INYOAG LLC
Pre-Feasibility Study
Darwin Mines Project

Darwin, California 93522

Prepared for

Jack Stone CEO.

SK-1300 QP

NV General Engineer A-0041596A

CA General Engineer A-1039470

MSHA-ARX

SME-4163278

ART Consulting Engineers ARTCONSULTINGENG@CS.COM 702.221.8869, 702.488.0697

ART Consulting Engineers

Civil Engineering✧ Land Planning✧ Traffic✧ Hydrology and Water-Right Surveyor
Commercial and Residential

Professional Engineer's Certification

INYOAG LLC
Darwin Mines Project
Darwin, California 93522

Prepared for: United States Department of Defense – Defense Production Act Title III Program
Prepared by: ART Consulting Engineers
Date: June 4, 2025

I, the undersigned, hereby certify that:

1. I am a licensed Professional Engineer in the State of Nevada, with license number 13986, and I am competent to evaluate and review the engineering content of this report.

2. I have reviewed the Darwin Mines Project Feasibility Study, prepared by INYOAG LLC, for the purpose of submission under the Defense Production Act (DPA) Title III program.

3. This review included an evaluation of:
   - The mine plan and mining methods (conventional underground),
   - The mineral resource and production estimates,
   - The metallurgical processing design and flowsheet for both oxide and sulfide circuits,
   - Infrastructure layout and supporting systems relevant to mine development,
   - Capital and operating cost estimates related to mine and plant operations.

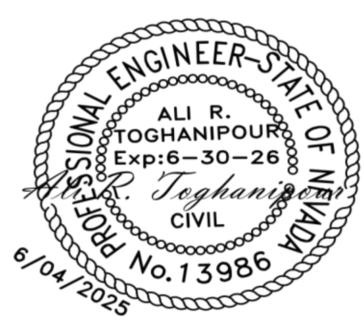
4. Based on my professional knowledge and review of the materials provided, I find the technical content described in this report to be prepared in a manner consistent with industry best practices and reasonable engineering judgment.

5. This certification is limited to the technical engineering aspects of the feasibility study and does not extend to legal, financial, permitting, or environmental regulatory opinions.

Signed:

Ali R. Toghanipour

Ali R. Toghanipour, P.E, WRS
Nevada License No. 13986



10733 Mentesana Ave
Las Vegas, Nevada 89166

24385 Terreno Drive
Temecula , California 92590

Table of Contents

1. Executive Summary

The Darwin Mine Project, located in Inyo County, California, marks the revival of one of the nation's most historically productive underground mining districts. Once a cornerstone of United States lead and silver production, Darwin is being reactivated by INYOAG LLC as a modern critical mineral hub aligned with domestic strategic supply chain priorities.

The project targets robust production of **zinc, copper, lead, silver**, and a suite of **critical minerals** including **tungsten, germanium, gallium, indium, tellurium, and cobalt**. These minerals are essential to the U.S. defense, clean tech, semiconductor, and communications sectors—yet face increasing global supply risk. Darwin is uniquely positioned to meet these demands through an accelerated two-phase redevelopment plan, starting with 500 tons per day (TPD) and scaling to 2,000 TPD in Year 6 (2031)**.**

Permits, Patents, Land Control, and Infrastructure

Darwin is one of the few privately held, fully permitted critical mineral projects in the western United States.

- **Permitting**: The site holds an active **Conditional Use Permit (CUP)** and **Air Quality Permits**, authorizing underground mining, surface operations, and mineral processing.
- **Land Position**: The mine spans 58 patented claims totaling 1,050 acres, securing both surface and mineral rights.
- **Infrastructure**: Includes over 150 miles of underground workings**,** a new and operational crushing system**,** a fully equipped lab**,** and a mill facility 65% complete.
- **Utilities**: Grid-connected site power is live**,** with remaining tie-ins pending for the well and final mill components. Water rights are secured and infrastructure is in place.
- **Backfill System**: Tailings are returned underground as structural fill—eliminating surface impoundments and aligning with best-in-class environmental practices.

Critical Minerals and Strategic Relevance

Darwin's mineralization is hosted in zinc and copper sulfides and was previously overlooked in historical production. Modern sampling confirms that zones enriched in **germanium**, **gallium**, and **indium** exist alongside traditional base metals. These are vital for:

Defense systems (night vision, satellites, IR optics)

Telecom and semiconductors

Green energy technologies (solar panels, wind turbines)

The project is a strong candidate for **Defense Production Act (DPA) Title III** support, offering the U.S. a secure domestic source of critical materials.

Agricultural Zinc Opportunity

Darwin also features high-grade oxide zinc, including **hydrozincite**, which will be processed into **zinc sulfate monohydrate powder and liquid**—a critical input for fertilizers. This enables the project to supply both strategic defense and agricultural sectors**,** a rare combination that diversifies revenue and enhances national food security.

Phased Production and Expansion

Phase 1 (2026–2030): 500 TPD throughput; oxide and accessible sulfide production; flotation and leach circuits operational

Phase 2 (2031–2035): Expansion to 2,000 TPD; deep zone mining and full critical mineral recovery; peak revenue years

---

Environmental, Social, and Governance (ESG) Commitment

The Darwin Mine is committed to high standards of ESG performance, emphasizing sustainability, safety, and community impact. The project will employ 350 workers during Phase 1, expanding to 1,500 by Phase 2, with a MOD factor of 3.8**,** reflecting a strong, proactive safety culture. Located within a federally designated **Opportunity Zone**, Darwin offers investors favorable tax treatment while supporting long-term regional economic growth and skilled job creation. With community partnerships, environmental stewardship, and transparent governance integrated into every stage of the project, Darwin stands out as both a strategic and socially responsible mining operation.

2. Corporate Overview

INYOAG LLC – Corporate Structure Overview

INYOAG LLC is the holding and controlling entity for all operations related to the Darwin Mine Project and associated critical mineral ventures. It does not have employees but maintains ownership and oversight of its operating subsidiaries, including **Stone Brothers Inc.**, which manages all field operations.

Stone Brothers Inc. – Operating Entity

Board of Directors

- **Jack Stone** – *Chairman of the Board*
- **Jim Oakes** – *Chief Executive Officer (CEO), Board Member*
- **Nicholas Stone** – *Chief Operating Officer (COO), Board Member*
- **Wally Sullivan** – *Opportunity Zone and Marketing Director*, *Board Member*
- **Earl Harrison** – *Chief Compliance Officer*, *Board Member*

Executive Management

Jim Oakes – *Chief Executive Officer (CEO)*

Jim has over 37 years of leadership in refractory metals and powder metallurgy, previously serving as Vice President at ATI and Teledyne. He holds BS and MS degrees in Engineering from the University of Delaware and an MBA from Middle Tennessee State University. Jim leads strategic growth, process development, and investor engagement.

Nicholas Stone – *Chief Operating Officer (COO)*

Nicholas has 17 years of operational leadership in mining and milling. He has improved safety and productivity across multiple crushing and underground systems. He oversees daily operations, equipment maintenance, and field personnel. He is an MSHA Certified Instructor and certified on all major mining and crushing systems.

Linda Stone – *Chief Financial Officer (CFO)*

Linda manages financial planning, compliance, and reporting, working closely with **RubinBrown LLP** to ensure audit readiness and Opportunity Zone accountability.

Earl Harrison – *Chief Compliance Officer*

Earl is a 40-year mining veteran and former Mine Manager at Ashdown and Mineral Ridge. Trained at Mackay School of Mines and University of Idaho, he leads all permitting, planning, and regulatory compliance across SMARA, MSHA, and EPA frameworks.

Chairman

Jack Stone – *Chairman of the Board*

With a long-standing career in underground development and mine infrastructure, Jack provides strategic oversight and board leadership. He has completed over $75 million in successful mining and mill projects across Nevada and California. Provides strategic oversight and governance. Also directs **Export, Trade, and Offtake Management**, managing contracts, logistics, and international sales relationships.

Technical & Field Operations

Guy Sacco – *Chief Engineer*

Guy directs all engineering efforts for site development, infrastructure, and integration of mechanical systems for production.

Brian Bishop – *Mine Engineer*

Brian designs mine layouts, oversees stope planning, and ensures alignment between extraction methods and geological modeling.

James Gim – *Process Engineer*

James, a former Boeing engineer, is a recognized innovator in advanced materials including graphene. He now leads flowsheet and metallurgical circuit design for critical minerals such as germanium, indium, and tellurium.

Special Programs and Outreach

Wally Sullivan – *Opportunity Zone and Marketing Director*

Wally leads the Opportunity Zone strategy and external communications, ensuring project alignment with government incentive programs and investor relations.

Supporting Legal, Environmental, and Financial Partners

- **Fabian VanCott LLP** – Legal counsel for corporate structuring, permitting, and Opportunity Zone compliance.
- **Environmental Division** – Oversees EIR filings, discharge permitting, and reclamation plans.
- **Rubin Brown LLP** – External accounting firm ensuring financial integrity, audit preparation, and OZ structuring.

Strategic Technical and Service Partners

- **Ballistic** – *Subcontractor for Underground Development and Operations*
  Handles underground headings, ground support, drifting, and advance in both oxide and sulfide ore bodies.
- **Ethos Geological** – Produces NI 43-101 reports, provides mapping, core logging, and resource modeling.
- **Woods Process Services** – Analytical testing and lab support for ore samples and recovery validation.
- **Kappes, Cassiday & Associates (KCA)** – Experts in leaching, flotation, and metallurgical circuit design.
- **AVI Metals** – Design and support for electrowinning, drying, and process engineering systems.
- **Clare Concrete** – Responsible for mill and site concrete work and paving.
- **Aldrich Electric** – Designs and installs electrical systems for surface and underground operations.
- **NV Rand** – Hoist renovation, mill mechanical infrastructure, and safety-critical upgrades.

3. Project Background Historical Strategic Role and Critical Mineral Potential of the Darwin Mining District

The Darwin Mining District, located in Inyo County, California, has a long history of base and strategic metal production, with documented output dating back to the 1870s. Initially developed during the silver boom, the district became a focal point for lead, zinc, and silver extraction, and later played a significant role in U.S. defense mineral supply during World War I & II and the Cold War era.

During the 1939–1945 war years, operations under the Strategic Materials Act, and later under the Defense Production Act (DPA) of 1950, enabled extensive underground development and infrastructure upgrades. The primary operator during mid-century activity, Defiance Mining Company, conducted widespread mining and milling operations, supported by federal programs including the War Production Board and the Defense Minerals Exploration Administration (DMEA). Concentrates were shipped to national smelters to support production of ammunition, electrical infrastructure, and critical alloys.

Much of Darwin's lead-silver concentrate was shipped to the ASARCO smelter in Tooele, Utah, one of the largest custom smelters in the western United States at the time. Tooele was capable of processing complex polymetallic ores and served as a major refining hub for domestic defense metals. The lead produced from Darwin's concentrates—some of which contained trace amounts of antimony, cadmium, and bismuth—was refined into ingots used in antimonial lead alloys for critical wartime applications such as ammunition, battery grids, and radiation shielding. The Darwin-to-Tooele supply chain formed part of a broader national industrial effort to secure strategic materials from domestic sources.

Although antimony was not a primary commodity at Darwin, moderate quantities of stibnite (Sb_2S_3) were encountered in deeper sulfide zones and gangue, often associated with tetrahedrite, chalcopyrite, and galena. These occurrences were not economically recovered on-site due to low concentration and lack of separation infrastructure. However, the antimony was blended into lead-silver concentrates and shipped to Tooele, where it was partially recovered during smelting and refining. Based on historical concentrate tonnage and known trace Sb content, it is estimated that Darwin may have indirectly supplied between 500 and 1,500 tons of contained antimony over its cumulative operating life. This material likely contributed to the national supply of antimonial lead, used extensively in military munitions, hardened battery plates, and specialty metal alloys.

Between 1940 and 1970, Darwin generated the equivalent of over $50 million in direct mineral value (adjusted to current dollars), primarily from lead, zinc, silver, and copper. Indirect contributions, including antimony, cadmium, tungsten, and bismuth—though not officially reported—strengthen Darwin's legacy as a significant participant in the U.S. strategic materials network. The district's workforce and tax base supported regional development, while its concentrates reinforced the domestic supply of metals essential to national defense and infrastructure.

Although early 20th-century assays and mill reports focused on major base metals, the ore mineralogy and geologic setting at Darwin strongly suggest that other now-strategic elements—such as germanium, indium, gallium, tellurium, and bismuth—were present in significant trace quantities within the polymetallic sulfide system. These elements, now defined as critical minerals by the U.S. Department of Energy and Department of Defense, were not recoverable using historic methods, but are economically extractable today through co-processing or selective hydrometallurgical recovery.

Project Darwin Ownership and Revitalization

Since 1996, the Darwin site has been held and operated under the ownership of Project Darwin LLC. In recent decades, the project team has undertaken extensive permitting work to overcome historical regulatory and access challenges. Modern efforts have focused on reopening and verifying legacy workings, rehabilitating key shafts, and updating the geological database through geophysical mapping, core re-logging, and channel sampling.

Numerous drill logs and legacy datasets—once fragmented—have now been digitized and integrated with new geological interpretations. The reestablishment of an accurate structural and lithological model is now central to evaluating not only the traditional base metal zones, but also a growing portfolio of critical mineral potential tied to unmined or backfilled zones.

Critical Mineral Potential: Germanium, Antimony, and Associated Elements

Recent attention has turned to the Darwin district's potential as a domestic source of critical minerals, particularly germanium and antimony, as well as co-occurring elements such as indium, gallium, tellurium, tungsten, and bismuth. These materials are all listed as strategic or critical by the U.S. government due to their role in defense, electronics, and renewable energy supply chains.

Germanium

Germanium is typically hosted in solid solution within sphalerite (ZnS) and is known to occur in minor amounts in Darwin ore bodies. Historical USGS and USBM reports from the 1950s document germanium concentrations in sphalerite ranging from 50 to 150 ppm in deeper sulfide-rich zones, particularly those associated with chalcopyrite and bornite. While these values were uneconomic to recover at the time, they are well within modern extraction thresholds when co-processed with zinc.

Antimony

Antimony occurs in minor quantities at Darwin, primarily as stibnite (Sb_2S_3) in contact zones within copper- and silver-bearing sulfides. Historical records from the Defiance Mining era and USBM indicate 1–3% Sb in selected sulfide gangue and lower mine levels. While never commercially recovered, these values were blended into base metal concentrates historically shipped to off-site smelters. Modern separation technologies, particularly when tied to reprocessing of bornite-sphalerite zones, could allow for targeted antimony recovery as a byproduct. With over 80% of current U.S. antimony supply imported from China and other geopolitical competitors, a viable domestic source would serve both economic and strategic objectives.

Indium, Gallium, Tellurium, and Bismuth

Recent re-evaluation of core logs, flotation tailings, and mineralogical reports suggests the presence of several additional critical elements not historically recovered but likely present in the Darwin system:

- Indium may occur in sphalerite and chalcopyrite zones at inferred levels of 10–50 ppm, based on analogs from similar polymetallic districts. Indium is essential for semiconductors, LCDs, and aerospace solder applications.
- Gallium is associated with sphalerite and may occur at 10–30 ppm. Gallium-based compounds are used in 5G infrastructure, solar technologies, and radar systems.
- Tellurium may occur with tetrahedrite and chalcopyrite. Though never assayed historically, tellurium is used in thermoelectrics, solar panels (CdTe), and IR optics.
- Bismuth has been inferred in minor amounts from galena- and tetrahedrite-rich zones. Bismuth is relevant for lead-free solders and medical shielding.

While these elements were never analyzed or recovered historically, current assays and testing shows their association with Darwin’s dominant sulfide mineralization suggests they are present in recoverable concentrations under modern processing regimes. Their co-recovery with zinc, copper, and lead could offer additional value and significantly increase the project's alignment with current national critical mineral priorities.

Historic Output and Legacy Infrastructure

Darwin's cumulative production over the past century includes:

Metal	Estimated Historic Output	Strategic Use
Lead	100,000+ tons	Ammunition, batteries
Zinc	75,000+ tons	Galvanization, alloys
Silver	10+ million ounces	Electronics, currency
Copper	~20,000 tons	Electrical systems
Tungsten	Minor	Armor, high-temp alloys
Cadmium	Minor	Avionics, plating
Antimony	Minor 500-1500 tons	Flame retardants, ordnance
Germanium	Trace (50–150 ppm in Zn ore)	Infrared optics, semiconductors
Indium	Inferred (10-50)	LCDs, semiconductors
Gallium	Inferred (10-30 ppm)	5G, radar, LEDs
Tellurium	Inferred	Solar panels, thermoelectrics
Bismuth	Possible	Lead-free solder, shielding

Remaining underground infrastructure includes backfilled stopes, shaft collars, rail systems, flotation residues, and partially intact mill foundations. Some zones rich in copper, silver, and zinc sulfides remain inaccessible or unsampled, representing a future opportunity for detailed geochemical investigation and pilot-scale recovery of critical mineral values.

Strategic Relevance and Forward Potential

Darwin's history aligns closely with the objectives of the Defense Production Act Title III, the National Defense Stockpile modernization effort, and recent DOE strategic materials initiatives. Reestablishing Darwin as a critical mineral source not only revives a proven U.S. mine site but supports broader national security goals by reducing dependence on foreign suppliers for germanium, antimony, indium, tellurium, and gallium—each vital to modern defense, electronics, and clean energy systems.

With permitting progress complete, legacy data being reinterpreted, and new sampling campaigns planned, Darwin is positioned for strategic redevelopment as both a domestic metals supplier and a low-risk reactivation target with existing infrastructure and extensive historical data.

Current Ownership: 100% patented claims owned by Project Darwin and leased to INYOAG LLC with a 5% NSR (Franco Nevada)

Geological and Production Comparison: Darwin vs. Apex Mine

The Darwin Mine in Inyo County, California, and the Apex Mine in Washington County, Utah, share several notable geological similarities, particularly in the way germanium and other critical elements are hosted and enriched. Both deposits are associated with altered carbonate sequences, iron oxide zones, and silica-rich jasperoid bodies—key indicators for supergene enrichment of critical minerals.

At Darwin, mineralization is structurally and stratigraphically controlled, with skarn-altered zones, breccia pipes, and iron-rich gossans developed along faulted and folded carbonate rocks. Extensive jasperoid development is observed in proximity to these structures, often associated with limonite and hematite. These zones not only show elevated levels of base metals such as zinc and lead, but are also prospective for germanium, gallium, indium, and tellurium—either through substitution in sulfide minerals like sphalerite and bornite or through adsorption onto oxide minerals such as goethite and hematite.

This geologic setting closely parallels that of the Apex Mine, which historically served as the United States' primary domestic source of germanium and gallium during the Cold War. At Apex, germanium was enriched in jasperoid breccias and iron oxide zones within Permian limestones. Recovery was based on leaching of adsorbed Ge and Ga from these oxide-rich zones, primarily through acid heap leach methods followed by solvent extraction.

While Apex's geological system was exploited for its relatively simple oxide-hosted germanium, Darwin presents a more complex system that includes both oxide and sulfide mineralization. This duality enhances Darwin's strategic advantage: it contains the same style of supergene-enriched jasperoid and gossan zones as Apex, but also includes deeper sulfide-rich zones that carry additional critical and base metals, enabling multi-product recovery from a single feedstock.

From a production standpoint, Apex's total germanium output was limited, with estimates ranging from 7 to 10 metric tons over its operating life. Annual recovery likely averaged around 1 ton of Ge per year. In contrast, Darwin's modern development plan projects approximately 20.5 metric tons of germanium per year once both oxide and sulfide circuits are operational. Over a 10-year life, Darwin is forecasted to produce over

200 tons of germanium, making it more than 20 times larger in projected output than Apex.

Furthermore, Darwin's potential for concurrent recovery of zinc, silver, lead, tellurium, and gallium significantly improves its economic profile relative to Apex, which was a single-metal operation reliant on federal procurement incentives of its time.

In summary, while Apex served as the benchmark for U.S. domestic germanium production, Darwin offers a geologically analogous but economically superior opportunity, with broader mineral potential, larger production volumes, and direct alignment with modern strategic material priorities.

4. Pre-Feasibility to Feasibility Study Plan

To advance the Darwin Mine Project from Pre-Feasibility Study (PFS) to a full Feasibility Study (FS), the following milestones and technical objectives are planned based on Ethos Geological recommendations and the 2025 Exploration Budget:

1. Geological Validation and Expansion Drilling

- Drill 3,500 meters of core in 2025, targeting:
  - A433/435 Oxide zone: Confirm WO_3, Zn, Ag grades along the anticline
  - B-458 Fissure: Down-dip sulfide extension (Zn 8.2%, Ag 9.3 opt confirmed)
  - 418D Oxide: Define volume and continuity across 400L to 1000L
- Rib and long hole sampling along key drifts and limbs
- Add two new underground core drill stations

2. Data Digitization and Modeling

- Finalize digitization of all historic logs, maps, and mine models
- Update 3D block model integrating 2025 assay results
- Begin geo-statistical validation for Indicated/Measured classification upgrade

3. Metallurgical Testing & Pilot Recovery

- Complete confirmatory flotation testing for WO_3, Ag, Zn, and Ge
- Initiate selective leaching tests for indium, gallium, and tellurium
- Conduct process optimization using blended oxide/sulfide feed trials

4. Geochemistry and QA/QC Program

- Execute full core and QA/QC assay program (~$85,000 budgeted)

- Include certified reference standards for germanium and tungsten
- Submit to third-party labs (Florin, ALS)

5. Technical Reporting and Compliance

- Prepare updated NI 43-101 / SK-1300-compliant mineral resource estimate
- Deliver full FS-level technical report with economic sensitivity analysis

6. Environmental and Regulatory Advancements

- Confirm no new disturbance triggers SMARA review
- Update CUP documentation with tonnage and employment scale-up
- Prepare technical ESIA update in line with Phase 2 planning

7. Timeline and Budget

- Total 2025 Exploration Budget: $1.3 million
- Duration: 117 days active field time
- FS draft report expected by Q2 2026

This FS advancement plan supports critical path milestones for off-take, DPA Title III eligibility, and debt/equity financing strategies.

Mineral Resource Statement – Source Attribution

This Mineral Resource Statement is based on data compiled, reviewed, and interpreted by independent geological and technical professionals using current industry-standard practices in accordance with **NI 43-101** and **SEC SK-1300** guidelines.

Source of Data

The Darwin Mine resource model integrates over 75 years of geological data, including both legacy records and modern QA/QC-verified sampling. While historical datasets provided broad spatial coverage and continuity across the underground workings, modern confirmation campaigns were undertaken to validate grade reliability and structural consistency.

Historical Data (1945–1995):

- **1,700+ drillholes** totaling ~160,000 feet of core (Anaconda, Blue Range, Quintana)
- Extensive underground **channel samples**, drift logs, and **geologic maps**
- **Production records** support grade continuity in key sulfide and oxide zones
- Mapping at 25–50 ft stope intervals and documented head grades from milling support original estimations.
- Limitation: Historical assays were not subjected to modern QA/QC; original sample custody procedures are undocumented

Modern Data (1991–2024):

- **140,000+ feet of historic core re-logged** and relabeled according to modern lithologic standards.
- Over **300 feet of new underground channel samples** across 8 target zones
- **Assayed at ALS and Florin Analytical**, using certified reference materials, blanks, and duplicates.
- Structural features mapped digitally; lithological and alteration models rebuilt in 3D
- Confirmation of **silver, zinc, and germanium grades** within 10–15% variance from historical intercepts in verified zones.

Conclusion:

The historical data forms a reliable foundation for Inferred and Indicated classification when integrated with modern mapping and sampling. Although the data density is extensive, it is not currently supported by twin holes, certified QA/QC from origin, or complete chain-of-custody. Therefore:

- **Inferred and Indicated classifications are fully supported.**
- **Measured classification is not assigned** until confirmatory core drilling and bulk density testing are completed.
- The existing model meets Pre-Feasibility confidence levels in line with **NI 43-101** and **SK-1300** standards.

Ethos Technical Report

Project Darwin

Inyo County, California

2024 Report

Ethos Geological Inc.
902 North Wallace, St A
Bozeman, MT 59715
www.ethosgeological.com

Zachary J. Black, SME-RM

Prepared for:
Project Darwin, LLC P.O. Box 29
4901 Darwin Mine Road Darwin, CA 93522
-report date- **April 12, 20**
-effective date **April 1, 2024**

LIST OF FIGURES

LIST OF TABLES

SUMMARY

Ethos Geological ("Ethos") was commissioned by Project Darwin, LLC (the "Company") to prepare a technical report for the Darwin Project located in Inyo County, California. The report aims to deliver a technical analysis and an evaluation of the current mineral resource. The methodology and analytical rigor align with industry standards.

Project Darwin is the issuer of this report, which presents the results of an assessment that includes an evaluation of geological settings, validation of historical exploration data, and review of mineral resource estimations. This comprehensive analysis and interpretation of available data aims to provide a detailed understanding of the status and potential of the mineral resource at the Darwin Project site.

Property Description and Ownership

Project Darwin, located in the Darwin Mining District, Inyo County, California, encompasses 1,050 acres of patented mining claims including a specific patented mill site claim. Situated 36 miles southeast of Lone Pine near the town of Darwin, the property is positioned between Sequoia National Park and Death Valley National Park, near the Nevada border. The mining claims are spread across townships within the Mount Diablo Meridian, with essential infrastructure for water and power facilitated through leases from the California State Lands Commission and the Bureau of Land Management.

The project is under a 5% Net Smelter Return (NSR) royalty to Franco Nevada Corporation, with ongoing negotiations to potentially reduce and partially purchase this royalty. The operational framework for Project Darwin includes a comprehensive suite of permits and licenses addressing land use, water rights, environmental, operational, and safety requirements. These include essential water and powerline right-of way leases, Water Right Permits for water diversion, and a suite of environmental compliance permits from regional and state authorities.

Environmental management at the site is regulated through a Draft Negative Declaration of Environmental Impact and a Conditional Use Permit that imposes specific operational conditions. The project is also subject to regulatory oversight from various environmental and safety agencies, ensuring compliance with federal and state mining safety standards. The operation's exemption from the California Surface Mining and Reclamation Act of 1975 (SMARA) is due to the project not anticipating new areas of disturbance exceeding one acre nor the removal of more than 1,000 cubic yards of material, with all waste rock and process tailings planned to be backfilled underground.

There are no known environmental liabilities attributable to Project Darwin, LLC. Potential liabilities associated with the reclaimed heap leach pad and tailings pond are assumed to reside with Arco Petroleum, successor to Anaconda and now part of British Petroleum.

Geology and Mineralization

The Darwin area's geological history extends from the Upper Paleozoic to the Cenozoic era, characterized by sedimentary deposition, structural deformation, plutonic intrusions, and metamorphic transformations. Specifically, Upper Paleozoic carbonate turbidites were subject to extensive folding due to the Late Triassic thrust faulting.

In the mid-Jurassic period, the intrusion of the Darwin stock resulted in a large areole of contact metamorphism in surrounding carbonate rocks, resulting in the formation of idocrase-wollastonite calcsilicate hornfels, garnet-rich skarnoid, and bleached marble, alongside tungsten-bearing skarns at contact zones with the Darwin stock.

Subsequent thrust faulting included the displacement of the upper plate by 1-3 km eastward by the Davis fault system, which telescoped the Coso batholith and associated Cu-skarns with the Darwin stock and associated W-skarns. Later, granite porphyry dikes and breccia pipes intruded along the northwest margin of the Darwin stock, coinciding with the formation of Pb-Zn-Ag skarns and veins, mainly on the west side of the stock, influenced by steeply dipping faults adjacent to breccia bodies. The Cenozoic reactivation of the Davis thrust, related to basin and range uplift, slightly deformed the nearby ore bodies.

Research findings suggest that tungsten (W), copper (Cu), and lead-zinc (Pb-Zn) skarn deposits in the Darwin area have distinct origins, indicating separate formation processes. This includes the rare occurrence of PbZn-W skarns, likely formed through tungsten remobilization. The presence of three different igneous systems and diverse calc-silicate/ore assemblages supports this model, confirming multiple independent plutonic hydrothermal events rather than strata bound contributions.

Typically associated with granite, scheelite skarns require extensive fractional crystallization in moderately deep settings, a condition observed in the Darwin stock. Small copper skarns tend to form near barren granitic stocks in deeper settings, consistent with patterns seen across the western U.S., including eastern California. Pb-Zn skarns in Darwin are found along faults and dikes, unrelated to the larger Darwin stock, possibly associated with unexposed granitic entities indicated by geophysical anomalies. This diversity underscores the presence of multiple, independent skarn-forming hydrothermal systems.

Vein ores may be related to skarn-forming episodes, particularly associated with shallow granite porphyries. Initial ore-forming fluids were likely magmatic, transitioning to mixed origins in later stages influenced by meteoric water cycling. The rare co-occurrence of Pb-Zn and W skarns reflects unique geological conditions and plutonic events, consistent with areas exhibiting contrasting multiphasic plutonic activities.

Status of Exploration

Project Darwin's efforts on the project have focused on organizing and understanding the historical data. Samples from the underground workings have been collected periodically to confirm zones of mineralization or in areas with potential for further development. The Company is in the process of digitizing all the historical maps and drill logs to incorporate them into a 3-dimensional model to facilitate exploration planning in areas showing promise. They have completed a detailed photogrammetric survey of the project area, began surveying accessible areas of the mine, and are in the process of drilling at least diamond drill hole targeting the Essex breccia pipe.

Mineral Resource Estimates

Mineral resources at the Darwin Project have been estimated by Project Darwin using a sectional method. This approach leverages all available historical data, including drill results, underground channel sampling, and historical production reports.

The mineral resource estimate is based on all data obtained as of April 1, 2024. Mineral resources are not mineral reserves and do not have demonstrated economic viability such as diluting materials and allowances for losses that may occur when material is mined or extracted; or modifying factors including but not restricted to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors. Ethos knows of no existing environmental, permitting, legal, title, taxation, socio-economic, or other relevant factors that might materially affect the mineral resource estimate. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated on the basis of limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

TABLE 1 DARWIN PROJECT MINERAL RESOURCE STATEMENT

Block	OX State	Tons	Ag		Pb		Zn		W		Cu	
		(x1000)	opt	oz (x1000)	%	lb. (x1000)	%	lb. (x1000)	%	lb. (x1000)	%	lb. (x1000)
Indicated Resource Blocks												
A433/435	Oxide	180.0	8.5	1,531.8	0.8	2,952.0	6.4	23,076.0	0.6	2,052.0	-	-
B-458 Fissure	Mixed	66.5	9.3	618.5	0.9	1,197.0	8.2	10,906.0	-	-	-	-
418D Oxide	Oxide	30.0	10.6	319.1	-	-	8.0	4,807.5	-	-	-	-
A-439 Stope	Oxide	30.0	11.2	337.0	3.4	2,066.4	2.6	1,538.4	-	-	-	-
Inferred Resource Blocks												
B-458 Fissure	Mixed	33.25	8.5	283.0	0.8	545.3	6.4	4,262.7	-	-	-	-
418D Oxide	Oxide	490.8	10.6	5,220.9	-	-	8.0	78,650.7	-	-	-	-
833/712 Zone	Mixed	30.0	2.6	76.6	7.4	4,428.0	7.7	4,615.2	-	-	-	-
1302 Drift	Mixed	33.75	4.2	140.7	2.6	1,771.2	7.6	5,148.8	-	-	-	-
Remnant Blocks	Mixed	114.85	6.5	746.5	7.8	17,916.6	7.1	16,308.7	-	-	-	-
B-458 Fissure Cu	Mixed	175.0	41.0	7,175.0	4.0	14,000.0	14.0	49,000.0	0.9	3,150.0	6.0	21,000.0

Notes: * "Reasonable prospects for economic extraction" are based on an 8 opt Ag equivalent cut-off grade. The estimation method restricts extrapolation to known mineralized areas within historical mine workings. Ag equivalent is calculated using $25/oz for silver, $0.9/lb. for lead, $1.1/lb. for zinc, and a 94% metallurgical recovery rate. Total costs, including mining, processing, G&A, and shipping and smelter fees, are estimated at $189.50 per ton of ore. Calculations assume use of underground long hole stoping and processing into a flotation concentrate.

Conclusions

The geology of the Darwin Project is well understood, and the appropriate deposit model is being applied for exploration. The conceptual geologic model is sound, aligns with historical drilling results, and indicates that mineralization is essentially open in all directions, suggesting significant potential to expand on the known mineralization. Significant potential exists to increase the known mineral resource with additional drilling, as well as to upgrade existing mineral resource classifications with infill drilling or systematic underground channel sampling. There is also considerable potential to encounter massive skarn and porphyry mineralization within the district. The current mineral resource at the Darwin Project is more than sufficient to warrant continued evaluation of the Project.

Recommendations

In the context of mineral exploration, understanding the potential of a property requires integrating multiple types of geoscience data. This integration involves aligning information from various surveys and sources to form a coherent picture. Each data layer, whether it's from magnetic surveys, gravity surveys, geochemical sampling, or detailed geological mapping, adds depth to the analysis, helping to identify and prioritize exploration targets. This layered approach is essential because it provides multiple perspectives on the potential geological formations and mineral deposits, enhancing the likelihood of a successful exploration effort.

To optimize exploration and decision-making processes, it is recommended that Project Darwin undertake the digitization of its extensive archive of historical documents related to the Anaconda mining operation and subsequent stakeholders. The archival materials, which include geologic maps, drillhole data, underground samples, production records, and surface exploration results, should be converted into a standardized digital format. This conversion should utilize reliable digital storage mediums such as databases or cloud storage solutions to ensure quick access, facilitate complex analyses, and support the seamless integration of diverse geological, geophysical, and geochemical data sets.

The digitization of these historical documents is crucial for constructing an accurate and comprehensive geological model. This model will serve as a foundational tool in supporting subsequent phases of mineral estimation and strategic exploration efforts. By implementing this recommendation, Project Darwin will enhance its ability to leverage historical data effectively, thereby improving the accuracy and efficiency of its resource management and exploration activities.

To bring the historical data in line with the current industry standards it requires further validation which may include, to the extent possible, re-locating the historical data in the field, and resampling and re-assaying of historical drill core or drilling samples and submission to a laboratory with certified reference materials. Historical drill hole data should be supported with newly completed drill holes and sampling.

The validation of the historical data should be done in conjunction with exploration targeting the areas outlined in the mineral resource estimates. Analysis of the data using a methodology appropriate for the base metals, tungsten, and the REEs associated with deposit.

After each phase of validation and exploration the models need to be updated with the new data and adjusted to guide the next phase.

Introduction

Ethos Geological ("Ethos") was commissioned by Project Darwin, LLC (the "Company") to prepare a technical report for the Darwin Project located in Inyo County, California. The report aims to deliver a technical analysis and an evaluation of the current mineral resource. The methodology and analytical rigor align with industry standards. Project Darwin is the issuer of this report, which presents the results of an assessment that includes an evaluation of geological settings, validation of historical exploration data, and review of mineral resource estimations. This comprehensive analysis and interpretation of available data aims to provide a detailed understanding of the current status and potential of the mineral resource at the Darwin Project site.

Zachary J. Black, SME-RM, undertook a site inspection from December 11-13, 2023, focusing on specific geological formations and areas of interest, such as the Mickey Summers Fissure and the Defiance Shaft area. This visit facilitated a direct assessment of the geological features, contributing to a robust analysis of the site's mineralization processes and geological structures. Observations made during the visit support the development theories of skarn and breccia bodies, corroborated by historical documentation from Anaconda.

The integration of site visit findings with a thorough examination of historical and recent data underpins the geological analysis presented in this report. This approach ensures a detailed and factual representation of the project's geological framework, enhancing the reliability of mineral resource estimates and providing a solid foundation for future exploration and development recommendations.

Reliance On Other Experts

Except for the author's firsthand observations on the surface and underground, all information presented herein is derived from a meticulous examination of published and unpublished sources, and a thorough review and projection from historical data.

The author asserts that these sources have provided information in a factual, truthful, and unbiased manner, without any reason to doubt the reliability of the information, ideas, and recommendations presented. Notably, the Anaconda Copper Mining Company, operator of the mine during its peak production period from 1945 to 1957, is renowned in the industry for its meticulous geologic mapping and analysis standards. The report places significant reliance on the comprehensive surface and underground maps, drill logs, assays, and other data meticulously maintained by Anaconda, which remained at the Darwin mine office until 1985.

Additionally, two anonymous reports retrieved from the Darwin mine office have been referenced, dating back to January 1990 and July 23, 2002. These reports, likely prepared by Blue Range Mining Co. and H. G. Brown, consultant to Project Darwin respectively, were found to align with and supplement information from other sources, further enriching the analysis.

Project Description and Location

Project Darwin is situated in the Darwin Mining District, Inyo County, California, positioned 36 miles southeast of Lone Pine near the unincorporated town of Darwin. Its precise location is at latitude 36.295074 N and longitude -117.596495 W, nestled between Sequoia National Park to the west and Death Valley National Park to the east, with the Nevada border approximately 80 miles to the east (FIGURE 1).

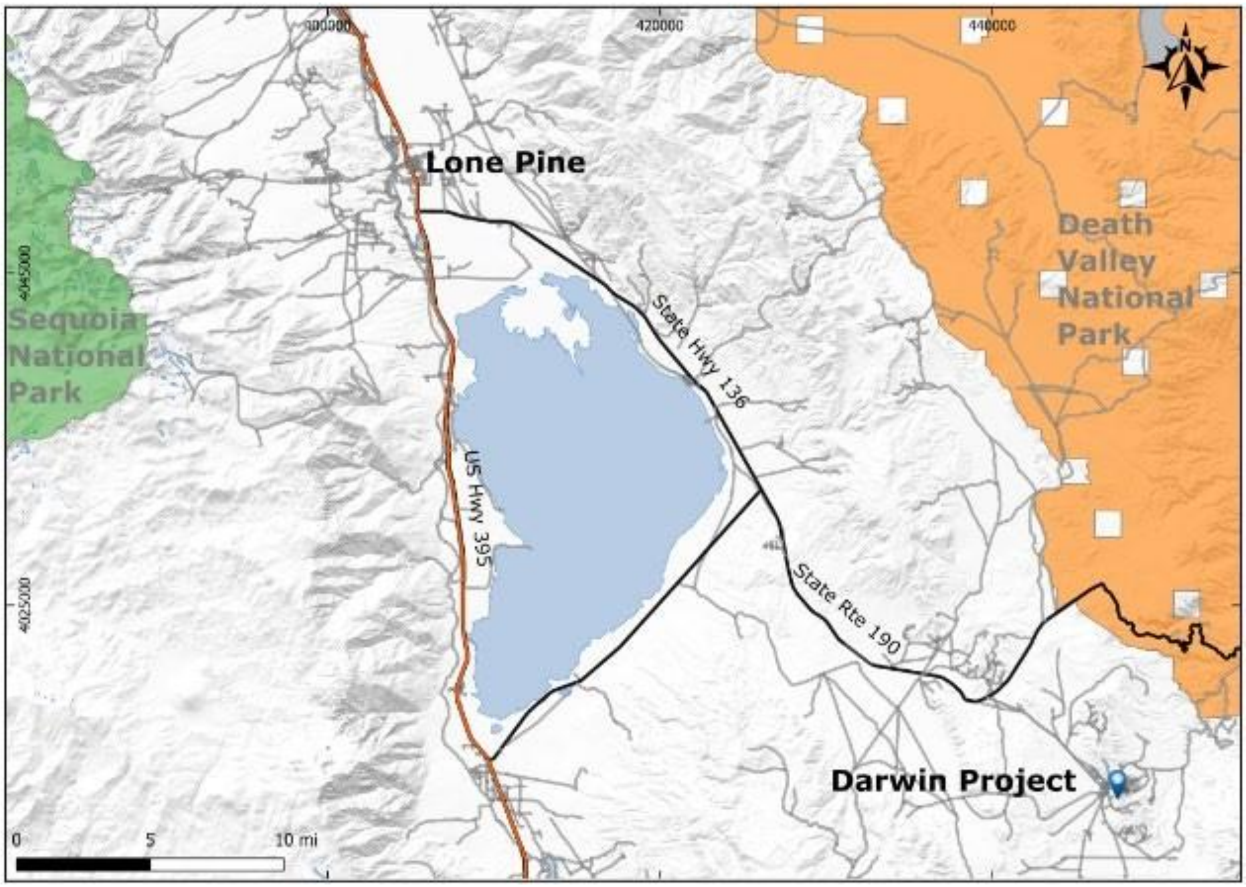


FIGURE 1 DARWIN REGIONAL LOCATION

Property Ownership, Mineral Tenure, Agreements and Encumbrances

The Property consists of 58 patented mining claims over 1,050 acres, including a specific patented mill site claim of 5.2 acres (Figure 2). Additionally, it holds a lease from the California State Lands Commission (PRC 4627.2) for water and powerline right-of-way across 12.97 acres, coupled with a Right-of-Way lease (CA8872) from the Bureau of Land Management for similar purposes over approximately 13,200 feet of BLM land. These patented claims are strategically situated in various sections across townships T. 19 S., R. 40 E., and T. 19 S., R. 41 E., with the water well centrally located in Section 16, T. 19 S., R. 41 E., all within the Mount Diablo Meridian.

The property is subject to a 5% Net Smelter Return (NSR) royalty on newly mined ore, currently owned by Franco Nevada Corporation. Project Darwin LLC is in negotiations with Franco Nevada to reduce this royalty and potentially purchase a part of the royalty interest. Furthermore, Project Darwin LLC pays annual property taxes to Inyo County.

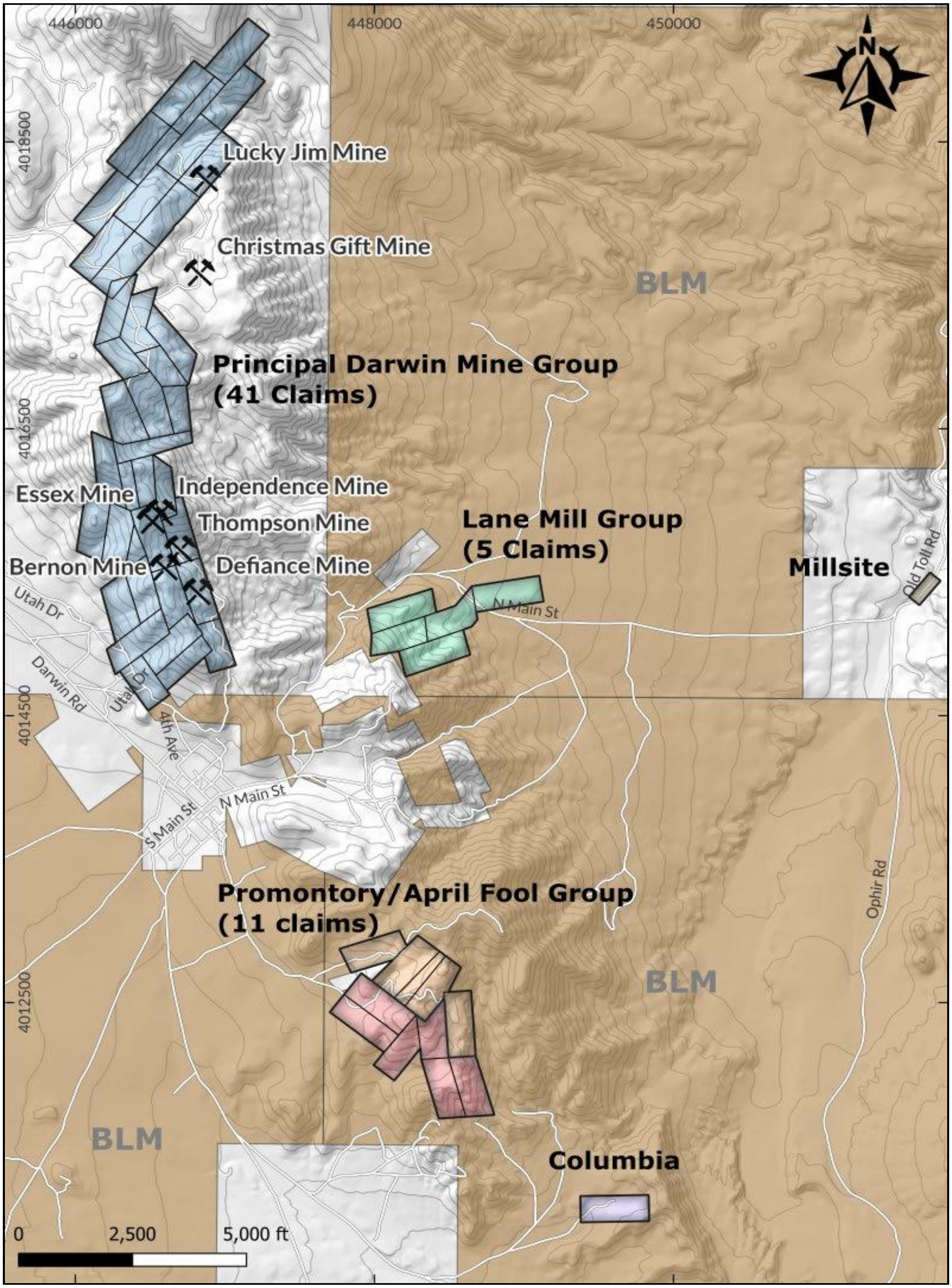


FIGURE 2 PATENTED LODE CLAIMS (SCALE 1:30,000)

Permitting

Operating a mine on patented claims in Inyo County, California, necessitates a suite of permits and licenses that address land use, water rights, environmental, operational, and safety requirements. Essential permissions include a water and powerline right-of-way lease from the California State Lands Commission (PRC 4627.2) and the Bureau of Land Management (CA-8872), along with Water Right Permits 1086 and 19497 for water diversion from Darwin Wash.

Environmental compliance is managed through a Draft Negative Declaration of Environmental Impact and a Conditional Use Permit (CUP #2007-04) from the Inyo County Planning Department, which imposes conditions on operations, including daylight work hours, weekday ore haulage, and a 500-ton daily production limit. Additionally, a General Stormwater Permit from the Lahontan Region California Regional Water Quality Control Board and an "Authority to Construct" from the Great Basin Unified Air Pollution Control District are essential for environmental protection.

Operational licensing includes permits from the Inyo County Health Department for water storage and septic systems, and an explosives license (9-NV-023-33-2M-00264) from the U.S. Bureau of Alcohol, Tobacco, Firearms, and Explosives. Compliance with the Federal Mine Safety and Health Administration (MSHA) covers all mining activities for safety. The operation's exemption from the California Surface Mining and Reclamation Act of 1975 (SMARA) is due to the project not anticipating new areas of disturbance exceeding one acre nor the removal of more than 1,000 cubic yards of material, with all waste rock and process tailings planned to be backfilled underground.

There are no known environmental liabilities attributable to Project Darwin, LLC. Potential liabilities associated with the reclaimed heap leach pad and tailings pond are assumed to reside with Arco Petroleum, successor to Anaconda and now part of British Petroleum.

Accessibility, Climate, Local Resource & Physiography

Accessibility

The Darwin Project is easily accessed year-round via the California State Highway 136/190 (the Death Valley Highway), which joins US 395 a mile south of Lone Pine. The paved Darwin Road turns south from 190 about 32 miles from the Highway 395 junction and runs south 5.6 miles to the mine site. The last half mile to the mine facilities is graded. Both Bakersfield and Independence have airports, with larger airports located in Las Vegas and Los Angeles.

Climate and Physiography

Climate is typical of the high desert of southeastern California; The average temperature variation ranges from 25 deg. F in the winter to 90 deg. F in the summer. Annual precipitation averages 18 inches.

Vegetation in the area is representative of the high desert ecosystem, with lodgepole and Jeffrey pine trees, sagebrush, and manzanita. Aspen trees are found near springs or areas with available water. Wildlife includes deer, turkey, mountain lion, squirrel, and many species of birds.

The Property elevation ranges from 3,600 feet at the well site in Darwin Wash to 6,000 feet at Ophir Mountain, a peak on the ridge in which the Darwin deposits are located.

Local Resources

Darwin, California, an unincorporated mining community within Inyo County, hosts a population of 74 according to the 2021 census, with no provided community services. The nearest population center is Lone Pine, California. Electrical power to the mine site is delivered via a 3-phase line from Olancho, supplied by the Los Angeles Department of Water and Power, at a voltage of 12,500 volts and a capacity of 2,500 kW. Although transformers for 3-phase power are present on-site, they have not been connected. The industrial power rate stands at 6.5 cents per kWh. The existing pipeline and power line infrastructure connecting the mine site to the well are in a state of disrepair, necessitating upgrades. The site is equipped with the necessary materials to refurbish the power line infrastructure. Water extraction from the well requires overcoming a vertical distance of approximately 1,800 feet (equivalent to around 780 psi static head) to traverse Darwin Ridge and reach the storage tanks. Two operational tanks situated above the mine office have a combined storage capacity of 38,000 gallons, facilitating gravity-fed water distribution to the plant area. Additional on-site water storage includes various tanks and a swimming pool, totaling 195,000 gallons.

The operational infrastructure encompasses a sizable residence, a mine office equipped with three fire-resistant walk-in vaults, storage warehouses, a shop building housing machinery and three large IR Imperial air compressors (lacking motors), and facilities at the 400 Level portal. The mine office also houses an extensive archive of engineering, geological, and production documents and maps accumulated by Anaconda and later operators. The Darwin Project possesses adequate surface rights for a small-scale underground mining and processing operation.

While a local workforce is accessible, there is a notable deficiency in personnel with mining and processing expertise. Recruiting a substantial workforce externally could significantly influence the demographic composition of Darwin, escalating the demand for critical services and infrastructure, including educational facilities, utilities, and water supply, to accommodate this growth.

FIGURE 3 PHYSIOGRAPHY OF PROJECT DARWIN, FROM THE DARWIN PROJECT AREA NORTHEAST OF RADIORE TUNNEL ENTRANCE, VIEWING NORTH-NORTHEAST.

History

Mining activities in the Darwin area, also known as the New Coso Mining District, trace back to the pre-1870s era, initiated by Mexican miners who commenced high-grade silver ore extraction. By 1870, the production landscape shifted with the displacement of Mexican miners and the onset of lead-silver ore extraction from mines such as Lucky Jim, Promontory, Columbia, Lane, Defiance, Thompson, and Essex. These mines, which constitute the patented claims of Project Darwin LLC, remained active until 1893, alongside the operation of several small smelters.

Following a period of dormancy, the Darwin Development Company embarked on property consolidation efforts in 1916, amidst a landscape marked by operational challenges including poor management, legal disputes, and financial instability. The subsequent formation of the Darwin Silver Company and later the Darwin Lead Company sought to address these issues.

Anaconda's involvement in 1939 entailed an evaluation of the Darwin Lead Company's operations, revealing significant lead-silver ore production, though profitability waned due to reliance on lower-grade ores and inefficient recovery processes. Despite attempts to enhance recovery rates, sustained profitability remained elusive. Further assessment in 1942-44 led to intermittent production until 1944, when operations achieved a notable milestone with continuous milling and ore shipments to a smelting plant in Midvale, Utah.

Anaconda's acquisition of the Darwin mine in 1945 marked a pivotal moment, with subsequent efforts focused on infrastructure enhancement, including the completion of the Radio ore Tunnel to facilitate ore transportation and processing. Despite sustained operations until 1957, declining lead and zinc prices prompted the mine's closure.

The exploration and understanding of the Darwin district's geology took a significant leap forward in 1980 with the collaborative work of R.J. Newberry from Stanford University and Anaconda geologists. Their efforts, particularly summarized in G.E. Wilson's 1981 report, unveiled a new structural model for the district, highlighting skarn bodies, breccia pipes, and the true nature of Paleozoic rocks. This period marked a shift towards a more systematic exploration approach, employing underground geochemistry and mapping to identify key mineralization trends and structural features, such as northwest-trending anticlines and synclines that contradicted previous theories of an overturned syncline. The discovery of skarn and breccia pipes enriched with minerals provided new targets for exploration, particularly for tungsten, which was identified as a future commodity of interest despite its exploration challenges.

In the early 1980s, deep core drilling efforts further informed the geological understanding of the district, with two +2000' holes drilled to test the porphyry-skarn potential and the depth extension of known mineralization zones. These efforts revealed complex zonation patterns and suggested the presence of significant, yet challenging, mineral deposits. Despite these insights, the variable mineralization with depth and the economic feasibility of mining these deeper resources remained a concern.

The Quintana period (1984-1989) followed the sale of the property by Anaconda's parent company, marking a shift towards potential heap leach operations to recover silver and gold, alongside a reevaluation of the mine's hard capital value and the potential for rediscovering bulk mineable ore bodies. Despite plans for near-surface drilling and exploration, the focus on silver prices and the eventual sale of the property to Blue Range Mining Co. underscored the fluctuating interests and strategies in the mine's development.

Blue Range's tenure (1989-1994?) brought updates in mine mapping, infrastructure, and preliminary mill work, alongside an estimation of reserves and potential ore. Despite limited exploration outside the immediate underground workings, the identification of breccia pipes and favorable geological structures hinted at unexplored potential within the district. However, the challenging economics of deep exploration and the focus on high-grade polymetallic deposits indicated a cautious approach to future mining endeavors.

The brief Cyprus Minerals period in the early 1990s, marked by an aborted joint venture and exploration initiative due to funding and market challenges, highlights the ongoing complexities of mining economics and the importance of strategic partnerships in advancing exploration and mining operations. This period of exploration, spanning over a decade, illustrates the evolving understanding of the district's geology and the continuous quest to balance mineral potential against economic viability. Project Darwin, Inc.'s acquisition in 1995 marked a new chapter, characterized by ongoing refurbishments, equipment acquisitions, and permit updates.

Historical Exploration

A comprehensive examination of numerous reports archived in the Darwin mine office reveals that Anaconda's underground drilling activities, integral to its mine production endeavors, were executed with a specific focus on developing reserves. However, the precise details regarding the type and extent of drilling procedures followed are not thoroughly documented. Anaconda's projections of reserves typically extended up to a year, drawing upon past experiences within the mining system and a fundamental understanding of the geological features, often based on one or two drill intercepts.

Blue Range and Quintana completed small drilling campaigns largely to confirm or outline known areas of mineralization. Like Anaconda, the precise details regarding the type and extent of drilling procedures followed are not thoroughly documented. However, assay certificates and weekly drilling reports are available for the Blue Range underground drilling project.

Anaconda and Blue Range drilled at least 1,700 underground holes, accumulating nearly 160,000 feet of AX and EX core alongside long hole sludge samples. Additionally, Quintana drilled eight shallow percussion holes near the Essex shaft to evaluate near surface silver mineralization. The azimuth, dip, and depth of the drill holes were adapted to the mine's varied geological conditions. Notably, many drill holes were oriented to intersect the Darwin Stock, as it was thought to exert significant control over mineralization at the time. In the deeper sections of the mine, drilling efforts were primarily concentrated on developing known ore zones between different levels associated with the Defiance and Essex breccia pipes.

Anaconda conducted two deep diamond core holes in 1982, with logging and assaying completed in the first quarter of 1983. Hole DA-1, drilled at a -77° angle with a total depth (TD) of 2,201 feet, showing increasing skarn development with depth and expected zonation from upper sulfide replacements to low-sulfide skarn near the Defiance pipe. There was no evidence of additional sulfide-rich mineralization or retrograde alteration. Hole DA-2, drilled at a -73° angle with a TD of 2,302 feet, intersected the Essex pipe skarn breccia approximately 400 feet beneath the deepest underground workings. It revealed largely pyritic replacements at depth, with intervals of significant silver grades and locally present tungsten. The reversal of normal zonation suggests a later volatile-rich mineralizing pulse, possibly related to fault contact with the Darwin stock. Mineralization and alteration between DA-1 and DA-2 varied significantly, indicating lateral zonation with DA-2 showing over twice the metal content compared to DA-1. The unexplored area adjacent to DA-2 shows promise for silver and/or tungsten mineralization.

Historical Production

The Darwin mine includes the workings owned by The Anaconda Company which are developed by the 6,300foot Radio ore adit on the 400-foot level. They are Bernon, Defiance, Essex, Independence, Rip van Winkle and Thompson workings and the Driver prospect, the most important of which are the Defiance, Thompson, Essex, and Independence. The last three are now grouped as the Thompson mine.

Anaconda's total production, comprising both oxide and sulfide ore, is estimated at approximately 962,000 tons. The sulfide ore extracted since 1942 had an average composition of 6% lead, 6% zinc, and 6 ounces per ton of silver, according to Hall & Mackevett (1962). After being idle until 1967, the mine saw activity when West Hills Exploration Co. leased it, extracting approximately 271,000 tons of ore from deeper levels before terminating the lease in 1970 due to unsatisfactory results. Subsequently, Mexicanus Colorado, Inc. leased the mine in April 1971, investing in enhancements but operated for less than a year, producing about 74,000 tons of lead, silver, and tungsten ore. Montecito Mining Company subleased portions, extracting tungsten ore from the 433-435 stope and lead-silver ore from the deeper Thompson workings. In 1972, Secundo Guasti leased tungsten extraction unsuccessfully, and other lessees, including Pacific Tungsten, West End Consolidated, and Brownstone Mining Company, primarily mined tungsten ore from the 433-435 stope. However, all leases were terminated in 1976.

Cumulatively, historic ore production spanning over a century from 1875 to 1976 yielded substantial quantities of silver, lead, zinc, copper, and gold, with an estimated total production of approximately 1.5 million tons of ore. Metal production during this period included about 10,500,000 ounces of silver, 840,300 tons of lead, 57,900 tons of zinc, 745 tons of copper, and 5,900 ounces of gold.

Regional Geology

The Darwin Project is situated within the Basin and Range metallogenic province and is largely composed of Paleozoic sedimentary rocks and carbonates, overlain by Mesozoic volcanic rocks and intruded by Mesozoic granodiorites. The intrusion of numerous igneous bodies throughout the region occurred as components of the Sierra Nevada batholith, associated with a major tectonic transition from a quiet passive continental margin during the Paleozoic era (from 545 to 242 Ma) to a convergent continental margin in the Mesozoic era (241 to 66 Ma). Intermittent periods of magmatic activity extended throughout the Cenozoic era (65 Ma to recent).

During the Mesezoic era, volcanic activity increased at surface, driven by interactions with the subducting Farallon oceanic plate. Continued subduction facilitated the ascent of intrusions into the upper crust, closely associated with the majority of vein-hosted gold deposits throughout California (Close, 2015).

In the Cenozoic era, the subsidence of convergent subduction gave rise to back-arc extension and crustal relaxation, propagating from eastern California through Nevada to Arizona, Utah, and Idaho. This extensional regime created the 'Great Basin and Range' of the United States. Early Cenezoic mineralization within this region predominantly ensued along transverse and extension-related fault systems, alongside the emplacement of intrusions and volcanic rocks (Fiero, 1986).

From 45 Ma to 30 Ma, an Eocene-aged pulse of magmatism throughout the entire western Cordillera brought a resurgence of fluid migration and gold mineralization. This epoch is characterized by many of the 'Carlinstyle' gold deposits in Nevada and other intrusion-related porphyry and vein deposits throughout western North America (Fiero, 1986).

During the Miocene, (from 24-4 Ma), several volcanic fields and intrusions also occurred along the California Nevada border. Volcanic rocks of this age and their associated intrusions occur throughout northern Mono County.

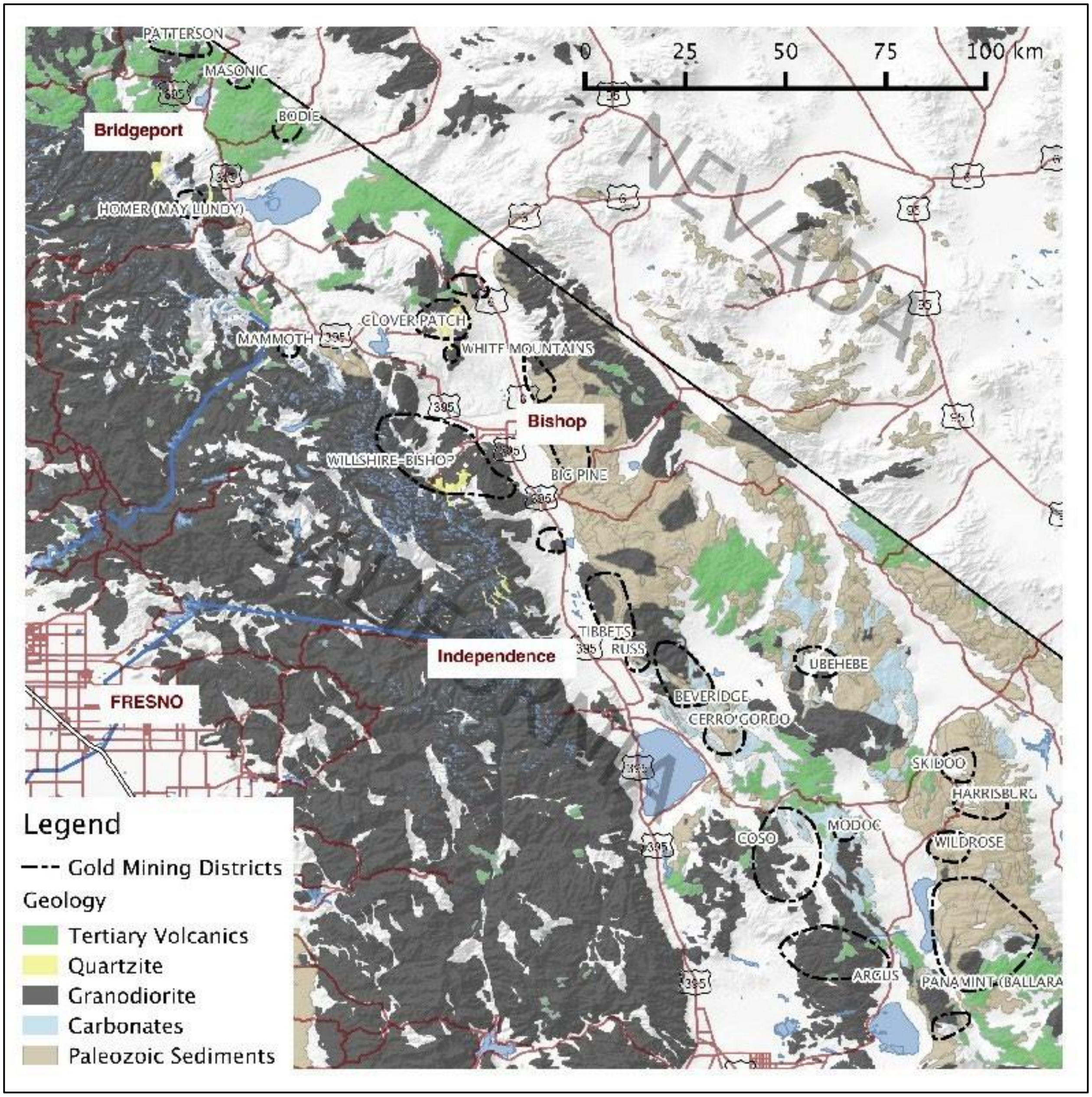


FIGURE 4 REGIONAL GEOLOGY SHOWING GENERALIZED LITHOLOGY, REGIONAL TRENDS, AND MINING DISTRICTS.

Regional Mineralization

Mineralization within Mono and Inyo counties is broadly categorized into four distinct geological domains based on their host environments and metal assemblages. These domains include volcanic-hosted Au-Ag, metamorphic Au-Ag-W-Lead, carbonate base metal-Au, and sediment-hosted Au-Ag domains, each presenting unique mineralization styles (Close, 2015).

The Darwin district (interchangeably referred to as the Coso Mining District), in SE California (Figure X) contains Pb-Zn-Ag as well as Cu-and W-rich veins and skarns (Hall and MacKevett, 1962; Newberry, 1987).

The volcanic-hosted domain is characterized by sulfide lodes and breccias within volcanic rocks, with mineralization predominantly occurring in highly silicified ore shoots and breccias (Close, 2015). Notable within this domain are the Bodie, Masonic, and Patterson districts, where gold and silver are found in association with granodiorite intrusions and volcanic host rocks.

The metamorphic domain hosts mineralization along metamorphic fabrics within quartzites, marbles, and metamorphosed intrusive rocks. Characterized by higher metamorphic grades and the presence of tungsten, alongside gold and silver, this domain spans several districts, including the Homer/May Lundy and Clover Patch. Mineralization is characterized by silicified structures, quartz veins, and breccias (Close, 2015). The districts in this domain are the Homer/May Lundy, Blind Springs, Clover Patch, Casa Diablo, Willshire-Bishop (tungsten district), and Fish Springs-Tinemaha. The most prominent mines in this Domain are the Homer/May Lundy, New Era Mine (Fish Springs) and the Bishop Creek/Cardinal Mine and the Pine Creek Tungsten mines of the Willshire-Bishop District.

The carbonate base metal-Au domain, spanning approximately 125 kilometers from north-central Inyo County to the southeast, extends through a carbonate and granodiorite sequence, with prominent mines in the Cerro Gordo, Coso (Darwin), and Argus districts. This domain is known for its high-grade assemblage of silver, lead, gold, zinc, and copper mineralization, characterized by skarn-type assemblages proximal to the intrusive-carbonate contacts (Close, 2015).

Lastly, the sediment-hosted domain encompasses gold mineralization primarily within Mesozoic-aged granodiorites and adjacent Paleozoic sediments. Gold is often associated with sulfides in thin veins and structurally related features across several districts, including the White Mountains and Big Pine (Close, 2015). Notably, this domain has historically yielded a significant amount of gold production, with continued exploration activity to the present day.

Regional Structure

The geological framework in California reflects a complex history of structural deformation. Many of these structures are attributed to the eastward subduction of the Farallon plate and subsequent transverse dextral (right-lateral) compression of the continental margin to the north- northeast (Close, 2015).

Major structures and folds strike north-northwest, and subsequent veins and secondary faults strike north northeast and exhibit dextral motion. This structural framework is ubiquitous across the majority of the gold veins and lode deposits in Mono and Inyo counties. However, it is imperative to acknowledge that localized stress refraction, stratigraphic variation, and other geological variables may produce deviations in structural attitudes from the overall trend (Close, 2015).

Property Geology

The Darwin area's geological history, as documented by sources including Dunne et al. (1978), Stone et al. (1989), and Newberry (1991), encapsulates a sequence of significant geological events from the Upper Paleozoic through the Cenozoic era, characterized by sedimentary deposition, structural deformation, plutonic intrusions, and metamorphic transformations. The Darwin area's geology, outlined in studies by Dunne et al. (1978) and Stone et al. (1989), encompasses a series of geological events from the Upper Paleozoic to the Cenozoic era, characterized by sedimentary deposition, deformation, plutonic intrusions, and mineralization. Upper Paleozoic carbonate turbidites, as described by Stevens (1986) and Stone (1984), were subject to broad folding attributed to the Late Triassic Last Chance thrust fault system, resulting in stratigraphic overthickening due to the substantial thickness of the fault's upper plate, now eroded.

Subsequent geological activity includes the intrusion of mid-Jurassic alkaline and calc-alkaline plutons— specifically, the 175 Ma Darwin stock and the 156 Ma Coso batholith (Chen, 1977)—located west of the alkaline plutons. These plutons, suggested by Sylvester et al. (1978) to be emplaced at depths of 4-6 km, initiated contact metamorphism in adjacent carbonate rocks, producing idocrase-wollastonite calc-silicate hornfels, garnet-rich skarnoid, and bleached marble around the Darwin stock. This metamorphic zone hosts tungsten-bearing skarns formed at contacts with the Darwin stock (Newberry, 1987).

The Davis fault system's activity around 154-148 Ma (Dunne et al., 1978) displaced the upper plate eastward by 1-3 km, juxtaposing the Coso batholith and its Cu-skarns with the Darwin stock and its W-skarns, and compacting the metamorphic assemblages—a process described as telescoping.

Following this phase, granite porphyry dikes and breccia pipes intruded the region along the northwest margin of the Darwin stock. This intrusion phase coincided with the formation of Pb-Zn-Ag skarns and veins, primarily on the west side of the Darwin stock, associated with steeply dipping faults adjacent to breccia bodies, less commonly along granite porphyry dike contacts, and along the Davis thrust. Cenozoic reactivation of the Davis thrust, in the context of basin and range uplift, slightly deformed nearby ore bodies.

Carbonate Stratigraphy

Within the project area, the stratigraphic framework is defined by a succession of carbonate units from the Upper Paleozoic to the Cenozoic, structured as follows:

Lee Flat Limestone

Originating in the late Mississippian to Early Pennsylvanian, the Lee Flat Limestone is exposed in the property's northwestern sector, directly underlying the Keeler Canyon Formation. Characterized by its dark gray, thinly bedded limestone with interspersed, sandy, iron-stained partings, this unit forms the base of the local stratigraphic sequence. Despite its geological significance, the Lee Flat Limestone is not directly associated with the area's mineralization events.

KEELER CANYON FORMATION

The Keeler Canyon Formation, conformably overlying the Lee Flat Limestone, is the dominant carbonate stratigraphic unit within the project, and serves as a receptive host for mineralization under the prevailing structural and metasomatic conditions present at Darwin. This formation encompasses a diverse lithology, including middle Paleozoic shelf carbonates and upper Paleozoic deep-water carbonates, alongside siliciclastic units, and discrete, lenticular carbonate bodies, which can extend up to 300 feet in strike length. Encapsulated frequently within sequences of carbonate turbidites and calc-pelites, these lenticular bodies denote a depositional environment extending from intertidal/supratidal zones, evidenced by a rich benthic fauna, to slope and basin environments, characterized by planktic fossils and sediment gravity flow deposits. Stratigraphically, the Keeler Canyon Formation is divided into upper and lower members, with the former comprising interbedded calcarenite, calcilutite, pink fissile shale, siltstone, and limestone pebble conglomerate. The lower member is distinguished by its calcarenite layers that include limestone pebble conglomerate beds, fusulinids, and spheroidal chert nodules of ½ to 1½ inches in diameter.

OWENS VALLEY FORMATION

This unit, conformably situated above the Keeler Canyon Formation with limited exposure in the property's northeast. The Owens Valley Formation is comprised of two members: the Upper Member, constituted of limestone conglomerate, siltstone, and sandstone, and the Lower Member, typified by brick-red and yellow-brown shale, siltstone, and blue-gray limestone.

This stratigraphic framework, from the Lee Flat Limestone through the Keeler Canyon Formation to the Owens Valley Formation, delineates the progression of carbonate sedimentary environments within the project area, each contributing distinct lithological and sedimentological characteristics to the region's geologic history and its potential for mineralization.

Intrusive Sequence

The intrusive sequence in the Darwin area is subdivided into the Darwin Stock, Coso Batholith, and Quartz Porphyry Suite, each associated with distinct metasomatic processes.

DARWIN STOCK

Dated at 174 million years (Chen, 1977), this stock spans approximately 1 square mile and displays quartzpoor, medium to coarse-grained textures, with a composition ranging from diorite to alkali-feldspar quartz syenite. Central zoning features felsic compositions. The stock includes diverse intrusive bodies such as coarse-grained dioritic to monzo-diorite, hornblende-biotite granodiorite, and a variable, fine-grained porphyritic unit. Despite initial assumptions, Newberry (1980) determined that the Darwin Stock does not serve as the primary source of mineralization, instead leading to the formation of non-mineralized contact metamorphic rocks and skarns.

COSO BATHOLITH

At 156 million years old (Chen, 1977), the Coso Batholith's calc-alkaline composition is primarily granitic to granodioritic. Exposed on the Davis Thrust's upper plate, it features quartz, feldspar, hornblende, and biotite, with accessory minerals like sphene, apatite, and magnetite. Characterized by a coarse-grained, light-colored, often sheared, and chloritized texture, the batholith is linked to the formation of Cu-bearing skarns.

QUARTZ PORPHYRY SUITE

This suite, distinct for its fluorine-rich and alkali-feldspar quartz-rich granite composition, contrasts with the Darwin Stock by lacking quartz-phenocrysts. It contains biotite, sphene, apatite, and occasionally interstitial minerals like fluorite and epidote. Exhibiting textural diversity including porphyry, aplitic, pegmatitic, and granophyric textures, the suite undergoes alterations to sericite-pyrite or calcite-chlorite-epidote. Positioned within and beyond the Darwin Stock, it forms the matrix in igneous breccias with scheelite-bearing skarn clasts, particularly in the district's west-central area (Newberry, 1987).

Contact Metamorphic Rocks

In the Darwin district, the metamorphic landscape is characterized by the presence of skarnoids, an intermediate between metamorphic and metasomatic rock classification of metamorphic rocks defined by their distinct mineralogy, primarily consisting of grossular, diopside, and idiocrase. These rocks, initially identified by Zharikov in 1970, emerge from a geological backdrop where the original sedimentary layers, composed of a mixture of carbonate and pelite (calc-turbidites), have been subjected to metamorphism. The transformation of these protoliths is driven by the thermal and to a lesser extent chemical influence of igneous activities in the region, leading to the extensive distribution of skarnoids across extensive strike lengths. These formations are noted for their layered to massive textures, the dominance of pale-colored, low-iron minerals, and the preservation of original sedimentary structures through pseudomorphism. Additionally, the skarnoids display a compact morphology without the presence of ore minerals. Any veins within these rocks contain minerals with compositions that mirror those of the surrounding host rocks, indicating a localized derivation of materials. These units exhibit isotropic textures, with anhedral, poikilitic crystals of garnet and idocrase that include inclusions of fine-grained pyroxene, wollastonite, calcite, and quartz.

The intrusion of the Darwin Stock is recognized as a significant factor in the development of a contact metamorphism zone within the calcareous-turbidite formations. However, the influence of subsequent intrusions on the metamorphic and metasomatic landscape of the area also warrants consideration. This zone of contact metamorphism, spanning approximately half a mile in width, demonstrates a diversity of metamorphic rock types, reflecting not only the thermal effects of the Darwin Stock but also the possible contributions of later magmatic phases.

Structure

The Darwin mineralization is situated within structurally controlled zones of crosscutting and bedded skarn, localized within Paleozoic turbidite carbonate and siliceous clastic units that have metamorphosed into marble or calc-silicate hornfels, notably near the west contact of the Darwin stock. These structural features significantly influence the localization and form of mineral deposits.

The upper Paleozoic carbonate rocks underwent deformation into broad folds, likely due to tectonic activity from the late Triassic Last Chance fault system, which was located above these formations and has since been eroded away. This deformation led to a pronounced increase in the thickness of the carbonate stratigraphy and resulted in the creation of the Darwin Anticline, which exhibits a trend of approximately 340°. The orientation of the Darwin Anticline's axis not only aligns with regional folding patterns but also runs parallel to the west contact of the Darwin Stock. The intrusion of the Darwin Stock has notably truncated the anticline's eastern limb. Additionally, several sill-like extensions from the Darwin Stock penetrate the sedimentary layers, conforming to the bedding planes as they extend into the anticline's western limb.

Two predominant structural orientations are observed: east-west (E-W) trending high-angle, left-lateral faults, and north-south (N-S) trending moderate-angle thrust faults. The thrust faults identified within the project area are recognized as part of the Davis thrust fault system. This system is a subset of the larger Swansea-Coso thrust system.

The thrust fault system consists of several west-dipping low-angle faults near the anticline's crest, disrupting the Paleozoic section and partially obscures the intrusive contact. This thrusting, which occurred around 20 million years after the emplacement of the Darwin stock, predates mineralization. These thrust faults are characterized by northeast (NE) transport directions with estimated aggregate displacements ranging from ½ to 2 miles.

The high angle fault system is divided into two groups based on their directional trends: a west-northwest (WNW) trending group, which is associated with displacements up to 950 feet and intersects the thrust faults, and an east-northeast (ENE) trending group, characterized by minor displacements and typically terminating at thrust faults. The structural interactions suggest that the main phase of movement along the high-angle faults, encompassing both the WNW and ENE trends, occurred prior to the development of thrust faulting. Subsequent minor structural movements on the WNW-trending (primary) faults likely happened in conjunction with or after the thrust faulting events.

The high angle faults are commonly associated with mineralization in the underground workings. The intersections of high-angle mineralizing structures with the hinge zone of the Darwin Anticline and secondary, doubly plunging north-trending folds along its flanks create zones conducive to significant bedded mineralization and serve as conduits for hydrothermal fluids. This interaction led to the development of reaction skarns (carbonate replacement), massive skarns, and quartz-carbonate-sulfide fissure/veins, with some containing scheelite.

Notable Pb-Ag-Zn mineralization is located along the Defiance and Essex fissures, characterized by steeply west-plunging skarn breccia pipes that surround Quartz Porphyry pipes intruded at the intersection of the high angle faults and the anticline’s hinge zone. The Defiance area extends to the

1,300 Level (approximately 1,500 feet below the surface), and the Essex area to the 900 Level (about 1,100 feet below the surface). Other important high angle fissures that impact mineralization include the Mickey Thompson, Water Tank, 434, Bernon, Copper, and A472 fissures.

Mineralization and Alteration

The Project area's mineralization is the result of at least three distinct mineralizing events, each associated with separate intrusions. These events have led to a complex overlay of metasomatic processes, culminating in the formation of mineralized zones rich in varying combinations of Cu, Ag, W, Pb, Zn, and Ag.

Proximal to the Darwin stock particularly evident on along eastern contact, skarn formations are delineated by their constrained dimensions and spatial distribution, with a notable association to W-anomalous, tourmaline-bearing quartz syenite phases within the stock. These formations are characterized by the presence of scheelite mineralization. The skarns identified exhibit widths ranging from 1 to 3 meters and are composed of medium-grained aggregates of brown, euhedral garnets with patchy anisotropic textures, in addition to green pyroxene. The aggregates include voids filled with quartz, calcite, and K-feldspar. Accessory minerals such as pyrite, amphibole, calcite, quartz, and sporadically, chalcopyrite are observed within later-stage alteration veins and disseminated formations. Tungsten concentrations within these skarns typically range between 0.1% and 0.5% WO_3.

Adjacent to the Davis thrust and proximal to Coso-type granitic intrusions, certain skarn formations demonstrate robust and compact structures with thicknesses up to 65 feet. These skarns predominantly consist of dark, isotropic garnet and a secondary presence of pale pyroxene, including sulfides where pyrite predominates over chalcopyrite, with occurrences of specular hematite. The mineral suite within these skarns encompasses epidote, actinolite, calcite, quartz, and sulfides, highlighting their exploration potential for copper.

Exo- and endoskarn-matrix breccias identified within the area feature pipe-shaped structures intersecting metamorphic strata and the Darwin stock, incorporating fragments of various rock types embedded within a matrix of fine-grained, pale grey-green garnet, idocrase, aluminous pyroxene, and calcite. These formations undergo extensive retrograde alteration defined by secondary mineral replacement, including calcite, quartz, epidote, white mica, actinolite, and pyrite. Tungsten concentrations of 0.1% WO_3 have been noted in areas covering approximately 2,400 square yards. These breccias also feature pyrite and occasional galena and sphalerite, with significant base metal concentrations associated with intersecting veins.

Pb-Zn-Ag skarn formations within the Darwin district are closely associated with east-west trending faults and faulted intrusive contacts, often proximal to granite porphyry-breccia dikes and pipes. These zones vary in mineral composition with depth, transitioning from pyroxene dominance to garnet predominance near the surface. Microscopic examinations reveal the detailed structure of garnets and the presence of disseminated galena, sphalerite, pyrite, chalcopyrite, and fluorite. The mineralization within the Darwin stock is characterized by vuggy textures with garnet-epidote-pyroxene mineralization that is limited to faulted and fractured zones, indicating the metasomatic process was driven by fluids derived elsewhere.

Vertical pipes intersecting the Pb-Zn-Ag skarn formations contain quartz, calcite, hematite, sulfides, fluorite, diopside, idocrase, bustamite, and fine-grained yellow garnet. Quartz-calcite-fluorite-barite-sulfide veins, rich in Ag-Pb-Bi-As, cut skarns, skarn pipes, and both igneous and metamorphic rocks.

In the underground workings of the Defiance and Essex areas, a vertical progression is observed from deep sulfide-bearing skarns, through sulfide-bearing breccia pipes, to sulfide-bearing veins (Newberry, 1987). The continuity among sulfide-bearing skarns, breccia pipes, and veins suggests a shared genesis across different types of Pb-Zn-Ag mineralization, indicating a spatially extensive hydrothermal system.

Pb-Zn vein and replacement bed mineralization, occurring both above and below the Davis thrust without intersecting it, are associated with both Cu Skarns and Pb-Zn-Ag skarns, suggesting a distal relationship to these skarns rather than a single hydrothermal source. Near Scheelite skarns, the mineralized veins or beds integrate tungsten into the Pb-Zn composition. This tungsten enrichment stems from the remobilization of scheelite within retrograde altered skarns.

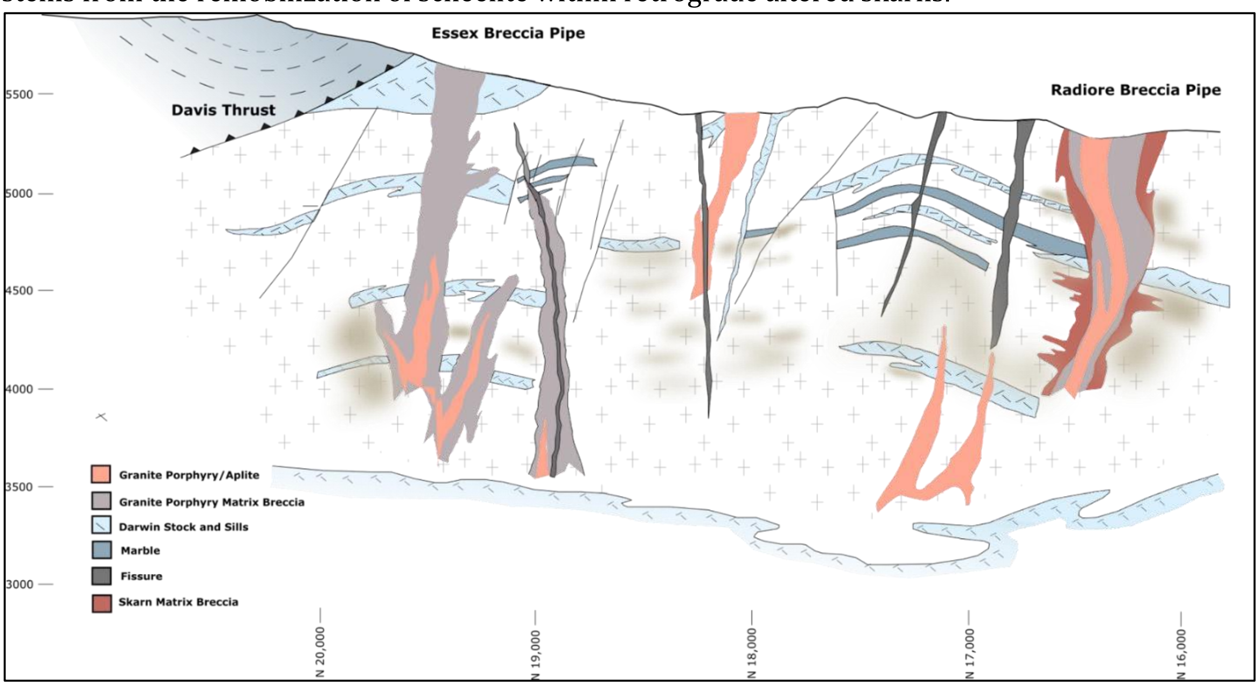


FIGURE 5 CONCEPTUAL LONG SECTION OF THE DARWIN MINERALIZATION

Mineral Zonation

The mineralogical and textural distribution within the Essex pipe delineates a clear zoning pattern, comprising both a systematic arrangement around a core zone and a broad vertical zoning, corroborated by geological mapping. These patterns in mineral paragenesis and zoning indicate the occurrence of multiple mineralization events throughout the formation of the skarn. Apart from the distribution of pyrrhotite—which might represent an early stage of marble bleaching associated with fluids related to the Darwin stock, prior to Pb-Zn skarn formation—there's no evidence to suggest that the skarns and skarn-related ores exhibit zoning directly influenced by the Darwin stock.

The breccia pipes, exemplified by the Essex and Defiance sites, display compositional zonation characterized by distinct gradients in mineral compositions, indicative of the controlling hydrothermal processes. Pyroxene minerals within these structures show an increase in iron (Fe) and manganese (Mn) contents from the core to the periphery, reflecting a thermal and chemical evolution of the system influenced by the hydrothermal fluid dynamics. Similarly, galena exhibits a zonal distribution with increasing bismuth (Bi) and antimony (Sb) away from the core, suggesting a process of selective deposition and fractionation over time.

In contrast, sphalerite maintains a consistent composition across the pipes, implying a stable sulfur fugacity environment during its formation despite the variability observed in other minerals. This uniformity suggests that the conditions affecting sphalerite crystallization were less influenced by the geochemical gradients impacting pyroxene and galena deposition.

Moreover, the symmetrical distribution of pyroxene compositional isopleths, aligning with the pipe outline and the underlying granite porphyry plugs, underscores the significance of the thermal and chemical contributions of these intrusions to the mineral zonation observed.

Deposit Types

The conceptual exploration model employed by Project Darwin targets a complex array of mineral deposit types, including lead-silver-zinc-bearing fissure veins, bedding replacement deposits, massive skarn, and breccia pipes characterized by a skarn matrix with garnet and pyroxene, sometimes centered by quartz porphyry bodies. A notable feature within these deposits is the presence of tungsten, specifically scheelite, which is found in veins and skarns along the contact with the Darwin stock and within certain bedded replacement ores.

Figure 6 illustrates a typical environment in which polymetallic skarns form. When a granite or porphyry stockwork intrudes into a carbonate sedimentary sequence, the fluids associated with the intrusion pass through the contact sediments. This creates prograde hydrothermal alteration of varying intensities as a function of the host sediment composition and reactivity of this with the fluids. Zonation is often characteristically evident in both the alteration suite and tenor of mineralization, both increasing toward the center of the intrusive stockwork.

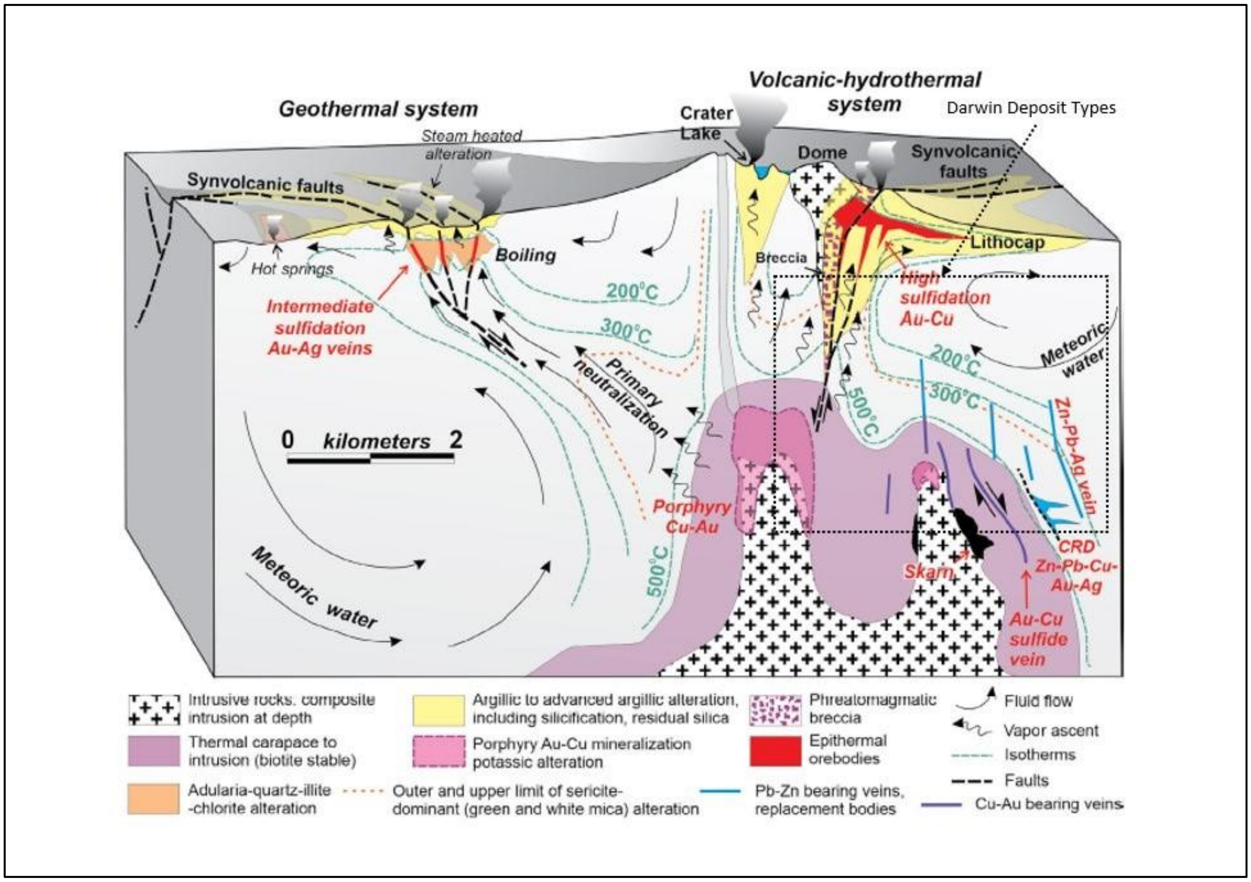


FIGURE 6 SCHEMATIC ILLUSTRATION OF THE GEOLOGICAL SETTING AND CHARACTERISTICS OF LOW- AND HIGH-SULFIDATION EPITHERMAL AU-CU DEPOSITS AND SUB-VOLCANIC PORPHYRY-TYPE CU-AU DEPOSITS (RHYS ET AL., 2020).

The geological model guiding exploration in the Darwin mine integrates several key concepts:

Intrusive-Related Mineralization: The model posits that the mineralization events, particularly for lead, silver, and zinc, are associated with late-stage felsic intrusive activities postdating the emplacement of the Darwin stock. This is supported by the spatial association of mineralized zones with quartz porphyry bodies and the alignment of mineral and metal zoning around these intrusives.

Structural Controls on Ore Localization: Exploration efforts have focused on understanding the structural complexities within the deposit, including the role of first- and second-order folds and crosscutting fault systems as major controls on ore distribution. The presence of small but high-grade orebodies, often delineated by faults or lithological boundaries, underscores the importance of detailed structural mapping in targeting exploration.

Zonal Mineralization Patterns: The observed concentric and vertical zoning of minerals and metals around breccia pipes and within the broader deposit area indicates a systematic variation in the chemical conditions of mineralization. These patterns, highlighting progressive changes in elements like zinc, lead, copper, and tungsten with depth and proximity to intrusive sources, are central to the exploration model.

Geochemical Gradients and Fluid Dynamics: The exploration model incorporates the concept of geochemical gradients, as evidenced by variations in the compositions of minerals like pyroxene, galena, and sphalerite. These gradients reflect changes in the temperature, pressure, and composition of the hydrothermal fluids responsible for mineralization, guiding drilling strategies to target zones of predicted mineral enrichment.

Empirical Exploration Approach: Historical exploration strategies, while informed by geological insights and drilling data, have been largely empirical, focusing on short-term reserve projections. Subsequent efforts aim to refine this approach by integrating advanced geological modeling and geochemical analysis to better predict orebody locations and qualities.

The exploration program at the Darwin mine is thus planned around these geological models and concepts, employing a range of techniques from geological mapping and geochemical sampling to geophysical surveys and targeted drilling. This comprehensive approach aims to delineate the extent of mineralization, understand the complex geologic setting of the deposits, and optimize the discovery and development of ore bodies within this multifaceted mineral deposit environment.

Exploration

In the most recent exploration efforts conducted between 2009 and 2010, underground sampling was undertaken on the 400 Level within the Defiance Pipe, along with an adjacent zinc oxide stope. Donald Strachan, acting as a consultant for World Industrial Minerals Inc., spearheaded these sampling activities. The findings from this endeavor have unveiled promising indications of the presence of tellurium as a potential byproduct in zinc production.

Sample Preparation, Analysis and Security

Samples collected from the 400 level by Project Darwin were analyzed using a multi-element ICP analyses performed by Inspectorate America Corp. and Florin Analytical Services in Reno, Nevada. Both laboratories are believed to follow standard QA-QC practice using blanks and certified standards, but there is no mention of QA/QC procedures followed by the Company.

Data Verification

A focused site visit to the project site and field office was conducted on December 11-13, 2024, by Zachary J. Black, SME-RM (qualified person - QP). Discussions were held with Darwin personnel to gain a thorough understanding of the project, available data, and the exploration and production history. This visit included:

- General Geologic Reconnaissance: Inspection of facilities, outcrops, and accessible underground mine locations to verify consistency with reported data and assess geologic continuity.
- Core Review: Examination of available core intervals across various lithologies and grade ranges. This confirmed alignment between observed lithologies and logged information for most samples.
- Field Observations: Field observations generally confirmed previous reports on project geology. Bedrock lithologies, alteration, and structures aligned with existing descriptions. No significant discrepancies were identified that could contradict the current geological interpretation.

Review of Historical Data and Internal Reports

The project's archival data, which was solely available in hard copy format within the Company's office, was reviewed in detail. The historical reports and internal company documents relevant to the project are well organized in filing cabinets. The archive includes detailed records on mine production, underground drilling, geologic mapping, historical survey, and an extensive library on the progression of the geologic understanding of the project. The more recent reports that included summaries of previous work were scanned and saved in digital format for review along with any sections, level plans, and surface geologic maps.

Historical Data

Historical drilling data from various programs conducted both during Anaconda's operation of the mine and post mining was validated to support its inclusion in the historical resource estimate and interpretations of potential mineralization zones. Here's a summary of the validation process:

Data from the historical programs were considered adequate for inclusion in this report, despite some limitations:

- Limited Collar Verification: While direct verification of drill hole collars through physical inspection was not possible due to mining activity, a two-pronged approach was employed for indirect verification:
- Historical Data Comparison: Collar coordinates recorded in survey and drill hole logs were compared against historical company maps to assess their internal consistency.
- Field Evidence Assessment: Despite the removal of much field evidence during mining, efforts were made to visit the collar locations and search for remnants of drilling activity, such as casing fragments in the ribs or floor. The presence of such remnants provided some confidence in the accuracy of the collar coordinates.

Consistency with supporting data: The available data (assay certificates and drill logs) provided sufficient details regarding drill orientation and sample types. Additionally, the assay results from these programs generally agreed with the overall grades reported during production.

Mineral Resource Estimates

Ethos has conducted a thorough review of all available data supporting the historical resource estimates and considers these estimates to be reasonable given the information at hand. However, it's important to acknowledge that there are inherent limitations associated with the validation of historical data focused on the immediate delineation of ore for production. The available data primarily originates from within the underground mine, which limits the development of a comprehensive geological model and the delineation mineralization boundaries. Examination of historic stope maps reveals that past mining activities are consistent with current conceptual geological models and interpretations. However, changes in economic conditions and metallurgical recoveries have a substantial impact on cut-off grades. Such limitations introduce a degree of ambiguity that can't be validated. Users of this information should exercise caution and not place undue reliance on these historical resource estimates without recognizing these uncertainties.

Estimation Methodology

Estimating the resources of the Darwin Project presents significant challenges due to the ore bodies' irregularities and the extensive areas of mineralized ground that have not been systematically sampled. The known historical resources are limited, largely because minimal development work has been conducted ahead of mining operations. The current estimate provided by the Company is organized into 8 resource blocks.

The historical mineral resource for the project initially consisted of 150 individual blocks, documented by Anaconda, Blue Range Mining, Western Industrial Minerals, and Montecito Minerals. The historical data for these blocks were systematically cataloged. Each resource block was then outlined and verified using available maps and data from mineralized drill intercepts (Figure . In areas that were accessible, sampling was conducted to verify the consistency of mineralization across different mine exposures and levels. Additionally, resource blocks previously outlined and mined by past operators were confirmed by visual inspection and subsequently excluded from the current estimate. This process resulted in a refined count of 52 individual resource blocks, each delineated based on historical production data.

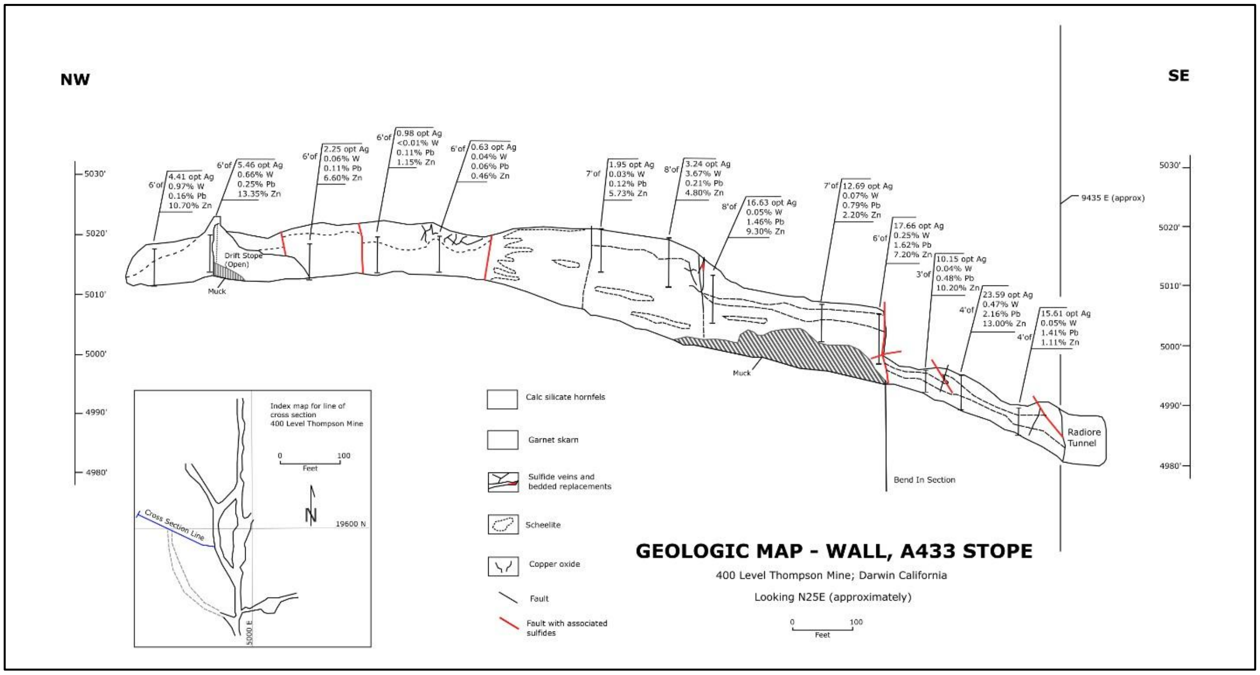


FIGURE 7 SECTIONAL INTERPRETATION OF THE A433/435 STOPE

The data related to the remaining 52 resource blocks facilitated the creation of hand-drawn maps and crosssectional diagrams, outlining the blocks in accordance with mining assumptions. The areas of sectional mineralization were tabulated using AutoCAD. Dimensions for height and length of the mineralized zones were estimated from the maps, drillhole intercepts or directly from measured exposures. Tonnage estimations for these blocks were calculated by multiplying the derived volume by the ore's specific tonnage factors, which are 9.6 cubic feet per ton for oxide, 9.0 cu ft per ton for mixed, and 8.4 cubic feet per ton for sulfide ores.

Grades for silver, lead, zinc, and occasionally tungsten and copper across these blocks were assigned the average of drill intercepts, with channel sample data from development phases integrated where feasible. The historical resources are categorized into two groups probable and possible. Probable resources are those with nearby development, while possible resources are the areas deeper in the mine along the Defiance and Essex breccia pipes. Their continuation at the lowest levels of the respective mine areas remains uncertain, although inspection of the historical stopes from levels above the area indicate a consistent and through going zone of mineralization. The current estimate provided by the Company is organized into 8 groups of resource blocks, following established estimation methods and assumptions regarding the mineralized zones. These blocks are summarized below:

- A433/435 Stope (Oxide)
  - Location: On the east limb of an anticline, connected to the 400L main tunnel.
  - Mineralization: Predominantly oxidized silver and zinc with locally significant scheelite. o Documentation: Sampled in detail by Anaconda, confirmed by Blue

Range. ○ Volume: Estimated at 1,728,000 cu ft ○ Average sample grades of 8.5 opt Ag, 0.8% Pb, 6.4% Zn, and 0.6% WO3.
  - This stope has been a major source of tungsten, with mineralization observed to extend northwest to the 458 fissure.

- B-458 Fissure (Mixed) ○ Location: Downdip extension along the B-458 fissure on the 400L, showing continuous mineralization with the A433/435 stope but under stronger structural control.
  - Characteristics: Partially oxidized zone, with drilling indicating an average of 9.3 opt Ag, 0.9% Pb, and 8.2% Zn.
  - Volume: Estimated at 1,197,000 cu ft based on extensive drilling and historical production data.
- 418D Oxide Shoot
  - Location: Exposed on the 400L within the Defiance mine, between the 418 and 450D crosscuts, and identifiable down to the 1000L. ○ Volume: Estimated at 4,999,689 cu ft.
  - Average sample grades of 10.6 opt Ag and 8.0% Zn,
- A-439 Stope
  - Location: Approximately 350 feet south of the A433/435 stope on the 400L, near the main Radio ore tunnel.
  - Documentation: Characterized by 45 drill holes, spanning multiple levels. ○ Volume: Estimated at 252,000 cu ft.
  - Average sample grades of 11.2 opt Ag, 3.4% Pb, and 2.6% Zn.
- 833/712 Zone
  - Location: Suggested unmined body deeper in the Independence/Essex area, identified by drilling on the 800L and 700L.
  - Mineralization: Notable for a thick mineralized zone encountered by Anaconda's core hole DA-2.
  - Volume: Estimated at 270,000 cu ft.
  - Average sample grades 2.6 opt Ag, 7.4% Pb, and 7.7% Zn.
- 1302 Drift
  - Location: Within the mineralized margin of the Defiance breccia pipe, explored at the 1200 and 1300 levels.
  - Characteristics: High zinc ore body, supported by 25 drill holes showing an increase in Zn:Pb ratio with depth.
  - Volume: Estimated at 303,750.
  - Average sample grades of 4.2 opt Ag, 2.6% Pb, and 7.6% Zn.
- Stope Remnant Blocks
  - General: Smaller blocks ranging in size and located across various levels. ○ Volume: Estimated at 1,033,650 cu ft.
  - Average realized grade from production 6.5 opt Ag, 7.8% Pb, and 7.1% Zn.
- B-458 Fissure (Copper)

- Location: North of the B-458 (Mixed) block ○ Volume: Estimated at 1,575,000 cu ft.
  - Average sample grades 41.0 opt Ag, 4% Pb, 14% Zn, 6% Cu, and 0.9% W

Mineral Resource Classification

Mineral resources are classified as either indicated or inferred. Resources classified as indicated are areas that had considerable analytical data to support the average grades applied to the block, where in areas accessible for visible inspection, and had detailed geologic and mine maps to support the sectional outline. The inferred blocks represent areas where their continuation at the lower levels of the respective mine areas remains uncertain, although inspection of the historical stopes from levels above the area indicate a consistent and through going zone of mineralization.

Mineral Resource Statement

The mineral resource estimate is based on all data obtained as of April 1, 2024. Mineral resources are not mineral reserves and do not have demonstrated economic viability such as diluting materials and allowances for losses that may occur when material is mined or extracted; or modifying factors including but not restricted to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors. Ethos knows of no existing environmental, permitting, legal, title, taxation, socio-economic, or other relevant factors that might materially affect the mineral resource estimate. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated on the basis of limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

TABLE 2 DARWIN PROJECT MINERAL RESOURCE STATEMENT

Block	OX State	Tons	Ag		Pb		Zn		W		Cu	
		(x1000)	opt	oz (x1000)	%	lb. (x1000)	%	lb. (x1000)	%	lb. (x1000)	%	lb. (x1000)
Indicated Resource Blocks												
A433/435	Oxide	180.0	8.5	1,531.8	0.8	2,952.0	6.4	23,076.0	0.6	2,052.0	-	-
B-458 Fissure	Mixed	66.5	9.3	618.5	0.9	1,197.0	8.2	10,906.0	-	-	-	-
418D Oxide	Oxide	30.0	10.6	319.1	-	-	8.0	4,807.5	-	-	-	-
A-439 Stope	Oxide	30.0	11.2	337.0	3.4	2,066.4	2.6	1,538.4	-	-	-	-
Inferred Resource Blocks												
B-458 Fissure	Mixed	33.25	8.5	283.0	0.8	545.3	6.4	4,262.7	-	-	-	-
418D Oxide	Oxide	490.8	10.6	5,220.9	-	-	8.0	78,650.7	-	-	-	-
833/712 Zone	Mixed	30.0	2.6	76.6	7.4	4,428.0	7.7	4,615.2	-	-	-	-
1302 Drift	Mixed	33.75	4.2	140.7	2.6	1,771.2	7.6	5,148.8	-	-	-	-
Remnant Blocks	Mixed	114.85	6.5	746.5	7.8	17,916.6	7.1	16,308.7	-	-	-	-
B-458 Fissure Cu	Mixed	175.0	41.0	7,175.0	4.0	14,000.0	14.0	49,000.0	0.9	3,150.0	6.0	21,000.0

Notes: * "Reasonable prospects for economic extraction" are based on an 8 opt Ag equivalent cut-off grade. The estimation method restricts extrapolation to known mineralized areas within historical mine workings. Ag equivalent is calculated using $25/oz for silver, $0.9/lb. for lead, $1.1/lb. for zinc, and a 94% metallurgical recovery rate. Total costs, including mining, processing, G&A, and shipping and smelter fees, are estimated at $189.50 per ton of ore. Calculations assume use of underground long hole stoping and processing into a flotation concentrate.

Adjacent Properties

The Darwin mine encompasses several historical operations, with the Lucky Jim, Promontory, and Lane mines noted as the other significant producers within the patented claims (Mackevett et al, 1958).

The Lucky Jim mine, positioned in Section 1, Township 19 South, Range 40 East, is approximately 2.25 miles directly north of the Darwin mine camp. This mine produced silver-lead-zinc ore within the northeast trending Lucky Jim fault zone and additional minor veins, intersecting granitic rocks and calc-hornfels. The orebody is accessed via a 320-foot shaft and internal winzes extending to a depth of 820 feet. Historical production records from 1915 to 1925 indicate extraction of over 40,000 tons of predominantly oxidized ore, resulting in approximately 10.2 million pounds of lead, 744,000 ounces of silver, and 457 ounces of gold. The Anaconda company assessed and refurbished the mine in 1948 without resuming production, and Quintana conducted further evaluations in 1987 (DeRuyter, 1987).

Located in Section 30, Township 19 South, Range 41 East, about 1.75 miles southeast of the mine camp, the Promontory mine produced oxidized lead-silver ore from bedding replacement mineralization in limestone. The mine infrastructure includes an incline shaft reaching a vertical depth of 283 feet. Production figures for this site are not specified individually as they were often aggregated with those of the Darwin Silver Co.

The Lane mine, situated in Section 18, Township 19 South, Range 41 East, approximately 1.25 miles east of the mine camp, originated around 1890. The site targeted predominantly oxidized lead-silver-copper minerals located within fissure veins in limestone and calc-silicate hornfels, reaching a depth of 800 feet. Historical operations ceased in 1948 after producing approximately 12,000 tons of ore, which yielded about 1.8 million pounds of lead, 76,000 ounces of silver, 39,000 pounds of copper, and 1,547 ounces of gold. Currently, Project Darwin LLC has no active plans to develop these properties.

Other Relevant Data and Information

This report summarizes all data and information material to the Darwin Project as of April 1, 2024. Ethos knows of no other relevant technical or other data or information that might materially impact the interpretations and conclusions presented herein, nor of any additional information necessary to make the report more understandable or not misleading.

Interpretation and Conclusions

The Darwin Project's exploration activities have been focused on organizing and understanding historical data, including the collection of samples from underground workings to confirm zones of mineralization and identify areas with potential for further development. The project has initiated the digitization of all historical maps and drill logs to integrate them into a 3-dimensional model to enhance exploration planning. A detailed photogrammetric survey of the project area has been completed, and surveys of accessible mine areas are underway, with plans for drilling at least one diamond drill hole targeting the Essex breccia pipe.

The geology of the Darwin Project is well understood, and the appropriate deposit model is being applied for exploration. The conceptual geologic model is sound, aligns with historical drilling results, and indicates that mineralization is essentially open in all directions, suggesting significant potential to expand on the known mineralization. Significant potential exists to increase the known mineral resource with additional drilling, as well as to upgrade existing mineral resource classifications with infill drilling or systematic underground channel sampling. There is also considerable potential to encounter massive skarn and porphyry mineralization within the district. The current mineral resource at the Darwin Project is more than sufficient to warrant continued evaluation of the Project.

The most significant geologic findings suggest that tungsten (W), copper (Cu), and lead-zinc (Pb-Zn) skarn deposits in the Darwin area have distinct origins, indicating separate formation processes. This includes the rare occurrence of Pb-Zn-W skarns, likely formed through tungsten remobilization. The presence of three different igneous systems and diverse calc-silicate/ore assemblages supports this model, confirming multiple independent plutonic-hydrothermal events rather than strata bound contributions.

Typically associated with granite, scheelite skarns require extensive fractional crystallization in moderately deep settings, a condition observed in the Darwin stock. Small copper skarns tend to form near barren granitic stocks in deeper settings, consistent with patterns seen across the western U.S., including eastern California. Pb-Zn skarns in Darwin are found along faults and dikes, unrelated to the larger Darwin stock, possibly associated with unexposed granitic entities indicated by geophysical anomalies. This diversity underscores the presence of multiple, independent skarn-forming hydrothermal systems.

Vein ores may be related to skarn-forming episodes, particularly associated with shallow granite porphyries. Initial ore-forming fluids were likely magmatic, transitioning to mixed origins in later stages influenced by meteoric water cycling. The rare co-occurrence of Pb-Zn and W skarns reflects unique geological conditions and plutonic events, consistent with areas exhibiting contrasting multiphasic plutonic activities.

In the context of mineral exploration, understanding the potential of a property requires the integration of multiple types of geoscience data, including magnetic surveys, gravity surveys, geochemical sampling, and detailed geological mapping. This integration helps form a coherent picture, adding depth to the analysis and assisting in the identification and prioritization of exploration targets. The project maintains an archive of historical documents related to the Anaconda mining operation and subsequent activities, which have contributed to a comprehensive understanding of geology and controls on mineralization.

These documents and data need conversion into a standard digital format on reliable mediums such as databases or cloud storage to allow for quick access, complex analyses, and easier integration of diverse data sets. Upon completion of an integrated data set and 3-dimensional models, the drill core stored underground should be inventoried and relogged. Samples should be analyzed by an independent certified laboratory using industry best practices, including a robust QA/QC program, with sample selection representative of the different ore types and mining zones within the project. The chosen analytical methods should be appropriate for the specific minerals of interest.

Mineral resources at the Darwin Project have been estimated by Project Darwin using a sectional method. This approach leverages all available historical data, including drill results, underground channel sampling, and historical production reports. The efficacy of this historically validated strategy underscores the inherent value of empirical data and operational insights in resource estimation. Despite its departure from modern estimation methods, the historical approach to mineralized block delineation and grade averaging, based on direct mining and exploration data, provides a credible and pragmatically grounded basis for the reliability of these estimates in understanding underground mine resources.

Risks and Uncertainties

In preparing this report, Ethos has undertaken extensive efforts to validate the historical data pertaining to the estimated mineral resources. This validation process was guided by rigorous methodologies and leveraged contemporary technologies and expert insights to ensure the highest possible accuracy and reliability of the historical data for analysis. Despite these thorough efforts, it is important to acknowledge that certain inherent risks and uncertainties remain when utilizing historical data for resource estimation. In general, the use of historical data needs consider the following risks and uncertainties:

- The nature of historical data—often collected with older methods, potentially under different regulatory standards, and stored in formats vulnerable to degradation—presents challenges that cannot always be fully mitigated. Although we have made significant adjustments and enhancements to this data, the potential for residual errors stemming from factors such as incomplete documentation, technological evolution, and methodological discrepancies between past and present practices cannot be entirely eliminated.
- Historical data collection and methods are not documented and may have been collected with methods that do not meet current industry standards. The accuracy, precision, and completeness of such data can be questionable, which can lead to errors in estimating the size, grade, and extent of a mineral deposit.

- The historical records do not include comprehensive documentation on how data was collected, processed, and interpreted. This lack of detailed records makes it difficult to assess the reliability of the data..
- Changes in technology and methodology over time mean that historical data might not be directly comparable with data collected using modern techniques. For instance, changes in drilling techniques, analytical methods (e.g., for assaying), and resource modeling methods can create discrepancies.
- Historical sampling might not have covered the deposit comprehensively or systematically, leading to potential bias in the results. Additionally, assay techniques from earlier periods might not have the same detection limits or accuracy as modern methods, affecting grade estimates.
- The standards for reporting mineral resources, such as those outlined by the JORC Code or NI 43-101, have evolved significantly. Historical data might not comply with these current standards, making it challenging to use this data in public reports without substantial reevaluation.
- Physical degradation of historical records, whether paper-based or early digital formats, can lead to loss of data. Moreover, accessing data that is stored in obsolete digital formats or physical locations can be difficult and costly.

- Historical spatial data might be inaccurate or based on surveying methods that lack the precision of modern GPS and GIS technologies. This can lead to errors in locating drill holes, trenches, and other geological features on current maps.

Despite these challenges, it's crucial to note that the historical data supports a long-standing mining operation, and the reported results align with the historical production records. This consistency provides a measure of validation and reliability to the historical data, suggesting that while there are inherent risks in its use, these risks are mitigated by the practical outcomes observed over the years of operation.

Recommendations

As the most significant risk to the mineral resource estimation resides with the use of historical data with limited confirmation data collected by the current operator, the focus needs to involve a more robust validation of this data.

To optimize exploration and decision-making processes, it is recommended that Project Darwin undertake the digitization of its extensive archive of historical documents related to the Anaconda mining operation and subsequent stakeholders. The archival materials, which include geologic maps, drillhole data, underground samples, production records, and surface exploration results, should be converted into a standardized digital format. This conversion should utilize reliable digital storage mediums such as databases or cloud storage solutions to ensure quick access, facilitate complex analyses, and support the seamless integration of diverse geological, geophysical, and geochemical data sets.

The digitization of these historical documents is crucial for constructing an accurate and comprehensive geological model. This model will serve as a foundational tool in supporting subsequent phases of mineral estimation and strategic exploration efforts. By implementing this recommendation, Project Darwin will enhance its ability to leverage historical data effectively, thereby improving the accuracy and efficiency of its resource management and exploration activities.

- To align historical data with current industry standards, a meticulous validation process is necessary. This process may encompass several critical actions, depending on the availability and condition of the historical data. The steps involved typically include:
- Re-locating Historical Data in the Field: This involves confirming the locations and details of historical geological features, drill hole collars, and sampling sites using modern GPS and GIS technologies to ensure spatial accuracy.
- Resampling and Re-assaying Historical Drill Core or Drilling Samples: Extract new samples from stored drill cores or from the field where historical samples were initially taken. These samples should be analyzed using current analytical methods to verify or refine the historical data.
- Use of Certified Reference Materials in Laboratory Analysis: Submit samples to a laboratory that uses certified reference materials (CRMs) for quality control. This ensures that the assay results are reliable and meet current industry standards.
- Supporting Historical Drill Hole Data with New Drilling: Augment historical drill hole data with new drilling campaigns. New drill holes should be strategically located to confirm and expand upon the historical data, providing a contemporary perspective on the mineralization.

Definition of “Historical Data”: For the purposes of this validation, "historical" refers to data that was acquired by previous operators of the mineral property. This data may have been collected under different technological, methodological, and regulatory conditions than those currently in effect.

By following these steps, the historical data can be effectively integrated into current exploration and development plans, enhancing the accuracy of resource estimates, and ensuring compliance with modern technical and regulatory standards.

The validation of the historical data should be done in conjunction with exploration targeting the areas outlined in the mineral resource estimates. Analysis of the data using a methodology appropriate for the base metals, tungsten, and the REEs associated with deposit.

After each phase of validation and exploration the models need to be updated with the new data and adjusted to guide the next phase.

References

Brown H. G, 2002, Darwin Mine, Inyo County, California, 7/23/02 (probably prepared for Project Darwin Inc.)

Chen, J. H.–Y., 1977, Uranium-lead isotopic ages from the southern Sierra Nevada batholith, and adjacent areas, California: PhD thesis, Univ. Calif. Santa Barbara.

Dunne, G. C., Gulliver, R. M., and Sylvester, A. G., 1978, Mesozoic evolution of rocks of the White, Inyo and Slate Ranges, eastern California, in Howell, D. G. and McDougall, K. A., eds., Mesozoic paleogeography of the western United States: Soc. Econ. Paleontologists and Mineralogists.

Kelley, v.c., December 1937, Origin of the Darwin Silver-Lead Deposits. Economic Geology, v. 32, No.8, p. 987-1008.

Newberry, R, 1987, Use of intrusive and calc-silicate compositional data to distinguish contrasting skarn types in the Darwin polymetallic skarn district, California, USA: Mineralium Deposita, Springer-Verlag, vol. 2, p 207-215.

Newberry, RJ., June 10, 1980, Progress report, Darwin Project (unpublished), 8 p.

Newberry, R.J., July 1981, Progress report, Darwin District (unpublished), 8 p.

Newberry, R.J., January 1982, Progress report, Darwin District (unpublished), 16 p.

Newberry, R.J., Spring 1982, Progress report, Darwin Mines (unpublished), lOp.

Rhys, D. A., Lewis, P. D., & Rowland, J. V. (n.d.). Structural controls on ore localization in epithermal goldsilver deposits: A mineral systems approach. Panterra Geoservices Inc.; Eldorado Gold Corporation.

Appendix A: Patented Lode Claims

Claim Name	Patent Number	Mineral Survey Number	Claimant
Grand View No. 2	4680210	6455	Darwin Project, LLC.
Grand View No. 1	4680210	6455	Darwin Project, LLC.
Grand View No. 3	4680210	6455	Darwin Project, LLC.
Ophir Hill	4680210	6455	Darwin Project, LLC.
Imperial Fraction No. 1	4680210	6455	Darwin Project, LLC.
Ophir Hill No. 4	4680210	6455	Darwin Project, LLC.
Ophir Hill No. 3	4680210	6455	Darwin Project, LLC.
Imperial Fraction No. 2	4680210	6455	Darwin Project, LLC.
Ophir Hill No. 2	4680210	6455	Darwin Project, LLC.
Reddy Millsite	1064099	6082	Darwin Project, LLC.
Last Chance	1055604	6051	Darwin Project, LLC.
Major Butt	1055605	6051	Darwin Project, LLC.
Rip Van Winkle No. 4	1055605	6050	Darwin Project, LLC.
Rip Van Winkle No. 3	1055605	6050	Darwin Project, LLC.
Rip Van Winkle No. 1	1055605	6050	Darwin Project, LLC.
Rip Van Winkle No. 2	1055605	6050	Darwin Project, LLC.
Vivian No. 1	1055605	6050	Darwin Project, LLC.
Big Ben	1055605	6050	Darwin Project, LLC.
Essex No. 1	1055605	6050	Darwin Project, LLC.
Essex No. 2	1055605	6050	Darwin Project, LLC.
Kerso	1055605	6050	Darwin Project, LLC.
April Fool No. 1	1055603	6047	Darwin Project, LLC.
April Fool	1055603	6047	Darwin Project, LLC.
April Fool No. 3	1055603	6047	Darwin Project, LLC.
Esperanza	1055603	6047	Darwin Project, LLC.
Esperanza No. 1	1055603	6047	Darwin Project, LLC.
June Bug	1055603	6047	Darwin Project, LLC.
Columbia	Not Available	6046	Darwin Project, LLC.
Carbonate	855888	5439	Darwin Project, LLC.
Lucky Bill	855888	5439	Darwin Project, LLC.
Wedge	855888	5439	Darwin Project, LLC.
Liberty	855888	5439	Darwin Project, LLC.
10	855888	5439	Darwin Project, LLC.
Last Chance No. 3	1055604	5439	Darwin Project, LLC.
New York	779068	5362	Darwin Project, LLC.
Michigan	779068	5362	Darwin Project, LLC.
Rayman	779068	5362	Darwin Project, LLC.
Virginia	779068	5362	Darwin Project, LLC.
Lone Star	779068	5362	Darwin Project, LLC.
Ohio	779068	5362	Darwin Project, LLC.
Lucky Jim	779068	5362	Darwin Project, LLC.
Vermont	779068	5362	Darwin Project, LLC.

Anthony	779068	5362	Darwin Project, LLC.
Lucky Jim Ext.	779068	5362	Darwin Project, LLC.
Defender	779068	5362	Darwin Project, LLC.
Thompson No. 2	779068	5362	Darwin Project, LLC.
Thompson No. 1	779068	5362	Darwin Project, LLC.
Thompson No. 4	779068	5362	Darwin Project, LLC.
Claim Name	Patent Number	Mineral Survey Number	Claimant
Thompson No. 3	779068	5362	Darwin Project, LLC.
Lead No. 1	779068	5362	Darwin Project, LLC.
Thompson No. 5	779068	5362	Darwin Project, LLC.
Lead No. 2	779068	5362	Darwin Project, LLC.
South West Mine	27816	3342	Darwin Project, LLC.
Last Chance No. 2	27815	3341	Darwin Project, LLC.
Driver	29613	44	Darwin Project, LLC.
Independence Ledge	30003	43	Darwin Project, LLC.
Bernon	20314	42	Darwin Project, LLC.
Buena Ventura & the Defiance	19500	38	Darwin Project, LLC.

5 Peter Hahn Technical Report w/ Graphics

TECHNICAL REPORT ON THE DARWIN MINE

INYO COUNTY

CALIFORNIA

WITH Graphic

Prepared for

PROJECT DARWIN LLC

By

PETER H. HAHN, CPG
1536 Brigadoon Park Drive
West Jordan, Utah 84088 (801)
664-2186 pands0710@infowest.com

July 26, 2010

Table of Contents

Summary

This Technical Report concerns the historic Darwin lead-silver-zinc mine located between Lone Pine and Death Valley in Inyo County, California. The mines that now comprise the Darwin mine workings have been worked since about 1870, and have a total recorded production of about 10.5 million ounces of silver, 840,300 tons of lead, 57,900 tons of zinc, 745 tons of copper and 5,900 ounces of gold. Anaconda Copper Company operated the mine and associate mill (since removed) continuously from 1945 to 1957, the period of greatest production. The property consists of 58 patented mining claims, about 1,050 acres, intact mine camp, and many miles of underground workings.

Base metal — silver deposits at Darwin occur in Upper Paleozoic carbonate rocks adjacent to a complex granodiorite stock. They are partially oxidized, and comprise fissure veins, Skarn breccia pipes centered by later quartz-porphyry intrusives, and complex, structurally controlled skarns.

Several geologic exploration targets for deep zinc-copper (+ lead-silver) skarn deposits are well documented, and ore grade material is in sight in numerous areas in the mine. A large body of engineering, geological and production data and maps is preserved in the mine . Excellent access is available through the +8,000 ft. long Radio ore.

Tunnel (400 Level) but there is no current usable to deeper levels. Hoists in the Defiance (to 1,300 Level) and Thompson (to 900 Level) are present but not currently powered or licensed.

It is recommended to cull, organize, preserve and digitize the useful geological data and maps, reactivate the two main shafts, and continue development of visible ore in the mine. More claims should be staked for protection and to cover potential exploration targets. Longer term goals are to explore several geologically favorable areas near the workings, and ultimately the deep Skarn targets.

Introduction

This report is prepared at the request of Mr. Jack E. Stone, President of Project Darwin LLC, for the purpose of bringing together data on the history, geology, historic production, metal resources, potential for future production and geologic exploration potential of the Darwin Mine. Information has been gathered by the author during several phases of investigation of the Darwin properties:

1. Extensive on-site investigations as Consulting Geologist for Western Zinc JV (Cyprus Exploration Company/Mitsui Metals) from August through December, 1991, during which the accessible portions of the mine were examined, surface skarn breccia exposures were mapped and sampled, some core was relogged, very extensive files left by Anaconda and subsequent operators were organized and reviewed, new concepts of genesis of the deposits were explored and confirmed, and recommendations made for exploration of deep zinc skarn targets; 2. Review of the files, which had been moved to Tonopah, Nevada, and field examination and sampling of areas on the 400 level of the mine and several other areas both on and near the Darwin property for tungsten potential, from June to October, 2007 as Chief Geologist for Galway Resources, Inc.; and
3. Review at Darwin of the relocated file and map archive and examination of several previously unseen mineral occurrences on the 400 level in July, 2010 as Consultant to Project Darwin LLC.

All in all, the author has spent approximately 35 days on-site at Darwin directly involved with Darwin mine investigations, and nearly as much time in omce review and reporting, since mid-1991. Much of the basic data compilation maps and ledgers for zinc potential done in 1991 has been lost, but monthly and progress reports, plus schematic long- and cross-sections which illustrate the deep geologic exploration targets are preserved.

This report is presented as closely as possible in the format prescribed for Technical Reports by the Canadian Securities Administrators under National Instrument 43-101 , although there are certain portions which cannot be certified as compliant by the author as a Qualified Person. These portions particularly include statements regarding mineral resources, which are entirely historical, probably incomplete because of missing documents and erratic reporting by operators subsequent to Anaconda, ore mined and resources not replaced by leasers, and reserve blocks poorly described as to location with respect to known locations underground (i.e. certain drill hole and stope locations, etc.)

Reliance On Other Experts

Except for the author's personal observations on the surface and underground, sampling, mapping, and description of potential geologic exploration targets, all the information presented herein is based on examination of published and unpublished sources, and review of and projection from historical data, together with interviews of several persons with long-time experience and familiarity with the Darwin mine. These persons include Jack Stone; Paul Skinner, who managed the mill for West Hills

Exploration Company and is currently setting up an analytical facility at the mine;

Dudley Davis, long-time Chief Geologist for Anaconda at Darwin; Rainer Newberry, consultant to Anaconda and currently Professor at University of Alaska; Rob Wetzel, geologic consultant to Blue Range Mining Company; Jerry Carr, I andman, who prepared a Preliminary Title History for Galway Resources Inc. in 2007; and Adena Fansler, Planner with the Inyo County Planning Department.

The author believes that these sources and persons have presented information factually, truthfully and without bias, and has no reason to doubt the applicability of information, ideas and recommendations presented. The Anaconda Copper Mining

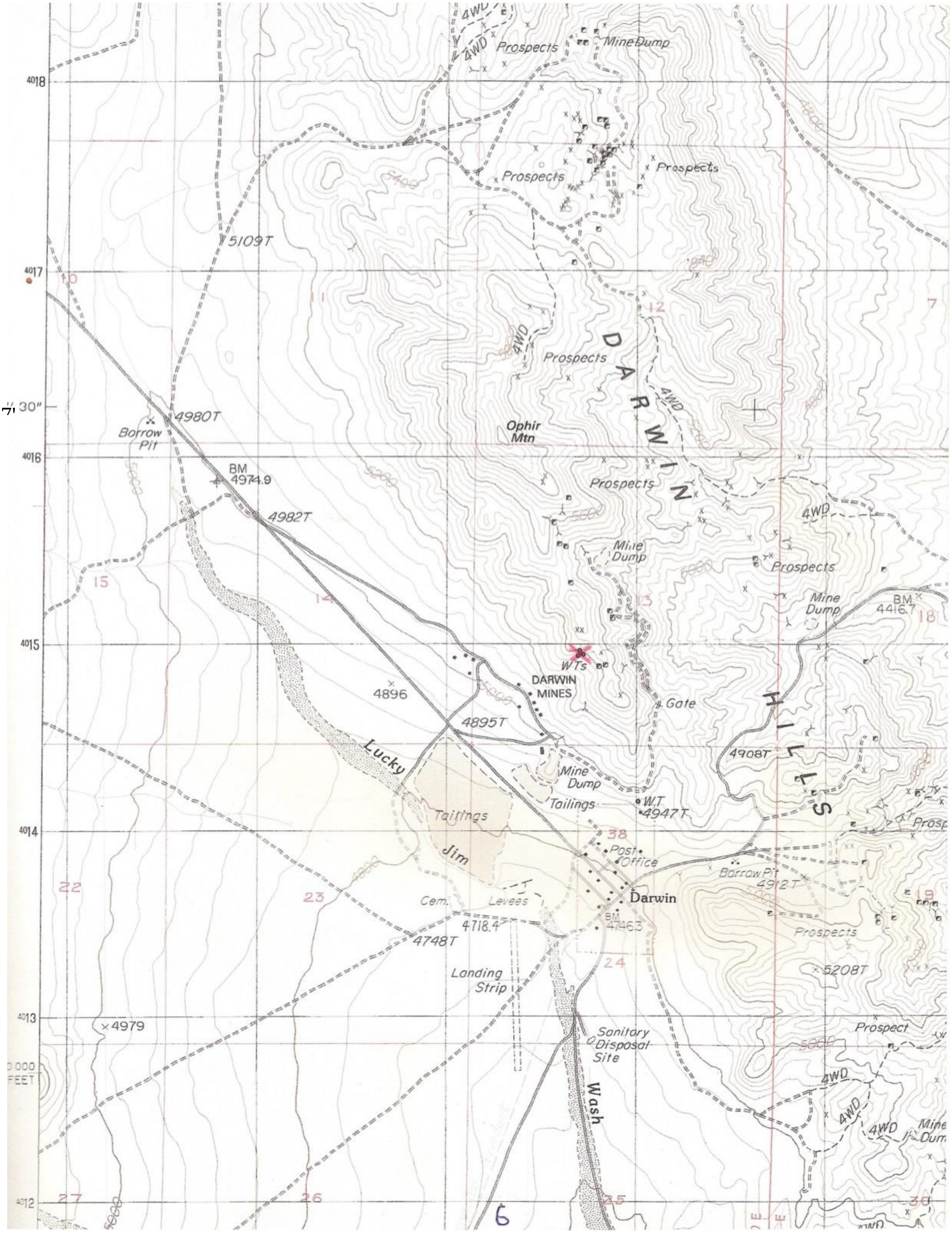
Company, operator of the mine during the period of greatest production from 1945 to 1957, is recognized within the industry for having maintained very high standards of geologic mapping and analysis, and great reliance has been placed on the excellent surface and underground maps, drill logs, assays, and other data which were left at the Darwin mine offce. Anaconda kept the property until 1985, although the mine or portions of it were leased to other operators from 1967 to 1972 It is believed that subsequent operators maintained the underground mapping, although authorship of later mapping is not indicated on the maps.

Two anonymous reports in the Darwin mine offce have also been drawn upon, as the contents agree with and are largely compiled from other sources. They are dated January, 1990, and July 23, 2002, and were probably prepared by Blue Range Mining Co.

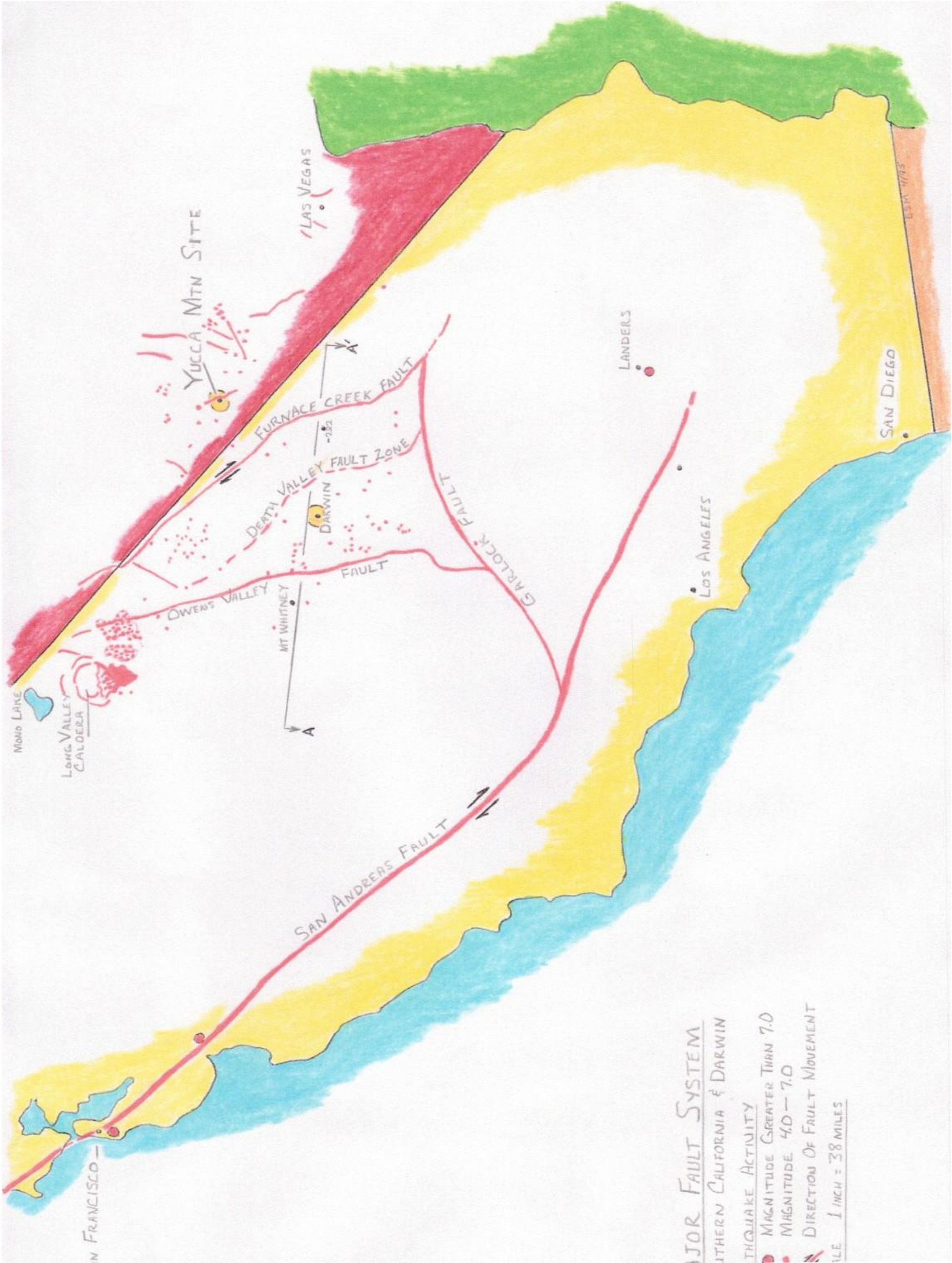
and by H. G. Brown, consultant to Project Darwin, respectively.

Property Description and Location

The Project Darwin LLC holdings consist of 58 patented mining claims, total about 1,050 acres; one patented mill site claim, about 5.2 acres; a lease (PRC 4627.2) from the California State Lands Commission for water/power line right-of-way, 12.97 acres; and a Right-of-Way lease CA-8872) from the Bureau of Land Management, for water/power line across about 13,200 feet of BLM luld. Also included are Water Right Permits 1086 (1957) and 19497 (1985) for 0.32 cubic feet per second (144 gallons per



4WD
Prospects
Mine Dump
4018
Prospects
5109T
4017
10
11
12
7
4WD
Prospects
DARWIN
30″
4980T
Borrow Pit
Ophir Mtn
4016
BM 4974.9
5200
Prospects
4982T
4WD
Mine Dump
15
Prospects
14
13
Mine Dump
BM 4416.7
18
4015
WTs
4896
DARWIN MINES
Gate
HILLS
4895T
Lucky
4908T
Mine Dump
Tailings
WT
4947T
Tailings
4014
38
Post Office
Jim
Borrow Pit
4912T
22
23
Cem.
Levees
Darwin
19
BM 4746.3
4718.4
4748T
24
Prospects
5208T
Landing Strip
4013
4979
Sanitary Disposal Site
Prospect
4WD
Wash
4WD
4WD
Mine Dum
4012
27
26
25
30


Yucca Mtn Site
Las Vegas
Furnace Creek Fault
Death Valley Fault Zone
Darwin
Owens Valley Fault
Mt Whitney
Garlock Fault
Landers
Los Angeles
San Diego
San Andreas Fault
Mono Lake
Long Valley Caldera
Francisco
A
A'
JOR FAULT SYSTEM
THERN CALIFORNIA & DARWIN
THQUAKE ACTIVITY
MAGNITUDE GREATER THAN 7.0
MAGNITUDE 4.0 – 7.0
DIRECTION OF FAULT MOVEMENT
1 INCH = 38 MILES

minute) to be diverted from a well in Darwin Wash, three miles east of the mine office. The patented claims are located in Secs. 1, 2, 11, 12, 13, 14, T. 19 S., R. 40 E. (the principal Darwin mine group of41 claims); Secs. 18 (Lane Mill group, 5 claims), 30

(Promontory/April Fool group, 11 claims), and 32 (Columbia, I claim), T. 19 S., R. 41 E. The water well is in the center of Sec. 16, T. 19 S., R. 41 E. (a California State School Section), all on Mount Diablo Meridian.

Improvements on the property include access roads, very extensive underground mine workings, and several serviceable structures which remain from the Anaconda and other operations and are currently in use by Project Darwin LLC, many semi-derelict structures some of which could be made useful, water tanks and water system, and active telephone and power distribution systems. Buildings in use include a large residence, mine omce with three fire-resistant walk-in vaults, warehouses, shop building with machinery and three large IR Imperial air compressors (no motors), and dry and shop buildings at the 400 Level portal. Also on site in the mine office is a very large collection of engineering, geological and production files and maps compiled by Anaconda and subsequent operators.

There is no mill or ore processing facility on site, the 500 TPD flotation mill having been tom down and the site reclaimed several years ago. The tailings pond and reclaimed heap leach pad west of the Darwin road are not on the property.

Annual property tax payable to Inyo County for FY 2009-10 is $2,639.50.

The mine carries a 5% Net Smelter Return royalty on newly mined ore; the royalty interest is currently by Franco NevÜ Corporation. Mr. Stone has entered into negotiations with Franco Nevada for reduction of the royalty and possible purchase of part of the royalty interest.

Accessibility. Climate. Local Resources. Infrastructure and Physiography

The community of Darwin and the Darwin mine are located in Inyo County, California, about 36 miles southeast of Lone Pine. Access is via California State Highway 136/190 (the Death Valley Highway), which joins US 395 a mile south of Lone Pine. The paved Darwin road turns south from 190 about 32 miles from the Highway

395 junction, and runs south 5.6 miles to the mine site. The last half mile to the mine facilities is graded.

Elevation on the properties ranges from 3,600 feet at the well site in Darwin Wash to 6,000 feet at Ophir Mountain, a peak on the ridge in which the Darwin deposits are located. Elevation at the portal of the Radio ore Tunnel / 400Level, the main haulage access to the mine, is about 4,890 feet. Climate is typical of the high desert of southeastern California; daytime summer temperatures can exceed 1 OOF, and in the winter it commonly falls below freezing at night. Average annual rainfall is about four inches.

Power is supplied to the mine site by Los Angeles Department of Water and

Power on a 3-phase power line from Olancha at 12,500 volts, capacity 2,500 kw. The transformers for 3-phase are on site but not hooked up. The power rate for industrial use is 6.5 cents per kWh. Water is currently trucked from the well in Darwin Wash, as both the pipeline and power line between mine site and well are derelict. Poles and equipment are on hand at the mine to rebuild the power line. Water must be pumped from the well against a head of about 1,800 feet (approx. 780 psi static head) to surmount Darwin Ridge above the water tanks. Two tanks above the mine office are in use with a capacity of 38,000 gallons, and water is distributed by gravity to the plant area. Elsewhere on site, there is water storage capacity of 195,000 gallons in several tanks and a swimming pool.

A small labor pool is available from Lone Pine and elsewhere in the Owens Valley, and there may be a limited number of experienced miners available from the Bishop area, where the large underground Pine Creek mine, formerly operated by Union Carbide, closed a few years ago. Bishop is 59 miles north of Lone Pine on Highway 395.

History

Mining in the Darwin area (New Coso Mining District) began before 1870 when Mexican miners began producing high grade silver ore. In 1870, the Mexicans were driven out and lead-silver ore was produced from the Lucky Jim, Promontory, Columbia, Lane, Defiance, Thompson and Essex mines; all these mines are included in the Project Darwin LLC patented claims and the latter three are incorporated in the present Darwin mine workings. These mines, and up to five small smelters, operated until 1893. The district then was dormant until 1916.

The Darwin Development Company began consolidating properties in the district in 1916; operations in succeeding years were marked by poor management, litigation, and financial difficulties. The Darwin Silver Company and later the Darwin Lead Company were formed to operate the mines.

Anaconda reviewed the operations of the Darwin Lead Company in 1939 and estimated that approximately 200,000 tons of lead-silver ore had been produced to that time by mines controlled by Darwin Development. Early on, the mines had produced high-grade lead-silver ore at a profit, but in later years dependence on lower grades and oxidized ore, together with recoveries at times as low as 35% and over-expenditure on surface plant made the operation less profitable. Anaconda concluded that as much as 100,000 tons of oxidized silver-lead ore might be developed, but that the properties could not be worked profitably at 1939 prices unless zinc, then avoided or going to tailings, could be recovered. Metallurgical changes just prior to 1939 had increased overall recovery to about 70%

Anaconda considered the Darwin Lead Company a second time in 1942-44. Production was intermittent until 1944, when the mill ran all year, treating 90 TPD of mixed oxide-sulfide ore, and shipping 35 tons of crude ore per day to the U.S. Smelting plant in Midvale, Utah. The direct-ship ore ran about 8 opt silver and 13% lead, while mill ore ran 3.26 opt silver, 3.74% lead and 4.86% zinc. It was estimated that total district production 1870 through 1944 was about $6 million in silver, lead and zinc. It was estimated (by Blue Range Mining Company, a later owner) that the total pre Anaconda ore production was about 250,000 tons.

Two new high grade bodies, along the Essex vein and on the downward extension of the Defiance zone to the Radio ore/400L area — recognition of the near-vertical plunge of the rich Skarn breccia "pipes" — made the district attractive to Anaconda. The Essex ore was running 60 opt silver and 31 % lead, with very low zinc, and Radio ore sulfide ore ran

10.5 opt silver, 16.2% lead and 12.2% zinc.

Anaconda completed purchase of the Darwin mine on August 1, 1945. They completed driving the Radio ore Tunnel on the 400 Level, which connects all the workings in the mine and through which ore could be delivered directly to the enlarged 500 TPD flotation mill (350 tons sulfide, 150 tons oxide) at the portal. This tunnel is at least 8,000 feet long and services the Defiance workings, with an internal shaft which now extends to the 1300 Level, and the Thompson-Essex-Independence workings, including a winze sunk to the 900 Level. At some time during the Anaconda mining years, long core holes were drilled from the 400 Level to penetrate the Darwin stock in a test for porphyry copper potential, but this effort was unsuccessful.

Anaconda operated the mine and mill continuously until milling lead and zinc prices forced shutdown on June I, 1957. Total Anaconda production is estimated at about 962,000 tons, combined oxide-sulfide ore. The sulfide ore mined since 1942 is reported to have averaged 6% lead, 6% zinc, and 6 opt silver (Hall & Mackevett, 1962.) "Reasonably Assured Ore" reserve at shutdown was estimated at about 155,000 tons, 4.9 opt silver, 6.1% lead, 8.5% zinc (Skiles, 1957)..

The mine was idle until 1967 when West Hills Exploration Co. leased the mine, deepened both the Defiance and Thompson shafts, and produced about 271 ,OOO tons of ore from several deeper levels in the mine, but gave up the lease due to "poor operating results" (Cormie, 1970). The mine was leased to Mexicanus Colorado, Inc., in April, 1971. This company invested several hundred thousand dollars in mine and mill improvements, but operated less than a year, reportedly producing about 74,000 tons or lead, silver and tungsten ore. During this period they subleased portions of the mine to Montecito Mining Company, who produced tungsten ore from the 433435 stope, and lead silver ore from the deeper Thompson workings Secundo Guasti leased the tailings and part of the Thompson workings from Mexicanus Colorado in 1972 for tungsten, but his operation was unsuccessful. Other leasers, principally mining tungsten ore from the 433-435 stope, included Pacific Tungsten, West End Consolidated and Brownstone Mining Company. All the leases were terminated in 1976.

Anaconda correspondence (G. A. Barber to R. L. Knight, 6/28/1977, in the Darwin office archive) states "Anaconda has not profited from any of the associations [with leasers] to date.. . "

Anaconda continued to hold the property, and began studies toward heap leach recovery of silver values in the tailings. They estimated that 2.3 million ounces of silver remaining in the tailings (1.69 million tons at 1.38 opt) by cyaniding and Merrill-Crowe recovery. A test heap was initiated in 1981-82, which was successful metallurgically, but low silver prices ended the operation. The heap has been reclaimed and covered.

In 1980, Anaconda began a detailed geological and geochemical re-evaluation of the mine and the district, employing Rainer Newberry, at that time a PhD candidate and Marco Einaudi's student at Stanford University, and new Anaconda geologists, principally Geoffrey Wilson. The progress and conclusions of this program are summarized in several internal reports and in two publications, Newberry (1987) and Newberry et al (1991) and are summarized in Section I I, "Mineralization." This intensive restudy of the geology and metal zoning culminated in the drilling of two deep angle core holes DA-I, 2,201 ft. TD and DA-2, 2,302 ft. TD, in 1982. Both holes were designed to test for deep bedding replacement and porphyry-skarn mineralization, particularly previously unrecognized tungsten zones. Both holes cut significant ore and near-ore grade silver and zinc mineralization at depth.

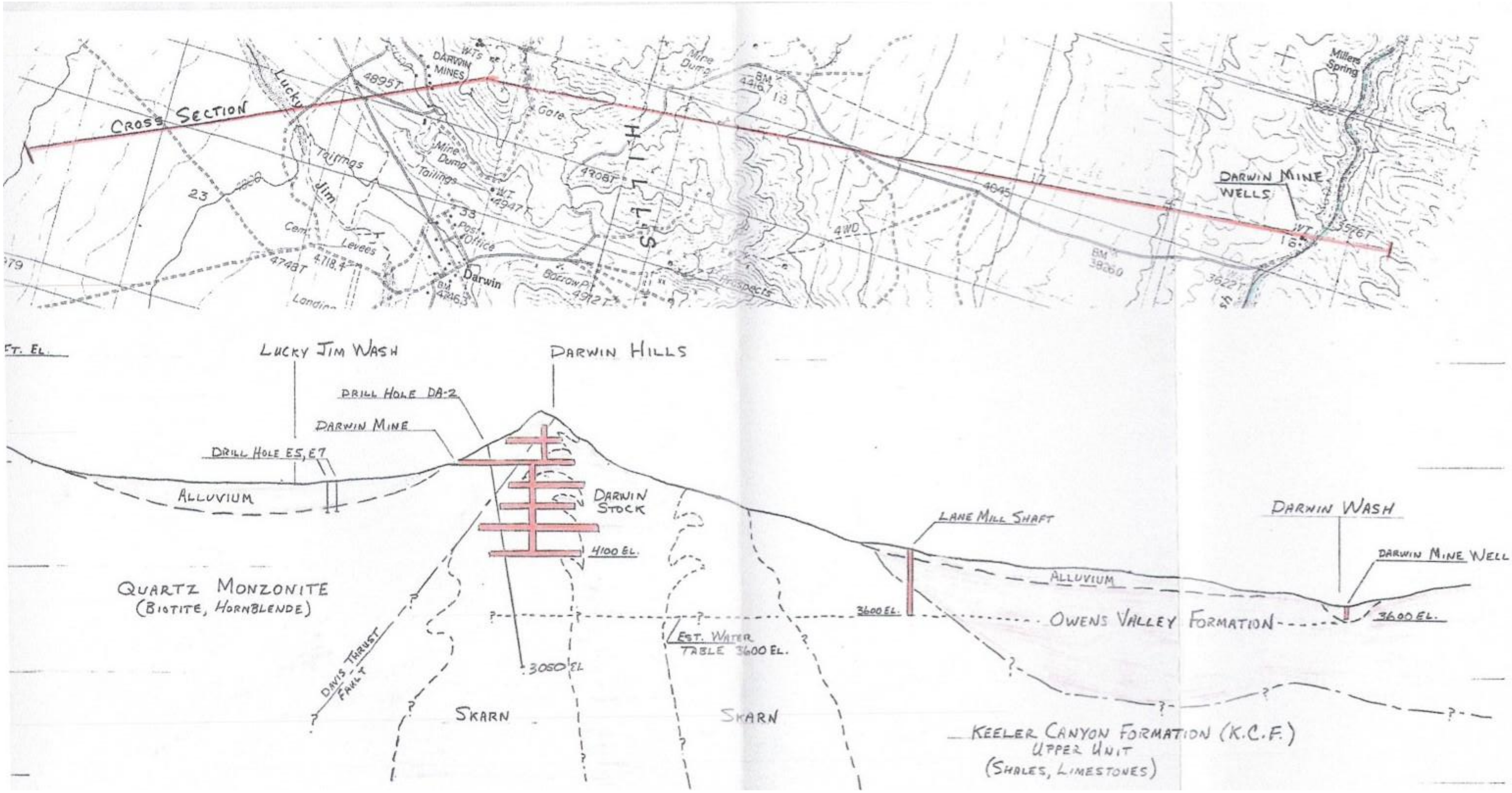
In 1983, Anaconda reviewed gold potential on the nearby Darwin Antimony mine and in the Lucky Jim mine area, without encouragement.

By 1984, Anaconda's acquisition by Atlantic Richfield Corporation was completed, and the Darwin property was sold to Quintana Minerals Corp. Anaconda retained a 5% net smelter return royalty on newly mined ore, and a sliding scale royalty on silver which might be produced from the tailings. Although Quintana drilled eight shallow percussion holes around the Essex shaft, investigated silver and gold possibilities (DeRuyter, 1986, 1987, 1988) and invited Cyprus Minerals Company (Seklemian, 1987) and perhaps others to consider bulk minable precious metal potential, no significant physical work was done. Quintana sold the property to Blue Range Mining Company in

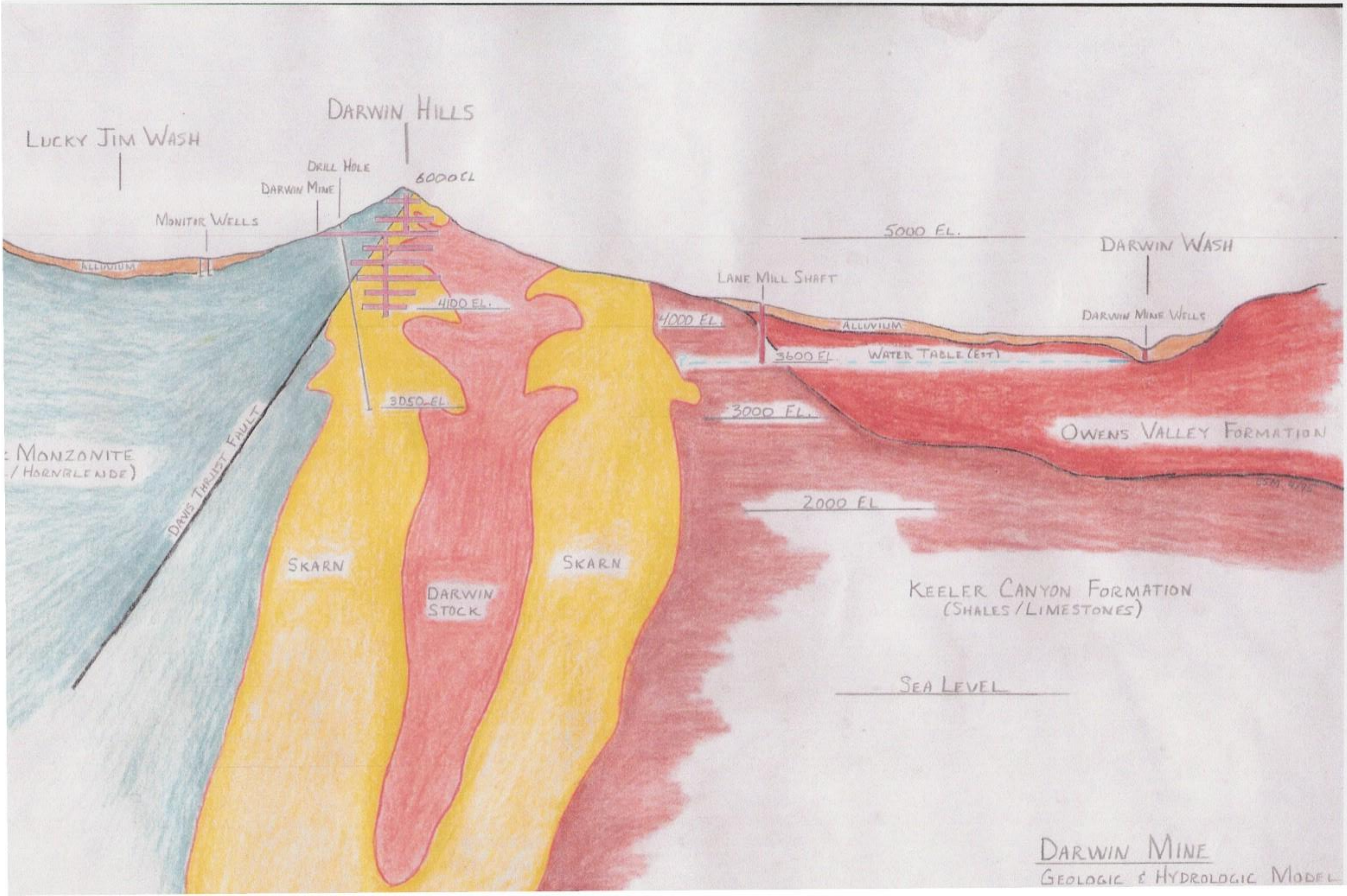
1989.

Work done by Blue Range included surface and underground mapping and sampling, long hole drilling, and upgrading of the underground utilities and mill facilities. By October 1989 Blue Range estimated that they had added about 30,000 tons of possible ore, resulting in perhaps 90,000 tons reserve at that time.

Early in 1991, Blue Range leased the property to Western Zinc Joint Venture (Cyprus-Mitsui) based on Cyprus geologist Chris Torrey's recognition of good potential for high-grade bedded replacement and skarn zinc mineralization which had been missed.



DARWIN MINE
GEOLOGIC & HYDROLOGIC MOD
VERTICAL SCALE 1" = 1000'
ESM 4/29/93
MRG 4/29/93
BRM/B



Darwin Hills
Lucky Jim Wash
Drill Hole
Darwin Mine
6000 EL
Monitor Wells
Alluvium
5000 EL.
Darwin Wash
Lane Mill Shaft
4100 EL.
4000 EL.
Alluvium
Darwin Mine Wells
3600 EL.
Water Table (Est)
3050 EL
3000 EL.
Owens Valley Formation
Monzonite
/Hornblende)
Davis Thrust Fault
2000 EL
Skarn
Darwin Stock
Skarn
Keeler Canyon Formation
(Shales/Limestones)
Sea Level
Darwin Mine
Geologic & Hydrologic Model

or avoided by Anaconda. Target for the JV was discovery of 10 million tons of 8-10% zinc (Torrey, 1991). The author of this report, as consultant to Cyprus, spent portions of four months working in the Anaconda mine office at Darwin, organizing and reviewing underground mapping, several hundred drill logs, and metal production reports; the data was plotted on Anaconda's 100-scale level maps, with particular attention to the zinc/lead ratio and incorporating metal zoning data from Newberry et al (1991). It was confirmed that Anaconda and others systematically avoided areas of high zinc drill intercepts, but without substantial lead and silver. Other work on site included surface mapping and sampling of several skarn pipe outcrops, and relogs of portions of the deep Anaconda holes. This project culminated in recommendations to the JV for several more deep drill holes to test further the roots of the Defiance and Essex breccias, which control the highest-grade replacement deposits. Early in 1992, Mitsui pulled out of the JV, and the property was returned to Blue Range.

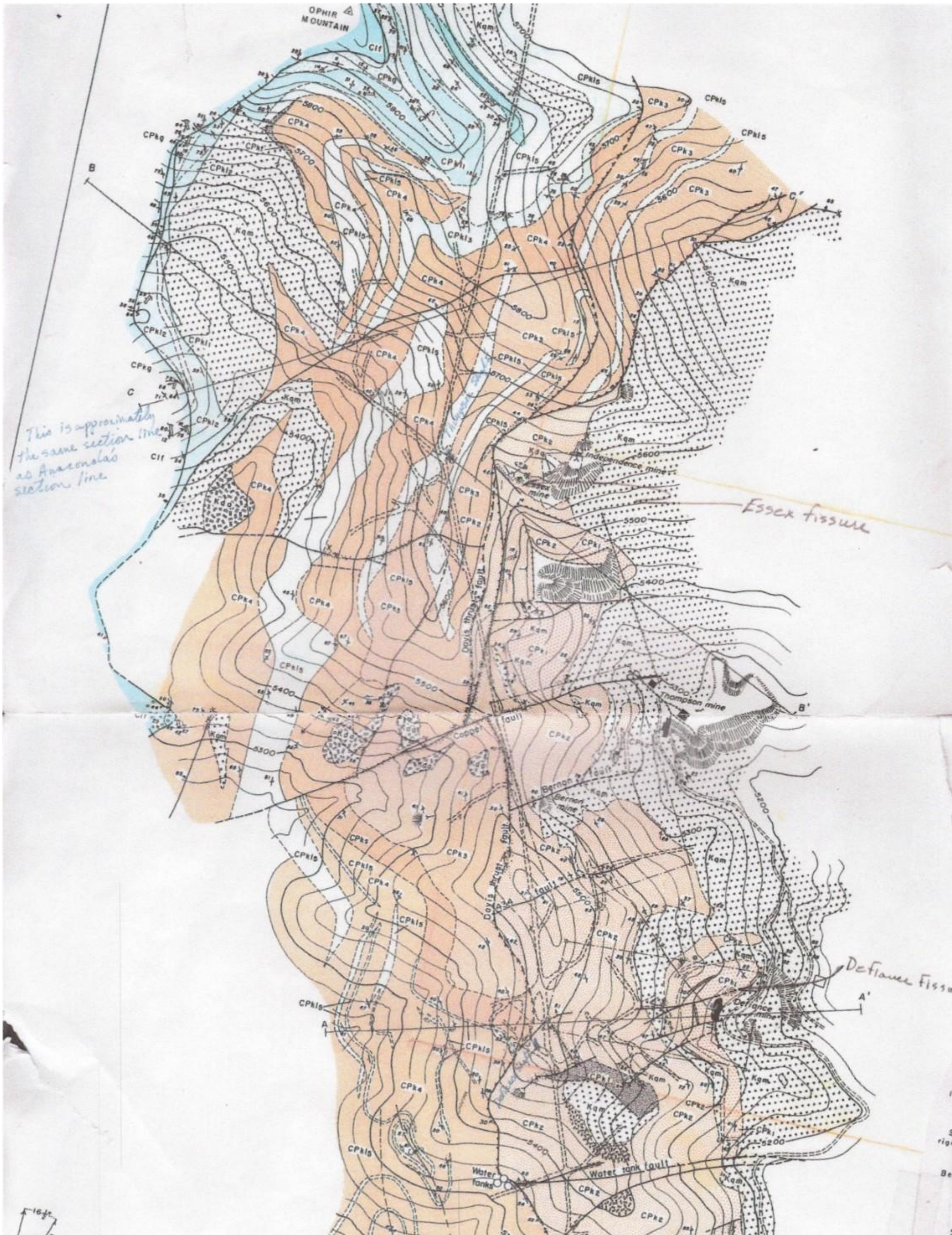
In May 1995, Blue Range sold the Darwin property to Project Darwin, Inc., which changed to Project Darwin LLC in 2006. Jack and Linda Stone, as Project Darwin LLC, acquired full ownership in June 2007.

During Project Darwin LLC's ownership, several groups, not all known to this author, have investigated the potential of the Darwin mine complex. In 2006, this author, then Chief Geologist for Galway Resources Inc., was contacted by Mr. Stone regarding the tungsten potential. Galway management became interested both for the tungsten and because of the deep zinc skarn targets proposed for the Cyprus group. Negotiations for participation in the property by Galway were unsuccessful.

Most recently, underground sampling in 2009 and 2010 on the 400 Level in the Defiance Pipe and a nearby zinc oxide stope by Donald Strachan as consultant for World Industrial Minerals Inc. suggest potential for tellurium as a byproduct of zinc production.

At the time of writing this report, Mr. Stone has refurbished several buildings in the mine plant complex, acquired equipment and supplies to rebuild the power line to the well in Darwin Wash, brought permits up to date, begun equipping an assay lab, purchased locomotives and cars, and is cleaning and repairing track in the Radio ore

Tunnel. Planning is proceeding to resume production of mixed sulfide-oxide lead-silver zinc-copper ore exposed on the 400 Level in the Defiance and Thompson mine areas.


OPHIR MOUNTAIN
This is approximately the same section line as Anaconda's section line
Essex fissure
Independence mine
Essex mine
Thompson mine
Bernon fault
Bernon mine
Copper fault
Davis thrust fault
Defiance Fissure
Defiance mine
Water tanks
Water tank fault
B
B'
C
C'
A
A'

Cumulative historic ore production from the district (of which most is from workings now incorporated into the Darwin mine) from 1875 to 1976 (102 years of intermittent mining activity) is about 1.5 million tons, and metal production is about.

10,500,000 ounces silver, 840,300 tons lead, 57,900 tons of zinc, 745 tons of copper, and 5,900 ounces gold (Brown? 2002). Tungsten production is unknown.

GEOLOGICAL SETTING

The Darwin mine, and geology of the surrounding Darwin Quadrangle, has perhaps been studied as intensively as any mining district in California with the exception of the Mother Lode. The principal published sources for regional geology and that of the mine are Hall & Mackevett (1958) and Hall et al (1962), although some structural interpretations therein have been revised by later investigators. Other published works include Rye et al (1974), Czamanske and Hall (1975), Dunne et al (1978), Newberry (1987), Stone et al (1989), and Newberry et al (1991). In addition, excellent work done by Anaconda geologists, particularly Dudley Davis, Geofrey Wilson and Robert Seklemian is preserved in the Darwin data archive.

The Darwin deposits are localized in several structurally controlled crosscutting and bedded skarn zones within Paleozoic turbidite carbonate and siliceous clastic units, metamorphosed to marble or calc-silicate hornfels, near the west contact of the Darwin stock. The sedimentary beds have been correlated with the Pennsylvanian-Permian

Keeler Canyon or Pennian Owens Valley Formations. The Darwin stock is about % by 2 miles in size and is alkaline, quartz-poor, and composed of monzonite and monzodiorite. It is dated by UfPb methods at 174 MA (Chen, 1977).

The upper Paleozoic carbonate rocks are folded into an anticline whose axis, which trends about N 20 W, parallels the west contact of the Darwin stock. Much of the east limb of this fold is cut out by the pluton. A number of sill-like intrusions occur within the sedimentary rocks and follow bedding into the west limb of the anticline. Several west-dipping low angle mults, termed the Davis thrust zone, cut the Paleozoic section near the crest of the anticline and in part cover the intrusive contact. The thrust is premineral, about 20 million years younger than the Darwin stock (Dunne et al, 1978).

A number of N 60-70 E trending steep faults cut both the sediments and the intrusive contact; strands of these faults both cut and are cut by the Davis thrust faults.

Minor mineralization, and carbonates locally altered to the same metamorphic facies, occurs in the upper plate. These structures control skarn development along the Darwin stock contact and also act as primary ore controls for fissure/vein, bedding replacements, and crosscutting "skarn pipe" mineralization. The most important northeast structures are, from north to south, the Mickey Thompson, Water Tank, Defiance, 434, Bernon, Copper and A472 fissures. In addition, the Essex fissure, which apparently is hidden on the surface by the upper plate of the Davis thrust, trends about N 60 W and controls important high-grade mineralization in the Thompson-Essex mine area.

The intersections of the high-angle mineralizing structures with the crest of the Darwin anticline and with second order, doubly plunging north-trending folds on the flanks of the anticline localize important bedded mineralization in favorable reactive units.

Orebodies at Darwin consist of bedding replacement deposits, skarns and quartz carbonate-sulfide +1- scheelite veins. The most important Pb-Ag-Zn deposits, localized along the Defiance fissure in the Defiance mine and along the Essex fissure in the Thompson mine, consist of steeply west-plunging chimneys or "skarn pipes" at or near the crest of the Darwin anticline. High galena-sphalerite replaces skarn-matrix breccia within the pipes, and mineralization also occurs in pyroxene and garnet skarns which replace favorable beds adjacent to these feeder structures. The Defiance deposit has been exposed and partially mined to the 1,300 Level, about 1,500 feet below the surface, and the Thompson area has been mined to the 900 Level, about 1,100 feet below the surface.

A substantial portion of the ore at Darwin is oxidized, and it is interesting to note that partially oxidized ore has been encountered in the deepest levels of the mine, and stopes of oxidized ore have been mined in all levels.

Early investigators attributed the mineralization at Darwin to contact metasomatism related to intrusion of the Darwin stock. However, the principal Darwin orebodies occur not at the stock contact but generally a few to several hundred feet to the west. The stock is essentially unaltered, weakly pyritic and exhibits only minor endoskarn development at the contact. Skarns do not occur in equivalent sedimentary rocks on the eastern or southern contacts of the stock.

The most recent publications on Darwin geology (Newberry, 1987; Newberry et al, 1991) attribute the lead-silver-zinc (but not the scheelite) bedded replacement, skarn and fissure ore in the district to a younger intrusive event, characterized by granite porphyry and aplite. That this felsic intrusive event is measurably younger than the

Darwin stock is proven by the fact that quartz porphyry cuts the Davis thrust. Rocks of this suite are present as dikes and pods both within and outside the stock; as the matrix of igneous breccias containing stock and scheelite-bearing skarn clasts; and as the core facies of igneous- and skarn-matrix breccia pipes. Newberry's isotopic and mineral zoning studies rule out a genetic tie between the Pb-Ag-Zn skarn and the Darwin stock; structural evidence indicates the skarns postdate the pluton by at least 20 Ma.

Recognition of this younger intrusive phase as the principal mineralizing event was one of the main premises for the 1991 exploration effort by Western Zinc JV.

Scheelite may have originally developed in garnet skarn at the main contact and have been remobilized with the later deposition of the base metal ores; the scheelite, particularly in the 433-435 stope, is intimately mixed with oxidized argentiferous lead and silver mineralization.

The surface geologic map included as Fig. 2 in Newberry et al (1991) indicates several localities where rocks of the granite porphyry suite crop out on the ridge above the Darwin mine workings, including a prominent zoned breccia pipe, about 500 by 700 feet in size, above the first bend in the Radio ore Tunnel at N16,800 E6,OOO (references are to the Anaconda mine grid). Other occurrences include several smaller outcrops on the crest of the ridge between the Bernon and Copper fissures at NI 8,000 E5,500, and a well-developed pipe of porphyry and skarn matrix breccia which cuts the stock east of the Independence mine at about N19,400 E5,800. In the Anaconda progress report

(Newberry, 1980) six "skarn pipes" are mentioned.

Deposit Types and Mineralization

Mineral deposits in the Darwin mine consist of lead-silver-zinc bearing, steeply dipping fissure veins, bedding replacements, and steeply plunging breccia pipes with a matrix composed of garnet-pyroxene skarn and igneous clasts. The pipes are locally cored by irregular quartz porphyry bodies. Tungsten (scheelite) occurs in veins and thin skarns on the contact of the Darwin stock in the Darwin mine and elsewhere in the

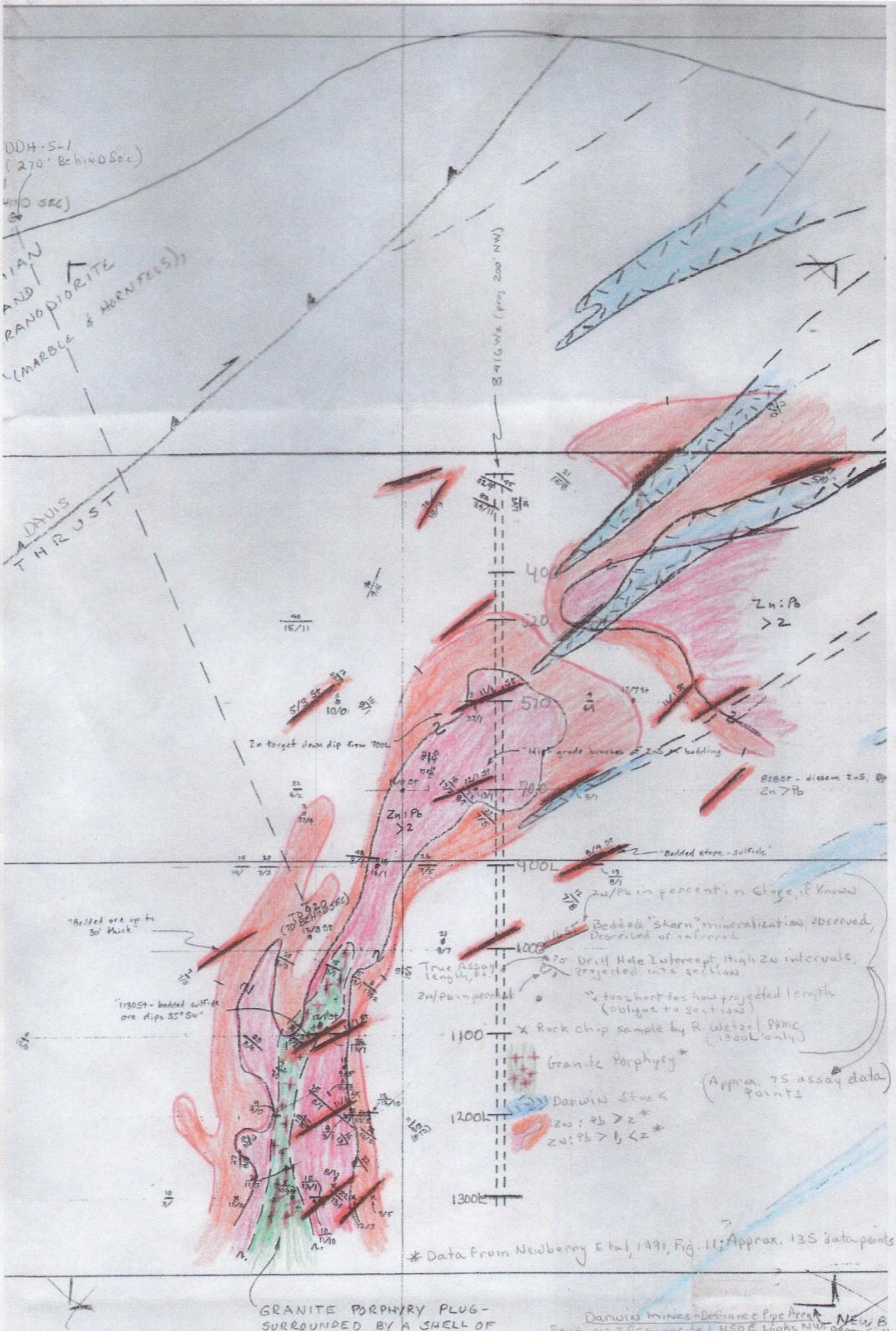
district, and locally is intimately mixed with bedded replacement ore in the mine, particularly in the 433-435 stope area.

Lead-silver ore mined in the district in the early days was near the surface and secondarily enriched, but only remnants of these bodies remains. A large, but unknown, portion of the ore mined by Anaconda and other later operators was oxidized, and mixed oxide-sulfide ore persists in the lowest levels of the mine.

Geological structure and ore controls within the several deposits which comprise the Darwin mine is very complex, and some of the relationships between the deposit types to lithology, first- and second-order folds, crosscutting fault structures which appear to be major ore controls, and the late aplite/quartz porphyry intrusive event which appears to be the source of the lead-silver-zinc mineralization are still not well understood. Individual orebodies of all types which have been mined by Anaconda and other operators are generally small but high grade, may be bounded by faults, intrusive rock and lithologic boundaries — or may have assay walls. Orebodies were usually discovered by following a drill intercept. Review of many reports in the Darwin mine omce suggests that Anaconda was able to project at best a year's reserve; the projected tonnage was based on past experience within the system and knowledge of the geology in the area of one or two drill intercepts. It is apparent in comparison of recommendations for stope development and later production records that the operator rarely knew how much was there until the stope was mined out.

Other deposit types related to the Darwin mine orebodies have been suggested by several investigators (Wilson, 1983; Newberry, 1991; Hahn, 1991-92; Brown?, 2002, and others.) As mentioned above in "History" Quintana and others reviewed the district for bulk-mineable gold, without much encouragement. The existence of a porphyry copper deposit, gradation into copper-zinc skarns, or higher zinc developed at the roots of the quartz porphyry cored breccia bodies have been postulated. The research by Newberry and Wilson for Anaconda and later published, proved that the lead-silver-zinc mineralization is younger than the Darwin stock and is related to the later felsic event. They also documented the mineral and metal zoning, both concentrically around the quartz porphyry breccia pipes and vertically. 'The skarn/breccia bodies in most cases expand with depth, but the total sulfide content decreases. This work, and the research by

1


DDH-S-1
(270' Behind Sec)
DAVIS
THRUST
Zn:Pb
>2
"Bedded stope - sulfide"
"Bedded ore up to
30' thick"
"1130St - bedded sulfide
ore dips 35° SW"
Zn target down dip from 700L
"High grade bunches of ZnS in bedding
400L
1000L
1100
1200L
1300L
Zn/Pb in percent in stope, if known
Bedded "skarn" mineralization, observed, described or inferred
Drill Hole Intercept, High Zn intervals, projected into section
True Assay length, Ft.
Zn/Pb in percent
too short to show projected length (oblique to section)
× Rock chip sample by R. Wetzel PRMC (1300L only)
Granite Porphyry*
Darwin Stock
Zn : Pb > 2*
Zn : Pb > 1, < 2*
(Approx. 75 assay data points)
* Data from Newberry Et al, 1991, Fig. 11; Approx. 135 data points
GRANITE PORPHYRY PLUG-
SURROUNDED BY A SHELL OF
Darwin Mines - Defiance Pipe Area
Section 3 (composite) N50E, Looks NW
Newberry Fig 11

Western Zinc JV (Hahn, 1992) proves that the zinc/lead increases downward in and around the Defiance and Essex pipes, and that zinc and copper also increase with depth. Both copper and tungsten are higher in the Thompson mine than in the Defiance.

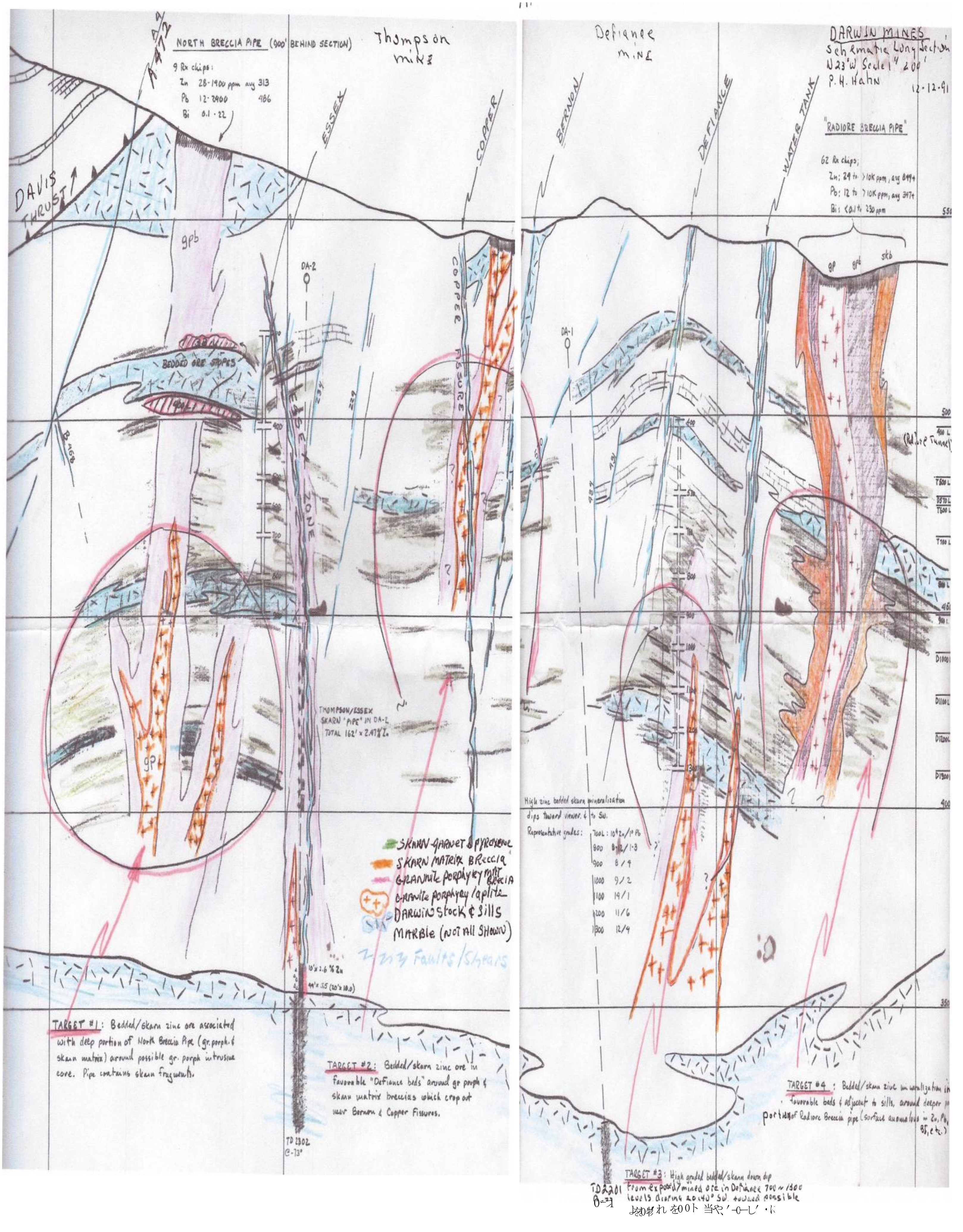
Detailed surface mapping by Anaconda geologists shows that the geologic section is not overturned as had been previously mapped by the USGS (Hall & Mackevett, 1962), suggesting that the more carbonate-rich portion of the Paleozoic section may be found in deeper levels of the mines, providing additional targets for skarn deposits. The deep holes drilled by Anaconda in 1982, steeply inclined toward the east, were designed to test for deep skarn zones, and particularly for tungsten. Hole DA-I cut the Defiance (south) pipe at depth; skarn intensity increases but sulfides decrease, and there is no evidence of late-stage sulfide mineralization. Hole DA-2, 1,350 feet north of DA-I cut the Essex pipe about 400 feet below the deepest Thompson mine 90rkings. The Essex pipe at that depth is largely replaced by large veins in skarn and irregular pyrite zones. The 500-foot mineralized interval includes 177 feet of2.6% zinc, partly oxidized. The Darwin stock was cut at 1,800 feet in DA-2, with strong skarn development and 44 feet of 5.5% zinc, including 20 feet of 10%. Tungsten was present but not notable.

Comparison of metal values in DA-2 vs. DA-I confirms the trend observed in the Thompson vs. Defiance workings. Assays in DA-2 and twice those in DA-I, with tungsten, zinc and silver significantly higher, and more intense alteration quartz veining (Brown? 2002). Recent discovery by Jack Stone of high grade, copper-rich (covellite/chalcocite/chalcopyrite) zones on the 400 Level north of the Thompson workings support the northward trend.

The Western Zinc JV work in 1991 was directed toward delineation of drill targets for a large bedded or skarn zinc deposit. Metal zoning already documented by Newberry's paper was confirmed by plotting Zn/Pb ratios and zinc grades from many drill holes, car samples and stope production records, on Anaconda's level maps.

> . . .more than 300 locations were identified with ore-grade zinc and the zinc: lead ratio over two. Many high-zinc headings and stopes, particularly on the Defiance 900-1200 levels, were producing +15% zinc and very low lead at the time of Anaconda 's shutdown. (Hahn, 1992)

16



DARWIN MINES
Schematic Long Section
N23°W Scale 1" 200'
P.H. Hahn
12-12-91
NORTH BRECCIA PIPE (900' BEHIND SECTION)
9 Rx chips:
Zn 28-1900 ppm avg 313
Pb 12-2900 486
Bi 0.1-22
Thompson mine
Defiance mine
DAVIS THRUST
ESSEX
COPPER
BERNON
DEFIANCE
WATER TANK
"RADIORE BRECCIA PIPE"
62 Rx chips;
Zn: 24 to >10K ppm, avg 8494
Pb: 12 to >10K ppm, avg 3474
Bi: <0.1 to 230 ppm
gp
gpb
skb
DA-2
DA-1
BEDDED ORE STOPES
ESSEX ZONE
COPPER FISSURE
THOMPSON/ESSEX SKARN "PIPE" IN DA-2
TOTAL 162' x 2.47% Zn
High zinc bedded skarn mineralization dips toward viewer & to SW.
Representative grades:
700L: 10% Zn/1% Pb
800 8-12/1-3
900 8/4
1000 9/2
1100 14/1
1200 11/6
1300 12/4
SKARN GARNET & PYROXENE
SKARN MATRIX BRECCIA
GRANITE PORPHYRY MATRIX BRECCIA
GRANITE PORPHYRY/APLITE
DARWIN STOCK & SILLS
MARBLE (NOT ALL SHOWN)
FAULTS/SHEARS
(Radiore Tunnel)
TARGET #1: Bedded/skarn zinc ore associated with deep portion of North Breccia Pipe (gr. porph. & skarn matrix) around possible gr. porph intrusive core. Pipe contains skarn fragments.
TARGET #2: Bedded/skarn zinc ore in favorable "Defiance beds" around gr porph & skarn matrix breccias which crop out near Bernon & Copper Fissures.
TARGET #3: High graded bedded/skarn down dip from exposed/mined ore in Defiance 700-1300 levels dipping 20-40° SW, toward possible
TARGET #4: Bedded/skarn zinc mineralization in favorable beds & adjacent to sills, around deeper portion of Radiore Breccia pipe (surface anomalous in Zn, Pb, Bi, etc.)
TD 2302
TD 2201

Four drill targets with good potential for discovery of bedded replacement/skarn zinc mineralization (and quite possible Pb-Ag-Cu, although these were not the metals sought in the study) were identified for the Western Zinc JV, detailed in Hahn (1992):

I. Northern Breccia Pipe; crops out within the Darwin stock, but the pipe contains skarn and skarn matrix breccia fragments. The roots of this pipe could be explored by a vertical hole at about N19,400 E5,500 directly west of the pipe outcrop, to a depth of 1,800 feet.

2. Thompson/Essex Zone; to cut a potential sill complex in a large unexplored area above the contact zone cut in DA-2. A 1,800 foot hole from the DA-2 site but at a shallower (-55 degree) angle was suggested.

3. Deep Defiance Target; all observers agree that bedded replacement zinc sulfide is increasingly well developed and thicker below the Defiance 900 Level.
The Zn/Pb >2 envelope increases in size downward, and at least 27 samples of +8% Zn were plotted on a composite section. The target is centered at NI 7,250

E4,250 at El. 3,700 ft., or about 300 feet below the Defiance 1,300 Level. An

1,800 ft. steep or vertical hole 600-800 feet east of the DA-I site was. recommended.

4. Radio ore Breccia Pipe; this porphyry/skarn pipe crops out on the ridge centered on the Water Tank fissure, covers about six acres and is strongly anomalous in zinc, lead and copper. Insumcient field work was accomplished to adequately document this target before the project was cancelled.

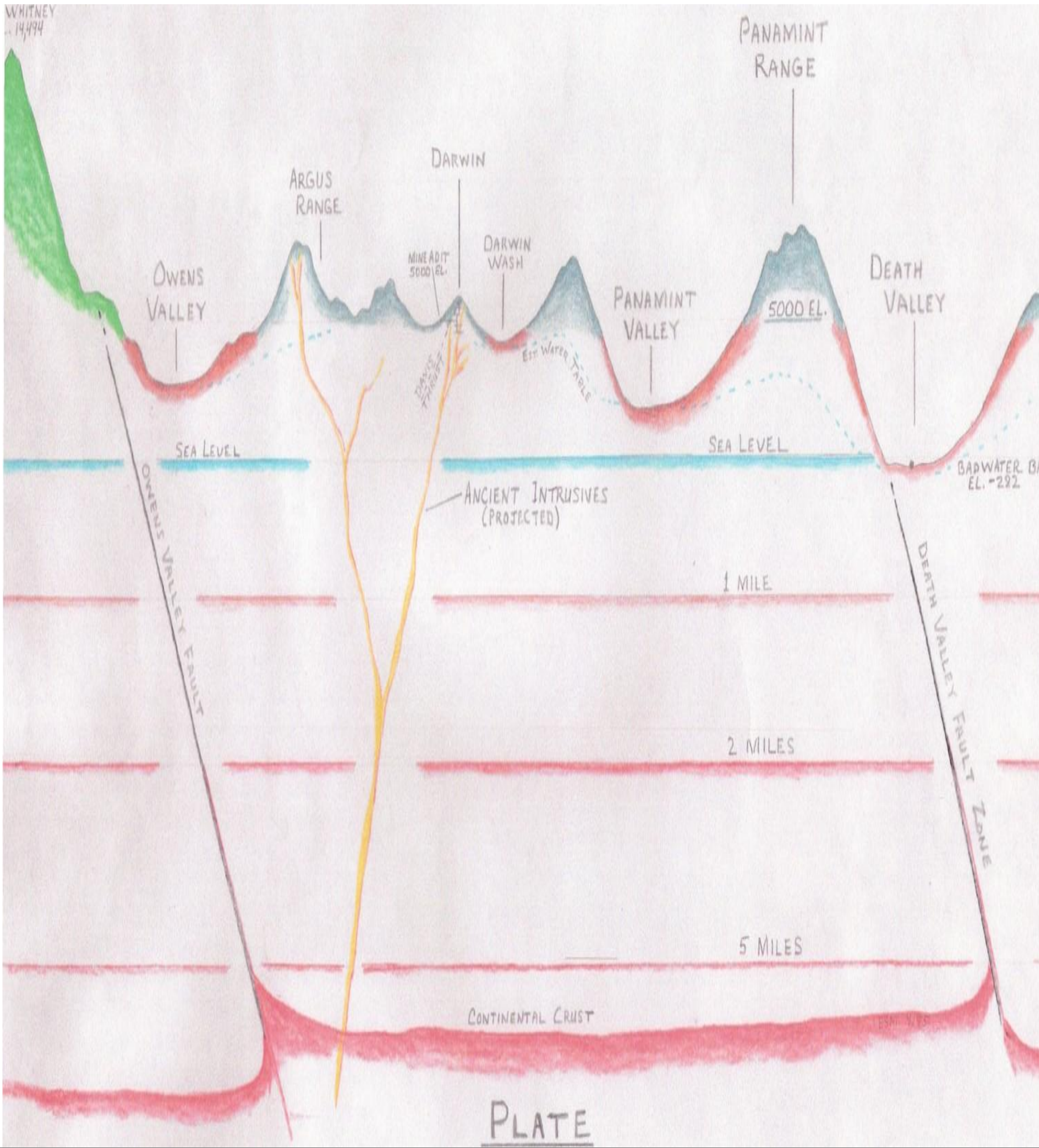
EXPLORATION DRILLING SAMPLING DATA VERIFICATION ETC.

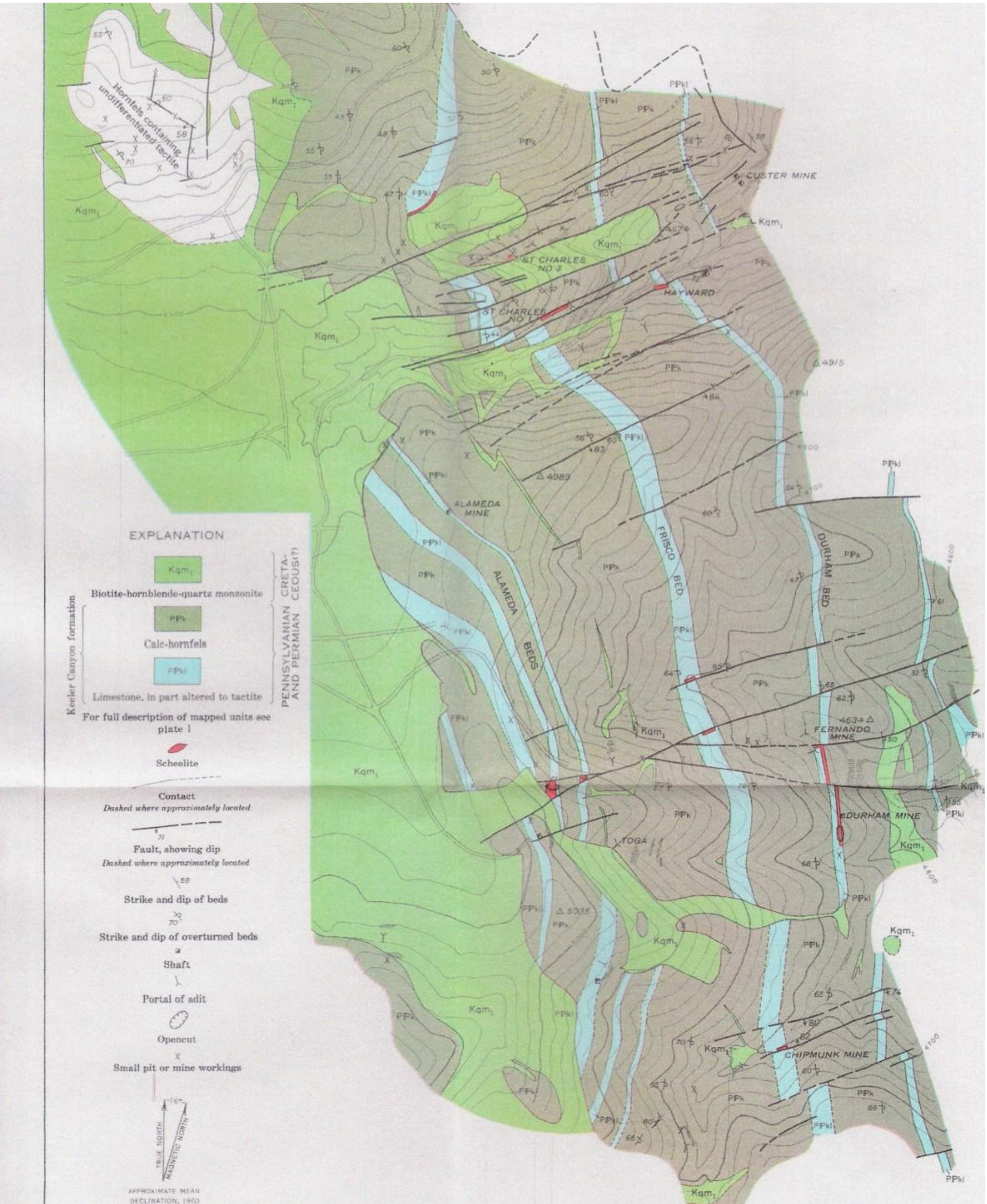
No drilling has been done since Blue Range's tenure, and the exploratory work done since Anaconda relinquished the property is detailed above. Exploration by the current owner of the property is limited to observation of exposed mineralization on the 400 Level, and short visits to a few deeper levels. Intermittent sampling (grab samples and channels) for tellurium in several high-zinc areas on the 400 Level continues, with multi-element ICP analyses performed by Inspectorate America Corp. and Florin Analytical Services in Reno, Nevada. Both laboratories are believed to follow standard QA-QC practice using blanks and certified standards.

ąRA'S

REGIONAL ȚF.CȚONIC
SECTION A-A'

14t) Młcrs




Hornfels containing undifferentiated tactite
CUSTER MINE
ST CHARLES NO 3
HAYWARD
ST CHARLES NO 1
ALAMEDA MINE
ALAMEDA BEDS
FRISCO BED
DURHAM BED
FERNANDO MINE
DURHAM MINE
TOGA
CHIPMUNK MINE
EXPLANATION
Kqm1
Biotite-hornblende-quartz monzonite
CRETACEOUS(?)
PPk
Calc-hornfels
PPkl
Limestone, in part altered to tactite
Keeler Canyon formation
PENNSYLVANIAN AND PERMIAN
For full description of mapped units see plate 1
Scheelite
Contact
Dashed where approximately located
Fault, showing dip
Dashed where approximately located
Strike and dip of beds
Strike and dip of overturned beds
Shaft
Portal of adit
Opencut
Small pit or mine workings
TRUE NORTH
MAGNETIC NORTH
APPROXIMATE MEAN DECLINATION, 1960

Adjacent Properties

Besides the several sets of workings in the Darwin mine itself, the only other mines on the patented claims with significant production are the Lucky Jim, Promontory and Lane mines (Mackevett et al, 1958).

The Lucky Jim mine is located in Sec. 1, T. 19 S. R. 40 E., about 2% miles due north of the Darwin mine camp. Silver-lead-zinc ore occurs as fissure fillings in the steep NE-trending lucky Jim fault zone and other smaller veins, which cut granitic rocks and calc-homfels. The mine is developed by a 320-foot shaft and internal winzes to 820 feet depth. Partial production figures 1915-1925 from mainly oxidized ore include over.

40,000 tons of ore which yielded about 10.2 million pounds of lead, 744,000 ounces of silver, and 457 ounces of gold. Anaconda renovated and evaluated the mine in 1948 but did not produce. Quintana re-examined the mine area in 1987 (DeRuyter, 1987).

The Promontory mine is in Sec. 30, T. 19 S. R.41 E., about 1% miles southeast of the mine camp. Oxidized lead-silver ore occurs as a bedding replacement in limestone; the mine is developed by an incline shaft with a vertical depth of 283 feet. Production is unknown, as most has been reported with that of the Darwin Silver Co.

The Lane mine, in Sec. 18, T. 19 S. R. 41 E., about 1% east of the mine camp, was developed about 1890. Largely oxidized lead-silver-copper minerals occur in fissure veins, developed to a depth of 800 feet, in limestone and calc-silicate hornfels. It was operated intermittently until 1948, and produced about 12,000 tons of ore, yielding 1.8 million pounds of lead, 76,000 ounces of silver, 39,000 pounds of copper, and 1,547 ounces of gold. Project Darwin LLC has no current plans for any of these properties.

Mineral Processing and Metallurgical Testing

There is currently no processing facility on the property, and Project Darwin LLC plans to mill future ore production off site.

Historically, the mixed sulfide/oxide mineralogy of the complex lead-silver-zinc ores of Darwin has presented problems for operators. Anaconda operated separate sulfide and oxide flotation circuits in their mill after the oxide section was built in 1951. It is assumed that before that time much or all of the oxide was shipped direct to the smelter. During Anaconda's ore production period, about 29.5% was processed in the oxide circuit and 70.5% in the sulfide circuit. Zinc oxide minerals were not recovered. Recovery for sulfide ore, 1948-1957, was 79.5% of the lead, 75.8% of the silver and 77.4% of the zinc. For the years 1945-1957, oxide recovery was 83.4% of the lead and 77.3% of the silver (Brown? 2002). When West Hills Exploration operated the mine, the separation of sulfide and oxide mill feed was made by visual inspection of the cars (Paul Skinner, pers. comm.).

There have been many improvements made recently in methods for processing oxidized zinc ores, and several large mines (i.e., Skorpion Zinc/Anglo American in

Namibia) have achieved good recovery from oxide deposits. Kappes Cassiday and Associates in Reno have considerable expertise in this area and have consulted for Project Darwin LLC. Both caustic and acid leach methods, together with modem resin extraction or precipitation reagents are used with success.

Mineral Resource/Mineral Reserves Estimates

There are currently no NI 43-101 compliant reserves or resources in the Darwin mine; numbers quoted in this section are historical in nature and should not be relied upon to represent mineable material present in the mine. These historical reserves have not been classified according to CIM resource/reserve categories. Project Darwin LLC is not treating any historical reserves as ppears current mineral resources/mineral reserves but considers the data to have been substantial and relevant at the time the calculations were done. An unknown, but probably substantial, tonnage quoted from each source has probably been removed by subsequent operators. The nature of the deposits as currently exposed in the mine and known from underground drilling and two deep core holes, with comparatively small but high-grade orebodies controlled by very complex faulting, folds and breccia pipes does not lend itself to "blocking out" ore tonnage. As explained above under "Deposit Types" Anaconda and other operators "followed the ore" and while Anaconda operated the mine, a great deal of underground core drilling was done, together with meticulous, detailed 40-scale and larger geologic mapping. Many sources have stated that the Anaconda operation never had more than a year's mining reserve ahead, but always managed to "keep ahead of the mill." Fortunately, these data are largely preserved in the Darwin mine office. Most, but possibly not all, of the subsequent operators kept up the mapping and drill logging standards. As an example of missing data, however, the zinc oxide stope on the 400 Level recently sampled to tellurium by D.

G. Strachan was not mapped on the geology compilation linens. Most of the field compilation ledger sheets and level maps compiled for Western Zinc JV by this author in 1991 have been lost, and many maps and sections which accompanied other reports are missing.

Anaconda calculated the reserves at Darwin each year as of January 1. A few of these reports in the Darwin files, but there are no maps with these copies. Most of these year-end reports are available in the Anaconda Collection in the American Heritage

Center at the University of Wyoming in Laramie. A request for a search of files keyed tc Darwin in 2009 produced a list of 135 records (among 796 files keyed to Inyo County, California). It is possible that review of the Darwin, and perhaps other Inyo County files, at the Laramie archive could produce other clues to future work at Darwin, but it a that the records there are limited to the Anaconda days. Any specific recommendations for exploration/exploitation of areas in the mine itself would be obsolete in view of the later work done by leasers. A single-person license to use this archive cost $1,750 in

2009-2010.

A detailed report (Skiles, 1957) on estimated ore reserves at the time of

Anaconda's shutdown is in the Darwin office files, but the 14 sections and section line index referenced in the report have not so far been found. A total of 150 separate ore blocks are listed, from 32 to 12,000 tons. Skiles classified reserves into "Positive,

Probable and Possible" tons and gave a total of "Reasonably Assured" tons by subtracting one-half the Possible tons. No explanation of his classification criteria was given. He further broke the numbers down into "Shipping Grade, Sulfide Milling and Oxide Milling." The total "Reasonably Assured" ore reserve at that time was 154,968 tons, 4.9 opt silver, 6.1% lead and 8.5% zinc. 30% of this tonnage was in stope and sill pillars. Sulfide milling ore accounted for 72% of the total, oxide milling for 27%, and direct ship less than 1%. It is notable that 56% of the total ore was in the Defiance mine in the 800 Level and lower. To repeat more detail of this calculation seems useless, as it is very likely that most of these reserves have been mined by subsequent operators.

Another detailed calculation was made by Cormie (1970) for Mexicanus Explorations Ltd. Cormie was assisted by Dudley Davis, long-time geologist at Darwin for Anaconda. He lists "Total Probable and Possible" ore at 129,375 tons, 5.7 opt silver,

4.1% lead and 10.0% zinc, in 39 blocks. General maps are in the report, but no detailed maps or sections by block have been found, and it is likely that part of this ore was removed by Mexicanus Colorado.

What is believed to be the last detailed reserve calculation found in the Darwin files (Musolf, 1971) was made in August 1971 for Mexicanus Colorado.

"Developed" ore, Defiance mine: 32,050 tons, 3.9 opt silver, 3.3% lead, 8.8% zinc.

"Possible" ore, Defiance mine: 19,400 tons, 7.1 opt silver, 2.7% lead, 1.7% zinc.

"Developed" ore, Thompson mine: 13,300 tons, 6.2 opt silver, 2.4% lead, 7.7% zinc

"Possible" ore, Thompson mine: 6,800 tons, 12.0 opt silver, 6.0% lead, 5.0% zinc

Total weighted average: 71,550 tons, 6.0 opt silver, 3.2% lead. 6.3 % zinc.

The Defiance reserves were distributed in stopes from the 570 to 1300 Level; the Thompson ore was in the 3B to 900 Levels. Zinc grade clearly increases downward in the Thompson mine. Again, part of this ore was no doubt mined subsequently.

It is believed that no more recent reserve calculations have been made.

Other Relevant Data and Information

Jack Stone and Adena Fansler, Associate Planner for the Inyo County Planning Commission, were interviewed at Darwin on July 13, 2010. This author is not a Qualified Person in the field of environmental assessment or mine permitting but was provided copies or relevant details of the permits and licenses currently in place, and assurance that all the necessary documents are in force.

In California, the lead agency in mine permitting is the county. The area of the mine is zoned Open Space, 40-Acre Minimum (OS40) and mining is a permitted use. Inyo County Planning Department issued a Draft Negative Declaration of Environmental

Impact for the project on September 28, 2007, and the key document, Conditional Use

Permit CUP #2007-04 was granted to Project Darwin LLC on December 12, 2007. The CUP provides that activities will be limited to one shift during daylight hours with ore haulage, weekdays only; ore will be trucked off site (to Nevada) for processing; and that Project Darwin LLC will pay to Inyo County its "applicable share" of the maintenance of the Darwin County Road from the mine site to the intersection of Highway 190. The CUP limits production/shipping to 500 tons per day, which is also the practical haulage limit imposed by the capacity of the Radio ore Tunnel.

All operations will occur on patented land, and BLM permitting is not required. The operation will not fall under the California Surface Mining and Reclamation Act of 1975 (SMARA); no new areas of disturbance will exceed one acre nor require the removal of more than 1,000 cubic yards of material. All waste rock will be backfilled underground.

Project Darwin LLC has received a General Stormwater Permit from the

Lahontan Region, California Regional Water Quality Control Board. No effect on groundwater is contemplated, as the static water table is at least 1,000 feet below the mine workings. The Project has an "Authority to Consmwt" issued by the Great Basin Unified Air Pollution Control District and permits from Inyo County Health Department for use of water storage tanks and operation of small water and septic systems.

A license 9-NV-023-33-2M-00264 to purchase, transport, store and use explosives has been issued to Stone Bros. Construction and Demolition, Inc. by the U. S.

Bureau of Alcohol, Tobacco, Firearms and Explosives, pursuant to 18 USC Ch. 40.

Any underground and surface work related to production will fall under the purview of the Federal Mine Safety and Health Administration (MSHA) and the equivalent California state agencies.

A seismic monitoring station operated by the U. S. Department of Energy is located in the Radio ore Tunnel near the portal. Rental of $12,000 per year is due to

Project Darwin LLC.

It is believed that Project Darwin LLC has no responsibility or potential liability concerning the reclaimed heap leach pad and tailings pond which are located entirely on BLM land. It is believed that Arco Petroleum, successor to Anaconda and now part of

British Petroleum may have some residual liability.

Interpretation and Conclusions

Throughout the history of mining in the Darwin lead-silver-zinc deposits, new ore reserves continued to be discovered as long as systematic exploration and development programs were followed. There has been no consistent exploration program in the district since Anaconda ceased operations in 1957, and the remaining reserves left by Anaconda (Skiles, 1957) were depleted by lessees 1967-1976.

Silver, lead and zinc grades were consistently high throughout the history of the district, including all lessee operations. The history of past mining at Darwin suggests that continuations of known ore can be proven, and that good potential exists for discovery of new deposits. The metal content, metal ratios, and mineralogical gradients suggest good potential for zinc rich or copper-zinc skarns at depth, and there remain unexplored portions of the system between and adjacent to the Thompson-Essex and Defiance skarn breccia pipes in levels already partially developed. These gradients are discussed in detail in Rainer Newberry's papers (1987, 1991) where Darwin is compared with other highly productive skarn and disseminated ore systems such as Cananea and Conception del Oro in Mexico.

The recent recognition of potential for byproduct elements not previously considered, such as germanium, tellurium and indium may contribute to the economics. These elements are well known to accompany zinc sulfides, particularly in skarn environments. Tungsten is another possible byproduct; high grade (+1%) scheelite ore has been produced from the Darwin mine, but the tungsten association with base metal silver ore is unusual and not well understood.

There are exposures underground of high-grade zinc-copper-lead-silver mineralization which not been followed; the "covellite area" at the north edge of the Thompson mine in the 400 Level discovered by Jack Stone, and the unmapped zinc oxide stope in the Thompson east of the 433-435 Stope are examples.

There are lead-silver-zinc skarn and fissure vein deposits known from the Promontory mine on the south to Lucky Jim on the north, near but probably not directly genetically related to the contact of the Darwin stock, a span of4h miles. Little of this area has been explored systematically.

There are two major detriments to near-term reduction of production from this mine. One is the lack of practical access to any deeper levels in the mine than the Radio ore Tunnel (400 Level). Developed levels, which have been major producers in the past, are present for 900 feet deeper in the Defiance Mine and 500 feet in the Thompson/ Essex workings. It is believed that the hoists and internal shafts in both areas are serviceable and could be brought up to safe working standards. Access to the deeper workings would increase the potential working faces by an order of magnitude; these are portions of the system which were producing high grade ore when the mine was closed by Anaconda and subsequent leasers due to low metal prices. The deep workings will also provide drill platforms for further exploration and expansion without the great cost of drilling from the surface. It may also be possible to connect the two mine areas, say on the 900 Level (bottom of the Thompson mine) to open up a large area for geologic mapping (as opposed to interpretation of drill results), additional drilling and discovery of new orebodies.

The other obstacle is the disarray of the very large body of engineering, geological, drilling, and production records, drill logs and maps. The Anaconda (plus whatever was added by leasers) data archive has been moved twice and much of the usable information is totally disorganized. Much may be duplicative and should be discarded or at least cataloged and stored separate, so as to make the remaining information suitable as working data to be used in a new operation.

Recommendations

The primary recommendations of this author are detailed above, to reactivate the hoists and make the shafts safe for use in the Defiance and Thompson shafts and cull the duplicative and totally obsolete data from the files and maps. These tasks must take priority in order to reopen large areas of the mine to inspection, and to come up with a working body of engineering and geology data which can be used on a day-to-day operating basis. When all areas of the mine which can be safely accessed are available, the deeper levels in particular should be carefully examined and compared with the master geologic maps, as some headings and stopes made by leasers may not be mapped.

To conform to modem methods of file/map preservation and deposit evaluation, the maps and reports which are deemed useful in a practical sense should be scanned into digital form. Digital data, engineering/survey work and particularly working copies of field geology sheets and geology compilations (such as the big Anaconda 40-scale linens:

should be routinely stored when not in use in the fireproof vault.

This author is not a Qualified Person with respect to GIS technology, computer mine and drill data manipulation, and related ore reserve determination currently in use by the industry. It is possible that the complex structure and variability of high-grade mineralization at Darwin may not lend itself to use of commonly employed deposit modeling programs such as Vulcan or Micromine. However, it is realistic to recommend that all data, including drill records, stope production, car and rib samples in a well explored portion of the mine be integrated into an applicable modeling program as a test.

This should be done by a geologist with expertise in computer modeling.

Before the general public in Inyo County becomes aware of active operations at Darwin, more mining claims should be staked for protection. On the west side of the patented property, many or all of the unpatented claims formerly held by Anaconda and other predecessors to Project Darwin LLC should be relocated, i.e., the Colorado and Imperial Colorado groups, Grandview, Anaconda, Vivian group, Aztec, Toltec,

Montezuna etc. To the east, at least 1,000 feet east-west adjoining the eastern boundary of the patents should be located, as there is discovery potential for additional breccia pipe zones in and near the Darwin stock. A portion of the claims recommended might correspond for convenience with Mineral Surveys which were never taken to patent.

Specific recommendations as to discrete areas in the Darwin mine which may yield near-term production cannot be made until the entire mine, or as much as safely possible, is available for examination. A complete chronological compilation of specific exploration recommendations made over the years by Kildale, Davis, Wilson and other Anaconda geologists has not been made and compared with what has been explored for, developed and mined. This will be a practical task once the files are reorganized. One of the last examples of specific exploration recommendations is offered by Wetzel (1989); targets include the Darwin anticline under the lower sill in the Thompson mine; deeper zones along the B458 fissure, and the Essex and Thompson fissure zones below the deepest workings. Wetzel (optimistically!) felt that together, these zones had potential to develop nearly four million tons of ore.

Long term geologic targets and exploration recommendations have been made by

Kildale (1955), Wilson (1982, 1983), DeRuyter (1987), Wetzel (1989), Newberry et al (1991), Hahn (1992), Brown? (2002), Saunders (2007), and others including Dudley Davis, long-time Anaconda geologist at Darwin. The geologic targets include the crest and west limb of the Darwin anticline between the Defiance and Thompson mines, particularly immediately below sill-like intrusive bodies which extend westward from the Darwin stock, and which may have served as dams to the upward migration of ore forming fluids. Areas where these geologic environments are cut by the 434, Bernon, and Copper fissures may be particularly favorable and have never been seen below the Radio ore Tunnel level.

The deepest levels of both mine areas, where high grade (particularly zinc) ore was being mined at shutdown should be re-examined. See Wetzel (1989). Sampling of high-zinc exposures for tellurium and indium should be continued.

As revealed in Newberry's work, the section is not overturned as it had been mapped by USGS geologists. It is likely that deeper limestone beds are present (Keeler Canyon Formation) and could be productive hosts for skarn.

Dependent on goals and funds available (and metal prices) follow up of the four deep zinc skarn drill targets recommended for Western Zinc JV (Hahn, 1992) and detailed in Section 10/11 above should be considered. If deeper sections of the mine can be accessed, exploration of these and other targets from underground drill locations may be more practical than drilling from the surface.

It seems likely that additional tungsten resources, such as the ore mined in the 433435 stopes, at least at the levels currently developed, may be discovered only by accident. Anaconda's deep exploration for tungsten in the DA-I and 2 holes was unsuccessful. No "typical" tungsten skarn are seen in the current mine. Nevertheless, tungsten should be considered in any future exploration, and development work, drill core/samples, and any other samples should be routinely lamped and possibly assayed for tungsten, particularly in deeper skarn intercepts. The multi-element ICP analyses listed by ALS-Chemex and other analytical labs can be used (or a Darwin-specific suite designed) for possible byproduct metals.

References

Most of the unpublished and company references will be found in the files in the Darwin mine omce.

Barber, G. A., 1977, Internal Anaconda memo to R. L. Knight, 6/29/77.

? 1990, Darwin Mine property: report prepared January 1990 (possibly by Rob Wetzel for Blue Range Mining Company.

Brown, H. G., 2002, Update, Darwin Mine, Inyo County, California, An Excellent Zinc Exploration Opportunity, 5/10/02.

Brown H. G (?), 2002, Darwin Mine, Inyo County, California, 7/23/02 (probably prepared for Project Darwin Inc.)

Chen, J. H.—Y., 1977, Uranium-lead isotopic ages from the southern Sierra Nevada batholith, and adjacent areas, California: PhD thesis, Univ. Calif. Santa Barbara.

Cormie, J. M., 1070, Darwin Mine, Inyo County, California, Geology, Ore Reserves, Proposed Development: Report 11/11/70 for Mexicanus Explorations Ltd.

Czamanske, G. K. and Hall, W. E., 1975, The Ag-Bi-Pb-Sb-S-Se-Te mineralogy of the Darwin lead-silver-zinc deposit, southern California: Econ. Geol., v. 70, p. 1092-1110.

DeRuyter, V., 1986, Bulk-minable (sic) silver-gold targets at the Darwin Project, Inyo County, California: report for Quintana, 8/86.

-----------, 1987, Results of sampling at Radio ore 400 Level and Lucky Jim area, Darwin Mines, Inyo County, California: report for Quintana, 9/87.

-----------, 1988, Results of drill work during 1988 in the Lucky Jim mine area, Darwin

Project, Inyo County, California: report for Quintana, 11/88.

Dunne, G. C., Gulliver, R. M., and Sylvester, A. G., 1978, Mesozoic evolution of rocks of the White, Inyo and Slate Ranges, eastern California, in Howell, D. G. and

McDougall, K. A., eds., Mesozoic paleogeography of the western United States: Soc. Econ. Paleontologists and Mineralogists.

Hahn, P. He, 1991, Monthly reports to Western Zinc JV/Cyprus Minerals.

-------, 1992, 1991 exploration program in the Darwin zinc-lead-silver mine, Inyo County, California: report for Western Zinc JV, 1/3-/92.

Hall, W. E., Mackevett, E. M., and Davis, D. L., 1958, Economic geology of the Darwin Quadrangle, Inyo County, California: California Division of Mines Special Report 51.

Hall, W. E., and Mackevett, E. M., 1962, Geology and ore deposits of the Darwin Quadrangle, Inyo County, California: U. S. Geol. Survey Prof. Paper 368.

Kildale, M. B., 1955?, Lead-zinc deposits of the Darwin district, California: Text of illustrated talk?, for Anaconda.

Musolf, K. R., 1971, Memo to E. Rudd, Manager, Mexicanus Colorado, Inc., 8/27/71

Newberry, R., 1986, Use of intrusive and calc-silicate compositional data to distinguish contrasting skarn types in the Darwin polymetallic district, California, USA: Mineralium Deposita, vol. 22, p 207-215.

Newberry, R., Einaudi, M. T., and Eastman, H. S., 1991, Zoning and genesis of the Darwin Pb-Zn-Ag skarn deposit, California: a reinterpretation based on new data: Econ. Geol., vol. 86, p. 960-982.

Rye, R. 0., Hall, W. E., and Ohmoto, H., 1974, Carbon, hydrogen, oxygen and sulfur isotopic study of the Darwin lead-silver-zinc deposit, southern California: Econ. Geol., vol. 69, p. 468-481.

Saunders, F., 2007, Darwin Property, Inyo County, California: review of data base: report 6/25/07 for Geneva Partners LLC.

Seklemian, R., 1987, Report of data evaluation, Darwin Mines property, Inyo County, California — a project of Cyprus Minerals Company: report for Cyprus, 11/87.

Skiles, R., 1957, Darwin Mines, ore reserves as of June 1, 1957: Report for Anaconda (with cover letter 1/16/59, M. B. Kildale to R. B. Mulchay)

Stone, P., 1984, Stratigraphy, depositional history, and paleogeographic significance of Pennsylvanian and Permian rocks in the Owens Valley-Death Valley region, California: PhD thesis, Stanford Univ.

Torrey, C. E., 1991, Darwin Mine, Inyo County, California: memo with exploration proposal and budget, to W. D. Burton, Cyprus Minerals Co., 1/23/91.

Wetzel, R., 1989, Darwin Mine, Inyo County, California: report for Blue Range Mining co., 12/30/89.

Wilson, G. E., 1982, Darwin Project, 1981 annual summary report: Anaconda report 5/82.

--------, 1983a, Darwin Project, first quarter and second quarter reports: Anaconda

--------, 1983b, Darwin Mines, General geology and ore deposits: Anaconda report, 7/07/83

Certificate of Author

Peter Hamilton Hahn

I, Peter H. Hahn, Certified Professional Geologist, to hereby certify that:

1. I am currently employed as a Consulting Geologist as a sole proprietor with an office at 1536 Brigadoon Park Drive, West Jordan, Utah 84088.

2. I graduated with a Bachelor of Science degree in Geology in 1957 and a Master of Science degree in Mining Geology from Stanford University in 1959.

3. I am a member of the American Institute of Professional Geologists and a Certified Professional Geologist #1 ()923 of that organization and have been since 2006.

4. I have been employed in the mineral exploration industry for 51 years since graduation from university, and have worked as a Geologist for Bear Creek Mining Co., District Geologist for Union Carbide Corp., Chief Geologist and Vice President for Nevada
Resources Inc., and Chief Geologist for Galway Resources US Inc. I have worked as Consulting Geologist for Utah International Inc., A. F. Budge Mining US Inc., Coeur Explorations, Cyprus Amax Metals, Tonogold Resources Inc. and others.

5. I have read the definition of "qualified Person" set out in National Instrument 43-101 ("NI 43-101 ") and certi$' that by reason of my education, affiliation with a professional organization (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-10-1.

6. I am responsible for all sections of the Technical Report titled Technical Report on the Darwin Mine, Inyo County, California and dated July 26, 2010. I have personally visited the Darwin Mine in the field on July 12-14, 2010, and other times in 1991 and 2007, for a total of approximately 35 days.

7. I have had prior involvement with the property that is the subject of the Technical Report by virtue of prior employment by Western Zinc JV / Cyprus-Amax and Galway Resources US Inc.

8. I am not aware Of any material fact or change with respect to the subject matter of the Technical Report that is not reflected therein, and as of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of the Issuer applying all the tests in Item 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

I I. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible to the public, of the Technical Report.

Dated this July 26, 2010, in West Jordan, Utah Signature of Qualified Person

Is/

Peter H. Hahn, CPG-10923

Printed name of Qualified Person

6 INYOAG LLC Mine Plan

DARWIN CALIFORNIA

MISSION STATEMENT

INYOAG LLC is committed to solving as many of the critical minerals & domestic metal production issues associated with clean energy independence, carbon neutral footprint, economic social governance, & job creation as it is within their realm of assets. The project consists of 1000+ acres of patented land. A mining district 3 miles wide by 8 miles long, it is a 24sq mile metals supermarket. With Zinc, Copper, Tungsten, Silver and Gold as well as 14 critical minerals we can help the USA and its allies to become a more self-reliant nation.

Overview

Project Overview: INYOAG LLC is a mining project located in Darwin, California. It involves reopening the historic Anaconda Mine, which operated from 1873 to 1976 and was closed due to market impacts from China. The mine is being redeveloped under the Opportunity Zone Program and is privately owned by INYOAG LLC. The project encompasses 1,050 acres of 58 patented mining claims including a specific patented mill site claim. Situated 36 miles southeast of Lone Pine near the town of Darwin, the property is positioned between Sequoia National Park and Death Valley National Park, near the Nevada border. The mining claims are spread across townships within the Mount Diablo Meridian, with essential infrastructure for water and power facilitated through leases from the California State Lands Commission and the Bureau of Land Management.

INYOAG LLC will produce zinc sulfate monohydrate flake as well as liquid zinc for the agriculture market, as well as copper, lead, tungsten, silver, germanium, tellurium, indium, gallium, and antimony. Once production is ramped up, we will work further on developing recovery for other critical minerals found in our unique ore structure.


San Francisco
Darwin, CA
Los Angeles
Sacramento
San Jose
Fresno
Bakersfield
Death Valley National Park
Inyo National Forest
Lancaster
Palmdale
Arizona
Flagstaff

Resource

Resource: Mineral resources at the Darwin Project have been estimated using a sectional method. This approach leverages all available historical data, including drill results, underground channel sampling, and historical production reports. The mineral resource estimate is based on all data obtained as of April 1, 2024. Inferred mineral resources are that part of the mineral resource for which quantity and grade or quality are estimated on the basis of limited geologic evidence and sampling, which is sufficient to imply but not verify grade or quality continuity. Inferred mineral resources may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration. The geology of the Darwin Project is well understood, and the appropriate deposit model is being applied for exploration. The conceptual geologic model is sound, aligns with historical drilling results, and indicates that mineralization is essentially open in all directions, suggesting significant potential to expand on the known mineralization. Significant potential exists to increase the known mineral resource with additional drilling, as well as to upgrade existing mineral resource classifications with infill drilling or systematic underground channel sampling. There is also considerable potential to encounter massive skarn and porphyry mineralization within the district.

TABLE 1 DARWIN PROJECT MINERAL RESOURCE STATEMENT

Block	OX State	Tons	Ag		Pb		Zn		W		Cu	
		(x1000)	opt	oz (x1000)	%	lb (x1000)	%	lb (x1000)	%	lb (x1000)	%	lb (x1000)
Indicated Resource Blocks												
A433/435	Oxide	180.0	8.5	1,531.8	0.8	2,952.0	6.4	23,076.0	0.6	2,052.0	-	-
B-458 Fissure	Mixed	66.5	9.3	618.5	0.9	1,197.0	8.2	10,906.0	-	-	-	-
418D Oxide	Oxide	30.0	10.6	319.1	-	-	8.0	4,807.5	-	-	-	-
A-439 Stope	Oxide	30.0	11.2	337.0	3.4	2,066.4	2.6	1,538.4	-	-	-	-
Inferred Resource Blocks												
B-458 Fissure	Mixed	33.25	8.5	283.0	0.8	545.3	6.4	4,262.7	-	-	-	-
418D Oxide	Oxide	490.8	10.6	5,220.9	-	-	8.0	78,650.7	-	-	-	-
833/712 Zone	Mixed	30.0	2.6	76.6	7.4	4,428.0	7.7	4,615.2	-	-	-	-
1302 Drift	Mixed	33.75	4.2	140.7	2.6	1,771.2	7.6	5,148.8	-	-	-	-
Remnant Blocks	Mixed	114.85	6.5	746.5	7.8	17,916.6	7.1	16,308.7	-	-	-	-
B-458 Fissure Cu	Mixed	175.0	41.0	7,175.0	4.0	14,000.0	14.0	49,000.0	0.9	3,150.0	6.0	21,000.0

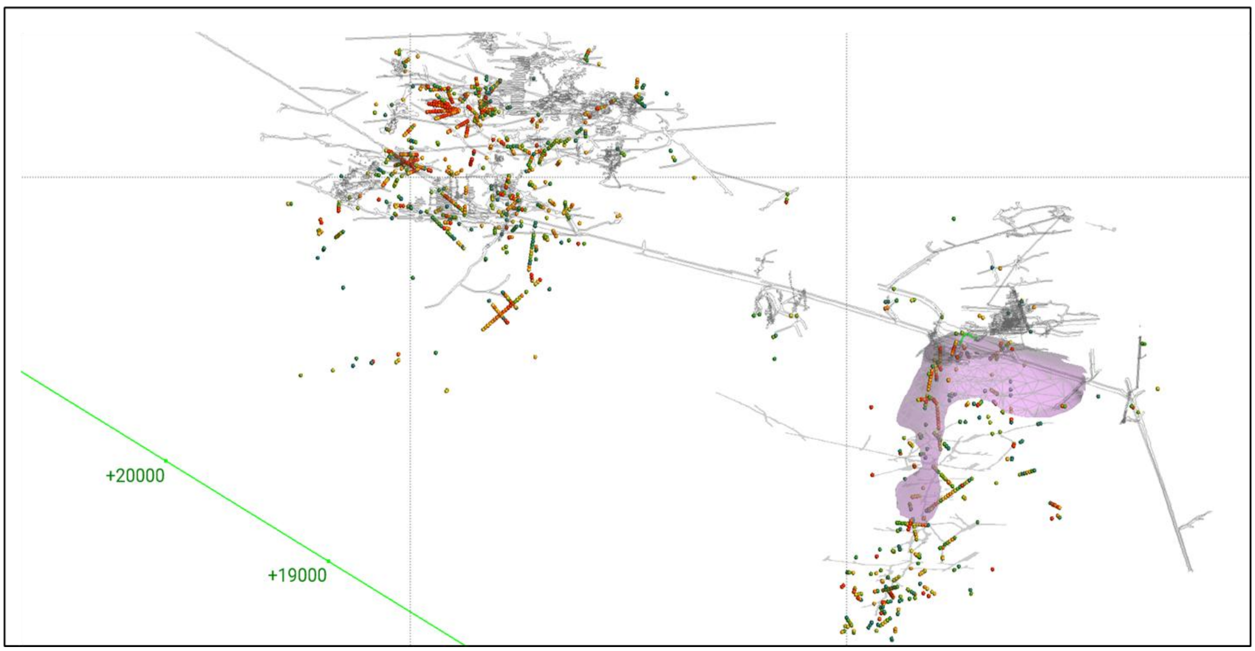
Notes: * "Reasonable prospects for economic extraction" are based on an 8 opt Ag equivalent cut-off grade. The estimation method restricts extrapolation to known mineralized areas within historical mine workings. Ag equivalent is calculated using $25/oz for silver, $0.9/lb for lead, $1.1/lb for zinc, and a 94% metallurgical recovery rate. Total costs, including mining, processing, G&A, and shipping and smelter fees, are estimated at $189.50 per ton of ore. Calculations assume use of underground longhole stoping and processing into a flotation concentrate.

Production and development

Production Plan The mining operation is balanced between oxide and sulfide production, with a daily capacity of 500 tons per day—300 tons per day sulfide and 200 tons per day oxide. Increasing to 2000 tons per day in year 5 to 1500 tons per day sulfide and 500 tons per day oxide.

Infrastructure and Existing Workings: The mine utilizes an extensive existing infrastructure of 150 miles of drift and 38 miles of rail. Two hoists—one in the Defiance developed to the 1300 level with hoist on the 400 level. One in the Thompson developed to the 900 level with the hoist on the 200 level—aid in transportation. Also, an incline winze with ore skip is developed to the 1000 level, with transfer chute on the 400 level.

The mine branches off the main haulage level, the 9,660-foot Radio Ore Tunnel, developed originally by the Radio Ore Company. The 400 level provides access to developments above and below.



The air for the mining operation is currently (2) 125 hp 750 cfm air compressors with dryer. There is a trailer mounted diesel unit for emergency use only. This compressor is 750 cfm as well. These compressors are plumbed to the mine on the 200 level of the Thompson through a 6" spiral steel main line. Future installations of two 750 cfm air compressors with dryers at the Mickey Summers will result in 1500 cfm feed to the mine. The air system on the 400 level remains intact and serviceable. Inspection of the lower levels is ongoing.

Underground temperature is 58 degrees with 17% humidity. No ground water is present on any level and not located via testing to 4000’. Natural ventilation to the mine is 21,500cfm. In addition, a mechanical fan has been installed to boost ventilation.

Power to the mine is through LADWP transmission lines that were sold to LADWP in 1984 an existing 33,000-volt substation is on the mine site. Plans for supplemental solar and a potential cogeneration from sulfuric acid production is being considered.

Water Rights:

There is a state land lease in place and current for transmission line and pipelines from the mill site claims to the end of the state lands section that we are in, this adjoins BLM right of way CA8872 that carries transmission and pipeline to the mine site. Two water certificates: 0.32 cfs predating BLM/state regs and a 1964 support certificate. Perfected in 1998. Source is 5-acre mill site 4.5 miles from the mine. Two wells produce 160 GPM combined. Infrastructure includes 12,000-gal tanks, 125 HP motor, and 850 psi pipeline. Water trucks are used pending pipeline repair. 250,000 gallons are stored on-site.

Initial Production Plan: Initial mining occurs above 400 level, eliminating hoisting for early production. Hoisting, manways, and timber recertification is ongoing. Incline ore skip to be reactivated.

Haulage System: Radio Ore Tunnel (400 level) is main haulage—9,660 feet. Thompson hoist developed to 900 level: ore removal at 400. Defiance developed to 1300, hoist at 400. Incline winds with ore skip to 1000 with transfer at 400. Independence and exit pipes show ore potential from 800 level.

Primary Zones: Defiance 418 and Thompson 433-435, both mixed, zones will come online first. The Independence and Essex pipes trend downward and offer ore potential which will be accessed from the 800 level in the future.

Exploration and Development: The mine has a large porphyry system at depth that was drilled by Anaconda, encountering 188-foot interval at 2200’ depth. This is nine hundred feet below the lowest development at the 1300’ level of the Defiance. It is difficult to drill this due to the extensive underground workings. Thus by drifting eight hundred feet on the Defiance 800’ level towards the Thompson 800’ level we will come under the Bernon, Essex, and Independence ore bodies. Two hundred feet of drift in the opposite direction puts us under the ore in the Mickey Summers Fault and Water Tank Fissure. This drift will provide better ventilation, secondary escape, allow for multiple drill stations and allow the Thompson mine to access the Defiance incline ore skip.

The 474 fault presents a new drill target, drifting encountered brecciated calcite with galena and tellurite bismuthsite.

The work performed by Don Strachan shows high levels of Antimony in the Defiance and Bernon areas. Records show Antimony production in the Independence as well as the Darwin Antimony Mine below the Lucky Jim as well as the Promontory.

The Lucky Jim presents a five-acre bulk mineable target, this work should be expanded. Prior production 170,000 tons with an average gold grade of .78 with sixteen ounces of Silver. The Lucky Jim was burnt to the ground in 1928, retimbered by Anaconda in 1954. It has not been touched since.

The radio ore tunnel nine thousand six hundred and sixty feet, when extended three thousand three hundred feet would day light below the original Lucky Jim shaft. This would give us second egress and allow for expanded capacity up to 2000 tons per day. A production shaft in the competent ground of the Lucky Jim extension will allow for recovery of ore from depth.

The existing ore will last approximately eight years, there should be no problem with blocking out future ore reserves. Peter Hahn had a target of fifteen million metric tons at fifteen percent zinc. There is a drill plan to move the oxide and sulfide ore to measured, at a cost of $1.3 million. (Details included)

Introduction

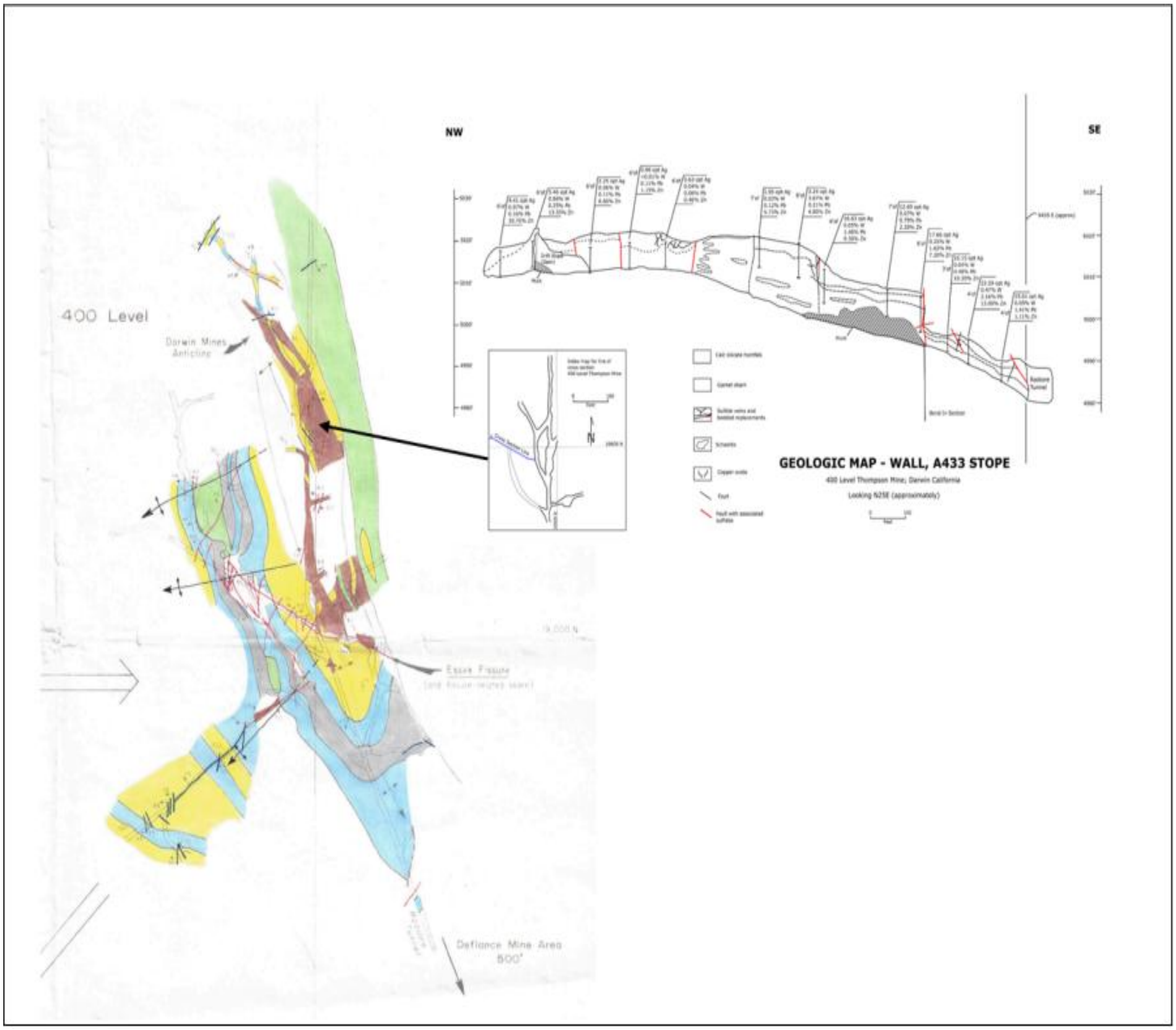
Ethos Geological has created a drill plan for the Darwin Mine in Inyo County, California. The goal of this plan is to identify enough material to sustain a mining operation for 7 years at a production rate of 500 tons per day. The A433 and 418D Oxide areas will be prioritized due to their high tonnage potential and strategic location, allowing access to other areas within the mine. Ethos and Jack Stone of Project Darwin have collaborated to ensure that the targeted areas are accessible for drilling with a core rig and that they also provide locations for collecting grade control samples using both long hole drills and rib sampling.

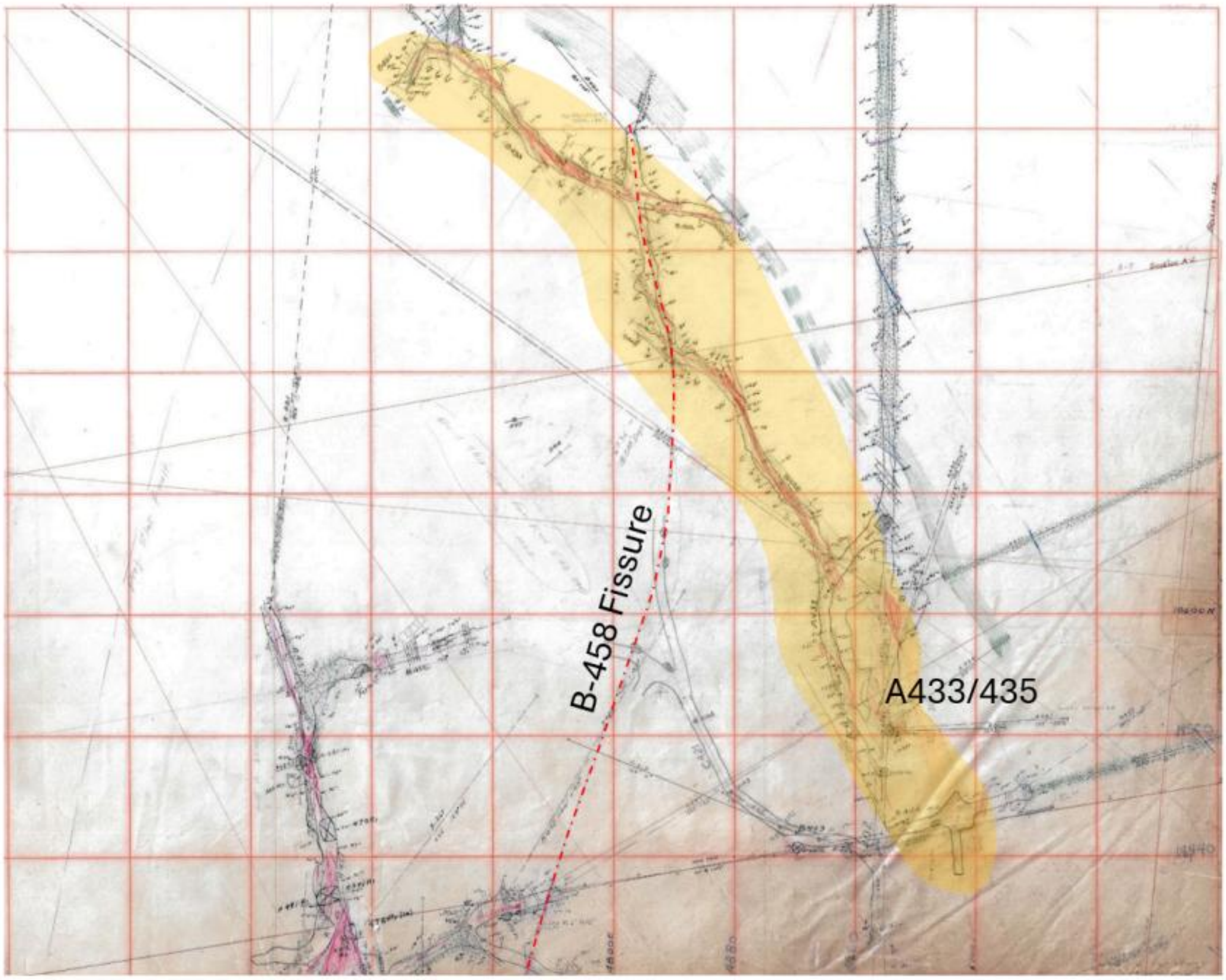
Project Description

The Darwin area's geological history includes sedimentary deposition, structural deformation, plutonic intrusions, and metamorphic transformations. The intrusion of the Darwin stock resulted in contact metamorphism and the formation of various minerals. Thrust faulting and the intrusion of granite porphyry dikes and breccia pipes further shaped the region's geology. Research suggests that tungsten (W), copper (Cu), and lead-zinc (Pb-Zn) skarn deposits in the area have distinct origins, indicating separate formation processes. The presence of three different igneous systems and diverse calc-silicate/ore assemblages supports this model. Vein ores may be related to skarn-forming episodes, and the rare co-occurrence of Pb-Zn and W skarns reflects unique geological conditions and plutonic events.

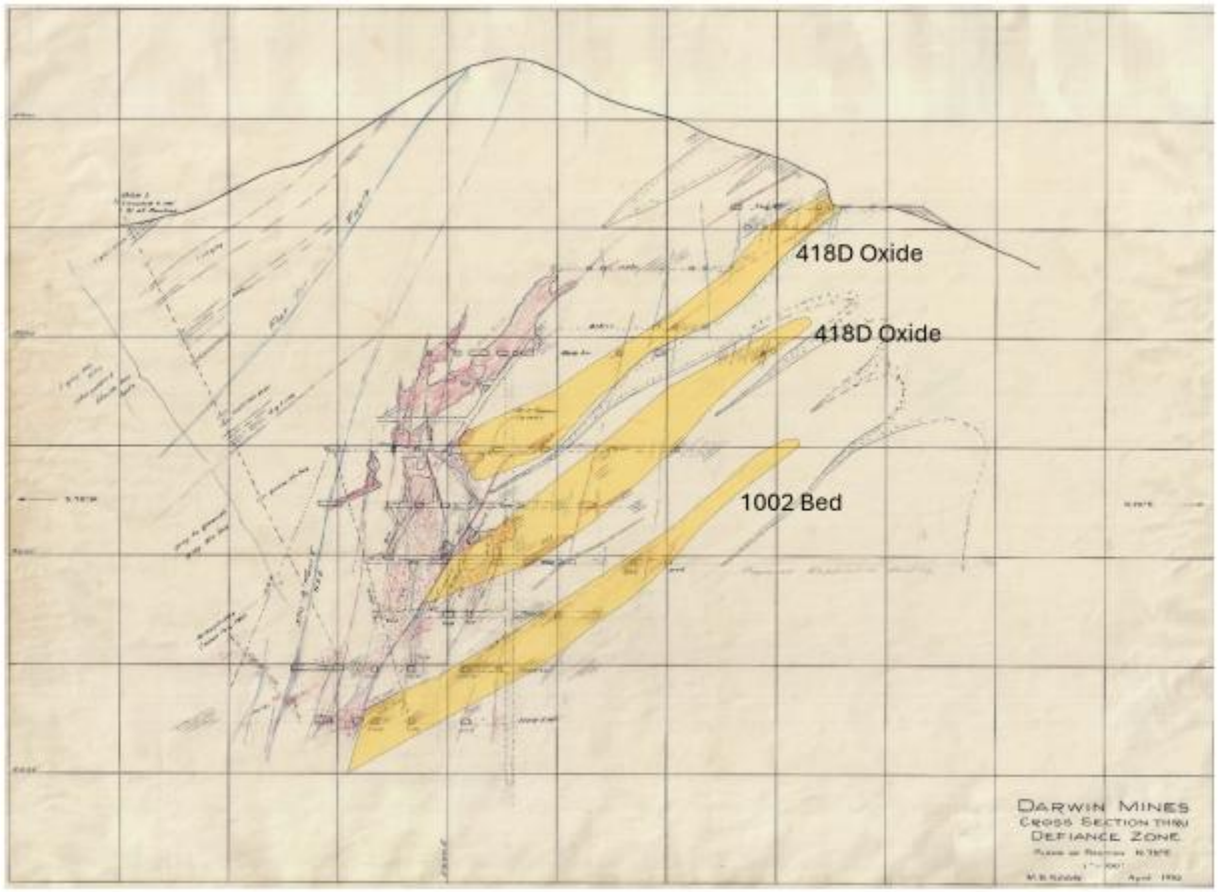
Exploration Targets

A433 Area the A433/435 stope is situated on the eastern limb of an anticline and is connected to the 400L main tunnel. The mineralization within this stope is predominantly oxidized silver and zinc, with locally significant scheelite occurrences. Detailed sampling conducted by Anaconda and confirmed by Blue Range has established an estimated volume of 1,728,000 cubic feet for this stope. Average sample grades are 8.5 opt Ag, 0.8% Pb, 6.4% Zn, and 0.6% WO_3. The A433/435 stope has been a major source of tungsten, and mineralization has been observed to extend northwest towards the 458 fissure. Rib samples will be collected along the B446 to B458 drift, spanning approximately 240 feet. Within the same drift, long holes will be drilled into the anticline limb on 50-foot centers, with cuttings retained for analysis. Additionally, two drill stations will be established to complete core holes through both limbs of the anticline.


400 Level
Darwin Mines Anticline
Estrk Fissure
Defiance Mine Area 500'
NW
SE
GEOLOGIC MAP - WALL, A433 STOPE
400 Level Thompson Mine; Darwin California
Looking N25E (approximately)
Bend in Section
Raiboré Tunnel



The B-458 fissure represents the downdip extension along the B-458 fissure on the 400L, exhibiting continuous mineralization with the A433/435 stope, but under stronger structural control. This zone is partially oxidized, with drilling indicating average grades of 9.3 opt Ag, 0.9% Pb, and 8.2% Zn. The estimated volume is 1,197,000 cubic feet, based on extensive drilling and historical production data. 418D Oxide Area The 418D Oxide Shoot is primarily exposed on the 400 level within the Defiance mine, situated between the 418 and 450D crosscuts, and can be identified down to the 1000 level. The volume of this shoot is estimated at 4,999,689 cubic feet, with average sample grades of 10.6 opt Ag and 8.0% Zn. Rib samples will be collected along the 462, 418, and 419 drifts, spanning approximately 180 feet. Within the same drift, a winkie drill will be utilized to intersect the 418D and 1002 Beds from the Defiance area. Additional drill stations will be established along the Radio ore tunnel to confirm mineralization along the strike of the beds.


418D Oxide
418D Oxide
1002 Bed
DARWIN MINES
CROSS SECTION THRU
DEFIANCE ZONE

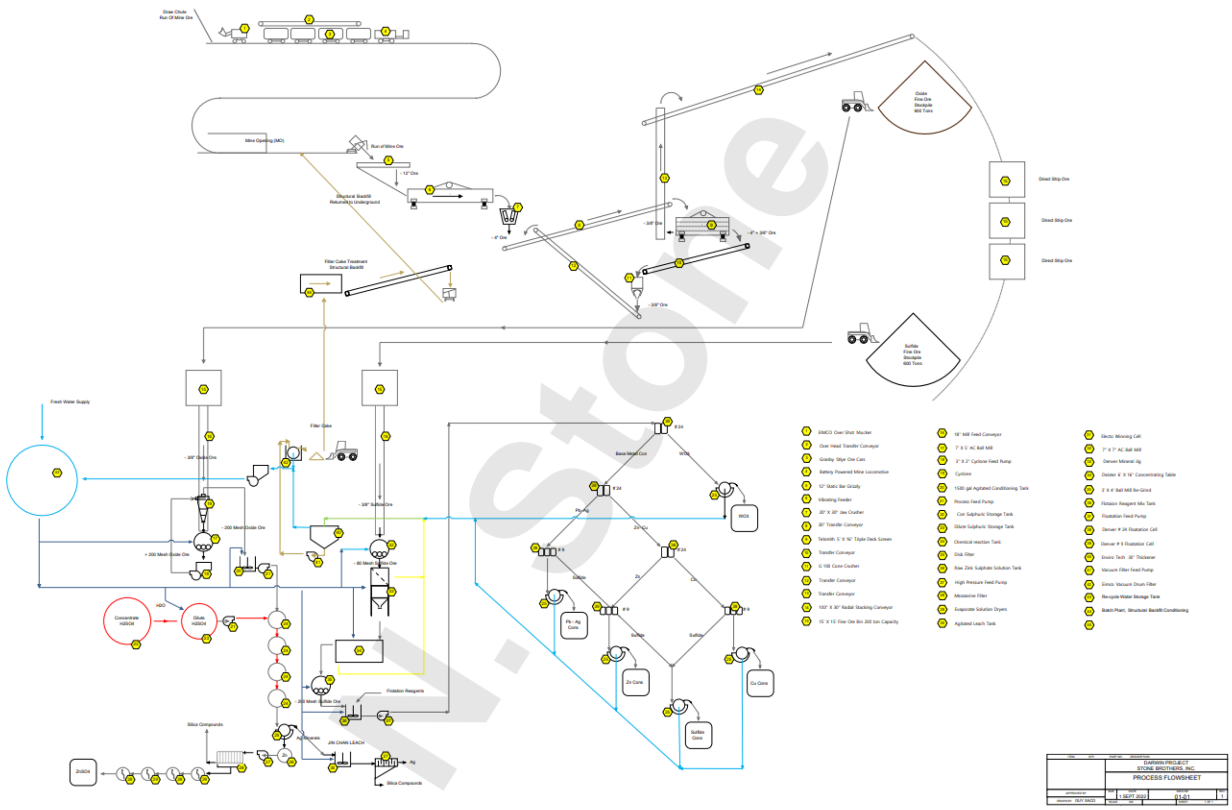
Mining methods/ Geological Context

Mining Methods: Mining will be done via conventional mining methods, best for high grade deposits. Jack legs, slushers, muckers, slick sheets, ore chutes, hoists and blasting methods will be the primary applications.

Geological Context: The 418-Defiance oxide bed dips at 28 degrees, is 120 feet wide, and varies in thickness from 39 feet (west) to 60 feet (east). It outcrops above the 160 level as a sulfide and includes occasional sulfide remnants. This bed has been touched by drilling at the 1000 level, with a potential of over 1.5 million tons. The oxide zone is soft and suitable for slushing with minimal blasting. Mining utilizes slusher buckets with double teeth and side rippers, bolted slusher cages, and conveyors feeding 60 cubic foot, 4.5-ton rail cars transported via upgraded battery locomotives on 40-pound rail to the crusher feed hopper.

433-435 Stope: Discovered in 1975, opposite the Radio Ore Tunnel, this area has produced 69,500 tons. Two ore beds are separated by a 15-foot bedrock. The upper lens is 10 feet thick, the lower 15 feet. Ore follows the anticline hinge like an inverted soup bowl. The first bed lies 20 feet above haulage. The ore continues to the 474 fault (north) and 475 fault (west). Mined using a modified room and pillar method with an undercut to accommodate limited slushing distance.

Transport: Ore dumped into 60 cubic foot (with 2.4 specific gravity) will result in 4.5-ton payload per car transported via upgraded battery locomotives on 40-pound rail to the crusher feed hopper. Locomotives will deliver 28.8 tons per trip.



Mineral Processing/Metallurgy

Crushing and milling

Crusher and Ore Handling: Ore is dumped into a feed hopper with vibrating and static grizzly. Oversize is reduced in a 20x30 Telsmith jaw crusher and sent to a 5x16 3-deck screen. Oversize goes to a GP100 Nordberg cone crusher in closed circuit. Final product is stacked on pads or directed to a 300-ton truck loadout bin.

Dust Suppression: All systems are permitted and utilize sonic tip suppression.

Mill Operations: Both oxide and sulfide mills are on-site.

Sulfide Mill: Processes 300 tons/day starting from a 300-ton fine ore bin with VFD rotary gate feeder. A 50-ft by 18-in conveyor feeds a 7x7 Allis-Chalmers open circuit ball mill. Product goes to a Denver mineral jig, with hutch to Diester table (3-day). The jig tailings are re-ground in a 5x5 mill to 200 mesh. Conditioned with sodium silicate, PAX, and 404, the bulk sulfide concentrate is floated. Tailings, including tungsten, are floated using red heart reagent. Differential flotation separates Pb-Zn-Cu.

Tailings circuit: Tailings are thickened, pressed to cake, and combined with limestone and cement. This neutralizes acidity and stabilizes for backfilling in underground voids using 640 EIMCO track muckers. Water is recycled.

Mining Operations: 50 tons/manshift benchmark. A roundhouse facilitates battery charging with an overhead lift. A crew of 14 miners and 14 helpers targets 700–750 tons/day, with split between oxide and sulfide. Three-shift operation ensures 24/7 milling. Tramming uses 5, 4.5, and 1.5-ton locomotives for long hauls and 3.5-ton Atlas Copco locomotives for shorter oxide runs. Tailing placement occurs on the second shift.

Oxide Mill: Starts with a 300-ton bin and rotary feeder to a 50-ft by 18-in conveyor, feeding a 7x5 Allis-Chalmers ball mill. Cyclone classifies to 80 mesh. Discharge to 2500-gal precoat tank before first-stage sulfuric acid leach. Two parallel cascading tanks with 389 RPM agitation ensure 98.7% Zn leaching. JCI filter press dewaters to cake. Pregnant solution goes to MDS membrane filter, clarified, then stored in 3000-gal tanks. Zinc sulfate is recovered via electrowinning and dried to monohydrate flake.
Filter Cake Leaching: Cake is leached with thiosulfate in six agitated tanks for one-hour reagent time. Multiple metals go into solution: Ag, Ge, Te, Co, Ni, Cd, Cu, Pb. They then enter EW electrowinning seen below. Indium and gallium are recovered using resin columns.

Tailings circuit: Tailings are thickened, pressed to cake, and combined with limestone and cement. This neutralizes acidity and stabilizes for backfilling in underground voids using 640 EIMCO track muckers. Water is recycled.

Below is 3rd party support for, Floatation testing, electrowinning and dryer, The RDI report, and the zinc leach by Kappes, Cassiday & Associates.

Floatation testing (Sulfide)

Key Observations:

- **Gold and Silver**:
    - The composite contains approximately **0.021 oz/ton gold** and **17.5 oz/ton silver**.
    - Across all tests, **gold recovery ranged from 75–88%** and **silver recovery from 83–90%**, with most precious metals reporting primarily to the lead concentrates.
- **Lead Flotation**:
    - Lead recovery improved with finer grinding (P80 = 75 µm) and more pulls in Flotation 2.
    - Switching from Aerofloat 7310 to Aero 3418A (Flotation 3) helped clean up the froth somewhat but **did not significantly reduce copper contamination** in the lead stage.
- **Copper Flotation**:

    - Copper remains challenging to separate cleanly. Much of the copper continues to report to the lead concentrate, suggesting close textural association with galena.
- **Zinc Flotation**:
    - Increasing zinc sulfate helped shift more zinc into the zinc flotation stage.
- **Mineralogical Analysis**:
    - The material is dominated by **garnet (41%)** and **sphalerite (26%)**, with minor **pyrite, galena (<3%)**, and **chalcopyrite (<2%)**.
    - Presence of **anglesite** suggests partial oxidation of galena, which could impact lead flotation behavior.
    - High silicate content could affect flotation mass pull and selectivity.

Conclusions and Recommendations:

- The flowsheet may benefit from a revised sequence: **lead flotation → zinc flotation → copper cleaning from Pb concentrate**.
- Sodium sulfite addition in the Pb stage did not sufficiently suppress copper. **Sodium metabisulfite** is on order and will be tested if additional work is requested.
- The mineralogical findings confirm why separation between lead and copper has been difficult **—** copper appears to be **texturally associated** with lead minerals rather than occurring as discrete free chalcopyrite.

April 28, 2025
Lab no. 225073

Ms. Jessica Link
Woods Process Services LLC
P.O. Box 20897
Reno, Nevada 89515

Dear Ms. Link:

Enclosed are the x-ray fluorescence (XRF) and the x-ray diffraction (XRD) results for sample "1096 HS" received last week. This report will be mailed and emailed to you.

A representative portion of the sample was ground to approximately -400 mesh in a steel swing mill and then analyzed by our standard XRF procedure for 31 major, minor and trace elements. The relative precision/accuracy for this procedure is ~5-10% for major-minor elements and ~10–15% for trace elements (those elements listed in ppm) at levels greater than twice the detection limit in samples of average geologic composition. A replicate sample and a standard reference material ("GSP-2", a USGS standard rock) were analyzed with the sample to demonstrate analytical reproducibility for your sample and analytical accuracy for a geologic standard, respectively. The accepted ("known") values for the quality control standard are listed with the XRF results.

A representative portion of the ground sample was packed into a well-type plastic holder and then scanned with the diffractometer over the range, 3-61°2θ using Cu-Kα radiation. The results of the scan are summarized as approximate mineral weight percent concentrations on the enclosed table. Estimates of mineral concentrations were made using our XRF-determined elemental composition and the relative peak areas on the XRD scan.The detection limit for an average mineral in this sample is ~1-3% and the analytical reproducibility is approximately equal to the square root of the amount. "Unidentified" accounts for that portion of the scan which could not be resolved and a "?" indicates doubt in both mineral identification and amount.

Thank you for the opportunity to be of service to Woods Process Services.

Sincerely,

Peggy Dalheim

Woods Process Services — **April 25, 2025**
XRF Results for Sample, "1096 HS" — **Lab no. 225073**

	Wt %												
IDENT	**Na_2O**	**MgO**	**Al_2O_3**	**SiO_2**	**P_2O_5**	**S**	**Cl**	**K_2O**	**CaO**	**TiO_2**	**MnO**	**Fe_2O_3**	**BaO**
1096 HS	0.27	1.97	3.33	24.3	0.20	12.0	< 0.02	0.12	21.9	0.24	0.59	11.9	0.01
Quality Control - Replicate (R) sample and standard reference material (GSP-2) analyzed with sample													
1096 HS(R)	0.23	1.95	3.33	24.3	0.20	12.0	< 0.02	0.12	21.9	0.24	0.59	11.9	0.04
GSP-2-XRF	3.11	1.12	14.6	66.2	0.33	0.07	0.06	5.38	2.10	0.59	0.04	4.37	0.14
GSP-2-known	2.78	0.96	14.9	66.6	0.29	----	----	5.38	2.10	0.66	0.04	4.90	0.15

	PPM												
IDENT	**V**	**Cr**	**Co**	**Ni**	**W**	**Cu**	**Zn**	**As**	**Sn**	**Pb**	**Mo**	**Sr**	**U**
1096 HS	40	309	86	149	< 10	2270	156000	< 20	< 20	27000	< 10	24	< 10
Quality Control													
1096 HS(R)	52	298	76	155	< 10	2280	156000	28	< 20	27000	< 10	23	< 10
GSP-2-XRF	41	23	< 10	12	< 10	44	103	< 20	< 20	33	< 10	215	< 10
GSP-2-known	52	20	7	17	--	43	120	--	--	42	--	240	2

	PPM				
Ident	**Th**	**Nb**	**Zr**	**Rb**	**Y**
1096 HS	< 10	< 10	49	< 10	< 10
Quality Control					
1096 HS(R)	< 10	< 10	48	< 10	< 10
GSP-2-XRF	100	21	509	208	30
GSP-2-known	105	27	550	245	28

Initial_____

Date_______

Analysis Performed By The Mineral Lab, Inc

Woods Process Services LLC
XRD Results for Sample “1096 HS”

April 28, 2025
Lab no. 225073

Mineral Name	Chemical Formula	Approx. Wt %
Sphalerite	ZnS	26
Pyrite	FeS_2	5
Galena	PbS	<3
Chalcopyrite	$CuFeS_2$	<2?
Anglesite	$PbSO_4$	<3
Gypsum	$CaSO_4 \bullet 2H_2O$	<3?
Garnet	$(Ca,Fe,Mg,Mn)_3(Al,Fe)_2(SiO_4)_{3-x}(OH)_{2x}$	41
Clinopyroxene	$Ca(Mg,Fe,Mn,Al,Ti)(Si,Al)_2O_6$	11
Quartz	SiO_2	6
“Unidentified”	?	<3

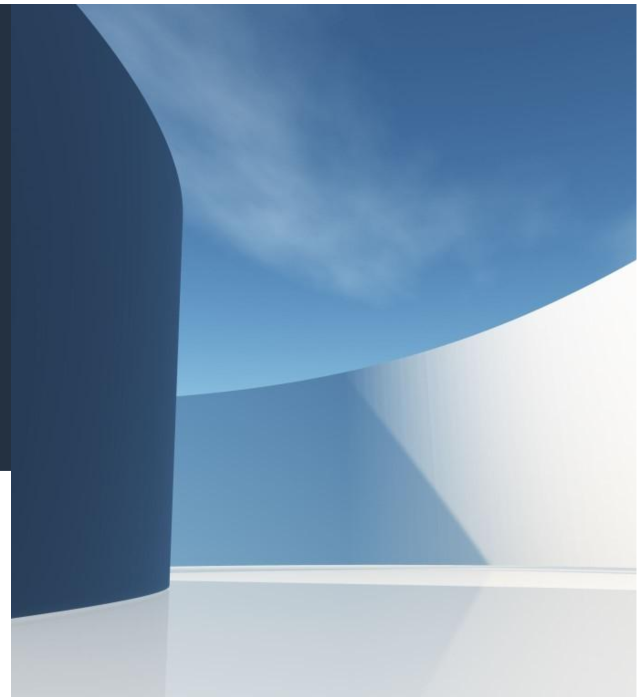
Initial ________

Date ________

Analysis performed by The Mineral Lab, Inc

1096 Darwin Flotation Update

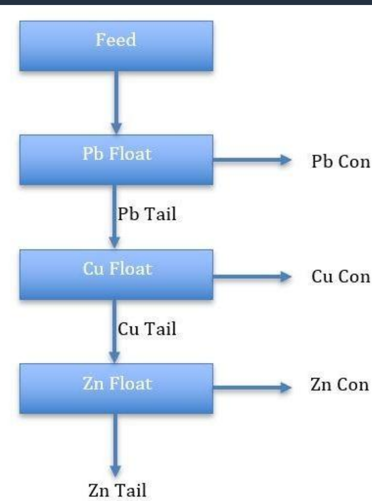
04/03/2025



WOODS PROCESS SERVICES, LLC

Goals

- Get 3 separate concentrates
  - Lead
  - Copper
  - Zinc
- Avoid using NaCN



Flotation 1

- Grind time 30 minutes
- P80 = 106µm
- MIBC for frother
- Milk of Lime for pH control

- Lead Float
  - 1 rougher for 5 minutes
  - 1 scavenger for 3 minutes
  - ~40 g/mt Aerofloat 7310
  - ~100 g/mt Zinc Sulfate
  - Target pH 9.5
- Copper Float
  - 1 rougher for 5 minutes
  - 1 scavenger for 3 minutes
  - ~40 g/mt Aero 3894
  - ~20 g/mt Aerofloat 238
  - Target pH 9
- Zinc Float
  - 1 rougher for 5 minutes
  - 1 scavenger for 3 minutes
  - ~750 g/mt Copper Sulfate
  - ~50 g/mt PAX
  - Target pH 11-11.5

Flotation 1 – Assays

- Each concentrate pull was kept and analyzed separately.
- The Pb Con 1 and 2 are combined **mathematically** to see how much total mass and metal was pulled in the 5 minute rougher pull and the 3 minute scavenger pull.
- Likewise for Cu Con 1 and 2 and Zn Con 1 and 2
- Sep. Wt % = Separate Weight Percent… combining the rougher and scavenger cons together by element by stage
- Cum. Wt. % = Cumulative Weight Percent is running total from all the pulls

Weight grams	Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	% Cu	% Zn
85.0	Pb Con 1	8.1	8.1	8.1	11.15	1.15	24.9
47.0	Pb Con 2	4.5	12.6	12.6	11.98	0.58	45.7
41.0	Cu Con 1	3.9	3.9	16.5	5.54	0.49	53.5
22.0	Cu Con 2	2.1	6.0	18.6	3.76	0.37	49.6
178.0	Zn Con 1	17.0	17.0	35.5	0.51	0.17	62.8
33.0	Zn Con 2	3.1	20.1	38.7	2.11	0.26	19.8
644.0	Tail	61.3		100.0	1.09	0.16	2.9
1050.00	Composite	100.0			2.56	0.28	20.2
				Assayed Head:	3.54	0.21	21.5

Mass Percentages

- 8.1 % of the mass was collected in Pb Con 1
- 12.6 % of the mass was collected in Pb Con 1 and 2
- 17.0% of the mass was collected in Zn Con 1
- 20.1 % of the mass was collected in Zn Con 1 and 2
- 38.7 % of the mass was collected in all the cons.

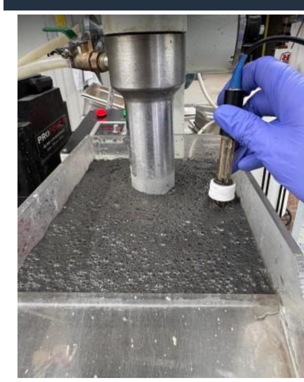
Flotation 1 – Lead

- Lead Stage
  - The 5 min rougher (Pb Con 1) pulled 35.3 % of the Pb.
  - The 3 min scavenger (Pb Con 2) pulled 21.0% of the Pb.
  - Combined they pulled 56.3 % of the Pb **in only 12.6% of the mass percent**
- Copper Stage
  - 11.5% of the Pb was collected in the Copper 5 min rougher and 3 min scavenger (Cu Con 1 and Cu Con 2)
- Zinc Stage
  - 6.0% of the Pb was collected in the Zn Con 1 and Zn Con 2.

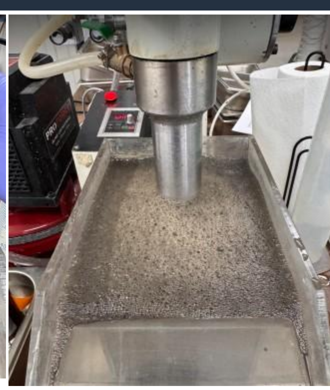
	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution			
Product	%	%	%	%	Sep. Cum %	Cum. %	Sep. Grade % Pb
Pb Con 1	8.1	8.1	8.1	35.3%	35.3%	35.3%	11.15
Pb Con 2	4.5	12.6	12.6	21.0%	56.3%	56.3%	11.45
Cu Con 1	3.9	3.9	16.5	8.5%	8.5%	64.7%	1.31
Cu Con 2	2.1	6.0	18.6	3.1%	11.5%	67.8%	1.59
Zn Con 1	17.0	17.0	35.5	3.4%	3.4%	71.2%	0.24
Zn Con 2	3.1	20.1	38.7	2.6%	6.0%	73.8%	0.40
Tail	61.3		100.0	26%		100%	2.56
Composite	100.0			100%			

- Total
  - 73.8% of the lead was recovered in the 3 stages
  - The lead grade is high
  - For Flotation 2 will have 4 pulls instead of 2 to try to recover more lead up front in the lead con.

Flotation 1 – Lead



Pre-Pb Con 1
Full Visible Sulfides



Post-Pb Con 1
Some Visible Sulfides

Pre-Pb Con 2
Some Visible Sulfides

Post-Pb Con 2
Some Visible Sulfides

Flotation 1 – Copper

Product	Wt. %	Separate Wt. %	Cum. Wt. %	Pb Distribution %	Pb Distribution Sep. Cum. %	Pb Distribution Cum. %	Pb Distribution Sep. Grade % Pb	Cu Distribution %	Cu Distribution Sep. Cum. %	Cu Distribution Cum. %	Cu Distribution Sep. Grade % Cu
Pb Con 1	8.1	8.1	8.1	35.3%	35.3%	35.3%	11.15	33.3%	33.3%	33.3%	1.15
Pb Con 2	4.5	12.6	12.6	21.0%	56.3%	56.3%	11.45	9.2%	42.5%	42.5%	0.94
Cu Con 1	3.9	3.9	16.5	8.5%	8.5%	64.7%	1.31	6.8%	6.8%	49.3%	0.12
Cu Con 2	2.1	6.0	18.6	3.1%	11.5%	67.8%	1.59	2.8%	9.6%	52.1%	0.14
Zn Con 1	17.0	17.0	35.5	3.4%	3.4%	71.2%	0.24	10.3%	10.3%	62.4%	0.08
Zn Con 2	3.1	20.1	38.7	2.6%	6.0%	73.8%	0.40	3.0%	13.3%	65.4%	0.10
Tail	61.3		100.0	26%		100%	2.56	35%		100%	0.28
Composite	100.0			100%				100%			

- Lead Stage
  - 33.3% of the copper was collected in Pb Con 1
  - 9.2% of the copper was collected in Pb Con 2
  - Total 42.5 % of the copper was pulled in the lead stage
- Copper Stage
  - 6.8% of the copper was collected in Cu Con 1
  - 2.8% of the copper was collected in Cu Con 2
  - Only 9.6% of the copper was pulled in the copper stage
- Zinc Stage
  - 10.3% of the copper was collected in Zn Con 1
  - 13.3% of the copper was collected in Zn Con 2
  - 13.3% of the copper was pulled in the zinc stage

Flotation 1 – Copper

Product	Wt. %	Separate Wt. %	Cum. Wt. %	Pb Distribution %	Pb Distribution Sep. Cum. %	Pb Distribution Cum. %	Pb Distribution Sep. Grade % Pb	Cu Distribution %	Cu Distribution Sep. Cum. %	Cu Distribution Cum. %	Cu Distribution Sep. Grade % Cu
Pb Con 1	8.1	8.1	8.1	35.3%	35.3%	35.3%	11.15	33.3%	33.3%	33.3%	1.15
Pb Con 2	4.5	12.6	12.6	21.0%	56.3%	56.3%	11.45	9.2%	42.5%	42.5%	0.94
Cu Con 1	3.9	3.9	16.5	8.5%	8.5%	64.7%	1.31	6.8%	6.8%	49.3%	0.12
Cu Con 2	2.1	6.0	18.6	3.1%	11.5%	67.8%	1.59	2.8%	9.6%	52.1%	0.14
Zn Con 1	17.0	17.0	35.5	3.4%	3.4%	71.2%	0.24	10.3%	10.3%	62.4%	0.08
Zn Con 2	3.1	20.1	38.7	2.6%	6.0%	73.8%	0.40	3.0%	13.3%	65.4%	0.10
Tail	61.3		100.0	26%		100%	2.56	35%		100%	0.28
Composite	100.0			100%				100%			

- 65.4% of the copper was recovered in all the stages.
- Too much copper reported to the lead and zinc stages and not enough in the copper stage
- Sodium Sulfite could be used in combination with zinc sulfate to depress sphalerite (Zn) and chalcopyrite (Cu) in the lead stage
- Will increase dosage of Aero 3894 to try to collect more copper in the copper stage
- Will take solution samples before and after each stage to analyze for copper in solution

Flotation 1 – Copper


Pre-Cu Con 1
Some Visible Sulfides


Post-Cu Con 1
Some Visible Sulfides


Pre-Cu Con 2
Some Visible Sulfides


Post-Cu Con 2
Very Little Visible Sulfides

Flotation 1 – Zinc

	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution				Cu Distribution				Zn Distribution			
Product	%	%	%	%	Sep. Cum. %	Cum. %	Sep. Grade % Pb	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn
Pb Con 1	8.1	8.1	8.1	35.3%	35.3%	35.3%	11.15	33.3%	33.3%	33.3%	1.15	10.0%	10.0%	10.0%	24.90
Pb Con 2	4.5	12.6	12.6	21.0%	56.3%	56.3%	11.45	9.2%	42.5%	42.5%	0.94	10.1%	20.1%	20.1%	32.30
Cu Con 1	3.9	3.9	16.5	8.5%	8.5%	64.7%	1.31	6.8%	6.8%	49.3%	0.12	10.3%	10.3%	30.4%	12.67
Cu Con 2	2.1	6.0	18.6	3.1%	11.5%	67.8%	1.59	2.8%	9.6%	52.1%	0.14	5.1%	15.4%	35.5%	16.83
Zn Con 1	17.0	17.0	35.5	3.4%	3.4%	71.2%	0.24	10.3%	10.3%	62.4%	0.08	52.6%	52.6%	88.1%	29.96
Zn Con 2	3.1	20.1	38.7	2.6%	6.0%	73.8%	0.40	3.0%	13.3%	65.4%	0.10	3.1%	55.7%	91.2%	29.13
Tail	61.3		100.0	26%		100%	2.56	35%		100%	0.28	9%		100%	20.24
Composite	100.0			100%				100%				100%			

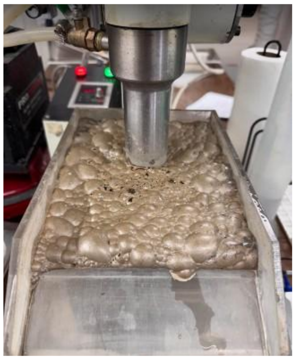
- Lead
  - 10% of the zinc was collected in Pb Con 1
  - 10.1% of the zinc was collected in Pb Con 2
  - Total 20.1 % of the copper was pulled in the lead stage
- Copper
  - 10.3% of the zinc was collected in Cu Con 1
  - 5.1% of the zinc was collected in Cu Con 2
  - 15.4% of the zinc was pulled in the copper stage
- Zinc
  - 52.6% of the zinc was collected in Zn Con 1
  - 3.1% of the zinc was collected in Zn Con 2
  - 55.7% of the zinc was pulled in the zinc stage!

Flotation 1 – Zinc

Product	Wt. %	Separate Wt. %	Cum. Wt. %	Pb Distribution %	Pb Sep. Cum. %	Pb Cum. %	Sep. Grade % Pb	Cu Distribution %	Cu Sep. Cum. %	Cu Cum. %	Sep. Grade % Cu	Zn Distribution %	Zn Sep. Cum. %	Zn Cum. %	Sep. Grade % Zn
Pb Con 1	8.1	8.1	8.1	35.3%	35.3%	35.3%	11.15	33.3%	33.3%	33.3%	1.15	10.0%	10.0%	10.0%	24.90
Pb Con 2	4.5	12.6	12.6	21.0%	56.3%	56.3%	11.45	9.2%	42.5%	42.5%	0.94	10.1%	20.1%	20.1%	32.30
Cu Con 1	3.9	3.9	16.5	8.5%	8.5%	64.7%	1.31	6.8%	6.8%	49.3%	0.12	10.3%	10.3%	30.4%	12.67
Cu Con 2	2.1	6.0	18.6	3.1%	11.5%	67.8%	1.59	2.8%	9.6%	52.1%	0.14	5.1%	15.4%	35.5%	16.83
Zn Con 1	17.0	17.0	35.5	3.4%	3.4%	71.2%	0.24	10.3%	10.3%	62.4%	0.08	52.6%	52.6%	88.1%	29.96
Zn Con 2	3.1	20.1	38.7	2.6%	6.0%	73.8%	0.40	3.0%	13.3%	65.4%	0.10	3.1%	55.7%	91.2%	29.13
Tail	61.3		100.0	26%		100%	2.56	35%		100%	0.28	9%		100%	20.24
Composite	100.0			100%				100%				100%			

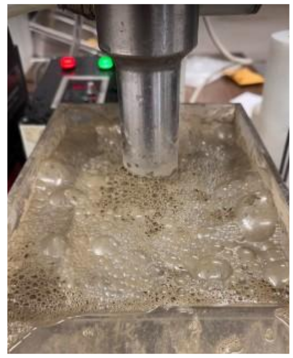
- 91.2% of the zinc was recovered in all the stages
- Too much zinc was pulled in the lead and copper stages – zinc could be tied up with the lead and copper minerals. Finer grinding will be tested in Flotation 2 to try and liberate the zinc separate from the lead and copper.
- Will adjust reagents as previously mentioned to clean up the lead and copper cons and see how the zinc responds in the zinc stage.
- Will increase the zinc sulfate in the lead stage and add it to the copper stage to depress zinc.

Flotation 1 – Zinc



Pre-Zn Con 1
Great PAX froth

Post-Zn Con 1



Pre-Zn Con 2
Not enough Zn for PAX to float

Post-Zn Con 2

Flotation 1 – Gold

	Pb Distribution				Cu Distribution				Zn Distribution				Au Distribution			
Product	%	Sep. Cum %	Cum. %	Sep. Grade % Pb	%	Sep. Cum %	Cum. %	Sep. Grade % Cu	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn	%	Sep. Cum. %	Cum %	Cum g/mt Au
Pb Con 1	35%	35.3%	35.3%	11.15	33%	33.3%	33.3%	1.15	10%	10.0%	10.0%	24.90	44%	43.7%	43.7%	3.73
Pb Con 2	21%	56.3%	56.3%	11.45	9%	42.5%	42.5%	0.94	10%	20.1%	20.1%	32.30	11%	55.0%	55.0%	3.02
Cu Con 1	8%	8.5%	64.7%	1.31	7%	6.8%	49.3%	0.12	10%	10.3%	30.4%	12.67	8%	7.5%	62.5%	2.62
Cu Con 2	3%	11.5%	67.8%	1.59	3%	9.6%	52.1%	0.14	5%	15.4%	35.5%	16.83	4%	11.2%	66.1%	2.46
Zn Con 1	3%	3.4%	71.2%	0.24	10%	10.3%	62.4%	0.08	53%	52.6%	88.1%	29.96	7%	6.5%	72.7%	1.41
Zn Con 2	3%	6.0%	73.8%	0.40	3%	13.3%	65.4%	0.10	3%	55.7%	91.2%	29.13	4%	10.2%	76.3%	1.36
Tail	26%		100%	2.56	35%		100%	0.28	9%		100%	20.24	24%		100.0%	0.69
Composite	100%				100%				100%				100%			

- 76.3% of the gold was recovered in all the stages
- 55.0% of the gold reported to the lead stage

Flotation 1 – Silver

Product	Cu Distribution %	Sep. Cum %	Cum %	Sep. Grade % Cu
Pb Con 1	33%	33.3%	33.3%	1.15
Pb Con 2	9%	42.9%	42.5%	0.94
Cu Con 1	7%	6.8%	49.3%	0.12
Cu Con 2	3%	9.6%	52.1%	0.14
Zn Con 1	10%	10.3%	62.4%	0.08
Zn Con 2	3%	13.3%	65.4%	0.10
Tail	35%		100%	0.28
Composite	100%			

Product	Zn Distribution %	Sep. Cum %	Cum %	Sep. Grade % Zn
Pb Con 1	10%	10.0%	10.0%	24.90
Pb Con 2	10%	20.1%	20.1%	32.30
Cu Con 1	10%	10.3%	30.4%	12.67
Cu Con 2	5%	15.4%	35.5%	16.83
Zn Con 1	53%	52.6%	88.1%	29.96
Zn Con 2	3%	55.7%	91.2%	29.13
Tail	9%		100%	20.24
Composite	100%			

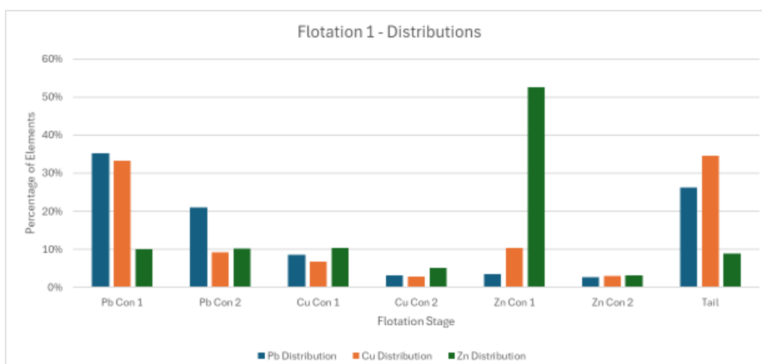
Product	Au Distribution %	Sep. Cum. %	Cum. %	Cum g/mt Au
Pb Con 1	44%	43.7%	43.7%	3.73
Pb Con 2	11%	55.0%	55.0%	3.02
Cu Con 1	8%	7.5%	62.5%	2.62
Cu Con 2	4%	11.2%	66.1%	2.46
Zn Con 1	7%	6.5%	72.7%	1.41
Zn Con 2	4%	10.2%	76.3%	1.36
Tail	24%		100.0%	0.69
Composite	100%			

Product	Ag Distribution %	Sep. Cum. %	Cum. %	Cum. g/mt. Ag
Pb Con 1	47.7%	47.7%	47.7%	3662.18
Pb Con 2	16.7%	64.4%	64.4%	3185.29
Cu Con 1	7.5%	7.5%	71.8%	2712.58
Cu Con 2	2.8%	10.3%	74.6%	2500.73
Zn Con 1	5.2%	5.2%	79.8%	1397.83
Zn Con 2	2.9%	8.0%	82.7%	1330.25
Tail	17.3%		100.0%	622.15
Composite	100%			

- 82.7 % of the silver was recovered in all the stages
- 64.4 % of the silver reported to the lead stage

Flotation 1 – Distributions

- Finer grinding can hopefully get the elements out of tails and into the cons
- Adjust reagents to get cleaner cons for each stage

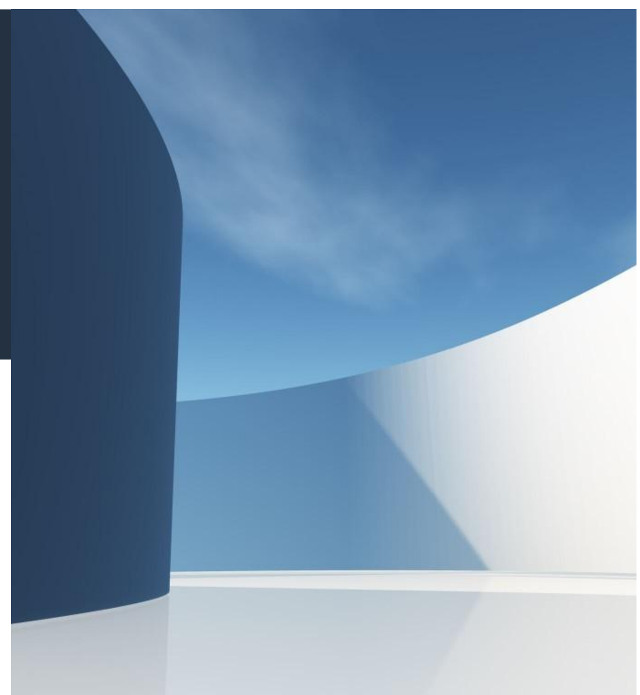


Conclusions & Plan for Flotation 2

- Lead and Zinc floats did ok for the first pass
- Too much copper floated in the lead and zinc stages
- Would like to decrease the amount of zinc reporting to lead and copper stages

- Will grind to 75μm to see if copper and zinc can be liberated from the lead minerals
  - Would strongly suggest reconsidering getting a mineralogy test done to see what forms these elements are in to make better reagent selections.
  - Finer grind will hopefully help reduce the about of lead and copper in the tails too.
- Will increase the number of pulls in the lead stage to collect more up front to try and eliminate lead from reporting to the Cu and Zn concentrates
- Will add sodium sulfite to depress the copper in the lead stage
- Will collect solution samples before and after each stage to see if the copper is activated and in solution or if it's in a mineral form (i.e. chalcopyrite)
- Will increase zinc sulfate in the lead stage and add it to the copper stage to depress more zinc.

1096 Darwin Flotation Update



04/14/2025

WOODS PROCESS SERVICES, LLC

Flotation 2

- Grind time 50 minutes
- P80 = 75µm
- MIBC for frother
- Milk of Lime for pH control

- Lead Float
  - 4 roughers for 5 minutes
  - ~80 g/mt Aerofloat 7310
  - ~2000 g/mt Zinc Sulfate
  - ~100 g/mt Sodium Sulfite
  - Target pH 9.5
- Copper Float
  - 1 rougher for 5 minutes
  - 1 scavenger for 3 minutes
  - ~40 g/mt Aero 3894
  - ~20 g/mt Aerofloat 238
  - Target pH 9
- Zinc Float
  - 1 rougher for 5 minutes
  - 1 scavenger for 3 minutes
  - ~750 g/mt lead Sulfate
  - ~50 g/mt PAX
  - Target pH 11-11.5

Flotation 2 - Assays

- Each concentrate pull was kept and analyzed separately.
- The Pb Con 1 -4 are combined **mathematically** to see how much total mass and metal was pulled in each 5 minute pull.
- Cu and Zn both had a 5 minute rougher and 3 minute scavenger pulls. Likewise combined mathematically.
- Sep. Wt % = Separate Weight Percent… combining the rougher and scavenger cons together by element by stage
- Cum. Wt. % = Cumulative Weight Percent is running total from all the pulls

Weight grams	Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	% Cu	% Zn
88.0	Pb Con 1	8.6	8.6	8.6	19.85	0.93	44.0
81.0	Pb Con 2	7.9	16.4	16.4	14.84	0.84	52.4
43.0	Pb Con 3	4.2	20.6	20.6	6.42	0.67	46.4
57.0	Pb Con 4	5.5	26.2	26.2	3.94	0.46	38.6
27.0	Cu Con 1	2.6	2.6	28.8	3.81	0.38	30.5
14.0	Cu Con 2	1.4	4.0	30.2	3.57	0.37	29.2
67.0	Zn Con 1	6.5	6.5	36.7	0.49	0.19	57.8
11.0	Zn Con 2	1.1	7.6	37.7	1.31	0.23	24.0
640.0	Tail	62.3		100.0	0.58	0.09	1.6
1028.00	Composite	100.0			3.91	0.28	18.2
				Assayed Head:	3.54	0.21	21.5

Mass Percentages
- 8.6 % of the mass was collected in Pb Con 1
- 26.2 % of the mass was collected in Pb Con 1-4
- 6.5 % of the mass was collected in Zn Con 1
- 7.6 % of the mass was collected in Zn Con 1 and 2
- 37.7 % of the mass was collected in all the cons.

Flotation 2 – Lead

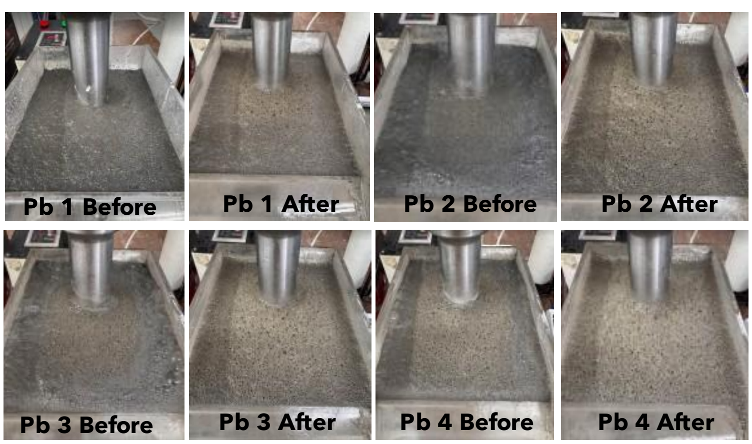
Product	Wt. %	Separate Wt. %	Cum. Wt. %	Pb Distribution %	Sep. Cum %	Cum. %	Sep. Grade % Pb
Pb Con 1	8.6	8.6	8.6	43.4%	43.4%	43.4%	19.85
Pb Con 2	7.9	16.4	16.4	29.9%	73.3%	73.3%	17.45
Pb Con 3	4.2	20.6	20.6	6.9%	80.2%	80.2%	6.97
Pb Con 4	5.5	26.2	26.2	5.6%	85.8%	85.8%	1.86
Cu Con 1	2.6	2.6	28.8	2.6%	2.6%	88.3%	0.35
Cu Con 2	1.4	4.0	30.2	1.2%	3.8%	89.6%	0.49
Zn Con 1	6.5	6.5	36.7	0.8%	0.8%	90.4%	0.09
Zn Con 2	1.1	7.6	37.7	0.4%	1.2%	90.7%	0.12
Tail	62.3		100.0	9.3%		100%	3.43
Composite	100.0			100%			

- Lead Stage
  - 43.4 % of the lead was collected in Pb Con 1
  - 29.9 % of the lead was collected in Pb Con 2
  - 6.9 % of the lead was collected in Pb Con 3
  - 5.6 % of the lead was collected in Pb Con 4
  - Total 85.8 % of the lead was pulled in the lead stage

- lead Stage
  - 2.6% of the lead was collected in Cu Con 1
  - 1.2% of the lead was collected in Cu Con 2
  - Only 3.8% of the lead was pulled in the lead stage

- Zinc Stage
  - 0.8% of the lead was collected in Zn Con 1
  - 0.4% of the lead was collected in Zn Con 2
  - 1.2% of the lead was pulled in the zinc stage

Flotation 2- Lead Pictures



Flotation 2 – Copper

	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution				Cu Distribution			
Product	%	%	%	%	Sep. Cum %	Cum. %	Sep. Grade % Pb	%	Sep. Cum %	Cum. %	Sep. Grade % Cu
Pb Con 1	8.6	8.6	8.6	43.4%	43.4%	43.4%	19.85	27.9%	27.9%	27.9%	0.93
Pb Con 2	7.9	16.4	16.4	29.9%	73.3%	73.3%	17.45	23.2%	51.1%	51.1%	0.88
Pb Con 3	4.2	20.6	20.6	6.9%	80.2%	80.2%	6.97	9.9%	61.0%	61.0%	0.46
Pb Con 4	5.5	26.2	26.2	5.6%	85.8%	85.8%	1.86	9.0%	70.0%	70.0%	0.21
Cu Con 1	2.6	2.6	28.8	2.6%	2.6%	88.3%	0.35	3.5%	3.5%	73.5%	0.03
Cu Con 2	1.4	4.0	30.2	1.2%	3.8%	89.6%	0.49	1.7%	5.3%	75.3%	0.05
Zn Con 1	6.5	6.5	36.7	0.8%	0.8%	90.4%	0.09	4.3%	4.3%	79.6%	0.03
Zn Con 2	1.1	7.6	37.7	0.4%	1.2%	90.7%	0.12	0.9%	5.2%	80.4%	0.04
Tail	62.3		100.0	9.3%		100%	3.43	19.6%		100%	0.23
Composite	100.0			100%				100.0%			

- Lead Stage
  - 27.9% of the copper was collected in Pb Con 1
  - 23.2% of the copper was collected in Pb Con 2
  - 9.9% of the copper was collected in Pb Con 3
  - 9.0% of the copper was collected in Pb Con 4
  - Total 70 % of the copper was pulled in the lead stage
- Copper Stage
  - 3.5% of the copper was collected in Cu Con 1
  - 1.7% of the Copper was collected in Cu Con 2
  - Only 5.3% of the copper was pulled in the copper stage
- Zinc Stage
  - 4.3% of the copper was collected in Zn Con 1
  - 0.9% of the copper was collected in Zn Con 2
  - 5.2% of the copper was pulled in the zinc stage

Flotation 2 – Copper


Cu Con 1 Before


Cu Con 1After


Cu Con 2 Before


Cu Con 2 After

Flotation 2 – Zinc

	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution				Cu Distribution				Zn Distribution			
Product	%	%	%	%	Sep. Cum. %	Cum. %	Sep. Grade % Pb	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn
Pb Con 1	8.6	8.6	8.6	43.4%	43.4%	43.4%	19.85	27.9%	27.9%	27.9%	0.93	20.7%	20.7%	20.7%	44.02
Pb Con 2	7.9	16.4	16.4	29.9%	73.3%	73.3%	17.45	23.2%	51.1%	51.1%	0.88	22.7%	43.4%	43.4%	48.02
Pb Con 3	4.2	20.6	20.6	6.9%	80.2%	80.2%	6.97	9.9%	61.0%	61.0%	0.46	10.7%	54.0%	54.0%	29.42
Pb Con 4	5.5	26.2	26.2	5.6%	85.8%	85.8%	1.86	9.0%	70.0%	70.0%	0.21	11.7%	65.8%	65.8%	15.59
Cu Con 1	2.6	2.6	28.8	2.6%	2.6%	88.3%	0.35	3.5%	3.5%	73.5%	0.03	4.4%	4.4%	70.2%	2.78
Cu Con 2	1.4	4.0	30.2	1.2%	3.8%	89.6%	0.49	1.7%	5.3%	75.3%	0.05	2.2%	6.6%	72.3%	3.98
Zn Con 1	6.5	6.5	36.7	0.8%	0.8%	90.4%	0.09	4.3%	4.3%	79.6%	0.03	20.7%	20.7%	93.0%	10.28
Zn Con 2	1.1	7.6	37.7	0.4%	1.2%	90.7%	0.12	0.9%	5.2%	80.4%	0.04	1.4%	22.1%	94.5%	10.67
Tail	62.3		100.0	9.3%		100%	3.43	19.6%		100%	0.23	5.5%		100%	14.13
Composite	100.0			100%				100.0%				100%			

- Lead
  - 20.7 % of the zinc was collected in Pb Con 1
  - 22.7 % of the zinc was collected in Pb Con 2
  - 10.7 % of the zinc was collected in Pb Con 3
  - 11.7 % of the zinc was collected in Pb Con 4
  - Total 65.8 % of the lead was pulled in the lead stage
- Copper
  - 4.4 % of the zinc was collected in Cu Con 1
  - 2.2 % of the zinc was collected in Cu Con 2
  - 6.6 % of the zinc was pulled in the copper stage
- Zinc
  - 20.7 % of the zinc was collected in Zn Con 1
  - 1.4 % of the zinc was collected in Zn Con 2
  - 22.1% of the zinc was pulled in the zinc stage!

Flotation 2 - Zinc

Zn Con 1 Before

Zn Con 1After

Zn Con 2 Before

Zn Con 2 After

Flotation 2 - Gold

	Pb Distribution				Cu Distribution			
Product	%	Sep. Cum %	Cum. %	Sep. Grade % Pb	%	Sep. Cum %	Cum. %	Sep. Grade % Cu
Pb Con 1	43.4%	43.4%	43.4%	19.85	27.9%	27.9%	27.9%	0.93
Pb Con 2	29.9%	73.3%	73.3%	17.45	23.2%	51.1%	51.1%	0.88
Pb Con 3	6.9%	80.2%	80.2%	6.97	9.9%	61.0%	61.0%	0.46
Pb Con 4	5.6%	85.8%	85.8%	1.86	9.0%	70.0%	70.0%	0.21
Cu Con 1	2.6%	2.6%	88.3%	0.35	3.5%	3.5%	73.5%	0.03
Cu Con 2	1.2%	3.8%	89.6%	0.49	1.7%	5.3%	75.3%	0.05
Zn Con 1	0.8%	0.8%	90.4%	0.09	4.3%	4.3%	79.6%	0.03
Zn Con 2	0.4%	1.2%	90.7%	0.12	0.9%	5.2%	80.4%	0.04
Tail	9.3%		100%	3.43	19.6%		100%	0.23
Composite	100%				100.0%			

	Zn Distribution				Au Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn	%	Sep. Cum %	Cum. %	Cum. g/mt Au
Pb Con 1	20.7%	20.7%	20.7%	44.02	36%	35.9%	35.9%	2.80
Pb Con 2	22.7%	43.4%	43.4%	48.02	24%	59.5%	59.5%	2.42
Pb Con 3	10.7%	54.0%	54.0%	29.42	9%	68.7%	68.7%	1.06
Pb Con 4	11.7%	65.8%	65.8%	15.59	10%	78.7%	78.7%	0.49
Cu Con 1	4.4%	4.4%	70.2%	2.78	5%	4.7%	83.4%	1.49
Cu Con 2	2.2%	6.6%	72.3%	3.98	2%	6.9%	85.6%	1.47
Zn Con 1	20.7%	20.7%	93.0%	10.28	1%	1.3%	86.9%	1.23
Zn Con 2	1.4%	22.1%	94.5%	10.67	1%	1.9%	87.6%	1.21
Tail	5.5%		100%	14.13	12%		100%	0.54
Composite	100%				100%			

- 87.6% of the gold was recovered in all the stages
- 78.7% of the gold reported to the lead stage

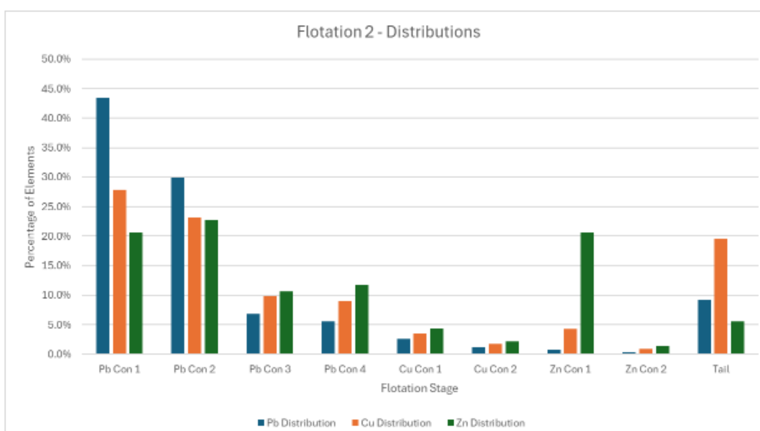
Flotation 2 - Silver

Product	Cu Distribution %	Cu Distribution Sep. Cum. %	Cu Distribution Cum. %	Sep. Grade % Cu	Zn Distribution %	Zn Distribution Sep. Cum. %	Zn Distribution Cum. %	Sep. Grade % Zn	Au Distribution %	Au Distribution Sep. Cum. %	Au Distribution Cum. %	Cum. g/mt Au	Ag Distribution %	Ag Distribution Sep. Cum. %	Ag Distribution Cum. %	Cum. g/mt Ag
Pb Con 1	27.9%	27.9%	27.9%	0.93	20.7%	20.7%	20.7%	44.02	36%	35.9%	35.9%	2.80	41.5%	41.5%	41.5%	3175.10
Pb Con 2	23.2%	51.1%	51.1%	0.88	22.7%	43.4%	43.4%	48.02	24%	59.5%	59.5%	2.42	26.9%	68.5%	68.5%	2724.31
Pb Con 3	9.9%	61.0%	61.0%	0.46	10.7%	54.0%	54.0%	29.42	9%	68.7%	68.7%	1.06	7.2%	75.7%	75.7%	2399.97
Pb Con 4	9.0%	70.0%	70.0%	0.21	11.7%	65.8%	65.8%	15.59	10%	78.7%	78.7%	0.49	7.0%	82.6%	82.6%	2066.34
Cu Con 1	3.5%	3.5%	73.5%	0.03	4.4%	4.4%	70.2%	2.78	5%	4.7%	83.4%	1.49	3.5%	3.5%	86.1%	1634.67
Cu Con 2	1.7%	5.3%	75.3%	0.05	2.2%	6.6%	72.3%	3.98	2%	6.9%	85.6%	1.47	1.4%	4.9%	87.6%	1591.63
Zn Con 1	4.3%	4.3%	79.6%	0.03	20.7%	20.7%	93.0%	10.28	1%	1.3%	86.9%	1.23	1.7%	1.7%	89.2%	1338.93
Zn Con 2	0.9%	5.2%	80.4%	0.04	1.4%	22.1%	94.5%	10.67	1%	1.9%	87.6%	1.21	0.6%	2.2%	89.8%	1310.54
Tail	19.6%		100%	0.23	5.5%		100%	14.13	12%		100%	0.54	10.2%		100%	561.38
Composite	100.0%				100%				100%				100%			

- 89.8 % of the silver was recovered in all the stages
- 82.6 % of the silver reported to the lead stage

Flotation 2 - Distributions

- Finer grinding did get the elements out of tails and into the cons
- Aerofloat 7310 collects too much of all the elements.



Conclusions & Plan for Flotation 3

- Grinding finer helped recover more metals, but Cu might still be locked with the Pb. Will grind finer.
- Would like to decrease the amount of copper & zinc reporting to lead stage.
  - The Aerofloat 7310 is probably not selective enough for just lead. Will switch to Aerofloat 3418A.
  - Will decrease zinc sulfate (sphalerite depression) and increase sodium sulfite (chalcopyrite depression) in lead stage.

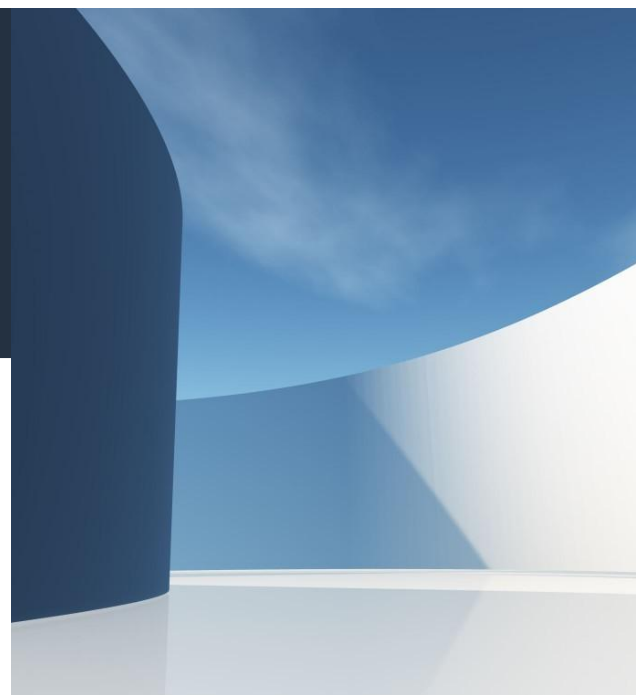
- 3418A is phosphine-based and much more selective for galena than 7310, especially when Cu/Zn are present
- Lower zinc sulfate managing sphalerite, may have overdosed a bit in lead stage coating zinc surfaces and causing them to float more.
- Higher sodium sulfite enhances depression of chalcopyrite and any reactive iron surfaces.
- Combined, this setup should yield a cleaner Pb concentrate with less Cu and Zn contamination.

- If copper (e.g., chalcopyrite) and zinc (e.g., sphalerite) are **locked with galena**, they'll float **with galena** regardless of reagent scheme.
- Would STILL strongly suggest reconsidering getting a mineralogy test done to see what forms these elements are in to make better reagent selections.

1096 Darwin Flotation Update

04/21/2025

WOODS PROCESS
SERVICES, LLC



Flotation 3

- Grind time 50 minutes
- P80 = 75µm
- MIBC for frother
- Milk of Lime for pH control

- Lead Float
    - 1 rougher for 5 minutes
    - 1 scavenger for 3 minutes
    - ~40 g/mt Aerofloat 3418A
    - ~770 g/mt Zinc Sulfate
    - ~1000 g/mt Sodium Sulfite
    - Target pH 9.0
- Copper Float
    - 1 rougher for 5 minutes
    - 1 scavenger for 3 minutes
    - ~40 g/mt Aero 3894
    - ~20 g/mt Aerofloat 238
    - ~1000 g/mt Zinc Sulfate
    - Target pH 9.0
- Zinc Float
    - 1 rougher for 5 minutes
    - 1 scavenger for 3 minutes
    - ~750 g/mt Copper Sulfate
    - ~50 g/mt PAX
    - Target pH 11-11.5

Flotation 3 - Assays

- Each concentrate pull was kept and analyzed separately.
- The Pb Con 1 -2 are combined **mathematically** to see how much total mass and metal was pulled in a 5 minute rougher and 3 minute scavenger.
- Cu and Zn both had a 5 minute rougher and 3 minute scavenger pulls. Likewise combined mathematically.
- Sep. Wt % = Separate Weight Percent… combining the rougher and scavenger cons together by element by stage
- Cum. Wt. % = Cumulative Weight Percent is running total from all the pulls

Weight		Wt.	Sep. Wt.	Cum. Wt.		Assay	
grams	Product	%	%	%	% Pb	% Cu	% Zn
33.0	Pb Con 1	3.3	3.3	3.3	24.98	1.70	27.5
43.0	Pb Con 2	4.2	7.5	7.5	23.76	0.75	27.2
44.0	Cu Con 1	4.3	4.3	11.8	9.45	0.57	27.7
27.0	Cu Con 2	2.7	7.0	14.5	11.23	0.36	24.8
164.0	Zn Con 1	16.2	16.2	30.6	0.44	0.15	53.9
58.0	Zn Con 2	5.7	21.9	36.4	0.15	0.05	0.2
646.0	Tail	63.6		100.0	0.37	0.06	1.4
1015.00	Composite	100.0			2.84	0.19	13.5

Mass Percentages
- 3.3 % of the mass was collected in Pb Con 1
- 7.5 % of the mass was collected in Pb Con 1 and 2
- 16.2 % of the mass was collected in Zn Con 1
- 21.9 % of the mass was collected in Zn Con 1 and 2
- 36.4 % of the mass was collected in all the cons.

Flotation 3 – Lead

	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution			
Product	%	%	%	%	Sep. Cum %	Cum %	Sep. Grade % Pb
Pb Con 1	3.3	3.3	3.3	28.6%	28.6%	28.6%	24.98
Pb Con 2	4.2	7.5	7.5	35.4%	64.0%	64.0%	24.29
Cu Con 1	4.3	4.3	11.8	14.4%	14.4%	78.4%	3.46
Cu Con 2	2.7	7.0	14.5	10.5%	24.9%	88.9%	4.89
Zn Con 1	16.2	16.2	30.6	2.5%	2.5%	91.4%	0.23
Zn Con 2	5.7	21.9	36.4	0.3%	2.8%	91.7%	0.22
Tail	63.6		100.0	8.3%		100%	2.84
Composite	100.0			100.0%			

- Lead Stage
  - 28.6 % of the lead was collected in Pb Con 1
  - 35.4 % of the lead was collected in Pb Con 2
  - Total 64.0 % of the lead was pulled in the lead stage
- Copper Stage
  - 14.4% of the lead was collected in Cu Con 1
  - 10.5% of the lead was collected in Cu Con 2
  - 24.9% of the lead was pulled in the copper stage
- Zinc Stage
  - 2.5% of the lead was collected in Zn Con 1
  - 0.3% of the lead was collected in Zn Con 2
  - 2.8% of the lead was pulled in the zinc stage

Flotation 3- Lead Pictures

Pb 1 Before

Pb 1 After

Pb 2 Before

Pb 2 After

Flotation 3 – Copper

	Wt.	Separate Wt.	Cum. Wt.	Pb Distribution				Cu Distribution			
Product	%	%	%	%	Sep. Cum %	Cum. %	Sep. Grade % Pb	%	Sep. Cum %	Cum %	Sep. Grade % Cu
Pb Con 1	3.3	3.3	3.3	28.6%	28.6%	28.6%	24.98	29.4%	29.4%	29.4%	1.70
Pb Con 2	4.2	7.5	7.5	35.4%	64.0%	64.0%	24.29	17.0%	46.5%	46.5%	1.16
Cu Con 1	4.3	4.3	11.8	14.4%	14.4%	78.4%	3.46	13.2%	13.2%	59.6%	0.21
Cu Con 2	2.7	7.0	14.5	10.5%	24.9%	88.9%	4.89	5.1%	18.2%	64.7%	0.24
Zn Con 1	16.2	16.2	30.6	2.5%	2.5%	91.4%	0.23	12.9%	12.9%	77.6%	0.08
Zn Con 2	5.7	21.9	36.4	0.3%	2.8%	91.7%	0.22	1.5%	14.4%	79.1%	0.07
Tail	63.6		100.0	8.3%		100%	2.84	20.9%		100%	0.19
Composite	100.0			100.0%				100.0%			

- Lead Stage
  - 29.4% of the copper was collected in Pb Con 1
  - 17.0% of the copper was collected in Pb Con 2
  - Total 46.5% of the copper was pulled in the lead stage
- Copper Stage
  - 13.2% of the copper was collected in Cu Con 1
  - 5.1% of the Copper was collected in Cu Con 2
  - 18.2% of the copper was pulled in the copper stage
- Zinc Stage
  - 12.9% of the copper was collected in Zn Con 1
  - 1.5% of the copper was collected in Zn Con 2
  - 14.4% of the copper was pulled in the zinc stage

Flotation 3 – Copper

Cu Con 1 Before

Cu Con 1 After

Cu Con 2 Before

Cu Con 2 After

Flotation 3 - Zinc

Product	Wt. %	Separate Wt. %	Cum. Wt. %	Pb Distribution %	Pb Sep. Cum %	Pb Cum. %	Sep. Grade % Pb	Cu Distribution %	Cu Sep. Cum %	Cu Cum. %	Sep. Grade % Cu	Zn Distribution %	Zn Sep. Cum %	Zn Cum. %	Sep. Grade % Zn
Pb Con 1	3.3	3.3	3.3	28.6%	28.6%	28.6%	24.98	29.4%	29.4%	29.4%	1.70	6.6%	6.6%	6.6%	27.50
Pb Con 2	4.2	7.5	7.5	35.4%	64.0%	64.0%	24.29	17.0%	46.5%	46.5%	1.16	8.5%	15.1%	15.1%	27.35
Cu Con 1	4.3	4.3	11.8	14.4%	14.4%	78.4%	3.46	13.2%	13.2%	59.6%	0.21	8.9%	8.9%	24.0%	10.14
Cu Con 2	2.7	7.0	14.5	10.5%	24.9%	88.9%	4.89	5.1%	18.2%	64.7%	0.24	4.9%	13.7%	28.9%	12.84
Zn Con 1	16.2	16.2	30.6	2.5%	2.5%	91.4%	0.23	12.9%	12.9%	77.6%	0.08	64.3%	64.3%	93.2%	28.41
Zn Con 2	5.7	21.9	36.4	0.3%	2.8%	91.7%	0.22	1.5%	14.4%	79.1%	0.07	0.1%	64.4%	93.3%	23.97
Tail	63.6		100.0	8.3%		100%	2.84	20.9%		100%	0.19	6.7%		100%	13.53
Composite	100.0			100.0%				100.0%				100%			

- Lead
  - 6.6 % of the zinc was collected in Pb Con 1
  - 8.5 % of the zinc was collected in Pb Con 2
  - Total 15.1 % of the zinc was pulled in the lead stage
- Copper
  - 8.9 % of the zinc was collected in Cu Con 1
  - 4.9 % of the zinc was collected in Cu Con 2
  - 13.7 % of the zinc was pulled in the copper stage
- Zinc
  - 64.3 % of the zinc was collected in Zn Con 1
  - 0.1 % of the zinc was collected in Zn Con 2
  - 64.4 % of the zinc was pulled in the zinc stage!

Flotation 3 - Zinc

Zn Con 1 Before

Zn Con 1After

Zn Con 2 Before

Zn Con 2 After

Flotation 3 – Gold

	Pb Distribution				Cu Distribution				Zn Distribution				Au Distribution			
Product	%	Sep. Cum %	Cum. %	Sep. Grade % Pb	%	Sep. Cum %	Cum %	Sep. Grade % Cu	%	Sep. Cum %	Cum %	Sep. Grade % Zn	%	Sep. Cum. %	Cum %	Cum g/mt Au
Pb Con 1	28.6%	28.6%	28.6%	24.98	29.4%	29.4%	29.4%	1.70	6.6%	6.6%	6.6%	27.50	28%	28.1%	28.1%	5.87
Pb Con 2	35.4%	64.0%	64.0%	24.29	17.0%	46.5%	46.5%	1.16	8.5%	15.1%	15.1%	27.35	17%	44.8%	44.8%	4.06
Cu Con 1	14.4%	14.4%	78.4%	3.46	13.2%	13.2%	59.6%	0.21	8.9%	8.9%	24.0%	10.14	14%	13.6%	58.4%	3.35
Cu Con 2	10.5%	24.9%	88.9%	4.89	5.1%	18.2%	64.7%	0.24	4.9%	13.7%	28.9%	12.84	7%	20.4%	65.2%	3.05
Zn Con 1	2.5%	2.5%	91.4%	0.23	12.9%	12.9%	77.6%	0.08	64.3%	64.3%	93.2%	28.41	6%	6.4%	71.6%	1.58
Zn Con 2	0.3%	2.8%	91.7%	0.22	1.5%	14.4%	79.1%	0.07	0.1%	64.4%	93.3%	23.97	3%	9.7%	75.0%	1.40
Tail	8.3%		100%	2.84	20.9%		100%	0.19	6.7%		100%	13.53	25%		100%	0.68
Composite	100.0%				100.0%				100%				100%			

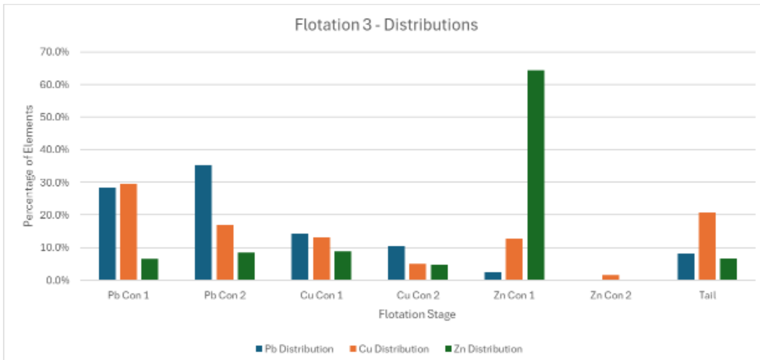
- 75.0 % of the gold was recovered in all the stages
- 44.8 % of the gold reported to the lead stage

Flotation 3 – Silver

	Cu Distribution				Zn Distribution				Au Distribution				Ag Distribution			
Product	%	Sep. Cum %	Cum. %	Sep. Grade % Cu	%	Sep. Cum %	Cum %	Sep. Grade % Zn	%	Sep. Cum %	Cum %	Cum. g/mt Au	%	Sep. Cum. %	Cum %	Cum. g/mt Ag
Pb Con 1	29.4%	29.4%	29.4%	1.70	6.6%	6.6%	6.6%	27.50	28%	28.1%	28.1%	5.87	28.1%	28.1%	28.1%	5146.19
Pb Con 2	17.0%	46.5%	46.5%	1.16	8.5%	15.1%	15.1%	27.35	17%	44.8%	44.8%	4.06	30.9%	59.1%	59.1%	4691.14
Cu Con 1	13.2%	13.2%	59.6%	0.21	8.9%	8.9%	24.0%	10.14	14%	13.6%	58.4%	3.35	14.0%	14.0%	73.1%	3676.24
Cu Con 2	5.1%	18.2%	64.7%	0.24	4.9%	13.7%	28.9%	12.84	7%	20.4%	65.2%	3.05	7.7%	21.7%	80.8%	3316.86
Zn Con 1	12.9%	12.9%	77.6%	0.08	64.3%	64.3%	93.2%	28.41	6%	6.4%	71.6%	1.58	4.1%	4.1%	84.9%	1648.07
Zn Con 2	1.5%	14.4%	79.1%	0.07	0.1%	64.4%	93.3%	23.97	3%	9.7%	75.0%	1.40	3.0%	7.2%	87.9%	1438.51
Tail	20.9%		100%	0.19	6.7%		100%	13.53	25%		100%	0.68	12.1%		100.0%	594.67
Composite	100.0%				100%				100%				100%			

- 87.9 % of the silver was recovered in all the stages
- 59.1 % of the silver reported to the lead stage

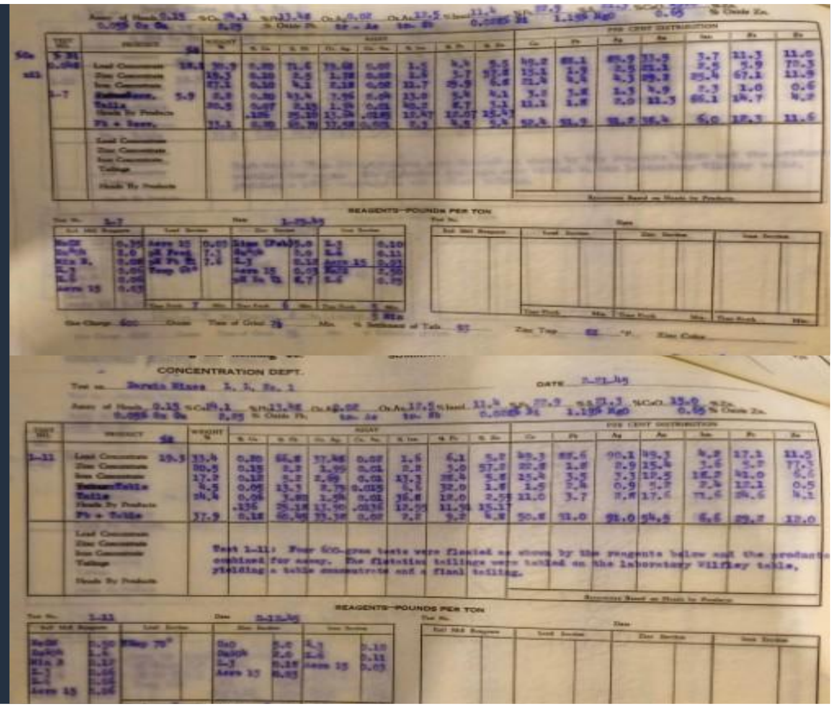
Flotation 3 – Distributions



Flotation 3 Conclusions & Suggestions

- Finer grinding helped get metals out of the tails and into the concentrates.
- In Flotation 3, the lead stage only had one 5 minute rougher and one 3 minute scavenger. Future tests should include additional pulls (like Flotation 2) to better recover lead earlier and minimize Pb in Cu/Zn stages.
- **Switching from Aerofloat 7310 to Aero 3418A didn't make much difference on the copper.** A large portion of Cu still reported to the lead concentrate, suggesting a stronger depressant or better liberation is needed.
- Increasing zinc sulfate in the lead and copper stages appears to have helped suppress zinc, as most of the Zn reported to the Zn stage.
- Increasing sodium sulfite in the lead stage did not significantly suppress copper. Additional tests using sodium sulfate (or other) are recommended to evaluate its effectiveness.
- Given the persistent copper in the Pb concentrate, it's possible the copper is still texturally bound to lead and floating regardless of reagent scheme.
- A revised flowsheet worth testing: **lead stage → zinc stage → copper cleaning from Pb concentrate.** This may allow a more targeted separation and help remove entrained Cu from the Pb con.
- Mineralogy results and fire assay for gold and silver results are pending.

Historical Concentrate Tickets



Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted by: Stone Brothers	Laboratory No.: 251289
PO Box 3010	Client Number: F577
Tonopah, Nevada 89049	Date Received: 11 Apr 2025
Attention: Mr. Jack Stone	Date Completed: 28 Apr 2025

Method: Lithium metaborate fusion, ICP analysis.

Lab code:	7041
Element:	Tungsten
Detection Limit:	0.01
Units:	%
Jacks Tungsten Spud	63.49
Blank	< 0.01
OREAS 701	2.45
Certified Reference Material	**W**
OREAS 701	2.43
95% Confidence Interval of Mean	0.04

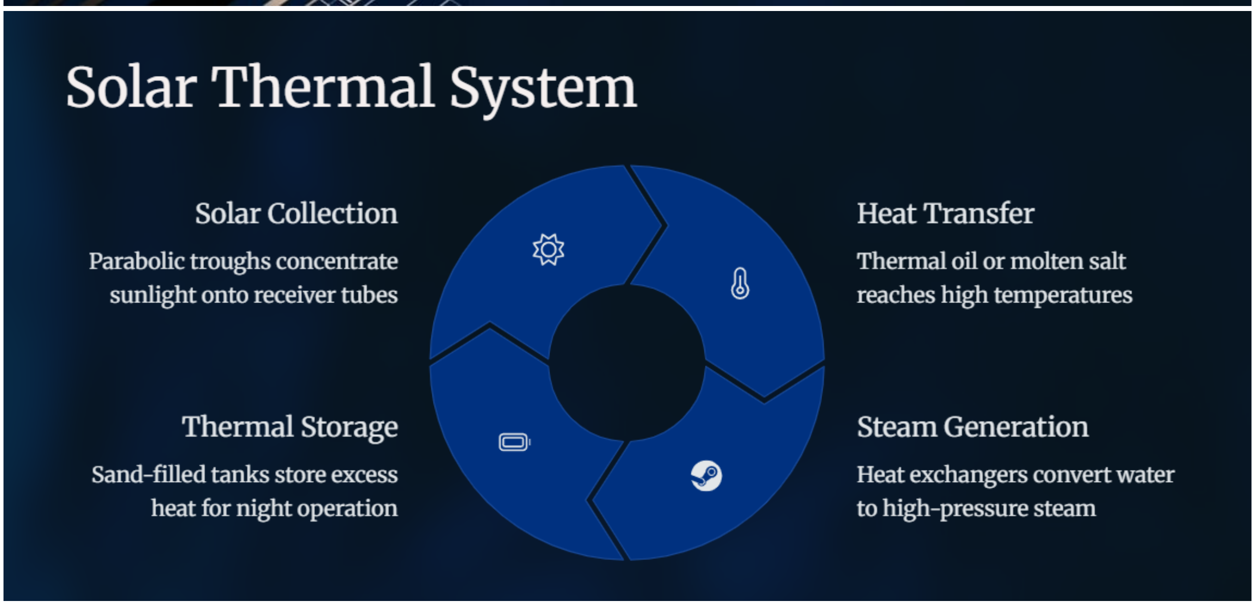
Karen Boldi

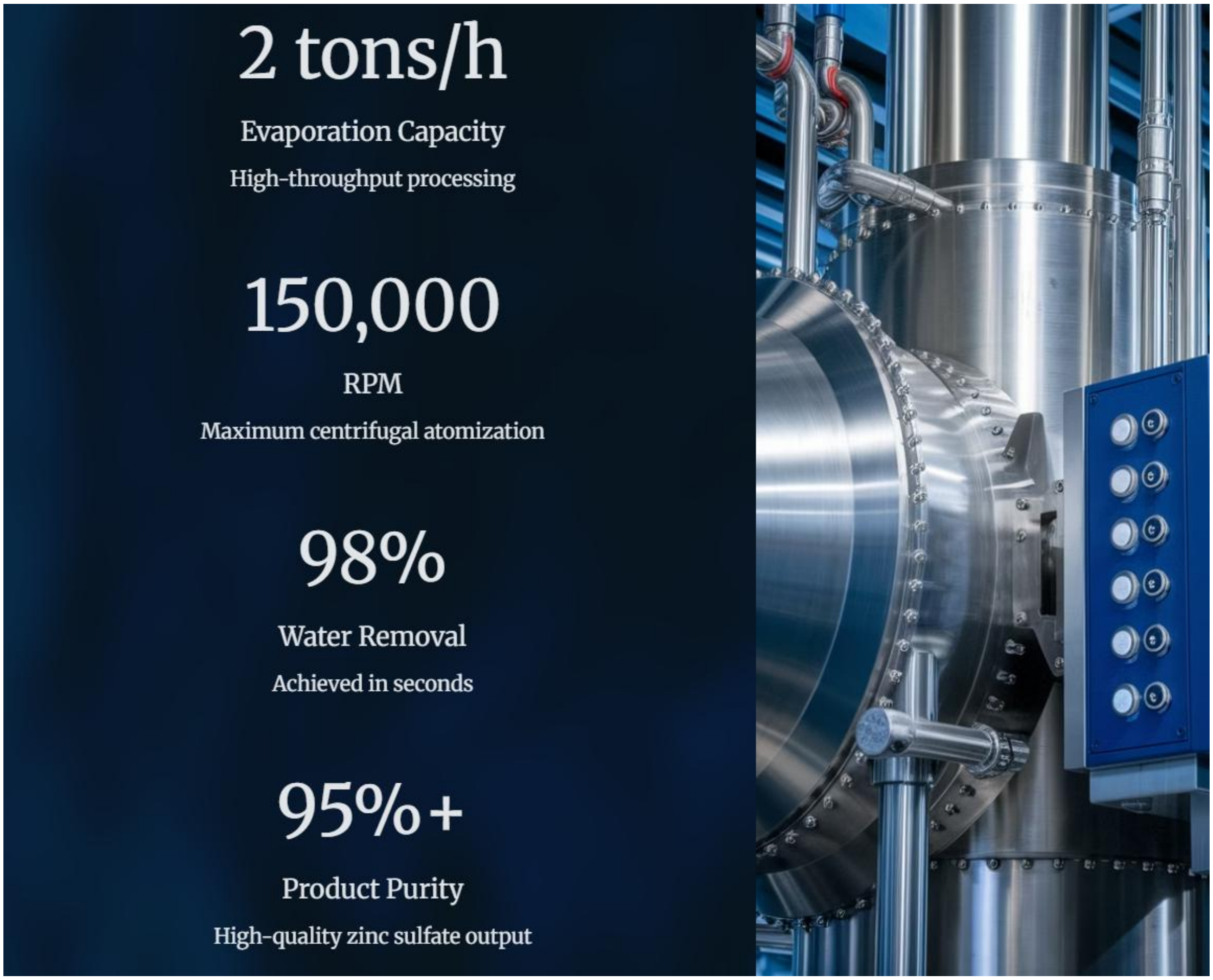
Karen Boldi, Quality Control Supervisor

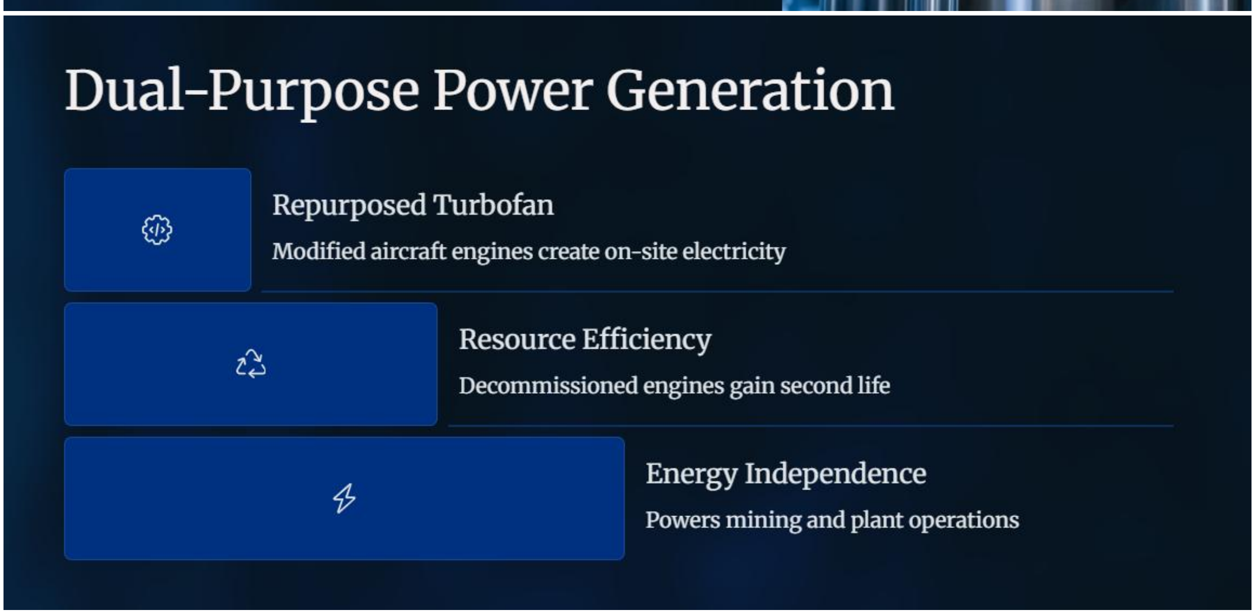
Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

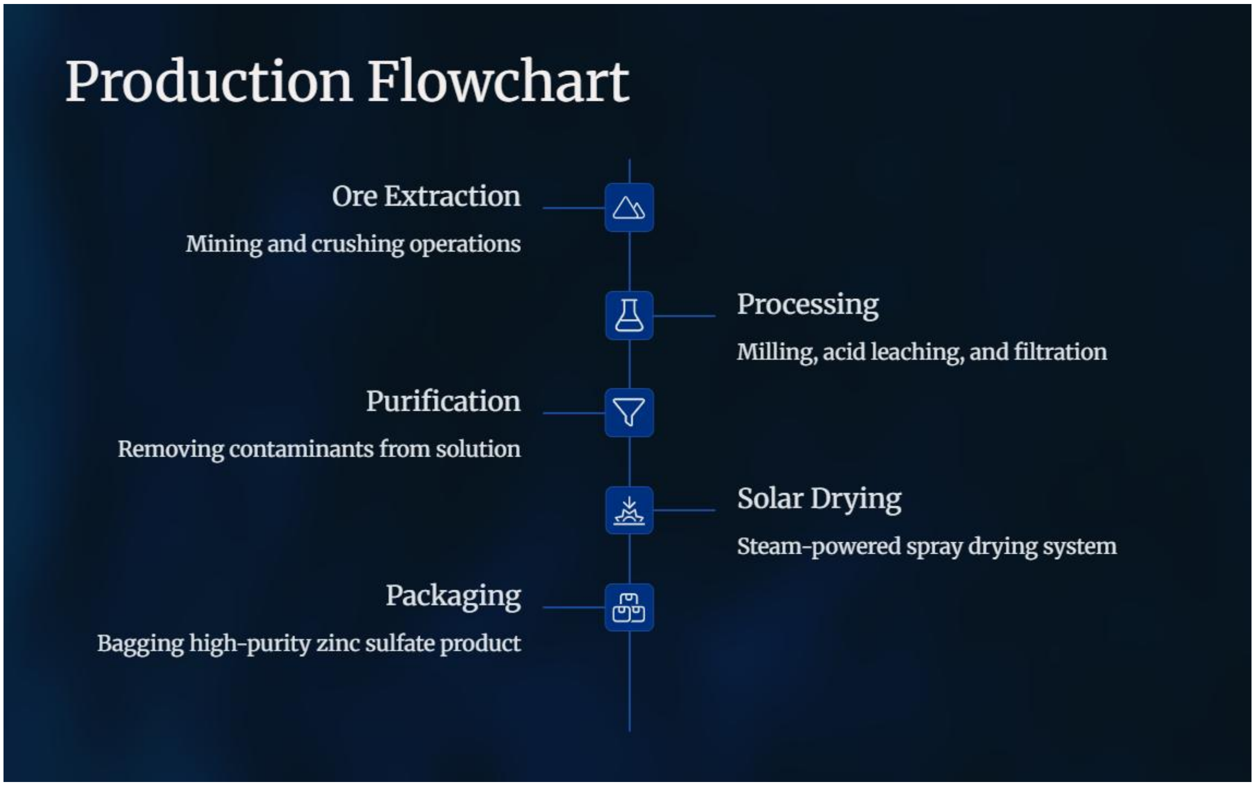
Dryer and Power solution

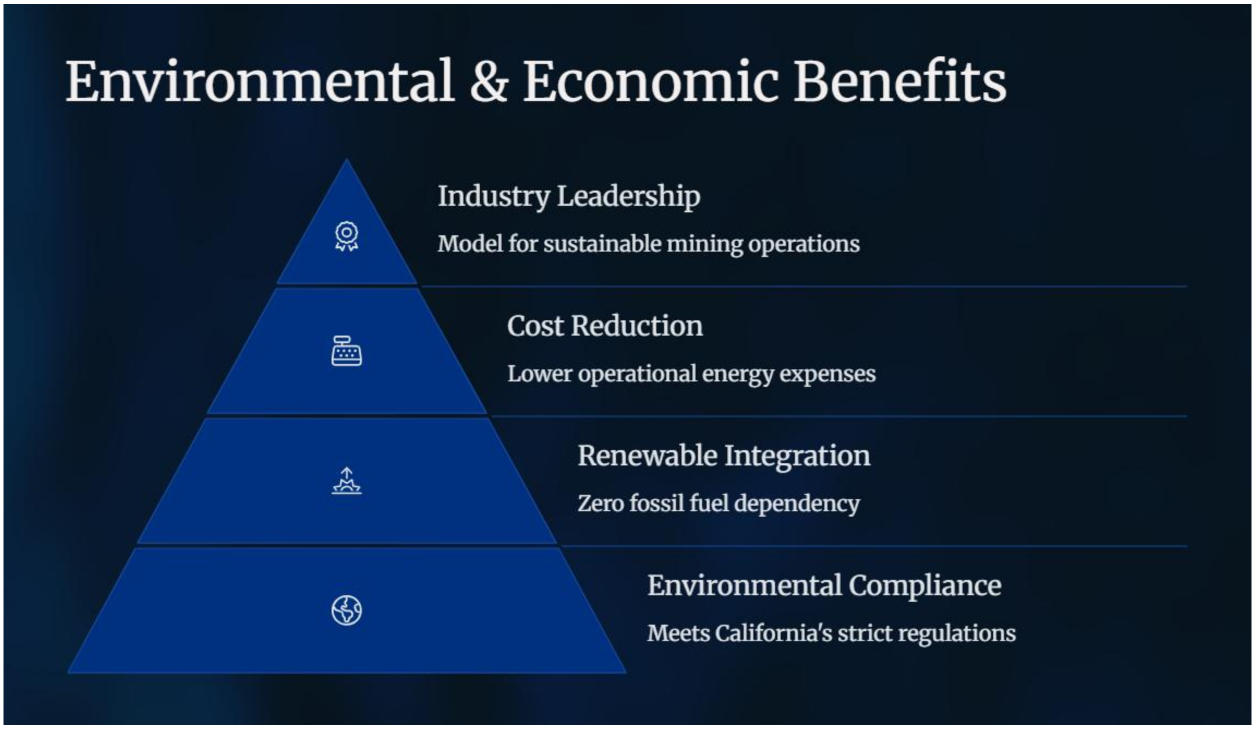

Regulatory Challenges
Fuel Restrictions
No LPG or natural gas available at the remote location.
Air Quality Permits
Stringent California regulations restrict diesel fuel usage.
Renewable Mandates
Compliance with aggressive clean energy requirements.


Solar Thermal System
Solar Collection
Parabolic troughs concentrate sunlight onto receiver tubes
Heat Transfer
Thermal oil or molten salt reaches high temperatures
Thermal Storage
Sand-filled tanks store excess heat for night operation
Steam Generation
Heat exchangers convert water to high-pressure steam


2 tons/h
Evaporation Capacity
High-throughput processing
150,000
RPM
Maximum centrifugal atomization
98%
Water Removal
Achieved in seconds
95%+
Product Purity
High-quality zinc sulfate output


Dual-Purpose Power Generation
Repurposed Turbofan
Modified aircraft engines create on-site electricity
Resource Efficiency
Decommissioned engines gain second life
Energy Independence
Powers mining and plant operations


Production Flowchart
Ore Extraction
Mining and crushing operations
Processing
Milling, acid leaching, and filtration
Purification
Removing contaminants from solution
Solar Drying
Steam-powered spray drying system
Packaging
Bagging high-purity zinc sulfate product


Environmental & Economic Benefits
Industry Leadership
Model for sustainable mining operations
Cost Reduction
Lower operational energy expenses
Renewable Integration
Zero fossil fuel dependency
Environmental Compliance
Meets California's strict regulations

Electrowinning

Process Proposal: Integrated EW System

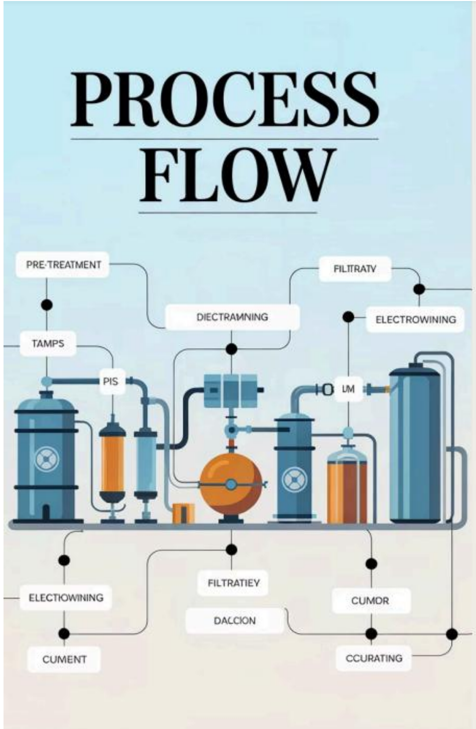
Our comprehensive solution integrates an Electrowinning (EW) System for Germanium, Indium, Tellurium, Antimony, and recovery of Gallium and Hafnium. This proposal includes a cleaner circuit for zinc sulfate, leaching of 8 tons/hour cake, a solar steam dryer, and cyclone EW cells.

This integrated approach maximizes efficiency while recovering valuable critical metals from process streams. The system is designed to be modular, environmentally sustainable, and economically advantageous.

Input Materials and Process Rates

Zinc Sulfate Solution	40,000 gallons (~151,400 liters) in circuit
Solid Feedstock ("Cake")	8 tons/hour
Cake Composition	Ag, Au, Ge, Te, Cu, In, Hf, Ga enriched residues
Leaching Reagent	Nitric Acid (HNO_3)

The process handles substantial volumes of zinc sulfate solution while continuously processing 8 tons per hour of metal-rich cake. The specialized composition of the feedstock contains valuable elements including silver, gold, germanium, tellurium, copper, indium, hafnium, and gallium.



Process Flow Overview

Pre-treatment (Cleaner Circuit)

Remove Pb, Co, Cd, Ni from zinc sulfate using lime neutralization, sulfide precipitation, and polishing filtration

Leaching System

Nitric acid leaching of solid cake at 60-80°C using solar-assisted steam heating to solubilize target elements

Steam Dryer System

Solar thermal parabolic trough system with molten salt or sand bed thermal storage for 24-hour operation

Electrowinning and Recovery

Cyclone cells (250mm x 2000mm) with improved electrolyte circulation, arranged in 2 banks of 12 cells

Steam Dryer System (Solar Heated)

Solar Type

Parabolic trough concentrators that efficiently capture and focus solar energy for steam generation

Thermal Storage

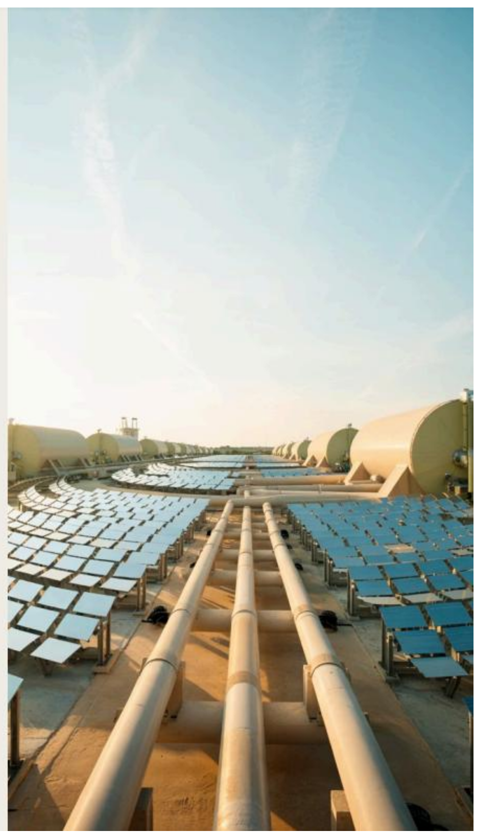
Sand bed providing approximately 10-12 hours of operation buffer

Heat Capacity

System designed to maintain steam temperatures of 150-200°C for continuous operation

Purpose

Dry leached cake and supply consistent heat to reactors, reducing conventional energy costs



Full Integrated Flow Diagram

1. **Solid Cake Feed**
   Initial input of 8 tons/hour of metal-rich cake material
2. **Nitric Leach Reactor (Solar Heated)**
   Dissolution of target metals using nitric acid at controlled temperatures
3. **Filtration & Solar Drying**
   Separation of solids and solution, with residue sent to solar dryer for moisture removal
4. **Selective Extraction/Electrowinning**
   Recovery of Ge, In, Te, Sb, Ga, and Hf using specialized cyclone cells
5. **Solution Recycling**
   Purified solution returned to zinc sulfate circuit or prepared for disposal

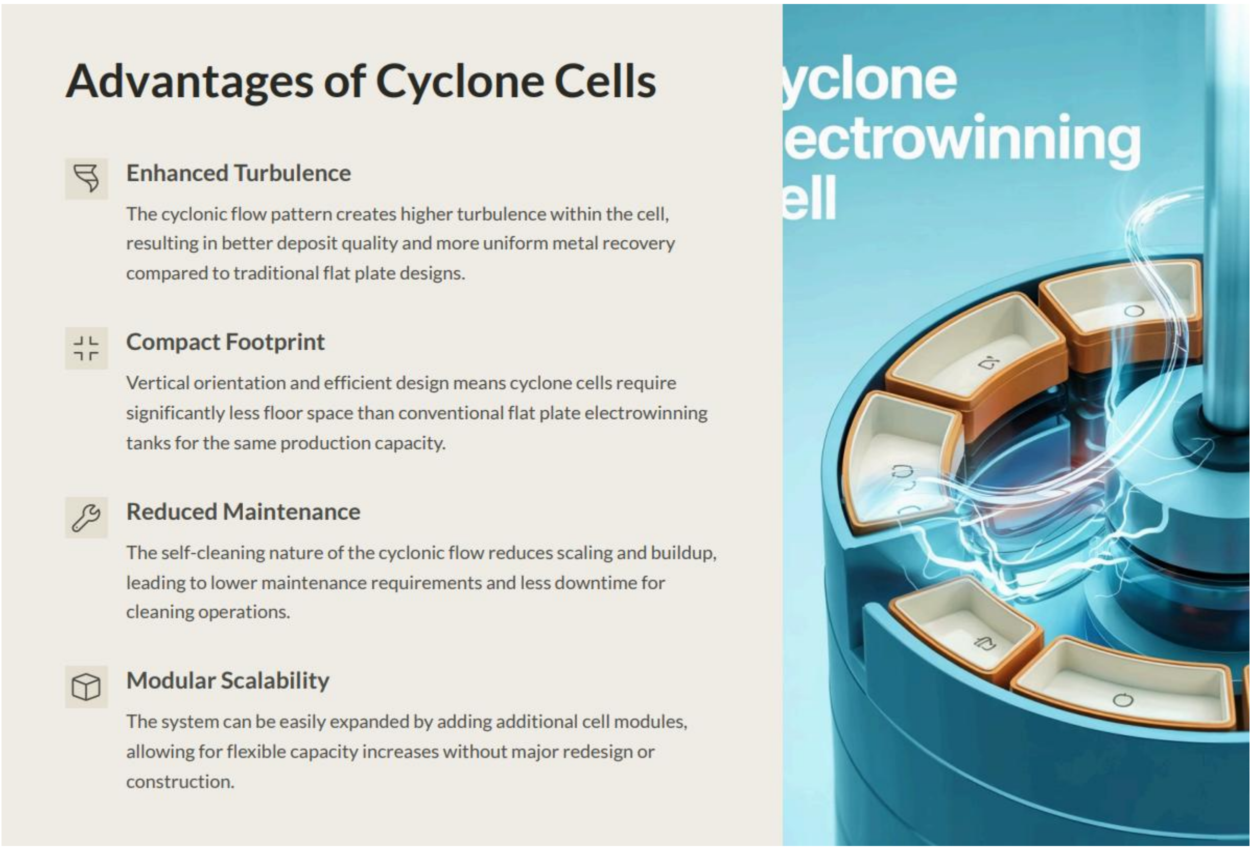
Electrowinning Cell System Details

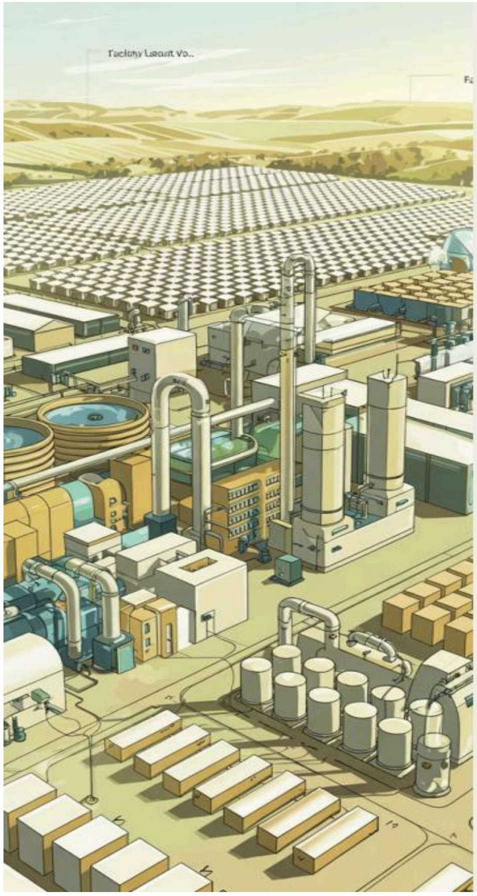
Physical Specifications

- Cyclone Cell Diameter: 250mm
- Cyclone Cell Height: 2000mm
- Electrodes: Titanium Cathode / Inert Anode (Ti/Ir coated)
- Cell Bank Configuration: 2 x 12 cell arrays

Operational Parameters

- Electrolyte Temperature: 45-65°C
- Current Density: 100-150 A/m²
- Power Supply: 6-12V, 300-500A per cell
- Cooling: Natural convection or optional air-cooled jackets


Advantages of Cyclone Cells
Enhanced Turbulence
The cyclonic flow pattern creates higher turbulence within the cell, resulting in better deposit quality and more uniform metal recovery compared to traditional flat plate designs.
Compact Footprint
Vertical orientation and efficient design means cyclone cells require significantly less floor space than conventional flat plate electrowinning tanks for the same production capacity.
Reduced Maintenance
The self-cleaning nature of the cyclonic flow reduces scaling and buildup, leading to lower maintenance requirements and less downtime for cleaning operations.
Modular Scalability
The system can be easily expanded by adding additional cell modules, allowing for flexible capacity increases without major redesign or construction.
yclone
ectrowinning
ell



Estimated System Sizing

24

Cyclone Cells

Total units required for full operation

400m²

Solar Collectors

Mirror surface area for parabolic troughs

30m³

Thermal Storage

Molten salt tank or sand thermal bed volume

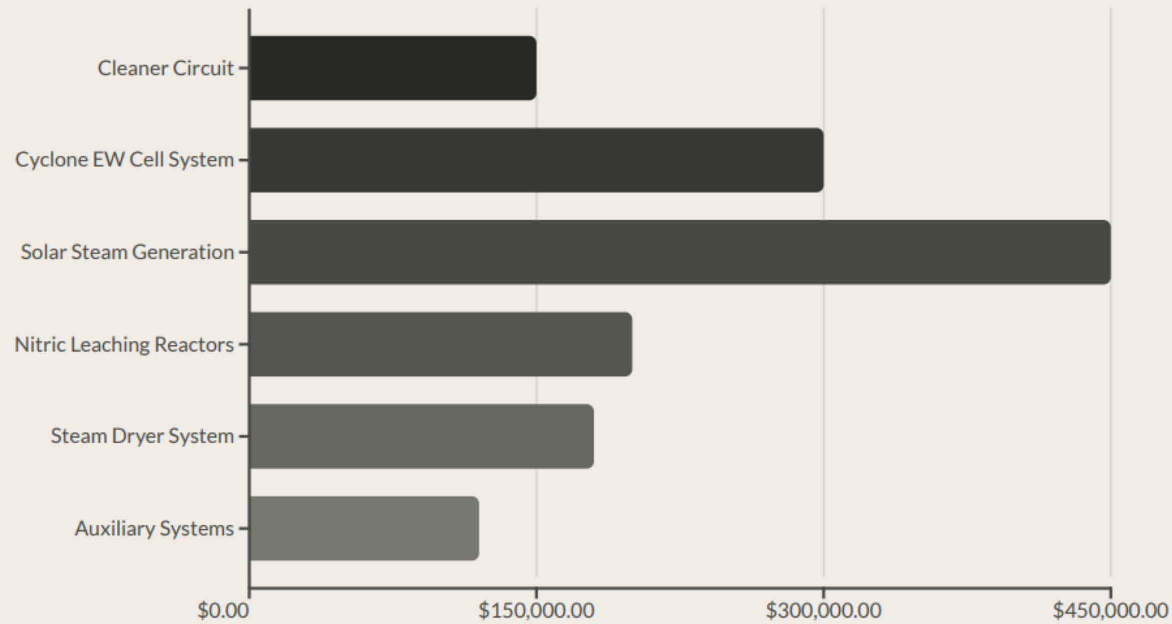
2-3

Leach Reactors

Continuous stirred reactors for nitric leaching

The system also includes one steam boiler with solar integration, capable of producing 2-3 tons per hour of steam. All components are sized to handle the 8 tons per hour throughput while maintaining operational flexibility.

Rough CAPEX Update



The total estimated capital expenditure is approximately $1.4 million. This is a very rough estimate based on a 100-200 tons per day full system with modular expandability. The largest single investment is in the solar steam generation system, which includes mirrors and thermal storage.

Key Notes

Modular Design

The system can scale by adding 12-cell cyclone banks, allowing for phased implementation and capacity expansion as needed without complete redesign.

Solar-Powered Efficiency

Solar-powered drying significantly reduces operational costs by minimizing diesel and electricity expenses while providing sustainable energy for continuous operation.

Strategic Value

Recovery of Ge, In, Te, Sb, Ga, and Hf provides access to critical metals with high market premiums, substantially improving project economics.

Cleaner Solution

The improved zinc sulfate solution reduces penalties when selling $ZnSO_4$ solution or cathodes, creating additional value beyond the recovered specialty metals.

Process Proposal: Integrated EW System

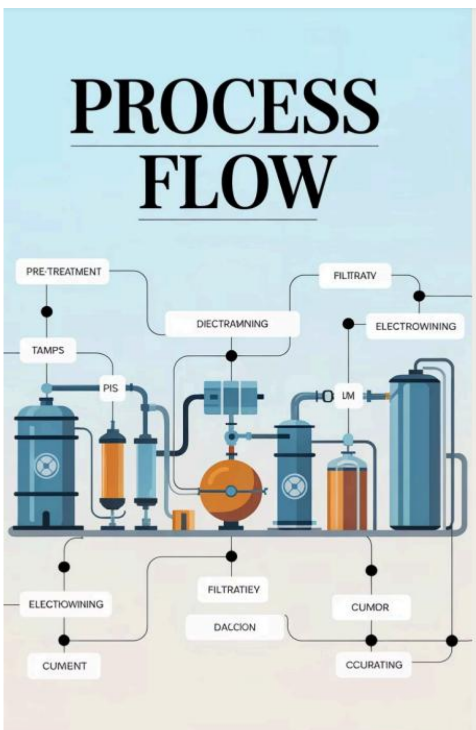
Our comprehensive solution integrates an Electrowinning (EW) System for Germanium, Indium, Tellurium, Antimony, and recovery of Gallium and Hafnium. This proposal includes a cleaner circuit for zinc sulfate, leaching of 8 tons/hour cake, a solar steam dryer, and cyclone EW cells.

This integrated approach maximizes efficiency while recovering valuable critical metals from process streams. The system is designed to be modular, environmentally sustainable, and economically advantageous.

Input Materials and Process Rates

Zinc Sulfate Solution	40,000 gallons (~151,400 liters) in circuit
Solid Feedstock ("Cake")	8 tons/hour
Cake Composition	Ag, Au, Ge, Te, Cu, In, Hf, Ga enriched residues
Leaching Reagent	Nitric Acid (HNO_3)

The process handles substantial volumes of zinc sulfate solution while continuously processing 8 tons per hour of metal-rich cake. The specialized composition of the feedstock contains valuable elements including silver, gold, germanium, tellurium, copper, indium, hafnium, and gallium.



Process Flow Overview

Pre-treatment (Cleaner Circuit)

Remove Pb, Co, Cd, Ni from zinc sulfate using lime neutralization, sulfide precipitation, and polishing filtration

Leaching System

Nitric acid leaching of solid cake at 60-80°C using solar-assisted steam heating to solubilize target elements

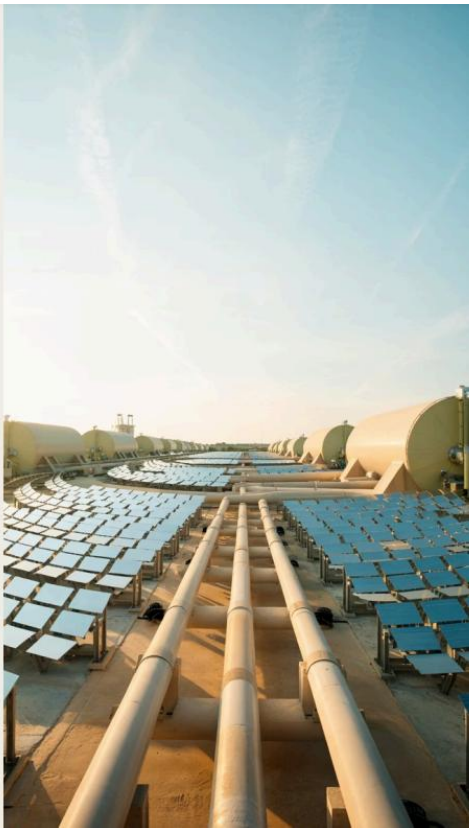
Steam Dryer System

Solar thermal parabolic trough system with molten salt or sand bed thermal storage for 24-hour operation

Electrowinning and Recovery

Cyclone cells (250mm x 2000mm) with improved electrolyte circulation, arranged in 2 banks of 12 cells

Steam Dryer System (Solar Heated)



Solar Type

Parabolic trough concentrators that efficiently capture and focus solar energy for steam generation

Thermal Storage

Sand bed providing approximately 10-12 hours of operation buffer

Heat Capacity

System designed to maintain steam temperatures of 150-200°C for continuous operation

Purpose

Dry leached cake and supply consistent heat to reactors, reducing conventional energy costs

Full Integrated Flow Diagram

1. **Solid Cake Feed**
   Initial input of 8 tons/hour of metal-rich cake material
2. **Nitric Leach Reactor (Solar Heated)**
   Dissolution of target metals using nitric acid at controlled temperatures
3. **Filtration & Solar Drying**
   Separation of solids and solution, with residue sent to solar dryer for moisture removal
4. **Selective Extraction/Electrowinning**
   Recovery of Ge, In, Te, Sb, Ga, and Hf using specialized cyclone cells
5. **Solution Recycling**
   Purified solution returned to zinc sulfate circuit or prepared for disposal

Electrowinning Cell System Details

Physical Specifications

- Cyclone Cell Diameter: 250mm
- Cyclone Cell Height: 2000mm
- Electrodes: Titanium Cathode / Inert Anode (Ti/Ir coated)
- Cell Bank Configuration: 2 x 12 cell arrays

Operational Parameters

- Electrolyte Temperature: 45-65°C
- Current Density: 100-150 A/m²
- Power Supply: 6-12V, 300-500A per cell
- Cooling: Natural convection or optional air-cooled jackets

Advantages of Cyclone Cells

Enhanced Turbulence

The cyclonic flow pattern creates higher turbulence within the cell, resulting in better deposit quality and more uniform metal recovery compared to traditional flat plate designs.

Compact Footprint

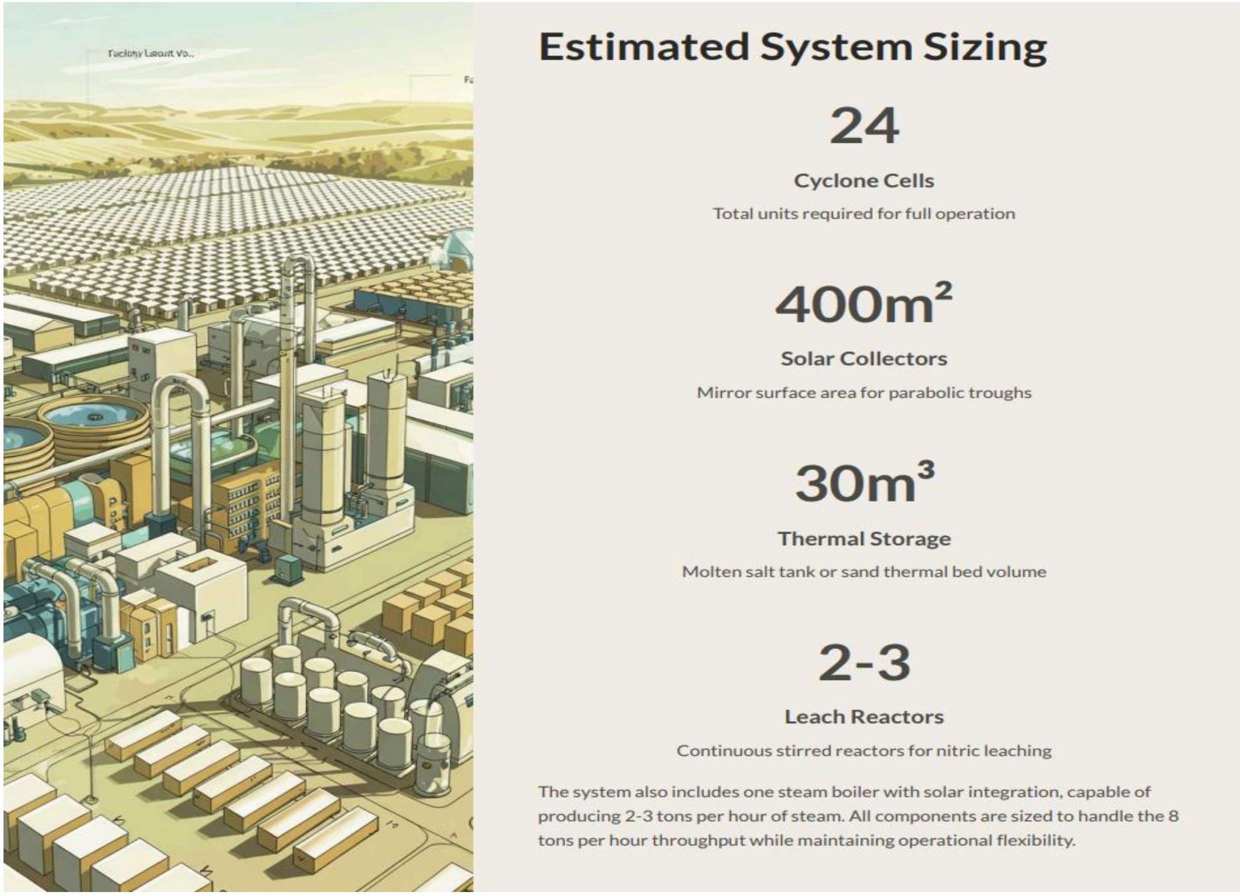
Vertical orientation and efficient design means cyclone cells require significantly less floor space than conventional flat plate electrowinning tanks for the same production capacity.

Reduced Maintenance

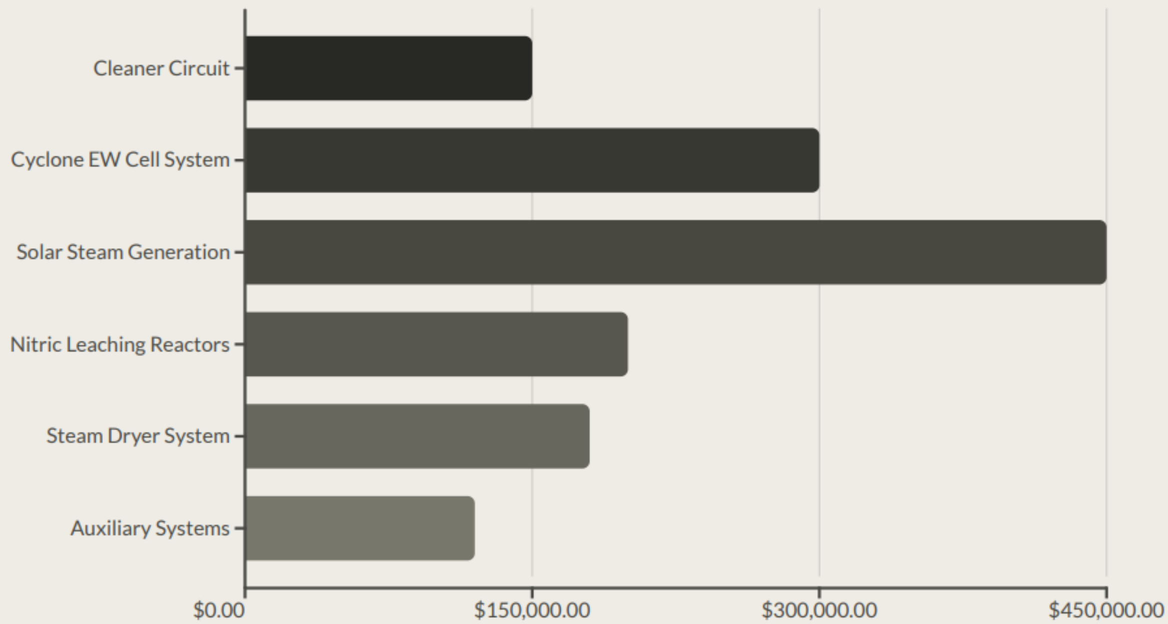
The self-cleaning nature of the cyclonic flow reduces scaling and buildup, leading to lower maintenance requirements and less downtime for cleaning operations.

Modular Scalability

The system can be easily expanded by adding additional cell modules, allowing for flexible capacity increases without major redesign or construction.


Estimated System Sizing
24
Cyclone Cells
Total units required for full operation
400m²
Solar Collectors
Mirror surface area for parabolic troughs
30m³
Thermal Storage
Molten salt tank or sand thermal bed volume
2-3
Leach Reactors
Continuous stirred reactors for nitric leaching
The system also includes one steam boiler with solar integration, capable of producing 2-3 tons per hour of steam. All components are sized to handle the 8 tons per hour throughput while maintaining operational flexibility.

Rough CAPEX Update



The total estimated capital expenditure is approximately $1.4 million. This is a very rough estimate based on a 100-200 tons per day full system with modular expandability. The largest single investment is in the solar steam generation system, which includes mirrors and thermal storage.

Key Notes

Modular Design

The system can scale by adding 12-cell cyclone banks, allowing for phased implementation and capacity expansion as needed without complete redesign.

Solar-Powered Efficiency

Solar-powered drying significantly reduces operational costs by minimizing diesel and electricity expenses while providing sustainable energy for continuous operation.

Strategic Value

Recovery of Ge, In, Te, Sb, Ga, and Hf provides access to critical metals with high market premiums, substantially improving project economics.

Cleaner Solution

The improved zinc sulfate solution reduces penalties when selling $ZnSO_4$ solution or cathodes, creating additional value beyond the recovered specialty metals.

RDI Report

Resource Development Inc

October 18, 2012

To: sstine@coronetmetals.com

Cc: dmalhotra@aol.com ; deepak@rdiminerals.com

From: tom@rdiminerals.com

Subject: Report of Preliminary Evaluation of Darwin Samples

Dear Steve,
This memorandum summarizes the results of testwork completed on two Darwin samples. The two samples are described in Table 1. The samples were received and weighted. Small hand specimens were taken out of each. The remaining material was blended and crushed through 6 mesh. One kilogram charges were split out, with one of them submitted for head analysis.

Table 1. Characterization of the Samples

Sample Name	Wt. Recd., kg	Head Analysis			
		Zn, %	Au, g/mt	Ag, g/mt	Te, ppm
DI - 5	28.5	8.97	0.140	53.4	27
C - 6	34.5	13.2	0.54	516	225

Additional analyses were performed on each sample head. Table 2 shows these results. The assay sheet can be found in Appendix A. As expected, the majority of the material appears to be silica (silicates), magnetite, limonite, limestone (calcite), some feldspars and zinc minerals. Environmentally unfriendly elements are generally not seen in appreciable amounts.

Table 2. Analysis of Head Samples by X-ray Fluorescence		
Assay	Sample	
	DI-5	C-6
Percent		
Na_2O	<0.5	<0.5
MgO	2	2
Al_2O_3	7	3
SiO_2	30	30
P_2O_5	0.7	<0.2
S	0.7	0.2
Cl	<0.1	<0.1
K_2O	0.6	1
CaO	16	11
TiO_2	<0.1	0.2
MnO_2	0.4	1.0
Fe_2O_3	14	14
BaO	<0.1	<0.1
ppm		
V	<50	<50
Cr	<200	<200
Co	<100	200
Ni	<100	<100
W	<100	<100
Cu	3000	400
Zn	100,000	130,000
As	<300	<300
Sn	<200	<200
Pb	30,000	6,000
Mo	<50	<50
Sr	<100	<100
U	<100	<100
Th	<100	<300
Nb	<100	<100
Zr	<100	<100
Rb	<100	<100
Y	<100	<100

Grind studies were performed on 1kg charges to determine grinding times required for the laboratory rod mill to achieve a P_{80} of 150 mesh. The data can be found in Appendix B.

Acid and ammonia leaching tests were run to evaluate the amenability of the zinc minerals to dissolution in strong acid and strong base. The zinc leaching tests were performed at a grind size of 80% passing 150 mesh in agitated containers with strong concentrated sulfuric acid (96.3% H_2SO_4) added to two of them. Strong concentrated ammonia hydroxide (29% NH_40H) was added to the other two. The resulting slurries were agitated for four hours. At the end of the four hour leach period, samples of pure pregnant solutions were taken from each slurry.

The residues were then washed extensively to remove any last traces of acid and ammonia prior to cyanide leaching. The acid leached residues required 5 to 6 separate high displacement washes with re-pulping at least three times during the process. The ammonia leached residues had better pulp characteristics and only required three high volume displacement washes with only one stage of repulping.

Again, the assumption was made that no zinc would dissolve during the cyanide leaching process, so a separate sample of zinc leach residue was unnecessary. Table 3 shows a summary of the conditions and results of the tests. Details of LT-1 through LT-4 can be found in Appendix C.

Table 3. Conditions and Results of Zinc Leaching Tests

Leach Test No.	Sample	Lixiviant	Recovery Zn, %	Cal. Head Zn	Wt. Loss, %	Reagent Added, kg/mt	
						H_2SO_4	NH_4OH_2
LT-1	DI - 5	Acid	97.3	11.5	19.5	433	-
LT-2	DI - 5	Ammonia	0.2	9.11	8.43	-	319
LT-3	C - 6	Acid	98.1	22.3	28.9	433	-
LT-4	C - 6	Ammonia	1.6	13.0	7.68	-	290

The acid dissolved almost all of the zinc (97.3% to 98.1%). The ammonia did not appreciably attack the zinc minerals (recovery of 0.2-1.6% Zn). It has been said that different species of zinc minerals are found in different parts of the mine. However, the source of the samples used in this test series is not known.

A second series of cyanide leach tests were conducted on the residues from the zinc leaching tests. Bottle roll tests were performed with strong sodium cyanide conditions (10g/l). A summary of the conditions and results of those tests are shown in Table 4 and the detailed leach sheets can be found in Appendix D.

Table 4. Summary of Gold and Silver Leaching Tests										
Test No.	Sample	Previous Test		Extraction %		Reagent Consumption, kg/mt			Cal. Head, g/mt	
		Acid	Ammonia	Au	Ag	NaCN	Lime	pH	Au	Ag
BR-1	DI - 5	X	-	74.2	71.0	5.25	59.0	11.0	0.256	65.3
BR-2	DI - 5	-	X	47.7	77.5	11.0	4.9	12.0	0.100	61.3
BR-3	C - 6	X	-	95.2	90.0	4.01	62.3	11.0	1.83	464.4
BR-4	C - 6	-	X	69.8	77.7	8.44	4.9	12.8	0.35	442.3

The extraction of gold ranged from 47.7% to 74.2% for the DI-5 sample and from 71.0% to 77.5% for the COR-6 sample. Likewise, silver extractions ranged from 69.8% to 95.2% and 77.7% to 90.0% for the DI-5 and COR-6 samples, respectively.

Other items to note include the very high (59.0 to 62.3 kg/tonne) lime consumed in the cyanide leach tests following acid leaching. This suggests the great difficulty in removing acid from the residue slurry. Generally, the slurries were very fine, thick and viscous and presented considerable difficulty in filtering and washing.

It is also interesting to note that the gold and silver recoveries were slightly enhanced in the tests following ammonia leaching. For these particular ore types, ammonia leaching would not be considered, so the enhancements would not be expected.

Additional analytical data are available that may shed some light on the mechanics of these processes. The pregnant solutions and residues from the acid and ammonia leach tests were analyzed by multi-element ICP for 22 elements. Tables 5 and 6 show the results of the assays, with the detailed analytical report sheets included in Appendix E.

Table 5. Results of ICP Analyses for Sample DI - 5						
Element	Assay, ppm					
	LT - 1 Acid			LT - 2 Ammonia		
	Feed[1]	Preg	Residue	Feed[1]	Preg	Residue
Al	37,100	5,318	29,000	37,100	8.6	38,700
As	<300	2.4	112	<300	0.1	93
Ba	<1,000	0.2	426	<1,000	<0.1	489
Bi		1.8	60		2.1	28
Ca	113,600	664	72,800	113,600	2.1	83,500
Cd		62.1	28		0.2	143
Co	<100	4.5	16	<100	<0.1	20
Cr	<200	5.2	146	<200	<0.1	112
Cu	3,000	1,042	779	3000	3.4	2,672
Fe	98,000	8805	99,700	98,000	4.7	97,700
K	4260	111	5,900	4260	16.7	5,200
Mg	12,000	818	7,400	12,000	1.0	8,100
Mn	2,520	1006	1,296	2,520	1.0	3,610
Mo	<50	0.7	15	<50	0.2	11
Na	<3,700	103	300	<3700	21.5	200
Ni	<100	13.1	92	<100	<0.1	72
Pb	30,000	7.2	33,656	30,000	0.9	29,762
Sr	<100	0	84	<100	0.3	77
Ti	<600	8.5	700	<600	<0.1	500
V	<50	4.0	46	<50	0.1	43
W	<100	11.3	172	<100	0.8	190
Zn	100,000	43,951	4,050	100,000	76	91,552

[1]Feed elements calculated from compounds - Al_2O_3, BaO, CaO, Fe_2O_3, K_2O, MgO, MnO_2, Na_2O and TiO_2

Table 6. Results of ICP Analyses for Sample C - 6						
Element	Assay, ppm					
	LT - 3 Acid			LT - 4 Ammonia		
	Feed(1)	Preg	Residue	Feed(1)	Preg	Residue
Al	15,900	1243	38,900	15,900	1.8	32,500
As	<300	36.1	106	<300	0.5	89
Ba	<1,000	0.1	342	<1,000	<0.1	288
Bi		712	1,300		2.1	1449
Ca	77,000	389	118,300	77,000	2.5	94,600
Cd		125	47		4.0	225
Co	200	73.6	97	200	0.2	199
Cr	<200	10.3	50	<200	<0.1	136
Cu	400	221	459	400	2.3	607
Fe	98,000	9,861	120,800	98,000	2.1	101,800
K	7,100	46.5	9,200	7,100	12.5	7,700
Mg	12,000	618	9,200	12,000	0.8	9,400
Mn	6,300	1667	10,225	6,300	0.8	11,307
Mo	<50	1.4	10	<50	0.5	11
Na	<3,700	106	300	<3,700	23.4	300
Ni	<100	42.7	72	<100	0.4	145
Pb	6,000	7.2	8,271	6,000	0.2	7,594
Sr	<100	0.3	72	<100	<0.1	67
Ti	1,200	21.5	2,100	1,200	0.1	1,600
V	<50	3.6	68	<50	0.4	55
W	<100	18.4	167	<100	2.4	143
Zn	130,000	84,240	5,547	130,000	1,048	129,318

Note: (1)Feed elements calculated from compounds - Al_2O_3, BaO, CaO, Fe_2O_3, K_2O, MgO, MnO_2, Na_2O and TiO_2

The data shows that the metals were leached more strongly with acid as opposed to ammonia. The elements that were dissolved in nearly equivalent amounts were calcium, manganese and sodium. It is also interesting to note that even with strong leaching environmentally sensitive elements such as arsenic, cadmium, bismuth, chromin and lead were not appreciably dissolved.

Sincerely Yours,

Tom Randall
Senior Project Manager
Resource Development Inc.
Phone: 303-422-1176
Website: www.rdiminerals.com

APPENDIX A

X-RAY FLUORESCENCE ANALSIS OF HEAD SAMPLES

The Mineral Lab, Inc.
12929 W. 26th, Suite 100
Golden, CO 80401
(303) 232-8708

June 26, 2012
Lab no. 212372

Mr. Ryan Allen
Resource Development, Inc
11475 West I-70 Frontage Road North
Wheat Ridge, Colorado 80033

Dear Mr. Allen:

Enclosed are the x-ray fluorescence (XRF) results for two "COR Head" samples. This report will be mailed and emailed to you and emailed to Deepak Malhotra, as usual.

A representative portion of each sample was ground to approximately -400 mesh in a steel swing mill and then analyzed by our standard XRF procedure for 31 major, minor and trace elements. The samples were diluted ten times with pure silica and re-run because of the high Zn concentration. The relative precision/accuracy for this procedure is ~5–10% for major–minor elements and ~10–15% for trace elements (those elements listed in ppm) at levels greater than twice the detection limit in samples of average geologic composition. This analytical procedure is best suited for the analysis of "average" geologic samples (ie, the matrix consists primarily of elements/oxides listed in the top portion of the table: Na_2O - BaO). Detection limits become higher and analytical accuracy becomes poorer the farther a sample deviates from this composition (ie, if any "trace" element such as Zn or Pb exceeds ~1% and/or if any element of significant concentration is not included in the analysis). These samples are far outside the compositional range for which this procedure is best suited. A replicate sample and a standard reference material ("GSP-2", a USGS standard rock) were analyzed with the samples to demonstrate analytical reproducibility for your samples and analytical accuracy for a geologic standard, respectively. The accepted ("known") values for the quality control standard are listed with the XRF results.

Thank you for the opportunity to be of continuing service to RDI.

Sincerely,

Peggy Dalheim

Peggy Dalheim

Resource Development, Inc
XRF Results for Samples (complete labels given below)

June 26, 2012
Lab no. 212372

IDENT	Na_2O	MgO	Al_2O_3	SiO_2	P_2O_5	S	Cl	K_2O	CaO	TiO_2	MnO_2	Fe_2O_3	BaO
	Wt %												
1	<0.5	2.	7.	30.	0.7	0.7	<0.1	0.6	16.	<0.1	0.4	14.	<0.1
2	<0.5	2.	3.	30.	<0.2	0.2	<0.1	1.	11.	0.2	1.0	14.	<0.1
Quality Control - Replicate (R) sample and standard reference material (GSP-2) analyzed with samples													
1(R)	<0.5	2.	8.	30.	0.7	0.7	<0.1	0.6	16.	<0.1	0.4	14.	<0.1
GSP-2-XRF	2.98	1.09	15.0	68.0	0.30	<0.05	0.03	5.69	2.14	0.68	0.04	4.97	0.17
GSP-2-known	2.78	0.96	14.9	66.6	0.29	----	----	5.38	2.10	0.66	0.04	4.90	0.15

IDENT	V	Cr	Co	Ni	W	Cu	Zn	As	Sn	Pb	Mo	Sr	U
	PPM												
1	<50	<200	<100	<100	<100	3000	100000	<300	<200	30000	<50	<100	<100
2	<50	<200	200	<100	<100	400	130000	<300	<200	6000	<50	<100	<100
Quality Control													
1(R)	<50	<200	<100	<100	<100	3000	90000	<300	<200	30000	<50	<100	<100
GSP-2-XRF	56	23	10	14	<10	49	138	<20	<50	49	<10	243	<20
GSP-2-known	52	20	7	17	--	43	120	--	--	42	--	240	2

Ident	Th	Nb	Zr	Rb	Y
	PPM				
1	<100	<100	<100	<100	<100
2	<300	<100	<100	<100	<100
Quality Control					
1(R)	<100	<100	<100	<100	<100
GSP-2-XRF	101	22	566	206	41
GSP-2-known	105	27	550	245	28

Ident	RDI Label
1	COR DI-5 Head
2	COR C-6 Head
1(R)	COR DI-5 Head

Note - Divide "ppm" (parts per million) by 10,000 to convert to "Wt %" (weight percent = parts per hundred)

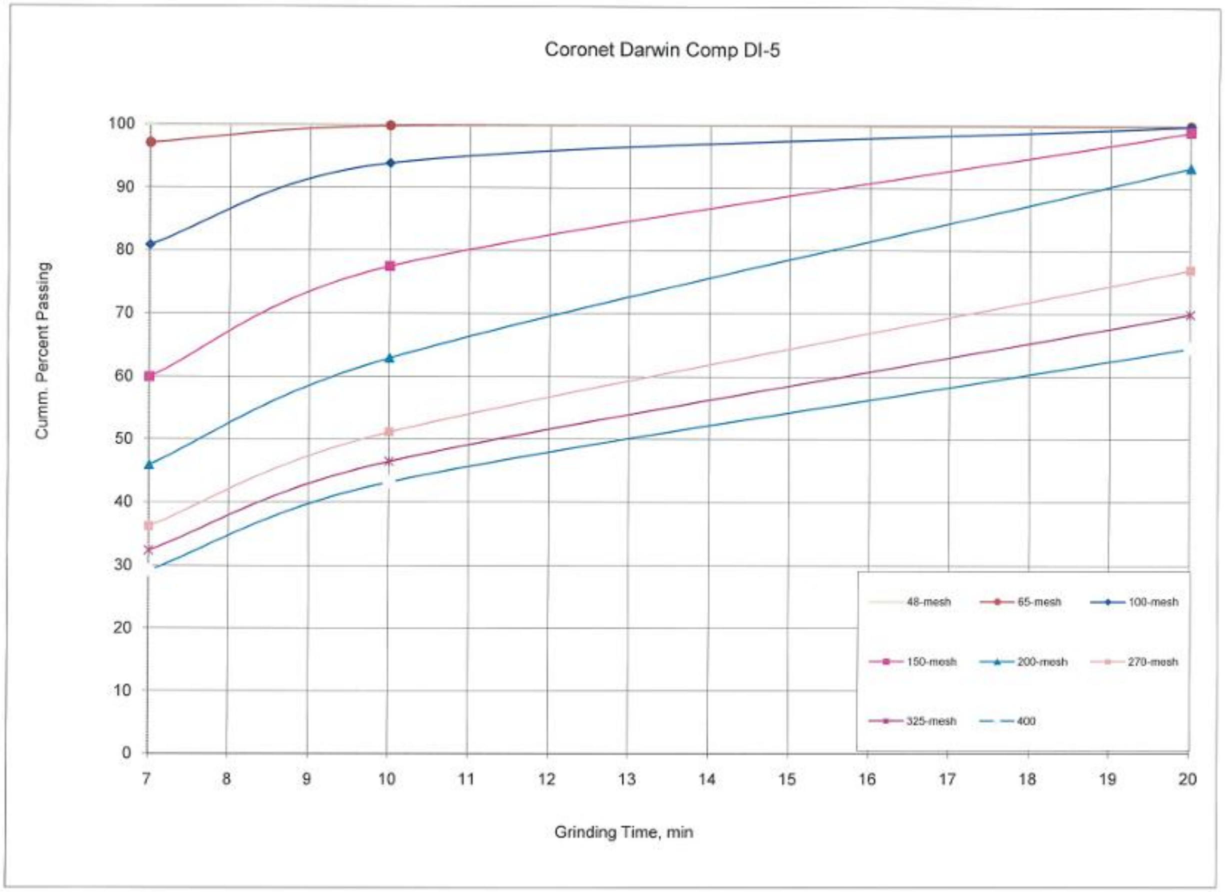
Initial PAQ

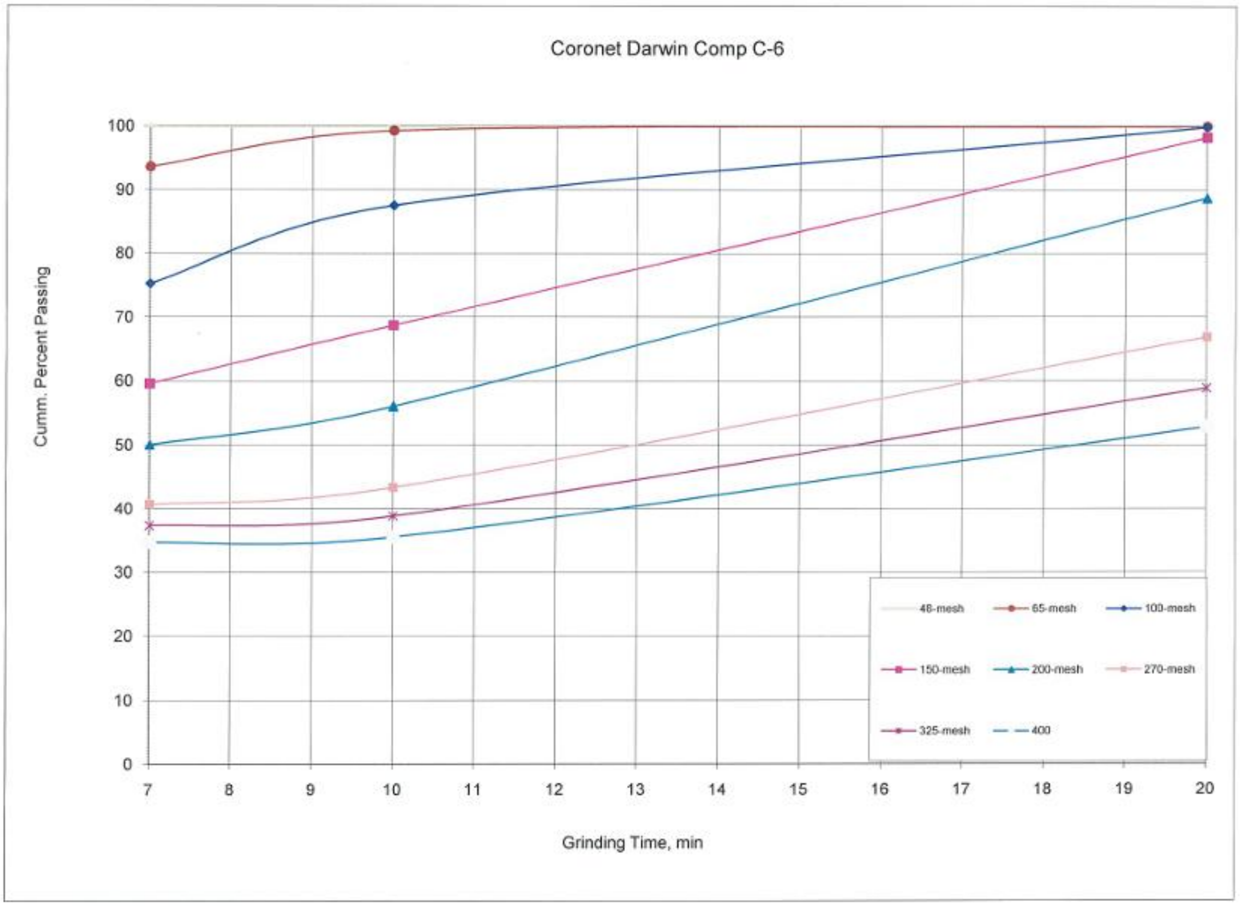
Date 26 June 2012

Analysis Performed By The Mineral Lab, Inc

APPENDIX B

GRIND STUDIES


Coronet Darwin Comp DI-5
Cumm. Percent Passing
Grinding Time, min
0
10
20
30
40
50
60
70
80
90
100
7
8
9
10
11
12
13
14
15
16
17
18
19
20
48-mesh
65-mesh
100-mesh
150-mesh
200-mesh
270-mesh
325-mesh
400


Coronet Darwin Comp C-6
Cumm. Percent Passing
100
90
80
70
60
50
40
30
20
10
0
7
8
9
10
11
12
13
14
15
16
17
18
19
20
Grinding Time, min
48-mesh
65-mesh
100-mesh
150-mesh
200-mesh
270-mesh
325-mesh
400

Appendix B

Particle Size Analysis 1 7 min **Project:** Coronet Nevada
Date: 4-Jun-12

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 7 minutes.

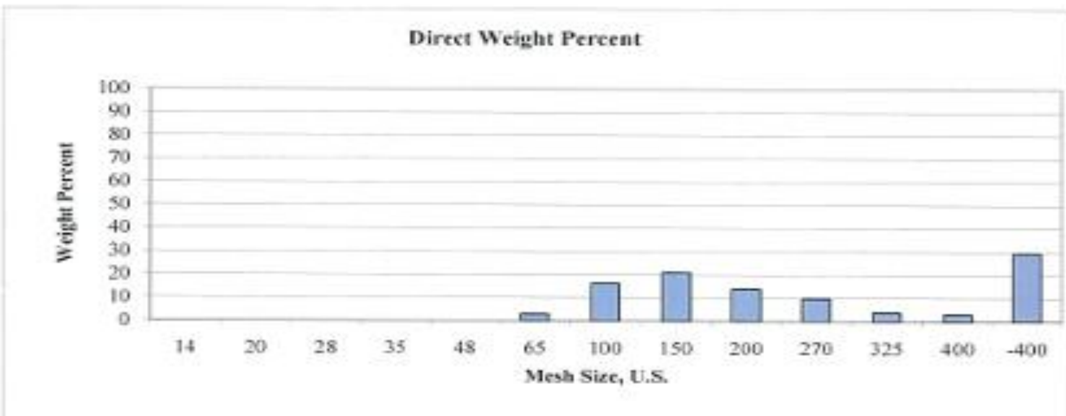
Sample: Approximately 1.0 kg of DI-5

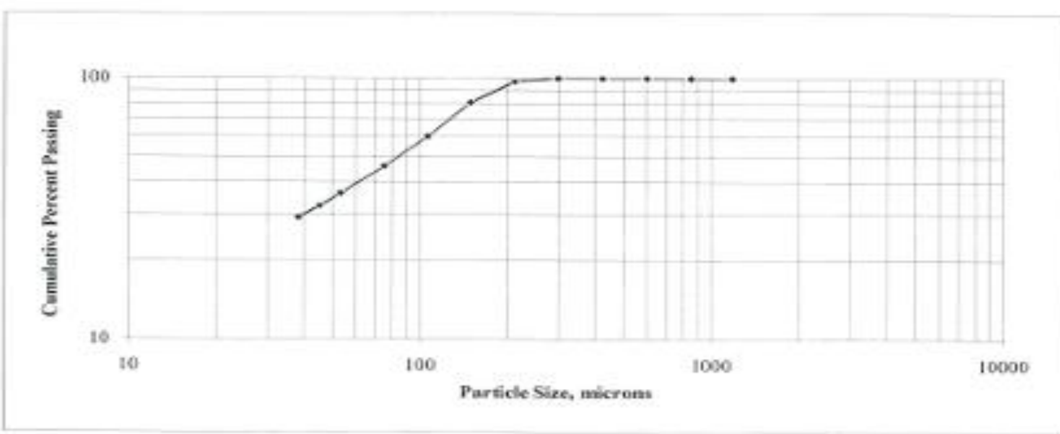
Procedure: The sample was rod mill ground for 7-minutes at 40% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
				Cumulative	
Mesh (Retained)	Microns	Grams	%	Passing	Retained
Feed (Calculated)		999.7	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	29.0	2.90	97.10	2.90
100	150	162.3	16.23	80.87	19.13
150	106	209.0	20.90	59.96	40.04
200	75	140.4	14.04	45.92	54.08
270	53	97.1	9.72	36.20	63.80
325	45	38.0	3.80	32.41	67.59
400	38	31.5	3.15	29.25	70.75
-400	-38	292.4	29.25	0.00	100.00

Product Size Passing 80% (P80): = 165 Microns





Appendix B

Particle Size Analysis 1 10 min

Project: Coronet Nevada
Date: 4-Jun-12

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 10 minutes.

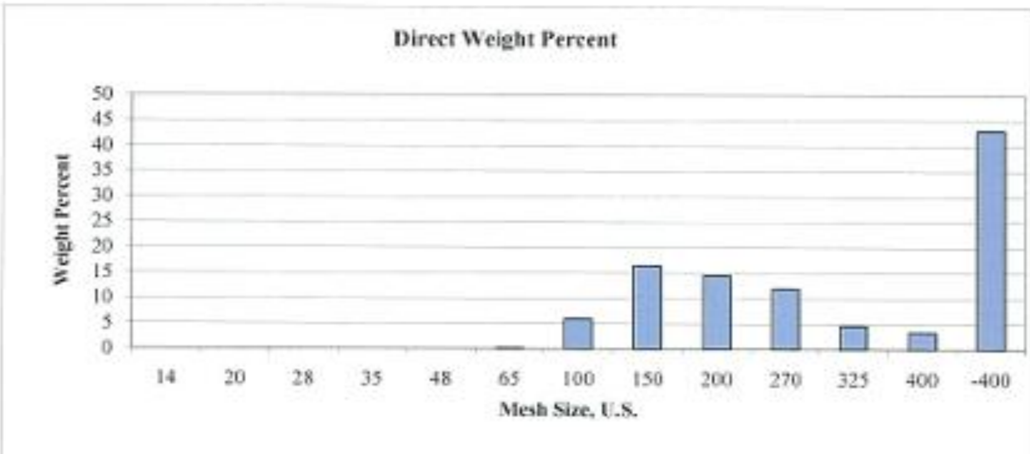
Sample: Approximately 1.0 kg of DI-5

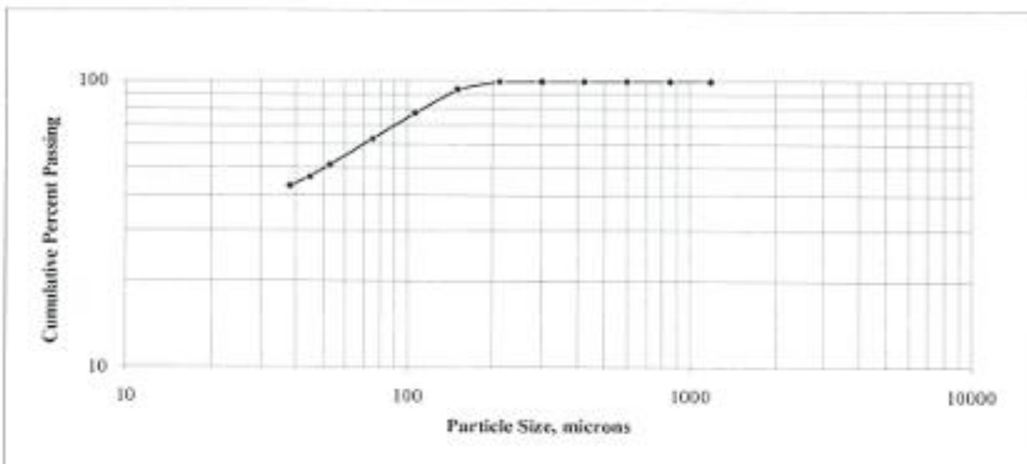
Procedure: The sample was rod mill ground for 10-minutes at 50% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		953.6	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	1.5	0.15	99.85	0.15
100	150	56.5	5.92	93.92	6.08
150	106	157.2	16.49	77.43	22.57
200	75	137.9	14.46	62.97	37.03
270	53	113.0	11.85	51.12	48.88
325	45	44.7	4.68	46.44	53.56
400	38	31.2	3.27	43.17	56.83
-400	-38	411.7	43.17	0.00	100.00

Product Size Passing 80% (P80): = 112 Microns





Appendix B

Particle Size Analysis 1 20 min

Project: Coronet Nevada
Date: 4-Jun-12

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 20 minutes.

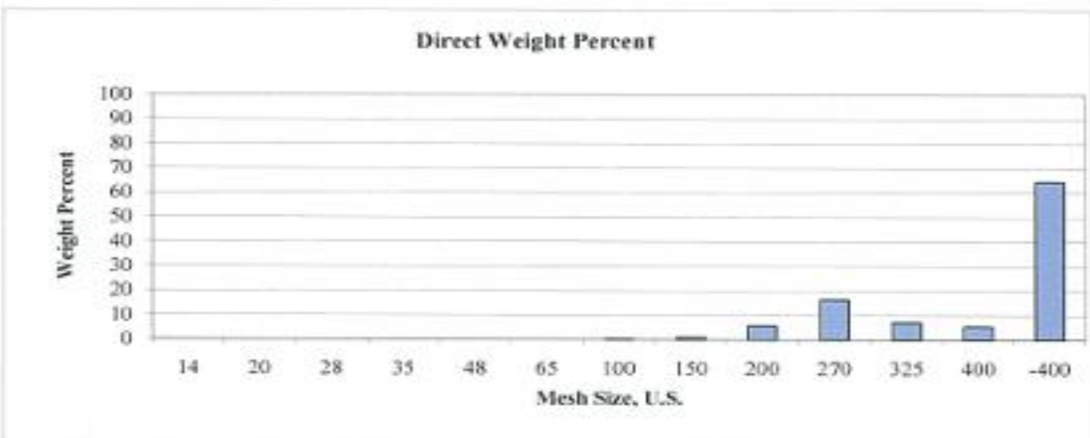
Sample: Approximately 1.0 kg of DI-5

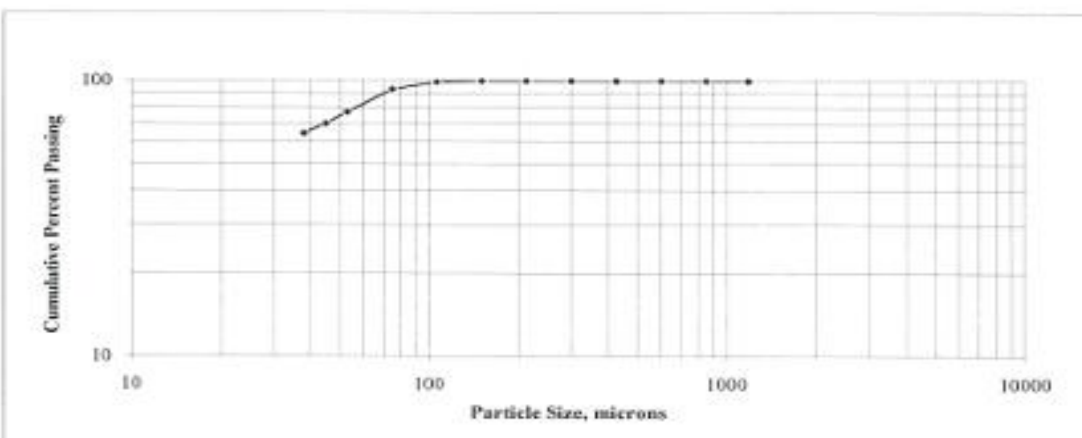
Procedure: The sample was rod mill ground for 20-minutes at 50% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		957.9	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	0.0	0.00	100.00	0.00
100	150	1.0	0.10	99.90	0.10
150	106	8.9	0.93	98.96	1.04
200	75	54.3	5.67	93.29	6.71
270	53	156.3	16.31	76.98	23.02
325	45	67.6	7.06	69.92	30.08
400	38	51.6	5.38	64.54	35.46
-400	-38	618.2	64.54	0.00	100.00

Product Size Passing 80% (P80) = 59 Microns





Appendix B

Particle Size Analysis 2 20 min **Project:** **Coronet**
Date: **4-Jun-12**

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 20 minutes.

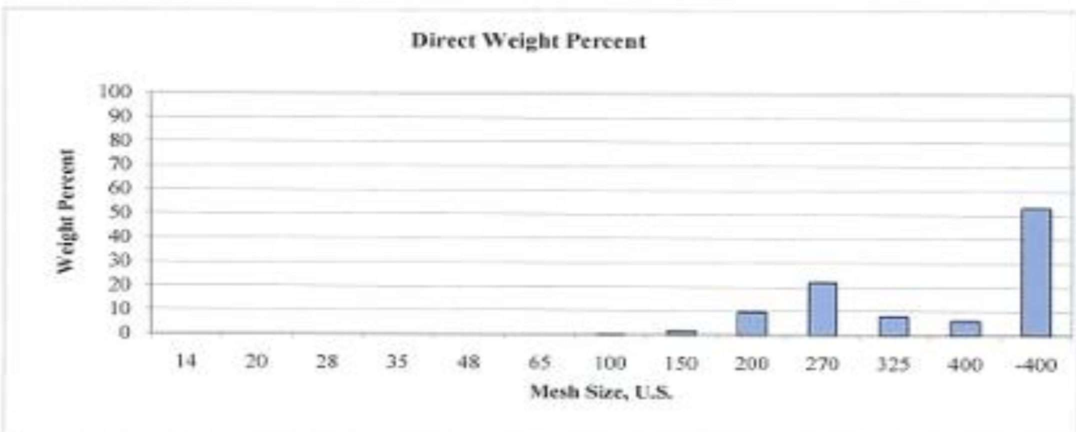
Sample: Approximately 1.0 kg of COR C-6

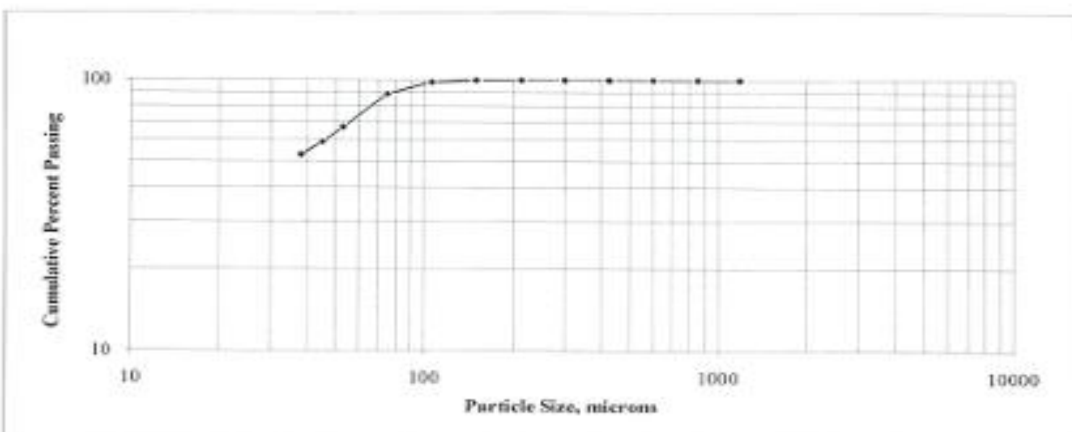
Procedure: The sample was rod mill ground for 20-minutes at 450% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		973.7	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	0.0	0.00	100.00	0.00
100	150	1.4	0.14	99.86	0.14
150	106	15.6	1.60	98.26	1.74
200	75	92.9	9.54	88.71	11.29
270	53	212.8	21.85	66.86	33.14
325	45	77.5	7.96	58.90	41.10
400	38	58.5	6.01	52.89	47.11
-400	-38	515.0	52.89	0.00	100.00

Product Size Passing 80% (P80): = 66 Microns





Appendix B

Particle Size Analysis 3 7 min **Project:** **Coronet** **Date:** **4-Jun-12**

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 7 minutes.

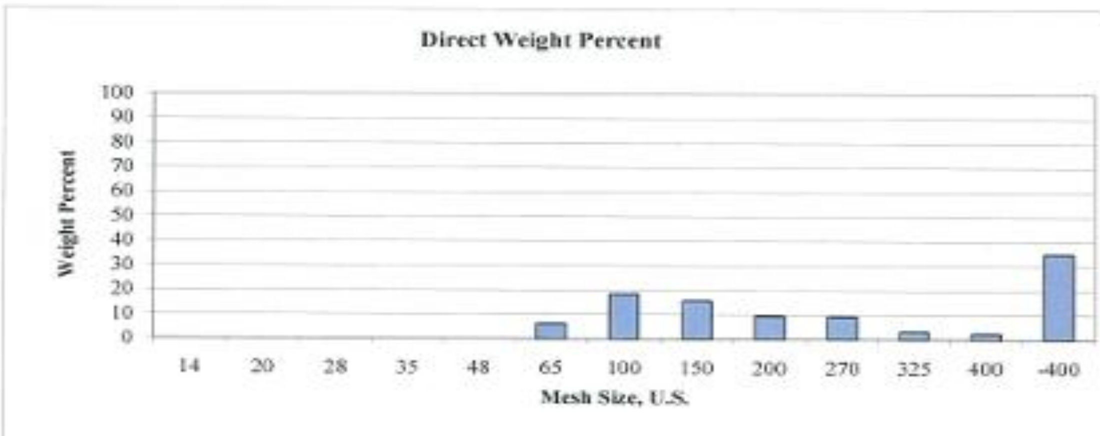
Sample: Approximately 1.0 kg of C-6

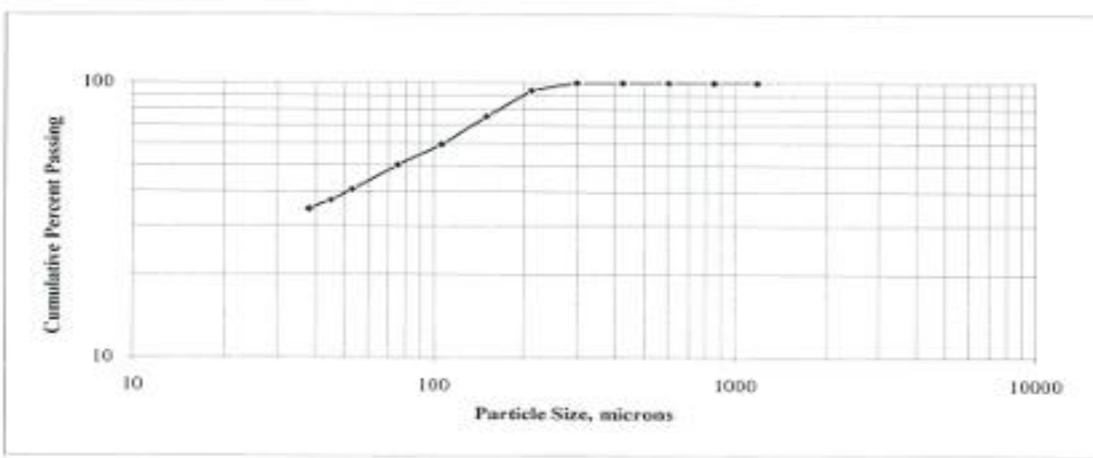
Procedure: The sample was rod mill ground for 7-minutes at 40% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		1,015.5	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	64.0	6.31	93.69	6.31
100	150	186.6	18.37	75.32	24.68
150	106	159.4	15.70	59.62	40.38
200	75	96.6	9.51	50.11	49.89
270	53	95.3	9.39	40.72	59.28
325	45	34.2	3.36	37.36	62.64
400	38	26.6	2.62	34.74	65.26
-400	-38	352.8	34.74	0.00	100.00

Product Size Passing 80% (P80): = 180 Microns





Appendix B

Particle Size Analysis 1 10 min **Project:** **Coronet** **Date:** **4-Jun-12**

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 10 minutes.

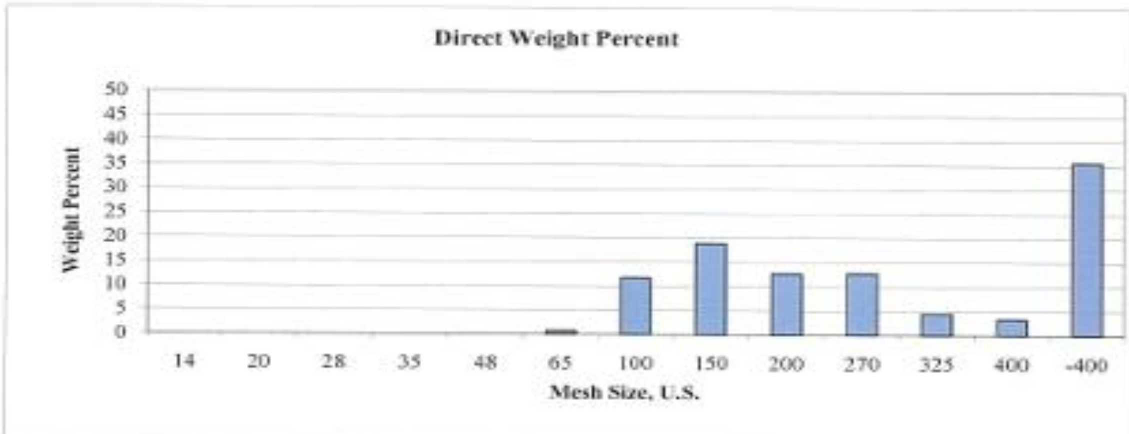
Sample: Approximately 1.0 kg of C-6

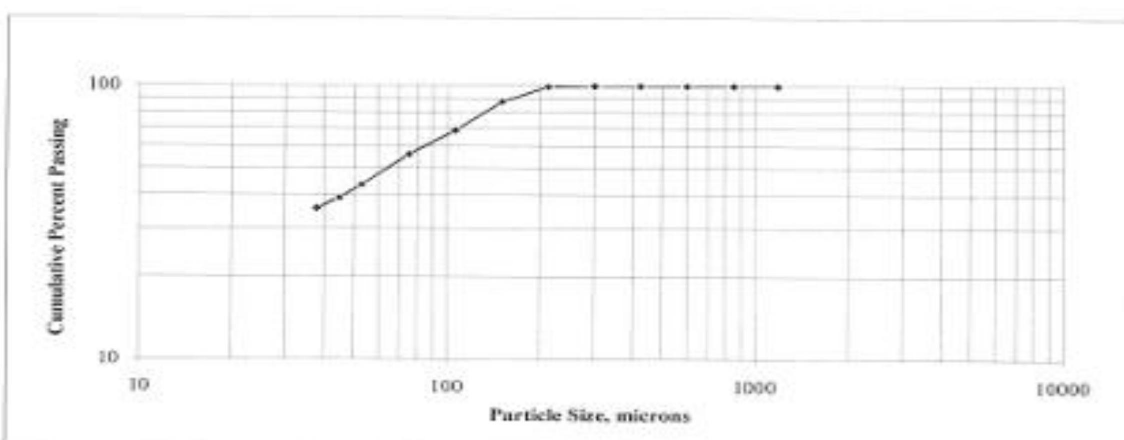
Procedure: The sample was rod mill ground for 10-minutes at 50% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		973.2	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	7.0	0.72	99.28	0.72
100	150	113.6	11.67	87.61	12.39
150	106	183.7	18.87	68.74	31.26
200	75	123.0	12.64	56.10	43.90
270	53	123.8	12.72	43.39	56.61
325	45	43.7	4.49	38.89	61.11
400	38	32.1	3.30	35.59	64.41
-400	-38	346.4	35.59	0.00	100.00

Product Size Passing 80% (P80): = 130 Microns





Appendix B

Particle Size Analysis 1 10 min **Project:** Coronet **Date:** 4-Jun-12

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 10 minutes.

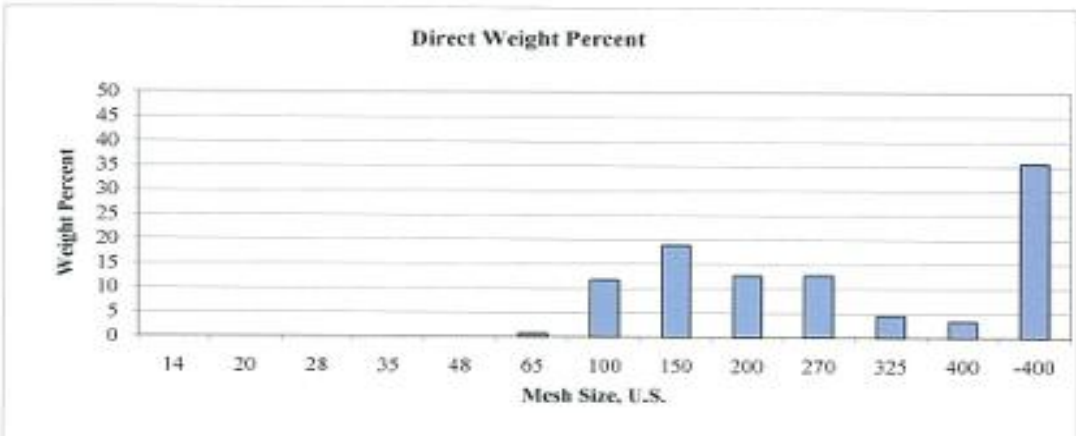
Sample: Approximately 1.0 kg of C-6

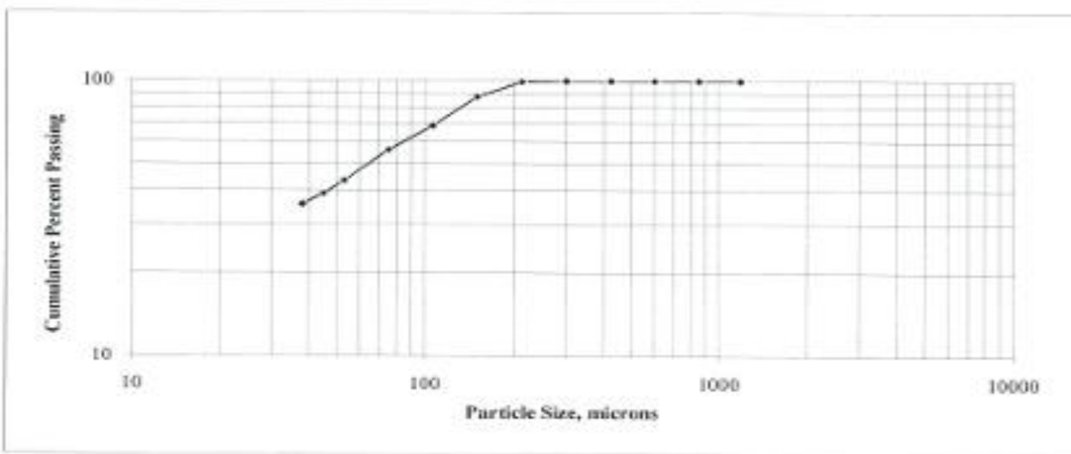
Procedure: The sample was rod mill ground for 10-minutes at 50% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		973.2	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	7.0	0.72	99.28	0.72
100	150	113.6	11.67	87.61	12.39
150	106	183.7	18.87	68.74	31.26
200	75	123.0	12.64	56.10	43.90
270	53	123.8	12.72	43.39	56.61
325	45	43.7	4.49	38.89	61.11
400	38	32.1	3.30	35.59	64.41
-400	-38	346.4	35.59	0.00	100.00

Product Size Passing 80% (P80): ~ 130 Microns





Appendix B

Particle Size Analysis 2 20 min **Project:** Coronet **Date:** 4-Jun-12

Purpose: To determine the particle size distribution of the sample after rod mill grinding for 20 minutes.

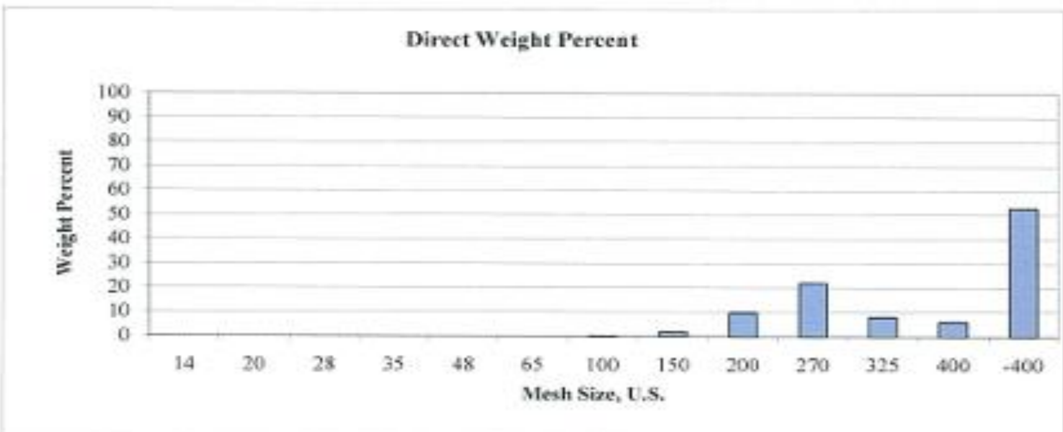
Sample: Approximately 1.0 kg of COR C-6

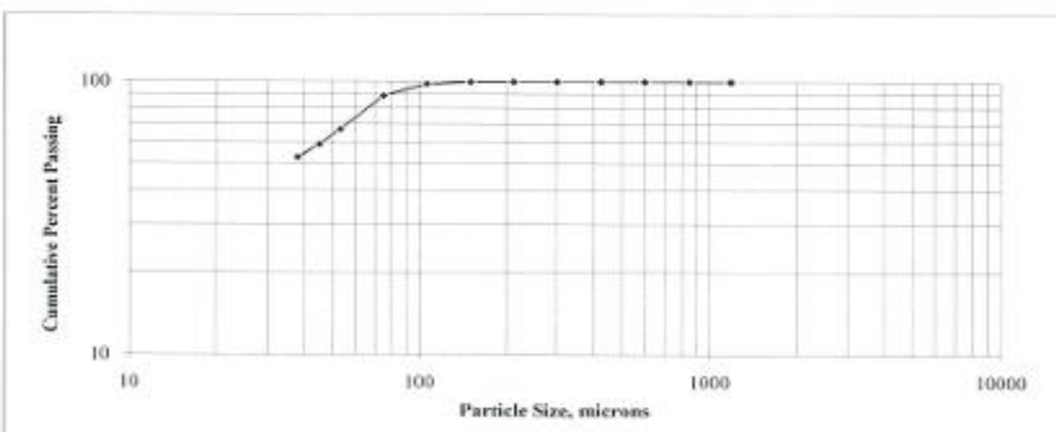
Procedure: The sample was rod mill ground for 20-minutes at 450% solids and wet screened at 400-mesh and the plus fraction dried. The dried plus 400-mesh material was dry screened (Ro-Tap) for 30-minutes on the screens shown. The minus 400-mesh fraction was allowed to settle, decanted and filtered. The screen fractions were weighed and the total particle size distribution calculated.

Results:

Product		Weight			
Mesh (Retained)	Microns	Grams	%	Cumulative Passing	Cumulative Retained
Feed (Calculated)		973.7	100.0		
14	1180	0.0	0.00	100.00	0.00
20	850	0.0	0.00	100.00	0.00
28	600	0.0	0.00	100.00	0.00
35	425	0.0	0.00	100.00	0.00
48	300	0.0	0.00	100.00	0.00
65	212	0.0	0.00	100.00	0.00
100	150	1.4	0.14	99.86	0.14
150	106	15.6	1.60	98.26	1.74
200	75	92.9	9.54	88.71	11.29
270	53	212.8	21.85	66.86	33.14
325	45	77.5	7.96	58.90	41.10
400	38	58.5	6.01	52.89	47.11
-400	-38	515.0	52.89	0.00	100.00

Product Size Passing 80% (P80): = 66 Microns





APPENDIX C

ZINC LEACHING TEST DATA

Appendix C

Agitated Leach Test No. 1

DI-5 ACID

RDi Project: Coronet Darwin
Date: 12-Jun-12

Purpose: To examine the dissolution of zinc from the sample

Sample: Approximately 1-kilogram DI-5 Sample

Procedure: The stage ground slurry was transferred to a stainless steel leach vessel and the percent solids adjusted to allow for addition of acid to a final pulp density of 40%. Concentrated sulfuric acid was added to give an acid concentration of 20%. Additional water was needed, however, due to the very thick and viscous nature of the pulp after acid had been added. The slurry was strongly agitated for four hours. A gross weight of vessel and slurry was taken. A sample was then taken out and filtered to yield a pure pregnant solution that was assayed for zinc. The residue was washed six times with re-pulping required three times. The resulting residue was used as feed for a subsequent cyanide leach test.

Conditions:

Grind	Leach Time	Acid Conc.	Initial % Solids	Final % Solids
p80 150 mesh	4 hours	18.4%	34%	32%

Summary of Results:

Parameter	Zn
Extraction, % (1)	97.3
Assayed Head, %	9.06
Calculated Head, %	11.5
Final Tail Assay, %	0.388

Extraction, % (1)		H_2SO_4 Added
Hr.	Zn	kg/mt
4	97.3	433

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g		
hrs	g	ml	100% H_2SO_4	Acid H_2O	Total Acid
Pre	2900	1900			
0	3350	2350	433	16.6	450
4	3355	2550			
Total			433	16.6	450

B. Products and Analyses

Leach Product	Weight g	Volume ml	Zinc %, ppm	Zinc mg/l
Feed (analyzed)	1000		9.06	
Feed (computed)	805.1		11.5	
4 hour Preg		2550		43,900
4 hour Dry Residue	805.1		3,880	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

Appendix C

Agitated Leach Test No. 2

DI-5 AMMONIA

RDi Project: Coronet Darwin
Date: 12-Jun-12

Purpose: To examine the dissolution of zinc from the sample

Sample: Approximately 1-kilogram DI-5 Sample

Procedure: The stage ground slurry was transferred to a stainless steel leach vessel and the percent solids adjusted to allow for addition of ammonia to a final pulp density of 50%. Concentrated ammonium hydroxide was added to give an ammonia concentration of 20%. The slurry was strongly agitated for four hours. A gross weight of vessel and slurry was taken. A sample was then taken out and filtered to yield a pure pregnant solution that was assayed for zinc. The residue was washed three times. The resulting residue was used as feed for a subsequent cyanide leach test.

Conditions:

Grind	Leach Time	NH4OH Conc.	Initial % Solids	Final % Solids
p80 150 mesh	4 hours	14.8%	49%	43%

Summary of Results:

Parameter	Zn
Extraction, % (1)	0.2
Assayed Head, %	9.06
Calculated Head, %	9.11
Final Tail Assay, %	9.93

Extraction, % (1)		NH_4OH Added
Hr.	Zn	kg/mt
4	0.2	319

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g		
hrs	g	ml	100% NH_4OH	NH_4OH Water	Total NH_4OH
Pre	2060	1060			
0	3161	2161	319	781	1100
4	3025	2109			
Total			319	781	1100

B. Products and Analyses

Leach Product	Weight g	Volume ml	Zinc %, ppm	Zinc mg/l
Feed (analyzed)	1000		9.06	
Feed (computed)	915.7		9.11	
4 hour Preg		2109		72
4 hour Dry Residue	915.7		99,300	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

Appendix C
Agitated Leach Test No. 3

C-6 ACID

RDi Project: Coronet Darwin
Date: 12-Jun-12

Purpose: To examine the dissolution of zinc from the sample

Sample: Approximately 1-kilogram C-6 Sample

Procedure: The stage ground slurry was transferred to a stainless steel leach vessel and the percent solids adjusted to allow for addition of acid to a final pulp density of 40%. Concentrated sulfuric acid was added to give an acid concentration of 20%. Additional water was needed, however, due to the very thick and viscous nature of the pulp after acid had been added. The slurry was strongly agitated for four hours. A gross weight of vessel and slurry was taken. A sample was then taken out and filtered to yield a pure pregnant solution that was assayed for zinc. The residue was washed six times with re-pulping required three times. The resulting residue was used as feed for a subsequent cyanide leach test.

Conditions:

Grind	Leach Time	Acid Conc.	Initial % Solids	Final % Solids
p80 150 mesh	4 hours	18.4%	34%	27%

Summary of Results:

Parameter	Zn
Extraction, % (1)	98.1
Assayed Head, %	13.2
Calculated Head, %	22.3
Final Tail Assay, %	0.584

Extraction, % (1)		H_2SO_4 Added
Hr.	Zn	kg/mt
4	98.1	433

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g		
hrs	g	ml	100% H_2SO_4	Acid H_2O	Total Acid
Pre	2900	1900			
0	3350	2350	433	16.6	450
4	3354	2643			
Total			433	16.6	450

B. Products and Analyses

Leach Product	Weight g	Volume ml	Zinc %, ppm	Zinc mg/l
Feed (analyzed)	1000		13.2	
Feed (computed)	710.8		22.3	
4 hour Preg		2643		82,650
4 hour Dry Residue	710.8		5,840	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

Appendix C
Agitated Leach Test No. 4

C-6 AMMONIA

RDi Project: Coronet Darwin
Date: 12-Jun-12

Purpose: To examine the dissolution of zinc from the sample

Sample: Approximately 1-kilogram DI-5 Sample

Procedure: The stage ground slurry was transferred to a stainless steel leach vessel and the percent solids adjusted to allow for addition of ammonia to a final pulp density of 50%. Concentrated ammonium hydroxide was added to give an ammonia concentration of 20%. The slurry was strongly agitated for four hours. A gross weight of vessel and slurry was taken. A sample was then taken out and filtered to yield a pure pregnant solution that was assayed for zinc. The residue was washed three times. The resulting residue was used as feed for a subsequent cyanide leach test.

Conditions:

Grind	Leach Time	NH4OH Conc.	Initial % Solids	Final % Solids
p80 150 mesh	4 hours	14.9%	51%	49%

Summary of Results:

Parameter	Zn
Extraction, % (1)	1.6
Assayed Head, %	13.2
Calculated Head, %	12.9
Final Tail Assay, %	13.8

Extraction, % (1)		NH_4OH Added
Hr.	Zn	kg/mt
4	1.6	290

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g		
hrs	g	ml	100% NH_4OH	NH_4OH Water	Total NH_4OH
Pre	1950	950			
0	2950	1950	290	710	1000
4	2814	1891			
Total			290	710	1000

B. Products and Analyses

Leach Product	Weight g	Volume ml	Zinc %, ppm	Zinc mg/l
Feed (analyzed)	1000		13.2	
Feed (computed)	923.2		12.9	
4 hour Preg		1891		1,115
4 hour Dry Residue	923.2		137,600	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

APPENDIX D

GOLD AND SILVER LEACHING TEST DATA

Appendix D

Bottle Roll Leaching Test BR 1

DI-5 from Acid

RDi Project: Coronet Darwin
Date: 12-Jun-12

Purpose: To examine the extraction of gold and silver from an acid leached residue.

Sample: Approximately 805 grams of washed residue from Acid Leach Test 1 performed on DI-5 Sample

Procedure: The material was transferred to a bottle and was adjusted to 33% (2) solids. The pH was adjusted to ~11 with hydrated lime. Sodium cyanide was then added to a calculated level of 10.0 g/l. At 6 and 24 hours, the pH and free cyanide were determined. In addition, solution samples were taken to determine the gold and silver contents. Sodium cyanide was added to return the level to 10.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
p80 150 mesh	48 hours	10.01 g/L	33% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	74.2	71.0
Assayed Head, g/mt		
Calculated Head, g/mt	0.21	52.6
Final Tail Assay, g/mt	0.07	19.0

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
6	56.9	77.4	3.45
24	65.2	70.0	5.16
48	74.2	71.0	5.25

Cyanide Consumption	5.25	kg NaCN/metric ton ore
Lime Added	59.0	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	Residual Reagents NaCN g/L	D.O. ppm	pH Initial	pH Adjust
0	2466	1661	16.80	45.0					11.1
6	2483	1678	2.76	1.00		8.36	4.80	10.8	11.0
24	2465	1660	1.21	1.50		9.28	4.75	10.7	11.1
48	2466	1661				9.96	4.60	10.8	
Total			20.77	47.50					
				(36.0)	CaO Equivalent				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L
Feed (analyzed)	805.1		0.140		53.4	
Feed (computed)			0.206		52.6	
6 hour Preg		1678		0.07		24.3
24 hour Preg		1660		0.08		21.8
48 hour Preg		1661		0.09		21.4
48 hour Dry Residue	805.1		0.066		19.0	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

Appendix D

Bottle Roll Leaching Test BR 2 | **DI-5 from Ammonia** | **RDi Project:** Coronet Darwin | **Date:** 12-Jun-12

Purpose: To examine the extraction of gold and silver from an acid leached residue.

Sample: Approximately 915 grams of washed residue from Ammonia Leach Test 2 performed on DI-5 Sample

Procedure: The material was transferred to a bottle and was adjusted to about 40% solids. The pH was adjusted to >11 with hydrated lime. Sodium cyanide was then added to a calculated level of 10.0 g/l. At 6 and 24 hours, the pH and free cyanide were determined. In addition, solution samples were taken to determine the gold and silver contents. Sodium cyanide was added to return the level to 10.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
p80 150 mesh	48 hours	9.82 g/L	39% Solids

Summary of Results:

Parameter		Au	Ag
Extraction, % (1)		47.7	77.5
Assayed Head, g/mt			
Calculated Head, g/mt		0.09	56.1
Final Tail Assay, g/mt		0.05	13.9
Cyanide Consumption	11.03	kg NaCN/metric ton ore	
Lime Added	4.9	kg $Ca(OH)_2$/metric ton ore	

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
6	46.8	83.6	8.65
24	47.0	76.3	10.2
48	47.7	77.5	11.0

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	Residual Reagents NaCN g/L	D.O. ppm	pH Initial	Adjust
0	2323	1407	13.84	4.5					11.8
6	2325	1409	8.02			4.20	6.33	11.9	11.9
24	2305	1389	1.33			9.04	4.89	12.1	12.1
48	2302	1386				9.44	5.35	12.2	
Total			23.19	4.50					
				(3.4)	CaO Equivalent				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L
Feed (analyzed)	915.7		0.140		53.4	
Feed (computed)			0.090		56.1	
6 hour Preg		1409		0.03		33.3
24 hour Preg		1389		0.03		30.2
48 hour Preg		1386		0.03		29.7
48 hour Dry Residue	915.7		0.052		13.9	

(1) Based on calculated head assays.
(2) Slurry thick, added 500 ml extra water at the start

Appendix D

Bottle Roll Leaching Test BR 3

C-6 from Acid

RDi Project: Coronet Darwin

Date: 12-Jun-12

Purpose: To examine the extraction of gold and silver from an acid leached residue.

Sample: Approximately 711 grams of washed residue from Acid Leach Test 3 performed on C-6 Sample

Procedure: The material was transferred to a bottle and was adjusted to 35% (2) solids. The pH was adjusted to ~11 with hydrated lime. Sodium cyanide was then added to a calculated level of 10.0 g/L At 6 and 24 hours, the pH and free cyanide were determined. In addition, solution samples were taken to determine the gold and silver contents. Sodium cyanide was added to return the level to 10.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
p80 150 mesh	48 hours	7.84 g/L	35% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	95.2	90.0
Assayed Head, g/mt		
Calculated Head, g/mt	1.30	330.1
Final Tail Assay, g/mt	0.09	46.5

Extraction, % (1)			NaCN Consumed
Hr.	Au	Ag	kg/mt
6	90.5	88.7	1.70
24	95.4	90.1	2.96
48	95.2	90.0	4.01

Cyanide Consumption 4.01 kg NaCN/metric ton ore

Lime Added 62.3 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g			Residual Reagents		pH	
hrs	g	ml	NaCN	$Ca(OH)_2$	Carbon	NaCN g/L	D.O. ppm	Initial	Adjust
0	2007	1297	10.28	42.0					10.8
6	2021	1311	4.07	1.00		6.92	6.85	10.8	11.0
24	2002	1292	0.69	1.30		9.48	6.49	10.7	11.0
48	1975	1265				9.64	5.63	10.7	
Total			15.04	44.30					
				(33.5)	CaO Equivalent				

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L
Feed (analyzed)	710.8		0.540		516.0	
Feed (computed)			1.30		330.1	
6 hour Preg		1311		0.90		223.5
24 hour Preg		1292		0.95		226.0
48 hour Preg		1265		0.95		212.0
48 hour Dry Residue	710.8		0.086		46.5	

(1) Based on calculated head assays.

(2) Slurry thick, added about 300 ml extra water at the start

APPENDIX E

ICP ANALYTICAL DATA ON LEACH PRODUCTS

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted by: Resource Development, Inc.
11475 West I-70 Frontage Road
North Wheat Ridge, Colorado 80033
Attention: Deepak Malhotra, Ryan Allen and Tom Randall

Laboratory No.: 122381
Client Number: F174
Date Received: 7/16/2012
Date Completed: 7/30/2012

Method: Serial dilutions, ICP analysis.
Lab code: 2072

Element:	Al	As	Ba	Bi	Ca	Cd	Co	Cr	Cu	Fe	K	Mg	Mn	Mo	Na	Ni	Pb	Sr	Ti	V	W	Zn
Detection Limit:	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Reporting Unit:	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
COR LT-1 PREG	5318	2.4	0.2	1.8	664	62.1	4.5	5.2	1042	8805	111	818	1006	0.7	103	13.1	7.2	-0.2	8.5	4.0	11.3	43951
COR LT-2 PREG	8.6	0.1	<0.1	2.1	2.1	0.2	<0.1	<0.1	3.4	4.7	16.7	1.0	1.0	0.2	21.5	<0.1	0.9	0.3	<0.1	0.1	0.8	76.0
COR LT-3 PREG	1243	36.1	0.1	712	389	125	73.6	10.3	221	9861	46.5	618	1667	1.4	106	42.7	7.2	0.3	21.5	3.6	18.4	84240
COR LT-4 PREG	1.8	0.5	<0.1	2.1	2.5	4.0	0.2	<0.1	2.3	2.1	12.5	0.8	0.8	0.5	23.4	0.4	0.2	<0.1	0.1	0.4	2.4	1048

Richard A. Grondin

Richard A. Grondin, QC Manager

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted by: Resource Development, Inc.
11475 West I-70 Frontage Road
North Wheat Ridge, Colorado 80033
Attention: Deepak Malhotra, Ryan Allen and Tom Randall

Laboratory No.: 122382
Client Number: F174
Date Received: 7/16/2012
Date Completed: 7/31/2012

Method: 4 Acid digestion, ICP analysis.
Lab code: 7045

Element:	Al	As	Ba	Bi	Ca	Cd	Co	Cr	Cu	Fe	K	Mg	Mn	Mo	Na	Ni	Pb	Sr	Ti	V	W	Zn
Detection Limit:	0.01	10	1	10	0.01	1	1	1	2	0.01	0.01	0.01	1	1	0.01	5	10	5	0.01	1	10	2
Reporting Unit:	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	%	%	ppm	ppm	%	ppm	ppm	ppm	%	ppm	ppm	ppm
COR LT-1 RESIDUE	2.90	112	426	60	7.28	28	16	146	779	9.97	0.59	0.74	1296	15	0.03	92	33656	84	0.07	46	172	4050
COR LT-2 RESIDUE	3.87	93	489	28	8.35	143	20	112	2672	9.77	0.52	0.81	3610	11	0.02	72	29762	77	0.05	43	190	91552
COR LT-3 RESIDUE	3.89	106	342	1300	11.83	47	97	50	459	12.08	0.92	0.92	10225	10	0.03	72	8271	72	0.21	68	167	5547
COR LT-4 RESIDUE	3.25	89	288	1449	9.46	225	199	136	607	10.18	0.77	0.94	11307	11	0.03	145	7594	67	0.16	55	143	129318

Richard A. Grondin

Richard A. Grondin, QC Manager

Zinc Leach work

Kappes, Cassiday & Associates
7950 Security Circle Reno, Nevada 89506
Telephone: (775) 972-7575 FAX: (775) 972-4567

20 April 2017

Mr. Paul Skinner
Brownstone Mining, LLC
P.O. Box 215
Lone Pine, CA 93545
T: 602-525-8731
E: flityre@aol.com

Project No. 102 - Brownstone / Report I.D. KCA0170023 / Mainfile No. 9116

Subject: Brownstone Acid Leach Test Work

Dear Mr. Skinner,

On 06 April 2017, the laboratory facility of Kappes, Cassiday & Associates (KCA) in Reno, Nevada received a small box containing one sample of minus 20 mesh material from Paul Skinner. This sample was prepared and utilized for head analyses and acid agitated leach test work.

From the received sample, two (2) 50 gram portions were split out and ring and puck pulverized to a target size of 80% passing 200 mesh Tyler. The pulverized portions were utilized for head analyses for silver, tungsten and zinc. From the received sample, a 400 gram portion was split out and utilized for acid agitated leach test work.

Head analyses were performed on the sample material. Two (2) 50 gram portions were individually ring and puck pulverized to a target size of 80% passing 200 mesh Tyler. The pulverized portions were assayed for silver, tungsten and zinc via ICP analyses.

The results of the head analyses are presented in Table 1.

A portion of as-received material was utilized for an acid agitated leach test. The test was performed as follows:

1. One 400 gram split of as-received material was placed into a 5 liter baffled reactor and slurried with 933 milliliters of 200 gram per liter H_2SO_4.
2. The slurry was mixed thoroughly with and overhead stirrer and the pH of the slurry checked.

3. The slurry was sampled at 90 minutes and analyzed for pH, Ag, W and Zn.

4. After completion of the leach period, the slurry was filtered. A small amount of rinse water was added to the reactor in order to remove the remaining tailings material. The pregnant leach solution (PLS) + rinse was then weighed and assayed for Ag, W and Zn. The cake was then washed with water and the wash water was then weighed and assayed for Ag, W and Zn. The cake was then oven dried.

From the dry tailings material, two (2) portions were ring and puck pulverized to a target size of 80% passing 200 mesh Tyler. The pulverized portions were assayed for Ag, W and Zn.

The results of the acid agitated leach test work are presented in Table 2.

It should be noted that the W assays were below the detection limit.

Submitted by,

Brandon Williams

Table 1.
Brownstone Project
Head Analyses – Silver, Tungsten and Zinc

KCA Sample No.	Assay 1, oz Ag/st	Assay 2, oz Ag/st	Average Assay, oz Ag/st
77580	18.17	18.17	18.17

KCA Sample No.	Assay 1, mg W/kg	Assay 2, mg W/kg	Average Assay, mg W/kg
77580	<10	<10	<10

KCA Sample No.	Assay 1, % Zn	Assay 2, % Zn	Average Assay, % Zn
77580	18.66	18.40	18.53

Table 2.
Brownstone Project
KCA Sample No. 77580
KCA Test No. 77581
Acid Agitated Leach Test Work
Target Sulfuric Acid Level – 20%

KCA Sample No.	KCA Test No.	Leach Product	pH	Density, g/cm³	Feed Ore Wt., grams	Final Tails Wt., grams	Removed Solution, grams	Volume Removed, mLs	Solution ICP, mg Ag/L	Solution ICP, mg W/L	Solution ICP, mg Zn/L	Extracted, oz Ag/st	Extracted[2], mg W/kg	Extracted, % Zn
77580	77581 B	90 minute	0.9	1.12	400	--	22.8	20.4	22.8	<1	53200	0.034	<10	0.27
		PLS+ Rinse	1.4	1.12	400	--	1,260	1,125	12.2	<1	43900	1.001	<10	12.35
		Wash	4.7	1.02	400	271.33	1,220	1,196	7.4	<1	6190	0.645	<10	1.85
Total Extracted (with wash):												1.680	<10	14.47
Tail Assays[1]:												15.930	<10	2.50
Calculated Head Grade:												17.610	<10	16.97
Assayed Head Grade:												18.170	<10	18.53
Mass Balance Error, %:												3%	--	8%
Percent Extracted (90 minute sample + PLS + Rinse), %:												6%	--	74%

Note(1): Tail assays calculated based on direct assays and head and tail weights
Note(2): All W assays were below detection limits

Distribution and costs

Mining costs are estimated at $41.29 per ton. Milling costs are estimated at $73.58 per ton. Crushing and water are estimated at $8.50 per ton. Total costs are $189.50 per mined ton.

The oxide has the same total cost estimate, but additional acid costs are offset by the loss of freight costs and smelting. The oxide products are picked up at the mine site, thus the drying and bagging deliver the same $189.50 costs per ton.

Darwin Mine - Mining Cost Detail

Cost Per Ton of Mining

Description	Cost Per Ton	Includes Labor
Labor Underground	28.00	Labor
Powder & Steel	6.00	Explosives
Ventilation	3.00	Utilities
Development Drilling	3.29	Labor
Safety Labor	1.00	Labor
Crushing	3.25	Labor
Maintenance	1.25	Labor
Consumables	0.50	Mine Supplies
Transport Fine Ore	0.50	Labor
Crushing Electric & Water	3.00	Utilities
Milling	19.60	Labor
Milling Maintenance	3.40	Labor
Chemicals	30.00	Mine Supplies
Tailing Treatment	13.00	Labor
Bags	2.58	Mine Supplies
Mill Electric & Water	3.50	Utilities
Security/Lab	1.50	Labor
Trucking	8.00	Transportation
Container Freight	6.00	Container Freight
Smelter Fees	27.50	Smelter Fees
G&A/Other/Contingency	16.63	Labor
Other	8.00	Corporate Costs
Total	189.50	

Our Copper concentrates are being shopped with Rio Tinto in Salt Lake City and Freeport McMoran in Bagdad Arizona. The Tellurium level is expected to be high and may benefit Rio Tinto as they have this capability. Tellurium could reach between 15 and 30 tons per year.

The Tungsten concentrates are being shopped between Global Tungsten Powder and Kennametal. Kennametal will probably be the best fit as we are already a supplier, they have a scheelite circuit at their Huntsville Alabama APT plant and use scheelite in Fallon Nevada 260 miles from the mine.

Other minor metals will be sold domestically from the cathodes.

Minor metals (see attached Don Strachan report). Germanium was first plated in 1948 and is soluble in sulfuric or nitric acid. This critical mineral has an annual market of 180 tons and there was a nine-ton shortfall in 2024. (See Germanium plating patent). We have not included Germanium in the Proforma but expect 20.25 tons per year of production.

The Ophir Mine, The Independence, The Promontory and the Darwin Antimony mine are all previous Antimony producers. Favorable assays results indicate that further work is required to develop this capability. A seventy-two percent Antimony hit in the Defiance is a good indication that this would be a worthwhile effort. (Ref USGS Antimony mine map).

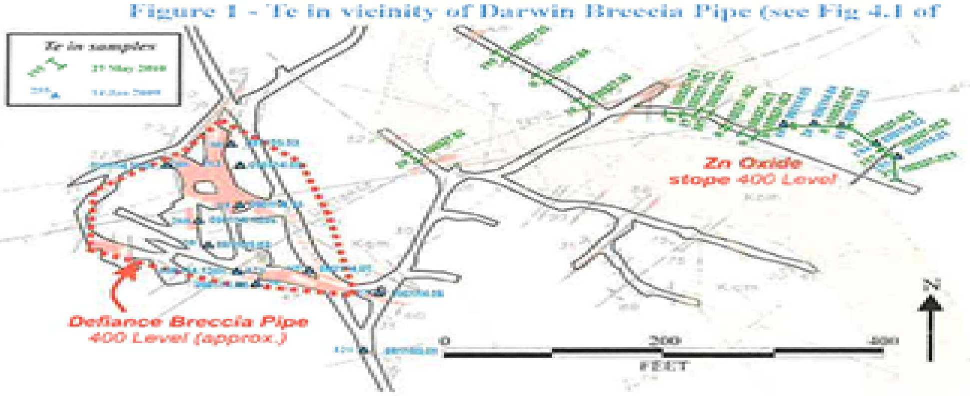
Indium levels have been all over the map, conservatively we should produce between 5 and 10 tons of Indium annually. Additionally, there is a potential for Gallium, Hafnium, Cobalt and Nickel, but at this point I am not ready to estimate production totals.

Don Strachan Report

Excerpt from June 30th monthly report by Don Strachan Darwin ZnAgPbWAuTe Inyo CA

Darwin ZnAgPbWAu Inyo CA

Figure 1 - Te in vicinity of Darwin Breccia Pipe (see Fig 4.1 of



On 14 January 2009, I took 11 grab samples were taken in the Defiance Breccia area and another 5 samples were taken in the "Zn Oxide" stope on the 400 Portal Level at Darwin (Figure 1). The Defiance Breccia samples returned an average of 89 ppm Te and a high of 208 ppm Te. The Zn Oxide samples averaged 103 ppm Te with a high of 255 ppm Te. These 16 samples were analyzed by Florin Labs of Reno and formed the basis of WIM's initial positive Te evaluation of Darwin. Unfortunately, Florin Labs soon after was found to have overstated Te in numerous samples from a New Brunswick Mo porphyry. WIM stopped using Florin Labs as a result, and considered the Darwin assays as flawed.

A year later, Jack Stone, Darwin's owner, submitted three 5-gallon bucket samples to Florin Labs for Te, Ge, and Ag analysis. Jack said he collected these samples from the Zn Oxide stope after a Chinese sampling delegation had shown repeated interest. In mid-May 2010, Florin reported Jack's three samples to average 329 mg/kg Te and range between 55 mg/kg Te and 757 mg/kg Te. These reported values were better than my previous samples of 14 January 2009. I immediately had Jack's three pulps re-assayed by Inspectorate-IPL and the rejects sent to WIM for petrographic analysis. Inspectorate-IPL reported an average of 275 ppm Te and values of 18 ppm Te, 98 ppm Te, and 708 ppm Te. WIM's petrographer identified Te in Bi sulfides (24 June conversation). The 2010 Florin and Inspectorate-IPL assays from the Zn Oxide stope correspond.

I returned to Darwin on May 27th to verify Jack's resampling with 10 grab samples in the Zn Oxide stope (100527-1C1to 7C1) and 4 grab samples (100527.02 to .05) in workings immediately west (Figure 1). I also reviewed Jack's intentions for a pilot metallurgical facility and a mill should his Darwin Project go forward. The Chalfant mill is 10 miles north of Bishop, maintained and guarded, and currently set up for making a gold concentrate from sulfide ore at 50 tpd, upgradeable to perhaps 100 tpd. An oxide ZnAgTe ore would require addition of an acid vat leach circuit. My report on the pilot and mill facilities is forthcoming.

The results of the May 27th assays were received from Inspectorate-IPL on June 26th. The 10 grab samples in the Zn Oxide stope (100527-1C1to 7C1) average 195 ppm Te and range between nil Te and 337 ppb Te. The 4 grab samples (100527.02 to .05) in workings immediately west of the Zn Oxide stope average 63 ppm Te and range between 20 ppm Te and 195 ppm Te.

The samples taken in 2010 at Darwin suggest the 2009 assays may not be as suspect as we once thought. The 2009 and 2010 assays in the Zn Oxide stope show excellent correspondence with spatial and geologic characteristics. The 194 and 255 ppm Te assays from 2009 correspond spatially with the 337 ppm Te assay from 2010. The correspondence appears due to a mineralized, NE-striking, down-to-SE near-vertical fault (gray NE line in Figure 1) similar to several associated with the Defiance Breccia Pipe.

All of the WIM samples taken to date at Darwin are high-graded grab samples or isolated vertical channel samples from the 400 Level to verify the presence of Te. Plotted locations are approximate and relative to workings mapped in the late 1970's. The Zn Oxide stope does not show up on these earlier maps. Samples from the Zn Oxide stope and adjacent area are all taken from strongly oxidized rock with occasional remnant sulfides.

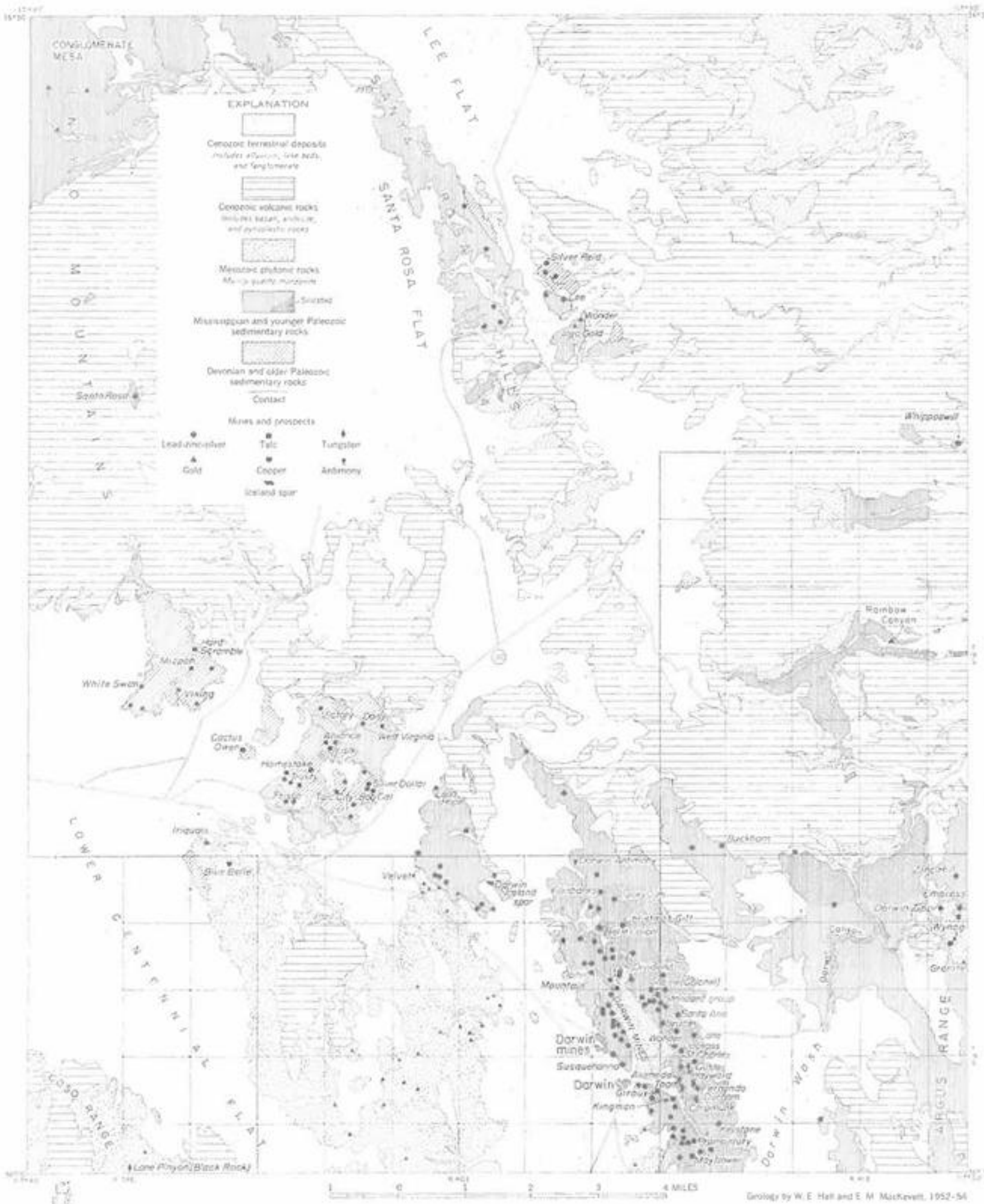
The Te assay results and observed geologic relationships in the Zn Oxide stope suggest Te-mineralized, steep NE faults are similar to others nearby that are mapped as co-genetic with the Defiance Breccia Pipe. The "spottiness" of the 400 Level Te values may also be due to Tedestructive oxidation, and perhaps remnant bismuth sulfides retain Te better under oxidizing conditions. The implication: The unmined, unoxidized portions of the Defiance Breccia Pipe at depth (below 1,000 Level?) may be a significant source of byproduct Te, as well as economic Ag-Zn-In-Au. These physical and geochemical relationships should be clarified once Pete Hahn concludes his data and map compilation started this past week for Jack Stone.

Pete Hahn says Rob Wetzel, a former Blue Range employee working for Lithic Resources, has recently found high indium numbers on the 400 Level. Pete thinks the deep Zn-In-Te skarn target at Darwin is a good shot, with the Zn/Pb increasing with depth. He expects better skarn ores cored by quartz porphyries are later and west of the main Darwin stock. These later quartz porphyries are also apparently responsible for the Te-bearing Defiance Breccia. Pete has also promised me to keep me fully informed as his own work progresses.

Concerning sampling logistics, Pete hopes to gain working access to the deeper levels of the Defiance Breccia Pipe during the field phase of his compilation work (later July?). A WIM sampling team should join him when he accesses these deeper levels.

USGS antimony mine map

PALEOZOIC ROCKS 7



Conclusion

The mine plan outlined above demonstrates a clear and achievable strategy for safely and efficiently extracting the identified orebody while maximizing resource recovery and maintaining operational flexibility. Given the geological continuity observed and the favorable mining conditions, the proposed methods are well suited to optimize production while controlling costs. Importantly, historical data from operations in this area show that for every ton mined, an additional ton was typically proven through subsequent drilling efforts. This trend suggests strong potential for ongoing resource growth and supports a long-term, sustainable mining operation. Continued exploration and close grade control will therefore be key components of the plan, ensuring that the full value of the deposit is realized over the life of the mine, which based on geological studies could reach 100 years or more.

Prepared by

Jack Stone

Chief Executive Officer

INYOAG LLC

Nicholas Stone

Chief Operations Officer

INYOAG LLC

4/28/2025

7 Mine Infrastructure

- Crusher: 20"x36" Telsmith jaw crusher Primary, GP100 cone Secondary.

- Lab Facility: Agilent ICP, AA, Microwave digester, Flotation Lab cells

-Hydrometallurgical Facility: 2 Feed hoppers, 2 Ball mills, large flotation cells, Leach tanks, Containment, Filter press, Drum filter, Thickener, EMEW cells, Tailings treatment plant, Water treatment, Batch plant for blending tailings.

- Workings: 150 miles of drifts and 38 miles of rail. Primary access via the 9,660-ft Radio

Ore Tunnel on the 400 level.

- Hoists: Thompson hoist to 900 level, Defiance hoist to 1300 level, and an incline winze

with ore skip to 1000 level.

- Ventilation: Natural airflows at 21,500 CFM, supplemented by a mechanical fan.

Underground temp is 58°F with 17% humidity. No groundwater encountered down to

4,000 feet.

- Air Supply: Two 125 HP 750 CFM compressors active, with expansion plans for two

additional units.

- Power: LADWP 33,000kV substation onsite, with supplemental solar and cogeneration options planned.

- Water: Two wells producing 160 GPM, supported by a 250,000-gallon onsite tank farm.

Initial Development Strategy

Mining will begin in accessible areas above the 400 level to avoid immediate hoisting requirements.

Expansion and Exploration

- Porphyry Target: Anaconda drilling intersected a 188-ft zone at 2,200 ft depth, warranting deep drift access.

- Drill Campaign: A $1.3M program is planned to upgrade inferred resources to measured and indicated. Drill stations from the Defiance and Radio Ore tunnel will intersect priority. targets.

- Lucky Jim Zone: A historically productive oxide/gold-silver zone with ~170,000 tons past production. Tunnel extension could daylight below this zone, enabling secondary egress and shaft development.

Life-of-Mine Outlook

- Current defined resources support an 8-year mine life at 500 TPD. Historically for every ton mined another ton was moved into the resource, this remains the plan for future production. Geological studies project a 100 year mine life, this will be supported by drilling throughout the life of the mine supporting the expanded production 200tpd.

- Peter Hahn's resource model targets 15 million metric tons at 15% Zn, supporting projections of 100 years of scalable production.

- Long-term success is supported by ongoing drilling, underground development, and geologic continuity in multiple directions.

Production plans

Inyoag will mine at a rate of 500 tons per day for the first 5 years while the radio tunnel is developed and opens a new access. All projections in this study are based on 500TPD production. Plans to expand in year 6 could result in 4X multiplier of all production if the mineralization remains. As well as plans to work with AVI metals on a domestic roaster in the United States.

Year	Total TPD	Sulfide TP	Oxide TPD
1–5	500	300	200
5+	2000	1500	500

8 Environmental Considerations and Framework

The project is structured in accordance with:
- California Environmental Quality Act (CEQA)
- National Environmental Policy Act (NEPA)
-California Environmental Reporting System (CERS) - California State Lands - Safe Drinking Water Act (SDWA)
-Department of toxic substances Control(DTSC) - Resource Conservation and Recovery Act (RCRA)
- Cal OSHA mine and Tunnel division
- Federal Mine Safety and Health Act (MSHA compliance)

INYOAG maintains ongoing coordination with the California Department of Fish and Wildlife (CDFW), State Water Resources Control Board, Inyo County Planning Department, Bureau of Land Management (BLM), and Lahontan Waste Water to ensure comprehensive environmental oversight and transparency.

Site-Specific Protections

- Zero discharge: All process water is recycled and reused within a closed-loop circuit. No contaminated water is released into the environment. -All acid processes happen within a containment facility
- Dust and emissions control: Water trucks till roads are paved, Sonic dust control, and particulate monitoring reduce dust and airborne emissions with a water conservation approach.
- Noise abatement: All underground activities operate within established decibel limits.
- Vegetation and wildlife preservation: Construction is confined to historically disturbed zones, protecting nearby habitats and maintaining migration corridors.

Tailings and Waste Management

- Tailings are thickened, filtered, and used as cemented backfill underground, eliminating the need for a surface tailings impoundment.
- All chemical and reagent use is tracked and contained in accordance with Hazardous Materials Business Plan (HMBP) regulations.
- A water treatment facility neutralizes all leachates along with limestone addition to the tails before distribution back underground.

Renewable Energy and Carbon Reduction

- Onsite solar arrays and future cogeneration units are designed to reduce grid dependence and cut emissions.
- Fleet electrification and the use of battery-powered locomotives underground eliminate diesel exhaust exposure for workers and reduce overall GHG output.

Reclamation and Closure Plan

A detailed Reclamation Plan has been submitted to Inyo County and includes:
- Backfilling of disturbed areas with non-toxic materials

- Long-term monitoring of water and soil quality post-closure
- Financial assurance instruments (bonding) to guarantee environmental performance.

Environmental, Social, and Governance (ESG) & Opportunity Zone Development

The Darwin Mine Project stands as a model of responsible resource development, blending environmental sustainability, strategic mineral production, and local economic revitalization. Located within a federally designated Opportunity Zone (OZ), Darwin is uniquely positioned to leverage federal tax incentives while delivering measurable impact through job creation, infrastructure improvement, and supply chain resilience. Backed by over $50 million in investment to date, the project represents a powerful convergence of national priorities and regional opportunity.

Since being recognized as America's Top Project in 2021, Darwin has attracted more than $15 million in private equity funding and an additional $35 million in direct capital investment from the project operators. These funds have supported installation of a modern crushing and screening system, construction of a fully equipped analytical laboratory, development of underground access, and advancement of the mill facility to 65% completion. Key permitting, grid power, water rights, and infrastructure are already in place, positioning Darwin for near-term production with scalable growth potential.

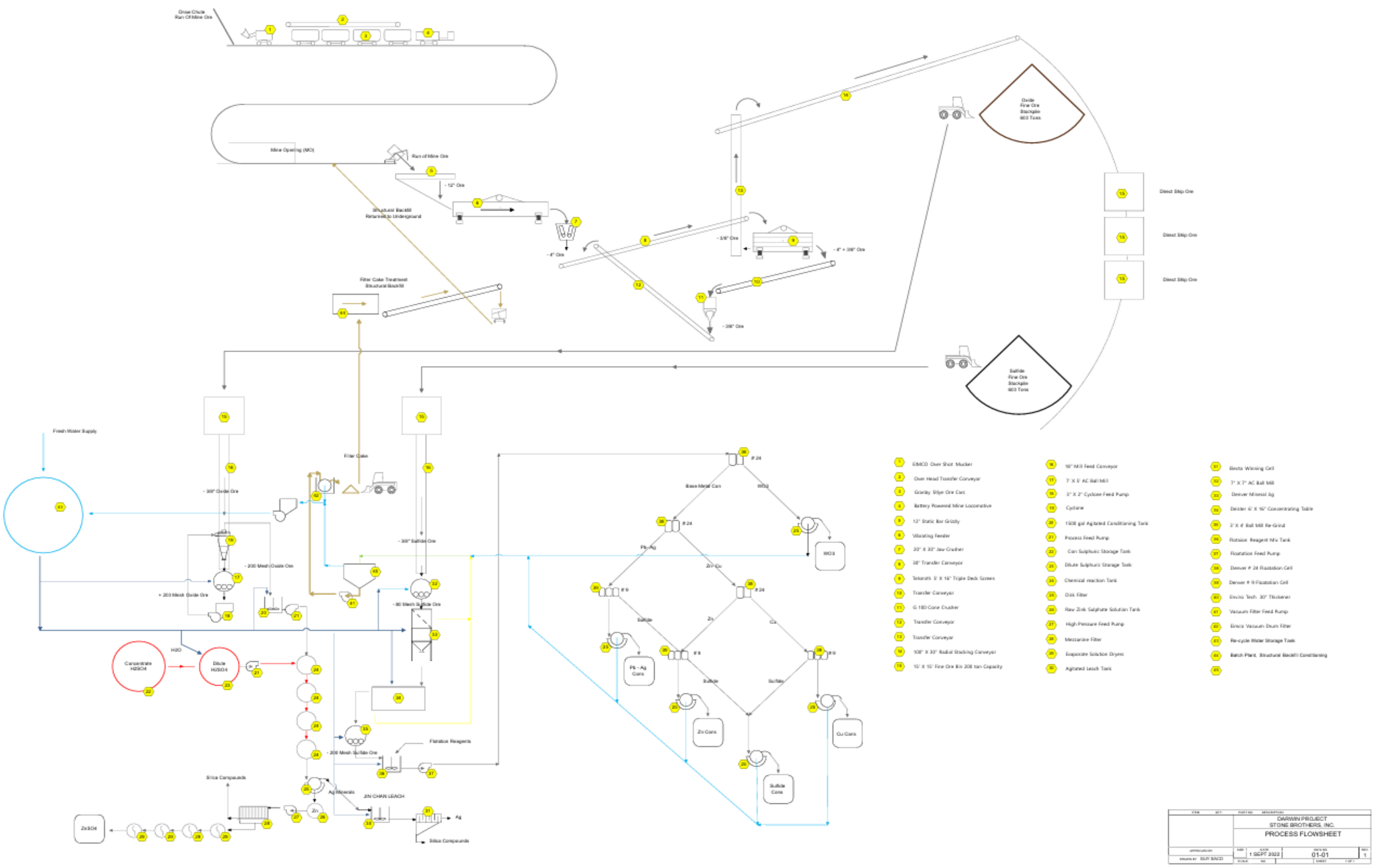
Darwin's Opportunity Zone status offers significant capital gains tax advantages for qualified investors under the Tax Cuts and Jobs Act of 2017. These benefits, including deferral and potential exclusion of capital gains, enhance returns for long-term investors while channeling capital into one of California's most economically underserved regions. The project is eligible for participation from ESG-focused and impact-oriented investment funds seeking both returns and measurable social outcomes.

The local and regional impact of Darwin is substantial. The project will create 350 direct jobs in Phase 1, increasing to 1,500 full-time roles during the expansion in Phase 2. These positions include skilled trades, operators, engineers, and support staff. More importantly, each direct job is expected to generate an additional 3.8 indirect or induced jobs in the surrounding economy—supporting employment across housing, food services, transportation, retail, education, and healthcare. At full scale, Darwin could support up to 5,700 jobs throughout Inyo County and the broader region.

From an ESG standpoint, the project adheres to strict environmental standards. All tailings are returned underground as structural backfill, eliminating the need for surface tailings storage. The site runs on grid-connected electric power, minimizing emissions, and reuses historic mine infrastructure to avoid expanding the environmental footprint. Socially, the project emphasizes safety, job quality, and inclusion, with a focus on local hiring and training in partnership with technical schools and tribal communities. On the governance front, INYOAG LLC maintains rigorous internal controls, third-party audits, and structured reporting aligned with the Defense Production Act (DPA) and U.S. critical mineral strategy.

The Darwin Mine is not just a mining operation—it is a high-impact economic engine, a responsible steward of environmental resources, and a cornerstone in the revitalization of a historic mining district.

9. Processing Overview



Summary of process

Ore extracted from underground.

Crushed to 3/8-

Sulfide Circuit: Feed → 7x7 mill → jig/table → flotation (Pb-Ag, Zn, Cu, W) →dried concentrates shipped to roaster.

→Tail

Oxide Circuit: 7x5 mill → sulfuric leach→ZnSO4·H20 (Liquid)→ EW/Resin columns→ ZnSO4 + critical solution (Ge, Te, In, Ga, Pb, Sb, Hf, Cu, Ag)

Tailings cemented and backfilled underground.

Process Overview – Extraction & Final Product by Mineral

Zinc (Zn)

Source: Both oxide and sulfide ores.
Process: Milled and subjected to sulfuric acid leaching. Leachate passes through filtration and electrowinning to recover high-purity zinc sulfate monohydrate powder.
- Sulfide Zone: Froth flotation produces a zinc sulfide concentrate (50–55% Zn) for smelting.
Final Products: Zinc sulfate powder, liquid zinc sulfate monohydrate, zinc concentrate.

Germanium (Ge)

Source: Associated with zinc oxides and hydrozincite minerals.
Process: Dissolved during zinc oxide leaching and recovered from pregnant leach solution via resin-in-pulp or selective precipitation. - Sulfide Zone: Froth flotation produces a Zn or Cu concentrate with Ge byproduct
Final Product: Germanium oxide or metal (>99.999% purity).

Silver (Ag)

Source: Both oxide and sulfide. Found with galena (PbS), sphalerite (ZnS), and native forms.
Process: Plated in EMEW oxide circuit
- Sulfide Zone: Froth flotation produces a Concentrates with lead and zinc during flotation, refined at a smelter.
Final Product: Silver dore or refined silver from concentrates.

Lead (Pb)

Source: Galena-rich sulfide ore zones minor in oxides.
Process: Plated in EMEW oxide circuit
- Sulfide Zone: Selective flotation into a lead concentrate (~60–65% Pb, high Ag), sent to smelter.
Final Product: Lead bullion with co-recovered silver.

Copper (Cu)

Source: Sulfide zones (e.g., chalcopyrite).
Process: Plated in EMEW oxide circuit
- Sulfide Zone: Floated after lead/zinc stages using depression and pH controls.
Final Product: Copper concentrate (20–25% Cu) for smelting.

Tungsten (W)

Source: Associated with scheelite ($CaWO_4$) and wolframite.
Process: Depressed sulfides allow selective flotation; gravity separation may assist.
Final Product: APT-grade tungsten concentrate (60–65% WO_3).

Tellurium (Te)

Source: Present in oxidized ores and minor sulfides.
Process: Leached with zinc oxide circuit and recovered using solvent extraction or cementation.

- Sulfide Zone: Froth flotation produces a or Cu concentrate with Te byproduct
Final Product: Tellurium metal or compound.

Indium (In)
Source: Trace inclusion in sphalerite.
Process: Recovered from flotation tailings or leach solutions via ion exchange.
Final Product: Indium ingots or salts.

Gallium (Ga)
Source: Associated with aluminum silicates and zinc ores.
Process: Dissolved in acidic leach and recovered by solvent extraction. - Sulfide Zone: Froth flotation produces a or Zn concentrate with Ga byproduct
Final Product: Gallium metal (4N–6N purity).

Cobalt (Co)
Source: Found in minor sulfide inclusions.
Process: Plated in EMEW oxide circut
- Sulfide Zone: Recovered during flotation or from Cu concentrate Co byproduct.
Final Product: Cobalt hydroxide or intermediate.

Antimony (Sb)
Source: Linked with Antimonial Lead, tetrahedrite or stibnite.
Process: Extracted via leaching and selective precipitation.
- Sulfide Zone: Selective flotation into a lead with Sb byproduct concentrate (~60–65% Pb, high Ag).
Final Product: Antimony trioxide or metal.

Hafnium (Hf)
Source: Trace inclusion in zircon or rare minerals.
Process: Not yet modeled; future pilot testing required.
Final Product: Hafnium oxide (HfO_2).

⚠ Hafnium and Antimony:
Preliminary exploration indicates potential for both minerals:
- Hafnium has shown 80 ppm presence but is not yet modeled. Additional drilling is required.
- Antimony has been found in historical antimonial lead and high ppm samples 71%. Further geologic validation is ongoing.

10. Woods Sulfide Floatation

Darwin Mines

Sequential Flotation Evaluation of Sphalerite-Dominant Composite

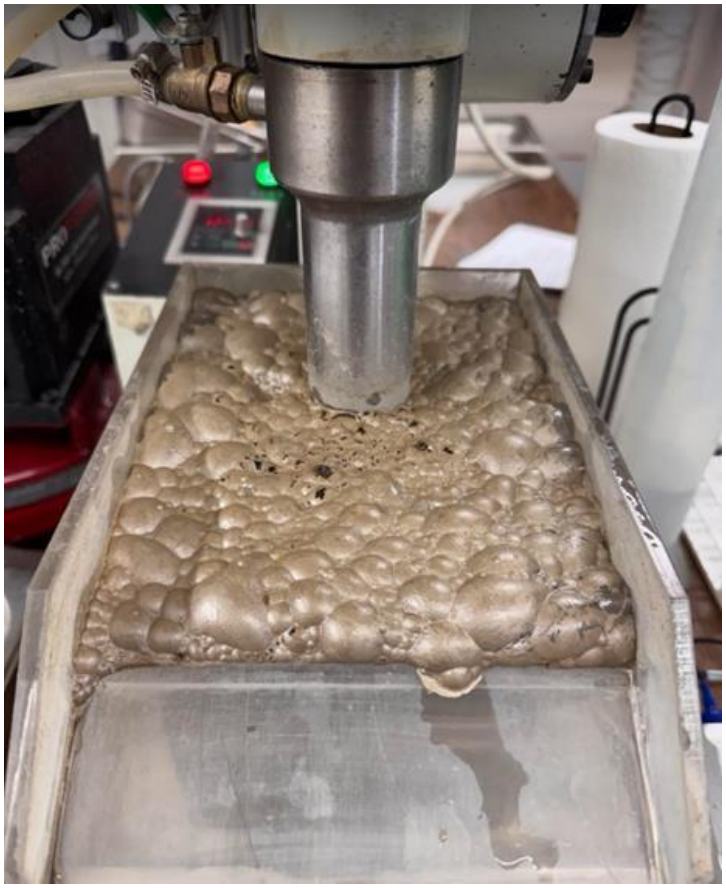


Figure 1.1 Flotation Test 1, Zinc Flotation

Prepared by:
Jessie Link, M.Sc., QP (MMSA)
Lab Manager/Consulting Metallurgist

Contents

Tables

Figures

List of Acronyms, Abbreviations, and Unit of Measure

Acronyms and Abbreviations

Ag...........................silver
Au...........................gold
Ca ....calcium
Calc'd………………Calculated
CS (LECO) .... Carbon and Sulfur Analyzer
Cu ....copper
Cum. ....Cumulative
GRG .... gravity-recoverable gold
ICP .... inductively coupled plasma
MIBC .... Methyl Isobutyl Carbinol
P80.........................80% passing
PAX .... Potassium Amyl Xanthate
Pb...........................lead
S.............................sulfur

S= ....sulfur sulfide
Sep.........................Separate
TS ....total sulfur
Wt...........................weight
XRF .... X-ray Diffraction
XRD ....X-ray Fluorescence
Zn .... zinc

Units of Measure

" .... inch
% .... percent
ft .... foot (feet)
g .... gram
g/L .... grams per litre
g/t .... grams per tonne
g/t Au...................... grams of gold per tonne
g/t Ag...................... grams of silver per tonne
gpm/ft^2 .... gallons per minute per square foot
h .... hour
kg .... kilogram
kg/t .... kilogram per tonne
lb/st .... pounds per short ton
L/min .... litres per minute
mL .... millilitre
t .... tonne
µm .... micrometre (micron)
mm .... millimeter
ppm .... parts per million, which equals g/t

Executive Summary

A composite sample from the Darwin project was delivered to Woods Process Services for a sequential flotation program targeting separate recovery of lead, copper, and zinc concentrates, along with precious metals. The program was conducted without the use of sodium cyanide and aimed to improve stage selectivity, enhance precious metal recovery, and minimize contamination across flotation stages. The composite material was crushed to approximately 10 mesh, split into 1000-gram charges, and tested in three flotation campaigns, each with adjustments to grind size, reagent schemes, and flotation sequence.

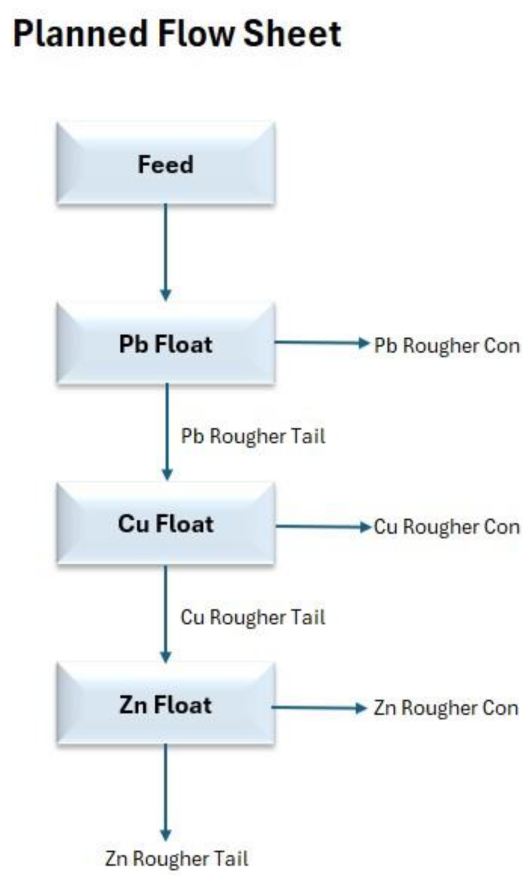


Figure 1.1 Planned Flow Sheet

The head sample assayed at 3.54% Pb, 0.21% Cu, 21.52% Zn, 0.7 g/t Au, and 599 g/t Ag. Sulfur speciation confirmed that the composite was largely unoxidized, containing 10.14% sulfide sulfur. Additional elemental analysis showed notable concentrations of calcium (12.92%), aluminum (2.08%), iron (8.89%), magnesium (1.01%), and manganese (0.43%), with minor contributions from trace elements including cobalt (135.48 ppm), chromium (113.41 ppm), nickel (212.96 ppm), and barium (99.40 ppm). Potassium, sodium, and lithium were also present at 668.05 ppm, 96.18 ppm, and 2.32 ppm, respectively, while strontium was detected at 32.25 ppm. These values suggest the presence of carbonate and silicate gangue minerals alongside the primary sulfides.

Table 1.1 ICP Results for Head Sample

Element	Unit	Value
Al	%	2.08
Ba	ppm	99.40
Ca	%	12.92
Co	ppm	135.48
Cr	ppm	113.41
Cu	%	0.21
Fe	%	8.89
K	ppm	668.05
Li	ppm	2.32
Mg	%	1.01
Mn	%	0.43
Na	ppm	96.18
Ni	ppm	212.96
Pb	%	3.54
Sr	ppm	32.25
Zn	%	21.52

Across the three flotation tests, progressive improvements were observed. Flotation Test 1 (F-1), run at a P80 of 106 µm, achieved 73.8% lead recovery, 65.4% copper recovery, and 91.2% zinc recovery. However, selectivity was limited: 42.5% of the copper and 20.1% of the zinc reported to the lead concentrate. Gold and silver recoveries were 76.3% and 82.7%, respectively, with more than half of each reporting to the lead stage. Iron distribution in F-1 was also spread across stages, with 14.7% recovered in the lead concentrate, 4.0% in the copper concentrate, and 9.6% in the zinc concentrate, totaling 28.4% recovery into flotation products.

Flotation Test 2 (F-2), conducted at a finer grind (P80 75 µm) and with increased depressant dosages, showed improved separation. Lead recovery increased to 85.8%, with reduced crossover of copper and zinc into the lead concentrate. Copper recovery rose to 80.4%, though 70.0% still floated in the lead stage. Zinc recovery increased to 94.5%, with 22.1% reporting to the zinc concentrate. Gold and silver recoveries improved to 87.6% and 89.8%, respectively. Iron distribution followed similar trends, with 19.1% recovered in the lead concentrate, 4.3% in the copper concentrate, and 4.1% in the zinc concentrate, for a total of 27.6% recovered into flotation products.

Flotation Test 3 (F-3) maintained the finer grind but implemented a revised reagent scheme using Aerofloat 3418A for improved lead selectivity. Lead recovery was 91.7%, with copper and zinc recovery at 87.9% and 93.3%, respectively. Zinc deportment improved significantly, with 64.4% reporting to the zinc concentrate, compared to just 22.1% in F-2. Copper separation remained a challenge, with 46.5% of the copper still reporting to the lead concentrate. Gold and silver recoveries were 75.0% and 87.9%, with most still reporting to the lead stage. Iron recovery in F-3 was slightly lower than previous tests, with 4.9% recovered in the lead concentrate, 7.7% in the copper concentrate, and 8.5% in the zinc concentrate, totaling 21.1% distributed among flotation products.

In the following tables and graphs, the separated cumulative percentage (Sep. Cum. %) for each element reflects the total recovery combined from all pulls into a single lead, copper, or zinc concentrate.

Table 1.2 Summary Table Lead Distribution

Pb Distribution						
	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	56.3%	56.3%	85.8%	85.8%	64.0%	64.0%
Cu Con	11.5%	67.8%	3.8%	89.6%	24.9%	88.9%
Zn Con	6.0%	73.8%	1.2%	90.7%	2.8%	91.7%

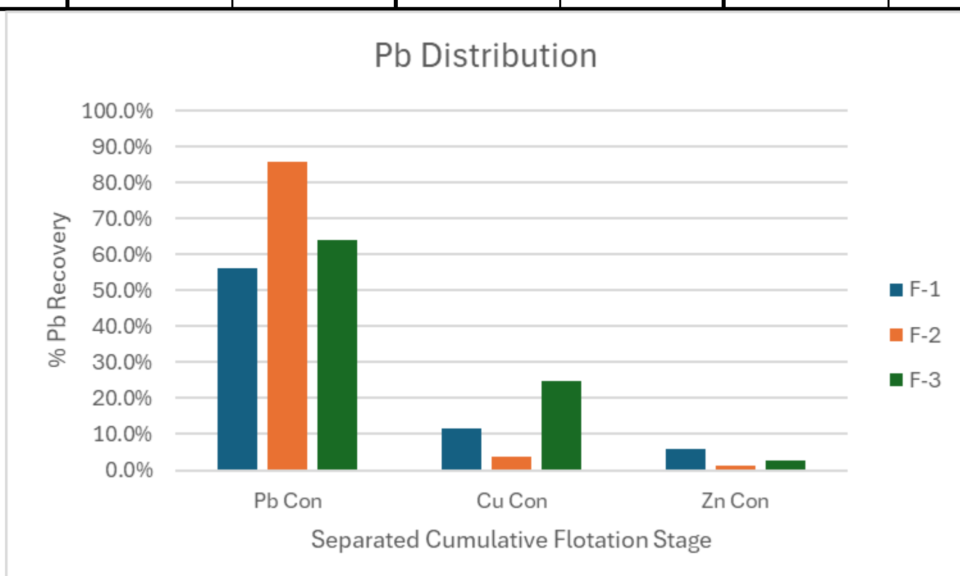


Figure 1.2 Summary Lead Distribution

Table 1.3 Summary Table Copper Distribution

Cu Distribution

	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	42.5%	42.5%	70.0%	70.0%	46.5%	46.5%
Cu Con	9.6%	52.1%	5.3%	75.3%	18.2%	64.7%
Zn Con	13.3%	65.4%	5.2%	80.4%	14.4%	79.1%

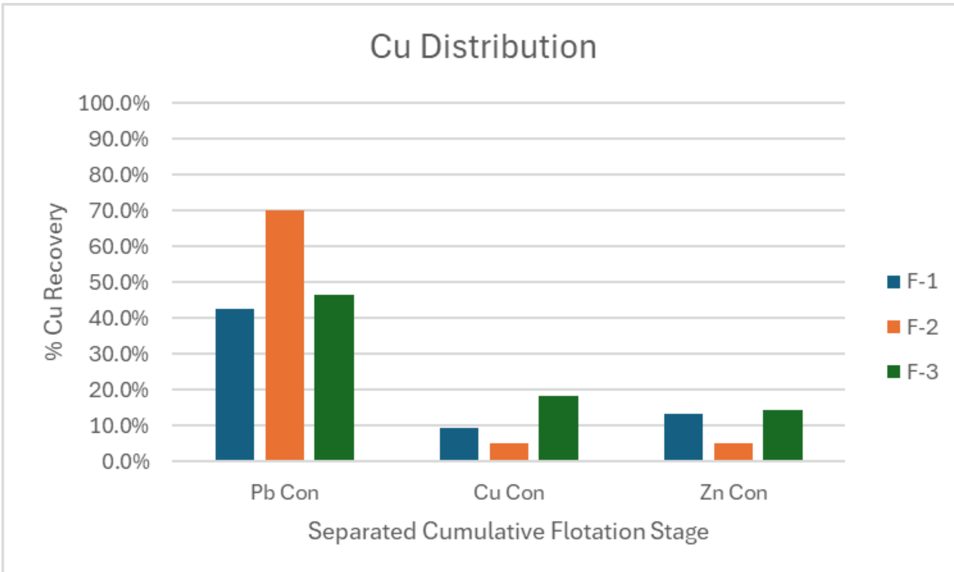


Figure 1.3 Summary Copper Distribution

1.4 Summary Table Zinc Distribution

Zn Distribution						
	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	20.1%	20.1%	65.8%	65.8%	15.1%	15.1%
Cu Con	15.4%	35.5%	6.6%	72.3%	13.7%	28.9%
Zn Con	55.7%	91.2%	22.1%	94.5%	64.4%	93.3%

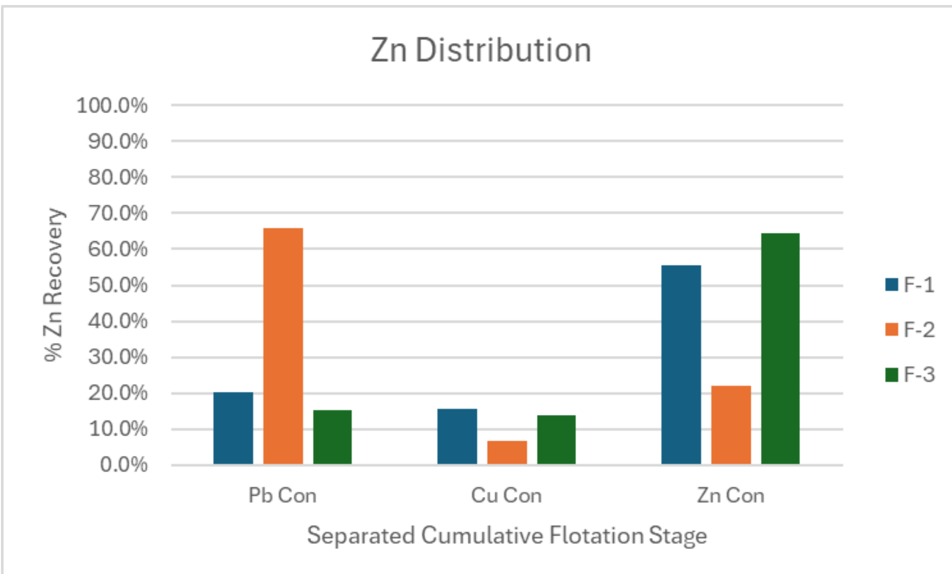


Figure 1.4 Summary Zinc Distribution

1.5 Summary Table Iron Distribution

Fe Distribution						
	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	14.7%	14.7%	19.1%	19.1%	4.9%	4.9%
Cu Con	4.0%	18.7%	4.3%	23.4%	7.7%	12.6%
Zn Con	9.6%	28.4%	4.1%	27.6%	8.5%	21.1%

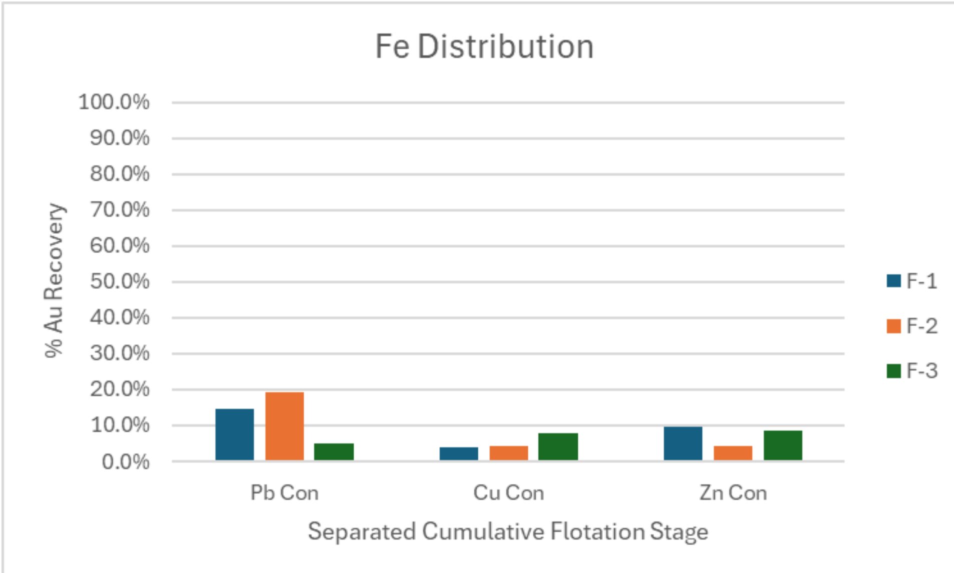


Figure 1.5 Summary Iron Distribution

1.6 Summary Table Gold Distribution

Au Distribution						
	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	55.0%	55.0%	78.7%	78.7%	44.8%	44.8%
Cu Con	11.2%	66.1%	6.9%	85.6%	20.4%	65.2%
Zn Con	10.2%	76.3%	1.9%	87.6%	9.7%	75.0%

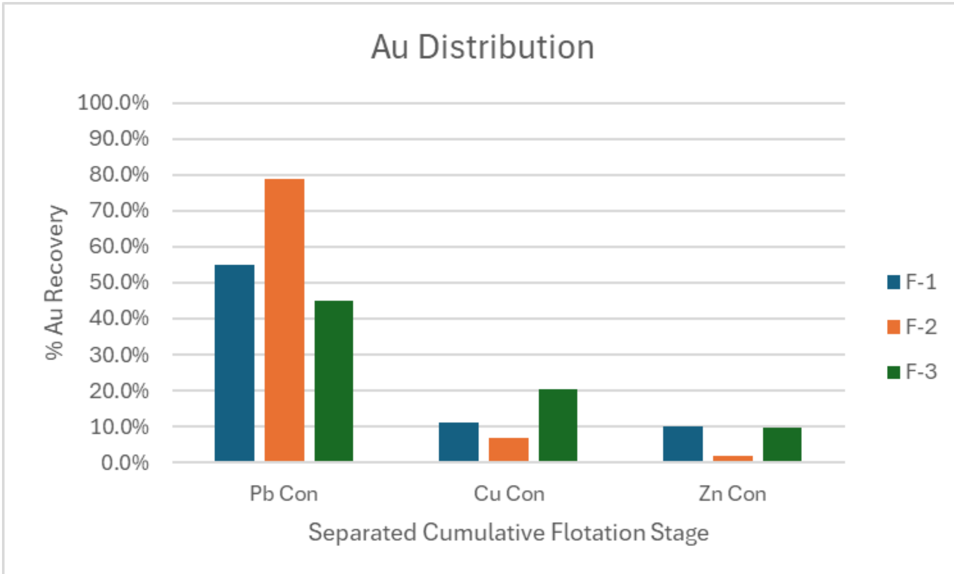


Figure 1.6 Summary Gold Distribution

1.7 Summary Table Silver Distribution

Ag Distribution						
	F-1		F-2		F-3	
Product	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %	Sep. Cum. %	Cum. %
Pb Con	64.4%	64.4%	82.6%	82.6%	59.1%	59.1%
Cu Con	10.3%	74.6%	4.9%	87.6%	21.7%	80.8%
Zn Con	8.0%	82.7%	2.2%	89.8%	7.2%	87.9%

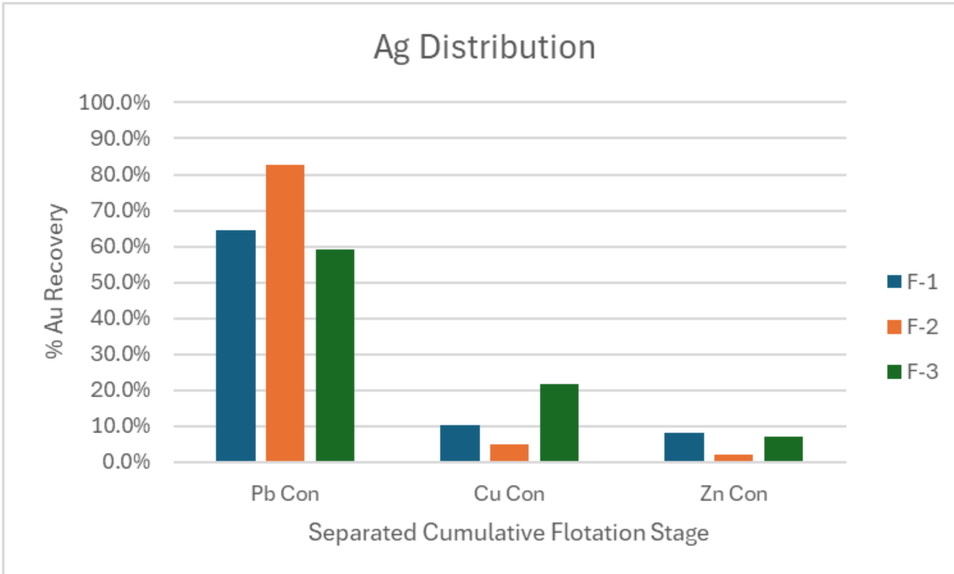


Figure 1.7 Summary Silver Distribution

These results demonstrate that finer grinding and reagent modifications enhanced lead and zinc recovery, though separating copper from lead remains a significant challenge. The findings from this test program will guide future flowsheet design and reagent strategy development for the Darwin composite. Supporting analytical work, including fire assay, ICP multi-element digestion, LECO CS speciation, XRF, and XRD, was used to characterize the material and monitor metallurgical performance. LECO sulfur analyses were performed on the head and tail samples only; sulfur values for the individual flotation concentrates were not directly measured but were instead back-calculated using mass balance and composite recovery data.

Composite Preparation and Head Analyses

2.1 Composite Prep

The bulk material delivered to Woods Process Services was processed to produce a consistent and representative composite for flotation testing. Initial size reduction was performed using a jaw crusher, followed by secondary crushing with a cone crusher to achieve a product size of approximately 10 mesh.

The crushed material was thoroughly homogenized and split using a riffle splitter to produce multiple 1000-gram charges. These charges were designated for flotation testing, head grade analysis, and mineralogical characterization. Care was taken during preparation to ensure sample integrity and minimize potential bias or segregation during splitting.

2.2 Head Analysis and Mineralogy

Representative portions of the prepared composite were selected for head analysis to establish baseline metal concentrations. Gold and silver were analyzed by fire assay, yielding head grades of 0.021 oz/ton Au (approximately 0.7 g/mt) and 17.5 oz/ton Ag (approximately 599.4 g/mt).

Base metal concentrations were determined using ICP following microwave digestion. The head grades were 3.54% Pb, 0.21% Cu, and 21.5% Zn. These values were consistent with results obtained via X-ray fluorescence (XRF) performed by The Mineral Lab, which confirmed high levels of zinc and significant lead and copper content.

Mineralogical characterization using X-ray diffraction (XRD) identified sphalerite as the dominant sulfide phase (~26 wt%), along with pyrite (~5 wt%), and trace chalcopyrite and galena. Major gangue minerals included garnet (~41 wt%), clinopyroxene (~11 wt%), and quartz (~6 wt%). The presence of garnet and intergrown sulfides suggests potential challenges in selective flotation, especially for copper. The full mineralogy report can be found in Appendix A.

Table 2.1 The Mineral Lab XRF Results Table 1

	Unit	Value
Na_2O	Wt %	0.27
MgO	Wt %	1.97
Al_2O_3	Wt %	3.33
SiO_2	Wt %	24.3
P_2O_5	Wt %	0.2
S	Wt %	12
Cl	Wt %	<0.02

K_2O	Wt %	0.12
CaO	Wt %	21.9
TiO_2	Wt %	0.24
MnO	Wt %	0.59
Fe_2O_3	Wt %	11.9
BaO	Wt %	0.01

Table 2.2 The Mineral Lab XRF Results Table 2

	Unit	Value
V	ppm	40
Cr	ppm	309
Co	ppm	86
Ni	ppm	149
W	ppm	<10
Cu	ppm	2270
Zn	ppm	156000
As	ppm	<20
Sn	ppm	<20
Pb	ppm	27000
Mo	ppm	<10
Sr	ppm	24
U	ppm	<10
Th	ppm	<10
Np	ppm	<10
Zr	ppm	49
Rb	ppm	<10
Y	ppm	<10

Table 2.3 The Mineral Lab XRD Results

Mineral Name	Chemical Formula	Approx. Wt %
Sphalerite	ZnS	26
Pyrite	FeS_2	5
Galena	PbS	<3
Chalcopyrite	$CuFeS_2$	<2?
Anglesite	$PbSO_4$	<3

Gypsum	$CaSO_4 \bullet 2H_2O$	<3?
Garnet	$(Ca,Fe,Mg,Mn)_3(Al,Fe)_2(SiO_4)_{3x}(OH)_{2x}$	41
Clinopyroxene	$Ca(Mg,Fe,Mn,Al,Ti)(Si,Al)_2O_6$	11
Quartz	SiO_2	6
"Unidentified"	?	<3

The LECO analysis of the head sample indicated a total sulfur content of 10.60%. Of this, 10.14% was identified as sulfide sulfur. The results confirm that sulfur in the composite is mainly present as sulfides, with minimal oxidation. Table 2.4 shows the results of the elements analyzed on the head sample.

Table 2.4 Head Sample Analysis

Element	Unit	Value
Pb	%	3.54
Cu	%	0.21
Zn	%	21.52
Fe	%	8.89
Au	ppm	0.73
Ag	ppm	599.36
S	%	10.60
S=	%	10.14

These analytical results guided reagent selection and flowsheet adjustments throughout the flotation test program.

3.1 Flotation Test 1

In Flotation Test 1, the material was ground to a P80 of 106 microns prior to flotation. A sequential reagent scheme was then applied to recover lead, copper, and zinc in separate stages. For the lead flotation stage, approximately 40 g/mt of Aerofloat 7310 was used as the primary collector, along with 100 g/mt of zinc sulfate to depress sphalerite. The pH was maintained at 9.5 using milk of lime. The copper flotation stage followed with the addition of 40 g/mt of Aero 3894 and 20 g/mt of Aerofloat 238 as collectors, with the pH adjusted to 9. In the zinc flotation stage, 750 g/mt of copper sulfate was used as an activator for sphalerite, and 50 g/mt of PAX (potassium amyl xanthate) was added as the collector. The pH during the zinc stage was elevated to between 11 and 11.5. MIBC was used as the frother across all stages.

The lead flotation stage recovered 73.8% of the total lead in the sample, with 56.3% captured in the first lead concentrate (Pb Con 1) and an additional 17.5% in the second lead concentrate (Pb Con 2). These two products together accounted for 12.6% of the total mass pulled, assaying at 11.15% and 11.98% Pb, respectively. Despite efforts at stage-wise separation, 11.5% of the lead reported to the copper flotation stage, and 6.0% to the zinc stage, indicating carryover and incomplete selectivity.

Copper recovery totaled 65.4%, but only 9.6% was recovered in the designated copper flotation stage, with Cu Con 1 and Cu Con 2 assaying 0.49% and 0.37% Cu, respectively. The majority of copper, 42.5%, reported to the lead concentrates, and an additional 13.3% was recovered in the zinc concentrates, again reflecting inadequate separation.

Zinc recovery was the highest of all target metals at 91.2%, with 55.7% recovered in the zinc concentrates. Zn Con 1, which represented 17% of the mass, assayed at 62.8% Zn, while Zn Con 2 contributed another 3.1% mass at 19.8% Zn. However, 20.1% of the zinc reported to the lead flotation stage and 15.4% to the copper stage.

Iron deportment followed a similar pattern. While the composite contained 9.6% Fe, the majority (72%) reported to the tailings, assaying at 11.3% Fe. The remaining 28.4% of iron was distributed across the concentrates, primarily in Pb Con 1 (11.2%) with an Fe grade of 13.32%. The iron content in the zinc concentrates was relatively low, with 9.6% of the iron distributed there at Fe grades between 1.81% and 2.41%, further supporting effective rejection of silicate gangue in those stages.

Table 3.1 Flotation Test 1 Rougher and Scavenger Concentrates – Lead, Copper, Zinc, Iron Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	Assay % Cu	Assay % Zn	Assay % Fe
Pb Con 1	8.1	8.1	8.1	11.15	1.15	24.9	13.3
Pb Con 2	4.5	12.6	12.6	11.98	0.58	45.7	7.5

Cu Con 1 Cu Con 2	3.9 2.1	3.9 6.0	16.5 18.6	5.54 3.76	0.49 0.37	53.5 49.6	6.9 5.8
Zn Con 1 Zn Con 2	17.0 3.1	17.0 20.1	35.5 38.7	0.51 2.11	0.17 0.26	62.8 19.8	3.8 9.1
Tail	61.3		100.0	1.09	0.16	2.9	11.3
Composite	100.0			2.56	0.28	20.2	9.6

Table 3.2 Flotation Test 1 Rougher and Scavenger Concentrates- Lead Distribution

	Pb Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Pb
Pb Con 1	35.3%	35.3%	35.3%	11.15
Pb Con 2	21.0%	56.3%	56.3%	11.45
Cu Con 1	8.5%	8.5%	64.7%	1.31
Cu Con 2	3.1%	11.5%	67.8%	1.59
Zn Con 1	3.4%	3.4%	71.2%	0.24
Zn Con 2	2.6%	6.0%	73.8%	0.40
Tail	26.2%		100%	2.56
Composite	100%			

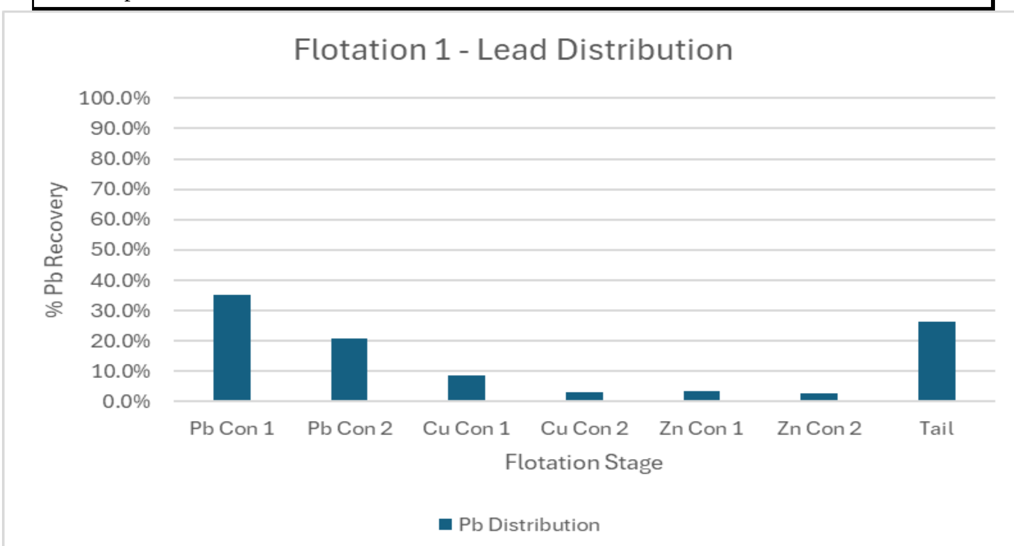


Figure 3.1 Flotation Test 1 Rougher and Scavenger Concentrates - Lead Distribution Graph

Table 3.3 Flotation Test 1 Rougher and Scavenger Concentrates – Copper Distribution

	Cu Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu
Pb Con 1	33%	33.3%	33.3%	1.15

Pb Con 2	9%	42.5%	42.5%	0.94
Cu Con 1	7%	6.8%	49.3%	0.12
Cu Con 2	3%	9.6%	52.1%	0.14
Zn Con 1	10%	10.3%	62.4%	0.08
Zn Con 2	3%	13.3%	65.4%	0.10
Tail	35%		100%	0.28
Composite	100%			

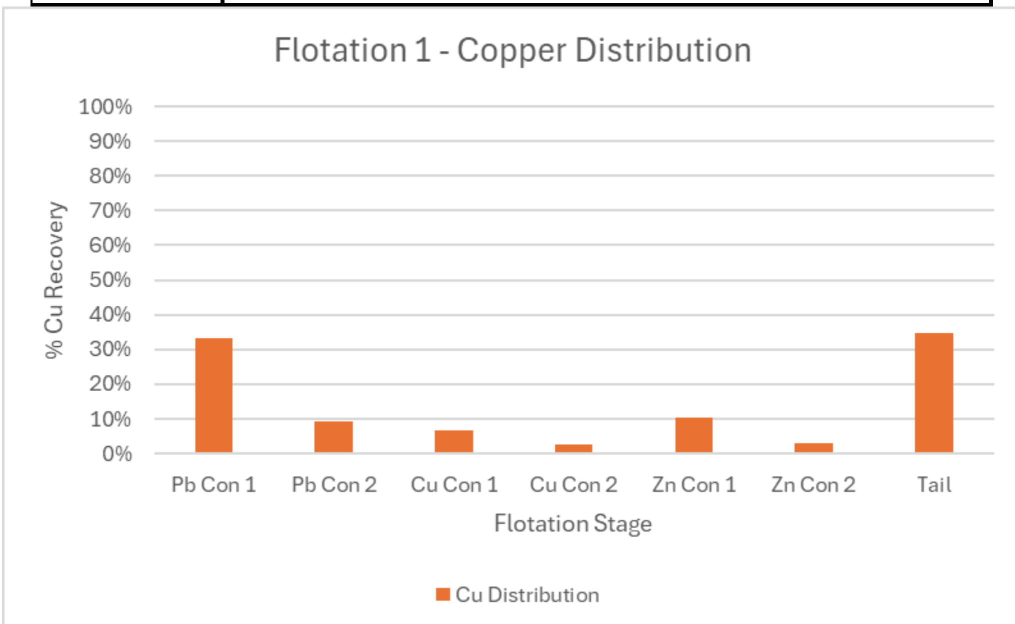


Figure 3.2 Flotation Test 1 Rougher and Scavenger Concentrates - Copper Distribution Graph

3.4 Flotation Test 1 Rougher and Scavenger Concentrates Zinc Distribution

Product	Zn Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Zn
Pb Con 1	10%	10.0%	10.0%	24.90
Pb Con 2	10%	20.1%	20.1%	32.30
Cu Con 1	10%	10.3%	30.4%	12.67
Cu Con 2	5%	15.4%	35.5%	16.83
Zn Con 1	53%	52.6%	88.1%	29.96
Zn Con 2	3%	55.7%	91.2%	29.13
Tail	9%		100%	20.24
Composite	100%			

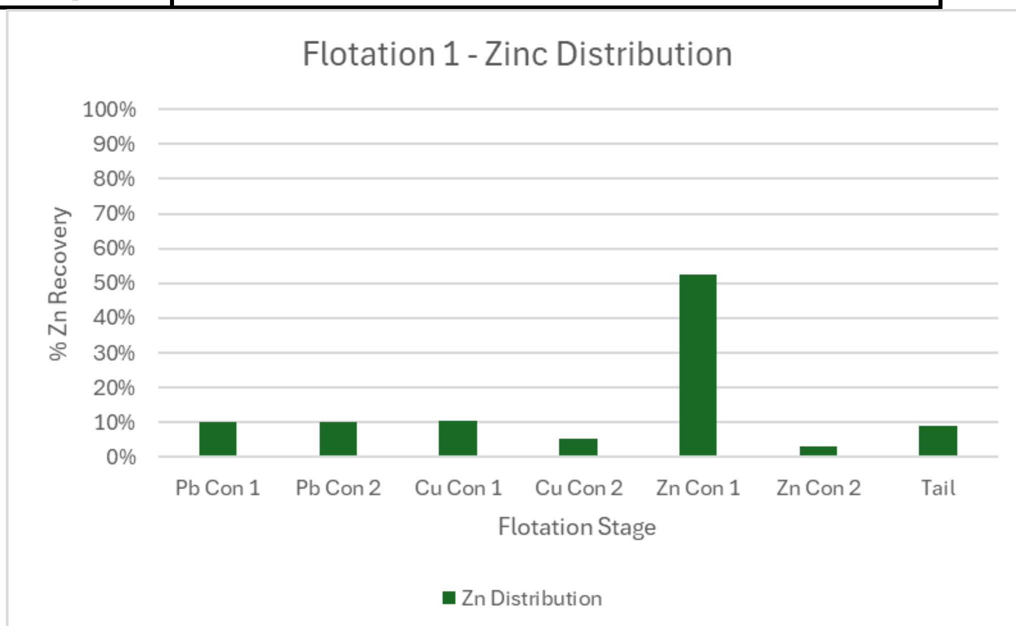


Figure 3.3 Flotation Test 1 Rougher and Scavenger Concentrates - Zinc Distribution Graph

3.5 Flotation Test 1 Rougher and Scavenger Concentrates Iron Distribution

Product	Fe Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Fe
Pb Con 1	11%	11.2%	11.2%	13.32
Pb Con 2	3%	14.7%	14.7%	11.25
Cu Con 1	3%	2.8%	17.5%	1.63
Cu Con 2	1%	4.0%	18.7%	2.10
Zn Con 1	7%	6.7%	25.4%	1.81

Zn Con 2	3%	9.6%	28.4%	2.41
Tail	72%		100%	9.64
Composite	100%			

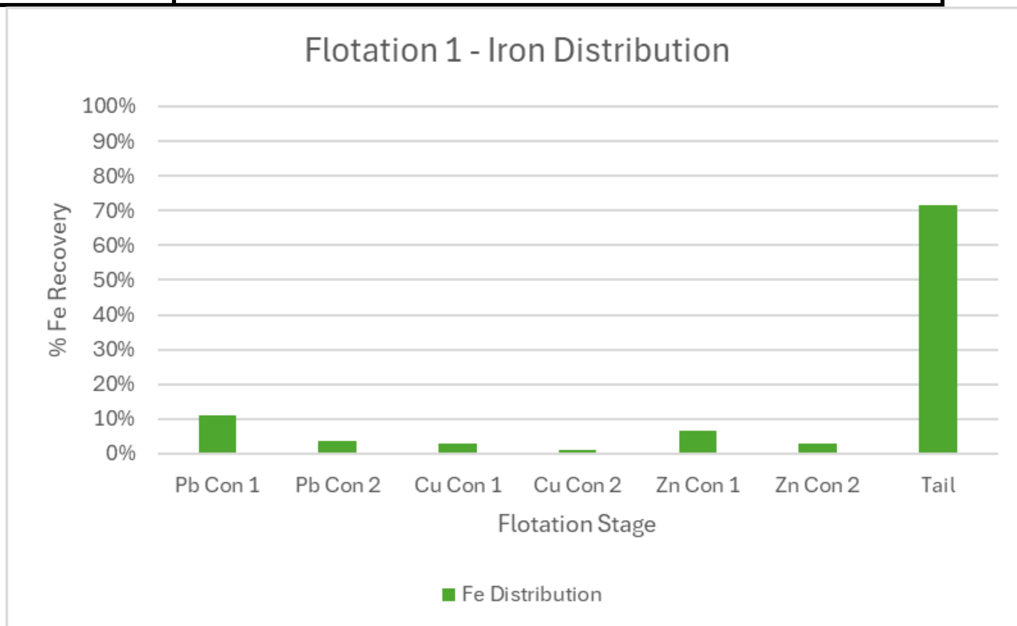


Figure 3.4 Flotation Test 1 Rougher and Scavenger Concentrates – Iron Distribution Graph

Precious metals followed similar patterns, with 76.3% of the gold and 82.7% of the silver recovered across all flotation stages. A large portion of both gold and silver reported to the lead flotation stage, with 55.0% of the gold and 64.4% of the silver captured in the lead rougher and scavenger pulls.

Table 3.6 Flotation Test 1 Rougher and Scavenger Concentrates – Gold and Silver Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	ppm Au	ppm Ag
Pb Con 1	8.1	8.1	8.1	3.7	3662.18
Pb Con 2	4.5	12.6	12.6	1.7	2322.83
Cu Con 1	3.9	3.9	16.5	1.3	1190.68
Cu Con 2	2.1	6.0	18.6	1.2	834.81
Zn Con 1	17.0	17.0	35.5	0.3	189.60
Zn Con 2	3.1	20.1	38.7	0.8	566.41
Tail	61.3		100.0	0.3	175.74
Composite	100.0			0.7	622.15

3.7 Flotation Test 1 Rougher and Scavenger Concentrates Gold Distribution

Product	Au Distribution			
	%	Sep. Cum. %	Cum. %	Cum. g/mt Au

Pb Con 1	44%	43.7%	43.7%	3.73
Pb Con 2	11%	55.0%	55.0%	3.02
Cu Con 1	8%	7.5%	62.5%	2.62
Cu Con 2	4%	11.2%	66.1%	2.46
Zn Con 1	7%	6.5%	72.7%	1.41
Zn Con 2	4%	10.2%	76.3%	1.36
Tail	24%		100.0%	0.69
Composite	100%			

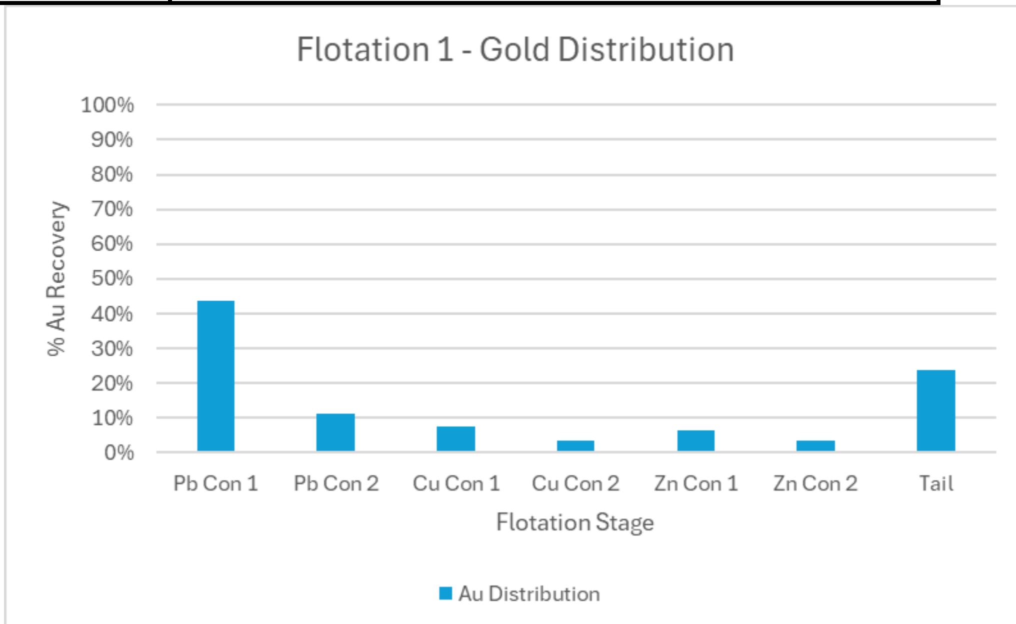


Figure 3.5 Flotation Test 1 Rougher and Scavenger Concentrates - Gold Distribution Graph

Table

3.8 Flotation Test 1 Rougher and Scavenger Concentrates - Silver Distribution

Product	Ag Distribution %	Sep. Cum. %	Cum. %	Cum. g/mt Ag
Pb Con 1	47.7%	47.7%	47.7%	3662.18
Pb Con 2	16.7%	64.4%	64.4%	3185.29
Cu Con 1	7.5%	7.5%	71.8%	2712.58
Cu Con 2	2.8%	10.3%	74.6%	2500.73
Zn Con 1	5.2%	5.2%	79.8%	1397.83
Zn Con 2	2.9%	8.0%	82.7%	1330.25
Tail	17.3%		100.0%	622.15
Composite	100%			

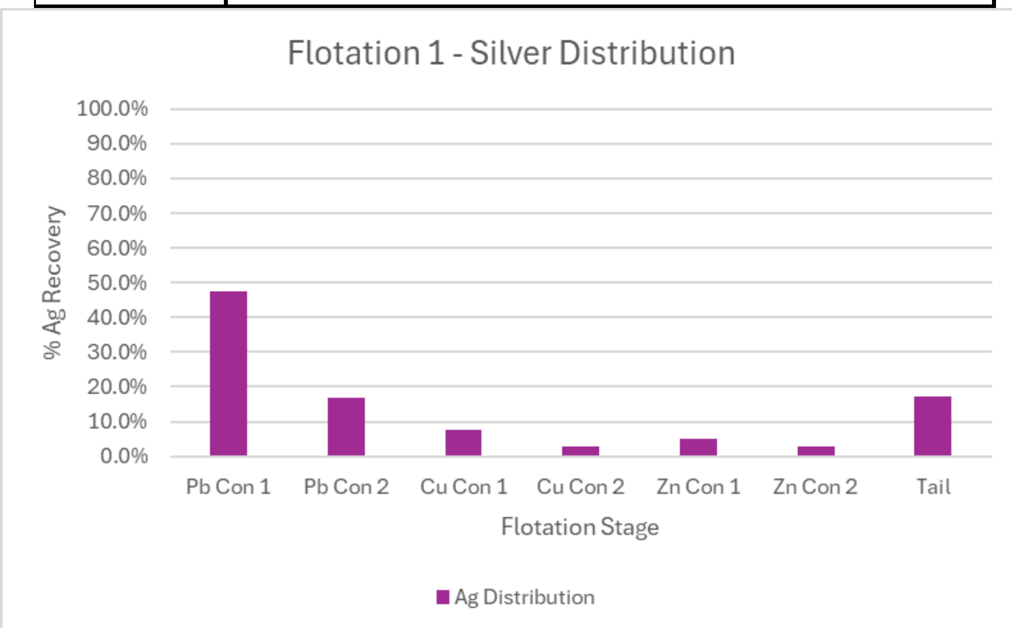


Figure 3.6 Flotation Test 1 Rougher and Scavenger Concentrates - Silver Distribution Graph

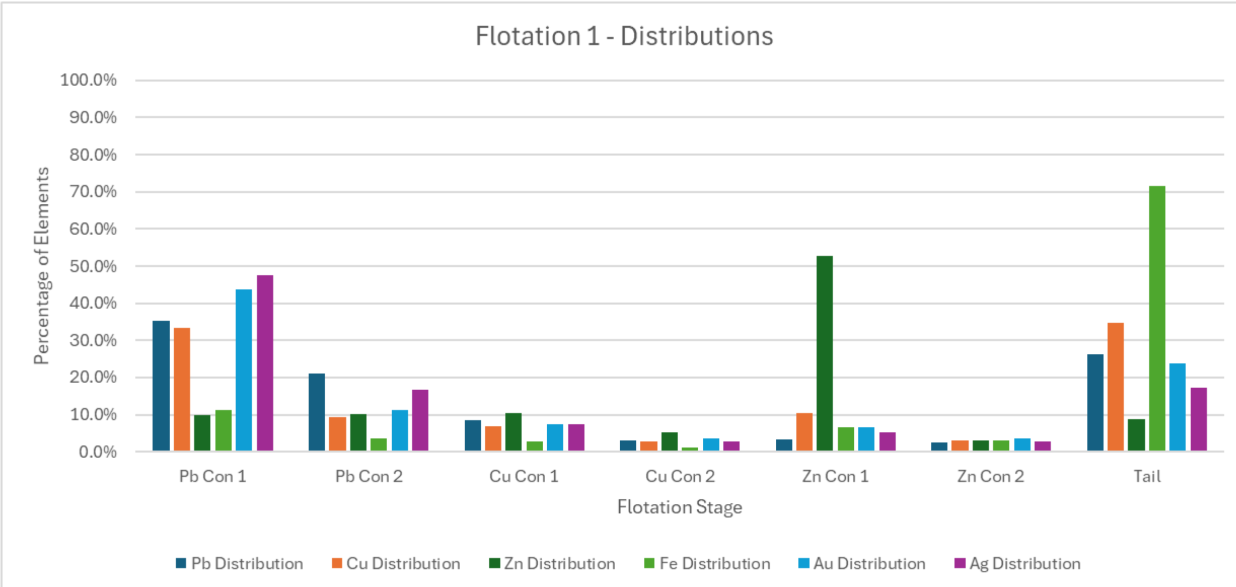


Figure 3.7 Flotation Test 1 Distributions

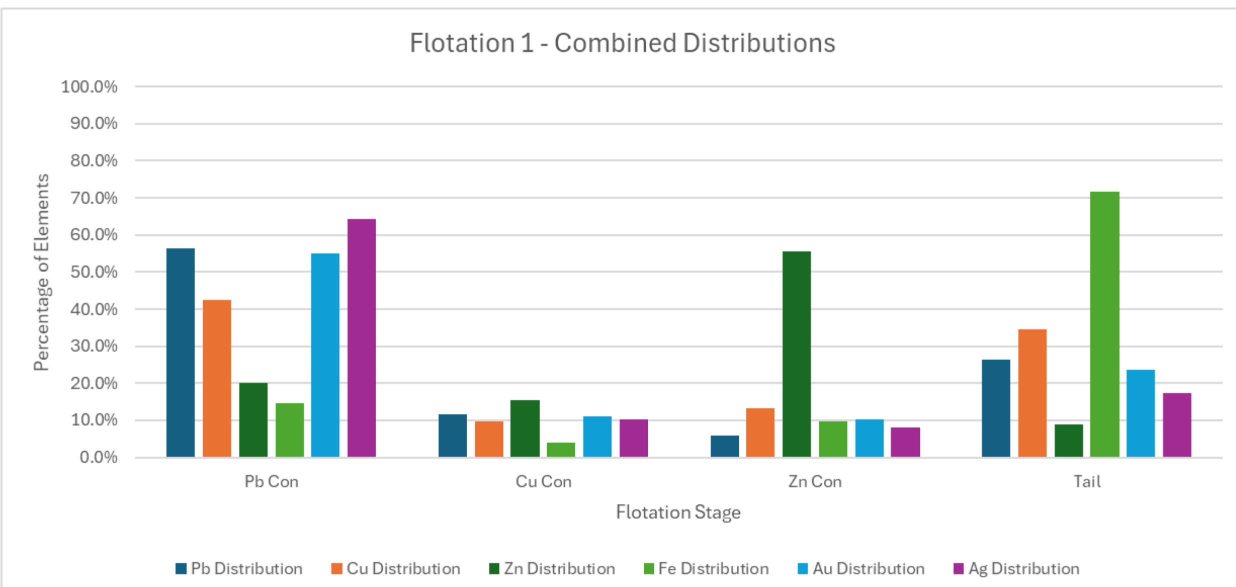


Figure 3.8 Flotation Test 1 Combined Distributions

A total of 94.5% of the sulfur in the composite reported to the combined concentrates, with only 5.5% remaining in the tailings, confirming efficient recovery of sulfur-bearing minerals through flotation. Meanwhile, 98.6% of the sulfide sulfur reported to the concentrates, with just 1.4% reporting to the tail, validating strong flotation performance for sulfide minerals such as galena, sphalerite, and chalcopyrite.

The near equivalence of cumulative total sulfur and sulfide sulfur values in the concentrates (both around 25.9%) further confirms that the recovered sulfur was almost entirely in sulfide form, with minimal oxidation. These results suggest effective reagent performance, minimal weathering of the feed material, and a flotation system well-tuned for targeting sulfide minerals.

Table 3.9 Flotation Test 1 Rougher and Scavenger Concentrates – Sulfur Speciation Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	% S	% S=
Pb Con 1	8.1	8.1	8.1		
Pb Con 2	4.5	12.6	12.6	25.91	25.86
Cu Con 1	3.9	3.9	16.5		
Cu Con 2	2.1	6.0	18.6		
Zn Con 1	17.0	17.0	35.5		
Zn Con 2	3.1	20.1	38.7		
Tail	61.3		100.0	0.95	0.23
Composite	100.0			10.60	10.14

Table 3.10 Flotation Test 1 Rougher and Scavenger Concentrates – Total Sulfur Distribution

	S Distribution		
Product	%	Cum. %	Cum. % TS
Pb Con 1			25.91
Pb Con 2			25.91
Cu Con 1	94.5%	94.5%	25.91
Cu Con 2			25.91
Zn Con 1			25.91
Zn Con 2			25.91
Tail	5.5%	100.0%	10.60
Composite	100%		

Table 3.11 Flotation Test 1 Rougher and Scavenger Concentrates – Sulfide Sulfur Distribution

	S= Distribution		
Product	%	Cum. %	Cum. % S=
Pb Con 1			25.86
Pb Con 2			25.86
Cu Con 1	98.6%	98.6%	25.86
Cu Con 2			25.86
Zn Con 1			25.86
Zn Con 2			25.86
Tail	1.4%	100.0%	10.14
Composite	100%		

The flotation results showed that Aerofloat 7310 lacked sufficient selectivity, allowing excessive copper and zinc to report to the lead concentrate. Copper in particular did not respond well to flotation in its designated stage, suggesting that it may be either mineralogically locked or insufficiently activated. Zinc recovery was high overall but was widely distributed across all stages. Based on these findings, the next round of testing would include finer grinding to improve liberation, increased zinc sulfate to depress sphalerite in the lead stage, and the introduction of sodium sulfite to depress chalcopyrite. Additional lead concentrate pulls would be added to recover more lead early and reduce contamination of downstream copper and zinc concentrates.

Figure 3.9 Flotation Test 1 – Lead Stage Froth Photographs

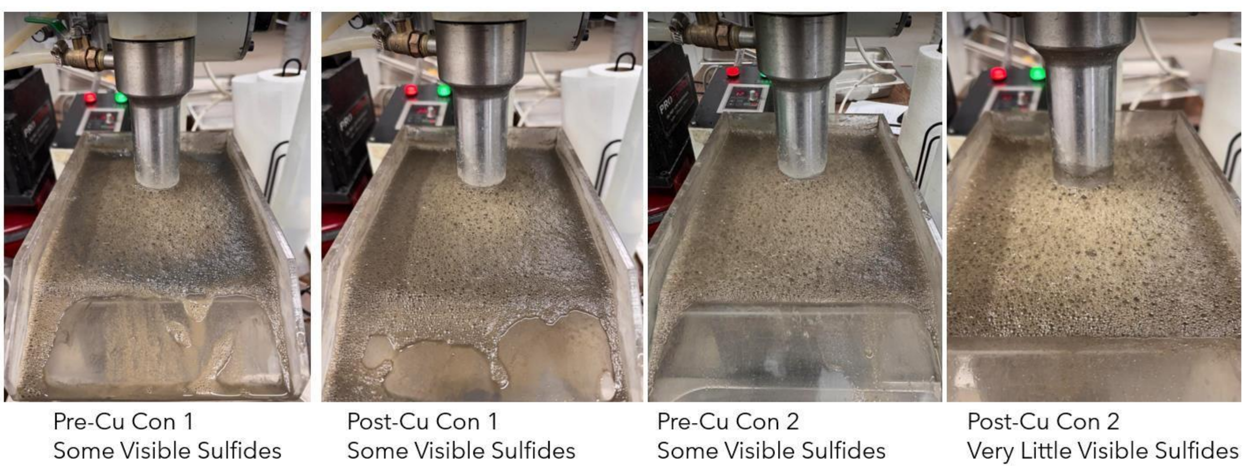
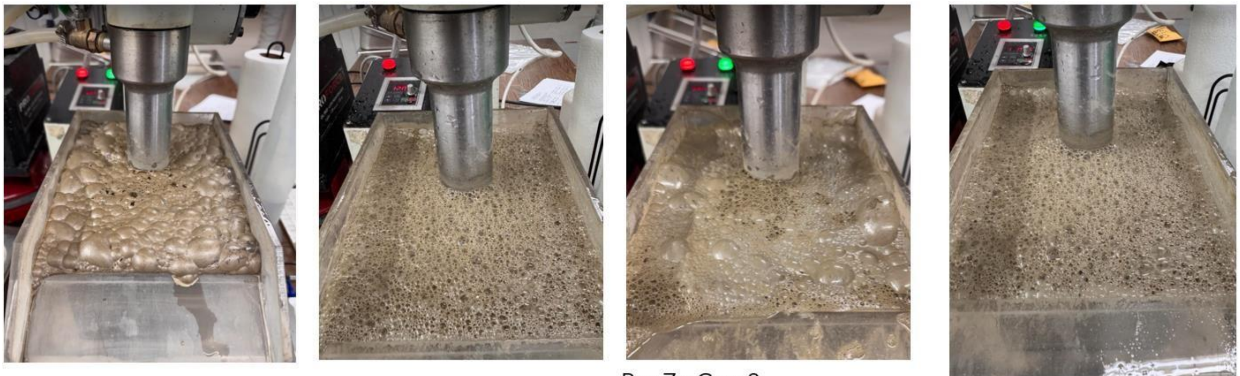


Figure 3.10 Flotation Test 1 – Copper Stage Frother Photographs



Pre-Zn Con 1
Great PAX froth

Post-Zn Con 1

Pre-Zn Con 2
Not enough Zn for PAX to float

Post-Zn Con 2

Figure 3.11 Flotation Test 1 – Zinc Stage Froth Photographs

3.2 Flotation Test 2

In Flotation Test 2, the composite was ground for 50 minutes to achieve a P80 of 75 microns, targeting improved liberation of sulfide minerals. The lead flotation stage was expanded to include four separate 5minute rougher pulls. Reagents for this stage included approximately 80 g/mt of Aerofloat 7310 as the collector, 2000 g/mt of zinc sulfate to depress sphalerite, and 100 g/mt of sodium sulfite to depress chalcopyrite. The pH was maintained at 9.5 using milk of lime. For the copper flotation stage, a single 5minute rougher and a 3-minute scavenger were conducted using 40 g/mt of Aero 3894 and 20 g/mt of Aerofloat 238, with the pH held at 9. Zinc sulfate was also included in the copper stage to continue sphalerite suppression. In the zinc flotation stage, which also included a rougher and scavenger, 750 g/mt of lead sulfate was used as an activator, and 50 g/mt of PAX (potassium amyl xanthate) was added as the collector. The pH was increased to 11–11.5. MIBC was used throughout the process as the frother.

Lead recovery improved substantially in Flotation Test 2, with 85.8% of the total lead recovered across four lead concentrate pulls. The first lead concentrate (Pb Con 1) accounted for 43.4% of the total lead at a grade of 19.85% Pb, followed by 29.9%, 6.9%, and 5.6% in Pb Con 2 through Pb Con 4, with lead grades declining from 17.45% to 12.82%. These products collectively represented 26.2% of the total mass. Only 3.8% of the lead reported to the copper flotation stage and 1.2% to the zinc stage, indicating the expanded lead recovery effort was effective at reducing lead crossover into subsequent flotation circuits.

Copper recovery reached 75.3%, though separation remained a challenge. A significant portion—70.0%—of the copper was still recovered in the lead flotation stage, with only 5.3% recovered in the copper flotation circuit. Copper concentrates (Cu Con 1 and Cu Con 2) assayed at 0.38% and 0.37% Cu, while the four lead concentrates averaged between 0.46% and 0.93% Cu, confirming persistent contamination. An additional 5.2% of copper reported to the zinc flotation stage.

Zinc recovery was high at 94.5%, with 22.1% of the zinc recovered in the designated zinc concentrates. Zn
Con 1 assayed at 57.8% Zn and Zn Con 2 at 24.0%, together comprising 7.6% of the total mass. However,

65.8% of the zinc still floated during the lead stage, and 6.6% was recovered during the copper stage, indicating limited sphalerite depression during the earlier stages.

Iron distribution was also evaluated. While 72.4% of the iron remained in the tailings, the remaining 27.6% was distributed primarily across the lead and zinc concentrates. Pb Con 1 and Pb Con 2 recovered 5.2% and 5.7% of the iron, with Fe grades of 5.53% and 6.07%, respectively. Zinc concentrates contributed 4.1% of the total iron, with Zn Con 1 and Zn Con 2 assaying at 0.82% and 1.00% Fe, respectively. This distribution suggests modest gangue entrainment and relatively clean zinc separation by the final stage.

Table 3.12 Flotation Test 2 Rougher and Scavenger Concentrates – Lead, Copper, Zinc, Iron Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	% Cu	% Zn	% Fe
Pb Con 1	8.6	8.6	8.6	19.85	0.93	44.0	5.5
Pb Con 2	7.9	16.4	16.4	14.84	0.84	52.4	6.7
Pb Con 3	4.2	20.6	20.6	6.42	0.67	46.4	6.7
Pb Con 4	5.5	26.2	26.2	3.94	0.46	38.6	8.3
Cu Con 1	2.6	2.6	28.8	3.81	0.38	30.5	9.2
Cu Con 2	1.4	4.0	30.2	3.57	0.37	29.2	11.2
Zn Con 1	6.5	6.5	36.7	0.49	0.19	57.8	4.6
Zn Con 2	1.1	7.6	37.7	1.31	0.23	24.0	7.3
Tail	62.3		100.0	0.58	0.09	1.6	10.6
Composite	100.0			3.91	0.28	18.2	9.1

Table 3.13 Flotation Test 2 Rougher and Scavenger Concentrates – Lead Distribution

Product	Pb Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Pb
Pb Con 1	43.4%	43.4%	43.4%	19.85
Pb Con 2	29.9%	73.3%	73.3%	17.45
Pb Con 3	6.9%	80.2%	80.2%	15.21
Pb Con 4	5.6%	85.8%	85.8%	12.82
Cu Con 1	2.6%	2.6%	88.3%	0.35
Cu Con 2	1.2%	3.8%	89.6%	0.49
Zn Con 1	0.8%	0.8%	90.4%	0.09
Zn Con 2	0.4%	1.2%	90.7%	0.12
Tail	9.3%		100%	3.91
Composite	100%			

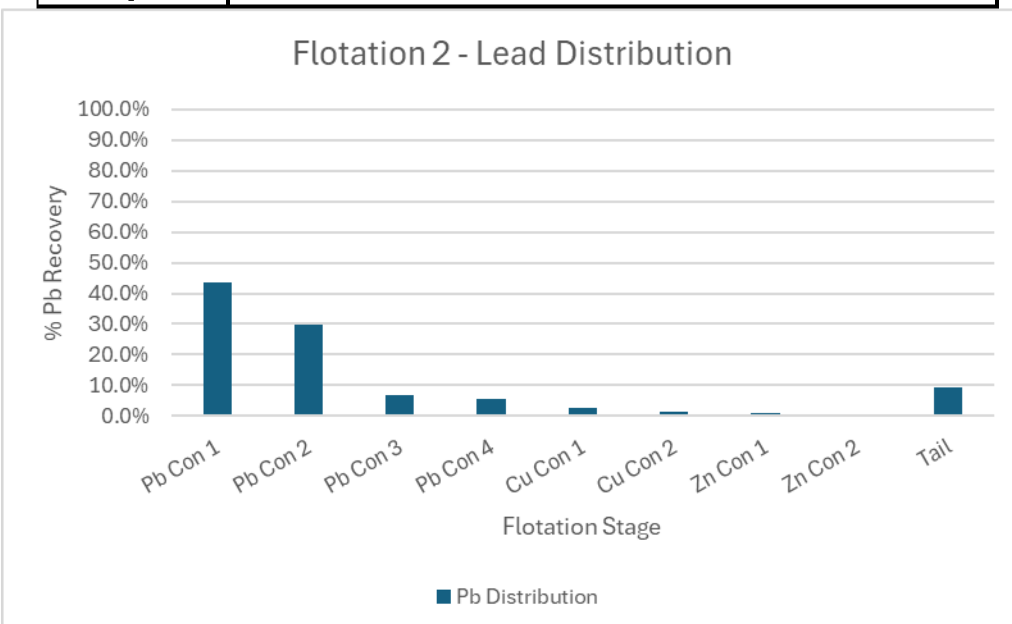


Figure 3.12 Flotation Test 2 Rougher and Scavenger Concentrates - Lead Distribution Graph

Table 3.14 Flotation Test 2 Rougher and Scavenger Concentrates – Copper Distribution

Product	Cu Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Cu
Pb Con 1	27.9%	27.9%	27.9%	0.93
Pb Con 2	23.2%	51.1%	51.1%	0.88
Pb Con 3	9.9%	61.0%	61.0%	0.84

Pb Con 4	9.0%	70.0%	70.0%	0.76
Cu Con 1	3.5%	3.5%	73.5%	0.03
Cu Con 2	1.7%	5.3%	75.3%	0.05
Zn Con 1	4.3%	4.3%	79.6%	0.03
Zn Con 2	0.9%	5.2%	80.4%	0.04
Tail	19.6%		100%	0.28
Composite	100.0%			

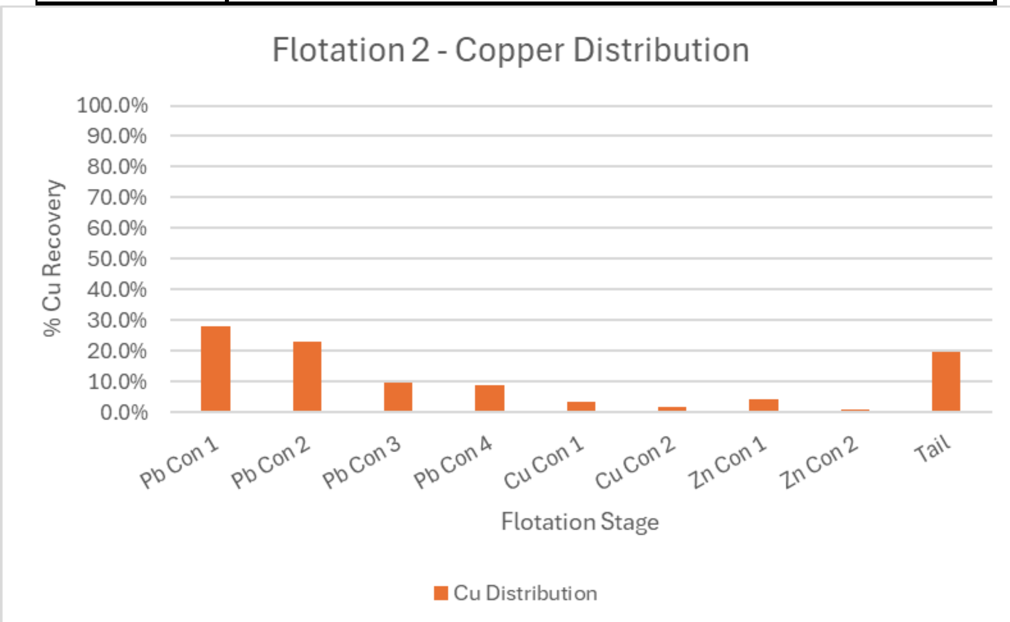


Figure 3.13 Flotation Test 2 Rougher and Scavenger Concentrates - Copper Distribution Graph

Table 3.15 Flotation Test 2 Rougher and Scavenger Concentrates – Zinc Distribution

	Zn Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn
Pb Con 1	20.7%	20.7%	20.7%	44.02
Pb Con 2	22.7%	43.4%	43.4%	48.02
Pb Con 3	10.7%	54.0%	54.0%	47.69
Pb Con 4	11.7%	65.8%	65.8%	45.76
Cu Con 1	4.4%	4.4%	70.2%	2.78
Cu Con 2	2.2%	6.6%	72.3%	3.98
Zn Con 1	20.7%	20.7%	93.0%	10.28
Zn Con 2	1.4%	22.1%	94.5%	10.67
Tail	5.5%		100%	18.21
Composite	100%			

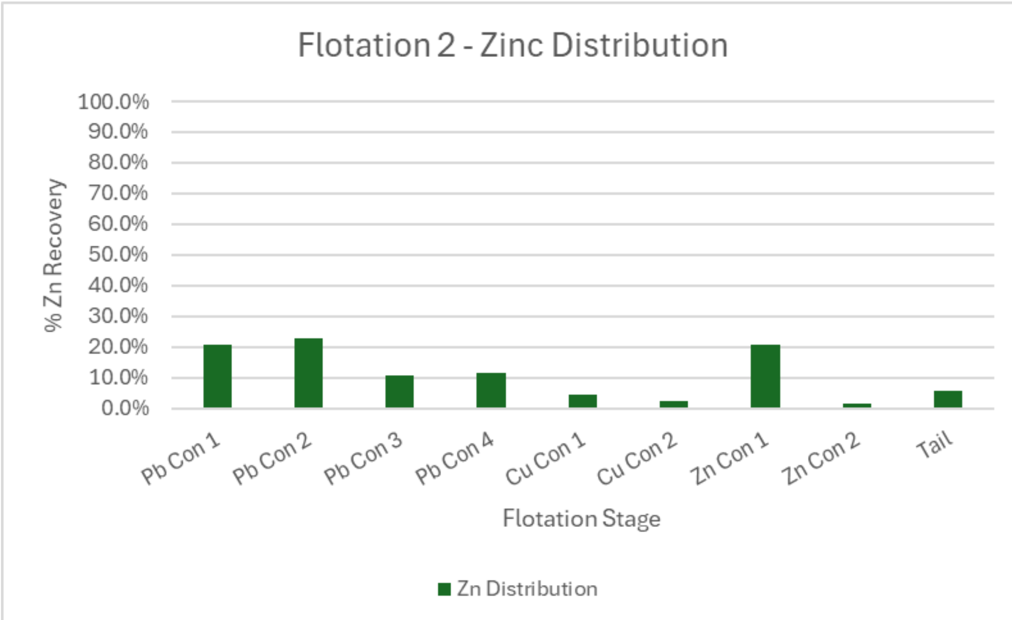


Figure 3.14 Flotation Test 2 Rougher and Scavenger Concentrates - Zinc Distribution Graph

Table 3.16 Flotation Test 2 Rougher and Scavenger Concentrates– Iron Distribution

	Fe Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Fe
Pb Con 1	5.2%	5.2%	5.2%	5.53
Pb Con 2	5.7%	10.9%	10.9%	6.07
Pb Con 3	3.1%	14.0%	14.0%	6.20
Pb Con 4	5.1%	19.1%	19.1%	6.65
Cu Con 1	2.7%	2.7%	21.8%	0.84
Cu Con 2	1.7%	4.3%	23.4%	1.31
Zn Con 1	3.3%	3.3%	26.7%	0.82
Zn Con 2	0.9%	4.1%	27.6%	1.00
Tail	72.4%		100%	9.12
Composite	100%			

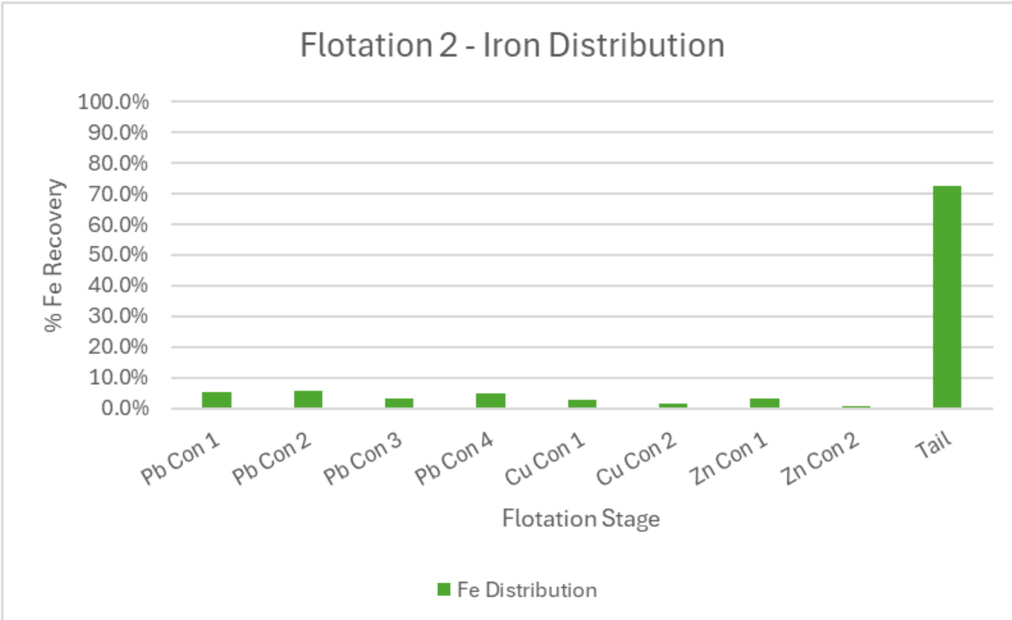


Figure 3.15 Flotation Test 2 Rougher and Scavenger Concentrates – Iron Distribution Graph

Gold and silver recovery also improved. A total of 87.6% of the gold was recovered, with 78.7% reporting to the lead stage. Silver recovery reached 89.8%, with 82.6% reporting to the lead stage. These trends continued to show that precious metals followed lead in flotation, enriching the lead concentrate and offering opportunities for further upgrading through downstream processing.

Table 3.17 Flotation Test 2 Rougher and Scavenger Concentrates – Gold and Silver Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	ppm Au	ppm Ag
Pb Con 1	8.6	8.6	8.6	2.8	3175.10
Pb Con 2	7.9	16.4	16.4	2.0	2234.56
Pb Con 3	4.2	20.6	20.6	1.5	1125.21
Pb Con 4	5.5	26.2	26.2	1.2	825.48
Cu Con 1	2.6	2.6	28.8	1.2	868.68
Cu Con 2	1.4	4.0	30.2	1.1	681.61
Zn Con 1	6.5	6.5	36.7	0.1	169.74
Zn Con 2	1.1	7.6	37.7	0.4	337.47
Tail	62.3		100.0	0.1	107.20
Composite	100.0			0.7	654.22

Table 3.18 Flotation Test 2 Rougher and Scavenger Concentrates – Gold Distribution

Product	Au Distribution %	Sep. Cum. %	Cum. %	Cum. g/mt Au
Pb Con 1	36%	35.9%	35.9%	2.80
Pb Con 2	24%	59.5%	59.5%	2.42
Pb Con 3	9%	68.7%	68.7%	2.22
Pb Con 4	10%	78.7%	78.7%	2.01
Cu Con 1	5%	4.7%	83.4%	1.93
Cu Con 2	2%	6.9%	85.6%	1.89
Zn Con 1	1%	1.3%	86.9%	1.58
Zn Con 2	1%	1.9%	87.6%	1.55
Tail	12%		100%	0.67
Composite	100%			

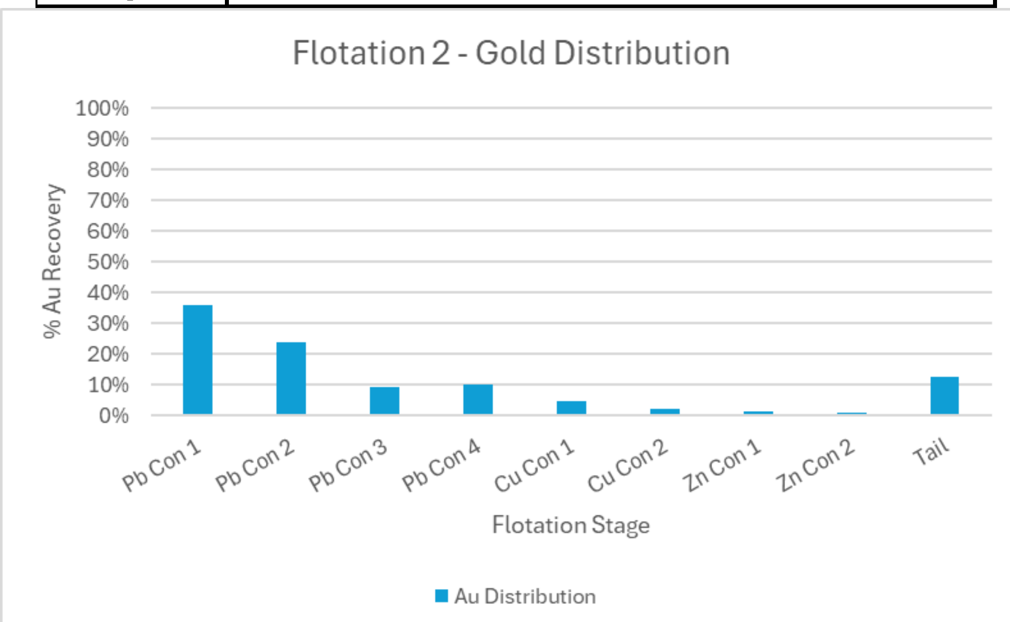


Figure 3.16 Flotation Test 2 Rougher and Scavenger Concentrates - Gold Distribution Graph

Table 3.19 Flotation Test 2 Rougher and Scavenger Concentrates – Silver Distribution

Product	Ag Distribution %	Sep. Cum. %	Cum. %	Cum. g/mt Ag
Pb Con 1	41.5%	41.5%	41.5%	3175.10
Pb Con 2	26.9%	68.5%	68.5%	2724.31

Pb Con 3	7.2%	75.7%	75.7%	2399.97
Pb Con 4	7.0%	82.6%	82.6%	2066.34
Cu Con 1	3.5%	3.5%	86.1%	1634.67
Cu Con 2	1.4%	4.9%	87.6%	1591.63
Zn Con 1	1.7%	1.7%	89.2%	1338.93
Zn Con 2	0.6%	2.2%	89.8%	1310.54
Tail	10.2%		100%	561.38
Composite	100%			

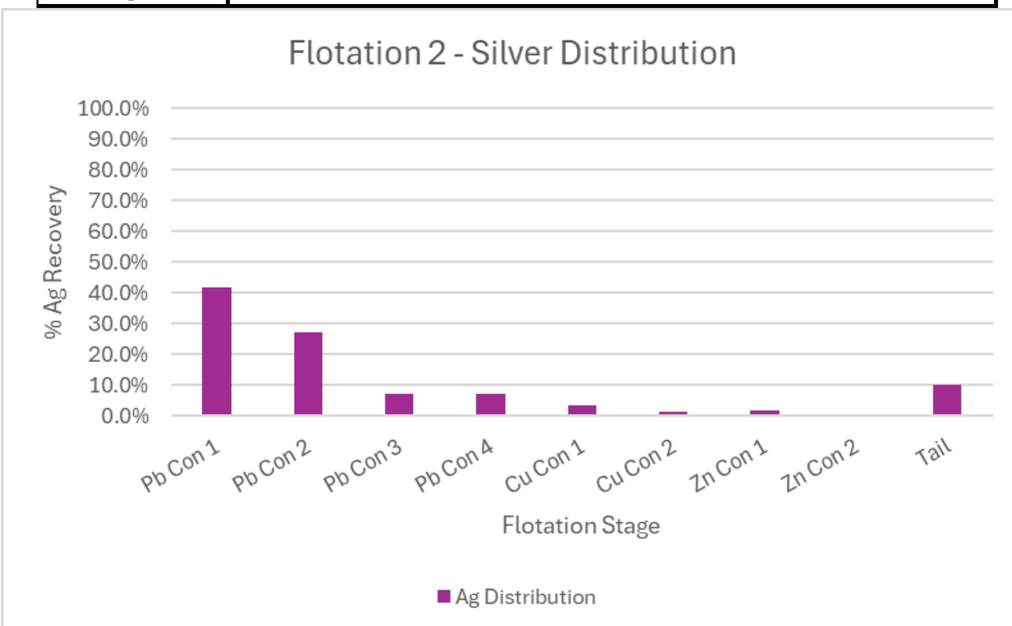


Figure 3.17 Flotation Test 2 Rougher and Scavenger Concentrates - Silver Distribution Graph

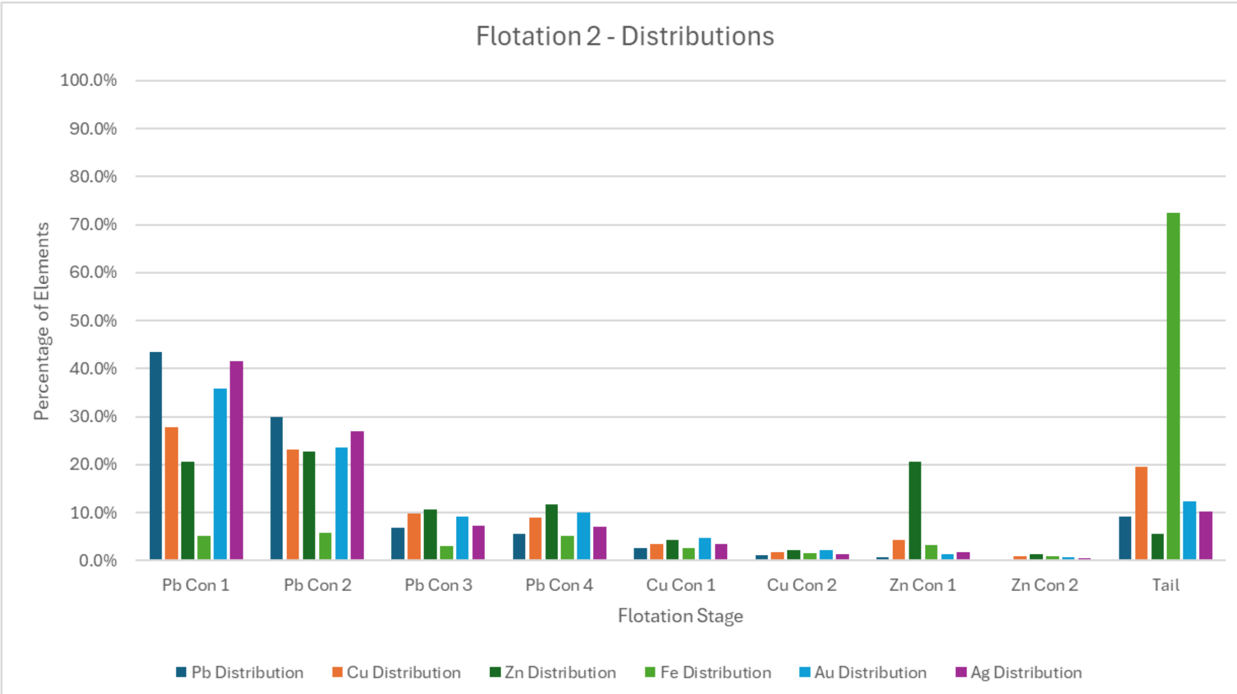


Figure 3.18 Flotation Test 2 Distributions

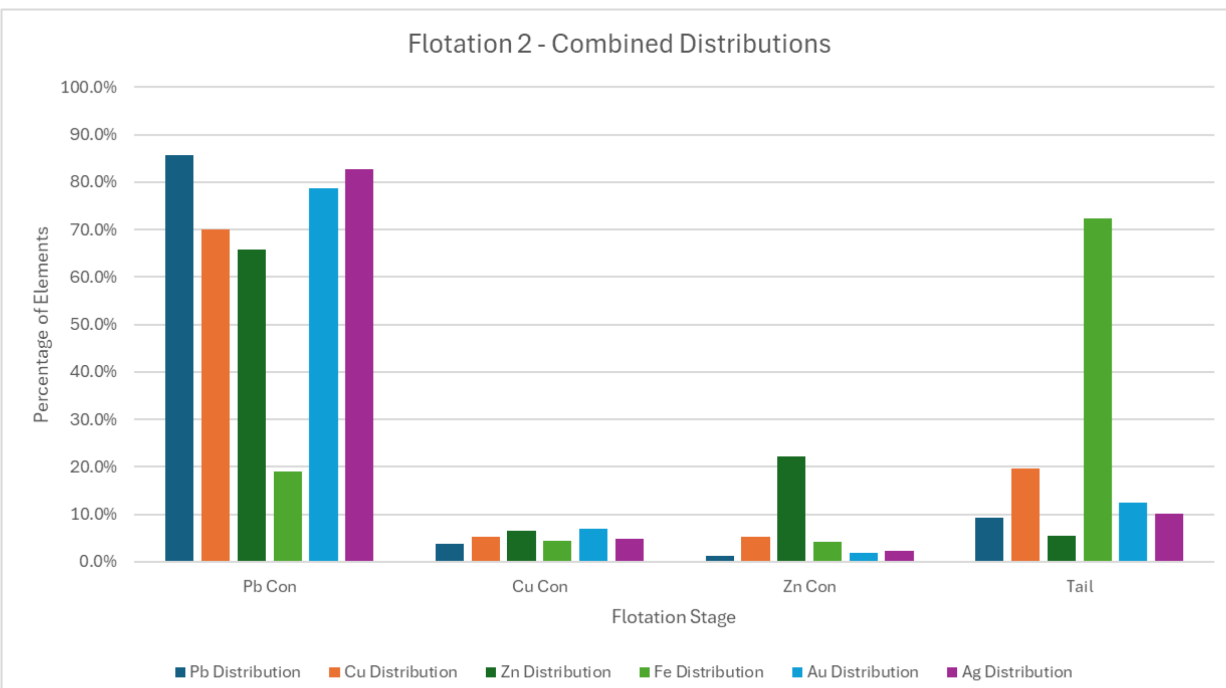


Figure 3.19 Flotation Test 2 Combined Distributions

A total of 92.7% of the total sulfur reported to the combined concentrates, while 7.3% remained in the tailings. This demonstrates a strong recovery of sulfide minerals during flotation. Sulfide sulfur recovery followed the expected trend for effective flotation, with 94.7% recovered in the concentrates and only 5.3% reporting to the tail. The nearly identical cumulative values of total sulfur and sulfide sulfur in the flotation products (both around 26.0%) suggest that nearly all recovered sulfur was in sulfide form. These results reinforce that flotation was highly effective in recovering sulfide minerals in Test 2.

Table 3.20 Flotation Test 2 Rougher and Scavenger Concentrates – Sulfur Speciation Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	% S	% S=
Pb Con 1	8.6	8.6	8.6	26.03	25.44
Pb Con 2	7.9	16.4	16.4		
Pb Con 3	4.2	20.6	20.6		
Pb Con 4	5.5	26.2	26.2		
Cu Con 1	2.6	2.6	28.8		
Cu Con 2	1.4	4.0	30.2		
Zn Con 1	6.5	6.5	36.7		
Zn Con 2	1.1	7.6	37.7		
Tail	62.3		100.0	1.25	0.86
Composite	100.0			10.60	10.14

Table 3.21 Flotation Test 2 Rougher and Scavenger Concentrates – Total Sulfur Distribution

Product	S Distribution %	Cum. %	Cum. % TS
Pb Con 1	92.7%	92.7%	26.03
Pb Con 2			26.03
Pb Con 3			26.03
Pb Con 4			26.03
Cu Con 1			26.03
Cu Con 2			26.03
Zn Con 1			26.03
Zn Con 2			26.03
Tail	7.3%	100.0%	10.60
Composite	100%		

Table 3.22 Flotation Test 2 Rougher and Scavenger Concentrates – Sulfide Sulfur Distribution

	S= Distribution		
Product	%	Cum. %	Cum. % S=
Pb Con 1	94.7%	94.7%	25.44
Pb Con 2			25.44
Pb Con 3			25.44
Pb Con 4			25.44
Cu Con 1			25.44
Cu Con 2			25.44
Zn Con 1			25.44
Zn Con 2			25.44
Tail	5.3%	100.0%	10.14
Composite	100%		

Overall, Flotation 2 demonstrated that finer grinding helped improve lead recovery and slightly reduced lead contamination in copper and zinc stages. However, copper still floated preferentially with lead, likely due to either insufficient depression or mineralogical locking. While zinc recovery remained high, the majority still reported to the lead flotation stage, suggesting the need for further reagent refinement. Planned changes for the next round of testing included switching to a more selective lead collector (Aero 3418A), adjusting zinc sulfate and sodium sulfite dosages, and continuing efforts to depress copper and zinc in the lead stage more effectively.

Figure 3.20 Flotation Test 2 – Lead Stage Froth Photographs

Figure 3.21 Flotation Test 2 – Copper Stage Froth Photographs

Figure 3.22 Flotation Test 2 – Zinc Stage Froth Photographs

3.3 Flotation Test 3

Flotation Test 3 was conducted using a 50-minute grind to achieve a P80 of 75 microns, consistent with Flotation 2, but with modified reagent schemes to improve selectivity. The lead flotation stage was reduced to a single 5-minute rougher and 3-minute scavenger pull, using 40 g/mt of Aerofloat 3418A as the collector, 770 g/mt of zinc sulfate to depress sphalerite, and 1000 g/mt of sodium sulfite to depress chalcopyrite. The pH was maintained at 9.0. The copper flotation stage also included a 5-minute rougher and 3-minute scavenger, with 40 g/mt of Aero 3894 and 20 g/mt of Aerofloat 238 as collectors, 1000 g/mt of zinc sulfate, and pH controlled at 9.0. The zinc stage, conducted at a pH of 11 to 11.5, utilized 750 g/mt of copper sulfate as an activator and 50 g/mt of PAX as the collector. MIBC was used as the frother across all stages.

Lead recovery totaled 91.7%, with 64.0% of the lead recovered in the lead flotation stage. This was split between Pb Con 1 and Pb Con 2, which together represented 7.5% of the mass and assayed 24.98% and 23.76% Pb, respectively. An additional 24.9% of the lead was recovered in the copper flotation stage, largely from Cu Con 1 and Cu Con 2, which together assayed between 9.45% and 11.23% Pb. A small amount, 2.8% of the lead, reported to the zinc flotation stage.

Copper recovery reached 87.9%, though only 18.2% was recovered in the designated copper concentrates. Cu Con 1 and Cu Con 2, which together represented 7.0% of the mass, assayed at 0.57% and 0.36% Cu, respectively. The majority of the copper, 46.5%, was still recovered during the lead flotation stage, with Pb Con 1 and Pb Con 2 assaying at 1.70% and 0.75% Cu. Another 14.4% of copper reported to the zinc stage, where Zn Con 1 and Zn Con 2 had relatively low copper grades of 0.15% and 0.05%, respectively.

Zinc recovery was strong at 93.3%, with 64.4% recovered in the zinc flotation stage. Zn Con 1, representing 16.2% of the mass, assayed at 53.9% Zn, while Zn Con 2 contained only 0.2% Zn. Zinc carryover was notable, with 15.1% recovered in the lead stage and 13.7% in the copper stage, due to incomplete sphalerite depression.

Gold and silver recoveries were 75.0% and 87.9%, respectively. The lead concentrates were the primary carriers of precious metals, with Pb Con 1 containing 5.9 ppm Au and 5146 ppm Ag, and Pb Con 2 containing 2.7 ppm Au and 4342 ppm Ag. Copper concentrates also contributed modestly to precious metal recovery, while zinc concentrates and tailings held only trace levels.

Iron distribution was dominated by the tailings, which retained 79% of the total iron at a grade of 8.18% Fe. The remaining 21.1% of iron was distributed across the concentrates, with 4.9% recovered in the lead stage, 7.7% in the copper stage, and 8.5% in the zinc stage. Iron grades in the concentrates were moderate, ranging from 1.92% in the zinc products to 5.32% in Pb Con 2 and 10.6% in Cu Con 2, reflecting a combination of entrained gangue and minor iron-bearing sulfides.

Table 3.23 Flotation Test 3 Rougher and Scavenger Concentrates – Lead, Copper, Zinc, Iron Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	Assay % Cu	Assay % Zn	Assay % Fe
Pb Con 1	3.3	3.3	3.3	24.98	1.70	27.5	4.9
Pb Con 2	4.2	7.5	7.5	23.76	0.75	27.2	5.6
Cu Con 1	4.3	4.3	11.8	9.45	0.57	27.7	8.1
Cu Con 2	2.7	7.0	14.5	11.23	0.36	24.8	10.6
Zn Con 1	16.2	16.2	30.6	0.44	0.15	53.9	3.6
Zn Con 2	5.7	21.9	36.4	0.15	0.05	0.2	1.9
Tail	63.6		100.0	0.37	0.06	1.4	10.1
Composite	100.0			2.84	0.19	13.5	8.2

Table 3.24 Flotation Test 3 Rougher and Scavenger Concentrates – Lead Distribution

Product	Pb Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Pb
Pb Con 1	28.6%	28.6%	28.6%	24.98
Pb Con 2	35.4%	64.0%	64.0%	24.29
Cu Con 1	14.4%	14.4%	78.4%	3.46
Cu Con 2	10.5%	24.9%	88.9%	4.89
Zn Con 1	2.5%	2.5%	91.4%	0.23
Zn Con 2	0.3%	2.8%	91.7%	0.22
Tail	8.3%		100%	2.84
Composite	100.0%			

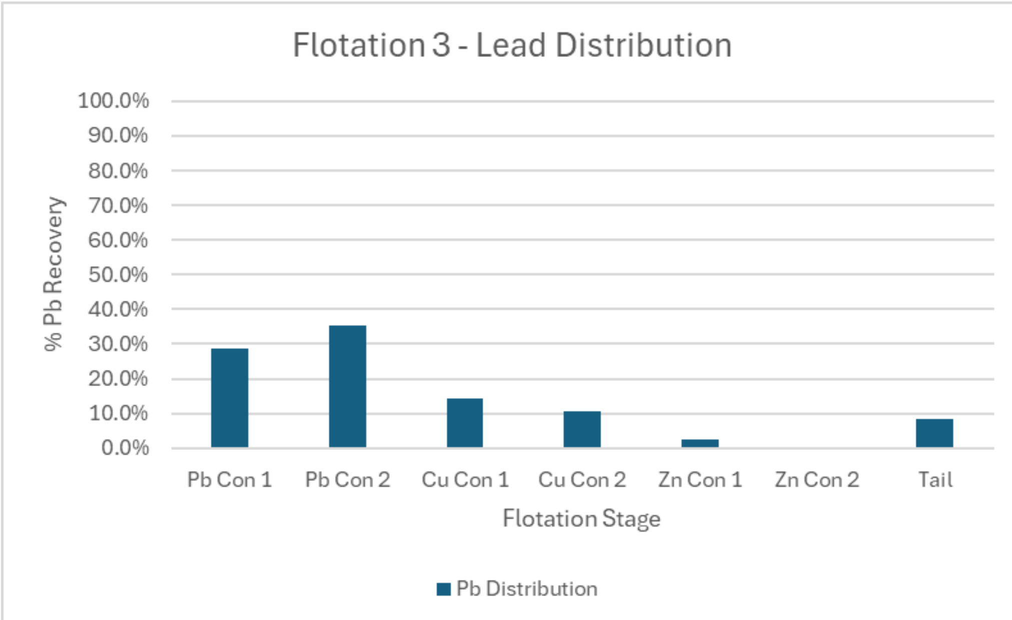


Figure 3.23 Flotation Test 3 Rougher and Scavenger Concentrates - Lead Distribution Graph

Table 3.25 Flotation Test 3 Rougher and Scavenger Concentrates – Copper Distribution

	Cu Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu
Pb Con 1	29.4%	29.4%	29.4%	1.70
Pb Con 2	17.0%	46.5%	46.5%	1.16
Cu Con 1	13.2%	13.2%	59.6%	0.21
Cu Con 2	5.1%	18.2%	64.7%	0.24
Zn Con 1	12.9%	12.9%	77.6%	0.08
Zn Con 2	1.5%	14.4%	79.1%	0.07
Tail	20.9%		100%	0.19
Composite	100.0%			

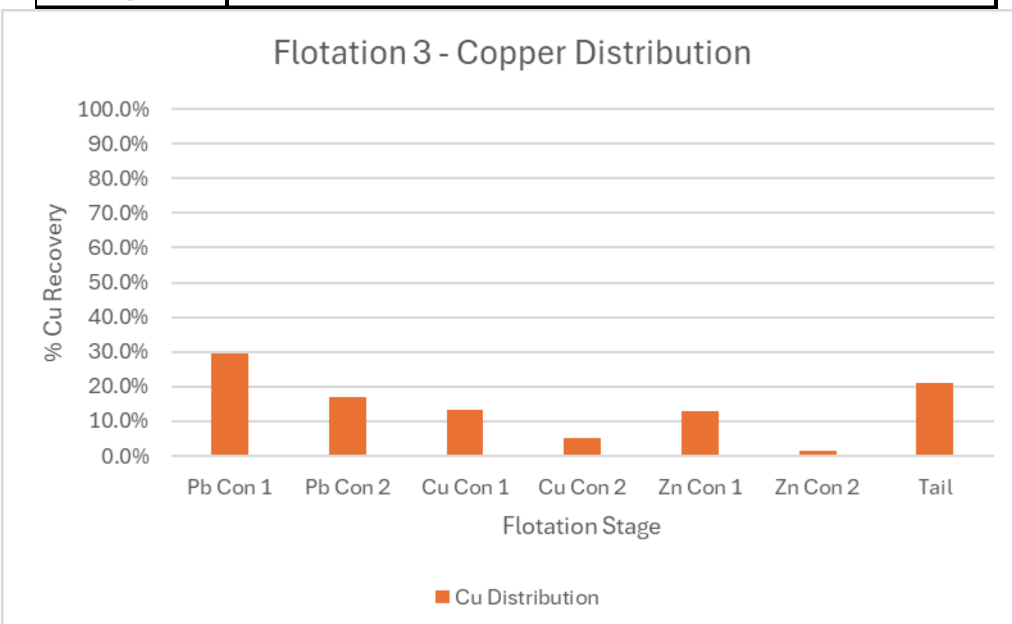


Figure 3.24 Flotation Test 3 Rougher and Scavenger Concentrates – Copper Distribution Graph

Table 3.26 Flotation Test 3 Rougher and Scavenger Concentrates – Zinc Distribution

	Zn Distribution

Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn
Pb Con 1	6.6%	6.6%	6.6%	27.50
Pb Con 2	8.5%	15.1%	15.1%	27.35
Cu Con 1	8.9%	8.9%	24.0%	10.14
Cu Con 2	4.9%	13.7%	28.9%	12.84
Zn Con 1	64.3%	64.3%	93.2%	28.41
Zn Con 2	0.1%	64.4%	93.3%	23.97
Tail	6.7%		100%	13.53
Composite	100%			

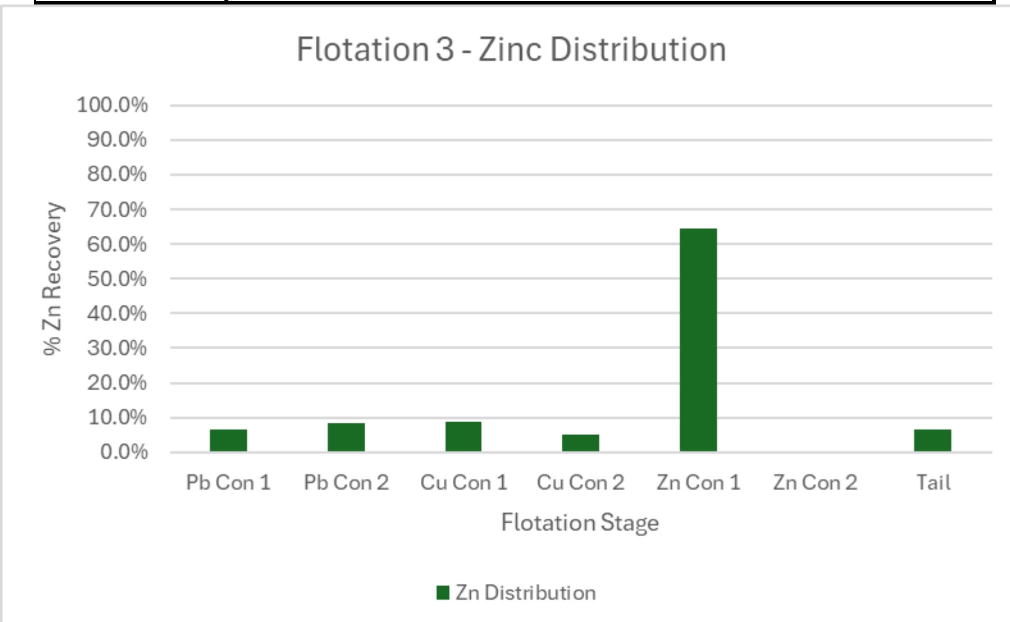


Figure 3.25 Flotation Test 3 Rougher and Scavenger Concentrates – Zinc Distribution Graph

Table 3.27 Flotation Test 3 Rougher and Scavenger Concentrates – Iron Distribution

Product	Fe Distribution			
	%	Sep. Cum. %	Cum. %	Sep. Grade % Fe
Pb Con 1	2%	2.0%	2.0%	4.92
Pb Con 2	3%	4.9%	4.9%	5.32
Cu Con 1	4%	4.3%	9.1%	2.96
Cu Con 2	3%	7.7%	12.6%	4.37
Zn Con 1	7%	7.2%	19.8%	1.92
Zn Con 2	1%	8.5%	21.1%	1.92

Tail	79%	100%	8.18
Composite	100%		

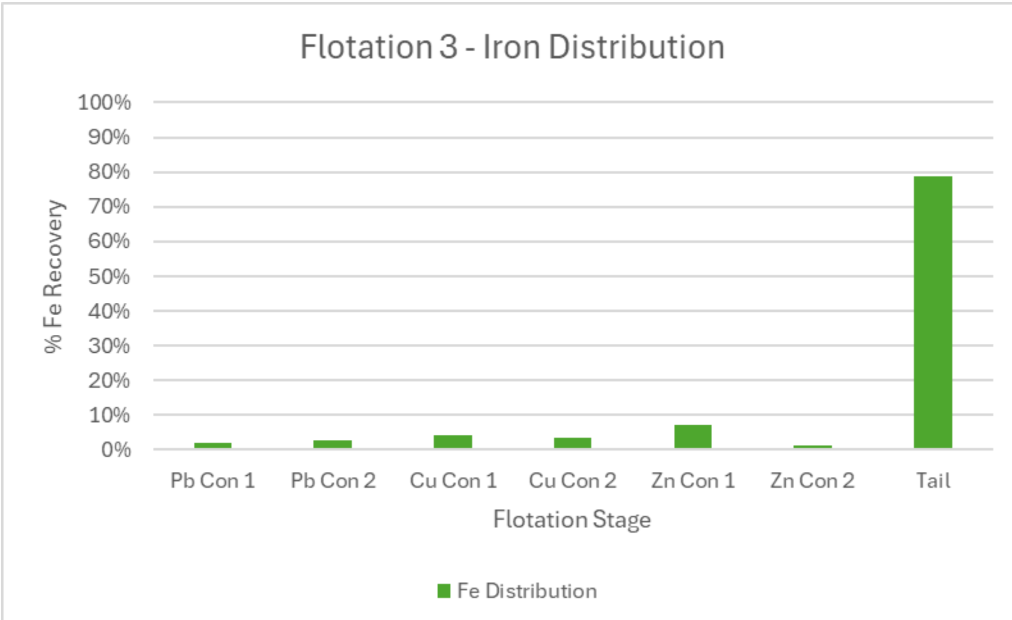


Figure 3.26 Flotation Test 3 Rougher and Scavenger Concentrates – Iron Distribution Graph

Table 3.28 Flotation Test 3 Rougher and Scavenger Concentrates – Gold and Silver Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	ppm Au	ppm Ag
Pb Con 1	3.3	3.3	3.3	5.9	5146.19
Pb Con 2	4.2	7.5	7.5	2.7	4341.92
Cu Con 1	4.3	4.3	11.8	2.1	1923.22
Cu Con 2	2.7	7.0	14.5	1.7	1719.62
Zn Con 1	16.2	16.2	30.6	0.3	152.27
Zn Con 2	5.7	21.9	36.4	0.4	314.80
Tail	63.6		100.0	0.3	112.67
Composite	100.0			0.7	594.67

Table 3.29 Flotation Test 3 Rougher and Scavenger Concentrates – Gold Distribution

	Au Distribution

Product	%	Sep. Cum. %	Cum. %	Cum. g/mt Au
Pb Con 1	28%	28.1%	28.1%	5.87
Pb Con 2	17%	44.8%	44.8%	4.06
Cu Con 1	14%	13.6%	58.4%	3.35
Cu Con 2	7%	20.4%	65.2%	3.05
Zn Con 1	6%	6.4%	71.6%	1.58
Zn Con 2	3%	9.7%	75.0%	1.40
Tail	25%		100%	0.68
Composite	100%			

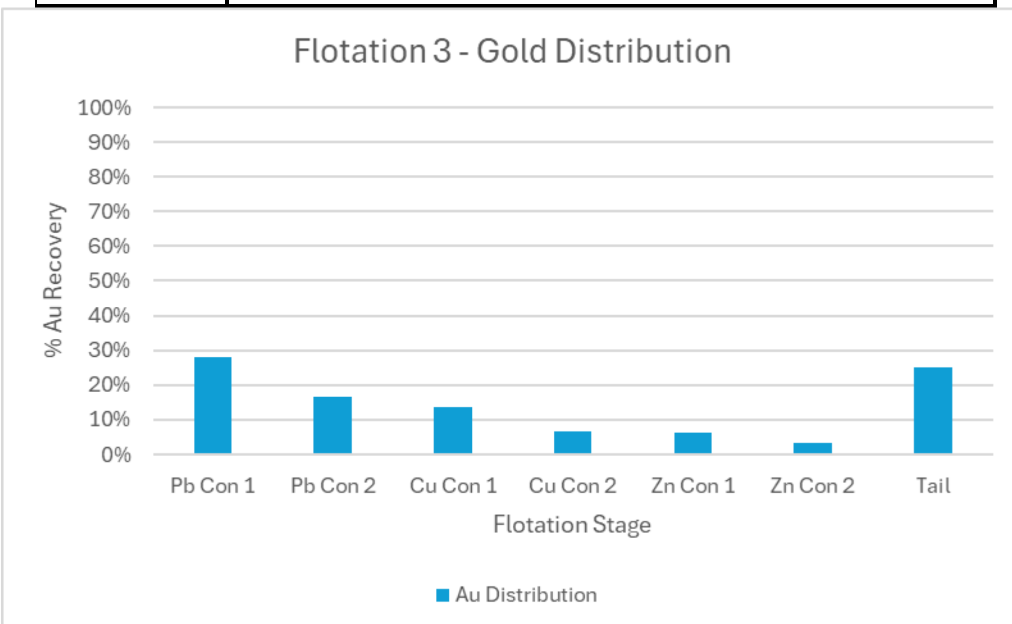


Figure 3.27 Flotation Test 3 Rougher and Scavenger Concentrates – Gold Distribution Graph

Table 3.30 Flotation Test 3 Rougher and Scavenger Concentrates – Silver Distribution

Product	Ag Distribution			
	%	Sep. Cum. %	Cum. %	Cum. g/mt Ag

Pb Con 1	28.1%	28.1%	28.1%	5146.19
Pb Con 2	30.9%	59.1%	59.1%	4691.14
Cu Con 1	14.0%	14.0%	73.1%	3676.24
Cu Con 2	7.7%	21.7%	80.8%	3316.86
Zn Con 1	4.1%	4.1%	84.9%	1648.07
Zn Con 2	3.0%	7.2%	87.9%	1438.51
Tail	12.1%		100.0%	594.67
Composite	100%			

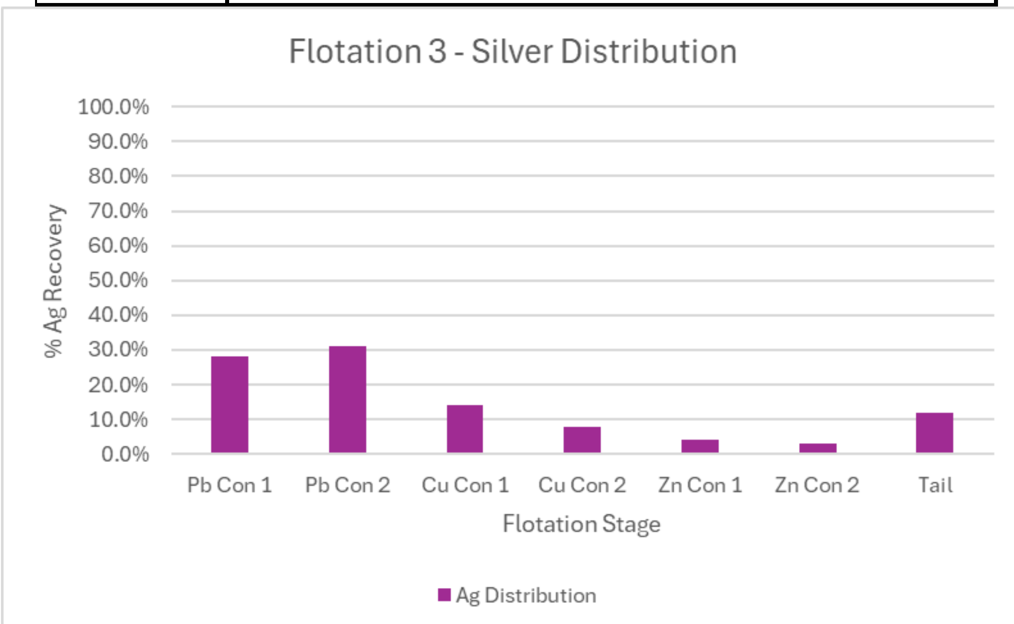


Figure 3.28 Flotation Test 3 Rougher and Scavenger Concentrates – Silver Distribution Graph

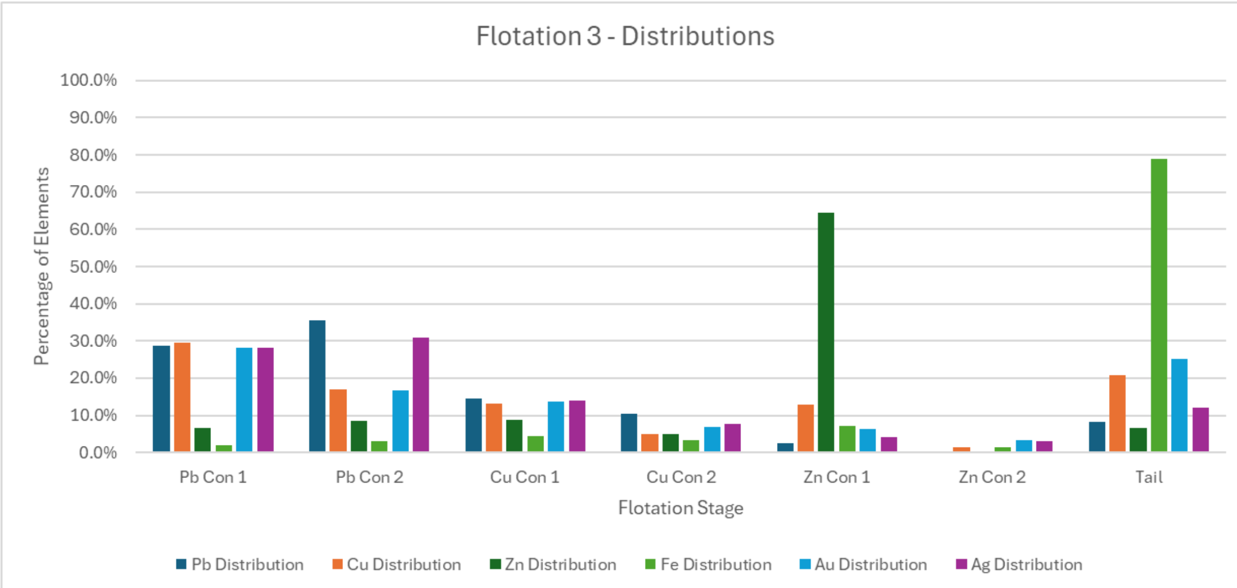


Figure 3.29 Flotation Test 3 Distributions

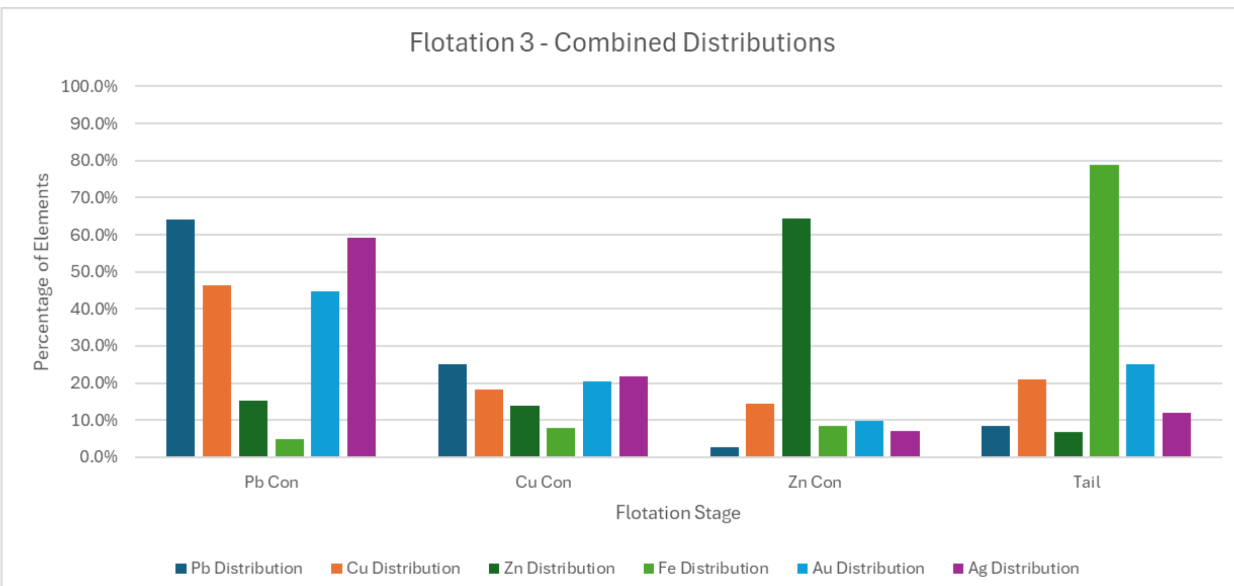


Figure 3.30 Flotation Test 3 Combined Distributions

A total of 86% of the total sulfur reported to the combined concentrates, while 14% remained in the tailings, representing slightly lower sulfur recovery than in previous tests. Sulfide sulfur recovery to the concentrates was 88%, with 12% reporting to the tail. The cumulative sulfide sulfur content in the flotation products was 24.62%, very close to the cumulative total sulfur value of 24.95%, which confirms that most of the sulfur in the recovered material was in sulfide form. These results suggest that while sulfide recovery remained effective, overall selectivity may have been slightly reduced compared to earlier tests, potentially indicating some carryover of non-sulfide components or reagent-related effects that could benefit from further flowsheet refinement.

Table 3.31 Flotation Test 3 Rougher and Scavenger Concentrates – Sulfur Speciation Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	% S	% S=
Pb Con 1	3.3	3.3	3.3		
Pb Con 2	4.2	7.5	7.5	24.95	24.62
Cu Con 1	4.3	4.3	11.8		
Cu Con 2	2.7	7.0	14.5		
Zn Con 1	16.2	16.2	30.6		
Zn Con 2	5.7	21.9	36.4		
Tail	63.6		100.0	2.40	1.87
Composite	100.0			10.60	10.14

Table 3.32 Flotation Test 3 Rougher and Scavenger Concentrates – Total Sulfur **Distribution**

Product	S Distribution %	Cum. %	Cum. % TS
Pb Con 1			24.95
Pb Con 2			24.95
Cu Con 1	86%	86%	24.95
Cu Con 2			24.95
Zn Con 1			24.95
Zn Con 2			24.95
Tail	14%	100%	10.60
Composite	100%		

Table 3.33 Flotation Test 3 Rougher and Scavenger Concentrates – Sulfide Sulfur Distribution

Product	S= Distribution %	Cum. %	Cum. % S=
Pb Con 1			24.62
Pb Con 2			24.62
Cu Con 1			24.62
Cu Con 2	88%	88%	24.62
Zn Con 1			24.62
Zn Con 2			24.62
Tail	12%	100%	10.14
Composite	100%		

Although switching to Aerofloat 3418A was intended to improve lead selectivity, copper continued to report strongly to the lead stage. Increasing zinc sulfate dosage in the lead and copper stages appeared to help direct zinc primarily to the zinc stage, reducing its presence in earlier stages. However, the high copper recovery in the lead concentrate indicated that sodium sulfite may not have been sufficient to suppress chalcopyrite under the current conditions. The test confirmed that while the finer grind and reagent adjustments improved zinc distribution, copper-laden lead concentrate remained a challenge.

Pb 1 Before | Pb 1 After | Pb 2 Before | Pb 2 After

Figure 3.31 Flotation Test 3 – Lead Stage Froth Photographs

Cu Con 1 Before | Cu Con 1After | Cu Con 2 Before | Cu Con 2 After

Figure 3.32 Flotation Test 3 – Copper Stage Froth Photographs

Zn Con 1 Before | Zn Con 1After | Zn Con 2 Before | Zn Con 2 After

Conclusions

Across the three sequential flotation tests conducted on the Darwin composite, significant progress was made in understanding the behavior of lead, copper, and zinc under various reagent regimes and grind sizes. Each test aimed to produce separate, high-grade concentrates for each base metal while maximizing recovery of gold and silver, and minimizing contamination across flotation stages.

Flotation Test 1, conducted at a grind size of P80 106 microns, demonstrated moderate recoveries but poor selectivity. A total of 73.8% of the lead, 65.4% of the copper, and 91.2% of the zinc were recovered, though most of the copper and a significant portion of zinc reported to the lead concentrate. This indicated a lack of mineral liberation and poor stage separation, especially for copper. Precious metal recovery was strong, with 76.3% of the gold and 82.7% of the silver recovered, primarily in the lead stage. Iron distribution showed 14.7% reporting to the lead concentrate, 4.0% to copper, and 9.6% to zinc, for a total of 28.4% recovered in flotation products.

Flotation Test 2 utilized a finer grind (P80 75 microns) and increased reagent dosages, particularly in the lead stage. This resulted in improved overall recoveries and better selectivity for lead, with 85.8% of the lead recovered in the lead stage. Zinc recovery was 94.5%, with 22.1% captured in the zinc stage, though the majority still floated with lead. Copper recovery increased to 80.4%, but 70.0% still reported to the lead flotation, confirming persistent issues with copper depression and potential mineralogical locking. Gold and silver recoveries improved to 87.6% and 89.8%, respectively, again dominated by the lead concentrate. Iron distribution remained consistent, with 19.1% reporting to lead, 4.3% to copper, and 4.1% to zinc, totaling 27.6% in flotation products.

Flotation Test 3 retained the P80 75 micron grind and introduced a revised reagent scheme, replacing Aerofloat 7310 with the more selective Aerofloat 3418A and increasing sodium sulfite for chalcopyrite depression. Zinc sulfate dosages were also optimized in both the lead and copper stages. While zinc deportment improved—with 64.4% reporting to the zinc stage and only 15.1% to lead—copper continued to report heavily to the lead concentrate, with only 18.2% recovered in the copper stage and 46.5% still in lead. Lead recovery remained consistent at 91.7%, while gold and silver recoveries were 75.0% and 87.9%, respectively. Iron recovery to flotation products was slightly lower in this test, with 4.9% reporting to lead, 7.7% to copper, and 8.5% to zinc, for a total of 21.1%.

Overall, the testing confirmed that finer grinding significantly improves metal liberation and recovery but does not fully resolve copper deportment in the lead concentrate. Zinc flotation performance improved with higher activator and depressant control. However, the limited response of copper to sodium sulfite suggests either insufficient depression or strong mineralogical association with lead minerals. Iron deportment trends were consistent across all tests, with the majority of iron reporting to tailings, but a notable fraction recovered in concentrates, particularly in earlier flotation stages, highlighting the potential for further refinement in managing gangue recovery.

Recommendation

Based on the results from all three flotation tests and the supporting mineralogical analysis, the following recommendations are proposed to improve metal recovery, enhance concentrate selectivity, and refine the overall flotation flowsheet:

- **Implement sodium metabisulfite** in place of or in addition to sodium sulfite to improve chalcopyrite depression during the lead flotation stage, as sodium sulfite alone was not effective.
- **Increase the number of lead concentrate pulls**, as demonstrated in Flotation 2, to maximize early lead recovery and reduce lead contamination in copper and zinc concentrates.
- **Test a revised flotation flowsheet** that includes lead flotation followed by zinc flotation and a copper cleaning step on the lead concentrate to improve separation and product quality.
- **Slightly reduce zinc sulfate dosage** in the lead stage to avoid overdosing, which may result in sphalerite surface coating and unintentional flotation.
- **Maintain or slightly reduce grind size below P80 75 µm** to improve mineral liberation and further reduce metal losses to tails.

Appendix A

Woods Process Services April 25, 2025
XRF Results for Sample, "1096 HS" Lab no. 225073

IDENT	Na_2O	MgO	Al_2O_3	SiO_2	P_2O_5	S	Cl	K_2O	CaO	TiO_2	MnO	Fe_2O_3	BaO
	Wt %												
1096 HS	0.27	1.97	3.33	24.3	0.20	12.0	< 0.02	0.12	21.9	0.24	0.59	11.9	0.01
Quality Control - Replicate (R) sample and standard reference material (GSP-2) analyzed with sample													
1096 HS(R)	0.23	1.95	3.33	24.3	0.20	12.0	< 0.02	0.12	21.9	0.24	0.59	11.9	0.04
GSP-2-XRF	3.11	1.12	14.6	66.2	0.33	0.07	0.06	5.38	2.10	0.59	0.04	4.37	0.14
GSP-2-known	2.78	0.96	14.9	66.6	0.29	----	----	5.38	2.10	0.66	0.04	4.90	0.15

IDENT	V	Cr	Co	Ni	W	Cu	Zn	As	Sn	Pb	Mo	Sr	U
	PPM												
1096 HS	40	309	86	149	< 10	2270	156000	< 20	< 20	27000	< 10	24	< 10
Quality Control													
1096 HS(R)	52	298	76	155	< 10	2280	156000	28	< 20	27000	< 10	23	< 10
GSP-2-XRF	41	23	< 10	12	< 10	44	103	< 20	< 20	33	< 10	215	< 10
GSP-2-known	52	20	7	17	--	43	120	--	--	42	--	240	2

Ident	Th	Nb	Zr	Rb	Y
	PPM				
1096 HS	< 10	< 10	49	< 10	< 10
Quality Control					
1096 HS(R)	< 10	< 10	48	< 10	< 10
GSP-2-XRF	100	21	509	208	30
GSP-2-known	105	27	550	245	28

Initial_____

Date________

Analysis Performed By The Mineral Lab, Inc

Figure 6.1 The Mineral Lab XRF Results

6-49

Woods Process Services LLC April 28, 2025
XRD Results for Sample "1096 HS" Lab no. 225073

Mineral Name	Chemical Formula	Approx. Wt %
Sphalerite	ZnS	26
Pyrite	FeS_2	5
Galena	PbS	<3
Chalcopyrite	$CuFeS_2$	<2?
Anglesite	$PbSO_4$	<3
Gypsum	$CaSO_4 \cdot 2H_2O$	<3?
Garnet	$(Ca,Fe,Mg,Mn)_3(Al,Fe)_2(SiO_4)_{3-x}(OH)_{2x}$	41
Clinopyroxene	$Ca(Mg,Fe,Mn,Al,Ti)(Si,Al)_2O_6$	11
Quartz	SiO_2	6
"Unidentified"	?	<3

Initial ________

Date ________

Analysis performed by The Mineral Lab, Inc

Figure 6.2 The Mineral Lab XRD Results

David Francisco Assay Services

2122 Fort Bridger Road, Fernley, Nevada 89408, 775-379-2284

Certificate of Analysis

Date: 4/21/2025

Job: WP-215

Client: Woods Process

Type: Flotation

Project: 1096

	Sample ID	Sample Weight (g)	Au mg	Ag mg	Au oz/t	Ag oz/t
1	Head Sample A	15.00	0.011	8.979	0.021	17.459
2	Head Sample B	15.00	0.011	9.003	0.021	17.506
3	Head Sample C	15.00	0.011	8.989	0.021	17.479
4	F2 PB1	7.50	0.028	27.466	0.109	106.813
5	F2 PB2	7.50	0.013	17.421	0.051	67.749
6	F2 CU1	7.50	0.010	8.930	0.039	34.728
7	F2 CU2	7.50	0.009	6.261	0.035	24.349
8	F2 ZN1	7.50	0.002	1.422	0.008	5.530
9	F2 ZN2	7.50	0.006	4.248	0.023	16.520
10	F2 Tails	15.00	0.004	2.636	0.008	5.126
11	F3 PB1	7.50	0.021	23.813	0.082	92.607
12	F3 PB2	7.50	0.015	16.759	0.058	65.175
13	F3 PB3	7.50	0.011	8.439	0.043	32.819
14	F3 PB4	7.50	0.009	6.191	0.035	24.076
15	F3 CU1	7.50	0.009	6.515	0.035	25.336
16	F3 CU2	7.50	0.008	5.112	0.031	19.880
17	F3 ZN1	7.50	0.001	1.273	0.004	4.951
18	F3 ZN2	7.50	0.003	2.531	0.012	9.843
19	F3 Tails	15.00	0.002	1.608	0.004	3.127
20	F4 PB1	7.50	0.044	38.596	0.171	150.097
21	F4 PB2	7.50	0.020	32.564	0.078	126.639
22	F4 CU1	7.50	0.016	14.424	0.062	56.094
23	F4 CU2	7.50	0.013	12.897	0.051	50.156
24	F4 ZN1	7.50	0.002	1.142	0.008	4.441
25	F4 ZN2	7.50	0.003	2.361	0.012	9.182
26	F4 Tails	15.00	0.004	1.690	0.008	3.286
	Blank				< 0.001	< 0.050
Measured	CRM SL77				0.150	0.822
Certified	**CRM SL77**				**0.151**	**0.850**

Approved By:

David Francisco

Figure 6.3 Fire Assay Results

WOODS PROCESS
SERVICES, LLC

7 Appendix B

Table 7.1 Flotation Test 1 – All Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	Assay % Pb	% Cu	% Zn	% Fe	ppm Au	ppm Ag	% S	% S=
Pb Con 1	8.1	8.1	8.1	11.15	1.15	24.9	13.3	3.7	3662.18		
Pb Con 2	4.5	12.6	12.6	11.98	0.58	45.7	7.5	1.7	2322.83		
Cu Con 1	3.9	3.9	16.5	5.54	0.49	53.5	6.9	1.3	1190.68	25.91	25.86
Cu Con 2	2.1	6.0	18.6	3.76	0.37	49.6	5.8	1.2	834.81		
Zn Con 1	17.0	17.0	35.5	0.51	0.17	62.8	3.8	0.3	189.60		
Zn Con 2	3.1	20.1	38.7	2.11	0.26	19.8	9.1	0.8	566.41		
Tail	61.3		100.0	1.09	0.16	2.9	11.3	0.3	175.74	0.95	0.23
Composite	100.0			2.56	0.28	20.2	9.6	0.7	622.15	10.60	10.14
			Assayed Head:	3.54	0.21	21.5	8.9	0.7	599.36	10.60	10.14
			XRF:	2.70	0.23	15.6					

Table 7.2 Flotation Test 1 – Lead and Copper Distributions

	Pb Distribution				Cu Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Pb	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu
Pb Con 1	35.3%	35.3%	35.3%	11.15	33%	33.3%	33.3%	1.15
Pb Con 2	21.0%	56.3%	56.3%	11.45	9%	42.5%	42.5%	0.94
Cu Con 1	8.5%	8.5%	64.7%	1.31	7%	6.8%	49.3%	0.12
Cu Con 2	3.1%	11.5%	67.8%	1.59	3%	9.6%	52.1%	0.14
Zn Con 1	3.4%	3.4%	71.2%	0.24	10%	10.3%	62.4%	0.08
Zn Con 2	2.6%	6.0%	73.8%	0.40	3%	13.3%	65.4%	0.10
Tail	26.2%		100%	2.56	35%		100%	0.28
Composite	100%				100%			

Table 7.3 Flotation Test 1 – Zinc and Iron Distributions

	Zn Distribution				Fe Distribution			
Product	%	Sep. Cum. %	Cum. %	Sep. Grade % Zn	%	Sep. Cum. %	Cum. %	Sep. Grade % Fe
Pb Con 1	10%	10.0%	10.0%	24.90	11%	11.2%	11.2%	13.32
Pb Con 2	10%	20.1%	20.1%	32.30	3%	14.7%	14.7%	11.25
Cu Con 1	10%	10.3%	30.4%	12.67	3%	2.8%	17.5%	1.63
Cu Con 2	5%	15.4%	35.5%	16.83	1%	4.0%	18.7%	2.10
Zn Con 1	53%	52.6%	88.1%	29.96	7%	6.7%	25.4%	1.81
Zn Con 2	3%	55.7%	91.2%	29.13	3%	9.6%	28.4%	2.41
Tail	9%		100%	20.24	72%		100%	9.64
Composite	100%				100%			

Table 7.4 Flotation Test 1 – Gold and Silver Distributions

	Au Distribution				Ag Distribution			
Product	%	Sep. Cum. %	Cum. %	Cum. g/mt Au	%	Sep. Cum. %	Cum. %	Cum. g/mt Ag
Pb Con 1	44%	43.7%	43.7%	3.73	47.7%	47.7%	47.7%	3662.18
Pb Con 2	11%	55.0%	55.0%	3.02	16.7%	64.4%	64.4%	3185.29
Cu Con 1	8%	7.5%	62.5%	2.62	7.5%	7.5%	71.8%	2712.58
Cu Con 2	4%	11.2%	66.1%	2.46	2.8%	10.3%	74.6%	2500.73
Zn Con 1	7%	6.5%	72.7%	1.41	5.2%	5.2%	79.8%	1397.83
Zn Con 2	4%	10.2%	76.3%	1.36	2.9%	8.0%	82.7%	1330.25
Tail	24%		100.0%	0.69	17.3%		100.0%	622.15
Composite	100%				100%			

7.5 Flotation Test 1 Sulfur Speciation Distributions

	S Distribution			S= Distribution		
Product	%	Cum. %	Cum. % TS	%	Cum. %	Cum. % S=

Pb Con 1	94.5%	94.5%	25.91	98.6%	98.6%	25.86
Pb Con 2			25.91			25.86
Cu Con 1			25.91			25.86
Cu Con 2			25.91			25.86
Zn Con 1			25.91			25.86
Zn Con 2			25.91			25.86
Tail	5.5%	100.0%	10.60	1.4%	100.0%	10.14
Composite	100%			100%		

Table 7.6 Flotation Test 2 – All Results

Product	Wt. %	Sep. Wt. %	Cum. Wt. %	% Pb	Assay % Cu	% Zn	% Fe	ppm Au	ppm Ag	% S	% S=
Pb Con 1	8.6	8.6	8.6	19.85	0.93	44.0	5.5	2.8	3175.10	26.03	25.44
Pb Con 2	7.9	16.4	16.4	14.84	0.84	52.4	6.7	2.0	2234.56		
Pb Con 3	4.2	20.6	20.6	6.42	0.67	46.4	6.7	1.5	1125.21		
Pb Con 4	5.5	26.2	26.2	3.94	0.46	38.6	8.3	1.2	825.48		
Cu Con 1	2.6	2.6	28.8	3.81	0.38	30.5	9.2	1.2	868.68		
Cu Con 2	1.4	4.0	30.2	3.57	0.37	29.2	11.2	1.1	681.61		
Zn Con 1	6.5	6.5	36.7	0.49	0.19	57.8	4.6	0.1	169.74		
Zn Con 2	1.1	7.6	37.7	1.31	0.23	24.0	7.3	0.4	337.47		
Tail	62.3		100.0	0.58	0.09	1.6	10.6	0.1	107.20	1.25	0.86
Composite	100.0			3.91	0.28	18.2	9.1	0.7	654.22	10.60	10.14
			Assayed Head:	3.54	0.21	21.5	8.9	0.7	599.4	10.60	10.14
			XRF:	2.70	0.23	15.6					

Table 7.7 Flotation Test 2 – Lead and Copper Distributions

Pb Distribution				Cu Distribution			
%	Sep. Cum. %	Cum. %	Sep. Grade % Pb	%	Sep. Cum. %	Cum. %	Sep. Grade % Cu
43.4%	43.4%	43.4%	19.85	27.9%	27.9%	27.9%	0.93
29.9%	73.3%	73.3%	17.45	23.2%	51.1%	51.1%	0.88
6.9%	80.2%	80.2%	15.21	9.9%	61.0%	61.0%	0.84
5.6%	85.8%	85.8%	12.82	9.0%	70.0%	70.0%	0.76
2.6%	2.6%	88.3%	0.35	3.5%	3.5%	73.5%	0.03
1.2%	3.8%	89.6%	0.49	1.7%	5.3%	75.3%	0.05
0.8%	0.8%	90.4%	0.09	4.3%	4.3%	79.6%	0.03
0.4%	1.2%	90.7%	0.12	0.9%	5.2%	80.4%	0.04
9.3%		100%	3.91	19.6%		100%	0.28
100%				100.0%			

Table 7.8 Flotation Test 2 – Zinc and Iron Distributions

Zn Distribution				Fe Distribution			
%	Sep. Cum. %	Cum. %	Sep. Grade % Zn	%	Sep. Cum. %	Cum. %	Sep. Grade % Fe
20.7%	20.7%	20.7%	44.02	5.2%	5.2%	5.2%	5.53
22.7%	43.4%	43.4%	48.02	5.7%	10.9%	10.9%	6.07
10.7%	54.0%	54.0%	47.69	3.1%	14.0%	14.0%	6.20
11.7%	65.8%	65.8%	45.76	5.1%	19.1%	19.1%	6.65
4.4%	4.4%	70.2%	2.78	2.7%	2.7%	21.8%	0.84
2.2%	6.6%	72.3%	3.98	1.7%	4.3%	23.4%	1.31
20.7%	20.7%	93.0%	10.28	3.3%	3.3%	26.7%	0.82
1.4%	22.1%	94.5%	10.67	0.9%	4.1%	27.6%	1.00
5.5%		100%	18.21	72.4%		100%	9.12
100%				100%			

7.9 Flotation Test 2 Gold and Silver Distributions

Au Distribution				Ag Distribution			
%	Sep. Cum. %	Cum. %	Cum. g/mt Au	%	Sep. Cum. %	Cum. %	Cum. g/mt Ag
36%	35.9%	35.9%	2.80	41.5%	41.5%	41.5%	3175.10
24%	59.5%	59.5%	2.42	26.9%	68.5%	68.5%	2724.31
9%	68.7%	68.7%	2.22	7.2%	75.7%	75.7%	2399.97
10%	78.7%	78.7%	2.01	7.0%	82.6%	82.6%	2066.34
5%	4.7%	83.4%	1.93	3.5%	3.5%	86.1%	1634.67
2%	6.9%	85.6%	1.89	1.4%	4.9%	87.6%	1591.63
1%	1.3%	86.9%	1.58	1.7%	1.7%	89.2%	1338.93
1%	1.9%	87.6%	1.55	0.6%	2.2%	89.8%	1310.54
12%		100%	0.67	10.2%		100%	561.38
100%				100%			

Table 7.10 Flotation Test 2 – Sulfur Speciation Distributions

	S Distribution			S= Distribution		
roduct	%	Cum. %	Cum. % TS	%	Cum. %	Cum. % S=
Con 1			26.03			25.44
Con 2			26.03			25.44
Con 3			26.03			25.44
Con 4			26.03			25.44
Con 1	92.7%	92.7%	26.03	94.7%	94.7%	25.44
Con 2			26.03			25.44
Con 1			26.03			25.44
Con 2			26.03			25.44
Tail	7.3%	100.0%	10.60	5.3%	100.0%	10.14
mposite	100%			100%		

Table 7.11 Flotation Test 3 – All Results

Sep. Wt. %	Cum. Wt. %	Assay % Pb	% Cu	% Zn	% Fe	ppm Au	ppm Ag	% S	% S=
3.3	3.3	24.98	1.70	27.5	4.9		5146.19		
7.5	7.5	23.76	0.75	27.2	5.6	5.9	4341.92		
						2.7			
4.3	11.8	9.45	0.57	27.7	8.1	2.1	1923.22	24.95	24.62
7.0	14.5	11.23	0.36	24.8	10.6	1.7	1719.62		
16.2	30.6	0.44	0.15	53.9	3.6	0.3	152.27		
21.9	36.4	0.15	0.05	0.2	1.9	0.4	314.80		
	100.0	0.37	0.06	1.4	10.1	0.3	112.67	2.40	1.87
		2.84	0.19	13.5	8.2	0.7	594.67	10.60	10.14
	Assayed Head:	3.54	0.21	21.5	8.9	0.7	599.36	10.60	10.14
	XRF:	2.70	0.23	15.6					

Table 7.12 Flotation Test 3 – Lead and Copper Distributions

Pb Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Pb	Cu Distribution %	Sep. Cum. %	Cum. %	Sep. Grade % Cu
28.6%	28.6%	28.6%	24.98	29.4%	29.4%	29.4%	1.70
35.4%	64.0%	64.0%	24.29	17.0%	46.5%	46.5%	1.16
14.4%	14.4%	78.4%	3.46	13.2%	13.2%	59.6%	0.21
10.5%	24.9%	88.9%	4.89	5.1%	18.2%	64.7%	0.24
2.5%	2.5%	91.4%	0.23	12.9%	12.9%	77.6%	0.08
0.3%	2.8%	91.7%	0.22	1.5%	14.4%	79.1%	0.07
8.3%		100%	2.84	20.9%		100%	0.19
100.0%				100.0%			

Table 7.13 Flotation Test 3 – Zinc and Iron Distributions

Zn Distribution			Fe Distribution			
Sep. Cum. %	Cum. %	Sep. Grade % Zn	%	Sep. Cum. %	Cum. %	Sep. Grade % Fe
6.6%	6.6%	27.50	2%	2.0%	2.0%	4.92
15.1%	15.1%	27.35	3%	4.9%	4.9%	5.32
8.9%	24.0%	10.14	4%	4.3%	9.1%	2.96
13.7%	28.9%	12.84	3%	7.7%	12.6%	4.37
64.3%	93.2%	28.41	7%	7.2%	19.8%	1.92
64.4%	93.3%	23.97	1%	8.5%	21.1%	1.92
	100%	13.53	79%		100%	8.18
			100%			

Table 7.14 Flotation Test 3 – Gold and Silver Distributions

Au Distribution			Ag Distribution			
Sep. Cum. %	Cum. %	Cum. g/mt Au	%	Sep. Cum. %	Cum. %	Cum. g/mt Ag
28.1%	28.1%	5.87	28.1%	28.1%	28.1%	5146.19
44.8%	44.8%	4.06	30.9%	59.1%	59.1%	4691.14
13.6%	58.4%	3.35	14.0%	14.0%	73.1%	3676.24
20.4%	65.2%	3.05	7.7%	21.7%	80.8%	3316.86
6.4%	71.6%	1.58	4.1%	4.1%	84.9%	1648.07
9.7%	75.0%	1.40	3.0%	7.2%	87.9%	1438.51
	100%	0.68	12.1%		100.0%	594.67
			100%			

Table 7.15 Flotation Test 3 – Sulfur Speciation Distributions

S Distribution			S= Distribution		
%	Cum. %	Cum. % TS	%	Cum. %	Cum. % S=

86%	86%	24.95 24.95 24.95 24.95 24.95 24.95	88%	88%	24.62 24.62 24.62 24.62 24.62 24.62
14%	100%	10.60	12%	100%	10.14
100%			100%		

Table 7.16 Head Sample and Flotation Test 2 ICP Raw Data

np	Ag 338.289 nm ppm	Al 396.152 nm ppm	Ba 455.403 nm ppm	Ca 393.366 nm ppm	Co 230.786 nm ppm	Cr 267.716 nm ppm	Cu 213.598 nm ppm	Fe 238.204 nm ppm	K 766.491 nm	Li 670.783 nm	Mg 280.270 nm ppm	Mn 260.568 nm ppm	Na 589.592 nm ppm	Ni 231.604 nm ppm	Pb 220.353 nm ppm	Sr 421.552 nm ppm	Zn 213.857 nm ppm	Y 371.029 nm Ratio
	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1
	--	--	0.01	--	0.01	0.01	0.01	--	--	0.01	--	0.01	0.01	0.01	--	0.01	0.01	1.01
	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	1.01
	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1.02
	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	1.01
	100	100	100	####	100	100	100	100	100	####	100	100	100	100	100	100	100	0.96
	0.02	0.00 u	0	0.05	0	0	0.01	-0.03 u	0.16	0	-0.01 u	0	0.02	0	0.01	0	-0.01 u	1
	0.88 Q	0.93	0.96	1.02	0.99	0.99	1.04	1.33 Q	0.98	0.94	1.19 Q	1.03	0.93	0.99	0.99	0.97	1.11 Q	1.01
	0.01	-0.01 u	0	0.05	0	0	0.01	-0.03 u	0.11	0	-0.01 u	0	0.01	0	0.01	0	-0.01 u	1.01
	694.54	20803.43	99.4	129158.7	135.48	113.41	2117.32	88948.2	668.05	2.32	10090.09	4342.18	96.18	212.96	35359.37	32.25	215165	--
	1488.11	1312.91	9.37	11467.34	630.16	25.12	11464.4	133242.1	49.67	0.27	875.97	2147.56	19.16	1456.79	111518.6	5.33	249032	--
	1380.93	7393.03	42.07	63208.07	276.13	47.43	5759.08	75048.96	314.86	1.28	4666.97	4269.47	61.95	395.08	119765.7	18.46	456906	--
	1402.17	4161.41	25.27	36295.95	328.95	67.13	4857.01	68685.34	182.34	0.87	2566.57	5283.31	46.59	333.83	55373.82	12.36	534523	--
	1472.42	4112.21	26.99	36329.38	319.65	38.64	4694.6	67190.17	180.18	0.9	2556.59	5131.62	50.18	322.52	54512.35	13.02	534631	--
	931.76	4774.54	34.19	41432.57	275.92	42.26	3679.08	57787.2	268.41	1.69	3336.38	5132.26	79.82	245.07	37629.72	19.65	495563	--

	144.86	1399.59	9.75	13740.38	303.51	10.49	1692.37	37938.54	15.11	0.17	1247.28	6152.02	16.43	93.5	5104.05	4.41	627856	--
	554.28	16952.54	49.14	167269.7	95.06	135.55	2644.9	91324.95	447.46	2.36	8587.77	3974.04	69.51	153.87	21104.59	27.34	197503	--
	10.04	66247.24	2835.11	11382.57	4.24	32.75	575.24	20988.12	28733	38.83	2159.86	282.97	3723.48	24.64	1708.42	220.81	1200.3	--
	190.73	37382.68	122.7	235779.4	22.28	131.69	1573.07	112639.4	506.39	1.96	23959.69	5490.12	118.72	160.94	10931	52.1	29095.5	--
	2.85	412.08	6.22	261191	0.91	59.4	26.48	852.54	162.61	2.37	103873.4	87.41	12.86	47.6	36.8	110.5	146.34	--
	29.84	8304.59	28.61	9225.25	2291.25	68.54	114537	128168.5	2445.2	7.09	11213.55	254.94	34.69	101.04	175.96	2.97	220.51	--
	1.10 Q	1.16 Q	1.24 Q	1.69 Q	1.23 Q	1.26 Q	1.25 Q	1.16 Q	0.97	1.17 Q	1.50 Q	1.25 Q	1.22 Q	1.26 Q	1.26 Q	1.24 Q	1.43 Q	0.98

Table 7.17 Flotation Test 2 ICP Raw Data

mp	Ag 338.289 nm ppm	Al 396.152 nm ppm	Ba 455.403 nm ppm	Ca 393.366 nm ppm	Co 230.786 nm ppm	Cr 267.716 nm ppm	Cu 213.598 nm ppm	Fe 238.204 nm ppm	K 766.491 nm ppm	Li 670.783 nm ppm	Mg 280.270 nm ppm	Mn 260.568 nm ppm	Na 589.592 nm ppm	Ni 231.604 nm ppm	Pb 220.353 nm ppm	Sr 421.552 nm ppm	Zn 213.857 nm ppm	Y 371.029 nm Ratio
5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1
5	--	0.01	0.01	--	0.01	0.01	0.01	0.01	0.01	0.01	--	0.01	0.01	0.01	--	0.01	0.01	1.01
5	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	1.02
5	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1.01
5	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	1.01
5	100	100	100	####	100	100	100	100	100	####	100	100	100	100	100	100	100	0.95
5	-0.01 u	0.03	0.01	-0.07 u	0.01	0.01	0.02	0.05	0.25	0.01	0.02	0.01	0.01	0.01	-0.01 u	0.01	0.00 u	1
5	1.01	1.06	1.11 Q	1.07	1.14 Q	1.11 Q	1.15 Q	1.12 Q	1.17 Q	1.02	1.25 Q	1.16 Q	1.09	1.13 Q	1.11 Q	1.14 Q	1.23 Q	1.01
5	-0.01 u	0.03	0.01	-0.07 u	0.01	0.01	0.02	0.05	0.19	0.01	0.02	0.01	0.01	0.01	0.00 u	0.01	0	1.01
5	1622.97	1043.12	24.96	6450.04	370.58	27.21	9263.93	55343.5	87.19	0.19	631.94	3946.31	34.91	627.34	198450.8	5.22	440192.7	--
5	1819.14	2290.14	33.67	13031.9	447.28	40.02	8376.23	66527.7	135.77	0.36	1221.53	5197.58	46.75	612.89	148403	7.34	523733.4	--
5	1463.59	1867.27	28.66	11062.84	367.57	33.16	6980.03	59894.25	105.49	0.26	1010.26	4282.39	39.93	501.6	130570.1	6.07	466486.1	--
5	1345	5775.73	83.1	31213.78	368.59	81.82	6714.18	67265.02	341.39	1.25	3317.76	5262.08	127.27	456.44	64196.82	19.09	463986.8	--
5	793.15	10935.36	123.08	56507.44	314.07	36.18	4639.85	83257.68	506.04	1.6	5833.11	5167.8	160.32	399.41	39388.95	24.69	385524.5	--
5	-451.61u	307.12	4.68	166438.7	0.74	45.62	1735.13	670.16	88.97	1.78	26018.21	69.42	8.49	29.84	5.79	93.34	90.64	--

5	747.29	11459.33	131.76	66700.26	278.95	126.45	3817.74	92419.23	539.18	1.76	6534.96	4131.76	166.91	487.1	38087.24	28.02	305178	--
5	690.38	14214.06	160.74	77039.01	314.55	46.77	3650.98	111776.1	696.81	2.33	7506.08	4378.78	224.35	528.04	35689.71	35.12	292419.7	--
5	3.96	61071.47	2854.85	9221.78	4.47	32.3	576.24	20861.34	30432.2	37.74	1937.43	284.07	3788.86	16.97	1657.53	225.73	828.21	--
5	116.04	3755.31	34.76	23514.47	335.9	51.52	1880.07	46037.42	153.84	0.38	2239.28	6640	44.51	130.26	4877.78	6.81	578277.1	--
5	287	15061.45	123.75	91868.59	119.01	46.69	2304.03	73007.84	551.35	2.27	7524.45	3931.93	176.46	121.87	13100.13	28.81	240177.4	--
5	93.93	34107.85	260.51	166518.3	28.91	60.85	893.35	106108.9	1150.31	3.19	11864.42	4193.27	250.74	93.95	5825.27	44.9	16219.89	--
5	1	1.12 Q	1.15 Q	1.1	1.16 Q	1.13 Q	1.18 Q	1.17 Q	1.01	1.02	1.24 Q	1.16 Q	1.14 Q	1.14 Q	1.13 Q	1.18 Q	1.29 Q	0.98

Table 7.18 Flotation Test 3 ICP Raw Data

Solution Label	Timestamp	Ag 338.289 nm ppm	Al 396.152 nm ppm	Ba 455.403 nm ppm	Ca 393.366 nm ppm	Co 230.786 nm ppm	Cr 267.716 nm ppm	Cu 213.598 nm ppm	Fe 238.204 nm ppm	K 766.491 nm ppm	Li 670.783 nm ppm	Mg 280.270 nm ppm	Mn 260.568 nm ppm	Na 589.592 nm ppm	Ni 231.604 nm ppm	Pb 220.353 nm ppm	Sr 421.552 nm ppm	Zn 213.857 nm ppm	Y 371.029 nm Ratio
Blank	4/18/2025 16:34	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1
Standard 1 - ICM-103	4/18/2025 16:35	0.01	--	0.01	0.01	0.01	0.01	0.01	--	--	0.01	0.01	0.01	0.01	--	0.01	0.01	0.01	1.01
Standard 2 - ICM-103	4/18/2025 16:37	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	1.01
Standard 3 - ICM-103/CCV	4/18/2025 16:38	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1.02
Standard 4 - ICM-103	4/18/2025 16:39	10	10	10	--	10	10	10	10	10	10	10	10	10	10	10	10	10	1.02
Standard 5 - ICM-103	4/18/2025 16:41	100	100	100	####	100	100	100	100	100	####	100	100	100	100	100	100	100	0.95
Wash Blank	4/18/2025 16:42	0.01	0.02	0.01	0.01	0.01	0.01	0.01	0.02	0.28	0.01	0.01	0.01	0.01	0.00 u	0.01	0.01	-0.01 u	1.01
ICM-103 CCV	4/18/2025 16:43	0.99	1.02	1	1.01	1.02	1.06	1.03	1.11 Q	1.07	0.98	1.21 Q	1.03	0.95	1.04	1.04	1.06	1.03	1.01
Wash Blank	4/18/2025 16:45	0.01	0.01	0	0.01	0	0	0.01	0.02	0.22	0	0	0	0.01	0.00 u	0	0	-0.01 u	1
1096 GS F4 Tails 60 min	4/18/2025 16:49	85.31	25656.33	81.52	55656.79	37.74	34.05	616.2	101361	762.32	2.79	9663.59	3079.38	78.15	95.28	3691.46	32.1	10409.52	--
1096 GS F4 Pb 1 Con 60 min	4/18/2025 16:55	1441.38	3103.77	36.47	7323.47	193.87	34.15	16973.41	49194.97	230.93	1.06	1963.56	2470.92	57.24	539.99	249763.3	12.95	258659.1	--
1096 GS F4 Pb 2 Con 60 min	4/18/2025 17:00	1331.4	4411.75	40.8	9927.49	244.87	40.21	7533.14	56305.96	228.12	0.97	2714.33	2738.95	68.65	546.58	237560.5	13.76	257991.1	--
1096 GS F4 Cu 1 Con 60 min	4/18/2025 17:05	1441.83	8045.07	68.49	16781.21	292.84	30.19	5698.3	80666.38	436.25	1.44	4655.13	3311.82	111.23	526.84	94484.2	24.71	262152.1	--
1096 GS F4 Cu 2 Con 60 min	4/18/2025 17:10	1463.42	8501.13	57.39	17801.96	336.88	25.96	3562.21	106208.7	379.38	1.27	4522.83	2995.96	87.35	625.81	112279.9	20.5	236915.1	--
Blank	4/18/2025 17:13	-56.79u	302.28	4.19	62974.31	-20.95u	4.6	2302.71	586.05	89.39	1.88	25445.48	60.04	6.92	1.13	33.17	87.06	135.9	--
STD 166	4/18/2025 17:18	116.64	1516.46	16.01	4125.21	259.17	5.34	1396.58	35180.98	96.47	0.21	1097.82	5380.96	25.68	70.58	4147.6	5.93	468545.9	--
1096 GS F4 Zn 1 Con 60 min	4/18/2025 17:23	106.9	1613.56	16.56	4007.01	274.97	4.92	1491.82	36324.59	69.15	0.21	1168.55	5648.75	24.72	74.99	4395.36	5.61	472154.5	--
1096 GS F4 Zn 1 Con 60 min - dup	4/18/2025 17:28	256.15	4569.5	15.32	10551.04	252.71	12.56	2119.77	47367.98	116.93	0.51	2766.82	4634.94	23.27	113.01	10782.4	6.67	417047.2	--
1096 GS F4 Zn 2 Con 60 min	4/18/2025 17:32	6.96	61442.17	2358.83	2997.87	3.3	14.91	496.37	19364.12	25893.6	32.99	1706.82	243.87	3026.5	5.79	1515.1	188.9	1738.65	--
STD 607b	4/18/2025 17:35	8.87	6737.74	22.43	2154.27	1806.1	51.84	93747.66	114330.7	1982.52	5.54	8358.62	196.66	25.53	67.86	145.35	2.45	166.56	--
ICM-103 CCV	4/18/2025 17:37	0.98	1.05	1.02	1.01	1.03	1.06	1.05	1.15 Q	1	0.97	1.20 Q	1.02	0.98	1.04	1.04	1.07	1.08	1.01

11. Market Study (U.S. Focus)

This section outlines the market positioning of the Darwin Mine Project with a focus on strategic and critical mineral contributions to the United States. It supports justification under the Defense Production Act (DPA) Title III for government engagement and national supply chain reinforcement.

Strategic Market Context

The Darwin Mine is poised to play a significant role in the domestic supply of critical minerals essential to U.S. economic security, clean energy transition, and defense manufacturing. Current global instability and reliance on foreign-controlled supply chains underscore the importance of domestic production capacity. This market study quantifies Darwin Mine's contribution to U.S. mineral demand across key categories prioritized by the Department of Energy, Department of Defense, and U.S. Geological Survey.

Germanium Supply Impact

Germanium is a vital semiconductor used in infrared optics, fiber optics, and military applications. The United States imports nearly all of its germanium supply. The Darwin Mine is projected to produce 20.35 tons annually—equivalent to 40.7% of current U.S. demand—qualifying it as a Tier 1 strategic asset for supply chain resilience.

Zinc Supply Chain Diversification

Zinc, a base and specialty metal, is produced at Darwin in multiple market-ready forms:
- Zinc Sulfate Monohydrate Powder: Agricultural input contributing 26.7% of the U.S. market
- Liquid Zinc Sulfate: Fertilizer-grade supply, representing 3.08% of U.S. market value
- Zinc in Sulfide Concentrate: Smelter feed, with production of 7,344.68 tons, contributing approximately 0.56% to the domestic zinc smelting supply chain.

Tungsten Market Position and National Security Role

Tungsten is a critical mineral designated by the U.S. government due to its use in armor-piercing munitions, high-temperature alloys, aerospace components, and electronics. Darwin Mine is projected to produce approximately 1,316,250 pounds (or 658.13 short tons) of tungsten annually. This volume represents approximately 14.3% of total U.S. demand.

Importantly, the tungsten recovered from Darwin can be converted to ammonium paratungstate (APT), the intermediate compound required for all downstream tungsten processing. The U.S. currently has no active primary APT production and relies heavily on imports from China. Darwin's tungsten production offers a unique opportunity to reestablish APT production domestically and reduce vulnerability to geopolitical supply disruptions.

Additional Critical Mineral Contributions

Darwin Mine’s output further enhances domestic access to tellurium, indium, gallium, cobalt, silver, copper, and lead. Notably, the project supplies:

- 48.1 tons of Tellurium (48.1% of U.S. demand)
- 15.93 tons of Indium (39.8%)
- 1.3 tons of Gallium (6.5%)

These minerals are critical to solar technologies, electronics, EVs, and aerospace defense components.

Competitive Landscape Analysis: Darwin Mine vs U.S. Mineral Supply Competitors

The Darwin Mine is uniquely positioned as a multi-commodity critical mineral hub with integrated infrastructure, ESG alignment, and the ability to serve both strategic defense and domestic agriculture markets. With a combination of on-site processing, hybrid domestic smelter partnerships, and a vertically integrated model, Darwin is poised to outpace its peers across three critical fronts:

- Critical Minerals (Germanium, Gallium, Indium, Tellurium, Cobalt)
- Agricultural Zinc Sulfate Monohydrate ($ZnSO_4 \cdot H_2O$)
- Ammonium Paratungstate (APT) from tungsten recovery

Competitive Summary Matrix

Darwin offers on-site or domestic processing access to five DPA-listed critical minerals. While some germanium and indium in Darwin's sulfide ores will be sent to domestic smelters for final recovery, this hybrid model reduces capex while maintaining full traceability and compliance with DPA sourcing rules.

In the agricultural zinc sulfate space, Darwin is the only U.S. mine converting zinc oxide ore into $ZnSO_4$ on-site, creating a vertically integrated supply for agricultural markets. Other suppliers like Nyrstar operate at the smelter level but do not control the mine-to-product supply chain.

For APT, Darwin will reintroduce U.S.-based production using legacy tungsten tailings. Pine Creek Mine, once the nation's top tungsten source, is now permanently closed due to hazardous radon contamination, eliminating it as a viable source.

The chart below visually represents how Darwin outpaces all current and former U.S. producers across critical minerals, APT, and $ZnSO_4$ output capabilities.

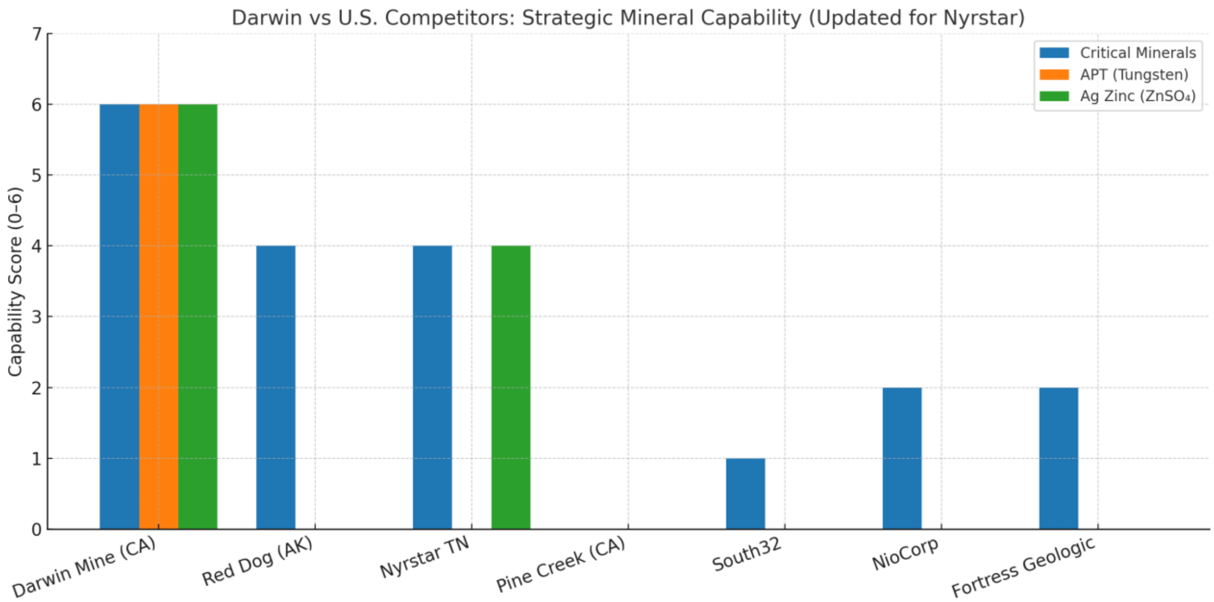
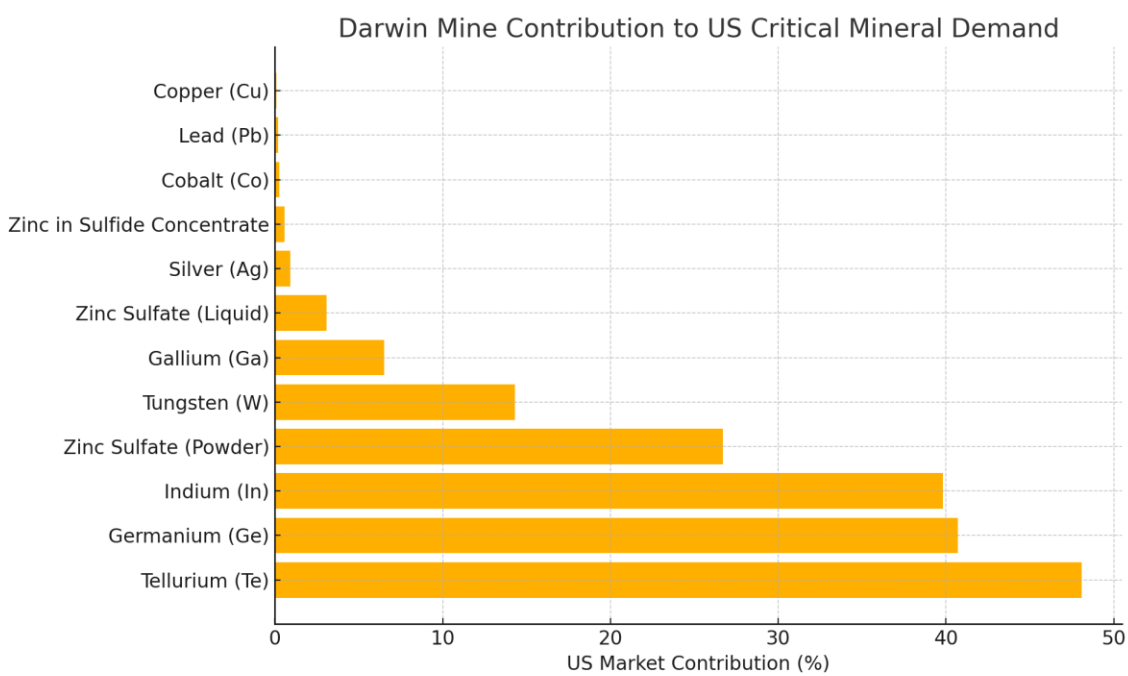


Figure: Comparative capability score (0–6) across Darwin and U.S. mining competitors, updated for Nyrstar zinc sulfate and critical mineral relevance.

Visual Market Summary

The following chart summarizes Darwin Mine's potential percentage contribution to U.S. demand across all applicable mineral categories.



Conclusion & National Security Relevance

The Darwin Mine Project qualifies as a nationally significant resource base capable of reshoring supply for several DPA Title III priority minerals. Its scale and diversification across metals make it a critical infrastructure asset. Government support under DPA or other federal programs will amplify its near-term activation potential and secure long-term supply chain reliability for essential U.S. industries.

Financial Analysis

The Darwin Mine Project demonstrates strong economic potential based on market conditions as of May 2025, historical development, and verified geological continuity. Financial projections supported by detailed modeling show a Net Present Value (NPV) of $2.76 billion over a 10-year period, with a site-wide operating cost of just $189.50 per ton—well below the U.S. underground mining average.

High-grade mineralization, minimal pre-stripping requirements, and access to existing infrastructure contribute to an exceptional cost structure. The project is designed to supply multiple critical and base metals—zinc, silver, lead, germanium, tellurium, indium, and gallium—with conventional selective mining and on-site processing facilities planned for scale-up.

This assessment outlines the current feasibility components and additional recommendations required to reach a fully bankable level for institutional investors, strategic buyers, and government stakeholders.

Capital & Operating Cost Estimates

The total initial capital expenditure (CAPEX) for Phase 1 is estimated at $51 million, with annual 20% of net revenue retained for CAPEX and OPEX. Operating costs average $189.50 per ton, based on detailed line-item analysis. Mining costs alone total $46.79 per ton, broken down as follows:

- Labor Underground: $28.00/ton
- Powder & Steel: $6.00/ton
- Ventilation: $3.00/ton
- Development Drilling: $3.29/ton
- Safety Labor: $1.00/ton
- Crushing: $3.25/ton
- Maintenance: $1.25/ton
- Consumables: $0.50/ton
- Transport Fine Ore: $0.50/ton

Additional processing reagents/acids, utilities, G&A, and contingency costs bring the all-in operating cost to $189.50/ton.

These costs reflect the use of conventional selective mining methods targeting high-grade zones (Zn 8–12%, Ge ~124 ppm), resulting in a more efficient and lower-cost operation compared to industry averages reliant on large-scale bulk mining.

Discounted Cash Flow (DCF) Model

The DCF model uses an 8% discount rate and a 10-year forecast horizon. Net Present Value (NPV) is $2.76 billion. Cash flows incorporate realistic CAPEX/OPEX assumptions, critical mineral pricing (May 2025), and conservative recovery factors. The project achieves strong positive free cash flow from Year 1 due to low initial capital requirements and robust early sulfide throughput.

Revenue Model by Product

Revenue is generated from saleable zinc, silver, lead, zinc sulfate monohydrate, germanium, tellurium, indium, and gallium. Pricing is based on May 2025 indexes:

- Zinc: $1.25/lb
- Zinc Sulfate Monohydrate ($ZnSO_4 \cdot H_2O$): $1,400/ton (contains ~35% zinc by weight in dry form; liquid formulations typically contain ~12% zinc)
- Silver: $34/oz
- Lead: $0.95/lb
- Germanium: $1,500/kg-$2,500/kg depending on purity
- Tellurium: $100/kg
- Indium: $250/kg
- Gallium: $300/kg

Annual revenue is calculated by multiplying expected tonnage and recovery by average grade and market price for each product. Concentrates are assumed to be shipped to domestic and international processors under tolling or off-take contracts.

Financial Sensitivity Analysis

Sensitivity analysis modeled ±20% shifts in commodity pricing. Even at -20% revenue levels, the project remains positive NPV with strong IRR, showcasing economic resilience. CAPEX overrun scenarios of +10% and +20% were also tested with tolerable impacts on return metrics.

Funding Strategy

The project will be funded through a combination of:

- Private equity capital raise
- Strategic government grants or investment under DPA Title III
- Optional commercial debt at later phases

The modular design and phased CAPEX reduce upfront investment risk. Operating cash flow from sulfide circuit supports expansion of oxide circuit without large financing rounds.

Cost & Grade Benchmarking

Darwin's estimated cost of $189.50/ton is significantly below U.S. underground mining averages ($250–$400/ton). This is due to:

- Selective extraction of high-grade material
- Existing infrastructure reducing CAPEX
- In-house lab and processing flow sheet design

Grade comparison highlights Darwin’s advantage:

- Zinc: 8–12% vs. industry avg. ~5–7%
- Silver: 4–8 oz/t vs. industry avg. ~2–4 oz/t
- Germanium: ~124 ppm vs. typical <50 ppm in polymetallic ores

This positioning enhances economic competitiveness and DPA relevance.

Component	Status	Notes
Capital & Operating Cost Estimates	**Complete**	Includes detailed cost breakdowns by category (mining, processing, labor, etc.) from proforma financial model
Discounted Cash Flow (DCF) Model	**Complete**	Based on 8% discount rate, 10-year horizon, reflects updated production profile and pricing assumptions
Revenue Model by Product	**Complete**	Covers base and critical minerals (Zn, Ag, Pb, Ge, Te, In, Ga); market prices based on May 2025 indexes
Financial Sensitivity Analysis	**Complete**	NPV stress test includes ±20% revenue swings, confirming positive returns under conservative scenarios
Funding Strategy	**Complete**	Blend of equity, DPA Title III support, and optional debt capital, based on modular CapEx ramp-up
Cost & Grade Benchmarking	**Complete**	Darwin compares favorably to peers with lower OPEX/ton and higher-grade resource (see Competitive Grade Comparison)
Year-by-Year Production Forecast	**Complete**	Derived from proforma; oxide and sulfide circuits detailed separately and combined below

---

Pending or Recommended for Full Bankability

Requirement	Status	Action Needed
NI 43-101 or SK-1300 Technical Report	**Partial**	Ethos Geological is preparing technical reports; additional sampling required for critical minerals not covered in historical records
Resource Classification (P&P)	**In Development**	Proved and Probable delineation to follow completion of 43-101 with critical mineral verification
Environmental & Permitting Summary	**Complete**	All necessary environmental and permitting requirements have been addressed, including air quality compliance, bonding strategies, and local land use approvals
Offtake Agreements or LOIs	**Partial**	Umicore has submitted a formal letter of support for germanium supply from Darwin. Additional LOIs from FLIR, Teledyne, or agricultural buyers will further support project de-risking.
Appendices and Supporting Docs	**In Progress**	Add technical appendices: proforma spreadsheet, assay lab results, sampling maps, circuit diagrams

---

Technical Clarification

Current resource estimates are supported by historical production and ongoing geological mapping. However, **critical minerals such as germanium, tellurium, indium, and gallium require targeted sampling** within remaining pillars and oxide zones due to their exclusion from past mine planning efforts. Channel sampling, hand-cobbing, and lab assays are ongoing to validate grades at commercial scale.

Ethos Geological and Don Strachan World Industrial Minerals are leading the technical reporting effort, aligned with 43-101 standards.

Conclusion

The Darwin Mine project presents a compelling financial and technical case at the prefeasibility stage. The project's low-cost structure, domestic strategic mineral profile, and modular production expansion make it a strong candidate for near-term investment and public-private partnership under Title III of the Defense Production Act. With final validation of critical mineral content and a formalized permitting plan, Darwin will meet or exceed the threshold for bankable project financing.

12. Risk analysis and mitigation

The Darwin Mine Project, operated under INYOAG LLC and supported by Stone Brothers and a range of subcontractors, aims to extract and process base and critical minerals including zinc, silver, lead, copper, tungsten, germanium, tellurium, indium, and gallium. Given its strategic mineral profile and historical production, the project presents strong upside potential. However, several categories of risk must be managed to ensure technical and economic viability.

Geological Risk

- Risk: Uncertainty in resource continuity, grade variability, or undiscovered fault zones.
- **Mitigation:**
  - Ongoing drill campaigns with updated block models.
  - Validation from third-party geologists (Ethos, Don Strachan).
  - Historical data leveraged from Anaconda operations pre-1976.

Operational Risk

- Risk: Equipment failure, labor shortages, underground hazards.
- **Mitigation:**
  - Redundant ventilation and haulage plans.
  - Use of proven mining methods (jacklegs, muckers, rail).
  - Certified MSHA workforce and contracted underground development (Ballistic).
  - Scheduled preventative maintenance with Nick Stone overseeing safety and operations.

Metallurgical Risk

- Risk: Variability in recoveries for complex ores (e.g., oxide vs sulfide zinc, germanium loss in leach).
- **Mitigation:**
  - Separate oxide and sulfide circuits.
  - Lab validation in-house + third-party labs (ALS, Kappes & Cassidy).
  - Process design supported by AVI Metals and Woods Process Services.
  - Emphasis on selective leaching and flotation to minimize critical metal loss.

Market Risk

- Risk: Fluctuations in global prices of critical minerals and base metals.
- **Mitigation:**
  - Multi-metal production allows diversification.
  - Targeting domestic markets (zinc sulfate, APT tungsten, rare metals).
  - Aligned with U.S. Defense Production Act demand (DPA Title III strategic mineral support).

Legal & Regulatory Risk

- Risk: Possible changes in regulation
- **Mitigation:**
    - Fully permitted, Water rights secured and right of ways
    - Legal support from Fabian Van Cott (opportunity zone expertise).
    - Local district liaison through public works and Inyo County officials.
    - Conditional Use Permits and CEQA compliance reviewed and approved.

Environmental & Social Risk

- Risk: Water use, tailings, community opposition, reclamation.
- **Mitigation:**
    - Closed-loop water system designs and cyanide-free recovery.
    - Permitted disposal sites and lined leach pads.
    - Historical mining legacy used to justify reactivation under modern standards.
    - Transparent engagement with local stakeholders.

Infrastructure & Logistics Risk

- Risk: Inadequate power, road access, or site preparation delays.
- **Mitigation:**
    - Electric infrastructure with Aldrich Electric.
    - Concrete and access road development by Clare Concrete.
    - Local infrastructure already partially in place from historic operations.

Financial Risk

- Risk: Capital overruns, lack of investor support, or delayed revenue.
- **Mitigation:**
    - Low capex startup using refurbished infrastructure.
    - Third-party validation of financial model.
    - DPA Title III and other federal funding channels being pursued.
    - Private equity markets pursued
    - Opportunity Zone Fund set up.

Critical Mineral Estimates Risk Assessment

Our critical mineral projections are primarily derived from sole channel samples, which provide localized but detailed insights into the mineralization. Historical production records from sphalerite, chalcopyrite, and bornite deposits indicate that germanium, tellurium, indium, and gallium are often found as trace elements within these ore bodies. These elements typically occur as substituents in the crystal lattice of primary sulfide minerals or as discrete mineral phases. The distribution and concentration of these critical minerals are inherently variable, leading to significant uncertainty in resource estimates.

Additionally, the presence of oxide material, such as hydrozincite, also influences the occurrence of these critical minerals. In oxidized zones, germanium and other critical elements can be found associated with secondary minerals, making their recovery and economic viability more complex.

Historical production focused heavily on silver and lead, leaving behind massive pillars of sphalerite and hydrozincite zones. The previous operations avoided oxidized zones and produced an antimonial lead, indicating potential antimony production. Our channel samples have been verified through lab results, but it's crucial to note that existing workings with remaining pillars and backfilled high-grade bornite, chalcopyrite, and sphalerite mineralization must be sampled in a hands-on manner. It remains uncertain whether the NI 43-101 standard supports this approach, as it heavily relies on in-depth hand sampling and assaying.

Mitigation

To address these uncertainties, we will implement a comprehensive program of further channel sampling, hand cobbing, and laboratory testing for these critical elements. This hands-on approach aims to improve the accuracy of our resource estimates and reduce the associated risks.

Risk Matrix

Risk Category	Likelihood	Impact	Rating	Mitigation Strategy
Geological	Medium	High	High	Continued drilling, 43-101 compliance
Metallurgical	Medium	High	High	Dual circuits, expert process support
Operational	Low	Medium	Medium	Experienced team, MSHA compliance
Market	Low	Medium-Broad minerals base	High	Target strategic markets, diversify products
Legal/Permitting	Low	Medium	Medium	Fully permitted, Vesting the mine for historical importance
Environmental/Social	Low	Medium	Medium	No wastewater, To tailings dam
Infrastructure	Low	Medium	Medium	Contractors already engaged; infrastructure mapped
Financial	Medium	High	High	DPA submission, phased development
Critical Minerals	Medium	High	High	channel sampling, hand cobbing, and laboratory testing for these critical elements

Risk Conclusion

The Darwin Mine Project holds strategic importance for U.S. critical mineral independence and economic security. The project has identifiable risks typical of underground mining and hydrometallurgical processing, but each is matched with specific, actionable mitigation strategies. With multiple built-in redundancies and validation partners, this risk assessment supports the feasibility and resilience of the project.

13. Conclusion

The Darwin Mine Project represents a singular opportunity to reestablish a historically significant mining district as a modern, vertically integrated source of critical and base metals. The findings of this pre-feasibility study confirm the project's technical and economic viability, underpinned by a robust resource base, well-understood geology, existing infrastructure, and alignment with both national security priorities and sustainable development goals.

With a proven foundation of historical production and extensive underground workings, Darwin offers both a low-risk near-term production scenario and substantial upside for expansion. The project's oxide and sulfide resources are supported by detailed mineralogical and flotation studies, with Phase 1 production (2026–2030) delivering early cash flow through a 500 TPD operation. The planned expansion to 2,000 TPD in Phase 2 (2031–2035) positions the project for peak critical mineral recovery and revenue generation, with projected total revenues exceeding $2.8 billion and an internal rate of return (IRR) of over 113%.

Importantly, Darwin stands out as one of the few privately controlled, fully permitted mining projects capable of producing a suite of strategic elements—germanium, tellurium, indium, gallium, cobalt, and tungsten—at a scale relevant to U.S. supply chain resilience efforts. This strategic positioning, combined with the project's location within a federally designated Opportunity Zone, makes Darwin a compelling candidate for DPA Title III investment and long-term government contracting.

From a technical standpoint, the geology of the deposit is well understood, and the mineralization remains open in multiple directions. Historical production data, in conjunction with modern sampling, supports both the indicated and inferred resource models. The combination of traditional base metals and high-value byproducts—particularly oxide zinc for agricultural use—provides a diversified product portfolio that mitigates commodity risk and enhances revenue stability.

Environmentally, the project employs best-in-class practices including tailings backfill, emissions controls, and carbon reduction strategies. Socially, the mine’s development will bring substantial employment to a region historically tied to mining, while fostering local infrastructure growth and skill development. Governance is structured around qualified third-party reporting, regulatory compliance, and transparent stakeholder engagement.

In conclusion, the Darwin Mine Project is not only economically sound but strategically essential. It has the potential to become a cornerstone of domestic critical mineral production while simultaneously supporting defense, clean energy, and food security objectives. With its combination of low technical risk, high strategic value, and strong financial returns, the project is well-positioned for immediate advancement into development and production phases.

Independent Technical Review Statement

This Prefeasibility Study (PFS) for the Darwin Mine Project has been reviewed by independent qualified professionals following accepted engineering and economic standards, including NI 43-101 and SEC SK-1300 guidance. The study provides a comprehensive and realistic evaluation of the technical and financial potential of the Darwin Mine, a historically productive site being redeveloped into a modern polymetallic and critical mineral operation.

Summary of Findings:

- **Geological and Metallurgical Integrity**
  The project's resource model is based on detailed historical data, modern sampling, and third-party verification. Grades include:
  - Zinc: **8–12%**
  - Germanium: **124 ppm**
  - Silver: **10.6 opt**
  - Tungsten: **0.22% WO_3**

  Metallurgical flowsheets are supported by flotation and leach test work, and include oxide/sulfide recovery, electrowinning, and zinc sulfate production. The technical approach is consistent with proven methods for similar deposit types.

- **Financial Model Realism**
  - **Initial Capital Expenditure (CapEx)**: $51 million
  - **Operating Expenditure (OpEx)**: $30 million/year for Years 1–5
  - **Operating cost per ton**: $189.50 (fully burdened)
  - **20% of net revenue is retained annually** to fund all sustaining CapEx and growth-related expenditures.

  These assumptions are consistent with comparable underground polymetallic projects. The financial model projects an **NPV of $2.7 billion** over a 10-year life, with internal rates of return in excess of typical investment thresholds, assuming current commodity prices and confirmed grades.

- **Critical Mineral Potential**
  Germanium is included in the financial model. Indium, gallium, and tellurium are present in quantifiable but conservative estimates, pending further metallurgical validation. These elements, all classified as critical by U.S. agencies, support future revenue expansion and off-take potential.

- **Phased Production Feasibility**
  The production scale—from 500 TPD oxide feed in Year 1 to 2,000 TPD combined oxide/sulfide by Year 6—is feasible given the site's existing underground infrastructure

and development strategy. Partnerships with qualified contractors and existing processing equipment further enhance execution capability.

- **Permitting, ESG, and Risk Mitigation**
  The project is fully permitted under a valid Conditional Use Permit (CUP), with secured water rights and power access. SMARA exemption and environmental backfill practices eliminate the need for tailings impoundments. The operation is located within a federally designated Opportunity Zone and includes workforce development, safety benchmarking, and long-term reclamation planning.

Conclusion:

The Darwin Mine Project, as detailed in this Prefeasibility Study, presents a technically credible and economically compelling opportunity. The financial model incorporates realistic costs, conservative recovery assumptions, and appropriate strategic flexibility. With an initial capital requirement of $51 million and an OpEx structure supported by robust projected margins, the project is well-positioned for advancement to a Feasibility Study and commercial execution.

References

- ASARCO Annual Reports (1942–1965)
- USBM Circular 8303 – Summary of Western U.S. Custom Smelter Operations
- USGS Bulletin 1027-F (1956) – Mineral Resources of Inyo County, California
- USGS Open File Report 75-454 – Geologic Map of the Darwin Quadrangle, Inyo County, California
- USBM Report of Investigations RI 5476 (1959) – Exploration of Polymetallic Deposits in the Darwin District
- Strategic and Critical Materials Reports to Congress (1951, 1954, 1958)
- Schwarz-Schampera, U. & Herzig, P.M. (2002). *Germanium: Geochemistry and Mineralogy*.
- Tooele County Historical Society Archives
- Defense Minerals Exploration Administration (DMEA) Project Files
- War Production Board – Defense Plant Corporation Files
- Department of Energy Critical Materials Strategy (2021 Update)
- U.S. Geological Survey Mineral Commodity Summaries (Multiple Years)
- Department of Defense Strategic and Critical Materials 100-Day Review (2021)
- Internal Project Darwin LLC Geological Database (1996–2024)
- Woods Process Services Internal Metallurgical Assessment (2022) – *Floatation Performance and Trace Element Evaluation, Darwin Tailings*
- Ethos Geological Consulting (2021–2023) – *Technical Reports, Mapping, and Core Re-Logging Logs (NI 43-101 Format)*
- Hahn, Peter (2014) – *Darwin Mine Mineral System Review: Structural, Lithological, and Critical Mineral Implications*
- Hahn, Peter; Ethos Technical Memo Series (2021–2023)
- Kappes, Cassiday & Associates – Internal Lab Reports and Ore Characterization Summaries (2021–2024)
- Skyline Assayers & ALS Laboratories – Third-Party Elemental Analysis Certificates (2022–2024)
- RubinBrown LLP – Tax Records Review for Darwin District Operations (2023)
- RDI Mineral Processing Engineering Report (2023) – *Bench-Scale and Pilot Circuit Design Evaluation for Darwin Critical Minerals*
- AVI Metals Engineering (2024–2025) – *Electrowinning Design, Drying Circuit Optimization, and Site Infrastructure Recommendations*
- Barber, G. A., 1977 – Internal Anaconda memo to R. L. Knight, 6/29/77
- Brown, H. G., 2002 – *Update, Darwin Mine, Inyo County, California: An Excellent Zinc Exploration Opportunity*, 5/10/02
- Brown, H. G. (?), 2002 – *Darwin Mine, Inyo County, California*, 7/23/02 (likely prepared for Project Darwin Inc.)
- Chen, J. H.—Y., 1977 – *Uranium-lead isotopic ages from the southern Sierra Nevada batholith, and adjacent areas, California*, PhD thesis, Univ. Calif. Santa Barbara
- Cormie, J. M., 1970 – *Darwin Mine, Inyo County, California: Geology, Ore Reserves, Proposed Development*, report for Mexicanus Explorations Ltd., 11/11/70
- Czamanske, G. K. & Hall, W. E., 1975 – *The Ag-Bi-Pb-Sb-S-Se-Te mineralogy of the Darwin lead-silver-zinc deposit, southern California*, Econ. Geol., v. 70, p. 1092–1110
- DeRuyter, V., 1986–1988 – Series of reports for Quintana on bulk-minable silver-gold targets, Radio ore and Lucky Jim sampling, and drill results (8/86, 9/87, 11/88)
- Dunne, G. C., Gulliver, R. M., & Sylvester, A. G., 1978 – *Mesozoic evolution of rocks of the White, Inyo and Slate Ranges, eastern California*, in SEPM Mesozoic paleogeography volume

- Hahn, P., 1991–1992 – Monthly reports and 1991 exploration summary for Western Zinc JV/Cyprus Minerals
- Hall, W. E., Mackevett, E. M., & Davis, D. L., 1958 – *Economic geology of the Darwin Quadrangle, Inyo County, California*, Calif. Div. of Mines Spec. Report 51
- Hall, W. E. & Mackevett, E. M., 1962 – *Geology and ore deposits of the Darwin Quadrangle, Inyo County, California*, USGS Prof. Paper 368
- Kildale, M. B., 1955? – *Lead-zinc deposits of the Darwin district, California*, Anaconda internal talk
- Musolf, K. R., 1971 – Memo to E. Rudd, Manager, Mexicanus Colorado Inc., 8/27/71
- Newberry, R., 1986 – *Use of intrusive and calc-silicate data to distinguish skarn types in the Darwin polymetallic district*, Mineralium, vol. 22, p. 207–215
- Newberry, R., Einaudi, M. T., & Eastman, H. S., 1991 – *Zoning and genesis of the Darwin Pb-Zn-Ag skarn deposit, California*, Econ. Geol., vol. 86, p. 960–982
- Rye, R. O., Hall, W. E., & Ohmoto, H., 1974 – *Isotopic study of the Darwin lead-silver-zinc deposit*, Econ. Geol., vol. 69, p. 468–481
- Saunders, F., 2007 – *Darwin Property, Inyo County, California: Review of data base*, report for Geneva Partners LLC, 6/25/07
- Seklemian, R., 1987 – *Data evaluation report, Darwin Mines property*, for Cyprus Minerals Co., 11/87
- Skiles, R., 1957 – *Darwin Mines, ore reserves as of June 1, 1957*, Anaconda report (cover letter dated 1/16/59)
- Stone, P., 1984 – *Stratigraphy and paleogeography of Pennsylvanian–Permian rocks, Owens Valley-Death Valley*, PhD thesis, Stanford Univ.
- Torrey, C. E., 1991 – *Memo and exploration proposal for Darwin Mine*, to W. D. Burton, Cyprus Minerals Co., 1/23/91
- Wetzel, R., 1989 – *Darwin Mine, Inyo County, California*, report for Blue Range Mining Co., 12/30/89
- Wilson, G. E., 1982–1983 – Anaconda reports on Darwin Project, annual and quarterly summaries, and general geology/ore deposits
- Kelley, V. C., 1937 – *Origin of the Darwin Silver-Lead Deposits*, Economic Geology, v. 32, no. 8, p. 987–1008
- Newberry, R. J., 1980 – *Progress report, Darwin Project*, June 10 (unpublished), 8 p.
- Newberry, R. J., 1981 – *Progress report, Darwin District*, July (unpublished), 8 p.
- Newberry, R. J., 1982a – *Progress report, Darwin District*, January (unpublished), 16 p.
- Newberry, R. J., 1982b – *Progress report, Darwin Mines*, Spring (unpublished), 10 p.
- Rhys, D. A., Lewis, P. D., & Rowland, J. V. (n.d.) – *Structural controls on ore localization in epithermal gold-silver deposits: A mineral systems approach*, Panterra Geoservices Inc. & Eldorado Gold Corporation