-49: Histogram of Drill Hole Sample Lengths**




Source: ASCU, 2022.

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.

### 14.2.7 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 conventional 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.

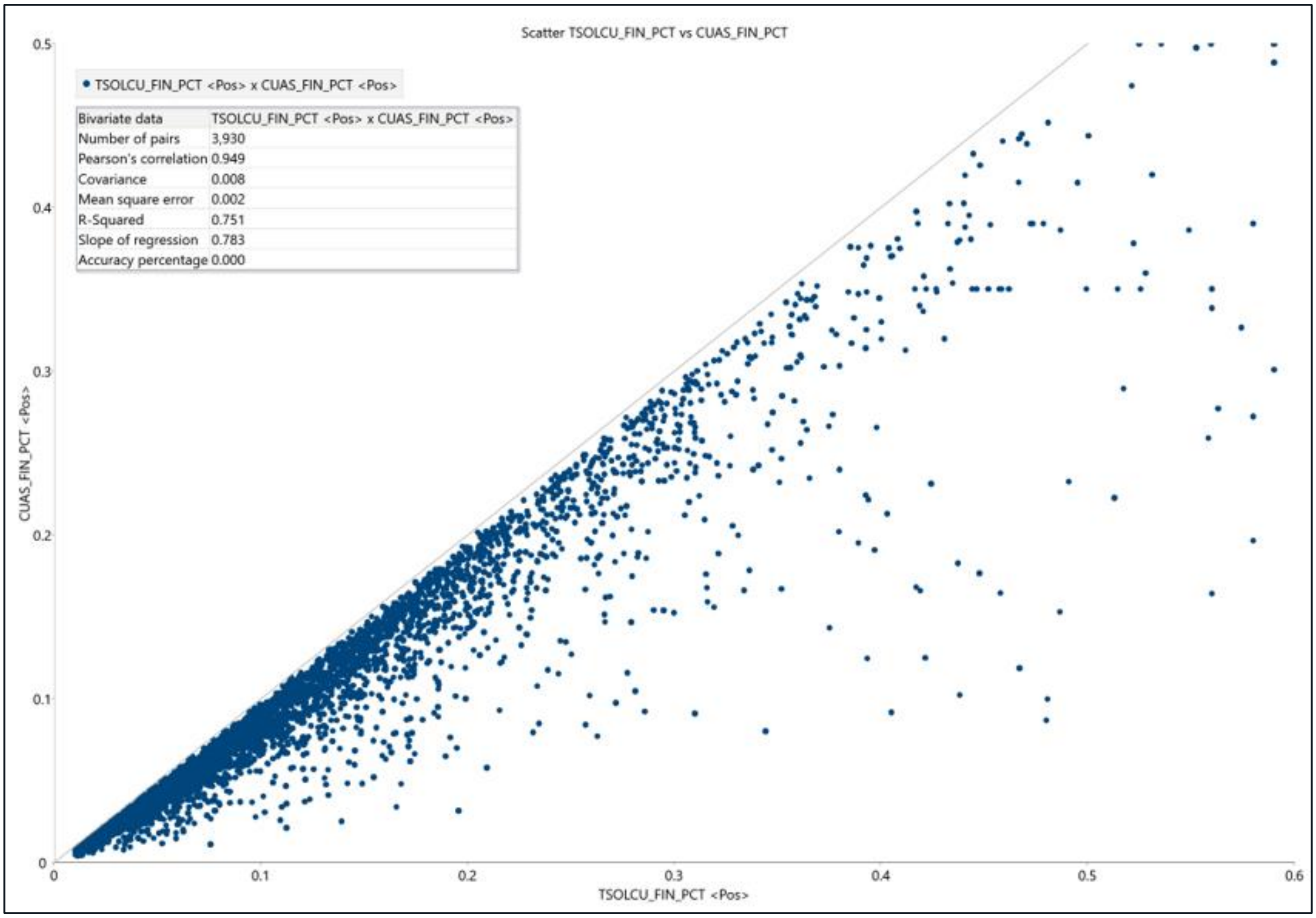
Univariate statistics were calculated for the mineralized material of the stockpile for CuAS, CuCN, Tsol, and CuT and results were reported for the entire stockpile and by individual lifts. The summary statistics are shown in Table 14-18. It can be seen from this table that mean grades decrease down 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 14-18: Lift Drill Hole 10 ft Composite Statistics for CuT, CuAS, CuCN, and Tsol**

| Variable Name | Count | Mean | Std Dev. | Variance | CV | Max | Upper Quartile | Median | Lower Quartile | Min |
|---|---|---|---|---|---|---|---|---|---|---|
| TCU_FIN_PCT LIFT1 | 736 | 0.130 | 0.119 | 0.014 | 0.922 | 1.495 | 0.164 | 0.103 | 0.057 | 0.002 |
| TCU_FIN_PCTLIFT2 | 1851 | 0.137 | 0.119 | 0.014 | 0.868 | 2.326 | 0.174 | 0.109 | 0.067 | 0.002 |
| TCU_FIN_PCT LIFT3 | 1352 | 0.187 | 0.135 | 0.018 | 0.722 | 1.609 | 0.247 | 0.158 | 0.092 | 0.002 |
| TCU_FIN_PCT LIFT4 | 24 | 0.340 | 0.153 | 0.024 | 0.452 | 0.805 | 0.389 | 0.338 | 0.282 | 0.095 |
| CUAS_FIN_PCT LIFT1 | 736 | 0.084 | 0.090 | 0.008 | 1.072 | 1.344 | 0.110 | 0.065 | 0.029 | 0.004 |
| CUAS_FIN_PCT LIFT2 | 1851 | 0.092 | 0.093 | 0.009 | 1.014 | 1.848 | 0.121 | 0.069 | 0.036 | 0.004 |
| CUAS_FIN_PCT LIFT3 | 1352 | 0.133 | 0.106 | 0.011 | 0.795 | 0.993 | 0.177 | 0.106 | 0.059 | 0.005 |
| CUAS_FIN_PCT LIFT4 | 24 | 0.254 | 0.142 | 0.020 | 0.558 | 0.754 | 0.279 | 0.237 | 0.199 | 0.062 |
| CUCN_SEQ_FIN_PCT LIFT1 | 736 | 0.024 | 0.048 | 0.002 | 2.025 | 0.797 | 0.021 | 0.011 | 0.006 | 0.002 |
| CUCN_SEQ_FIN_PCT LIFT2 | 1851 | 0.023 | 0.041 | 0.002 | 1.795 | 0.692 | 0.023 | 0.012 | 0.007 | 0.002 |
| CUCN_SEQ_FIN_PCT LIFT3 | 1352 | 0.026 | 0.036 | 0.001 | 1.359 | 0.587 | 0.031 | 0.017 | 0.010 | 0.002 |
| CUCN_SEQ_FIN_PCT LIFT4 | 24 | 0.058 | 0.032 | 0.001 | 0.553 | 0.119 | 0.084 | 0.060 | 0.030 | 0.006 |
| TSOL_FIN_PCT LIFT1 | 736 | 0.108 | 0.113 | 0.013 | 1.046 | 1.372 | 0.136 | 0.081 | 0.039 | 0.010 |
| TSOL_FIN_PCT LIFT2 | 1851 | 0.144 | 0.113 | 0.013 | 0.990 | 2.248 | 0.148 | 0.086 | 0.048 | 0.010 |
| TSOL_FIN_PCT LIFT3 | 1352 | 0.159 | 0.125 | 0.016 | 0.784 | 1.580 | 0.208 | 0.131 | 0.073 | 0.011 |
| TSOL_FIN_PCT LIFT4 | 24 | 0.312 | 0.149 | 0.022 | 0.478 | 0.772 | 0.772 | 0.303 | 0.258 | 0.068 |

Figure 14-50 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.

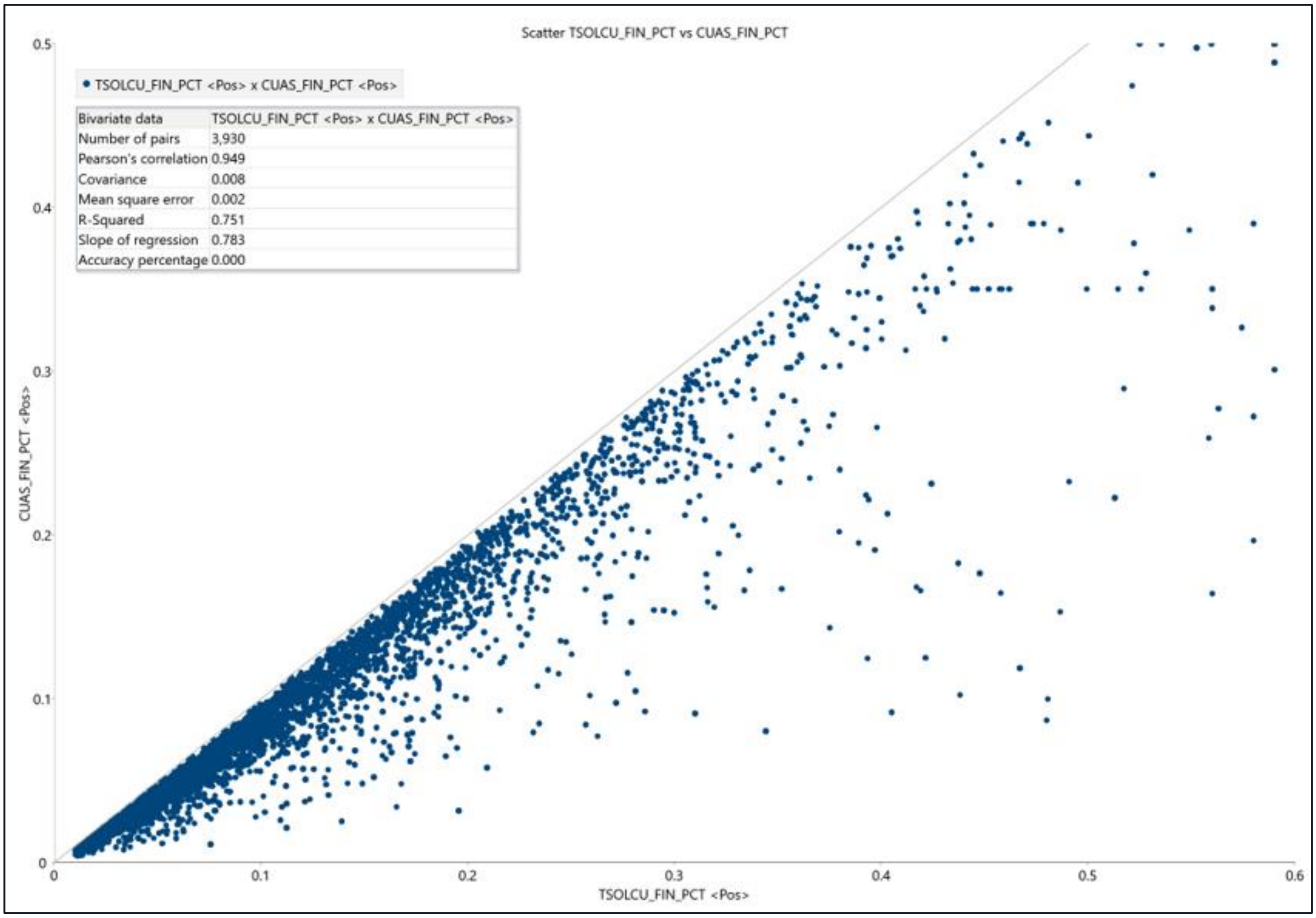
**Figure 14-50: Scatter Plot of CuAS Against Tsol**




Source: ASCU, 2022.

Figure 14-51 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.

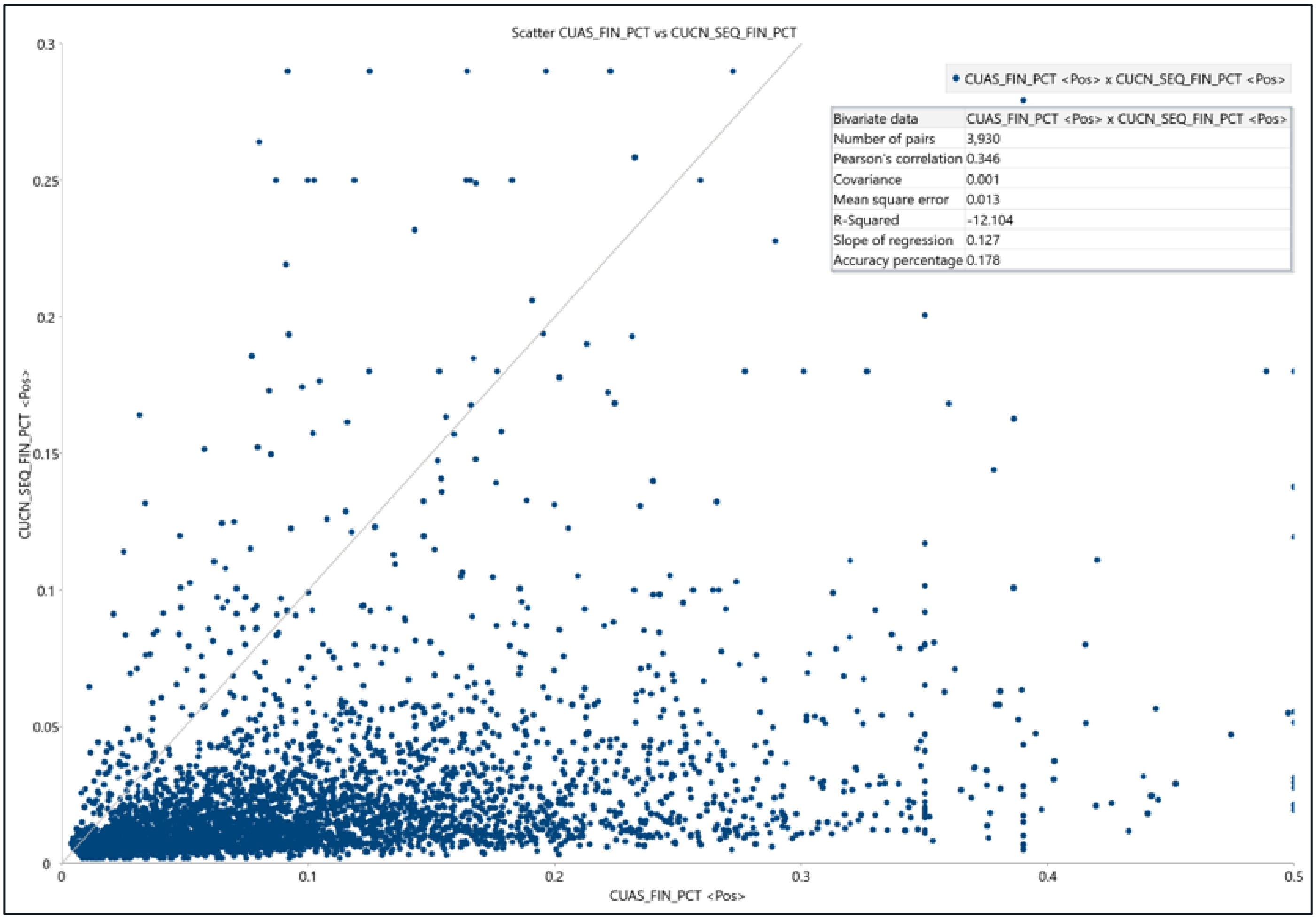
**Figure 14-51: Scatter Plot of Tsol Against CuT**




Source: ASCU, 2022.

Figure 14-52 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.

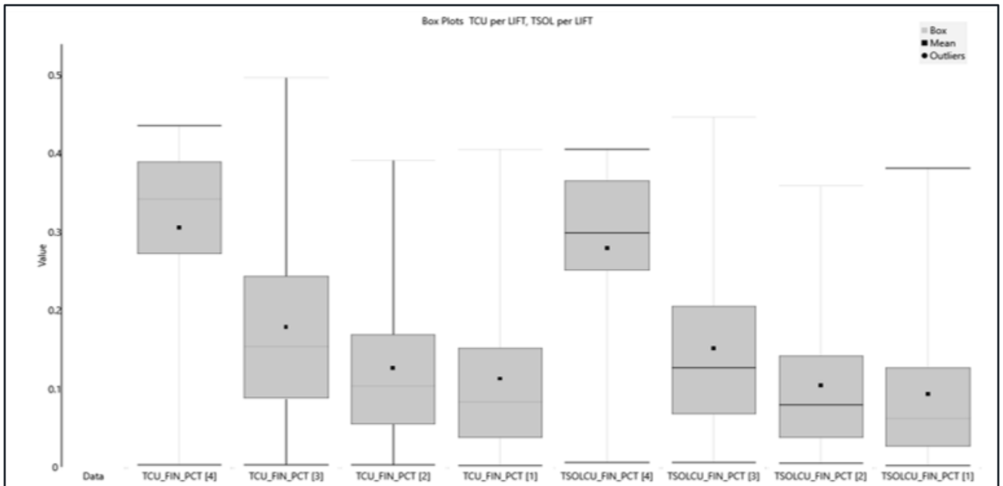
**Figure 14-52: Scatter Plot of CuCN Against CuAS**




Source: ASCU, 2022.

Figure 14-53 and Figure 14-54 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. Lift 4 represents the "bowl' are of lift 3 that was created more recently due to rehabilitation works Figure 14-57 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.

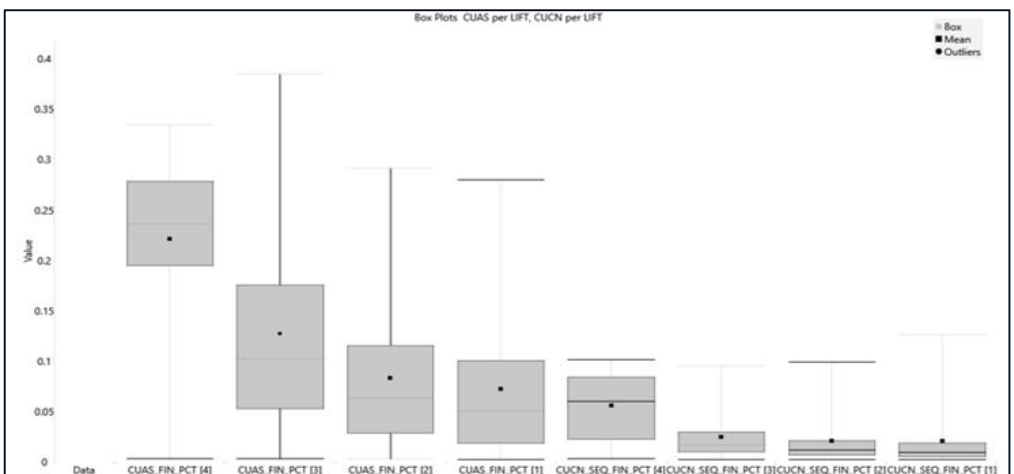
**Figure 14-53: Box Plots for CuT and Tsol Grouped by Lift Showing the Grade Reduction Down Through the Stockpile Lifts**




| Data | TCU_FIN_PCT [4] | TCU_FIN_PCT [3] | TCU_FIN_PCT [2] | TCU_FIN_PCT [1] | TSOLCU_FIN_PCT [4] | TSOLCU_FIN_PCT [3] | TSOLCU_FIN_PCT [2] | TSOLCU_FIN_PCT [1] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 25 | 1,415 | 2,020 | 852 | 25 | 1,415 | 2,020 | 852 |
| Max | 0.430 | 0.610 | 0.650 | 0.650 | 0.400 | 0.590 | 0.560 | 0.580 |
| 97.5% quantile | 0.430 | 0.490 | 0.386 | 0.400 | 0.400 | 0.441 | 0.354 | 0.376 |
| Upper quartile | 0.384 | 0.241 | 0.167 | 0.150 | 0.361 | 0.203 | 0.141 | 0.125 |
| Median | 0.337 | 0.152 | 0.102 | 0.082 | 0.295 | 0.125 | 0.079 | 0.061 |
| Lower quartile | 0.268 | 0.086 | 0.054 | 0.037 | 0.248 | 0.067 | 0.038 | 0.026 |
| 2.5% quantile | 0.003 | 0.003 | 0.003 | 0.002 | 0.006 | 0.006 | 0.005 | 0.002 |
| Min | 0.003 | 0.002 | 0.002 | 0.002 | 0.006 | 0.002 | 0.002 | 0.002 |
| Mean | 0.302 | 0.176 | 0.125 | 0.111 | 0.276 | 0.150 | 0.103 | 0.092 |

Source: ASCU, 2022.

**Figure 14-54: Box Plots for CuAS and CuCN Grouped by Lift**




| Data | CUAS_FIN_PCT [4] | CUAS_FIN_PCT [3] | CUAS_FIN_PCT [2] | CUAS_FIN_PCT [1] | CUCN_SEQ_FIN_PCT [4] | CUCN_SEQ_FIN_PCT [3] | CUCN_SEQ_FIN_PCT [2] | CUCN_SEQ_FIN_PCT [1] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 25 | 1,415 | 2,020 | 852 | 25 | 1,415 | 2,020 | 852 |
| Max | 0.330 | 0.500 | 0.350 | 0.390 | 0.100 | 0.180 | 0.250 | 0.290 |
| 97.5% quantile | 0.330 | 0.380 | 0.288 | 0.276 | 0.100 | 0.094 | 0.098 | 0.124 |
| Upper quartile | 0.275 | 0.174 | 0.114 | 0.100 | 0.084 | 0.030 | 0.021 | 0.019 |
| Median | 0.233 | 0.101 | 0.062 | 0.050 | 0.059 | 0.016 | 0.011 | 0.009 |
| Lower quartile | 0.192 | 0.052 | 0.028 | 0.018 | 0.022 | 0.009 | 0.006 | 0.005 |
| 2.5% quantile | 0.003 | 0.003 | 0.002 | 0.002 | 0.003 | 0.002 | 0.002 | 0.002 |
| Min | 0.003 | 0.002 | 0.002 | 0.002 | 0.003 | 0.002 | 0.002 | 0.002 |
| Mean | 0.219 | 0.125 | 0.082 | 0.071 | 0.055 | 0.024 | 0.020 | 0.020 |

Source: ASCU, 2022.

### 14.2.8 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.

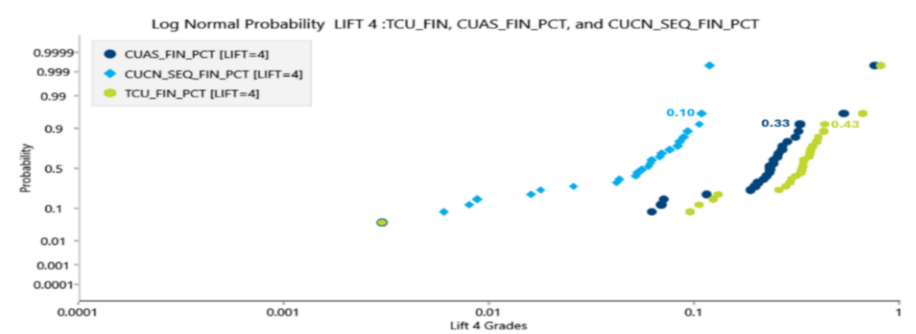
Log normal probability plots were generated per lift to define applicable capping levels within each lift. Due to the grade distribution differences between lifts a single threshold defined for the global population was not appropriate.

Table 14-19 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 14-55 through Figure 14-58). Lift 4 represents only a very limited dataset with its own characteristics.

**Table 14-19: Capping Threshold Values Applied per Lift to the Estimation of CuT, CuAS, and CuCN**

| Lift | CuT | CuAS | CuCN |
|---|---|---|---|
| Lift 4 | 0.43 | 0.33 | 0.10 |
| Lift 3 | 0.61 | 0.50 | 0.18 |
| Lift 2 | 0.65 | 0.35 | 0.25 |
| Lift 1 | 0.65 | 0.39 | 0.29 |

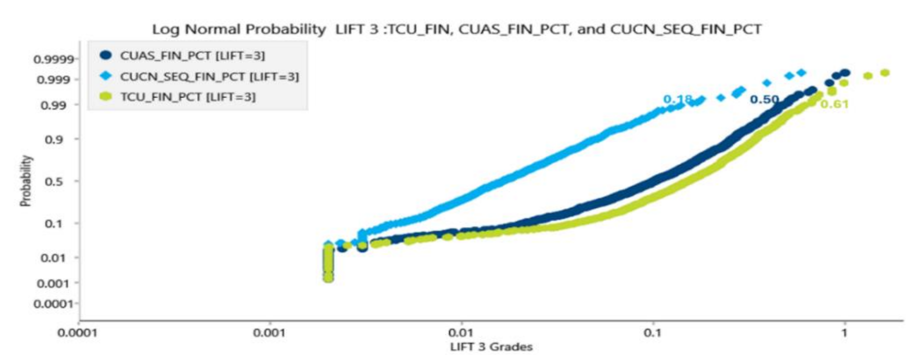
**Figure 14-55: Log Normal Probability Plot of Lift 4 Copper Assays with Capping Grades**




| Data | CUAS_FIN_PCT [LIFT=4] | CUCN_SEQ_FIN_PCT [LIFT=4] | TCU_FIN_PCT [LIFT=4] |
|---|---|---|---|
| Number of samples | 25 | 25 | 25 |
| Mean | 0.244 | 0.056 | 0.326 |
| Standard Deviation | 0.147 | 0.033 | 0.164 |
| Variance | 0.022 | 0.001 | 0.027 |
| CV | 0.604 | 0.596 | 0.503 |
| Max | 0.754 | 0.119 | 0.805 |
| Upper quartile | 0.275 | 0.084 | 0.384 |
| Median | 0.233 | 0.059 | 0.337 |
| Lower quartile | 0.192 | 0.022 | 0.268 |
| Min | 0.003 | 0.003 | 0.003 |

Source: ASCU, 2022.

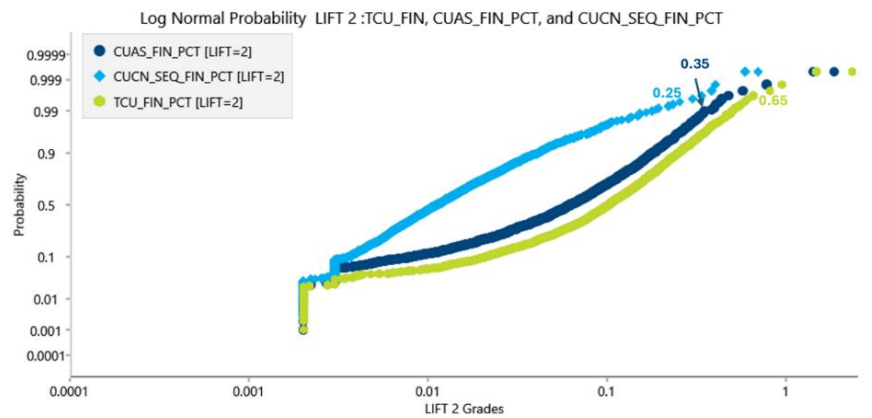
**Figure 14-56: Log Normal Probability Plot of Lift 3 Copper Assays with Capping Grades**




| Data | CUAS_FIN_PCT [LIFT=3] | CUCN_SEQ_FIN_PCT [LIFT=3] | TCU_FIN_PCT [LIFT=3] |
|---|---|---|---|
| Number of samples | 1,415 | 1,415 | 1,415 |
| Mean | 0.127 | 0.025 | 0.179 |
| Standard Deviation | 0.107 | 0.035 | 0.137 |
| Variance | 0.011 | 0.001 | 0.019 |
| CV | 0.840 | 1.398 | 0.767 |
| Max | 0.993 | 0.587 | 1.609 |
| Upper quartile | 0.174 | 0.030 | 0.241 |
| Median | 0.101 | 0.016 | 0.152 |
| Lower quartile | 0.052 | 0.009 | 0.086 |
| Min | 0.002 | 0.002 | 0.002 |

Source: ASCU, 2022.

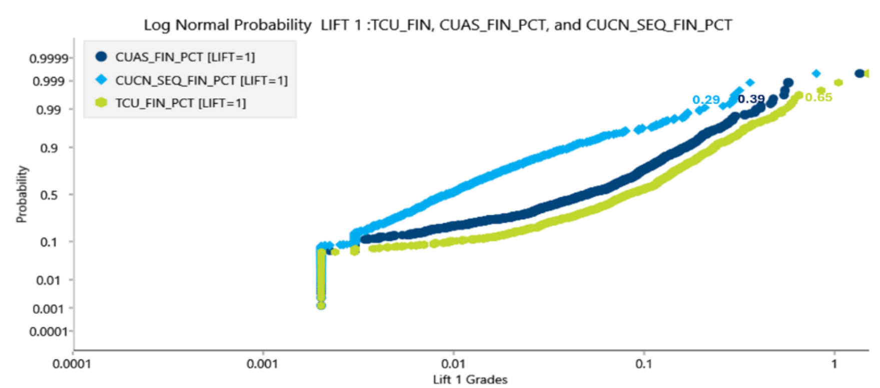
**Figure 14-57: Log Normal Probability Plot of Lift 2 Copper Assays with Capping Grades**




| Data | CUAS_FIN_PCT [LIFT=2] | CUCN_SEQ_FIN_PCT [LIFT=2] | TCU_FIN_PCT [LIFT=2] |
|---|---|---|---|
| Number of samples | 2,020 | 2,020 | 2,020 |
| Mean | 0.084 | 0.021 | 0.126 |
| Standard Deviation | 0.092 | 0.039 | 0.120 |
| Variance | 0.009 | 0.002 | 0.014 |
| CV | 1.095 | 1.871 | 0.945 |
| Max | 1.848 | 0.692 | 2.326 |
| Upper quartile | 0.114 | 0.021 | 0.167 |
| Median | 0.062 | 0.011 | 0.102 |
| Lower quartile | 0.028 | 0.006 | 0.054 |
| Min | 0.002 | 0.002 | 0.002 |

Source: ASCU, 2022.

**Figure 14-58: Log Normal Probability Plot of Lift 1 Copper Assays with Capping Grades**




| Data | CUAS_FIN_PCT [LIFT=1] | CUCN_SEQ_FIN_PCT [LIFT=1] | TCU_FIN_PCT [LIFT=1] |
|---|---|---|---|
| Number of samples | 852 | 852 | 852 |
| Mean | 0.073 | 0.021 | 0.113 |
| Standard Deviation | 0.089 | 0.045 | 0.119 |
| Variance | 0.008 | 0.002 | 0.014 |
| CV | 1.208 | 2.169 | 1.049 |
| Max | 1.344 | 0.797 | 1.495 |
| Upper quartile | 0.100 | 0.019 | 0.150 |
| Median | 0.050 | 0.009 | 0.082 |
| Lower quartile | 0.018 | 0.005 | 0.037 |
| Min | 0.002 | 0.002 | 0.002 |

Source: ASCU, 2022.

### 14.2.9 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.

### 14.2.10 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. 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 could be utilized to potentially mine the stockpile (two benches per lift). Table 14-20 displays the key block definition parameters.

**Table 14-20: Block Model Definition Parameters**

| Parameter | 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 | 25.0 | 25.0 | 10.00 |
| Sub-block Block Size | 2.5 | 2.5 | 0.25 |
| **Total Blocks** | | | **4,290,456** |

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 volume the total mineralized stockpile. Results are reported in Table 14-21.

**Table 14-21: Block Model Volumes Compared to Triangulation Volumes**

| Material | Block Volume | Triangulation Volume | Difference |
|---|---|---|---|
| **Total** | **2,371,321,695** | **2,371,318,913** | **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 was backfilled with historical sulphide 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).

Review of historical dump maps indicated six major periods of time that reflected the presence of long-lived dump faces that should be honoured by the stockpile estimates. These periods were designated as 1973, 1975, 1976, 1979, 1980, and 1984.

Twenty-foot (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.

### 14.2.11 Estimation Plan

For combination of lift and time period 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 supporting.

Significant parameters used in the copper estimates included the following:

- Domain combinations of stockpile lifts and time periods were estimated with soft boundaries being implemented generally between adjacent time periods.
- The estimation was undertaken using two passes. The first pass focused estimating the 200 ft (61 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 for the bulk of the estimation based on 200ft drilling.
- 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 300 ft (91.4 m) × 300 ft (91.4 m) × 30 ft (9.1 m) for the first pass. The search ellipse was set to 500 ft (57 m) × 500 ft (57 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 neighbour value was assigned to the blocks during the estimation process for use in validations of the estimate.

### 14.2.12 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.

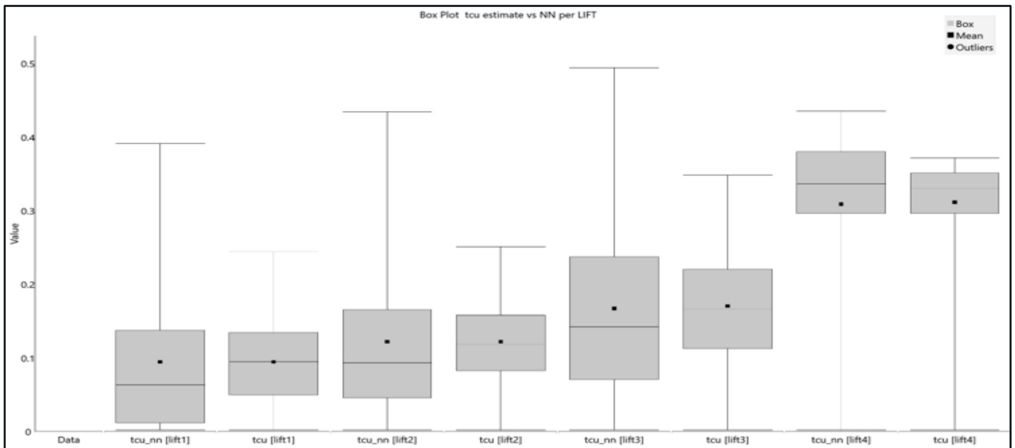
### 14.2.13 Validations

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

#### 14.2.13.1 Box Plots

Box plots were created for CuT, CuAS, and CuCN mean grades and distributions within each lift to compare against the nearest neighbour (representing declustered composites) in Figure 14-59 through Figure 14-61, respectively. All comparisons show very similar mean grades between the estimated blocks and the nearest neighbour. The adjustment from a nearest neighbour 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 neighbour plots. No maximum values of the nearest neighbour statistics are reported higher than the planned top cuts, indicating that the top cut was applied to the estimation as planned.

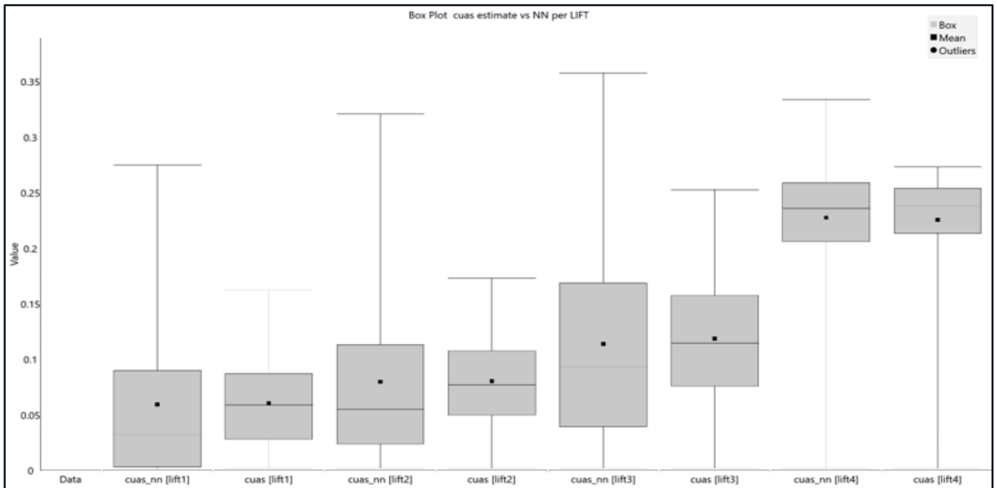
**Figure 14-59: Box Plots Comparing CuT for the Cactus Stockpile Project Against the Nearest Neighbour Grouped by Lift**




| Data | tcu_nn [lift1] | tcu [lift1] | tcu_nn [lift2] | tcu [lift2] | tcu_nn [lift3] | tcu [lift3] | tcu_nn [lift4] | tcu [lift4] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 817,887 | 817,887 | 878,835 | 878,835 | 719,554 | 719,554 | 34,710 | 34,710 |
| Max | 0.650 | 0.495 | 0.650 | 0.584 | 0.610 | 0.549 | 0.430 | 0.399 |
| 97.5% quantile | 0.386 | 0.241 | 0.429 | 0.248 | 0.488 | 0.344 | 0.430 | 0.367 |
| Upper quartile | 0.136 | 0.133 | 0.164 | 0.156 | 0.235 | 0.218 | 0.376 | 0.347 |
| Median | 0.063 | 0.094 | 0.092 | 0.117 | 0.140 | 0.164 | 0.333 | 0.326 |
| Lower quartile | 0.011 | 0.049 | 0.045 | 0.082 | 0.069 | 0.111 | 0.292 | 0.293 |
| 2.5% quantile | 0.002 | 0.002 | 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 | 0.093 | 0.094 | 0.120 | 0.121 | 0.165 | 0.168 | 0.305 | 0.308 |

Source: ASCU, 2022.

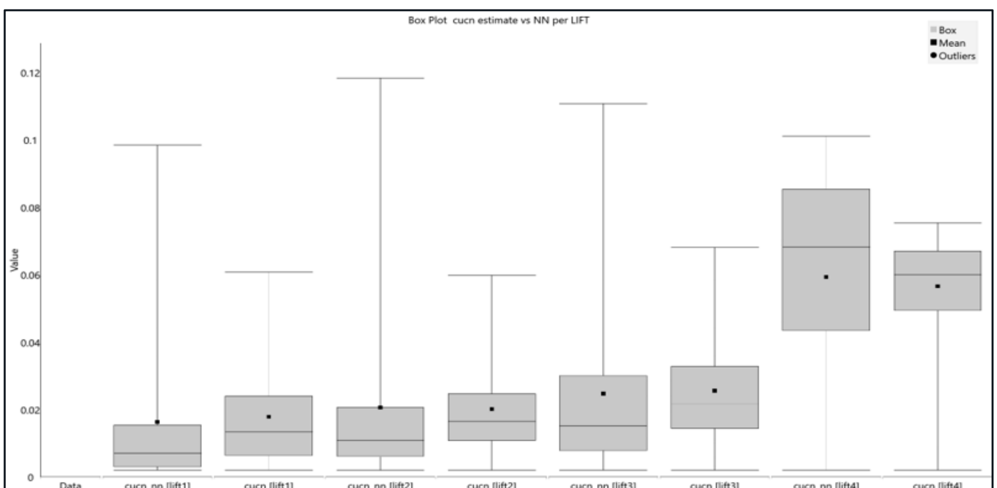
**Figure 14-60: Box Plots Comparing CuAS for the Cactus Stockpile Project Against the Nearest Neighbour Grouped by Lift**




| | cuas_nn [lift1] | cuas [lift1] | cuas_nn [lift2] | cuas [lift2] | cuas_nn [lift3] | cuas [lift3] | cuas_nn [lift4] | cuas [lift4] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 817,887 | 817,887 | 878,835 | 878,835 | 719,554 | 719,554 | 34,710 | 34,710 |
| Max | 0.390 | 0.270 | 0.390 | 0.318 | 0.500 | 0.400 | 0.330 | 0.301 |
| 97.5% quantile | 0.272 | 0.161 | 0.317 | 0.171 | 0.354 | 0.250 | 0.330 | 0.270 |
| Upper quartile | 0.089 | 0.086 | 0.112 | 0.107 | 0.167 | 0.156 | 0.256 | 0.251 |
| Median | 0.032 | 0.058 | 0.054 | 0.076 | 0.092 | 0.113 | 0.233 | 0.235 |
| Lower quartile | 0.003 | 0.028 | 0.023 | 0.049 | 0.038 | 0.075 | 0.203 | 0.211 |
| 2.5% quantile | 0.002 | 0.002 | 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 | 0.058 | 0.060 | 0.079 | 0.079 | 0.112 | 0.117 | 0.225 | 0.223 |

Source: ASCU, 2022.

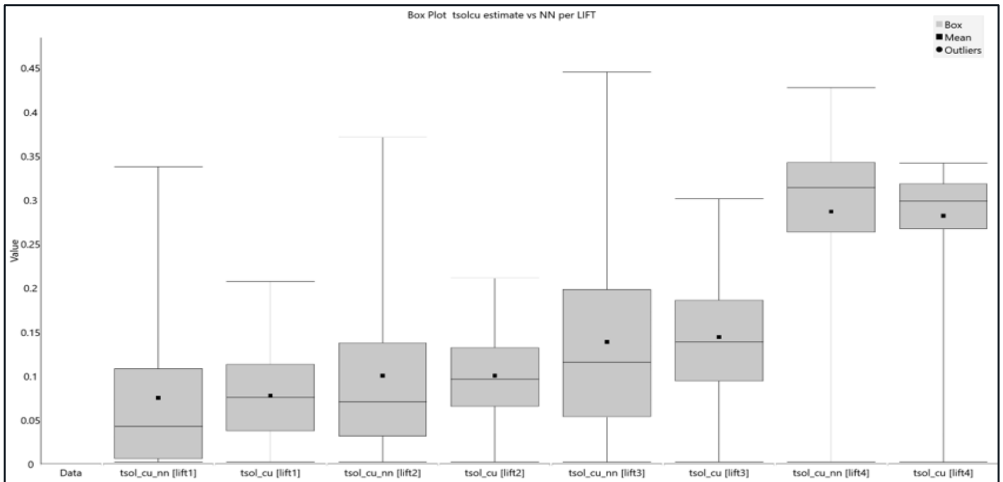
**Figure 14-61: Box Plots Comparing CuCN for the Cactus Stockpile Project Against the Nearest Neighbour Grouped by Lift**




| | cucn_nn [lift1] | cucn [lift1] | cucn_nn [lift2] | cucn [lift2] | cucn_nn [lift3] | cucn [lift3] | cucn_nn [lift4] | cucn [lift4] |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 817,887 | 817,887 | 878,835 | 878,835 | 719,554 | 719,554 | 34,710 | 34,710 |
| Max | 0.290 | 0.227 | 0.279 | 0.219 | 0.180 | 0.115 | 0.100 | 0.085 |
| 97.5% quantile | 0.097 | 0.060 | 0.117 | 0.059 | 0.109 | 0.067 | 0.100 | 0.075 |
| Upper quartile | 0.015 | 0.024 | 0.020 | 0.024 | 0.030 | 0.033 | 0.084 | 0.066 |
| Median | 0.007 | 0.013 | 0.011 | 0.016 | 0.015 | 0.021 | 0.068 | 0.059 |
| Lower quartile | 0.003 | 0.006 | 0.006 | 0.011 | 0.008 | 0.014 | 0.043 | 0.049 |
| 2.5% quantile | 0.002 | 0.002 | 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 | 0.016 | 0.018 | 0.020 | 0.020 | 0.024 | 0.025 | 0.059 | 0.056 |

Source: ASCU, 2022.

**Figure 14-62: Box Plots Comparing Tsol for the Cactus Stockpile Project Against the Nearest Neighbour as an Independent Cross Check Grouped by Lift**




| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Number of samples | 817,887 | 817,887 | 878,835 | 878,835 | 719,554 | 719,554 | 34,710 | 34,710 |
| Max | 0.669 | 0.432 | 0.669 | 0.536 | 0.680 | 0.516 | 0.423 | 0.383 |
| 97.5% quantile | 0.334 | 0.205 | 0.367 | 0.209 | 0.440 | 0.298 | 0.423 | 0.337 |
| Upper quartile | 0.107 | 0.112 | 0.136 | 0.131 | 0.196 | 0.184 | 0.339 | 0.314 |
| Median | 0.042 | 0.075 | 0.070 | 0.095 | 0.114 | 0.137 | 0.310 | 0.295 |
| Lower quartile | 0.006 | 0.037 | 0.031 | 0.065 | 0.053 | 0.093 | 0.260 | 0.264 |
| 2.5% quantile | 0.002 | 0.002 | 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 | 0.074 | 0.077 | 0.099 | 0.099 | 0.137 | 0.142 | 0.283 | 0.279 |

Source: ASCU, 2022.

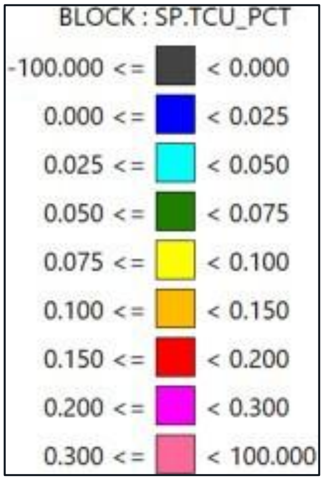
As an independent check on the grade estimates, box plots were created for Tsol mean grades and distributions within each lift to compare against the nearest neighbour.

Figure 14-62 shows the box plots and confirms similar mean grades and smoothed distributions in line with the composite distributions.

### 14.2.13.2 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 14-63 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 14-64 through Figure 14-69. 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.

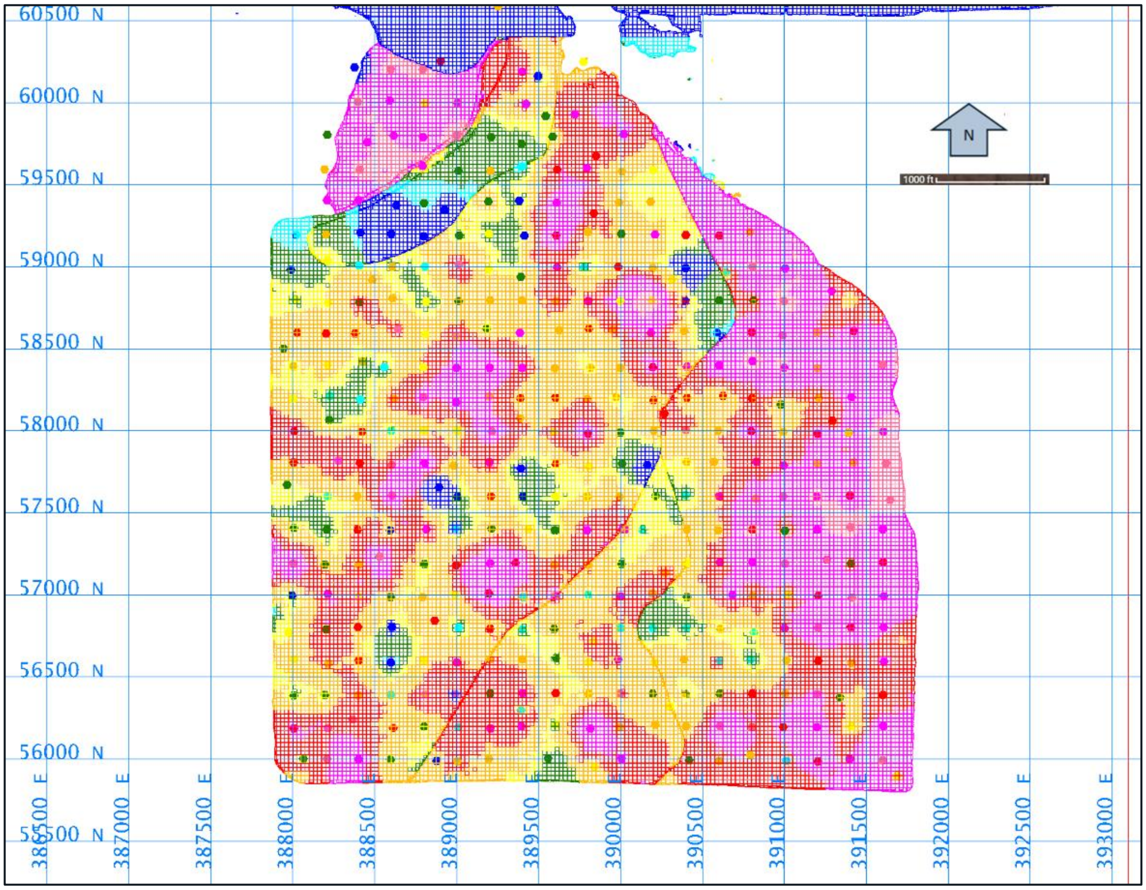
**Figure 14-63: Legend for all Copper Grade Sections**




Source: ASCU, 2022.

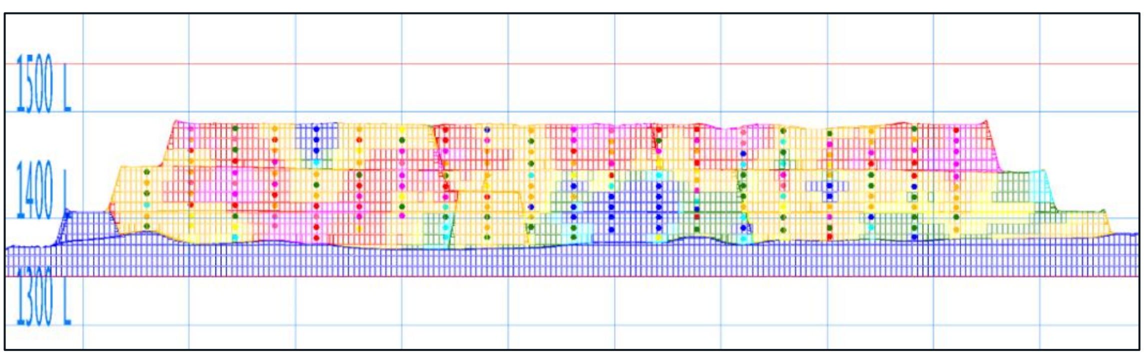
As an independent check on the grade estimates, a visual comparison of block grades to composite grades was also performed for the Tsol grades (see Figure 14-70 and Figure 14-71). Examination confirms appropriate agreement and that overall grade trends are represented as planned.

**Figure 14-64: Plan View Lift 3 (1455) for CuT Grade Comparing Blocks to Sample Composites**



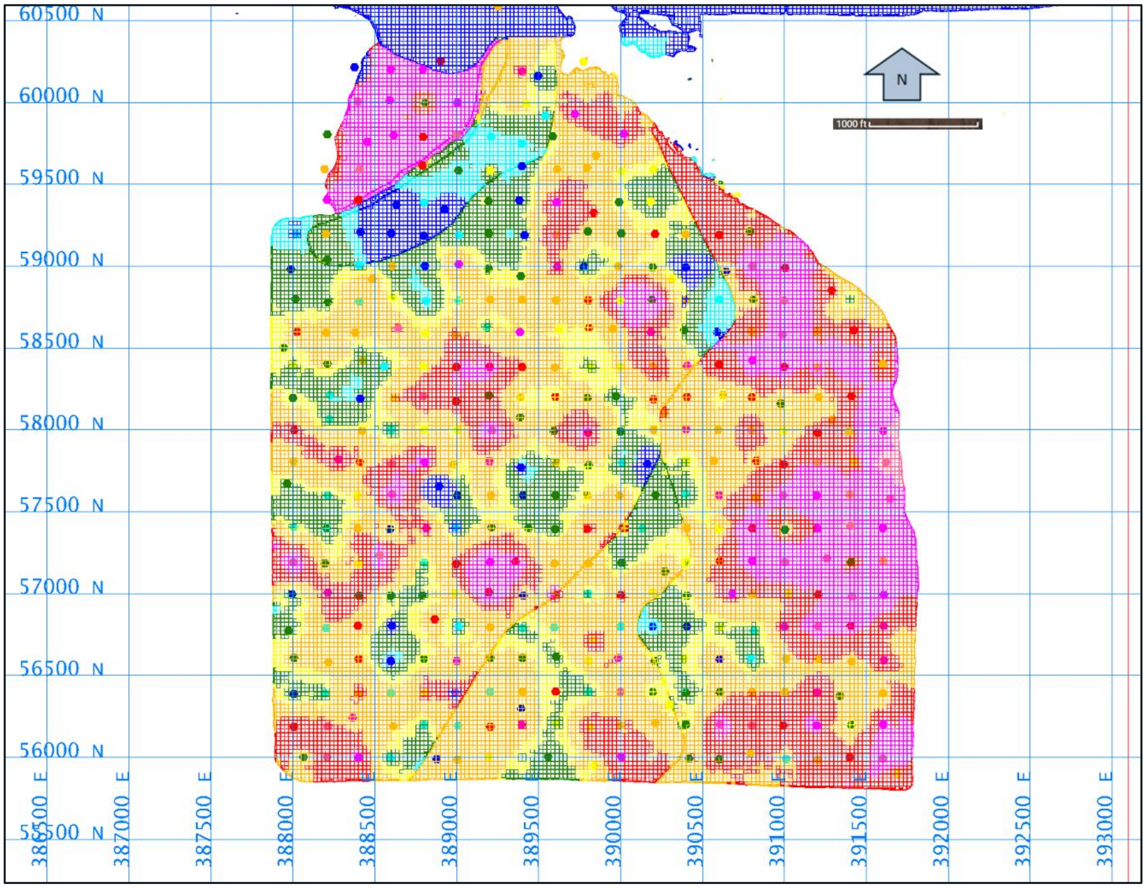

Note: Clipping is 5 ft either side of the section. Source: ASCU, 2022.

**Figure 14-65: Cross Section view (56600N) for CuT Grade Comparing Blocks to Sample Composites**



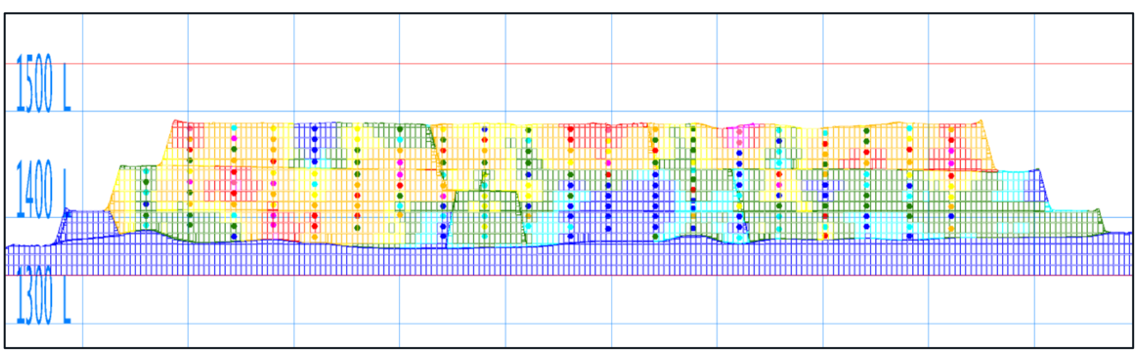

Note: Clipping is 75 ft either side of the section. Vertical exaggeration is set to 500. Source: ASCU, 2022.

**Figure 14-66: Plan View Lift 3 (1455) for CuAS Grade Comparing Blocks to Sample Composites**



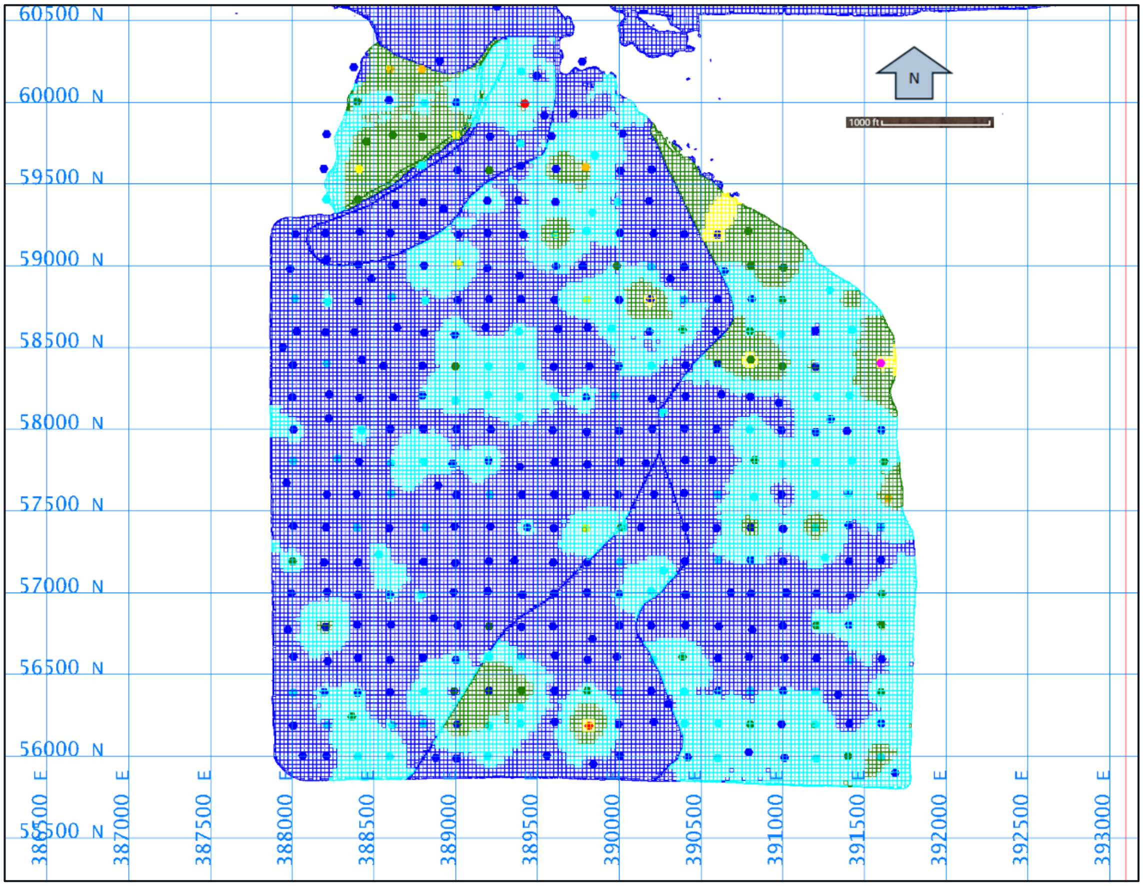

Note: Clipping is 5 ft either side of the section. Source: ASCU, 2022.

**Figure 14-67: Cross Section View (56600N) for CuAS Grade Comparing Blocks to Sample Composites**



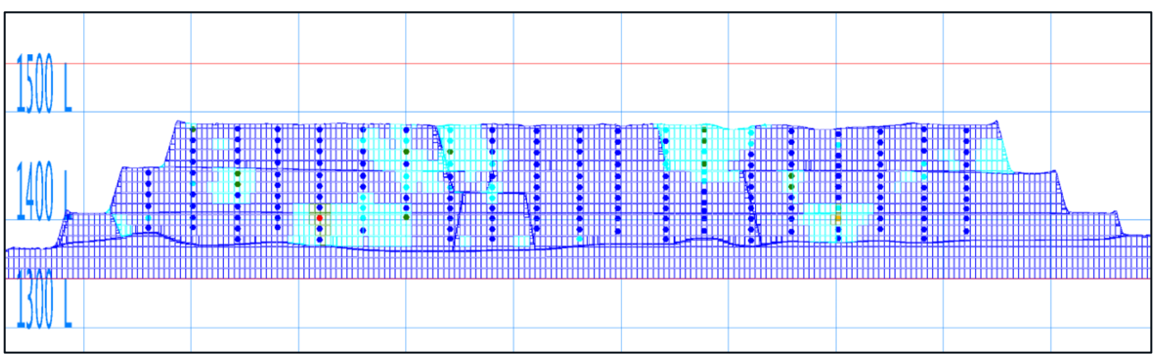

Note: Clipping is 75 ft either side of the section. Vertical exaggeration is set to 500. Source: ASCU, 2022.

**Figure 14-68: Plan View Lift 3 (1455) for CuCN Grade Comparing Blocks to Sample Composites**



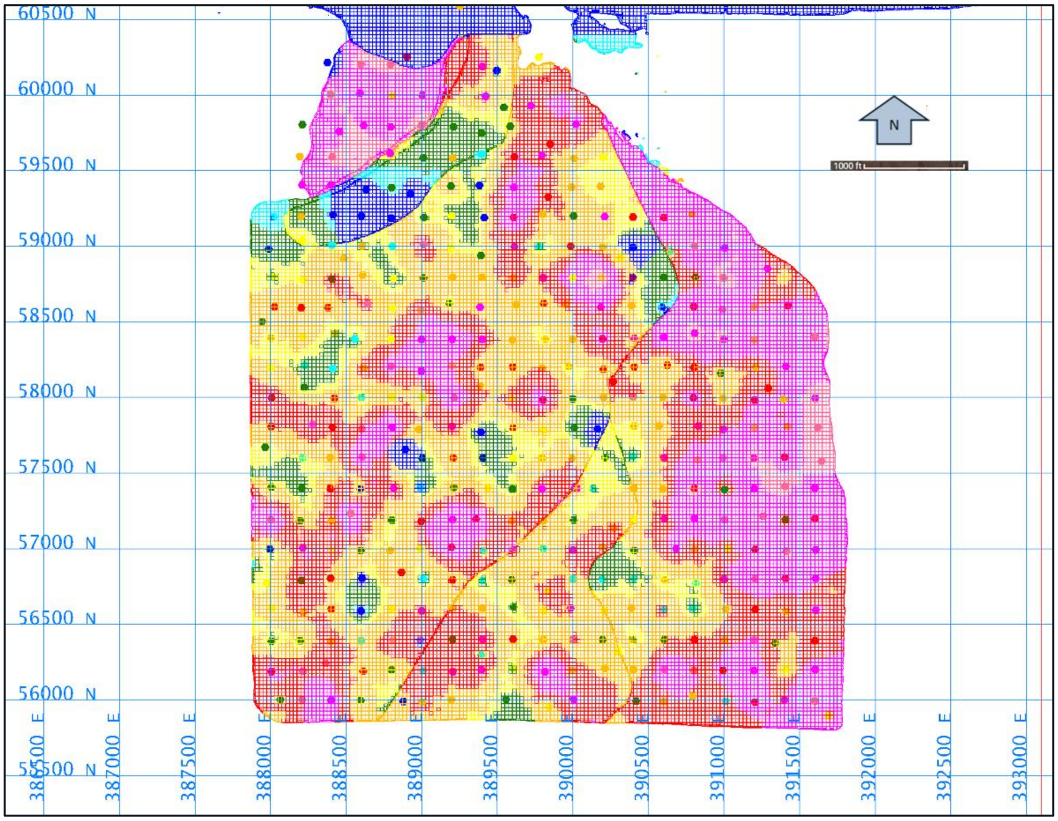

Note: Clipping is 5 ft either side of the section. Source: ASCU, 2022.

**Figure 14-69: Cross Section View (56600N) for CuCN Grade Comparing Blocks to Sample Composites**



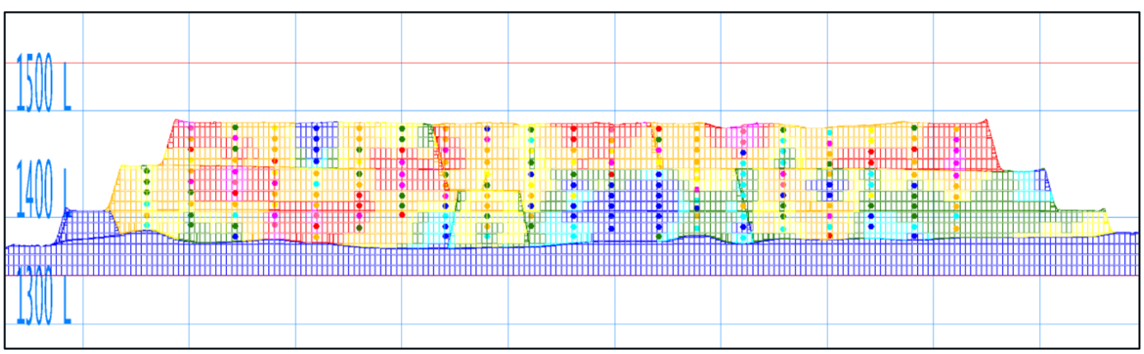

Note: Clipping is 75 ft either side of the section. Vertical exaggeration is set to 500. Source: ASCU, 2022.

**Figure 14-70: Plan View Lift 3 (1455) for Tsol Grade Comparing Blocks to Sample Composites**




Note: Clipping is 5 ft either side of the section. Source: ASCU, 2022.

**Figure 14-71: Cross Section View (56600N) for Tsol Grade Comparing Blocks to Sample Composites**



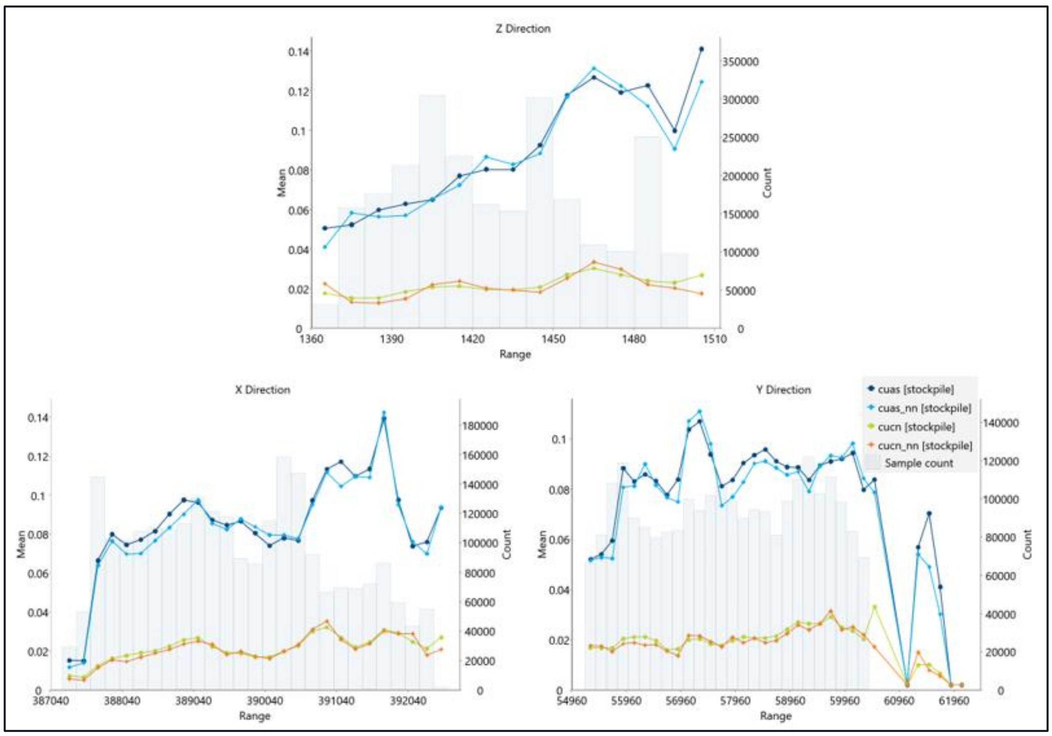

Note: Clipping is 75 ft either side of the section. Vertical exaggeration is set to 500. Source: ASCU, 2022.

#### 14.2.13.3 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 neighbour model. As an independent check on the estimates, swath plots were also generated for Tsol.

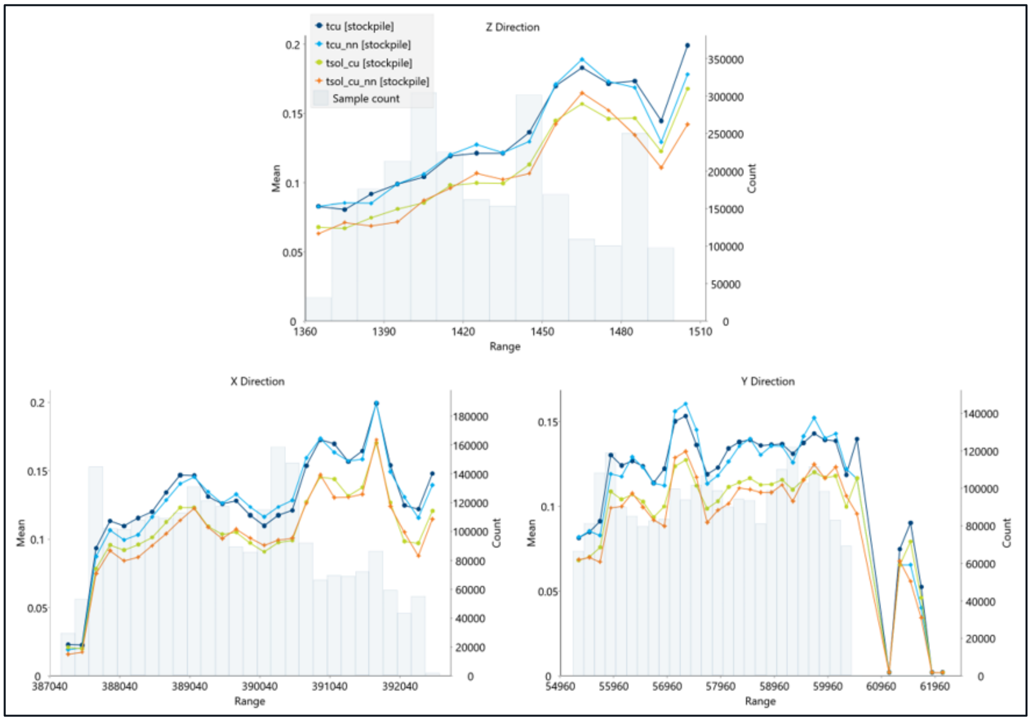
Comparisons for CuAS and CuCN are shown in Figure 14-72. Comparisons for CuT and Tsol are shown in Figure 14-73.

**Figure 14-72: Swath Plots through Cactus West Comparison with Associated Nearest Neighbour Grade Trends**




Source: ASCU, 2022.

**Figure 14-73: Swath Plots through the Cactus Stockpile Project with Associated Nearest Neighbour Grade Trends**




Source: ASCU, 2022.

### 14.2.14 Resource Classification

The drill spacing for the Cactus Stockpile Project has been reduced from approximately 750 ft (229 m) to 200 ft (61 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 has been assigned an Indicated status. Of particular note is that through the process of significantly reducing the drill spacing and significantly increasing the number of drill holes, there has been little change to the grade tonnage curve and global resource from that previously reported in 2020 based on the 750 ft (229 m) drill spacing.

## 14.3 Resource Reporting

### 14.3.1 Resource Cutoff Grades

To meet a Reasonable Prospects for Eventual Economic Extraction (RPEEE) requirement, as stated in CIM 2019 Best Practices, Cutoff Grades (CoGs) were applied to a potential open pit across the Cactus deposit, a potential underground mine at depth in Cactus East, and a potential underground mine at Parks/Salyer. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

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 sulphides in the primary material were based on CuT assays. High- level cost analysis for the Cactus open pit suggested CoGs of 0.099% Tsol for the oxides, and 0.092% Tsol for the enriched material. A cutoff of 0.226% 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.

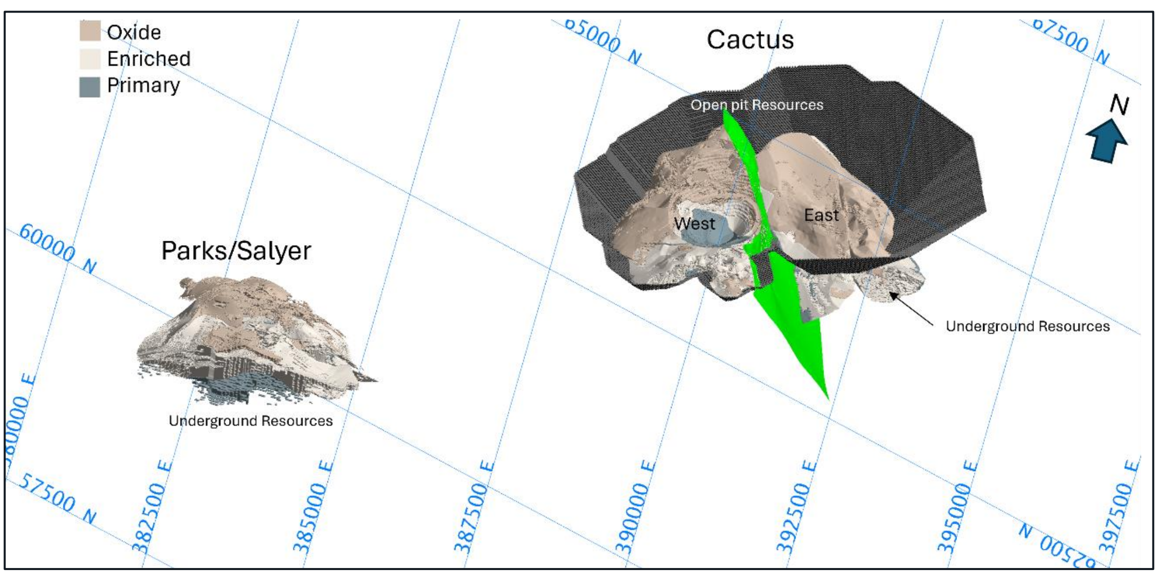
Additional mineral resources outside of the Whittle pit in Cactus East have the potential to be amenable to underground mining. High-level analysis of the material yielded cutoffs of 0.549% Tsol for the oxides and 0.522% Tsol for the enriched. The primary had a 0.691% cutoff applied to the CuT grade for potential recovery in a flotation mill.

Mineral resources for Parks/Salyer were also determined based on its amenability to underground mining and vary due to applicable royalties. High-level analysis of the material yielded cutoffs of 0.549% Tsol for the oxides, 0.522% Tsol for the enriched and for primary on ASCU owned land. Cutoffs of 0.545% Tsol for the oxides. 0.518% Tsol for the enriched, and 0.686% for primary apply to the state land. Cutoffs of 0.532% Tsol for the oxides. 0.505% Tsol for the enriched, and 0.669% for primary apply to the MainSpring property for which no royalties apply. l.

Stockpile Project mineral resources were defined using a CoG of 0.095% Tsol.

Figure 14-74 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. The Sacaton fault which offsets the Cactus West and Cactus East ore bodies is defined in green.

**Figure 14-74: Oblique Image Displaying Open Pit and Underground Resources for Cactus West, Cactus East, and Parks/Salyer and Material Types**



Source: ASCU, 2022.

### 14.3.2 Resource Tables

Table 14-22 details the breakdown of resources for Cactus West and Cactus East by mineral zone and classification within the Whittle pit. Table 14-23 through Table 14-25 have the same breakdown for the potential underground mineral resources for Cactus East and Parks/Salyer. Table 14-26 shows the combined total of the two previous tables.

**Table 14-22 Cactus West and Cactus East Open Pit Measured, Indicated, and Inferred Resource**

| Cactus O/P Resource | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons (kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| **MEASURED** | | | | |
| Oxide | 200 | | 0.137 | 500 |
| Enriched | 8,900 | | 0.232 | 41,400 |
| Primary | 1,300 | 0.315 | | 8,000 |
| **Total Measured** | **10,400** | **0.241** | | **49,800** |
| **INDICATED** | | | | |
| Oxide | 72,300 | | 0.346 | 500,300 |
| Enriched | 66,000 | | 0.631 | 831,800 |
| Primary | 72,000 | 0.339 | | 486,000 |
| **Total Indicated** | **210,300** | **0.432** | | **1,818,100** |
| **M&I** | | | | |
| Oxide | 72,500 | | 0.346 | 500,800 |
| Enriched | 74,900 | | 0.583 | 873,200 |
| Primary | 73,000 | 0.339 | | 494,000 |
| **Total M&I** | **220,300** | **0.468** | | **1,868,000** |
| **INFERRED** | | | | |
| Oxide | 32,600 | | 0.320 | 208,400 |
| Enriched | 24,900 | | 0.308 | 153,400 |
| Primary | 120,400 | 0.335 | | 806,900 |
| **Total Inferred** | **177,900** | **0.328** | | **1,168,700** |

Note: Refer to Table 14-23 for applicable notes to the open pit resource parameters and assumptions. Totals may not add up due to rounding.

**Table 14-23: Cactus East Underground Indicated and Inferred Resource**

| Cactus U/G Resource | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons (kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| **INDICATED** | | | | |
| Oxide | 1,000 | | 0.773 | 15,400 |

| Cactus U/G Resource | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons (kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| Enriched | 8,000 | | 0.906 | 145,600 |
| Primary | 1,300 | 0.816 | | 21,600 |
| **Total Indicated** | **10,400** | **0.882** | | **182,600** |
| **INFERRED** | | | | |
| Oxide | 400 | | 0.740 | 6,500 |
| Enriched | 4,100 | | 0.770 | 63,400 |
| Primary | 1,800 | 0.831 | | 30,200 |
| **Total Inferred** | **6,400** | **0.782** | | **100,100** |

Note: Refer to Table 14-23 for applicable notes to the underground resource parameters and assumptions. Totals may not add up due to rounding.

**Table 14-24: Parks/Salyer Indicated and Inferred Resource**

| Parks/Salyer U/G Resource | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons (kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| **INDICATED** | | | | |
| Oxide | 10,000 | | 0.921 | 183,700 |
| Enriched | 120,200 | | 1.037 | 2,793,000 |
| Primary | 13,800 | 0.833 | | 229,400 |
| **Total Indicated** | **143,900** | **1.117** | | **2,906,100** |
| **INFERRED** | | | | |
| Oxide | 8,700 | | 0.925 | 161,700 |
| Enriched | 35,700 | | 0.996 | 711,500 |
| Primary | 3,900 | 0.797 | | 62,900 |
| **Total Inferred** | **48,400** | **1.033** | | **936,100** |

Note: Refer to Table 14-23 for applicable notes to the underground resource parameters and assumptions. Totals may not add up due to rounding.

**Table 14-25: Cactus Stockpile Project Inferred Resource**

| Cactus Stockpile | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons (kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| **INDICATED** | | | | |
| Oxide | 71,000 | | 0.153 | 217,300 |
| **INFERRED** | | | | |
| Oxide | 1,200 | | 0.127 | 3,000 |

Note: Refer to Table 14-23 for applicable notes to the stockpile resource parameters and assumptions. Totals may not add up due to rounding.

**Table 14-26: Cactus Project Total Measured, Indicated and Inferred Resource**

| Total Resources | | | | |
|---|---|---|---|---|
| **Material Type** | **ktons(kt)** | **CuT (%)** | **Tsol (%)** | **Contained Cu (k lbs)** |
| **MEASURED** | | | | |
| Total Leachable | 9,100 | | 0.230 | 41,900 |
| Total Primary | 1,300 | 0.315 | | 8,000 |
| **Total Measured** | **10,400** | **0.241** | | **49,800** |
| **INDICATED** | | | | |
| Total Leachable | 348,500 | | 0629 | 4,387,200 |
| Total Primary | 86,800 | 0.425 | | 737,000 |
| **Total Indicated** | **435,300** | **0.588** | | **5,124,200** |
| **M&I** | | | | |
| Total Leachable | 357,600 | | 0.619 | 4,429,000 |
| Total Primary | 88,000 | 0.423 | | 745,000 |
| **Total M&I** | **445,700** | **0.580** | | **5,174,000** |
| **INFERRED** | | | | |
| Total Leachable | 107,700 | | 0.607 | 1,307,900 |
| Total Primary | 126,200 | 0.357 | | 900,000 |
| **Total Inferred** | **233,800** | **0.472** | | **2,207,900** |

Notes:

1. Leachable copper grades are reported using sequential assaying to calculate the soluble copper grade. Primary copper grades are reported as total copper, Total category grades reported as weighted average copper grades of soluble copper grades for leachable material and total copper grades for primary material. Tons are reported as short tons.
2. Stockpile resource estimates have an effective date of 1 March 2022, Cactus resource estimates have an effective date of 29th April 2022, Parks/Salyer resource estimates have an effective date of 19th May 2023. All resources use a copper price of US$3.75/lb.
3. Technical and economic parameters defining resource pit shell: mining cost US$2.43/t; G&A US$0.55/t, 10% dilution, and 44°-46° pit slope angle.
4. Technical and economic parameters defining underground resource: mining cost US$27.62/t, G&A US$0.55/t, and 5% dilution,
5. Technical and economic parameters defining processing: Oxide heap leach (HL) processing cost of US$2.24/t assuming 86.3% recoveries, enriched HL processing cost of US$2.13/t assuming 90.5% recoveries, Primary mill processing cost of US$8.50/t assuming 92% recoveries. HL selling cost of US$0.27/lb; Mill selling cost of US$0.62/lb.
6. Royalties of 3.18% and 2.5% apply to the ASCU properties and state land respectively. No royalties apply to the MainSpring (Parks/Salyer South) property.
7. For Cactus: Variable cutoff grades were reported depending on material type, potential mining method, and potential processing method. Oxide material within resource pit shell = 0.099% Tsol; enriched material within resource pit shell = 0.092% Tsol; primary material within resource pit shell = 0.226% CuT; oxide underground material outside resource pit shell = 0.549% Tsol; enriched underground material outside resource pit shell = 0.522% Tsol; primary underground material outside resource pit shell = 0.691% CuT.
8. For Parks/Salyer: Variable cut-off grades were reported depending on material type, associated potential processing method, and applicable royalties. For ASCU properties - Oxide underground material = 0.549% Tsol; enriched underground material = 0.522% Tsol; primary underground material = 0.691% CuT. For state land property - Oxide underground material = 0.545% Tsol; enriched underground material = 0.518% Tsol; primary underground material = 0.686% CuT. For MainSpring (Parks/Salyer South) properties - Oxide underground material = 0.532% Tsol; enriched underground material = 0.505% Tsol; primary underground material = 0.669% CuT.
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. Totals may not add up due to rounding.

# 15 MINERAL RESERVE ESTIMATES

## 15.1 Introduction

The mineral reserve estimates for the ASCU Cactus Project were prepared in accordance with the guidelines of NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum definition Standards for Mineral Resources and Mineral Reserves (CIM Standards).

The mineral reserve estimates are based on the conversion of Measured and Indicated resources from two surface mining sources (Cactus West and Historic Stockpile) and two underground sublevel caving mines (Cactus East and Parks/Salyer). Mineral reserves are reported from engineered mine designs and the LOM plan. A total proven and probable mineral reserve of 276 Mton @ 0.549 Tcu%, containing 3,031 M lbs Cu has been scheduled. The mineral reserves for the Cactus West pit, Historic Stockpile, and the Cactus East and Parks/Salyer underground are presented in Table 15-5.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves.

## 15.2 Open Pit

### 15.2.1 Geotechnical Considerations

The geotechnical investigation for the Cactus West Pit has been provided by Call & Nicholas Inc. (CNI). The open pit consists of three primary material types which include Alluvium, Gila Conglomerate, and Oracle Granite. The Alluvium and Gila Conglomerate have an inter ramp slope angle of 50 degrees with a 27-foot safety bench placed at every 60 ft of bench height. The Oracle Granite contains four domain sectors, North, Northeast, Northwest and South. The Northeast and Northwest sectors have an inter-ramp slope angle of 47 degrees, while the North and South sectors have an inter-ramp slope angle of 45 degrees. All sectors in the Oracle Granite include a 27-foot safety bench placed at every 60 ft of bench height.

### 15.2.2 Economic Pit Shell Development

Ultimate pit designs are based on pit shells generated using the Lerchs-Grossman method in Datamine's Studio NPVS software. The parameters for the pit shells are shown in Table 15-1.

A series of pit optimization shells were produced from a range of copper prices to produce an industry standard pit-by-pit graph. The optimization shells were run at $0.10/lb copper price increments with $3.70/lb copper used as the base price economics. Pit shells were generated up to a $5.00/lb copper price. The Cactus West ultimate pit design is based on the $2.90/copper price shell with an internal phase designed using the $2.00/lb copper shell. Only Measured and Indicated blocks were considered as potential ore and assigned a recovery during pit optimization.

**Table 15-1: Open Pit Design Parameters**

| Description | Units | Value |
|---|---|---|
| **Resource Model** | | |
| Block classification used | | M+I |
| **Metal Prices** | | |
| Price | $/lb | 3.70 |
| Royalty | % | 2.54 |
| **SX/EW, Selling costs** | | |
| Product Grade | | LME cathode |
| Payable | % | 100 |
| Selling Costs | $/lb | 0.04 |
| SX-EW Refining Cost | $/lb | 0.23 |
| **Cost Information** | | |
| Mining Cost * | | Ore / Waste |
| Mining Cost base rate – 1400' elevation | $/ton | 2.12 / 2.62 |
| Incremental rate - above | $/ton/20' bench | 0 |
| Incremental rate - below | $/ton/20' bench | 0.0127 |
| **Processing Costs include Base Cost + Net Acid Consumption Cost**** | | |
| Processing Base Cost – Tertiary Crush – Oxide/Leached | $/ton leach | 0.48 |
| Processing Base Cost – Tertiary Crush – Enriched | $/ton leach | 1.20 |
| Processing Base Cost – Primary Crush – Oxide/Leached | $/ton leach | 0.48 |
| Processing Base Cost – Primary Crush – Enriched | $/ton leach | 1.20 |
| Acid Cost | $/lb acid | 0.08 |
| **Metallurgical Information – Leach Cap and Oxide Alteration Zones** | | |
| Recovery of Acid-Soluble Copper – Primary Crush | % | 82 |
| Recovery of Cyanide-Soluble Copper – Primary Crush | % | 41 |
| Recovery of Acid-Soluble Copper – Tertiary Crush | % | 91 |
| Recovery of Cyanide-Soluble Copper – Tertiary Crush | % | 55 |
| Bulk Acid Consumption | acid lbs./ton ore | 22 |
| Acid Credit lbs acid produced/lb copper recovered | factor | 1.54 |
| **Metallurgical Information – Enriched Alteration Zones** | | |
| Recovery of Acid-Soluble Copper – Primary Crush | % | 85 |
| Recovery of Cyanide-Soluble Copper – Primary Crush | % | 74 |
| Recovery of Acid-Soluble Copper – Tertiary Crush | % | 94.3 |
| Recovery of Cyanide-Soluble Copper – Tertiary Crush | % | 89.6 |
| Bulk Acid Consumption | acid lbs/ton ore | 21 |
| Acid Credit lbs acid produced/lb copper recovered | factor | 1.54 |
| **General and Administrative Cost** | | |
| G&A cost | $/ton leach | 0.47 |

Note: *mining costs based on using 100 t haul trucks. **process costs based on 22 Mt/y dry throughput.
Source: AGP, 2023.

### 15.2.3 Cutoff Grade

Cutoff grade decisions for Cactus West and Historic Stockpile are based on a block value calculation in the mine schedule, which is effectively a net-smelter return with expected processing, G/A, and royalty cost removed. The cutoff block value employed was a marginal cut-off grade of $0/t, meaning that any block which would generate a net positive value was either processed on the heap leach or placed into stockpiles. This cutoff calculation does not include mining costs which are considered sunk for the purpose of cutoff grade determination.

### 15.2.4 Dilution

No dilution is applied to the Cactus West and the Stockpile Mining. The Mineral Resource block model for the Cactus West deposit has block dimensions of 20’ x 20’ x 20’ and the nature of the ore body and cutoff grades is such that ore-waste contacts are relatively infrequent and gradational in nature. It was determined that the Mineral Resource block model is suitable for use in stating Mineral Reserves without any secondary factors for ore loss or dilution applied. In the Stockpile area, there is a very low ratio of internal waste in the model, and as such it was similarly determined that no secondary dilution or ore loss was required.

### 15.2.5 Mine Design

The Cactus West pit and historic stockpile each provide 12 Mt/y of ore to the leach pad (24 Mt/y total) from Year 1 through Year 6. During preproduction, 3.0 Mt of historic stockpile and 0.5 Mt of Cactus West ore is placed on the leach pad. The Cactus West pit is mined in two phases while the stockpile is mined in a sequence from east to south to west in order to facilitate construction of the heap leach facility.

The detailed designs for the Cactus West pit are based on wall slope parameters received from CNI in September 2023. Equipment sizing for ramps and working benches is based on the use of 150-ton rigid frame trucks. The road width is 100 ft with a maximum ramp gradient of 10%. Working benches are designed at 20 ft with placement of a 27-ft safety berm for every 60 ft of bench height.

A total of 217.9 Mton of material is mined from the Cactus West pit, including 75.5 Mton of proven and probable leach ore at a 0.307% total copper grade and a strip ratio of 1.9:1. The Stockpile Project mines 76.8 Mton of probable ore at a 0.163% total copper grade along with 5.5 Mton of waste for a strip ratio of 0.1:1.

## 15.3 Underground

### 15.3.1 Estimation Procedure

The underground reserves for the SLC mines consider the mixing of Indicated resources with dilution from low-grade and barren material originating from within the sublevel cave outline and from overlying material. Estimations were derived from flow modelling simulations using the Power Geotechnical Cellular Automata (PGCA) program. Inferred resources above the mining shapes that contribute to mixing and therefore dilution have been assigned zero grade in the flow model.

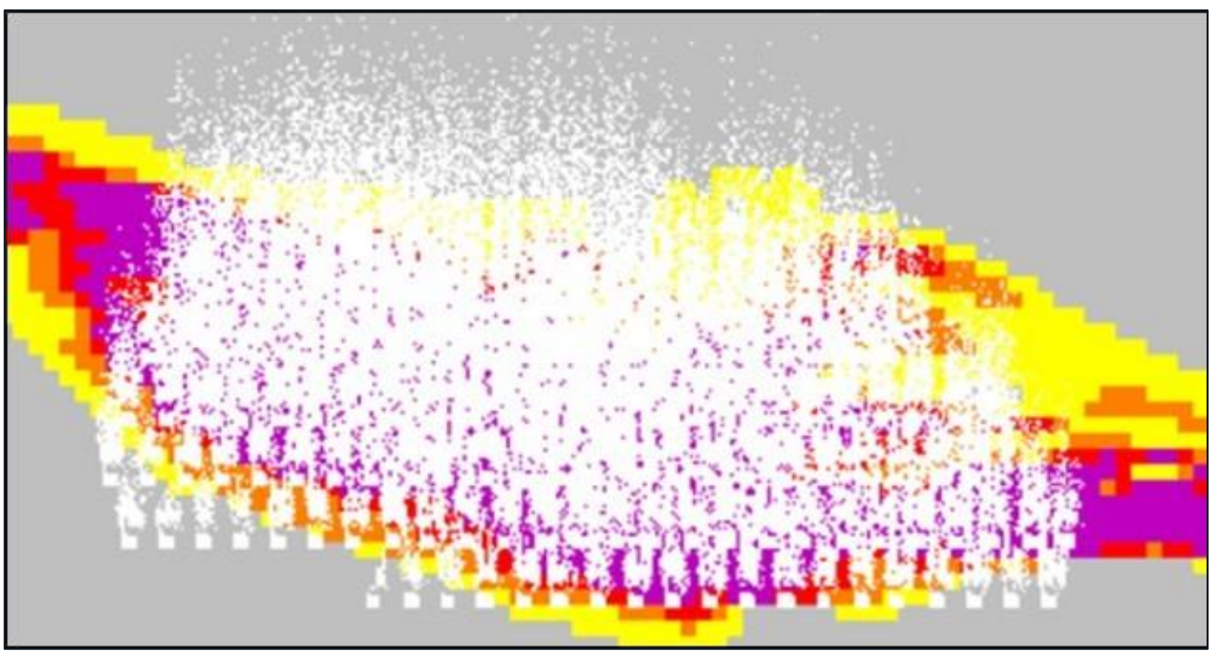
PGCA is a stochastic cellular or 'probabilistic cellular automata' (PCA) model which defines discrete blocks representing broken rock. In PGCA the cells are defined per material states, such as void, static solids, or movable rock (and when required, different classes of movable rock, e.g., fines, rock type A, rock type B etc.) The rules defining movement of individual blocks can be driven by several variables, some of which are user defined, including:

- relative particle location
- material properties
- caving behaviour
- blasting behaviour
- fragmentation

The PGCA block model has been developed from the Reserve block model used for the SLC delimitation and development interrogation. The PGCA block model is a regularised version (5 m x 5 m x 5 m) of the main block model. The regularisation covers all the parameters for each block cell (rescat, lithology, alteration, grades, etc.). The block model is then subdivided into smaller block cells, (multiple of 1.25) before the flowing/mixing takes place (Figure 15-1).

The flow models for the Cactus East and Parks/Salyer reserves models assume there will be uniform flow characteristics for the different lithologies, however, all geotechnical, constraints and recommendations provided by CNI, and mining sequence (principal stress direction and SLC mining guidelines) were considered on the development of the flow model.

**Figure 15-1: Illustration of PGCA Flow Model in Cactus East**



Source: ASCU, 2022.

PGCA has been used in numerous SLC operations including Ernest Henry, Northparkes, Ridgeway, Carrapateena, Finch, Ekati and Black Rock mines and is an industry accepted program for estimation of ore reserves, mine sequencing and scheduling. PGCA model calibrations against mill reconciled production sources has been reported to be very good at several operational mines.

### 15.3.2 Cutoff Grade

SLC footprints were designed to a shutoff dollar value (CFTC1) of $27.62, however, a minor quantity of subeconomic material was incorporated into the upper levels in Cactus East to establish mineable shapes for SLC mining and to accommodate the shallow plunge of the orebody. Drawpoints were shut-off when the grade value fell below a CFTC1 of $27.62 following the necessary removal of swell material within the footprint regardless of grade.

### 15.3.3 Dilution

Sub-level caving is a non-selective bulk mining method where subeconomic material (dilution) is mixed into the flow modelling process and accepted to recover the blasted ore rings. Dilution from production activities is quantified through cave flow modelling and is included in the reported Ore Reserves.

The draw rates have been restricted in the upper three sublevels during SLC startup or where significant step-out distances are required from one sublevel to the next. This strategy enables a blasted “ore” blanket to be created above the production horizon which will delay and limit dilution entry from the overburden. The restricted draw rates will also help to control caveability and reduce air blast risk during startup.

The extraction ratio, or total ore recovered from SLC rings within the mining outlines, is estimated to be 84.7% for Parks Salyer and 86% for Cactus East (Table 15-2). Dilution is derived from the mixing process in the flow model. The overall total dilution (internal and external) for SLC rings valued >$27.62 is estimated to be 11.1% for Parks/Salyer and 11.2% for Cactus East.

**Table 15-2: PGCA SLC Extraction Ratio and Dilution Bins**

| Extraction Ratio | | |
|---|---|---|
| | **Parks Salyer (%)** | **Cactus East (%)** |
| Ore - > $27.62 Extraction Ratio | 84.7 | 86.0 |
| **Dilution Bins** | | |
| Ore - > $27.62 Overall Internal and External Dilution | 11.1 | 11.2 |
| Marginal Ore Dilution > $19.31 | 5.8 | 6.0 |
| Mineralised Waste External Dilution > $3.71 | 1.2 | 1.0 |

### 15.3.4 Underground Modifying Factors

Modifying factors for the SLC mine are based on the geotechnical constraints for mine design and sequencing provided by CNI (Table 15-3). The draw cone dimensions have been assigned a maximum draw width of 8.5m based on expected fragmentation. All lithologies are assumed to have similar mobility (flow) rates.

**Table 15-3: Sublevel Cave Design Parameters and Modifying Factors**

| Design Parameter | | Recommendation |
|---|---|---|
| Sublevel Vertical Spacing | PS | 65 – 80 ft |
| | CE | 80 ft |
| Sublevel Horizontal Spacing – Centerline to Centerline (ft) | | 45 ft |
| Sublevel Drift Width (ft) | | 16.5 ft |
| Distance from Nearest Brow to Ramp Access (ft) | PS | 120 ft |
| | CE | 100 ft |
| Vertical Echelon (Horizontal Distance between Vertical Faces on Adjacent Sublevels) | | > 50 ft |
| Horizontal Echelon (Horizontal Distance between Vertical Faces on Same Sublevel) | | 2 – 8 Burden Rings |
| Hydraulic Radius (m) for Caving | PS | 18 m |
| | CE | 21 m |
| Retreat Direction (Azimuth in Deg.) | PS | 180° |
| | CE | 335° |
| Max Panel Width (ft) | | 800 ft |
| Panel Transition Zone Thickness (ft) Buffer zones between SLC panels removed from Reserve | | 74 – 119 ft |
| Subsidence Limits (Composite Angle in Deg.) | | 65° |
| Startup draw rates including for step-out areas | | First Level ~40% (swell only)<br>Second Level ~60%<br>Third level ~ 100% |
| Maximum draw cone width in flow model for PS and CE | | 8.5 m |
| Mobility (flow) factor for Lithologies | | Uniform |

## 15.4 Mineral Reserve Statement

The mineral reserves are derived from the Measured and Indicated mineral resources prepared by ALS Geo Resources. Measured and Indicated resources for the underground operations were converted to Proven and Probable Reserves based on design guidelines and applicable modifying factors.

The total reserves for the Cactus Project are shown in Table 15-4. Some variation may exist due to rounding.

Table 15-4: Mineral Reserve Inventory

| Metal | Unit | Cactus West Open Pit | Stockpile Open Pit | Cactus East Underground | Parks/Salyer Underground | Total |
|---|---|---|---|---|---|---|
| Proven | Tons | 3,600,000 | | | | 3,600,000 |
| | TCU (%) | 0.249 | | | | 0.249 |
| | CU-AS (%) | 0.052 | | | | 0.052 |
| | CU-CN (%) | 0.173 | | | | 0.173 |
| | Cu (M lbs) | 17.9 | | | | 17.9 |
| Probable | Tons | 71,921,000 | 76,777,000 | 27,739,000 | 96,248,000 | 272,686,000 |
| | TCU (%) | 0.310 | 0.163 | 0.950 | 0.930 | 0.552 |
| | CU-AS (%) | 0.138 | 0.112 | 0.333 | 0.110 | 0.141 |
| | CU-CN (%) | 0.122 | 0.024 | 0.552 | 0.710 | 0.346 |
| | Cu (M lbs) | 445.4 | 251.0 | 527.0 | 1,789.7 | 3,013.0 |
| Proven + Probable | Tons | 75,521,000 | 76,777,000 | 27,739,000 | 96,248,000 | 276,286,000 |
| | TCU (%) | 0.307 | 0.163 | 0.950 | 0.930 | 0.549 |
| | CU-AS (%) | 0.134 | 0.112 | 0.333 | 0.110 | 0.140 |
| | CU-CN (%) | 0.125 | 0.024 | 0.552 | 0.710 | 0.344 |
| | Cu (M lbs) | 463.3 | 251.0 | 527.0 | 1,789.7 | 3,031.0 |

Notes to accompany Reserves table:

1. Mineral Reserves have an effective date of November 10, 2023. The Qualified Person for the underground estimates of Cactus East and Parks/Salyer is Nat Burgio of AGP Mining Consultants Inc. The Qualified Person for the open pit estimates of Cactus West and Stockpile is Gordon Zurowski of AGP Mining Consultants Inc.
2. The Mineral Reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves.
3. The Mineral Reserves are supported by a combined open pit and underground mine plan, based on open pit and underground designs and schedules, guided by relevant optimization procedures. Inputs to that process are:
   - Metal prices of Cu $3.70/lb.
   - Processing costs which are variable and based upon material type, processing destination, copper grade, and copper recovery. Processing costs include a fixed unit cost component, a net acid consumption cost, and a cost for refining and selling copper cathode.
   - General and administration cost of $0.47/t processed.
   - Royalty cost of 2.5% for Parks/Salyer and 2.54% for Cactus/Stockpile Ores.
   - Process recoveries which are variable depending upon mineralization type, sequential copper grades, and comminution size.
   - Open pit geotechnical design criteria from Call and Nicholas.
   - Underground geotechnical design criteria from Call and Nicholas.
   - Open pit mining costs including an escalation factor with pit depth.
   - Underground mining costs of $27.62.
4. The footprint delineations for the Cactus East and Park Slayer mines were based on a resource model block cash flow dollar value (CFTC1) of $27.62 (net of process, G/A and royalties). Drawpoints were shut-off when the grade value fell below a CFTC1 of $27.62 following the necessary removal of swell material within the footprint.
5. Dilution and mining loss adjustments are incorporated into the underground mining inventories by way of cave flow modelling software. Inferred resources included in the mixing process have been assigned zero grade. No allowance for mining dilution or ore loss has been provided in the open pit mining inventories.
6. Ore/Waste delineation in open pit areas was based on a Block Value cutoff of $0/t considering metal prices, recoveries, royalties, process, and G&A costs as per LG shell parameters stated above. Ore/Waste delineation in the underground was similarly applied with a $0/t cutoff for a material mined from the cave, though mine targeting employed a higher optimization cut-off reflecting the expected mining costs.
7. The life-of-mine (LOM) stripping ratio in tonnes is 0.97:1 for open pit mining areas.
8. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

## 15.5 Factors that May Affect the Mineral Reserves

### 15.5.1 Underground Geotechnical Factors

The underground mine design parameters are based on geotechnical recommendations and guidelines provided by CNI based on the information available (refer Section 16.2). The factors that could impact on the Mineral Reserves include:

- Ore recoveries, at the drawpoint, assume the mine design and ground support measures are adequate to maintain the stability of the production drives and brows to enable production to proceed as scheduled. Note that the Parks/Salyer deposit will have some of the weakest rock mass conditions that will be developed by the SLC method. Poorer than expected ground conditions, either due to faults or weak rock strengths, could hamper the ability to maintain brow stability in some areas. Increased ground support measures have been assigned to zones of poorer ground based on geotechnical information, however, there remain some uncertainties in the expected behaviour of the rock mass response due to the absence of a detailed structural model. Further data collection and geotechnical modelling is required to define potential problem areas, consider appropriate mitigating measures, and adjust the modifying factors accordingly.
- The PGCA flow models assume uniform flow behaviour for the different rock types and overburden. Changes in fragmentation could, however, result in differential flow which may impact ore recoveries and dilution. For example,

  (1) The restricted draw rates for the upper SLC levels will help delay and limit dilution ingress of the overburden. This is expected to buffer the impacts of differential draw during the early stages of mining.

  (2) Coarser fragmentation is possible from the siliceous cap rocks towards upper sections of the enriched zone at Parks Salyer could assist in minimising dilution ingress further than current estimates

  (3) The overburden waste material (Gila conglomerates) will be exposed in the initial development across the southwest corner of the Cactus East footprint. The presence of pure waste immediately adjacent to the production front will need closely monitored and controlled to avoid excessive dilution in this area.

The sensitivity of the Ore Reserves due to differential draw behaviour will require further review to evaluate the possible range of outcomes.

- The reserves reported for sublevel caving are based on what can be recovered from the blasted rings on each sublevel. This largely depends on the flow behaviour of the material within the draw zones above a draw point. The size and shape of the draw zone depends on a range of factors including fragmentation, design layout and spacings, rates of draw and mine sequencing strategies. In general, finer fragmentation results in smaller draw-cone dimensions. Preliminary analysis suggests that the impact on ore recovery due to changes in draw size are expected to be low assuming conditions remain dry. The base case assumes a maximum draw diameter of 28 ft (8.5m) for both Cactus East and Parks/Salyer. A sensitivity analysis using draw diameters of 20 ft (6.0m) and 36 ft (11.0 m) indicates that the SLC drive spacings for Cactus East can accommodate variability in fragmentation without significant impact on potential recoveries (Table 15-5). A 20-ft (6.0 m) case was also assessed for Parks/Salyer which showed negligible impacts according to the PGCA flow model.

**Table 15-5: Impact of Draw Cone Dimensions using PGCA Model**

| | Cactus East | | Parks/Salyer | |
|---|---|---|---|---|
| **Draw Size (m)** | **Tonnes** | **TCu %** | **Tonnes** | **TCu %** |
| 11.0 | 25,762,700 | 0.88 | - | - |
| 8.5 (Base Case) | 24,560,200 | 0.87 | 88,753,900 | 0.91 |
| 6.0 | 23,526,800 | 0.86 | 87,179,800 | 0.90 |

- The proportion of "fines" (<5 cm;<2 in) material is expected to increase over time as a result of comminution within the cave column. This may have implications for dilution entry, ore recovery and mud formation in the long term if the rock mass were saturated by ground or meteoric waters.
- The mine sequencing, particularly at Parks/ Salyer, results in extended periods of inactive draw for some areas of the mining footprint. This could lead to the risk of re-compaction of the caved rock which may impact flow behaviour. The production rates are not expected to be affected if there are sufficient mining fronts available, but recovery may be impacted if zones of stagnant rock are not recoverable. Further work in optimization the draw sequence may be necessary to maintain caving mobility.
- The significant step out distances at Parks/Salyer will generate multiple caving fronts. There is some uncertainty in the behaviour of the rock mass in areas where the caves will merge. Provisions have been made in the schedule to address this issue, i.e., reducing draw rates, creating buffers (temporary pillars), and mass blasting through critical areas. Further studies are required to assess the rock mass response in order to optimise mining schedule and adjust ground support provisions. Ore recoveries and dilution may be impacted by such changes.

Changes in the following factors and assumptions may affect the Mineral Reserve estimate:

- metal prices.
- mining, process and operating costs.
- leach recovery and acid consumption.
- interpretation of grade, mineralization type and continuity of mineralization zones.
- geotechnical and hydrogeological assumptions; and
- ability to obtain and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

# 16 MINING METHODS

## 16.1 Overview Mine Design

The Cactus Project is comprised of the Cactus and Parks/Salyer deposits and an existing surface stockpile (Stockpile Project) of previously mined material. The deposits are planned to be developed using conventional open pit mining and underground mining methods. The surface mining portion of the Project includes Cactus West and the Stockpile. Underground mining methods will be used for the Cactus East and Parks/Salyer deposits.

The Project located at the historic Sacaton Mine is 40 road miles south southeast of the Greater Phoenix metropolitan area and approximately 3 mi (5 km) northwest of the city of Casa Grande in Pinal County, Arizona. The property operated as a conventional open pit copper mine mining the Cactus deposit from 1974 until 1984.

The mine schedule for open pit mining at Cactus West consists of 75.6 Mton of leach ore grading 0.307% CuT over 8 years of mining, including one year of pre-production. The stockpile project has 76.5 Mton of leach ore grading 0.163 % CuT with a mine life of slightly more than 8 years. Initial ore from the Stockpile Project and Cactus West will be placed on the leach pad near the end of pre-production period, Year -1. The Cactus West pit and Stockpile Project will be mined concurrently each providing 12 Mt/y of leach material or 24 Mt/y of leach material combined. An additional 142.4 Mton of waste material is mined in Cactus West, while the stockpile project contains 5.5 Mton of waste.

The sublevel caving method was deemed to offer the best opportunity to maximise the conversion of resources to reserves whilst offering more favourable economic and production capacity outcomes compared to the other underground mining options considered for the Cactus East and Parks/Salyer deposits. Total Probable Reserves amount to 27.7 Mton @ 0.95 CuT% for Cactus East, and 96.2 Mton@ 0.93 CuT% for Parks/Salyer.

The initial Cactus East SLC level will commence 1,345 ft below surface and be comprised of 7 sublevels to a final depth 1,845 ft below surface. Access will be via a single decline with a portal located within the existing Cactus West pit. Ore haulage to surface will be via a vertical conveyor which can be supplemented with truck haulage to surface via the open pit if necessary. Production will continue for 11 years and will peak at 3.9 Mt/y.

The initial Parks/Salyer SLC level will commence 1,120 ft (341 m) below surface and include 11 sublevels to a final depth of 1,930 ft (588 m) below surface. Access to the Parks/Salyer deposit will be via a surface portal and twin declines. One will be dedicated to ore haulage using an inclined conveyor and the other providing access for personnel and equipment. Production will continue for 19 years and will peak at 6.9 Mt/y.

## 16.2 Geotechnical Considerations

CNI performed a geotechnical evaluation in support of a PFS of the Cactus Mine Project located in Pinal County of southern Arizona. The purpose of the study was to characterize ground conditions at the project site based on existing data and new data collected between January and July 2023, and to provide geotechnical design parameters for open pit mining of the Cactus West deposit and sublevel cave mining of the Cactus East and Parks Salyer underground deposits.

### 16.2.1 Dataset

As part of the Project, drilling was conducted and geomechanical logging was performed by CNI engineers and ASCU geologists on the drill core. Laboratory testing was conducted on core samples collected from geotechnical drill holes. The data used as the basis for the geotechnical analysis and recommendations came both from historic data provided by ASCU and from data collected during the study by both CNI and ASCU personnel. The historic dataset provided by ASCU includes:

- basic geotechnical logging (RQD and recovery),
- lithology and mineral domain solids,
- historic rock strength testing data, and
- modeled major faults.

November 2022 Mineral Resource Estimate and Technical Report Data collected by ASCU and CNI during this study includes:

- detailed geotechnical logging (RQD, Q parameters, piece lengths),
- structural core logging from acoustic televiewer surveys,
- structural data collected from mapping of a drone survey point cloud,
- additional rock strength testing, and
- water pressure and temperature data from vibrating wire piezometers installed in three holes in the Parks Salyer area.

Development of geotechnical domains considered lithology, mineralization, alteration, and fault blocks/spatial locations. After review, comparisons, and discussions with ASCU geology staff, it was determined that the mineral domains (leached/oxide, chalcocite enriched, and primary/hypogene) were suitable for use as the basis for geotechnical domains. The overburden package (alluvium and Gila Conglomerate) was separated from the mineral domains as a discrete geotechnical domain.

The Cactus East and Parks Salyer areas are planned for mining using a SLC method. Because the previous drill campaign had limited drilling within these target areas, logging of historical drill cores was performed to supplement the underground geotechnical analyses. However, because drill core from past exploration drilling has already been split and could not be physically logged, geotechnical data for the CE area was logged from core photos. The photo logging focused on collecting data for rock characterization using Barton Lien and Lunde’s Modified Rock Tunneling Quality Index Q’ (1974). Nine historic drill holes totalling 3,340 ft were logged using photos. Photo logging was limited to mineralized intervals.

A detailed geotechnical core drilling program consisting of thirteen holes and totaling 26,110 ft (796 m) was undertaken to obtain geologic, geomechanical, and rock fabric data from open pit and underground targets, as presented in Table 16-1. All holes were planned by ASCU with the goal of improving the resource model. Drilling was conducted by Ruen using standard diamond core drilling methods. CNI and ASCU personnel collected geotechnical data at the Cactus logging facility as the holes were being drilled.

**Table 16-1: 2023 Geotechnical Drilling Summary**

| Drillhole Name | Target Area | Data Collection | Azimuth | Dip | Target Depth (ft) | Detailed Logged Footage (ft) |
|---|---|---|---|---|---|---|
| ECE-143 | Cactus East | GMX/ATV | 92.31° | -80.60 | 2,274 | 1,617 |
| ECE-146 | Cactus East | GMX/ATV | 318.69° | -79.93 | 2,096 | 1,770 |
| ECE-149 | Cactus East | GMX/ATV | 332.66° | -79.57 | 2,000 | 1,900 |
| ECP-129 | Parks Salyer | GMX/ATV | 253.65° | -80.77 | 2,300 | 2,316 |
| ECP-132 | Parks Salyer | GMX/ATV/VWP | 231.23° | -80.87 | 2,430 | 2,314 |
| ECP-135 | Parks Salyer | GMX/ATV | 257.71° | -78.98 | 2,086 | 2,078 |
| ECP-138 | Parks Salyer | GMX/ATV | 114.84° | -81.36 | 2,248 | 2,138 |
| ECP-140 | Parks Salyer | GMX/ATV | 255.68° | -80.9 | 2,333 | 2,234 |
| ECW-150 | Cactus West | GMX/ATV | 42.34° | -66.18 | 2,155 | 2,156 |
| ECW-151 | Cactus West | GMX/ATV | 358.57° | -58.98 | 2,017 | 2,017 |
| ECW-153 | Cactus West | GMX/ATV | 172.82° | -65.52 | 1,844 | 1,769 |
| ECW-154 | Cactus West | GMX/ATV | 178.65° | -66.15 | 1,877 | 1,771 |
| ECW-157 | Cactus West | GMX/ATV | 86.84° | -81.27 | 2,030 | 2,033 |
| **Total Detailed Logged Footage:** | | | | **26,112** | | |

Source: CNI, 2023.

### 16.2.2 Material Properties

Samples collected from the geotechnical core holes were sent for testing at CNI's geomechanics laboratory located in Tucson, Arizona. The purpose of the laboratory testing was to determine strength parameters for use in stability analyses. Laboratory testing was conducted to ASTM standards and included uniaxial compression tests, triaxial compression tests, Brazilian disk tension tests, and small-scale direct-shear tests. A summary of the number of tests is listed in Table 16-2.

**Table 16-2: Material Property Testing**

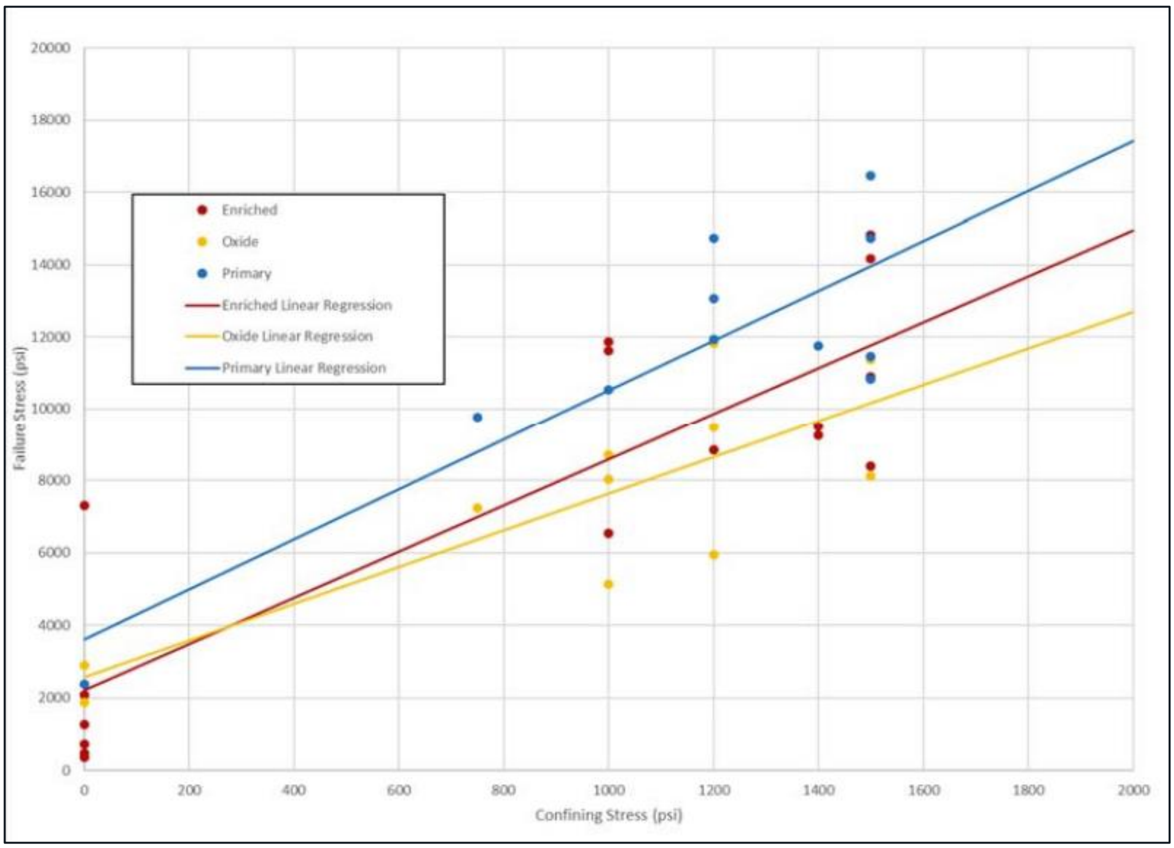
| Testing Method | Number of Tests |
|---|---|
| Uniaxial Compression Test | 32 tests (19 tested in 2022) |
| Triaxial Compression Test | 92 tests |
| Brazilian Disk Tension Test | 98 tests |
| Small-Scale Direct- Results Shear Test | 12 tests |

Source: CNI, 2023.

Some unconfined compressive strength (UCS) samples were instrumented with gauges to measure strain to calculate elastic modulus and Poisson's ratio. Triaxial compression was done at confinement stresses which varied between 200 and 1500 psi and was utilized with uniaxial compression test data to calculate intact shear strength by rock type. Brazilian disk tension tests were conducted on disks cut from the ends of uniaxial and triaxial compression test samples and were used to determine the tensile strength of the sample. Small-scale direct-shear testing was conducted on joint surfaces to determine the shear strength of natural joints.

Most mining is planned in the mineralized domains of the Oracle Granite, and as a result, these mineral domains were the focus of the laboratory testing campaign. Figure 16-1 presents a summary of intact rock strengths based on UCS and triaxial compressive strength testing. While all mineral domains are similar in intact strength, the chalcocite enriched, and primary mineral domains demonstrate slightly superior intact strength.

**Figure 16-1: Intact Strength for Underground Strength Estimation**




Source: CNI, 2023.

Rock-mass strengths (Table 16-3) were evaluated by applying a linear approximation to a Hoek-Brown strength envelope for each target underground mining area by depth using laboratory strength data.

Table 16-3: Rock-Mass Strength Summary

| Mineral Domain | Intact | | Source | GSI / RMR76 | Sig3 (ksf) at 1,000ft depth | Hoek-Brown | | | Rock Mass | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | PHI (Deg.) | Cohesion (psi) | | | | mi | UCS (ksf) | UCS (psi) | PHI (Deg.) | Coh (ksf) | Coh (psi) |
| Oxide | 42.0 | 571 | CE + PS | 38 | 150 | 17.6 | 340.7 | 2366.1 | 27.24 | 24.561 | 170.6 |
| Chalcocite Enriched | 46.8 | 437 | CE + PS | 38 | | 30.9 | 295.8 | 2054.4 | 30.82 | 28.387 | 197.1 |
| Primary | 49.2 | 415 | CE + PS | 38 | | 38.1 | 358.9 | 2492.7 | 34.23 | 32.37 | 224.8 |

Source: CNI, 2023.

For the open pit stability analyses, strength estimates considered test results from the Cactus West area only, except for the chalcocite enriched mineral domain as limited tested existed in the Cactus West area. Intact, fracture and rock mass strengths used in open pit analyses are presented in Table 16-4.

Table 16-4: Open Pit Rock-Mass

| Min Domain | Source | Density (pcf) | Intact | | | Tension (psi) | Fracture | | RQD | | Rock Mass | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | UC$ | PHI (*) | Coh (psi) | | PHI (*) | Coh (psi) | Reliability | RQD | PHI (*) | Coh (psi) | Coh (psf) |
| Oxide-Leached | ECW | 151.94 | 2204 | 44.57 | 461.26 | 303 | 25.12 | 2.06 | 50% | 46.1 | 32.0 | 39.1 | 5630.4 |
| C.Enr. | ECW | 157.39 | 1355 | 44.66* | 247.45* | 363 | 28.64 | 3.44 | 50% | 50.6 | 34.9 | 24.8 | 3571.2 |
| Primary | ECW | 158.85 | 2613 | 38.37 | 631.84 | 185 | 26.77 | 2.31 | 50% | 23 | 29.4 | 29.8 | 4291.2 |
| Gila | ECW | 149.75 | 743 | 52.67 | 125.52 | 73 | 36.00 | 10.0 | - | - | 44.3 | 67.8 | 9757.4 |

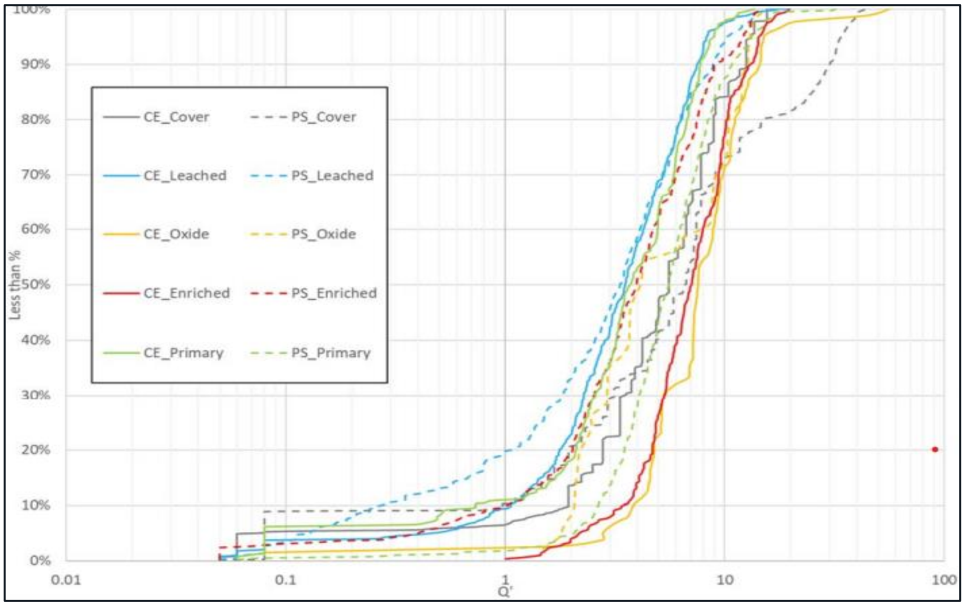
Source: CNI, 2023.

### 16.2.3 Rock Mass Classification

For underground analyses using empirical methods, Barton's Q' (1974) rock tunnelling quality index was calculated from the logged parameters. During the logging, a significant amount of the core was identified to have ISRM rock hardness less than R2, indicating very low rock strengths, which is corroborated by the laboratory testing. Barton's Q rock tunnelling quality index does not consider the rock strength in the logged parameters, and this could affect some of the results of the underground analyses. To avoid discrepancies in the data, CNI applied some corrections to the dataset, the details of which are outlined in the CNI report.

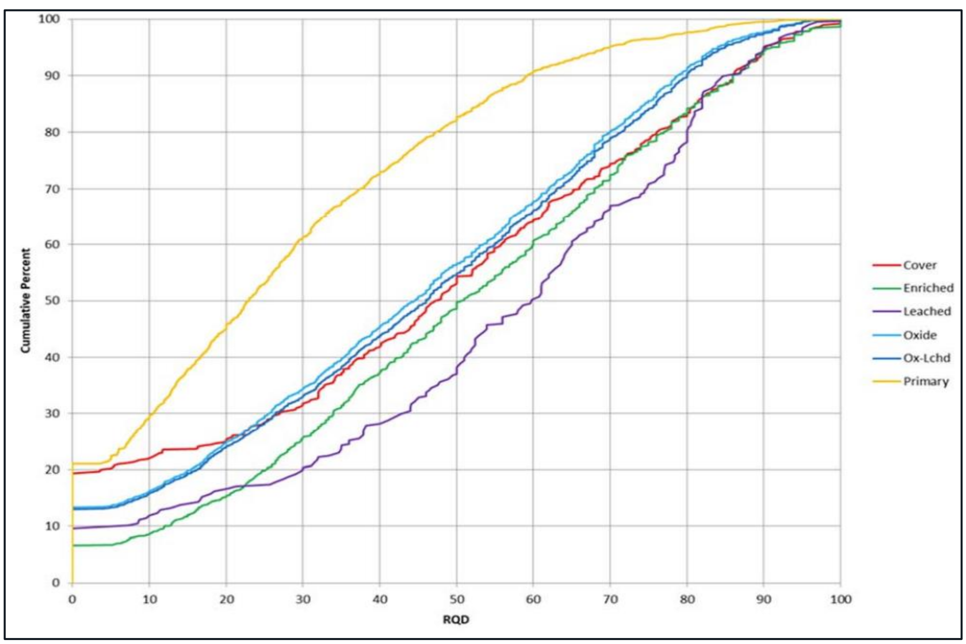
Figure 16-2 and Figure 16-3 present cumulative distributions of all logged Q' data for Cactus East and Parks Salyer, and RQD data for Cactus West by mineral domain.

**Figure 16-2: Q' Cumulative Distributions by Mineral Domain in PS and CE**




Source: CNI, 2023.

**Figure 16-3: RQD Distribution for Cactus West Drillholes**




Source: CNI, 2023.

Geotechnical block models were generated for predicting rock conditions at Parks Salyer and Cactus (East and West) project areas. The summary statistics of RQD by lithology are summarised in Table 16-5 and Table 16-6.

**Table 16-5: Parks Salyer RQD Summary Statistics by Lithology**

| Lithology | Lith Codes | Model Code | Statistics | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Count | Min | Max | Mean | Variance | Std. Dev. | C.o.V. | Q1 | Q2 | Q3 |
| Basement | 0-14 | 0 | 764 | 0 | 92.2 | 36.8 | 604.0 | 24.6 | 0.7 | 16.8 | 36.3 | 54.6 |
| Granite | 20-49 | 2 | 23522 | 0 | 117.6 | 43.7 | 698.2 | 26.4 | 0.6 | 23.3 | 43.0 | 63.2 |
| Conglomerate | 70 | 3 | 1530 | 0 | 100.0 | 45.6 | 735.6 | 27.1 | 0.6 | 30.0 | 50.0 | 67.0 |
| Alluvium | 80 | 6 | 91 | 0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Dacite | 60 | 7 | 563 | 0 | 102.0 | 53.1 | 543.7 | 23.3 | 0..4 | 38.0 | 54.0 | 70.6 |

Source: CNI, 2023.

**Table 16-6: Cactus RQD Summary Statistics by Lithology**

| Lithology | Lith Codes | Model Code | Statistics | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Count | Min | Max | Mean | Variance | Std. Dev. | C.o.V. | Q1 | Q2 | Q3 |
| Basement | 10-19 | 0 | 489 | 0 | 118.8 | 52.5 | 725.2 | 26.9 | 0.5 | 31.8 | 56.3 | 71.3 |
| Monzonite Porphyry | 40-49 | 1 | 6515 | 0 | 100.0 | 41.4 | 718.2 | 26.8 | 0.6 | 18.7 | 41.2 | 64.0 |
| Granite | 20-29 | 2 | 6674 | 0 | 102.9 | 50.7 | 738.1 | 27.2 | 0.5 | 28.9 | 52.6 | 73.4 |
| Conglomerate | 70 | 3 | 2315 | 0 | 104.0 | 58.5 | 438.0 | 20.9 | 0.4 | 45.1 | 60.0 | 74.0 |
| Alluvium | 80 | 6 | 145 | 0 | 89.1 | 4.1 | 115.8 | 10.8 | 2.6 | 0.0 | 0.0 | 0.0 |
| Dacite Porphyry | 60 | 7 | 84 | 0 | 86.0 | 24.2 | 557.2 | 23.6 | 1.0 | 0.3 | 20.2 | 37.7 |
| Andesite | 61 | 8 | 165 | 0 | 100.8 | 58.2 | 658.8 | 25.7 | 0.4 | 46.9 | 62.7 | 76.7 |
| Diabase | 30-39 | 9 | 137 | 0 | 92.0 | 28.1 | 691.9 | 26.3 | 0.9 | 0.2 | 21.1 | 49.2 |

Source: CNI, 2023.

### 16.2.4 Cavability

Cavability was estimated using Laubscher's cavability chart. The range of Laubscher RMR values are presented in Table 16-7. Adjustments to the RMR (MRMR) were applied based on joint orientation and the orientation of development relative major structures. No adjustment was applied to account for mining-induced stresses as these are not currently well understood. Laubscher's curves for predicting critical hydraulic radii for caving is shown on Figure 16-4.

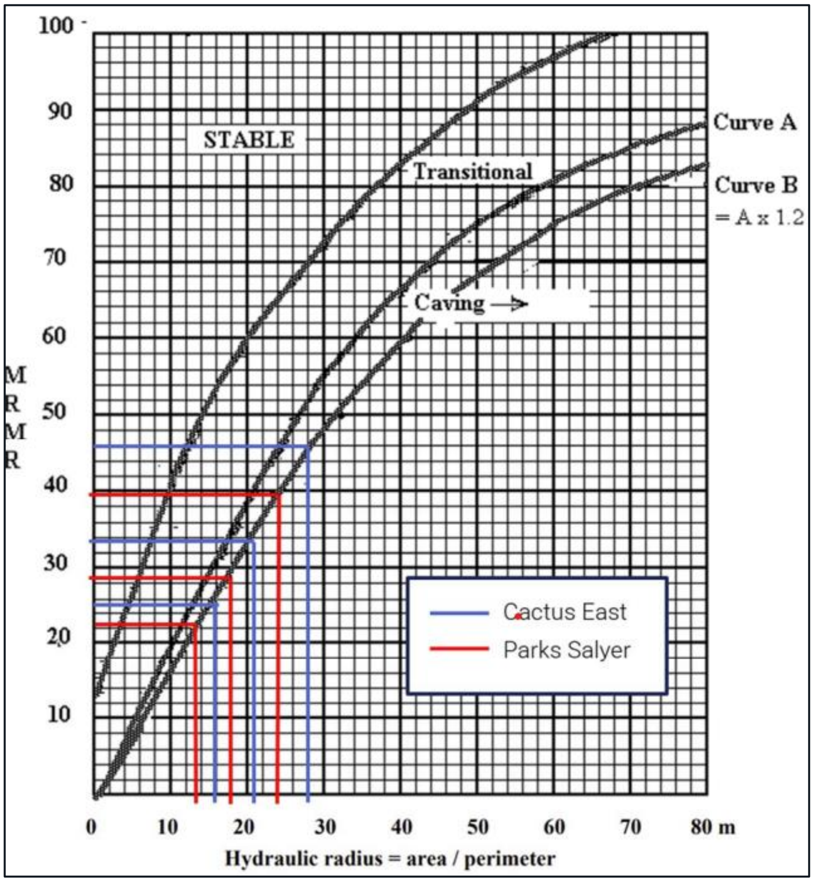
Both deposits are expected to begin sustained caving at a hydraulic radius of 65.6 ft (20 m) (equivalent to a 260 ft (79.2 m) by 260 ft (79.2 m) square) based on central estimates. Both the Cactus East and Parks Salyer areas have footprints which far exceed the minimum hydraulic radius for caving. Consequently, SLC backs are expected to cave naturally with no preconditioning necessary.

**Table 16-7: Laubscher RMR and MRMR Estimates**

| Laubscher 1990 RMR | Cactus East | | | Parks/Salyer | | |
|---|---|---|---|---|---|---|
| Parameter | MIN | MID | MAX | MIN | MID | MAX |
| IRS (MPa) | 6 | 7 | 8 | 4 | 4.5 | 5 |
| RQD | 6 | 9 | 12 | 4 | 6 | 10 |
| Fracture Frequency | 11.3 | 14 | 16.3 | 10 | 11 | 12 |
| Joint Condition | 13.5 | 17 | 20.5 | 14.5 | 17 | 20.5 |
| RMR | 36.8 | 47 | 56.8 | 32.5 | 38.5 | 47.5 |
| Joint Orientation Adjustment (%) | 75 | 80 | 90 | 75 | 80 | 90 |
| Shear Zone Orientation Adjustment (%) | 90 | 90 | 90 | 90 | 90 | 90 |
| Adjusted Rock Mass Rating (MRMR) | 24.8 | 33.8 | 46.0 | 21.9 | 27.7 | 38.5 |
| H.R. for CAVING | 16 | 21 | 28 | 13 | 18 | 24 |

Source: CNI, 2023.

**Figure 16-4: Cavability Prediction**




Source: CNI, 2023.

### 16.2.5 Fragmentation

The rock mass overlying the SLC development areas at Parks/Salyer and Cactus East will generate very fine fragmentation as it caves to surface. Most of the ore recovered by SLC mining will be derived from blasted rock, however, the fragmentation generated from the caved overburden is important in understanding issues such as:

- ingress of dilution (note that dilution is here defined as any material outside the SLC blasted volumes and can therefore, be economic or sub-economic in grade),
- assessing the impact of differential flow rates,
- assessing the proportion of fines material and potential for mud rushes in the cave column over time,
- understanding the porosity/permeability of the caved rock mass, and
- estimating the proportion of oversize material requiring secondary blasting at the drawpoint.

Estimations of primary fragmentation based on measured core piece lengths suggests that a medium volume size of 1.2 $ft^2$ (0.37 m) and 0.54 $ft^2$ (0.16 m) could be generated for Cactus East and Parks/Salyer respectively based on length weighted averages.

An assessment of secondary fragmentation was undertaken by AGP for the Cactus East and Parks/Salyer deposits using the Block Cave Fragmentation program BCF (V3.05) developed by Dr. Essie Esterhuizen. BCF has been widely used in the caving industry for estimating fragmentation. The program relies on information derived from core logging, mapping data and material properties. Primary fragmentation is defined as primary blocks that are formed and released from the cave back and is influenced by the joint distribution, orientation, and joint condition characteristics. If stress levels exceed the rock mass strength, then additional stress induced fractures are also included. Secondary fragmentation is an indication of the rock size reporting to the draw points and is influenced by cave shape, stress magnitudes, rock strength, height of draw and production rates.

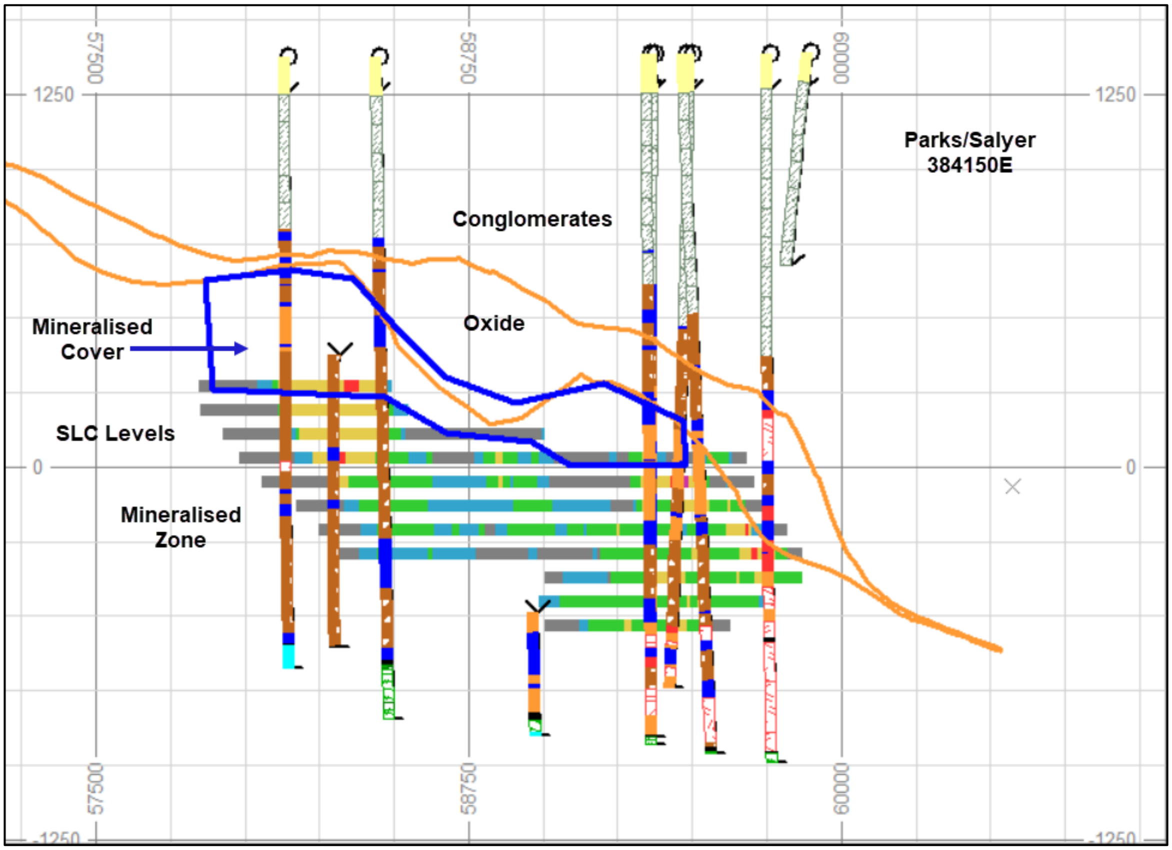
The data inputs used for this assessment were sourced from a combination of preliminary summary data reports prepared by CNI, however, several assumptions and parameters were also included to address information gaps. The results should therefore be regarded as indicative.

Fragmentation assessments were made for the following areas:

- Parks/Salyer mineralized zone (all areas)
- Parks/Salyer mineralized cover (between top SLC level and overlying conglomerates)
- Cactus East mineralized zone (all areas)
- Conglomerates above Cactus East and Parks/Salyer

For this assessment the mineralised zone at Parks/Salyer was divided into two zones. (1) the mineralised zone which is to be mined by SLC and (2) the mineralised cover rocks which indicate improved rock quality conditions (Figure 16-5).

**Figure 16-5: Parks/Salyer Fragmentation Domains (Looking West)**




Source: AGP, 2023.

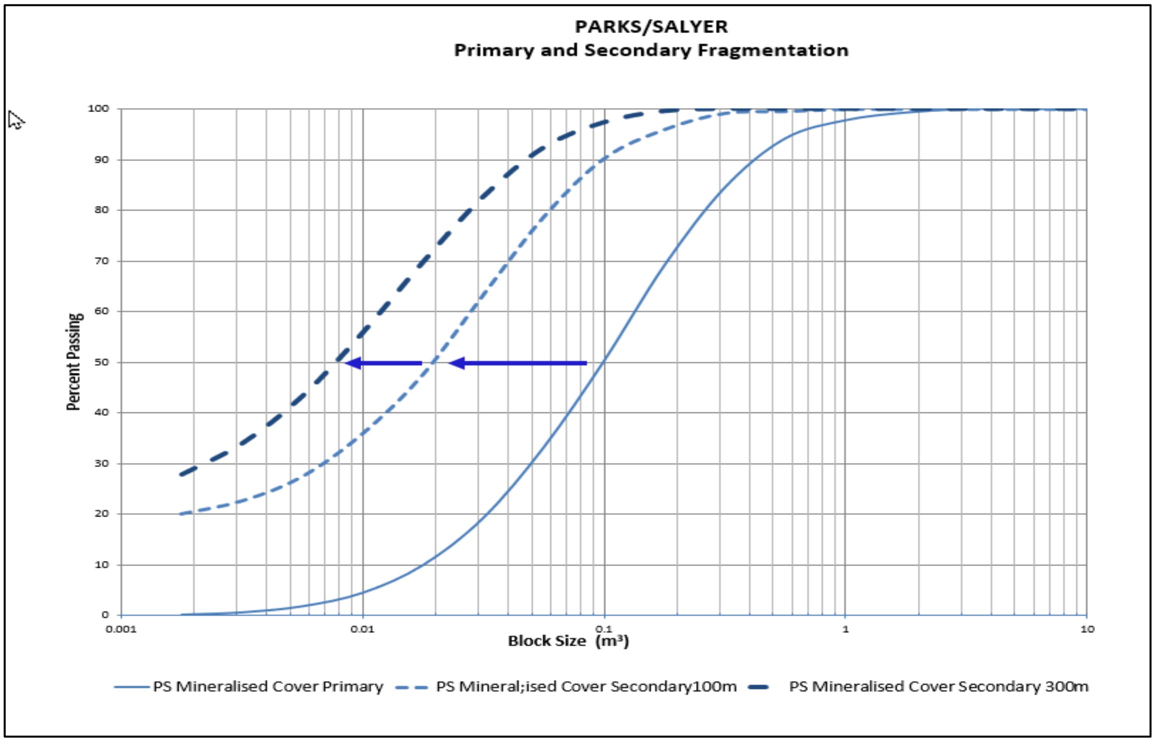
The Parks/Salyer and Cactus East deposits are expected to generate fine primary fragmentation (Table 16-8). The mineralised rocks above the Parks/Salyer SLC development area and the Cactus East mineralised zone have similar fragmentation profiles.

**Table 16-8: Primary Fragmentation**

| Zone | Average Volume | $P_{50}$ | $P_{80}$ |
|---|---|---|---|
| PS Mineralised Zone (Block Caving scenario only) | 0.21 $ft^3$ (0.006 $m^3$ ) | <0.6 $ft^3$ (0.017 $m^3$) | <5.3 $ft^3$ (0.15 $m^3$) |
| PS Mineralised Cover (above SLC) | 1.3 $ft^3$ (0.037 $m^3$) | <3.5 $ft^3$ (0.1 $m^3$) | <9.5 $ft^3$ (0.27 $m^3$ ) |
| CE Mineralised Zone | 1.62 $ft^3$ (0.046 $m^3$) | <4.2 $ft^3$ (0.12 $m^3$) | <8.8 $ft^3$ (0.25 $m^3$) |

Secondary fragmentation profiles were generated for the rock mass after the equivalent drawdown of 328 ft (100 m) and 984 ft (300 m). There is a significant reduction in rock size due to the impacts of comminution and breakup of the rock fragments, principally due to the inherit weak rock strength (Figure 16-6 and Table 16-9).

**Figure 16-6: Parks/Salyer Primary and Secondary Fragmentation**




Source: AGP,2023.

**Table 16-9: Secondary Fragmentation**

| Zone | Average Volume 100 m Draw | Average Volume 300 m Draw |
|---|---|---|
| PS Mineralised Zone (Block Caving scenario only) | 0.07 $ft^3$ (0.0019 $m^3$) | 0.02 $ft^3$ (0.00046 $m^3$) |
| PS Mineralised Cover (above SLC) | 0.35 $ft^3$ (0.01 $m^3$) | 0.12 $ft^3$ (0.0034 $m^3$) |
| CE Mineralised Zone | 0.35 $ft^3$ (0.01 $m^3$) | 0.16 $ft^3$ (0.0046 $m^3$) |

Of particular importance is the high proportion of "fines" (<5 cm;<2 in) expected in the rock mass in the order of 20-30%. The fines component is a subjective estimate based on RQD results and visual inspection of selected core photos. This indicates that the rock mass is highly prone to formation of mud columns if the cave were to be saturated by ground or meteoric waters. This may have implications for ore recoveries towards the later stages of the mine life when high levels of overdraw are scheduled to be mined from the SLC.

The very weak rock strengths and lack of dependable structural information precluded a reliable assessment of the fragmentation for the overlying conglomerates using the BCF method. The breakdown and denigration of the conglomerates in the cave column is expected to result in the generation of significant fines over time. The vertical movement of these rocks are expected to flow more readily than the underlying oxides, leached and mineralised zones assuming dry conditions.

Stress induced fracturing is expected to be minor due to the low stress environment within an extensional tectonic setting. The largest impact to secondary fragmentation will be the rapid degradation of the rock mass as it flows within the cave column. Later stages of SLC mining could encounter higher levels of dilution when overdraw strategies are implemented. Future studies should consider the impact of differential draw by changing the mobility factors for overlying caved rocks.

The flow behaviour of the overburden could be unpredictable under saturated conditions. The cohesive strength of the saturated clays could hamper flow and increase the risk of mud forming columns in the overburden. Draw strategies and ground monitoring programs will be critical in minimising and identifying mud rush risks.

#### 16.2.6 Subsidence

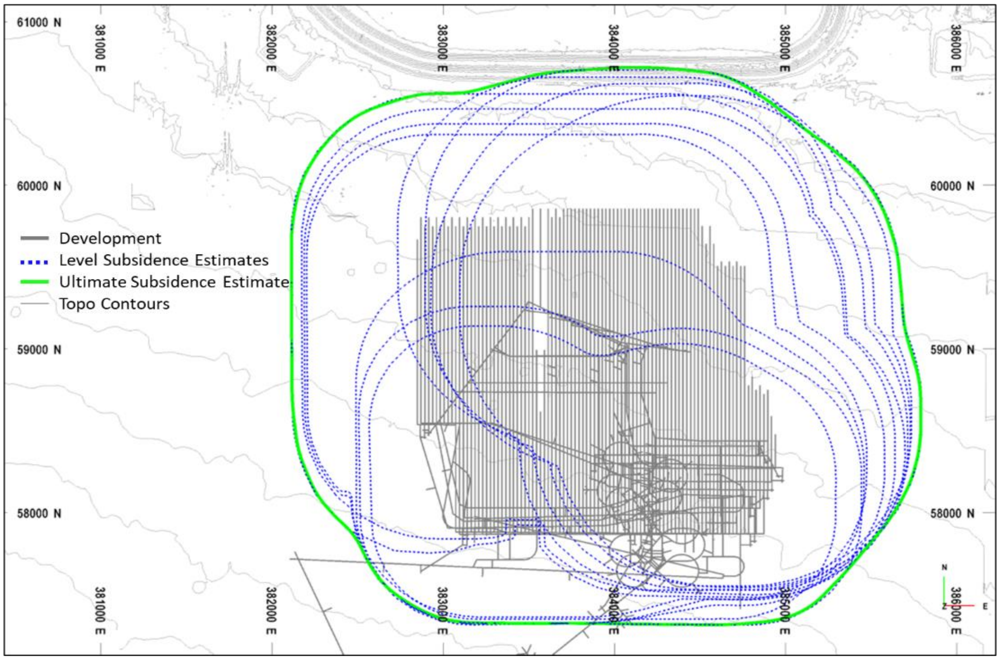
As material is drawn from the underground using the SLC method, a surface depression will occur from subsidence. Any disturbance to rock on the surface can impact the viability and stability of open pit targets or other surface infrastructure. As a result, the ultimate extents of surface disturbance must be considered. The extent of vertical subsidence decreases with distance from the centre of the surface depression, as summarized below:

- Glory hole (nominal 80-degree cone) – zone where the depression is most extensive and native topography is dropped downward.
- Crack limits (nominal 70-degree cone) – zone outboard of the glory hole where tension cracks are visible.
- Zone of influence (nominal 65-degree cone) – zone where the surface is disturbed, although it may be difficult to observe visually.

For the prediction of subsidence affecting surface mining targets and infrastructure, CNI used a 65-degree composite subsidence angle which is inclusive of all three zones of deformation (glory hole, crack limits, and zone of influence). These composite angles are based on both site-specific observations (slope audits in Gila), as well as prior experience with mining projects in Arizona overlain by Gila Conglomerate, including the Lakeshore (Tohono) panel cave operation, San Manuel, and the various Miami block cave operations.

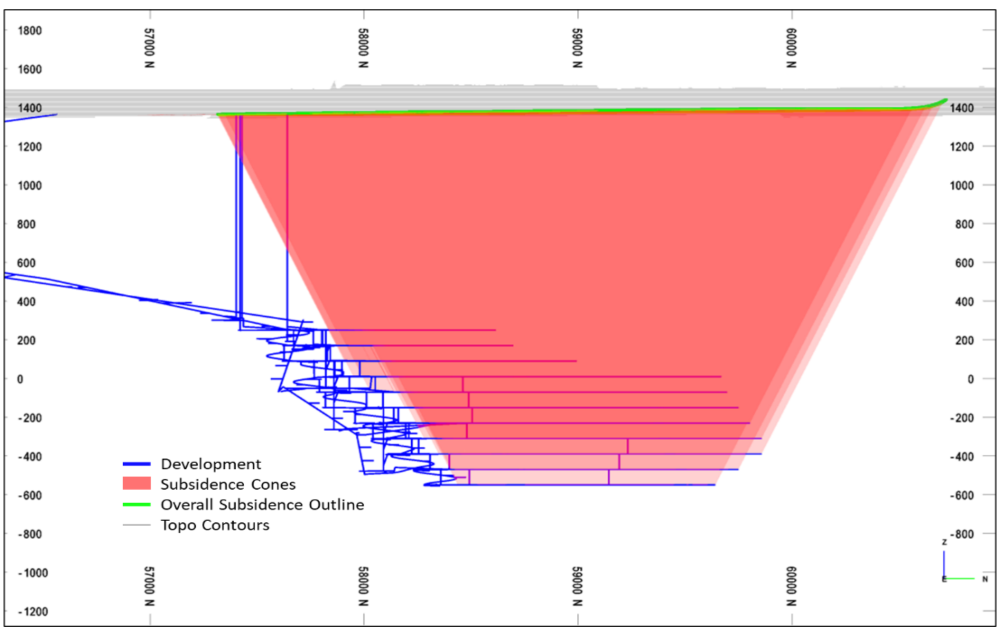
Figure 16-7 and Figure 16-8 present the estimated ultimate subsidence extents for Parks Salyer. Figure 16-9 and Figure 16-10 present the estimated ultimate subsidence extents for Cactus East.

**Figure 16-7: Surface Subsidence Predictions – PS, Plan View**



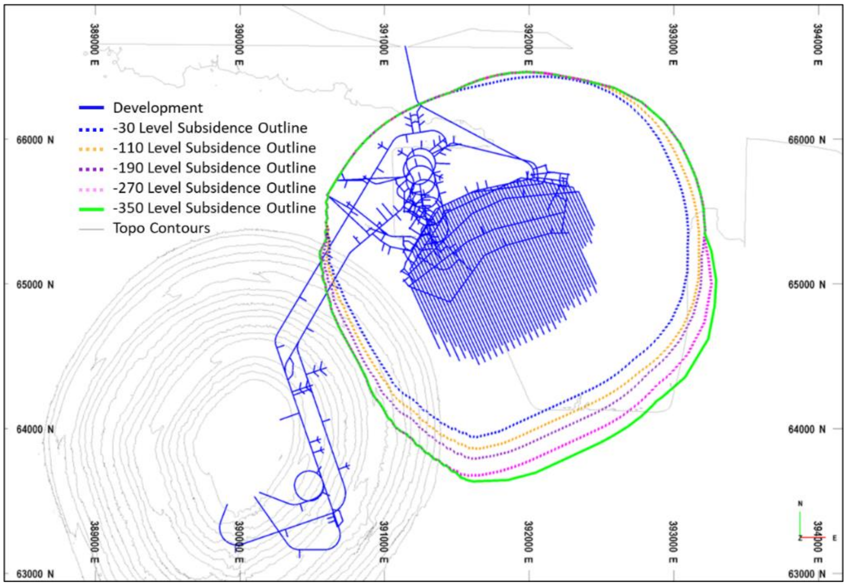

Source: CNI, 2023.

**Figure 16-8: Surface Subsidence Predictions – PS, Section View Looking West**



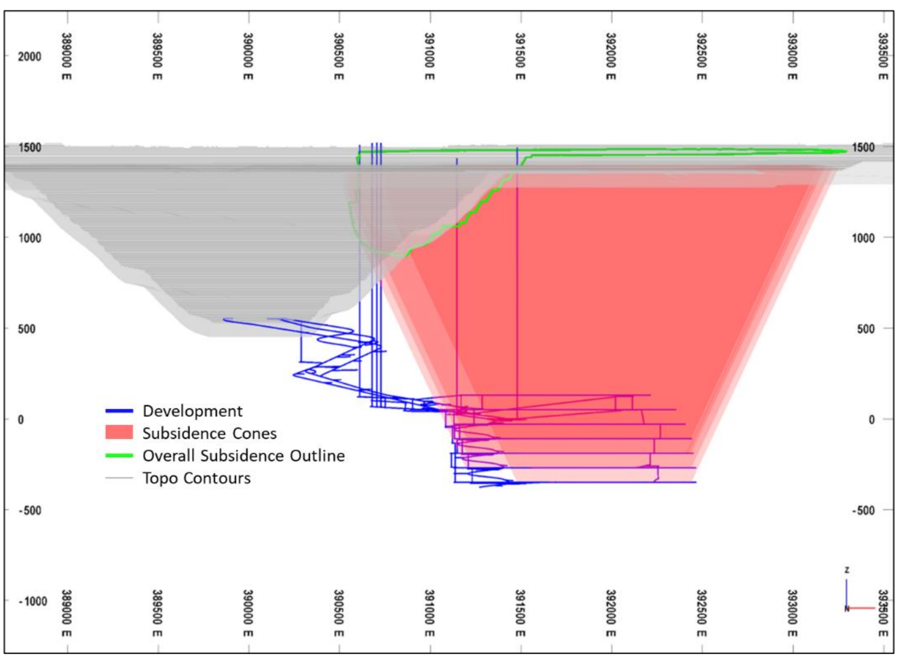

Source: CNI, 2023.

**Figure 16-9: Surface Subsidence Predictions – CE, Plan View**




Source: CNI,2023.

**Figure 16-10: Surface Subsidence Predictions – CE, Looking North**




Source: CNI,2023.

### 16.2.7 Ground Support Provisions

Ground support for capital development is considered permanent and requires a high level of stability. Table 16-10 presents the ground support categories for long term development.

**Table 16-10: Long Term Development Ground Support Categories**

| Support Category | Q Value | D&B Advance Length (ft) | Roadheader Unsupported Advance Length (ft) | Support Type |
|---|---|---|---|---|
| Category 1 | > 3.0 | 13.0 | 13.0 | 8-ft #7 rebar on 4-ft spacing with welded mesh (4 in/6 Ga) to within 5 ft of sill |
| Category 2 | 0.8 – 3.0 | 10.0 | 13.0 | 8-ft #7 rebar on 4-ft x 4-ft spacing with welded mesh (4 in/6 Ga.) and 2 in of shotcrete to within 5 ft of sill |
| Category 3 | 0.07 – 0.8 | 8.0 | 10.5 | 4 in of fiber reinforced shotcrete (FRS) and 8-ft #7 rebar on4-ft x 4-ft spacing with welded mesh (4 in/Ga.) down to sill |
| Category 4 | < 0.07 | 4.0 | 5.5 | 6 in of FRS and 8-ft #7 rebar on 4-ft x 4-ft spacing with welded mesh (4 in/Ga.) down to sill with 6 count #7 rebar arch spaced each 8 ft and fully encased in shotcrete; forepoling (spilling) |

Notes: For intersections additional secondary support is required.

Ground support for the sublevel drifts is considered temporary and requires less security than long term openings. Production heading ground support will consist of split set type friction bolts (SS-39), welded wire mesh (4 in/6 Ga.), and shotcrete as summarized in Table 16-11. In areas of particularly poor rock quality (Q ≤ 2.0), secondary support of high-capacity bolts should be installed within each brow and shotcrete carried down to the sill level, as detailed in Table 16-11.

**Table 16-11: Production Ground Support Categories**

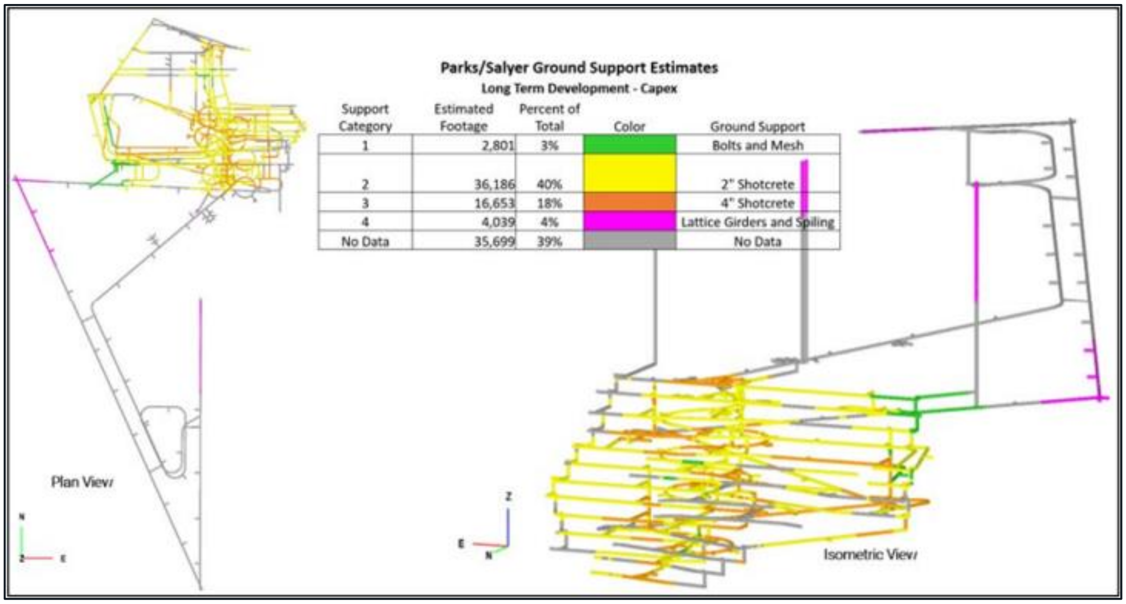
| Production Ground Support - SLC Drives | | | | |
|---|---|---|---|---|
| Support Category | Q value | Advance Length (ft) | Primary/ Secondary | Support Type |
| Category 1 | > 2.0 | 9.0 -13.0 | Primary Support | 8-ft #7 Split Sets (SS39) on 3-ft x 3-ft spacing with welded mesh (3-in spacing - 7 SWG/W2.7) sill to sill |
| | | | Secondary Support | 2 in of shotcrete on back only |
| Category 2 | 0.7 - 2.0 | 8 | Primary Support | 8-ft #7 Split Sets (SS39) on 3-ft x 3-ft spacing with welded mesh (3-in spacing - 7 SWG/W2.7) and 2 in of shotcrete sill to sill |
| | | | Secondary Support | Two rows of 12-ft high-capacity bolts on 4-ft spacing at each production brow |

Notes:
1. 13-ft advances are possible when Q > 3.0.

2. High-capacity bolts include Super Swellex, #8 rebar or DIWYDAG, cable bolts, or R28 self-drilling hollow-core bolts.

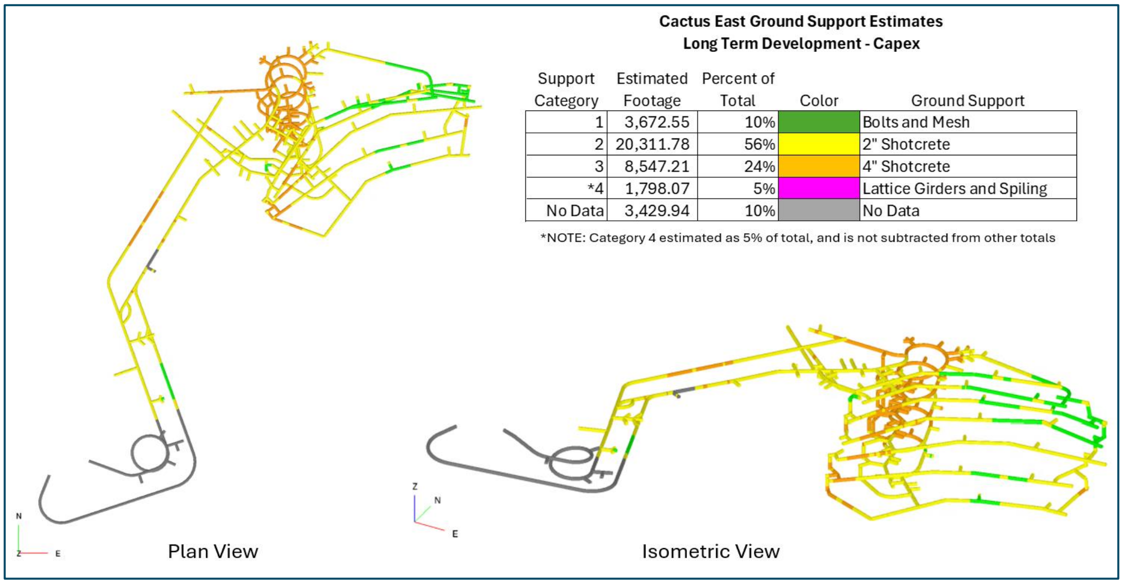
Estimated proportions of the Cactus East and Parks Salyer capital development by ground support category are presented in Figure 16-11 and Figure 16-12.

**Figure 16-11: PS Ground Support Category Estimates – Long Term**




Source: CNI, 2023.

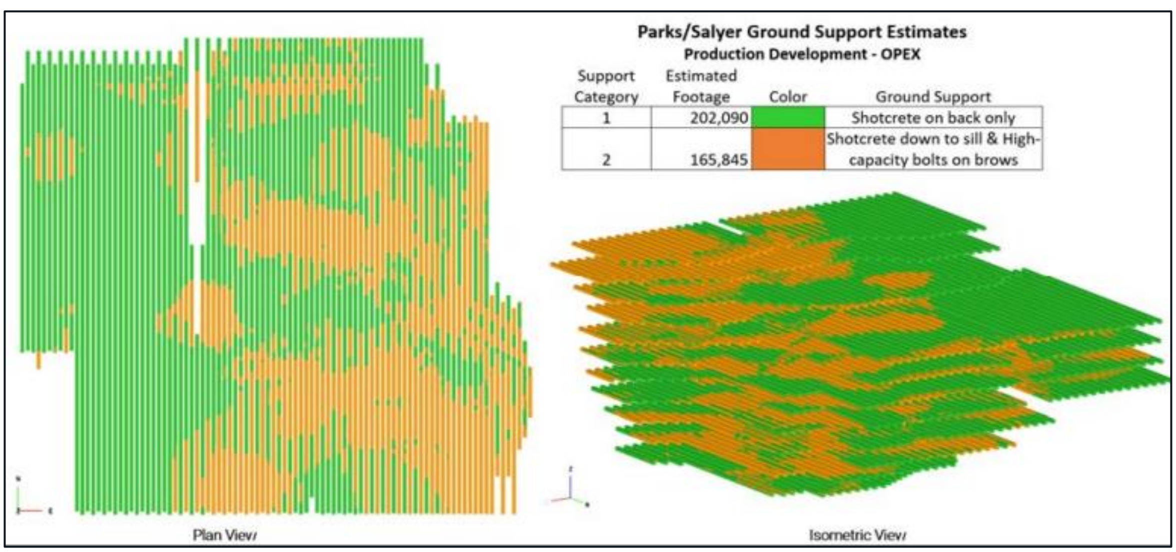
**Figure 16-12: CE Ground Support Category Estimates – Long Term**




Source: CNI, 2023.

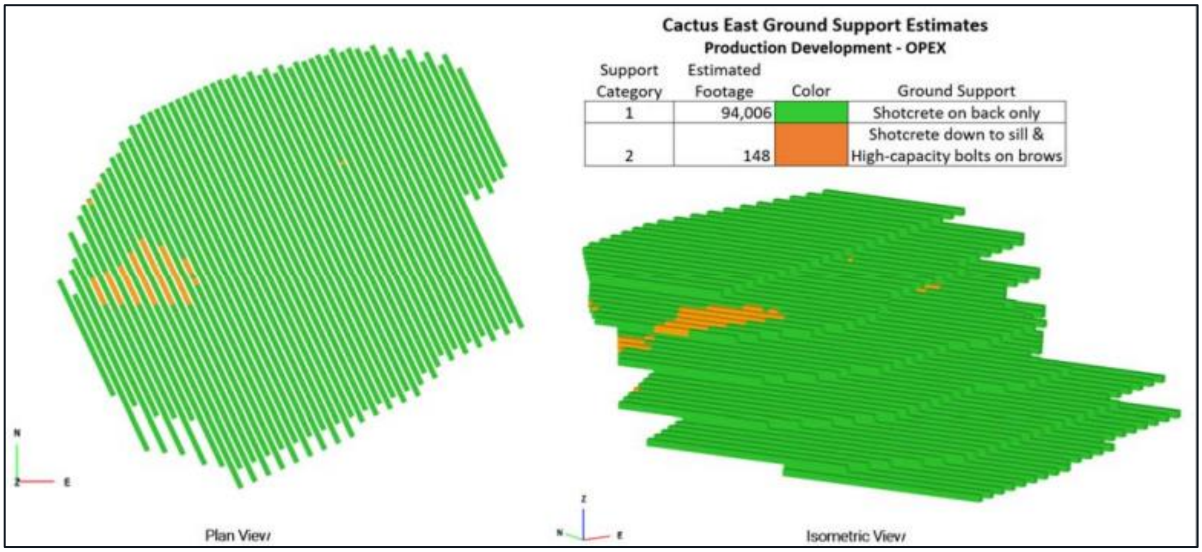
Estimated proportions of the Cactus East and Parks/Salyer production development by ground support category are presented in Figure 16-13 and Figure 16-14.

**Figure 16-13: Parks Salyer Ground Support Category Estimates - Production**




Source: CNI, 2023.

**Figure 16-14: Cactus East Ground Support Category Estimates - Production**




Source: CNI, 2023.

### 16.2.8 Underground Pillar Stability

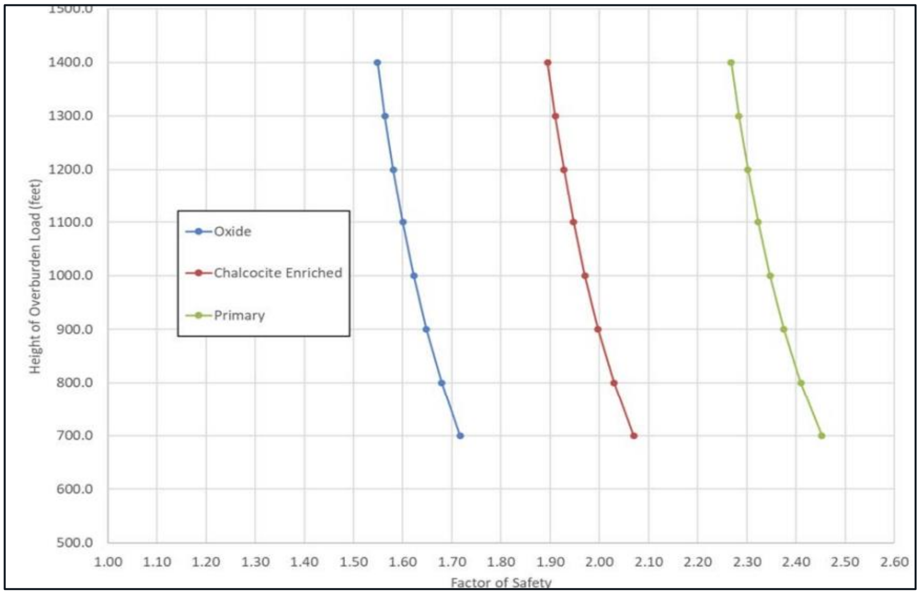
To estimate rock pillar strength between sublevel drifts, CNI used Wilson's (1972) confined core method of pillar stability. Pillar stability must be maintained for optimized sublevel drift spacing if sublevel cave is a viable mining method. Stability analyses were conducted for the mineral domains using the strengths summarized in Table 16-12. The primary mineral domain was included to account for cases where sublevel pillars are within the primary just below the chalcocite enriched zone. CNI has assumed a k value (ratio of horizontal stress to vertical stress) equal to 0.8 based on the project area being in an extensional environment and in-situ stress measurements at other project sites in southern Arizona.

**Table 16-12: Rock-Mass Strength Used in Pillar Stability Analyses**

| Mineral Domain | Rock Mass Strength | |
|---|---|---|
| | PHI (Deg.) | Coh (psi) |
| Oxide | 27.24 | 170.6 |
| Chalcocite Enriched | 30.28 | 197.1 |
| Primary | 34.23 | 224.8 |

Pillar loads were estimated as tributary area loads equal to the overburden (caved zone) thickness which can range between 700 ft to 1,400 ft. The results of the stability analysis are presented in Figure 16-15. All mineral domains achieve an acceptable safety criterion at the recommended sublevel drift spacing with the base case pillar load assumption.

**Figure 16-15: Stability Results of Sublevel Drift Pillars**




Source: CNI, 2023.

### 16.2.9 Open Pit-Geotechnical

CNI undertook a prefeasibility slope stability study for an expansion of the existing open pit at the Cactus Mine near Casa Grande, AZ. The main objective of this study was to recommend interramp slope angles for mine planning and determine overall slope stability for a larger pit that will be closer to the property boundary and a number of future facilities. Access to the Cactus East underground development may also occur from the east side of the pit.

The existing open pit at Cactus was mined by ASARCO in 1972 to 1984; the legacy pit bottom is at approximately the 400-ft elevation, with a total slope height of approximately 1,000 ft (305 m). Bench heights mined by ASARCO were 40 ft (12.2 m). Despite the age of the slopes, there is no evidence of large-scale slope instability other than long term erosion of bench faces and fault traces. However, due to safety concerns and legal status of the mine closure, access to the pit was not possible for this study.

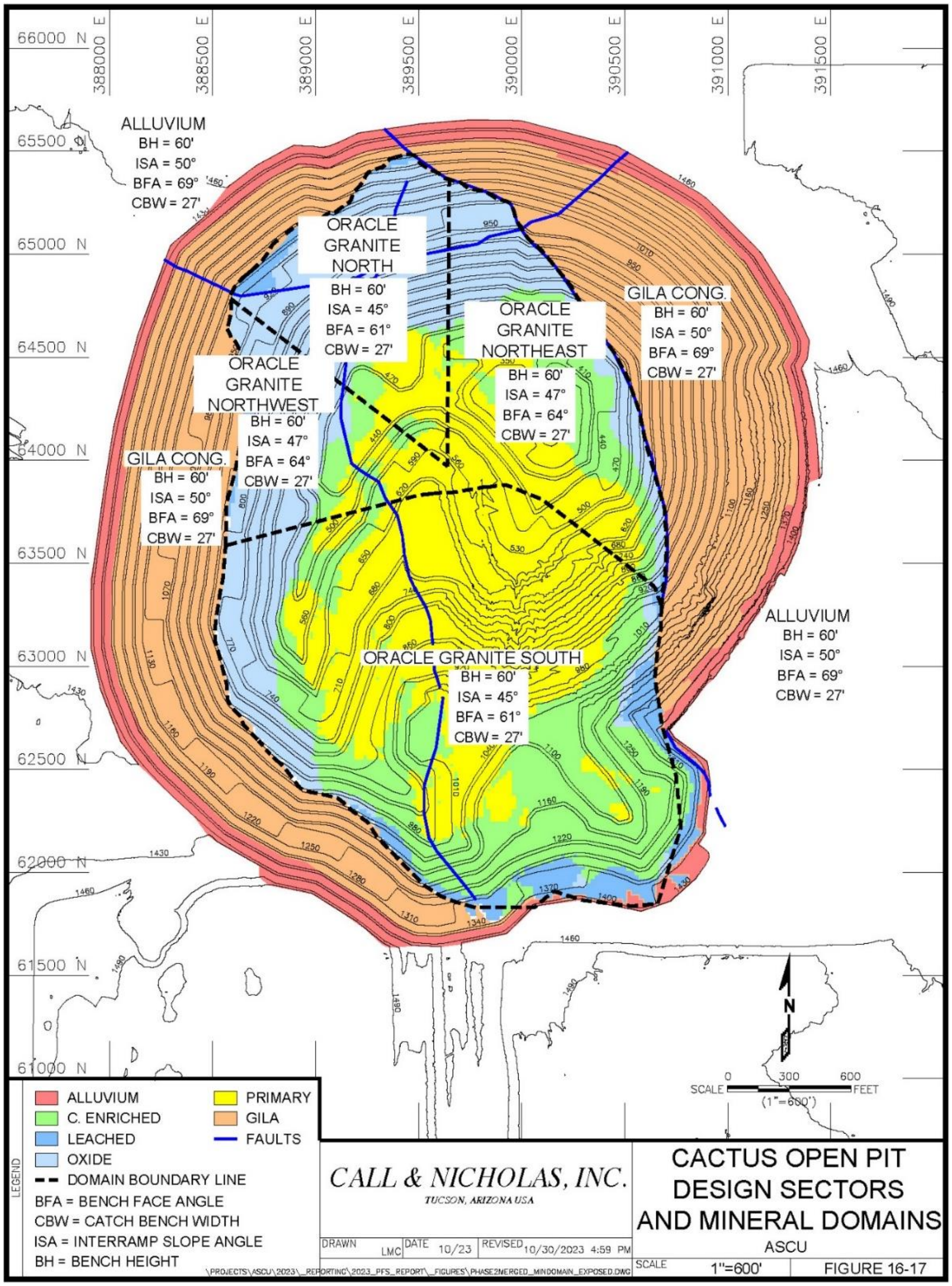
The design assumption is for drill and blast on 20-ft increments with a 60-ft-high final (triple) bench. The design acceptance criteria for the slope design are an 80% bench slope reliability, a 90% interramp slope reliability, and an overall slope factor of safety of 1.2 using two-dimensional slope analysis. CNI has identified a domain in the north and northwest side of the pit south of the Sacaton Fault that is sensitive to overall slope angle and the impact from saturated slopes. The remainder of the pit is controlled by the rock fabric and structures that impact benches and interramp slope angles.

Based on the analyses conducted, the recommended interramp slope angles for the pit expansion are 45˚ in the north sector of the pit, 45˚ in the south sector, and 47˚ in the northeast and northwest. The Gila Conglomerate slope angle is recommended at 50°. The recommended angles by sector are shown in Figure 16-16 and listed in Table 16-13.

**Table 16-13: Cactus Open Pit PFS Slope Recommendations**

| Design Sector | Inter-ramp Slope Angle (deg.) | Bench Layout | | | | |
|---|---|---|---|---|---|---|
| | | Bench Height (ft) | Layout Bench Face Angle (deg.) | Layout Catch Bench Width (ft) | Minimum Bench Face Angle (deg.) | Minimum Catch Bench Width (ft) |
| Alluvium | 50 | 60 | 70 | 28.5 | 69 | 27 |
| Gila Conglomerate | 50 | 60 | 70 | 28.5 | 69 | 27 |
| Oracle Granite Northeast | 47 | 60 | 70 | 34.1 | 64 | 27 |
| Oracle Granite Northwest | 47 | 60 | 70 | 34.1 | 64 | 27 |
| Oracle Granite North | 45 | 60 | 70 | 38.2 | 61 | 27 |
| Oracle Granite South | 45 | 60 | 70 | 38.2 | 61 | 27 |

**Figure 16-16: Cactus Open Pit Design Sectors and Mineral Domains on the Final Pit**

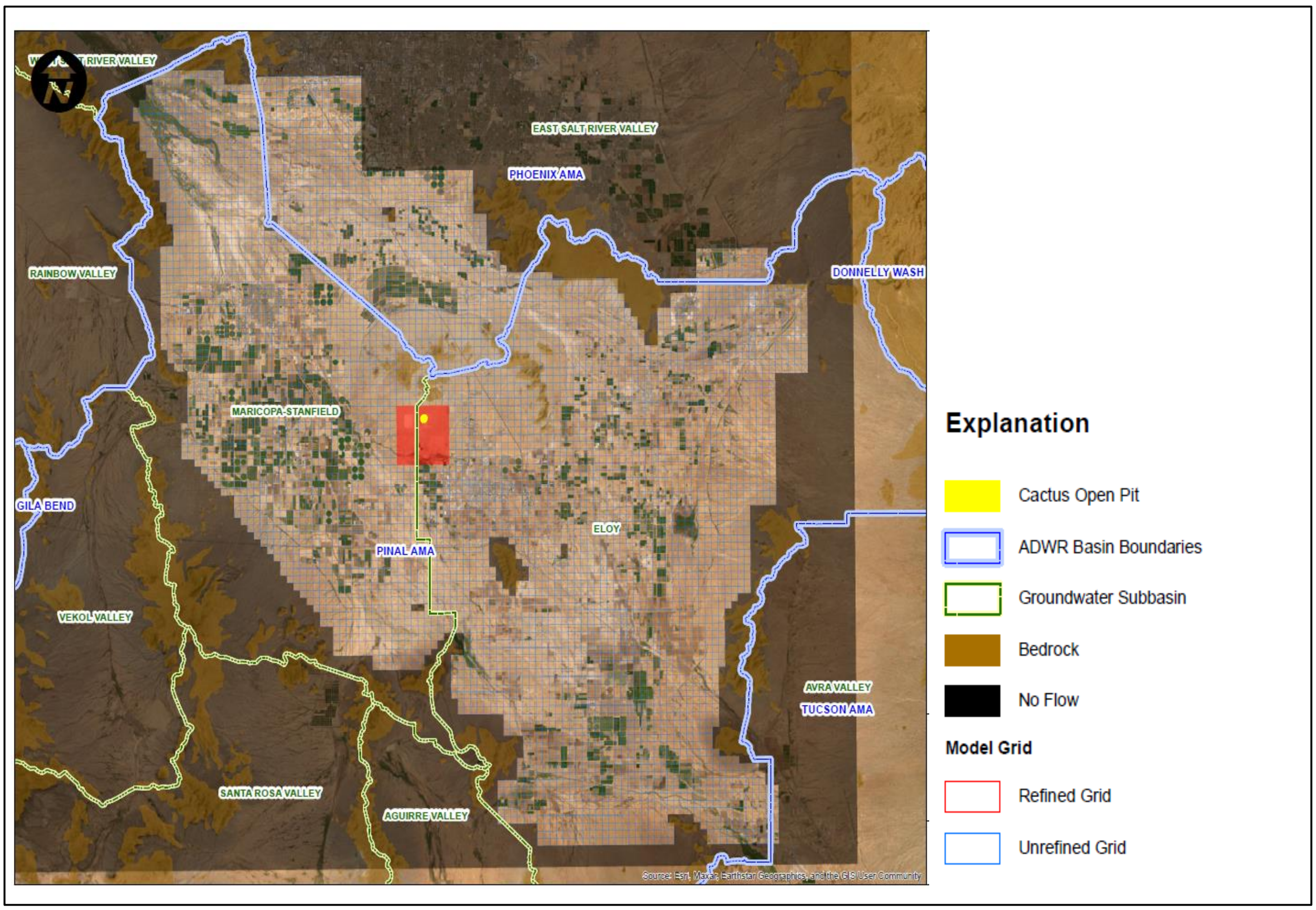



Source: CNI, 2023.

## 16.3 Hydrogeological Considerations

To assess the groundwater conditions and the potential dewatering rates associated with the Cactus East and the Parks/Salyer underground operations, a computer groundwater model was constructed. The model constructed for this project was based upon the Arizona Department of Water Resources (ADWR) Pinal Active Management Area (Pinal AMA) Groundwater Flow Model (Pinal Model), which was released in 2014 (Liu, et al, 2014). The aquifer in the Pinal AMA consists of three units, the upper alluvial unit (UAU), the middle silt and clay unit (MSCU), and the lower conglomerate unit (LCU). In the Eloy Subbasin near and east of the mine, the aquifer is divided into just two units, an Upper Aquifer, and a Lower Aquifer (Hammett, 1992). The Upper Aquifer is the primary aquifer for groundwater production within the Eloy sub-basin, although wells also produce from the LCU.

**Figure 16-17: Refined Model Grid**



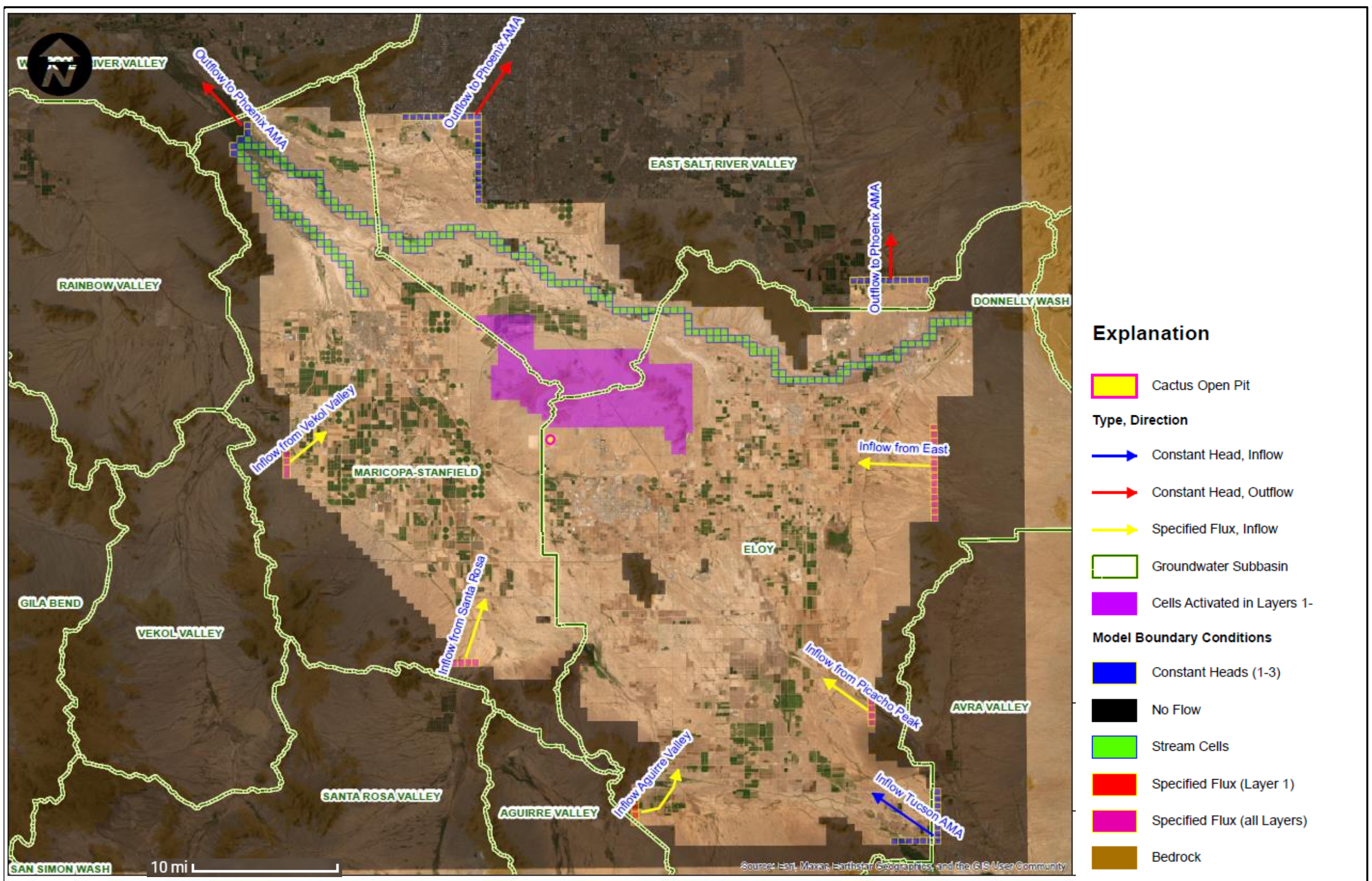

Source: Clear Creek Associates, 2024.

#### 16.3.1 Model Development

The Pinal AMA model was modified by adding two model layers to simulate the upper bedrock (ore deposit layer) and the basement rocks (Pinal Schist) at the site. The model was also converted to use the MODFLOW-Unstructured Grid (MODFLOW-USG) model code, which allows for selective refinement of the model grid. A refinement of the model grid was included in the area of the mine site (Figure 16-18). This level of detail allows for representation of the underground mining activities, such as the excavation of declines and mining of ore deposits and individual pumping wells.

Figure 16-18 shows the model boundary conditions derived from the Pinal Model, which includes specified flux inflow and outflow, constant head outflow and stream cells representing the Gila River and lower reaches of the Santa Cruz River. Groundwater generally flows south to north across the model domain, although significant pumping in the Maricopa-Stanfield area has caused a cone of depression which acts as a groundwater sink.

**Figure 16-18: Model Boundary Conditions**

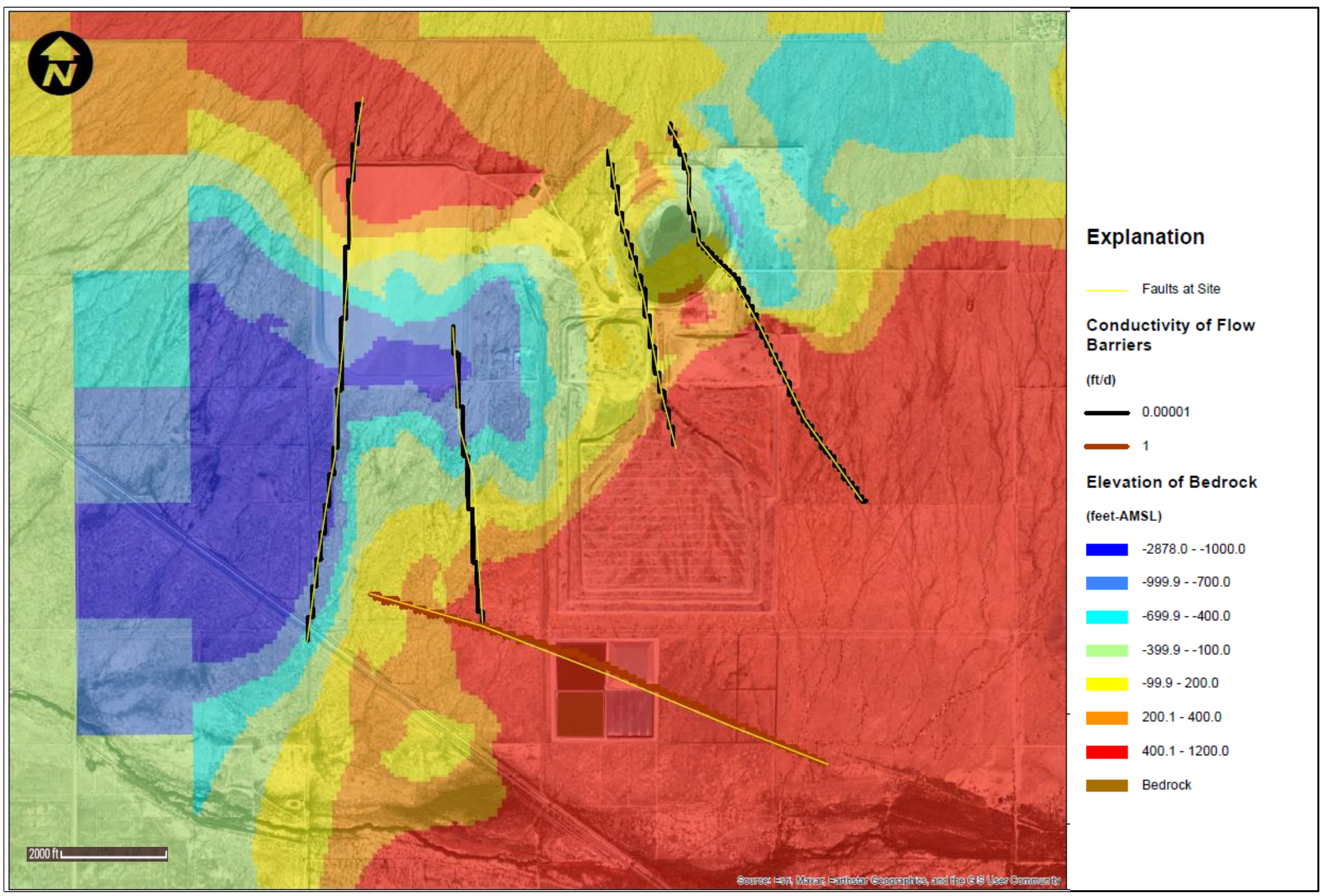



Source: Clear Creek Associates, 2024.

No modifications were made to the Pinal AMA model boundary flow conditions, with rates and levels remaining unchanged. Model layer elevations were modified in the area of the mine using compiled 3-dimensional surfaces representing the bottom of the conglomerate (Layer 3), and the surface representing the basement fault (which separates model layers 4 and 5) with elevations based upon the ASCU exploration drilling program. Model layers 4 and 5 are separated by the basement fault, a low angle normal fault which separates the older basement rocks of the Precambrian Pinal Schist from the younger Oracle Granite, Sacaton Granite and Three Peaks Monzonite.

Data from the site suggests that the faults that cut the conglomerate (Layer 3) may act as barriers to horizontal groundwater flow. Figure 16-19 shows hydrogeologically significant mapped faults at the site, based on available geologic information. Horizontal flow barriers were inserted in layers 3 and 4 to reflect these faults and allow for their simulation as impediments to flow. Horizontal flow barriers change the cell-to-cell conductance, by altering the hydraulic conductivity between cells.

**Figure 16-19: Faults and Horizontal Flow Barriers**




Source: Clear Creek Associates, 2024.

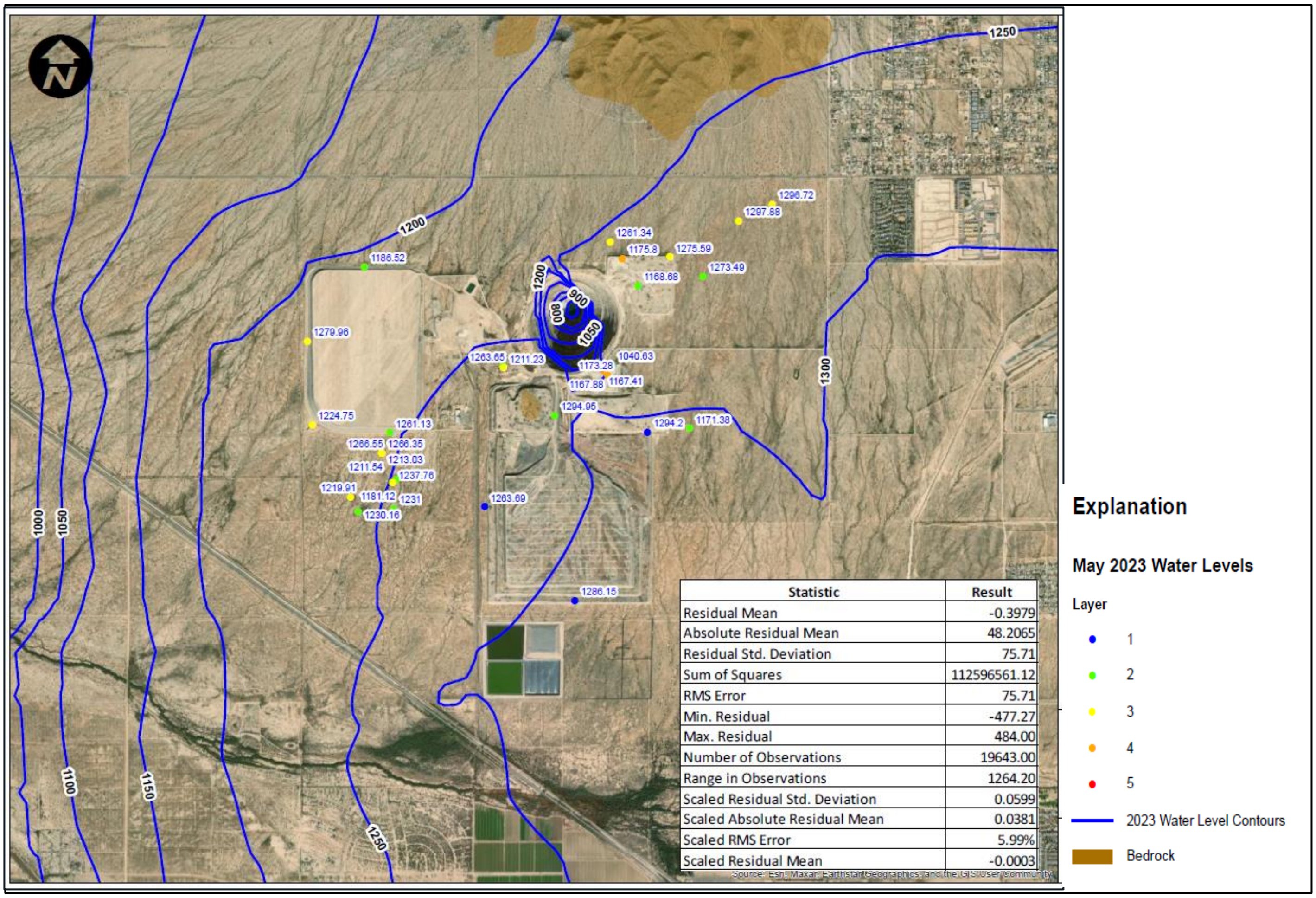
### 16.3.2 Transient Simulation 1984 to 2023

A predictive model simulation was conducted to condition the model with the pit lake and establish local hydrologic conditions with the pit in place. The model was run from 1984 to 2023, representing 40 years of transient conditions. Pumping stresses were updated from the pumpage database of reported Registry of Groundwater Rights (ROGR) and estimates based upon San Carlos Irrigation Project (SCIP) surface water deliveries and estimated pumpage for 1984 to 2021. Rates were then held constant through 2023. This extended the Pinal AMA model from 2009 to 2023, although other boundary stresses, such as recharge, stream flows and specified flux boundaries were not updated after 2009.

Figure 16-20 shows the simulated heads for 2023, showing the influence of the pit. The overall calibration statistics are also shown, indicating an overall scaled root mean square error (RMSE) of 5.99%. Because the purpose of this model was to evaluate drainage flows in the pit and proposed underground workings, the calibration statistics were deemed acceptable.

Simulated inflow to the constant head cells representing the open pit is 35 gpm, at the end of the transient simulation. This is similar to the previous estimates (M&A, 1986), for inflow into the pit, which indicated an average of 33 gpm.

**Figure 16-20: Simulated Water Levels for 2023**




| Statistic | Result |
|---|---|
| Residual Mean | -0.3979 |
| Absolute Residual Mean | 48.2065 |
| Residual Std. Deviation | 75.71 |
| Sum of Squares | 112596561.12 |
| RMS Error | 75.71 |
| Min. Residual | -477.27 |
| Max. Residual | 484.00 |
| Number of Observations | 19643.00 |
| Range in Observations | 1264.20 |
| Scaled Residual Std. Deviation | 0.0599 |
| Scaled Absolute Residual Mean | 0.0381 |
| Scaled RMS Error | 5.99% |
| Scaled Residual Mean | -0.0003 |

Source: Clear Creek Associates, 2024.

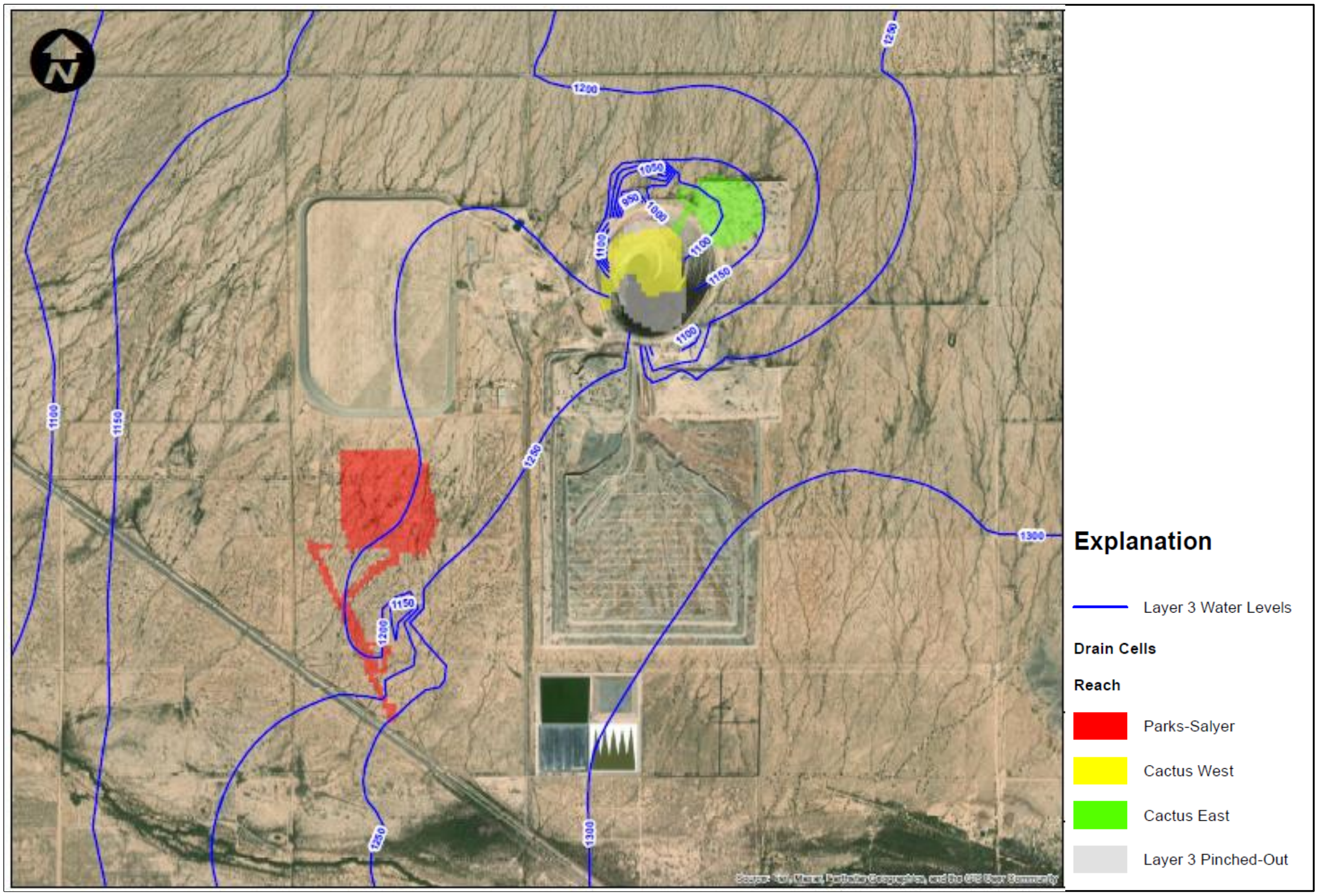
### 16.3.3 Simulation of Mining Activities

The model was designed to simulate three principal proposed phases of mining at the site:

- continued mining in the existing Cactus West open pit.
- development of the Parks-Salyer deposit, and
- development of the Cactus East orebody.

The predictive simulation runs through the projected end of mining in mining Year 20. Figure 16-21 shows the simulated water level for the end of Year 20 for Layer 3. Water level impacts from the mining at Cactus East and Cactus West show up clearly in this figure. Impacts from Parks-Salyer do not impact water levels in Layer 3 significantly.

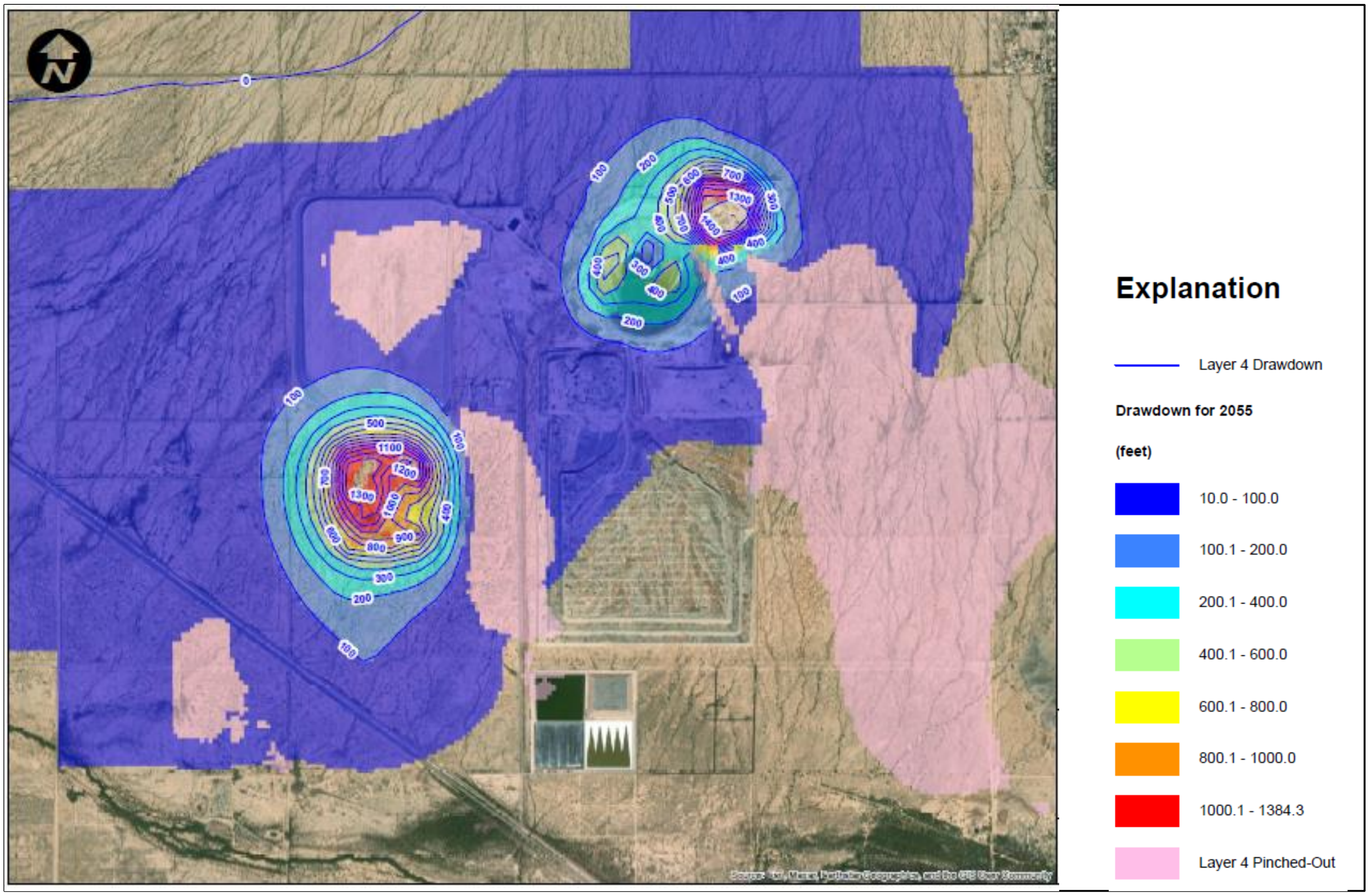
**Figure 16-21: Water Level Elevation for Mining Year 20**




Source: Clear Creek Associates, 2024.

Figure 16-22 shows the drawdown simulated for the end of mining Year 20. The simulated drawdown exceeds 1,200 ft (366 m).

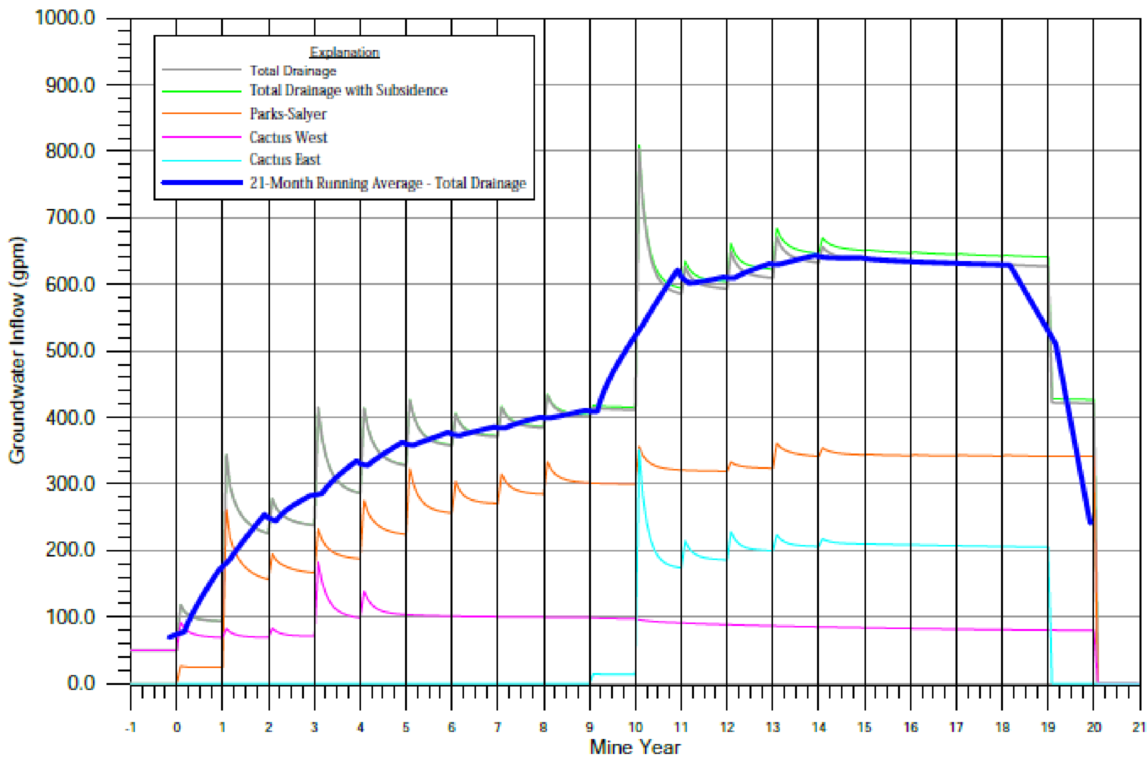
**Figure 16-22: Drawdown for Mining Year 20, Layer 4: Bedrock**




Source: Clear Creek Associates, 2024.

In addition to evaluating drainage from the surrounding aquifer, the impact of subsidence was evaluated by assuming that the hydraulic conductivity in the subsidence zone increases as the mining progresses. Subsidence will propagate upward into the overlying conglomerate and alluvium. Figure 16-23 shows the impact of higher hydraulic conductivity as the green line labeled “Total Drainage with Subsidence.” The impact appears to be small, approximately 9 gpm. Subsidence does not appear to significantly increase drainage flows, based on these assumptions. This may be explained by the fact that much of the rock and conglomerate overlying the underground workings are primarily desaturated by time subsidence occurs.

**Figure 16-23: Simulated Drainage into Workings**




Drainage drops to zero when pumping stops after the end of mining in Year 20 at Parks-Salyer and Year 19 for Cactus-East.

Source: Clear Creek Associates, 2024.

### 16.3.4 Conclusions

An assessment of the model results indicates that the total flow into the workings reaches a maximum of about 640 gpm by mining Year 14, if the hydraulic properties that define the drainage remain similar to the current pit drainage inflow rates. Inflow to the Parks-Salyer underground workings is predicted to peak in mining Year 14 at about 340 gpm and inflow to the Cactus East workings is predicted to peak in mining Year 11 at slightly more than 200 gpm. Flow into the expanded open pit is expected to peak in mining year 4 at about 100 gpm declining to 80 gpm by end of mining. If the assumed conductance in the representation of mine drain cells is increased by 100 times, the maximum flow into the workings would increase to 1600 gpm. If the mined orebody is much more hydraulically conductive than the current pit, flows could also be higher.

Subsidence that is expected to occur over each underground operation is not expected to significantly increase inflow rates and therefore dewatering rates. Subsidence will, however, allow a direct connection between the ground surface and the underground workings. Of concern is the potential for large rain events to contribute water to the workings. A

24-hour, 100-year rain event in this area would result in 3.66 (9.25 cm) inches of rain. Given the area of subsidence over each ore body, it is calculated that a total of 1,240,000 gallons of water would enter the Parks-Salyer operation and a total of 25,500,000 gallons would enter the Cactus East operation. The volume of the water entering the Cactus East operation is larger than the Parks-Salyer operation because the subsidence zone at Cactus East intersects the open pit effectively enlarging the catchment of the subsidence zone. Depending on where the subsidence breaches the open pit, the pit may retain storage capacity that could prevent some of the rainwater from entering the mine workings of the Cactus East underground operation therefore we believe the volumetric estimate of water entering the Cactus East operation is conservatively high.

## 16.4 Open Pit Mining Methods

Open pit mining methods have been selected for the extraction of Mineral Resources in the Cactus West and historical Stockpile areas of the Cactus Project based on the size of the resource, grade tenor, grade distribution and proximity to topography, while Parks/Salyer and Cactus East will be mined using underground methods.

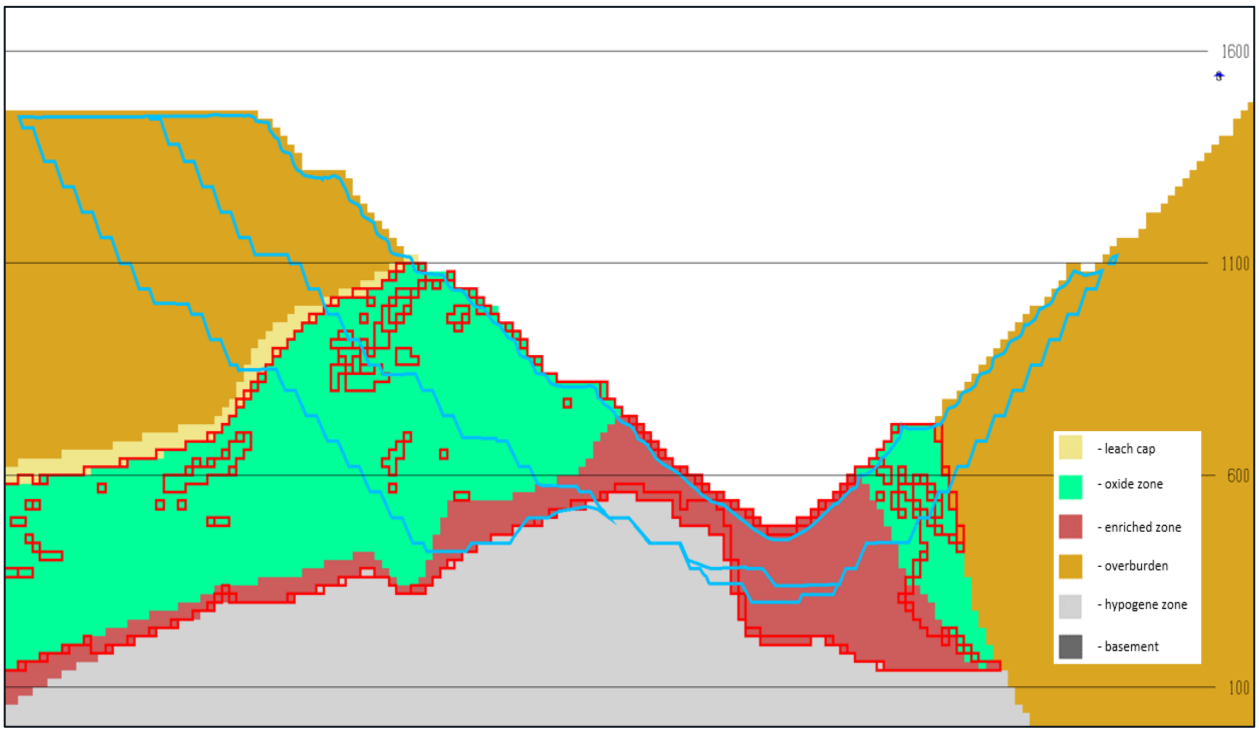
The Cactus West orebodies lie adjacent to and beneath the historically mined Cactus Pit which has a demonstrated open pit geotechnical suitability, with existing pit walls relatively unchanged since mining ceased approximately 30 years ago. The Cactus West orebody includes several different lithological units, including Alluvium and Gila Conglomerate waste overburden which typically range in depth from 50-150 ft (15.2 m to 45.7 m) for Alluvium and 0-600 ft (0 m to 182.9 m) thick for Gila Conglomerate. Intrusive granites and porphyries underly the overburden and include oxide and enriched porphyry zones, as well as minor amounts of hypogene porphyry which are mined in the open pit and not processed. These hypogene porphyry areas represent potential future expansion opportunities if considered with a suitable processing approach. A cross section showing the Cactus West mining phases and mineralization types is shown in Figure 16-24.

The Stockpile mining area is a historical waste dump which contains significant quantities of oxide copper mineralization. This material was considered waste in the historical operation because the sole processing method on site was a flotation mill which could not recover oxide copper mineralization. The Stockpile area has recently been drilled to define a Mineral Resource block model which was used for mine planning. This block model includes the same planning framework which was applied to Cactus West. There are portions of the Stockpile area which have inclusions of non-mineralized waste, but typically the strip ratios are very low. The depth of the Stockpile area varies from approximately 30 ft to 130ft (9.1 m to 39.6 m).

Ore processing in the mine schedules involves all ore material types from Cactus West and the Historic stockpile being processed on a single heap leach facility after multistage crushing. Two primary crushers will be located in close proximity to each other just to the south of the Cactus West pit area. Some stockpiling of ore is envisaged in the mine schedule to help smooth the stripping and ore release profile from Cactus West. These lower-grade stockpiles will be placed west of the Cactus West pit.

Waste from Cactus West and the Stockpile area will be placed into multiple locations, but primarily to the north-east of Cactus West where a view shed berm and waste dump are planned. Some waste will also be used for construction fills in support of heap leach construction, and also into a supplemental waste dump that wraps around the southern edge of the historical tailing's storage facility, partially above and adjacent to the expected subsidence area from the Parks/Salyer sublevel cave.

**Figure 16-24: North-looking cross Section of CW Pit showing Phase 1 and 2 Pits (against lithological model with ore outlined in red)**




Source: AGP 2023.

### 16.4.1 Geological Model Importation

The initial Cactus West and Stockpile resource block models were provided to AGP from Ausenco (Stantec / ALS Geo Resources) in Hexagon MinePlan® block model format which is the same format used by AGP for the mining portion of the PFS including pit and WRF design and mine scheduling tools. Items imported from the Ausenco models are shown in Table 16-14. Framework details for the open pit block models are provided in Table 16-15. The final open pit mine planning model items are displayed in Table 16-16.

**Table 16-14: Imported Model Items**

| Field Name | Min | Max | Precision | Units | Comments |
|---|---|---|---|---|---|
| CuT | 0 | 9.9 | 0.001 | % | Total copper grade |
| CuAS | 0 | 9.9 | 0.001 | % | Acid-soluble copper grade |
| CuCN | 0 | 9.9 | 0.001 | % | Cyanide-soluble copper grade |

| Field Name | Min | Max | Precision | Units | Comments |
|---|---|---|---|---|---|
| CLASS | 0 | 9 | 1 | - | Resource category where 1=Measured; 2=Indicated; 3=Inferred |
| MINZN | 0 | 9 | 1 | - | Mineralization type |
| LITH | 0 | 9 | 1 | - | Lithology domain |
| SG | 0 | 0.99 | 0.001 | Tons/CUY | Density |

**Table 16-15: Open Pit Model Framework**

| Framework Description | Cactus West Open Pit Model (Value) | Stockpile Area Open Pit Model (Value) |
|---|---|---|
| MinePlan file 10 (control file) | CACT10.dat | STKP10.dat |
| MinePlan file 15 (model file) | CACT15.pln | STKP15.pln |
| X origin (ft) | 385,900 | 387,000 |
| Y origin (ft) | 60,800 | 55,000 |
| Z origin (ft) (min) | -1000 | 1345 |
| Rotation (degrees clockwise) | 0 | 0 |
| Number of blocks in X direction | 455 | 244 |
| Number of blocks in Y direction | 405 | 300 |
| Number of blocks in Z direction | 150 | 20 |
| X block size (ft) | 20 | 25 |
| Y block size (ft) | 20 | 25 |
| Z block size (ft) | 20 | 10 |

**Table 16-16: Resource Model Item Descriptions (items are the same in both models)**

| Field Name | Min | Max | Precision | Units | Comments |
|---|---|---|---|---|---|
| CuT | 0 | 9.9 | 0.001 | % | Total copper grade |
| CuAS | 0 | 9.9 | 0.001 | % | Acid-soluble copper grade |
| CuCN | 0 | 9.9 | 0.001 | % | Cyanide-soluble copper grade |
| CLASS | 0 | 9 | 1 | - | Resource category where 1=Measured; 2=Indicated; 3=Inferred |
| MINZN | 0 | 9 | 1 | - | Mineral domain where 1=Leached; 2=Oxide; 3=Enriched; 4=Hypogene; 5=Basement; 6=Overburden |
| LITH | 0 | 9 | 1 | - | Lithology domain |
| SG | 0 | 0.99 | 0.001 | Tons/CUY | Density |
| RCUTC | 0 | 9.9 | 0.001 | % | Recovered copper grade when tertiary-crushed |
| RCUPC | 0 | 9.9 | 0.001 | % | Recovered copper grade when primary-crushed |
| ACDTC | -99 | 99 | 0.1 | lbs/ton | Net acid consumption when material is tertiary crushed |

| Field Name | Min | Max | Precision | Units | Comments |
|---|---|---|---|---|---|
| ACDPC | -99 | 99 | 0.1 | lbs/ton | Net acid consumption when material is primary crushed |
| CFTC1 | -99 | 999 | 0.01 | $/ton | Block cash flow when tertiary crushed - $3.75/lb Cu |
| CFTC2 | -99 | 999 | 0.01 | $/ton | Block cash flow when tertiary crushed - $3.00/lb Cu |
| CFPC1 | -99 | 999 | 0.01 | $/ton | Block cash flow when primary crushed - $3.75/lb Cu |
| CFPC2 | -99 | 999 | 0.01 | $/ton | Block cash flow when primary crushed - $3.00/lb Cu |
| MINE1 | 0 | 9.99 | 0.001 | $/ton | Open pit mining cost estimate |
| SLPCD | 0 | 9 | 1 | - | Geotechnical zone code |
| OSA | 0 | 90 | 0.1 | Degrees | Overall slope angle for pit limits analysis |
| IRA | 0 | 90 | 0.1 | Degrees | Inter-ramp angle for pit design |
| FACE | 0 | 90 | 0.1 | Degrees | Face slope angle for pit design |
| BERM | 0 | 99 | 0.1 | ft | Catchment berm width |

### 16.4.2 Economic Pit Shell Development

The open pit ultimate size and phasing requirements were determined with various input parameters including estimates of the expected mining, processing and G&A costs, as well as metallurgical recoveries, pit slopes and reasonable long-term metal price assumptions. AGP worked together with ASCU personnel to select appropriate operating cost parameters for the proposed Cactus West open pit. The mining costs are estimates based on cost estimates for equipment from vendors and previous studies completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various dump locations. Process costs and a portion of the G&A costs were provided by Samuel and ASCU based on preliminary costing results.

The parameters used are shown in Table 16-17. The net value calculations are in US$ unless otherwise noted. The mining cost estimates are based on the use of 100-ton trucks using an approximate WRSF configuration to determine incremental hauls for mineralized material and waste.

**Table 16-17: Pit Shell Parameter Assumptions**

| Description | Units | Value | Copper Value |
|---|---|---|---|
| Resource Model | | | |
| Block classification used | | M+I | |
| Block Model height | ft | 20 | |
| Mining Bench height | ft | 20 | |
| Metal Prices | | | |
| Price | $/lb | | 3.75 |
| Royalty | % | | 2.54% |
| Refining, Transportation Terms | | | |

| Description | Units | Value | Copper Value |
|---|---|---|---|
| Product Grade | | LME cathode | |
| Payable | % | 100% | |
| Selling Costs | $/lb | 0.04 | |
| SX/EW Cost | $/lb | 0.23 | |
| Metallurgical Information – Leach Cap and Oxide Alteration Zones | | | |
| Recovery of Acid-Soluble Copper – Primary Crush | % | 82 | |
| Recovery of Cyanide-Soluble Copper – Primary Crush | % | 41 | |
| Recovery of Acid-Soluble Copper – Tertiary Crush | % | 91 | |
| Recovery of Cyanide-Soluble Copper – Tertiary Crush | % | 55 | |
| Bulk Acid Consumption | Acid lbs/ton ore | 22 | |
| Acid Production | lbs/lbs Recovered Cu | 1.54 | |
| Metallurgical Information – Enriched Alteration Zones | | | |
| Recovery of Acid-Soluble Copper – Primary Crush | % | 85 | |
| Recovery of Cyanide-Soluble Copper – Primary Crush | % | 74 | |
| Recovery of Acid-Soluble Copper – Tertiary Crush | % | 94.3 | |
| Recovery of Cyanide-Soluble Copper – Tertiary Crush | % | 89.6 | |
| Bulk Acid Consumption | Acid lbs/ton ore | 21 | |
| Cost Information | | | |
| Mining Cost * | | Ore | Waste |
| Mining Cost base rate – 1400' elevation | $/t | 2.12 | 2.62 |
| Incremental rate - above | $/t/20 in bench | 0 | 0 |
| Incremental rate - below | $/t/20 in bench | 0.0127 | 0.0127 |
| Processing Costs include Base Cost + Net Acid Consumption | | | |
| Processing Base Cost – Tertiary Crush – Oxide/Leached | $/ton leach | $0.48 | |
| Processing Base Cost – Tertiary Crush – Enriched | $/ton leach | $1.20 | |
| Processing Base Cost – Primary Crush – Oxide/Leached | $/ton leach | $0.48 | |
| Processing Base Cost – Primary Crush – Enriched | $/ton leach | $1.20 | |
| Acid Cost | $/lb acid | $0.08 | |
| General and Administrative Cost | | | |
| G&A cost | $/ton leach | $0.47 | |

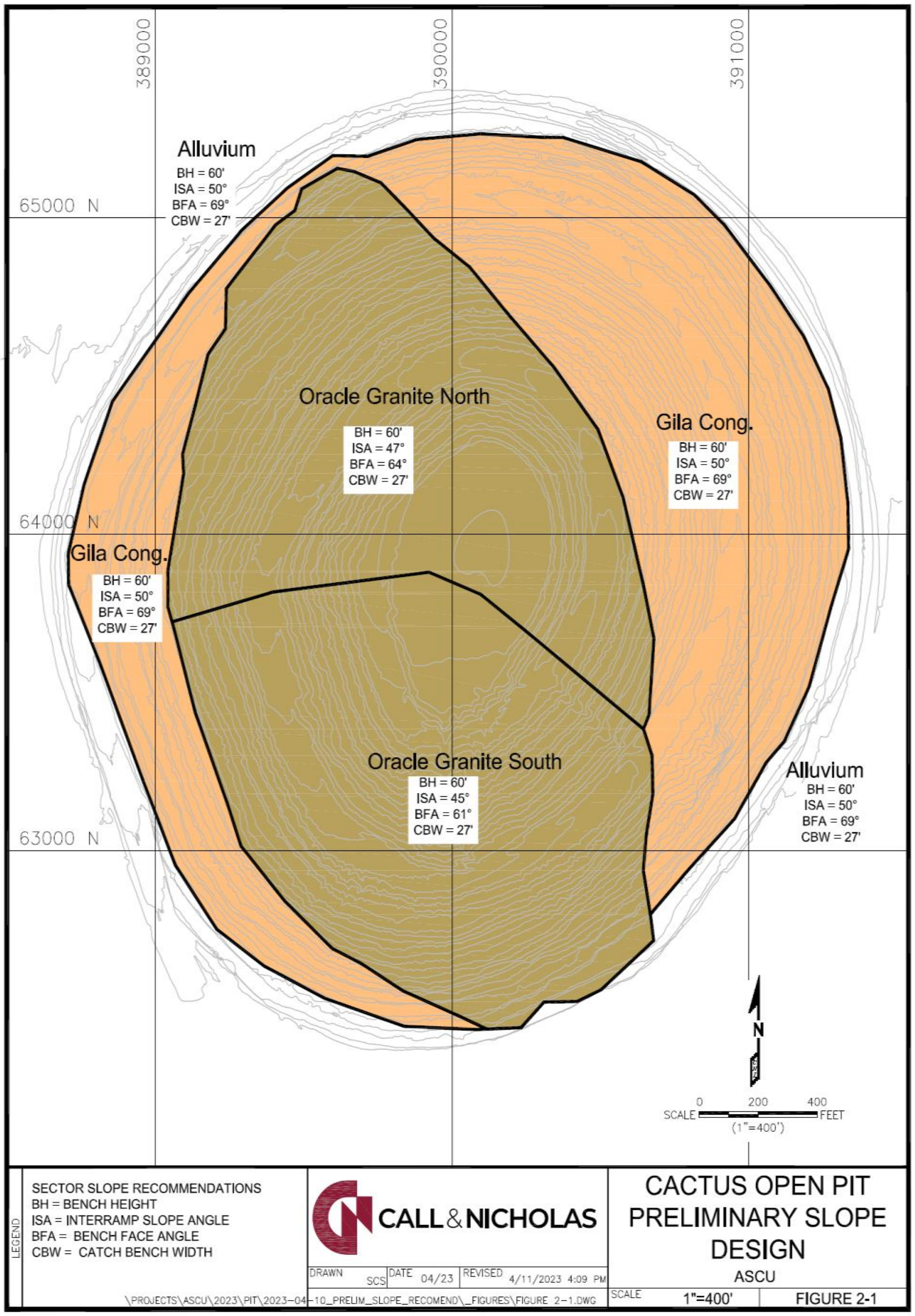
Note: * mining costs based on using 100-ton haul trucks. ** process costs based on 22 Mt/y dry throughput

Wall slopes for pit optimization were based on guidance from CNI. A design sector map was created which was defined by structural domains and dominant geotechnical units, as shown in Figure 16-25. Solids were used to code the model SLPCD item, then overall slopes were applied by code as shown in Table 16-18. Subsequent to the pit limits analysis and pit shell selection, a revision to the geotechnical guidance was provided which resulted in an adjustment to portions of the north wall in granite. These adjustments are reflected in Section 16.4.4 which discusses the pit design parameters.

**Table 16-18: Pit Shell Slopes**

| Structural Domain | SLP Code | Azimuths | | Angle (°) |
|---|---|---|---|---|
| | | Start (°) | End (°) | |
| Alluvium | 1 | All orientations | 30 | |
| Gila Conglomerate | 2 | All orientations | 45 | |
| Granite North | 3 | 260 | 135 | 42 |
| Granite South | 4 | 135 | 260 | 40 |

**Figure 16-25: Geotechnical Pit Domains for Pit Optimization**



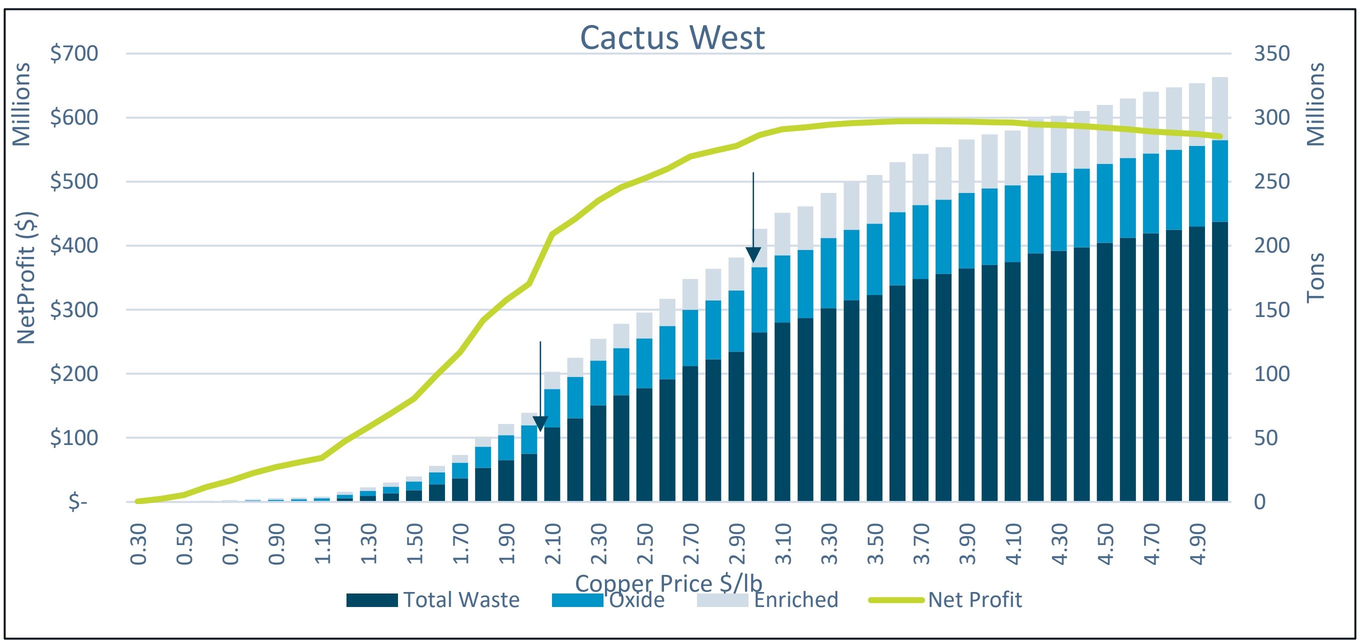

Source: CNI, 2023.

Nested L–G pit shells were generated to examine sensitivity to the copper prices with a target of US$3.70/lb Cu. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Measured and Indicated resources were used in the analysis. The nested pit shells were run using copper price factors to US$5.00/lb Cu at US$0.10/lb Cu increments. The resulting nested pit shells assist in visualizing natural breakpoints in the deposit and selecting shells to act as design guidance for phase design. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell. The pit shells were restricted to the Cactus West part of the deposit and not allowed to value the deeper Cactus East resource. The inclusion of the Cactus East resource, which generates a pit at higher copper prices would have skewed the nested pit analysis. Leach material, waste tonnages, and potential net profit were plotted against the copper price and are displayed in Figure 16-26.

Figure 16-26 shows distinct break points in the pit shells. These were used as a guide for sequencing pit phase designs. The first break point shown at US$2.00/lb Cu is selected for the initial phase This break point represented 57% of the net value of the US$3.70/lb pit, includes 33% of the leach ore and only 22% of the waste of the larger pit shell. The cumulative waste tonnage is 37.6 Mton, with a corresponding leach ore tonnage of 31.9 Mton or a strip ratio of 1.2:1.

The second and pit shell selected for the ultimate pit is at US$2.90/lb Cu. This US$2.90/lb Cu break point represented 94% of the net value of a US$3.70/lb pit but with only 67% of the waste of the larger pit shell. Limited potential pit value was available beyond this pit shell to cover the associated costs of developing another phase. The cumulative waste tonnage is 79.7 Mton, with a corresponding leach ore tonnage of 41.5 Mton or a strip ratio of 1.9:1. A visual check of the selected pit shells shows sufficient mining room exists between the selected phase shells.

**Figure 16-26: Cactus West Net Profit vs. Price by Pit Shell**




Source: AGP, 2023.

### 16.4.3 Dilution

The Mineral Resource block model for the Cactus Project has block dimensions of 20 ft (6.1 m) x 20 ft (6.1 m) x 20 ft (6.1 m). This was determined to be an acceptable selective mining unit for the project under the assumption that mining will be undertaken with relatively small equipment and the nature of the ore body and cut-off grades is such that ore-waste contacts are relatively infrequent and gradational in nature. As a result of low processing costs and good recoveries, cutoff grades are low and there is a relatively low penalty cost to any dilution incurred. Given this dynamic it was determined that the Mineral Resource block model is suitable for use in stating Mineral Reserves without any secondary factors for ore loss or dilution applied.

In the Stockpile area, there is a very low ratio of internal waste in the model, and as such it was similarly determined that no secondary dilution or ore loss was required.

### 16.4.4 Pit Design

Open pit designs were completed in MinePlan software according to geotechnical design parameters provided by Call and Nicholas, with design assumptions for road and minimum mining widths provided by AGP. The geotechnical design parameters employed are described in Table 16-19 and shown in Figure 16-27. Haul road widths were selected to accommodate 150-ton class trucks with a two-way design width of 100 ft (30.5 m) applied to the pit designs. Minimum mining widths of approximately 100 ft (30.5 m) were employed in certain areas of the pit, but typical mining widths range from 150 ft (45.2 m) to greater than 500 ft (152.4m).

The Phase 1 pit design incorporates a primary haul ramp which exists the pit on the south pit rim proximal to the crushers. The ramp runs down the south and west wall before switching back on the north wall. Ramps were positioned on the west walls due to the geometry of the ore body which dips at a flatter angle than the pit slopes in this area. On the north and north-east sides of the pit, mining widths are minimized, targeting only a narrow piece of high-grade ore at depth. There is a section of ramp near the pit bottom where a small amount of backfill will be required to establish the permanent ramp across the historically mined pit bottom.

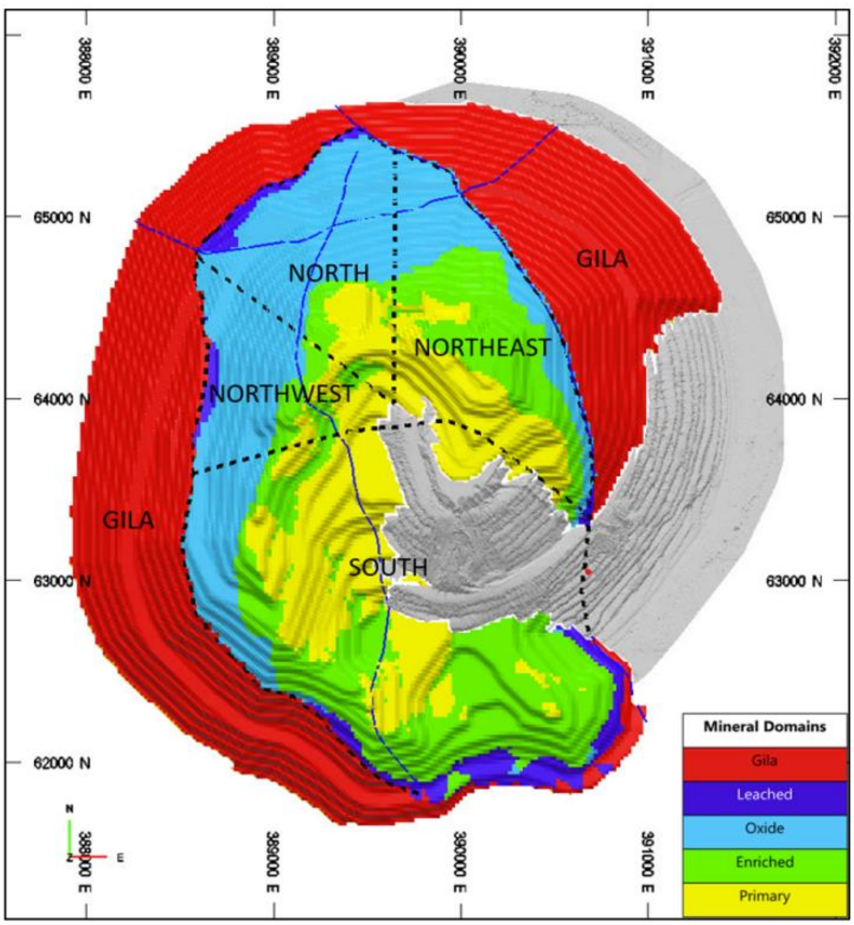
The Phase 2 pit design incorporates a very similar ramp configuration to Phase 1, pushing the pit back to the west and south. Phase 2 also requires the use the same backfill ramp to facilitate truck access across portions of the Phase 1 and historically mined open pit areas.

The potential for underground mining of Cactus East to destabilize the east and north-east sections of the pit wall was also considered in the ramp placement for both Phase 1 and Phase 2. The current pit designs do not incorporate any ramps on the east or northeast sides of the pit except when accessing the bottom two benches which carry minor value relative to the total pit inventory (less than 1% of total ore tons). It is expected that timing of respective open pit and underground operations and geotechnical slope monitoring and roll-out protection measures should enable the entire pit inventory to be extracted. Phase 1 which mines material on the east and northeast sides of the pit is completed by early Year 4, and Phase 2 pit is completed in Year 6, which is two years before Cactus East development begins.

Table 16-19: Open Pit Slope Design Parameters

| Design Sector | Slope Azimuth | | Pit Design Geometry | | | |
|---|---|---|---|---|---|---|
| | | | Face Height | Face Angle | Berm Width | Inter-Ramp Angle |
| | Start (°) | End (°) | Bh (ft) | Da (°) | Dw (ft) | Ia (°) |
| Alluvium | All | - | 69 | 27 | 50 | |
| Gila Conglomerate | All | - | 69 | 27 | 50 | |
| Granite North | 310 | 360 | 60 | 61 | 27 | 45 |
| Granite South | 135 | 260 | 60 | 61 | 27 | 45 |
| Granite NE | 260 | 310 | 60 | 64 | 27 | 47 |
| Granite NW | 0 | 135 | 60 | 64 | 27 | 47 |

Figure 16-27: Pit Design Slope Sectors



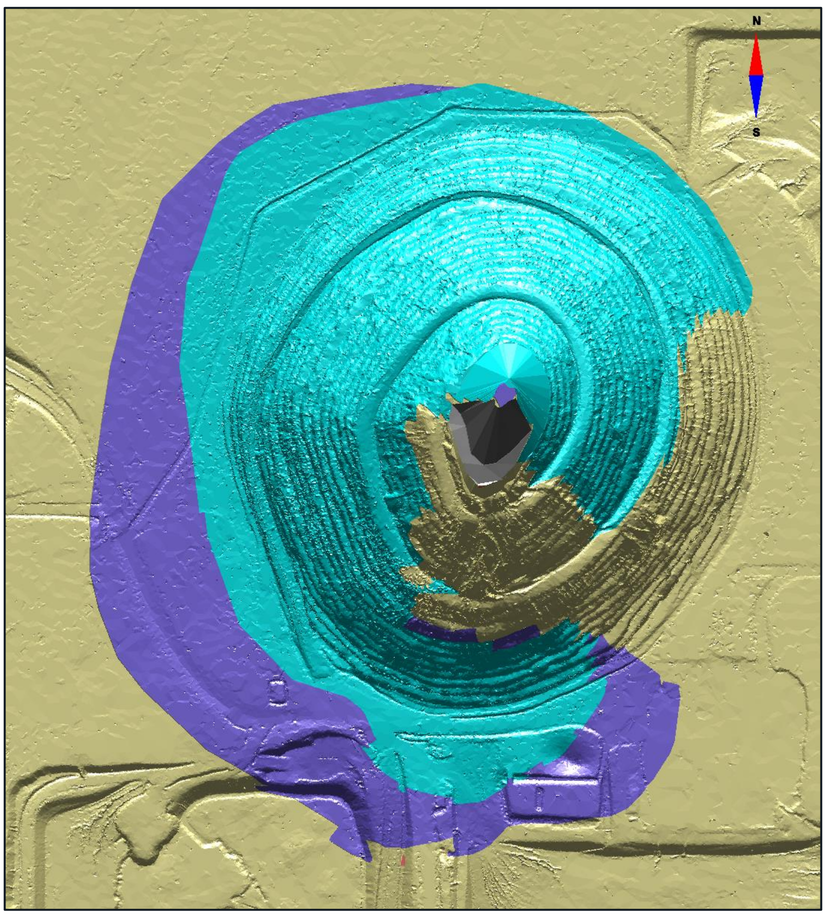

Source: Call and Nicholas, 2023.

Tons and grade for the Cactus West phase designs and the Stockpile Project are reported in Table 16-20. Only Measured and Indicated Mineral Resources were included in the leach ore inventory. The Cactus West phases, and sequence are shown in Figure 16-28.

**Table 16-20: Cactus West Phase and Stockpile, Tons, and Grade**

| Phase | Leach Ore (Mton) | TCU (%) | CUAS (%) | CUCN (%) | Waste (Mton) | Total (Mton) | Strip Ratio (w:o) |
|---|---|---|---|---|---|---|---|
| CW-PH1 | 33.0 | 0.327 | 0.147 | 0.137 | 63.0 | 96.0 | 1.9 |
| CW-PH2 | 42.5 | 0.291 | 0.124 | 0.116 | 79.4 | 121.9 | 1.9 |
| CW -Total | 75.5 | 0.307 | 0.070 | 0.065 | 142.4 | 217.9 | 1.9 |
| Stockpile | 76.8 | 0.163 | 0.112 | 0.024 | 5.5 | 82.2 | 0.1 |

**Figure 16-28: Cactus West Mining Phases**

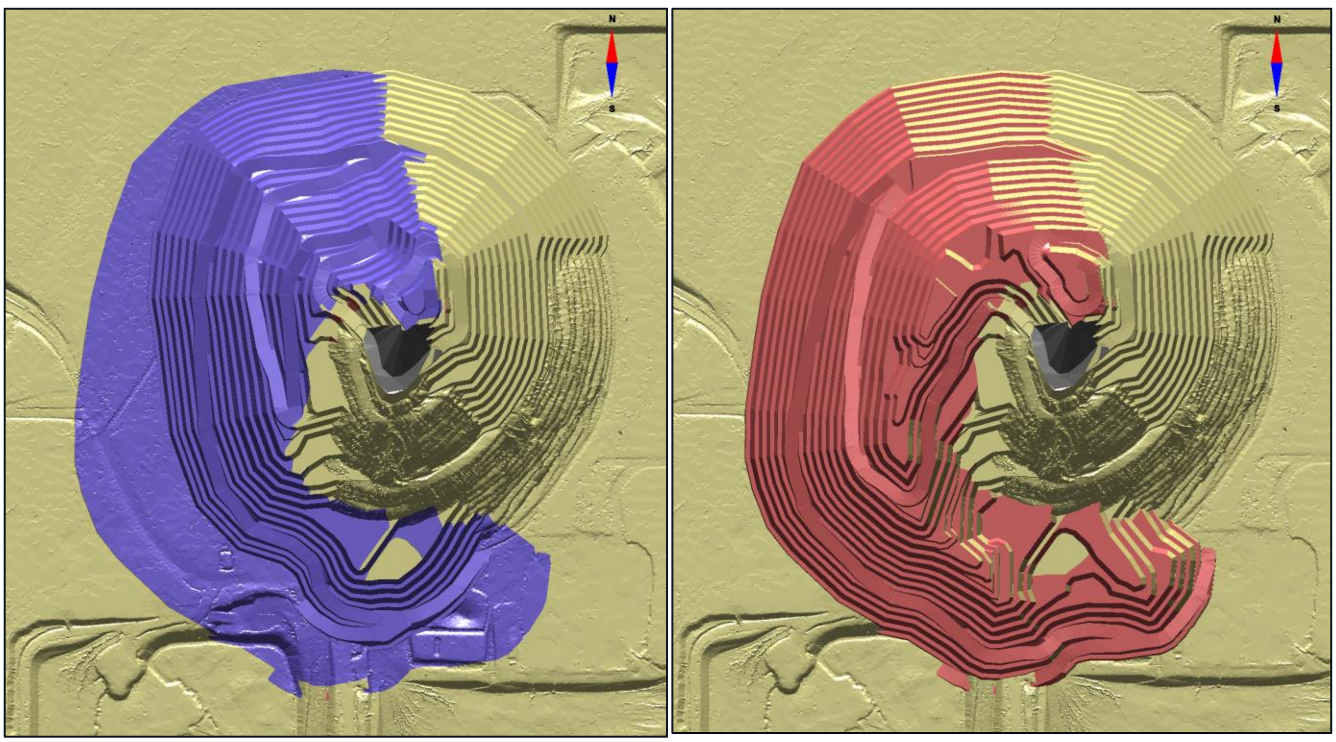



Source: AGP 2023

Phase 1, shown in Figure 16-29 (in purple), is located west and south of the existing pit and will start mining during the pre-production period, Year-1. Bench elevations range from the 1,440-ft (439 m) level down to 380-ft (116 m) level with the primary ramp access located on the south side of the pit near the existing stockpile project and ore crushers. Waste material will be routed via the primary access to the view shed berms and Northeast waste rock facility although some waste material in the upper benches may be direct hauled to the view shed berm from the north end of the phase. Ore mined and not processed will be sent to the stockpile for processing in later years. Mining in Phase 1 is completed in Year 4.

Phase 2, shown in Figure 16-29 (in red), extends the pit to south and west of Phase 1. Bench elevations range from 1,440 ft (439 m) to 380 ft (116 m). Mining begins in Year 3 and Phase 2 is mined out in Year 7. Similar to the initial phase, the access ramp exits the pit on the south side of the pit near the ore crushers and stockpile project.

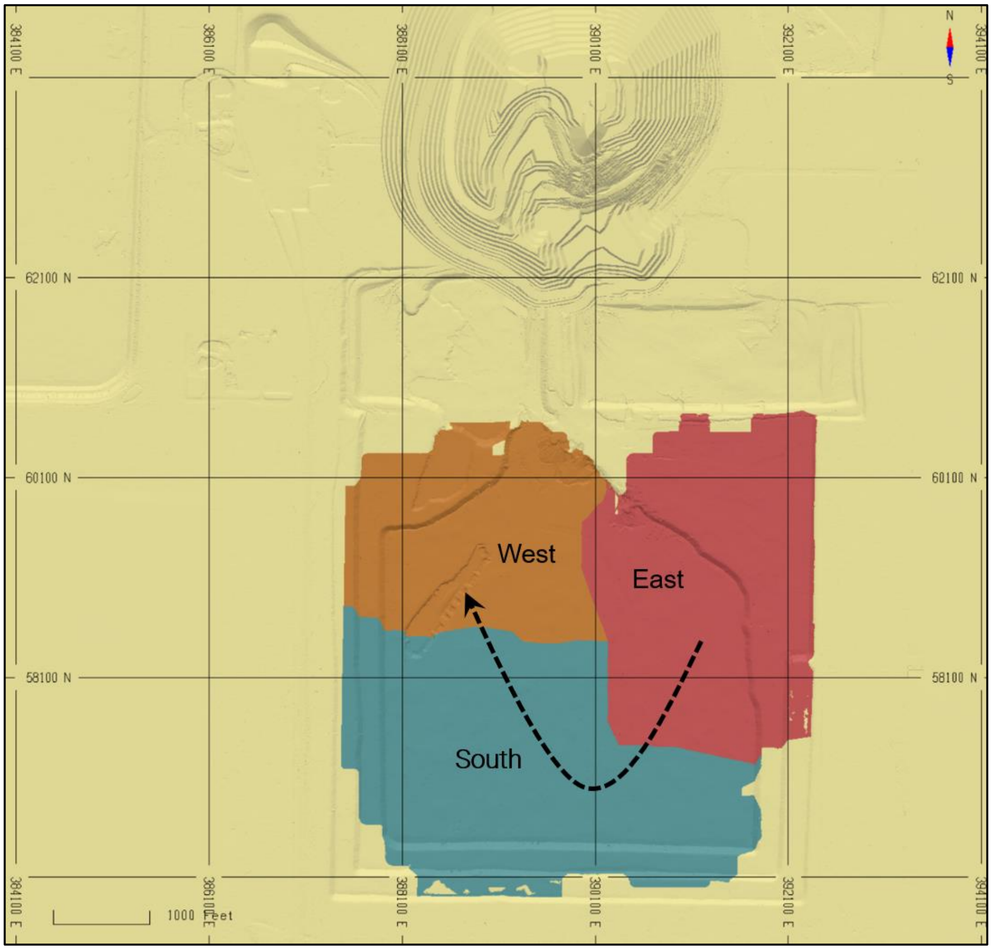
**Figure 16-29: Cactus West After Phase 1, Phase 2**



Source: AGP 2023.

The historic stockpile was divided into three phases for mining: the east phase, south phase, and west phase. Mining starts in the east phase followed by south phase with the west phase mined last. This stockpile mining sequence was chosen due to higher average grades mined upfront and to make room for construction of the heap leach pad. Mining the stockpile in this order is required to ensure space is available for leach pad construction. The stockpile mining sequence is shown in Figure 16-30.

**Figure 16-30: Stockpile Mining**




Source: AGP 2023.

### 16.4.5 Cutoff Grade Calculations

Cutoff grade decisions on the mine schedule are based on a block value calculation, which is effectively a net-smelter return with expected processing, G/A, and royalty cost removed. This block value calculation employed all the same assumptions as outlined in Table 16-21, including a $3.70/lb copper price. The cutoff block value employed was a marginal cutoff grade of $0/t, meaning that any block which would generate a net positive value was either processed on the heap leach or placed into stockpiles. This cutoff calculation does not include mining costs which are considered sunk for the purpose of cut-off grade determination.

Cutoff grade determinations consider the projected recoveries generated by each block. These recoveries are strongly influenced by the proportion of each type of copper speciation present in the block (acid-soluble, cyanide-soluble, and non-soluble). As such, it is not possible to state a generic cutoff grade for the mine schedule, as the ratio of copper speciation is variable resulting in variable cutoff grades for different materials. As a reference point, for the typical copper speciation in oxide and enriched materials, the cutoff grades are approximately 0.050% and 0.055% acid-soluble plus cyanide-soluble copper, respectively.

Over the course of the open pit mine schedule, approximately 11.8 Mton of low-grade ore is stockpiled and reclaimed in order to smooth the ore release from the open pits. This amount includes approximately 2.4 Mton of material stockpiled in the first three years of mining, and then processed in Year 3, and another 9 Mton stockpiled later in the mine schedule before being reclaimed in Years 6 and 7.

### 16.4.6 Waste Rock Facilities

Waste materials generated from mining Cactus West and the Stockpile areas will be composed of predominantly Gila Conglomerate and Alluvium overburden (80%) with the remainder being granite and porphyry rock with lower copper grades, or unfavourable metallurgical characteristics (including 4 Mton of waste bearing primary copper mineralization).

No waste segregation is required in the mine schedule aside from the stockpiling of primary copper mineralization, and as such, different waste types can be placed into any of the available waste facilities as required by scheduling and fleet optimization constraints.

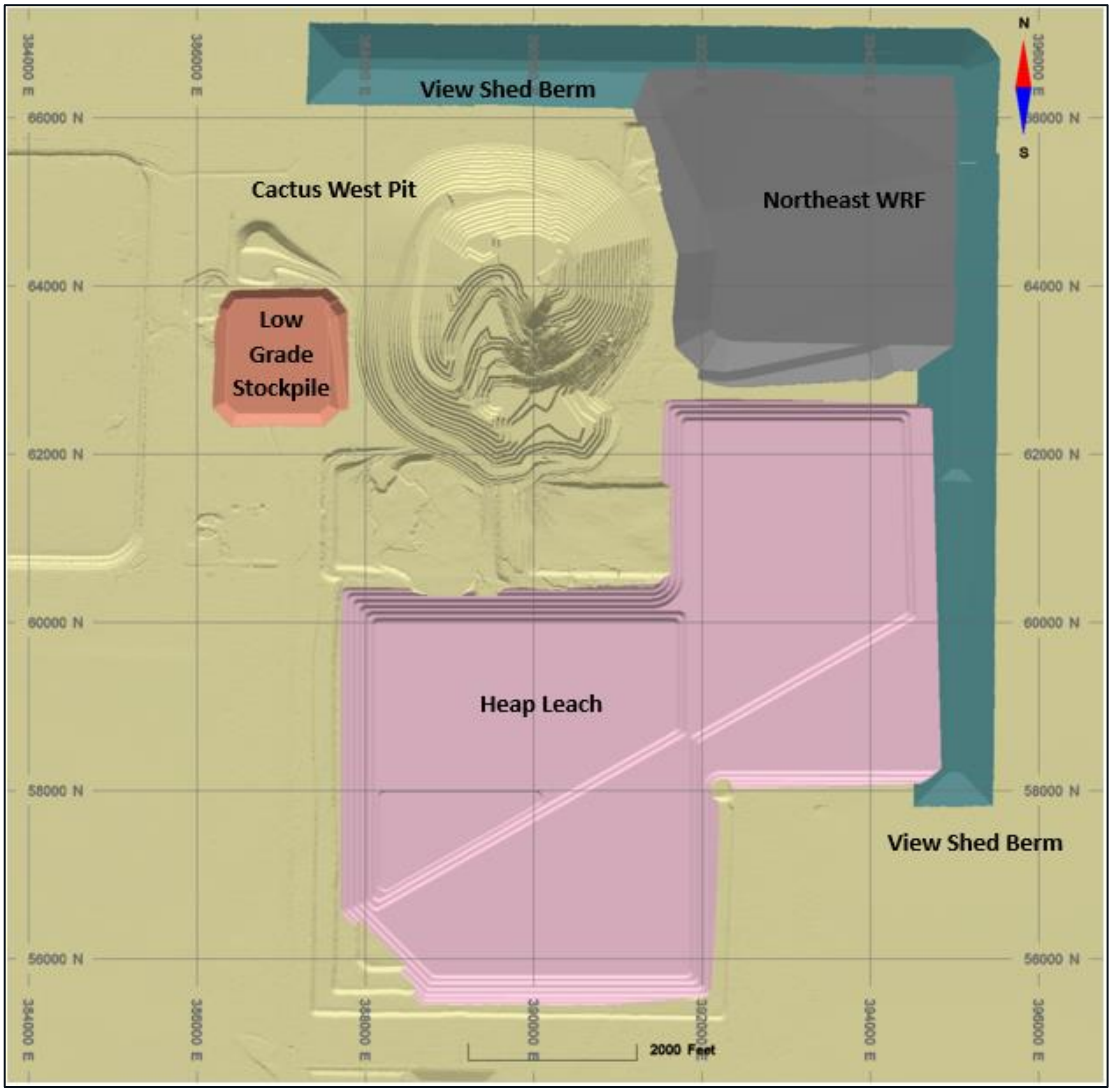
Four primary waste rock storage facilities are employed in the mine schedule, including a view Shed Berm, the North-East WRF, and assorted construction fills as required for initial and ongoing heap leach construction. A small, segregated area for primary copper mineralized waste will be left on the edge of the North-East WRF to enable future access to this material should a processing method become available in the future.

The View Shed Berm is a long narrow waste facility which wraps around the East and North sides of the property to a maximum height of 125 ft (38.1 m). The purpose of this structure is to hide the mine infrastructure from communities to the east and north, with the outer slopes reclaimed. This berm will also serve to mitigate noise levels from the mine. A separate constraint on the height of the View Shed Berm is a flight path for a local airport to the east of the mine. The height of the berm has been reduced at its southern end to respect relevant regulations.

The North-East WRF will fill in the space between the View Shed Berm and the Cactus West open pits, allowing for a 250 ft (76.2 m) offset to the pit rim and respecting the footprint of the heap leach facility and haul roads to the south. The dump will be constructed in 20-ft (6.1 m) high lifts. Portions of this WRF overly the Cactus East Sublevel Cave footprint and will be subject to future subsidence. The areas subject to this subsidence will be constructed as quickly as possible following the View Shed Berm to utilize the shorted available hauls and to ensure adequate safe dumping locations are available once subsidence of the area has begun.

Some of the mine waste, primarily consisting of alluvium from the pre-strip period of the mine will be utilized to complete construction fills in support of the heap leach facilities. Figure 16-31 shows the final configuration of the View Shed berm, NE waste rock facility and the Heap Leach.

**Figure 16-31: Waste and Heap Leach Facilities**




Source: AGP, 2023.

### 16.4.7 Mine Equipment Selection

The mining equipment selected to meet the required production schedule is conventional mining equipment, with additional support equipment for site maintenance.

Primary production drilling will be completed with six down the hole hammer (DTH) drills using 6 ¾ in bits. This will provide the capability to drill patterns for either 20 ft (6.1 m) or 40 ft (12.2 m) bench heights. Two smaller drills using 5 ½ in bits will be utilized to perform wall control drilling in the form of buffer patterns.

Production mining will be completed with two 30 $yd^3$ hydraulic shovel, five 15 $yd^3$ loaders, and twenty-four 150-ton rigid body trucks. It is expected that the larger hydraulic shovels will be utilized in the Cactus West Pit, while the front-end loaders will support mining in the Stockpile area and supplement in Cactus West. Two of the front-end loaders and eight of the haul trucks will be utilized in the pre-production period to assist in heap leach and other construction activities.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements and is composed of 14-ft graders, track dozers, and assorted auxiliary fleet.

The proposed equipment requirements for the LOMP are included in Section 21.

### 16.4.8 Blasting and Explosives

Blasting will be undertaken using 6 ¾ in blastholes on a 12 ft (3.6 m) x 14 ft (4.3 m) pattern spacing. Blasting will be performed on 20 ft (6.1 m) benches heights in ore and mixed materials and using 40 ft (12.2 m) bench heights in waste and overburden areas. Bulk explosives are expected to be 80% ANFO and 20% emulsion, with emulsion potentially required in deeper benches if water is encountered. Three rows of smaller 5 ½ in buffer holes will be drilled around pit wall contacts to minimize blast damage.

### 16.4.9 Grade Control

Grade control assaying will be performed using cuttings from production blastholes. A cost allowance has been included which assumes that approximately 100% of the blastholes in ore areas and 25% of blastholes in waste areas will be assayed. Assaying will be completed at an offsite lab, with the results used to generate ore control polygons which will be surveyed in the field to guide mine operations. Assaying will include sequential copper grades (acid-soluble, cyanide-soluble, and total copper) in order to best model process performance. All ore from the Cactus West Open Pit is planned to be processed together after multi-stage crushing, while all ore from the historical Stockpile will be processed on the same heap leach in segregated areas after undergoing single-stage crushing. All ore will be placed on the leach facilities by stacking conveyors.

## 16.5 Underground Mining Operations

### 16.5.1 Introduction

As part of the initial phase of the Pre-Feasibility Study AGP undertook a high-level review of underground mining options which included, sublevel open stoping, room and pillar, inclined caving, block caving and the SLC method.

Sublevel caving was selected as the preferred underground mining method for the Cactus East and Parks/Salyer deposits. The mine designs are based on geotechnical recommendations and ore is recovered by blasting rings between sublevels in a staggered retreat direction towards the material handling system.

SLC is a common method with which design criteria such as drive configurations, ring designs, ramp up profiles, draw rate and flow behaviour are generally well understood and have been effective in achieving good productivity and ore recovery in mine operations worldwide.

The SLC method commences close to the top of the orebody and is mined by drilling and blasting a ring pattern between sublevels horizons. At Cactus East and Parks/Salyer the overburden between the top of the orebody and the surface is expected to cave naturally in response to the ore volume being extracted, thereby supplying the waste fill into the mine. A subsidence zone (crater) will form and expand at the surface in response to continued mining.

### 16.5.2 Cutoff Grade

The footprint delineations for the Cactus East and Park Salyer mines were based on a resource model block cash flow dollar value (CFTC1) of $27.62 (net of process, G/A and royalties) assuming a copper price of $3.70/lb. The drawpoints were shut-off when the grade value of the drawn material falls below a CFTC1 value of $27.62. There is further opportunity of optimising the shut-off value based on a Hill of Value and NPV study in later stages.

Breakeven cut off grades (values) using the study mine operating and sustaining capital cost estimates for both mines are summarised in Table 16-21.

**Table 16-21: Summary of Break-even Cutoff Analysis**

| Description | Unit | Value | Cactus East | | Park Salyer | |
|---|---|---|---|---|---|---|
| | | | Oxide | Enriched | Oxide | Enriched |
| **Insitu Cut-Off Grade** | | | | | | |
| Total Soluble Cu | % Tsol | | 0.60 | 0.59 | 0.45 | 0.46 |
| Copper Grade CuAS | % CuAS | | 0.53 | 0.09 | 0.43 | 0.04 |
| Copper Grade CuCN | % CuCN | | 0.06 | 0.50 | 0.02 | 0.42 |
| Contained Metal Value | US$ | | 44.14 | 43.64 | 33.32 | 34.27 |
| Mining Dilution | % | | 21.0 | 21.0 | 21.0 | 21.0 |
| **Ore To Process Plant** | | | | | | |
| Copper Grade | % Tsol | | 0.50 | 0.50 | 0.38 | 0.39 |
| | % CuAS | | 0.45 | 0.08 | 0.37 | 0.04% |
| | % CuCN | | 0.05 | 0.42 | 0.01 | 0.3 |
| Contained Metal Value | US$ | | 37.12 | 36.70 | 28.02 | 28.83 |
| **Metal Price** | | | | | | |
| Copper | US$/lb | | 3.70 | 3.70 | 3.70 | 3.70 |
| REVENUES | | | | | | |
| Recovery of CuAS | % of CuAS | | 91.0 | 94.3 | 91.0 | 94.3 |
| Recovery of CuCN | % of CuCN | | 55.0 | 89.6 | 55.0 | 89.6 |
| Total Cu Recovered | lbs Cu | | 8.74 | 8.96 | 6.80 | 7.01 |

| Description | Unit | Value | Cactus East | | Park Salyer | |
|---|---|---|---|---|---|---|
| | | | Oxide | Enriched | Oxide | Enriched |
| Gross Metal Value | US$ | 99.9% | 32.32 | 33.12 | 25.13 | 25.92 |
| **Downstream Charges** | | | | | | |
| Selling Cost | US$/lb recovered Cu | 0.04 | 0.35 | 0.36 | 0.27 | 0.28 |
| SXEW | US$/lb recovered Cu | 0.23 | 2.01 | 2.06 | 1.56 | 1.61 |
| COPPER CHARGES | US$ | | 2.36 | 2.42 | 1.84 | 1.89 |
| NET REVENUE | US$/ton processed | | 29.96 | 30.70 | 23.29 | 24.03 |
| **Operating Costs** | | | US$ | US$ | US$ | US$ |
| U/G Mining Costs Used | US$/ton Processed | | 23.16 | 23.16 | 19.52 | 19.52 |
| Sustaining Capital (Mining) | US$/ton Processed | | 4.79 | 4.79 | 1.93 | 1.93 |
| Surface Haulage | US$/ton Processed | | 0.30 | 0.30 | 0.30 | 0.30 |
| Crushing, Stack, Leach | US$/ton Processed | | 0.48 | 1.20 | 0.48 | 1.20 |
| Acid Consumption | US$/ton Processed | | 0.00 | 0.00 | 0.00 | 0.00 |
| G & A | US$/ton Processed | | 0.47 | 0.47 | 0.47 | 0.47 |
| Copper Royalty | Cu Net Revenue | 2.54% | 0.76 | 0.78 | 0.59 | 0.61 |
| **Total Operating Cost** | **US$/ton processed** | | **29.96** | **30.70** | **23.29** | **24.03** |

### 16.5.3 Application of Modifying Factors

Table 16-22 presents a summary of geotechnical design parameters for mine planning using a sublevel cave (SLC) mining method.

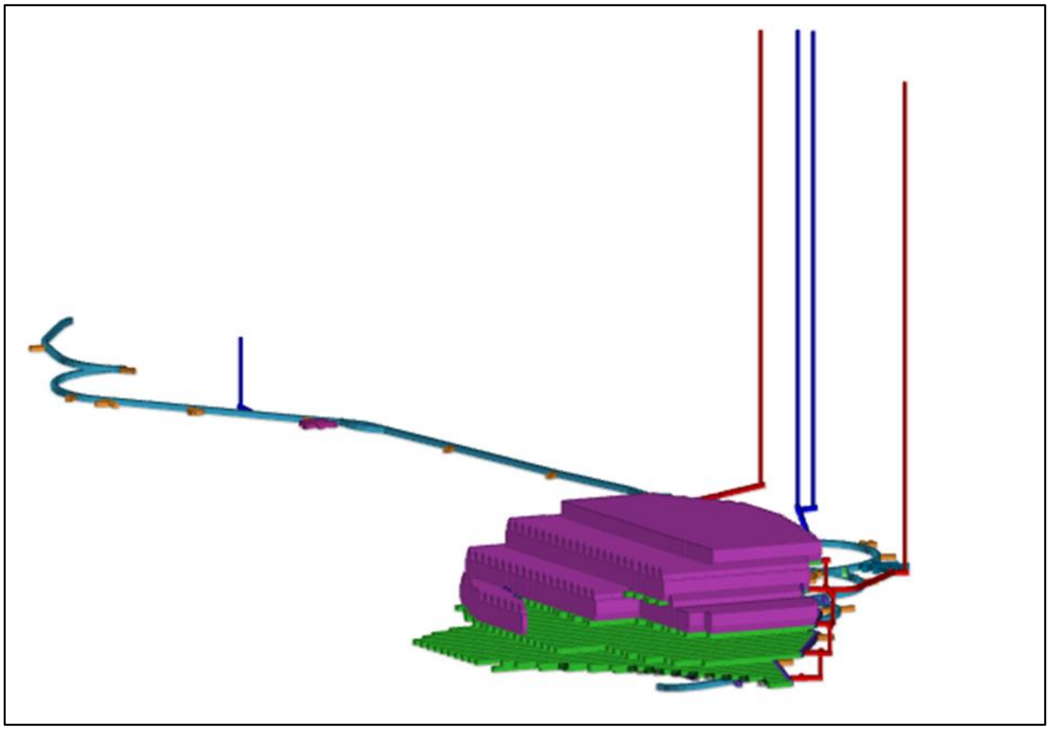
**Table 16-22: Sublevel Cave Mining Recommendations**

| Design Parameter | | Recommendation |
|---|---|---|
| Sublevel Vertical Spacing (ft) | PS | 65 – 80 |
| | CE | 80 |
| Sublevel Horizontal Spacing - Centerline to Centerline (ft) | | 45 |
| Sublevel Drift Width (ft) | | 16.5 |
| Distance from Nearest Brow to Ramp Access (ft) | PS | 120 |
| | CE | 100 |
| Vertical Echelon (Horizontal Distance between Vertical Faces on Adjacent Sublevels) | | > 50 |
| Horizontal Echelon (Horizontal Distance between Vertical Faces on Same Sublevel) | | 2 – 8 Burden Rings |
| Hydraulic Radius (m) for Caving | PS | 18 |
| | CE | 21 |
| Retreat Direction (Azimuth in Deg.) | PS | 180 |
| | CE | 335 |
| Max Panel Width (ft) | | 800 |
| Panel Transition Zone Thickness (ft) | | 74 - 119 |
| Total Draw to Recover 90% of Ore (% of ore tons) | | 135 |
| Subsidence Limits (Composite Angle in Deg.) | | 65 |

### 16.5.4 Underground Mining Design

The initial Cactus East SLC will commence at a depth of 1,325 feet below the surface and will consist of seven sub-levels, reaching a final depth of 1,845 feet. Access to the SLC will be facilitated through a single decline, with a portal situated within the existing Cactus West pit. Ore haulage to the surface will primarily utilize a vertical conveyor system, with the option to supplement it with truck haulage via the open pit if required. The final configuration for the Cactus East SLC mine is illustrated in Figure 16-32.

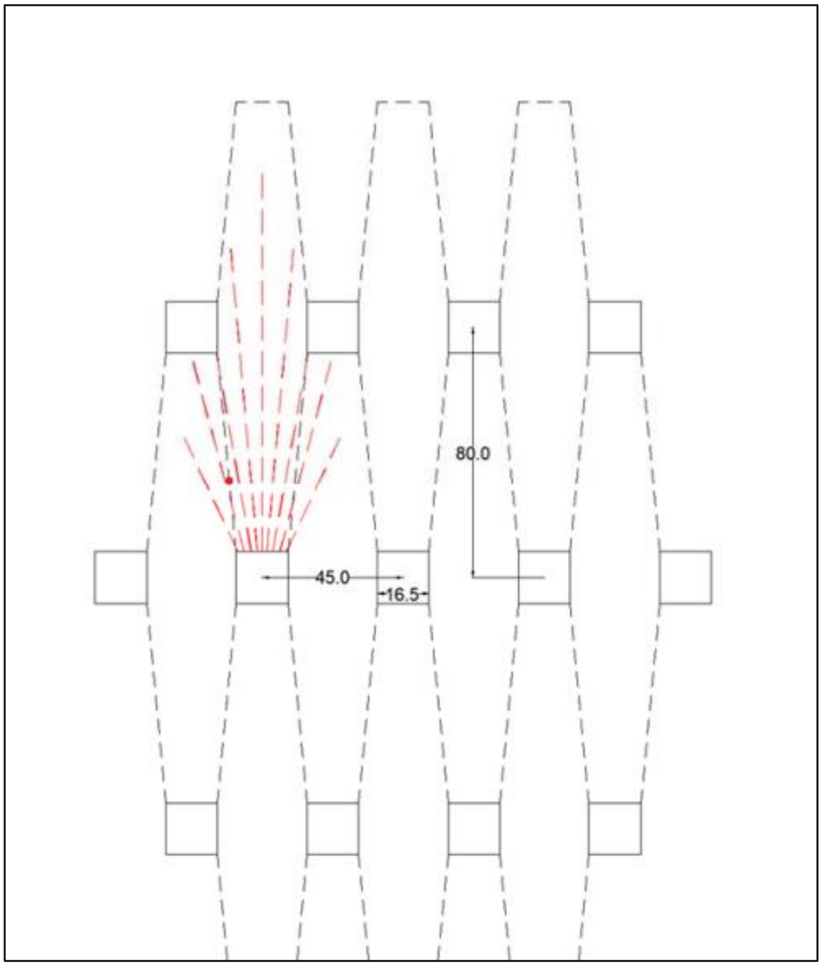
**Figure 16-32: Cactus East Mine**



Source: AGP, 2023

The design parameters for the SLC production drives at Cactus East are in line with standard practices in similar operations. Production crosscuts have been strategically designed to horizontally offset drives from the levels above and below, maximizing ore recovery. The production drives are 16.5 feet wide, spaced 45 feet (13.75 meters) apart, center to center, enhancing recoveries by accounting for expected finer fragmentation. The vertical separation between production levels is maintained at 80 feet (25 meters), consistent with other SLC mining operations. Typical production drive profile and level spacings are shown in Figure 16-33.

**Figure 16-33: General Arrangement – Section View**




Source: CNI, 2023.

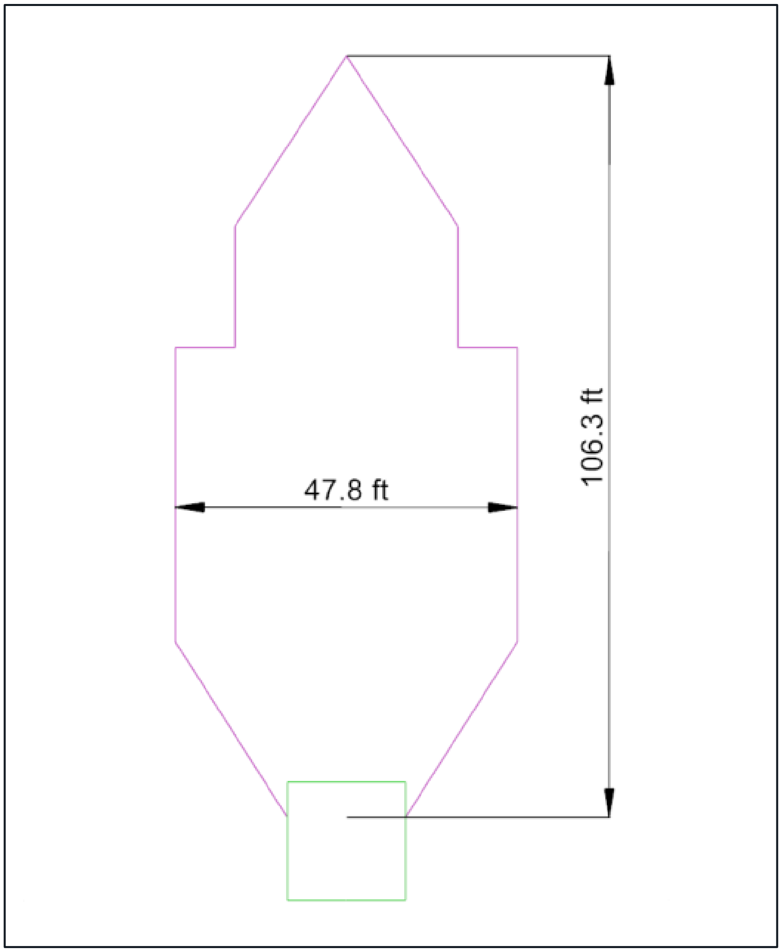
The production drives in Cactus East have predominantly been designed with each level horizontally offset from the levels both above and below. The typical practice is to advance the SLC face position on the upper level ahead of the SLC face position on the lower level. This is necessary to mitigate hazards related to drilling breakthroughs into the level above, manage the draw of ore and waste through the cave, and prevent the above level from being 'undercut,' which could lead to a loss of access. The sub-level caving advances primarily from south to north.

Typical SLC production ring parameters:

- 3.5” to 4” hole diameter
- 8.5ft ring burden (horizontal)
- 75° – 80° ring inclination

A standard SLC ring configuration is shown in Figure 16-34.

**Figure 16-34: SLC Ring Layout**




Source: AGP, 2023.

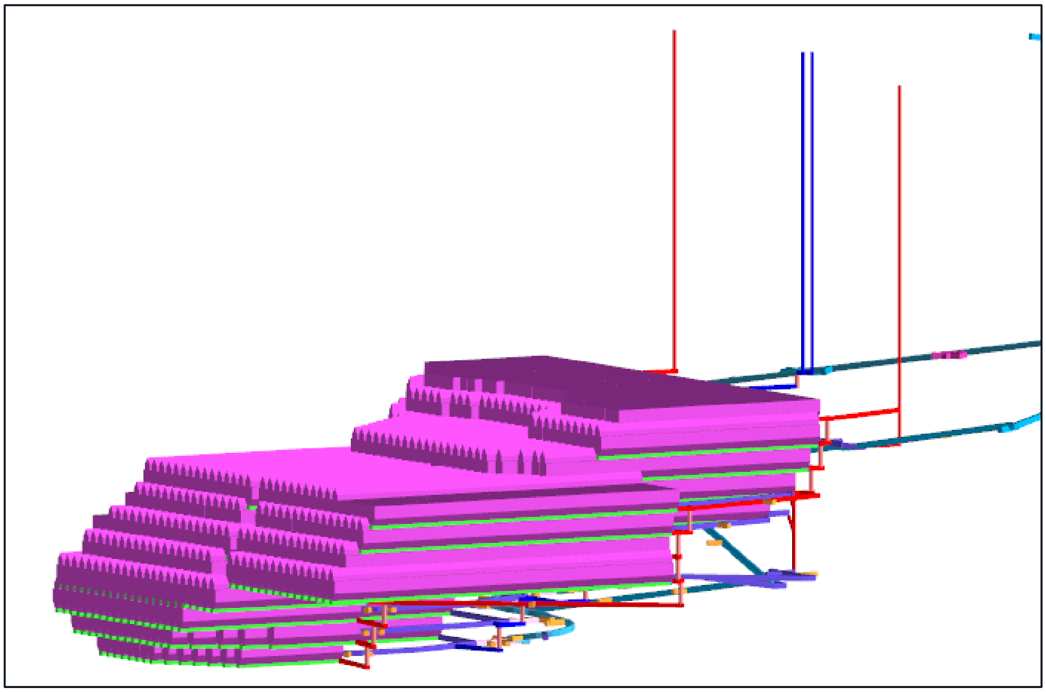
Table 16-23 lists the range of design profiles used in the underground mine design. The development profiles have been chosen based on minimising the excavation size to facilitate the selected mining method, whilst still being able to operate equipment of a size necessary to achieve required rates of productivity. The maximum decline gradient in the design was set to 15% to ensure that mobile equipment considered would be able to operate effectively throughout the mine.

**Table 16-23**: Development Drive Profiles

| Drive Type | Width (ft) | Height (ft) | Profile |
|---|---|---|---|
| Conveyor Decline | 18.0 | 20.0 | Arched |
| Crusher Ore Pass (Raisebore) | 16.5 | - | Circle |
| Crusher Reclaim Chamber | 18.0 | 24.0 | Arched |
| Conveyor Transfer Chamber | 20.0 | 60.0 | Arched |
| Decline | 18.0 | 20.0 | Arched |
| Permanent Electrical Bay | 16.5 | 16.5 | Arched |
| Escape Raise (Raisebore) | 8.0 | - | Circle |
| Escape Raise Access | 15.0 | 15.0 | Arched |
| Escape Raise (Drop Raise) | 12.0 | - | Circle |
| Finger Raise (Drop Raise) | 10.0 | - | Circle |
| Footwall Drive | 16.5 | 18.0 | Arched |
| Haulage Drive | 18.0 | 20.0 | Arched |
| Loader Crosscut | 16.5 | 18.0 | Arched |
| Level Access | 16.5 | 18.0 | Arched |
| Laydown | 16.5 | 16.5 | Arched |
| Magazine | 16.5 | 18.0 | Arched |
| Maintenance Bay | 16.5 | 18.0 | Arched |
| Production Drive | 16.5 | 16.5 | Square |
| Permanent Refuge | 16.5 | 16.5 | Arched |
| Remuck | 16.5 | 16.5 | Arched |
| Ramp | 18.0 | 20.0 | Arched |
| Sump | 16.5 | 20.0 | Arched |
| Truck Loadout | 18.0 | 20.0 | Arched |
| Transfer Raise (Raisebore) | 11.5 | - | Circle |
| Transfer Raise (Drop Raise) | 11.5 | - | Circle |
| Vent Raise (Raisebore) | 11.5 | - | Circle |
| Vent Raise Access | 15.0 | 15.0 | Arched |
| Vent Raise (Drop Raise) | 15.0 | - | Circle |
| Washbay | 16.5 | 20.0 | Arched |
| Crosscut | 18.0 | 20.0 | Arched |

Parks/Salyer has a similar configuration as Cactus East Mine which also utilises sub-level caving mining method to extract the ore from the underground. The initial Parks/Salyer SLC level will commence 1210 ft below surface and includes 11 sublevels, reaching a final depth below surface of 1930 ft. Access to the Parks/Salyer deposit will be via a surface portal and twin declines. One of the declines will be dedicated to ore haulage via an inclined conveyor while the other providing access for personnel and equipment. The final configuration for the Parks/Salyer mine is illustrated in Figure 16-35.

**Figure 16-35: Parks Salyer Mine**

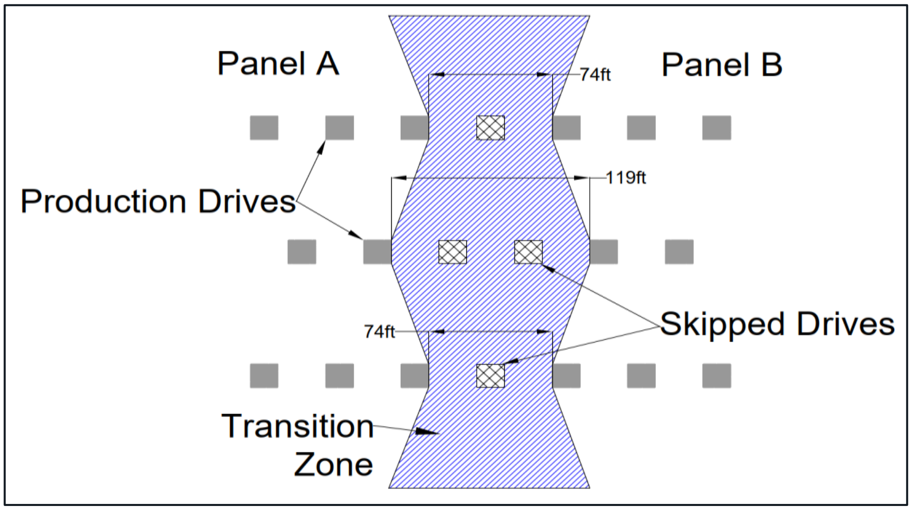


Source: AGP, 2023.

The design of Parks/Salyer development drives follows the same parameters as that of Cactus East wherein production drives are strategically designed to have horizontal offset from the levels above and below with 45 feet spacing in between drives. Consistent with usual SLC mining operations, the vertical spacing between levels is kept at 80 feet.

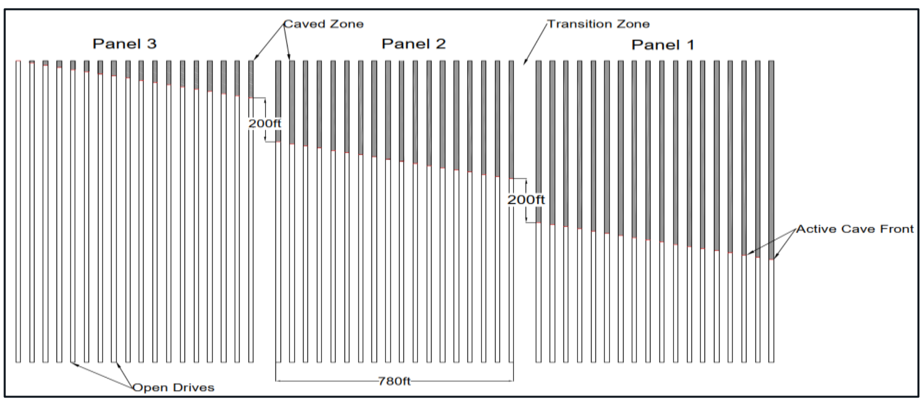
There are, however, some unique differences such as the large step-out distances between successive sublevels and the requirement to split the mining sequence into SLC panels due to the large size of the footprint at Parks/Salyer. As a result, the mine plan incorporates panels with widths around 800 feet, separated by "transition zones" where no sublevel drifts are developed. Figure 16-36 presents a lay-out of these transition zones. The potential recovery of material from these zones is considered, exploring the option of increased draw from the sublevel drifts adjacent to them. A cross sectional layout of the transition zone where drives are not developed is shown in Figure 16-36 and Figure 16-37.

**Figure 16-36: Conceptual Cross-Section of Transition Zone**




Source: CNI, 2023.

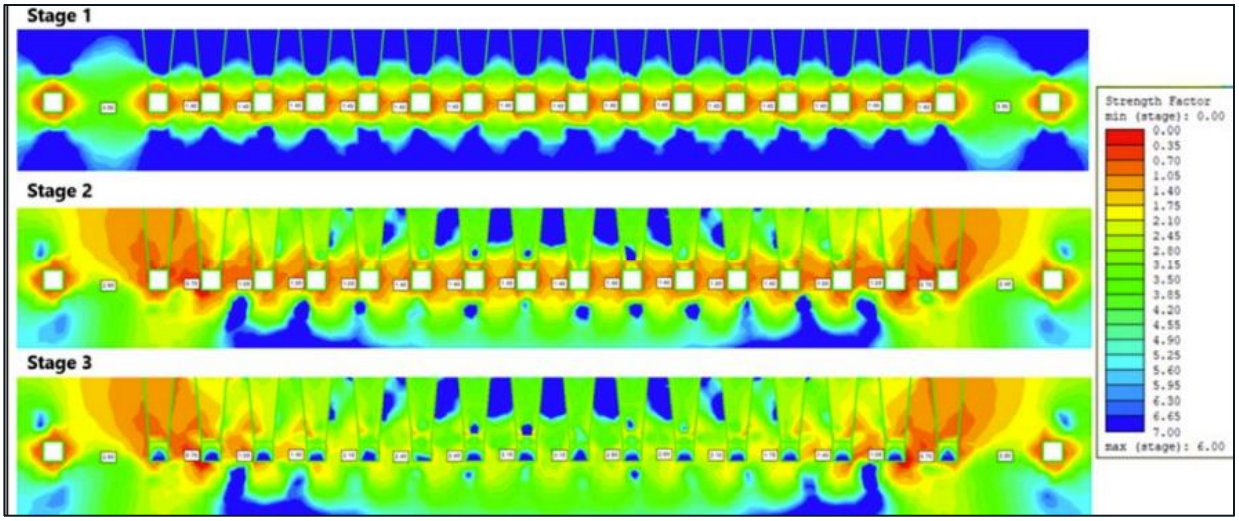
**Figure 16-37: Conceptual Plan View Drawing of Pane Arrangements**




Source: CNI, 2023.

CNI conducted a simple 2-D finite element numerical model analysis, which indicates that the sublevel drift spacing and 90-ft buffer between panels are appropriate (Figure 16-38).

**Figure 16-38: Stability Results of Sublevel Drift Pillars and Panel Transition Zones**



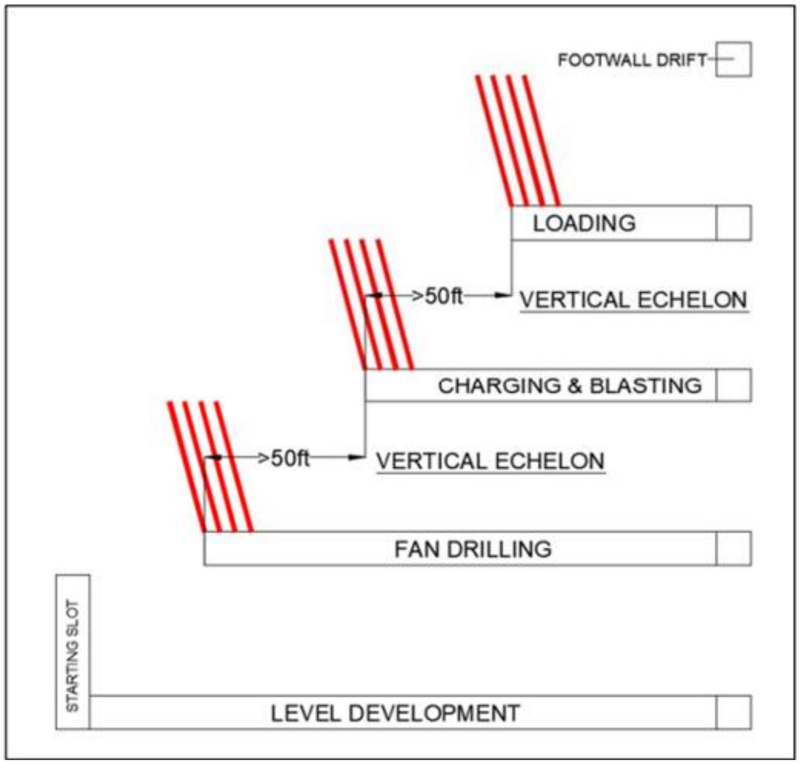

Source: CNI, 2023.

Key takeaways from modeling include:

- pillars between sublevel drifts achieve a nominal 1.4 strength factor during development,
- a 90-ft transition zone to separate panels achieves a nominal 2.4 strength factor; and
- increased abutment loading is expected on outboard pillars within the panels (0.7 strength factor).

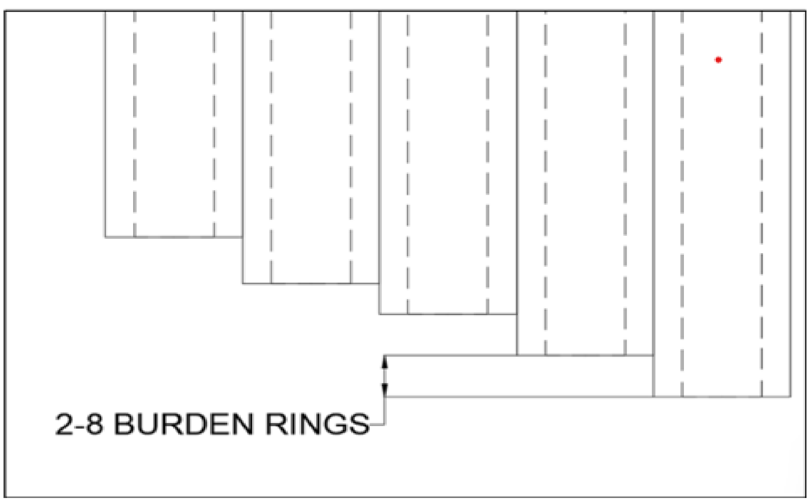
The vertical echelon between sublevels is greater than 50 ft (15.2 m). This is defined as the horizontal distance between vertical faces on adjacent sublevels (above/below), as presented in Figure 16-39. The horizontal echelon, which is the horizontal distance between vertical faces on the same sublevel, shall be at minimum, approximately two burden rings but not to exceed eight burden rings, as presented in Figure 16-40. The horizontal echelon within each caving panel shall be executed in one approximately straight line.

**Figure 16-39 Vertical Echelon – Long Section View (N.T.S.)**




Source: CNI, 2023.

**Figure 16-40: Horizontal Echelon – Plan View (N.T.S.)**




Source: CNI, 2023.

The decline translates to an access drive closest to the leading drive on the SLC footprint. This enables an efficient opening of the level for production purposes. The ventilation is driven by a fresh air drive developed from the access drive, in which the fresh air will be splitting right and left to connect to the return air drives at the extremities of the footprint. This allows natural flow of ventilation through the entire footprint.

#### 16.5.5 SLC Initiation

The amount of ore to be extracted will be limited in the upper three production levels to the following proportions:

- First Level ~40% (swell only)
- Second Level ~60%
- Third level ~100%
- Lower levels >100% to shutoff grades or dollar values.

The restricted draw rates on the upper levels are used to establish an ore blanket above the production area, control caveability, and minimise the formation of air gaps. The main aim of the draw strategy is to limit the draw rate in upper levels to avoid early ingress of waste (dilution) into the cave (SLC envelope) and preserve the high-grade ore material in the drawpoints. These restricted draw rates also apply to areas where large step-outs distances are required from one sublevel to the next.

The cave will begin to propagate towards the surface once the mining span exceeds the critical hydraulic radius required to induce caving. Both deposits are expected to begin sustained caving at a hydraulic radius of 65.6 ft (20 m) (equivalent to a 260 ft (79.2 m) by 260 ft (79.2 m) square). The mechanism for caving is expected to be gravity driven, hence the rate of propagation will depend on the natural bulking factor of the overburden and the quantity of ore extracted from the sublevels.

#### 16.5.6 Material Handling Systems

At Cactus East the mine production stages involve first the mucking of the SLC rings fired from the ore drive drawpoint with a load haul dump (LHD) loader. Rings are retreated towards the access following an interlevel lead/lag rule. The ore is then trammed to the closest stockpile within the perimeter drive. Trucks are then loaded on the level, through the main decline and to the crusher at the bottom of the vertical conveyor loading pocket or waste to the surface through the main decline.

In Parks/Salyer mine the mine production stages involve first the mucking of the SLC rings fired from the ore drive drawpoint with an LHD. Rings are retreated towards the access following an interlevel lead/lag rule. The ore is then trammed to the closest orepass within the perimeter drive, in which the ore drops down to the transfer level. Another loader in the transfer level then trams the material to the loading pocket in the inclined conveyor belt system. Each production level within the mine has direct access to a northern and southern orepass system. The conveyor belt system then transports all the ore material up to surface for processing.

#### 16.5.6.1 Cactus East Ore/Waste Handling System

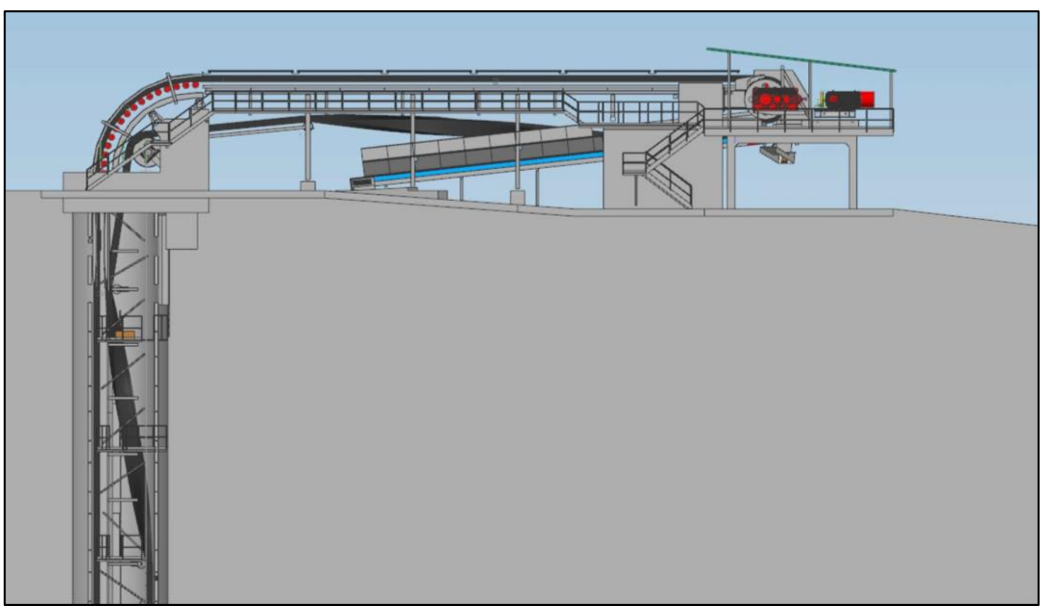
The Cactus East Ore/Waste Handling System consists of a crusher station and a 1,600 ft (488 m) vertical conveyor with a capacity of 630 tons per hour that will convey ore from the top of the orebody to surface via a vertical raise feeding an overland conveyor. Ore will be hauled by 55-ton diesel trucks to a sizer located adjacent to the bottom of the vertical conveyor. Ore will be crushed to maximum 6-in dimension. A short conveyor from the sizer will feed the vertical conveyor.

The vertical conveyor could necessitate a shaft diameter of up to 15 ft (4.6 m). CNI estimate that a shaft of this span has a nominal 85% to 90% reliability of successful execution. Detailed design of the vertical conveyor and configuration within the shaft is necessary to minimise final raise dimensions. Any vertical conveyor through the overburden conglomerate should be fully lined with either concrete or steel cans. Due to the weakness of the overburden materials which would not achieve good bond strength, CNI does not recommend anchoring utilities or other infrastructure to the rock wall inside the shaft.

Waste will be trucked to the portal for disposal within the Cactus West open pit.

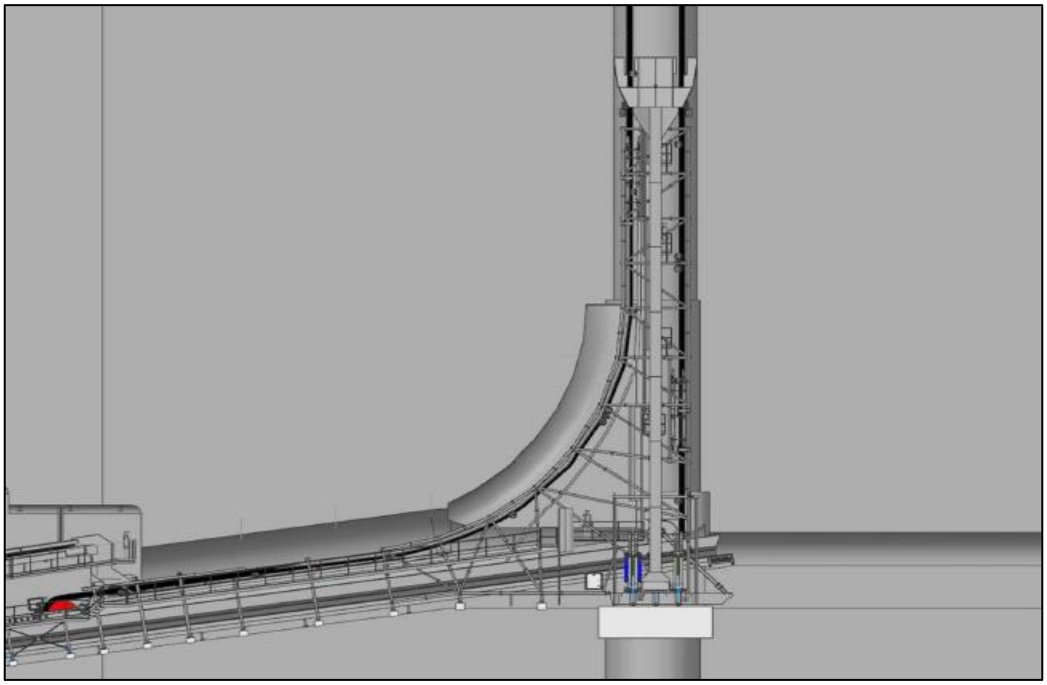
General arrangements for the top and bottom of the vertical conveyor are shown in Figure 16-41 and Figure 16-42.

**Figure 16-41: General Arrangement at Top of Vertical Conveyor**



Source: AGP 2023.

**Figure 16-42: General Arrangement at Bottom of Vertical Conveyor**



Source: AGP 2023.

Four locked, steel wire track ropes form the line structure of the conveyor system within the raise. The track ropes which form the line structure on which the conveyor belt travels are tightly tensioned and therefore must be anchored at both ends.

The conveyor transports the material on a continuous cross-reinforced flat belt with corrugated side walls. The belt is manufactured in fire retardant rubber. Additional to the sidewalls, cleats are fixed to the belt in which the ore is transported. Belt tensioning is carried out at the underground loading station, using a similar system than for conventional conveyor systems. The conveyor belt can be re-tensioned via the return drum by means of the hydraulic tensioning equipment. When the conveyor belt has passed the unloading point, the belt turning unit turns the belt by 180° to bring the soiled side of the belt upwards again. This prevents soiling of the track. Loose particles are then cleaned from the belt after turning. The conveyor belt is turned once more before it reaches the loading point.

Drive units are located on the surface area, where maintenance activities can be performed safely and easily.

#### 16.5.6.2 Parks/Salyer Ore/Waste Handling System Description

An ore/waste handling system is required for the Parks/Salyer orebody. The mine plan for this orebody consists of two ramps with one dedicated for material handling. The ore/waste handling system consists of a series of initially four, extending to five switchback conveyors and two crushing sizers on -270 L, one of which will subsequently be relocated at the -470 L. that will deliver material from the mine working levels to the surface portal, from where materials will then be transported on surface via an overland conveyor.

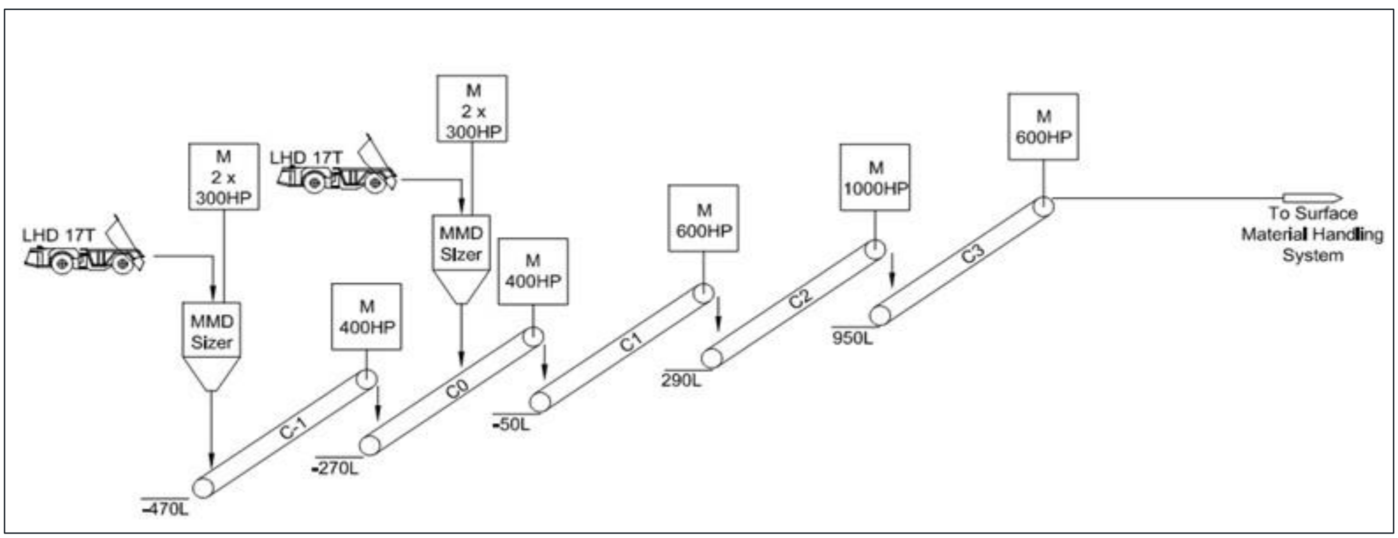
Ore will be transferred from the working faces to ore passes to a remuck bay adjacent to each crusher location by a 17-ton scoop tram. The ore will be re-handled from the remuck bay to the crusher.

Material will be fed through sizers to reduce lump size to around -6 inch, which then discharges to the conveyor system. Two feed sizers will initially be located on -270L. As the mine develops, one of the two sizers will be relocated to -470 L. The sizers will discharge material onto a series of five switchback conveyors that will transport materials to the portal. A process flow diagram illustrating the material handling system is shown in Figure 16-43.

Design assumptions for the conveyor system were as follows:

- System flow capacity = 950 tons/h
- The required system flow capacity will not increase. If there is an increase in the system's flow capacity, the sizer's feed hopper will require review.
- Material insitu dry density = 0.785 tons/ft$^3$
- Dry broken density at 35% swell factor = 0.580 tons/ft$^3$
- Wet broken density with 4% moisture = 0.605 tons/ft$^3$
- LHD 17T is used to load the sizer hopper.

**Figure 16-43: Schematic of Parks/Salyer Conveyor System**




Source: AGP, 2023.

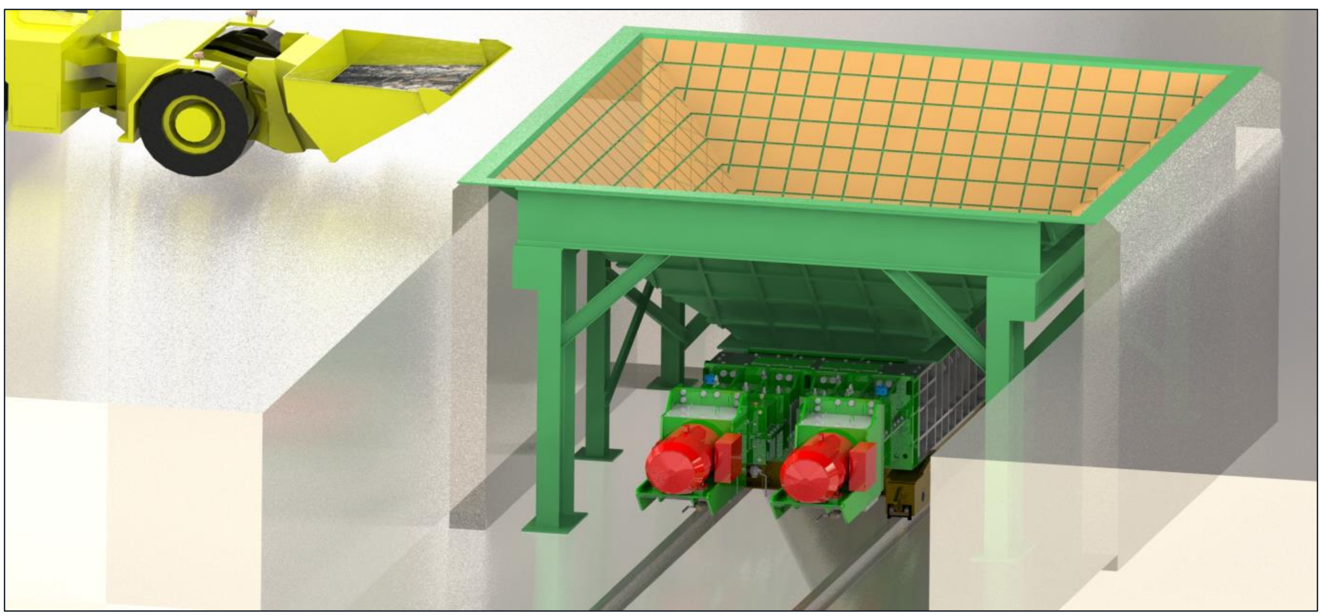
At Parks/Salyer a total of five conveyors are required. See Table 16-24 for a summary of conveyor requirements.

**Table 16-24: Park Salyer Conveyor Requirements**

| Conveyor Designation | C-1 | C0 | C1 | C2 | C3 |
|---|---|---|---|---|---|
| Lift [ft] | 215 | 247 | 371 | 656 | 425 |
| Length [ft] | 1466 | 1473 | 2559 | 4518 | 2913 |
| Belt Width [in] | 42 | | | | |
| Belt Speed [fpm] | 300 | | | | |
| Belt Specification | ST1400 | ST1600 | ST2250 | ST4000 | ST2500 |
| Demand Power [HP] | 317 | 354 | 528 | 933 | 602 |

There are three crusher stations within the Parks/Salyer mine, each consisting of a 25-ton hopper and an MMD sizer. The first two sizers will be located at -270 L with one of these sizers being re-located to the -470 L later in the mine life. The material loading room will contain enough space to allow a LHD 17T to dump material onto the sizer from a single side. The hopper will be flush with the ground, discharging onto to the MMD sizer that is located at a level below. The sizer will be supported by two concrete pads with an embedded rail which allows easy transport for maintenance and installation purposes. The hopper will be supported by concrete footings. A general arrangement is shown in Figure 16-44.

**Figure 16-44: General Arrangement for MMD Sizer**



Source: AGP 2023.

### 16.5.7 Ventilation

The function of the ventilation system is to cool the air, dilute/remove airborne dust, diesel emissions, and explosive gases at levels necessary to ensure safe production throughout the life of the mine. The ventilation phases described generally follows major ventilation milestones.

The ventilation system for the project was modeled using Ventsim Visual™ Advanced. This software provides for 3D visualization of a network and uses a form of the Hardy-Cross method for the ventilation network calculations. From the Ventsim Visual™ manual:

> *"The [Hardy-Cross Method is] used by Ventsim Visual™ to perform the calculation of airflows in a network. It uses an iterative estimation method that adjusts the airflows through a network until the estimation errors lie within acceptable limits. Ventsim Visual™ Advanced uses a modified method which considers density changes and mass flow balances."*

The ventilation network analysis for the project was undertaken by importing the mine design from the Deswik 3D program and then applying attributes for each of the airways relative to their dimensions, frictional resistance, length, etc.

Table 16-25 outlines the velocity design criteria for the project. These upper limit values are comparable to industry standards used in the US and elsewhere and generally align with industry best practice. The estimated friction factor k values used in the ventilation model and for calculations, are based on recommendations by M. J. McPherson and industry best practice. Table 16-26 lists the friction factor k values used in modeling.

**Table 16-25: Air Velocity Design Criteria**

| Type of Opening | Velocity Limits (m/s) | Comments |
|---|---|---|
| Fresh Air Decline | 6.5 | above stopes, vehicular traffic only |
| Return Air Decline | 6.5 | no pedestrian access |
| Stope and Level Accesses | 6.5 | in mining areas to minimize dust |
| Return Air Raises | 20 | rule of thumb, airway economics |
| Return Air Raises | 7.0 to 12.0 | design outside this range to minimize water blanketing |

**Table 16-26: Friction k-Factor Values**

| Airway Description | Friction Factor, k ($kg/m^3$) | Comments |
|---|---|---|
| Arched drifts | 0.0120 | |
| Drop or longhole raises | 0.0120 | if contains a ladderway area is decreased by $1.0m^2$ |
| PVC type Ventilation Duct | 0.0035 | lay-flat type ducting |
| Plastic Ventilation Duct | 0.0019 | plastic duct |

| Airway Description | Friction Factor, k ($kg/m^3$) | Comments |
| --- | --- | --- |
| Steel Spiral Ventilation Duct | 0.0029 | |

The air volume supplied must be able to dilute and remove noxious gases as well as diesel particulate matter, exhaust and heat generated using such equipment. The amount of air required is largely determined by the number and size of diesel equipment operating underground.

The mines will use MSHA rates for airflow requirements for diesel powered equipment, using specific engine models. An example of this analysis referencing Cactus East is shown in Table 16-27.

The number of equipment items and required airflow is shown graphically in Figure 16-45 and Figure 16-46 for Cactus East and Parks/Salyer, respectively.

The Cactus Project is located within a very hot surface temperature area and well-known geothermal area of Arizona. The ventilation system will be designed to limit the wet-bulb temperature to 85.1°F (29.5°C) in any location.

The current planned maximum depth of the Project is about 2,000 ft (~600m) below surface. The virgin rock temperature at depth will approach 104°F (40°C). A preliminary level heat evaluation was modeled, and it was determined that refrigeration would be required for both the development and production phases of the project. Heat modelling thermal parameters are shown in Table 16-28.

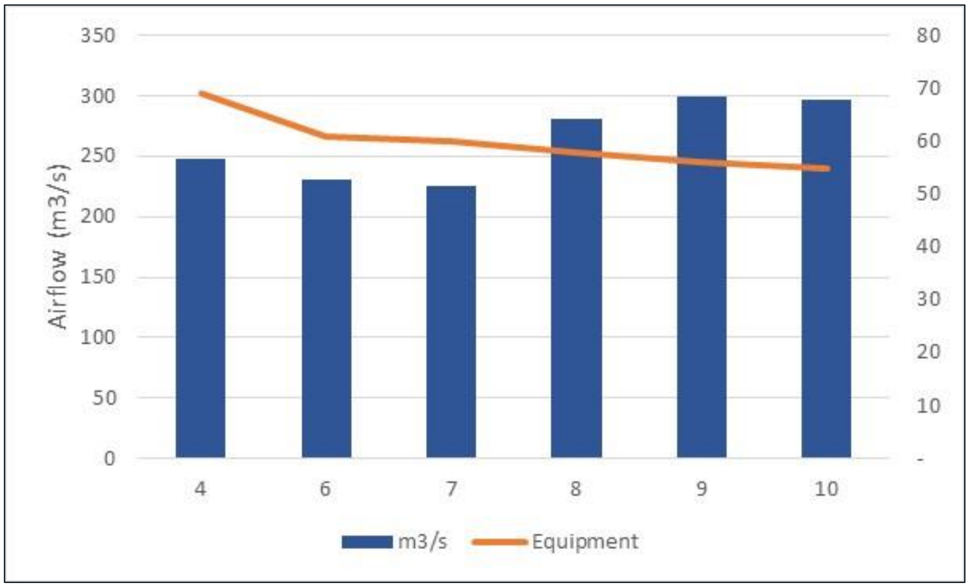
Figure 16-47 graphically identifies the origin of the greatest heat gains. This confirms the large amounts of heat created using diesel equipment, as compared to other sources. One area which is of concern and cannot be determined based on available data is if there is any inflow of hot water. This will have a major effect on cooling and wet-bulb temperatures. Therefore, it is recommended that for the next level of study the presence or non-presence of geothermal water be determined.

**Table 16-27: Cactus East (Vertical Conveyor) Air Volume Requirements for Selected Years**

| Equipment Model | MSHA Total $m^3/s$ per unit | MSHA Total Utilized $m^3/s$ per unit | Equip Utilization (%) | Equip Qty Year 4 | MSHA Total $m^3/s$ | Equip Qty Year 6 | MSHA Total $m^3/s$ | Equip Qty Year 7 | MSHA Total $m^3/s$ | Equip Qty Year 8 | MSHA Total $m^3/s$ | Equip Qty Year 9 | MSHA Total $m^3/s$ | Equip Qty Year 10 | MSHA Total $m^3/s$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 2-500ER Emulsion Loader | 3.1 | 1.5 | 50 | 3 | 4.6 | 2 | 3.1 | 3 | 4.6 | 3 | 4.6 | 2 | 3.1 | 2 | 3.1 |
| 8 Man Landcruiser | 4.3 | 4.3 | 100 | 3 | 12.9 | 3 | 12.9 | 3 | 12.9 | 3 | 12.9 | 3 | 12.9 | 3 | 12.9 |
| A-64 Flatdeck/Pallets | 3.1 | 1.5 | 50 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 |
| A-64 Fuel | 3.1 | 1.5 | 50 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 |
| A64 HDR60 | 3.1 | 1.5 | 50 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 |
| A-64 Scissor | 3.1 | 1.5 | 50 | 7 | 10.7 | 5 | 7.7 | 5 | 7.7 | 3 | 4.6 | 2 | 3.1 | 2 | 3.1 |
| CAT 120K | 5.0 | 2.5 | 50 | 1 | 2.5 | 1 | 2.5 | 1 | 2.5 | 1 | 2.5 | 1 | 2.5 | 1 | 2.5 |
| DD422i | 3.3 | 0.8 | 25 | 4 | 3.3 | 3 | 2.5 | 3 | 2.5 | 2 | 1.7 | 1 | 0.8 | 1 | 0.8 |
| DL 422i | 3.3 | 1.7 | 50 | 4 | 6.6 | 3 | 5.0 | 4 | 6.6 | 4 | 6.6 | 5 | 8.3 | 4 | 6.6 |
| DS411 | 3.1 | 1.5 | 50 | 6 | 9.2 | 4 | 6.1 | 4 | 6.1 | 3 | 4.6 | 2 | 3.1 | 2 | 3.1 |
| Elec Landcruiser | 4.3 | 4.3 | 100 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 |
| LH307 | 3.5 | 3.5 | 100 | 2 | 7.1 | 2 | 7.1 | 2 | 7.1 | 1 | 3.5 | 1 | 3.5 | 1 | 3.5 |
| LH410 | 7.6 | 7.6 | 100 | 4 | 30.2 | 3 | 22.7 | 3 | 22.7 | 2 | 15.1 | 2 | 15.1 | 2 | 15.1 |
| LH517i | 8.7 | 8.7 | 100 | 3 | 26.2 | 3 | 26.2 | 4 | 34.9 | 5 | 43.7 | 5 | 43.7 | 5 | 43.7 |
| Mech Landcruiser | 4.3 | 4.3 | 100 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 | 2 | 8.6 |
| Mine Mate SL3/Pipe Handler | 3.3 | 0.8 | 25 | 1 | 0.8 | 1 | 0.8 | 1 | 0.8 | 1 | 0.8 | 1 | 0.8 | 1 | 0.8 |
| Mine Mate WS3 | 3.3 | 1.7 | 50 | 1 | 1.7 | 1 | 1.7 | 1 | 1.7 | 1 | 1.7 | 1 | 1.7 | 1 | 1.7 |
| SST Shotcrete | 3.1 | 1.5 | 50 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 | 2 | 3.1 |

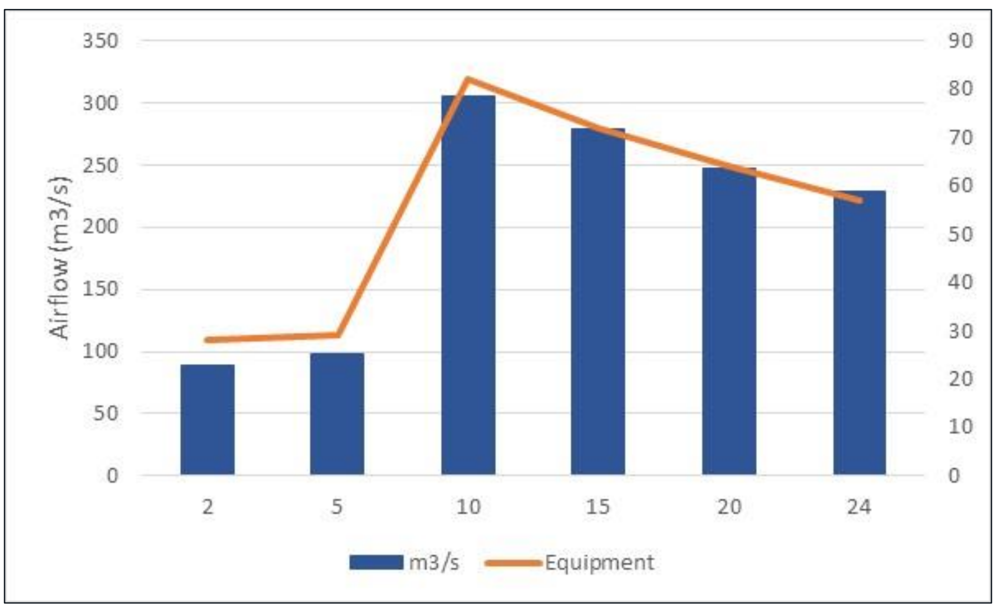
| Equipment Model | MSHA Total m³/s per unit | MSHA Total Utilized m³/s per unit | Equip Utilization (%) | Equip Qty Year 4 | MSHA Total m³/s | Equip Qty Year 6 | MSHA Total m³/s | Equip Qty Year 7 | MSHA Total m³/s | Equip Qty Year 8 | MSHA Total m³/s | Equip Qty Year 9 | MSHA Total m³/s | Equip Qty Year 10 | MSHA Total m³/s |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Supv Landcruiser | 4.3 | 4.3 | 100 | 14 | 60.2 | 14 | 60.2 | 10 | 43.0 | 10 | 43.0 | 10 | 43.0 | 10 | 43.0 |
| TH551i | 21.2 | 21.2 | 100 | 2 | 42.5 | 2 | 42.5 | 2 | 42.5 | 5 | 106.2 | 6 | 127.4 | 6 | 127.4 |
| TM15XH Mobile Rockbreaker | 2.1 | 0.5 | 25% | 1 | 0.5 | 1 | 0.5 | 1 | 0.5 | 1 | 0.5 | 1 | 0.5 | 1 | 0.5 |
| Toyota Rescue | 4.3 | - | | 1 | - | 1 | - | 1 | - | 1 | - | 1 | - | 1 | - |
| **Total** | | | | **69** | **248** | **61** | **231** | **60** | **225** | **58** | **281** | **56** | **299** | **55** | **297** |

**Figure 16-45: Cactus East Airflow Requirements**




Source: AGP, 2023.

**Figure 16-46: Parks/Salyer Airflow Requirements**

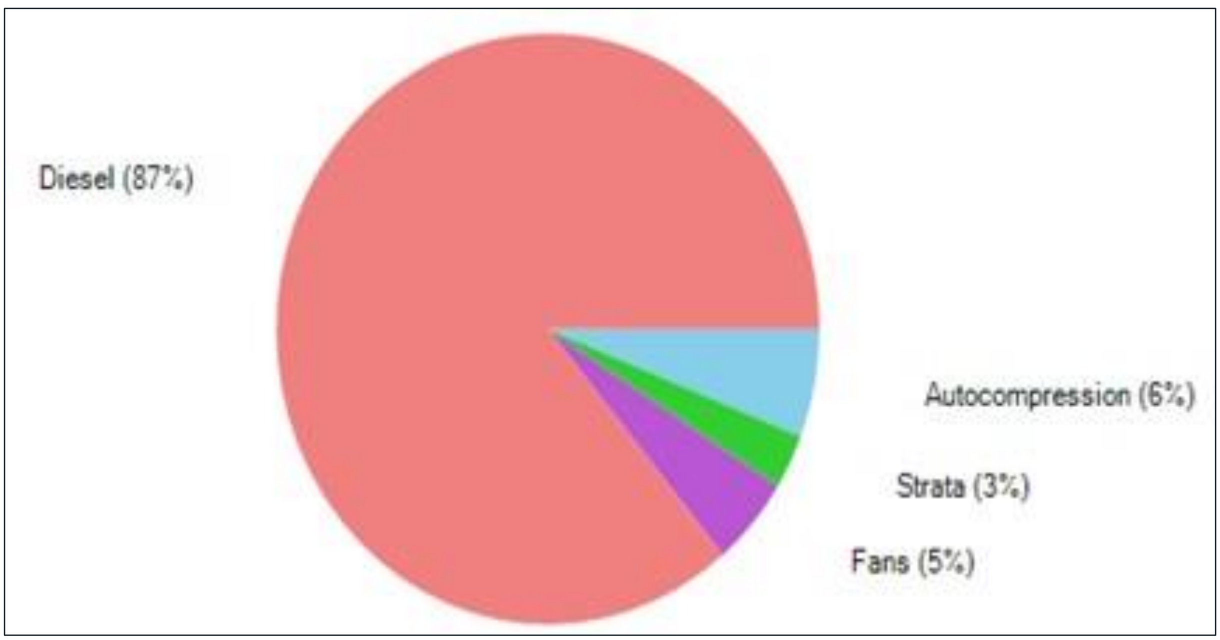



Source: AGP, 2023.

**Table 16-28: Heat Modelling Thermal Parameters**

| Thermal Parameter | Value | Comments |
|---|---|---|
| Maximum UG Wet Bulb Temperature (°C) | 29.5 | |
| Maximum UG Dry Bulb Temperature (°C) | 45 | |
| Surface Elevation (masl) | 420 | |
| Geothermal gradient °C /100m) | 2.80 | |
| Rock Density (kg/m$^3$) | 2700 | average |
| Rock Wetness Fraction | 0.50 | 0.5 indicates a 50% wet surface |
| Rock Thermal Conductivity (W/mC) | 3.20 | average |
| Surface Rock Temp (°C) | 20 | |
| Surface Relative Humidity (%) | 20 | |
| Surface Temperature-Wet Bulb (°C) | 16.5 | |
| Surface Temperature-Dry Bulb (°C) | 32.0 | |

**Figure 16-47: Heat**




Source: AGP, 2023.

#### 16.5.7.1 Ventilation Strategy

##### 16.5.7.1.1 Ventilation Shafts, Main Fans and Air Cooling

Ventilation raises will be excavated using a raisebore drill within the Gila Conglomerate overburden. To evaluate the use of a raisebore drill to excavate raises, CNI utilized the Stacey and McCracken method (1989) of assessing geotechnical risk for large diameter raisebored shafts. The details of this evaluation are presented in CNI's memo Raisebore Stability within Gila Conglomerate at ASCU (August 2023).

The maximum stable span based on the estimated rock quality is 11.6 ft (3.5 m) and ventilation fresh and return raises have been planned at this diameter. This estimate is based on Stacey and McCracken's recommendation for a ventilation shaft with a service life of 10 years and is considered permanent (RSR = 1.3) and achieves a 95% reliability (Probability of Failure = 5%). Because the actual site conditions are unknown, these evaluations are considered preliminary. To have a better understanding of the rock and soil types and their quality at any raise location, a pilot hole should be core-drilled so that cores can be carefully logged for geomechanical properties and lithology, and raise stability reassessed.

At Cactus East the Intake and West Exhaust fan installations are located on surface and the East Exhaust fan installation will be underground due to surface constraints. All installations will be controlled with variable frequency drives (VFD) to allow fluctuation in air volumes during the LOM (see Table 16-29 and Table 16-30).

**Table 16-29: East Main Fans**

| Location | Fan Type | Fan Type | Raise Diameter ft (m) | Flowft$^3$/s ($m^3$/s) | Fan Power (kW) | Est Pressure (Pa) |
|---|---|---|---|---|---|---|
| Intake | Surface | Axial | 11.5 (3.5) | 7946 (225) | 825 | 2725 |
| West Exhaust | Surface | Axial | 11.5 (3.5) | 5827 (165) | 350 | 975 |
| East Exhaust | Underground | Axial | 11.5 (3.5) | 4662 (132) | 175 | 1000 |

At Parks/Salyer all intake and exhaust fan installations will be located on surface. All installations will be controlled with variable frequency drives (VFD) to allow fluctuation in air volumes during the LOM.

**Table 16-30: Parks/Salyer Main Fans**

| Location | Fan Type | Fan Type | Raise Diameter ft (m) | Flow ft$^3$/s ($m^3$/s) | Fan Power (kW) | Est Pressure (Pa) |
|---|---|---|---|---|---|---|
| West Exhaust | Surface | Axial | 11.5 (3.5 | 6180 (175) | 375 | 1475 |
| East Exhaust | Surface | Axial | 11.5 (3.5) | 6180 (175) | 400 | 1600 |
| Intake | Surface | Axial | 11.5 (3.5) | 6180 (175) | 175 | 625 |

Air cooling requirements are shown in Table 16-31.

**Table 16-31: Overall Air-Cooling Requirements**

| Orebody/Development | Location | Estimated Cooling Capacity (kW) | Estimated Power Requirements (kW) |
|---|---|---|---|
| Cactus East | Surface | 10,000 | 3,000 |
| Parks/Salyer | Surface | 7,500 | 2,250 |

##### 16.5.7.1.2 Auxiliary Ventilation

Auxiliary fans will be required for access development and production.

For decline development, it is estimated that these fans will operate at a volume of up to 169,500 cfm (80 $m^3$/s) and at an estimated pressure of up to 12 in water gauge (WG) (3,000 Pa), their motor size will be approximately 125-150 kW. These fans may operate singly or in a series configuration for longer duct runs. These fans will allow provide sufficient volumes for 1 truck and 1 LHD and ancillary equipment per heading. The duct will be PVC plastic with a diameter of 4.5 ft (1.37 m). Decline development fan installations will operate in a pull or exhaust type configuration.

It is estimated that these fans for production will operate at a volume of 1,060 $ft^3$/s (30 $m^3$/s) and at an estimated pressure of 1,100 Pa, their motor size will be approximately 50 kW. These fans will allow provide sufficient volumes for 1 LHD per heading as loading will be outside of all ore drives. The duct will be PVC plastic with a diameter of 3.5 ft (1.07 m).

As a contingency duct leakage of 15% has been added to the auxiliary fan requirements.

##### 16.5.7.1.3 Access Development Ventilation Phasing

The Cactus East will be developed using a single decline system. This results in a requirement for a temporary raise into the pit at approximately the half-way point to the orebody, which will establish a surface through ventilation circuit. Initially the single decline will require a small refrigeration plant at the portal, but only until the pit raise is established, after which time decline refrigeration will not be required until the main ventilation cooling system is established for the orebody.

The access ramp auxiliary fans and duct will operate in a pull-type arrangement pulling hot air and blast gases from the face in an exhaust type installation. This provides a benefit in minimising blast re-entry times improving development advance rates.

The access development at Cactus East was divided into phases as follows:

###### 16.5.7.1.3.1 Cactus East Phase 1

The initial decline development will employ a pull or exhaust type of ventilation. Initially this will be accomplished using a single 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 $m^3$/s) at an estimated maximum pressure of 12 in. WG (3,000 Pa). No refrigeration is required for this stage (see Figure 16-48).

**Figure 16-48:** **Phase 1 Ventilation**

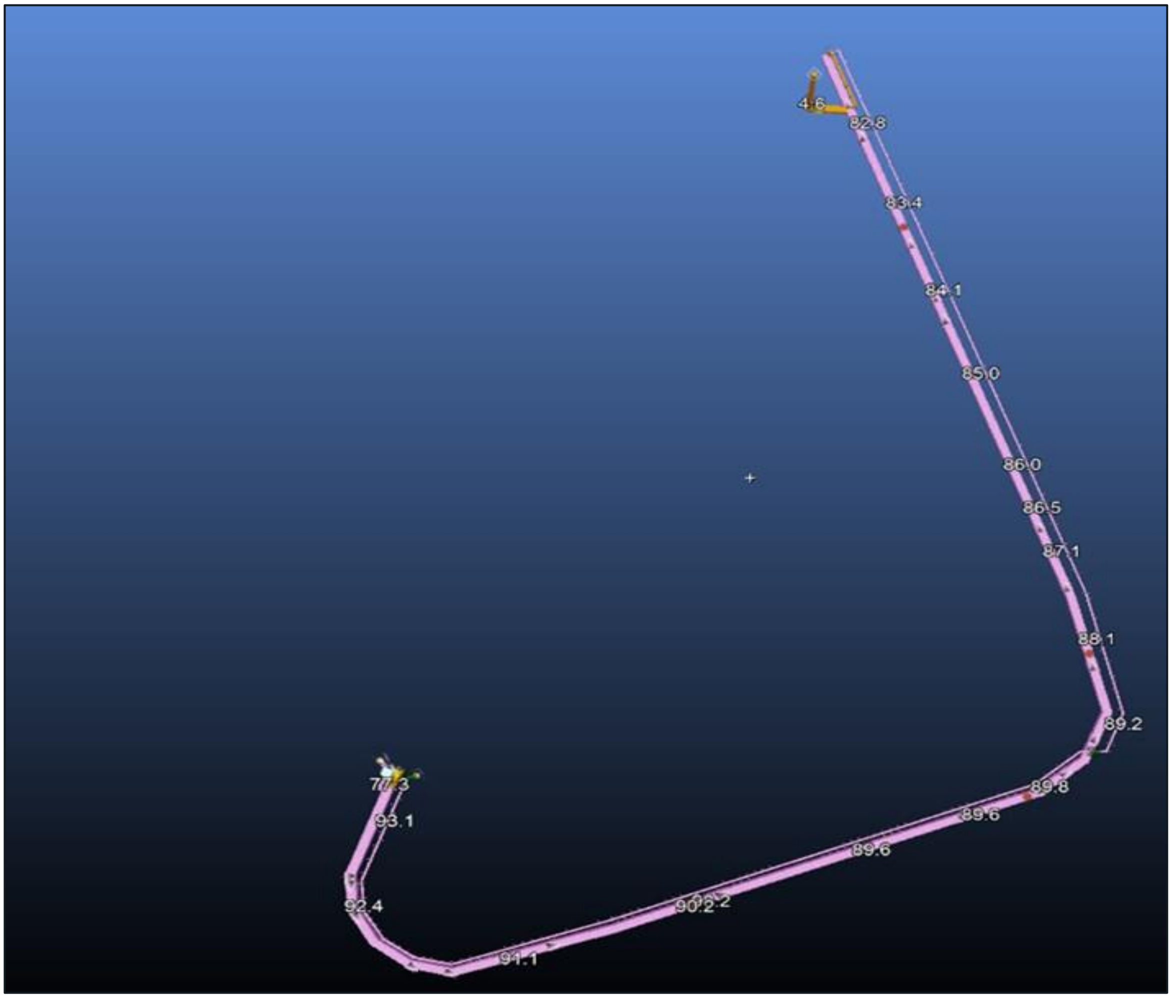

Source: BBA 2023.

16.5.7.1.3.2 Cactus East Phase 2a

Phase 2a decline development will continue to employ a pull or exhaust type of ventilation. Initially this will be accomplished using a single 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 $m^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). Refrigeration of 1,000 kW is required for this stage (see Figure 16-49).

The milestone at the end of this stage is the breakthrough to surface of a midpoint intake raise allowing for a through ventilation circuit to be established and will remove the requirement for refrigeration in later development stages.

**Figure 16-49: Cactus East Phase 2a Ventilation**

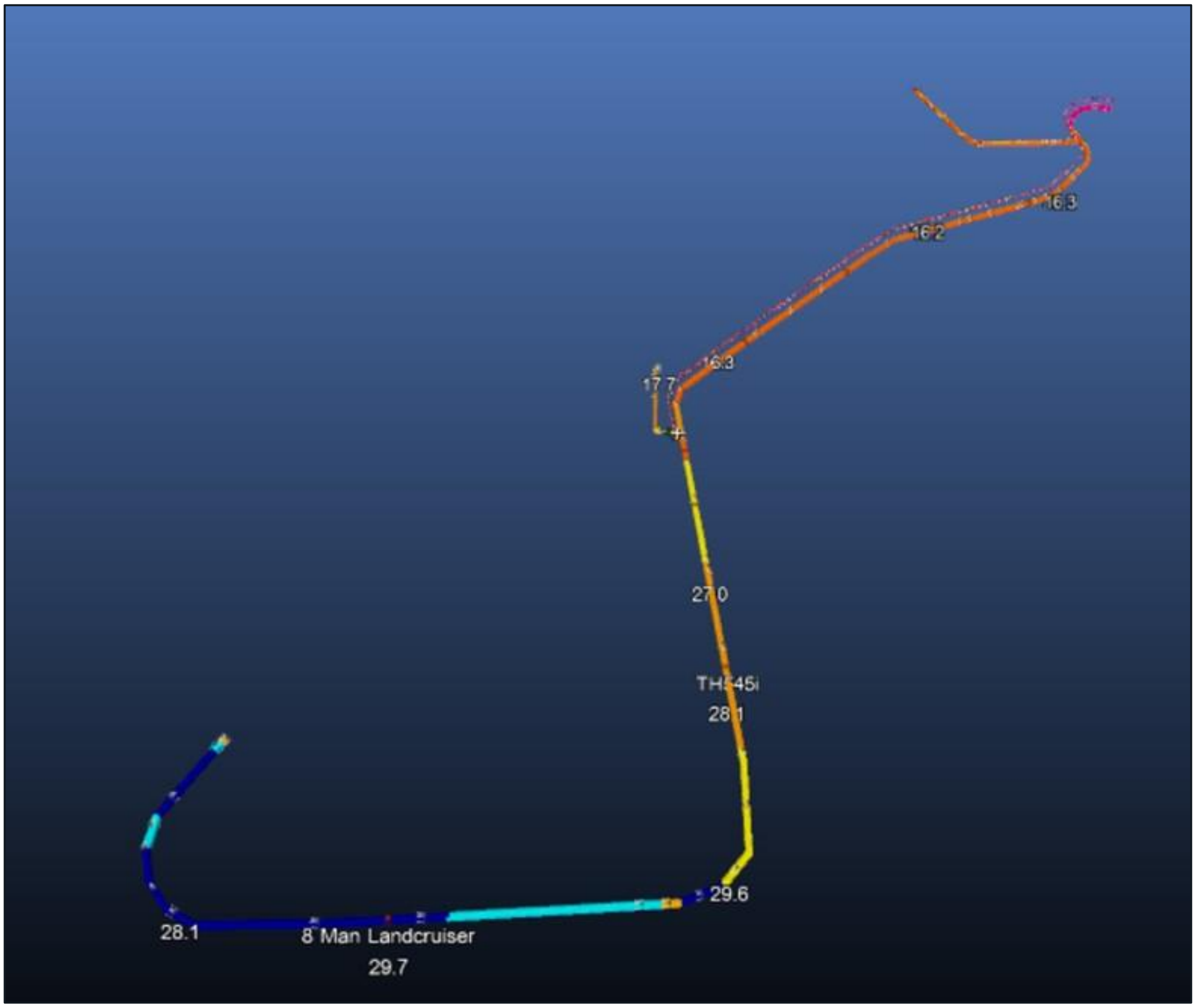



Source: BBA 2023.

16.5.7.1.3.3 Cactus East Phase 2b

Phase 2b commences with the breakthrough to surface of an open pit intake raise and installation of an intake fan at the bottom of this raise. This will eliminate the requirement for ramp auxiliary ventilation above this point and will allow development to continue without refrigeration to the bottom of the permanent intake raise. Auxiliary ventilation will switch to a push type of ventilation using a single 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 $m^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa) (see Figure 16-50).

**Figure 16-50: Cactus East Phase 2b Ventilation**




Source: BBA 2023.

#### 16.5.7.1.3.4 Cactus East Phase 3

Phase 3 commences with the breakthrough to surface of the main intake raise and installation of a surface intake fan supplying 5,300 ft3/s (150 $m^3$/s) and the first phase of the main refrigeration/cooling plant with an output of 3,000 kW. These installations allow for development to continue to the orebody and connection to the first main exhaust raise. Auxiliary ventilation will continue to be a push type of ventilation using a single 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 $m^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa) (see Figure 16-51).

**Figure 16-51: Cactus East Phase 3 Ventilation**

Source: BBA 2023.

### 16.5.7.1.3.5 Parks/Salyer Ventilation

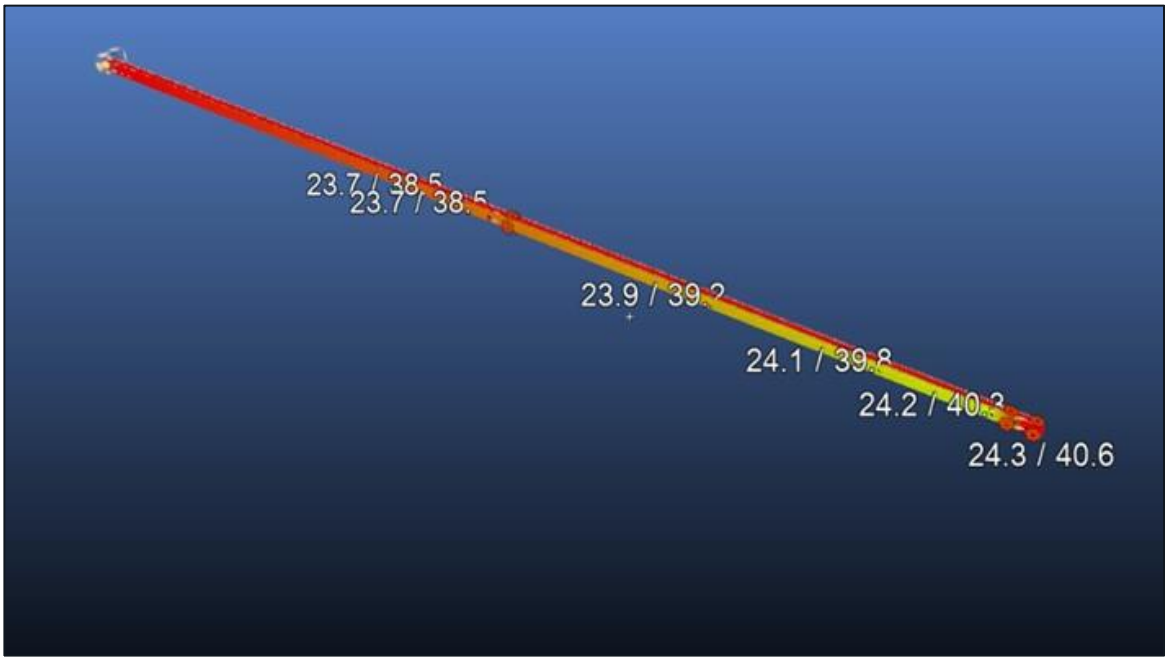
Parks/Salyer will be developed using a twin decline system from surface with dedicated cooled, fresh air and dedicated return air declines. The general strategy is a leapfrog approach in which cooled, fresh air travels down one side of the decline, picked up by auxiliary fans, pushed to the face in a pull-type duct system, and hot, exhaust air migrates back up the opposite side of the decline to surface. This scenario is repeated until the main ventilation circuit is established. This scenario also uses an internal raise and secondary surface raise. The length of this development requires a large surface cooling plant for development.

### 16.5.7.1.3.6 Parks/Salyer Phase 1

The initial decline development will employ a pull or exhaust type of ventilation. Initially this will be accomplished using twin 4.5 ft (1.37 m) diameter PVC duct lines and 2” series installed 125 kW axial fans supplying up to 169,500 cfm

(80 $m^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). No refrigeration is required for this stage (see Figure 16-52).

**Figure 16-52: Parks/Salyer Phase 1 Ventilation**

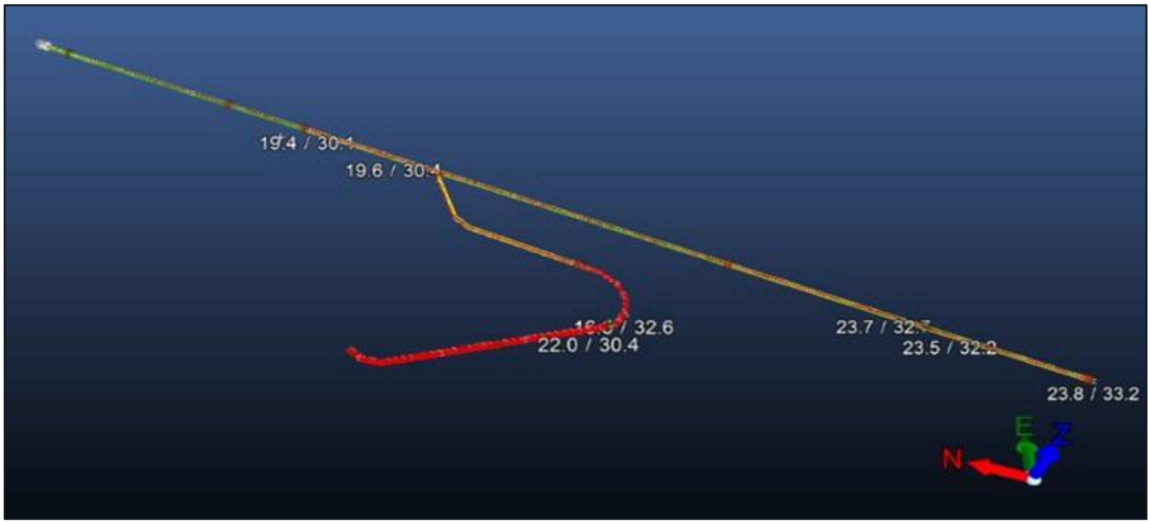



Source: BBA 2023.

16.5.7.1.3.7 Parks/Salyer Phase 2

The initial decline development will employ a pull or exhaust type of ventilation. Initially this will be accomplished using twin 4.5 ft (1.37m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 m3/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). The pull or exhaust type of overlap auxiliary ventilation is used for all development. Refrigeration of 1,000 kW is required for this stage (see Figure 16-53). This refrigeration plant will be modular for growth and may be used in the function of the final main refrigeration plant. In this stage the ramps spilt off in different directions.

**Figure 16-53: Parks/Salyer Phase 2 Ventilation**

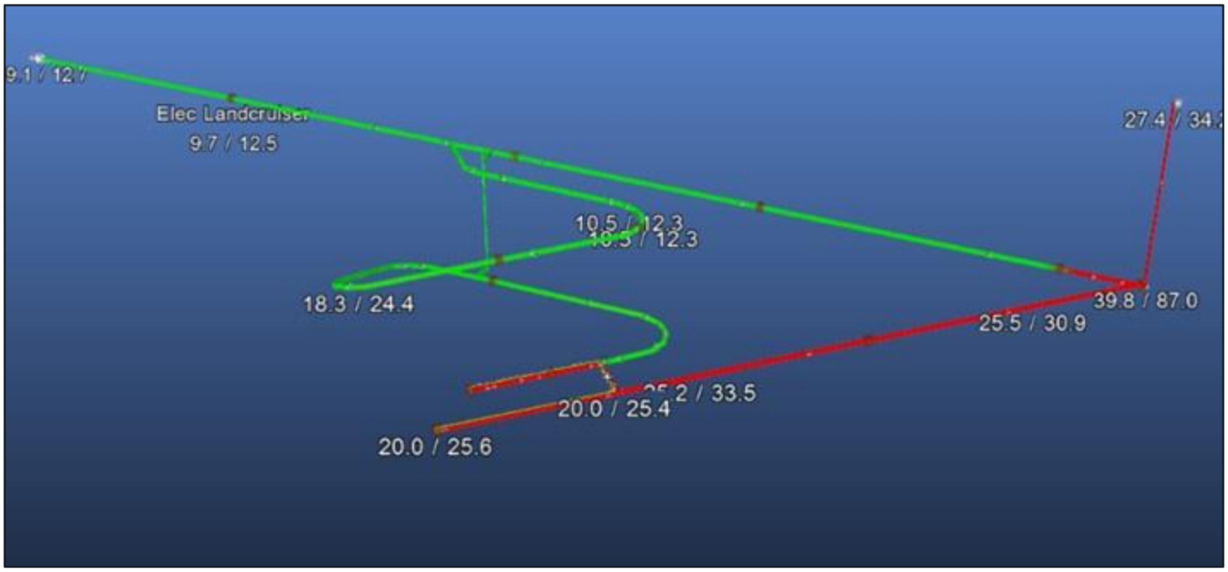



Source: BBA 2023.

16.5.7.1.3.8 Parks/Salyer Phase 3

Phase 3 commences with the development of a raise at the first corner of the conveyor drift. This raise will exhaust 3,530 $ft^3/s$ (100 $m^3/s$), allowing for the removal of some auxiliary ducting and creating a through ventilation circuit with the main ramp. The same type of pull auxiliary ventilation will be used consisting of twin 4.5 ft (1.37 m) diameter PVC duct lines and 2i n series installed 125 kW axial fans supplying up to 169,500 cfm (80 $m^3/s$) at an estimated maximum pressure of 12 in WG (3,000 Pa). Refrigeration of 2,000 kW is required for this stage (see Figure 16-54). In this stage the ramps spilt off in different directions and a small raise will go from the main ramp to the lower haulage ramp. In addition, in this stage, the conveyor and the ramp begin a parallel descent and the first crosscut is established.

**Figure 16-54: Parks/Salyer Phase 3 Ventilation**

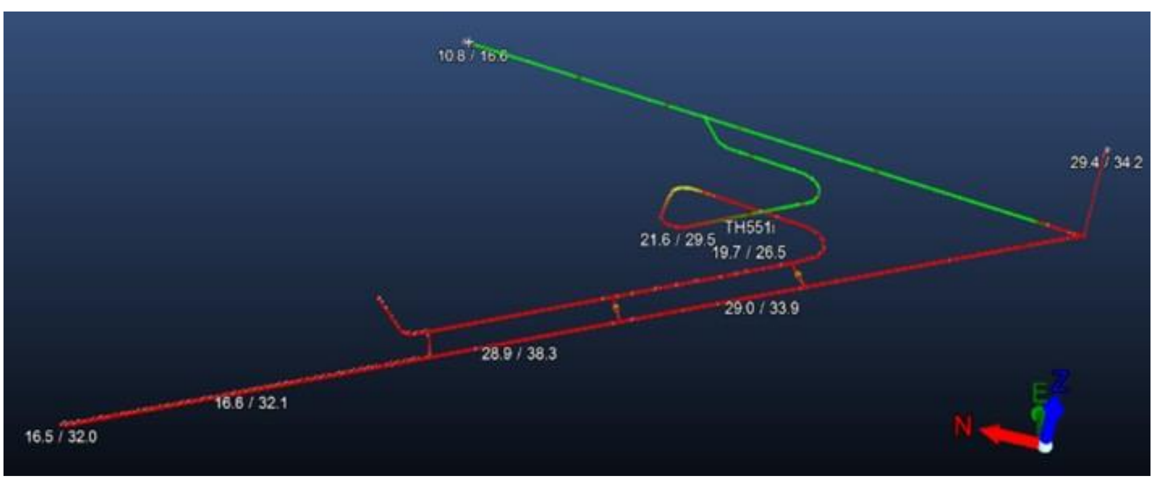



Source: BBA 2023.

16.5.7.1.3.9 Parks/Salyer Phase 4

The same type of pull auxiliary ventilation will be used consisting of twin 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 12 5 kW axial fans supplying up to 169,500 cfm (80 $m^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). Refrigeration of 2,000 kW is required for this stage (see Figure 16-55).

**Figure 16-55: Parks/Salyer Phase 4 Ventilation**

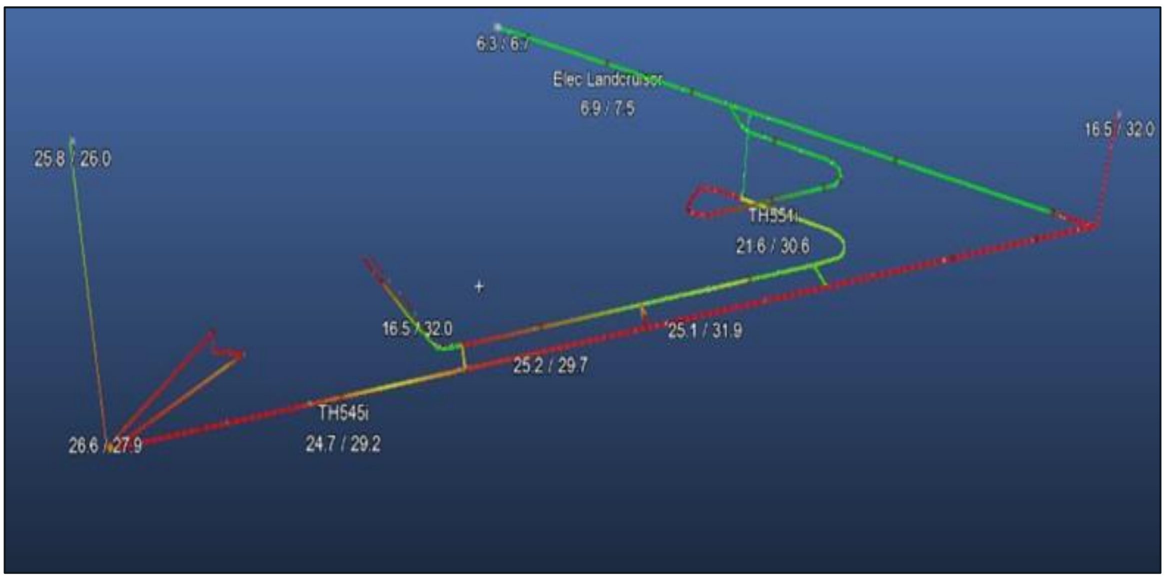



Source: BBA 2023.

#### 16.5.7.1.3.10 Parks/Salyer Phase 5

Phase 5 commences with the connection to the West exhaust raise and the installation of a surface exhaust fan to pull 5,280 ft$^3$/s (150 m$^3$/s) through the current development. This will eliminate a large requirement for auxiliary ducting. The same type of pull auxiliary ventilation will be used when required consisting of twin 4.5 ft (1.37 m) diameter PVC duct lines and 2-(remove dash) in series installed 170hp (125 kW) axial fans supplying up to 169,500cfm (80 m3/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). Refrigeration of 712 TR (2,500 kW) is required for this stage (see Figure 16-56).

**Figure 16-56: Parks/Salyer Phase 5 Ventilation**

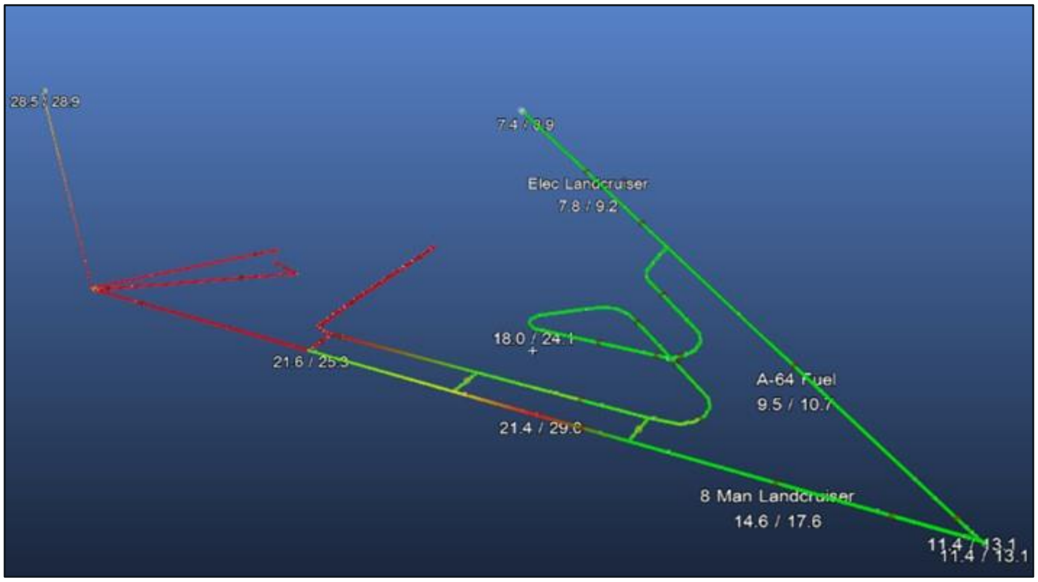



Source: BBA 2023.

#### 16.5.7.1.3.11 Parks/Salyer Phase 6

Development towards the orebody continues. The same type of pull auxiliary ventilation will be used when required consisting of twin 4.5 ft (1.37 m) diameter PVC duct lines and 2 in series installed 125 kW axial fans supplying up to 169,500 cfm (80 m$^3$/s) at an estimated maximum pressure of 12 in WG (3,000 Pa). Refrigeration of 3,500 kW is required for this stage (see Figure 16-57).

**Figure 16-57: Parks/Salyer Phase 6 Ventilation**

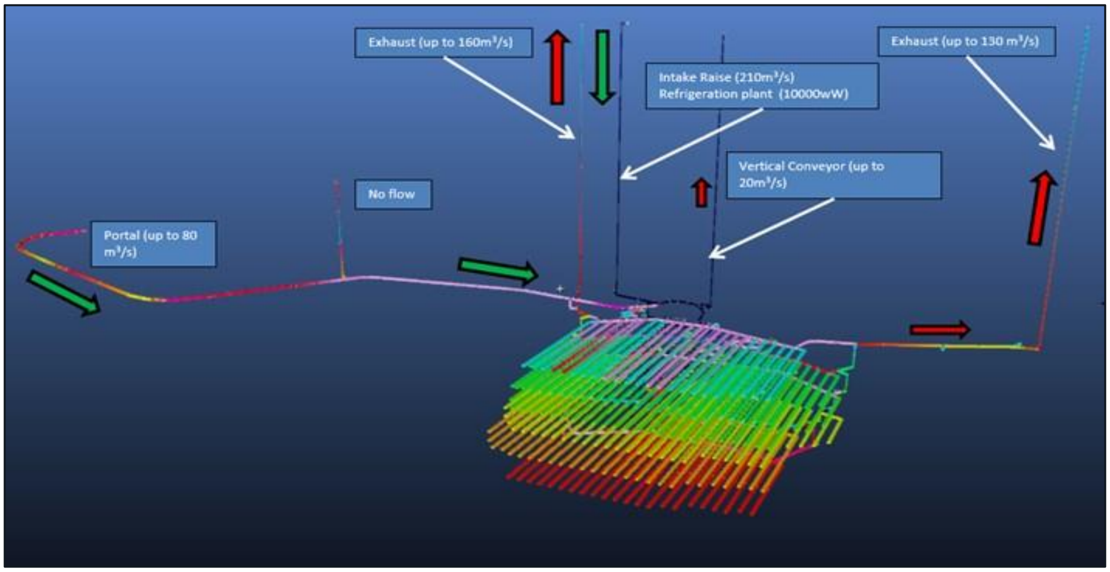



Source: BBA 2023.

#### 16.5.7.1.4 Steady State Production

Cactus East production ventilation will rely on cooled fresh air in a flow-through ventilation design across the orebody controlled by automated regulators at each end of the foot wall drifts. These will be regulated to allow for up to 1,410 $ft^3/s$ (40 $m^3/s$) up each exhaust raise on each footwall drift for a total of up to 169,500 cfm (80 $m^3/s$) on each production level. This quantity will be sufficient to allow for 710 $ft^3/s$ (20 $m^3/s$) for staggered LHD production activities off the footwall drift (see Figure 16-58).

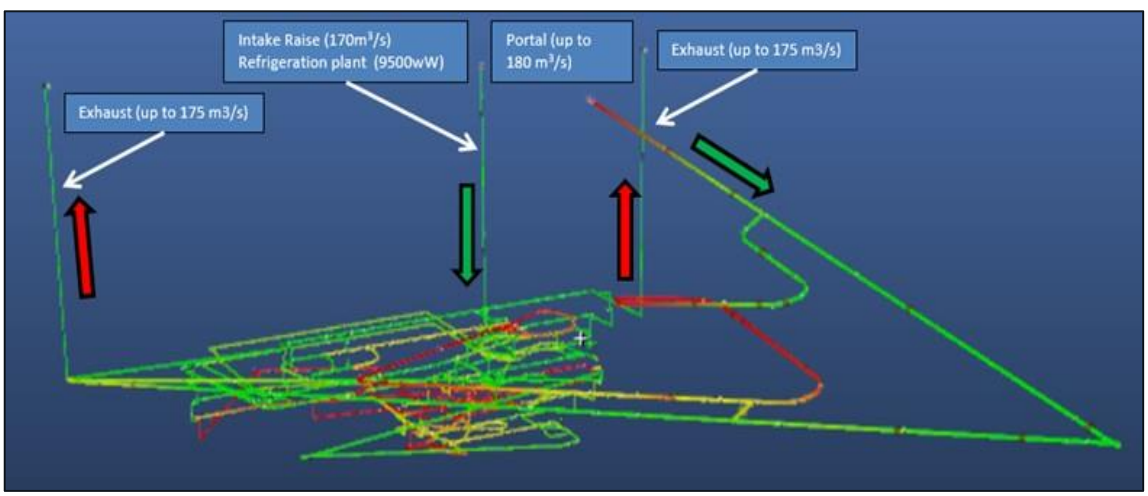
**Figure 16-58: Cactus East Maximum Ventilation Requirement**




Source: BBA 2023.

Parks/Salyer production ventilation will also rely on cooled fresh air in a flow-through ventilation design across the orebody controlled by automated regulators at each end of the foot wall drifts. These will be regulated to allow for up to 1,240 ft$^3$/s (35 m$^3$/s) up each exhaust raise on each footwall drift for a total of up to 2,470 ft$^3$/s (70 m$^3$/s) on each production level. This quantity will be sufficient to allow for 710 ft$^3$/s (20 m$^3$/s) for staggered LHD production activities off the footwall drift (see Figure 16-59).

**Figure 16-59: Parks/Salyer Maximum Ventilation Requirements**




Source: BBA 2023.

#### 16.5.8 Dewatering

The underground dewatering systems for both Cactus East and Parks/Salyer have been designed based on a steady inflow pumping requirement of 500 gpm at the main level sump and pump stations with the capacity to support an emergency outflow condition of 1,500 gpm in the event of a sudden inflow caused by rainfall or aquifer intersection.

Main level sumps are spaced to allow the same set of pumps to be used at each main pump station located throughout both orebodies. Level sumps located on each mining level in between the main level sumps will cascade their flow downwards via borehole and by means of gravity to the nearest sump, in turn relaying the reporting flows to the nearest main level sump. The location of the lowest main sump is set to be above the bottom mining level, in both orebodies, in order to allow for the bottom of the mine to flood in the event that the required pumping rate at any main level sump exceeds 1,500 gpm. A 15 hp submersible sump will be used to pump fluids from the mine's bottom level sump up to the nearest main level sump.

Each main level sump arrangement will consist of excavated drifts, with concrete partitions to form one clean and two dirty sumps. Each dirty sump is provided with a means of access via LHD for mucking out. The intent is to alternate the use of dirty sumps to have one in use while the other is cleaned/maintained.

The dirty sumps are dimensioned to provide one square foot of surface area for each gpm of expected flow during normal operating conditions (500 gpm outflow) based on a drift profile of 16.5 ft (5 m) wide by 16.5 ft (5 m) high. The total length required for each dirty sump is 30.3 ft (9.2 m).

Each clean water sump is sized to provide one full eight-hour shift of filling, at a rate of 400 gpm, in the event of a power interruption, which renders the pumps inoperable. This requires a sump live volume of 192,000 gal. As the drifts are dimensions at 16.5 ft (5 m) wide by 16.5 ft (5 m) high, the total required length of the clean sump is 158 ft.

An array of four x 200 hp horizontal centrifugal pumps, comprising (three duty and one installed spare), will pump the clean water sump via flooded suction pipe that draws water through a pipe penetration in a concrete dam wall. During normal operations, a single pump operates and discharges water through a 6 in pipe to the next main level sump. In an emergency operating condition, three pumps operate through two pump discharge pipes, 6 in and 8 in, to provide a combined outflow rate of 1,500 gpm. There will be one installed spare pump at each pump station.

Water from the mines will discharge at the portal locations.

##### 16.5.8.1 Cactus East Dewatering

The Cactus East dewatering system consists of three typical level sumps and three main level sumps.

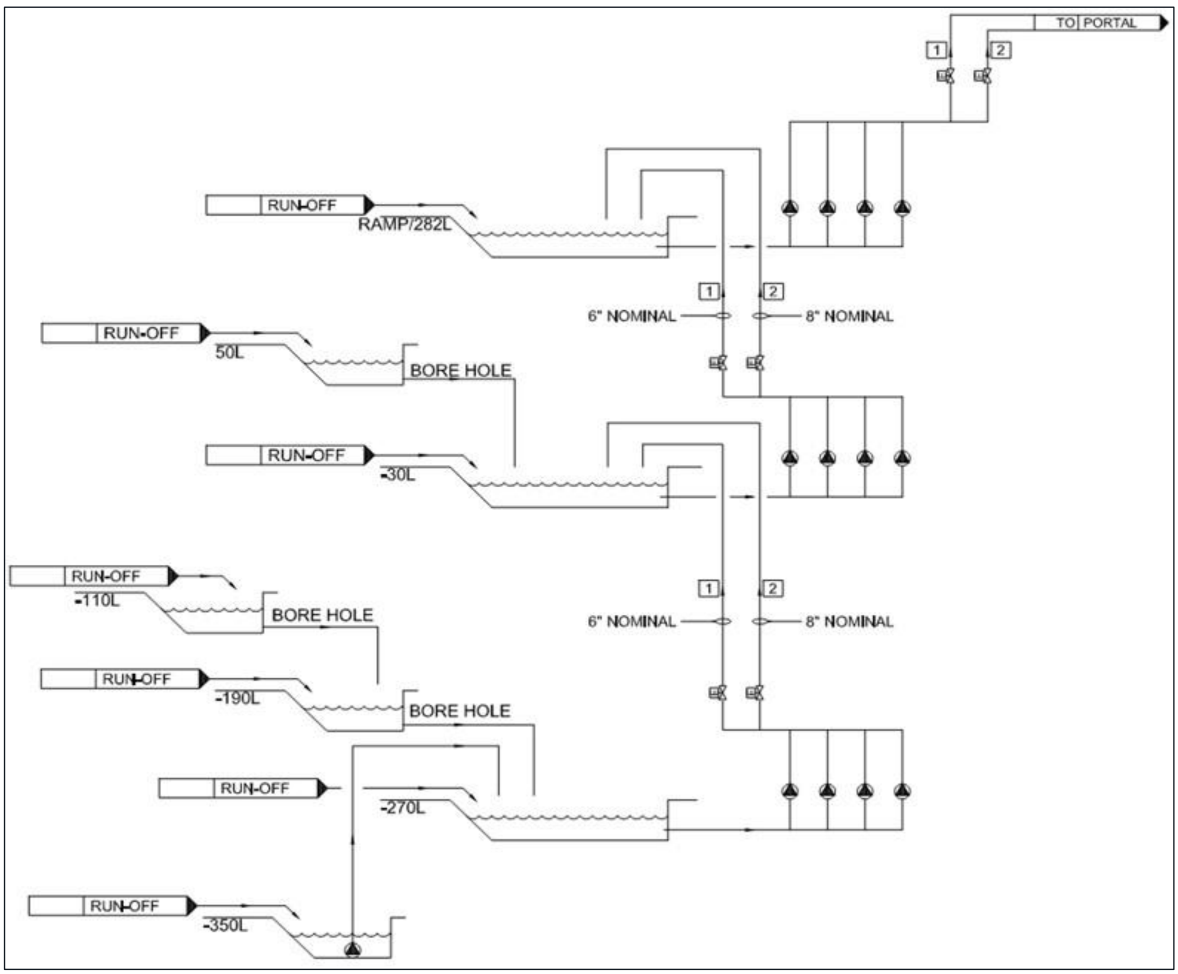
Figure 16-60 is a schematic of the dewatering system Level sumps will cascade their flow downwards via borehole and by means of gravity to the nearest sump, in turn relaying the reporting flows to the nearest main level sump. The level sump located on -350 L will be equipped with a submersible pump directing its flow to the main level sump on -270 L.

Table 16-32 provides a summary of sump locations and pumps installed.

**Table 16-32: Summary of Cactus East Dewatering Sump Requirements**

| Type of Sump | Number of Sumps | Location (s) | Pump Description |
|---|---|---|---|
| Typical Level Sump | 3 | 350 Level (bottom of mine) | 1 x 15 hp submersible |
| | | -190 Level | - |
| | | 50 Level | - |
| Main Level Sump | 3 | -270 Level | 4 x 200 hp horizontal centrifugal |
| | | -30 Level | 4 x 200 hp horizontal centrifugal |
| | | 282 Level (off the ramp) | 4 x 200 hp horizontal centrifugal |

**Figure 16-60: Schematic of Cactus East Dewatering Arrangement**




Source: AGP, 2023.

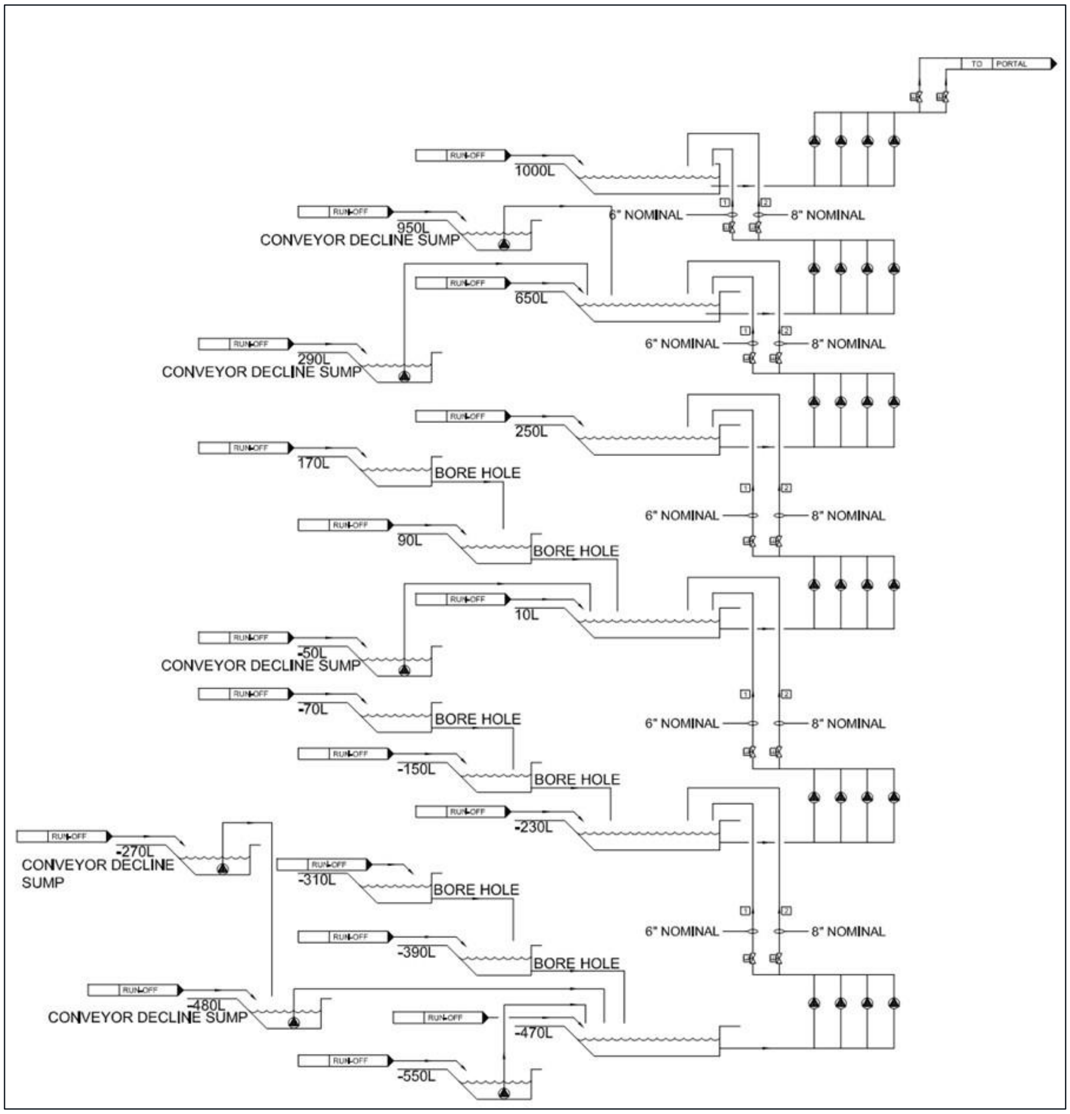
#### 16.5.8.2 Parks/Salyer Dewatering

The Parks/Salyer orebody will be accessed by twin ramps, as there is a separate ramp for the conveyor system. As such, sumps along the conveyor decline are required, each equipped with a 15 hp submersible sump to deliver water to the nearest main level sump, or the nearest conveyor sump, as illustrated in Figure 16-61. The dewatering system consists of seven typical level sumps, five level sumps located along the conveyor decline ramp, and main sumps. Table 16-33 provides a summary of sump locations. Level sumps will cascade their flow downwards via borehole and by means of gravity to the nearest sump, in turn relaying the reporting flows to the nearest main level sump. The level sump located on -550 L will be equipped with a 15 hp submersible pump directing its flow to the main level sump on -470 L.

**Table 16-33: Summary of Parks/Salyer Dewatering Sump Requirements**

| Type of Sump | Number of Sumps | Location (s) | Pump Description |
|---|---|---|---|
| Typical Level Sump | 7 | - -550m Level (bottom) | 1 x 15 hp submersible |
| | | -390m Level | - |
| | | -310m Level | - |
| | | 150m Level | - |
| | | 70m Level | - |
| | | 90m Level | - |
| | | 170m Level | - |
| Conveyor Decline Sump | 5 | -480 Level | 1 x 15 hp submersible |
| | | -270 Level | 1 x 15 hp submersible |
| | | -50 Level | 1 x 15 hp submersible |
| | | 290 Level | 1 x 15 hp submersible |
| | | 950 Level | 1 x 15 hp submersible |
| Main Level Sump | 6 | -470 Level | 4 x 200 hp horizontal centrifugal |
| | | -230 Level | 4 x 200 hp horizontal centrifugal |
| | | 10 Level | 4 x 200 hp horizontal centrifugal |
| | | 250 Level | 4 x 200 hp horizontal centrifugal |
| | | 650 Level | 4 x 200 hp horizontal centrifugal |
| | | 1000 Level | 4 x 200 hp horizontal centrifugal |

**Figure 16-61: Schematic of Parks/Salyer Dewatering Arrangement**




Source: AGP, 2023.

### 16.5.9 Power Distribution

The mine electrical infrastructure includes provisions to support access portal development, dewatering pump stations, conveyors, crushers, ventilation, exploration activities and underground communications. All underground feeds for both Parks/Salyer and Cactus East terminate in switchgears on surface for connection to one or more overhead (surface) power distribution feeders. As the facility is grid connected, substation upgrades will be required. Estimated loads at the connection point to the 13.8 kV gear are expected to peak at:

- East Cactus – 10,425 kVA
- Parks/Salyer – 17,750 kVA

Should the feed be supplied by two feeders, it would be likely that the bus would be split with a normally open tie breaker connecting both bus sections.

Most of the power distribution design is looped (redundant) in nature, not only to supply load levels, but to provide failover should a cable develop a fault or require maintenance. Loop feeds are brought to the surface switch gear to connection to one or more surface power feeders. Most loads on the site will be fed with a high resistance ground system, created by installing a neutral grounding resistor on the secondary (4,160 V or 480 V) side of the neutral of each power transformer. This will limit fault currents for single-line-ground faults. However, 120/208 V auxiliary systems will be solidly grounded to permit line to neutral (120 V) loads.

High resistance grounded (HRG) systems are very common in industrial and mining applications. HRG systems provide high availability, reliability and safety for personnel and equipment. Ground fault protection on these systems provides additional safety. It should also be noted that the grounding resistors do require monitoring equipment to ensure proper operation and the ability to respond to failures.

For underground power distribution, mine power feeders are proposed to leave the surface substation and feed the underground workings (through the portal) through a set of fuses or breakers, this will provide isolation and protection for the underground portion of the circuit. The underground feeder cable will be spliced with junction boxes at each proposed underground substation and will also include sectionalizing load break switches at regular intervals so that portions of the system may be de-energized for faults or maintenance (typically at each mine power station).

A looped configuration for both mines is proposed; allowing damaged sections of cabling to be isolated and repaired and allowing flexibility in the power distribution system in order to limit impacts to the mining operations. However, it should be noted that when peak loading is expected (eg. after a large rainfall) some load curtailment may be required to ensure all cable segments remain within loading limits.

As Parks/Salyer is a relatively larger and a more complex deposit with conveyors and crushers; power will be distributed via two sets of cables (four feeders) to supply two loops. One loop will follow the ramp and feed primarily pumping loads, while the other will primarily feed conveying and crushing loads.

The underground mine power centres contain breakers; while they may be ordered with motor starters, it was assumed that pumps and other loads will have VFDs, or starters supplied in external cabinets. Fuses and disconnects will be provided for portable mine construction power substations and radial feeder branches to remote substations.

Surface loads in close proximity to the portal are connected directly to the surface power distribution switchgear. Substations are provided to supply the cooling, ventilation, and hoisting loads directly at 4,160 V.

By adding current transformers, potential transformers, and metering equipment; greater operational flexibility can be attained by allowing the central control room to monitor current on all key pieces of infrastructure and avoiding overload by curtailing loads as required. It would be assumed that metering would be added at the surface switchgear at a minimum on each primary underground feeder cable.

#### 16.5.9.1 Cactus East Underground Power Distribution

Nine main substations will be established, each supplied with a local Mine Power Centre (MPC) installed at a nearby ESS:

- Surface Level (Switching Station with Connection for Mobile Construction Substation MPC)
- Surface Level Cooling and Hoisting MPCs (x2)
- 282 Level
- -30 Level
- -170 Level
- -270 Level (Pumping)
- -270 Level (Primary underground fan)
- -350 Level

Additionally, there will be two portable construction mine power centres (MPCs). These are to be connected to a spare fused disconnect at 13.8 kV at various locations throughout the mine as construction and development progresses. The intent is for these units to be relocated as development progresses. Further, these units allow high voltage power to be brought closer to the load centres to reduce copper costs and cabling requirements of extensive 480 V distribution.

Each permanent substation is equipped with several gang operated switches to sectionalize the main feeder loop within the mine and provide redundancy. Switching will allow sections of cable to be taken out of service for faults or maintenance activities. Fused disconnect switches are provided to feed radial sections of cable for remote mine substations and other loads located at a distance from the primary underground cabling. Switches are also provided with a connector for quick connection of the portable MPCs.

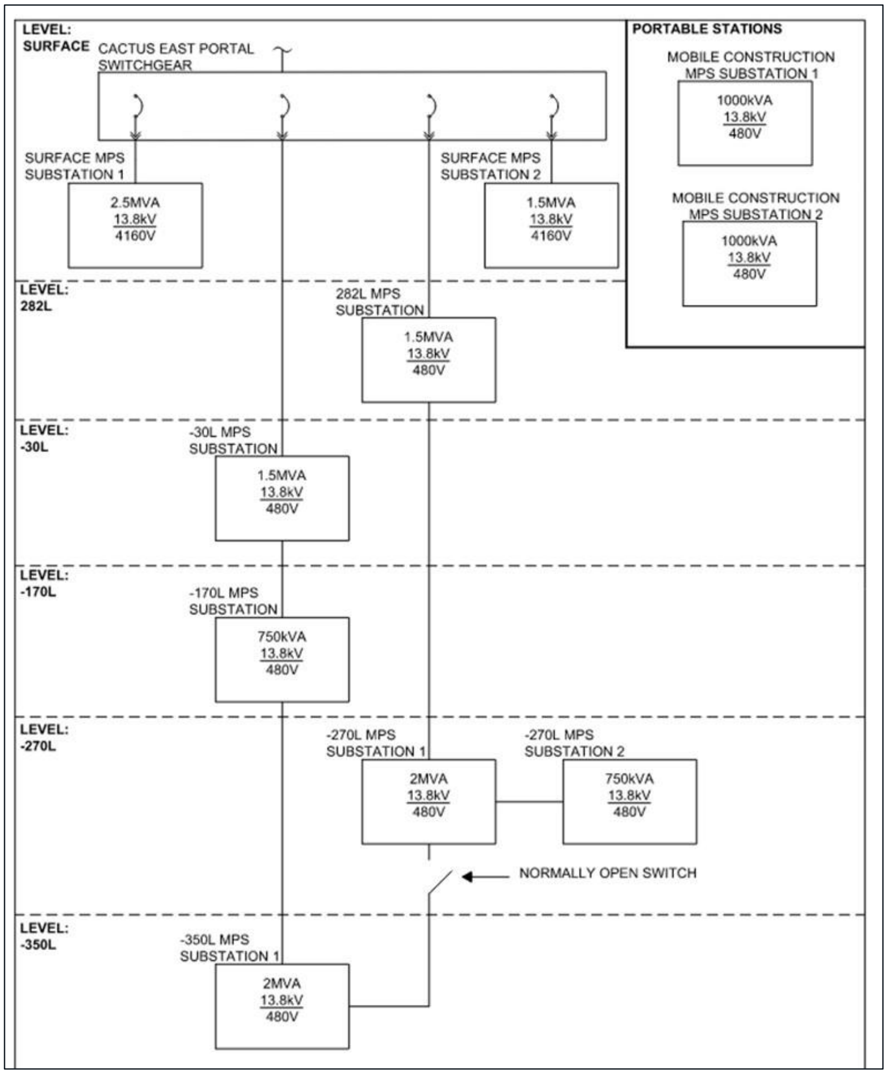
Assumptions have been made to include auxiliary fans connected to several of the permanent MPCs, while others may be temporary and connected to the portable construction MPCs at 480 V. Pumps are assumed to be supplied with a soft starter as part of the mechanical supply package. Power Take Off (PTO) panels have also been included at many points along the ramp for connection of bolters and other mining equipment.

On surface, one mine power substation is provided to supply cooling units and a primary fan at 4160 V. The cooling plants are assumed to be delivered as a prepackaged unit, where incoming power will be provided directly from the 4160 V bus. However, the primary fan will require a VFD, and a cost has been carried in the electrical cost estimate. Additionally, a second MPC will provide 4160 V to a vertical conveyor motor on surface.

The 120/208V loads would be supplied by local dry type distribution transformers and panel boards. IT equipment, local lighting and other ancillary loads would be connected to these panels.

Shops and maintenance facilities would be supplied at 480 V from the nearest available MPC. A high-level overview of the power system for the Cactus East deposit is shown in Figure 16-62.

**Figure 16-62: High Level Overview of Cactus East Power Distribution**




Source: AGP, 2023.

#### 16.5.9.2 Parks/Salyer Underground Power Distribution

Twenty-two main substations will be established, each supplied with a local Mine Power Centre (MPC) installed at a nearby ESS. There will be two loops fed by four feeders emanating from the main Surface Level Switching Station.

Loop 1 is assumed to be comprised of the stations:

- Surface Level (Switching Station with Connection for Mobile Construction Substation MPC)
- Surface Level Conveyor Substation

- Surface Level Cooling MPCs (x3)
- 950 Level
- 950 Level Conveyor Substation
- 290 Level
- 290 Level Conveyor Substation
- -50 Level
- -50 Level Conveyor Substation
- -270 Level Conveyor Substation
- -310 Level

Loop 2 is assumed to be comprised of the following stations:

- Surface Level (Switching Station with Connection for Mobile Construction Substation MPC)
- 1,000 Level
- 650 Level
- 250 Level
- 10 Level
- -230 Level
- -470 Level
- -480 Level
- -550 Level

Additionally, there will be four portable construction mine power centres (MPCs). These are to be connected to a spare fused disconnect at 13.8 kV at various locations throughout the mine as construction and development progresses. The intent is for these units to be relocated as development progresses. Further, these units allow high voltage power to be brought closer to the load centres to reduce copper costs and cabling requirements of extensive 480 V distribution.

Each permanent substation is equipped with several gang operated switches to sectionalize the main feeder loop within the mine and provide redundancy. Switching will allow sections of cable to be taken out of service for faults or maintenance activities. Fused disconnect switches are provided to feed radial sections of cable for remote mine substations and other loads located at a distance from the primary underground cabling. Switches are also provided with a connector for quick connection of the portable MPCs.

Assumptions have been made to include auxiliary fans connected to several of the permanent MPCs, while others may be temporary and connected to the portable construction MPCs at 480 V. Pumps are assumed to be supplied with a soft starter as part of the mechanical supply package. Power Take Off (PTO) panels have also been included at many points along the ramp for connection of bolters and other mining equipment.

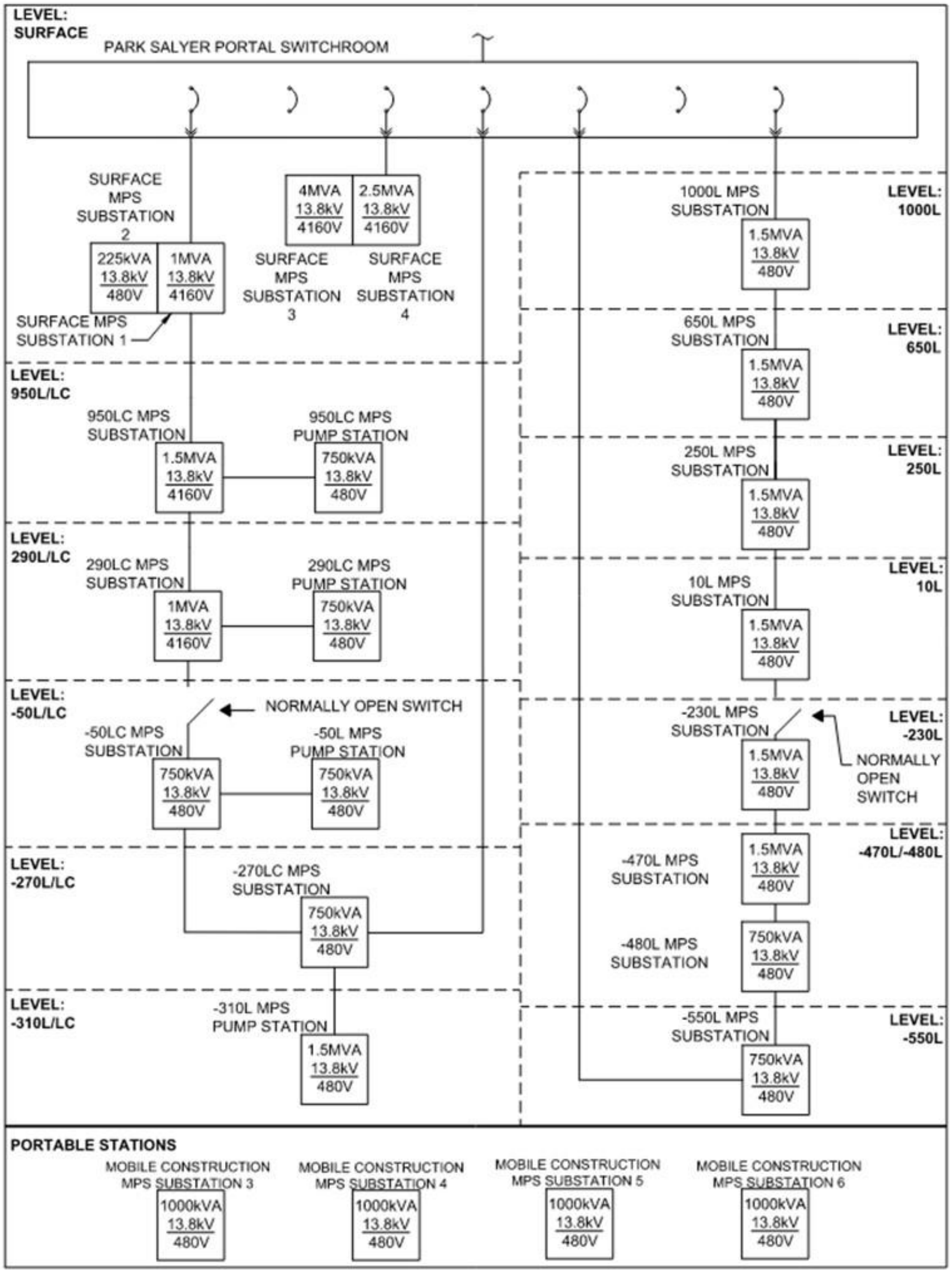
Conveying equipment will be served from dedicated Conveyor Substations. These units will be similar to the 480V stations, but will operate at 4160 V to supply larger conveyor motors in the range of 600-1000 HP. Where no other substation is within close proximity, local step-down transformers to 480 V and/or 120/208 V will be provided to supply local loads within the vicinity of the conveyor system.

The 120/208 V loads in proximity to pumping and production areas would be supplied by local dry type distribution transformers and panel boards located at the proposed substations. IT equipment, local lighting and other ancillary loads would be connected to these panels.

Shops and maintenance facilities would be supplied at 480 V from the nearest available MPC.

A high-level overview of the power system for the Parks/Sayler deposit is shown in Figure 16-63.

**Figure 16-63: High Level Overview of Parks/Salyer Power Distribution**




Source: AGP, 2023.

#### 16.5.10 Mine Communications

The underground communications systems are proposed to be ethernet based and leaky feeder based for radio communications. The proposed topology for communications and automation will be through a primary fiber optic trunk network. Fiber interface panels will be located at key locations to provide interconnectivity to associated users. Primary user networks will be:

- Voice – VOIP phones for communications in shop spaces and other strategic locations
- Business Network – provide wired and wireless connectivity to the business network for mine resources to connect to the internet, email, and other business applications while in the mine.
- Process Network – provide a communications network for programmable logic controllers (PLCs), distributed control systems (DCS) and other process specific devices. This network would be secured and access to the business network and internet would be restricted by firewall to protect the integrity of the control systems. Other systems on this network include geotechnical monitoring, personnel tracking, and vehicle telemetry.
- Security – provide a dedicated VLAN for the connection of underground cameras to a remote digital video recorder (DVR).

The fiber/ethernet network is designed with site communications in mind; however, a router and connection to external networking would be provided in either the surface substation or in the office/administrative space.

Leaky feeder coverage is also assumed to be for underground operations only. By providing leaky feeder communications, mine wide Wi-Fi network coverage is not required, and reliable communications can be extended into each mining zone. As new zones are developed, leaky feeder coax can be T-tapped and spliced throughout the mine by site electricians.

Both the leaky feeder system and ethernet/fiber system cables are extended as the mine drift and ramps are developed and will connect to devices as required by the mine instrumentation personnel.

#### 16.5.11 Safety

##### 16.5.11.1 Stench Gas System

The emergency notification system will be in the form of a stench gas system which can be released into the fresh air stream at both the main portals and in the main fresh air raises. These systems are simple and very effective in a high-volume intake system as proposed at both mines.

##### 16.5.11.2 Emergency Egress

At Parks/Salyer the main ramp is planned to provide primary egress from the underground workings. The conveyor drift ramp has been located in excess of 100 ft from the main ramp to facilitate an independent secondary emergency egress. Connections between the main ramp and the return air ramp are mined at approximately 500-foot intervals. A ventilation control bulkhead with man-access will be installed in each connection.

Ladderway systems will also be installed in the return air raise from each sub-level to the level above providing secondary egress from each sub-level in every mining area.

At Cactus East, there is a single main ramp access. In order to provide a secondary emergency egress to surface a ladderway will be installed in one of the fresh air intake raises. Again, ladderway systems will also be installed in the return air raise from each sub-level to the level above providing secondary egress from each sub-level in every mining area.

#### 16.5.11.3 Refuge Stations

Purpose built self-contained portable refuge stations will be installed at specific locations within the underground workings. The refuge stations can be moved to new locations as the mine expands and areas of activity change. It has assumed that a total of six 16-man capacity refuges will be provided. The refuge stations will be equipped with compressed air, potable water, and first aid equipment. They will also be equipped with a fixed telephone line and emergency lighting. The refuge chambers will be sealable to prevent the entry of gases.

As initial development is completed permanent refuges/lunchrooms will be provided at suitable locations in each mine.

Self-rescuers will be allocated to personnel to ensure safe passage to refuge chambers in case of smoke or gas.

#### 16.5.11.4 Mine Rescue

A fully trained and equipped Mine Rescue Team is essential to the safe operation of any mine and shall be provided at the Cactus Project with a dedicated rescue centre and equipment. Team members will be drawn from volunteers from the mine workforce. The mine rescue team will be trained for surface and underground emergencies. A dedicated mine rescue/first aid vehicle will be purchased.

#### 16.5.11.5 Fire Prevention

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground refuge stations, electrical substations, pump stations, fuelling stations, explosive magazines, and other strategic areas. Every vehicle would carry at least one fire extinguisher; the correct size and type will depend on the type of vehicle. All underground heavy equipment will be equipped with automatic fire suppression systems (Ansul system).

#### 16.5.11.6 Traffic Control

A traffic control system will be installed in the main access ramp and at other strategic locations at both Cactus East and Park Salyer. Provision for this system has been included in the mine communications systems.

### 16.5.12 Underground Mine Development and Production Schedules

The development schedule within Deswik employs an Effort driven methodology. A resource is assigned as Effort-driven if the resource's production rate, as opposed to the task duration, will determine the number of resources used to complete a task. When a resource is assigned as Effort-driven:

- The duration of the task is not changed to maximize the use of the resource if the resource is not used to its full potential.
- Then additional resources must be assigned if one resource is not enough to complete a task over the task's duration.

The development was resourced accordingly to support the production profile requirement. Each task had their specific task rate based on heading availability. Development zones were developed based on the geotechnical domains provided to constrain advance through the SLC mine.

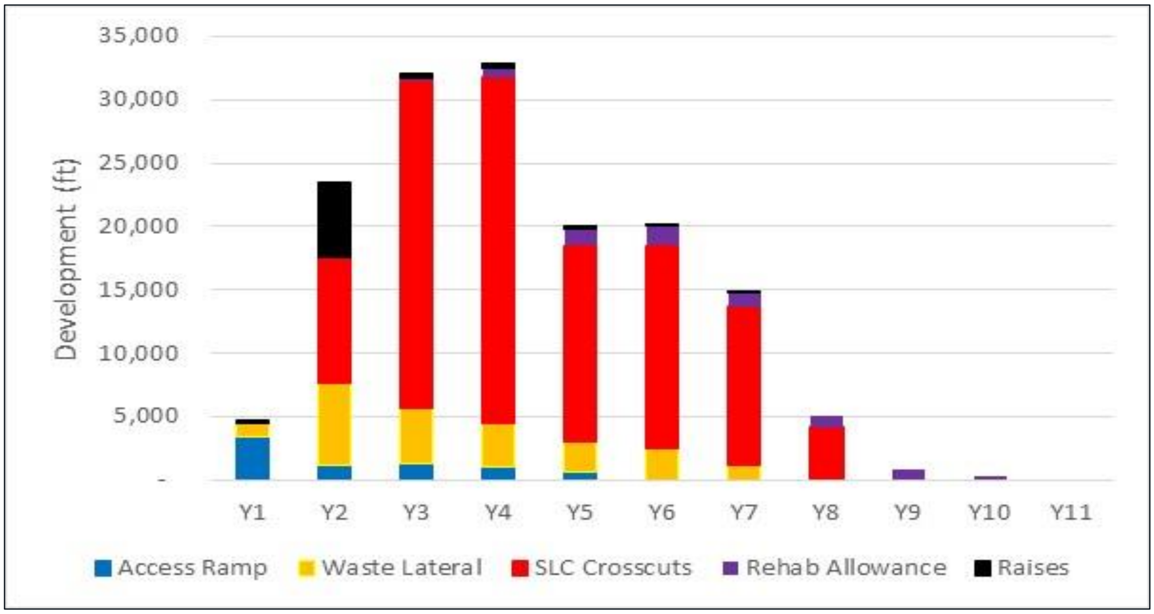
The production schedule within Deswik employs a driving scheduling methodology. A resource is assigned as Driving if the duration of the task will be determined by the production rate and the number of resources available. The footprint size and cave front propagation was used for the determination of equipment requirement. This included the loaders, production drills and raisebores.

The production was resourced accordingly to the drawpoint availability, numbers of levels open for production and production ramp up of the SLC to reach maximum the throughput rate. The through rate was then maintained consistently across the life of mine. The resources were reduced based on the decline on production rates across the mine life.

All development and production tasks were linked, creating dependencies, based on mining SLC guidelines and requirements and geotechnical guidelines for caving progression and stress regimes.

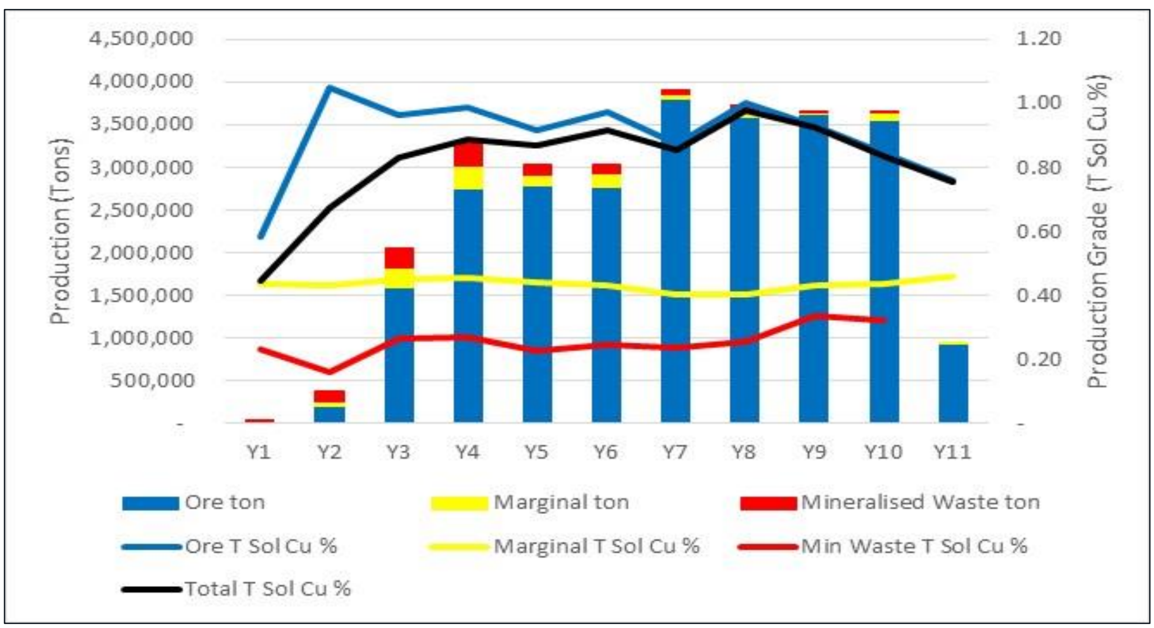
Each mine was scheduled separately and individually commencing with a Year 1. Subsequently the schedules were included in the consolidated project schedules commencing in the best calendar years to meet corporate objectives. Year 1 for Cactus East was taken to be 2029 and for Year 1 at Parks/Salyer 2031 was selected (See Section 16.6). The development and production schedules for Cactus East and Parks/Salyer are shown in Figure 16-64 through Figure 16-67.

**Figure 16-64: Cactus East Development Schedule**



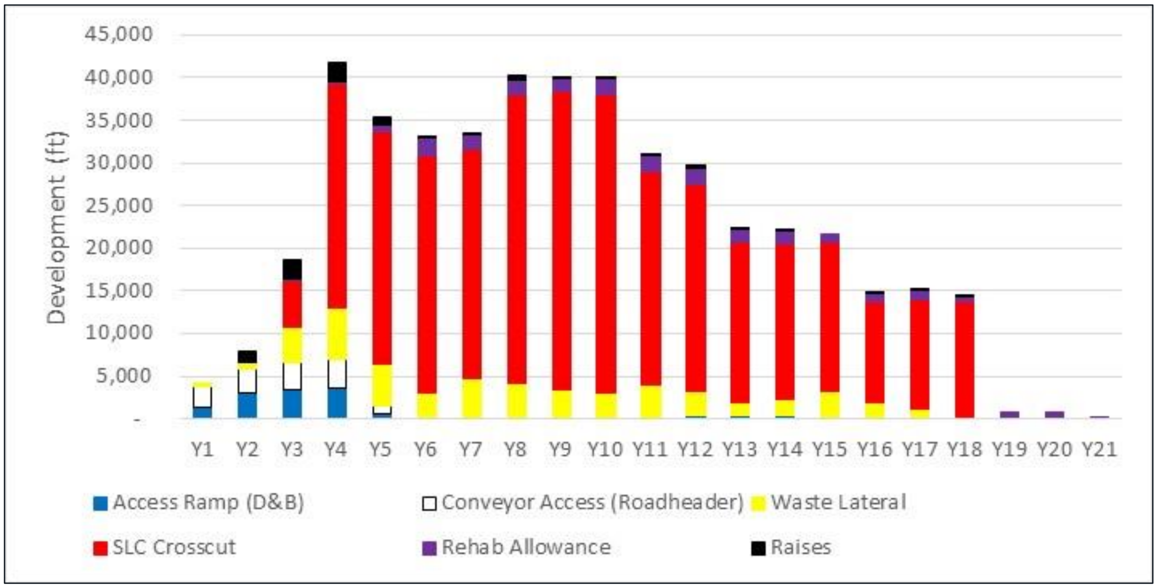

Source: AGP, 2023.

**Figure 16-65: Cactus East Production Schedule**



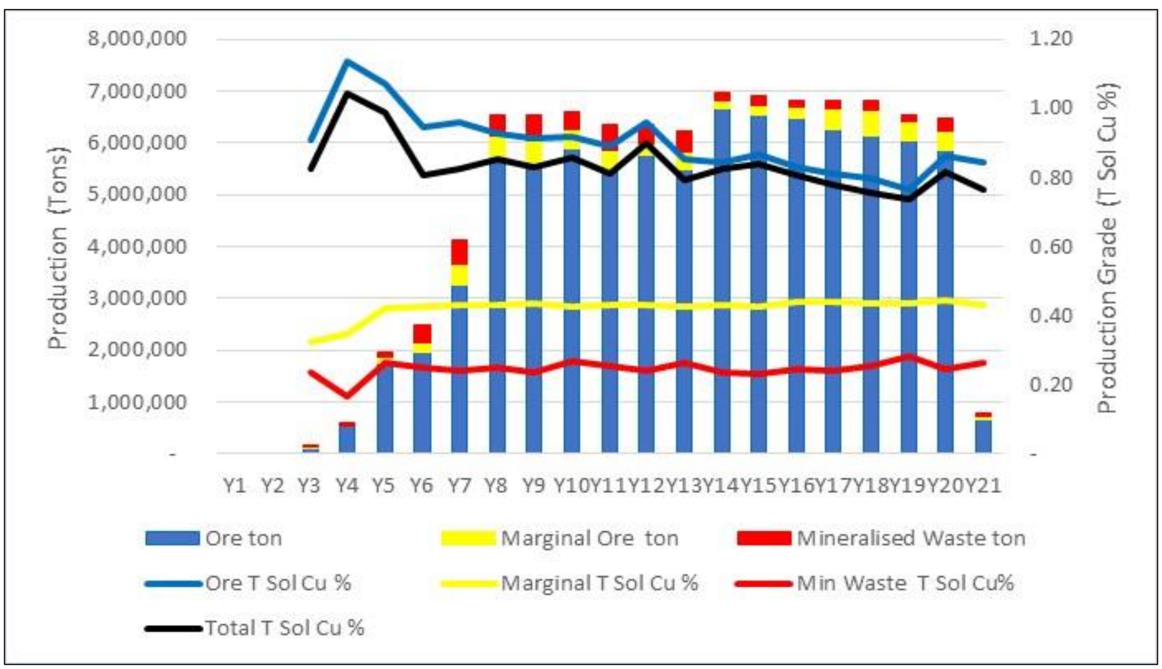

Source: AGP, 2023.

**Figure 16-66: Parks/Salyer Development Schedule**




Source: AGP, 2023.

**Figure 16-67: Parks/Salyer Production Schedule**




Source: AGP, 2023.

### 16.5.13 Underground Mine Costing Methodology

The direct capital development and operating costs for the underground mines were generated from first principal unit cost models. Each of the models was developed using the mine design criteria and other general engineering estimates of performance. The mine was assumed to operate two 12-hour shifts per day, 365 days per year. Costs were estimated on a quarterly basis for the first ten years and annually thereafter. All costs were modelled in 2023 Q2 US dollars.

Wherever possible the mine consumable cost database was updated locally during the course of the study. Labour costs were derived from a recent underground feasibility study in Arizona. Budget quotations were provided by mobile equipment suppliers.

Separate drill and blast development cost models included detailed design and ground support assumptions for each mine and each different rock type as provided by CNI. Separate models were created for conveyor drift development by roadheader at Parks/Salyer. Other models were developed for application to the other mine activities, raising, stope drilling and blasting, stope mucking, trucking, and delineation drilling. The unit rates were applied to the scheduled quantities in order to estimate the direct costs.

Initial development to first main stoping production was assumed to be undertaken by contractors. The contractors will provide all labour, consumables and equipment until Year 3 Q2 at Cactus East and Year 5 Q1 at Parks/Salyer, during these periods ASCU will provide only contract supervision and technical services. Thereafter all activities will be undertaken by owner crews apart from roadheader development and raising which will continue to be undertaken by reduced contractor crews.

Additional models were designed to reflect overhead-type activities at the mines:

- Mine Services (including Labour, supplies and equipment for construction, materials transport, road maintenance and sanitation). Diesel maintenance Labour costs are also included.
- Vertical Conveying and Sizing at Cactus East.
- Ramp conveying and Sizing at Parks/Salyer.
- Owners Mine Supervision and Technical (including mine management, production supervision, maintenance supervision, and mine technical and safety staff).
- Air Cooling.
- Mine Power (developed from aggregation of mine loads and estimated usage).

Overheads were estimated by quarter and applied as a fixed daily cost. The overheads for each period were split between operating and capital development estimates in the ratio of the respective direct costs.

The models were also used to track Labour and equipment hours to identify annual requirements in each Labour category and equipment type.

All owner mobile equipment will be leased with 15% downpayment followed by a five-year lease at 8.3% pa interest.

Replacement capital for fixed plant was included in the daily overhead cost estimates. Replacement capital for mobile mining equipment was estimated by tracking equipment fleet operating hours with a mid-life rebuild equivalent to 50% of the purchase price. Rebuild capital is not subject to leasing. Replacement equipment will be purchased after assumed useful life, subject to the normal leasing criteria.

Equipment downpayments and rebuild costs are capitalised. Lease payments are divided between capital development and operating costs in the ratio of each period's direct costs.

#### 16.5.14 Underground Labour

Hourly paid employees will workday, afternoon and night shifts, each of 12 hours. Two crews will be employed to allow for continuous operations, 365 mine operating days per year. Labour requirements were tracked during the cost modelling process. The estimate of effective working hours during the shift for hourly paid underground workers is shown in Table 16-34.

**Table 16-34: Effective Working Time**

| Factor | Unit | Value |
|---|---|---|
| Shift length | h | 12.0 |
| Travel time | h | 1.00 |
| Safety huddle | h | 0.50 |
| Breaks | h | 1.00 |
| Efficiency Factor (50 min/h) | % | 83 |
| Effective Hours | h/shift | 7.9 |

The most senior managers and superintendents may have shared responsibility between the two underground and open pit operations, however, for the purposes of this study each of the underground operations assumes its own dedicated workforce. The majority of staff will work 8 hours per day, 5 days per week. Some job categories will be filled one or two shifts per day basis depending on the position to support continuous operations. A duty roster and call-out system will be employed to ensure effective coverage for ongoing operation during off-duty time.

Hourly paid employed labour for selected periods during the Cactus East life of mine are shown in Table 16-35. Employed staff for the same periods are shown in Table 16-36.

**Table 16-35: Cactus East Employed Hourly Labour**

| | Yr 2 Q1 | Yr 4 Q1 | Yr 6 Q1 | Yr 8 Q1 | Yr 11 |
|---|---|---|---|---|---|
| Development Miner, Miner 1 | - | 5 | 4 | 2 | - |
| Longhole Drilling Miner 1 | - | 21 | 20 | 27 | 27 |
| Development Miner, Miner 1 Lead | - | 28 | 17 | 7 | - |
| Scoop Driver | - | 20 | 16 | 19 | 20 |
| Stope Blasting Miner 1 | - | 10 | 10 | 13 | 13 |
| Construction Lead | - | 4 | 4 | 4 | 4 |
| Construction | - | - | - | - | - |
| Materials | - | - | - | - | - |
| Truck Driver | - | 2 | 3 | 13 | 19 |
| Refrigeration UG | - | 4 | 4 | 4 | 4 |
| Pumps | - | 4 | 4 | 4 | 4 |
| Road Maintenance | - | 2 | 2 | 2 | 2 |
| Diesel Mechanic I | - | 24 | 18 | 18 | 16 |
| Diesel Mechanic 2 | - | 24 | 18 | 18 | 16 |
| Diesel Mechanic 3 | - | 24 | 18 | 18 | 16 |
| Mechanic I | - | 5 | 5 | 5 | 5 |
| Mechanic 2 | - | 5 | 5 | 5 | 5 |
| Mechanic 3 | - | 4 | 4 | 4 | 4 |
| Electrician I | - | 4 | 4 | 4 | 4 |
| Electrician 2 | - | 5 | 5 | 5 | 5 |
| Electrician 3 | - | 5 | 5 | 5 | 5 |
| Welder | - | 4 | 4 | 4 | 4 |
| Drill Maintenance | - | 4 | 4 | 4 | 4 |
| Conveyor Operator | - | 6 | 6 | 6 | 6 |
| UG Helper | - | 66 | 54 | 54 | 48 |
| Crusher Operator | - | 2 | 2 | 2 | 2 |
| **TOTAL** | **-** | **282** | **236** | **247** | **233** |

**Table 16-36: Cactus East Employed Staff**

| | Yr 2 Q1 | Yr 4 Q1 | Yr 6 Q1 | Yr 8 Q1 | Yr 11 |
|---|---|---|---|---|---|
| Maintenance Supt | - | 1 | 1 | 1 | 1 |
| Maintenance Foreman | 1 | 2 | 2 | 2 | 2 |
| Maintenance Planner | - | 1 | 1 | 1 | 1 |
| Mine Superintendent | 1 | 1 | 1 | 1 | 1 |
| Mine Foreman | - | 2 | 2 | 2 | 2 |
| Shift Boss | 4 | 16 | 16 | 16 | 16 |

| | Yr 2 Q1 | Yr 4 Q1 | Yr 6 Q1 | Yr 8 Q1 | Yr 11 |
|---|---|---|---|---|---|
| Mine Dry/Lamps/Bits | 4 | 6 | 6 | 6 | 6 |
| Secretary/Clerk/Stores | 2 | 4 | 4 | 4 | 4 |
| Mine Trainers/Safety | - | 1 | 1 | 1 | 1 |
| Safety | 2 | 4 | 4 | 4 | 4 |
| Technical Services Manager | 1 | 1 | 1 | 1 | 1 |
| Senior Mine Eng/Geo/Geotech/Hydro | 2 | 3 | 3 | 3 | 3 |
| Mine Geologist | 1 | 3 | 3 | 3 | 3 |
| Mine/Hydro Technician | 2 | 3 | 3 | 3 | 3 |
| Geology Technician/Grade Control | 2 | 3 | 3 | 3 | 3 |
| Mine Engineer | 2 | 3 | 3 | 3 | 3 |
| Surveyor | 2 | 4 | 4 | 4 | 4 |
| Survey Helper | 4 | 8 | 8 | 8 | 8 |
| Ventilation /Samplers/Rock Mechanics Asst | 4 | 8 | 8 | 8 | 8 |
| **Total Staff** | **34** | **74** | **74** | **74** | **74** |

Hourly paid employed labour for selected periods during the Parks/Salyer life of mine are shown in Table 16-37. Employed staff for the same periods are shown in Table 16-38.

**Table 16-37: Parks/Salyer Employed Hourly Labour**

| | Yr 3 Q1 | Yr 8 Q1 | Yr12 | Yr 16 | Yr 20 |
|---|---|---|---|---|---|
| Development Miner, Miner 1 | | 6 | 4 | 3 | 1 |
| Longhole Drilling Miner 1 | | 33 | 33 | 37 | 37 |
| Development Miner, Miner 1 Lead | | 34 | 25 | 13 | 1 |
| Scoop Driver | | 39 | 41 | 39 | 34 |
| Stope Blasting Miner 1 | | 20 | 20 | 23 | 23 |
| Construction Lead | | 8 | 8 | 8 | 8 |
| Construction | | 16 | 16 | 16 | 16 |
| Materials | | 8 | 8 | 8 | 8 |
| Truck Driver | | 1 | 1 | 1 | 7 |
| Refrigeration UG | 2 | 4 | 4 | 4 | 4 |
| Pumps | | 4 | 4 | 4 | 4 |
| Road Maintenance | | 4 | 4 | 4 | 4 |
| Diesel Mechanic I | | 26 | 26 | 24 | 20 |
| Diesel Mechanic 2 | | 26 | 26 | 24 | 20 |
| Diesel Mechanic 3 | | 26 | 26 | 24 | 20 |
| Mechanic I | 1 | 7 | 7 | 7 | 7 |
| Mechanic 2 | | 7 | 7 | 7 | 7 |

| | Yr 3 Q1 | Yr 8 Q1 | Yr12 | Yr 16 | Yr 20 |
|---|---|---|---|---|---|
| Mechanic 3 | | 6 | 6 | 6 | 6 |
| Electrician I | 1 | 7 | 7 | 7 | 7 |
| Electrician 2 | | 7 | 7 | 7 | 7 |
| Electrician 3 | | 6 | 6 | 6 | 6 |
| Welder | | 6 | 6 | 6 | 6 |
| Drill Maintenance | | 6 | 6 | 6 | 6 |
| Conveyor Operator | | 8 | 8 | 8 | 8 |
| UG Helper | | 87 | 78 | 72 | 61 |
| Crusher Operator | | 8 | 8 | 8 | 8 |
| **TOTAL** | **4** | **410** | **392** | **372** | **336** |

**Table 16-38: Parks/Salyer Employed Staff**

| | Yr 3 Q1 | Yr 8 Q1 | Yr12 | Yr 16 | Yr 20 |
|---|---|---|---|---|---|
| Maintenance Supt | | 1 | 1 | 1 | 1 |
| Maintenance Foreman | | 3 | 3 | 3 | 3 |
| Maintenance Planner | | 1 | 1 | 1 | 1 |
| Mine Superintendent | 1 | 1 | 1 | 1 | 1 |
| Mine Foreman | | 3 | 3 | 3 | 3 |
| Shift Boss | 4 | 24 | 24 | 24 | 16 |
| Mine Dry/Lamps/Bits | 4 | 6 | 6 | 6 | 6 |
| Secretary/Clerk/Stores | 2 | 4 | 4 | 4 | 4 |
| Mine Trainers/Safety | | 2 | 2 | 2 | 2 |
| Safety | 2 | 4 | 4 | 4 | 4 |
| Technical Services Manager | 1 | 1 | 1 | 1 | 1 |
| Senior Mine Eng/Geo/Geotech/Hydro | 2 | 3 | 3 | 3 | 3 |
| Mine Geologist | 2 | 4 | 4 | 4 | 3 |
| Mine/Hydro Technician | 2 | 4 | 4 | 4 | 3 |
| Geology Technician/Grade Control | 2 | 6 | 6 | 6 | 4 |
| Mine Engineer | 2 | 3 | 3 | 3 | 2 |
| Surveyor | 2 | 4 | 4 | 4 | 2 |
| Survey Helper | 4 | 8 | 8 | 8 | 4 |
| Ventilation /Samplers/Rockmechanics Asst | 4 | 12 | 12 | 12 | 8 |
| **Total Staff** | **34** | **94** | **94** | **94** | **71** |

### 16.5.15 Underground Mine Equipment

Modelled equipment requirements are based on operational hours. Some potential may remain for fleets to be reduced by rescheduling and optimization of activities during the feasibility study process. It is planned that the development contractor will provide all mobile equipment during the project capital phase for each mine. The owner will purchase equipment required for operations thereafter under a lease arrangement.

Typical current leasing terms were provided by a major supplier during the study and comprise a 15% down payment on acquisition followed by a five-year lease at the rate of 8.3% Pa. The leasing cost estimate was derived from quotations for the PFS and other recent AGP projects were used for the equipment types selected. These quotations were escalated by 3% Pa from the date of quotation where necessary. An allowance for initial spare parts (5%) was included in the purchase price used for modeling, but freight to site was excluded. Mechanical availability, utilisation and operational life were estimated by AGP for each equipment type and the hourly operating costs were assessed. A mid-life rebuild equivalent to 50% of the purchase price was included in the capital estimate to increase operational life.

The mobile equipment requirements during selected periods of the mine life are provided in Table 16-39 for Cactus East and Table 16-40 for Parks/Salyer.

**Table 16-39: Cactus East Owner Mobile Equipment Requirements**

| | Yr2 Q1 | Yr 4 Q1 | Yr 6 Q1 | Yr 8 Q1 | Yr11 |
|---|---|---|---|---|---|
| 4 yd Scoop | | 2 | 2 | 1 | 1 |
| 11yd Scoop | | 5 | 3 | 5 | 6 |
| 6yd Scoop | | 4 | 3 | 2 | 1 |
| 55-ton Diesel Truck | | 6 | 2 | 5 | 6 |
| Development Jumbo | | 4 | 3 | 2 | 1 |
| Longhole Drill | | 6 | 6 | 7 | 8 |
| Rockbolter | | 6 | 4 | 3 | 1 |
| Scissors | | 7 | 5 | 3 | 1 |
| Fuel & Lube Service | | 2 | 2 | 2 | 2 |
| Flatdeck | | 2 | 2 | 2 | 2 |
| Supv Landcruiser | | 14 | 14 | 10 | 10 |
| 8 Man Landcruiser | | 3 | 3 | 3 | 3 |
| Mech Landcruiser | | 2 | 2 | 2 | 2 |
| Elec Landcruiser | | 2 | 2 | 2 | 2 |
| Emulsion Loader | | 4 | 4 | 4 | 4 |
| Mobile Rockbreaker | | 1 | 1 | 1 | 1 |
| Rescue/First Aid | | 1 | 1 | 1 | 1 |
| Grader | | 1 | 1 | 1 | 1 |
| Transmixer | | 2 | 2 | 2 | 1 |

| | Yr2 Q1 | Yr 4 Q1 | Yr 6 Q1 | Yr 8 Q1 | Yr11 |
|---|---|---|---|---|---|
| Shotcrete | | 2 | 2 | 2 | 1 |
| Scissors/Pipe Handler | | 1 | 1 | 1 | 1 |
| Water Truck | | 1 | 1 | 1 | 1 |

**Table 16-40: Parks/Salyer Owner Mobile Equipment Requirements**

| | Yr 3 Q1 | Yr 8 Q1 | Yr12 | Yr 16 | Yr 20 |
|---|---|---|---|---|---|
| 4 yd Scoop | | 3 | 3 | 3 | 2 |
| 11yd Scoop | | 10 | 13 | 14 | 14 |
| 6yd Scoop | | 3 | 3 | 2 | 1 |
| 45-ton Diesel Truck | | 2 | 2 | 2 | 2 |
| Development Jumbo | | 4 | 4 | 3 | 1 |
| MH621 Roadheader | | | | | |
| Longhole Drill | | 9 | 9 | 10 | 10 |
| Rockbolter | | 6 | 4 | 3 | 1 |
| Scissors | | 6 | 6 | 5 | 4 |
| Fuel & Lube Service | | 2 | 2 | 2 | 2 |
| Flatdeck | | 2 | 2 | 2 | 2 |
| Supv Landcruiser | | 14 | 14 | 14 | 10 |
| 8 Man Landcruiser | | 3 | 3 | 3 | 3 |
| Mech Landcruiser | | 4 | 4 | 4 | 4 |
| Elec Landcruiser | | 3 | 3 | 3 | 3 |
| Emulsion Loader | | 7 | 7 | 7 | 7 |
| Mobile Rockbreaker | | 1 | 1 | 1 | 1 |
| Rescue/First Aid | | 1 | 1 | 1 | 1 |
| Grader | | 1 | 1 | 1 | 1 |
| Transmixer | | 2 | 2 | 2 | 2 |
| Shotcrete | | 2 | 2 | 2 | 2 |
| Scissors/Pipe Handler | | 2 | 2 | 2 | 2 |
| Water Truck | | 1 | 1 | 1 | 1 |

### 16.5.16 Underground Power

A load list was compiled for the underground mine. Estimated total installed power for Cactus East is shown in Table 16-41.

**Table 16-41: Cactus East**

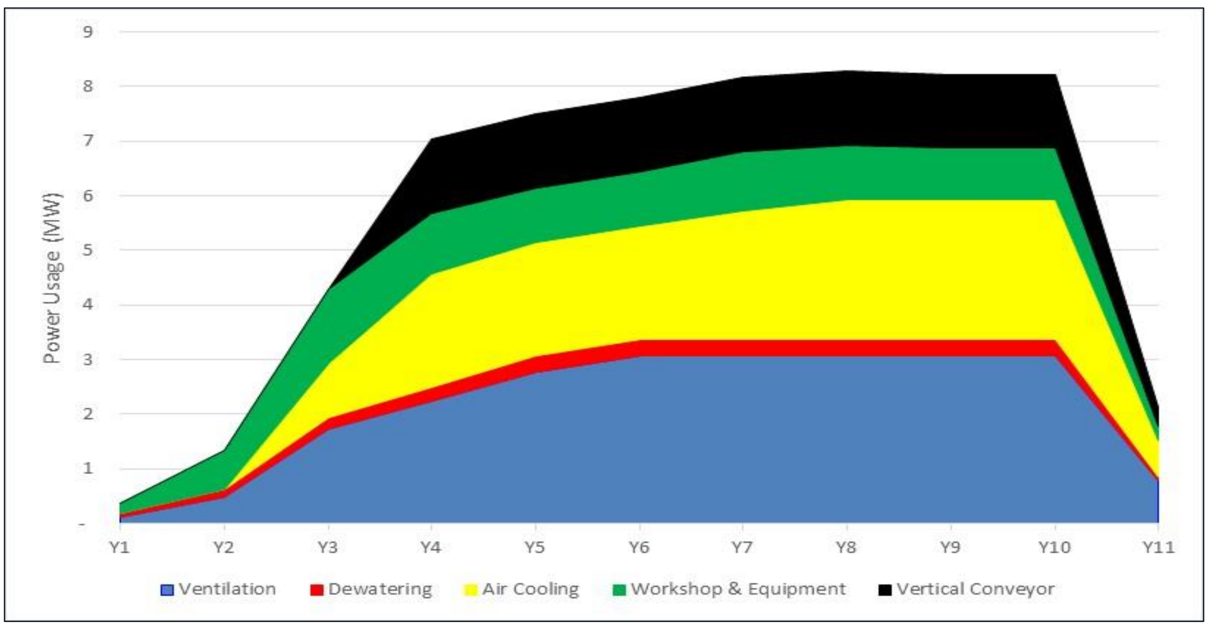
| Area | Installed Power (kW) |
|---|---|
| Ventilation | 3,900 |
| Dewatering | 898 |
| Air Cooling | 4,000 |
| Workshop & Equipment | 3,594 |
| Vertical Conveyor System | 2,296 |
| **Total Installed kW** | **14,688** |

With reference to the activity schedules and milestone achievements AGP reviewed each line item of the load list on a period-by-period basis to estimate the power requirements by quarter.

A power cost of $0.071/kWh was then applied to estimate the power cost.

The resulting schedule of utilised power for Cactus East is shown in Figure 16-68.

**Figure 16-68: Cactus East Power Utilized**



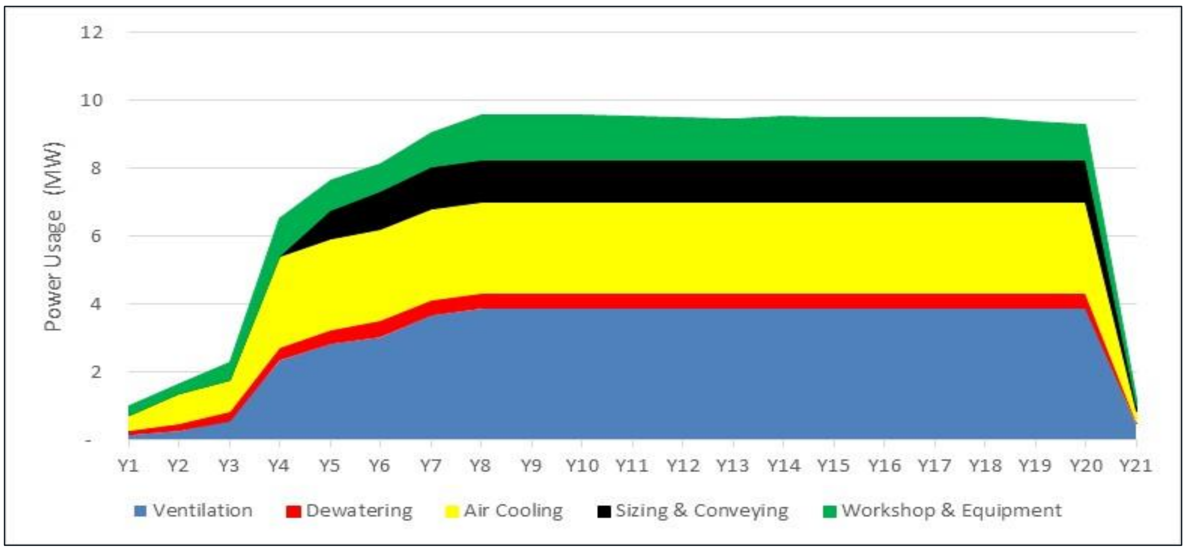

Source: AGP 2023.

Similar data for Parks/Salyer is shown in Table 16-42 and Figure 16-69.

**Table 16-42: Parks/Salyer Total Installed Power**

| Area | Installed Power (kW) |
|---|---|
| Ventilation | 4,809 |
| Dewatering | 1,635 |
| Air Cooling | 4,500 |
| Sizing & Conveying | 2,730 |
| Workshop & Equipment | 4,416 |
| **Total Installed Power** | **18,091** |

**Figure 16-69: Parks/Salyer Power Utilized**




Source: AGP 2023.

## 16.6 Combined Production Schedule

The Cactus Mine combined schedule includes production from four separate mining areas: Cactus West Open Pits, Historical Stockpiles, Cactus East Underground, and Parks/Salyer Underground. The mine production schedule is initially focused on surface sources of ore to enable ramp up of the processing facilities, with underground ore sources being developed sequentially later in the mine life to partially replace the open pit ore supply.

Total surface material movement from Year 1 to Year 4 represents the period of peak activity and averages approximately 55 Mt/y Approximately 24 Mton of pre-stripping is planned for Cactus West in Year -1 to source construction materials and to enable full ore processing Year 1. From Year 1 to Year 6 both the crushing circuits at Cactus will be operating at their full capacity of 12 Mt/y each before ramping down with Cactus West and Stockpile ore sources being depleted and the ore stream becoming exclusively underground.

The schedule details are shown in Table 16-43 to Table 16-45.

**Table 16-43: Total Tons Mined by Area (ore and waste)**

| Phase | Total Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 |
| CW Ph-1 | 96.0 | | 24.1 | 37.2 | 22.7 | 11.5 | 0.4 | | | | | | | |
| CW Ph-2 | 121.9 | | | 2.0 | 24.6 | 33.0 | 34.1 | 21.3 | 6.9 | | | | | |
| Stockpile | 82.2 | | 3.2 | 13.0 | 12.6 | 12.3 | 12.6 | 13.1 | 13.4 | 2.0 | | | | |
| CE UG | 28.4 | | | | | | | | | | | 0.1 | 0.5 | 2.2 |
| PS UG | 98.5 | | 0.1 | 0.3 | 0.6 | 0.8 | 2.9 | 2.6 | 4.3 | 6.7 | 6.7 | 6.7 | 6.5 | 6.4 |
| **Total** | **427.1** | | **27.4** | **52.6** | **60.6** | **57.7** | **50.0** | **37.0** | **24.6** | **8.7** | **6.7** | **6.8** | **6.9** | **8.6** |
| **Phase** | **Total Tonnage (Mton)** | | | | | | | | | | | | | |
| | **12** | **13** | **14** | **15** | **16** | **17** | **18** | **19** | **20** | **21** | **22** | **23** | **24** | **25** |
| CW Ph-1 | | | | | | | | | | | | | | |
| CW Ph-2 | | | | | | | | | | | | | | |
| Stockpile | | | | | | | | | | | | | | |
| CE UG | 3.4 | 3.2 | 3.1 | 3.9 | 3.7 | 3.7 | 3.7 | 1.0 | | | | | | |
| PS UG | 6.3 | 7.0 | 7.0 | 6.9 | 6.9 | 6.6 | 6.6 | 6.6 | 0.1 | | | | | |
| **Total** | **9.7** | **10.2** | **10.2** | **10.8** | **10.6** | **10.3** | **10.2** | **7.5** | **0.1** | | | | | |

**Table 16-44: Ore Tons Mined by Area**

| Phase | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 |
| CW Ph-1 | 33.0 | | 0.5 | 12.7 | 12.8 | 6.7 | 0.3 | | | | | | | |
| CW Ph-2 | 42.5 | | | | 1.1 | 2.7 | 15.6 | 18.1 | 4.1 | | | | | |
| Stockpile | 76.8 | | 3.0 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 1.8 | | | | |
| CE UG | 27.7 | | | | | | | | | | | 0.4 | 3.7 | 2.0 |
| PS UG | 96.2 | | 0.0 | 0.1 | 0.3 | 0.6 | 2.8 | 2.5 | 4.2 | 6.6 | 6.5 | 6.6 | 6.3 | 6.4 |
| **Total** | **267.3** | | **3.5** | **24.8** | **26.3** | **22.0** | **30.6** | **32.6** | **21.1** | **8.4** | **6.5** | **7.0** | **10.1** | **8.4** |
| **Phase** | **Ore Tonnage (Mton)** | | | | | | | | | | | | | |
| | **12** | **13** | **14** | **15** | **16** | **17** | **18** | **19** | **20** | **21** | **22** | **23** | **24** | **25** |
| CW Ph-1 | | | | | | | | | | | | | | |
| CW Ph-2 | | | | | | | | | | | | | | |
| Stockpile | | | | | | | | | | | | | | |
| CE UG | 3.3 | 3.0 | 3.0 | 3.9 | 3.7 | 3.7 | 3.7 | 1.0 | | | | | | |
| PS UG | 6.2 | 6.9 | 6.9 | 6.8 | 6.8 | 6.6 | 6.5 | 6.5 | 0.1 | | | | | |
| **Total** | **9.5** | **10.0** | **10.0** | **10.7** | **10.6** | **10.3** | **10.2** | **7.4** | **0.1** | | | | | |

**Table 16-45: Ore Processed by Mining Area**

| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **CW OP** | **Total** | **Y-2** | **Y-1** | **Y1** | **Y2** | **Y3** | **Y4** | **Y5** | **Y6** | **Y7** | **Y8** | **Y9** | **Y10** | **Y11** |
| Tons (Mt) | 75.5 | | 0.5 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 3.0 | | | | |
| TCU (%) | 0.31 | | 0.36 | 0.26 | 0.30 | 0.39 | 0.31 | 0.33 | 0.28 | 0.18 | | | | |
| CU-AS (%) | 0.13 | | 0.08 | 0.14 | 0.14 | 0.12 | 0.15 | 0.14 | 0.11 | 0.12 | | | | |
| CU-CN (%) | 0.12 | | 0.26 | 0.07 | 0.10 | 0.22 | 0.11 | 0.13 | 0.12 | 0.03 | | | | |

| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Stockpile** | **Total** | **Y-2** | **Y-1** | **Y1** | **Y2** | **Y3** | **Y4** | **Y5** | **Y6** | **Y7** | **Y8** | **Y9** | **Y10** | **Y11** |
| Tons (Mt) | 76.8 | | 3.0 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 12.0 | 1.8 | | | | |
| TCU (%) | 0.16 | | 0.23 | 0.20 | 0.16 | 0.17 | 0.14 | 0.15 | 0.14 | 0.14 | | | | |
| CU-AS (%) | 0.11 | | 0.17 | 0.14 | 0.11 | 0.12 | 0.10 | 0.10 | 0.10 | 0.09 | | | | |
| CU-CN (%) | 0.02 | | 0.03 | 0.03 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | | | | |

| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **CE UG** | **Total** | **Y-2** | **Y-1** | **Y1** | **Y2** | **Y3** | **Y4** | **Y5** | **Y6** | **Y7** | **Y8** | **Y9** | **Y10** | **Y11** |
| Tons (Mt) | 27.7 | | | | | | | | | | | 0.04 | 0.4 | 2.0 |
| TCU (%) | 0.95 | | | | | | | | | | | 0.42 | 0.71 | 0.87 |
| CU-AS (%) | 0.33 | | | | | | | | | | | 0.07 | 0.18 | 0.47 |
| CU-CN (%) | 0.55 | | | | | | | | | | | 0.28 | 0.47 | 0.35 |

| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **CE UG** | **12** | **13** | **14** | **15** | **16** | **17** | **18** | **19** | **20** | **21** | **22** | **23** | **24** | **25** |
| Tons (Mt) | 3.3 | 3.0 | 3.0 | 3.9 | 3.7 | 3.7 | 3.7 | 1.0 | | | | | | |
| TCU (%) | 0.94 | 0.93 | 0.98 | 0.91 | 1.03 | 1.00 | 0.94 | 0.89 | | | | | | |
| CU-AS (%) | 0.41 | 0.38 | 0.37 | 0.34 | 0.34 | 0.28 | 0.22 | 0.19 | | | | | | |
| CU-CN (%) | 0.48 | 0.49 | 0.55 | 0.51 | 0.65 | 0.66 | 0.93 | 0.58 | | | | | | |

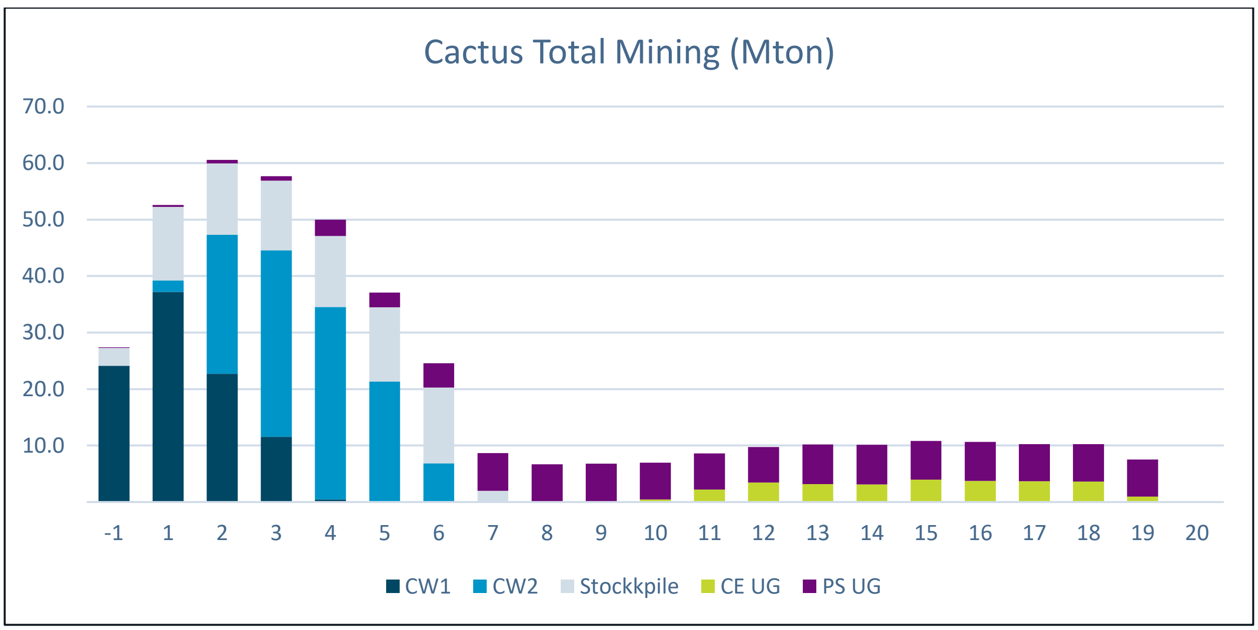
| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **PS UG** | **Total** | **Y-2** | **Y-1** | **Y1** | **Y2** | **Y3** | **Y4** | **Y5** | **Y6** | **Y7** | **Y8** | **Y9** | **Y10** | **Y11** |
| Tons (Mt) | 96.2 | | | 0.1 | 0.6 | 0.6 | 2.8 | 2.5 | 4.2 | 6.6 | 6.5 | 6.6 | 6.3 | 6.4 |
| TCU (%) | 0.93 | | | 0.07 | 0.89 | 1.02 | 1.04 | 0.93 | 0.92 | 0.95 | 0.95 | 0.94 | 0.89 | 1.05 |
| CU-AS (%) | 0.11 | | | 0.01 | 0.06 | 0.10 | 0.11 | 0.10 | 0.09 | 0.10 | 0.13 | 0.14 | 0.15 | 0.11 |
| CU-CN (%) | 0.71 | | | 0.06 | 0.75 | 0.82 | 0.86 | 0.76 | 0.73 | 0.77 | 0.74 | 0.70 | 0.65 | 0.82 |

| | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **PS UG** | **12** | **13** | **14** | **15** | **16** | **17** | **18** | **19** | **20** | **21** | **22** | **23** | **24** | **25** |
| Tons (Mt) | 6.2 | 6.9 | 6.9 | 6.8 | 6.8 | 6.6 | 6.5 | 6.5 | 0.1 | | | | | |
| TCU (%) | 0.95 | 0.87 | 0.92 | 0.83 | 0.83 | 0.91 | 0.90 | 1.08 | 1.03 | | | | | |
| CU-AS (%) | 0.11 | 0.10 | 0.14 | 0.12 | 0.10 | 0.10 | 0.08 | 0.06 | 0.04 | | | | | |
| CU-CN (%) | 0.75 | 0.64 | 0.69 | 0.61 | 0.93 | 0.70 | 0.66 | 0.79 | 0.64 | | | | | |

| All Ore | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 |
| Tons (Mt) | 276 | | 3.5 | 24.1 | 24.3 | 24.6 | 26.8 | 26.5 | 28.2 | 11.4 | 6.5 | 6.6 | 6.7 | 8.4 |
| TCU (%) | 0.55 | | 0.25 | 0.23 | 0.24 | 0.30 | 0.30 | 0.31 | 0.32 | 0.62 | 0.62 | 0.93 | 0.88 | 1.01 |
| CU-AS (%) | 0.14 | | 0.16 | 0.14 | 0.13 | 0.12 | 0.12 | 0.12 | 0.10 | 0.10 | 0.13 | 0.13 | 0.14 | 0.23 |
| CU-CN (%) | 0.34 | | 0.07 | 0.05 | 0.08 | 0.15 | 0.14 | 0.14 | 0.17 | 0.46 | 0.70 | 0.70 | 0.64 | 0.71 |

| All Ore | Ore Tonnage (Mton) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 |
| Tons (Mt) | 0.95 | 10.0 | 10.0 | 10.7 | 10.6 | 10.3 | 10.2 | 7.4 | 0.14 | | | | | |
| TCU (%) | 0.94 | 0.89 | 0.94 | 0.86 | 0.90 | 0.94 | 0.91 | 1.05 | 1.03 | | | | | |
| CU-AS (%) | 0.22 | 0.19 | 0.21 | 0.20 | 0.19 | 0.16 | 0.13 | 0.07 | 0.04 | | | | | |
| CU-CN (%) | 0.66 | 0.59 | 0.65 | 0.58 | 0.63 | 0.68 | 0.65 | 0.76 | 0.64 | | | | | |

**Figure 16-70: Tons Mined by Area**

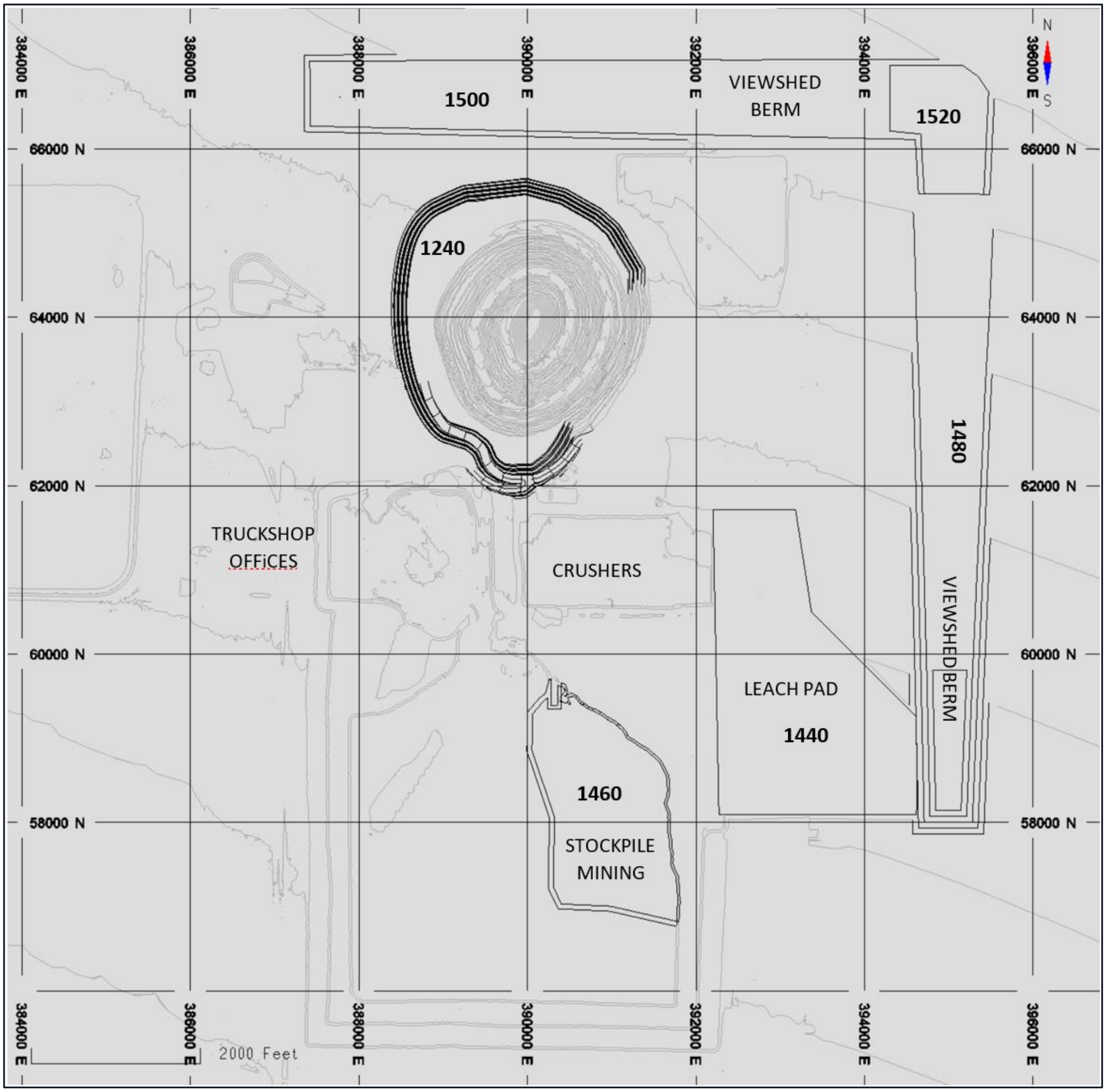



Source: AGP, 2023.

## 16.7 End of Period Plans – Open Pit

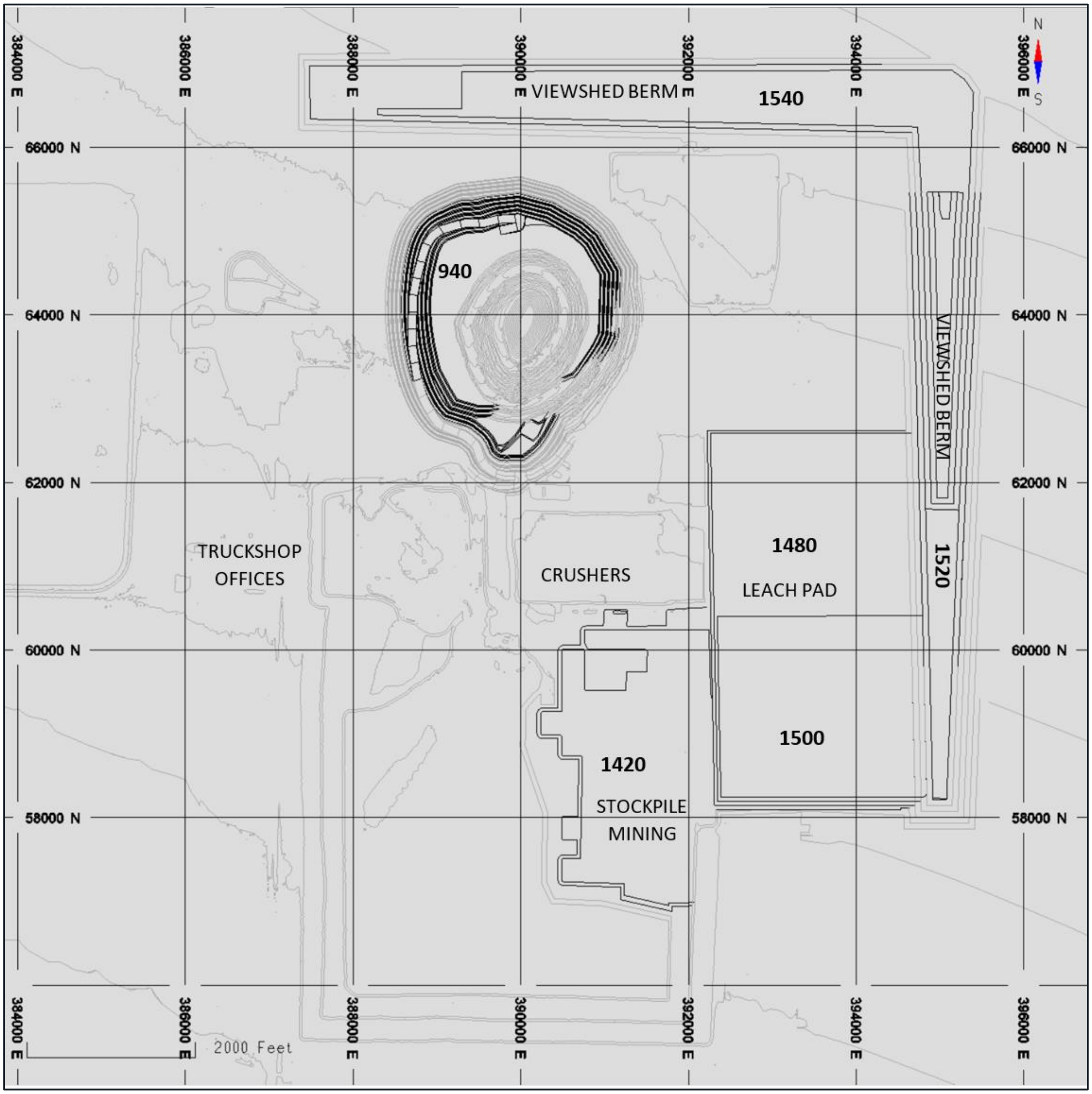
Figure 16-71 through Figure 16-78.

**Figure 16-71: End of Preproduction Period- Year-1**



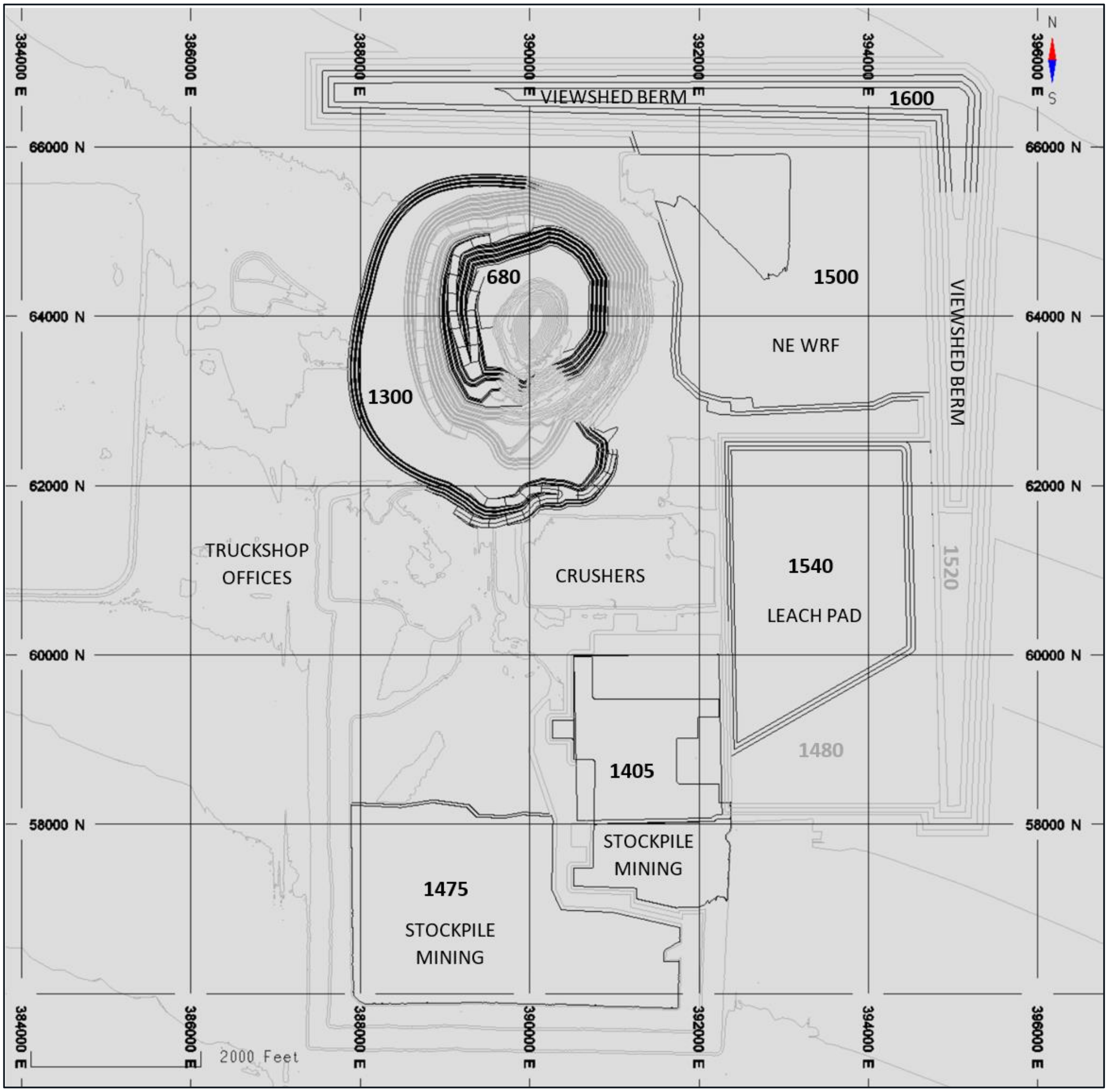

Source: AGP 2023.

**Figure 16-72: End of Year 1**



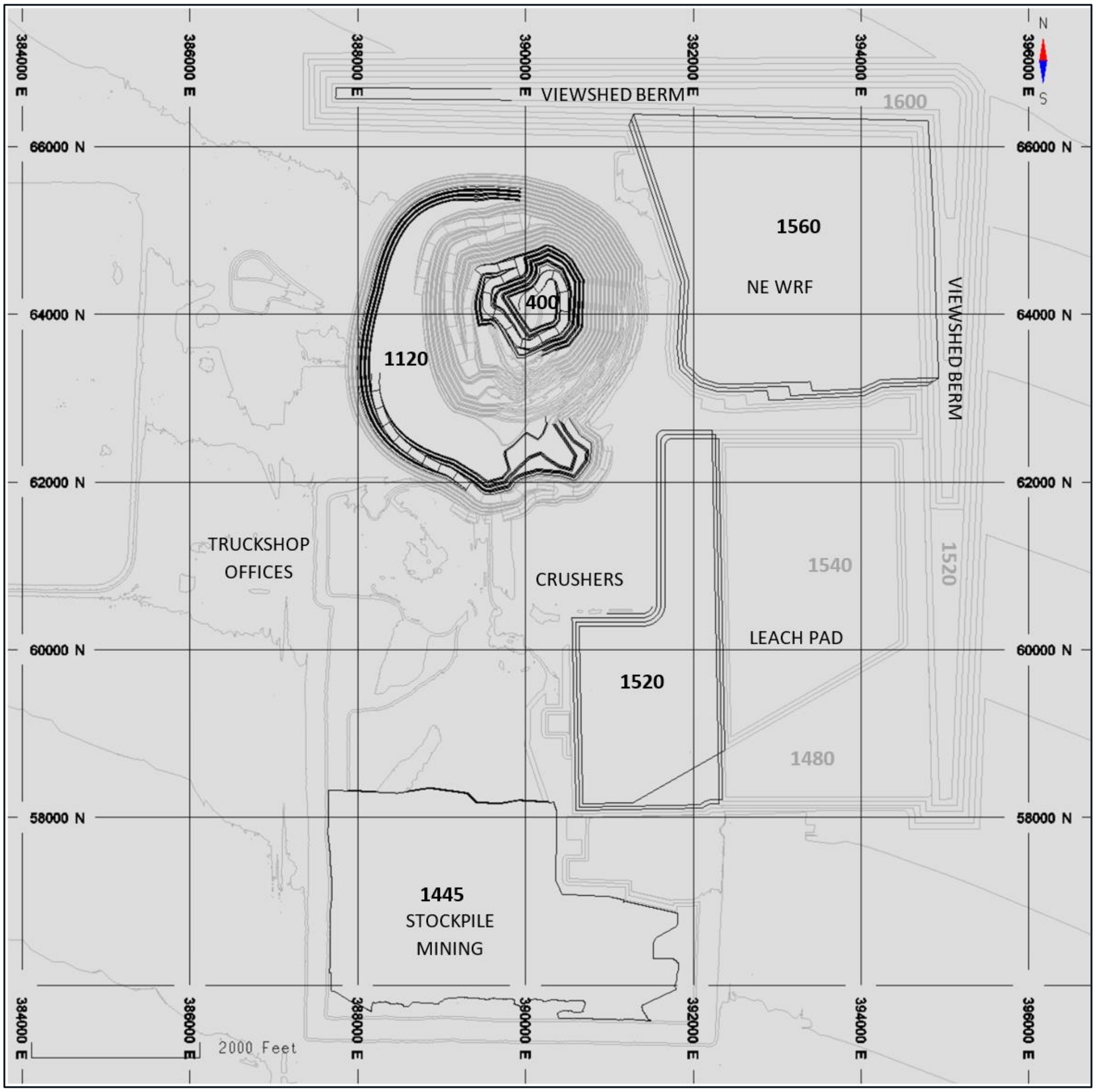

Source: AGP 2023.

**Figure 16-73: End of Year 2**



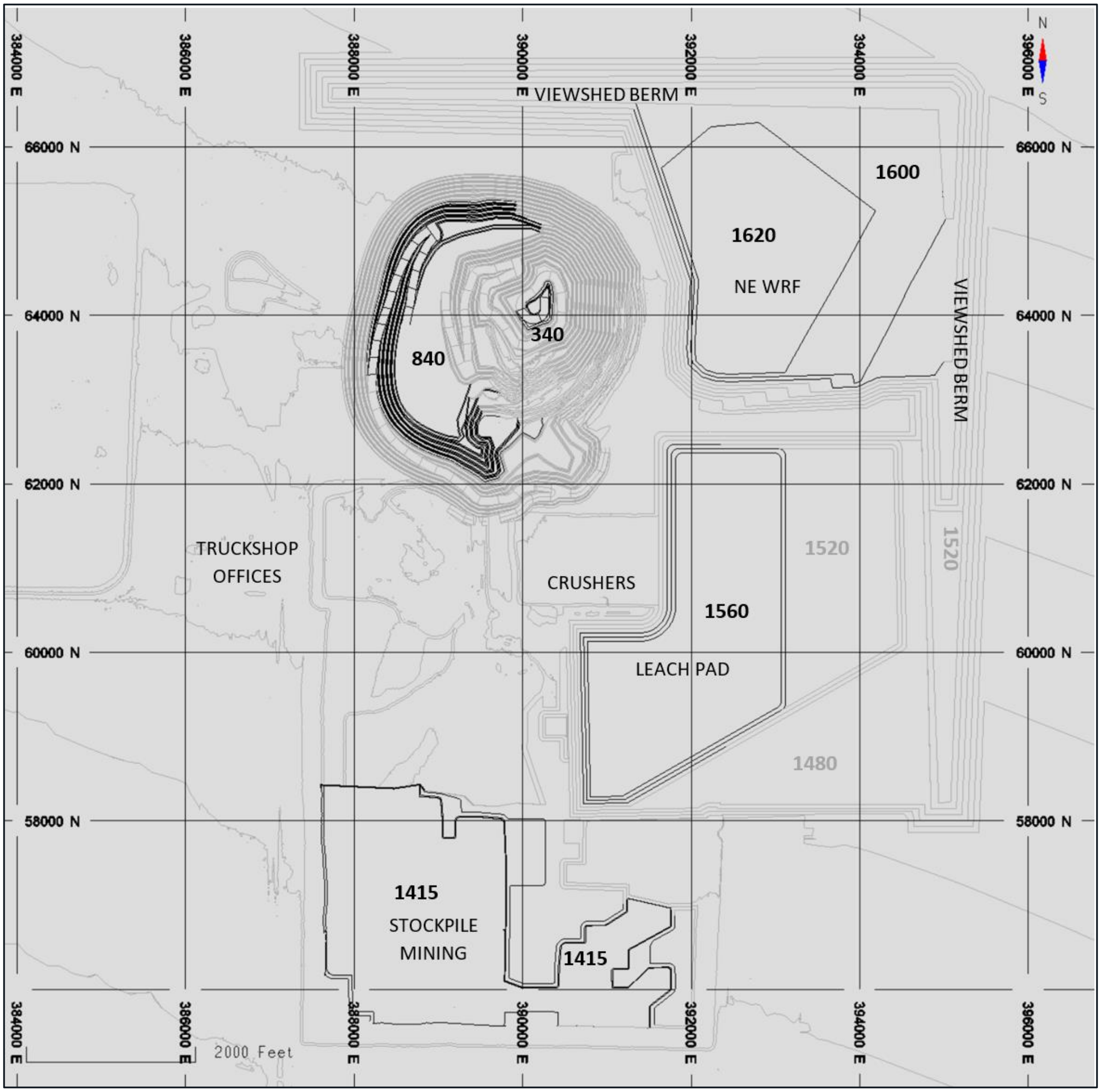

Source: AGP 2023.

**Figure 16-74: End of Year 3**



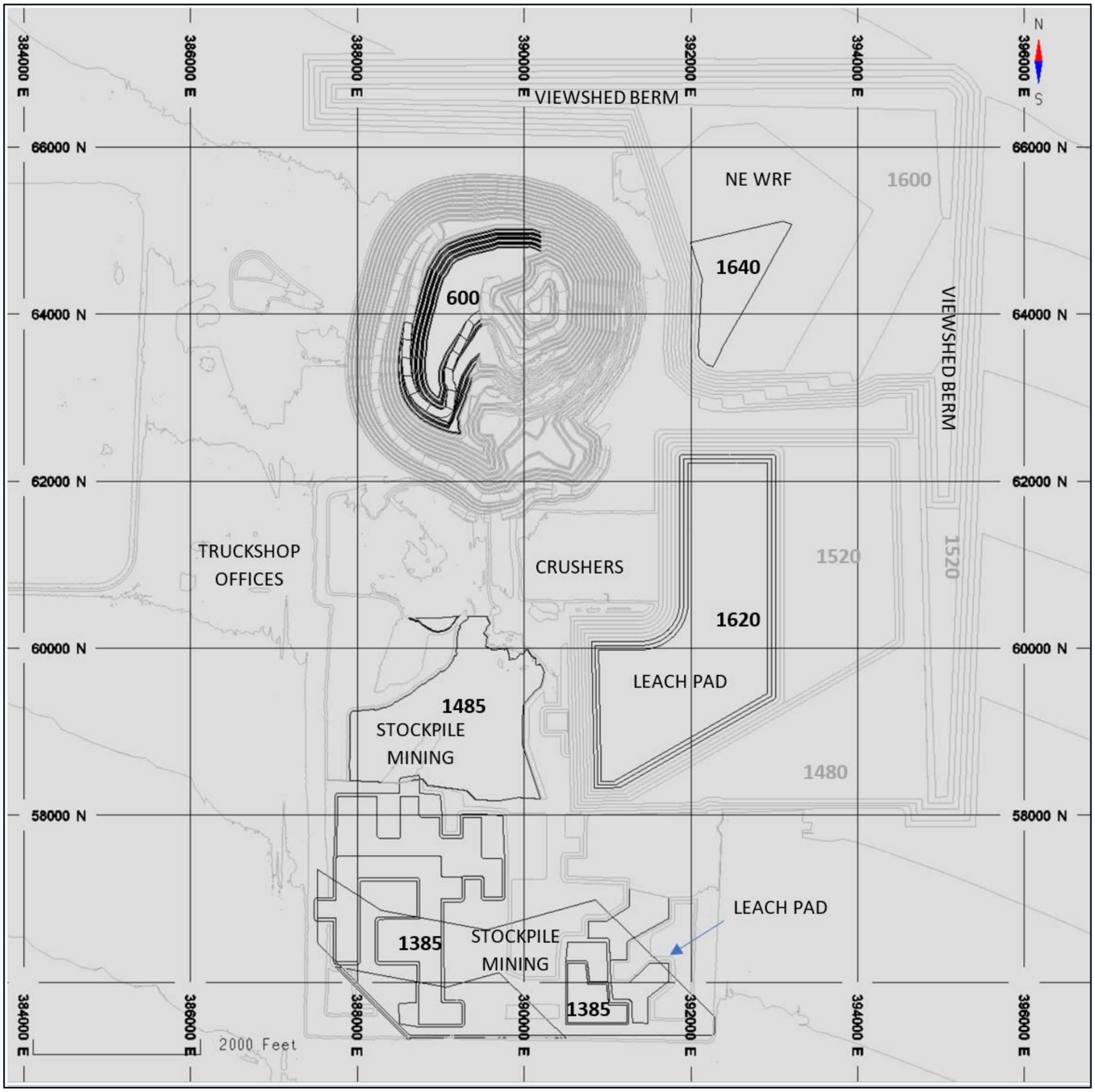

Source: AGP 2023.

**Figure 16-75: End of Year 4**



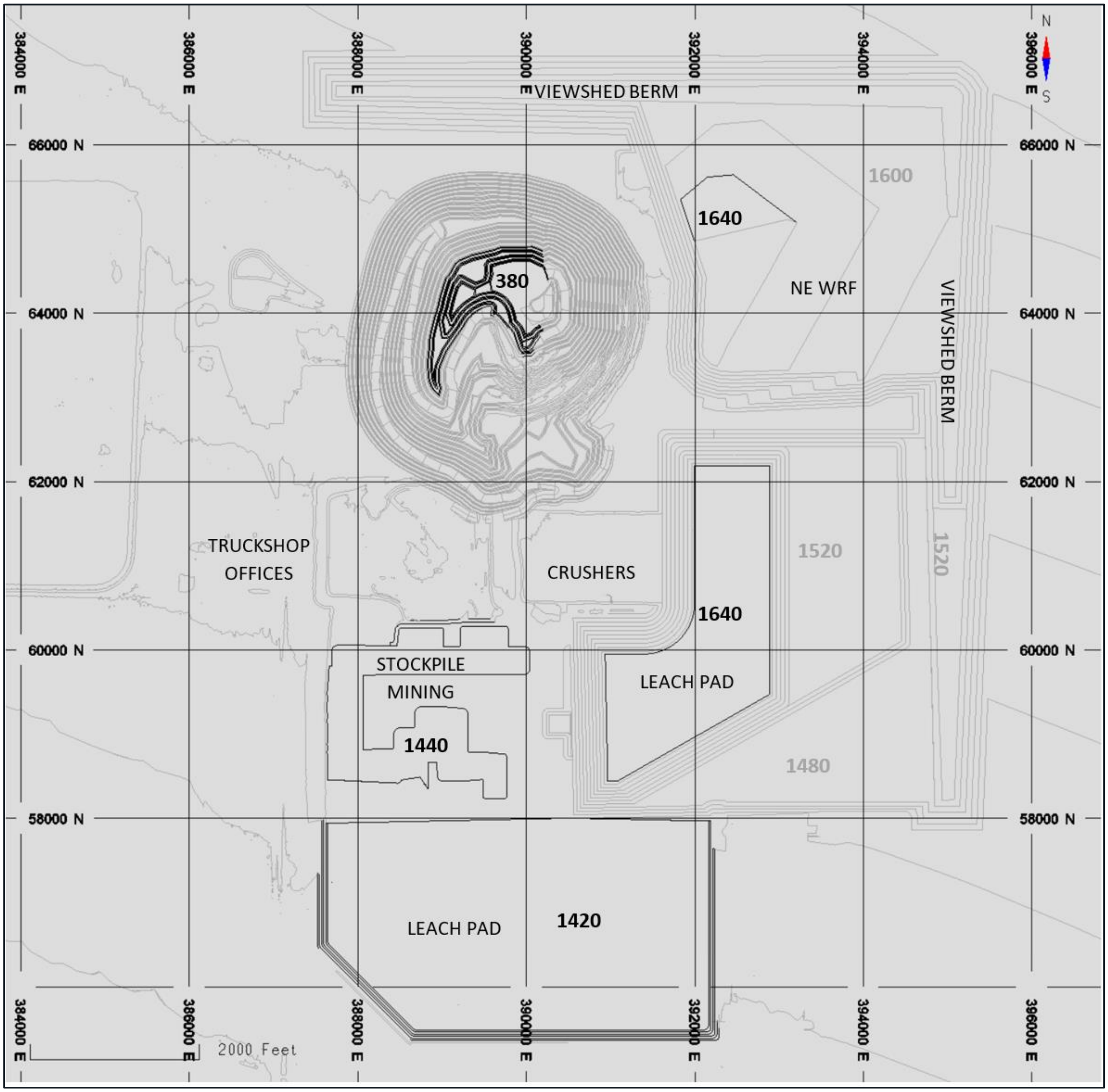

Source: AGP 2023.

**Figure 16-76: End of Year 5**



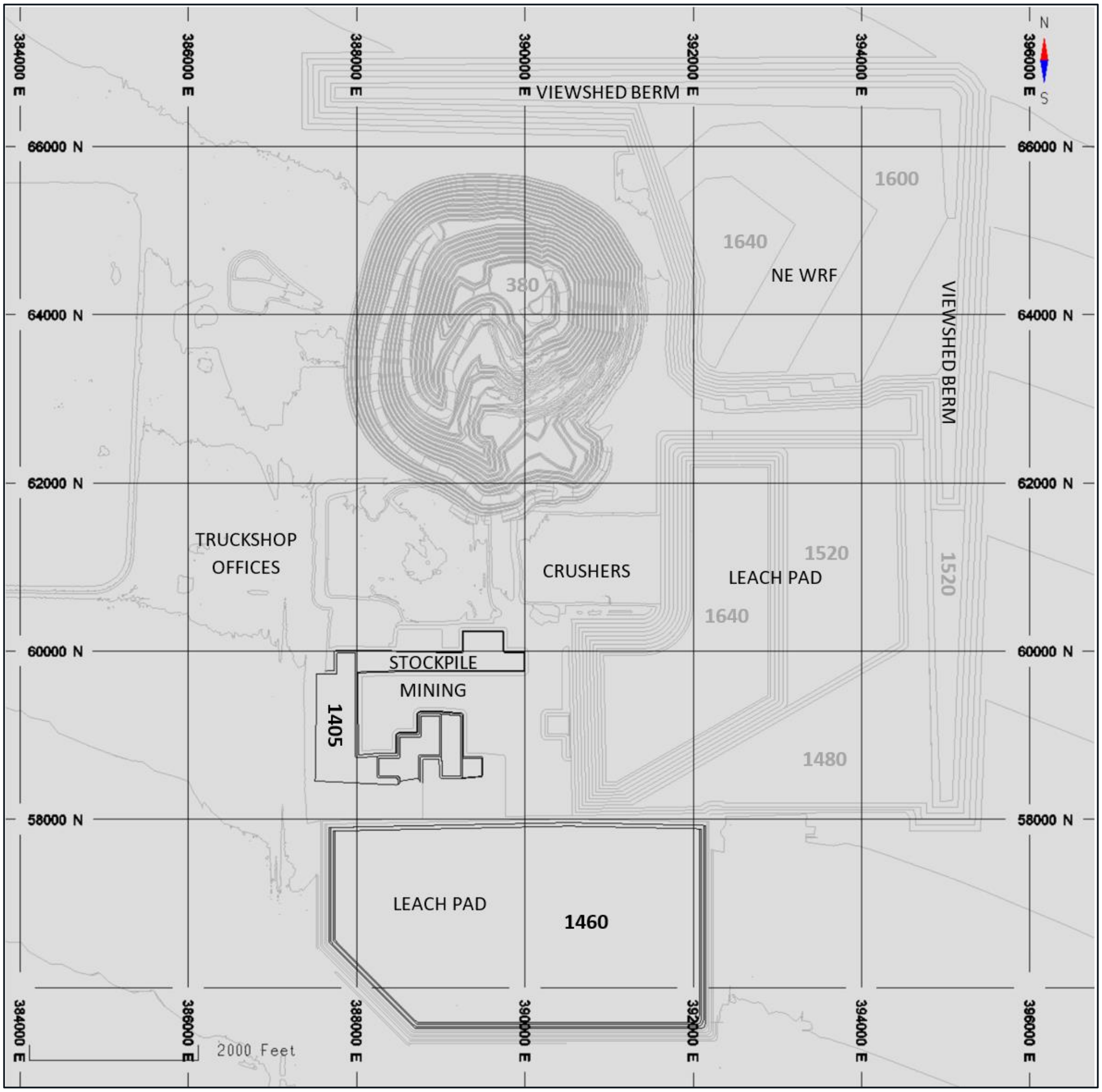

Source: AGP 2023.

**Figure 16-77: End of Year 6**




Source: AGP 2023.

**Figure 16-78: End of Year 7**

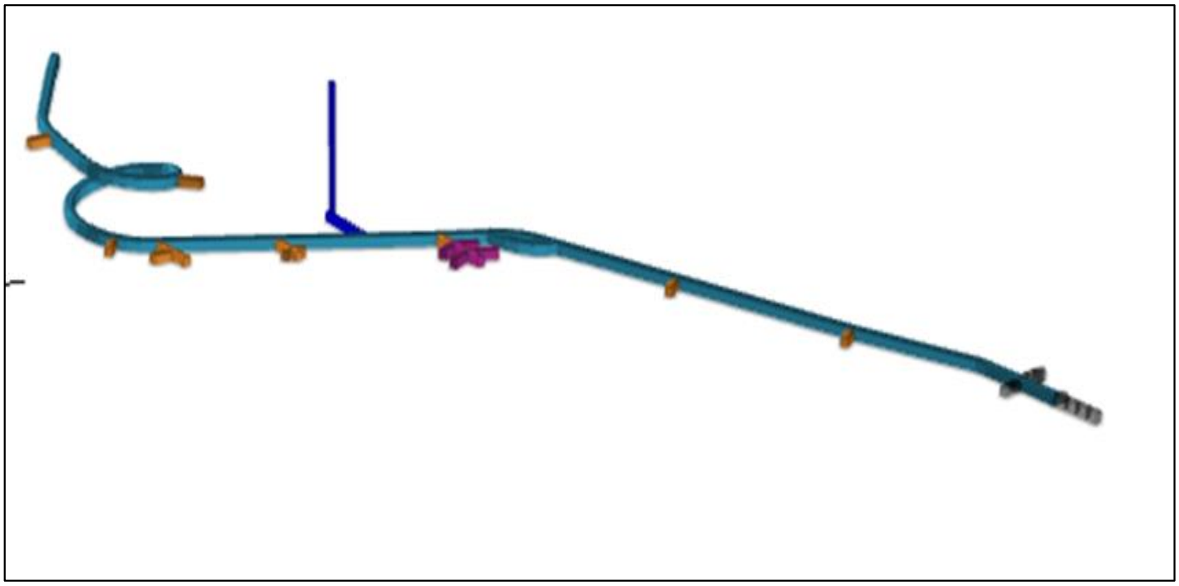



Source: AGP 2023.

## 16.8 End of Period Plans – Underground

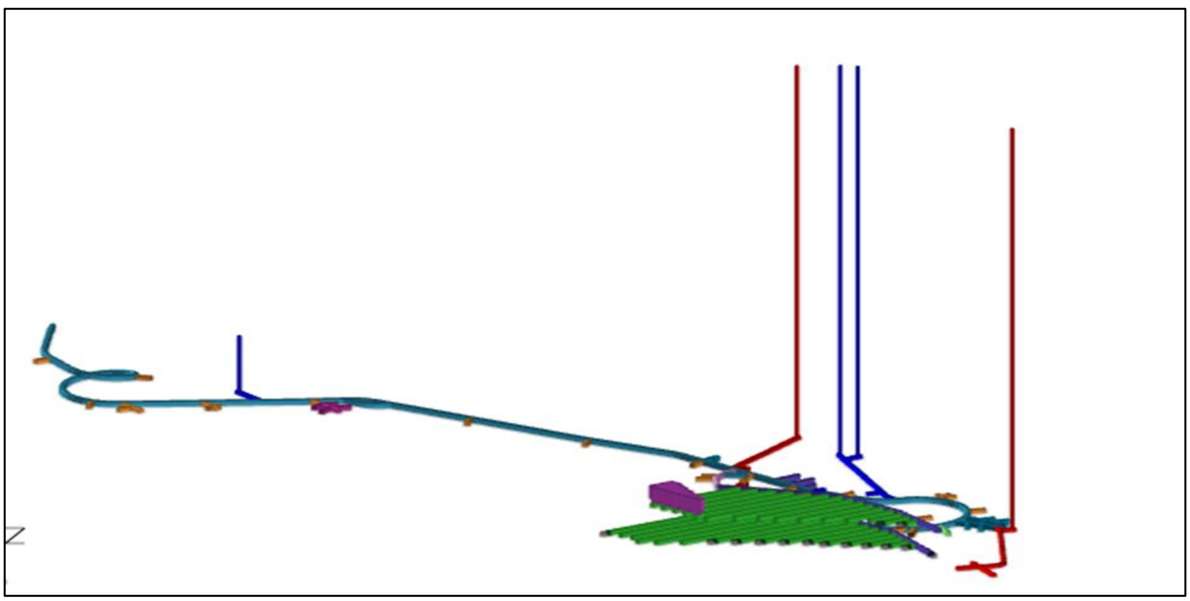
Figure 16-79 through Figure 16-92.

**Figure 16-79: Cactus East – Year 9 (looking Northwest)**



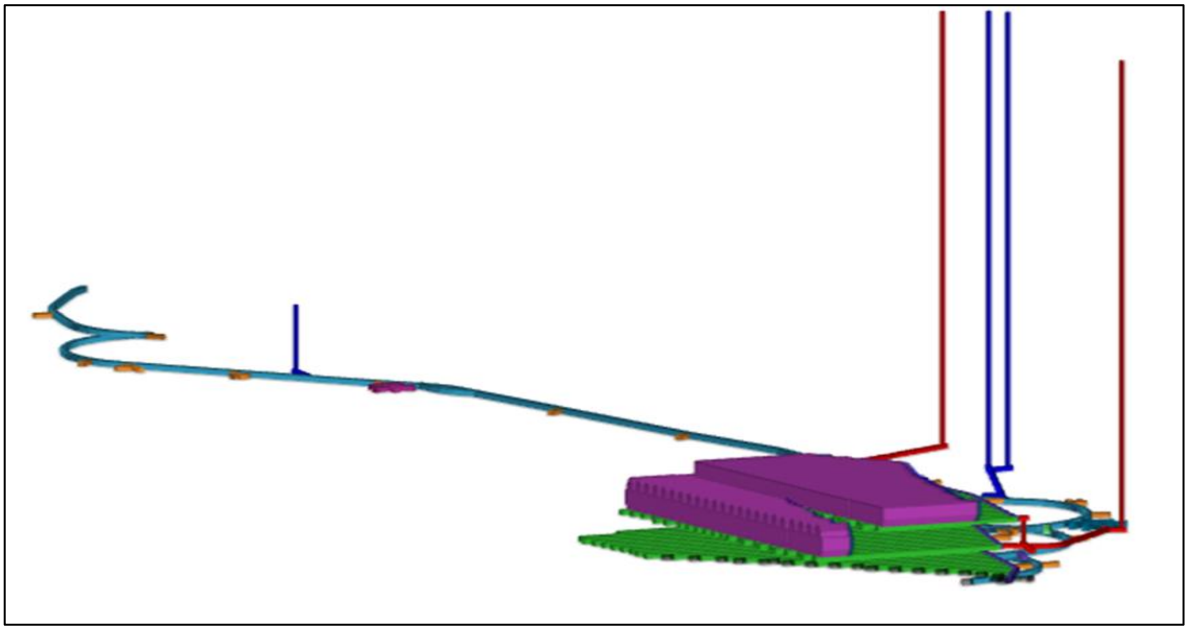
Source: AGP 2023.

**Figure 16-80: Cactus East –Year 10 (looking Northwest)**



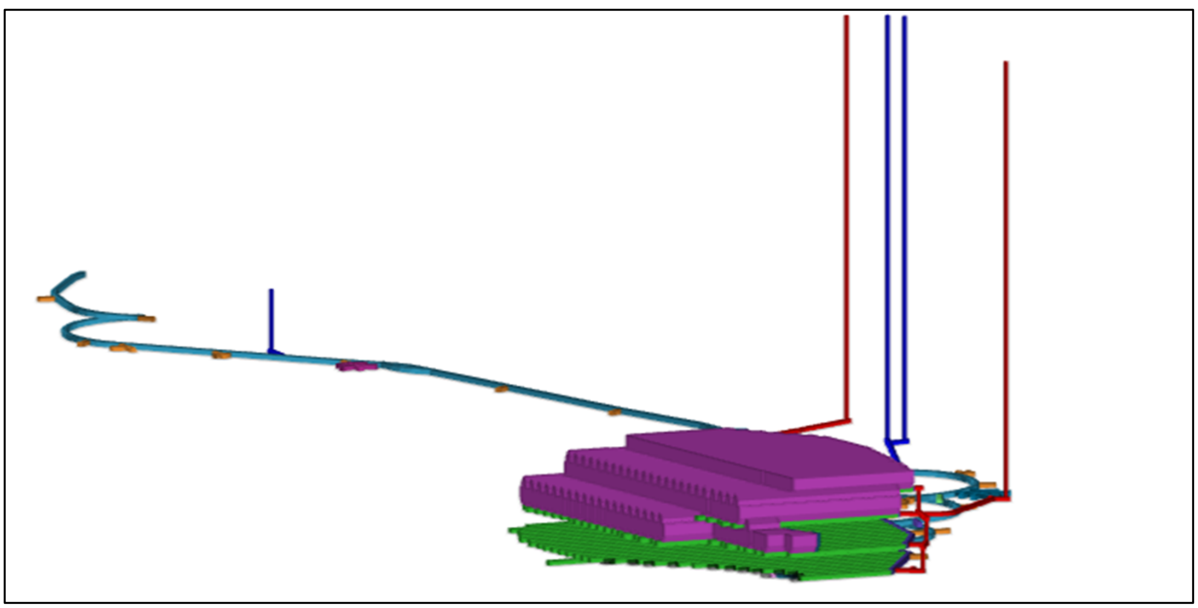
Source: AGP 2023.

**Figure 16-81: Cactus East –Year 11 (looking Northwest)**



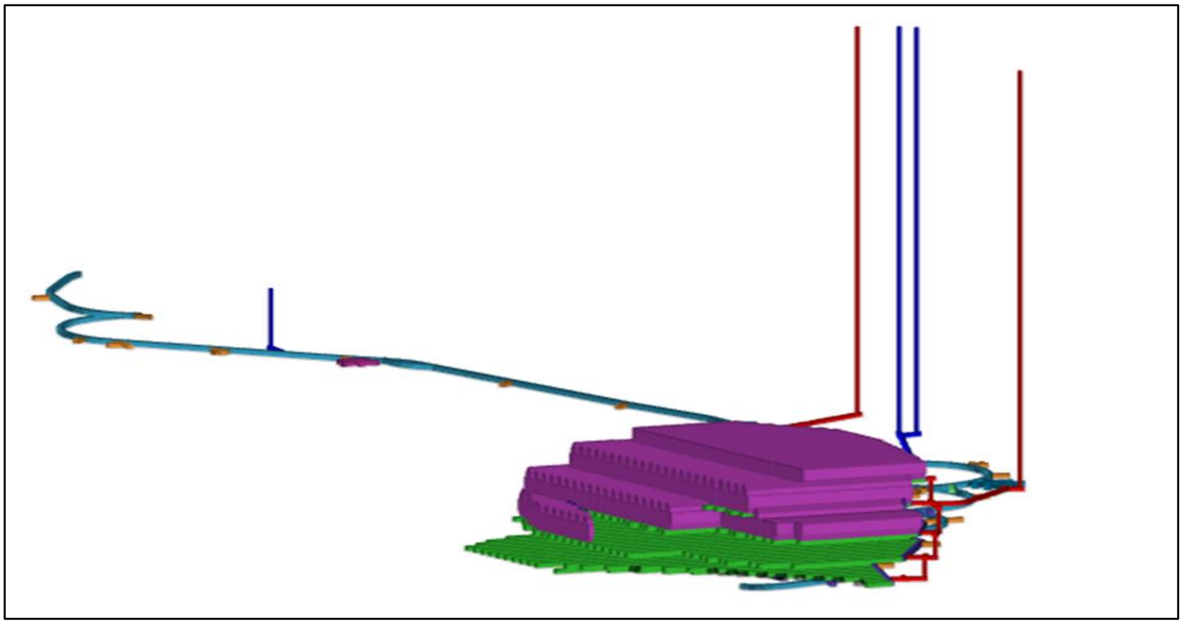
Source: AGP 2023.

**Figure 16-82: Cactus East –Year 12 (looking Northwest)**



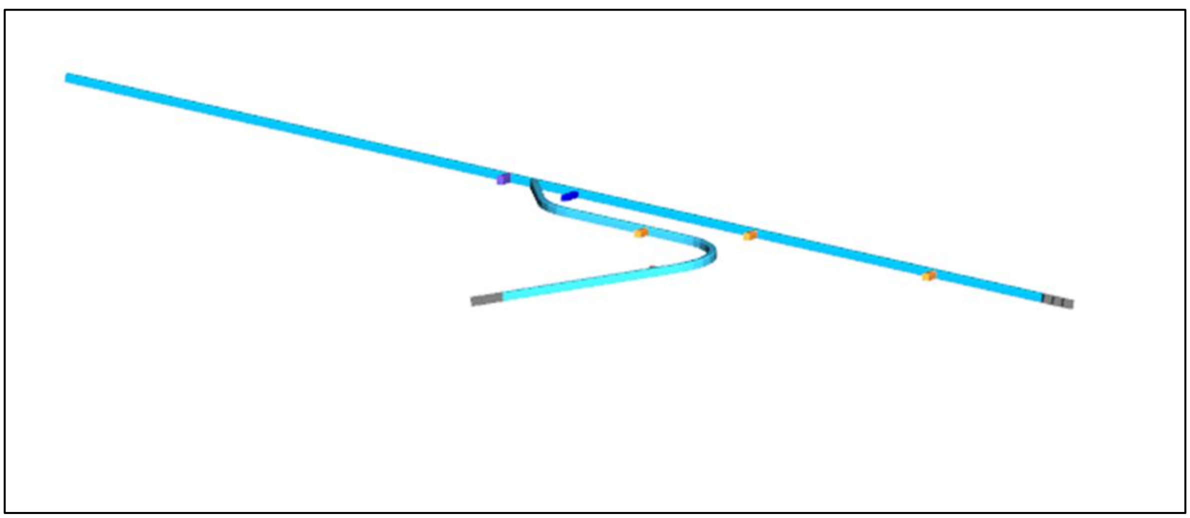
Source: AGP 2023.

**Figure 16-83: Cactus East –Year 13 (looking Northwest)**



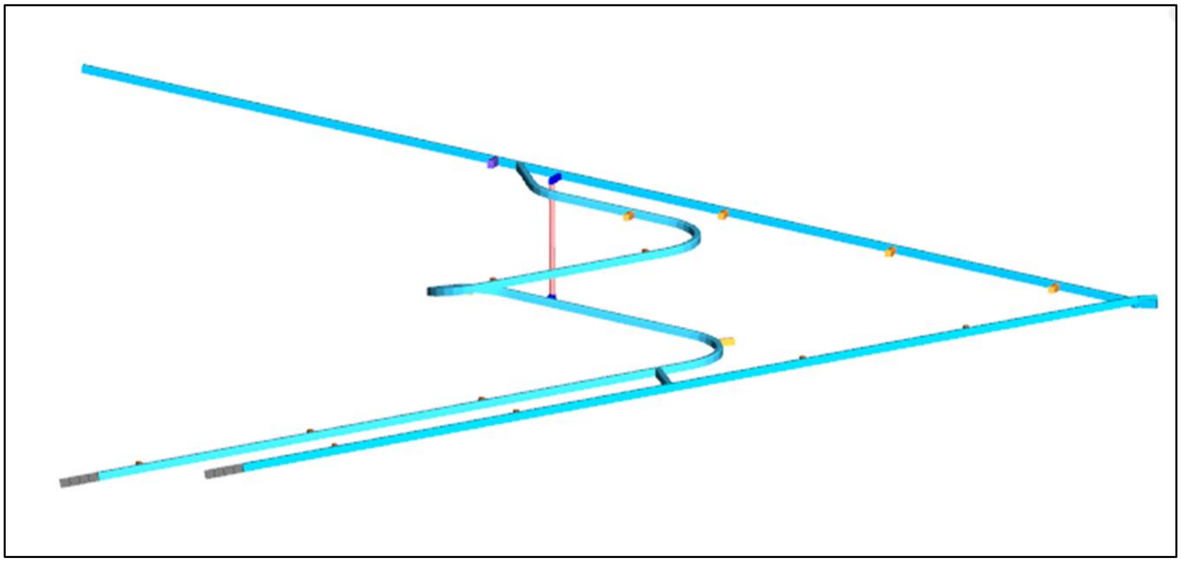
Source: AGP 2023.

**Figure 16-84: Parks/Salyer –Year -1 (looking Southeast)**



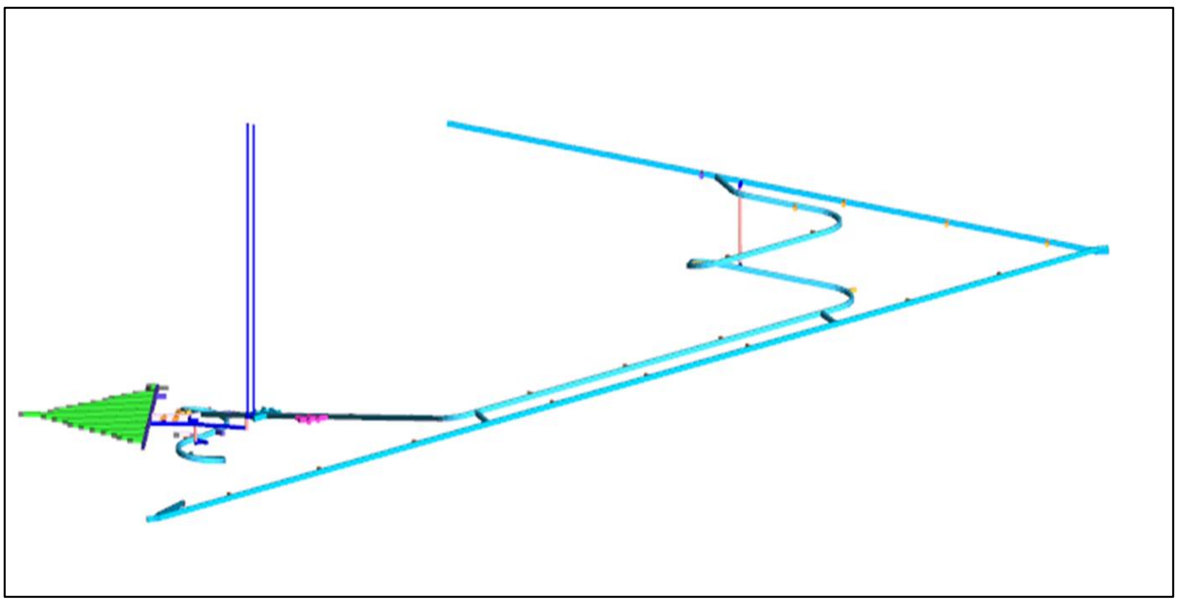
Source: AGP 2023.

**Figure 16-85: Parks/Salyer –Year 1 (looking Southeast)**



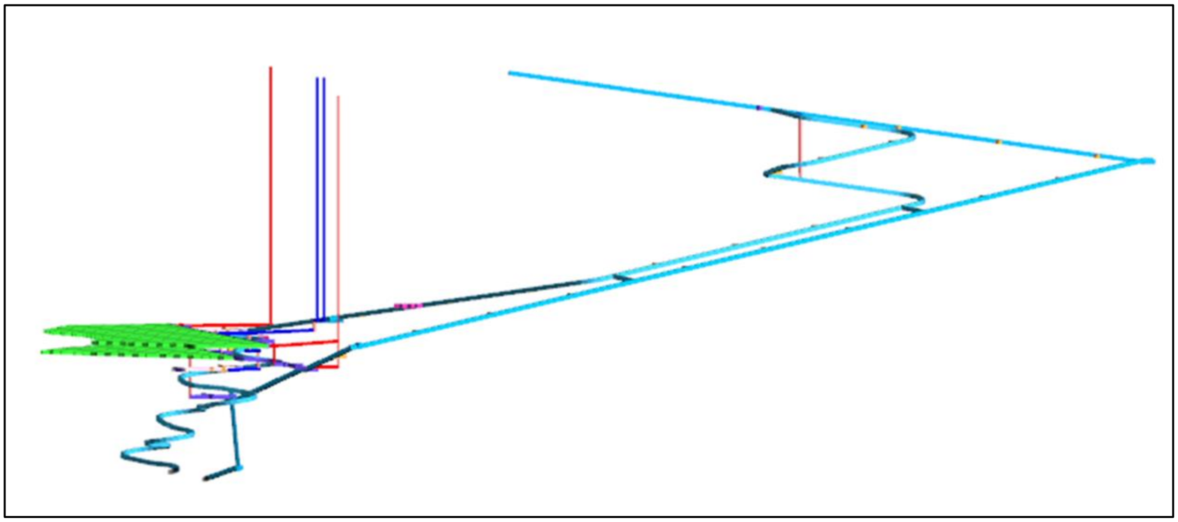
Source: AGP 2023.

**Figure 16-86: Parks/Salyer –Year 2 (looking Southeast)**



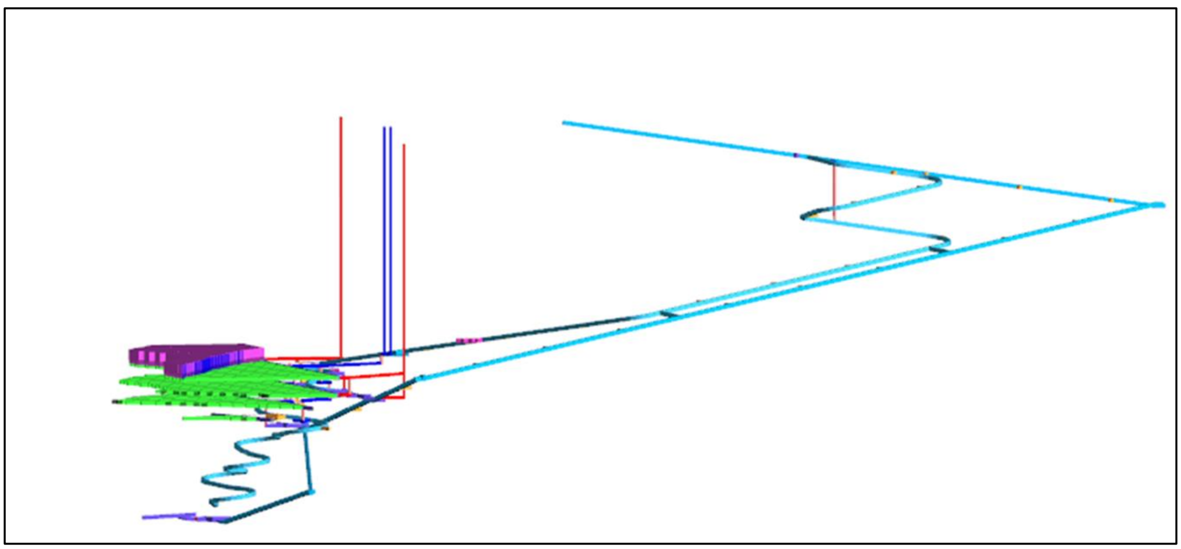
Source: AGP 2023.

**Figure 16-87: Parks/Salyer –Year 3 (looking Southeast)**



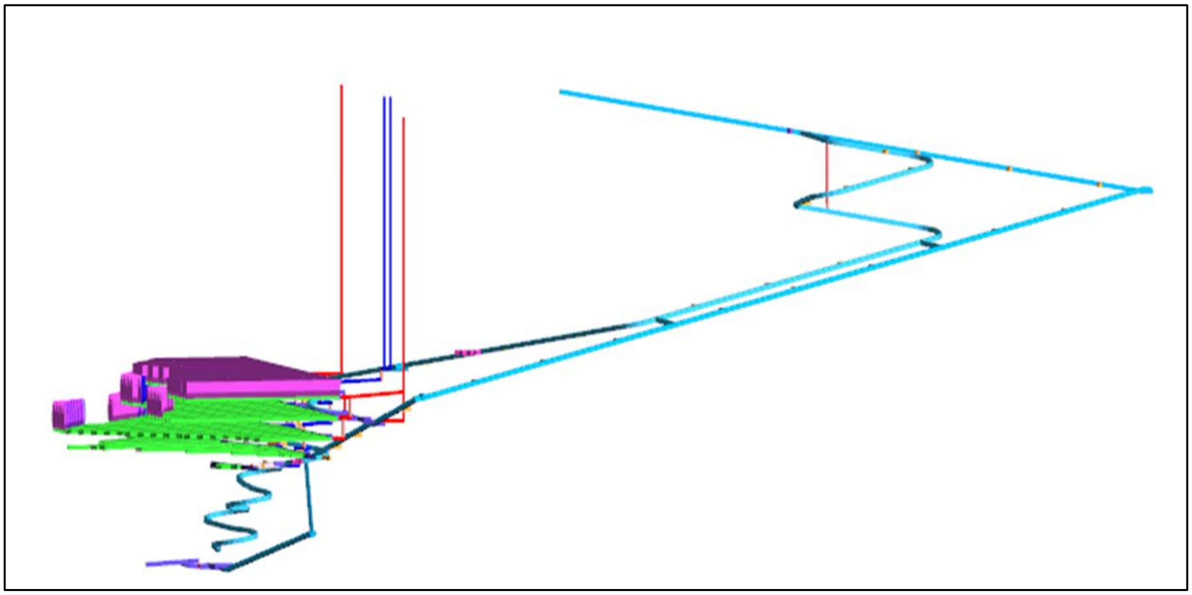
Source: AGP 2023.

**Figure 16-88: Parks/Salyer –Year 4 (looking Southeast)**



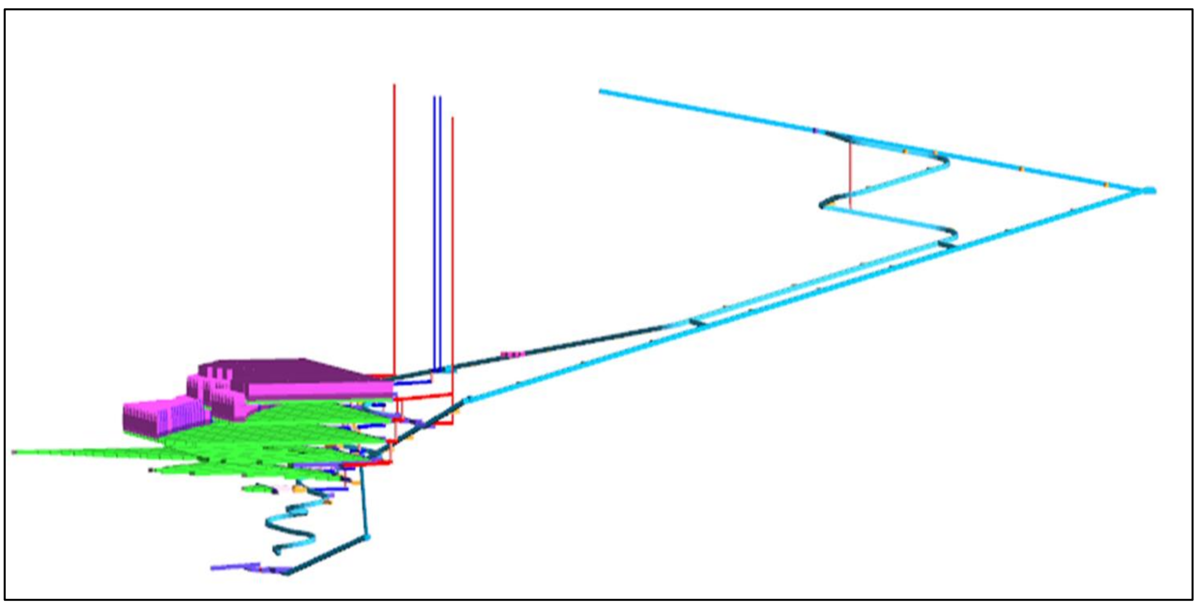
Source: AGP 2023.

**Figure 16-89: Parks/Salyer –Year 5 (looking Southeast)**



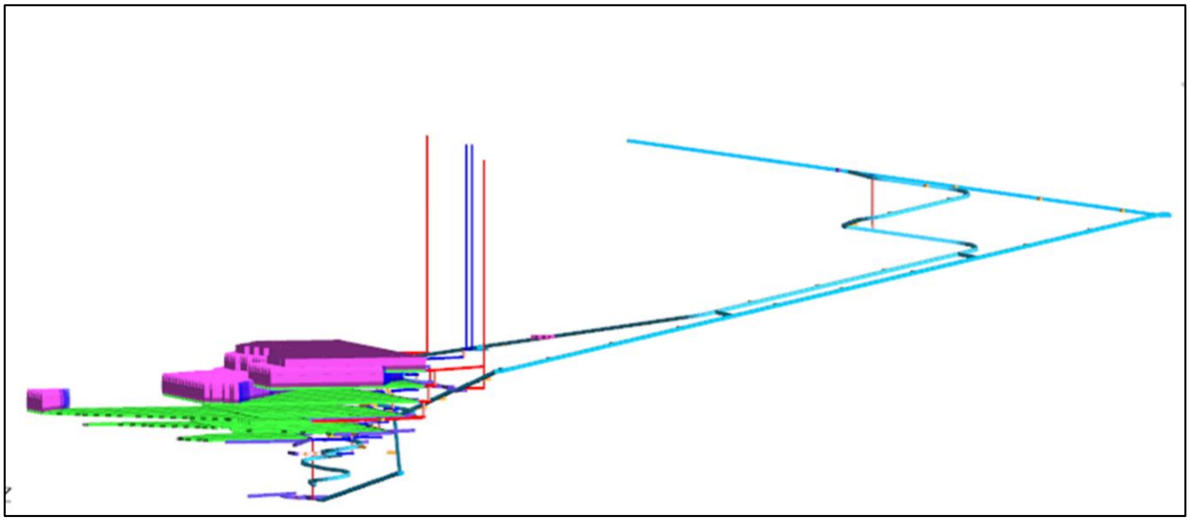
Source: AGP 2023.

**Figure 16-90: Parks/Salyer –Year 6 (looking Southeast)**



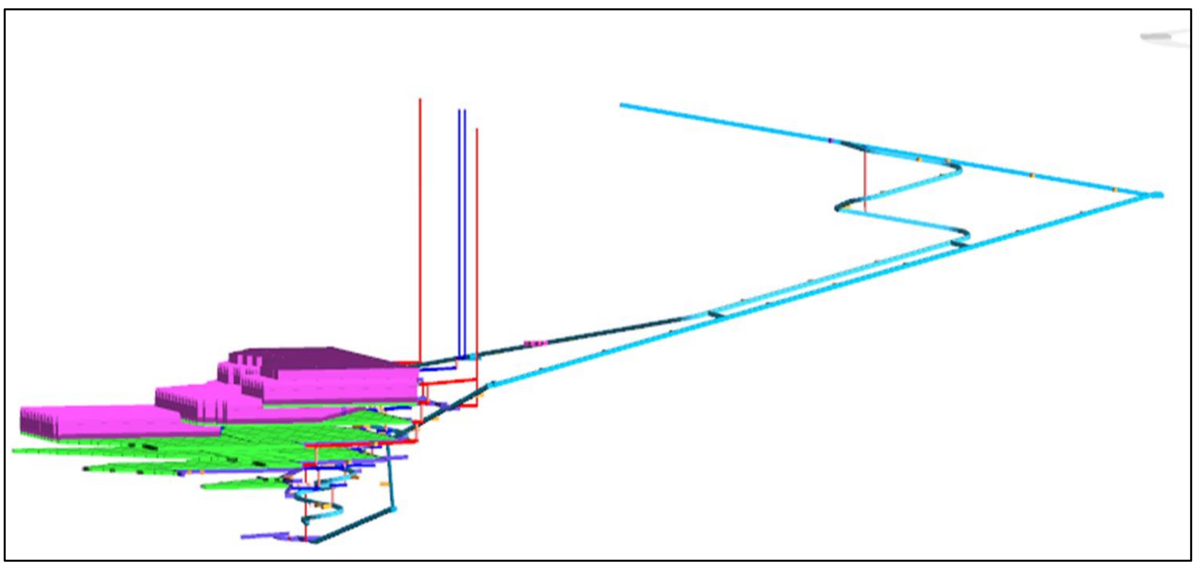
Source: AGP 2023.

**Figure 16-91: Parks/Salyer –Year 7 (looking Southeast)**



Source: AGP 2023.

**Figure 16-92: Parks/Salyer –Year 8 (looking Southeast)**



Source: AGP 2023.

# 17 RECOVERY METHODS

## 17.1 Process Plant Description and Flowsheet

Potential ore sources considered in this report are related to four sources:

- An existing mine stockpile built during the development and operation of a copper open pit and milling facility from 1974 to 1984. The stockpile includes oxide and lower grade sulphide material containing primarily copper mineralization.
- Further development of the existing Cactus West open pit containing oxide and lower grade sulphide material.
- The underground resource called Cactus East located northeast immediately adjacent to the Cactus West open pit and at a depth of 1,200 ft. This resource contains mostly low-grade sulphide material.
- The underground resource called Park Salyer located about 1 mile to the southwest of the Cactus West open pit at a depth of 1,500 ft. This resource contains mostly low-grade sulphide material.

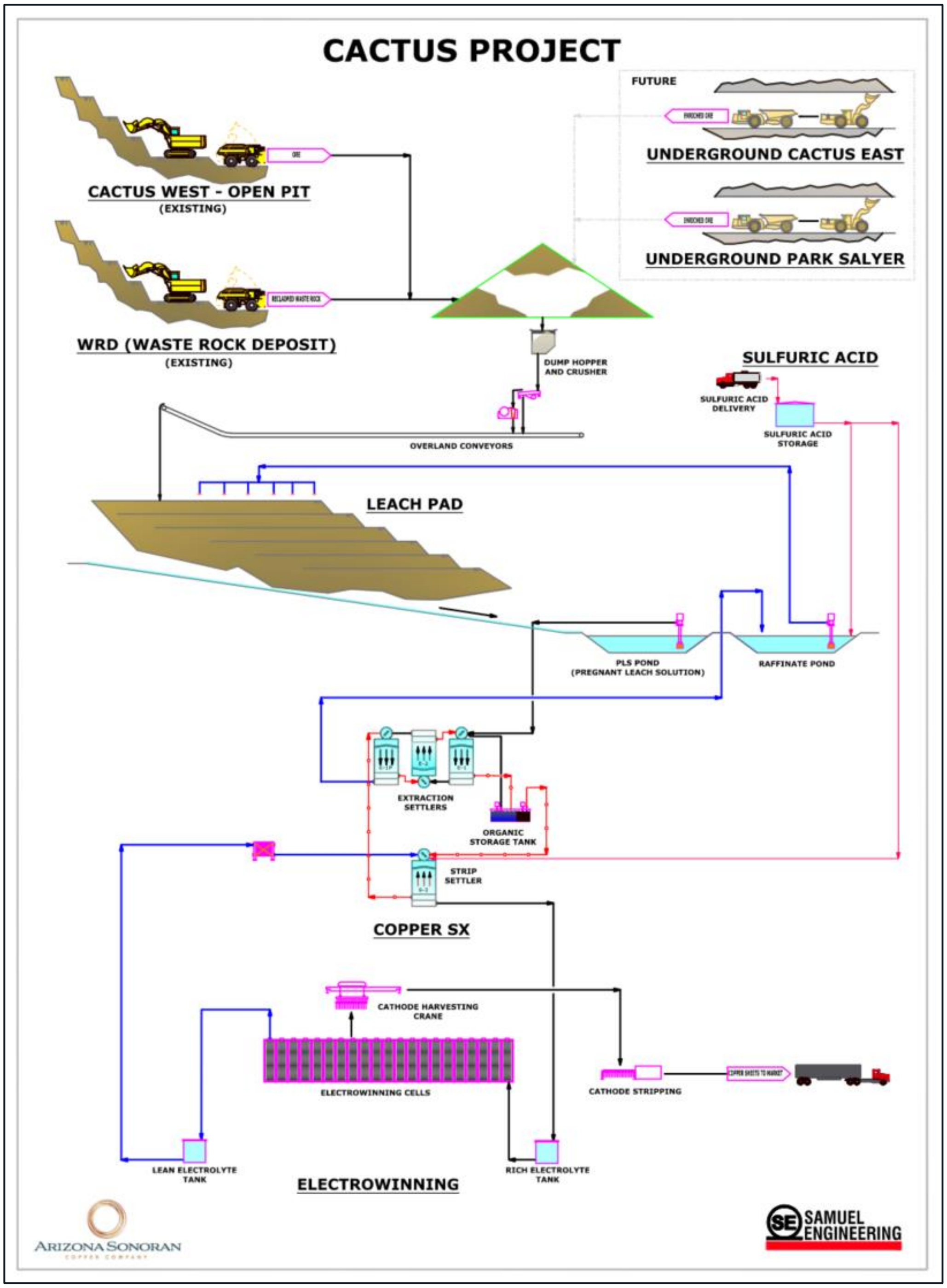
The materials are believed to be suitable for treatment in a heap leach, solvent extraction, and electrowinning (SX/EW) process facility to produce copper cathodes at LME Grade A quality standards ASTM B115-10 - Cathode Grade 1. Figure 17-1 provides a conceptual overview of the operation.

Material mined from the existing stockpile will be placed in 20-ft lifts and material from all other sources will be stacked in 30-ft lifts. Material will be reclaimed and transferred by haul truck to the crushing circuit where it will be crushed down to ¾-in. From the crushing circuit the material will transfer by overland conveyor to the agglomeration drums, mobile transfer conveyors, and mobile radial stacker to be placed on the lined heap leach pad. Leaching solutions, containing dilute sulfuric acid will be pumped and applied to the top of each lift and allowed to percolate though the copper leach material. Copper is dissolved into the solution while acid is consumed at approximately 13.6 lb/ton of material leached. Acid consumption is net of regenerated acid in the SX/EW process. The height of the leach material on the pad will eventually reach 200 ft in overall height.

The pregnant leach solution from the heap leach ponds will be pumped for processing in a copper SX/EW plant capable of producing initially up to 30,000 ton/y of copper cathodes with a design PLS flow of up to 12,000 gpm and grade at approximately 3.0 gpl copper based on a 71% CuT recovery from the heap leaching methods. The electrowinning circuit will expand in Year 3, doubling in size so the overall plant capacity will increase to 60,000 t/y of copper cathodes.

Metalex Technologies (METALEX), a company based in Santiago Chile, designs, and supplies small, modular, relocatable, and standard solvent extraction and electrowinning plants for the recovery of copper. METALEX plants are designed to have a low capital cost and be easily transportable, everything fitting onto trucks or containers for easy transportation of equipment. The transportable nature of the plants is often of interest to projects such as Cactus to keep capital cost lower. They also have resale value should the decision be made to sell the equipment after operation has begun. 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.

**Figure 17-1: Process Flowsheet (Conceptual Flow Diagram)**




Source: Samuel Engineering, 2024.

METALEX has based the SXEW 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 ASCU with a package that maximizes amount of preassembly that can be done, thereby minimizing the time needed onsite for field installation.

The solvent extraction plant is designed to be operated in a series, parallel, or series-parallel configuration with a single stage of stripping. The optionality of the solvent extraction plant will allow the plant to operate at 4,000 gpm, 8,000 gpm, or 12,000 gpm based on the copper grade seen in the mine plan. Two minutes mixing time per mixer-settler unit is anticipated. No wash stages or after-settlers are anticipated or included in the design. A loaded organic tank and diluent storage tank are collocated with the solvent extraction mixer settlers.

Copper electrowinning is expected to initially require 52 cells, then expanding to 104 cells, constructed of polymer concrete, and containing 81 cathodes and 82 anodes each, operating in series and connected to two parallel rectifier transformer units. Two additional rectifier transformer units will be added during the electrowinning expansion. Expected current efficiency is 92% operating at a nominal 28 amps per square foot current density. Cathode stripping from the permanent stainless-steel blanks will be done by a stripping machine that is of a semi-automatic, robotic design.

Copper cathode bundles of up to 5,000 lbs 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 electrowinning operation will be housed in a pre-engineered steel building fitted with an overhead crane for copper production material handling. Siding will be fiberglass or protected steel.

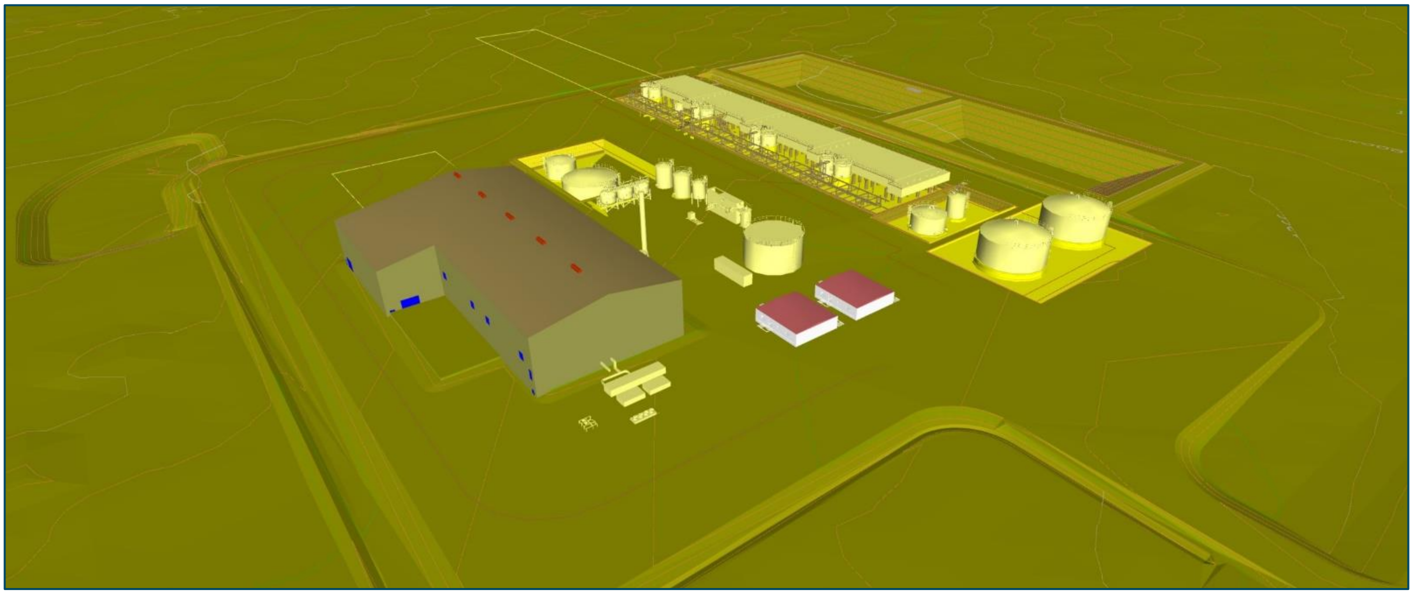
An Administration/Control Building will consist of a new prefabricated double-wide prefabricated structure. The process control room will be located in this building as well as a small wet laboratory for process control assays and mine grade control stockpile sample assays.

The facilities also include a tank farm area comprised of electrolyte solution tanks, electrolyte filters, crud handling system and a solution management holding tank.

A foam-based fire protection system is included for the SX area and diesel back-up pump is included in the fire water system for the processing areas.

The general layout of the proposed processing facilities is provided in Figure 17-2. Figure 17- shows the initial process facilities general arangement plan view with areas shown for future expansion.

**Figure 17-2: General Process Plant Layout**



Source: Samuel Engineering, 2023.

The leach pad will be constructed in four phases and is described in Section 18.10. The initial build out (Phase 1) will be in operation for 3 years. Phases 2, 3 and 4 will be brought online during years 4 and 13 of operation. The leach pad and pond layouts are shown in Figure 17-4. In addition to the first phase of the leach pad, there are three ponds that will also need to be constructed to initiate operations: the Raffinate Pond, the PLS Pond and an Event Pond. 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. Figure 17-5 shows the ultimate leach pad and process facilities envisioned.

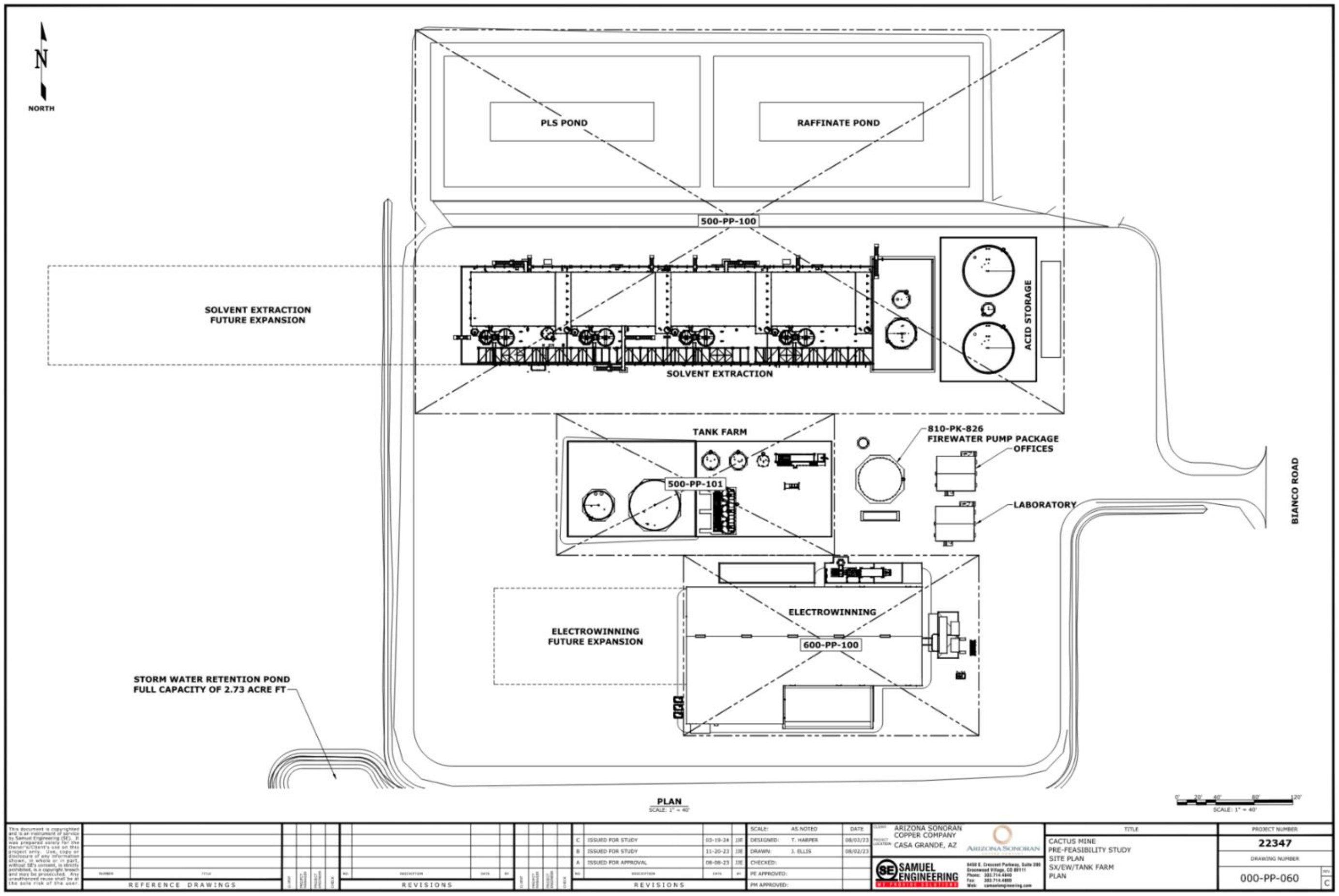
In order to apply the recovery estimates in a timeframe for a production heap leach operation, a distribution over two years was considered with a primary leach cycle of 180 days and extended leaching occurring as subsequent lifts are placed and leached over top of the lift. The distributions are shown in Table 25-1 for the initial year of placement and a second year of leaching time.

Based on the recommended copper recovery and processing assumptions stated in Section 13.7.1, the resulting production plan for the Cactus processing facilities considered from the heap leach and SXEW plant is presented in Table 17-1.

**Table 17-1: Cactus Project Copper Production Plan**

| Material/Product | | Unit | Total | Year -1 | 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 | Year 21 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide HL Placed Material | Crushed | ktons | 76,777 | 2,970 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 1,807 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Placed | ktons | 76,777 | 2,970 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 1,807 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Enriched HL Placed Material | Crushed | ktons | 199,508 | 517 | 12,055 | 12,324 | 12,593 | 14,763 | 14,508 | 16,163 | 9,594 | 6,511 | 6,634 | 6,720 | 8,397 | 9,528 | 9,992 | 9,982 | 10,700 | 10,565 | 10,261 | 10,151 | 7,408 | 143 | 0 |
| | Placed | ktons | 199,508 | 517 | 12,055 | 12,324 | 12,593 | 14,763 | 14,508 | 16,163 | 9,594 | 6,511 | 6,634 | 6,720 | 8,397 | 9,528 | 9,992 | 9,982 | 10,700 | 10,565 | 10,261 | 10,151 | 7,408 | 143 | 0 |
| Copper Recovered to Cathode - Oxide | | ton | 92,768 | 0 | 19,408 | 14,977 | 14,941 | 12,967 | 12,738 | 12,789 | 4,508 | 440 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | | klb | 185,537 | 0 | 38,815 | 29,954 | 29,883 | 25,934 | 25,476 | 25,578 | 9,016 | 881 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Copper Recovered to Cathode - Enriched | | ton | 1,061,647 | 0 | 17,090 | 27,115 | 36,159 | 46,608 | 47,721 | 51,188 | 51,960 | 49,366 | 47,470 | 45,831 | 59,896 | 69,126 | 68,431 | 71,510 | 71,913 | 73,654 | 74,314 | 69,971 | 58,393 | 19,796 | 2,895 |
| | | klb | 2,123,293 | 0 | 34,180 | 54,230 | 72,317 | 93,216 | 95,442 | 102,375 | 103,921 | 98,733 | 94,939 | 91,661 | 119,792 | 138,253 | 136,862 | 143,021 | 143,826 | 147,308 | 148,628 | 139,942 | 116,786 | 39,592 | 5,791 |

**Figure 17-3: Schematic Layout, Process Plant**

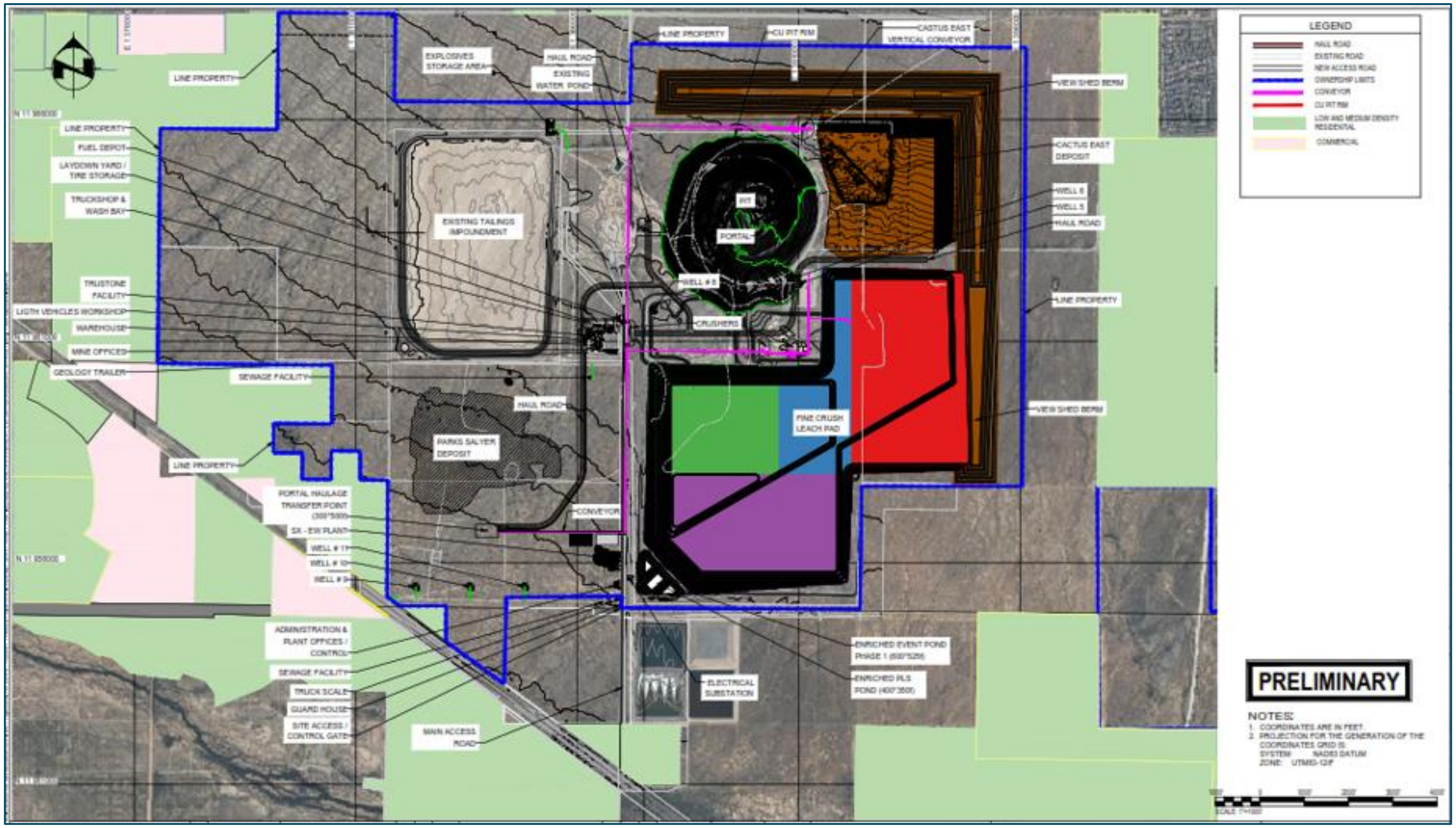



Source: Samuel Engineering, 2023

**Figure 17-4: Pad Design**


Source: Ausenco, 2024.

**Figure 17-5: Overall Site Plan**




Source: Ausenco, 2024.

## 17.2 Reagents, Water, and Power

Projected reagent and operating consumables requirements for the Project are summarized as:

- Energy: 1.9 kWh/lb Cu produced.
- Makeup fresh water: 634 gpm.
- Crushing wear material 0.24 pounds of steel per crusher kWh energy usage.
- Sulfuric Acid: 123 tons/d.
- SX Reagents.
  - Extractant: 37 gpd.
  - Diluent: 269 gpd.
- EW Reagents.
  - Cobalt Sulphate: 2.28 lbs/ton Cu produced.
  - Guar: 0.55 lbs/ton Cu produced;

The estimated average water requirements for the Cactus Mine Process Areas at average full production rates is provided in Table 17-2

**Table 17-2: Process Area Average Annual Fresh Water Use**

| Water Usage Source | Quantity | Units |
|---|---|---|
| Evaporation (net) | 1410 | gpm |
| | 741.1M | gal/y |
| Ore consumption (avg) | 357 | gpm |
| | 187.7M | gal/y |
| SXEW Plant | 100 | gpm |
| | 52.6M | gal/y |
| **Average Annual Totals** | | |
| Flowrate | 1867 | gpm |
| Gallons per Year | 981.3M | gal/y |
| Acre-Feet per year | 3,011 | afy |

Evaporation usage is based on 0.1% heap leach irrigation losses. Evaporation loses occur from active leaching area, non-leaching area, and the ponds. Evaporation from the active leaching area uses an evaporation factor of 0.60, the non-leaching area evaporation factor is 0.40, and the evaporation factor for the ponds is 0.80. Ore consumption is based on a 10% terminal moisture content, a 7% moisture retention in the ore, and 3% initial ore moisture content.

Water supply is described in Section 18 and already available via buried pipeline to the property boundary as a result of prior mining and commercial operations.

Approximately 25 MW of power will be required for the process areas as shown in Table 17-3.

**Table 17-3: Projected Electric Power Usage**

| Area | Unit | LOM Avg | Unit | LOM Avg |
|---|---|---|---|---|
| EW | kWh/lb | 1.32 | MWhr/y | 152,410 |
| SX/TF & Utilities | kWh/lb | 0.36 | MWhr/y | 39,540 |
| Crushing & Leaching | kWh/lb | 0.21 | MWhr/y | 25,840 |

The expected sulfuric acid consumption is approximately 124 ton/d on a 100% basis. Based on quotes from the acid the prospective provider, the delivered concentration is expected to be 94.5%. The heap leach acid consumption estimate is included in Table 17-4.

**Table 17-4: Acid Consumption Heap Leach Operations**

| Acid Consumption/Regeneration | Consumption | Units |
|---|---|---|
| Effective Acid Consumption | 19.34 | lbs $H_2SO_4$/ton ore |
| | 697,140 | lbs $H_2SO_4$/d |
| Regenerated from SX Operation | 463,380 | lbs $H_2SO_4$/d |
| Net Acid Consumption | 233,760 | lbs $H_2SO_4$/d |
| Net Unit Consumption | 0.78 | lbs $H_2SO_4$/lb Cu |
| | 6.5 | lbs $H_2SO_4$/ton ore |

An additional 2 ton/d 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.

## 17.3 Plant Design

The solvent extraction electrowinning plant process design as described in Section 17.1 will include three extraction settlers, one strip settler, a tank house, and initial electrowinning cathode capacity of 30 tpy. The electrowinning will be expanded in year 3 doubling in size to a capacity of 60 tpy. The rectifier turn-up capacity is 20% to account for annual production changes and catch-up.

The plant has been design based on common SX/EW technology utilizing a vendor who designs low capital cost and easily transportable equipment. Equipment will be modular and relocatable in nature to lower the final installed cost of the process plant. The overall Process Design Criteria used for the crushing, conveying and SX/EW facilities is provided in Table 17-5.

**Table 17-5: Process Design Criteria**

| Description | Units | Data | |
|---|---|---|---|
| | | Nominal | Design |
| Site Data | | | |
| Location | | Casa Grande, AZ | |
| Elevation | ft | 1,400 | |
| Ambient Temperature Range | | | |
| Maximum | ˚F | 117.0 | |
| Minimum | ˚F | 19.0 | |
| Average | ˚F | 70.0 | |
| Weather | | | |
| Precipitation, Annual Average | in | 9.2 | |
| Precipitation, Month Average | in/d | 0.7 | |
| Precipitation, Month Average Maximum | in/d | 1.5 | |
| Precipitation, Month Average Minimum | in/d | 0.1 | |
| Relative Humidity, Average | % | 36.0 | |
| Relative Humidity, Maximum | % | 51 | |
| Relative Humidity, Minimum | % | 19 | |
| Atmospheric Pressure, Average | mmHg | 30 | |
| Evaporation, Hourly Maximum | in/hr | 0.021 | |
| Evaporation, Hourly Average | in/hr | 0.012 | |
| Annual Production | | | |
| Production (Copper Cathode) - Initial | tpy | 30,000 | 36,000 |
| Production (Copper Cathode) - Expanded | tpy | 60,000 | 72,000 |
| Operating Schedule | | | |
| Operating Days per Year | d | 365 | |
| Shifts Per Day | shifts/d | 2 | |
| Hours per Shift | h/shift | 12 | |
| Operating Days per Week | d/w | 7.0 | |
| Plant Availability | | | |
| Crushing/Materials Handling | % | 85 | |
| SXEW | % | 98 | |
| Plant Utilization | | | |
| Crushing/Materials Handling | % | 85 | |
| SXEW | % | 98 | |
| Personnel Availability | % | 100 | |
| Overall Plant Availability | | | |
| Crushing/Materials Handling | % | 72 | |

| Description | Units | Data | |
|---|---|---|---|
| | | Nominal | Design |
| SXEW | % | 96 | |
| Feed to Plant (PLS) | gpm | 12,000 | |
| Copper Recovery | | | |
| LOM Copper Recovery (% Total Copper) | % | 76.1 | |
| LOM Copper Recovery (% Soluble Copper) | % | 86.3 | |
| Sulfuric Acid Consumption (Gross) | | | |
| Oxide Heap Leach | lbs/ton | 22 | |
| Enriched Heap Leach (Cactus East/West) | lbs/ton | 21 | |
| Enriched Heap Leach (Parks Salyer) | lbs/ton | 16 | |
| Ore Characteristics | | | |
| Ore Specific Gravity | - | 2.63 | |
| Ore Bulk Density | t/ft3 | 0.057 | |
| Maximum ROM Ore Size | in | 24.00 | |
| Ore Moisture Content | % | 3.00 | |
| Crusher Work Index (CWi) | | | |
| CWi | kWh/st | 3.90 | |
| Classification | | Very Soft | |
| Bond Work Index (BWi) | | | |
| BWi | kWh/st | 11.3 | |
| Classification | - | Medium | |
| Abrasion Index (Ai) | | | |
| Ai | grams | 0.047 | |
| Classification | | Lightly Abrasive | |

## 17.4 Materials Handling

As described in Section 17.1, material will be transferred to the crushing circuit from one of the four ore resources and introduced to the crushing circuit. The primary crusher will be a sizer located just east of the Truestone facility capable of sizing 2,740 tons/h. Material will then be conveyed by overland conveyor to the secondary screen with oversize feeding the secondary crusher. The secondary crusher will have the capacity of 1,436 tons/h then feeding the two tertiary crushers each having a capacity of 718 tons/h. All material will have a $P_{80}$ size of ¾".

Once crushed, all material will be agglomerated through two agglomeration drums with the overall capacity of 2740 tons/h. After agglomeration the ore will be transferred to the leach pads by overland tripper conveyor and tripper car tied to a series of mobile grasshopper conveyors, index conveyor, and radial stacker.

Table 17-6 is a list of the conveying equipment that will be used to transfer from the primary crusher to the secondary/tertiary crushing plant, agglomeration, and then for placement onto the leach pad.

The crushing and conveying system included in the project design are based on used equipment that ASCU is currently negotiating the purchase of with an equipment broker for a facility located in Namibia. The broker AMKING based in Oroville, California has an exclusive agreement with Orano to sell the various assets that make up the Trekkopje project. The AREVA ORANO Trekkapje material handling facilities located in Namibia has an oversized throughput of 7.5k tph for the Cactus application criteria and is partially installed, however has not been commercially operated.

**Table 17-6: Proposed Conveying/Stacking Equipment List**

| Conveyor Number | Qty | Description | Beltwidth (in) | Horizontal Length(ft) | Vertical Lift (ft) | Installed Motor Power (kW) |
|---|---|---|---|---|---|---|
| 425-CV-310 | 1 | Fine Crushed Ore Primary Crusher Discharge Conveyor | 54 | 200 | 49 | 150 |
| 425-CV-311 | 1 | Fine Crushed Ore Secondary Crusher Discharge Conveyor | 42 | 200 | 49 | 100 |
| 425-CV-312 | 1 | Fine Crushed Ore Tertiary Crusher Return Conveyor | 42 | 200 | 30 | 100 |
| 425-CV-314 | 1 | Fine Crushed Ore Overland Tripper Conveyor | 48 | 3950 | 13 | 373 |
| 425-CV-315 | 1 | Fine Crushed Ore Tripper Car | 54 | 32 | 10 | 37 |
| 425-CV-319 | 1 | Fine Crushed Ore Overland Conveyor | 48 | 2000 | 98 | 373 |
| 425-CV-366-393 | 28 | Fine Crushed Ore Portable "Grasshopper" Conveyor | 54 | 151 | 10 | 56 |
| 425-CV-394 | 1 | Fine Crushed Ore Index Feed Conveyor | 54 | 138 | 11 | 112 |
| 425-CV-395 | 1 | Fine Crushed Ore Index Conveyor | 54 | 134 | 0 | 56 |
| 425-CV-396 | 1 | Fine Crushed Ore Radial Stacking Conveyor | 54 | 144 | 26 | 112 |
| NA | 1 | Electrical Control Design and Supply | | | | |

The Trekkopje project MAXI Phase represents the largest fraction of the installation and incorporates all mechanical and electrical gear specific to twin Primary Crusher Relocatable Sizer Stations, twin secondary/tertiary crushing and screening circuits, three parallel agglomeration circuits, all interconnecting in-plant conveyors and feed mechanisms for a combined design capacity of 7,870stph. This system includes both a plant compressed air system and uninstalled Donaldson dust extraction system with six separate baghouses which were intended to provide collection at the various process steps and material transfer points. Figure 17.5 shows the Trekkopje crushing and screening plant.

The processing plant was designed to crush ore at a rate of 7,870stph to a product size of 100% passing 1.5”. The crushing and screening plant consists of two parallel circuits from primary through to tertiaries, each circuit with a throughput design capacity of 3,935stph. Although the crushing/screening plant is sized for the full 7,870stph production rate, the leach pad equipment of interest is per the MIDI design and stacking equipment supply in this area was limited to one circuit sized for half tonnage, approximately 3,935stph.

An onsite contractor has determined an estimate for the disassembly, transport of 70 miles to the port at Walvis Bay and ocean freight to Houston USA.

**Figure 17-6: Trekkopje Crushing and Screening Plant**

Source: https://inventory.amking.com/, 2024.

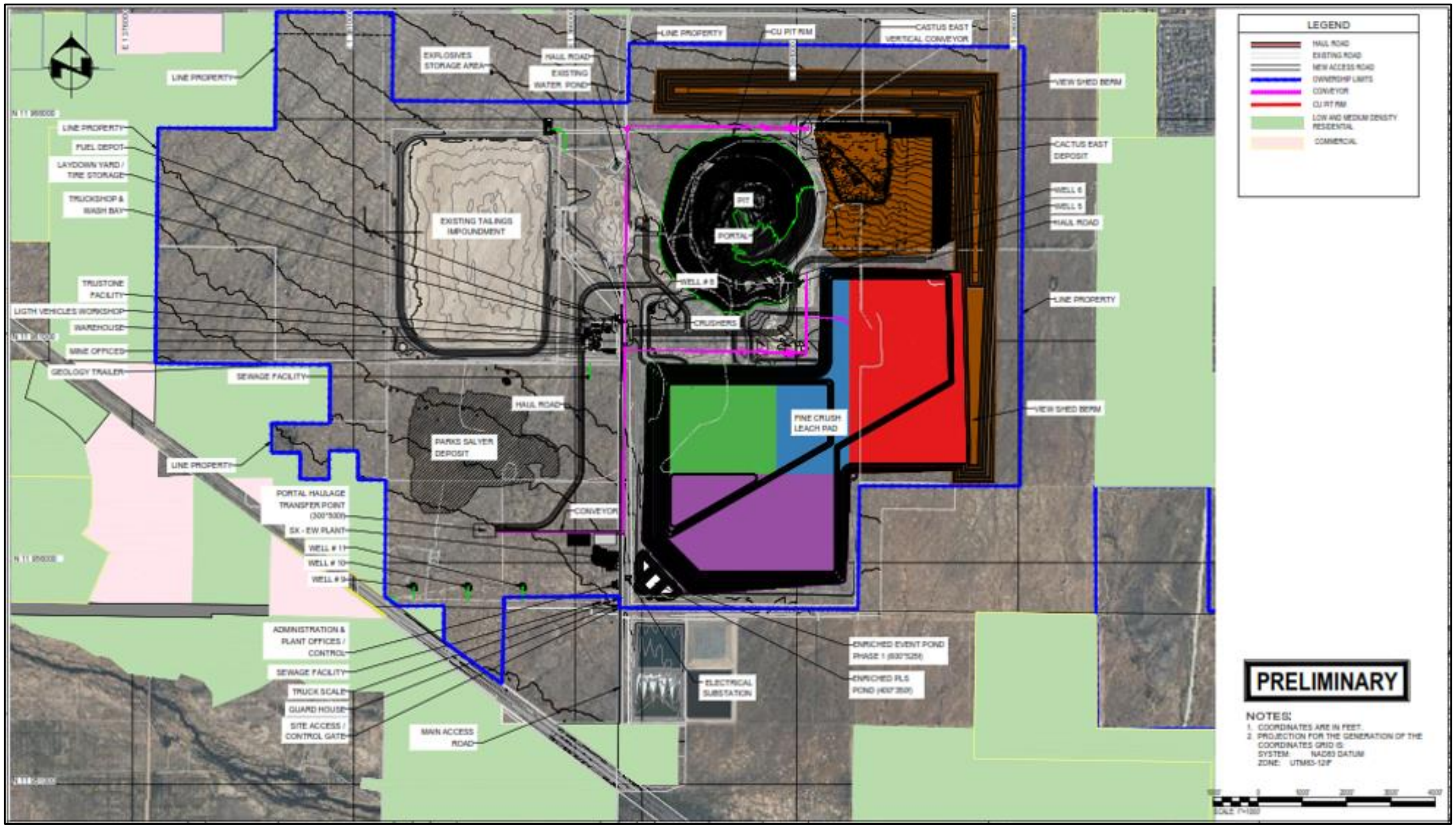
# 18 PROJECT INFRASTRUCTURE

## 18.1 Introduction

The Cactus Mine project, located at the historic Sacaton Mine, is 40 road miles southeast of the Greater Phoenix metropolitan and 3 mi northeast of the city of Casa Grande, Pinal County, Arizona. The site is accessible from W Maricopa Casa Grande Highway Road via Bianco Road, a 2.2-mile paved access road. The site will require the following facilities as listed below and shown in Figure 18-1.

- Mining facilities including administration offices, change house, truck shop, explosives storage, fuel storage and distribution, ore stockpiles, waste stockpiles, and truck wash.
- Process facilities including the SX/EW process plant, crushing facilities, process plant workshop, change house, assay laboratory, and freshwater infrastructure.
- Heap leach pads and ponds.
- Power supply and distribution.
- General facilities include a gatehouse, administration building, and weighing scale.
- Catchments, ponds, water wells, drainage, and other site water management infrastructure.
- The location of site facilities was based on the following criteria:
- Locate the facilities within the claim boundaries.
- Consider locations of existing features such as roads, buildings. power lines, open pit, tailings, stockpiles, and waste rock areas.
- Comply with flight path requirements outlined by nearby Casa Grande Municipal Airport due to the minesite's proximity.
- Utilizing existing infrastructure like buildings, access roads, and power supply to greatest extent possible.
- Locate the rock storage facilities near the mine pits to reduce haul distance.

**Figure 18-1: Infrastructure Layout Plan**



Source: Ausenco, 2024.

## 18.2 Roads and Logistics

### 18.2.1 Site Access

The property is accessed from W Maricopa Casa Grande Highway that links Casa Grande and Maricopa, Arizona. Bianco road (the primary access road) currently extends North to an existing building known as Truestone facility, other offices, a water storage pond and a small substation.

The following buildings and facilities will be accessible from the primary access road:

- Control Gate, Guard House and weigh scale.
- SX/EW plant, plant office, workshop, assay lab and electrical substation.
- Mine administration office, truck shop, warehouse, light vehicle workshop, tire storage, and fuel storage facility.

From the Highway, the primary access road is a paved road of 28.9 ft width and 9,946.9 ft length. It will be repaired and upgraded with an additional asphalt layer to ensure suitability for daily operational traffic.

Copper cathode produced at the mine will be picked up by a copper broker for transport and sale. Copper bundles will be prepared, stored and loaded onsite for shipment by truck. The storage and loadout facilities are included as part of the processing plant. Existing roadways will be used for transport. No additional infrastructure is required to facilitate transport of product from the mine.

Existing unpaved maintenance roads originating from primary access road will be repaired to ensure suitable light vehicle traffic. Additional maintenance roads to connect explosive storage and water wells to existing unpaved roads will be constructed.

A list of roads with their details is provided in Table 18-1.

**Table 18-1: Access and Haul Roads**

| Road | Condition | Description | Width (ft) |
|---|---|---|---|
| Primary Access Road | Paved | Existing | 28.90 |
| Maintenance Roads | Unpaved | Existing | 24.00 |
| Maintenance Roads | Unpaved | New | 24.00 |
| Haul Road | Unpaved | New | 121.50 |

### 18.2.2 Airports

There is no airport at project site. Nearby airport facilities are listed in Table 18-2.

**Table 18-2: Nearby Airports**

| Airport | Distance to Site (Road Travel) (mi) |
|---|---|
| Ak-Chin Regional Airport | 9.5 |
| Casa Grande Municipal Airport | 10.5 |
| Eloy Municipal Airport | 20.6 |
| Chandler Municipal Airport | 35.7 |
| Coolidge-Randolph Municipal Airport | 36.6 |
| Phoenix-Mesa Gateway Airport | 45.7 |
| Phoenix Sky Harbour International Airport | 48.7 |
| Gila Bend Municipal Airport | 63.5 |

Given the site's proximity to the Casa Grande Municipal Airport, the maximum height of the site facilities will be in accordance with the Federal Aviation Act of 1958. A summary of the relevant Federal Aviation Regulations (FAR) Part 77 Section 77.9 is provided as follows:

> *{77.9} – Any person/organisation 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.
- Any construction or alteration within 10,000 ft of a public use or military airport that exceeds a 50:1 surface from any point on the runway of each airport with its longest runway no more than 3,200 ft.
- Any construction or alteration within 5,000 ft of a public use heliport that 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 the height or location.

### 18.2.3 Rail

There is an existing isolated rain spur that dead ends in front of the remaining processing plant building from historic Sacaton mining operations. It is not connected to the main line that runs parallel to the W Maricopa Casa Grande Highway. There are no current plans to reconnect or use the rail line.

### 18.2.4 Security

The site will be accessible year-around via primary access road off W Maricopa Casa Grande Highway.

Access to the processing plant, mining areas, workshops, administrative buildings and other process facilities will be controlled by a control gate and guard post at the entrance of the site. The site has existing peripheral wire fencing.

A view shed berm with width of 80 ft and height of 120 ft, for a length of 1,375 ft in the east and northeast sides of the project will be constructed to mitigate the visual impacts and noise from mining activities.

#### 18.2.5 Accommodation

Due to the close proximity to the town of Casa Grande and the city of Phoenix, personnel will be housed offsite. Vehicles will be provided to personnel for on-site transportation.

## 18.3 Stockpiles

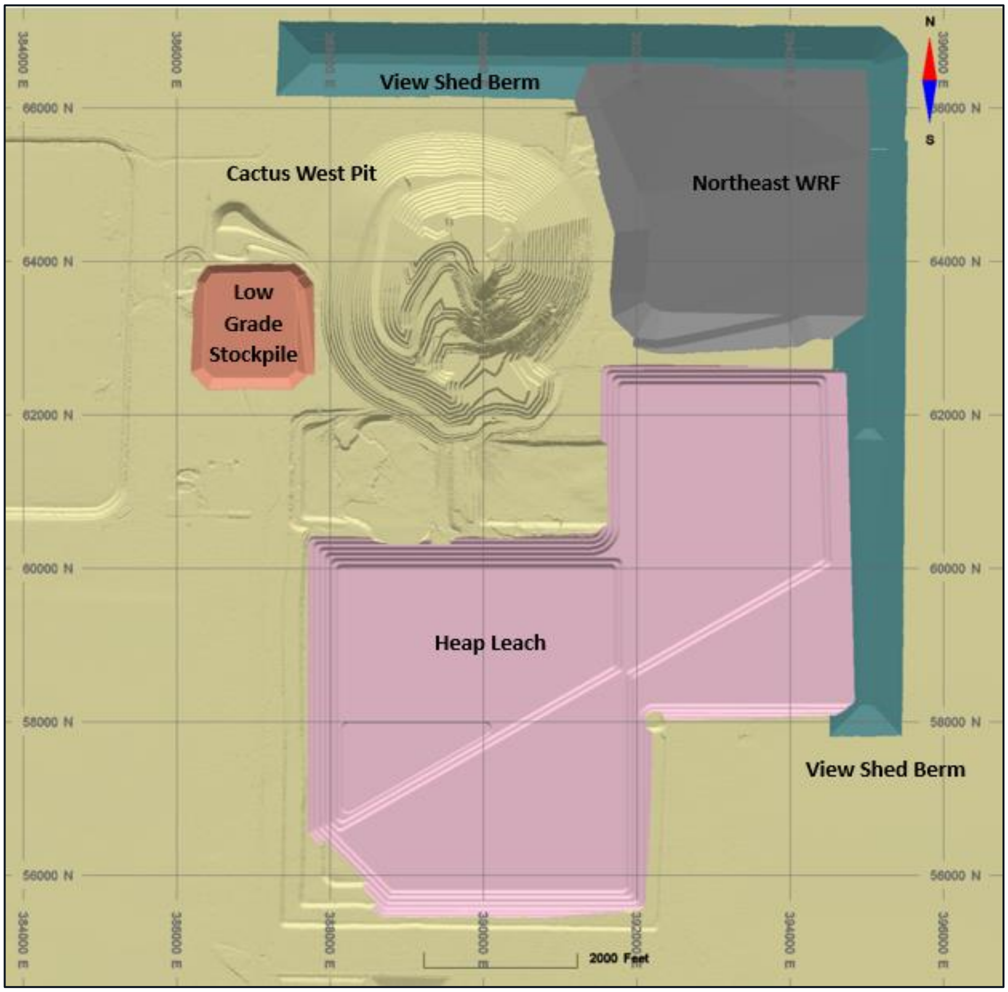
Up to 12 Mton of Cactus West low grade and pre-production ore over the life of the mine will be cycled through a low-grade stockpile to help manage feed grade and work around availability of the crushing units. This material will be sent to the crusher throughout the open pit mine life and at the end of mining. The low-grade stockpile is located north of the mining truck shop facilities west of the Cactus pit. The peak stockpile size in the mine schedule is 9 Mton.

## 18.4 Waste Rock Storage Facilities

Waste materials generated from mining Cactus West and the Stockpile areas will be composed of predominantly Gila Conglomerate and Alluvium overburden (80%) with the remainder being granite and porphyry rock with lower copper grades. No segregation of waste material is required.

In general, design considerations assumed an overall reclaimed slope of 3.5:1 for the view shed berms and 2.5:1 slope for the North-East Waste Rock Facility. A swell factor of 30% was applied to all waste material. The North and East view shed berms have a capacity of 32 Mton and 28 Mton respectively. The North-East Waste Rock Facility capacity is 90 Mton. Waste rock facilities are shown in Figure 18-2.

**Figure 18-2: Waste Rock Storage and Ore Stockpile**




Source: AGP, 2023.

## 18.5 Built Infrastructure

### 18.5.1 Haul Roads

Haul roads will be constructed for heavy vehicle movement to the underground mine portals, truck shop, fuel storage, crushing facility, mine pit, and rock storage facilities.

The roads will be constructed to the following specifications:

- Haul roads are designed to accommodate two 141-ton class haul trucks with assumed operating width of 21.78 ft.

- A total running width of 121.5 ft was allocated for two-way traffic.
- Safety berms are considered of 22.63 ft width (3/4 tire height) on both sides of the haul roads.

### 18.5.2 Support Buildings

As shown in the site infrastructure layout in Figure 18-1, the mine will require several support buildings. A list of support buildings is shown in the Table 18-3.

**Table 18-3: Description of On-Site Buildings**

| Building Name | Construction Type | L (ft) | W (ft) | H (ft) | Area ($ft^2$) |
|---|---|---|---|---|---|
| Administration Building | Modular Building | 128 | 60 | 9 | 7,680 |
| Change House | Modular Building | 42 | 22 | 9 | 924 |
| Communications, IT and Computing | Modular Building (In Mine Office) | 20 | 12 | 9 | 240 |
| Gatehouse & Weighbridge | Modular Building | 20 | 12 | 9 | 240 |
| Plant Workshop and Warehouse | Pre-engineered building | 144 | 74 | 26 | 10,656 |
| Assay Laboratory | Modular Building | 78 | 30 | 9 | 2,340 |
| MineOffice/Administration/ Changehouse | Modular Building | 175 | 60 | 9 | 10,500 |
| Mine Maintenance Office | Modular Building | 48 | 34 | 9 | 1,632 |
| Mine Truck Shop | Truestone building modifications | 140 | 80 | 50 | 11,200 |
| Heavy Equipment Maintenance | Pre-engineered building | 131 | 107 | 26 | 14,017 |

### 18.5.3 Explosives Facilities

Explosive magazine will be located near the northeast corner of the existing tailings impoundment. A 24 ft wide access road will be constructed from existing unpaved roads in the area to transport the explosives and accessories to the pit and mining portals.

Explosive facilities are limited to a fenced in area with a concrete pad. Electric power and water will be available for use by a registered explosives contractor.

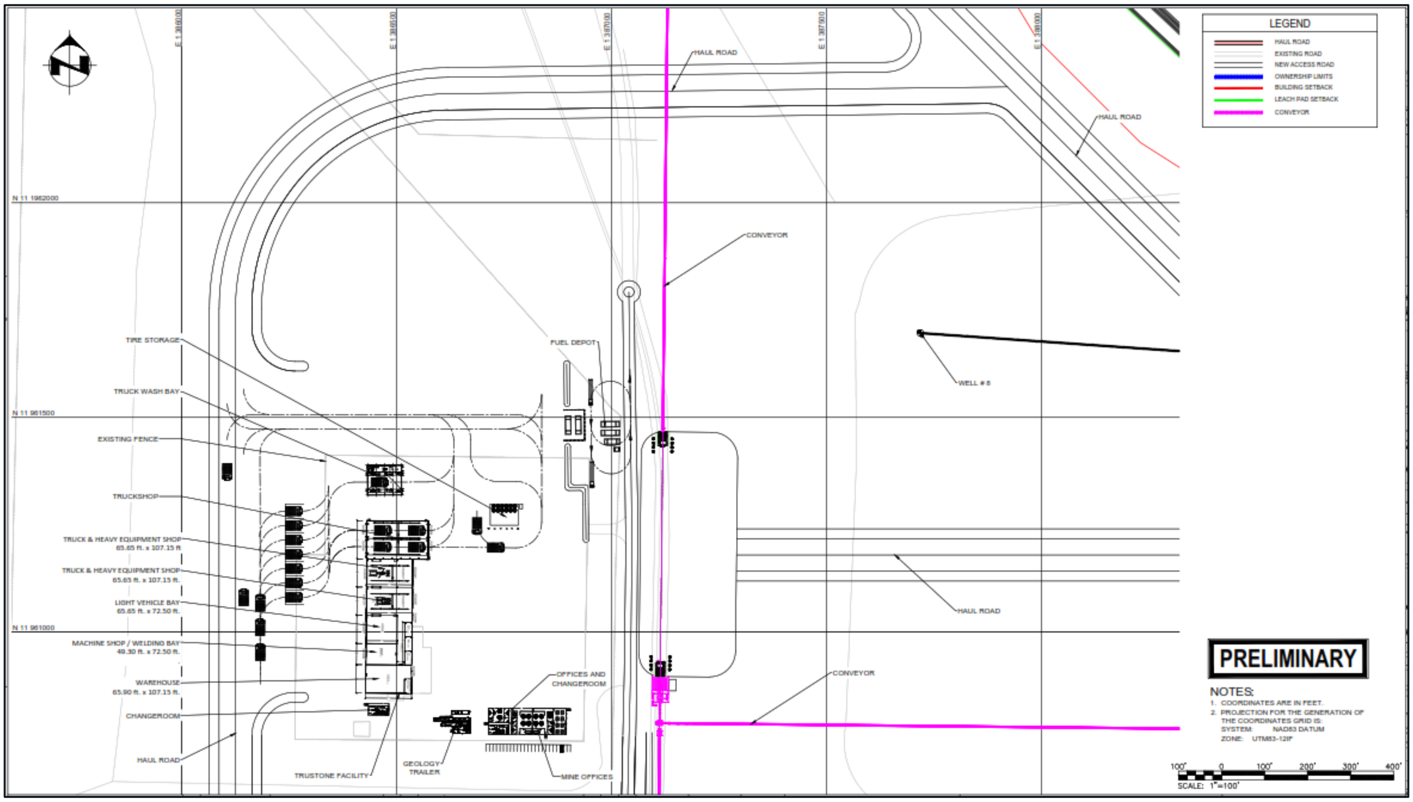
### 18.5.4 Truck Shop and Wash

Truck shop and truck wash facilities are located adjacent to Fuel storage within the Truestone complex. The truck shop will be created by modifying the existing Truestone building. It will accommodate one 25-ton crane to provide maintenance to four trucks similar to a CAT 785D. Inside the truck shop, there will also be two closed warehouses and storage areas to store the spare parts for truck maintenance.

The truck wash is an open area (without rooftop) located on the same platform as the truck shop. It will have the capacity to wash one truck at a time and can also be used to wash lightweight vehicles.

Figure 18-3 shows the truck shop, wash bay, fuel storage, tire storage, light vehicle workshop and my office.

**Figure 18-3: Building Layout**




Source: Ausenco, 2023.

### 18.5.5 Mine Office / Administration / Change House

The Mine office / administration / change house is a modular building of 10,500 $ft^2$ area located near the existing Truestone facility. It will be divided in three areas: mine dry area, mining office area, and lunchroom. Each area will have one entrance door.

The building will contain:

- changing rooms with lockers and showers for men.
- changing rooms with lockers and showers for women.
- 12 closed office rooms.
- four conference rooms.

- three open areas with desks; and
- a lunchroom of 80 people capacity.

### 18.5.6 Administration Building

Administration building is a modular building of 7,680 $ft^2$ area located near the SX-EW facility. It will be divided in four areas: dry area, plant office area, lunchroom, and infirmary. The building will have three entrance doors.

The building will contain:

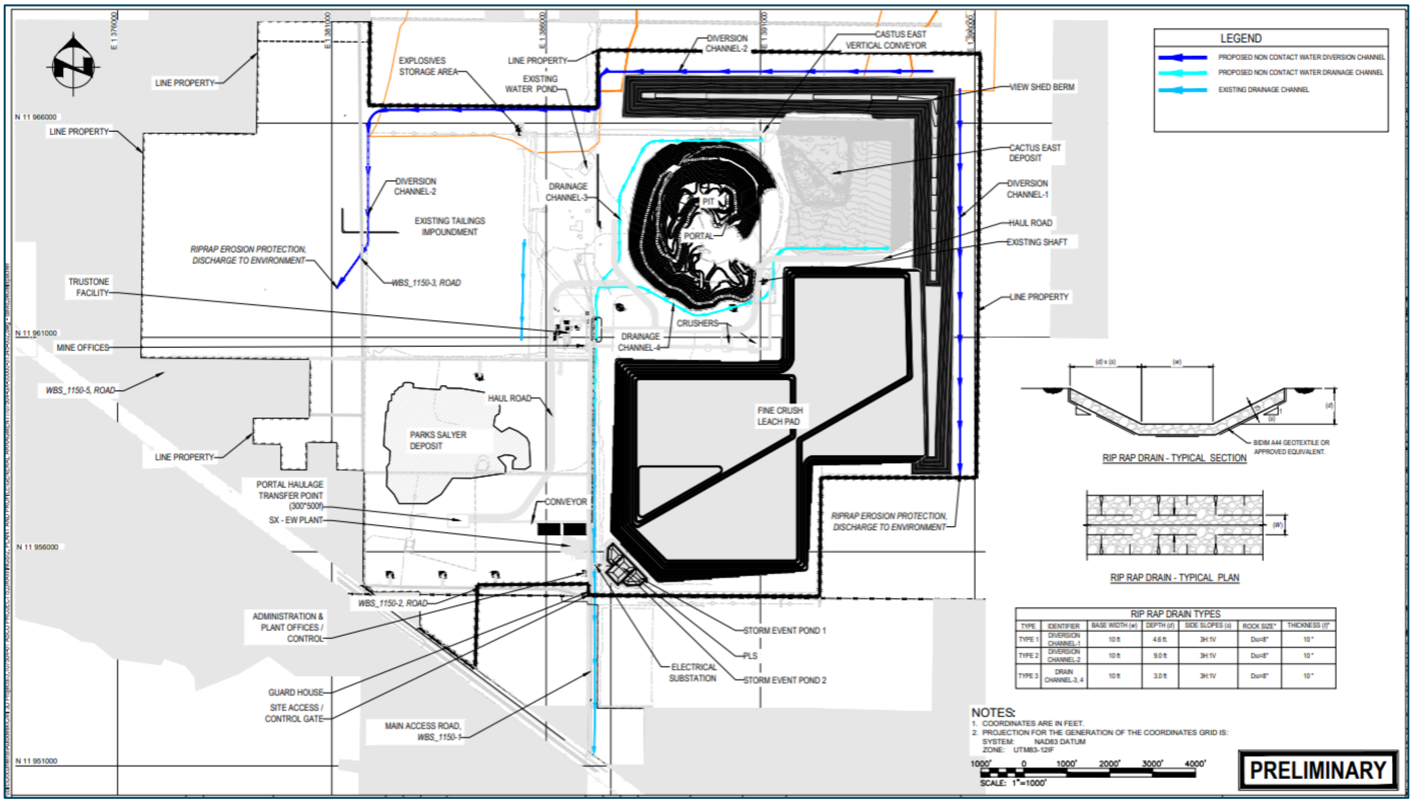
- changing rooms with lockers and showers for men.
- changing rooms with lockers and showers for women.
- 12 closed office rooms.
- four conference rooms.
- three open areas with desks; and
- an infirmary with separate entrance.

### 18.5.7 Stormwater Controls

A surface water inflow and outflow analysis were prepared for the Cactus Mine Facility. As part of this analysis, stormwater inlet and outlet tests were evaluated. Calculations were made to evaluate the run on/off infrastructure that would be required for the development of the Facility.

The surface water analysis was divided into two phases: 1) external drainage (i.e. drainage) and 2) on-site drainage (i.e. runoff). The purpose of the ongoing analysis was to determine the peak discharge rates entering the facility from the north and design infrastructure to route that discharge around the facility and to existing natural drains. The objective of the second-round analysis was to determine the maximum discharge rates leaving the facilities and design infrastructure to route and store that runoff where necessary. Generally, facility inflows will be collected from existing sheet flow and shallow drainages. These sources will be directed into existing internal and external drainages that will not require substantial modification as the majority of the infrastructure (roads, graded areas, drainageways and culverts) are existing. New culverts and minor grading have been included in the estimates. Figure 18-4 shows the extent of the required stormwater control infrastructure.

**Figure 18-4: Stormwater Management Plan**




| RIP RAP DRAIN TYPES | | | | | | |
|---|---|---|---|---|---|---|
| TYPE | IDENTIFIER | BASE WIDTH (w) | DEPTH (d) | SIDE SLOPES (s) | ROCK SIZE* | THICKNESS (t)* |
| TYPE 1 | DIVERSION CHANNEL-1 | 10 ft | 4.6 ft. | 3H:1V | D50=8" | 10" |
| TYPE 2 | DIVERSION CHANNEL-2 | 10 ft | 9.0 ft | 3H:1V | D50=8" | 10" |
| TYPE 3 | DRAIN CHANNEL-3, 4 | 10 ft | 3.0 ft | 3H:1V | D50=8" | 10" |

Source: Ausenco, 2024.

Rainfall data used in the Facility's hydrological analysis was obtained from the National Oceanic and Atmospheric Administration (NOAA) Atlas 14 Rainfall Frequency Database. Upon review of the rainfall frequency map, the 100-year, 24-hour maximum storm event was determined to be 3.68 inches.

## 18.6 Power Supply

The HV transmission from the grid will be connected by distribution overhead power line to a 69 kV/13.8 kV new substation on site, to cover the electrical requirements needed for the process design. This substation will require a footprint that is 350 ft by 350 ft. The advantage of the 69 kV line is that it runs very close to the main substation and does not require a lot of infrastructure for connection. The connection from the existing 69 kV line to the main substation will be through a short section of overhead power line with a single circuit structure.

The main substation consists of a small 69 kV switchyard to feed a 50/62.5 MVA transformer and feed with medium voltage cables to a 13.8 kV MV Switchgear. The transformer will feed 80% of the total plant load under normal operation with bus tie open. The main substation is considered to be installed in a central area of the project to facilitate the distribution of the 13.8 kV overhead distribution power lines.

## 18.7 Electrical Distribution

The substation will distribute power in 13.8 kV to all areas of the project including the process plant, administration building, the crushing/conveying facilities, PLS pumps, raffinate pumps and the underground portal areas. Table 18-4 shows the distribution of power to different areas of the mine.

**Table 18-4: Electrical Load List**

| WBS | INFRASTRUCTURE | MISC LOADS | CONNECTED LOAD | | OPERATING LOAD | | ANNUAL ENERGY CONSUMPTION | |
|---|---|---|---|---|---|---|---|---|
| | | kW | kVA | kW | kVA | kW | HOURS | MWh |
| 1510 | Administration Building | 63 | 105 | 90 | 84 | 72 | 4,380 | 316 |
| 1520 | Change House | 10 | 17 | 15 | 14 | 12 | 4,380 | 54 |
| 1610 | Security Gatehouse & Weighbridge | 9 | 15 | 13 | 12 | 10 | 8,760 | 90 |
| 1620 | Plant Workshop and Warehouse | 32 | 53 | 46 | 42 | 36 | 4,380 | 160 |
| 1630 | Laboratory | 23 | 39 | 34 | 31 | 27 | 4,380 | 118 |
| 1720 | Main Substation | 51 | 85 | 73 | 68 | 58 | 8,760 | 512 |
| 1910 | Water Wells | | 260 | 224 | 221 | 257 | 8,760 | 2,253 |
| 2920 | Mine Maintenance Office | 22 | 36 | 31 | 29 | 25 | 4,380 | 108 |
| 2930 | Mine Truck Shop | 166 | 277 | 238 | 222 | 191 | 4,380 | 835 |
| 2940 | Truck Wash | 44 | 74 | 64 | 59 | 51 | 4,380 | 223 |
| 2950 | Heavy Equipment Maintenance | 95 | 159 | 137 | 127 | 109 | 4,380 | 479 |
| 2960 | Tire Workshop | 26 | 43 | 37 | 34 | 30 | 4,380 | 130 |
| 4200 | HLP Ore Handling Area | 3,829 | 4,504 | 3,873 | 3,603 | 3,099 | 8,760 | 27,145 |
| 4300 | Pregnant Leach Solution Management | 839 | 987 | 849 | 790 | 679 | 8,760 | 5,949 |

| WBS | INFRASTRUCTURE | MISC LOADS | CONNECTED LOAD | | OPERATING LOAD | | ANNUAL ENERGY CONSUMPTION | |
|---|---|---|---|---|---|---|---|---|
| | | kW | kVA | kW | kVA | kW | HOURS | MWh |
| 5000/6000 | Solvent Extraction & Electrowinning Area (SX/EW) | 21,131 | 24,860 | 21,380 | 19,888 | 17,104 | 8,760 | 149,828 |
| 7000 | Reagents | 218 | 257 | 221 | 206 | 177 | 8,760 | 1,549 |
| 2500 | Underground Cactus East | 15,985 | 18,806 | 16,173 | 15,045 | 12,939 | 8,760 | 113,342 |
| 2600 | Underground Parks/Salyer | 15,985 | 18,806 | 16,173 | 15,045 | 12,939 | 8,760 | 113,342 |
| **TOTAL** | | **58,528** | **69,383** | **59,670** | **55,520** | **47,814** | **118,260** | **416,115** |

Six distribution lines will be constructed at the project site to provide stepped-down power to the site administration, process facilities, underground mines, and well water distribution system.

Individual E-house/MCC buildings and MV to LV transformers are located strategically around the site to provide power and control to individual areas and processes, and to minimize distance of LV power runs.

## 18.8 Fuel

Three 80,000-L tanks of diesel will be used to fuel equipment onsite during operations. This is approximately three days of fuel consumption for mining and site equipment. These tanks will be located north-east of the Truck Shop area, out of any main vehicle travel path, and on a concrete pad with containment.

## 18.9 Water Supply and Management

### 18.9.1 Potable Water

Potable water will be supplied by Arizona Water Company from an existing 12-inch pipeline, which currently terminates approximately a mile south-east of the planned site operations. Existing valves are available to facilitate a future connection. Current potable water needs are estimated based on a total of 480 employees occupying multiple facilities to include a guardhouse, administration building, a mining office with a geology laboratory, and separate areas for mining employees from both Parks/Salyer and Cactus East.

A demand of 20 gallons per employee per day, equivalent to an average of 6.67 gpm, was implemented per EPA guidelines to size a site reservoir. A 10,000-gallon tank will satisfy the potable needs of the mining operations. The tank will be filled using a centrifugal booster pump and will provide 24 hours of retention.

Facility distribution was determined thorough peak flow usage, which is based on volumetric flow rate and requires a diversification factor. One-third of the 480 employees, or 160, are expected to be onsite at any given time. One-half of that figure, or 80, are conservatively estimated to be engaged in an hourly shift-change period, and one-half in turn of that figure, or 40, are expected to be actively using potable water facilities and receptacles. The duration of peak flow is approximately 10 minutes, which occurs during shift change. Peak flow was also based on the requirements of fixtures such as flushomatic valves, which are typical with industrial-use toilets and urinals. Toilets for this facility will be planned with gravity tanks, which simplifies system design. Urinals with a flushomatic valve use approximately one gallon per flush, but the brevity of the flush scales to a flow rate of approximately 12 gpm, which is the highest flow rate of any potable receptacle. Thus, the distribution piping needs to be able to provide a minimum of 12 gpm to any facility containing a urinal, although the overall consumption may be higher or significantly lower at a given time. Typical receptacles, including flow and pressure requirements, are presented Table 18-5.

**Table 18-5: Typical Industrial Water Fixtures, Receptables, and Requirements**

| Fixture or Receptacle | Flow (gpm) | Minimum Pressure (psig) |
|---|---|---|
| Toilet (tank) | 2 | 20 |
| Urinal (flushomatic valve) | 12 | 25 |
| Shower head | 2.5 | 20 |
| Sink faucet | 0.5 | 8 |

Source: Pienta, 2017 and IPC, 2021.

Final numbers will depend on the number of potable fixtures and receptacles installed. The flow requirement can be reduced significantly for example by using water-free urinals. Peak flow averages are distributed among the facilities as shown in Table 18-6, which totals 180 gpm. Conservative estimations were made in lieu of available data.

**Table 18-6: Facility Distribution of Peak Potable Water Flow**

| Facility | Employees | Active | Toilet | Urinal | Sink | Shower | Potable Peak Usage (gpm) |
|---|---|---|---|---|---|---|---|
| Guardhouse | 2 | 1 | 1 | - | | – | 2 |
| Admin. Building | 14 | 2 | | 1 | 1 | | 13 |
| Parks Salyer Mining Operations | 150 | 13 | 3 | 3 | 2 | 5 | 56 |
| Mining Office and Geology Laboratory | 14 | 2 | 1 | 1 | | – | 13 |
| Cactus East Mining Operations | 300 | 25 | 5 | 5 | 5 | 10 | 98 |

The facility also contains two sewage plants, the capacity given as 64 gpm each. The estimated daily flow rate of 6.67 gpm is well within this range. The peak flow rate of 180 gpm exceeds the combined plant capacity but will rarely occur and be of short duration, allowing the systems to stabilize over time.

### 18.9.2 Plant Water

Plant (non-potable) water will be supplied by several sources, both active and planned. The sources first considered will provide water appropriate for process, utility and sewage uses. Another critical purpose is to dewater the underground region in the immediate vicinity of the mines to avoid operational complications. Removing and best utilizing the recovered water is a priority; utilization will depend on chemical analyses. Rainfall is not considered in this analysis.

Existing water supply facilities include a 16-inch pipeline that currently supplies up to 600 gpm to an existing onsite pond from offsite well sources. The abandoned and flooded production shaft likewise was rated at a production flowrate of 600 gpm based on previous performance analyses. New water supply infrastructure includes a collection system from the existing wastewater treatment facility (WWTF) in Casa Grande. This source is capable of providing up to 5 Mgpd (3,472.2 gpm) depending on project requirements. Four dewatering wells were also included in the scope. Based on the hydrology report, these dewatering wells have the potential to produce 200 gpm from Cactus East and 350 gpm from Parks/Salyer. The water sources will either supply directly to a distribution manifold or discharge to the existing onsite Plant Water Pond (PWP) depending on location, application, and demand. These sources are summarized in Table 18-7.

**Table 18-7: Fresh Water Sources, Available and Planned**

| Source | Provider | Flow (maximum gpm) |
|---|---|---|
| 16-inch main | Existing | 600 |
| Production shaft | Existing | 600 |
| Cactus East dewatering | New well construction | 200 |
| Parks/Salyer dewatering | New well construction | 350 |
| WWTF | City of Casa Grande | 3,472 |
| **TOTAL AVAILABLE** | | **5,222** |

Capacities for the wells were assessed based on the hydrologic model results presented in Section 16.3. The quantity presented for the WWTF exceeds that required for a balance of flow. Facility distribution will be accomplished through a piping network that will be connected to the PWP. Table 18-8 defines water requirements for all users at the mine; infrastructure, mining and process.

**Table 18-8: Cactus Mine Fresh Water Users**

| User | Source | Demand (maximum gpm) | Comment |
|---|---|---|---|
| SX Plant | Manifold | 100 | |
| Parks/Salyer Underground | Manifold | 450 | |
| Truck Wash | Manifold | 5 | |

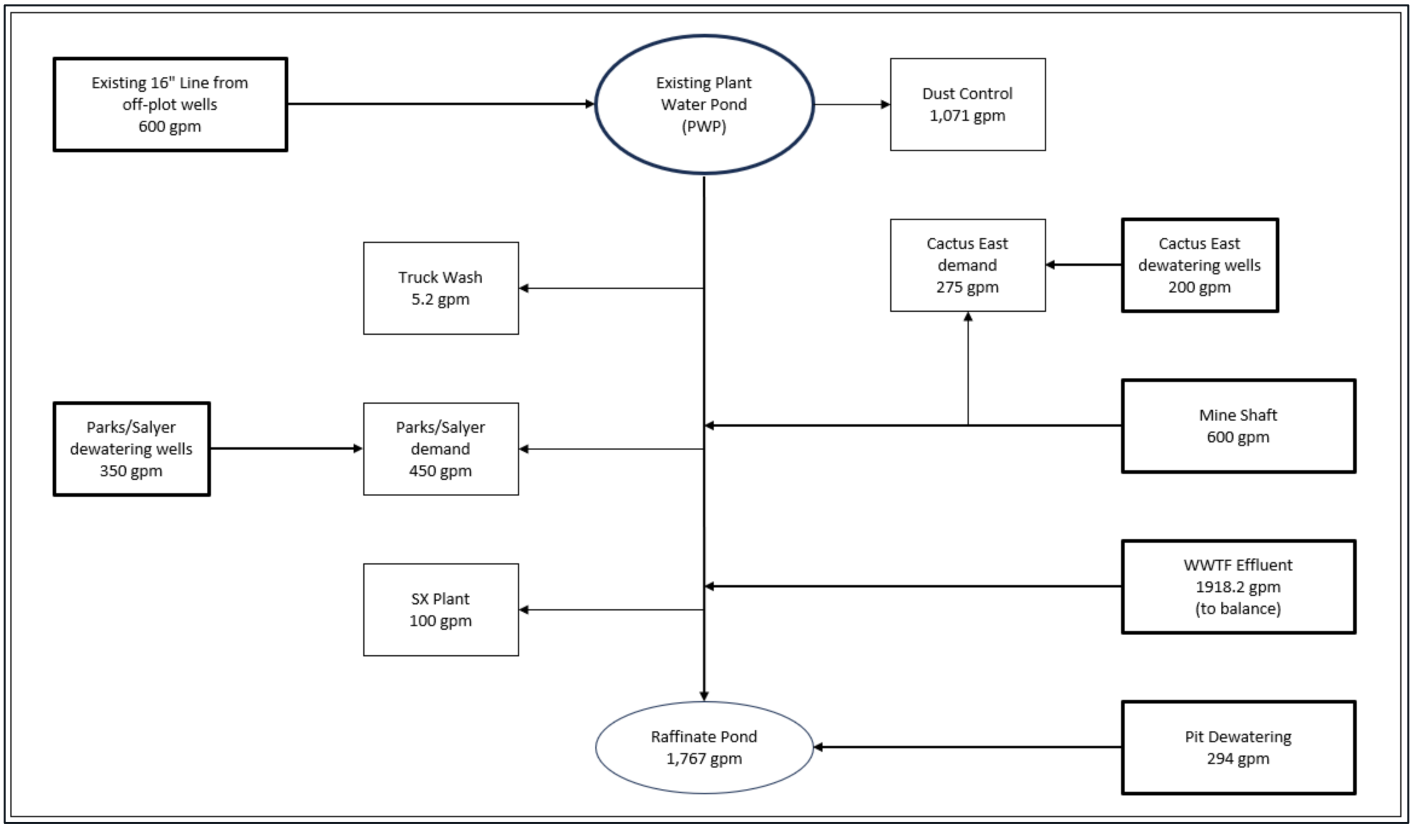
| User | Source | Demand (maximum gpm) | Comment |
|---|---|---|---|
| Truck Water Fill (dust control) | PWP | 1,071 | Fed by a dedicated pump at the PMP |
| Cactus East Underground | Production Shaft | 275 | Demand is not immediate |
| Leach Pad | Manifold and pit water | 1,767 | Maximum occurs in year 18 |
| **TOTAL DEMAND** | | **3,668** | |

Most of the source requirements were specified. The Truck Water Fill station used for dust control operations will be supplied directly from the PWP due to the high flow rates required. The amount of watering needed for this application required a calculation, which first obtained the surface area of all haul and conveyor access roads. A wetting factor of one quart of water per square foot was subsequently applied, and watering was assumed to take place twice per day. The average flow rate determined for adequate dust control of the haul roads was approximately 1,005 gpm, and that for the conveyor access roads was 65 gpm. These numbers may be reduced with gravel and surfactants, and review of historic rainfall.

An additional water source deeper within the mine, specifically the Sacaton Pit, was analyzed and deemed unsuitable for utility use due to its copper content and acidic pH. This water, however, is suitable for leaching applications and will be provided directly to the raffinate pond. The maximum expected demand for the leach pad is 1,767 gpm, which occurs in year 18. This requirement will be met by providing plant water, supplemented with water from the pit.

A block flow diagram of the water balance follows in Figure 18-5. Sources are represented with thick lines. Pond retention was not considered for purposes of the balance.

**Figure 18-5: Block Diagram, Plant Water Balance**




Source: Ausenco, 2023.

### 18.9.3 Water System Hydraulic Design

The following basis was used the design of the potable and plant water systems. Table 18-9 presents the water properties used. An ambient average temperature of 77°F was assumed given the climate of the region.

Table 18-10 presents pipeline properties of the materials used, including roughness and selected pipe grade and compound. Table 18-11 presents design and safety factors used in modeling.

**Table 18-9: Water Properties**

| Water Properties[1] | Value |
|---|---|
| Temperature (°F) | 77 |
| Specific gravity | 0.997 |
| Viscosity (cP) | 0.890 |

[1]www.engineeringtoolbox.com

**Table 18-10: Pipeline Properties**

| Pipeline Properties[1] | Value |
|---|---|
| Roughness, epoxy-lined Carbon Steel (in) | 0.000139 |
| Roughness, HDPE (in) | 0.00084 |
| Roughness, mortar-lined DIP (in) | 0.000139 |
| API 5LX | B |
| HDPE compound | PE4710 |

[1]https://www.engineersedge.com/fluid_flow/pipe-roughness.htm

**Table 18-11: Design and Safety Factors**

| Pipeline Design and Safety Factors | Value |
|---|---|
| Carbon Steel Design Factor[1] | 0.80 |
| HDPE Design Factor[2] | 0.96 |
| Hydraulic Safety Factor | 1.1 |
| Length Safety Factor | 1.05 |
| Pump Efficiency | 75% |

[1]Pipeline Transportation Systems for Liquids and Slurries, B31.4, 2022
[2]Handbook of Polyethylene Pipe, Second Edition, Plastics Pipe Institute, CLVR Company, 2012.

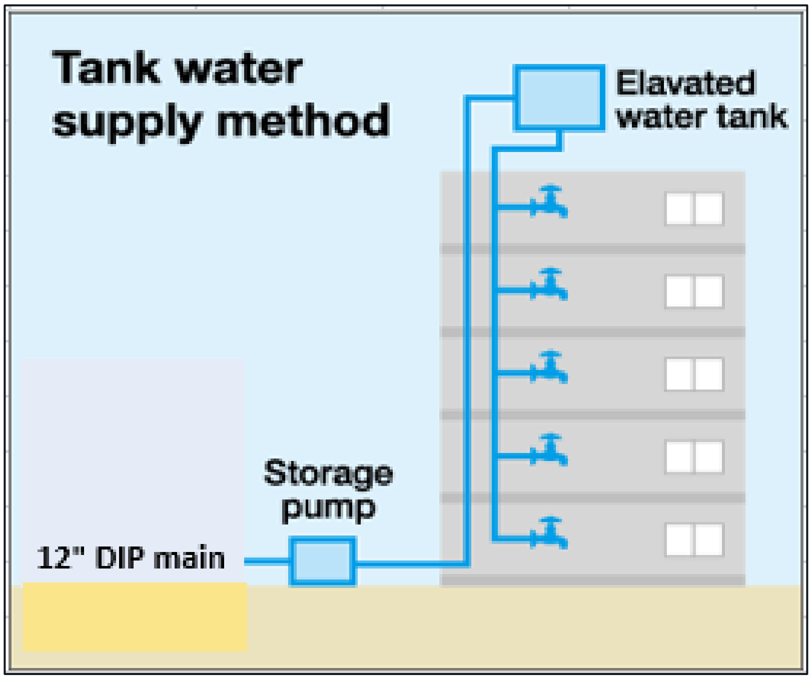
The HDPE design factor accounts for temperature derating, which was obtained from interpolation of available data. The hydraulic safety factor provides a margin of error for hydraulic calculations, and the length safety factor accounts for uncertainty in length measurements, both based on Ausenco engineering experience. A 75% efficiency was assumed for all centrifugal pumps.

Google Earth was used to obtain elevations and distances of all relevant project locations the Bernoulli Equation was used to calculate the pressure drop between source and user locations. Friction losses were calculated using the Darcy-Weisbach equations where the Reynolds number was confirmed to be within the turbulent range, validating the analysis.

#### 18.9.3.1 Potable Water

The system will receive potable water supplied by Arizona Water Co. from a 12-inch cement-lined DIP header. This pipe size and material must be used to connect the header with the water meter at the property limit. The pressure at the planned header tie-in was given as 70 psig when installed in 2006. This is expected to be sufficient for service at the guardhouse, which requires a minimum of 25 psig. At the property limit, a 4-inch HDPE DR 13.5 header approximately 11,500 ft in length will connect the 12-inch DIP pipe with an atmospheric water tank located on the existing tailings impoundment. The tank will be cylindrical and manufactured from fiberglass with a height of 18 ft and a diameter of 10 ft. This size will provide 24 hours of retention under daily usage. A discharge nominal diameter of 6” was selected to meet the minimum drainpipe size requirement. A schematic of the water supply method is presented in Figure 18-6.

**Figure 18-6: Potable Water Supply Method**




Source: Internet, 2023.

A DR of 13.5 was selected to protect pipe integrity from a vacuum. The header has the following take-offs to five destinations (users):

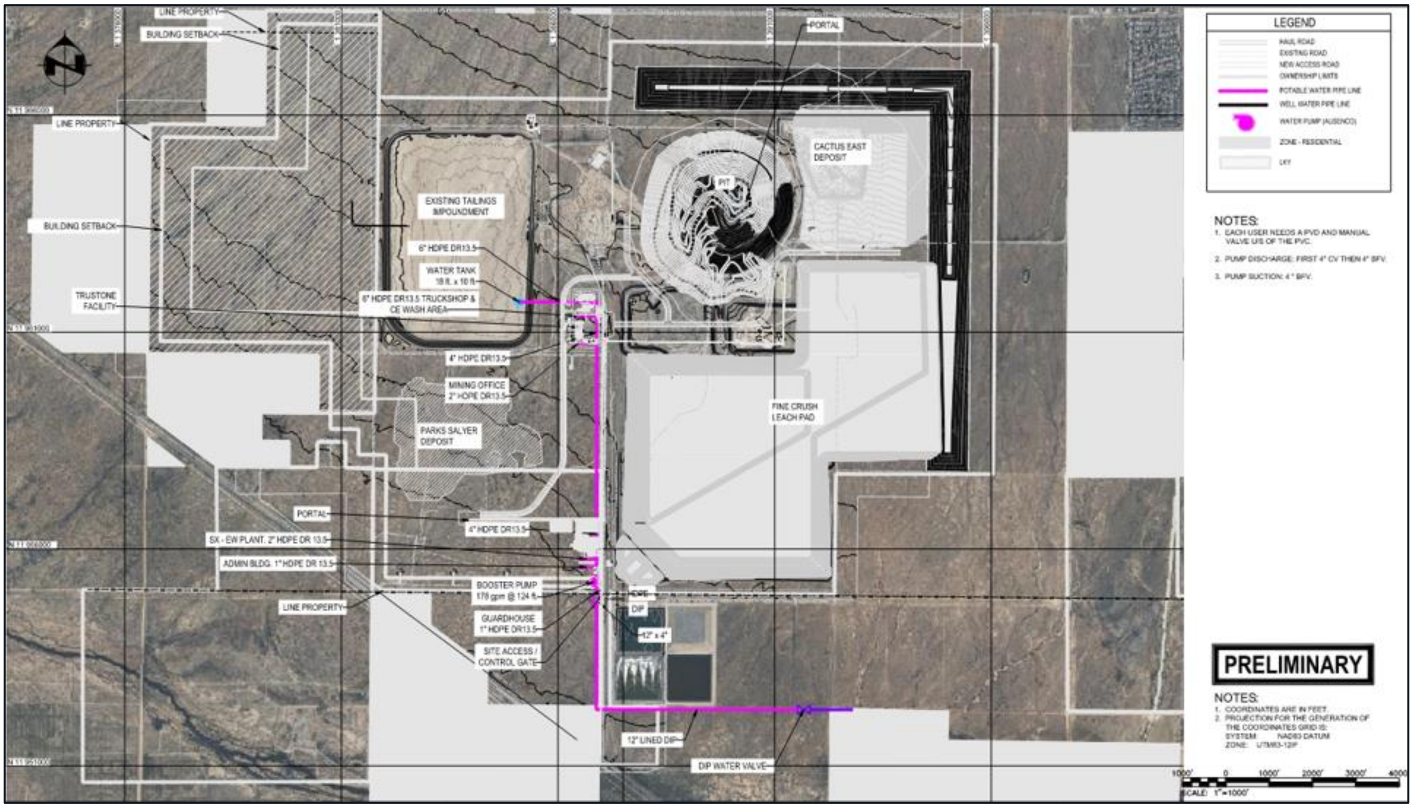
- Guardhouse, 1-inch HDPE DR 13.5
- Administration Building, 1-inch HDPE DR 13.5
- Rest area for Parks/Salyer mining staff in the vicinity of the SX Plant, 2-in HDPE DR 13.5
- Mining Office with Geology lab, 2-in HDPE DR 13.5
- Rest area for Cactus East mining staff in the vicinity of the Truckshop area, 4-in HDPE DR 13.5

A booster pump will be located near the Guardhouse. This pump will provide additional pressure to the system (upstream of the guardhouse) when city water pressure is low. The booster provides 178 gpm at 124 ft of head and will fill the tank while delivering flow to the users; once the tank is filled, the pump will shut off and the tank will supply the system. When the tank is emptied by one-third, or approximately 8 hours of average daily use, the pump will switch on and the cycle repeated. A liquid height of 12 ft in the tank is necessary to be maintained to ensure all users receive the required minimum pressure.

Although the Mining Office and Administration Building have comparable requirements, the Mining Office requires a larger diameter pipe to ensure adequate pressure is received when the tank supplies water.

Figure 18-7 presents a distribution plan of the system, which includes an approximate topographical overview.

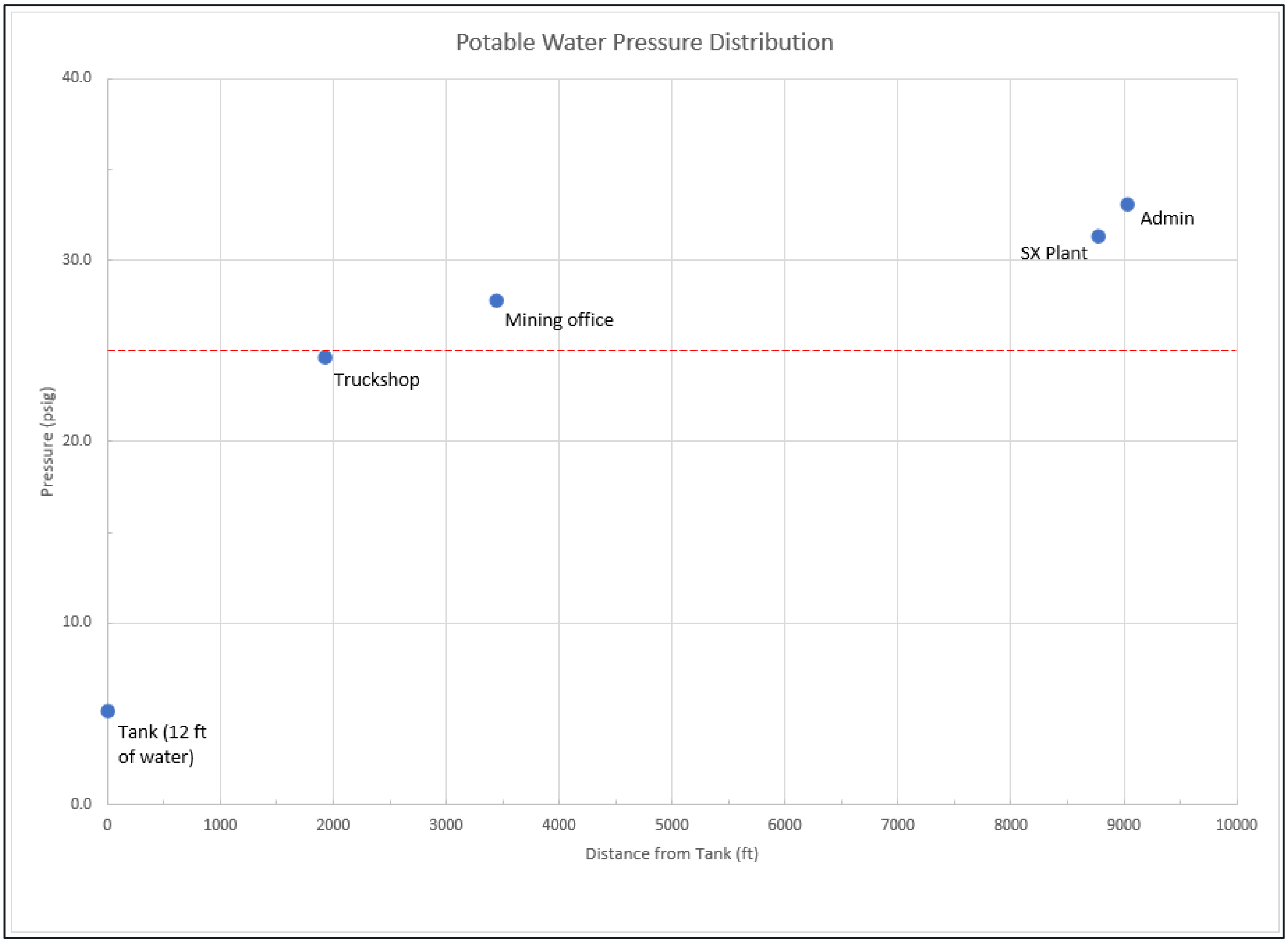
**Figure 18-7: Potable Water System Distribution Plan**




Source: Ausenco, 2024.

Section 18.9.1 presents the minimum pressure requirements for the users. Figure 18-8 displays the pressure distribution from the tank as a function of distance under peak flow conditions.

**Figure 18-8: Pressure Distribution with Distance, Potable Water Tank**




Source: Ausenco, 2024.

The dashed red line indicates a pressure of 25 psig, which is the minimum that must be met. The pressure at the Truckshop is at the low end of the requirement (25 psig with roundoff), but a conservative peak flow rate is assumed, which is not a frequent occurrence. Gravity ensures adequate pressure to the additional users downstream.

#### 18.9.3.2 Plant Water

Plant Water is collected from multiple sources including effluent from a municipal wastewater treatment plant, groundwater from existing and proposed wells, and surface water from the existing Sacaton Pit.

The backbone of the plant water system is a 5.15-mi main piping header connecting, at the north end, to an existing Plant Water Pond (PWP), and at the at the south end to the Casa Grande wastewater treatment facility (WWTF). The

WWTF can provide from one to five million gallons (US) per day (Mgpd). A centrifugal pump that produces 1,500 gpm at 279 ft of head satisfies the current requirement. Additional pumping capacity may be required in the latter years of the project to meet the maximum demand of 1,918 gpm. The header consists of 10-in CS piping that primarily runs along the west end of the Park Salyer Mine and north towards the Cactus Mine and receives water from multiple sources. The header is joined by an additional 4-in line designed to convey up to 350 gpm from Wells 9, 10, and 11. These wells are expected to have a static water level (swl) of 1,500 ft; submersible centrifugal pumps will be positioned 500 ft below the swl, each producing about 2,300 ft of head. 3-in CS piping will be connected to each pump and routed to the 4-in header.

Additional water feed to the system includes water from Well 8, a production shaft, and a 16-in main that supplies water from offsite wells. These sources will feed the PWP directly. Well 8 is expected to have a capacity of 200 gpm, which will be provided by a submersible centrifugal pump placed 500 ft below its swl that develops 2,100 ft of head. 3-in CS piping will connect the pump with the main piping header. The production shaft has a measured swl of 500 ft, a capacity of 600 gpm, and will provide water with a submersible centrifugal pump positioned 500 ft below its swl that develops 1,200 ft of head. The production shaft provides 275 gpm to the Cactus Mine and the balance to the main piping header through separate 6-in and 4-in CS piping.

All pumps were sized to provide atmospheric pressure delivery to the PWP.

Distribution sources from the main header include:

- SX Plant, which will receive 100 gpm from 4-in lined CS piping
- Park Salyer portal, which will receive 450 gpm from 6-in CS piping
- Truck Service Station (washing), which will receive 5.2 gpm from 1-in CS piping
- PWP

Water collected in the PWP will be routed through a 10-in/8-in CS main to the Raffinate Pond as make-up, with an 8-in CS branch provided to the dust control station. A centrifugal pump will supply 2,275 gpm at 90 ft, to trucks for dust control.

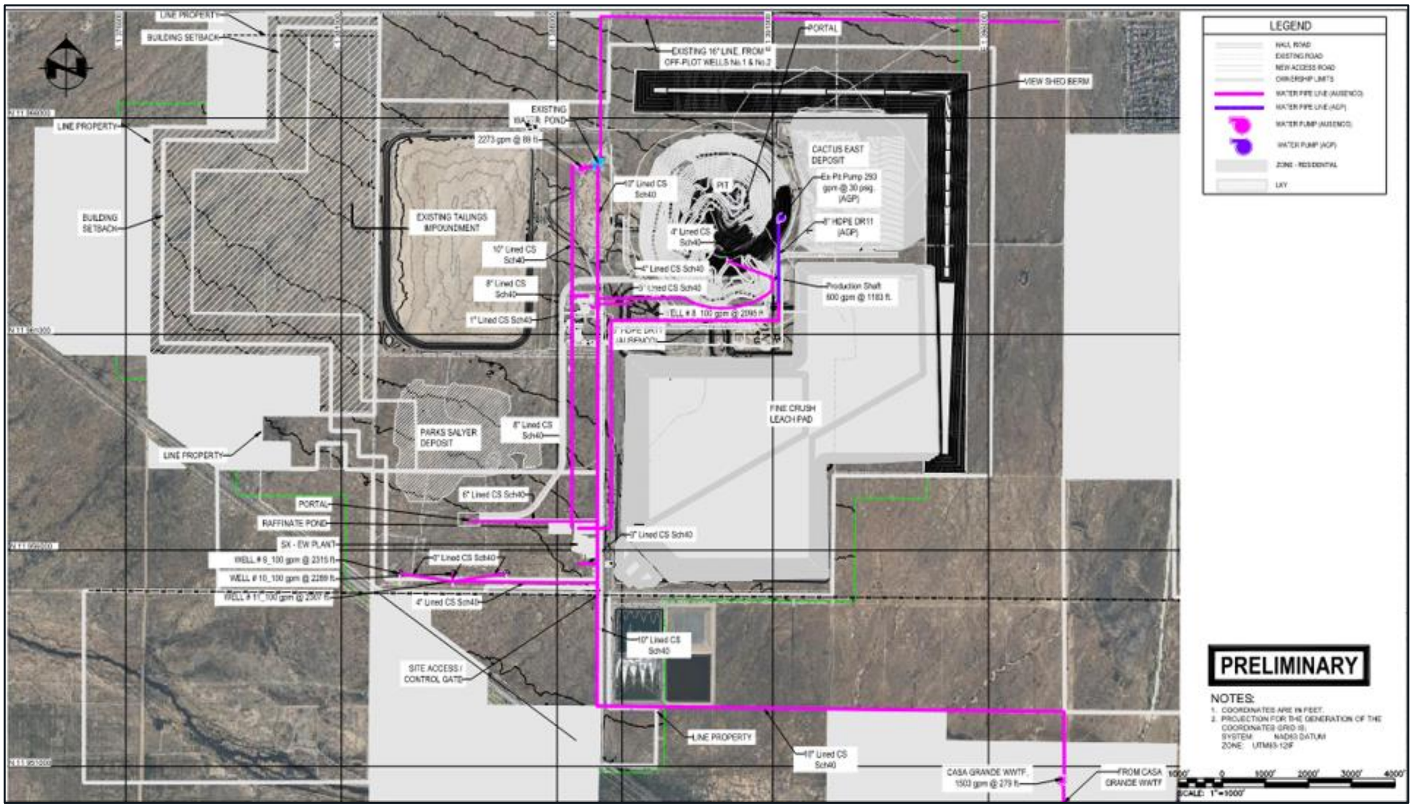
#### 18.9.3.3 Pit Water

The system to transport surface water out of the Sacaton Pit was included in AGP's scope. AGP has provided a pump rated at 293 gpm and 4,918 ft of 8-in HDPE pipe. An additional 4,600 ft of 8-in HDPE piping was required to connect AGP's piping to the final discharge point at the Raffinate Pond. This piping is included in Ausenco's scope.

The design flow from the pit is 293 gpm, which will be dedicated as raffinate make-up water (due to its acid and copper content). The maximum requirement for make-up water is 1,767 gpm. Additional water will be supplied via the plant water system as described is Section 8.9.3.2.

Figure 18-9 presents a distribution plan of the system with an approximate topographical overview .

**Figure 18-9: Plant Water System Distribution Plan**



Source: Ausenco, 2024.Heap Leach Facility

## 18.10 Heap Leach Pad

The HLP will be constructed in four phases, has an approximate final footprint area of 37.9 M $ft^2$ and will support approximately 276 Mton of leach material. It is designed to be operated as a fully drained system with no leachate solution storage within the pad. The leach pad has a composite liner system to prevent seepage to the environment. Above the liner is a series of solution collection pipes encapsulated in an overliner to rapidly collect pregnant solution and transport it to the double lined pregnant leach solution (PLS) pond. In addition, there is also a double lined raffinate pond and a single lined event ponds. Crushed ore materials will be stacked in 30 ft lifts to a maximum height of 200 ft with exterior slopes of 2.5:1. The collected pregnant solution will be pumped to the SX/EW circuit.

Phase 1 is located east of the existing stockpile due to the relatively flat area that facilitates construction and allows for mining of the existing stockpile to liberate space for the consecutive phases of construction. Phase 1 will have an area of 11 M $ft^2$. and hold 61.7 Mton of ore. Phase 1 will support two types of material, oxide ore and fine crushed ore. To support early copper production the stacked material on the first cell will be oxide from the existing stockpile and it will hold approximately 2.67 Mton stacked on a single 20 ft lift with 30 ft lifts after that. Construction of the first phase will start in year -2 with a total time of 3.2 years to reach its capacity.

Consecutive phases will be constructed west of Phase 1 on top of the existing stockpile area. The following phases (2, 3 and 4) will be constructed as each phase is stacked to its capacity and mining of the stockpile liberates the required area. Table 18-12 shows the capacities and stacking times of each phase. Figure 18-10 shows the location of each of the phases of the HLP and Ponds.

The anticipated ore production will be approximately 58,000 ton/d for the first seven years and reduced to 22,500 ton/d after that for the LOM (20 years). The pad will be loaded with conveyor belts coming in from the west along the northern side of the pad to discharge to the eastern area of the pad (Phase 1). First cell of phase 1 will be stacked as a single 20 ft lift and will have a leach cycle of 90 days. The following lifts will be 30 ft and require 180-day leach cycles.

**Table 18-12: HLP Capacity by Phase**

| HLP Phase | Capacity (tons) | Avg. Throughput (Mt/y) | Stacking time (y) |
|---|---|---|---|
| Phase 1 | 61,778,987 | 19,114,953 | 3.23 |
| Phase 2 | 45,053,606 | 25,678,040 | 1.75 |
| Phase 3 | 100,726,363 | 12,982,660 | 7.75 |
| Phase 4 | 68,717,724 | 8,747,688 | 7.86 |
| **Total** | **276,285,915** | | **20.59** |

**Figure 18-10: Heap Leach Facility Phasing**


Source: Ausenco, 2024

## 18.10.1 Leach Pad Liner System

The liner system involves placing a 1 ft (minimum) thick low permeability soil layer followed by a 60 mils thick smooth Low Linear Density Polyethylene (LLDPE) geomembrane that will be deployed on top of this soil liner. It shall then be covered with 2 ft thick overliner layer (select ore).

## 18.10.2 Low Permeability Soil Layer

Soil liner material (low permeability soil that consists of clayey soils such as clay, clayey sand, and clayey gravel) shall be conditioned to adequate moisture and compacted according to the requirements indicated in the Technical

Specifications. The upper 4 in of this soil liner layer shall be free of angular gravel greater than 1 in that may damage the geomembrane during installation or ore stacking and leaching operations.

According to the quantity estimation, volume of low permeability soil material necessary is approximately 1,359,220 $yd^3$. It is indicated by ASCU that construction materials for the low permeability soil layer will be sourced from the existing stockpiles. However, if the moisture condition is adequate to utilize the materials coming from closer sources, they shall be evaluated to verify if the requirements indicated in the technical specifications are attained.

#### 18.10.3 60 MIL LLDPE Smooth Geomembrane Liner

For the Fine Crushed leach pad liner system, a smooth low linear density polyethylene (LLDPE) geomembrane of 60 mils in thickness will be utilized. This type of geomembrane has been chosen due to its flexibility and puncture resistance against the load (or weight) of the ore.

Contractor shall provide temporary and permanent anchorage of outer edges of the geomembrane. Temporary anchorage may consist of sandbags or other ballast material which are necessary for the liner materials to avoid significant displacements and uplift due to high winds during deployment and welding activities.

Permanent anchorage will consist of placing the outer edges of the geomembrane in anchor trenches backfilled and compacted with the spoils from the trench excavation. Geomembrane liner along the view shed berm slope shall be placed and properly anchored as the berm is constructed. The view shed berm shall be constructed in 30 ft lifts and shall maintain at least 10 ft height above the stacked ore.

#### 18.10.4 Overliner

A 2 ft minimum thick overliner layer shall be placed over the geomembrane to protect geomembrane liner and solution collection pipes from possible damage caused by transport and ore spread system on the pads. The overliner also serves the purpose of facilitating solution collection by acting as a drainage element.

Overliner materials shall consist of selected and durable granular ore with relatively high permeability coefficients, which shall be placed around the collection system to protect pipes and geomembrane liner. Origin of this material shall be delimited to the existing stockpile area and shall have the CQA Engineer approval during construction.

#### 18.10.5 Solution Collection System

The purpose of the solution collection system, which will be installed in the heap leach pad to provide a relatively quick evacuation of leach solution and storm water that reaches the liner system. The pipe network has been designed to minimize the solution height over the liner system, as well as to facilitate and accelerate solution collection.

Leach pad solution collection system has been configured to independently collect flows coming from each phase leach process. Defined slopes in the grading plan direct the solution by gravity to the PLS pond located at the south-west extreme.

The solution collection system consists of perforated dual wall header pipes (variable diameter as function of the estimated solution flows) and perforated dual wall lateral pipes of 4 in diameter extending out from the header pipes in a “herring bone” configuration. At the edge of each phase the solution collection headers will transition to 18-inch non-perforated dual wall pipes with watertight bell and spigot connections. These headers will be installed in a geomembrane lined ditch (secondary containment) and convey pregnant solution along the perimeter of the facility conveying solution flows to the PLS and Event ponds.

## 18.11 Ponds

Raffinate, PLS, and Event ponds have been designed to handle the solution and stormwater involved in the leaching cycle. PLS and raffinate ponds are designed to provide storage for solution to be pumped to and from the SW/EW plant. The PLS pond is situated immediately down-gradient in the southwest corner of the HLP and solution is conveyed to this pond through the collection system pipes by gravity. During major storm events, excess solution will be diverted to the Event ponds via spillways. A starter Event Pond will be built during the first phase of the project, and it will have capacity to hold excess solution from Phases 1, 2, and 3. A second event pond will be constructed when Phase 4 comes online. The following design criteria has been considered for the process ponds:

- Process ponds (PLS and Raffinate) are designed to contain up to 24 hours of solution at the maximum heap irrigation rate.
- 2 ft freeboard.

The liner system for the PLS and raffinate ponds will include a leak collection and recovery system (LCRS) and consists of the following components (from top to bottom):

- 60 MIL HDPE Smooth Geomembrane Liner.
- Geonet (including a sump, riser, and pump located in a corner of these ponds).
- 60 MIL HDPE Smooth Geomembrane Liner.
- Geosynthetic Clay Liner.
- Prepared Subgrade.

The Event Ponds liner system will be a single liner system consisting of the following layers (from top to bottom):

- 60 MIL HDPE Smooth Geomembrane Liner.
- Geosynthetic Clay Liner.
- Prepared Subgrade.

## 18.12 Geotechnical Parameters

### 18.12.1 Alluvial, Conglomerate and Ore

Two geotechnical investigations were performed to determine the physical and mechanical properties of the alluvial and conglomerate subgrade below the HLP. In addition, the geotechnical investigations also looked at the physical and mechanical properties of the historical ore stockpile. The results of the investigations are shown in Table 18-13.

**Table 18-13: Geotechnical Parameters**

| Description | Wet Density (lbs/ft$^3$) | Sat Density (lbs/ft$^3$) | % Gravel | % Sand | % Fines | C' (psf) | Friction Angle Φ |
|---|---|---|---|---|---|---|---|
| Ore | 114 | 114 | 42 | 51 | 7 | 0 | 36 |
| Alluvial | 121 | 124 | 1 | 61.5 | 37.5 | 0 | 31 |
| Conglomerate | 127 | 131 | | | | 2506 | 34 |

## 18.13 Stability Analysis

Two sections were identified as critical sections for slope stability analysis. These sections cover the extents of the HLP. Section 1-1 runs north to south along phase 1 of the pad and Section 2-2' runs west to east through phases 4,2, and 1 of the pad. Analyses were undertaken for both static and pseudo-static (earthquake loading), and post-earthquake conditions with the calculated factors of safety (FOS) higher than the minimum required values per the BADCT of 1.3 FOS for static, 1.0 FOS for pseudo-static, deformations not affecting geomembranes shall be less than or equal to 1 ft, and deformations affecting geomembranes shall be less than or equal to 6 inches.

# 19 MARKET STUDIES AND CONTRACTS

## 19.1 Market Studies

No market studies or product valuations were completed as part of the 2024 PFS. Market price assumptions were based on a review of public information, industry consensus, standard practices and specific information from comparable operations in the region.

Copper cathodes are widely traded and can be marketed domestically with significant optionality regarding the ultimate customer base. It is assumed that Cactus will produce a LME deliverable copper cathode quality.

## 19.2 Commodity Price Projections

Project economics were estimated based on long-term flat metal prices of US$3.90/lb Cu. This copper price is in accordance with consensus market forecasts from various financial institutions and are consistent with historic prices, shown in Table 19-1, sourced from Capital IQ on September 25, 2023. The QP also considers the prices used in this study to be consistent with the range of prices being used for other project studies.

**Table 19-1: Summary of Historic Commodity Pricing (Sept 25, 2023)**

| Metal | 1-Year Average | 2-Year Average | 3-Year Average |
|---|---|---|---|
| Copper (US$/lb) | 3.81 | 3.88 | 4.03 |

## 19.3 Contracts

No contracts for transportation or off-take of the copper cathode are currently in place, but if they are negotiated, they are expected to be within the industry norms. Similarly, there are no contracts currently in place for the supply of reagents, utilities, or other bulk commodities required to construct and operate the Project.

# 20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

## 20.1 Environmental Considerations

In 2009, approximately 15 years after the Cactus Mine ceased operation, the mine was conveyed to the ASARCO Multi-State Environmental Custodial Trust (the Trust) as part of ASARCO bankruptcy proceedings. The Trust entered the property into the Voluntary Remediation Program (VRP) with Arizona Department of Environmental Quality in 2010. In the following years, structures were demolished and reclaimed, and characterization studies were conducted. Environmental studies were conducted as part of the VRP program and included sampling and testing of groundwater quality and pit lake water quality, and whether the pit lake is a terminal sink (Tetra Tech, 2017a, 2017b, 2018a, 2019a, 2019b, 2019c).

Limited historic analytical data was included in reports reviewed by Tetra Tech (2017a). Based on narrative information, sampling and analysis of groundwater at and near the Sacaton Mine began prior to the mine's operations in the early 1970's. A reference to historic groundwater quality conditions suggests that the water quality in the tailings pond was better than the underlying groundwater system and that discharges from the site were "freshening" the groundwater system (Montgomery & Associates, 1986). The most complete set of analytical data located is found in a November 1986 preliminary report titled "Hydrogeologic Conditions, Asarco Sacaton Open-Pit Mine, Pinal County, Arizona," prepared by Errol L. Montgomery & Associates (Montgomery & Associates, 1986). Montgomery & Associates also conducted field investigations in 2013 on behalf of Russell Mining Corporation as part of a due diligence study and reported results in an Interim Data Report (Montgomery & Associates, 2013).

The Tetra Tech investigations of groundwater and soil chemical quality resulted in the following conclusions:

- The initial Hydrogeology Investigation prepared by TetraTech (2017b) for the Trust dated December 21, 2017, demonstrated that the open pit is a hydraulic sink and does not, therefore, contribute to groundwater chemical degradation.
- A comprehensive facility inspection by TetraTech (2018a) was submitted to ADEQ Voluntary Remediation Program (VRP) Project Manager John Patricki on July 15, 2018. This report identified eight areas of stained soil for further investigation. This inspection was conducted in accordance with Part 4.3 of the Arizona Pollutant Discharge Elimination System General Permit for Stormwater Areas previously identified as potential source areas in the work plan, and areas of stained soil were inspected. Discharges were found to be contained within the site.
- A comprehensive facility inspection of the TruStone facility was carried out by TetraTech (2018b). A report was submitted to ADEQ VRP Project manager John Patricki on July 15, 2018. This inspection was conducted in accordance with Part 4.3 of the Arizona Pollutant Discharge Elimination System General Permit for Stormwater Discharges from non-mining industrial facilities. No signs of discharge from the sediment basins were observed, nor was evidence of discharges to any drainages or washes.
- Tetra Tech (2019a) prepared a Demolition Completion report for the Trust dated March 11, 2019. This report documented removal of buildings and other structures that posed health and safety risks. Asbestos containing materials, electrical components containing PCBs, and lead-based paint were investigated and addressed.

- A Site Improvement Plan (SIP) was prepared by TetraTech (2019b) for the Trust and dated March 11, 2019. The objectives of the SIP were to (1) mitigate potential human/ecological health hazards; (2) mitigate offsite transport of tailings/waste rock sediments and wind-blown dust; and (3) stabilize the TSF, WRD and the underground mine workings area. The SIP states that "TruStone facility was not considered in the creation of the SIP. Any future activities proposed for the TruStone facility will be addressed under a separate scope of work."
- In 2019-2020, nine monitoring wells were installed at the site and one older monitoring well (TM-1) was rehabilitated to characterize hydrogeologic conditions and to establish baseline conditions for an Aquifer Protection permit for future mining operations. Clear Creek was provided with a table summarizing the groundwater quality and well completion logs.
- TetraTech prepared the Sacaton Mine Site Construction Completion report (dated January 27, 2020, and an addendum dated February 14, 2020). The report documented the environmental issues that were remediated and addressed at the site. It also documents the baseline groundwater quality from the monitor wells installed at the mine site. ADEQ approved these reports (ADEQ, 2020). Based on the information reported, ADEQ agreed not to sue the Trust under Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA), Resource Conservation and Recovery Act (RCRA) or Water Quality Assurance Revolving Fund (WQARF) based on "any claim or cause of action arising out of the ownership or performance of the remedial activities at the Sacaton Mine Site."

Based on the results of the characterization studies and reclamation work, in August 2019, Elim entered into a Prospective Purchaser Agreement (PPA) with ADEQ. The PPA, which ADEQ issued because of the substantial public benefit to the remedial work conducted at the site, released Elim from potential liabilities related to existing, known contamination under CERCLA, WQARF, and RCRA. The PPA does not cover unidentified environmental conditions or contamination.

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

## 20.2 Permitting Considerations

The Project includes exploration and mining on private land and on five Arizona State Land Department (ASLD) leases. There is no federal nexus for permitting of the Project.

The primary permit with the longest permitting timeframe is anticipated to be the Aquifer Protection Permit Amendment (APP). ASCU currently has an APP (no. P-513324) for the following facilities: oxide leach pad, enriched leach pad, oxide PLS pond, enriched PLS pond, raffinate pond, oxide events pond, enriched events pond, site runoff pond 1, site runoff pond 2, and the waste rock stockpile runoff pond. ASCU will apply for amendments to the APP for additional discharging facilities, as needed. An APP Significant Amendment (without a public hearing) has a licensing timeframe of 221 business days.

ASCU has agreed to post a bond for the APP of $1,144,576.00 prior to beginning project construction.

Other permits/authorizations/notifications for which ASCU has applied for or currently holds include:

- Mineral Leases on Arizona State Land: ASCU has five mineral leases on State land.

- Permit for exploration on Arizona State Land: ASCU currently has five prospecting Permits with the ASLD for exploration operations on State land.
- Dust Permit: ASCU has been issued dust permit # DUSTW-24-0029 by Pinal County.
- Industrial Permit from Pinal County: ASCU has been issued the Industrial Air Permit on May 12, 2023, Permit #C31407.000 This applies to any industrial operation that has the potential to emit 5.5 pounds per day or 1 ton per year of any regulated air pollutant. Generators, stationary fuel burning equipment, and petroleum storage tanks are regulated under Industrial Permits.
- Mined Land Reclamation plan (MLRP). An MLRP is required for surface disturbances on private land greater than 5 acres. Financial assurance requirements such as bonding apply. An MLRP was issued on March 27, 2023, by the Arizona State Mine Inspector. The bond has been submitted and the bond amount was $4,797,829. ASCU plans to update the MLRP as necessary to reflect changes to the mine plan.
- Type 2 Grandfathered Water Right: ASCU currently has a right for 136 afy.
- Groundwater Withdrawal Permit: ASCU currently has Permit 59-233782.0000 for 3,600 afy.
- Special Land Use Permit for use of State Surface to construct facilities for mining operations has been granted by ASLD (permit no. 23-123266-03).
- Stormwater Pollution Prevention Plan (SWPPP) – LTF/ID 95924.
- Permits/ authorizations that ASCU may need to apply for include the following:
- Notice of Intent to Clear Land: ASCU will notify the Arizona Department of Agriculture regarding potential destruction of protected native plants.
- Notice of Startup: ASCU will submit a notice to the Arizona State Mine Inspector prior to commencing mining. ASMI issues permits for underground diesel equipment, inspects and permits elevators, enforces fuel storage rules.
- EPA Hazardous Waste Generator: ASCU will apply for an EPA ID number as required by RCRA when needed.
- Cultural Resources: ASCU must notify Arizona State Museum of cultural artifacts are found on private property. Arizona State Land Department will consult with the State Historic Preservation officer regarding potential impacts to resources on State Land. ASLD has provided a “letter of no survey” for Section 34, Township 5 South, Range 5 East.
- ADWR requires that industrial facilities including mines submit a Conservation Plan for water use if the water demand is greater than 500 afy. ASCU has prepared a plan and will submit it to ADWR when and if water use reaches the 500 afy threshold.
- ADWR also requires that a Notice of Intent to Drill and Abandon an Exploration/Specialty Well Permit be obtained by an Arizona Licenced Well Driller for any drilling deeper than 100 ft on private land.
- ASCU does not anticipate having to apply for an Arizona Pollution Discharge Elimination System (AZPDES) permit. The US Army Corps of Engineers conducted a Jurisdictional Determination and found that there are no “waters of the US” (WOTUS) on the project site.

Permitted Facilities and Best Available Demonstrated Control Technologies (BADCT) as per the Aquifer Protection Permit Amendment (APP). ASCU currently has an APP (no. P-513324)

A list of the permitted facilities and their coordinate locations included in the APP for the site are listed in Table 20-1.

**Table 20-1: Discharging Facilities**

| Facility | Latitude | Longitude |
|---|---|---|
| **New Facilities** | | |
| Oxide Leach Pad | 32° 56' 54.81" N | 111° 49' 1.103" W |
| Enriched Leach Pad | 32° 57' 1.357" N | 111° 49' 38.80" W |
| Oxide PLS Pond | 32° 56' 12.24" N | 111° 49' 30.16" W |
| Enriched PLS Pond | 32° 56' 54.81" N | 111° 49' 38.66" W |
| Raffinate Pond | 32° 56' 52.36" N | 111° 49' 40.87" W |
| Oxide Events Pond | 32° 56' 7.228" N | 111° 49' 29.83" W |
| Enriched Events Pond | 32° 56' 54.35" N | 111° 49' 42.22" W |
| Site Runoff Pond 1 | 32° 56' 41.03" N | 111° 49' 30.13" W |
| Site Runoff Pond 2 | 32° 57' 0.1398" N | 111° 48' 32.17" W |
| Waste Rock Stockpile Runoff Pond | 32° 56' 18.76" N | 111° 48' 28.30" W |

Descriptions of the BADCT planned for each facility described in the APP follows:

### 20.2.1 Enriched Leach Pad – Prescriptive BADCT

The Enriched Leach Pad will be situated between the open pit and the tailings facility with the Phase 1 pad located at the south end of the ultimate footprint. The pad is slated to be constructed once the pit expansion has exposed the enriched ore body approximately two years after the start of the Oxide pad operations. The Enriched Pad will receive ore from the pit as it is expanded, and the enriched ore is exposed. The pad will cover an area of 2.2 million square feet and extend to a maximum initial elevation of 1,518 feet for Phase 1. The ore will be stacked typically in 20-foot lifts. Inter-lift setbacks of approximately 40 feet width and an angle of repose of the ore material of 1.5H:1V will result in an overall slope of 3.5H:1V.

The leach pad construction shall consist of a compacted subgrade overlain by a minimum 12 inches of soil liner that is compacted to achieve a maximum permeability of 10-6 cm/sec and a 80-mil textured LLDPE primary geomembrane. The pad will be divided into four cells. The cells will be separated by divider berms running from north to south. The divider berms will be slightly lower than the perimeter berm. The cells will slope from north to south at an average grade of 0.5 to 1 percent at the southern toe of the pad. Each cell will have a gravity flow solution collection system installed. The solution collection system will consist of a network of pipes within a layer of drainage gravel. The pipes will be arranged in a herringbone pattern with smaller lateral pipes connecting to larger collector pipes located on the upgradient side of the divider berm. The system utilizes perforated and corrugated polyethylene (CPE) collector and lateral pipes for solution collection. All CPE pipes will be of double-walled N12 construction by ADS or equal. The drainage layer will consist of a minimum 18 inches of free draining 3/4-inch minus crushed rock. The rock must be chemically stable under acidic conditions. The Oxide pad collector pipes are sized at 14-inches in diameter and the lateral pipes feeding the collectors are 4-inch diameter. The 4inch diameter laterals will be placed on nominal 30-foot centers to keep the maximum and average hydrostatic heads on the liner to less than 5 feet and 2 feet. The collector pipes for both leach pads will transition to solid pipe at the perimeter berm and routed in an HDPE lined channel to the PLS Pond. In the event that solution flows exceed the capacity of the collector pipes, solution head will build up inside the leach pad and overtop a spillway in the perimeter berm and flow into the HDPE lined channel. The channel will discharge into the Stormwater Pond adjacent to the PLS Pond thereby minimizing dilution of the PLS.

### 20.2.2 Oxide Leach Pad – Prescriptive BADCT

The Phase 1 Oxide Leach Pad will be located at the north end of the existing WRS, on an area currently occupied by waste rock and alluvial soil stockpiles. The alluvial soil stockpiles and waste rock on the Phase 1 Oxide Pad footprint will be removed for its construction. The waste rock reclaim will proceed south and as the stockpile material is placed on the pad, space will be created for the construction of the next phase. Leach-grade waste rock will be temporarily moved south onto the existing stockpile as needed. The total lined area of the pad will be 10.6 million square feet with a maximum stacked height of 100 feet in Phase 1. The ore will be stacked typically in 20-foot lifts. Inter-lift setbacks of approximately 40 feet width and an angle of repose of the ore material of

1.5H:1V will result in an overall slope of 3.5H:1V. The first lift will be 25 feet high in order to meet the initial quantity requirements. All leach solutions will drain to the PLS Pond at the Oxide pad from where they will initially be pumped to the PLS pond at the SX/EW plant for processing. Once the Enriched Pad comes into operation, the PLS from the Oxide pad will be pumped onto the Enriched pad with the solution draining to the PLS pond at the SX/EW plant. Solutions in the Raffinate Pond will be pumped to the Oxide pad for re-application to the Oxide heap. Storm run-on flows due to the 100-year storm will be intercepted by the leach pad perimeter berm and diverted to a site runoff pond.

The leach pad construction shall consist of a compacted subgrade overlain by a minimum 12 inches of soil liner that is compacted to achieve a maximum permeability of 10-6 cm/sec and a 80-mil textured LLDPE primary geomembrane. The pad will be divided into four cells. The cells will be separated by divider berms running from north to south. The divider berms will be slightly lower than the perimeter berm. The cells will slope from north to south at an average grade of 0.5 to 1 percent at the southern toe of network of pipes within a layer of drainage gravel. The pipes will be arranged in a herringbone pattern with smaller lateral pipes connecting to larger collector pipes located on the upgradient side of the divider berm. The system utilizes perforated and corrugated polyethylene (CPE) collector.

### 20.2.3 Oxide Leach Pad – Prescriptive BADCT

The Phase 1 Oxide Leach Pad will be located at the north end of the existing WRS, on an area currently occupied by waste rock and alluvial soil stockpiles. The alluvial soil stockpiles and waste rock on the Phase 1 Oxide Pad footprint will be removed for its construction. The waste rock reclaim will proceed south and as the stockpile material is placed on the pad, space will be created for the construction of the next phase. Leach-grade waste rock will be temporarily moved south onto the existing stockpile as needed. The total lined area of the pad will be 10.6 million square feet with a maximum stacked height of 100 feet in Phase 1. The ore will be stacked typically in 20-foot lifts. Inter-lift setbacks of approximately 40 feet width and an angle of repose of the ore material of 1.5H:1V will result in an overall slope of

3.5H:1V. The first lift will be 25 feet high in order to meet the initial quantity requirements. All leach solutions will drain to the PLS Pond at the Oxide pad from where they will initially be pumped to the PLS pond at the SX/EW plant for processing. Once the Enriched Pad comes into operation, the PLS from the Oxide pad will be pumped onto the Enriched pad with the solution draining to the PLS pond at the SX/EW plant. Solutions in the Raffinate Pond will be pumped to the Oxide pad for re-application to the Oxide heap. Storm run-on flows due to the 100-year storm will be intercepted by the leach pad perimeter berm and diverted to a site runoff pond.

The leach pad construction shall consist of a compacted subgrade overlain by a minimum 12 inches of soil liner that is compacted to achieve a maximum permeability of 10-6 cm/sec and a 80-mil textured LLDPE primary geomembrane. The pad will be divided into four cells. The cells will be separated by divider berms running from north to south. The divider berms will be slightly lower than the perimeter berm. The cells will slope from north to south at an average grade of 0.5 to 1 percent at the southern toe of network of pipes within a layer of drainage gravel. The pipes will be arranged in a herringbone pattern with smaller lateral pipes connecting to larger collector pipes located on the upgradient side of the divider berm. The system utilizes perforated and corrugated polyethylene (CPE) collector and lateral pipes for solution collection. All CPE pipes will be of double-walled N12 construction by ADS or equal. The drainage layer will consist of a minimum 18 inches of free draining 3/4-inch minus crushed rock. The rock must be chemically stable under acidic conditions. The Oxide pad collector pipes are sized at 14-inches in diameter and the lateral pipes feeding the collectors are 4-inch diameter. The 4inch diameter laterals will be placed on nominal 30-foot centers to keep the maximum and average hydrostatic heads on the liner to less than 5 feet and 2 feet. The collector pipes for both leach pads will transition to solid pipe at the perimeter berm and routed in an HDPE lined channel to the PLS Pond. In the event that solution flows exceed the capacity of the collector pipes, solution head will build up inside the leach pad and overtop a spillway in the perimeter berm and flow into the HDPE lined channel. The channel will discharge into the Stormwater Pond adjacent to the PLS Pond thereby minimizing dilution of the PLS.

### 20.2.4 Raffinate Pond – Prescriptive BADCT

The PLS Pond will contain flows initially from the Oxide Pad and then from the Enriched Leach Pad when it becomes operational. Once the Enriched Pad is operational, the Oxide PLS will be pumped to the Raffinate Pond at the SX/EW Plant and from there to the Enriched Pad ore providing a pseudo-intermediate or counter-current leach cycle. Raffinate solution, partly enriched, will also be pumped to the Oxide Pad to complete the leaching cycle. Both the PLS and the Raffinate ponds are sized according to the criteria for the PLS Pond. The ponds will be double lined with 80-mil HDPE primary liner and 60-mil HDPE secondary liner separated by a layer of geonet for leak detection. Soil liner six (6) inches thick compacted to a minimum permeability of 10-6 cm/sec will underlie the secondary liner.

### 20.2.5 Oxide PLS Pond

The Oxide PLS Pond will be located towards the south end of the final Oxide Pad layout on an area currently occupied by the WRS. The pond will receive process solution from the Oxide Leach Pad. The pond location is governed by the fall required for gravity flow in the solution pipes and stormwater channel for all phases of the Oxide Leach Pad. The pond layout is restricted on the west side by a high voltage powerline. The pond will be formed mostly by excavation and the crest will be above adjacent ground to prevent surface water run-on. The pond will be double lined with 80-mil HDPE primary liner and 60-mil HDPE secondary liner separated by a layer of geonet for leak detection. Soil liner six

(6) inches thick compacted to a minimum permeability of 10-6 cm/sec will underlie the secondary liner. Barge mounted pumps will be used to transfer pregnant solution from the PLS Pond to the Raffinate Pond at the Enriched Pad. The pond will contain PLS flows from the Oxide Leach Pad and direct precipitation from the 100-year, 24-hour storm. The pond shall have a capacity of 419,217 CF (3,135,740 Gallons) with 2 feet of pond storage allocated to freeboard. The pond dimensions are 240 x 170 x 20 feet deep, with side slopes of 3H:1V.

### 20.2.6 Enriched PLS Pond

The PLS Pond will contain flows initially from the Oxide Pad and then from the Enriched Leach Pad when it becomes operational. Once the Enriched Pad is operational, the Oxide PLS will be pumped to the Raffinate Pond at the SX/EW Plant and from there to the Enriched Pad ore providing a pseudo-intermediate or counter-current leach cycle. Raffinate solution, partly enriched, will also be pumped to the Oxide Pad to complete the leaching cycle. Both the PLS and the Raffinate ponds are sized according to the criteria for the PLS Pond. The ponds will be double lined with 80-mil HDPE primary liner and 60-mil HDPE secondary liner separated by a layer of geonet for leak detection. Soil liner six (6) inches thick compacted to a minimum permeability of 10-6 cm/sec will underlie the secondary liner.

### 20.2.7 Event Ponds

#### 20.2.7.1 Oxide Events Pond

The Oxide Events Pond will be located adjacent to the PLS Pond towards the south end of the final Oxide Pad layout on an area currently occupied by the Waste Rock Stockpile. The pond will receive direct precipitation, excess stormwater runoff from the Oxide Leach Pad and process solution overflows from the PLS Pond. Excess stormwater from the leach pad will be conveyed in an HDPE-lined channel to the Events Pond. An HDPE-lined spillway channel connects the pond to the PLS Pond. The pond is restricted on the west side by a high voltage powerline, similar to the PLS Pond. The pond elevation is governed by the fall required for the stormwater channel to gravity flow from all phases of the Oxide Leach Pad. The pond will be formed mainly by excavation and the crest slightly elevated around the perimeter to prevent surface water run-on. The pond will be single lined with 80-mil HDPE liner on a 6-inch layer of 3/8” minus compacted bedding. The pond will contain excess stormwater runoff from the Oxide Leach Pad and direct precipitation due to the 100-year, 24-hour storm. Two (2) feet of pond storage are allocated to freeboard. The pond will be normally empty to maintain maximum available storage volume for runoff and will provide containment of the 100-year, 24-hour storm event from the Oxide Leach Pad. The pond shall have a capacity of 2,651,050 CF (17,181,600 Gallons) with side slopes of 3H:1V.

#### 20.2.7.2 Enriched Events Pond

The Enriched Events Pond will be located adjacent to, and west of the PLS Pond at the southwestern end of the Enriched Pad. The pond will receive direct precipitation, excess stormwater runoff from the Enriched Leach Pad, and any process solution overflows from the PLS Pond. The storm event pond has been sized based off the current estimated full build out of the enriched heap leach facility, an area of 431.5 acres. Excess stormwater from the leach pad will be conveyed in a HDPE lined channel to the Stormwater Pond. An HDPE lined spillway channel connects the PLS Pond to the storm event pond should the PLS and Raffinate Ponds exceed their combined capacities. The pond will be formed in cut with the crest slightly elevated around the perimeter to prevent surface water run-on. The pond will be single lined with 80 mil HDPE liner on a 12-inch layer of 3/8" minus compacted bedding. The storm event pond will contain excess stormwater runoff from the Enriched Leach Pad and direct precipitation due to the 100-year, 24-hour storm. Two feet of pond storage are allocated to freeboard. The pond will be normally empty to maintain maximum available storage volume. The pond has a volume of 1,115,834 CF (8,346,437 Gallons) with side slopes of 3H:1V.

### 20.2.8 Runoff Ponds

#### 20.2.8.1 Site Runoff Pond 1

Site Runoff Pond 1 will be located adjacent to the west side of the Oxide Leach Pad, north of the Oxide Pad PLS Pond. The pond will receive direct precipitation and stormwater runoff from the site to the west and south of the open pit. Most of the drainage area is natural ground but facilities including the SX/EW Plant, storage tanks, and access roads lie within the drainage area and runoff from these is considered non-stormwater. The pond is restricted on the west side by a high voltage powerline as for the PLS Pond. The pond will be formed in cut with the crest slightly elevated around the perimeter to prevent surface water run-on. The pond will be single lined with 80-mil HDPE liner on a 12-inch layer of 3/8" minus compacted bedding. A concrete ramp and sump are included in the pond design for sediment cleanout. The ramp and sump are located on the northeast corner and will also form the entry point for incoming flow. The concrete will be placed on a 6-inch layer of low permeability soil compacted to achieve a maximum permeability of 10-6 cm/sec. The pond will be normally empty to maintain maximum available storage volume for stormwater runoff and will provide containment of the 100-year, 24-hour storm event from the drainage area. The pond has a volume of 2,278,736 CF with side slopes of 3H:1V.

#### 20.2.8.2 Site Runoff Pond 2

Site Runoff Pond 2 will be located north of the Oxide Pad and adjacent to the west side of the new WRS. The pond will receive direct precipitation and stormwater runoff from the area between the open pit and the View Shed Berm. Most of the drainage area is natural ground but facilities including the truck shop, truck wash, and access roads lie within the drainage area and runoff from them is considered non-stormwater. The pond will be formed in cut with the crest slightly elevated around the perimeter to prevent surface water run-on. The pond will be single lined with 80-mil HDPE liner on a 12-inch layer of 3/8" minus compacted bedding. A concrete ramp and sump are included in the pond design for sediment cleanout. The ramp and sump are located on the northeast corner and will also form the entry point for incoming flow. The concrete will be placed on a 6" layer of low permeability soil compacted to achieve a maximum permeability of 10-6 cm/sec. The pond will be normally empty to maintain maximum available storage volume for stormwater runoff and will provide containment of the 100-year, 24- hour storm event from the drainage area. The pond has a volume of 1,743,862 CF with side slopes of 3H:1V.

#### 20.2.8.3 Waste Rock Stockpile Runoff Pond

The Phase 1 Waste Rock Stockpile (WRS) will be comprised mostly of alluvial soil from existing stockpiles and the expansion of the pit. It is intended to cover the surface of the WRS with alluvial soil a minimum of 2 feet thick to provide an evapo-transpirative capping. Runoff from the side slopes of the first lift will be contained in the non-stormwater pond south of the WRS. The Runoff Pond will be located south of the WRS. The pond will receive direct precipitation and stormwater runoff from the slope of the first WRS lift and adjacent ground. The drainage area consists of the side slope of the first WRS lift and adjacent ground. Runoff from the area is considered contact stormwater. The pond will be formed in cut with the crest slightly elevated around the perimeter to prevent uncontrolled surface water run-on. The pond will be single lined with 80-mil HDPE liner on a 6-inch layer of 3/8" minus compacted bedding. A concrete ramp and sump are included in the pond design for sediment cleanout. The ramp and sump are located on the northwest corner and will also form the entry point for incoming flow. The concrete will be placed on a 6-inch layer of low permeability soil compacted to achieve a maximum permeability of 10-6 cm/sec. The pond will be normally empty to maintain maximum available storage volume for stormwater runoff and will provide containment of the 100-year, 24-hour storm event from the drainage area. The pond has a volume of 265,724 CF with side slopes of 3H:1V.

## 20.3 Reporting and Recordkeeping Requirements

### 20.3.1 Self-Monitoring Report Form (from APP P-513324 issued July 29th, 2021)

1. The permittee shall complete the Self-Monitoring Reporting Forms (SMRFs) provided by ADEQ and submit the completed report through the myDEQ online reporting system. The permittee shall use the format devised by ADEQ.
2. The permittee shall complete the SMRF to the extent that the information reported may be entered on the form. If no information is required during a reporting period, the permittee shall enter "not required" on the form, include an explanation, and submit the form to the Groundwater Protection Value Stream.
3. The tables contained in APP Section 4.0 list the monitoring parameters and the frequencies for reporting results on the SMRF:
   a. APP Table 11: QUARTERLY GROUNDWATER MONITORING
   b. APP Table 12: SEMI-ANNUAL GROUNDWATER MONITORING

The parameters listed in the above-identified tables from APP Section 4.0 are the only parameters for which SMRF reporting is required.

### 20.3.2 Operation Inspection / Logbook Recordkeeping

A signed copy of this permit shall be maintained at all times at the location where day-to-day decisions regarding the operation of the facility are made. A logbook (paper copies, forms, or electronic data) of the inspections and measurements required by this permit shall be maintained at the location where day-to-day decisions are made regarding the operation of the facility. The logbook shall be retained for ten years from the date of each inspection, and upon request, the permit and the log book shall be made immediately available for review by ADEQ personnel. The information in the logbook shall include, but not be limited to, the following information as applicable:

- Name of inspector.
- Date and shift inspection was conducted.

- Condition of applicable facility components.
- Any damage or malfunction, and the date and time any repairs were performed.
- Documentation of sampling date and time.
- Any other information required by this permit to be entered in the logbook; and
- Monitoring records for each measurement shall comply with A.A.C. R18-9-A206(B)(2).

### 20.3.3 Permit Violation and Alert Level Status Reporting

- The permittee shall notify the Groundwater Protection Value Stream within 5 days (except as provided in APP Section 2.6.5) of becoming aware of an AL exceedance, or violation of any permit condition, AQL, or DL for which notification requirements are not specified in APP Sections 2.6.2 through 2.6.5.
- The permittee shall submit a written report to the Groundwater Protection Value Stream within 30 days of becoming aware of the violation of any permit condition, AQL, or DL. The report shall document all of the following:
- Identification and description of the permit condition for which there has been a violation and a description of the cause; the period of violation including exact date(s) and time(s), if known, and the anticipated time period during which the violation is expected to continue.
- Any corrective action taken or planned to mitigate the effects of the violation, or to eliminate or prevent a recurrence of the violation.
- Any monitoring activity or other information which indicates that any pollutants would be reasonably expected to cause a violation of an AWQS.
- Proposed changes to the monitoring which include changes in constituents or increased frequency of monitoring.
- Description of any malfunction or failure of pollution control devices or other equipment or processes.

### 20.3.4 Operational, Other or Miscellaneous Reporting

The permittee shall record the information as required in APP Section 0, Table 13: FACILITY INSPECTION AND OPERATIONAL MONITORING in the facility log book as per APP Section 2.7.2, and report to the Groundwater Protection Value Stream any violations or exceedances as per APP Section 2.7.3.

### 20.3.5 Annual Report

If an Alert Level #1 has been exceeded discussed in APP Section 2.6.2.2, the permittee shall submit an annual report that summarizes the results of the liner assessment. The Liner Leakage Assessment Report shall also include information including but not limited to the following: number and location of holes identified; and a table summarizing alert level exceedances including the frequency and quantity of fluid removed, and corrective actions taken.

### 20.3.6 Reporting Location

All Self-Monitoring Report Forms (SMRFs) shall be submitted through the myDEQ portal accessible on the ADEQ website at: http://www.azdeq.gov/welcome-mydeq. Contact at 602-771-4571 for any inquiry related to the SMRFs.

5-day and 30-day contingency notification and reports, laboratory reports, and verification sampling results required by this permit should be submitted through the myDEQ portal accessible on the ADEQ website at: http://www.azdeq.gov/welcome-mydeq.

If the required reports cannot be submitted, or require further documentation that cannot be submitted on the myDEQ portal, then submit items to groundwaterpermits@azdeq.gov or the address listed below:

The Arizona Department of Environmental Quality

Groundwater Protection Value Stream Mail Code 5415B-3

1110 West Washington Street Phoenix, Arizona 85007

Phone (602) 771-4999

### 20.3.7 Reporting Deadline

The following table lists the quarterly SMRF report due dates:

**Table 20-2: Quarterly Reporting Deadlines**

| Monitoring Conducted During Quarter: | Quarterly Report Due By: |
|---|---|
| January-March | April 30 |
| April-June | July 30 |
| July-September | October 30 |
| October-December | January 30 |

The following table lists the semi-annual and annual SMRF report due dates (if applicable):

**Table 20-3: (Semi-)Annual Reporting Deadlines**

| Monitoring Conducted: | Report Due By: |
|---|---|
| Semi-annual: January-June | July 30 |
| Semi-annual: July-December | January 30 |
| Annual: January-December | January 30 |

### 20.3.8 Changes to Facility Information

The Groundwater Protection Value Stream shall be notified within ten days of any change of facility information including Facility Name, Permittee Name, Mailing or Street Address, Facility Contact Person, or Emergency Telephone Number.

## 20.4 Compliance or Operational Monitoring

Parameters in Table 20-4 shall also be used for contingency monitoring. Metals shall be analyzed as total metals.

**Table 20-4: Discharge Monitoring**

| Facility | Latitude | Longitude |
|---|---|---|
| Oxide PLS Pond | 32° 56' 12.24" N | 111° 49' 30.16" W |
| Enriched PLS Pond | 32° 56' 54.81" N | 111° 49' 38.66" W |
| Raffinate Pond | 32° 56' 52.36" N | 111° 49' 40.87" W |
| **Parameters for One-Time Discharge Monitoring (in mg/L unless otherwise noted)** | | |
| pH – field & lab (SU) | Magnesium | Mercury |
| Specific Conductance – field and lab (µmhos/cm) | Potassium | Cadmium |
| Total Dissolved Solids | Sodium | Cobalt |
| Total Alkalinity | Iron | Copper |
| Carbonate | Aluminum | Lead |
| Bicarbonate | Antimony | Nickel |
| Total Nitrogen | Arsenic | Selenium |
| Nitrate as N | Barium | Thallium |
| Nitrite as N | Cadmium | Zinc |
| Nitrate + Nitrite | Chromium | Total Uranium |
| Sulfate | Cobalt | Gross Alpha Particle Activity (pCi/L) |
| Chloride | Copper | Radium 226 + Radium 228 (pCi/L) |
| Fluoride | Lead | Uranium-Isotopes (pCi/L) |
| Calcium | Manganese | Ammonia |

**Table 20-5: Leak Collection and Removal System Monitoring**

| Facility Name | Parameter | Alert Level 1 (GPD)[1] | Alert Level 2 (GPD) | Monitoring Method | Monitoring Frequency |
|---|---|---|---|---|---|
| Alert Levels for Operational Depth of 15 feet | | | | | |
| Enriched PLS Pond | Liquid Pumped | 1,167 | 7,797 | Calculated | Weekly |
| Oxide PLS Pond | Liquid Pumped | 1,167 | 7,797 | Calculated | Weekly |
| Raffinate Pond | Liquid Pumped | 1,167 | 7,797 | Calculated | Weekly |
| Alert Levels at Freeboard of 28 feet | | | | | |
| Enriched PLS Pond | Liquid Pumped | 1,423 | 9,507 | Calculated | Weekly |
| Oxide PLS Pond | Liquid Pumped | 1,423 | 9,507 | Calculated | Weekly |
| Raffinate Pond | Liquid Pumped | 1,423 | 9,507 | Calculated | Weekly |

**Table 20-6: Parameters for Ambient Groundwater Monitoring**

| | | |
|---|---|---|
| pH – field & lab (SU) | Cadmium | para-Dichlorobenzene |
| Specific Conductance – field & lab (µmhos/cm) | Chromium | 1,2-Dichloroethane |
| Total Dissolved Solids | Cobalt | 1,1-Dichloroethylene |
| Total Alkalinity | Copper | Cis-1,2-Dichloroethylene |
| Carbonate | Lead | trans-1,2-Dichloroethylene |
| Bicarbonate | Manganese | Dichloromethane |
| Total Nitrogen | Mercury | 1,2-Dichloropropane |
| Nitrate as N | Chromium | Ethylbenzene |
| Nitrite as N | Cobalt | Hexachlorobenzene |
| Nitrate + Nitrite | Copper | Hexachlorocyclopentadiene |
| Sulfate | Lead | Monochlorobenzene |
| Chloride | Nickel | Styrene |
| Fluoride | Selenium | Tetrachloroethylene |
| Calcium | Thallium | Toluene |
| Ammonia | Zinc | Trihalomethanes (total) |
| Magnesium | Total Uranium | 1,1,1-Trichloroethane |
| Potassium | Gross Alpha Particle Activity (pCi/L) | 1,2,4 – Trichlorobenzene |
| Sodium | Gross Beta (pCi/L) | 1,1,2 – Trichloroethane |
| Iron | Radium 226 + Radium 228 (pCi/L) | Trichloroethylene |
| Aluminum | Uranium-Isotopes (pCi/L) | Vinyl Chloride |
| Antimony | Benzene | Xylenes (Total) |
| Arsenic | Carbon tetrachloride | o-Dichlorobenzene |
| Barium | Beryllium | |

Note: Units in mg/L unless otherwise noted. Metals must be analyzed as dissolved metals.

**Table 20-7: Quarterly Groundwater Monitoring**

| Parameter | Units | Future POC Wells Determined by CSI #20 | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | AL | AL | AQL | AL | AQL | AL | AQL | AL |
| Depth to Groundwater | ft. bgs | Monitor[2] | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Water Level Elevation | ft. amsl | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Temperature – Field | ºF | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| pH (lab) | SU | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Specific Conductance - lab | mmhos/c m | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Total Dissolved Solids | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Cyanide, Total | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Total Alkalinity | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Bicarbonate | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Carbonate | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Chloride | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Fluoride | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |

[2] Monitoring required, but no AQL or AL will be established in the permit.

**Table 20-8: Quarterly Groundwater Monitoring (continued)**

| Parameter | Units | Future POC Wells Determined by CSI #20 | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | AL | AL | AQL | AL | AQL | AL | AQL | AL |
| Sulfate | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Total Nitrogen[5] | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Nitrate as N | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Nitrite as N | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Nitrate + Nitrite | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Ammonia | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Metals (Dissolved) | | | | | | | | | |
| Aluminum | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Antimony | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Arsenic | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Barium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Beryllium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Cadmium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Calcium | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Chromium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Copper | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Iron | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Lead | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Magnesium | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Manganese | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Mercury | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Molybdenum | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Nickel | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Potassium | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Selenium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Thallium | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Zinc | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Uranium, Total | mg/l | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |
| Radionuclides | | | | | | | | | |
| Adjusted Gross Alpha[6] | pCi/L | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Gross Beta | pCi/L | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Radium 226 +228 | pCi/L | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Uranium- Isotopes[7] | pCi/L | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor | Monitor |

[5] Total Nitrogen is the sum of Nitrate as N, Nitrite as N, and Total Kjeldahl Nitrogen (TKN)

[6] The adjusted gross alpha particle activity is the gross alpha particle activity, including radium 226, and any other alpha emitters, if present in the water sample, minus radon and total uranium (the sum of uranium 238, uranium 235 and uranium 234 isotopes). The gross alpha analytical procedure (evaporation technique: EPA Method 900.0) drives off radon gas in the water samples. Therefore, the Adjusted Gross Alpha should be calculated using the following formula: (Laboratory Reported Gross Alpha MINUS Sum of the Uranium Isotopes).

[7] Uranium Isotope activity results must be used for calculating Adjusted Gross Alpha. SMRF reporting is required after completion of ambient groundwater monitoring.

**Table 20-9: Semi-Annual Groundwater Monitoring**

| | Parameter | | Units | Future POC Wells Determined by CSI #20 | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | AL | AL | AQL | AL | AQL | AL | AQL | AL |
| Benzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Carbon tetrachloride | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| o-Dichlorobenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| para-Dichlorobenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,2-Dichloroethane | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,1-Dichloroethylene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| cis-1,2-Dichloroethylene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| trans-1,2-Dichloroethylene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Dichloromethane | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,2-Dichloropropane | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Ethylbenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Hexachlorobenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Hexachlorocyclopentadiene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Monochlorobenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Styrene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Tetrachloroethylene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Toluene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Trihalomethanes (total)[8] | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,1,1-Trichloroethane | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,2,4 - Trichlorobenzene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| 1,1,2 – Trichloroethane | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Trichloroethylene | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Vinyl Chloride | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |
| Xylenes (Total) | mg/l | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved | Reserved |

The Aquifer Protection Permit Compliance Schedule is in Table 20-10.

**Table 20-10: Compliance Schedule Items**

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 1 | The permittee shall submit an amendment application along with a financial assurance mechanism as per A.A.C. R18-9-A203(C) for the estimated closure and post-closure costs for the APP facilities as per Section 2.1.2 of this permit. | A minimum of 60 days prior to any construction of the Heap Leach Pad and/or Ponds. | Yes |

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 2 | The permittee shall submit a demonstration that the financial assurance mechanism listed in Section 2.1, Financial Capability, is being maintained as per A.R.S. 49-243.N.4 and A.A.C. R18-9-A203(H) for all estimated closure and post-closure costs including updated costs submitted under Section 3.0, No. 2 below. The demonstration shall include a statement that the closure and post-closure strategy has not changed, the discharging facilities listed in the permit have not been altered in a manner that would affect the closure and post-closure costs and discharging facilities have not been added. The demonstration shall also include information in support of a "performance surety bond" as required as per A.A.C. R18-9-A203(C)(2).<br>NOTE: The financial assurance mechanism due on the date specified in CSI No. 3, may be provided following ADEQ's approval of the closure and post-closure costs. When submitting the closure and post-closure costs, permittee may provide a statement for the type of mechanism intended to be provided. | Every 6 years from the date of permit signature, for the duration of the permit. | No |
| 3 | The permittee shall submit updated cost estimates for facility closure and post-closure, as per A.A.C. R18-9- A201(B)(5) and A.R.S. 49-243.N.2.a.<br>NOTE: When submitting the closure and post-closure costs the permittee may provide a statement for the type of mechanism they intended provide (see CSI No. 2). The financial assurance mechanism, may then be submitted following ADEQ's approval of the closure and post-closure costs. | Every 6 years from the date of permit signature, for the duration of the permit. | Yes |
| 4 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Oxide Leach Pad to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 5 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Enriched Leach Pad to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 6 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Oxide PLS Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 7 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Enriched PLS Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 8 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Raffinate Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 9 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Oxide Events Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 10 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Enriched Events Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 11 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Site Runoff Pond 1 to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 12 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for Site Runoff Pond 2 to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 13 | The permittee shall submit a construction report along with as-built drawings and QA/QC documentation sealed by an Arizona registered professional engineer for the Waste Rock Stockpile Runoff Pond to confirm that the facility was constructed in accordance with the design report, engineering plans and specifications submitted in the application. | Prior to discharging under this permit and within 90 days of completion of construction. | No |
| 14 | The permittee shall conduct discharge monitoring on a one-time basis at the Oxide PLS Pond following steady flow of Oxide PLS solutions from the Oxide Heap Leach Pad. Notify ADEQ on the day the discharge monitoring is conducted. | Between 90 and 180 days after starting of leaching operations. | No |
| 15 | The permittee shall submit the results (including laboratory report) of the discharge monitoring conducted at the Oxide PLS Pond as per CSI No. 14. | Within 30 days of receipt of laboratory analytical results | No |
| 16 | The permittee shall conduct discharge monitoring on a one-time basis at the Enriched PLS Pond following steady flow of Enriched PLS solutions from the Enriched Heap Leach Pad. Notify ADEQ on the day the discharge monitoring is conducted. | Between 90 and 180 days after starting of leaching operations. | No |
| 17 | The permittee shall submit the results (including laboratory report) of the discharge monitoring conducted at the Enriched PLS Pond as per CSI No. 16. | Within 30 days of receipt of laboratory analytical results | No |

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 18 | The permittee shall conduct discharge monitoring on a one-time basis at the Raffinate Pond following steady flow of raffinate solutions from the stripper unit. Notify ADEQ on the day the discharge monitoring is conducted. | Between 90 and 180 days after starting of commencement of SX/EW operations. | No |
| 19 | The permittee shall submit the results (including laboratory report) of the discharge monitoring conducted at the Raffinate Pond as per CSI No. 18. | Within 30 days of receipt of laboratory analytical results | No |
| 20 | The permittee shall submit to ADEQ within 30 days of permit issuance plans for a new POC monitoring well including well construction details, depth to water, and latitude and longitude for approval. | in 30 days of permit issuance. | No |
| 21 | Within 30 days after ADEQ's approval of the proposed POC well location, the permittee shall submit an application for an Other Amendment to designate the POC well and begin the ambient monitoring period. | Within 30 days of ADEQ approval of the POC well. | Yes |
| 22 | The permittee shall submit an APP amendment application and an ambient groundwater monitoring report to establish ALs and AQLs for the POC well. At a minimum, the report shall contain copies of all ADWR documents related to the well, as-built diagrams of the well, and latitude and longitude of each well. The report shall be sealed by an Arizona Registered Geologist or other qualified registrant. | Within 90 days of completion of ambient monitoring. | Yes |
| 23 | The permittee shall submit a model report to the Groundwater Protection Value Stream demonstrating passive containment of the pit lake. The collection, analysis and interpretation of groundwater elevation, gradient information and or groundwater quality collected shall be included in the report, which qualifies as a Professional Document and requires supervision and seals from each professional AZ registrant responsible for the work. | 180 days before any activity in the mine pit takes place. | No |

| No. | Description | Due By: | Permit Amendment Required? |
|---|---|---|---|
| 24 | The permittee shall submit a model report to the Groundwater Protection Value Stream demonstrating passive containment of the pit lake, including any revisions resulting from hydrologic or operational changes observed during the re-evaluation every five (5) years, after the initial demonstration. The collection, analysis and interpretation of groundwater elevation, gradient information and or groundwater quality shall be included in the report, which qualifies as a Professional Document and requires supervision and seals from each professional AZ registrant responsible for the work. | Every five years after the initial evaluation in CSI No. 25 | No |
| 25 | Prior to any activity within the Cactus Mine Pit the permittee shall submit a closure plan and post-closure monitoring plan for the Pit. | 6 months prior to any work being performed within the Cactus Mine Pit. | Yes |
| 26 | Based on the results of CSI #25 above, permittee shall update the closure/post-closure cost estimate and update the Financial Assurance Mechanism based in the change in closure/post-closure cost estimates. | 6 months prior to any work being performed within the Cactus Mine Pit. | Yes |
| 27 | Before any construction activity on the waste rock stockpile (WRS), and before any materials are placed on the new WRS, it should be added to the permit. | 180 days before planned construction of the WRS or placing any materials on it. | Yes |

Industrial Air Permit Requirements as per Permit #C31407.000:

ASCU is required to submit a Semi-annual Report documenting emissions and use of emission-generating materials from January thru June and July thru December each year.

## 20.5 Social Considerations

In keeping with ASCU's community engagement and partnership standards, the Project will be developed with a plan to establish and maintain the support of our host communities. ASCU has commenced community outreach at the earliest stages of the Project and is currently evaluating and building partnerships within the community. As the Project's permits will involve a public process and are based on the permit submission and review schedule, ASCU understands the importance of outreach during the permitting process and throughout the life of the mine. ASCU is encouraged by the positive response to the project from the community. Its status as a “brownfields” project makes it a more appealing project than a new mine might be.

## 20.6 Closure and Reclamation Planning

A Mined Land Reclamation Permit (MLRP) was issued by the state in 2023 and an Amended Aquifer Protection Permit was issued in 2021 based on ASCU’s original PEA design. ASCU has posted a bond of $4,797,829.00 for the MLRP reclamation costs and has an $1,144,576.00 bond with APP that has not been posted yet but will be posted prior to construction. ASCU will need to amend these permits to reflect changes from this PFS. The APP will cover closure and remediation of the leach pads, which consists of rinsing and caping the leach pads, and the ponds, which consists of draining and treating any residual fluids, then removing the liners. The MLRP covers the removal of any buildings, scarification and revegetating existing roads, capping of waste rock disposal sites, and safeguarding access to the pit and any underground access.

ASCU estimates that the new closure bond estimates for both the APP and MLRP will be $23M based the increase in production from the Parks/Salyer deposit and the increase in leach pads and waste rock disposal.

# 21 CAPITAL AND OPERATING COSTS

## 21.1 Introduction

The capital cost estimates for this PFS were developed with a -20% to +30% accuracy with a level of total contingency from 15% according to the association of the Advancement of Cost Engineering International's (AACE) Class 4 estimate requirements.

ASCU has engaged third-party consultants to contribute to the total project scope of work and overall capital cost estimate. Capital and operating estimates for infrastructure (WBS code 1000) and leach pad construction (WBS code 4000) are within Ausenco's scope. Capital and Operating estimates for mining (WBS 2000) were assigned to AGP. Corresponding estimates for the required processing facilities (WBS codes 3000-8000) are the responsibility of Samuel Engineering. Finally, Ausenco incorporated the third-party contributions into an overall pre-feasibility study cost estimate.

All third-party contributors are accountable for the development and quality of their cost estimates, they are inclusive of all direct costs, growth allowances, project indirect costs, and associated contingency within their scope of work, but separately identified. Each aligns with the overall project WBS numbering system (as defined in Section 2.7.1).

## 21.2 Estimate Structure and Definitions

The capital cost estimates are broken out by direct and indirect costs and further defined by initial and sustaining costs.

Direct costs are generally quantity based and include equipment and materials associated with the physical construction of the facility. Construction contractor's costs are contained within each discipline's all-in rates and considered as direct costs.

Project indirect costs include all costs associated with the implementation of the plant and incurred by the owner, engineer, or consultants in the design, procurement, construction, and commissioning of the project.

Initial capital is the capital expenditure required to start up a business or in this case, complete the construction of a facility to a standard where it is ready for initial production.

Sustaining capital is the capital cost associated with the periodic addition of new plant equipment or services that are required to maintain production and operations.

These estimates are derived from data as shown on the drawings, models, MTOs, quotes and historical data. They include the associated infrastructure as defined within the individual scopes of work.

### 21.2.1 Direct Costs

Direct costs are those costs that pertain to the permanent equipment, materials and Labour associated with the physical construction of the process facility, infrastructure, utilities, buildings, etc. All inclusions are identified by WBS and discipline and grouped into initial and sustaining capital.

The following items are included:

- Permanent equipment
- Bulk materials
- Consumables (Job Materials)
- Growth (Design Allowance)
- Freight
- Subcontractor costs
- Direct Man-hours
- Contractor Direct Labour (Field Labour)
- Contractor Labour Distributable (Indirect costs)

## 21.3 Capital Cost Estimates

### 21.3.1 Capital Cost Estimate Summaries

The project capital cost estimate was compiled by Ausenco Engineering USA South Inc. (“Ausenco”) with input from AGP and Samuel Engineering for the open pit, underground mining operation, SXEW process plant, conveying, crushing and screening equipment, site sub-station, site power distribution, access roads, heap leach facilities and associated infrastructure. all direct costs, growth allowances, project indirect costs, and associated contingency within their scope of work, but separately identified. Initial capital costs and sustaining development costs are summarized in Table 21-1.

**Table 21-1: Initial and Sustaining Capital Costs**

| Capitalized Costs | Initial ($M) | Sustaining ($M) |
|---|---|---|
| **Mining and Processing** | **174** | **905** |
| Processing | 4 | 0 |
| Mining (Pre-Stripping) | 78 | 0 |
| Mining – Open Pit – Cactus West | 24 | 20 |
| Mining – Underground – Cactus East | 0 | 341 |

| Capitalized Costs | Initial ($M) | Sustaining ($M) |
|---|---|---|
| Mining – Underground – Parks/Salyer | 57 | 544 |
| Mining – Underground – Combined/Shared | 11 | 0 |
| **Other** | **342** | **315** |
| Infrastructure | 56 | 0.3 |
| Crushing And Conveying | 29 | 6 |
| Leaching & Waste Rock Storage | 66 | 126 |
| Solvent Extraction (SX) | 30 | 0 |
| Electrowinning (EW) | 26 | 14 |
| Reagents | 1 | 0 |
| Process Plant Services and Utilities | 4 | 0 |
| Project Execution | 54 | 8 |
| Provisions | 75 | 160 |
| **PROJECT TOTAL** | **515** | **1,221** |

The capital cost estimates have been combined and summarized to the levels indicated in Table 21-2 and Table 21-3. This data is stated in United States Dollar (US$) with a base date of 4th Quarter 2023 and with no provision for forward escalation.

**Table 21-2: Total Project Costs Summary – Level 1 Major Facility**

| WBS Level 1 | WBS Description | Initial ($M) | Sustaining ($M) | Total Cost ($M) |
|---|---|---|---|---|
| 1000 | Infrastructure | 56.2 | 0.3 | 56.5 |
| 2000 | Mining | 169.8 | 905.4 | 1,075.2 |
| 3000 | Crushing And Conveying | 29.1 | 6.5 | 35.6 |
| 4000 | Leaching & Waste Rock Storage | 65.8 | 126.7 | 192.5 |
| 5000 | Solvent Extraction (SX) | 30.5 | 0 | 30.5 |
| 6000 | Electrowinning (EW) | 26.3 | 14.4 | 40.7 |
| 7000 | Reagents | 1.2 | 0 | 1.2 |
| 8000 | Process Plant Services and Utilities | 3.5 | 0 | 3.5 |
| | **Subtotal Direct Costs** | **382.4** | **1,053.3** | **1,435.8** |
| 9000 | Project Execution – Project Indirect | 43.7 | 5.7 | 49.5 |
| 9000 | Project Execution – Owner's Costs | 10.4 | 1.9 | 12.3 |
| 9000 | Project Execution – Provision | 75.3 | 160.2 | 235.5 |
| | **Subtotal Indirect Costs** | **129.4** | **167.8** | **297.2** |
| | Capitalized Process Cost | 3.6 | 0 | 3.6 |
| | **PROJECT TOTAL** | **515.4** | **1,221.1** | **1,736.5** |

Table 21-3: Total Project Costs Summary – Responsible Party

| Scope | Owner | Initial ($M) | Sustaining ($M) | Total Cost ($M) |
|---|---|---|---|---|
| Heap Leach and Infrastructure | Ausenco | 148.7 | 145.1 | 293.7 |
| Mining Fleet and Mine Development | AGP | 172.4 | 1,033.1 | 1,205.5 |
| Process Plant (including Capitalized Process Cost) | Samuel Engineering | 184 | 41.0 | 225.0 |
| Owner | ASCU | 10.4 | 1.9 | 12.3 |
| | **PROJECT TOTAL** | **515.4** | **1,221.1** | **1,736.5** |

#### 21.3.1.1 Total Direct Hours by Responsible Party

Total initial and sustaining site hours are shown in Table 21-4 that summarises the total direct hours for installation, categorized by responsible party.

Table 21-4: Total Direct Hours by Responsible Party

| Responsible Party | Initial Direct Hours | Sustaining Direct Hours | Total Direct Hours | % of Total |
|---|---|---|---|---|
| Ausenco - Heap Leach and Infrastructure | 478,371 | 1,176,591 | 1,654,962 | 81 |
| Samuel Eng - Process Plant | 332,155 | 43,522 | 375,677 | 19 |
| AGP - Mining Fleet and Mine Development | 0 | 0 | 0 | 0 |
| **Total Direct & Indirect Hours** | **810,562** | **1,220,113** | **2,030,639** | **100** |

### 21.3.2 Infrastructure and Leach Pads

Ausenco's capital cost estimate for infrastructure and leach pad construction is summarized to the levels indicated in Table 21-5 for initial capital, and Table 21-7 for sustaining capital. These values are stated in United States Dollar (US$) with a base date of 4th Quarter 2023 and no provision for forward escalation. A summary for the supply cost with the % total cost from quoted, historical, estimated, and allowance is shown in Table 21-5

Table 21-5: Supply cost source summary

| Pricing Basis | Initial ($M) | Sustaining ($M) | Total Cost ($M) | % of Total |
|---|---|---|---|---|
| Quoted – Budget | 43.1 | 56.6 | 99.7 | 71.4 |
| Quoted – Preliminary | 6.3 | 0 | 6.3 | 4.5 |
| Historical Data | 16.0 | 15.0 | 31.0 | 22.2 |
| Estimated | 1.6 | 0 | 1.6 | 1.2 |
| Allowance | 1.0 | 0 | 1.0 | 0.7 |
| **Total Supplies** | **68.1** | **71.6** | **139.7** | **100%** |

Table 21-6: Ausenco's Cost Summary – Initial Capital

| Cost Type | WBS LVL 1 | LVL 1 Description | Total (US$M) |
|---|---|---|---|
| Direct | 1000 | Infrastructure | 53.2 |
| | 2000 | Mining | 11.1 |
| | 4000 | Leaching and Waste Rock Storage | 45.7 |
| **Direct Total** | | | **110.0** |
| Indirect | 9000 | Project Execution | 14.0 |
| **Indirect Total** | | | **14.0** |
| Provision | 9000 | Project Execution | 24.7 |
| **Provision Total** | | | **24.7** |
| **Ausenco's Total Initial Capital Cost** | | | **148.7** |

Table 21-7: Ausenco's Cost Summary – Sustaining Capital

| Cost Type | WBS LVL 1 | LVL 1 Description | Total (US$M) |
|---|---|---|---|
| Direct | 1000 | Infrastructure | 0.3 |
| | 4000 | Leaching and Waste Rock Storage | 120.6 |
| **Direct Total** | | | **120.9** |
| Indirect | 9000 | Project Execution | |
| **Indirect Total** | | | |
| Provision | 9000 | Project Execution | 24.2 |
| **Provision Total** | | | **24.2** |
| **Ausenco's Total Sustaining Capital Cost** | | | **145.1** |

#### 21.3.2.1 Scope

The items within Ausenco's scope include the facilities required to support the mining operations. Ausenco is also responsible to provide services (utilities) to the facilities that fall under all third-party consultants. Capital costs estimate for Infrastructure and heap leach construction includes:

- Development of new access roads and repair of existing roads.
- Construction of a new haul road for heavy mining equipment.
- Buildings such as offices, warehouses, shops, and labs.
- Services including potable water, plant water, sewage disposal and power.
- Main power distribution to all facilities including mining and processing.
- An explosives storage area.
- A single leach pad along with the associated collection and distribution ponds.

- Stormwater controls, diversions, culverts, and channels.

#### 21.3.2.2 Exclusions

The following items were not considered in Ausenco's Class 4 PFS capital cost estimate:

- Residual value of temporary equipment and facilities.
- Cost of this study or any further studies.
- Special incentives (schedule, safety, or others).
- Management reserve.
- Replacement capital.
- Scope changes.
- Operating costs.
- Environmental approvals.
- Environmental impact assessment.
- No allowance has been made for loss of productivity and/or disruption due to religious, union, social and/or cultural activities.
- Force majeure issues.
- Foreign exchange exposure.
- Permitting.
- Escalation costs beyond 4$^{th}$ Quarter 2023.
- Sunk costs.
- Costs associated with geotechnical field investigations.
- Land acquisition.
- Final closure plan and estimate.
- Management of other consultants.
- Taxes and duties.

#### 21.3.2.3 Estimating Methodology

The methodology applied and source data used to develop the estimate is as follows:

- Defined the scope of work.

- Quantified the work in accordance with standard commodities.
- Organized the estimate structure in accordance with an agreed WBS.
- Calculated “all in” Labour rates for construction work.
- Determined the bulk materials supply pricing from recent returned contractor’s rates from similar projects.
- Determined the installation cost for equipment and bulks, based on recent returned contractor’s rates from similar projects.
- Established requirements for freight.
- Determined foreign exchange content and exchange rates.
- Determined growth allowances for each estimate line item.
- Integrated third party information into estimate.
- Determined the costs to carry out detailed engineering design and project management.
- Determined the construction Indirect capital cost.
- Determined the estimate contingency value.
- Undertook internal peer review, finalize the estimate, estimate basis and obtain sign off by the Project Manager and Qualified Professional.

Source data used in the development of the Class 4 estimate includes:

- Scope of work.
- Design criteria by discipline.
- General arrangement drawings.
- Layout drawings.
- Sketches.
- Electrical Equipment lists.
- Contractor cost data (bulk material pricing, Labour rates, installation hours).
- Material take-offs.
- Construction and permanent camp requirements.
- Third party estimate stakeholder’s information.
- Project Schedule
- Applicable factors and historical data.

The following basic information pertains to the estimate:

- The estimate base date is 4th Quarter 2023.
- The estimate is expressed in United States Dollars (US$).
- Imperial units of measure are used throughout the estimate.
- Actual estimate accuracy is defined by the stated maturity of the information available.

#### 21.3.2.4 Architectural Buildings (Commodity A)

The estimate includes the supply and installation of all the buildings within the on-site infrastructure area. The list of the buildings is based on document # 107504-GX-0000-24000-001, Building List.

The building list was developed to describe the requirements of the buildings, including sizing, load requirements, and features. Cost of the buildings included design, supply, delivery, and installation. The modular buildings are based on a budget quoted and the pre-engineered buildings are based on Ausenco's historical data for recent projects.

Table 21-8 shows the cost per unit and the total cost including supply, installation, and delivery. All buildings listed below are included in the Initial phase of the project.

**Table 21-8: Building List and Cost**

| Building Type | Buildings List | Total Cost ($M) Exclude Growth |
|---|---|---|
| Modular | Administration building - modular building - length: 128 - width: 60 - area (ft2): 7,680 - volume (ft3): 69,120 - include design, fabrication, transportation and installation | 1.7 |
| Modular | Security gatehouse & weighbridge - modular building - length: 20 - width: 12 - area (ft2): 240 - volume (ft3): 2,160 - include design, fabrication, transportation, and installation | 0.1 |
| Pre-Engineered | Plant workshop and warehouse - pre-engineered building - steel structure w/ Insulated metal panel in roofing & siding - length: 144 - width: 74 - area (ft2): 10,656 - volume (ft3): 277,056include design, fabrication, transportation and installation | 1.6 |
| Modular | Laboratory - modular building - length: 78 - width: 30 - area (ft2): 2,340 - volume (ft3): 21,060 - include design, fabrication, transportation, and installation | 0.5 |
| Modular | Mine office / administration / change house - modular building - length: 175 - width: 60 - area (ft2): 10,500 - volume (ft3): 94500 - include design, fabrication, transportation, and installation | 2.6 |
| Modular | Mine maintenance office - modular building - length: 48 - width: 34 - area (ft2): 1,632 - volume (ft3): 14,688 - include design, fabrication, transportation, and installation | 0.4 |
| | **Total Buildings cost** | **6.9** |

#### 21.3.2.5 Earthworks (Commodity B)

All earthwork quantities were estimated from quantity take-offs from the Civil 3D software model and/or engineering calculations from 2D drawings.

Quantities cover the Ausenco's scope as noted in document; Bulk Earthworks / Site Development, and 107504-EB-00000-18000-001 Low Grade HLP and Fine Crush HLP.

Sub-contract rates are used in the estimate for bulk earthworks requirements. Prices carried in the estimate are based on Ausenco's historical rates for recent projects in the area.

Detailed earthworks, such as detailed excavation and backfill associated with the installation of concrete have been included and itemized separately in the concrete discipline costs.

Major earthworks quantities and all-in rates are in Table 21-9.

**Table 21-9: Earthworks Quantities and Rates**

| Commodity - Earthworks | UoM | Unit Installed Rate | Initial Qty | Sustaining Qty |
|---|---|---|---|---|
| Bulk Earthworks / Site Development – Infrastructure Facility | | | | |
| Clear and grub Light/medium wooded | Acre | 1,306.1 | 130.3 | 0 |
| Topsoil removal | $yd^3$ | 6.9 | 198,000 | 0 |
| Bulk cut to fill (common) - under 1.5 miles | $yd^3$ | 2.5 | 122,000 | 0 |
| Relocate material by dozer (or other) to fill within 0.5 miles | $yd^3$ | In mining rates* | 36,100 | 0 |
| Bulk Cut (Rock Rippable) | $yd^3$ | 16.1 | 6,000 | 0 |
| Haul cut material - load/cart and dump | $yd^3$ | In mining rates* | 49,100 | 0 |
| Backfill with Cut Material | $yd^3$ | In mining rates* | 37,700 | 0 |
| Backfill with imported structural material | $yd^3$ | 36.3 | 177.8 | 0 |
| Backfill (common) with local side cast or waste pile - place & compact | $yd^3$ | 19.5 | 9,500 | 0 |
| Crushed/Graded Fill | $yd^3$ | 9.2 | 46,400 | 0 |
| Riprap Protection | $yd^3$ | 45.0 | 52,400 | 0 |
| Trenching Excavation | $yd^3$ | 37.0 | 137.0 | 0 |
| Trenching Backfill | $yd^3$ | 50.6 | 643.0 | 0 |
| Road Sub-Base preparation | $yd^3$ | 10.4 | 53,100 | 0 |
| Road Base preparation | $yd^3$ | 62.3 | 22,600 | 0 |
| Road surface - bitumen | $ft^2$ | 12.70 | 287,000 | 0 |
| Drainage Dich | ft | 15.0 | 23,200 | 0 |
| PVC sewers drain DN 250 | ft | 30.0 | 3,100 | 0 |
| Bulk Earthworks / Site Development – Leach Pad & Waste rock Storage Facilities | | | | |
| Clear and grub Light/medium wooded | Acre | 1,306.09 | 221.5 | 16.8 |
| Topsoil removal | $yd^3$ | 6.9 | 179,000 | 13,549.0 |

| Commodity - Earthworks | UoM | Unit Installed Rate | Initial Qty | Sustaining Qty |
|---|---|---|---|---|
| Bulk cut (common) | $yd^3$ | 2.5 | 2,830,000 | 15,870,000.0 |
| Haul cut material - load/cart and dump | $yd^3$ | In mining rates* | 608,000 | 11,700,000 |
| Backfill with Cut Material | $yd^3$ | In mining rates* | 6,260,000 | 4,420,000 |
| Trenching excavation | $yd^3$ | 37.0 | 19,400 | 73,500 |
| Trenching backfill | $yd^3$ | 43.6 | 15,200 | 57.500 |
| 4" HDPE Perforated dual wall pipe | ft | 49.1 | 106,900 | 236,200 |

### 21.3.2.6 Concrete (Commodity C)

The scope of the concrete works allows for all concrete work in the on-site facilities which pertains to Ausenco's scope of work, as detailed in the WBS. All concrete quantities were prepared by engineering and are based on calculation derived from general arrangement drawings and sketches.

The basis for the development of installed concrete is the product of concrete material supply and installation costs based on similar recent project in the Southwest United States. Labour costs include the necessary consumables, reinforcement bar, and formwork.

In the case of existing infrastructure, namely the Truestone facility, it is assumed that the existing concrete foundations and slabs are in good working order and are of sufficient design to be used without modification. No allowance has been given for concrete work within existing structures.

All concrete works are included in the initial phase of the project, no new concrete work has been assigned to sustaining capital.

Major concrete quantities and all-in rates are noted in Table 21-10.

**Table 21-10: Concrete Quantities & All-in Rates (Supply and Install)**

| Commodity - Concrete | UoM | Unit Installed Rate | Initial Qty | Sustaining Qty |
|---|---|---|---|---|
| Structural Excavation | $yd^3$ | 20.5 | 1.246.8 | 17.3 |
| Structural Backfill | $yd^3$ | In mining rates* | 375.5 | 6.2 |
| Mass Concrete 15 Mpa | $m^3$ | 514.6 | 236.0 | 0.7 |
| Pad Footings | $yd^3$ | 1,371.9 | 145.0 | |
| Equipment Foundation 32Mpa | $yd^3$ | 1,510.0 | 51.8 | |
| Hold Down Bolts Steel (over) M24 | t | 31,690.7 | 3.9 | 0.1 |
| Pedestals 32Mpa | $yd^3$ | 2,105.8 | 372.6 | 20.7 |
| Slab on Grade 30Mpa | $yd^3$ | 1,118.4 | 988.1 | |
| Walls Structural 30Mpa | $yd^3$ | 2,036.3 | 32.7 | |
| Walls bunds 30Mpa | $yd^3$ | 1,620.6 | 60.0 | |
| Elevated Slabs | $yd^3$ | 2,674.6 | 125.6 | |

The total required volume of structural concrete at initial phase is 3,638 $yd^3$.

#### 21.3.2.7 Pipework (Commodity P)

The off-plot pipelines have been preliminary designed to account for flows, specifications, sizing, and lengths of each line.

Quantities are based on the document ‘Overland Pipeline MTO’.

The supply pricing for off-plot piping is based on recent pricing for the supply and delivery; fitting costs were factored into the pipe unit rates based on a typical complexity. Valve cost and pumps has been quantified and included by pipeline and by WBS area.

All piping works are included in the Initial phase of the project, no new development was assigned to sustaining capital.

A summary of the off-plot pipelines is highlighted in Table 21-11.

**Table 21-11: Off-plot Pipelines**

| Commodity – Off-Plot Pipelines | UoM | Unit Installed Rate | Initial Qty |
|---|---|---|---|
| SIZE: 1" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 24.6 | 251 |
| SIZE: 3" - Carbon steel lined (CSOL) Pipe, sched 40 | ft | 49.3 | 12,413 |
| SIZE: 4" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 49.3 | 3,264 |
| SIZE: 6" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 81.1 | 7,649 |
| SIZE: 8" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 113.0 | 6,995 |
| SIZE: 10" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 143.6 | 27,204 |
| SIZE: 12" - Carbon steel lined (CSOL) pipe, sched 40 | ft | 159.1 | 4,538 |
| SIZE: 1" - HDPE, PE4710, DR13.5, ASTM D3035 | ft | 34.4 | 244 |
| SIZE: 2" - HDPE, PE4710, DR13.5, ASTM D3035 | ft | 34.4 | 567 |
| SIZE: 4" - HDPE, PE4710, DR13.5, ASTM F714 | ft | 46.6 | 7,549 |
| SIZE: 6" - HDPE, PE4710, DR13.5, ASTM F714 | ft | 130.7 | 1,722 |
| SIZE: 8" - HDPE, PE4710, DR13.5, ASTM F714 | ft | 130.7 | 4,619 |
| SIZE: 12" - Ductile iron with cement lined | ft | 378.3 | 6,084 |
| **Total Off-Plot Pipelines (ft)** | | | **83,099** |

#### 21.3.2.8 Electrical Equipment (Commodity E)

The estimate allows for the supply and installation of all new electrical equipment for the on-site facilities as detailed in the WBS within Ausenco’s scope.

The list was developed by the electrical engineering department from the following sources:

- Site/plant layout drawings
- Single line drawings
- Load List
- Mechanical equipment list.

Major equipment prices were quoted by vendors. These quotes were technically and commercially evaluated by engineering. The costs included in the estimate are based on information from document 107504-EE-00000-24242-001, Electrical Equipment List.

Table 21-12 notes the pricing basis for the electrical equipment supply.

**Table 21-12: Electrical Equipment Supply Price Basis**

| Commodity – Electrical Equipment | Initial ($M) | % of Total |
|---|---|---|
| Quote - Budget | 17.8 | **94.6** |
| Estimated | 1.0 | **5.4** |
| **Total Electrical Equipment Supply** | **18.8** | **100** |

#### 21.3.2.9 Electrical Bulks (Commodity L)

An allowance for the remaining electrical bulks (grounding, cable trays, conduits, and cable) has been developed by factoring the total electrical equipment cost. A total of US$1.0M, is carried in the estimate, which is the blended factor of total supplied equipment cost.

#### 21.3.2.10 Growth Allowance

Each line item of the estimate is developed initially as a bare quantity and cost. A growth allowance has then been allocated to each element of those line items' costs to reflect the level of definition of design (Quantity Maturity) and pricing strategy (Cost Maturity).

Estimate growth accounts for:

- Items that cannot be quantified based on the current engineering status but are empirically known to appear, essentially bridging the gap from study to constructed quantities/costs.
- The accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at Pre-Feasibility Study level.
- Pricing growth for the likely increase in cost due to the development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the project.

Where an allowance has been used which is the result of factoring, no growth has been applied as the factor has been surmised from a total cost.

Growth has been calculated at the line-item level by evaluating the status of the engineering scope definition, maturity, and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The total cost of growth is presented in Table 21-13.

**Table 21-13: Total Growth Allowance – Ausenco Scope**

| Growth Allowance | Initial | | Sustaining | |
|---|---|---|---|---|
| | ($M) | % Blended (Qty & Price) | ($M) | % Blended (Qty & Price) |
| Architectural | 0.7 | 8.4 | 0 | 0 |
| Earthworks | 6.3 | 10.7 | 12.7 | 10.5 |
| Concrete | 0.4 | 11.0 | 0 | 0 |
| Electrical Equipment & Bulks | 1.7 | 7.9 | 0 | 0 |
| Mechanical Equipment & Platework | 0.3 | 7.1 | 0 | 0 |
| Piping And Fittings (Off-Plot Pipe) | 1.2 | 10.2 | 0 | 0 |
| Structural Steel | 0.3 | 12.2 | 0 | 0 |
| **Total Growth Allowance** | **10.9** | **9.8** | **12.7** | **11.3** |

#### 21.3.2.11 Freight Costs

Freight costs include all costs associated with the delivery to site of all permanent and job materials, including the following:

- inland transportation
- overseas freight
- export packing.
- bulking factors
- port charges and demurrage costs
- air freight

The freight costs were calculated in the direct costs as a separate cost element for each line item based on various freight percentages. No freight logistics plan was developed for this PFS.

The estimated freight costs have been determined by applying a percentage to the material and equipment supply costs by line item. Import duties are excluded from the estimate.

The percentage carried in the estimate have been highlighted in Table 21-14 and are based on current market conditions and Ausenco's current project experiences.

**Table 21-14: Freight Percentages – Ausenco Scope**

| Freight Description | % applied to purchase price |
|---|---|
| Inland Freight – USA to site | 10.0 |

Table 21-15 shows the distribution of freight costs by commodity.

**Table 21-15: Freight Cost Distribution**

| Commodity | Description | Initial ($M) | Sust. ($M) |
|---|---|---|---|
| Architectural / Buildings | Included in Sub-contractor rate by contractor | 0 | 0 |
| Earthworks | Factored on Supply Rates | 0.2 | 0.3 |
| Concrete | Factored on Supply Rates | 0.1 | 0 |
| Electrical Equipment | Factored on Supply Rates | 0.5 | 0 |
| Piping (Off-Plot), pumps & fiberglass tank | Factored on Supply Rates | 0.3 | 0 |
| | **Total Freight - Blended %** | **1.4** | **0.3** |

### 21.3.2.12 Taxes & Duties

Taxes and duties are excluded from the cost estimate.

### 21.3.2.13 Direct Hours by Discipline

Direct site hours shown in Table 21-16 summarise the total direct hours for installation, categorized by discipline.

**Table 21-16: Direct Hours by Discipline – Ausenco Scope**

| Commodity Description | Initial Direct Hours | Sustaining Direct Hours | Total Direct Hours | % of Total Infrastructure |
|---|---|---|---|---|
| Architectural | 8,464 | 222 | 9,336 | 0.5 |
| Earthworks | 402,019 | 1,176,120 | 1,578,139 | 95.5 |
| Concrete | 12,390 | 249 | 12,639 | 0.8 |
| Structural Steelwork | 5,722 | 0 | 5,722 | 0.3 |
| Platework | 148 | 0 | 148 | 0.0 |
| Mechanical Equipment | 1,990 | 0 | 1,990 | 0.1 |
| Piping And Fittings (A/G) & (U/G) | 39,747 | 0 | 42,181 | 2.4 |
| Electrical Equipment | 4,777 | 0 | 4,770 | 0.3 |
| **Total Direct Hours** | **475,257** | **1,176,591** | **1,651,848** | **100** |

#### 21.3.2.14 Labour Productivity

Productivity factors are used to capture the productivity loss due to conditions experienced in the project's region and specifically the project site. They are used to fill the gap between normal installation or construction periods and the actual time spent. Below is a list of items that can affect lost time or productivity.

- Level of worker skills
- Labour availability
- Union meetings
- Toolbox meetings
- Normal weather pattern for the site
- On-site travel time & Logistics
- Job complexity
- Interference with other crews
- Working hours (Roster)
- Limited and difficult access to area
- Greenfield work
- Site general conditions

Site productivity has been assessed for each discipline; unit-man hours are multiplied by the productivity factors for total manhours per line item.

The assessed productivity factors, by discipline, applied in the cost estimates are shown in the following Table 21-17.

**Table 21-17: Discipline Productivity Factors**

| Discipline | P.F. |
|---|---|
| Earthworks & Civils | 1.20 |
| Concrete | 1.20 |
| Architectural | 1.20 |
| Mechanical equipment | 1.20 |
| Platework | 1.20 |
| Piping - Off-Plot | 1.20 |
| Electrical Equipment | 1.20 |
| Electrical Bulks | 1.20 |
| Instrumentation | 1.20 |

#### 21.3.2.15 Indirect Capital Costs

Indirect costs include all costs that are necessary for project completion but not related to the direct construction cost.

Major cost categories covering the Ausenco indirect costs are listed in Table 21-18.

**Table 21-18: Indirect Capital Cost**

| Indirect Cost | Initial | |
|---|---|---|
| | ($M) | Ratio of Direct Cost (%) |
| Temporary Construction Facilities and Services | 4.5 | 4.2 |
| Commissioning Reps and Assistance | 0.4 | 0.4 |
| Spares | 0.4 | 0.4 |
| First Fills & Initial Charges | 0.2 | 0.2 |
| Project Delivery - EP | 7.7 | 7.0 |
| **Total Indirect Cost** | **13.2** | |

#### 21.3.2.16 Vendors Representatives

Costs for vendor representatives for commissioning have been calculated from the mechanical and electrical equipment supply costs, 1.0% for construction vendor representatives and 1.0% for commissioning vendor representatives have been included in the estimate.

#### 21.3.2.17 Commissioning

Commissioning assistance from mechanical completion to handover was developed along with Ausenco's CM costs. In addition, a modification squad has been included in the estimate.

The contractor support has been carried out to assist the commissioning team to make minor modifications or provide Labour assistance for commissioning. The contractor support allowance has been estimated assuming 6 personnel at 8 h/day for 20 days at $225/h.

#### 21.3.2.18 Spare parts

The following types of spares are typically included with capital cost estimates:

- Commissioning Spares: Major mechanical and electrical spares for commissioning purposes have been factored by 1.5% of the mechanical and electrical supply costs based on Ausenco's historical data.
- Initial Operational Spares: Major mechanical and electrical spares for operational purposes have been factored by 0.6% of the mechanical and electrical supply costs based on Ausenco's historical data.
- Capital (Insurance) Spares: Major mechanical and electrical spares for capital/insurance purposes have been factored by 3.0% of the mechanical and electrical supply costs based on Ausenco's historical data.

#### 21.3.2.19 First Fills

First fills include the costs for the initial construction, installed equipment and process first fills (these consist of fuels, lubricants, etc.).

First fills have been calculated from the mechanical and electrical equipment supply costs; total first fills equate to 1.0% for construction first fills and 1.5% for commissioning/first fills.

#### 21.3.2.20 Project Delivery - EP

EP services cost covers engineering and procurement services (home office-based).

The overall EP budget has been calculated as a percentage of the direct costs. The overall percentage is 7% of the total direct cost which is inclusive of other direct costs and general expenses.

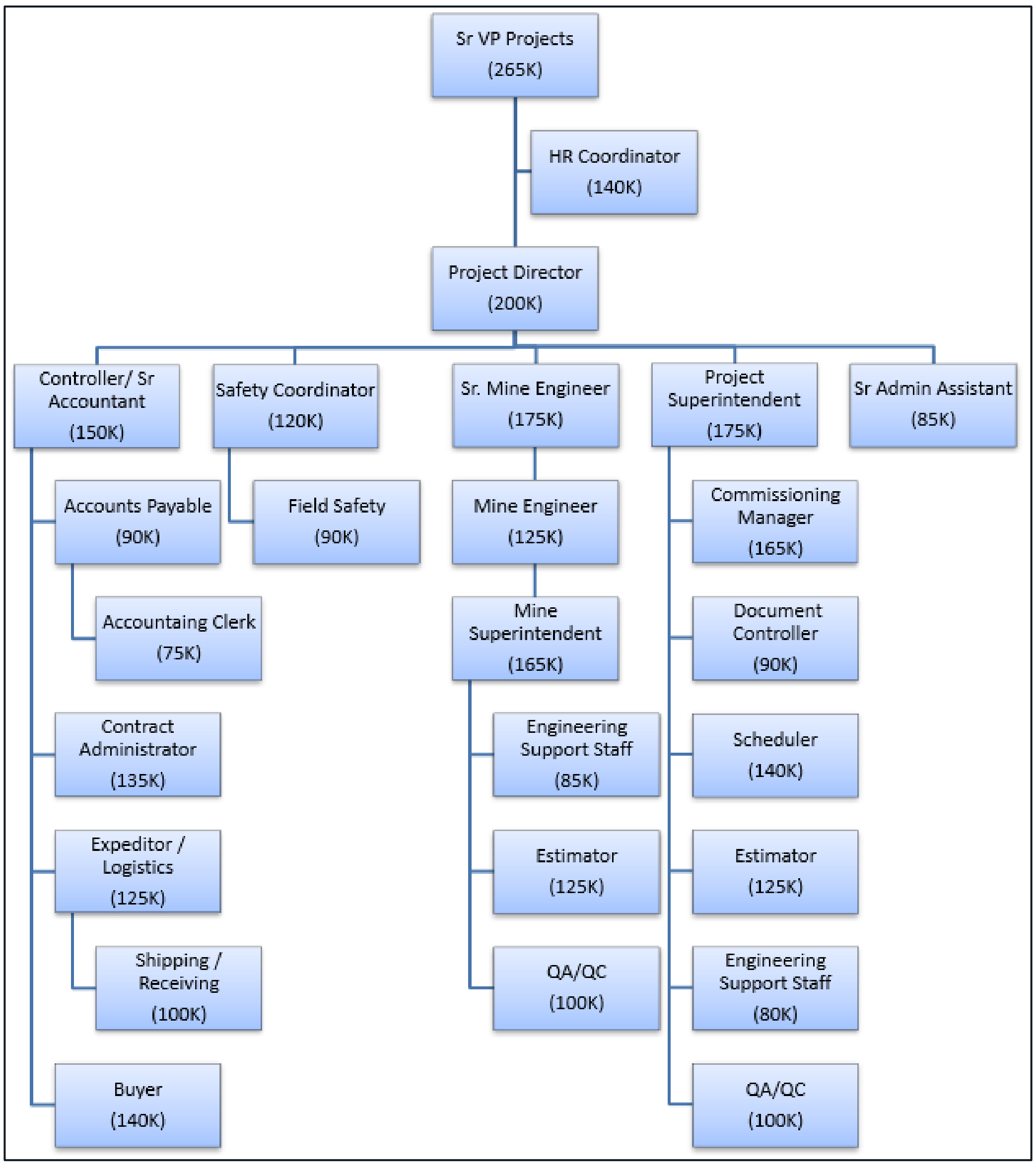
#### 21.3.2.21 Owners Costs

This cost is associated with the direct and indirect costs that the owner will incur to implement and execute Ausenco's scope of work. The G&A portion of the owner's cost has been included as 2% of the total direct cost. This accounts for $2.5M of the Owner's Initial Capital.

For the study, self-performing the construction management of the project is the most efficient way to decrease project timelines and save on project construction cost. The location of the project provides direct access to nearby experienced, well-skilled, and adequately equipped mining and processing resources. The owner-operator construction team will be directly involved overseeing the construction site, and with input from the engineering procurement, selecting experienced local subcontractors. In addition, the owner-operator construction team will improve the timeline and efficiency of the start-up and prove-out of the operations due to familiarity with the installed equipment and infrastructure.

Figure 22-1 illustrates the self-perform organizational structure for construction activities for the Cactus project. Table 21-19 depicts the estimated compensation, which includes benefit packages, for the project team.

**Figure 21-1: Construction Management Organization Chart**




Source: ASCU, 2024.

**Table 21-19: Owner's Construction Costs**

| Position | Annual Salary ($) | Benefits % Allocation | Total Annual Salary + Benefits ($) |
|---|---|---|---|
| Administration | | | |
| Sr. VP Projects | 265 | 20 | 318 |
| Human Resource Coordinator | 140 | 25 | 175 |
| Project Director | 200 | 20 | 240 |
| Sr. Administrative Assistant | 85 | 25 | 106 |
| Accounting Department | | | |
| Controller/Sr. Accountant | 150 | 20 | 180 |
| Accounts Payable | 90 | 25 | 113 |
| Accounting Clerk | 75 | 25 | 94 |
| Contract Administrator | 135 | 25 | 169 |
| Expeditor / Logistics | 125 | 25 | 156 |
| Shipping / Receiving | 100 | 25 | 125 |
| Engineering - Mining | | | |
| Sr. Mine Engineer | 175 | 25 | 219 |
| Mine Superintendent | 165 | 25 | 206 |
| Mine Engineer | 125 | 25 | 156 |
| Estimator | 125 | 25 | 156 |
| QA/QC | 100 | 25 | 125 |
| Engineering Support Staff | 85 | 25 | 106 |
| Safety Department | | | |
| Safety Coordinator | 120 | 25 | 150 |
| Field Safety | 90 | 25 | 113 |
| Subtotal - Annual Salary | 2,350 | | |
| Avg % Benefit Allocation | | 18 | |
| **TOTAL - Salary + Benefits** | | | **2,907** |
| 18-Month Build Schedule | | | 4,360 |

#### 21.3.2.22 Contingency

Contingency is a provision of funds for unforeseen or inestimable costs within the defined project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy and applied to provide an overall level of confidence in costs and schedule outcomes. The contingency is meant to cover events or incidents that occur during the project which cannot be quantified during the estimate preparation and does not include any allowance for project risk.

It is important to note that contingency does not cover scope changes, force majeure, adverse weather conditions, and changes in government policies, currency fluctuations, escalation, and other project risks.

The blended estimate contingency for Ausenco's scope of work is based on a deterministic calculation that equates to 20% of the total direct and indirect costs which is within the typical AACE Class 4 PFS range of 15 to 25% of total direct plus indirect costs. A summary of the contingency is noted in Table 21-20.

**Table 21-20: Contingency by Party**

| Party | Initial ($M) | | Sustaining ($M) | |
|---|---|---|---|---|
| | Contingency | Ratio of Direct + Indirect | Contingency | Ratio of Direct + Indirect |
| Ausenco | 24.7 | 20.0% | 24.2 | 20.0% |
| AGP | 13.6 | 9.0% | 133.8 | 15.0% |
| Samuel Engineering | 37.0 | 26.0% | 8.3 | 26.0% |
| **Total Contingency** | **75.3** | | **166.3** | |

#### 21.3.2.23 Escalation

No escalation has been included in the estimate beyond the base date of 4th Quarter 2023.

#### 21.3.2.24 Qualifications/Assumptions

The assumptions made for the design are as follows:

- All design data meets the requirements of Ausenco standard specifications and discipline design criteria.
- Ausenco has only priced the scope of work as noted in the proposal.
- A construction camp is not required.

### 21.3.3 Mining

The mining capital cost estimate is grouped into three main areas each with its own capital cost categories. They are broken into the following with their range of WBS numbers:

- Open Pit – WBS 2110 to 2430
- Underground (Cactus East) – WBS 2510 to 2580
- Underground (Parks/Salyer) – WBS 2610 to 2680

A summary of the breakdown is shown in Table 21-21.

**Table 21-21: Mine Capital Cost Estimate (US$M)**

| Area | Mining Capital Category | WBS | Initial Cost ($M) | | Sustaining Cost ($M) | Total Capital Cost ($M) |
|---|---|---|---|---|---|---|
| | | | Y-2 | Y-1 | Y 1 – Y 25 | |
| Open Pit | Pre-Production Stripping | 2210 | 6.7 | 71.0 | - | 77.7 |
| | Major Mine Equipment Capital | 2310 | 6.5 | 6.5 | 13.5 | 26.5 |
| | Support/Auxiliary Mine Capital | 2320 | 3.5 | 6.6 | 1.9 | 12.0 |
| | Mine Infrastructure | 2430 | - | 1.1 | 5.1 | 6.2 |
| Cactus East | Portal/Development | 2510,2520 | - | - | 195.6 | 195.6 |
| | Mine Equipment | 2530 | - | - | 40.3 | 40.3 |
| | Mine Ventilation | 2550 | - | - | 20.6 | 20.6 |
| | Mine Dewatering | 2560 | - | - | 3.6 | 3.6 |
| | Mine Electrical | 2570 | - | - | 16.2 | 16.2 |
| | Mine Infrastructure | 2580 | - | - | 64.5 | 64.5 |
| Parks/Salyer | Portal/Development | 2610,2620 | - | 41.0 | 347.7 | 388.7 |
| | Mine Equipment | 2630 | - | - | 86.5 | 86.5 |
| | Mine Ventilation | 2650 | - | 1.3 | 24.5 | 25.8 |
| | Mine Dewatering | 2660 | - | 0.3 | 7.8 | 8.1 |
| | Mine Electrical | 2670 | - | 7.6 | 31.7 | 39.3 |
| | Mine Infrastructure | 2680 | - | 6.5 | 45.8 | 52.3 |
| **Total** | | | **16.7** | **142.0** | **905.4** | **1,064.2** |

Source: AGP Mining, 2023.

#### 21.3.3.1 Open Pit - Pre-Production Stripping (2210)

Mining activity commences in two open pit locations in advance of the processing facility commissioning. This includes the mining of the first phase in Cactus West to prepare for heap leach feed release in Year 1. Portions of the waste from this mining will be used for infrastructure purposes such as heap leach facility construction, roads and the view shed berm. The waste rock storage facility will also be initiated. The Cactus West prestripping includes 23.6 M tons of waste and stockpiling of 0.5 M tons of feed when the crushers are available.

The historic stockpile will also be mined in Year -1 as part of prestripping. This is primarily feed material and will be placed on the heap facility for commissioning purposes. A total of 2.9 M tons will be placed on the pads with 0.2 M tons of waste sent to the waste rock storage facility.

The prestripping cost covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology Labour, grade control costs, and mobilization costs. It also includes any finance costs that have been added to the operating cost for that period.

##### 21.3.3.2 Open Pit Major - Mine Equipment Capital (2310)

The open pit mining equipment capital is determined based on the use of an operating lease. The capital portion of the lease requires a 20% downpayment with the remainder plus a set interest rate applied to the operating cost as a lease payment. Only the downpayment is included in the capital. The major equipment fleet for the Project was purchased in this manner. Certain items such as spare buckets for the shovels and trays for the trucks were not leased but considered as a capital purchase. The total initial capital cost was $13.0M and sustaining capital was $13.5M.

##### 21.3.3.3 Open Pit - Support/Auxiliary Mine Capital (2320)

The mine support equipment was a mixture of capital purchase and lease depending on the unit type and ability to lease that unit. In the case of pickup trucks, buses, road maintenance loaders, etc. these were leased with the appropriate downpayment included in this category. Other items such as the ambulance and firetruck with the necessary supplies were purchased outright. The rough terrain cranes and certain site-specific trucks were also assumed as a capital purchase.

Additional items in this category include the engineering office equipment (computers, drones, mining specific software) as well as the dispatch system hardware purchase. The preparation of the waste storage facility foundation and initial mine access roads are included as capital purchases.

The initial capital for the WBS 2320 category was $10.1M with sustaining capital of $1.9M.

##### 21.3.3.4 Open Pit - Mine Infrastructure (2430)

This category of capital purchase is the dewatering system planned for use in the Cactus West pit and any dewatering needs mining the Historical Stockpile. The assumption is the use of high lift diesel pumps with associated piping. The system cost covers the equipment necessary to move the water to the pit rim where it ties into the proposed project water system for distribution to the appropriate area.

While not required in initial capital, an annual program of horizontal drain holes in the Cactus West pit wall is included in the sustaining capital. This is to assist in the depressurization of the wall slopes to assist in stability.

At the end of the open pit life, an additional dewatering system is included in the sustaining capital to deal with potential storm events that may rapidly fill the open pit and potentially be an issue to the Cactus East underground. This allowance has been made to reduce infiltration to the underground mine adjacent to the pit.

Initial capital costs were $1.1M with sustaining capital totaling $5.1M.

#### 21.3.3.5 Cactus East - Portal/Development (2510,2520)

Costs associated with this category are for the initial preparation of the portal. In comparison to Parks/Salyer, the capital cost needed for the Cactus East development is significantly lower. The cost includes the preparation of the rock face in the existing pit, already in rock, and support around this opening. While a sustaining cost for the project due to its timing, it is part of the initial costs necessary for production from Cactus East. This portion of the cost is estimated at $1.1M for WBS 2510.

The single decline development prior to production and the ongoing development necessary to access the various sublevels is included in WBS 2520. The length of the access decline is 7,440 ft. There is also a portion of the operating cost that has been capitalized as part of the mine development plan. The internal split between initial capital and sustaining capital for Cactus East was considered 2.5 years into the mine development. Small quantities of ore are produced as part of the development in the first two years of underground mining (400,000 tons) but in the third year of development (Year 11 in the overall project) the project ore release rises to 2.1 Mton. This is internally considered to be the split between Initial and Sustaining capital for the Cactus East mine.

The total capital cost in WBS 2520 is $194.5M. Using the logic discussed, the initial capital totals $142.7M. In that total, $73.1M is capital development and the remaining $69.6M is operating cost transferred to capital.

The remaining capital cost, internally called sustaining, is strictly the development charge. This is a further $51.8M over the life of the mine.

It is important to note that waste development is included in capital and ore development is included in the operating costs.

#### 21.3.3.6 Cactus East - Mine Equipment (2530)

Mine production equipment capital costs are included in this category. Major equipment was leased in a manner similar to the open pit major equipment, but with different rates. The vendors of the underground equipment provided terms that allowed for a 15% downpayment, but slightly higher interest rate then used in the open pit equipment.

The mine equipment includes the initial purchase amount as well as any rebuilds during the life of the equipment.

For Cactus East, the life of mine capital cost is estimated at $40.3M. The initial capital during the production ramp up is $15.9M with the remaining $24.4M as sustaining capital for Cactus East.

#### 21.3.3.7 Cactus East - Mine Ventilation (2550)

The cost of ventilation equipment installed in the mine is covered in this category. The vent raises are in development capital. This cost area includes all fans, doors, ducting, and refrigeration necessary to ensure the Cactus East mine is safe and efficient.

Life of mine the capital cost is $20.6M for the necessary items and their installation. Initial capital is estimated to be $13.6M with the sustaining cost at $7.0M.

#### 21.3.3.8 Cactus East - Mine Dewatering (2560)

The mine dewatering system is developed as the mine progresses in depth. The system will use pump stations and transfer pumps to bring the water to the portal. From this point it will be combined with the open pit dewatering system for removal from the area. The costs only consider the system to the portal.

Life of mine the capital cost estimated is $3.6M with $2.6M in initial capital and the remaining million in sustaining.

#### 21.3.3.9 Cactus East - Mine Electrical (2570)

The mine electrical system will carry the loads for the electrical mine equipment but also the ventilation, cooling and vertical conveyor. The system is costed from the portal of the Cactus East mine to the interior of the mine at the various levels. This includes all necessary switchgear and distribution lines.

The life of mine electrical budget for Cactus East is $16.2M with $11.1M initial and $5.1M in sustaining cost.

#### 21.3.3.10 Cactus East - Mine Infrastructure (2580)

Mine infrastructure in Cactus East totals $64.5M life of mine. The majority of the infrastructure cost is in the initial capital period of the first three years and totals $61.2M. This is primarily the cost of the vertical conveyor system and sizer which are commissioned just prior to the major rise in mine feed release in Year 6. The conveyor and sizer total $39.4M. The remaining infrastructure is $21.8M and includes the underground workshop, lunchrooms, refuges, etc.

The sustaining capital is less at $3.3M.

#### 21.3.3.11 Parks/Salyer - Portal/Development (2610/2620)

Parks/Salyer portal development differs from Cactus East as a traditional box cut needs to be excavated for Parks/Salyer. Cactus East benefits from access in the old Cactus West pit to reduce development. The Parks/Salyer decline is a dual decline approach to provide access and a dedicated conveyor access. The access decline is designed at 13,230 ft long and the conveyor decline is 13,040 ft long.

During the development, the two declines are linked to provide cross access and also for ventilation purposes. The decline has a ventilation raise for use during construction due to the length. Four additional raises are located in the footwall of the Parks/Salyer deposit in the upper levels.

Life of mine the total cost for the portal and development is $388.7M due to the size of the Parks/Salyer deposit. The total initial capital cost for development and the portal is $303.8M which includes $3.8M for the portal (WBS 2610) and $156.1M of operating cost capitalized as part of the development. The remaining $84.9M is considered to be sustaining capital.

Like Cactus East, the waste development is included in capital and ore development is included in the operating costs at Parks/Salyer.

#### 21.3.3.12 Parks/Salyer - Mine Equipment (2630)

Mine production equipment capital costs are included in this category in the same manner as Cactus East. Major equipment was leased in a manner with the vendors of the underground equipment provided terms that allowed for a 15% downpayment, but slightly higher interest rate then used in the open pit equipment.

The mine equipment includes the initial purchase amount as well as any rebuilds during the life of the equipment.

For Parks/Salyer, the life of mine capital cost is estimated at $86.5M. The initial capital during the production ramp up is $8.1M with the remaining $78.4M as sustaining capital for Parks/Salyer. This is different to Cactus East as a contract road header is used on the conveyor drift development in the initial capital and helps defer the cost of equipment.

#### 21.3.3.13 Parks/Salyer - Mine Ventilation (2650)

The larger areal extent and depth of the Parks/Salyer deposit requires more capital for ventilation and cooling than Cactus East although the dual decline system helps in managing the cost. The vent raises are also in development capital. This cost area includes all fans, doors, ducting, and refrigeration necessary for the Parks/Salyer mine.

LOM capital cost is $25.8M for the necessary items and their installation. Initial capital is estimated to be $24.2M with the sustaining cost at $1.6M.

#### 21.3.3.14 Parks/Salyer – Mine Dewatering (2660)

Low expected water inflows help keep the Parks/Salyer dewatering cost lower. The same type of infrastructure is developed like Cactus East with the sump stations, pumping and piping. It has been sized to accommodate 3 times the expected inflow but has less of a concern of water ingress from an open pit as none is adjacent.

LOM cost is $8.1M with initial costs of $5.5M and sustaining of $2.6M.

#### 21.3.3.15 Parks/Salyer – Mine Electrical (2670)

The mine electrical system for Parks/Salyer includes the normal ventilation, cooling and equipment loads but also includes the conveying system from the mine which is much longer than the vertical system employed at Cactus East. The system is costed from the twin portals to the interior of the mine at the various levels. This includes all necessary switchgear and distribution lines and substations at the various levels.

The LOM electrical budget for Parks/Salyer is $39.3M with $33.2M initial and $6.1M in sustaining cost.

More detail on the electrical design is provided in Section 16.

#### 21.3.3.16 Parks/Salyer – Mine Infrastructure (2680)

Parks/Salyer Mine infrastructure in totals $52.3M life of mine and includes the conveyor for ore transportation.

The underground conveyor is a major part of the cost life of mine with initial conveyor costs of $16.5M then extensions to the system adding another $3.1M for a total conveyor cost of $19.7M. The conventional conveyor cost is much lower and higher has a higher capacity than the vertical conveyor system employed at Cactus East.

The remaining mine infrastructure capital is $32.6M life of mine with $27.9M being initial capital for the underground workshop, lunchrooms, refuges, etc. The sustaining portion of this is $4.7M.

### 21.3.4 Processing Facilities

Samuel Engineering (SE) has been tasked to develop a +/-25% accuracy total installed cost (TIC) estimate for ASCU. The capital cost addresses the development, construction, and start-up of a greenfield plant capable of initially producing 30,000 tons/y of Grade A LME copper cathode from the existing stockpile, open pit, and underground resources and ramping up to produce over 60,000 tons/y. The key objectives of the capital cost estimate are to:

- Support the economic evaluation and assessment of the Project.
- Identify and assess the processes and facilities that will provide the most favorable return on investment.
- Establish a budget for financing and forecasting.

The total estimated capital cost to design, procure, construct, and commission the facilities described in this basis of estimate is summarized by WBS/Area in Table 21-22.

**Table 21-22: Capital Cost Summary – New Equipment**

| Commodity | Total Cost ($M) |
|---|---|
| Bulk Earthworks (Rough Grading) | 1,186,072 |
| Administrations building | 1,280,000 |
| Power Supply to Process Plant | 529,448 |
| Fine Crushed PH1 Leach Pad | 6,650,154 |
| Crushing and Conveying | 52,570,765 |
| Fine Crushed PH1 HLP PLS Pumping | 5,730,486 |
| Raffinate Pumping | 7,704,038 |
| Fine Crushed Leach Pad Event Pond | 63,823 |
| Solvent Extraction | 24,537,688 |
| Tank Farm | 5,931,027 |
| Electrowinning | 17,634,631 |
| Cathode Storage | 2,618,215 |

| Commodity | Total Cost ($M) |
|---|---|
| Electrowinning Building | 6,016,194 |
| Sulfuric Acid | 656,488 |
| Guar | 175,003 |
| Cobalt | 164,502 |
| SX Diluent | 158,297 |
| SX Extractant | 34,628 |
| Plant Water Services | 2,996,423 |
| Air Services | 536,643 |
| **Directs Total** | **137,174,525** |
| Contractor Indirects | 5,455,794 |
| Construction Equipment | 4,650,020 |
| Construction Permits | 1,028,809 |
| Engineering And Procurement | 9,602,217 |
| Construction Management & Commissioning | - |
| Pre-Operational Testing | 409,185 |
| Vendor Representatives | 1,083,310 |
| Freight | 4,349,793 |
| Spare Parts | 2,366,620 |
| First Fills | 1,500,000 |
| Owners Cost | - |
| Contingency | 41,905,068 |
| **Total** | **209,525,340** |

Late in the Prefeasibility Phase, a used crushing plant with all plant conveyors and an overland conveyor was discovered to be available. The equipment is located at the Trekkopje Mine in Namibia and ASCU immediately began working to secure the plant or a portion of it. The used plant has a crushing capacity of about 7 ktph which is twice what is required for the Cactus Mine project. The capital estimate in Table 21-21 is a summary of the process equipment which includes the used purchase price for the crushers and conveyors from the Trekkopje site. Disassembly of the used equipment is being negotiated by ASCU with the equipment provider and not included in Table 21-24. These costs were incorporated into the financial model. Purchasing the used equipment presents an opportunity to save upwards of twenty-five million dollars once disassembly is considered.

**Table 21-23: Capital Cost Summary – Used Equipment from Trekkopje Mine**

| Commodity | Total Cost ($M) |
|---|---|
| Bulk Earthworks (Rough Grading) | 1,186,072 |
| Administrations building | 1,280,000 |
| Power Supply to Process Plant | 529,448 |
| Fine Crushed PH1 Leach Pad | 6,650,154 |
| Crushing and Conveying | 29,116,788 |
| Fine Crushed PH1 HLP PLS Pumping | 5,730,486 |
| Raffinate Pumping | 7,704,038 |
| Fine Crushed Leach Pad Event Pond | 63,823 |
| Solvent Extraction | 24,537,688 |
| Tank Farm | 5,931,027 |
| Electrowinning | 17,634,631 |
| Cathode Storage | 2,618,215 |
| Electrowinning Building | 6,016,194 |
| Sulfuric Acid | 656,488 |
| Guar | 175,003 |
| Cobalt | 164,502 |
| SX Diluent | 158,297 |
| SX Extractant | 34,628 |
| Plant Water Services | 2,996,423 |
| Air Services | 536,643 |
| **Directs Total** | **113,720,548** |
| Contractor Indirects | 5,455,794 |
| Construction Equipment | 4,650,020 |
| Construction Permits | 852,904 |
| Engineering And Procurement | 7,960,438 |
| Construction Management & Commissioning | - |
| Pre-Operational Testing | 409,185 |
| Vendor Representatives | 1,083,310 |
| Freight | 5,425,793 |
| Spare Parts | 2,366,620 |
| First Fills | 1,500,000 |
| Owners Cost | - |
| Contingency | 35,856,153 |
| **Total** | **179,280,764** |

#### 21.3.4.1 Scope

The Cactus Mine Project is located midway between Phoenix and Tucson, Arizona in the American southwest copper belt. Paved highways, rail, and power access are immediately adjacent to the property. The property is the former Sacaton Mine operated by ASARCO, which is currently in the closure and reclamation stage. Existing stockpile material and the Cactus West open pit at site will provide a source of feed for the lower grade, coarse crush heap leach. Additional feed for the high-grade, fine crush heap leach will come from underground expansion into the neighbouring Parks/Salyer property and the Cactus East deposit. Combined the current resource totals 275 Mton of soluble copper.

The initial solvent extraction and electrowinning (SX/EW) facility will be designed to produce 30,000 tons/d of copper cathode. The SX/EW plant will expand the electrowinning capacity, which will impact the overall capacity of the plant bringing it to 60,000 tons/d by Year 4 of production. Copper recovery from the heap leach stacked ore is expected to be 84% for high-grade material and 82% for lower grade stockpile feed.

The processing plant and heap leach operations for this estimate include:

- Crushing, conveying, and stacking equipment.
- Pumping for raffinate and PLS.
- Solvent Extraction and Tank Farm.
- Electrolyte Filtration.
- Electrowinning and Refining.
- Reagents; and
- Associated MCC’s to support the above areas.

All ore will be fed through a primary sizer, then conveyed to the secondary and tertiary crushing circuits where the ore will be crushed down to a $P_{80}$ of ¾ in. Once crushed, the ore will be agglomerated within two agglomeration drums before transferred to the overland tripper conveyor and then out to the leach pad. Mobile grasshopper conveyers will transfer the ore across the leach pad to the index conveyor and radial stacker. The stacking system will recede back towards the tripper conveyor has material is placed on the leach pad. Whether the new or used equipment is purchased, the crushing and conveying operation will remain the same as described above.

A PLS pond and event pond will be located below the leach pad to collect PLS coming from the pad, where it will be pumped to the PLS pond at the SX/EW plant. The enriched pregnant solution will then be introduced to the solvent extraction and electrowinning process producing a saleable copper cathode product.

The estimate is based on the outlined scope of work and as defined by the following:

- Process Design Criteria
- Process Flow Diagrams

- Mechanical Equipment List
- Mechanical equipment specifications
- Plot plans
- Civil model development
- Electrical one-line diagrams
- Electrical Equipment List
- Material take-offs (MTOs), generated from GA models and layouts
- Quotations from vendors and contractors
- In-house historical data and database information

#### 21.3.4.2 Exclusions

Items not included in the capital estimate are as follows:

- Site access road construction or upgrades
- Underground mine development or Mine costs (portals, access roads, facilities, etc.)
- Leach Pad and Ponds
- Substation
- Rail System
- Landfill facility, if required
- Owner's costs
- Taxes
- Sunk costs (costs prior to start of detailed design)
- Disposal/clean-up of hazardous materials (none have been identified)
- Allowance for special incentives (schedule, safety, etc.)
- Interest and financing cost
- Working capital and sustaining capital
- Closure, reclamation, and salvage costs
- Escalation beyond 4$^{th}$ quarter 2023
- Risk due to political upheaval, government policy changes, labour disputes, permitting delays or any other force majeure occurrences, including COVID-19.

#### 21.3.4.3 Currency

The estimate is expressed in third-Quarter 2023 United States dollars. No provision has been included to offset future escalation.

The value of the US dollar against other world currencies could influence future project cost depending upon the location of equipment sourcing. No allowance has been made in the estimate to offset potential currency fluctuations.

#### 21.3.4.4 Estimating Methodology

The estimate is built up by cost centres as defined by the project's WBS for Area designations as well as by prime commodity accounts, which include earthwork, concrete, structural steel, buildings, mechanical equipment, piping, electrical and instrumentation.

Table 21-24 is the responsibility matrix for the Project. Only items for which SE is responsible for are included in this estimate.

**Table 21-24: Capital Cost Estimate Contributors**

| Scope/Responsibility | Firm |
|---|---|
| Site Infrastructure | Ausenco |
| Mine | AGP |
| Leach Pads & Ponds | Ausenco |
| Process Plant | Samuel Engineering |
| Substation | Ausenco |
| Utilities | Ausenco |

Imperial units of measure have been used throughout the estimate.

Costs were derived from various sources including unit rates provided by contractors, equipment and material quotations, in-house historical data, published databases, factors and estimators' judgment (allowances).

The estimate assumes that equipment and materials will be purchased on a competitive basis, and installation contracts will be awarded in defined packages on a time and materials, unit price or lump sum basis. It is also assumed that any equipment listed on the Mechanical and Electrical Equipment Lists will be purchased by the Owner, or Owner's Agent, without markup and be provided (free issue) to the construction contractor(s) for installation. It is anticipated that the construction contractors will supply most of the bulk materials.

Manufacturer's standard designs and warranties on equipment and materials are assumed to be satisfactory.

In Section 21.3.4.5 through Section 21.3.4.28, is a discussion of how various estimating methodologies have been applied within each commodity or WBS group.

#### 21.3.4.5 Site Civil Work

Existing topographical mapping and data of the site's terrain is how SE based the civil design for the process facility and equipment. Site benches for the process facility and crushing areas were developed utilizing 3-dimensional general arrangement models of the new Facility.

Cut and fill quantities have been developed from three-dimensional topographic models for both existing and proposed surfaces using Digital Terrain Models (DTMs). The triangle computations calculated the difference between the existing ground surface and the proposed designed surface.

The civil work includes:

- Clear and grub
- Topsoil stripping and stockpiling
- Excavation and embankment placement
- Erosion sedimentation control
- Process area drainage conveyance
- Surfacing

Approximately 10 acres need to be cleared and grubbed to account for the plant area. Additionally, the primary crushing area and screening and secondary crushing areas will account for roughly 50 acres.

Since it is a brownfield site all material is rippable for the bench development for the above facilities, and no drilling and blasting is required outside of the mine pit.

The process plant location is relatively flat with limited earthwork for the pad construction once the area is cleared and grubbed. All fill material needed will be derived from the cut activities being performed and from on-site borrow sources. Depending on the material encountered during construction, both cut and fill slopes may require adjustment based on soils stability and erosion maintenance. Cut ditches will be employed at the top of cut slopes to both minimize erosion along the slope and divert water prior to impacting the slope. Cut slope benching will be required where cut slopes exceed five meters (5 m) in elevation.

Base course surfacing has been included for the plant road and bench. It is assumed all surfacing material will be sourced on-site. No allowance has been made for encountering hazardous waste or other buried items.

The cost for performing topsoil stockpiling, surfacing, and mechanically stabilized earth (MSE) works is based on in house unit rates for work in the region. Remaining civil work activities are based on US average unit rates from RSMeans Construction database.

##### 21.3.4.6 Concrete

Concrete quantities were derived from the plant model and include foundations for major equipment, pipe racks and buildings. There is approximately 9,000 $yd^3$ of cast-in-place concrete. The average cost used in the estimate is \$2,000 $yd^3$. Unit rates are assumed to include installation labour, forming, rebar, batch plant, cement, aggregate, stripping, backfill as well as contractor overhead and profit.

No seismic factors have been accounted for in the concrete quantities.

No concrete coatings have been included for the EW area.

##### 21.3.4.7 Structural Steel

Tonnages for structural steel encompass equipment supports, pipe racks, access structures and platforms, grating and handrail. Structural steel quantities were derived from the plant model and include major facilities, accessways, equipment supports and pipe racks. Allowances have been made for areas in which no modeling is available.

There are approximately 60 tons of steel. The steel consists of individual equipment supports, access platforms and miscellaneous pipe and electrical supports.

Unit rates for steel fabrication have been included at an average rate of \$5500/ton and are based on total weight. The unit rate includes materials, detailing, fabrication, prime painting, and finish coat epoxy painting and is based on recent in-house data.

##### 21.3.4.8 Buildings

Given the proximity to the city of Casa Grande, limited non-process facilities are required. Corporate administration will be stay in Chandler, Arizona. The scope of work for the ancillary facilities includes the following buildings:

- Process (SX/EW) Building
- SX/EW Site Office, Control Room, and Security
- Truestone building – being refurbished and reused for site administration.

Pricing for the buildings is based on a budgetary quotation for pre-engineered metal buildings (shell only). Where needed, pricing has been adjusted to account for changes in building dimensions. The cost of the building includes supply, erection, and transportation to site.

Interior finishes and services (walls, ceilings, fire protection, plumbing, electrical) are based on square meter costs from published databases. No allowances have been made to account for noise reduction within the buildings.

The process building will only have a tenant finish in the control room. Costs for control room hardware are included with the Instrumentation and Automation costs.

#### 21.3.4.9 Mechanical

Pricing for mechanical equipment is from vendor quotations as specifications or data sheets were prepared for all major equipment. Quotations (+/-20%) have been obtained for approximately 89% (by value) of the new equipment itemized on the mechanical equipment list. The remaining equipment costs are based on budgetary vendor quotations, historical data, or allowances.

New equipment quotes include drives and motors where drives and motors are required, as well as on-equipment instrumentation, on-skid piping, etc. Some vendors provided separate equipment cost, spare parts cost, vendor erector hours and cost, and packaging and shipping costs. Costs used in the estimate are from SE recommended vendors, plus any optional scope of supply costs where applicable.

Prices include vendor engineering, documentation and tagging, as well as SE recommended options and adders. FOB factory pricing is included in the direct cost section and all freight has been allocated separately in the indirect section of the estimate.

It is the intent of the direct man-hour quantities to cover all required operations for the installation of individual pieces of equipment. This includes unloading from delivery trucks, storing in yard or warehouse, unpacking, hauling to erection site, rigging, lifting, setting, aligning, calibrating, and checking out of all items included with the equipment supply.

Mechanical man-hours exclude installation of piping and electrical items and hook-up.

Anchor bolts, grout, and shims are accounted for in the materials portion of the mechanical equipment section as an average of 2% of the mechanical equipment cost per plant area.

Pricing for the used crusher and conveying equipment was obtained by ASCU from the Trekkopje Mine site. The used equipment capital cost estimate was developed based on these used crusher and conveyor unit price quotes. Samuel Engineering confirmed that the capacities of the crushers and conveyors equaled or exceeded the design capacities of the new equipment specifications that were develop for the project. To develop the used equipment capital cost, estimate the same estimate structure was used as the new equipment capital cost estimate, only new equipment unit rates were replaced by the used equipment unit rates. All factors for construction costs were left unchanged during the analysis. Disassembly of the equipment at the Trekkopje Mine site is not included in the capital cost build up in this section but is accounted for in the financial model.

#### 21.3.4.10 Piping

Material take-offs were developed for the piping and quoted. All major pipe runs for overland piping are also included. Allowances have been made for valves and fittings not included in the take-off.

#### 21.3.4.11 Electrical

The capital cost includes electrical equipment necessary to operate the ancillary facilities and process areas, as well as equipment necessary to distribute power from the new APS Substation to all areas of the project. This cost estimate does not include electrical equipment costs for the underground mining facilities. The capital cost estimate for this portion of the project will be provided by AGP as part of their scope of supply.

Major electrical equipment included in this cost estimate has been priced with budgetary vendor quotations. Prices include vendor engineering, documentation and tagging. Freight costs have not been included. Quotes are FOB factory, and all freight has been allocated separately in the indirect section of the estimate.

The project will utilize Power Distribution Centres (PDCs), each housing the main electrical equipment for their areas. The PDCs will be prefabricated buildings with all equipment, including battery systems, motor control centres, relay panels and switchgear, pre-installed & wired at the factory prior to shipment. (Due to the size, some PDCs may be shipped in multiple sections.) Hours for on-site electrical equipment installation are not included in this estimate.

No material take-offs have been generated for bulk materials. All material costs for grounding, wire, terminations, cable tray, conduit, lighting, lightning protection, etc. have been factored at 5% of mechanical equipment. Labour is included at 75% of the material cost.

#### 21.3.4.12 Instrumentation & Controls

No material take-offs have been generated for instrument wiring, control cable, communication cables, etc. These material costs have been factored at 1.5% of mechanical equipment. Labour is included at 40% of the material cost.

The operator and engineering control stations for the Facility will be in the main process building in a dedicated control room. It is assumed that there will be two Distributed Control System consoles. Costs for DCS consoles, main control panel, the network rack, Remote I/O (RIO) panels and DeltaV software licensing is from a vendor quotation based on a preliminary controls system specification. RIO panels will be installed in the PDCs, and other high density I/O areas strategically located throughout the plant site. Field instruments will be routed to the RIO panel located in their respective area. The network rack and control panel will be installed in the control room.

#### 21.3.4.13 Labour Rates & Productivity

The man-hours associated with materials and equipment are based on North American standard productivity. Hours cover all necessary operations for the installation of individual components. This includes unloading storage (yard or warehouse), unpacking, hauling to erection site, rigging, lifting, setting, welding, aligning, calibrating, and checking out of all items included with the supply.

Average labour billing rates for a local contractor are based upon data from RSMeans for third quarter 2023 with a location factor for Arizona. These rates do not include contractor field indirect costs such as mobilization, demobilization, temporary facilities, temporary utilities, surveying, or on-site administration. These items are included with the construction indirect cost. Rates for each commodity type are a blending of various labour and skill levels to derive reasonable crew mixes. Table 21-25 shows the average crew rates by discipline:

**Table 21-25: Average Crew Labour Rates**

| Discipline | All-in rate ($) |
|---|---|
| Earthwork | 62.96 |
| Concrete | 67.74 |
| Steel | 67.63 |
| Buildings | 66.47 |
| Mechanical | 67.84 |
| Piping | 69.20 |
| Electrical | 70.49 |
| Instrumentation | 73.36 |

The wage rates reflect a 6-day workweek of 60 hours and include the following:

- Basic wage
- Overtime premium for 50-hour work week
- Payroll burdens and benefits
- Small Tools and Consumables
- Safety
- Transportation to Site
- Overhead and profit

The labour rates do not include a construction camp or catering due to the proximity of the project to large cities.

#### 21.3.4.14 Common Distributable and Contracted Indirect Costs

Common distributable and contracted indirect costs apply to multiple parties (suppliers, contractors, service providers, etc.) across multiple areas of the project. Various methodologies have been used to arrive at costs which are discussed below.

#### 21.3.4.15 Contractor Indirect Costs

The construction contractor's indirect costs for the Facility have been included at an overall rate of 38% of the direct field labour cost. Items included with contractor's indirect costs include:

- Mobilization and demobilization

- Supervision, safety, and administrative support costs
- Maintenance of temporary construction facilities (offices, etc.)
- Warehousing and distribution control
- Temporary toilets maintenance
- Construction personnel vehicles, fuel, and maintenance
- Construction power/utilities hook-up
- Heating and hoarding
- Dust suppression and snow removal
- Cleanup and waste removal
- Insurances
- Temporary communications
- Construction surveying

It is assumed that no performance bond will be required of contractors.

It is assumed that construction power and water will be provided to the contractors free of charge by ASCU. Contractors will be responsible for temporary utility tie-ins and distribution to work areas.

#### 21.3.4.16 Construction Equipment

intended to cover the cost of heavy lifting cranes, forklifts, man-lifts, flat-bed trucks, gensets, scissor lifts, dewatering pumps, light plants, scaffolding, etc. for both contractor owned and rented equipment plus fuel and maintenance.

Where subcontract unit rates have been used (earthwork, etc.) the cost for construction equipment is included in the various subcontract unit rates. Construction equipment costs are included at $14.00 per direct manhour.

#### 21.3.4.17 Construction Permits

Construction permits required during construction have been included at 0.75% of the total direct costs.

#### 21.3.4.18 Engineering and Procurement Services

Costs for engineering and procurement services have been included at seven percent (7%) of the direct cost.

#### 21.3.4.19 Construction Management and Commissioning Services

All construction management and commissioning services have been included under the Owner's Cost and developed by ASCU.

#### 21.3.4.20 Pre-Operations Testing

An allowance of 3% of the direct labour has been included to cover the cost of pre-operational craft labour activities. The cost for the pre-operations testing manager and engineering team is assumed to be included with the construction management cost.

The operations personnel, utilities and reagents expended during this period are considered an operating cost and are not included in the estimate.

#### 21.3.4.21 Vendor Representatives

Vendor representative assistance has been included for supervision of specialized equipment installation, pre-commissioning, commissioning, startup, and testing. An allowance of 2% of mechanical equipment costs is included for installation supervision and commissioning support. This is intended to cover labour, travel expenses, per diem, meals, accommodations, etc.

#### 21.3.4.22 Freight

Transportation costs have been included for the delivery of equipment and materials to the site. No considerations have been made for road improvements, bridges, power lines, etc. for transportation routes. Freight cost has been included at 8% of equipment cost or 6% of material cost, where applicable.

#### 21.3.4.23 Spare Parts

An allowance of 4% of the mechanical equipment has been allocated for spare parts inventory. It has been assumed that spare parts required for start-up and commissioning are included. One-year spare parts are not included in the estimate.

#### 21.3.4.24 Initial Fills

Initial fills for reagents and other consumables have been allocated a cost of $1.5M based on benchmarking from other projects of similar size. Lubricants, for gearboxes, and other miscellaneous initial fill consumables are assumed to be included in this allowance.

#### 21.3.4.25 Owner's Costs

Owner's costs are excluded from this estimate. The categories ASCU should consider accounting for include, but are not limited to, the following:

- Project Insurances
- Owner's Project Management Team
- Travel Expenses
- Start-up Operations Staff
- Owner's Commissioning Team
- Temporary Facilities
- Camp and catering services
- Construction Water and Power
- Administrative Offices
- Pickup Trucks for CM staff
- Medical & Safety Supplies for CM staff
- Training
- Corporate Services
- Environmental (Monitoring, Reporting, Preservation, Studies, Etc.)
- Permitting (other than construction permits)
- Legal Services and Fees
- Community Relations

#### 21.3.4.26 Taxes

Taxes are excluded from the estimate.

#### 21.3.4.27 Contingency

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 contingency has been assessed by considering the quality of quantities, scope definition and pricing obtained for each commodity of the estimate. Each component is assigned a percentage rate based on the best judgment of the project team.

In recognition of the degree of detail on which the estimates are based, a contingency of 20% has been included.

##### 21.3.4.28 Accuracy

The estimate has been developed to a level sufficient to assess/evaluate the project concept, various development options and the overall project viability. After inclusion of the recommended contingency and excluding any scope changes, the capital cost estimate is considered to have a level of accuracy in the range of -25% to +25%.

Estimate accuracy ranges are projections based upon preliminary designs, assumptions and cost estimating methods and are not a guarantee of actual project cost.

### 21.3.5 Closure Costs

Estimated closure requirements inclusive of all necessary demolition, rehabilitation, revegetation, earth grading/contouring, scrap metal disposal/tipping fees, as well as post-closure monitoring. The total closure cost was calculated to be US$23M, with salvage credits of US$97M.

## 21.4 Operating Cost Estimates

### 21.4.1 Operating Cost Estimate Summary

The project OPEX estimate encompasses mine operating costs, process plant operating costs, and general and administrative (G&A) costs. Cash costs are expressed in dollars per short ton ($/t) of heap feed or dollars per pound of ($/lb) cathode produced. Total cash costs encompass royalties and transportation charges. Additionally, the All-In Sustaining Costs (AISC) and the All-In Costs (AIC) incorporate non-sustaining Capex, closure, and reclamation CAPEX, respectively. A summary of these costs is presented in Table 21-26.

**Table 21-26: Operating Cost, AISC and AIC Summary**

| Total Production Cost Item | LOM | | |
|---|---|---|---|
| | ($/t Placed) | ($/lb Cathode Produced) | ($M) |
| Mining | 11.51 | 1.38 | 3,180 |
| Processing | 2.93 | 0.35 | 809 |
| Infrastructure | 0.03 | 0.00 | 8 |
| G&A | 0.12 | 0.01 | 32 |
| **Cash Cost** | **14.58** | **1.75** | **4,029** |
| Royalties | 0.79 | 0.09 | 218 |

| Total Production Cost Item | LOM | | |
|---|---|---|---|
| | ($/t Placed) | ($/lb Cathode Produced) | ($M) |
| Refining and Transportation | 0.00 | 0.00 | 0 |
| **Total Cash Cost** | **15.37** | **1.84** | |
| Sustaining CAPEX | 4.42 | 0.53 | 1,221 |
| Reclamation and Closure | 0.08 | 0.01 | 23 |
| Salvage | 0.35 | 0.04 | 97 |
| **All-In Sustaining Costs** | **19.52** | **2.34** | |
| Property Taxes | 0.69 | 0.08 | 191 |
| Initial (non-sustaining) CAPEX | 1.86 | 0.22 | 515 |
| **All-In Costs** | **22.08** | **2.64** | |

### 21.4.2 Mining

The mine operating costs for both the open pit and underground estimates have been estimated from local mine equipment vendors and include locally supplied consumables.

The underground costs have been detailed in Section 16 and a summary will be provided here. The open pit operating costs will be discussed in detail in this section. Both areas consider leasing as part of the cost strategy for their respective fleets with the lease payments included in the operating costs.

The open pit equipment is diesel powered. The underground equipment is a mix of diesel and electric equipment. The diesel price for the operating cost estimate is $3.49/ gal.

#### 21.4.2.1 Open Pit Operating Cost

The open pit operating cost covers the mining of the Cactus West Pit, the historic Stockpile and rehandle as part of the mine schedule. Material movement within the mining cost is used to build the view shed berm, haul roads for mine use and placement and compaction of the base layer of the heap leach facilities. This material is considered free issue to the project construction team.

The vertical conveyor at Cactus East transports ore to the surface where it is placed on an overland conveyor for transport to the crushers. At various times in the mine schedule that conveyor is not available, and the mine will be responsible for moving the underground ore to the crushers. This occurs while the Cactus East underground mine is being developed. The mine fleet will be used to transport that material to its final destination.

##### 21.4.2.1.1 Labour

Labour costs for the various job classifications were obtained from review of other operations and discussion with their personnel. A burden rate of 30% was applied to the staff rates and 40% to the hourly rates. Labour was estimated for both staff and hourly on a 12-hour shift basis. Mine staff positions and salaries are shown in Table 21-27.

The mine staff Labour remains constant from Year 1 until Year 4, when positions are removed as the open pit mining winds down.

Hourly employee Labour force levels in mine operations and maintenance fluctuate with production requirements. The hourly Labour requirements for Year 3 are shown in Table 21-28. Labour costs are based on Owner-operated mining with Arizona Sonoran responsible for the equipment with its own employees.

Overseeing all the mine operations is the General Manager who is costed as part of the G&A and not the mining cost. This person would have the maintenance and pit shift supervisors and the Chief engineer and Chief geologist reporting directly to them.

The mine will have four mine operations crews. Over the mine life, there will also be a road crew/services supervisor responsible for roads, drainage, and pumping around the mine. This person would also be a backup mine shift supervisor. There are four junior shift supervisors from Year 1 to Year 4 due to the large area and volume of material being moved. The mine operations department will have its own administration clerk.

**Table 21-27: Hourly Labour Requirements and Annual Salaries (Year 3)**

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| **Mine Maintenance** | | |
| Maintenance Shift Supervisor | 4 | 150,800 |
| Maintenance Planner/Contract Administration | 2 | 121,300 |
| Clerk | 1 | 74,400 |
| Subtotal | 7 | - |
| **Mine Operations** | | |
| Mine Shift Supervisor | 4 | 149,800 |
| Junior Shift Supervisor | 4 | 130,000 |
| Road Crew/Services Supervisor | 1 | 149,800 |
| Clerk | 1 | 74,400 |
| Subtotal | 10 | - |
| **Mine Engineering** | | |
| Chief Engineer | 1 | 158,000 |
| Senior Engineer | 1 | 136,500 |
| Open Pit Planning Engineer | 1 | 113,500 |
| Blasting Engineer | 1 | 113,500 |
| Blasting/Geotechnical Technician | 1 | 83,500 |
| Surveyor/Mining Technician | 2 | 98,200 |
| Surveyor/Mining Technician Helper | 2 | 83,500 |
| Subtotal | 9 | - |

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| **Geology** | | |
| Chief Geologist | 1 | 158,000 |
| Senior Geologist | 1 | 136,500 |
| Grade Control Geologist/Modeller | 1 | 113,500 |
| Sampling/Geology Technician | 2 | 98,200 |
| Subtotal | 5 | - |
| **Total** | **31** | |

**Table 21-28: Hourly Labour Requirements and Annual Salaries (Year 3)**

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| **Mine General** | | |
| General Equipment Operator | 4 | 98,900 |
| Road/Pump Crew | 4 | 86,300 |
| General Mine Labourer | 4 | 67,900 |
| Trainee | 4 | 65,900 |
| Tire Technician | 4 | 86,300 |
| Light Duty Mechanic | 2 | 107,100 |
| Lube Truck Driver | 4 | 86,300 |
| Subtotal | 26 | - |
| **Mine Operations** | | |
| Driller | 32 | 90,600 |
| Blaster | 2 | 107,100 |
| Blast Helper | 4 | 85,700 |
| Loader Operator | 20 | 111,100 |
| Hydraulic Shovel Operator | 8 | 111,100 |
| Haul Truck Driver | 100 | 86,300 |
| Dozer Operator | 14 | 107,100 |
| Grader Operator | 6 | 107,100 |
| Crusher Loader Operator | 3 | 107,100 |
| Water Truck | 6 | 86,300 |
| Subtotal | 195 | - |
| **Mine Maintenance** | | |
| Heavy Duty Mechanics | 47 | 107,100 |
| Welder | 29 | 107,100 |
| Electrician | 2 | 111,100 |
| Apprentice | 7 | 89,000 |

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| Subtotal | 85 | - |
| **Total Hourly** | **306** | |

The chief engineer will have one senior engineer and one open pit engineer reporting to them. The blasting engineer would be included in the short-range planning group and would double as geotechnical engineer as required.

The short-range planning group in engineering will have two surveyor/mine technicians and two surveyors/mine helpers. These employees will assist in the field with staking, surveying, and sample collection with the geology group.

In the geology department, there will be one senior geologist reporting to the chief geologist. There will also be one grade control geologist/modeller. Between the senior geologist and the grade control geologist they will manage the short range and grade control drilling, and long range/reserves. There will also be two grade control/sampling technicians.

Four mine maintenance shift supervisors will report to the Mine General Manager. There will be two maintenance planners/contract administrators and a clerk.

The hourly Labour force includes positions for the light duty mechanic, tire technician, and lube truck drivers. These positions will all report to maintenance. There will generally be one of each position per crew. Other general Labour includes general mine Labourers (one per crew) and trainees (one per crew until Year 4) plus four road/pump crew personnel per crew for water management/road maintenance.

The drilling Labour force is based on one operator per drill, per crew while operating. This peaks at 32 drillers in Year 2 and holds until Year 4 then drops slowly to 12 in Year 6 the last year of drilling required.

Shovel and loader operators peak at 28 in Year 2 and Year 3 and then starts to decline until the end of the mine life. Haulage truck drivers peak at 100 in Year 2 and Year 3 and then tapers off to the end of the mine life.

Maintenance factors are used to determine the number of heavy-duty mechanics, welders and electricians are required and are based on the number of equipment operators. Heavy duty mechanic requirements work out to 0.25 mechanics required for each drill operator for example. Welders are 0.25 per operator and electricians are 0.05 per operator.

The number of loader, truck and support equipment operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit, to match the mine crews.

#### 21.4.2.1.2 Equipment Operating Costs

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Cactus project. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the R&M are expressed in US dollars per hour.

Tire costs were also collected from various vendors for the sizes expected to be used. Estimates of tire life are based on AGP's experience. The operating cost of the tires is also expressed in dollars per hour. The life of the haulage truck tires is estimated at 5,000 hours per tire for the 150 t trucks with proper rotation from front to back. Each truck tire for the 150-ton truck costs $23,900 so the cost per hour for tires is $28.68 per hour for the truck using six tires in the calculation.

The cost for ground-engaging tools (GET) is estimated from other projects and is an area that will be fine-tuned when the project is operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 76 ft/h for the smaller drill and 75 ft/h for the larger drill for both heap feed and waste. The equipment costs used in the estimate are shown in Table 21-29.

**Table 21-29: Major Equipment Operating Costs – No Labour ($/h)**

| Equipment | Fuel/Power | Lube/Oil | Tires/ Undercarriage | Repair & Maintenance | GET/ Consumables | Total |
|---|---|---|---|---|---|---|
| Production Drill – 5 ½ inch | 55.32 | 5.53 | 3.00 | 55.00 | 77.12 | 195.97 |
| Production Drill – 6 ¾ inch | 69.15 | 6.92 | 6.00 | 45.00 | 86.75 | 213.82 |
| Production/Crusher Loader - 15 $yd^3$ | 79.30 | 7.93 | 29.76 | 98.25 | 10.00 | 225.24 |
| Hydraulic Shovel – 30 $yd^3$ | 230.52 | 34.58 | 55.00 | 195.28 | 40.00 | 555.37 |
| Production/Crusher Loader - 15 $y^3$ | 79.30 | 7.93 | 29.76 | 98.25 | 10.00 | 225.24 |
| Haulage Truck – 150 t | 92.21 | 9.22 | 28.68 | 99.00 | 4.00 | 233.11 |
| Track Dozer | 54.40 | 5.44 | 15.00 | 74.00 | 7.00 | 155.84 |
| Grader – 14' | 13.83 | 1.38 | 2.53 | 14.00 | 2.00 | 33.75 |

### 21.4.2.1.3 Drilling

Drilling in the open pit will use down-the-hole hammer drill rigs. The initial drill platforms will be developed by the smaller drill with the main production by the 6 ¾ inch drill. The pattern size varies between the drills but is the same for heap feed and waste for each drill. The material will be smaller and finer to improve productivity and reduce maintenance costs as well as improve crusher performance. The drilling pattern parameters are shown in Table 21-30.

**Table 21-30: Drill Pattern Specifications**

| | | Drill 5 ½ inch | | Drill 6 ¾ inch | |
|---|---|---|---|---|---|
| Specification | Unit | Heap Feed | Waste | Heap Feed | Waste |
| Bench Height | Ft | 20 | 20 | 20 | 20 |
| Sub-drill | Ft | 3.6 | 3.6 | 4.3 | 4.3 |

| | | Drill 5 ½ inch | | Drill 6 ¾ inch | |
|---|---|---|---|---|---|
| **Specification** | **Unit** | **Heap Feed** | **Waste** | **Heap Feed** | **Waste** |
| Blasthole Diameter | Inches | 5 ½ | 5 ½ | 6 ¾ | 6 ¾ |
| Pattern Spacing – Staggered | Ft | 14.1 | 14.1 | 16.7 | 16.7 |
| Pattern Burden – Staggered | Ft | 12.1 | 12.1 | 14.4 | 14.4 |
| Hole Depth | Ft | 23.6 | 23.6 | 24.3 | 24.3 |

The sub-drill is included to allow for caving of the holes in weaker zones, reducing re-drill requirements or short holes that would affect bench floor conditions.

The parameters used to estimate drill productivity are shown in Table 21-31.

**Table 21-31: Drill Productivity Calculation**

| Drill Activity | Unit | Drill 5 ½ inch | | Drill 6 ¾ inch | |
|---|---|---|---|---|---|
| | | Heap Feed | Waste | Heap Feed | Waste |
| Pure Penetration Rate | ft/min | 1.8 | 1.8 | 1.6 | 1.6 |
| Hole Depth | Ft | 23.6 | 23.6 | 24.3 | 24.3 |
| Drill Time | min | 13.09 | 13.09 | 14.80 | 14.80 |
| Move, Spot and Collar Hole | min | 3.00 | 3.00 | 3.00 | 3.00 |
| Level Drill | min | 0.50 | 0.50 | 0.50 | 0.50 |
| Add Steel | min | 0.50 | 0.50 | 0.00 | 0.00 |
| Pull Drill Rods | min | 1.50 | 1.50 | 1.00 | 1.00 |
| Total Setup/Breakdown Time | min | 5.50 | 5.50 | 4.50 | 4.50 |
| Total Drill Time per Hole | min | 18.6 | 18.6 | 19.3 | 19.3 |
| Drill Productivity | ft/h | 76.1 | 76.1 | 75.5 | 75.5 |

##### 21.4.2.1.4 Blasting

Quotations from local explosive vendors were obtained which included delivery to the blasthole. The explosives cost includes monthly fees from the explosive vendor for magazine rental and all costs associated with delivering the product to the open pit and down the hole.

Powder factors that result from the proposed equipment are shown in Table 21-32. The cost for blasting is approximately $0.21 per ton mined over the life of mine. This is $12.0M per year on average for the first 4 years, decreasing thereafter as material movement requirements drop.

**Table 21-32: Design Powder Factors**

| Description | Unit | Heap Feed | Waste |
|---|---|---|---|
| Powder Factor | $Lb/yd^3$ | 1.12 | 1.12 |
| Powder Factor | Lb/t | 0.52 | 0.52 |

#### 21.4.2.1.5 Loading

Loading costs for both heap feed and waste are based on the use of hydraulic shovels and front-end loaders. Due to the length of the mine life more loaders with their lower capital cost were considered. They also have mobility to various locations around the mine site which can be beneficial. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-33. This highlights the balanced nature of the loading units.

"Trucks present at the loading unit" refers to the percentage of time a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the loading unit. One of the largest operating cost items is haulage and minimizing this cost by maximizing the truck productivity is crucial to lower operating costs. The value of 75-80% comes from the standby time shovels/loaders typically encounter due to a lack of trucks.

**Table 21-33: Loading Parameters – Year 3**

| Description | Unit | Hydraulic Shovel | Front End Loader |
|---|---|---|---|
| Bucket Capacity | $yd^3$ | 30 | 15 |
| Truck Capacity Loaded | t | 150 | 150 |
| Waste Tonnage Loaded | % | 51 | 49 |
| Mill Feed Tonnage Loaded | % | 50 | 50 |
| Bucket Fill Factor | % | 88 | 87 |
| Cycle Time | sec | 35 | 40 |
| Trucks Present at Loading Unit | % | 80 | 75 |
| Loading Time | min | 2.45 | 5.37 |

#### 21.4.2.1.6 Hauling

Haulage profiles were determined for each pit phase for the primary crusher, waste rock facility, view shed berm and heap material from Cactus East and Parks/Salyer. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on the trucks is limited to 30 m/h for tire life and safety reasons. Calculation speeds for various segments are shown in Table 21-34.

**Table 21-34: Haulage Cycle Speeds**

| Flat (0%) on Surface | Flat (0%) In-pit, Crusher, Dump | Slope Up (5%) | Slope Up (10%) | Slope Down (5%) | Slope Down (10%) | Flat (0%) on Surface |
|---|---|---|---|---|---|---|
| Loaded (m/h) | 25 | 10 | 7.5 | 19 | 19 | 30 |
| Empty (m/h) | 25 | 22 | 15.5 | 22 | 22 | 30 |

#### 21.4.2.1.7 Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PFS, some of the factors used are shown in Table 21-35.

These factors resulted in the need for five track dozers, three graders, and small support backhoe with hammer. Their tasks will include clean-up of the loader faces, roads, WRSFs, and blast patterns. The graders will maintain the crusher and waste haul routes. In addition, water trucks will have the responsibility for patrolling the haul roads controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for maintaining roads, ditches and pumping facilities.

The hours generated in this manner were applied to the individual operating costs for each piece of equipment. Many of these units will be support equipment, so no direct Labour is allocated to them due to their variable function. The operators will come from the General Equipment operator pool.

**Table 21-35: Support Equipment Operating Factors**

| Mine Equipment | Factor | Factor Units |
|---|---|---|
| Track Dozer | 15% | Of haulage hours to maximum of 5 dozers |
| Grader | 10% | Of haulage hours to maximum of 3 graders |
| Crusher Loader | 40% | Of loading hours to maximum of 1 loader |
| Water Truck | 10% | Of haulage hours to maximum of 3 trucks |
| Road Crew Backhoe | 2 | hours/day/unit |
| Road Crew Dump Truck | 2 | hours/day/unit |
| Road Crew Loader | 2 | hours/day/unit |
| Lube/Fuel Truck | 6 | hours/day/unit |
| Mechanics Truck | 12 | hours/day/unit |
| Integrated Tool Carrier | 4 | hours/day/unit |
| Light Plants | 12 | hours/day/unit |
| Pickup Trucks | 10 | hours/day/unit |

###### 21.4.2.1.8 Grade Control

The grade control program will be completed with blast hole cuttings. Known heap feed samples will be collected in addition to 25% of the waste samples to identify new mineralized zones. Samples will be sent to the assay laboratory with the results applied to the short-range mining model.

If additional grade control is required, a reverse-circulation drilling program can be incorporated but is not considered at this time.

Annual samples are expected to average 51,000 per year for the first 5 years. The total grade control program is estimated to cost approximately $700,000 annually or about $0.02 per ton mined.

###### 21.4.2.1.9 Leasing

Leasing of the mine fleet is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment. Both Caterpillar and Komatsu have the ability, and desire, to allow leasing of their product lines.

Indicative terms for leasing provided by the vendors are as follows:

- Down payment = 20% of equipment cost
- Term length = 3 to 5 years (depending on equipment)
- Interest rate = SOFR plus a percentage
- Residual = $0.

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.20 to equate to the rate. It does not consider a declining balance on the interest, but rather the full amount of interest paid over the term, equally distributed over those years. The calculation is as follows:

Annual Lease Cost = {[(Initial Capital Cost) x 80%] x 1.20} / term in years.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All the major mine equipment, and most of the support equipment where it was considered reasonable, was assumed to be leased. If the equipment had a life greater than the lease term length, then the years after the lease did not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

Using the leasing option adds $0.38/t to the mine operating cost over the life of the mine or $0.70/t of heap feed.

#### 21.4.2.1.10 Dewatering

The dewatering quantity is currently estimated at 154 million gallons per year. Two in-pit diesel pumps will remove this water from the pit and another diesel pump will direct it horizontally to the transfer pond where it joins the site water system. Normal pumping rates are estimated at 422,000 gal/d with peak rates of 924,000 gal/d during the wetter part of the year. Additional dewatering in the form of horizontal drain holes is included in the dewatering cost. These holes will be campaigned and included in sustaining capital. The dewatering operating cost is expected to be approximately $392,000 per year.

### 21.4.2.2 Total Open Pit Operating Costs

The total life-of-mine operating costs per tonne of material mined (in situ and rehandling) is $2.47/ton. The cost per ton milled is estimated at $4.53/ton of open pit material. The costs for the PFS are shown in Table 21-36 and Table 21-37.

**Table 21-36: Open Pit Operating Costs – with Leasing ($/ton mined)**

| Open Pit Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| General Mine and Engineering | $/t mined | 0.15 | 0.13 | 0.17 | 0.16 |
| Drilling | $/t mined | 0.25 | 0.25 | 0.21 | 0.24 |
| Blasting | $/t mined | 0.22 | 0.23 | 0.18 | 0.21 |
| Loading | $/t mined | 0.30 | 0.32 | 0.31 | 0.33 |
| Hauling | $/t mined | 0.88 | 0.83 | 0.93 | 0.88 |
| Support | $/t mined | 0.23 | 0.21 | 0.29 | 0.24 |
| Grade control | $/t mined | 0.01 | 0.01 | 0.02 | 0.02 |
| Leasing costs | $/t mined | 0.45 | 0.44 | 0.34 | 0.38 |
| Dewatering | $/t mined | 0.01 | 0.01 | 0.02 | 0.01 |
| **Total** | **$/t mined** | **2.50** | **2.43** | **2.49** | **2.47** |

**Table 21-37: Open Pit Operating Costs – with Leasing ($/ton open pit heap feed)**

| Open Pit Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| General Mine and Engineering | $/t heap feed | 0.32 | 0.32 | 0.25 | 0.30 |
| Drilling | $/t heap feed | 0.54 | 0.59 | 0.31 | 0.43 |
| Blasting | $/t heap feed | 0.49 | 0.55 | 0.26 | 0.39 |
| Loading | $/t heap feed | 0.65 | 0.75 | 0.45 | 0.60 |
| Hauling | $/t heap feed | 1.92 | 1.98 | 1.33 | 1.61 |
| Support | $/t heap feed | 0.50 | 0.50 | 0.42 | 0.45 |

| Open Pit Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| Grade control | $/t heap feed | 0.03 | 0.03 | 0.04 | 0.03 |
| Leasing costs | $/t heap feed | 0.99 | 1.04 | 0.49 | 0.70 |
| Dewatering | $/t heap feed | 0.02 | 0.02 | 0.02 | 0.02 |
| **Total** | **$/t heap feed** | **5.45** | **5.77** | **3.57** | **4.53** |

#### 21.4.2.3 Underground Operating Costs

The direct operating costs for the underground mines were generated from first principal unit cost models. Each of the models was developed using the mine design criteria and other general engineering estimates of performance. The mine was assumed to operate two 12-hour shifts per day, 365 days per year.

Costs were estimated on a quarterly basis for the first ten years and annually thereafter.

Wherever possible the mine consumable cost database was updated locally during the course of the study. Labour costs were derived from a recent underground feasibility study in Arizona. Budget quotations were provided by mobile equipment suppliers.

Separate drill and blast development cost models included detailed design and ground support assumptions for each mine and each different rock type as provided by CNI. Separate models were created for conveyor drift development by roadheader at Parks/Salyer. Other models were developed for application to the other mine activities, raising, stope drilling and blasting, stope mucking, trucking, and delineation drilling. The unit rates were applied to the scheduled quantities in order to estimate the direct costs.

Initial development to first main stoping production was assumed to be undertaken by contractors. The contractors will provide all labour, consumables and equipment until Year 3 Q2 at Cactus East and Year 5 Q1 at Parks/Salyer, during these periods ASCU will provide only contract supervision and technical services. Thereafter all activities will be undertaken by owner crews apart from roadheader development and raising which will continue to be undertaken by reduced contractor crews.

Additional models were designed to reflect overhead-type activities at the mines:

- Mine Services (including Labour, supplies and equipment for construction, materials transport, road maintenance and sanitation). Diesel maintenance Labour costs are also included.
- Vertical Conveying and Sizing at Cactus East.
- Ramp conveying and Sizing at Parks/Salyer.
- Owners Mine Supervision and Technical (including mine management, production supervision, maintenance supervision, and mine technical and safety staff).

- Air Cooling
- Mine Power (developed from aggregation of mine loads and estimated usage).

Overheads were estimated by quarter and applied as a fixed daily cost. The overheads for each period were split between operating and capital development estimates in the ratio of the respective direct costs.

The models were also used to track Labour and equipment hours to identify annual requirements in each Labour category and equipment type.

All owner mobile equipment will be leased with 15% downpayment followed by a five-year lease at 8.3% pa interest.

The detailed underground operating cost can be reviewed in Section 16.5.12 of this report but has been summarized for the reader in Table 21-38.

**Table 21-38: Underground Operating Costs by Area ($/t ore)**

| Open Pit Category | Unit | Cactus East | Parks/Salyer |
|---|---|---|---|
| Ore Development | $/t ton ore | 2.92 | 3.13 |
| Stoping and Mucking | $/t ton ore | 5.84 | 6.28 |
| Truck Haulage | $/t ton ore | 1.25 | 0.16 |
| Delineation Drilling | $/t ton ore | 0.16 | 0.16 |
| Mine Services | $/t ton ore | 4.02 | 3.38 |
| Sizing and Conveying | $/t ton ore | 1.24 | 1.22 |
| Refrigeration | $/t ton ore | 0.25 | 0.26 |
| Equipment Leasing | $/t ton ore | 2.94 | 1.95 |
| Supervision and Technical | $/t ton ore | 3.27 | 1.93 |
| Power | $/t ton ore | 1.25 | 0.90 |
| **Total** | **$/t ton ore** | **23.13** | **19.38** |

### 21.4.3 Processing Facilities

The process operating costs for the Project were developed based on direct build-up of metallurgical parameters from test work, chemical and material supply costs, and typical unit consumptions and costs. The processing costs reflect a heap leach with a planned maximum placement of 28M tpy including crushing, agglomeration, and stacking. The acidic leach solution is processed through an SX/EW facility with a maximum annual cathode production rate of 72,000 tons.

The average LOM process operating cost is $2.96 per ton processed on the heap leach pad. Crushing and heap leach costs, SX/EW costs, and power costs are outline in Table 21-39, Table 21-40, and Table 21-41. Power will be sourced from the local Casa Grand power grid supplying 69kv to site. Average power cost is approximately $0.07/kWh through the first 16 years of the project escalating to $0.093/kWh in year 17 and beyond.

**Table 21-39: Crushing & Heap Leach Costs**

| Operating Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| Crushing Consumables | $/t heap feed | 0.04 | 0.04 | 0.04 | 0.04 |
| Crushing Maintenance | $/t heap feed | 0.14 | 0.14 | 0.14 | 0.14 |
| Stacking System Maintenance | $/t heap feed | 0.01 | 0.01 | 0.01 | 0.01 |
| Heap Consumables | $/t heap feed | 0.02 | 0.02 | 0.02 | 0.02 |
| Net Acid Consumption | $/t heap feed | 0.97 | 1.05 | 1.33 | 0.70 |
| **Total** | **$/t heap feed** | **1.19** | **1.26** | **1.54** | **0.91** |

**Table 21-40: SX/EW Costs**

| Operating Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| CoSO4 | $/lb Cu Prod | 0.007 | 0.007 | 0.007 | 0.007 |
| Guar | $/lb Cu Prod | 0.001 | 0.001 | 0.001 | 0.001 |
| Diluent | $/lb Cu Prod | 0.006 | 0.004 | 0.004 | 0.004 |
| Extractant | $/lb Cu Prod | 0.008 | 0.005 | 0.005 | 0.005 |
| Sulfuric Acid | $/lb Cu Prod | 0.002 | 0.001 | 0.001 | 0.001 |
| SX/EW Maintenance | $/lb Cu Prod | 0.010 | 0.010 | 0.010 | 0.010 |
| **Total** | **$/lb Cu Prod** | **0.034** | **0.029** | **0.027** | **0.028** |

**Table 21-41: Process Facilities Power Consumption**

| Operating Category | Unit | Year 1 | Year 3 | Year 5 | LOM Average |
|---|---|---|---|---|---|
| Crushing & Heap Leach | MW | 4.9 | 5.1 | 5.4 | 2.9 |
| SX/TF Power & Utilities | MW | 4.5 | 4.5 | 4.5 | 4.5 |
| Electrowinning | MW | 11.0 | 15.5 | 18.3 | 18.4 |
| **Total** | **MW** | **20.5** | **25.0** | **28.3** | **25.8** |
| Crushing & Heap Leach | $/t heap feed | 0.15 | 0.13 | 0.13 | 0.13 |
| SX/TF Power & Utilities | $/lb Cu Prod | 0.05 | 0.03 | 0.02 | 0.03 |
| Electrowinning | $/lb Cu Prod | 0.11 | 0.10 | 0.09 | 0.10 |
| **Total** | **$/lb Cu Prod** | **0.034** | **0.029** | **0.027** | **0.028** |

Labour costs for the various job positions were baselined from other operations in Arizona and discussion with ASCU personnel. A burden rate of 30% was applied to the staff rates and 40% to the hourly rates. Labour was estimated for both staff and hourly on a 12-hour shift basis. Process staff positions and salaries are shown in Table 21-42. Operating and mainentnace personnel and salaries are shown in Table 21-43. Annual salary figures are inclusive of burden.

Table 21-42: Management and Engineering Labour Requirements and Annual Salaries

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| **Process Management** | | |
| Process Manager | 1 | 234,000 |
| Chief Metallurgist | 1 | 195,000 |
| Subtotal | 2 | |
| **Operations/Maintenance Supervision** | | |
| General Foreman | 1 | 162,500 |
| SXEW Operations Foreman | 4 | 130,000 |
| Leach Operations Foreman | 1 | 130,000 |
| Maintenance Foreman | 2 | 130,000 |
| Maintenance Planner | 2 | 117,000 |
| Electrical Foreman/Planner | 1 | 136,500 |
| Subtotal | 11 | |
| **Engineering/Assay Lab** | | |
| Metallurgist | 1 | 149,500 |
| Chief Chemist | 1 | 162,500 |
| Assay Lab Technicians | 6 | 64,064 |
| Subtotal | 8 | |
| **Total** | **21** | **2,568,384** |

Table 21-43: Operating Labour Requirements and Annual Salaries

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| **Process Operations** | | |
| SX Operators | 12 | 100,433 |
| EW Crew | 5 | 87,360 |
| Crushing Operators | 4 | 100,433 |
| Agglomeration Operators | 4 | 100,433 |
| Stacking Equipment Operators | 4 | 100,433 |
| Subtotal | 29 | |
| **Process Maintenance** | | |
| SXEW Millwright/Pipefitters | 4 | 107,128 |
| SXEW Electricians | 3 | 107,128 |
| SXEW Instrument Techs | 3 | 107,128 |
| Crushing/Stacking Millwrights | 4 | 107,128 |

| Position | Employees | Annual Salary (US$/y) |
|---|---|---|
| Crushing/Stacking Electricians | 4 | 107,128 |
| Crushing/Stacking Instrument Techs | 4 | 107,128 |
| Subtotal | 22 | |
| **Operations Support** | | |
| Security | 4 | 73,651 |
| SXEW Labour | 5 | 69,888 |
| Crushing Labour | 8 | 100,433 |
| Leach Pad Labour | 4 | 69,888 |
| Subtotal | 21 | |
| **Total Hourly** | **72** | **6,931,075** |

### 21.4.4 Infrastructure

The infrastructure operating cost are the ongoing expenses associated with the operation of infrastructure facilities. These costs include expenditures related to electrical utilities. The estimate of infrastructure operating costs is provided in Table 21-44. The infrastructure operating cost was estimated at US$369K per year. Operating cost of the heap leach facility is included with the processing facilities estimate by Samuel Engineering.

**Table 21-44: Infrastructure Operating Cost**

| Area Description | Installed (kW) | Operating (kW) | Operating hours per year (h/y) | Consumption (MWh/y) | Cost ($/y) |
|---|---|---|---|---|---|
| 1510 - Administration Building | 90 | 72 | 4,380 | 316 | 22,149 |
| 1520 - Change House | 15 | 12 | 4,380 | 54 | 3,753 |
| 1610 - Security Gatehouse & Weighbridge | 13 | 10 | 8,760 | 90 | 6,328 |
| 1620 - Plant Workshop and Warehouse | 46 | 36 | 4,380 | 160 | 11,180 |
| 1630 - Laboratory | 34 | 27 | 4,380 | 118 | 8,227 |
| 1720 - Main Substation | 73 | 58 | 8,760 | 512 | 35,860 |
| 1910 - Water Wells | 224 | 257 | 8,760 | 2,253 | 157,719 |
| 2920 - Mine Maintenance Office | 31 | 25 | 4,380 | 108 | 7,594 |
| 2930 - Mine Truck Shop | 238 | 191 | 4,380 | 835 | 58,431 |
| 2940 - Truck Wash | 64 | 51 | 4,380 | 223 | 15,610 |
| 2950 - Heavy Equipment Maintenance | 137 | 109 | 4,380 | 479 | 33,540 |
| 2960 - Tire Workshop | 37 | 30 | 4,380 | 130 | 9,070 |
| **Total** | **1,000** | **879** | | **5,278** | **369,459** |

### 21.4.5 General and Administrative Operating Costs

Operating G&A has been defined by ASCU with input from the sub-consultants for Infrastructure, Process and Mining. An allocation of $833K/yr was included for Infrastructure G&A and $723K/yr for Process. Mining operating cost unit rate estimates include a non-itemized G&A allowance of $0.16/ton.

# 22 ECONOMIC ANALYSIS

## 22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this Section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward looking includes the following:

- Mineral resource estimate.
- Assumed commodity prices and exchange rates.
- The proposed mine production plan.
- Projected mining and process recovery rates.
- Assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged the timing and amount of estimated future production.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rates.
- Accidents, Labour disputes and other risks of the mining industry.
- Geotechnical or hydrogeological considerations during mining being different from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment, or processes to operate as anticipated.
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis.
- Ability to maintain the social license to operate.
- Changes to interest rates.

- Changes to tax rates.

## 22.2 Methodologies Used

The project has been evaluated using a discounted cash flow (DCF) analysis based on an 8% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including pre-production costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections.

Cash flows are taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in copper price, discount rate, head grade, recovery, total operating cost, and total capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21. The economic analysis has been run on a constant dollar basis with no inflation.

## 22.3 Financial Model Parameters

The economic analysis was performed assuming the copper price of US$3.90/lb; this price is based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metals price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- Construction period of two years
- Total mine life of 21.0 years
- Cost estimates in constant Q4 2023 US$ with no inflation or escalation factors considered.
- Results based on 100% ownership with a series of NSR royalties applicable to distinct portions of the mineralized material.
- Capital cost funded with 100% equity (no financing cost assumed)
- All cash flows discounted to start of construction period using mid-period discounting convention.
- All metal products are sold in the same year they are produced.
- Project revenue is derived from the sale of copper cathode with no other metal credits payable.
- No contractual arrangements for refining currently exist.

### 22.3.1 Taxes

The project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model calculations are based on the tax regime as of the date of the PFS technical report. At the effective date of this report, the Project is assumed to be subject to the Arizona Property Tax, Arizona Severance Tax, and Federal and State Income taxes resulting in a total estimated tax payable of US$692.2M over the life of mine.

Mining Tax Plan LLC (MTP) has prepared the U.S federal and state income tax computation based on the Internal Revenue Code of 1986, as amended and the regulations thereunder including Arizona Revised Statutes as in effect as of December 31, 2023. Any subsequent changes or modifications to U.S. federal or state tax statutes, regulations or to the judicial and administrative interpretations thereof may impact the federal and state income tax computations. MTP have not audited or verified any of the economic or operating assumptions of the Preliminary Feasibility Study Model but have made inquiries to properly classified revenue, expenses and capital expenditures consistent with federal and state income tax statutes, regulations and case law.

The following is a summary of tax elections incorporated into this tax computation:

- The overall effective federal and state income tax rate for Arizona Sonoran Copper Company USA Inc. is 24.9 percent which is comprised of 21 percent for federal and for Arizona 4.9 percent net of federal tax deduction.
- The surface and underground mines of the Cactus Copper Mine Project will be treated as separate depletable properties under Section 614.
- The Cactus Copper Mine Project will opt out of bonus depreciation under Section 168(k) for 2027, the last year of allowed bonus depreciation under the phase out and elect 150 DB MACRS Section 168(a) for all subsequent years.
- The Cactus Copper Mine Project will deduct mine development costs as incurred under Section 616(a) subject to Section 291(b)(2) limitation for corporate preferences.
- The Cactus East underground development capital expenditures from years 7-8 are treated as tax mine development and for years 9 and later are treated as inventoriable cost. The Park/Salyer underground development capital expenditures are treated as inventoriable cost for all years.
- All metal sales will occur at the mine site and therefore will not be eligible for Section 250 FDII deduction available on exported goods.
- No Section 382 ownership change will occur during the construction or operation of the mine which could limit the tax attributes available.
- The severance tax liability has been computed in accordance with the Arizona Department of Revenue statutes and regulations. The tax rate is 2.5 percent and is applied to 50 percent of the gross margins on metal sales.
- The property tax liability has been computed in accordance with Arizona Department of Revenue statutes, regulations, guidelines and discussions with the State for the Cactus Copper Mine Project. Under these provisions the cost approach was used for years 1 through 5, a 60/40 ratio split between the income and cost approaches utilizing a $3.60 copper price for years 6 through 27, and again the cost approach was utilized for the final 5 years of the mine life.

## 22.4 Economic Analysis

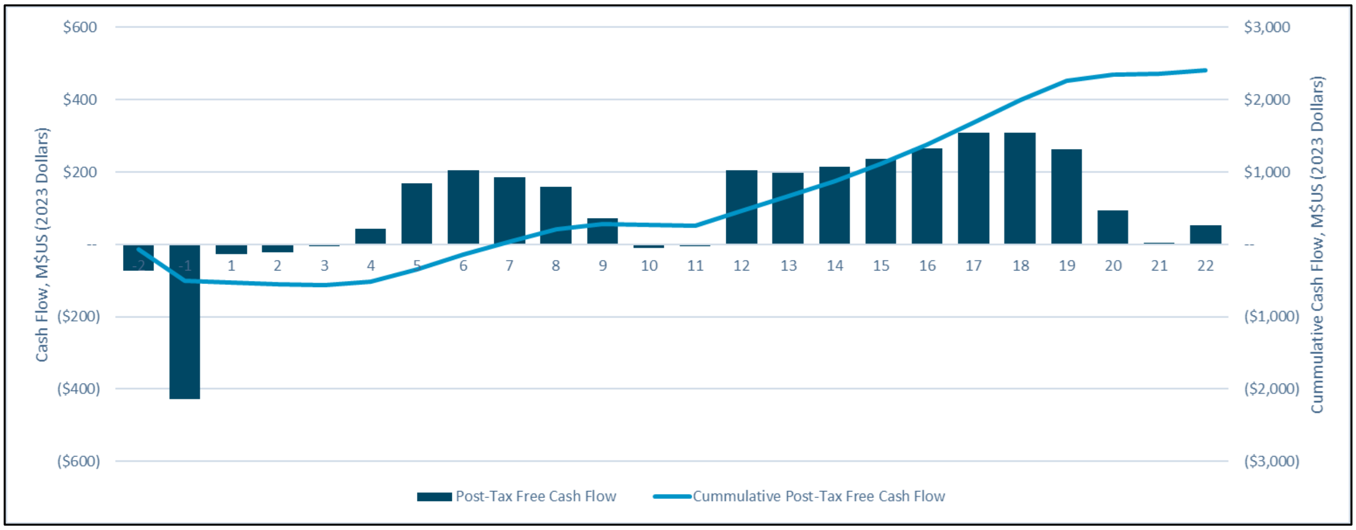
The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV discounted at 8% is US$733.3M; the internal rate of return (IRR) is 17.7%, and payback period is 6.3 years. On a post-tax basis, the NPV discounted at 8% is US$508.7M; the IRR is 15.3%, and the payback period is 6.8 years. A summary of project economics is tabulated in Table 22-1 The analysis was done on an annual cashflow basis; the cashflow output is shown in Table 22-2 and cashflow is represented graphically in Figure 22-1 on a post-tax basis.

**Table 22-1: Economic Analysis Summary Table**

| General | Units | LOM Total / Avg. | |
|---|---|---|---|
| Copper Price | US$/lb | 3.90 | |
| Mine Life | Years | 21.0 | |
| Total Mineralized Material Processed | Kt | 276,286 | |
| Total Waste | Kt | 147,561 | |
| Avg. CuAS Head Grade | % | 0.14 | |
| Avg. CuCN Head Grade | % | 0.34 | |
| Avg. Acid Consumption | lb/t | 18.99 | |
| **Production** | **Units** | **LOM Total / Avg.** | |
| Avg. Head Grade – CuAS | % | 0.14 | |
| Avg. Head Grade – CuCN | % | 0.34 | |
| Avg. Acid Consumption | lb/t | 18.99 | |
| Avg. Recovery Rate – CuAS | % | 90.8 | |
| Avg. Recovery Rate – CuCN | % | 84.5 | |
| Total Payable Copper | M lb | 2,306 | |
| Annual Payable Copper | M lb/y | 110 | |
| **Operating Costs** | **Units** | **LOM Total / Avg.** | |
| Mining Cost | US$/t mined | 7.50 | |
| Mining Cost | US$/t processed | 11.51 | |
| Processing Cost | US$/t processed | 2.96 | |
| G&A Cost | US$/t processed | 0.12 | |
| Operating Cash Costs* | US$/lb Cu | 1.75 | |
| C1 Cash Costs** | US$/lb Cu | 1.84 | |
| C3 Cash Costs (AISC)*** | US$/lb Cu | 2.34 | |
| **Capital Costs** | **Units** | **LOM Total / Avg.** | |
| Initial Capital (Incl. Capitalized Opex) | US$M | 515 | |
| Capitalized Processing Costs | US$M | 4 | |
| Sustaining Capital | US$M | 1,221 | |
| Closure Costs | US$M | 23 | |
| Salvage Value | US$M | 97 | |
| **Financials** | **Units** | **Pre-Tax** | **Post-Tax** |
| NPV (8%) | US$M | 733.3 | 508.7 |
| IRR | % | 17.7 | 15.3 |
| Payback | Years | 6.3 | 6.8 |

*Operating cash costs consist of mining costs, processing costs, and G&A.
**Total cash coasts consist of operating cash costs plus transportation cost, royalties, treatment, and refinancing.
***AISC consist of total cash costs pls sustaining capital, closure cost, and salvage value.

**Figure 22-1: Free Cash Flow – Post Tax**




Source: Ausenco, 2024.

**Table 22-2: Cashflow Statement on an Annualized Basis**

| | Units | Total / Avg. | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Copper Price | US$/lb | $3.90 | -- | -- | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 |
| **Revenue** | **US$M** | **$8,994** | **--** | **--** | **$283** | **$328** | **$399** | **$465** | **$472** | **$499** | **$440** | **$388** | **$370** | **$357** | **$467** | **$539** | **$534** | **$558** | **$561** | **$574** | **$580** | **$546** | **$455** | **$154** | **$23** | **$0** |
| Operating Cost | US$M | ($4,029) | -- | -- | ($184) | ($202) | ($214) | ($246) | ($254) | ($248) | ($195) | ($168) | ($203) | ($245) | ($237) | ($237) | ($249) | ($243) | ($227) | ($226) | ($166) | ($138) | ($109) | ($23) | ($16) | ($0) |
| Off-Site Costs | US$M | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Royalties | US$M | ($218) | -- | -- | ($7) | ($8) | ($10) | ($11) | ($11) | ($12) | ($11) | ($9) | ($9) | ($9) | ($11) | ($13) | ($13) | ($14) | ($14) | ($14) | ($14) | ($13) | ($11) | ($4) | ($0) | ($0) |
| **EBITDA** | **US$M** | **$4,746** | **--** | **--** | **$92** | **$118** | **$175** | **$207** | **$206** | **$239** | **$234** | **$211** | **$159** | **$104** | **$219** | **$289** | **$272** | **$302** | **$321** | **$335** | **$400** | **$395** | **$335** | **$128** | **$6** | **$0** |
| Initial Capex | US$M | ($515) | ($97) | ($419) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Sustaining Capex | US$M | ($1,221) | -- | -- | ($125) | ($137) | ($168) | ($153) | ($21) | ($15) | ($27) | ($33) | ($75) | ($106) | ($207) | ($46) | ($34) | ($36) | ($25) | ($6) | ($5) | ($1) | -- | -- | -- | -- |
| Closure Capex | US$M | ($23) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | ($23.0) |
| Salvage Value (Incl. Proceeds from Disposal of Non-Impacted Land) | US$M | $97 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | $97.5 |
| Royalty Buyback | US$M | ($9) | -- | ($9.4) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Proceeds From Disposal of Non-Impacted Land | US$M | $23 | $23.2 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Change in Working Capital | US$M | ($0) | -- | -- | $15.7 | $1.6 | $1.1 | $2.8 | $0.7 | ($0.5) | ($4.5) | ($2.4) | $2.8 | $3.5 | ($0.4) | $0.1 | $0.9 | ($0.4) | ($1.3) | ($0.1) | ($4.9) | ($2.3) | ($2.5) | ($7.7) | ($0.8) | ($1.3) |
| **Pre-Tax Unlevered Free Cash Flow** | **US$M** | **$3,099** | **($73)** | **($428)** | **($17)** | **($17)** | **$8** | **$57** | **$185** | **$224** | **$203** | **$176** | **$87** | **$1** | **$11** | **$243** | **$239** | **$265** | **$294** | **$329** | **$390** | **$391** | **$333** | **$120** | **$6** | **$73** |
| Cumulative Pre-Tax Unlevered Free Cash Flow | US$M | -- | ($73) | ($501) | ($518) | ($536) | ($527) | ($470) | ($285) | ($62) | $142 | $318 | $404 | $405 | $416 | $659 | $898 | $1,163 | $1,458 | $1,786 | $2,176 | $2,567 | $2,900 | $3,020 | $3,026 | $3,099 |
| Income Tax, Property Tax, and Severance | US$M | $692 | $0 | $0 | $10 | $10 | $11 | $14 | $16 | $18 | $18 | $17 | $15 | $11 | $16 | $38 | $42 | $52 | $59 | $64 | $81 | $82 | $70 | $26 | $1 | $21 |
| **Post-Tax Unlevered Free Cash Flow** | **US$M** | **$2,407** | **($74)** | **($428)** | **($27)** | **($27)** | **($3)** | **$43** | **$169** | **$206** | **$186** | **$159** | **$71** | **($10)** | **($5)** | **$205** | **$197** | **$213** | **$235** | **$265** | **$308** | **$309** | **$263** | **$94** | **$4** | **$53** |
| Post-Tax Unlevered Free Cash Flow | US$M | $4,592 | ($74) | ($502) | ($530) | ($553) | ($558) | ($516) | ($347) | ($141) | $45 | $205 | $276 | $266 | $261 | $465 | $662 | $875 | $1,110 | $1,375 | $1,683 | $1,992 | $2,255 | $2,349 | $2,354 | $2,407 |
| **Production Summary** | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Waste Mined and Moved Total | kt | 147,841 | -- | 23,767 | 27,525 | 33,989 | 35,469 | 19,235 | 4,351 | 3,316 | 189 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mineralized Material Mined - Existing Stockpile | kt | 76,777 | -- | 2,970 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 1,807 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mineralized Material Mined - Cactus West | kt | 75,521 | -- | 517 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 12,000 | 3,003 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mineralized Material Mined - Park Salyer | kt | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mineralized Material Mined - Cactus East | kt | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Total Mill Feed | kt | 152,298 | -- | 3,487 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 4,811 | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Project Life | yrs | 21.0 | -- | -- | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 1.0 | 0.0 |
| **Processing Summary** | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Total Leach Feed - Cu Grade (CuAS) | % | 0.14% | -- | 0.16% | 0.14% | 0.13% | 0.12% | 0.12% | 0.12% | 0.10% | 0.10% | 0.13% | 0.13% | 0.14% | 0.23% | 0.22% | 0.19% | 0.21% | 0.20% | 0.19% | 0.16% | 0.13% | 0.07% | 0.04% | -- | -- |
| Total Leach Feed - Cu Grade (CuCN) | % | 0.34% | -- | 0.07% | 0.05% | 0.08% | 0.14% | 0.15% | 0.14% | 0.17% | 0.46% | 0.74% | 0.70% | 0.64% | 0.71% | 0.66% | 0.59% | 0.65% | 0.58% | 0.63% | 0.68% | 0.65% | 0.76% | 0.64% | -- | -- |
| **Total Leach Feed - Cu Content (CuAS)** | **mlbs** | **772** | **--** | **10.9** | **67.5** | **61.5** | **60.5** | **64.1** | **61.7** | **57.8** | **23.2** | **17.5** | **16.9** | **19.2** | **38.0** | **41.0** | **37.4** | **41.5** | **42.6** | **39.4** | **33.5** | **26.0** | **11.1** | **0.1** | **--** | **--** |
| **Total Leach Feed - Cu Content (CuCN)** | **mlbs** | **1,900** | **--** | **4.6** | **23.4** | **38.0** | **69.1** | **78.3** | **75.7** | **95.7** | **103.9** | **96.2** | **92.7** | **86.1** | **119.3** | **125.3** | **118.2** | **129.4** | **123.3** | **134.1** | **139.8** | **131.9** | **113.3** | **1.8** | **--** | **--** |
| Copper Recovery (CuAS)* | % | 90.8% | -- | 88.0% | 90.4% | 90.5% | 90.0% | 90.8% | 90.6% | 90.4% | 91.3% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | 91.2% | -- | -- |

| | Units | Total / Avg. | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Copper Recovery (CuCN)* | % | 84.5% | -- | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | 84.5% | -- | -- |
| Total Copper Produced | mlbs | 2,306 | -- | -- | 72.6 | 84.2 | 102.2 | 119.2 | 120.9 | 128.0 | 112.9 | 99.6 | 94.9 | 91.7 | 119.8 | 138.3 | 136.9 | 143.0 | 143.8 | 147.3 | 148.6 | 139.9 | 116.8 | 39.6 | 5.8 | 0.1 |
| **Total Copper Payable** | **mlbs** | **2,306** | **--** | **--** | **72.6** | **84.2** | **102.2** | **119.2** | **120.9** | **128.0** | **112.9** | **99.6** | **94.9** | **91.7** | **119.8** | **138.3** | **136.9** | **143.0** | **143.8** | **147.3** | **148.6** | **139.9** | **116.8** | **39.6** | **5.8** | **0.1** |
| **Total Operating Costs** | **US$M** | **($4,029)** | **--** | **--** | **($184)** | **($202)** | **($214)** | **($246)** | **($254)** | **($248)** | **($195)** | **($168)** | **($203)** | **($245)** | **($237)** | **($237)** | **($249)** | **($243)** | **($227)** | **($226)** | **($166)** | **($138)** | **($109)** | **($23)** | **($16)** | **($0)** |
| Mine Operating Costs | US$M | (3,180) | -- | -- | (131) | (142) | (155) | (184) | (182) | (171) | (151) | (141) | (177) | (219) | (208) | (205) | (217) | (210) | (194) | (191) | (126) | (99) | (75) | (2) | -- | -- |
| Processing Costs | US$M | (831.9) | -- | -- | (52.6) | (59.6) | (58.2) | (61.3) | (71.6) | (76.5) | (43.8) | (25.8) | (25.2) | (25.0) | (28.5) | (30.9) | (30.8) | (31.5) | (31.7) | (33.8) | (39.1) | (37.8) | (33.5) | (19.8) | (15.0) | (0.0) |
| Total Unit Operating Costs | US$/t Process | (14.6) | -- | -- | (7.7) | (8.3) | (8.7) | (9.2) | (9.6) | (8.8) | (17.1) | (25.8) | (30.5) | (36.5) | (28.2) | (24.9) | (24.9) | (24.3) | (21.2) | (21.4) | (16.1) | (13.6) | (14.7) | (159.1) | -- | -- |
| **Total Royalties** | **US$M** | **($218.2)** | **--** | **--** | **($6.8)** | **($7.9)** | **($9.7)** | **($11.2)** | **($11.5)** | **($12.2)** | **($10.7)** | **($9.4)** | **($9.0)** | **($8.7)** | **($11.4)** | **($13.2)** | **($13.0)** | **($13.6)** | **($13.7)** | **($14.0)** | **($14.0)** | **($13.2)** | **($11.0)** | **($3.6)** | **($0.4)** | **($0.0)** |
| Tembo Royalty | US$M | (193.7) | -- | -- | (6.8) | (7.9) | (9.5) | (10.7) | (10.8) | (11.2) | (9.2) | (7.7) | (7.3) | (7.1) | (9.6) | (11.4) | (11.4) | (11.9) | (12.1) | (12.4) | (12.4) | (11.7) | (9.4) | (3.0) | (0.3) | (0.0) |
| ASLD Royalty | US$M | (19.6) | -- | -- | (0.0) | (0.0) | (0.1) | (0.5) | (0.6) | (0.7) | (1.2) | (1.4) | (1.4) | (1.3) | (1.4) | (1.4) | (1.3) | (1.4) | (1.3) | (1.3) | (1.3) | (1.2) | (1.3) | (0.5) | (0.1) | (0.0) |
| Bronco Creek Royalty | US$M | (4.9) | -- | -- | (0.0) | (0.0) | (0.0) | (0.1) | (0.1) | (0.2) | (0.3) | (0.3) | (0.3) | (0.3) | (0.4) | (0.4) | (0.3) | (0.3) | (0.3) | (0.3) | (0.3) | (0.3) | (0.3) | (0.1) | (0.0) | (0.0) |
| **Cash Costs** | | | | | | | | | | | | | | | | | | | | | | | | | | |
| C1 Cash Cost** | US$/lb Cu | $1.84 | -- | -- | $2.63 | $2.50 | $2.19 | $2.16 | $2.20 | $2.03 | $1.82 | $1.78 | $2.23 | $2.77 | $2.07 | $1.81 | $1.91 | $1.79 | $1.67 | $1.63 | $1.21 | $1.08 | $1.03 | $0.67 | $2.80 | $0.43 |
| C3 Cash Cost*** | US$/lb Cu | $2.34 | -- | -- | $4.37 | $4.07 | $3.85 | $3.45 | $2.38 | $2.15 | $2.06 | $2.11 | $3.02 | $3.93 | $3.80 | $2.14 | $2.16 | $2.04 | $1.84 | $1.67 | $1.25 | $1.09 | $1.03 | $0.67 | $2.80 | ($896.52) |
| **Total Initial Capital** | **US$M** | **($515)** | **($97)** | **($419)** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** |
| Process Capitalized Operating Cost | US$M | ($4) | -- | ($4) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mine Pre-Strip and Pre-Production Mining | US$M | ($78) | ($7) | ($71) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Mine Capital Cost | US$M | ($92) | ($11) | ($81) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Process Capital Direct Costs | US$M | ($213) | ($49) | ($164) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Process Capital Indirect Costs | US$M | ($54) | ($13) | ($41) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Contingency Costs | US$M | ($75) | ($17) | ($58) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| **Total Sustaining Capital** | **US$M** | **($1,221)** | **--** | **--** | **($126)** | **($133)** | **($170)** | **($154)** | **($22)** | **($15)** | **($27)** | **($33)** | **($75)** | **($106)** | **($207)** | **($46)** | **($34)** | **($36)** | **($25)** | **($6)** | **($5)** | **($1)** | **--** | **--** | **--** | **--** |
| Mine Capital Cost | US$M | ($905) | -- | -- | ($103) | ($99) | ($128) | ($68) | ($18.6) | ($12.8) | ($23.4) | ($19.5) | ($65.2) | ($92.3) | ($142) | ($40) | ($30) | ($31) | ($22) | ($5) | ($5) | ($1) | -- | -- | -- | -- |
| Process Capital Direct Costs | US$M | ($147) | -- | -- | ($6.1) | ($20.6) | ($14.4) | ($63.3) | ($0.3) | -- | -- | ($6.5) | -- | -- | ($36.2) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Process Capital Indirect Costs | US$M | ($8) | -- | -- | ($1) | -- | ($4) | -- | -- | -- | -- | ($2) | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- | -- |
| Contingency Costs | US$M | ($160) | -- | -- | ($15) | ($17) | ($22) | ($22) | ($3) | ($2) | ($3) | ($5) | ($10) | ($14) | ($29) | ($6) | ($4) | ($5) | ($3) | ($1) | ($1) | ($0) | -- | -- | -- | -- |
| **Salvage Value** | **US$M** | **$97** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **$97** |
| **Closure Cost** | **US$M** | **($23)** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **--** | **($23)** |
| **Total Capital Expenditures Including Salvage Value** | **US$M** | **($1,661)** | **($97)** | **($419)** | **($125)** | **($137)** | **($168)** | **($153)** | **($21)** | **($15)** | **($27)** | **($33)** | **($75)** | **($106)** | **($207)** | **($46)** | **($34)** | **($36)** | **($25)** | **($6)** | **($5)** | **($1)** | **--** | **--** | **--** | **$74** |

Dollar figures in Real 2023 $mm unless otherwise noted.
* Recovery stated is net of 3-year copper distribution.
** C1 costs consist of mining costs, processing costs, mine-level G&A and transportation cost.
*** C3 includes cash costs plus sustaining capital, expansion capital, royalties, and closure costs
Source: Ausenco, 2024

## 22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax $NPV_{8\%}$ and IRR of the Project using the following variables: metal price, discount rate, total operating cost, and initial capital cost. Table 22-3 shows a summary of the post-tax sensitivity results, Table 22-4 shows a summary of the pre-tax sensitivity results.

As shown in Figure 22-2 and Figure 22-3, the sensitivity analysis revealed that the Project is most sensitive to commodity price, head grade, and operating cost and less sensitive to initial capital cost.

**Table 22-3: Post-Tax Sensitivity Summary**

**Post-Tax NPV (US$M) Sensitivity to Discount Rate**

| Discount Rate | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| 3.0% | $373 | $886 | $1,389 | $1,883 | $2,374 |
| 5.0% | $115 | $538 | $950 | $1,351 | $1,749 |
| 8.0% | ($128) | $195 | $509 | $811 | $1,109 |
| 10.0% | ($228) | $46 | $311 | $566 | $816 |
| 12.0% | ($297) | ($62) | $165 | $382 | $594 |

**Post-Tax IRR % Sensitivity to Discount Rate**

| Discount Rate | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| 3.0% | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 5.0% | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 8.0% | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 10.0% | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 12.0% | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |

**Post-Tax NPV (US$M) Sensitivity to Recovery**

| Recovery | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | ($661) | ($372) | ($102) | $156 | $408 |
| (10%) | ($387) | ($83) | $208 | $490 | $763 |
| -- | ($128) | $195 | $509 | $811 | $1,109 |
| 10% | $111 | $458 | $791 | $1,118 | $1,440 |
| 20% | $252 | $610 | $956 | $1,298 | $1,636 |

**Post-Tax IRR Sensitivity to Recovery**

| Recovery | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | 0.0% | 2.9% | 6.6% | 10.2% | 13.8% |
| (10%) | 2.7% | 6.8% | 10.9% | 15.0% | 19.0% |
| -- | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 10% | 9.6% | 14.6% | 19.4% | 24.2% | 28.9% |
| 20% | 11.5% | 16.7% | 21.7% | 26.6% | 31.4% |

**Post-Tax NPV (US$M) Sensitivity to Initial Capital Cost**

| Initial Capital Cost | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | ($34) | $289 | $602 | $904 | $1,202 |
| (10%) | ($81) | $242 | $555 | $858 | $1,155 |
| -- | ($128) | $195 | $509 | $811 | $1,109 |
| 10% | ($174) | $149 | $462 | $764 | $1,062 |
| 20% | ($221) | $102 | $416 | $718 | $1,016 |

**Post-Tax IRR Sensitivity to Initial Capital Cost**

| Initial Capital Cost | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | 7.5% | 12.6% | 17.8% | 23.0% | 28.2% |
| (10%) | 6.8% | 11.6% | 16.5% | 21.3% | 26.0% |
| -- | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 10% | 5.7% | 10.0% | 14.3% | 18.4% | 22.5% |
| 20% | 5.2% | 9.3% | 13.4% | 17.3% | 21.0% |

**Post-Tax NPV (US$M) Sensitivity to Operating Cost**

| Operating Cost | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | $232 | $555 | $869 | $1,171 | $1,469 |
| (10%) | $52 | $375 | $689 | $991 | $1,289 |
| -- | ($128) | $195 | $509 | $811 | $1,109 |
| 10% | ($307) | $16 | $329 | $631 | $929 |
| 20% | ($486) | ($163) | $150 | $452 | $750 |

**Post-Tax IRR Sensitivity to Operating Cost**

| Operating Cost | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | 11.3% | 16.0% | 20.7% | 25.2% | 29.7% |
| (10%) | 8.7% | 13.4% | 18.0% | 22.5% | 26.9% |
| -- | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 10% | 3.7% | 8.2% | 12.7% | 17.0% | 21.3% |
| 20% | 1.3% | 5.7% | 10.1% | 14.4% | 18.6% |

**Post-Tax NPV (US$M) Sensitivity to Head Grade**

| Head Grade | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | ($661) | ($372) | ($102) | $156 | $408 |
| (10%) | ($387) | ($83) | $208 | $490 | $763 |
| -- | ($128) | $195 | $509 | $811 | $1,109 |
| 10% | $120 | $467 | $801 | $1,128 | $1,452 |
| 20% | $364 | $731 | $1,089 | $1,442 | $1,794 |

**Post-Tax IRR Sensitivity to Head Grade**

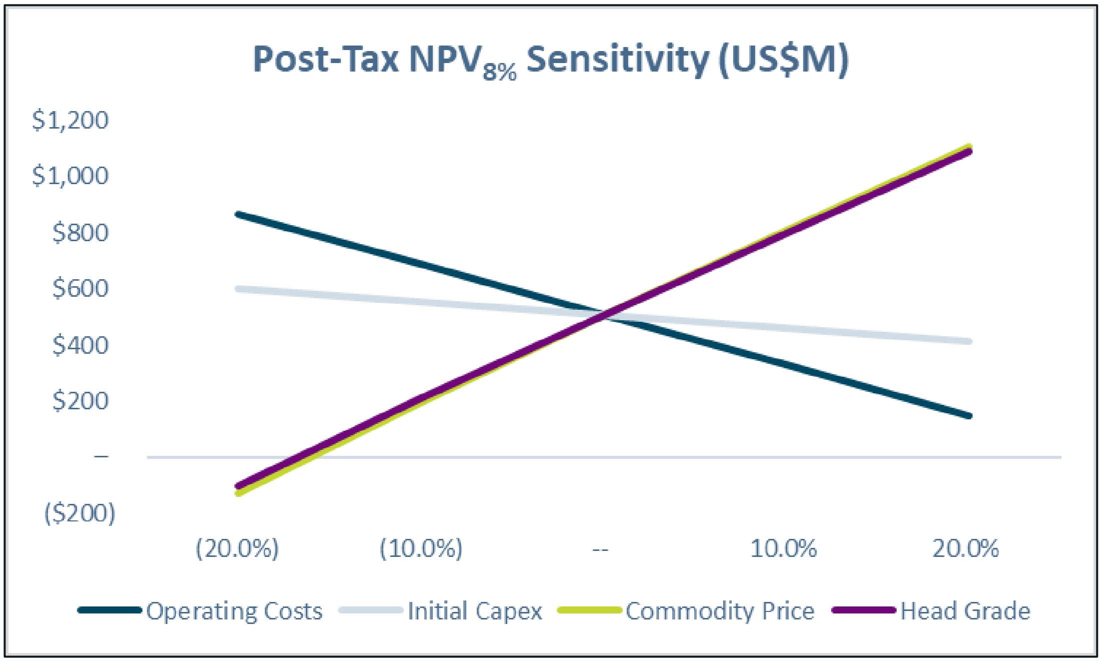
| Head Grade | Commodity Price: -20% | -10% | 0% | 10% | 20% |
|---|---|---|---|---|---|
| (20%) | 0.0% | 2.9% | 6.6% | 10.2% | 13.8% |
| (10%) | 2.7% | 6.8% | 10.9% | 15.0% | 19.0% |
| -- | 6.2% | 10.8% | 15.3% | 19.7% | 24.1% |
| 10% | 9.7% | 14.7% | 19.6% | 24.4% | 29.2% |
| 20% | 13.3% | 18.7% | 23.9% | 29.1% | 34.2% |

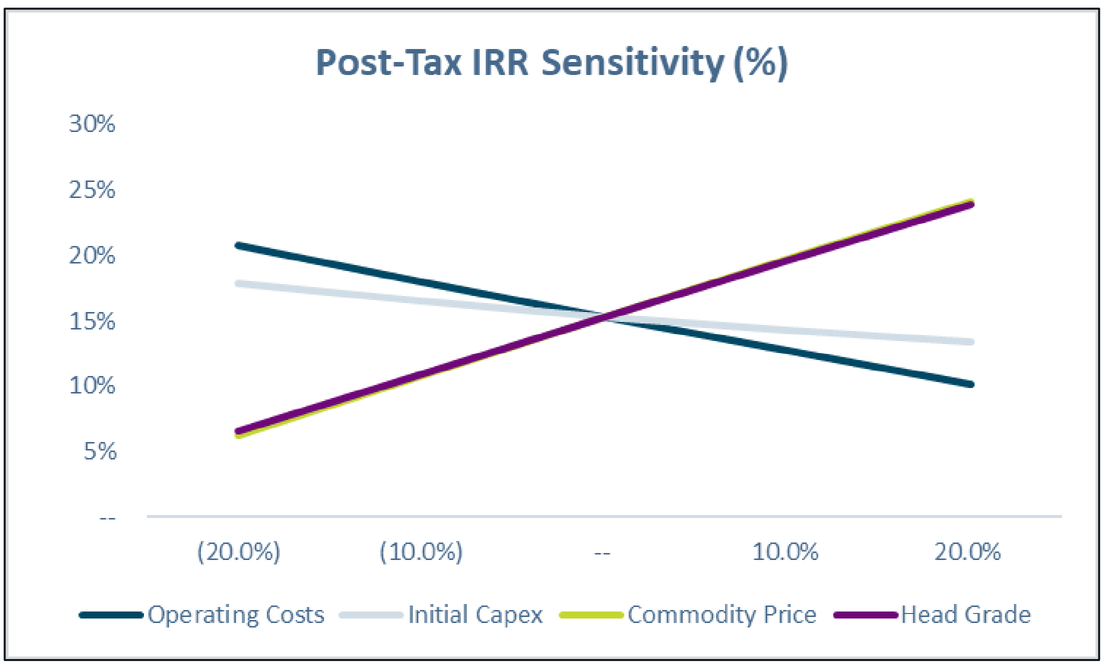
**Table 22-4: Pre-Tax Sensitivity Summary**

| | Pre-Tax NPV (US$M) Sensitivity to Discount Rate | | | | | |
|---|---|---|---|---|---|---|
| | | Commodity Price | | | | |
| Discount Rate | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | 3.0% | $600 | $1,215 | $1,830 | $2,445 | $3,060 |
| | 5.0% | $292 | $787 | $1,282 | $1,777 | $2,273 |
| | 8.0% | ($2) | $366 | $733 | $1,101 | $1,469 |
| | 10.0% | ($126) | $181 | $488 | $794 | $1,101 |
| | 12.0% | ($212) | $47 | $306 | $565 | $824 |
| | **Pre-Tax NPV (US$M) Sensitivity to Recovery** | | | | | |
| | | **Commodity Price** | | | | |
| Recovery | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | ($582) | ($288) | $6 | $301 | $595 |
| | (10%) | ($292) | $39 | $370 | $701 | $1,032 |
| | -- | ($2) | $366 | $733 | $1,101 | $1,469 |
| | 10% | $278 | $682 | $1,085 | $1,488 | $1,892 |
| | 20% | $445 | $869 | $1,294 | $1,719 | $2,143 |
| | **Pre-Tax NPV (US$M) Sensitivity to Initial Capital Cost** | | | | | |
| | | **Commodity Price** | | | | |
| Initial Capital Cost | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | $91 | $459 | $827 | $1,194 | $1,562 |
| | (10%) | $44 | $412 | $780 | $1,148 | $1,516 |
| | -- | ($2) | $366 | $733 | $1,101 | $1,469 |
| | 10% | ($49) | $319 | $687 | $1,055 | $1,422 |
| | 20% | ($95) | $272 | $640 | $1,008 | $1,376 |
| | **Pre-Tax NPV (US$M) Sensitivity to Operating Cost** | | | | | |
| | | **Commodity Price** | | | | |
| Operating Cost | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | $363 | $731 | $1,099 | $1,467 | $1,834 |
| | (10%) | $181 | $548 | $916 | $1,284 | $1,652 |
| | -- | ($2) | $366 | $733 | $1,101 | $1,469 |
| | 10% | ($185) | $183 | $551 | $919 | $1,286 |
| | 20% | ($368) | $0 | $368 | $736 | $1,104 |
| | **Pre-Tax NPV (US$M) Sensitivity to Head Grade** | | | | | |
| | | **Commodity Price** | | | | |
| Head Grade | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | ($582) | ($288) | $6 | $301 | $595 |
| | (10%) | ($292) | $39 | $370 | $701 | $1,032 |
| | -- | ($2) | $366 | $733 | $1,101 | $1,469 |
| | 10% | $288 | $692 | $1,097 | $1,501 | $1,906 |
| | 20% | $577 | $1,019 | $1,460 | $1,901 | $2,343 |

| | Pre-Tax IRR % Sensitivity to Discount Rate | | | | | |
|---|---|---|---|---|---|---|
| | | Commodity Price | | | | |
| Discount Rate | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | 3.0% | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 5.0% | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 8.0% | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 10.0% | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 12.0% | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | **Pre-Tax IRR Sensitivity to Recovery** | | | | | |
| | | **Commodity Price** | | | | |
| Recovery | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | 0.0% | 4.1% | 8.1% | 12.0% | 15.9% |
| | (10%) | 4.0% | 8.5% | 12.9% | 17.3% | 21.6% |
| | -- | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 10% | 11.7% | 17.0% | 22.3% | 27.6% | 32.9% |
| | 20% | 13.8% | 19.3% | 24.8% | 30.3% | 35.7% |
| | **Pre-Tax IRR Sensitivity to Initial Capital Cost** | | | | | |
| | | **Commodity Price** | | | | |
| Initial Capital Cost | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | 9.4% | 14.9% | 20.5% | 26.2% | 31.9% |
| | (10%) | 8.6% | 13.8% | 19.0% | 24.2% | 29.5% |
| | -- | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 10% | 7.4% | 12.0% | 16.6% | 21.1% | 25.6% |
| | 20% | 6.8% | 11.3% | 15.6% | 19.9% | 24.1% |
| | **Pre-Tax IRR Sensitivity to Operating Cost** | | | | | |
| | | **Commodity Price** | | | | |
| Operating Cost | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | 13.0% | 18.0% | 22.9% | 27.9% | 32.8% |
| | (10%) | 10.5% | 15.4% | 20.3% | 25.2% | 30.1% |
| | -- | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 10% | 5.5% | 10.4% | 15.2% | 20.0% | 24.8% |
| | 20% | 3.1% | 8.0% | 12.7% | 17.5% | 22.2% |
| | **Pre-Tax IRR Sensitivity to Head Grade** | | | | | |
| | | **Commodity Price** | | | | |
| Head Grade | | (20.0%) | (10.0%) | -- | 10.0% | 20.0% |
| | (20%) | 0.0% | 4.1% | 8.1% | 12.0% | 15.9% |
| | (10%) | 4.0% | 8.5% | 12.9% | 17.3% | 21.6% |
| | -- | 8.0% | 12.9% | 17.7% | 22.6% | 27.4% |
| | 10% | 11.9% | 17.2% | 22.5% | 27.9% | 33.2% |
| | 20% | 15.7% | 21.5% | 27.4% | 33.2% | 39.0% |

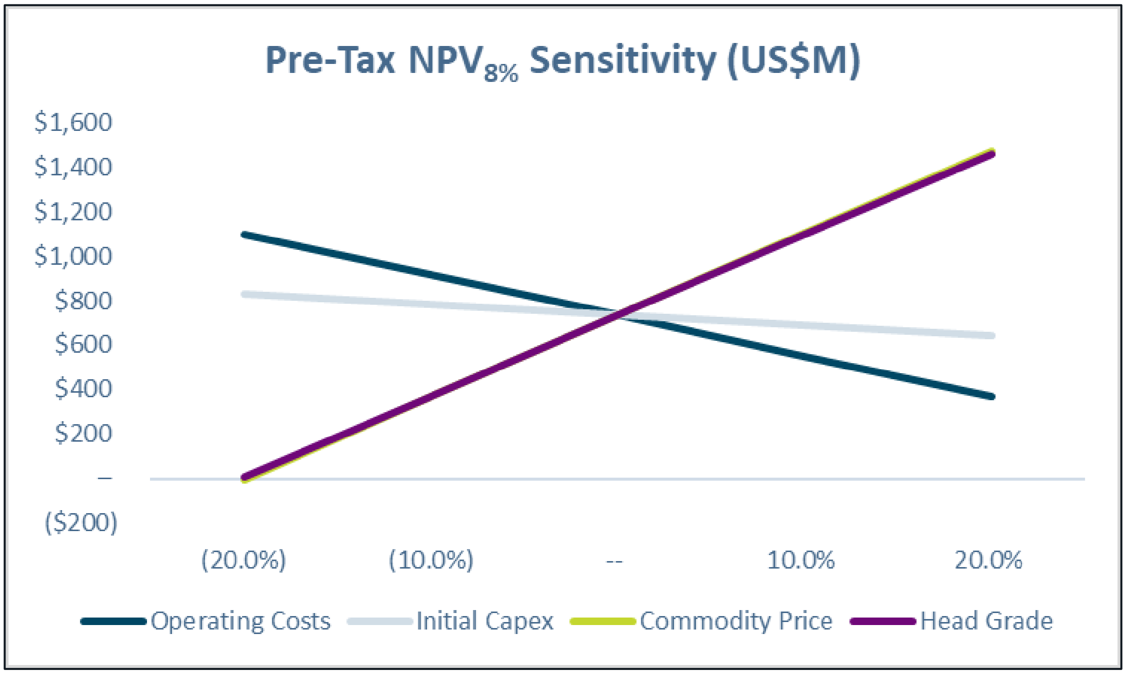
**Figure 22-2: Post-Tax NPV and IRR Sensitivity Results**

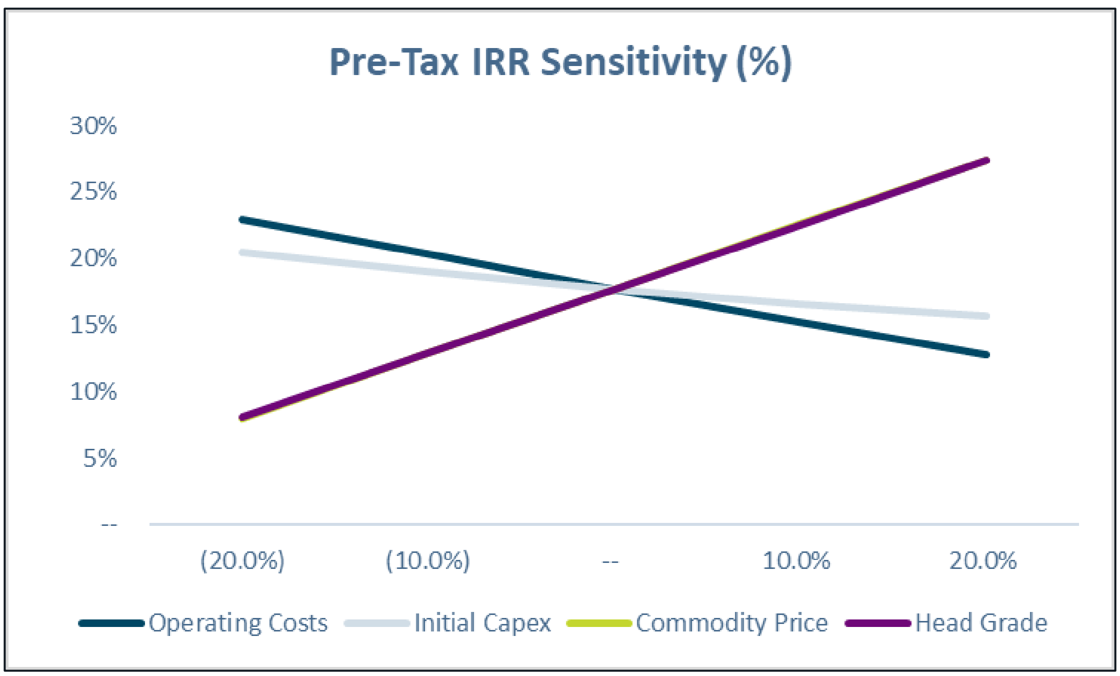



Source: Ausenco, 2024.

**Figure 22-3: Pre-Tax NPV and IRR Sensitivity Results**




Source: Ausenco, 2024.

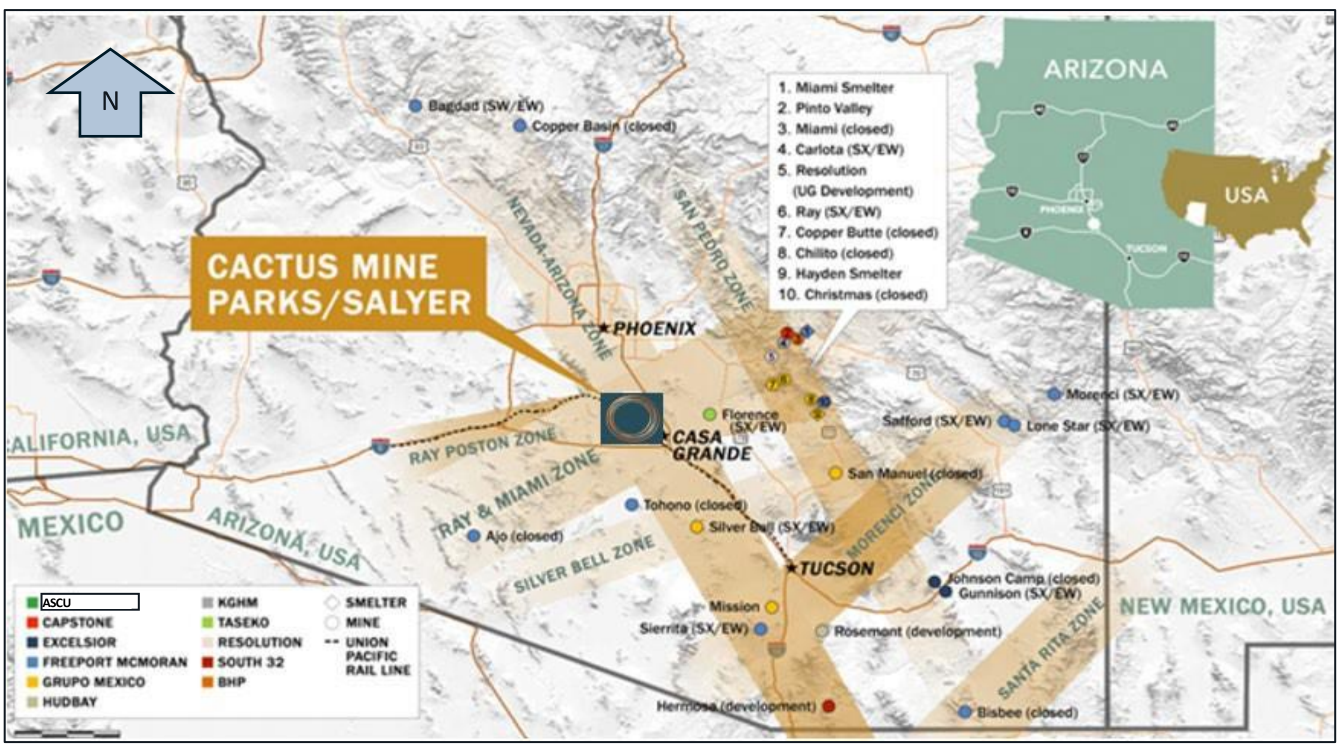
# 23 ADJACENT PROPERTIES

The Project, as shown in Figure 23-1, is surrounded by other, current and past-producing, copper deposit mines and similar processing facilities.

The nearest adjacent mineral property is the Santa Cruz copper porphyry deposit just over 2 mi (3 Km) southeast of the Cactus site and 7 mi (11 Km) 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 mi (11 Km) long and about a mile (1.6 Km) 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 mi (40 Km) ENE and the Ray Mine, owned, and operated by ASARCO LLC, a subsidiary to Grupo Mexico (approximately 50 mi ENE) of the Cactus Mine.

**Figure 23-1: Regional Copper Mines and Processing Facilities**




Source: ASCU, 2022.

# 24 OTHER RELEVANT DATA AND INFORMATION

There are no other relevant data and information required for this project.

# 25 INTERPRETATION AND CONCLUSIONS

## 25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

## 25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The Project is 100% controlled by ASCU through its wholly owned subsidiary Cactus 110 LLC, encompassing an area of approximately 5,370 acres of that (acres) is fee simple land, three ASLD prospecting permits that the State has surface and minerals (649.12 acres), two ASLD prospecting permits that the State has minerals only with ASCU owning the surface (797.5 acres) and 18 BLM unpatented mining claims, this is for mineral only as ASCU owns the surface rights (320 acres). The BLM unpatented mining claims are outside of the known mineralization and there are currently no plans for mining in this area.

The ASCU Cactus Mine project is a viable copper mining opportunity in a community that supports mining. The challenges facing the project are consistent with almost any mining operation in Arizona including risks associated with broad economic cycles that can impact profitability, unforeseen legislative and regulatory changes and maintaining a long-term water supply in a basin that has many groundwater users. None of the identified risks are insurmountable.

## 25.3 Geology and Mineralization

The Cactus and Parks/Salyer 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 and Parks/Salyer 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 sulphide 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.

## 25.4 Exploration, Drilling and Analytical Data Collection Supporting Mineral Resource Estimation

The Cactus and Parks/Salyer 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 and separate deposit at Parks/Salyer. 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 PFS. In addition, Arizona Sonoran drilled 184 core holes on the project 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). Five hundred eleven sonic holes have been drilled on the Stockpile Project to infill to approximately 200 ft (61 m) centres. 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.

## 25.5 Mineral Resource Estimate

The assays and geological logging described in the previous sections were used to generate three individual Mineral Resource Estimates (MRE). This involved the update and expansion of three previously generated resource block models. These models were created and updated using Vulcan Mine Planning Software. The three MREs referenced in this reports have the effective dates of;

- Cactus East and Cactus West – April 29, 2022
- Parks/Salyer – May 19, 2023
- Stockpile – March 1, 2022

Each of these models was updated with all available analyitial and geologic data available at the time of the effective date. All data used to generate and update the MRE followed the format, checks, and balances outlines in the CIM Best Practices Guidelines (2019).

The QP believes the geologic and analytical data collected to date is sufficient to support the generation of the resource statements for the Cactus, Parks/Salyer, and Stockpile deposits used in this report.

### 25.6 Metallurgical Testwork

Overall, the risks associated with the predicted metallurgical performance of the various resources at Cactus are consistent with other copper leaching projects. Copper recovery is expected be within a +/-5% (absolute recovery) window of certainty. Similarly, acid consumption requirements are also to be within a +/-10% (absolute net consumption) window of certainty. Metallurgical testing continues to further optimize the leaching protocols for the commercial operations.

A significant amount of metallurgical performance information has been developed for the design basis for the stockpile, Cactus East, and Cactus West resources. The work completed for these deposits is considered adequate for the level of study undertaken, PFS. Further optimization work related to reducing the acid consumption requirements is recommended.

Only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit. The work completed represents only a minimal metallurgical understanding of this deposit and additional confirmatory work is required to better understand the deposit variability.

Approximately 45 column tests have been completed (Stockpile - 25, Cactus – 14, Parks/Salyer - 6) covering the resources identified in the current study effort for processing. In addition, over 150 bottle roll tests, mineralogical analyses and other metallurgical and materials property testing has been completed.

Testing designed to support the final commercial protocols envisioned for the resources contemplated as the project basis for the PFS were developed and conducted by the ASCU technical staff in their facility located on site. A significant effort was expended to ensure adequate QA/QC records existed and test data integrity have not been compromised. The impacts are not considered high risk, but there still exists more risk than would normally be expected in the information developed. The next phase of testing should repeat these tests as part of the work to ensure that the results are repeatable and fully validated.

The QP believes the metallurgical testing and data collected to date is sufficient to establish the required supporting metallurgical performance expectations used in estimating the project Reserves for the Stockpile, Cactus East, Cactus West and Parks/Salyer deposits. However, only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit. The work completed represents only a minimal metallurgical understanding of this deposit and additional confirmatory work is required to better understand the Parks/Salyer deposit variability.

#### 25.6.1 Copper Recovery

The current testing does not show meaningful differences overall for each material type (stockpile, predominantly oxide and enriched) relative to lithologic types. The sequential assay method for copper mineralogical variability is an adequate proxy for copper leaching kinetic variability for each material type.

With the leaching configuration in mind, a maximum 3-year leaching cycle has been assumed (3 lifts) as the practical limit for effective recovery based on experience and preliminary hydrodynamic analysis of the materials by HGS. The copper leaching metallurgical test data has been extrapolated from the testing data at one year based on the rates prevailing after one year using a logarithmic curve fit projection that considers the decaying rate of copper extraction.

Scalability has been considered by 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. The recommended copper recovery projections include the efficiency factor applied to the expected extraction from column testing.

Based on the above, the recommended copper extraction estimates for use in evaluating the Cactus Project resources is presented in Table 25-1.

A production timing has been assigned for each material type corresponding to material mined in one year and the expected delays in achieving the two- or three-year final recovery vales. This factor is intended to account for material placement timing over the course of a year and leach cycle delays in subsequent new lift placements.

**Table 25-1: Cactus Project Copper Recovery & Production Timing Distribution Recommendations**

| Resource Area | Units | Value |
|---|---|---|
| Stockpile Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 87.7 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Leach Cycle Distribution - Year 1 | % | 75.0 |
| Leach Cycle Distribution - Year 2 | % | 25.0 |
| Leach Cycle Distribution - Year 3 | % | 0.0 |
| Oxide Heap (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 93.1 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Leach Cycle Distribution - Year 1 | % | 75.0 |
| Leach Cycle Distribution - Year 2 | % | 25.0 |
| Leach Cycle Distribution - Year 3 | % | 0.0 |
| Enriched Heap Leach (3/4" Crush) | | |
| Acid Soluble Copper Recovery | % | 91.2 |
| Cyanide Soluble Copper Recovery | % | 84.5 |
| Leach Cycle Distribution - Year 1 | % | 65.0 |
| Leach Cycle Distribution - Year 2 | % | 30.0 |
| Leach Cycle Distribution - Year 3 | % | 5.0 |

Applying these extraction criteria, the calculated overall soluble copper (Tsol) recovery to cathodes is 86.3% and the corresponding total copper recovery is 76.1% for the resources contained in the mine plan.

### 25.6.2 Acid Consumption

Sulfuric acid consumption per ton of materail leached is dependent on several factors. Gross acid consumption varies by materail type in each deposit. Net acid consumption acounts for acid regenrated in the electrowinning process when copper is plated to product. Net acid consumption per ton of material is dependent on recoverable copper content with a stochiometric converion of 1.54 tons of acid generated per ton of copper plated in electrowinning.

The stockpile material is more complex given that there are no geologic constraints to apply and waste materials have been mixed in. Calcium was determined to provide a measurable indicator for acid consumption and a calcium distribution model was developed and applied to estimate gross acid consumption. For the materails included in the mine plan, the average gross acid consumption averages 22 lbs of acid per ton of material and ranges from 16.7 lbs/ton to 25.7 lbs/ton depending on the average calcium content annually.

For Cactus East and West materials, the gross acid consumption for the oxide dominant material is 22 lbs/ton from the column testing. Enriched material gross acid consumption is slightly lower at 21 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals.

For the Parks Salyer enriched material, gross acid consumption is lower at 16 lbs/ton due to the contribution of sulphur contained in the sulfide copper minerals, lower clay, biotite and calcite mineralogy when compared to Cactus samples tested.

Applying the specific gross acid consumption for each material the overall LOM gross acid consumption is calcualted to be 19.3 lbs per ton and varies from 27.0 lbs/ton to 15.7 lbs/ton in a given year. The LOM Net acid consumption is calculated to be 6.5 lbs per ton and varies from 15.7 lbs/ton to net acid generating in a given year. Years where acid regenrated exceeds acid required to be consumed will need to be attenuated with low grade/high calcium content material from the stockpile or tailings.

Similar to copper recovery, acid consumption is distributed over a two-year period with 75% of the estimated acid requirement consumed in the year of placement and 25% of the requirement in the following year.

## 25.7 Mineral Reserve Estimate

The mineral reserve estimate for the Arizona Sonoran Copper (ASCU) Cactus Project conforms to industry-accepted practices and were prepared in accordance with the guidelines of NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum definition Standards for Mineral Resources and Mineral Reserves (“CIM Standards”).

Factors that may affect the estimate include: changes to long-term metal price assumptions, metal recovery assumptions, mine design and ground support, open pit slopes, ore recovery and dilution, mine sequencing, ability of the mining operation to meet the annual production rate, operating cost assumptions, leach pad performance, the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

There is an opportunity to increase reserves if additional near surface resources continue to be identified at Parks/Salyer which may allow additional options to be considered including open pit mining transitioning to underground methods.

## 25.8 Mining Methods

The mineable resource for the Cactus Project includes the Cactus deposit (West and East), Parks/Salyer deposit and the Historical stockpile. The deposits are planned to be developed using conventional open pit mining and underground mining methods. Cactus West (CW) and the Stockpile Project (SP) will be developed using conventional open pit mining. Underground, the sub level caving method (SLC) will be used for the Cactus East and Parks/Salyer deposits.

The mine schedule plans to deliver 276.3 Mt of heap leach feed grading 0.549% total copper over a project life of 25 years.

The open pit mine schedule consists of 75.5 Mt of proven and probable ore at a 0.31% total copper grade from the Cactus West pit plus an additional of 76.8 Mt of probable ore grading 0.16% total copper from the historical stockpile. The Cactus West pit will be mined over seven years including one year of pre-production. The Historic stockpile will be mined concurrently with Cactus west pit for slightly more than eight years. Waste tonnage totaling 147.9Mt from the Cactus pit and Historic stockpile will be placed in the viewshed berm and northeast waste storage area. The Cactus West pit strip ratio is 1.9:1 (waste:ore), whereas the Historical stockpile has a 0.1:1 (waste:ore) strip ratio.

Underground mineable reserves are classified as probable and amount to 27.7 Mt at 0.95% total copper for Cactus East, and 96.2 Mt at 0.93% total copper for Parks/Salyer. Underground development begins at Cactus East in year 9 with ore production from caving ramping up during year 11. Parks/Salyer development starts in year-1 with minor tonnages of ore released in year 1-to year 3. Significant and sustained ore release in excess of 2.5 million tons per year starts in year 5 onwards. Production from Cactus East ends in year 19 with Park/Salyer production lasting into year 20 of the project schedule. The mining layout and sequence for Parks/Salyer will be one of the largest footprints attempted using the SLC method within comparatively weak rock mass conditions. Current mine designs are based on geotechnical guidance; however, more advanced geotechnical modelling will be required to better understand the rock mass response and flow behaviour in order to optimise the mine plan.

Heap leach ore will be tertiary crushed and stacked on a single heap leach facility that will provide copper in solution to the SX/EW facility.

SLC footprints are highly variable in size and production capacity which is important when selecting industry examples for comparative benchmarking. However, the essential design criteria such as drive configurations, ring designs, ramp up profiles, draw rate and flow behaviour are generally well understood and have been effective in achieving good productivity and ore recovery. The factors which impact production tend to be related to geotechnical conditions and fragmentation. Strict draw control practices are also required as deviations to the optimum draw sequence can lead to elevated dilution and reduced ore recovery. Cave fronts on a level shall ideally lead the fronts on the next lower level in a 45° angle. Within an operating level, there are also lead/lag rules of rings offset and dependencies. There is potential for optimising the lead angles and dependencies in future studies.

The Cactus East SLC footprint size is within typical spans that have been adopted elsewhere. The Ridgeway, Telfer, Carrapateena and Ernest Henry SLC mines had similar mining spans and have achieved peak production rates close to 6 Mt/y (16,500 ton/d),Table 25-2. These mines have twice the rock strength of Cactus East, however this is not expected to impeded production or recovery based on current information.

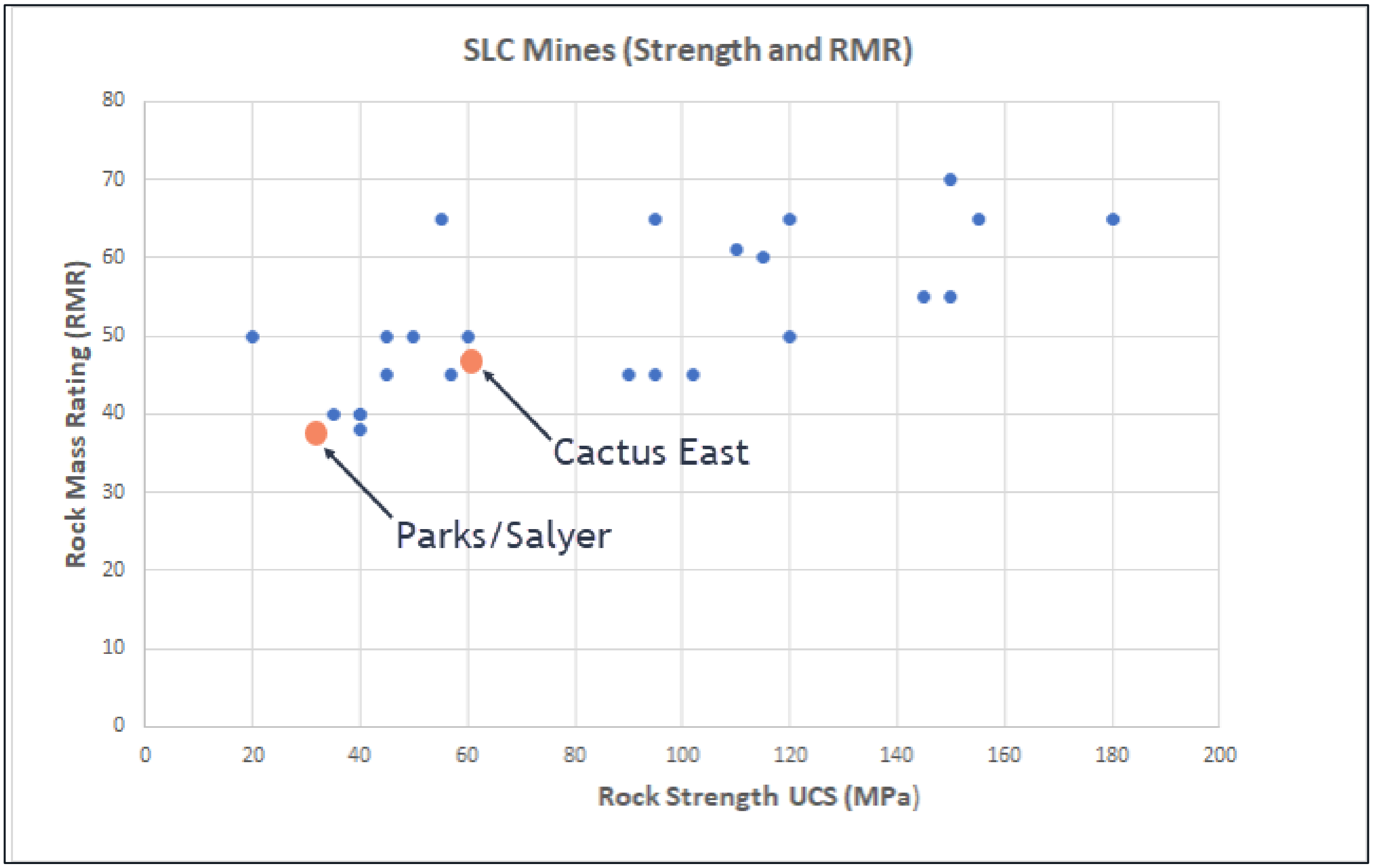
**Table 25-2: SLC Mining Spans and Production Rates**

| | Approx Width (ft) | Approx Crosscut Lengths (ft) | Peak Rates (Mt/y) |
|---|---|---|---|
| Cactus East | 1050 | 500-650 | 3.5-3.9 |
| Carrapateena | 1000 | 650 | 4 |
| Ridgeway SLC | 1000-1150 | 650 | 6 |
| Telfer | 2600 | 650 | 5.6 |
| Ernest Henry | 800 | 750 | 6.8 |
| SLC with Weak Rock Mass | | | |
| Parks/Salyer | 600 | 500-1650 | 6.5-7.0 |
| Black Rock | 500 | 230 | 0.42 |
| Diavik/Ekati | 650 | <325 | <1.0 MTPA |
| Koffefontein | N/A | N/A | 1.1 |
| Perseverance | 650 | 500 | 1.5 |

The Parks/Salyer deposit will have some of the weakest rock mass conditions that will be developed by the SLC method (Figure 25-1). There are no other SLC mines that can be directly compared against Parks/Salyer when considering a combination of geometry and geotechnical conditions. SLC mines with weak rock mass conditions similar to Parks/Salyer have considerably smaller geometries with production capacities under 1.5 Mton/y.

Provisions have been made in the mining schedule for slower advance rates and increased ground support costs in weaker zones. The stability and integrity of the production drives will be essential in ensuring effective ore recovery and controlling dilution.

**Figure 25-1: Rock Strength and RMR at SLC Mines**




Source: NCL 2023.

A review of benchmarking subsidence zones indicates that each site is unique geologically and will therefore respond differently according to local geotechnical conditions, the mining geometry and draw strategy applied. Major faults have been shown elsewhere to influence the rate of cave propagation and the final breakback angles. In some cases, vertical faults can steepen the cave profiles, whilst in other cases shallow dipping structures cause low angle breakouts particularly when they are close to surface. Changes in cave angles is not expected to impact reserves directly but there may be impacts on capital costs and production constraints if critical infrastructure is affected.

Numerical modelling simulations are required to better understand the likely local rock mass response to caving once the necessary structural and geotechnical information becomes available. The interaction of the Cactus East cave with the Cactus West pit and the likely influence of major faults needs to be analysed to confirm locations for critical infrastructure such as the in-pit portal and vent shafts.

Wet muck is a risk to the project, as several of the conditions conducive to wet muck are present. General criteria for liquifiable soil leading to wet muck events is presented in Table 25-3 along with soil conditions from the Casa Grande region.

**Table 25-3: General Criteria for Liquifiable Soil**

| Liquifiable Soil Conditions | Average Conditions in Casa Grande Region | |
|---|---|---|
| Liquid Limit (LL) | < 35 | 25 |
| Plasticity Index (PI) | < 9 | 10 |
| % Sand | > 20 | 60 |
| % Clay | < 15 | 20 |
| Saturation | > 80% | N/A |

*80% saturation is typically 10-13% moisture content for sand.

## 25.9 Recovery Methods

The current processing plant design is adequate for the mining and leaching plans considered in this PFS. A modular plant design has been considered using prefabricated components and fiberglass construction due to the potential for high chloride content in the leaching solutions and make-up water.

The solvent extraction facility is designed to operate in series, series-parallel and all parallel configurations allowing for variations in PLS flowrates and copper content that are expected over the course of the project life. PLS flowrate configuration options are 4,000 gpm, 8,000 gpm and 12,000 gpm.

The crushing and screening plant employs used equipment that has been partially installed but never operated. While used equipment is typically not considered at this stage of study, ASCU has advanced commitments to the equipment broker to establish as reasonable expectation for acquisition. The Trekkopje project MAXI Phase incorporates all mechanical and electrical gear specific to twin Primary Crusher Relocatable Sizer Stations, twin secondary/tertiary crushing and screening circuits, three parallel agglomeration circuits, all interconnecting in-plant conveyors and feed mechanisms for a combined design capacity of 7,870stph. This system includes both a plant compressed air system and uninstalled Donaldson dust extraction system with six separate baghouses which were intended to provide collection at the various process steps and material transfer points.

The designs considered are believed to be suitable for treatment in a crushed ore heap leach, solvent extraction, and electrowinning (SX/EW) process facility to produce copper cathodes at LME Grade A quality standards ASTM B115-10 - Cathode Grade 1 on consistent basis with appropriate operating practices.

The SXEW plant layout allows for the future production increases. The addition of one SX train and associated tank farm equipment can be included adjacent to the current train. The EW facility is designed to be doubled in capacity with the addition of a second production bay opposite to the stripping machine and initial production bay.

No work was completed in terms of SXEW performance or piloting, and none is deemed necessary given the well understood nature of the process and design conditions. No deleterious elements have been found in the ore samples tested or the resulting leach solutions and residues analyzed.

There does exist a potential for higher chloride content in process solutions related to the ground water and possible make-up water sources. Continued development of this possible impact to the SX plant configuration (an organic wash stage may be required) and materials of construction is warranted. In anticipation of potential concerns and future technological options, the SX plant is contemplated as a fiberglass-based design.

As higher-grade copper ores are leached, the amount of acid regenerated through the SXEW operation will increase. There is a potential for an excess amount of acid can be returned to the leaching system than can be consumed by the gangue materials. The result would be lowering of the PLS acid pH (increased acid content) and significantly reducing the extraction efficiency of the SX operation. Balancing high and lower grades in the mining operation may be an option to avoid this possibility. Additional, higher acid consuming materials (e.g. high calcium content stockpile materials) may also be considered are part of the acid management strategy.

## 25.10 Infrastructure

### 25.10.1 Project Infrastructure

Planned infrastructure for the mine facility includes a truck shop and wash area, a mine office with two change rooms, explosives storage facilities, a diesel fuel island, ROM (run-of- mine), wells and two underground portals, and an underground crushing circuit with transportation to the surface where it will connect with the land conveyor that also serves a crushing facility near the well. All associated mining and crushing electrical infrastructure will be located near this facility.

Common facilities include an entry/exit guard house that will house site security and medical/health and safety personnel, a general site administration building, fresh water distribution systems, a main substation and associated power generation and distribution. facilities, communications area and sanitation systems.

The site design is configured to optimize material handling synergies between existing mine stockpile, existing Cactus West open pit and the underground resource called Cactus East located northeast immediately adjacent to the existing Cactus open pit, minimizing the environmental footprint and prioritizing utilization of nearby land and existing facilities to ensure operational scalability following resource expansion. The project plans will take advantage of existing infrastructure, such as high voltage power supply near the property, existing and paved access roads, as well as dirt roads that will lead to different infrastructure within the property. In conclusion the proposed infrastructure is inline with the Cactus Mine initial needs for start-up operations taking advantage of some of the existing infrastructure.

### 25.10.2 Heap Leach Facility

The HLF design can hold the current production volumes. The facility phasing will work with the construction schedules to allow for the appropriate leaching cycles while keeping a lower capital cost. However, based on the on-going testing, it is possible that additional area for leaching may be required to support the additional ore reserves that may arise throughout the mine life. The current HLF layout allows for a future HLP to be built on the SE corner of the current design and utilize the southern pipe corridor to convey future flows to the PLS and Event ponds.

## 25.11 Environmental, Permitting and Social Considerations

Due to historic mining operations, the project site is considered a Brownfields Project. ADEQ entered into a Prospective Purchaser Agreement (PPA) with Elim, ASCU's predecessor, because of the substantial public benefit to the remedial work conducted at the site. The PPA releases ASCU from potential liabilities related to existing, known contamination under CERCLA, WQARF, and RCRA, but does not cover unidentified environmental conditions or contamination. No environmental fatal flaws that would materially impede the advancement of the project have been identified.

There is no federal nexus for permitting of the project, reducing potential permitting delays. Of the permits/authorizations/notifications listed in Section 21.2, the APP will likely require the most review time by regulators. An APP Significant Amendment (without a public hearing) has a licensing timeframe of 221 business days. Other permits/ authorizations/and notifications have relatively short turnaround times.

## 25.12 Capital Cost Estimate

The initial and sustaining capital cost estimate conforms to Class 4 guidelines for a prefeasibility-level estimate with a -20% to +30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q4 2023 US$ based on budgetary quotations for equipment, contractor's costs, in-house data from projects and studies as well as experience from similar operations.

## 25.13 Operating Cost Estimate

The operating cost estimate was developed in Q4 2023 dollars from budgetary quotations and in-house database of projects and studies as well as experience from similar operations. Mine operating costs have been estimated from base principals using quotations from local mine equipment vendors plus local supply consumables. The accuracy of the operating cost estimate is -20% to +30%. The estimate includes mining, processing, and general and administration (G&A) costs. For more details, refer to Section 21.5.

## 25.14 Economic Analysis

Based on the assumptions and parameters in this report, the PFS shows positive economics of US$508.7M post-tax NPV (8%) and 15.3% post-tax IRR.

## 25.15 Risks and Opportunities

### 25.15.1 Risks

The risks and uncertainties associated with the project are related to litigation, economics, regulatory developments, and financing.

#### 25.15.1.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

ASCU is in litigation with RAMM Power Group, which wishes to acquire the project site through imminent domain. This risk is considered low, as the cost to acquire the property, considering the value of the mineral resource, is prohibitive.

#### 25.15.1.2 Metallurgical Testwork

Overall, the risks associated with the predicted metallurgical performance of the various resources at Cactus are consistent with other copper leaching projects. Copper recovery is expected be within a +/-5% (absolute recovery) window of certainty. Similarly, acid consumption requirements are also to be considered to be within a +/-5% (absolute consumption) window of certainty. Metallurgical testing continues to further optimize the leaching protocols for the commercial operations.

A significant amount of metallurgical performance information has been developed for the design basis for the stockpile, Cactus East and Cactus West resources. The work completed for these deposits is considered adequate for the level of study undertaken, PFS. Further optimization work related to reducing the acid consumption requirements is recommended.

Only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit. The work completed represents only a minimal metallurgical understanding of this deposit and additional confirmatory work is required to better understand the deposit variability.

Testing designed to support the final commercial protocols envisioned for the resources contemplated as the project basis for the PFS were developed and conducted by the ASCU technical staff in their facility located on site in an existing building (Truestone facility). The testing facility was visited, and testing protocols reviewed with the ASCU team by the QP.

During the conductance of the site-based work, records and information were not kept to the same standard as commercial laboratories and a significant effort was expended to ensure adequate QA/QC records existed and test data integrity have not been compromised. While most of the information was eventually pieced together and validated, there remain concerns about some aspects of the information. The impacts are not considered high risk, but there still exists more risk than would normally be expected in the information developed. The next phase of testing should repeat these tests as part of the work to ensure that the results are repeatable and fully validated.

#### 25.15.1.3 Mineral Reserve Estimate

- The underground mineral reserve estimate is based on strict adherance to the production sequence and draw rates as detailed in the production schedules. Deviations to the mining sequence and/or local geotechnical conditions may impede access or alter recoveries in some areas.
- The dilution and ore recovery estimates for the SLC method assume uniform flow behaviour within the cave. Variations in flow rates due to changes in fragmentation could impact rate of dilution ingress. These impacts are expected to be minimal as significant overdraw rates, where dilution could vary due to differential draw, are not planned until the lowermost levels are mined.Prolonged inactive draw in some areas could encourage re-compaction of the caved rock which may impact flow behavior and recoveries due to stagnant flow. Further work in optimization the draw sequence may be necessary to maintain caving mobility.
- There is some uncertainty in the behaviour of the rock mass due to the signficant step-out distances between sublevels at Parkes/Salyer. Some provisions have been made in the mining schedule, however, further studies are required to assess the rock mass response to optimise the mining schedule and adjust ground support provisions. Ore recoveries and dilution may be impacted by such changes.

#### 25.15.1.4 Mineral Resource Estimate

##### 25.15.1.4.1 Resource Expansion

- 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 within 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.

##### 25.15.1.4.2 Stockpile Resource Estimation

- 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 within 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.

#### 25.15.1.5 Mining Methods

- Mine design and modifying factors for the SLC mine are based on the geotechnical constraints. More detailed geotechnical analysis is required assess the rock mass response to mine development and planning. This could impact design configurations, production layouts, and mine sequencing.
- There is presently limited drilling information along the access development to the underground resources and related infrastructure. A targetted drilling program is required to assess structural and geotechnical conditions. The results of future work may alter the decline access path and critical surface and subsurface infrastructure locations.
- The Cactus East portal location is planned to be located in the Cactus West open pit. Numerical modelling of the subsidence zone and pit wall interactions is required to verify the suitabilty of the portal location as well as the position of ventilation raises and production shafts. Changes to portal or vertical development locations could impact development costs and mine scheduling.

#### 25.15.1.6 Geotechnical

- Much of the Gila Conglomerate contains large clasts (up to several ft in diameter). This will cause delays in roadheader advance rates and delays due to additional ground support requirements where large clasts are dislodged, leaving unstable pockets.
- There are portions of the proposed Cactus East decline parallel to and within 200 ft of the LOM pit shell. This will likely position the decline within the zone of rock mass yielding. Numerical modelling will be required to understand the extent of this “no go” zone so it can be avoided to reduce ground support requirements.
- There will always be uncertainty between the predicted ground conditions and the actual field conditions. Additional drilling is ongoing to better characterize ground conditions throughout the project area and improve confidence in predictions.
- The geotechnical data (Q) necessary to estimate ground support requirements is inadequate. Due to this, the geotechnical block model estimation of Q relies heavily on drill holes which have only RQD data to estimate NGI Q system parameters. This estimation method has uncertainty and support requirements could vary substantially from what is currently predicted. Where possible, the methodology to estimate Q from RQD was compared to

logged values and the estimation method was found to under predict Q, which suggests there could be opportunity to reduce ground support requirements with additional geotechnical drilling.

- The geotechnical block model and all analyses are based on logged geotechnical data from core holes which includes fracture statistics (RQD) and joint conditions (number of joint sets, joint alteration, and joint roughness) to estimate the modified NGI Q system of rock classification. However, many empirical methods rely on characterization using the full NGI Q system which also considers in-situ conditions of the rock, such as stress and water factors, that are not captured in core logging. The decline pathway in particular, being within conglomerate, may be of poorer quality than currently predicted because these stress and water factors have not been considered. Additional evaluation should be conducted to assess the impact of the weak rock mass and residual water inflows and pore pressures during tunnelling.
- All analyses assume generally dry conditions and that the mining areas are effectively depressurized. If there is residual water within the rock-mass surrounding the excavations, or depressurization is incomplete, then the stability of openings and ground support performance will be less than predicted.
- Wet muck is a risk for the sublevel caving operations. While left unchecked, this poses a significant risk to personnel and equipment. Managing wet muck can be achieved by allowing proper drainage time; however, this will result in delays and reduced production.
- Structural geology is not currently well understood in the underground mining targets. Major faults have been modeled but have not been characterized, and secondary faults and dikes are not well understood or identified in most areas outside of the existing open pit.
- The northeast side of the Phase 2 Cactus pit has a tall slope in Gila Conglomerate. Additional drilling in this wall for sampling, laboratory testing, and piezometer installations is recommended to confirm Gila Conglomerate stability.
- Overall slope stability analyses do not include strength anisotropy. Once access to the pit is reestablished, mapping should be conducted to further characterize the rock fabric in the Oracle Granite and determine structural control on slope stability.

#### 25.15.1.7 Recovery Methods

No work was completed in terms of SXEW performance testing and piloting, and none is deemed necessary given the well understood nature of the process and design conditions. No deleterious elements have been found in the ore samples tested or the resulting leach solutions and residues.

There does exist a potential for higher chloride content in process solutions related to the ground water and possible make-up water sources. Continued development of this possible impact to the SX plant configuration (an organic wash stage may be required) and materials of construction is warranted. In anticipation of potential concerns and future technological options, the SX plant is contemplated as a fiberglass-based design.

As higher-grade copper ores are leached, the amount of acid regenerated through the SXEW operation will increase. There is a potential for an excess amount of acid can be returned to the leaching system than can be consumed by the gangue materials. The result would be lowering of the PLS acid pH (increased acid content) and significantly reducing the extraction efficiency of the SX operation. Balancing high and lower grades in the mining operation may be an option to avoid this possibility. Additionally, higher acid consuming materials (e.g. high calcium content stockpile materials) may also be considered are part of the acid management strategy.

The crushing and conveying system included in the project design are based on used equipment ASCU is negotiating the purchase of with a broker (A.M. King) for the Trekkopje project materials handling facility located in Namibia. The used facility has a slightly oversized throughput, is partially installed and has not been operated. There is a risk that the negotiations may not be concluded, and the equipment is not available as included. The relative capital cost differences between the used equipment and "new" comparable equipment are shown below:

1. PFS Crushing & Conveying (3.0k tph) CapEx = $52.6M (new equipment basis)
2. Namibia Plant (3.9k tph) CapEx = $29.1M (same equipment list as above), with Namibia plant "used" pricing from A.M. King)
3. Namibia Plant (7.5k tph) CapEx = $56.8M (same equipment list as above), with Namibia plant "used" pricing from A.M. King)

Should "new" equipment be necessitated for the materials handling systems, initially approximately $23.5M additional capital will be required.

Securing the complete Trekkopje circuit will provide additional benefits in terms of project schedule and cost reduction, project execution and the ability to easily expand the circuit at marginal cost to 7,870stph, consistent with the increased capacity requirements that would be required for the treatment of the additional tonnage primary ore business cases as proposed by NUTON. For the expanded facility, an additional project capital cost of $27.7M is estimated to be required.

#### 25.15.1.8 Infrastructure

##### 25.15.1.8.1 Project Infrastructure

Risks related to project infrastructure are:

- Support building facilities not sized for increased production.
- The concrete floor in the Truestone building is assumed to be in good condition and designed to accommodate heavy vehicle traffic.

##### 25.15.1.8.2 Potable Water

The elevation of the potable water tank should be verified. The current design provides sufficient hydraulic force if the level of water in the tank is at least 12 ft deep. This is based on topographical data from Google Earth. If the local elevation is higher, the tank design and water supply method will have greater flexibility.

#### 25.15.1.8.3 Plant Water

The WWTF has the capacity to provide ample make-up water, but the contract with the city needs to be confirmed.

The quantities of water expected from the geotechnical assessments have not been confirmed. The actual amounts available may be less than those predicted. The quality of water obtained from planned wells needs to be tested to ensure its adequate safety for the intended equipment applications such as washing trucks, grinders, etc.

The water obtained from the Sacaton Pit is acidic with an approximate pH of 2.6 and in addition a copper concentration of 2 g/L, characteristics that require processing in the SX Plant facility for leaching applications. This fluid must not be allowed to mix with other plant water sources designated for washing, grinding, or dust control.

#### 25.15.1.8.4 Heap Leach Facility

Risks associated with the HLF are related to the possibility of an increase in ore reserves which could require additional facilities. The current design is restricted in height and area, any additional ore reserves shall be placed in a separate HLF and within ASCU property.

#### 25.15.1.9 Environmental, Permitting and Social Considerations

Economic risks include copper prices, stock market volatility, and interest and currency rates. These factors are not controllable by ASCU. However, the outlook for copper demand is generally positive. Higher interest rates will affect financing costs; ASCU has factored this into the economic model.

Legislative and regulatory developments are a potential risk. However, ASCU knows of no planned or pending legislation that will adversely affect the project.

### 25.15.2 Opportunities

#### 25.15.2.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Additional copper resources may exist in the area and would provide a substantial opportunity for future expansion.

#### 25.15.2.2 Metallurgical Testwork

The current acid consumption prediction approach uses average of all non-duplicative tests to date – a more refined methodology may be possible now with a calcium distribution model for the stockpile and additional testing inputs. Calcium is present as calcite, gypsum and other minerals in the stockpile and other Cactus deposit areas.

Building from the current testing protocols, additional testing was completed to show the potential to improve gross acid consumption by continuing to minimizing excess acid available in the leaching system with both flowrate and acid content reductions.

At average stockpile copper (~0.20% Cu) and calcium contents (~ 0.20% Ca), historical and current testing supports potential for reduced acid consumption based on calcium resource modeled for the stockpile upper lifts, there is a potential to reduce acid consumption by 20% to 40% (from a gross acid consumption of 22 lbs/ton to approximately 15 lbs/ton) by lowering leach solution flowrates to 6 L/h/$m^2$ and acid concentration (5 gpL to) the stockpile materials.

Based on the current calcium modeling information, the upper two lifts of the stockpile contain a majority of the copper while also having lower calcium content. The current mine plan extracts material from the stockpile through a sequence of eight cuts. The initial four cuts contain 41.4 million tons of leach material with lower calcium content.

Further work is required to verify this opportunity, the main elements are:

- Resource calcium model and drill spacing needs to be improved from 400 ft to better understand calcium distributions in an "as-mined" distribution and annual/monthly production averages.
- Additional column testing is required to further delineate the methods and quantify the opportunity to manage acid usage for the stockpile, Cactus and Park Salyer leach materials.

Stockpile material with higher calcium content and resulting acid consumption (mining cuts 5-8) can be stored for future use as acid mitigation for higher grade ores, a separate pad area is advised.

Net acid consumption (new acid required to be purchased) is influenced by the amount of copper recovered from each ton of material leached and plated in the electrowinning operation. The chemical reaction in this operation hydrolyzes water as copper is plated to regenerate sulfuric acid that is transferred to the SX operation as electrolytes are recirculated in the stripping part of the operation. Acid is transferred to the leach systems as copper is transferred from the leaching solutions and replaced with hydrogen ions generating acid returning to the leaching operation. The more coper that can be leached from a ton of material leached, the higher the return acid content will be off setting the gangue acid consumption.

The project is studying the potential for incorporation of the Nuton™ copper leaching technology. The technology is currently being tested at the Nuton technical facilities as a parallel project development. Economic leaching of primary copper mineralization could significantly change the mining plan concepts/sequence and improve overall project economic value.

#### 25.15.2.3 Mineral Reserve Estimate

- There may be an opportunity to increase reserves if additional near surface resources continue to be identified at Parks/Salyer which may allow for additional options to be considered including expansion of open pit mining and/or commencing SLC production from higher elevations.
- The mineralised zones, particularly at Parks/Salyer, appear to be more highly fractured compared to the cover rocks. Higher mobilities of flow from the mineralised zone may assist in limiting or reducing dilution entry from the non-mineralised cover assemblage.

#### 25.15.2.4 Geotechnical

- Accelerated development rates may be possible through the use of roadheaders based on the high estimated instantaneous cutting rates. Additional study of overall advance rates that account for pick consumption, installation of ground support, and utility advancement is required to confirm this.
- Alternative ground support types should be considered which could optimize lengths and installation density of bolting options.
- The Gila Conglomerate is a weak but generally massive unit, with sub horizontal bedding partings. Due to this, there is opportunity to support the ribs with only mesh and fibercrete and minimize or eliminate rib bolting.
- In this study, bench analyses have been conducted via photogrammetry on weathered benches. In-pit mapping is needed to confirm the structural fabric controlling slope stability. Freshly blasted benches may perform better than estimated. Controlled blasting, including pre-split blasting, may provide opportunity for steeper slope angles in the Oracle Granite.

#### 25.15.2.5 Recovery Methods

Opportunities related to the processing areas are limited to continued optimization of the equipment selection and requirements.

Used equipment that can be verified and confirmed through an executable agreement may have some applications in areas other than the materials handling facilities currently contemplated.

Incorporation of the Nuton™ leaching technology is being studied for primary copper mineralization and the plant design development should consider any impacts related to that application when introduced.

Nuton is a Rio Tinto venture bio-heap leaching technology that has the potential to produce copper from sulfide copper resources that were previously too technically challenging or too costly to process in any other way. This unique technology has shown promising results with primary sulphide ore which underlies the oxide and enriched sulfide (primarily chalcocite) at the Cactus project.

Nuton doesn't generate tailings and eliminates the need for concentrating, smelting, and refining of sulphide copper. In a single integrated process, Nuton technology has the potential to produce a high-quality copper cathode on site from an estimated 380 million tons of primary sulfide through SX-EW processing.

#### 25.15.2.6 Infrastructure

##### 25.15.2.6.1 Project Infrastructure

Opportunities related to project infrastructure include:

- A detailed evaluation of the primary access road could provide opportunity to reduce repairs to specific areas that are in need of repair. For this study it was assumed that the entire road surface would be rehabilitated.

- Plant water use includes a high demand for dust suppression. It is possible to reduce the use of water by adding surfactants, gravel, or pavement to reduce dust from the roadways. This should be evaluated as a way to minimize the environmental impact and preserve water resources.

#### 25.15.2.6.2 Heap Leach Facility

With the possibility of an increase in ore reserves, ASCU has the opportunity to acquire additional real estate to accommodate a new HLF. The current configuration of the facility will allow for a new pad to be built at the southern area of the HLF and utilize the pipe corridor to convey any new solution flows to the existing ponds.

#### 25.15.2.7 Environmental, Permitting and Social Considerations

The site's status as a pre-existing mine is helpful to engendering support from the community. Mining projects on previously undeveloped land generally raise concerns regarding habitat and other environmental impacts from nearby residents and environmental groups. ADEQ, through a prospective Purchaser Agreement, has released ASCU from any potential liability associated with the legacy environmental issues at the site, based on investigations and remedial efforts conducted. The "brownfields" status of the project presents an opportunity for ASCU to engage with the community regarding the work that has been done to address legacy environmental issues.

# 26 RECOMMENDATIONS

## 26.1 Introduction

The QPs note the following recommendations for their respective areas of expertise, based on the review of data available for this report.

**Table 26-1: Summary of Budget for Recommendations**

| Budget Item | ($M) |
|---|---|
| Exploration and Drilling | 20.0 |
| Metallurgy and Process Design | 0.1 |
| Metallurgical Testwork | 0.9 |
| Mineral Resource Estimates | 0.05 |
| Mineral Reserve Estimates | 0.1 |
| Open Pit Mine Design and Scheduling | 0.3 |
| Underground Mine Design and Scheduling | 0.8 |
| Mine Capital and Operating Cost Estimation | 0.1 |
| Geotechnical | 1.5 |
| Recovery Methods | 1.0 |
| Roads and Logistics | 0.07 |
| Heap Leach Facility | 0.4 |
| Environmental, Permitting, and Social Recommendations | 1.0 |
| **Total** | **26.32** |

Note: Numbers may not add due to rounding.

## 26.2 Exploration and 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.

## 26.3 Metallurgy & Process Design

The current acid consumption prediction approach uses average of all non-duplicative tests to date – a more refined methodology may be possible now with a calcium distribution model for the stockpile and additional testing inputs. Calcium is present as calcite, gypsum and other minerals in the stockpile and other Cactus deposit areas.

Building from the current testing protocols, additional testing was completed to show the potential to improve gross acid consumption by continuing to minimizing excess acid available in the leaching system with both flowrate and acid content reductions.

At average stockpile copper (~0.20% Cu) and calcium contents (~ 0.20% Ca), historical and current testing supports potential for reduced acid consumption based on calcium resource modeled for the stockpile upper lifts, there is a potential to reduce acid consumption by 20% to 40% (from a gross acid consumption of 22 lbs/ton to approximately 15 lbs/ton) by lowering leach solution flowrates to 6 L/h/m2 and acid concentration (5 gpL to) the stockpile materials.

Based on the current calcium modeling information, the upper two lifts of the stockpile contains a majority of the copper while also having lower calcium content. The current mine plan extracts material from the stockpile through a sequence of eight (8) cuts. The initial four cuts contain 41.4 million tons of leach material with lower calcium content.

Further work is required to verify this opportunity, the main elements are:

- Resource calcium model and drill spacing needs to be improved from 400 ft to better understand calcium distributions in an “as-mined” distribution and annual/monthly production averages.
- Additional column testing is required to further delineate the methods and quantify the opportunity to manage acid usage for the stockpile, Cactus and Park/Salyer leach materials.
- Stockpile material with higher calcium content and resulting acid consumption (mining cuts 5-8) can be stored for future use as acid mitigation for higher grade ores, a separate pad area is advised.

This additional analysis can be estimated to be $100,000.

## 26.4 Metallurgical Testwork

Only a small amount of metallurgical testing has been completed for the Parks/Salyer deposit. The work completed represents only a minimal metallurgical understanding of this deposit and additional confirmatory work is required to better understand the deposit variability. This work should include testing of material from lithologically and mineralogically variable areas known to exist. The main are of interest is a higher covellite content portion of the deposit. Covellite mineralization is leachable using the current methods, however kinetics are expected to be slower, and this impact will need to be confirmed.

Preliminary testing of an opportunity to reduce acid consumption has shown positive results in areas of lower calcium content. Additional column work is required to verify this opportunity, the main elements are:

- Resource calcium modeling in the stockpile material and drill spacing needs to be improved from 400 ft to better understand calcium distributions in an “as-mined” distribution and better quantify how calcium content will present in annual/monthly production averages.
- Additional column testing is required to further delineate the commercial production methods and quantify the opportunity to manage acid usage for the stockpile, Cactus and Parks/Salyer leachable materials. Testing of baseline and variability samples in duplicate should be completed to assess the relative differences in results by changing the leach flowrates from 12 L/H/$m^2$ to 6 L/H/$m^2$ and utilization of a constant leaching acid content of 5 g/L.

Testing designed to support the final commercial protocols envisioned for the resources contemplated as the project basis for the PFS were developed and conducted by the ASCU technical staff in their facility located on site. During the conductance of the work, records and information were not kept to the same standard as commercial laboratories and a significant effort was expended to ensure adequate QA/QC records existed and test data integrity have not been compromised. The next phase of testing should repeat these tests as part of the work to ensure that the results are repeatable and fully validated.

Approximately 20 additional column tests are required to complete the work above. Based on commercial laboratory rates, the costs so sample prep, sample material characterizations, chemical and mineralogical analyses, test conductance for up to 180 days and reporting is estimated at $50,000 per column test. A total budget of $900,000 should be considered.

If additional testing is to be completed at the Truestone facility on site, adequate resources in terms of additional qualified personnel will need to be sourced to ensure testing protocols and data are managed in a more diligent manner.

## 26.5 Mineral Resource Estimates

As drill hole spacing decreasing with continued in-fill drilling there will be an opportunity to use more sophisticated estimation techniques such as kriging to better define grade distribution within the known resource. Variographic analysis of the growing drilling database should be used to validate the use of these estimation methodologies.

## 26.6 Mineral Reserve Estimates

The mineral reserve estimate for the Arizona Sonoran Copper (ASCU) Cactus Project conforms to industry-accepted practices and were prepared in accordance with the guidelines of NI 43-101 and the Canadian Institute of Mine Metallurgy and Petroleum definition Standards for Mineral Resources and Mineral Reserves (“CIM Standards”).

Factors that may affect the estimate include: changes to long-term metal price assumptions, metal recovery assumptions, mine design and ground support, open pit slopes, ore recovery and dilution, mine sequencing, ability of the mining operation to meet the annual production rate, operating cost assumptions, leach pad performance, the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

There is an opportunity to increase reserves if additional near surface resources continue to be identified at Parks/Salyer which may allow additional options to be considered including transitioning from underground methods to open pit mining.

Review and analysis of estimated reserves for the Feasibility study with the inclusion of near surface resources is expected to cost $0.1 million.

## 26.7 Open Pit Mine Design and Scheduling

The mineable resource for the Cactus Project includes the Cactus deposit (West and East), Parks/Salyer deposit and the Historical stockpile. The deposits are planned to be developed using conventional open pit mining and underground mining methods in the PFS. Cactus West (CW) and the Stockpile Project (SP) will be developed using conventional open pit mining. Underground, the sub level caving method (SLC) were used for the Cactus East and Parks/Salyer deposits.

Access to the existing Cactus West pit walls was not possible due to safety concerns and legal status of the mine closure. Detailed mapping and geotechnical review of the existing pit walls when access is available and recommendations to reduce stripping while providing a safe work environment is required.

Optimization of the mining schedule and design should be completed with updated metallurgical inputs resulting from this study and ongoing and planned test work. Increased understanding of acid balance (net acid consumption, net acid producing) and recovered metal recovery can be used to improve the overall mine plan.

Realize and incorporate upside potential from nearby opportunities such as the Mainspring deposit.

Consider isolating primary mineralization for potential future processing and evaluate processing of primary mineralization by conventional milling or sulphide leaching.

With recent drilling the opportunity for mining of the MainSpring and Parks/Salyer deposits by open pit methods needs to be examined. This will include selection of appropriate equipment, updated slope parameters for use in these areas and ranking of the various areas as part of an exercise to enhance project economics. This will require coordination with all groups including processing and infrastructure.

The updates to the open pit mine designs and scheduling are estimated to cost $0.3 million.

## 26.8 Underground Mine Design and Scheduling

The drill coverage along the proposed decline paths and major areas where infrastructure is planned is minimal to non-existent. Large portions of the long-term development are too far from drilling data to estimate rock mass quality. A detailed drilling program is required to test portal locations, decline paths, vertical development, transfer, and crusher stations for detailed mine planning.

The geological knowledge concerning the definition and characterisation of major faults varies significantly between ore sources. For example, mapping information from historical mining provides good information for the Cactus West open pit and adjacent areas, however, no fault interpretations were made available within the mining geometry at Parks/Salyer for this study. Explicit definition and characterisation of major faults are required in the Parks/Salyer area in order to assist geotechnical modelling and mine planning.

The majority of the drilling orientation is subvertical. Some angled holes at strategic locations to better define structural and mineral boundaries and provide lateral geological and geotechnical coverage would be useful. For example, a more accurate delineation of the apparent NE trending dykes and adjacent geotechnically weak rocks would be useful for mine planning at Parks/Salyer. Similarly, better definition of boundaries between good and poor rock classes in the footwall areas of Cactus East will help optimise the access development locations.

Geotechnical domains are currently based on gross lithology and mineralogy changes. i.e. (Oxide, Enriched, Primary etc.). There, however, remains a high degree of variability within these domains which require further definition. Efforts should be made to establish sub-domains within this broader category to define zones of variability. This will allow mine planning and ground support provisions to be more specifically targeted.

The geotechnical block models should be revised once new data, geotechnical domains and fault interpretations are updated. A closer link and association between the geotechnical data and geological models will help establish more meaningful data interpolation trends which will improve the accuracy for the forecasting of local ground conditions.

The significant step out distances between subsequent sublevels at Parks/Salyer is an unusual aspect of the mine design for Parks/Salyer. Based on the current mining sequence, this will create conditions where cave fronts are required to merge. Current geotechnical recommendations are based on empirical assessments and 2D analysis. More advanced 3D modelling using updated information is required to optimise mine planning, sequencing, and ground support provisions particularly for Parks/Salyer.

Areas of surface subsidence generated by the underground mining activities are based on nominal break back angles derived from industry experience. The interaction of the Cactus East cave with the Cactus West pit does not currently consider the influence of major faults. Numerical modelling of the interaction between the underground and pit is required to finalise the location of the Cactus East portal, vent raises and production shafts.

The high clay content within the Parks/Salyer mineralised system could impact flow behaviour and ore handling under wet conditions. Further studies such as durability and weathering tests on materials that may degrade in the draw column to form mud and plasticity and dispersion tests on soils in the weathered zone that may result in mud rushes is required. The impact of stockpiling waste above the Cactus East SLC mine needs to be considered. A review of industry experience in clay management and risk mitigation measures would be useful to better understand the potential impacts in order to minimise operational risks.

The PGCA flow modelling work assumes all rock types have the same flow mobility characteristics. The impacts of differential flow rates between the different geological units require to be evaluated to determine dilution sensitivities particularly at Cactus East where the waste rocks are exposed along some sections of the mining front at the initial stages mining.

Detailed planning of geotechnical monitoring strategies supported by trigger and response protocols are needed for operational guidance for areas including ground stability, cave propagation, air gap detection, fragmentation, water balance and micro seismics.

The geologic, hydrogeological and geotechnical information collection cost is included in their respective area. The data suggested to be collected for the FS will form the design parameters for the updated underground designs and scheduling at Parks/Salyer and Cactus East. This work is expected to cost $0.8 million to complete to an FS level.

## 26.9 Mine Capital and Operating Cost Estimation

Fluctuations in mine capital and operating costs were noted as a result of worldwide economic pressures during the preparation of the PFS. These forces seem to have stabilized and retreated in some instances. Updated detailed quotes for equipment capital costs, repair and maintenance parts and consumables need to be collected. This new information including revised leasing terms will need to be applied to the updated open pit and underground mine designs and their expenditure scheduled. In some instances vendors may require payment for the detailed information needed and this has been considered in the cost estimate.

This update of the mine capital and operating cost is expected to cost $0.1 million.

## 26.10 Geotechnical

The following are recommendations for future work to advance to a feasibility level of study:

- Pit access should be re-established, and in-pit mapping should be conducted, and piezometers installed.
- Additional rock strength testing will be needed to advance the study both for the pit and the underground.
- The pit slope interaction with ground water needs additional study. Stability in the north region of the pit will be impacted by ground water. Ground water of the pit expansion is needed to develop dewatering targets and for determining dewatering methodology.
- Additional geotechnical drilling is required to advance the study to feasibility. Geotechnical logging (parameters necessary to calculate NGI Q and RMR) should be conducted on all in-fill holes and on dedicated holes drilled at locations of critical infrastructure, such as raise locations, decline pathways, underground workshops, etc. In particular, the decline pathways require drilling to characterize geotechnical and hydrogeological conditions.
- Geotechnical logging should be standard practice on all ore delineation holes throughout the property. This should include the Gila Conglomerate.
- Mineral domains were used as the geotechnical domains for this study. Their interpretation has an impact on the design recommendations. Consequently, mineral domain interpretations should be updated with additional drilling.
- Once portal sites are located, discrete ground support designs for the portals should be conducted.
- Numerical stress modeling is recommended to identify the general timing and locations of the detrimental stress redistributions so that they can be accounted for in the mine plan and sequencing.

- Measurement of in-situ stress is recommended. In-situ stress measurements can be conducted in drilled holes from surface. Estimates of the in-situ stress orientations and magnitudes are necessary for the numerical modeling work and to improve the geological understanding of the deposit.

Addressing these recommendations is estimated to cost $1.5M.

## 26.11 Recovery Methods

ASCU is considering the acquisition of a used crushing, screening and conveying facility partially erected in an African location. The equipment that is included can support most of the requirements for the Cactus materials handling requirements. A set of commercial terms have been developed and the pricing is considered in the current study work. ASCU will need to finalize an agreement and field verify the current condition of all components, registration and licensing, manufacturer’s warranties, and code compliance for an installation in the United States if included in an ensuing Feasibility Study. Field verification should include vendor representation.

The used equipment considered will need to be optimized for the duty at Cactus. Given the possibility of increased ore reserves, ASCU should verify adequacy of the equipment for any reconfiguration of the leaching design and throughput capacity in the next stage of study. Finalizing these details can be estimated to cost $1M.

## 26.12 Infrastructure

### 26.12.1 Roads and Logistics

A transportation study should be included with the next phase of the work. Existing traffic on the highway approaching the site is not typically excessive, however the increased traffic load may require the addition of turning lanes or similar upgrades. Construction costs for this type of work would be on the order of $0.7M per lane for a total road improvement project cost of $1.4M. Study fees to determine the extent of the required work is estimated at 5% of the construction value, or $0.07M.

### 26.12.2 Heap Leach Facility

ASCU must perform a geotechnical investigation within the footprint of the HLF to expand the understanding of the underlying foundation. It is also recommended that a seismic hazard analysis is performed to further develop the geotechnical stability sections.

Given the possibility of increased ore reserves, ASCU should expand the geotechnical investigation to the possible areas of expansion for the leach pads, including, the existing tailings facility. This investigation must consider boreholes and test pits to further understand the geotechnical properties of the soils contained in the TSF and determine the suitability for the construction of an HLF.

It is estimated that a total of 20 boreholes and 40 test pits will be required to expand the surface geotechnical investigation with an approximate cost of $0.2M. The site-specific seismic hazard analysis is expected to cost $0.2M for a total of $0.4M in recommendations for the Heap Leach Facility.

## 26.13 Environmental, Permitting, and Social Recommendations

ASCU must maintain compliance with the monitoring requirements specified in the Aquifer Protection Permit as well as additional monitoring that may be required when the permit is updated with the Parks-Salyer mine plan. In addition, ASCU will need to continue its engagement with the local community to maintain a positive relationship with key local stakeholders. It is estimated that future environmental costs for installation of two additional monitor wells will cost about $1M.

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