

Initial Assessment
SK 1300 Technical Report on the Longstreet Gold & Silver Project
Nye County, Nevada, USA

Prepared for:



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1.0 Executive Summary

1.1 Property Description and Location

The Longstreet Project (the Project, or Project) is located in northern Nye County, in the central part of the State of Nevada, USA. The Project consists of 142, unpatented mining claims covering approximately 2,500 acres, located on United States Forest Service (USFS) and United States Bureau of Land Management controlled land. The resource and primary exploration areas fall within land controlled by the USFS.

Star Gold has staked 26 claims (Leach Pad Claims) adjacent to the eastern boundary of the property, with the objective of providing the site for leach pads planned for future development of the Main Zone. These claims are included in the total of 142 claims making up the property holdings. These claims fall within land controlled by the BLM. In addition, Star Gold has staked 12 claims along a corridor leading from the main Longstreet property to the Leach Pad Claims.

Star Gold Corp. (Star) entered into an exclusive option agreement with MinQuest on the Longstreet Project in December, 2009. This option agreement gave Star the right to obtain a 100% interest in the Longstreet property, minus a 3% NSR royalty, by meeting the financial obligations outlined in the agreement.

Star Gold holds an option with MinQuest to purchase up to one-half of the 3% NSR, resulting in a carried 1.5% NSR, for a one time cash price of \$1.5 million USD. A separate agreement covering 5 unpatented claims (the Clifford agreement) has also been satisfactorily completed so that Star now owns 100% of those claims.

The Project is located approximately 170 miles northwest of Las Vegas and approximately 50 miles northeast of Tonopah, a town of approximately 2,500 people, in west-central Nevada. The northeast-southwest oriented property extends approximately 1.8 miles along strike within the Monitor Range. The geographic coordinates of the central part of the property are approximately 38°22'00"N Latitude and 116°40'00"W Longitude.

Access to the Longstreet Project area is by paved and gravel road. Access is by two-lane paved highway (Nevada HWY 6 to Warm Springs) approximately 30 miles east from Tonopah, and then by gravel road (Stone Cabin Road) approximately 25 miles north and a further 2.5 miles west along a trail to Windy Canyon to reach the Longstreet property. The total distance from Tonopah to the property is approximately 57 miles. Supplies and heavy equipment are brought to the site by trucks, or other four-wheel drive vehicles. Currently, there is no permanent camp at the site. Casual labor would likely be sourced from Tonopah, Reno, Elko or Ely, Nevada. Experienced

mining operations would likely be via contract and be sourced from Reno, Salt Lake City, Utah or other mining centers.

Production mining will be conducted by contract miners in order to reduce startup capital requirements. Star has already obtained a bid and proposal from N. A. Degerstrom of Elko, NV for contract mining and crushing. Other contract mining companies active in the region include Leducor, Turner Mining Group, and Small Mine Development (SMD).

1.2 Geology and Mineralization

The Longstreet Project is located in Nevada's Basin and Range Province, within the southern Monitor Range and adjacent to the Walker Lane volcanic-hosted gold trend. The regional geology reflects Paleozoic–Mesozoic compressional deformation followed by Cenozoic extension, magmatism, and widespread Oligocene–Miocene ash-flow volcanism. More than 90% of the exposed bedrock in the southern Monitor Range consists of Tertiary volcanic rocks, predominantly rhyolitic to dacitic ash-flow tuffs.

The property is principally underlain by a moderately to poorly welded rhyolitic ash-flow tuff unit, referred to as the Tat unit, which may be approximately 3,000 ft thick. This unit is intruded by rhyolite porphyry dikes and locally overlain by volcanoclastic and siliceous sedimentary rocks, including sinter, followed by a younger, strongly welded ash-flow tuff.

Gold-silver mineralization is interpreted as a low-sulfidation epithermal system comparable in host rock and mineralization style to the nearby Round Mountain deposit. Mineralization is associated with:

- Quartz-adularia veins and sheeted vein systems
- Silicification and dense quartz veinlets
- Argillic and potassic alteration
- Brecciation and limonitic fractures
- Steeply dipping structural zones

Three principal structural orientations have been identified: northwest, east, and west-northwest. An east-trending fault dipping approximately 40° to 55° is associated with the highest-grade mineralization identified to date. Most mineralization at the Main Zone occurs in multiple steeply dipping vein sets parallel or subparallel to this structure.

At least nine mineralized target areas have been recognized. The report considers targets associated with east-trending veins and structures, including Red Knob, Cyprus Ridge, and Opal Ridge, to be high-priority areas for future exploration.

1.3 Exploration and Drilling

Exploration has focused on evaluating epithermal gold-silver mineralization hosted by altered and brecciated rhyolitic tuffs. The work has included geological mapping, lithogeochemical rock-chip sampling, reverse-circulation drilling, and diamond drilling.

At least 107 lithogeochemical samples were collected. Surface sampling generally showed moderate to good correlation between gold and silver, with an approximate ratio of 1 Au to 10 Ag.

Important exploration targets include:

- Opal Ridge: Quartz-adularia vein systems and limonitic fractures occur beneath an opalized sinter cap. Historic drilling identified anomalous gold and silver over substantial intervals, and the sinter suggests that favorable mineralization may occur at depth.
- Northeast Main: An undrilled target containing steep sheeted veins and limonitic fractures. Rock-chip sampling returned values up to 0.93 ppm Au and 20.6 ppm Ag.
- North Zone: Mineralization extends over approximately 1,800 by 1,100 ft. Historic drilling and surface samples indicate potentially economic mineralization, including rock-chip values up to ppm g/t Au and 49.6 ppm Ag.
- Spire: An east-trending sheeted quartz-vein system with rock-chip values up to 0.45 ppm Au and 20.0 ppm Ag.
- Red Knob: A well-developed quartz-adularia vein system with reported values up to 4.1 ppm Au historically and 2.96 ppm Au from MinQuest sampling.
- Cyprus Ridge: A large sheeted-vein system extending more than 1.7 miles, with quartz veins locally up to 15 ft thick. Sampling returned gold values up to 11.2 ppm.
- North Slope and West Main: Structurally favorable vein systems that have received limited geological investigation and no drilling.

The report states that more than 430 RC holes totaling approximately 84,000 ft and 19 diamond holes totaling 4,295 ft have been completed by Star Gold and previous operators. Most drilling has focused on the Main Zone. Of the reported 437 RC and 19 diamond holes across six target areas, 414 intersected mineralization exceeding 0.2 g/t Au and 1.0 ppm Ag over intervals ranging from more than 10 ft to approximately 275 ft.

Star Gold completed systematic exploration from 2011 through 2014, including approximately 26,685 ft of RC drilling in 68 holes and 1,295 ft of diamond drilling in four holes. RC drilling and logging procedures were considered consistent with industry standards.

1.4 Data Verification

Data-verification records are unavailable for some of the earliest exploration programs conducted during the 1980s and 1990s. Nevertheless, the more recent Star Gold data were reviewed by MinQuest, the previous technical-report authors, and the current Qualified Person. The QP concludes that the data used for the mineral-resource estimate are adequate and reliable.

The verification work included the following:

- **Naneco twin drilling:** Ten historic holes were twinned during the 1984–1987 campaigns. Silver grades generally compared well, but gold grades showed poor correlation, possibly due to sampling issues or irregular gold distribution.
- **MinQuest QA/QC:** The program included blanks and two certified standards at a frequency of approximately one in 20 samples, B samples for unexpected results, duplicate assays for samples exceeding 1 g/t Au, and monitoring of standard results.
- **Duplicate assays:** Gold and silver duplicate results generally demonstrated good reproducibility, with only a few exceptions.
- **Standards and blanks:** Most blank results were below detection limits. Results for the two gold standards were generally close to their expected values and within acceptable standard-deviation limits.
- **ALS laboratory QA/QC:** ALS conducted regular check assays using multiple internal standards on RC-chip and diamond-core samples.
- **Star Gold twin drilling:** Four diamond holes were drilled near historic RC holes. Although the exact historic collar locations were uncertain, average grades from the diamond and RC holes were generally within the same order of magnitude.
- **Database verification:** Agnerian compared laboratory certificates with the assay database. Because recent ALS results were downloaded directly into MinQuest's digital logging system, the report concludes that the 2011–2013 database is free of data-entry and transfer errors. Historic assay certificates, however, were unavailable.
- **Independent sampling:** Agnerian collected 15 samples, including 13 diamond-core samples and two adit grab samples, and submitted them to SGS. The results generally compared well with Star Gold assays, although individual differences and a positive gold bias were observed. Average silver grades were similar: 14.4 ppm for Star Gold versus approximately 15.9 ppm for Agnerian.

The principal data limitations are the absence of historic assay certificates and incomplete documentation from early drilling programs. For future programs, the report recommends submitting approximately 10% of the original samples to an independent laboratory, conducting check assays on 5% of previous assays, and implementing a more systematic standards-and-blanks protocol for every sample batch.

1.5 Mineral Processing and Metallurgical Testing

Metallurgical testing was completed by Kappes, Cassiday & Associates (KCA) in 1984 and 1987 and by McClelland Laboratories in 2014. The programs evaluated cyanidation, heap-leach amenability, crushing requirements, reagent consumption, percolation, comminution characteristics, and gold-silver mineralogy. The Qualified Person considers the available test work adequate to support a Preliminary Economic Assessment-level process design.

1.5.1 Historical KCA Test Work

The 1984 test work compared oxide and sulfide samples:

- The oxide material achieved 76% gold recovery after 60 days in a bucket-leach test at a nominal 1.5-inch crush.
- The sulfide material achieved only 29% gold recovery after 75 days, suggesting that some gold was locked or encapsulated in pyrite.
- Bottle-roll testing produced gold recoveries of 89% from oxide material and 76% from sulfide material.
- Silver recovery was variable, reflecting mineralogical differences and liberation limitations.
- Cyanide consumption was substantially higher for the sulfide material.

The 1987 column tests demonstrated a strong relationship between crushing and recovery:

- Coarse 3- to 14-inch material averaged 57% Au and 3% Ag recovery.
- Minus 3-inch material averaged 78% Au and 13% Ag recovery.
- Material crushed to minus ¼ inch and agglomerated averaged 88% Au and 24% Ag recovery.
- Percolation testing indicated that material crushed to minus ¼ inch would require cement agglomeration.
- Bottle-roll testing produced approximately 92% Au and 64% Ag recovery.

1.5.2 McClelland 2014 Test Work

The 2014 program used underground, surface, and blended composites. Direct head assays showed considerable variability, interpreted as reflecting the heterogeneous or “spotty” distribution of gold rather than analytical problems.

Column tests were performed at 80% passing ¾ inch for 190 days:

Table 1-1
McClelland Summary of Columns Test Work

Composite	Leach Time (days)	Au Rec. %	Ag Rec. %	NaCN Consumed lb/ton	Lime Added lb/ton
Surface	190	88.9	20	3.12	3.4
Underground	190	84.6	15.4	3.86	5.4
Blended	190	86.3	16.7	3.72	4.0

Gold extraction was largely complete within approximately 30–40 days. Silver leached more slowly, but extending the test beyond 190 days was not expected to materially improve silver recovery.

Gold recovery showed limited sensitivity to crush size, whereas silver recovery improved significantly with fine grinding. A bottle-roll test at 80% passing 200 mesh and elevated cyanide concentration achieved 60.6% silver recovery. Mineralogical examination attributed the low heap-leach silver recovery to extremely fine silver-sulfide particles, approximately 0.5–5 µm, and refractory silver associated with jarosite minerals.

The test work supports conventional heap leaching at 80% passing ¾ inch. The design recoveries selected for the Initial Assessment (IA) are 84% for gold and 13% for silver. Cement agglomeration is not included at this crush size. The reported abrasion index is 0.2431.

1.6 Mineral Resource Estimate

The Mineral Resource Estimate was prepared by independent Qualified Person James Ashton, P.E., with an effective date of May 29, 2026. No Mineral Reserves were estimated.

The estimate incorporates 437 drill holes containing 18,278 assay intervals and more than 93,071 ft of RC and core drilling. It also includes 12 holes drilled in 2014 that were not incorporated into earlier estimates.

Estimation methodology

The resource model was developed in MineSight using:

- Gold and silver mineralized domains modeled separately
- Twenty-four cross-sections spaced at 50-ft intervals
- Block dimensions of 20 × 20 × 20 ft
- Ten-foot downhole composites
- An average specific gravity of 2.37

- A tonnage factor of 13.518 ft³/t
- Locally varying anisotropy
- Inverse-distance weighting as the reporting method
- Ordinary kriging and nearest-neighbor estimates for validation

The broad gold domain was defined at approximately 0.003 oz Au/t, with an internal high-grade domain at approximately 0.030 oz Au/t. Silver domains were developed using approximately 0.20 and 2.0 oz Ag/t boundaries. Grade capping was evaluated but was not considered necessary.

Resource classification was based on composite proximity, number of composites, and number of drill holes. Blocks meeting the more stringent criteria were classified as Measured or Indicated; the remaining estimated blocks were classified as Inferred.

The Mineral Resources were constrained by a Lerchs-Grossmann optimized pit and reported using a diluted gold-equivalent cutoff of 0.005 oz AuEq/t. Metal prices and recoveries used in the gold-equivalent calculation were \$3,300/oz Au, \$40/oz Ag, 84% Au recovery, and 13% Ag recovery.

Table 1-2
Longstreet Gold and Silver Mineral Resources

Measured & Indicated Resources							
Classification (oz. Au/ton)	Tons	Grade (oz/t)			Ounces	Ounces	Ounces
		Gold	Silver	Gold Eq.	Gold	Silver	Gold Eq.
Measured (.005)	1,050,336	0.019	0.382	0.020	20,000	401,200	20,700
Indicated (.005)	5,246,679	0.016	0.427	0.017	83,900	2,240,300	88,200
Total Measured & Indicated Resources							
Total M&I	6,297,015	0.016	0.419	0.017	103,900	2,641,500	108,900
Inferred Resources							
Classification (oz. Au/ton)	Tons	Grade (oz/t)			Ounces	Ounces	Ounces
		Gold	Silver	Gold Eq.	Gold	Silver	Gold Eq.
Inferred (.005)	481,517	0.014	0.400	0.015	6,700	192,600	7,100
Total MI&I	6,778,532	0.016	0.418	0.017	110,600	2,834,100	116,000

The block model was validated through comparisons with geological-domain volumes, alternative interpolation methods, composite grade distributions, swath plots, and visual reviews. The model generally reproduces the observed gold and silver trends, although the highest-grade portions may be understated.

Recommended future work includes additional density measurements, improved interpretation of high-grade controls, verification of the oxide-sulfide boundary, infill drilling with increased diamond-core coverage, downhole surveys, and geotechnical core logging.

1.7 Mining Methods

The IA assumes conventional open-pit mining by a mining contractor. The pit design was developed from an optimized pit shell using Measured, Indicated, and Inferred Mineral Resources; therefore, the resulting mine plan does not constitute a Mineral Reserve.

Because no project-specific geotechnical study has been completed, pit-slope parameters were developed by comparison with the nearby Manhattan and Round Mountain open pits. Principal design parameters include:

- Nominal bench height: 20 ft
- Triple-bench rock face: 60 ft
- Bench face angle: 63°
- Inter-ramp slope: 49°
- Two-way haul-road width: 60 ft at a maximum 10% grade
- Single-lane road width: 45 ft at a maximum 12% grade
- Optimization overall slope: 45°

The economic pit optimization used heap-leach processing, gold and silver recoveries of 84% and 13%, respectively, and a minimum cutoff grade of 0.005 oz AuEq/t. The report's mining optimization parameters include gold and silver prices of \$3,600/oz and \$48/oz.

In-pit inventory and schedule

The designed pit contains approximately:

- 6.55 million tons of leach material
- 5.31 million tons of waste
- 11.86 million total tons mined
- Overall strip ratio of 0.81:1
- 105,461 contained ounces of gold
- 2.73 million contained ounces of silver

Mining and processing are scheduled over four years, with residual metal production continuing into Year 5 because of the heap-leach recovery lag. No separate preproduction waste-stripping

period is required because mineralized material occurs near surface. Initial crushed material will be used as protective overliner on the leach-pad liner.

The mining rate ramps up during Year 1 and reaches approximately 13,000 t/d during the second half 1 and continues through years 2 and 3. Heap placement reaches approximately 2.0 million tons annually during peak production. Estimated life-of-mine production is:

- 88,587 oz Au at 84% recovery
- 354,701 oz Ag at 13% recovery

Mining will be performed on a top-down bench sequence. The design includes a lined 6.5-million-ton heap-leach facility and a waste-rock storage area. Known historic underground workings will require additional investigation and appropriate ground-control measures during mining.

1.8 Processing and Recovery Methods

The selected process is three-stage crushing followed by conventional cyanide heap leaching, carbon-in-column adsorption, and off-site stripping or ashing of loaded carbon.

Six alternatives were evaluated, including on-site versus toll stripping, gold-only versus gold-and-silver recovery, HPGR crushing, contract crushing, diesel power, and portable crushing. Option 6 (diesel generators with portable crushers and off-site carbon treatment) produced the strongest modeled economics and was selected as the basis for this Technical Report.

1.8.1 Process configuration

ROM material will be fed by front-end loader to a portable three-stage crushing plant. The circuit comprises a primary jaw crusher followed by secondary and tertiary cone crushers operating in closed circuit with screens. The plant will produce material at 80% passing ¾ inch.

Key design criteria include:

- Average crushing and heap-placement rate: approximately 5,500 t/d
- Annual heap placement: approximately 2.0 million tons
- Crushing-plant availability: 60%
- Leach cycle: 140 days
- Solution application rate: 0.004 gpm/ft²
- Nominal solution flow: 2,279 gpm
- Gold recovery: 84%

- Silver recovery: 13%
- Lime addition: 3.58 lb/t
- Cyanide consumption: 1.31 lb/t

Crushed material will be stockpiled, reclaimed by feeders, dosed with pebble lime, and conveyor-stacked in 32.8-ft lifts. The heap will have a maximum height of five lifts and an overall slope of approximately 2.5H:1V.

The leach pad is designed for approximately 6.5 million tons over 2.38 million ft². Its composite liner includes overliner, a 60-mil textured LLDPE geomembrane, a low-permeability soil layer, leak detection, and prepared subgrade. A suitable clay source has not yet been identified.

Pregnant solution will flow by gravity to the pregnant pond and then be pumped through one train of five open-top CIC columns. Each column will contain approximately 3.3 tons of carbon. Loaded carbon is expected to contain approximately 67 oz/t Au and 58 oz/t Ag.

Loaded carbon will be shipped off-site for stripping or ashing and refining. Stripped carbon will not be returned; fresh carbon will replace each shipment because reuse without on-site reactivation was considered less efficient.

Solution-management facilities include:

- Pregnant pond: approximately 9.58 million gal
- Event pond: approximately 6.9 million gal
- Barren-solution tank: 45,000 gal
- 30% liquid-cyanide storage tank: 45,000 gal
- Pebble-lime silo: 105 tons

The project will use diesel generators for electrical power and groundwater wells for process makeup, dust suppression, fire protection, and potable-water requirements.

1.9 Environmental and Permitting

Star Gold has staked and maintains 139 unpatented mineral exploration claims on United States Forest Service (USFS) and Bureau of Land Management (BLM) lands. It also has the right to mine on an additional 5 claims held by Clifford, et al. The Company has an active Plan of Operations with USFS and BLM to drill several water supply and monitor wells as well as additional core drilling. Star Gold also has begun work to permit a large initial baseline core drilling program for the “Red Cyprus Ridge” prospect located on the Company’s USFS claims. Updates to the Project’s baseline studies have been ongoing since June of 2025. Star Gold is planning

on submitting a complete Plan of Operations Application to USFS and BLM early in 2027 to facilitate full mining operations.

The heap leach pads and most of the required support infrastructure will be located on BLM claims with mining operations occurring on USFS claims. Star Gold had previously decided to locate the heap leach pads in a small canyon immediately adjacent to the proposed mining area on USFS claims but decided to relocate the heap leach pads to BLM claims due to engineering and water management considerations with the canyon location. This location is also near additional potentially mineralized zones and may be utilized in the future.

Star Gold is planning on submitting a complete mine Plan of Operations Application to USFS and BLM early in 2027 to facilitate full mining operations. Based on recent permitting in Nevada and the current permitting climate being promoted by the Federal Government, the time required to secure permits, prior to the construction of facilities and mine pre-stripping, is estimated to be up to 12-14 months. This schedule is bolstered by the fact that all baseline studies are completed and are currently being updated (where required), hydrogeologic and geochemical study work has commenced and will be completed by the end of 2026. No significant objections have been raised thus far by members of the public, including indigenous peoples, environmental groups or other government agencies (note that the U.S. EPA conducts a review of all Environmental Assessment/Environmental impact statements).

Star Gold is fully funded to complete the required baseline environmental study updates, the additional technical studies underway as all engineering work and Plan of Operations preparation on an accelerated schedule as much of the required work is being completed simultaneously.

1.9.1 Cultural Resources

The Baseline Cultural Resources Study was completed on the entire claim block (both USFS and BLM claims) in 2015 and identified many Cultural Resources sites. Most of these are non-significant Native American camp locations and artefacts along with remnants of the Longstreet settlement within the Project boundaries. The most significant Native American sites are located far from the mine, leach pads and other Project impacts. Remediation plans will be developed for the sites that would be impacted, as required. Mining and leaching activities will not be impacted by any sites deemed worthy of preservation. Updates to the Cultural Resources Study are completed on an as needed basis during the current exploration and well drilling activities. Star Gold is utilizing on-site monitoring by a cultural resources' expert, on as needed basis, during the current drilling program to minimize any cultural resource impacts.

1.9.2 Hydrology

The Longstreet Mine is in the Monitor Range of the Great Basin section of the Basin and Range physiographic region. The Basin and Range is largely an arid region encompassing a majority of the western United States; its topography is characterized by alternating narrow faulted mountain ranges and flat fault bound valleys (basins). The Great Basin, as a section of the Basin and Range region, follows the same topographic characteristics with notable internal surface and subsurface hydrologic drainage.

Star Gold has secured, through two long term leases, 2471 acre/feet of water rights from current owners of these water rights in Stone Cabin Valley. The acre/feet of water leased is at least 150% larger than what is anticipated to be required for mining and ore leaching applications.

Star Gold has an approved Plan of Operations with the USFS and BLM to conduct water supply and monitor well drilling in a favorable location near the Project site. The well drilling began in late August 2026 and will continue until November or December of 2026. Once water has been located, a point of diversion application will be applied for with the Nevada Division of Water Resources. Based on the results of the well drilling program and pump testing, a baseline hydrogeologic study will be developed.

The required seeps and springs location study and sampling work commenced at the Longstreet site in July 2026 and will continue a quarterly basis until June 2027. No adverse conditions have been noted thus far during the study work.

1.10 Capital and Operating Costs

The capital and operating cost estimate is based on third-quarter 2026 U.S. dollars. Costs were developed from vendor quotations, contractor pricing, recent Nevada project data, KCA's internal database, and first-principles estimates. The selected basis assumes portable crushing, diesel-generated power, contract mining, CIC recovery, and off-site carbon treatment.

The total Life of Mine (LOM) capital cost for the Project is \$84.2 million and is summarized in Table 1-3 below.

Table 1-3
Capital Cost Summary

Description	Cost (US\$ M)
Pre-Production Process Capital	\$57.6
Mining Capital	\$4.2
Subtotal Capital	\$61.8

Working Capital & Initial Fills	\$7.8
Total Initial Capital	\$69.6
Sustaining Capital – Mine & Process	\$4.6
Reclamation & Closure	\$10.0
Total	\$84.2

Process and infrastructure direct costs are approximately \$40.0 million before spares and contingency. Major capital areas include:

- Crushing: \$7.9 million
- Stockpile, reclaim, and stacking: \$5.3 million
- Heap-leach pad and ponds: \$7.6 million
- Carbon adsorption: \$4.5 million
- Laboratory: \$2.5 million
- Diesel power generation: approximately \$6.3 million
- Plant mobile equipment: \$2.75 million

Process and infrastructure contingency averages approximately 16.9%, with discipline-specific allowances of 10%–20%. Working capital is based on approximately 60 days of operating costs. Reclamation and closure are estimated at \$10 million.

The average LOM operating cost for the Project is US\$22.98 per ton of ore processed. Table 1-4 below presents the LOM operating cost requirements for the Longstreet Project.

Table 1-4
Operating Cost Summary

Description	Cost (US\$ M)
Mine	\$7.49
Process & Support Services	\$13.27
Site G&A	\$2.22
Total	\$22.98

Mining costs are based on a contract-mining quotation. Process costs were developed from staffing, power, reagents, consumables, maintenance, laboratory, mobile equipment, and off-site carbon-treatment requirements.

Important operating assumptions include:

- 44 process personnel and 10 G&A personnel
- Connected process load of approximately 2.9 MW
- Average operating load of approximately 1.3 MW

- Energy consumption of approximately 4.63 kWh/t
- Generated-power cost of \$0.38/kWh
- Diesel price of \$3.34/gal
- Cyanide consumption of 1.31 lb/t at \$1.45/lb
- Lime consumption of 3.58 lb/t at \$0.204/lb
- Complete replacement of loaded carbon
- Average annual G&A expense of approximately \$3.3 million

1.11 Economic Analysis

A summary of key economic parameters is shown in Table 1-5

Table 1-5
Key Economic Parameters

Item	Value	Units
Au Price	3,600	US\$/oz
Ag Price	48	US\$/oz
Au Avg. Recovery	84	%
Ag Avg. Recovery	13	%
Treatment Rate	5,500	ton/d
Annual Produced eqAu, Avg.	21.1	koz
Royalties	1.5	%
Mine Life	4	years
Payback	1.8	years

A summary of the economic model is presented in Table 1-6

Table 1-6
Economic Analysis Summary

Economic Analysis		
Internal Rate of Return (IRR), Pre-Tax	48.30%	
Internal Rate of Return (IRR), After-Tax	39.57%	
Average Annual Cashflow (Pre-Tax)	\$43	M
NPV @ 5% (Pre-Tax)	\$87	M
Average Annual Cashflow (After-Tax)	\$38	M
NPV @ 5% (After-Tax)	\$67	M
Gold Price Assumption	\$3,600	/Ounce
Silver Price Assumption	\$48	/Ounce
Pay-Back Period (Years based on After-Tax)	1.8	Years
Capital Costs		

Initial Capital	\$62	M
Working Capital & Initial Fills	\$8	M
LOM Sustaining Capital	\$5	M
Operating Costs (Average LOM)		
Mining	\$7.49	/Ton processed
Processing & Support	\$13.27	/Ton processed
G&A	\$2.22	/Ton processed
Total Operating Cost	\$22.98	/Ton processed
Total Cash Cost	\$1,617	/Ounce Au
All-in Sustaining Cost	\$1,729	/Ounce Au
Production Data		
Life of Mine	4	Years
Total Tons to Crusher	6,552,050	Ton
Grade Au (Avg.)	0.016	ounces per ton
Grade Ag (Avg.)	0.416	ounces per ton
Contained Au oz	105,461	Ounces
Contained Ag oz	2,728,471	Ounces
Mine Throughput per day	13,000	Tons/day
Metallurgical Recovery Au (Overall)	84%	
Metallurgical Recovery Ag (Overall)	13%	
Average Annual Silver Production	80,310	Ounces
Average Annual Gold Production	20,057	Ounces
Total Silver Produced	354,701	Ounces
Total Gold Produced	88,587	Ounces
LOM Strip Ratio	0.81:1	

1.11.1 Gold Production

Total gold produced for the Longstreet deposit is estimated at 88,587 payable ounces of gold. Annual production profiles for gold are presented in Figure 19-4 with 20,057 payable ounces of gold recovered annually on average.

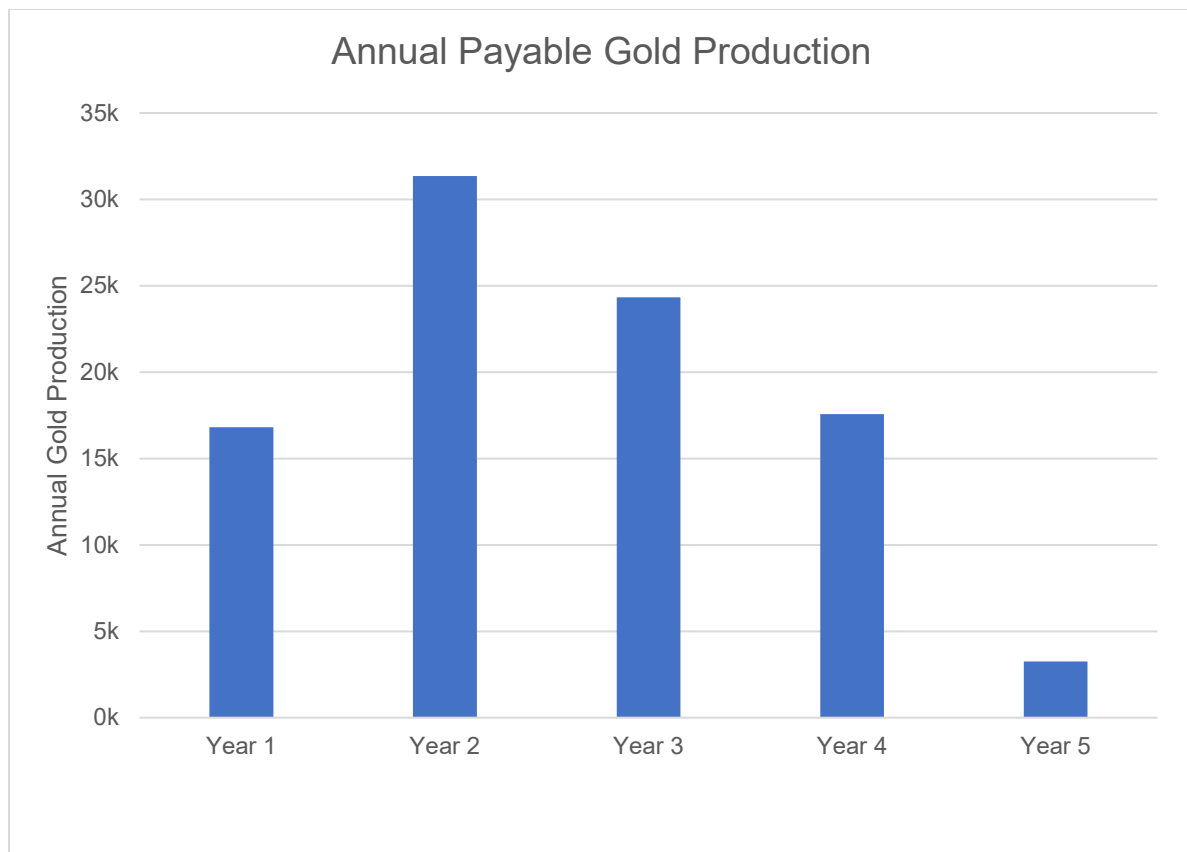


Figure 1-1
Annual Gold Production

1.11.2 Economic Cash Flow

The Base Case After-tax NPV for the Longstreet Project is US\$67M at 5% with an IRR of 40% using a gold price of US\$3,600/oz. The base case life of mine (LOM) all in sustaining cost is US\$1,729.

1.12 Conclusions and Recommendations

1.12.1 Conclusions

The Project location, access, claim position, and availability of regional mining services are adequate to support continued development evaluation. No location-related issue identified in the report precludes advancement of the Project. Federal land administration, permitting, surface-use arrangements, water availability, and access to the proposed leach-pad area remain important development considerations.

The available geological evidence supports the interpretation of Longstreet as a structurally controlled, low-sulfidation epithermal gold-silver deposit. The geological model is appropriate for the current Mineral Resource Estimate. Multiple incompletely tested targets provide meaningful opportunities to expand the known mineralized system.

Available testwork supports conventional heap leaching of material crushed to 80% passing $\frac{3}{4}$ inch using design recoveries of 84% for gold and 13% for silver. Gold extraction is favorable, whereas silver extraction is limited by mineralogy and liberation. Additional variability, permeability, and crush-size testing is required to support the next study stage and determine whether HPGR or finer crushing would provide an economic benefit.

The proposed contractor-operated open pit is a reasonable Preliminary Economic Assessment-level mining approach. The low strip ratio and near-surface mineralization are favorable characteristics. Project-specific geotechnical investigations, detailed characterization of historic underground workings, and higher-confidence resource classification are required before completing a final pit design and Mineral Reserve estimate.

Given the current information regarding this Project, it is estimated that the permitting timeline could be up to 12-14 months once the complete mine operating Plan of Operations is submitted. This timeline will likely be reduced somewhat by the modest size of the proposed operation and its relatively small impacts.

Baseline studies for flora and fauna and cultural resources have been completed with hydrogeologic, geotechnical and geochemical studies are scheduled for completion in 2026. These studies will be accomplished during the continued advanced engineering phase of the Project. Kappes Cassiday and Associates concludes that there are no recognized potential environmental or permitting fatal or material flaws regarding this Project.

The portable crushing, heap-leach, CIC, and off-site carbon-treatment flowsheet is technically feasible at the Preliminary Economic Assessment level and is appropriate for the Project scale. The process concept remains subject to confirmation of site conditions, water supply, commercial toll-treatment arrangements, and final equipment design criteria.

The capital and operating cost estimates are adequate for a Preliminary Economic Assessment and support comparison of the development alternatives considered. Project costs are particularly exposed to contract mining, diesel fuel, power generation, cyanide, toll treatment, transportation, and fixed G&A requirements.

The base-case economic analysis demonstrates potentially favorable Project economics, with an after-tax NPV5 of approximately \$67.3 million, an after-tax IRR of approximately 40%, and an after-tax payback period of approximately 1.8 years.

1.12.2 Recommendations

- Advance coordination with the USFS and BLM and prepare a comprehensive permitting register and schedule identifying the required federal, state, and county approvals, baseline studies, critical-path activities, and responsible parties.
- Refine the three-dimensional geological and structural model, with particular emphasis on the east-trending high-grade structure, crosscutting vein sets, alteration domains, oxide-sulfide boundaries, and controls on locally high gold grades.
- Complete detailed mapping and systematic prioritization of Red Knob, Cyprus Ridge, Opal Ridge, Northeast Main, North Zone, Spire, North Slope, and West Main using consistent geological, geochemical, and geophysical criteria.
- Conduct additional column-leach tests on spatially and geologically representative drill-core composites crushed to 80% passing $\frac{3}{4}$ inch. The test program should confirm recovery, leach kinetics, cyanide and lime consumption, and metallurgical variability.
- Complete comparative column tests using conventional and HPGR crushing at product sizes ranging from 80% passing $\frac{1}{2}$ inch to $\frac{1}{4}$ inch. Use the results to determine whether additional recovery from finer crushing would justify the corresponding capital and operating costs.
- Perform compacted permeability and load-percolation tests at the planned ultimate heap height.
- Advance the process design through preparation of updated equipment sizing calculations; site layouts; piping and instrumentation diagrams; electrical load lists; and a process-control philosophy.
- Confirm the design storm, pond capacities, leak-detection system, liner configuration, foundation conditions, and the availability and suitability of low-permeability soil and overliner material.
- Complete water-supply drilling and pump testing. Evaluate water quality for potable use and environmental management.
- Obtain firm commercial proposals for off-site carbon treatment, refining, carbon supply, transportation, and security.
- Update vendor and contractor quotations and develop a consistent estimate basis, scope matrix, battery limits, escalation date, contingency methodology, construction schedule, and owner-cost basis.
- Update operating costs using current quotations for contract mining, diesel fuel, power generation, activated carbon, toll treatment, transportation, labor, maintenance, and laboratory services.
- Evaluate opportunities to reduce operating costs through carbon reuse or regeneration, alternative crushing strategies, and optimization of fixed staffing requirements over the relatively short mine life.

- Advance the Project to a prefeasibility-level study after completing the recommended drilling, resource estimation, geotechnical, metallurgical, hydrogeological, environmental, permitting, engineering, and cost-estimation work.

2.0 Introduction

This SK 1300 Initial Assessment Technical Report was prepared for Star Gold Corp. (“Star Gold”), which trades on the OTCQB Market under the symbol SRGZ and holds a 100% interest in the Longstreet Gold and Silver Project. This report was prepared by Kappes, Cassiday & Associates (KCA), James W. Ashton, P.E. and Reinis N. Sipols

2.1 Terms of Reference

The purposes of this Technical Report are as follows:

- Present the results of an Initial Assessment (IA) for the implementation of open pit mining and heap leaching to recover the gold and silver mineralization, and
- Propose additional work required for Preliminary Feasibility or Feasibility level studies.

The property description, including reporting on historical exploration work, geology and mineralization, environmental review and adjacent properties have been published by Paul D. Noland in the previous SK1300 Resource Estimation in 2025.

The report was written in compliance with the disclosure and reporting requirements set forth by the U.S. Securities and Exchange Commission (SEC) S-K regulations (Title 17, Part 229).

2.2 Units of Measure and Acronyms

The units of measure presented in this report, unless noted otherwise, are in imperial units. The currency used for all costs is presented in US Dollars (US\$ or \$), unless specified otherwise. The costs were estimated based on quotes and cost data as of third-quarter 2026. Economic evaluation is done on a Project basis and from the point of view of a private investor, after deductions for royalties, income taxes, and various mining taxes and duties paid

Table 2-1
Units of Measure and Acronyms

Abbreviation	Definition
Ag	Silver
Au	Gold
AuEq	Gold equivalent
°F	Degrees Fahrenheit
G&A	General and administrative costs

GPS	Global Positioning System – satellite system used for ground location. Also colloquially refers to the receivers that obtain such locations from the system.
m	Meters
km	Kilometer
km ²	Square kilometers
kVA	Kilo volt ampere
lb	pound
gal	US Gallons
ha	hectares
in.	inch
"	inch (Where applicable)
'	Feet (Where applicable)
ft	Feet
ft ²	Square Feet
opt	Troy ounces per short ton
oz Au/t	Troy ounces of gold per short ton
oz Ag/t	Troy ounces of silver per short ton
ppm	Parts per million
ppb	Parts per billion
BLM	United States Bureau of Land Management
USFS	United States Forest Service
FWS	United States Fish and Wildlife Service
SFA	Sagebrush Focal Area – an area proposed as having outstanding Sage Grouse habitat
NEPA	National Environment Policy Act
DH	Drill Hole
RC	Reverse Circulation Drilling
RQD	Rock-quality designation
QA/QC	Quality assurance and quality control
Ma	Mega annum = Million years old
NI 43-101	Canadian Nation Instrument 43-101
FA	Fire Assay
AAS	Atomic Absorption Spectroscopy-analytical technique for multi-element analysis
ICP	Inductively Coupled Plasma, an analytical technique
ISO	International Standards Organization
NSR	Net Smelter Return, a type of Royalty
NAD27	North American Map Datum 1927
NAD83	North American Map Datum 1983
t	short ton (2,000 lb)
ton	short ton (2,000 lb)
tpd	Tons per day
IA	Initial Assessment
g/t	Gram per metric tonne

2.3 Source of Information

KCA has taken all reasonable care in producing the information contained in this report. The information, conclusions and estimates contained in this report are consistent with information available at the time of preparation, the data supplied by outside sources and assumptions, conditions and qualifications set forth in this report. The authors of this report are Travis Manning, James W. Ashton, P.E. and Reinis N. Sipols, and Richard Kern, each of whom is a Qualified Person.

The information in this report is not a substitute for independent professional advice before making any investment decisions. Any information in this report cannot be modified without the express written permission from KCA.

The primary sources of information used for this technical report are set out in Section 24.0, References, and include:

- SK1300 Resource Estimate, December 8th, 2025 (Paul D. Noland)
- 43-101 Preliminary Economic Assessment, February 2nd, 2021 (A-Z Mining Limited)
- Metallurgical testing developed by KCA and McClelland Laboratories
- The digital drillhole database.
- The original assay certificates for the holes.
- Various reports, including previous reports on sampling methodology, quality control and quality assurance (QA/QC)

2.4 Qualified Persons

Travis J. Manning of Kappes, Cassiday & Associates (KCA), James W. Ashton, Reinis N. Sipols and Richard Kern known collectively as the “Qualified Person(s)” or the “Author(s)” have prepared this Technical Report on the Longstreet Project at the request of Star Gold Corp.

Technical data and information used in the preparation of this report also included some documents prepared by third party contractors. The author sourced information from referenced documents as cited in the text and listed in the References section 24.0 of this report.

The processing studies, cost estimations, and financial analysis and review of current and historical metallurgical data were conducted by KCA under the auspices of Travis Manning, PE, of Reno, NV. Mr. Manning has no direct association or ownership with either Star or the Property. Mr. Manning is a Registered Member of SME and a Professional Engineer in the state of Utah and has over 22 years of experience in the mining industry. Mr. Manning is an independent

Qualified Person as stated in 17 CFR section 229.1300 and is responsible for Sections 1.5, 1.8, 1.9, 1.10, 1.11, 2, 3, 4, 10, 14, 15, 18, 19, 22.1.5, 22.1.8, 22.1.9, 22.1.10, 22.2.5, 22.2.8, 22.2.9, 22.2.10, 23.1.5, 23.1.8, 23.1.9, 23.1.10.

Reinis N. Sipols is a licensed Professional Engineer in the state of New York, New Jersey and Pennsylvania. Mr. Sipols is a Qualified Person (QP) for mining and environmental permitting, compliance, and community impacts; member of the Mining and Metallurgical Society of America (Member Number 01440 QP) and has over 44 years of experience. Mr. Sipols is an independent Qualified Person as stated in 17. CFR section 229.1300 and is responsible for Section 1.9, 17 and 22.2.9

James W. Ashton P.E. is an independent Qualified Person as stated in 17 CFR section 229.1300 and is responsible for Sections 1.2, 1.3, 1.4, 1.6, 1.7, 7, 8, 9, 11, 13, 22.1.1, 22.1.2, 22.1.3, 22.1.4, 22.1.6, 22.1.7, 22.2.1, 22.2.2, 22.2.3, 22.2.4, 22.2.6, 22.2.7, 23.1.2, 23.1.3, 23.1.4, 23.1.6, 23.1.7.

Richard Kern is an independent consultant as stated in 17 CFR section 229.1300 and is responsible for Sections 1.2, 5, 6, 20, 22.1.2, 22.2.2, 23.2

3.0 Property Description and Location

3.1 Property Description

The Longstreet Project (the Project, or Project) is located in northern Nye County, in the central part of the State of Nevada, USA. The Project consists of 142, unpatented mining claims covering approximately 2,500 acres, located on United States Forest Service (USFS) and United States Bureau of Land Management controlled land. The resource and primary exploration areas fall within land controlled by the USFS.

The regional structure setting is shown in Figure 3-1

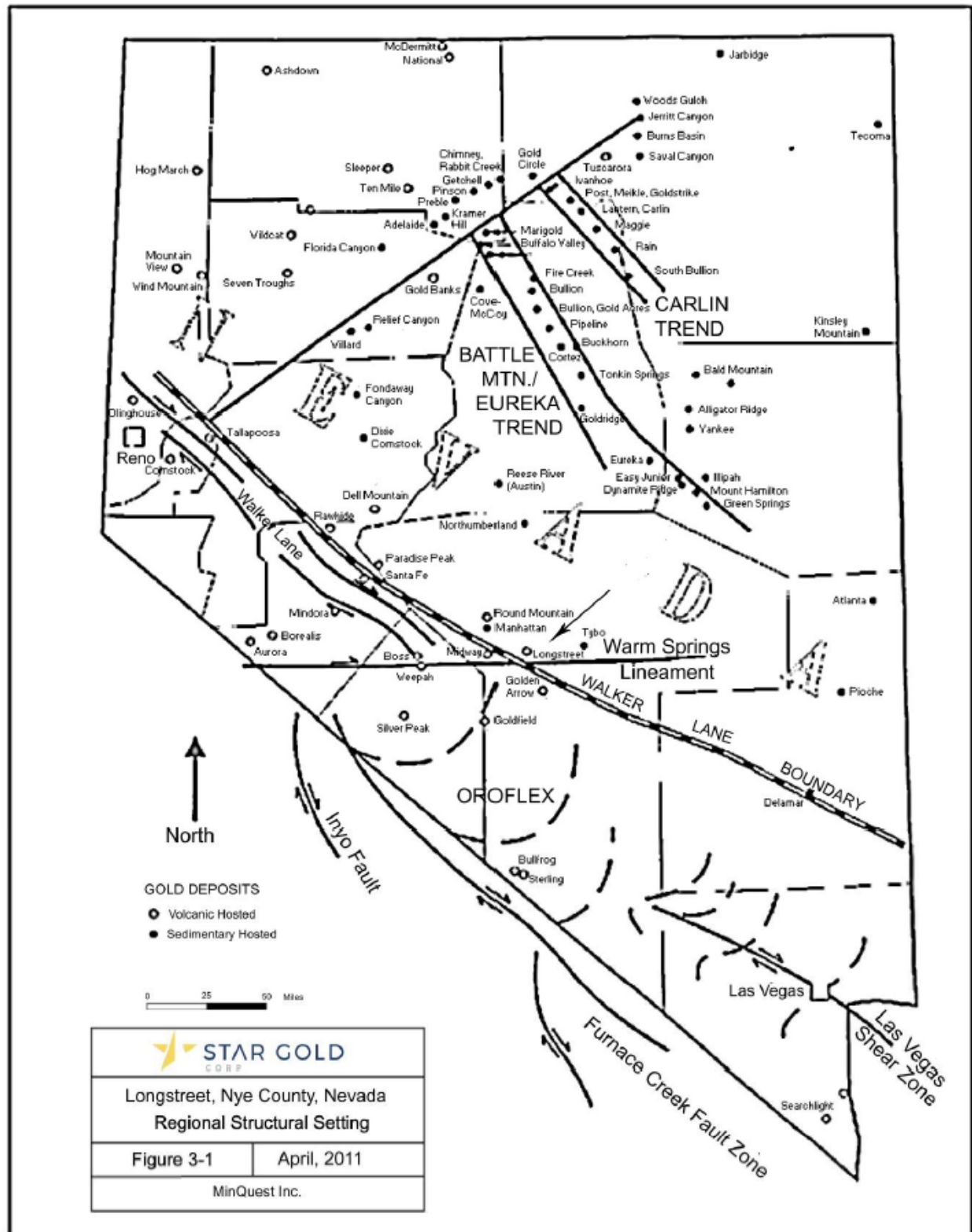


Figure 3-1
Regional Structural Setting

Star Gold has staked 26 claims (Leach Pad Claims) adjacent to the eastern boundary of the property, with the objective of providing the site for leach pads planned for future development of the Main Zone. These claims are included in the total of 142 claims making up the property holdings. These claims fall within land controlled by the BLM. In addition, Star Gold has staked 12 claims along a corridor leading from the main Longstreet property to the Leach Pad Claims.

The claims are located in Sections 9, 10, 15, 16, 17, 20 and 21 of T6N, R47E, MDB&M, Nye County, Nevada (Figure 3-2). A list of claims is provided in Appendix 1 (Section 27.1). This location falls on the eastern slope of the Monitor Range and within the Monitor Range portion of the Humboldt-Toiyabe National Forest.

Unpatented mining claims allow the owner to explore for and mine mineral commodities on public land. Surface disturbances and mining activities are regulated by a series of permits issued by the State of Nevada and the Federal Government (BLM and USFS).

The Project location and access are shown in Figure 3-2. The Longstreet claim map is shown in Figure 3-3.

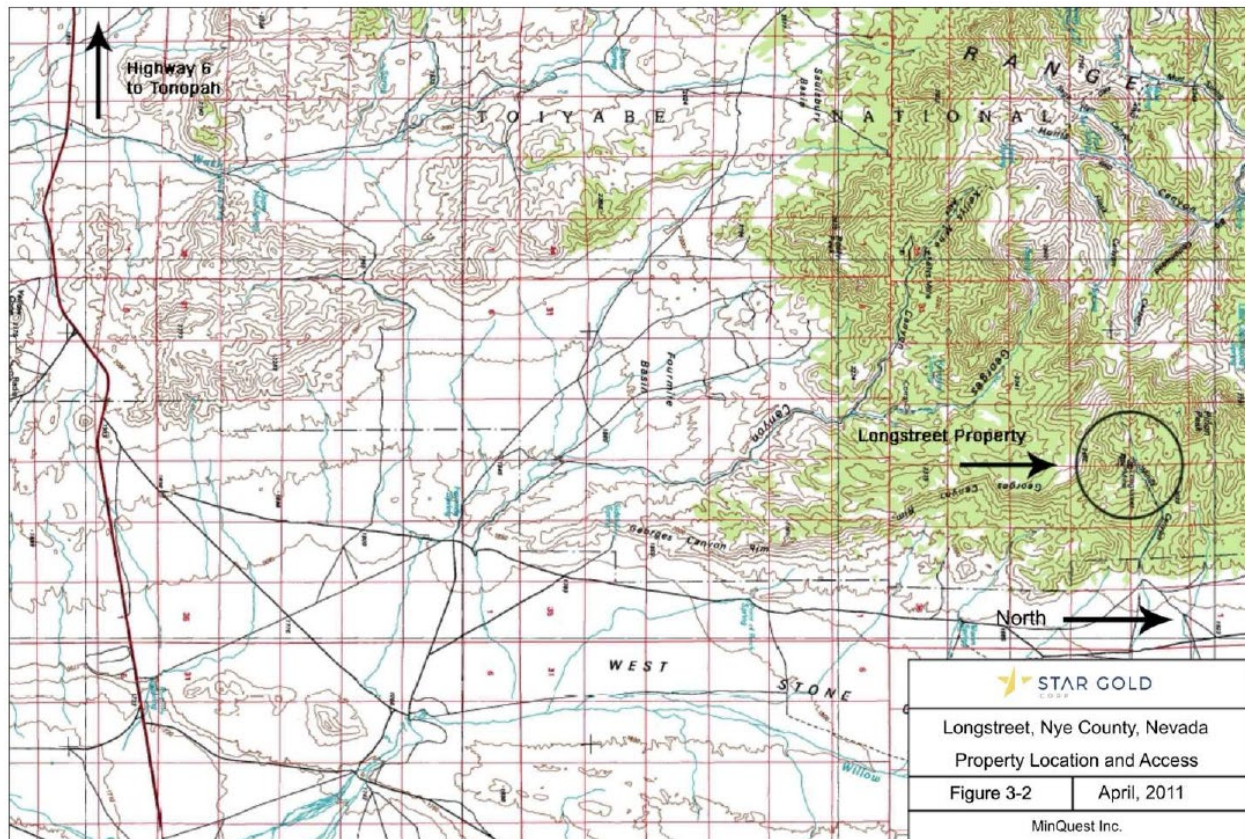


Figure 3-2
Location and Access

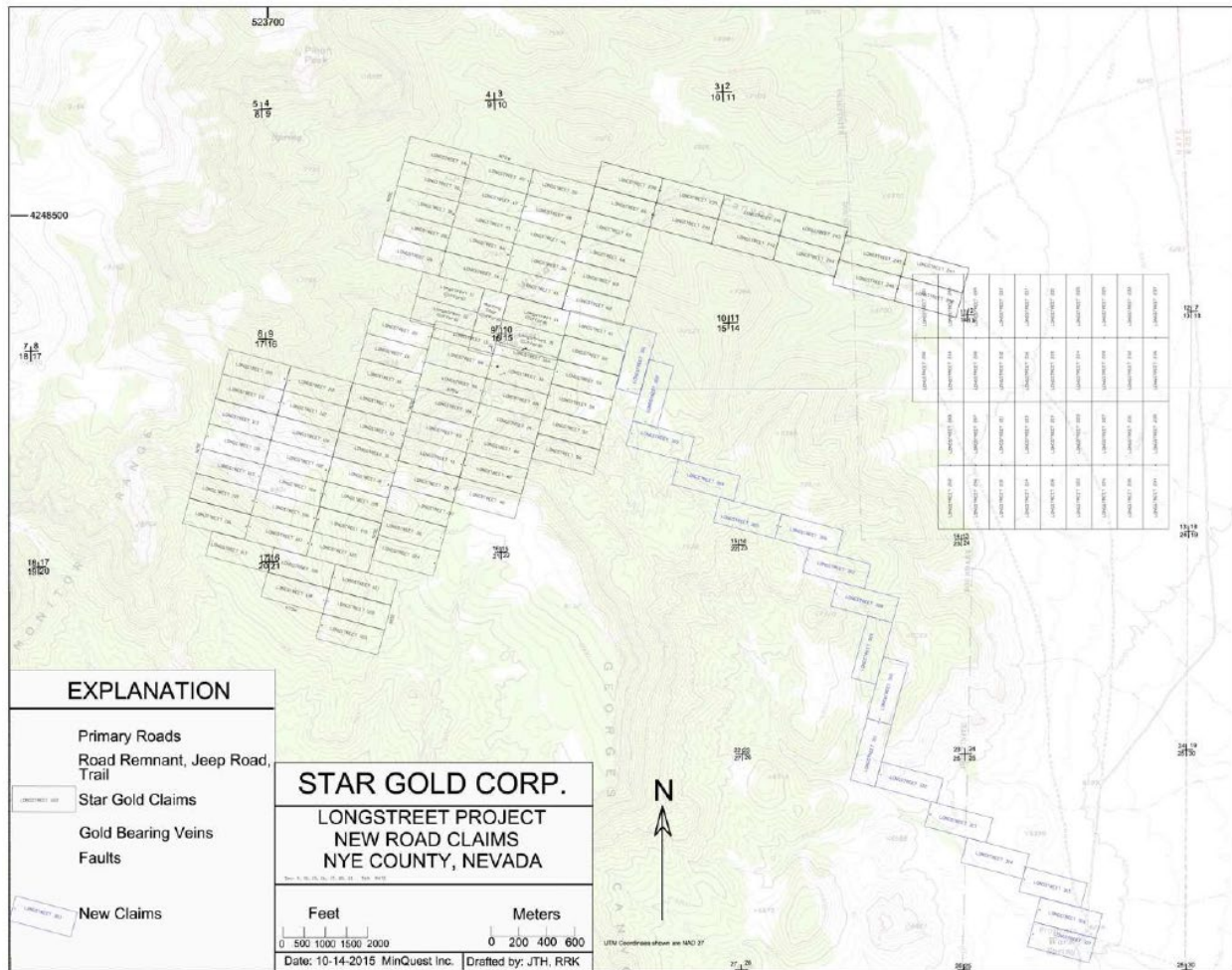


Figure 3-3
Longstreet Claim Map

3.2 Mineral Tenure

Star Gold Corp. (Star) entered into an exclusive option agreement with MinQuest on the Longstreet Project in December, 2009. This option agreement gave Star the right to obtain a 100% interest in the Longstreet property, minus a 3% NSR royalty, by meeting the financial obligations outlined in the agreement. Refer to Table 3-1 below, for a summary of the payment schedule and expenditure obligations in the Star-MinQuest option agreement.

Subsequent the execution of the MinQuest option agreement, all terms and payments have been satisfied. Consequently, Star now controls 100% ownership in the property minus the 3% Net Smelter Royalty (NSR) as per the initial MinQuest agreement. Star Gold holds an option with MinQuest to purchase up to one-half of the 3% NSR, resulting in a carried 1.5% NSR, for a one

time cash price of \$1.5 million USD. A separate agreement covering 5 unpatented claims (the Clifford agreement) has also been satisfactorily completed so that Star now owns 100% of those claims.

Figure 3-3 illustrates the land holdings. All the unpatented claims within the Longstreet Project have both U.S. Bureau of Land Management (BLM) and Nye County, Nevada fees paid and current through August 2026. Copies of receipts for the most recent annual claims fees for both Nye County, NV and the BLM are provided in the Appendices (Section 27.0).

The U.S. Forest Service is responsible for the surface and subsurface mineral estate in the active exploration areas. Star Gold has conducted intermittent exploration activities, including exploration drilling, under a Plan of Operation with the local U. S. Forest Service (USFS) office in Tonopah. In July 2025, Star submitted an expanded Plan of Operation to the USFS to allow for additional resource and exploration drilling, water production well drilling, and monitor well construction. The Plan of Operation describes the proposed exploration and pre-mining activities and anticipated surface disturbances that will require reclamation.

A reclamation plan and bond are part of the Plan of Operation. The recently submitted Plan of Operation is awaiting USFS approval. USFS approval is anticipated sometime in the first quarter of 2027. Meanwhile, Star continues to be permitted and bonded for casual exploration activities through the previous Plan of Operation. USFS and BLM roads are open to the general public without permit requirement or fees.

Table 3-1
Summary of Star Gold Option Agreement from 2009

Year No.	Payment (Cash)	Payment (stock options)*	Expenditure Obligation
On Signing	\$20,000	50,000	
1	\$0	50,000	\$200,000
2	\$20,000	50,000	\$250,000
3	\$30,000	50,000	\$350,000
4	\$30,000	50,000	\$450,000
5	\$50,000	50,000	\$550,000
6	\$50,000	50,000	\$750,000
7	\$50,000	50,000	\$1,000,000

*Stock and option payments for each of the year is broken into 25,000 shares of stock and 25,000 options 'at fair market value'

3.3 Property Location and Access

The 142 claims making up the project are located in Sections 9, 10, 15, 16, 17, 20 and 21 of T6N, R47E, MDB&M, Nye County, Nevada (Figure 3-2). A list of claims is provided in Appendix 1

(Section 27.1) and a claim map of the property is provided as Figure 3-3. This location falls on the eastern slope of the Monitor Range and within the Monitor Range portion of the Humboldt-Toiyabe National Forest. The Longstreet Au-Ag Project is located approximately 170 miles northwest of Las Vegas and approximately 50 miles northeast of Tonopah, a town of approximately 2,500 people, in west-central Nevada. The northeast-southwest oriented property extends approximately 1.8 miles along strike within the Monitor Range. The geographic coordinates of the central part of the property are approximately 38°22'00"N Latitude and 116°40'00"W Longitude.

Access to the Longstreet Project area is by paved and gravel road. Access is by two-lane paved highway (Nevada HWY 6 to Warm Springs) approximately 30 miles east from Tonopah, and then by gravel road (Stone Cabin Road) approximately 25 miles north and a further 2.5 miles west along a trail to Windy Canyon to reach the Longstreet property. The total distance from Tonopah to the property is approximately 57 miles. Supplies and heavy equipment are brought to the site by trucks, or other four-wheel drive vehicles. Currently, there is no permanent camp at the site. Casual labor would likely be sourced from Tonopah, Reno, Elko or Ely, Nevada. Experienced mining operations would likely be via contract and be sourced from Reno, Salt Lake City, Utah or other mining centers.

Production mining will be conducted by contract miners in order to reduce startup capital requirements. Star has already obtained a bid and proposal from N. A. Degerstrom of Elko, NV for contract mining and crushing. Other contract mining companies active in the region include Ledcor, Turner Mining Group, and Small Mine Development (SMD).

4.0 Accessibility, Climate, Local Resources, Infrastructure And Physiography

4.1 Physiography

The Monitor Range is about 100 miles long and from 5 to 15 miles wide. Summit elevations for most of its length are between 6,500 and 9,000 feet, with the highest point being approximately 10,000 feet. The Longstreet Project is located in rugged topography at the head of Windy Canyon, on the eastern slope of the Monitor Range (Figure 3-2). Elevations on the property range from about 6,000 feet in the northeast to 8,000 feet in the southwest part of the property. The hills are sparsely covered with pinion pine, juniper, sagebrush and greasewood. A mixture of mountain mahogany, willows, wild roses and various grasses occupy the valley bottoms.

4.2 Climate

Arid to semi-arid climatic conditions prevail in this part of Nevada. The annual precipitation is in the order of 5 inches (measured at Tonopah). Winter snowfall is typically about 6 inches, with the largest accumulations at elevations above 7,500 feet. Summer temperatures average 75 degrees F. in Tonopah, and winter temperatures average 23 degrees F. Prevailing climatic conditions allow year-round exploration and mining activities.

4.3 Infrastructure

Local infrastructure is available at Tonopah, Las Vegas or Reno. There is no infrastructure at the site, and electric power is provided by diesel generators. Infrastructure at Tonopah includes electrical power, internet service and limited road building equipment, and cell phone network. Reno and Las Vegas, although more distant, provide a full range of services and infrastructure typically associated with metropolitan areas of the Western USA. Potable water is provided in bottles, and industrial water is drawn from wells. Diamond and RC drilling equipment is available in Reno, and Elko, NV, and is also brought from other cities in neighboring States, such as Montana, Utah, Arizona or Idaho. For drilling programs, water is brought in by trucks from the Clifford Ranch. There is an airstrip close to Tonopah, but there is no regular commercial air service between Tonopah and Las Vegas or Reno.

Star intends to permit for and construct production water wells during a subsequent phase of work at Longstreet in preparation for mining. Once at least one of these wells is completed, water for drilling and all other needs will be supplied on site.

Electrical power requirements for any future mining operations on the Longstreet Project need to rely on diesel generators, and process water can be obtained from aquifers located in Windy Canyon. The detailed technical study by MDA in 1988 indicates that an open pit heap-leach mine exploiting the Longstreet Main Zone would have sufficient space for waste and tailing disposal storage (MDA, 1988).

5.0 History

5.1 Property History Prior to MinQuest and Star Gold Corporation

The Property was discovered in the early 1900's, but had limited development work until the 1920's. A 1929 report and maps show development of the "Golden Lion Mine" on two levels spaced 75 meters apart vertically (J.M. Butler, 1935). The report indicates the Golden Lion Mine operators targeted '300,000 tons of vein material' averaging 0.20 oz/ton (6.8 g/t) gold and 8 oz/ton (274 g/t) silver. The basis for this tonnage is not clear, but grades are confirmed by select underground sampling. A mill was constructed, the remnants of which are still on the property. However, the small stopes underground and very limited tailings material below the mill indicate very little mining was done and the operation was abandoned. For this reason, the current resource does not take into account the removal of any significant material by historic mining activities. This is explained in greater detail in the 'Resource' section of this report.

The following discussion provides several accounts of historic 'resource' estimates at the Longstreet property. Each is identified by source and references are provided. The resource estimates were conducted by reputable companies in an attempt to arrive at economic evaluations of mining potential at Longstreet. For this reason, they are considered relevant and reliable. Each of the historic resources appears to utilize drill data available at the time, and underground sampling in combination to derive what they refer to as 'indicated', or 'drill inferred' resources. However, the historical reports presented here do not define 'indicated' or 'inferred', nor do they always distinguish between 'resource' and 'reserve'.

Therefore, the categories utilized in these historic reports do not adhere to current NI 43-101 or SK 1300 categories or definitions of 'resource' or 'reserve'. The differences are not certain, since the methods used in the historic accounts are not sufficiently detailed. It was determined that a current, NI 43-101 or SK 1300 compliant resource estimate required more modern drilling accompanied by a documented QA/QC program, and that these historic resources could not be 'upgraded' or 'converted' to NI 43-101 or SK 1300 compliant resources by any available means.

One of the objectives of recent drilling at Longstreet has been to confirm much of the historic data utilizing methods suggested by NI 43-101 and SK 1300 standards. Neither the current author nor any other Qualified Person has done sufficient work on the historic estimates to classify them as current resources or reserves. Consequently, none of the historic resources or reserves reported below are treated by Star, or in this report, as 'current mineral resources or mineral reserves'.

The property lay idle until 1980 when Keradamex Inc. and E & B Exploration formed a joint venture to explore the property. The venture conducted soil and rock chip geochemical surveys, limited

underground sampling and drilled seven (one was abandoned) angle core holes into the Main mine workings area. Details for much of the historic drilling are missing or only partially available.

A summary of known drilling activity during significant exploration efforts is summarized in Table 3-1. Keradamex drilling revealed the presence of fracture related gold mineralization up to 36 meters thick extending into the hanging wall of the vein structure. Gold mineralization within this zone averaged 0.02- 0.528 oz/ton (0.7-18.1 g/t) and revealed the presence of a heap leachable target.

Table 5-1
Historic Drilling Summary

Date	Company	No. of Holes	Total Footage
1980	Keradamex	7	NA
1982-1983	Minerva	0	UG Sampling, no drilling
1984-1997	Naneco	approx. 500	NA, RC and air track
1987	Cyprus	7	3000
2002-2005	R.E.M.	30	11000

None of the historic resource estimates described below have been verified by current author, and are not treated or presented as current mineral resource estimates. They do not adhere to any of the NI43-101 or SK 1300 resource categories. None of the historic resources listed above or below in this section of the report meet the criteria of any of the 'resource' categories defined by NI43-101 standards, CIM or SK 1300 definitions. Consequently, these historic resource estimates should not be relied upon for current property evaluations or investment recommendations.

In 1982 Minerva Exploration optioned the property and initiated an underground sampling program. In 1983 a joint venture was formed with Geomex Canada Resources Ltd. Derry, Michener, and Booth were commissioned to assess the property and conducted underground sampling, bulk sampling and metallurgical testing. They concluded there were reserves* of 60,000 tons averaging 0.11 oz/ton (3.8 g/t) gold and 5 oz/ton (171 g/t) silver.

In early 1984 Naneco Resources Ltd., an Alberta company, acquired all of the assets of Minerva and an additional 10 percent interest in the property from Geomex. As operator, Naneco immediately initiated drilling. In 1985, with over 200 RC holes drilled, the venture announced an oxidized 'drill inferred reserve'* (not defined) of 850,000 tons averaging 0.079 oz/ton (2.7 g/t) gold and 1.1 oz/ton (38 g/t) silver along with an additional low grade reserve* of 1.5 million tons averaging 0.021 oz/ton (0.72 g/t) gold and 0.4 oz/ton (14 g/t) silver.

During the next few years Naneco increased its interest from 53 percent to 100 percent, conducted additional metallurgy, economic evaluation and drilling. In 1988, Naneco retained Mine

Development Associates (MDA) of Sparks, Nevada, to carry out a Prefeasibility study on the Longstreet property. As part of this Prefeasibility study, Kappes, Cassiday and Associates (KCA) carried out metallurgical testwork (bottle roll) on a representative composite of mineralized material from 31 RC drill holes (Prenn, 1988). Results of this metallurgical testing is provided in a later section of this report. Naneco's last reported resource indicates a “drill proven reserve”^{*} of 140,000 ounces of gold.

At least 492 RC holes have been drilled, most within the Main resource area. Naneco's last announcement also states that total property potential is “considerably higher than the 280,000 ounces currently believed to exist”. Unable to raise money because of falling gold prices and strapped with high land payments to the claim owners, Naneco dropped the property in 1998.

MinQuest acquired it shortly thereafter. The Cyprus Ridge target, which was evaluated by Cyprus Minerals Company in 1987 was acquired by MinQuest in early 2002. The property was optioned to Rare Earth Metals Corp. (REM) in May of 2002. REM later changed its name to Harvest Gold. Mapping and geochemical sampling of the 7 targets shown in Figure 6-3, Longstreet Targets, was completed in October, 2002. From 2003 through 2005 REM drilled 30 holes into Main totaling 3,350 meters (11,000 feet). The drill holes were angled toward the intersection of the two primary sheeted vein sets. Results showed a 20% improvement in average grade over vertical drilling. REM relinquished rights to the Longstreet property back to MinQuest in August 2009.

MinQuest and Star Gold Corp. agreed to terms for an option/purchase agreement in late 2009. The agreement between MinQuest and Star Gold was executed in January 2010.

^{*}These stated historical ‘reserves’ are not defined and do not meet standards set by NI 43-101 or SK 1300 for mineral resources or reserves. They are presented here for reference.

5.2 Property Exploration Under MinQuest and Rare Earth Metals 2002-2005

From 1998 to 2002, MinQuest carried out detailed geological mapping and lithogeochemical sampling over various target areas. Assay values of the 107 samples collected by MinQuest ranged from 0.02 g/t Au to 35.45 g/t Au and from 0.1 g/t Ag to 108 g/t Ag. At the Cyprus Ridge target area, assay values of 50 samples ranged from 0.03 g/t Au to 11.6 g/t Au, with an average value of approximately 0.5 g/t Au and 0.1 g/t Ag to 47 g/t Ag, with an average value of 7.2 g/t Ag.

Based on these results, MinQuest concluded that “the gold values are leakage anomalies from a deeper boiling zone” and considered the Cyprus Ridge as a high priority target area (Noland, 2012).

MinQuest crews sent the samples to ALS Minerals (ALS) in Reno, Nevada, for sample preparation, and then to ALS Minerals Laboratories in North Vancouver, British Columbia for Au and Ag assays. Results from this program are provided in are summarized in Table 5-2 below.

Table 5-2
Surface Geochemical Sampling Results (1998-2002)

Target Area	No. of Samples	Au ppm			Ag ppm		
		From	To	Average	From	To	Average
Main Zone	3	0.38	35.45	12.4	3.5	108	48
Opal	15	0.02	0.27	0.15	0.6	9.2	2.3
NE Main	2	0.11	0.93	0.52	4	20.6	12.3
North	12	0.03	18.14	1.93	0.6	49.6	15.2
Spire	8	0.03	0.45	0.24	0.4	20	9.7
Knob Hill	17	0.03	2.97	0.61	0.1	27.4	7.5
Cyprus Ridge	50	0.03	11.16	0.9	0.1	47	7.2
Total	107	0.02	35.45		0.1	108	

In May 2002, Rare Earth Metals Corp. (REM) optioned the property from MinQuest and carried out geological mapping and geochemical sampling over the Main Zone as well as six other target areas. Later in that year, REM changed its name to Harvest Gold Inc. (Harvest Gold), and from 2003 to 2005, Harvest Gold completed approximately 11,285 ft (3,440 m) in 32 inclined RC drill holes on the Main Zone. In August 2009, Harvest Gold returned the property to MinQuest.

Results from the MinQuest lithogeochemical sampling program indicate that, with few exceptions, there is good correlation between gold and silver values, as shown in Figure 5-1.

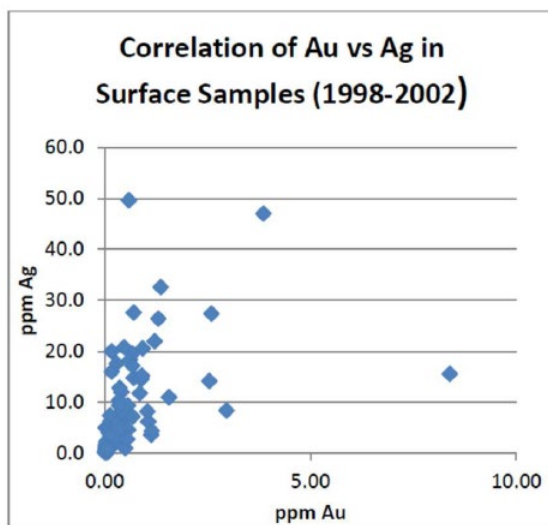


Figure 5-1
Au vs Ag Correlation

5.3 Property Exploration Star Gold 2010-2013

In January 2010, Star Gold entered into an option agreement with Messrs. Kern and Duerr of MinQuest to earn a 100% interest in the Longstreet property, and commenced a systematic exploration program, including 5,270 ft (1,728 m). of drilling in 16 RC holes and lithogeochemical sampling completed in 2011. All of the 2011 drilling by Star Gold was done on the Main Zone.

From August 5 to October 19, 2012, Star Gold completed 10,240 ft (3,122 m) of drilling in 23 RC holes and 1,295 ft (395 m) of diamond drilling in four holes (LS-1216C, LS-1217C, LS-1222C, and LS1224C). Detailed discussion on exploration by Star Gold is provided in the Drilling section.

From May 8 to July 29, 2013, Star Gold completed approximately 6,930 ft (2,123 m) of drilling in 20 RC holes. In 2014, Star Gold drilled 12 RC holes (4,445 ft) in the main resource area. Detailed discussion on exploration by Star Gold is provided in Section 7.2.

Exploration data indicate that work done to date has been concentrated on the Main Zone in the central part of the property, and the target areas tested by drilling cover less than 10% of the total area interpreted to host gold-bearing veins and fracture zones within the rhyolitic tuffs of the Longstreet property. The Author is of the opinion that additional drill testing of the remaining target areas is warranted.

To date, Star Gold has spent approximately \$1,500,000 on exploration on the Longstreet property, including option payments (Star Gold, 2012 and 2013). Table 5-3 is a summary of the exploration on the Longstreet property.

Additional details of historical exploration and drilling is provided in Section 7.0 of this report.

Table 5-3
Exploration History

Year	Company	Drilling						Remarks
		Geology	Lithogeo-chemical Sampling	RC		DD		
				ft.	No. of Holes	ft.	No. of Holes	
1903		v						Discovery of mineralized boulders?
1929	Gold Coin Company							Development of Golden Lion Mine (Main Zone)
1980	Keradamex / E & B	v	v			N/A	8	
1982	Minerva Exploration							Bulk sampling and resource estimation
1984-87	Naneco Resources Ltd.		v	54,221	332			Resource estimation, metallurgical test work & PFS
1987	Cyprus Minerals Company					3,000	7	Property evaluation
2002	MinQuest Inc.	v	107					
2003	REM/Harvest Gold			11,285	32			Metallurgical testwork
2011	Star Gold Corp.	v	v	5,270	16			Property evaluation and preliminary resource estimation
2012	Star Gold Corp.			10,240	23	1,295	4	
2013	Star Gold Corp.			6,930	20			
2014	Star Gold Corp.			4,445	12			
Totals			107+	92,391	435	4,293+	19	

6.0 Geological Setting, Mineralization and Deposit

6.1 Regional Geology

The Longstreet Project is located in the Nevada portion of the Basin and Range Province. This geological province is characterized by repeated episodes of compressional deformation in Paleozoic and Mesozoic time followed by extensional deformation and extensive magmatism and volcanism in Cenozoic time. Gold deposits are often described as being associated with 'mineralization trends', that are a reflection of deep crustal structures and magmatism, such as the 'Walker Lane' and the 'Carlin Trend'. The Longstreet Project is located in the Monitor Range, adjacent to the northwest trending Walker Lane volcanic-hosted gold trend that includes such world-class deposits as the Comstock and Goldfields mining camps (Figure 3-1).

The Monitor Range is a westward-tilted fault block that has been elevated by normal faults along its eastern front, and is typical of the uplifted mountains of the Basin and Range Province. The ranges are topographic highs rising above alluvium-filled valleys generated by Tertiary extensional tectonics. Central Nevada was an area of intense Oligocene – Miocene ash-flow volcanism that created numerous calderas and their outflow products. At least 13 calderas that range in age between 32 and 22 Ma have been mapped or interpreted in the area extending from the Shoshone Mountains eastward to the Monitor Range (see Figure 6-1). The southern Monitor Range consists mainly of Tertiary age volcanics and hypabyssal rocks related to the eruption of the Big Ten Peak volcano and a nearby unnamed 29 Ma caldera (Kleinhampl and Ziony, 1985) intruding and overlying Paleozoic sedimentary and metamorphic rocks (Figure 6-1).

The Paleozoic rocks are thrust-faulted marine sedimentary rocks comprised of quartzite, argillite and limestone of Cambrian, Ordovician and Silurian age. Minor amounts of Permian marine sediments (Palmetto Formation?) are also present in the Georges Canyon area.

In the southern Monitor Range Tertiary age volcanic rocks comprise more than 90% of the exposed bedrock (Figure 6-2). These rocks are more than 1 km thick and are predominantly flatlying.

Early Oligocene to early Miocene rhyolitic to dacitic ash-flow tuffs, with rhyolitic welded tuff are the thickest and most extensive units. Most of the Tertiary intrusions in the region are rhyolitic, but several small dacitic to andesitic dikes are present in the Georges Canyon area.

Mineral deposits in this part of the Basin and Range Province are varied and widespread and some of them have (had) substantial metal production. The producing Round Mountain gold deposit is about 25 miles northwest, and the past-producing Manhattan Mining Camp (gold/silver) is about 20 miles west-northwest of the Longstreet Property.

The Round Mountain Mine is a giant among epithermal precious metal deposits hosted by volcanic rocks, and the mineralization is a classic example of low sulphidation epithermal gold mineralization (White and Hedenquist, 1995). Gold deposits were discovered at Round Mountain in 1906 (Shawe, 1982) and by 1959 about 410 thousand ounces (troy ounces) of gold had been produced from placer and narrow vein lode deposits. Current production by open-pit mining methods commenced in 1977. Kinross (2010) reported an annual production for 2010 at 184,554 ounces of AuEq, with over 66 million tons of proven and probable reserves.

The oxidized ore is described as a closely spaced set of steeply dipping veins and veinlets following northwest-trending faults and associated joints over broad areas. Significant gold mineralization is not found in northeast-trending faults and fractures. The vein/veinlet system contains quartz, adularia, limonite (oxidized from pyrite), manganese oxide and associated native free gold. Flat veins are similar to the steep veins in character and mineral content, but with more brecciation of the wall rocks. Gold contents also appear to be higher in the flat veins. The adularia in the ore related veins is dated at 25.9 to 26.6 Ma, which is indistinguishable from the age of the enclosing 'Tuffs of Round Mountain' welded ash flow tuffs.

These tuffs were erupted from the Round Mountain caldera and were deposited within the caldera (Henry, Castor and Elson, 1996). The author notes here that both the nature of the host rocks and style of mineralization at Longstreet is very similar to that at the Round Mountain Mine. Hydrothermal alteration associated with the bulk mineable ore is evidenced by silicification and the replacement of magmatic feldspar by hydrothermal feldspar engendered by a potassium-rich hydrothermal fluid (Sander, 1988). Resources at Round Mountain as of December 31, 2010 included measured and indicated resources of 46 million tons grading 0.75 ppm Au, and 'Proven and Probable' reserves of 66 million tons grading 0.62 ppm Au (Kinross 2010 Annual Report). The resource categories are NI 43-101 compliant.

The Manhattan gold / silver camp is located approximately 20 miles west-northwest of the Longstreet Mine and is an example of Tertiary epithermal mineralization superimposed on Paleozoic sedimentary rocks. Gold / silver deposits were discovered at Manhattan in 1905 (Shawe, 1982) and by 1959 about 337,583 ounces (10,500 kg) of gold and 141,463 ounces (4,400 kg) of silver had been produced from placer and lode deposits. The lode deposits in the Manhattan district are of a variety of types, although they occur together in a coherent belt about 0.62 miles (1 km) wide, which follows the south side of the Manhattan caldera for about 6.21 miles (10 km). The most productive deposits formed in strongly faulted argillite and quartzite of the Cambrian age Gold Hill Formation. The generally northtrending zones of mineralized fractures are stockworks containing quartz, adularia, pyrite (oxidized to limonite) and native gold similar to the sheeted zones at Round Mountain. The silver production recorded for this camp is related to electrum and various silver-bearing sulphosalts.

The Clipper Mine located approximately 5 miles southwest of the Longstreet Mine near Murphy Camp was discovered in 1903 and was worked intermittently until 1943. The mine was initially developed during World War I and included a 175 foot shaft and a 370 foot adit. Recorded production is about \$12,000 (current 1951dollars) from mineralization having a gold: silver ratio of 1:1 and assaying from \$34-124 per ton (1951 dollars). Host rocks are welded rhyolite ash-flow tuffs similar to the Longstreet mine (Kleinhampl and Ziony, 1984). The Little Joe Claim located 6 miles south-southwest of the Longstreet Mine was developed by a 75 foot inclined shaft. Gold-bearing veins in 'rhyolitic tuff' were mined but production details are lacking.

Mineralization on the Last Chance claims located 11 miles west-northwest of the Longstreet Mine and southwest of Big Ten Peak was discovered in the 1920s. Mineralization consists of argentiferous galena, minor sphalerite and pyrite occurring in irregular pipes and chimneys generally at the intersection of cross faults within a northwest-trending shear zone in pre-Tertiary rocks. This property was developed by a 98.4 ft (30 m) two compartment shaft and a 200 ft (61 m) adit. Production in the late 1920s is recorded as 13.6 tons containing an average of 720 ppm Ag, 21% Pb and 2% Zn. A further 18.1 tons produced in 1938 contained 240-275 ppm Ag and 8% Pb (Kleinhampl and Ziony, 1984).

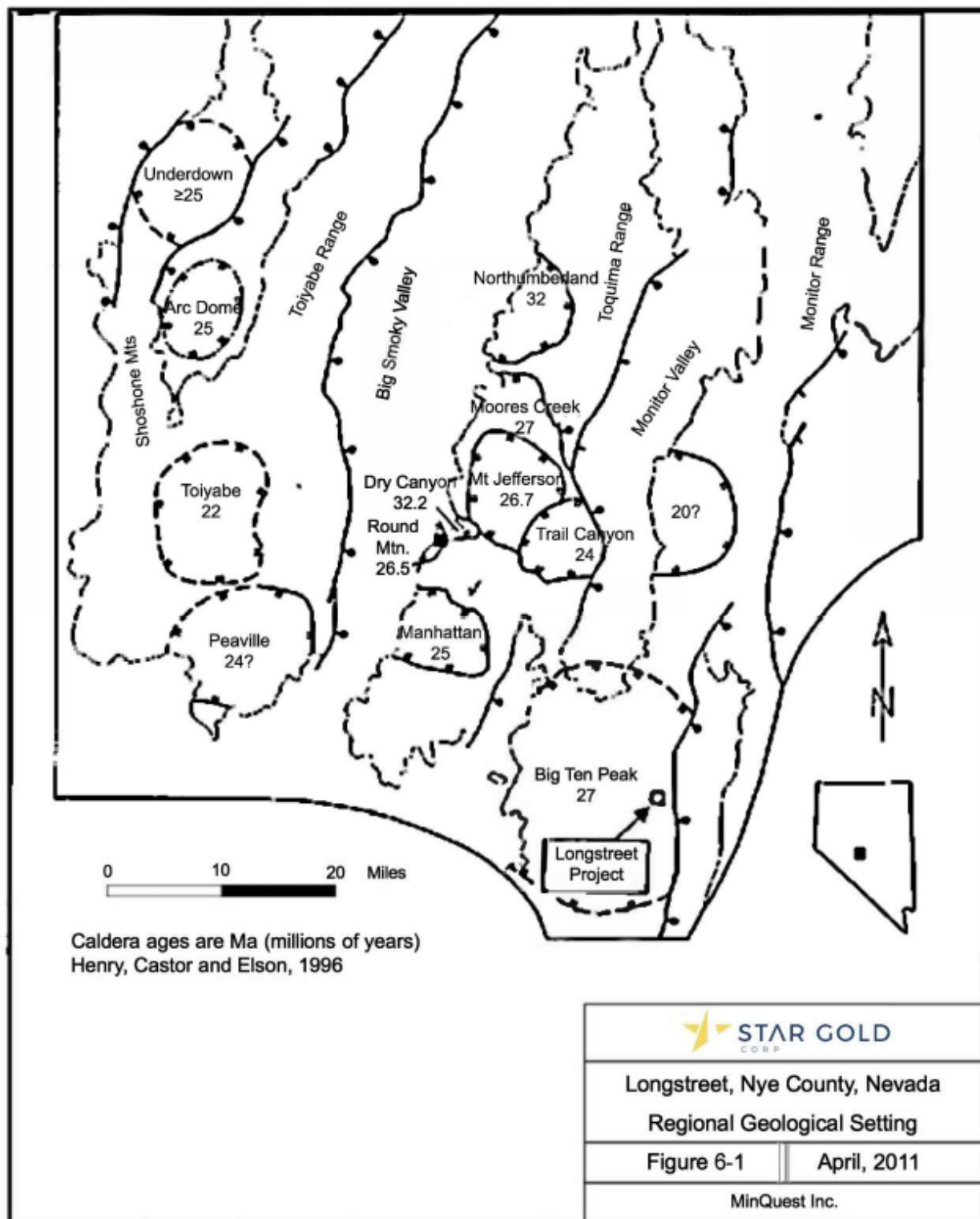


Figure 6-1
Regional Geological Setting

6.2 Local Geology

Outcrops of Cambrian to Jurassic metasedimentary rocks and volcanic rocks occur in the general area of the Longstreet property. The Tertiary volcanic rocks have been deposited on the basement rocks, and include fine-to-medium-grained, felsic tuff and breccia. The contact zones between the extrusive rocks and metasedimentary rocks are favorable for gold and silver mineralization, as evidenced by a number of surface showings.

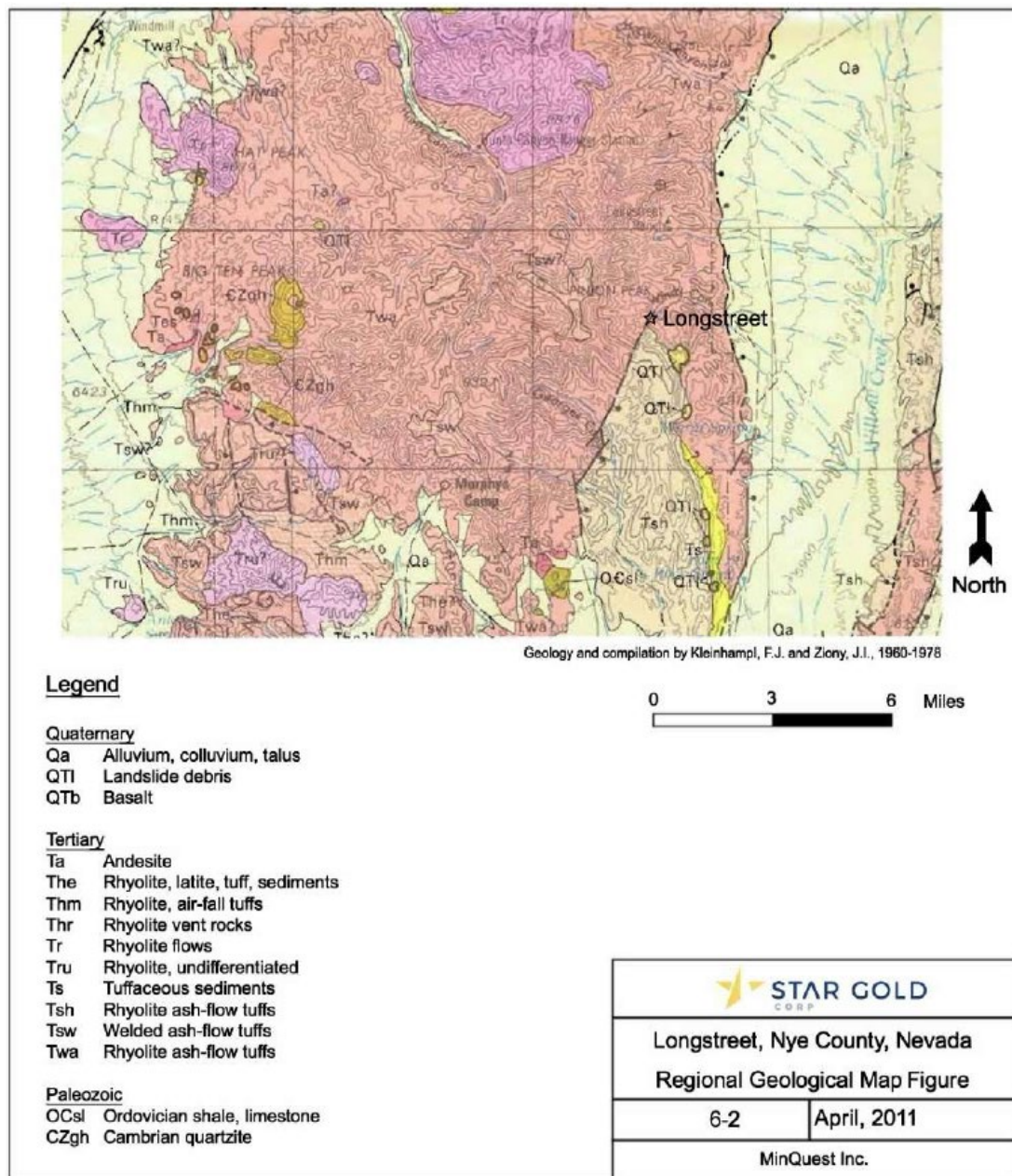


Figure 6-2
Regional Geological Map

6.3 Property Geology

Geologic mapping by MinQuest since 2002 indicates that the majority of the Longstreet Project is underlain by moderately to poorly welded rhyolite ash-flow tuff ('Tat') containing conspicuous exotic lithic fragments and pumice (Figures 6-4 through 6-6). The ash-flow tuff unit is buff to gray, and contains <10% quartz phenocrysts, 15% feldspar phenocrysts, 5-15% pumice and 5-20% other exotic fragments in an aphanitic groundmass (Liedtke, 1984). Hydrothermal alteration is prevalent and consists of argillic (bleaching and clay mineral development), silicic (pervasive silica flooding, or extremely high veinlet density) and potassic (adularia in quartz veinlets). Limonite and goethite development is considered to be weathering phenomena. These felsic ash-flow tuffs of Oligocene age are similar in age and character to the 'tuffs of Round Mountain', which host the Round Mountain Mine.

The Tat tuff unit (Figure 6-3 through Figure 6-5) displays horizontal bedding and may be in the order of 3,000 feet thick. The ash-flow tuff is intruded by rhyolite porphyry dykes ('Trp') exhibiting various orientations, and may represent feeder conduits to now-eroded rhyolitic lithologies higher in the stratigraphy.

A thin discontinuous unit of volcanoclastic and siliceous sediments ('Ts'), including sinter is deposited upon the ash-flow tuff unit. The unit is white, yellowish and light gray, bedded in part and probably represents a hiatus in volcanism. Siliceous alteration resulting in the development of sheeted quartz vein systems affects the Tat, Ts and Trp rock units. Overlying the Tat tuff and the Ts sediments is a black to brown strongly welded ash-flow tuff ('Trt') that forms bluffs and caps ridges. This unit has a distinctive thin (about 10 feet) vitrophyre zone near its base. This unit is estimated to be 300 to 450 feet thick and possibly a correlative of the Saulsbury Wash Formation (21.6 +/- 0.6 Ma).

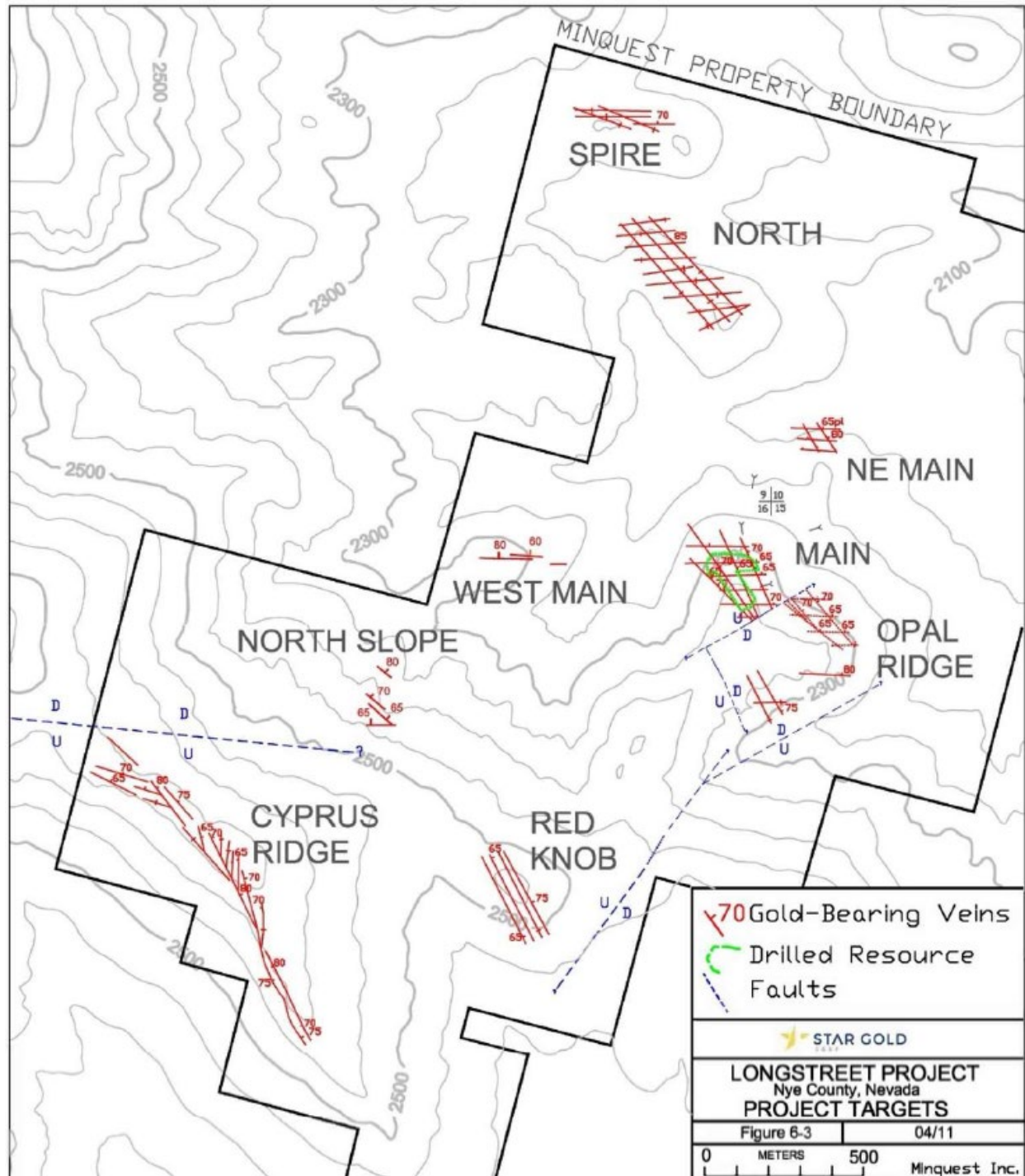


Figure 6-3
Project Targets

6.4 Structural Setting

The structural setting of the Longstreet area is not well understood. Regional geological mapping indicates that there are three sets of mineralized veinlet and fracture systems within the Longstreet property, which include at least nine mineralized target areas. These fracture systems are:

- Northwest trending vein and fracture system: a structural feature commonly present at the Main Zone, Opal Ridge, Red Knob, North Slope, and Cyprus Ridge target areas in the central and southwestern parts of the explored area, and Northeast Main, North, and Spire target areas in the northern part of the explored area.
- East trending vein and fracture system: a structural feature commonly present at the Main Zone, Opal Ridge, Red Knob, North Slope, Northeast Main, North, West Main, and Cyprus Ridge target areas.
- West-northwest trending vein and fracture system: a structural feature present at the Spire target in the northern part and northern portion of the Cyprus Ridge target area in the southwestern part of the property.

An east-trending fault dipping 40-55° is associated with the highest-grade gold / silver mineralization known to date. The bulk of the gold / silver mineralization in the Longstreet Mine is contained in steeply dipping multiple vein sets parallel to or semi-parallel to the fault. The author is of the opinion that east-trending vein and structure systems targets such as Red Knob, Cyprus Ridge, Red Knob and Opal Ridge should be highest priority targets for future exploration and drilling.

Liedtke (1984) indicates that similar fault directions are known 4,600 feet south and 2,800 feet north of the Longstreet Mine, which may host similar high-grade gold / silver mineralization.

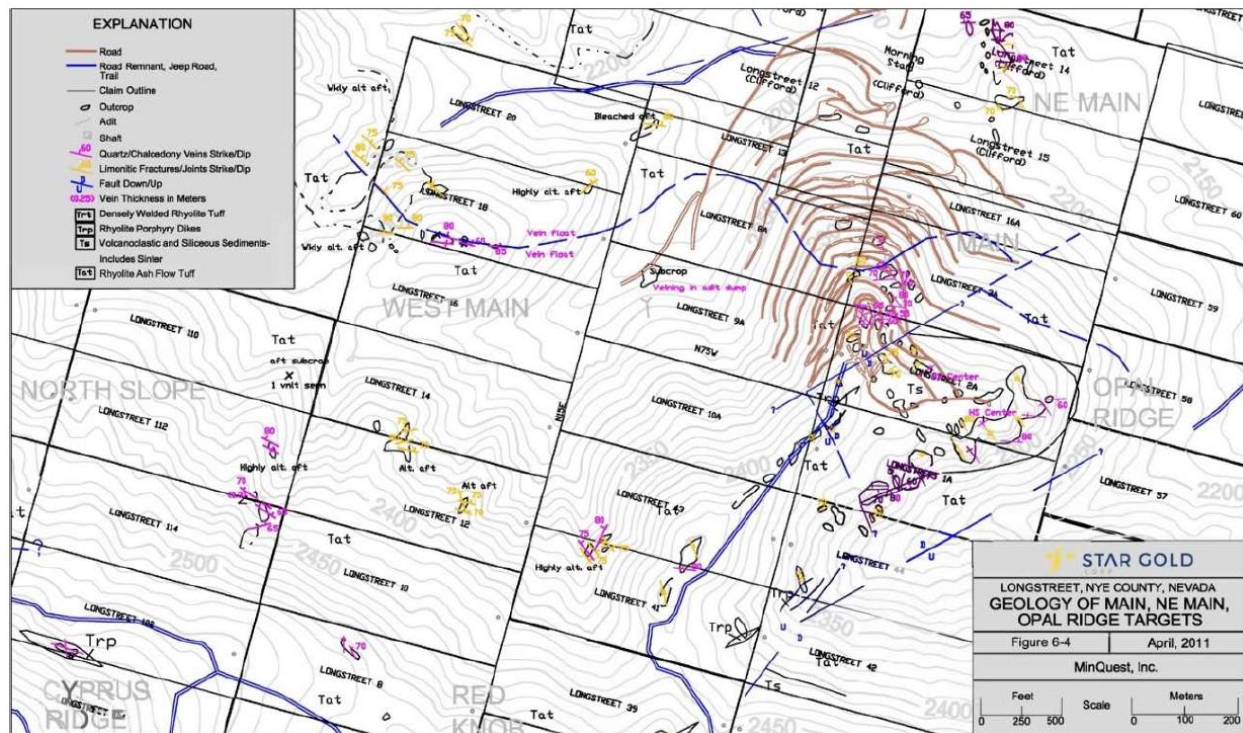


Figure 6-4
Main, NE Main

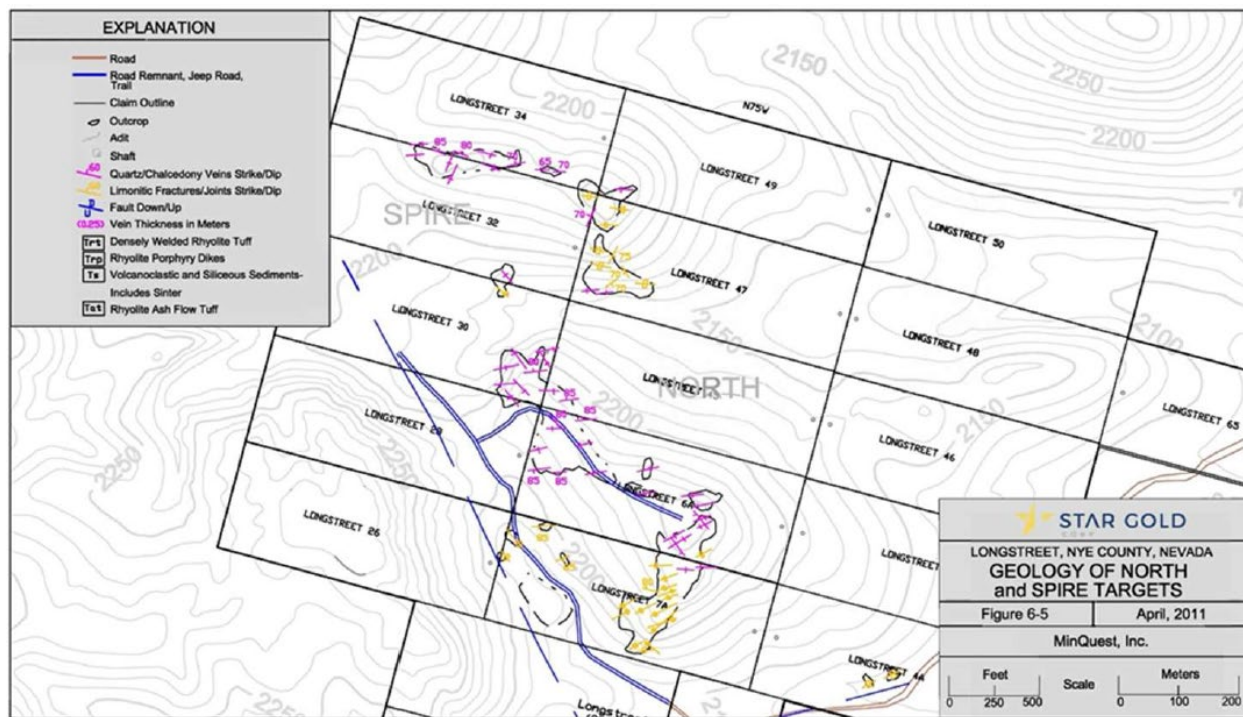


Figure 6-5
North Hill and Spire Targets

7.0 Exploration

The exploration methodology applied in the past by early operators, and during recent exploration programs by Star Gold, has been to evaluate the mineralized zones by drilling, and determine favorable areas for epithermal Au-Ag mineralization of the host poorly to moderately welded tuffs, which are moderately altered and brecciated. To date, at least 107 lithogeochemical (rock chip) samples have been collected, and approximately 422 mostly RC drill holes have been completed by various operators on the property.

In December 2009, Star Gold entered into a property option agreement with Messrs. Kern and Duerr of MinQuest to earn a 100% interest in the Longstreet property, and commenced a systematic exploration program, including 26,885 ft. of RC drilling in 71 RC holes, 1,295 ft. of diamond drilling in 4 holes, and lithogeochemical sampling, which was completed in years 2011 - 2014. All of the drilling by Star Gold was done on the Main Zone except for 3 RC holes drilled in the North target area.

Figure 7-1 shows the exploration target areas surrounding the Main Zone.

7.1 Lithogeochemical Sampling

Upon signing the option agreement with MinQuest, Star Gold commenced a program of sampling mineral showings at several target areas and structures on the Longstreet property. Star Gold contracted MinQuest to carry out this work. MinQuest crews sent the samples to ALS in Reno, Nevada, for Au and Ag assays. Results from this program indicate that, with few exceptions, the gold and silver values in surface samples show moderate to good correlation, with a ratio of approximately 1Au:10Ag, as discussed in Section 5.0 – History. Figure 7-2 shows the location of the lithogeochemical sampling. Detailed tables of the lithogeochemical sampling can be found in the December 8, 2025 Technical Report

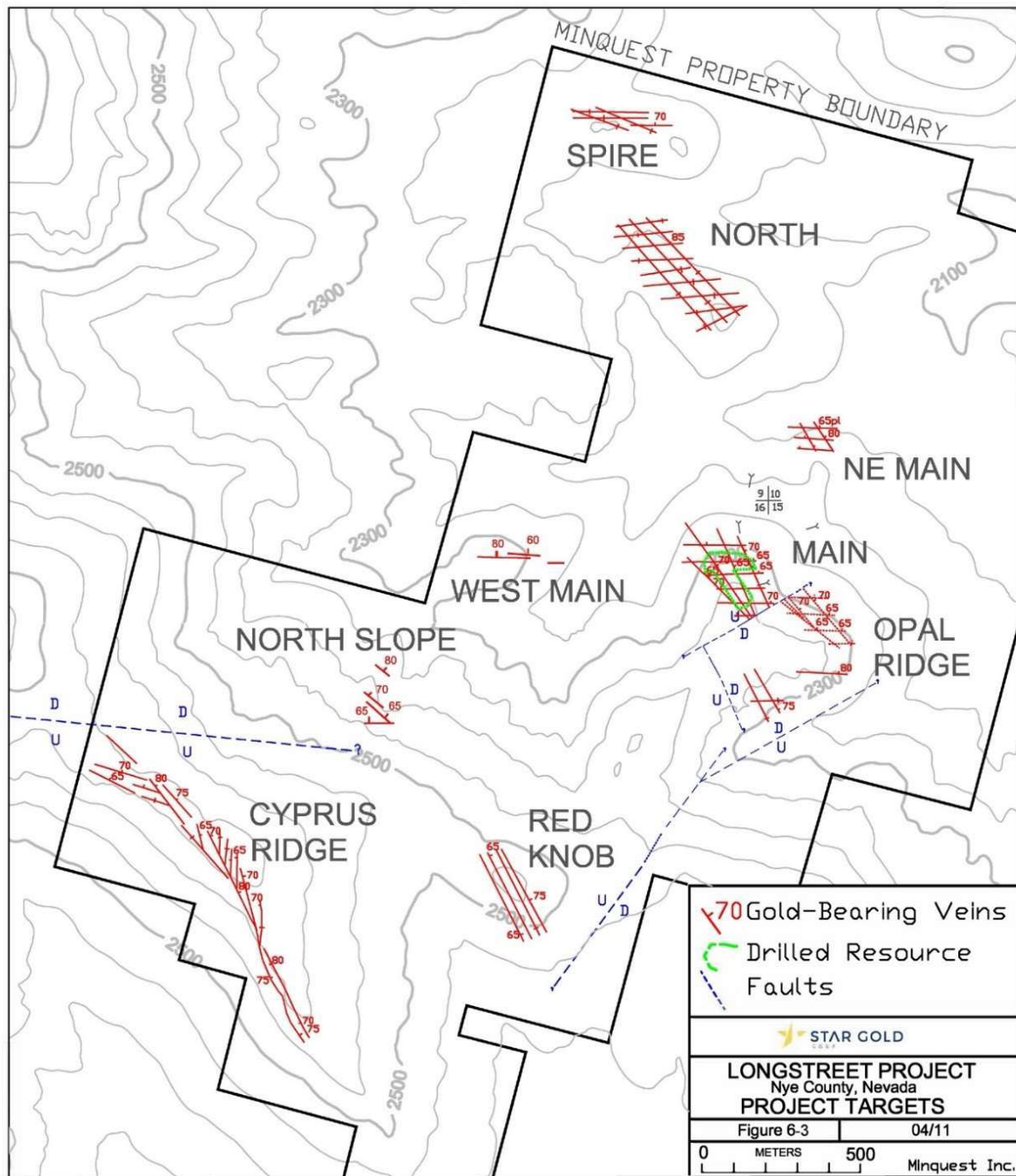


Figure 7-1
Exploration Target Areas for the Longstreet Gold Project

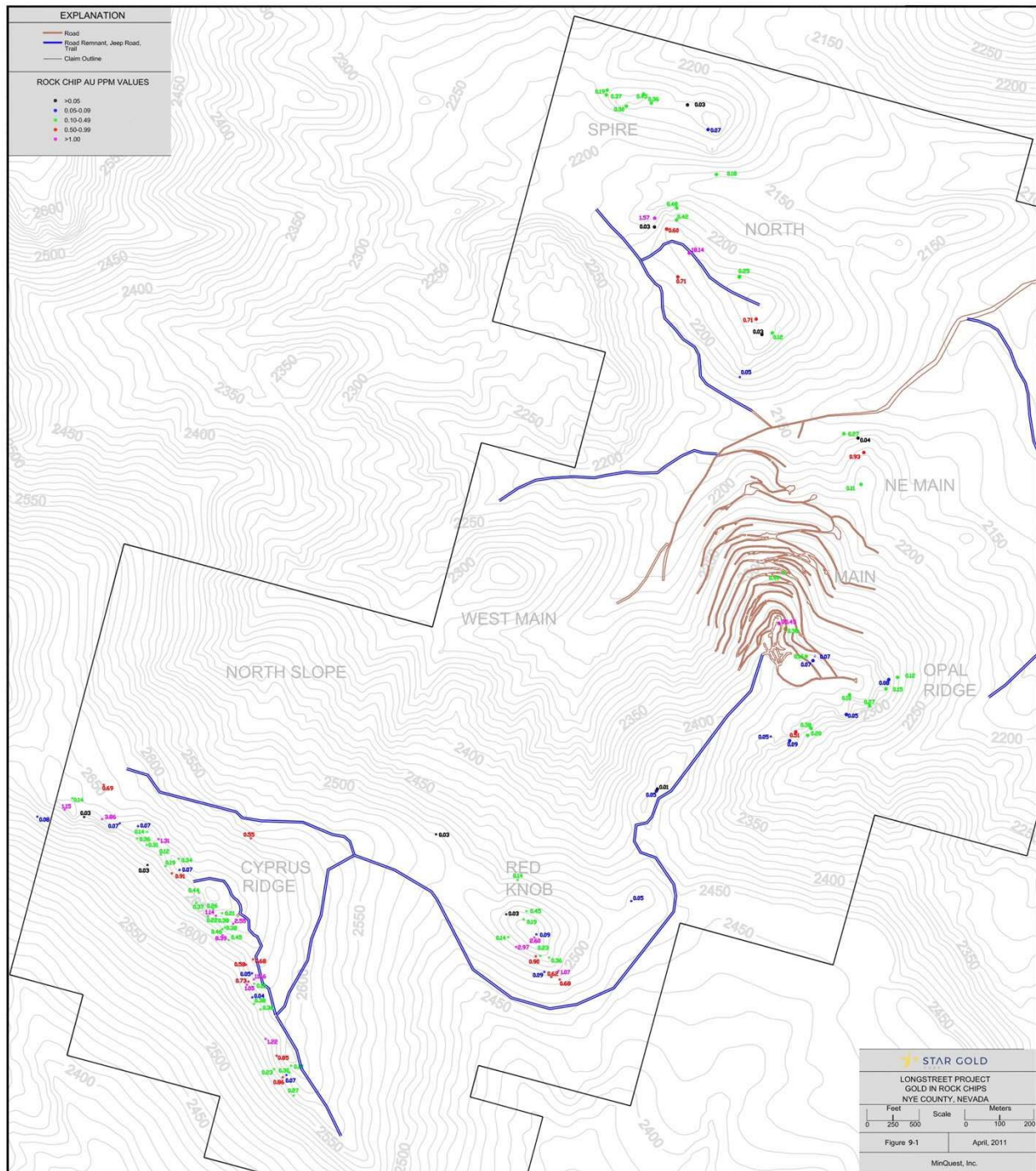


Figure 7-2
Lithogeochemical Sampling Locations

7.1.1 Opal Ridge

The Opal Ridge Zone occurs in a detached fault block located along a ridge at a mean elevation of 7,675 feet. This zone is adjacent to and southeast of the Longstreet Main zone mineralization (Figure 7-1). Bounding faults trend northeast and geological observations indicate an apparent 200-foot vertical displacement and a northward displacement of less than 30 feet for this block. The bedrock consists of moderately welded ash-flow tuff and epiclastic sediments capped by an opalized sinter apron. Quartz-adularia sheeted vein systems and limonitic fractures occur over an area measuring approximately 650 feet by 650 feet. Both steeply dipping northeast and northwest-trending vein systems with limonitic fractures are present.

Two vertical RCHs (S-1 and S-2) completed in this area by Naneco in 1984 reported anomalous Au and Ag values over 400 feet (Liedtke, 1984). 15 surface samples collected by MinQuest in 2002 have reported values similar to the drill intersections. However, the presence of opal (sinter) at surface implies that physiochemical conditions appropriate for the deposition of Au and Ag mineralization will be located at some depth beneath the sinter.

7.1.2 NE Main Zone

The NE Main Zone is located at a mean elevation of 7125 ft, approximately 1500 feet northeast of the Longstreet Mine (Figure 6-4). The mineralization is poorly exposed over an area measuring about 500 ft by about 230 ft along a 340° axial direction, and has not been drill tested. Sheeted vein systems and limonitic fractures trend both 90° and 315° with steep dips to the north and southwest, respectively. Preliminary rock chip sampling yielded a best value of 0.93 ppm Au and 20.6 ppm Ag.

7.1.3 North Zone

The North Zone is located at a mean elevation of 7350 ft about 3300 feet north-northwest of the Longstreet Mine (Figure 6-5). An east-northeast striking fault of unknown character separates this mineralization from the Longstreet Mine. Both northwest trending and east to northeast-trending vein systems and related limonitic fractures are present in outcrops. The gold mineralization is known over an area measuring 1800 ft by 1100 ft. MDA (1988) reports that 22 vertical RCs (3700 ft) were drilled at this location, with potentially economic mineralization found in four holes.

MinQuest collected 12 surface samples from the North target area in 2002. Gold values in the range of 0.031 to 18.1 ppm and Ag values in the range of 1.0 to 49.6 ppm are recorded at this location. The highest Au value is from an east-west trending sheeted quartz vein system near the northwestern end of the zone.

The gold values recorded in the RCs and the surface samples indicate that the area contains potentially economic mineralization.

7.1.4 Spire Zone

MinQuest visited the Spire Zone area in 2002 and collected eight rock samples. The analytical results indicate gold values in the range of 0.031 to 0.45 ppm with Ag values in the range of 1.0 to 20.0 ppm over an area measuring approximately 1000 ft by 150 ft along an east-trending ridge. The oxidized mineralization is hosted in an east-northeast-trending, steeply north dipping to vertical sheeted quartz vein system. Northwest-trending veins are present, but not abundant.

7.1.5 Red Knob

The Red Knob Zone is located between 8250 ft and 8400 ft elevation about 4600 ft south-southwest of the Longstreet Mine (Figure 7-1). The mineralization occurs in a well-developed sheeted quartz-adularia vein system that trends northwest and dips steeply both northeast and southwest. Liedtke (1984) reported a best grab sample assay of 4.1 ppm Au from this location. Quartz vein material > 3 feet in width was noted by MinQuest in 2002 in talus at this location. A best gold value of 2.96 ppm Au in a vein sample hosted in Tat is reported from the 16 samples collected in this target area by MinQuest.

7.1.6 Cyprus Ridge Zone

The Cyprus Ridge Zone is located between elevations of 8400 and 8850 ft. along the crest of a southwest sloping ridge approximately 1.5 miles southwest of the Longstreet Mine. The Tat unit hosts quartz veins up to 15 feet thick in the southeastern part of the northwest trending sheeted vein system, which measure more than 1.7 miles long by 150 feet wide. The quartz vein system is described as a hanging wall vein dipping steeply northeast. Conjugate veins developed in the hanging wall are parallel structures with southwest dips. In the northwestern part of the vein system the hanging wall vein is supplanted by northwest-trending northeast dipping, and an east-trending north-dipping quartz vein system. MinQuest mapped the intricate vein system in 2002 and collected 41 surface samples. A best gold value of 11.2 ppm is recorded from this target area.

7.1.7 North Slope

The North Slope Zone is situated approximately 0.6 miles west-southwest of the Main Zone in the western part of the Longstreet property. A number of northwest trending and steeply to moderately

northeast dipping quartz veins (up to 3 feet thick) outline an area 650 feet long and 325 feet wide. This area has received little geological investigation and no drill testing in the past.

7.1.8 West Main

The West Main Zone is situated approximately 0.3 miles west of the Main Zone in the western part of the Longstreet property. A number of east trending and steeply north dipping sheeted quartz veins outline an area 650 feet long and 160 feet wide. This area has received some geological investigation in the past, as evidenced by old workings, but no drill testing of targets.

7.2 Drilling

More than 430 RC holes (approximately 84,000 ft. or 25,600 m) and 4,295 ft. (1,310 m) of diamond drilling in 19 holes have been completed by Star Gold and previous operators in the general area of the Longstreet property during the past 32 years. Most of the drilling was done on the Main Zone and includes 16 RC holes completed by Star Gold in 2011, 23 RC drill holes, and four diamond drill holes completed by Star Gold in 2012. Three of the 2012 RC holes tested the North Zone, and one RC hole tested the Opal Ridge Zone. For the 2011, 2012, and 2013 campaigns, the drilling contractor was O'Keefe Drilling Company, Inc. (O'Keefe) of Butte, Montana. During the historic and recent RC drilling campaigns, drill chips were retrieved and sent for assays. Figure 7-3 shows the drill hole collar locations.

Of the 19 diamond drill holes and 437 RC holes completed on the six target areas, the majority (414) of the holes intersected significant mineralization of more than 0.2 g/t Au and 1.0 g/t Ag over intervals ranging from more than 10 ft to approximately 275 ft.

7.3 RC Drilling

To date, approximately 364 RC holes by previous operators and 68 RC holes by Star Gold have been completed on the Longstreet property during the past 32 years. The drilling contractor, methodology, or procedures of sampling in previous campaigns are not available at this time. During the 2011, 2012, and 2013 drilling programs, the RC drilling contractor was O'Keefe. In 2014 Star Gold used Harris Drilling. Star Gold used similar truck-mounted mud rotary equipment (with hole diameters ranging from 4 ¾ in. to 5 ¾ in.) using local Reno, Nevada based drilling contractors. The procedures used during the RC drilling programs are summarized, as follows:

- The collar locations of all drill holes were surveyed and marked in the field. A Geographic Positioning System (GPS) instrument was used to mark the collar locations of drill holes. This survey was carried out by MinQuest.

- Lithologic logging of drill core and geotechnical observations were provided by Mr. David Eastwood, Star Gold contract geologist, on loan from MinQuest. Logging is done by depicting all down-hole data and assay values. All information is recorded on previously prepared logs using LOGPLOT software developed by RockWare, Inc. (RockWare) of Denver, Colorado. This includes marking:
 - Lithologic contacts.
 - Descriptive geology.
 - Intensity of various alteration types.

Agnerian and the qualified Person for this section are of the opinion that logging procedures for RC holes used by Star Gold are in keeping with industry standards.

7.3.1 Diamond Drilling

To date, nineteen diamond drill holes have been completed on the Longstreet property; eight holes by Keradamex/E&B, seven holes by Cyprus on the Cyprus Ridge target area, and four holes by Star Gold on the Main Zone (Prenn, 1988, Nolan 2012, and Kern, 2012a). The drilling contractor or the size of drill core recovered for Keradamex/E&B in 1980, and Cyprus during the 2003-2005 drilling program, are not available.

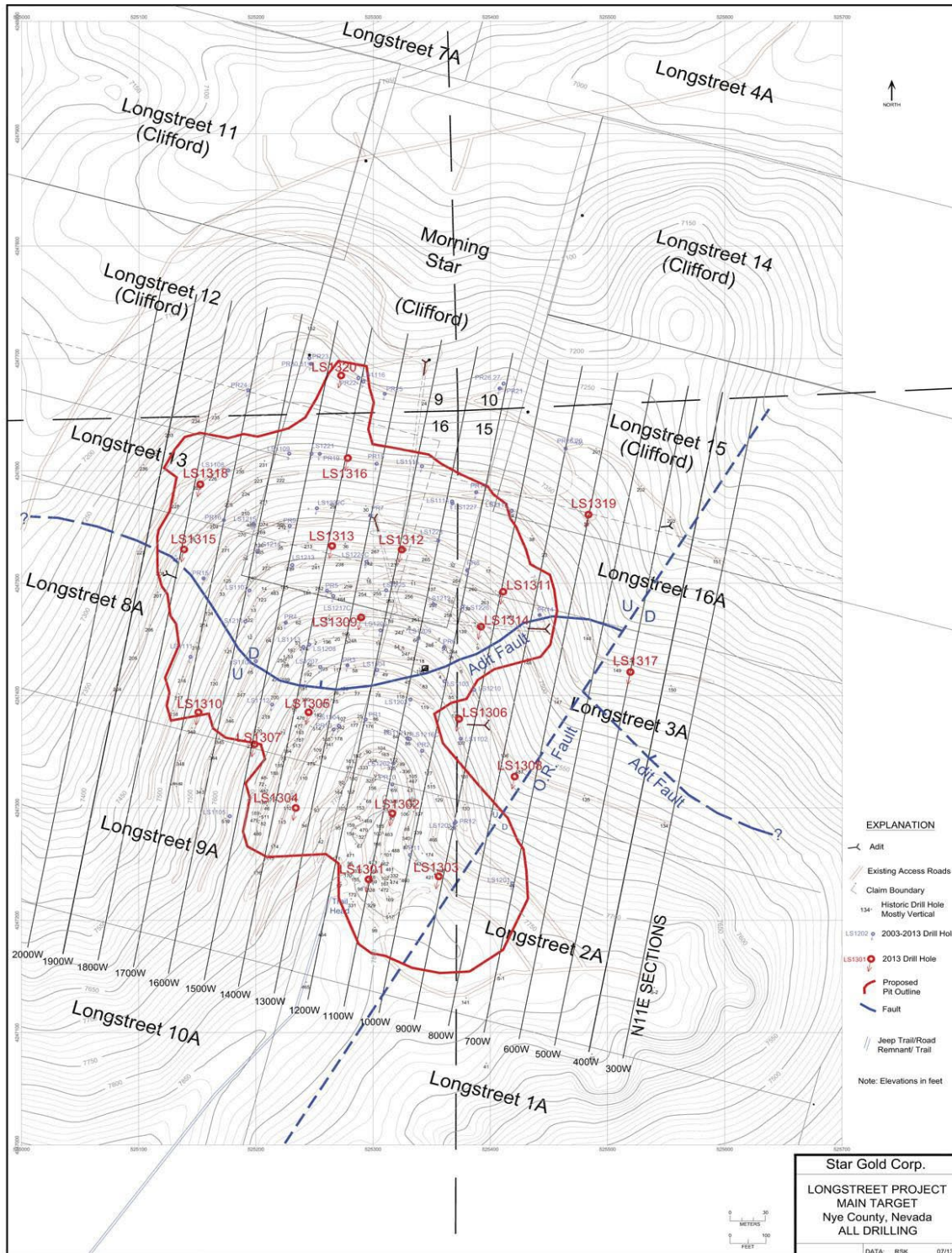
During the 2012 exploration program, diamond drilling was done by O'Keefe. The drilling crew worked two 12-hour shifts and recovered NQ core throughout the hole. Drill hole collar coordinates and collar elevations were determined using GPS instrument. For down-hole surveying the holes, Star Gold used the International Directional Services (IDS) system from Elko, Nevada, which is similar to the digital Maxibore instrument. Measurements of the hole inclination and azimuth were taken continuously down the hole, thus a complete record of down-hole measurements is available at intervals of five feet.

The procedures used during the diamond drilling programs are summarized as follows:

- The collar locations of all drill holes were surveyed and marked in the field using GPS. Agnerian notes that in future programs the collar locations need to be corrected by a topographic survey to allow for a more precise topographic control for resource estimation and development of the Main Zone deposit.
- Lithologic logging of drill core and geotechnical observations was provided by Mr. Richard Kern, President of MinQuest. Logging was done by Mr. David Eastwood, contract geologist on loan from MinQuest. Logging was done by depicting all down-hole data including assay values. Similar to RC holes, drill holes are logged using LOGPLOT software. All information was recorded on handwritten logs. This includes marking:
 - Lithologic contacts.

- Descriptive geology.
- Intensity of various alteration types.
- Structural features, such as foliation, fracture and brecciated zones
- Core angles.
- Core diameter.
- Downhole inclination.
- Percent core recovery record.
- Recording geotechnical data, such as RQD measurements.
- Down-hole survey using the IDS system survey tool.

Agnerian reviewed drill logs of three 2012 diamond drill holes at the Reno, Nevada, core storage area during the second visit. Agnerian is of the opinion that the lithologic logging procedures are somewhat in keeping with industry standards. Agnerian, nevertheless, recommends density measurements to be carried out at regular intervals throughout the drill holes during future campaigns. Agnerian also recommends maintaining a photographic record of the core with a digital camera. Photographs should be taken of all exploration drill core and key information must be summarized in a digital database.



Source: Kern, 2012a.

Note: Proposed pit outline is based on 2012 drilling results.

Figure 7-3
Drill Hole Location Map

8.0 Sample Preparation, Analysis and Security

The following section has been re-printed from the Preliminary Economic Assessment of the Longstreet Gold Project dated January 12, 2021 by A-Z Mining Professionals Limited (AMPL). The Qualified Person for this section concurs with the findings by AMPL.

Information on sampling procedures during geochemical sampling and RC drilling programs carried out by Keradamex/E & B, Minerva, Naneco, and REM/Harvest Gold in the 1980s and 1990s, is not available at this time. Sampling was done at 2 ft., 5 ft., and 10 ft. intervals. The bulk of the sampling, however, was done at 5 ft. intervals, and sampling was done throughout the entire hole. In general, samples from earlier drilling were assayed mainly for gold and silver. The exact methods of gold and silver determinations, however, are not available.

8.8 RC Drilling

During the recent work by Star Gold, sampling of drill chips was done by MinQuest technical personnel contracted by Star Gold. Sampling procedures during the drilling programs included splitting the drill chips using a two-way wet sample splitter, at 5 ft. sample intervals, under the supervision of the Project Geologist. Material from one half of the sample (A sample) was put in securely sealed bags and sent to the ALS sample preparation laboratory in Reno, Nevada. Samples were numbered on the sample bags, according to the drill hole number and footage of the hole. The other half of the sample (B sample) was kept at the site for future reference. The chain of custody of logging and sampling was the responsibility of the Project Geologist. Details of sampling are provided in Item 31, Appendix 2 (Section 27.2).

Agnerian notes that sample numbers are assigned in accordance with the footage of the drill hole, e.g., for sample from 20 ft. to 25 ft. in Hole LS-1205, the sample number is LS-1205 20-25. Agnerian is of the opinion that the sample numbers must be unique (commonly those with the book of sample tags either purchased independently or provided by the laboratory) so that the hole number is not disclosed.

8.9 Diamond Drilling

Diamond drill core samples were cut longitudinally using a diamond saw at 5 ft. intervals, and the numbering system was the same as for the RC holes.

Based on its review of exploration data and sampling procedures, Agnerian makes the following recommendations:

- For RC holes, water resistant sample tags, bearing unique numbers, are inserted in the sample bags in order to maintain the integrity of the samples, and avoid any possibility of assay information being disclosed to unauthorized people.
- For diamond drill holes, one of the sample tags (of a book of sample tags) is fastened at the end of each sample in the core boxes. This will allow for easy identification of samples in the core box.

During the 2011, 2012, 2013, and 2014 exploration programs by Star Gold, samples were sent to the ALS laboratory in Reno, Nevada, where samples were crushed and ground. Sample pulps were then assayed for gold and silver using the one assay-ton (30 g sample) fire assay method and the atomic absorption (AA) finish. Samples that contained more than 10 g/t Au were re-assayed by the fire assay method and gravimetric finish. In addition, at the discretion of the Project Geologist, samples adjacent to intervals with significant gold, were re-assayed.

Agnerian notes that ALS is an ISO 9001 recognised laboratory and the procedures used at ALS Chemex laboratories are similar to those used at many commercial laboratories in Canada. In particular, they include:

- Drying the split sample and preparing by particle size reduction to produce a homogeneous sub-sample, which is representative of the original sample.
- Crushing the split sample to 10 mesh and grinding it to 200 mesh, 85% passing <75µ.
- Cleaning the pulverizer after each sample using cleaner sand to avoid cross contamination of samples.
- Determinations of the gold and silver contents are carried out using the Aqua Regia Digestion Method, including sulphuric acid, nitric acid, and hydrochloric acid.
- Sample size is generally <250 g.

Agnerian is of the opinion that the sample preparation and assay procedures at ALS are in keeping with industry standards.

9.0 Data Verification

Information on data verification during exploration programs carried out by Keradamex/E & B, Minerva, and REM/Harvest Gold in the 1980s and 1990s is not available. During its extensive RC drilling program, Naneco carried out a twin drilling program and results are discussed below. During the recent exploration programs, the quality and reliability of the Star Gold data were reviewed and verified by Mr. Richard Kern of MinQuest and Mr. Paul Noland, the author of the 2012 and 2025 technical report, with cooperation by Mr. Scott Jenkins of Star Gold. In addition, the Qualified Person for this section has reviewed and verified the data and is of the opinion that the data used to calculate the resource in this report are adequate and reliable.

9.1 Naneco Twin Drilling Program

During its drilling campaigns from 1984 to 1987, Naneco carried out a program of twin drilling. This comprised of ten twinned holes, as shown in Table 9-1 and Figure 9-1.

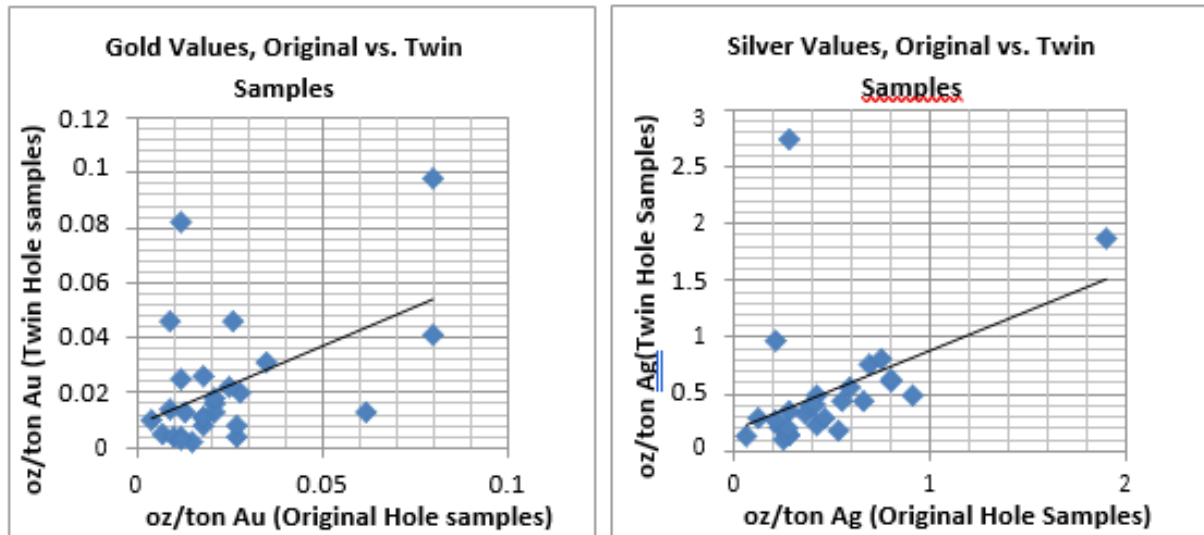
Table 9-1
Naneco Twin Drilling Results, Main Zone Drilling 1987

Original Drill Hole					Twin Drill Hole				
Hole ID	Interval (ft.)		Au opt	Ag opt	Hole ID	Interval (ft.)		Au opt	Ag opt
	From	To				From	To		
LRH-1	0	35	0.018	0.42	LRH-53	0	35	0.008	0.23
LRH-1	35	105	0.080	0.80	LRH-53	35	105	0.041	0.61
LRH-1	105	115	0.027	0.28	LRH-53	105	115	0.004	0.14
LRH-1	0	35	0.018	0.42	LRH-250*	0	35	0.026	0.49
LRH-1	35	105	0.080	0.80	LRH-250*	35	105	0.098	0.63
LRH-1	105	115	0.027	0.28	LRH-250*	105	115	0.008	0.14
LRH-3	0	30	0.021	0.36	LRH-248*	0	30	0.016	0.33
LRH-3	30	70	0.012	0.25	LRH-248*	30	70	0.004	0.10
LRH-3	70	180	0.035	0.69	LRH-248*	70	180	0.031	0.76
LRH-5	0	50	0.028	0.23	LRH-247*	0	50	0.020	0.21
LRH-5	50	65	0.007	0.27	LRH-247*	50	65	0.005	0.14
LRH-5	65	90	0.198	0.53	LRH-247*	65	90	0.029	0.18
LRH-5	90	120	0.021	0.75	LRH-247*	90	120	0.013	0.81
LRH-5	120	195	0.010	0.59	LRH-247*	120	162	0.004	0.56
LRH-7	0	130	0.026	0.46	LRH-246	0	100	0.046	0.29
LRH-8	0	135	0.021	0.39	LRH-243*	0	155	0.018	0.40
LRH-8	135	155	0.025	1.90	LRH-243*	155	175	0.022	1.87
LRH-8	155	190	0.015	0.66	LRH-243*	175	190	0.002	0.44
LRH-18	0	15	0.013	0.41	LRH-245	0	15	0.013	0.41
LRH-18	40	95	0.018	0.55	LRH-245	40	95	0.011	0.44
LRH-50	0	50	0.009	0.21	LRH-244	0	50	0.014	0.27
LRH-50	50	65	0.012	0.28	LRH-244	50	65	0.082	2.74
LRH-50	0	50	0.009	0.21	LRH-21	0	50	0.046	0.97
LRH-50	50	65	0.012	0.28	LRH-21	50	65	0.003	0.35
LRH-51	0	20	0.012	0.12	LRH-249*	0	20	0.025	0.29
LRH-51	20	80	0.018	0.27	LRH-249*	20	80	0.011	0.20
LRH-51	80	100	0.004	0.06	LRH-249*	80	100	0.010	0.13
LRH-51	100	135	0.062	0.91	LRH-249*	100	135	0.013	0.49

Total length	1,260	0.031	0.49	Total length	1,197	0.027	0.49
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Source: Prenn, 1988. Note: Holes with asterisk (*) were sampled at 2 ft. intervals.

Results of the twin drilling by Naneco indicate that except for one pair of values, the silver assays from twin holes were, in general, comparable to the ones from the original holes. The gold assays, on the other hand, show poor correlation. This may indicate either poor sampling or irregular gold distribution in the host rocks.



Source: Prenn, 1987

Figure 9-1
Naneco Twin Drilling Results (1987)

9.2 Assay Quality Assurance and Quality Control

9.2.1 Data Verification by MinQuest

Data verification is done by MinQuest personnel. These include comparing the list of samples sent to the laboratory and sampling results from the laboratory. In particular, the MinQuest quality assurance and quality control (QA/QC) program includes:

- Insertion of a blank and two standards (pulp) of known gold and silver concentration at the frequency of approximately one in every 20 samples.
- Insertion of a B sample, if unexpected high or low values are encountered, at the frequency of one in every 30 samples, and sent to the laboratory together with the A samples.
- Duplicate assays, i.e., re-assay of values higher than 1 ppm Au.
- Analysis of assay results of the standards.

9.2.1.1 Duplicate Assays

Results of the duplicate assays indicate good assay reproducibility, i.e., except for a few assays, the original and duplicate assays are comparable to the original assays, as shown in Figure 9-2 and Figure 9-3.

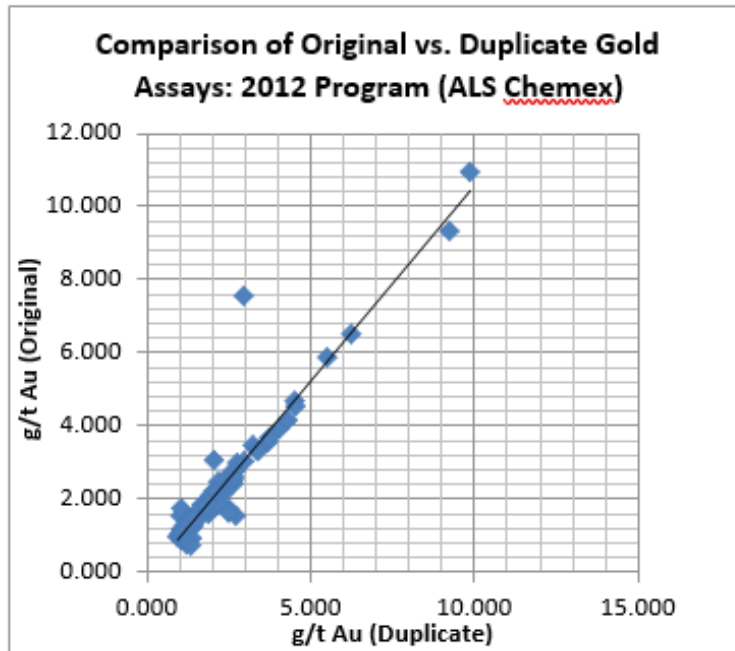


Figure 9-2
Duplicate Gold Assay Results

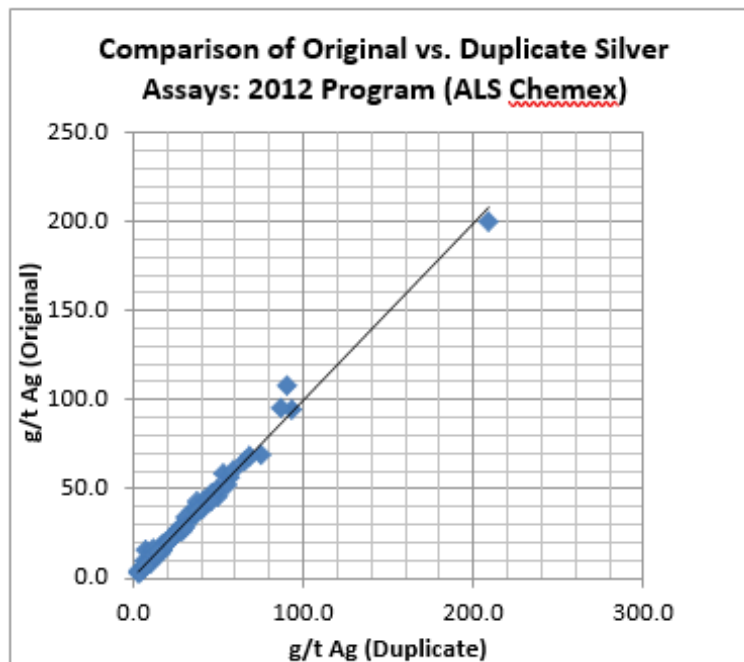


Figure 9-3
Duplicate Silver Assay Results

9.2.1.2 *Standards and Blank*

MinQuest inserted two standards (S2 and S3) with expected concentrations of 0.181 g/t Au and 2.099 g/t Au, respectively, and a blank sample with expected nil value. Results are shown in Figures 9-4.

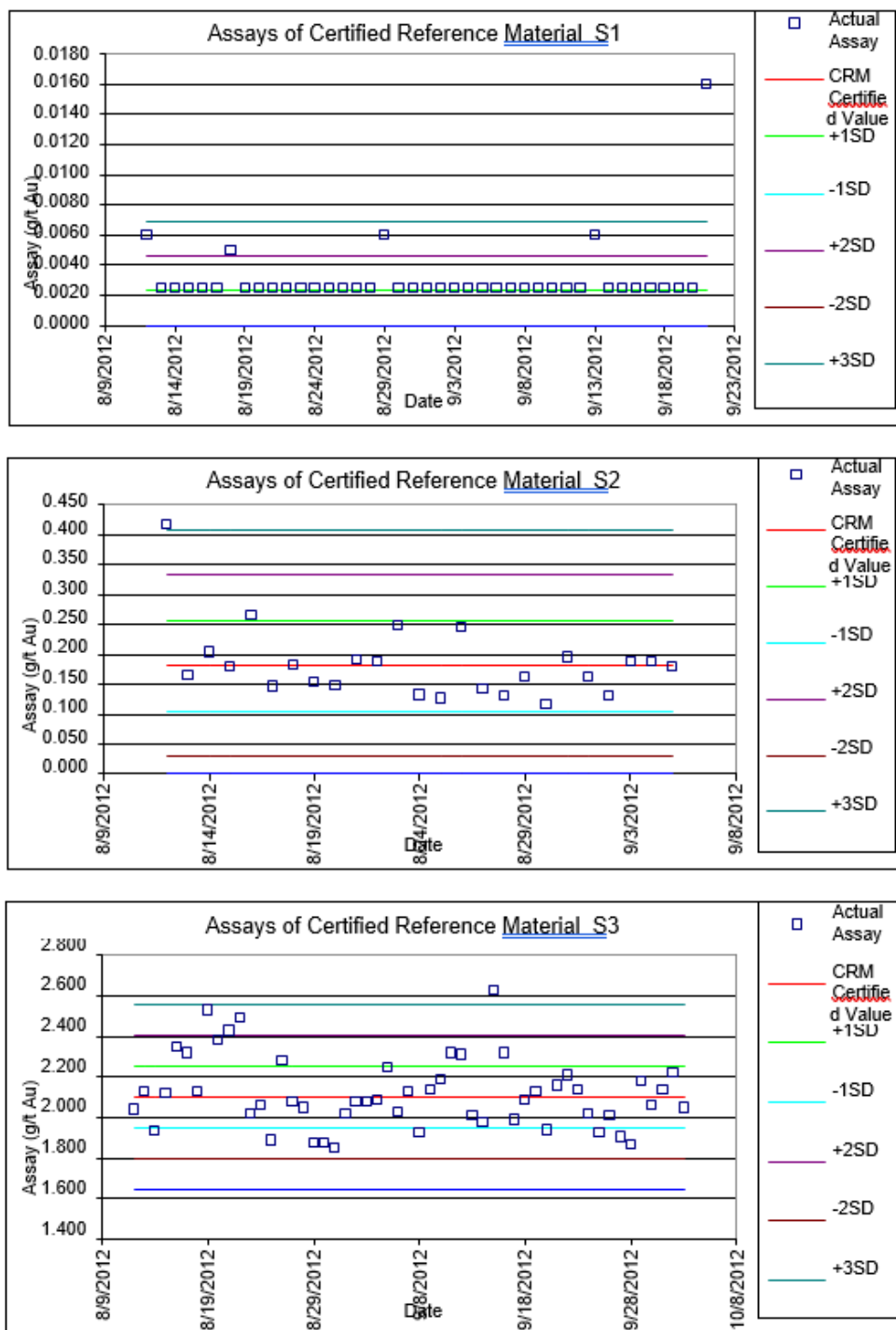


Figure 9-4
Standard S1 (Blank), S2, and S3 Gold Assay Results

Results of the MinQuest check assay program indicate that:

- For Standard S1 (Blank), all but five of the forty-one determinations were below the detection limit of 0.005 ppm Au. Four samples have values ranging from 0.005 ppm Au to 0.006 ppm Au, and one sample had a value of 0.016 ppm Au, which may reflect data entry error, or actual low grade gold value.
- For Standard S2, all but one of the 25 determinations had values within one standard deviation of the expected value of 0.181 ppm Au.
- For Standard S3, all but one 53 determinations were within three standard deviations and all but four were within two standard deviations of the expected value of 2.099 ppm Au.

For future drilling programs, Agnerian recommends that Star Gold request results of ALS Minerals' internal QA/QC program on internal and external standards. Agnerian also recommends that Star Gold carry out a program of internal check assays on standards and blanks with each batch of samples. These should include:

- Analysis of laboratory assays to check for unusual high and low values, and comparison of A and B (duplicate) sample values.
- Check assays on 5% of all previous assays.
- Check assays using three standards and blank determinations, including:
 - High Grade Standard, with an expected value in the order of 1.5 ppm Au.
 - Medium Grade Standard, with an expected value in the order of 0.5 ppm Au.
 - Low Grade Standard, with an expected value in the order of 0.2 ppm Au.
 - Determinations on Coarse Blank samples.
 - Determinations on pulps of blank samples.

Each batch of Star Gold's samples should contain 45 samples, including 40 regular samples, one high grade standard, one medium grade standard, one low grade standard, one coarse blank, and one fine blank. The procedure and methodology of inserting samples of standards and blanks with regular sample batches are as follows:

- Samples one to ten comprise regular samples.
- Sample No. 11 is a low-grade standard.
- Sample Nos. 12 to 21 comprises regular samples.
- Sample No. 22 is a fine blank.
- Sample No. 23 is a coarse blank.
- Sample Nos. 24 to 33 comprises regular samples.
- Sample No. 34 is a medium grade standard.
- Sample Nos. 35 to 44 comprises regular samples.
- Sample No. 45 is a high-grade standard.

In terms of accepting or rejecting check assay data, Star Gold should use the following criteria:

- If the assays for one standard fall within two standard deviations, and those for another standard within three standard deviations, then the batch results are accepted.
- If the assays for one standard fall within three standard deviations, then the standard is considered as a failure.
- If the assays for two or more standards fall within two to three standard deviations, then the batch results are rejected.
- If the assays for a standard and the nearest blank in a batch are failures, then the batch results are rejected.
- If the assays for both blanks (coarse and fine) are beyond the warning line, then the batch results are rejected.

Agnerian considers the above approach to be important in view of the relatively low grade of the Longstreet deposit.

9.2.2 Check Assay Program at ALS Minerals

ALS carried out regular check assays on samples of RC chips and diamond drill core submitted by MinQuest. Table 9-2 lists the various standards used by ALS, and results of the gold values of the ALS check assay program for three of the standards (MG-12, OxN 92. And OxK 95) are shown in Figure 9-5. Detailed results of various standards used for the 2011 and 2012 drilling programs are provided in the AMPL Technical Report dated January 12, 2021.

Table 9-2
Statistics of Check Assay Programs, Main Zone Drilling 2011-2012

Standard	Statistic (ppm Au)						
	Number	Expected Value Value	Maximum	Minimum	Average	Median	Standard Deviation
S1 (Blank)	41		0.016	0.0025	0.0025	0.0031	0.0023
S2	25		0.418	0.117	0.184	0.179	0.062
S3	53		2.630	1.850	2.121	2.090	0.177
MG-12	56		0.949	0.834	0.892	0.891	0.021
OREAS 65A	20		0.547	0.513	0.526	0.523	0.009
OREAS 16B	5		2.350	2.130	2.220	2.210	0.080
OREAS 503	38		0.731	0.664	0.694	0.695	0.015
OREAS 67A	1		2.210	2.230	2.210	2.210	
OREAS 68A	25		4.140	3.470	3.886	3.920	0.152
OREAS 501	49		0.219	0.172	0.207	0.208	0.008
OxN 92	75		8.090	7.190	7.769	7.760	0.175
OxK 95	48		3.740	3.470	3.564	3.560	0.062
OxP 61	2		14.950	14.400	14.675	14.675	0.389

OxC 88	38		0.215	0.188	0.204	0.205	0.005
OxL 78	20		6.140	5.700	5.945	5.960	0.156
ALS Blank	150		0.008	0.0025	0.0026	0.0025	0.0006
Standard A	33		2.440	0.005	0.954	0.245	0.986

Source: ALS Chemex, 2012

Notes:

1. S1, S2, S3 and A standards are inserted by MinQuest.
2. Values of 0.0025 ppm Au for S1 and ALS blank are half of the detection limit of 0.005 ppm Au.
3. All other standards are used by ALS.

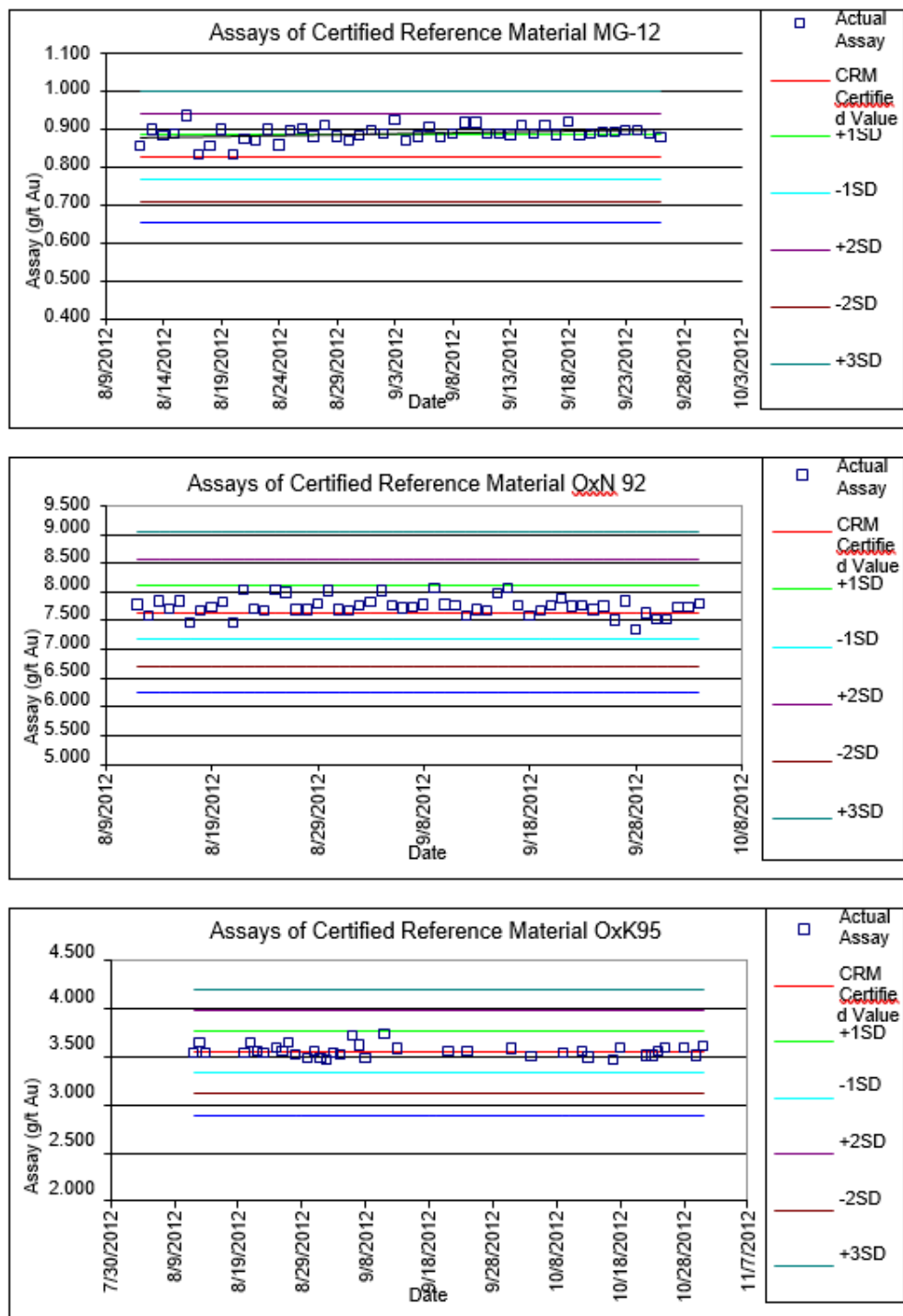


Figure 9-5
Check Assays on Standards by ALS

9.3 Twin Drilling by Star Gold

As part of the 2012 drilling program, MinQuest attempted to verify past drilling results by twin drilling four diamond drill holes, LS-1216C, 1217C, 1222C, and 1224C. Since the collar locations of historic RC holes are only approximate, as there are no pickets or concrete monuments for the old holes, MinQuest spotted holes in the general area of the old holes.

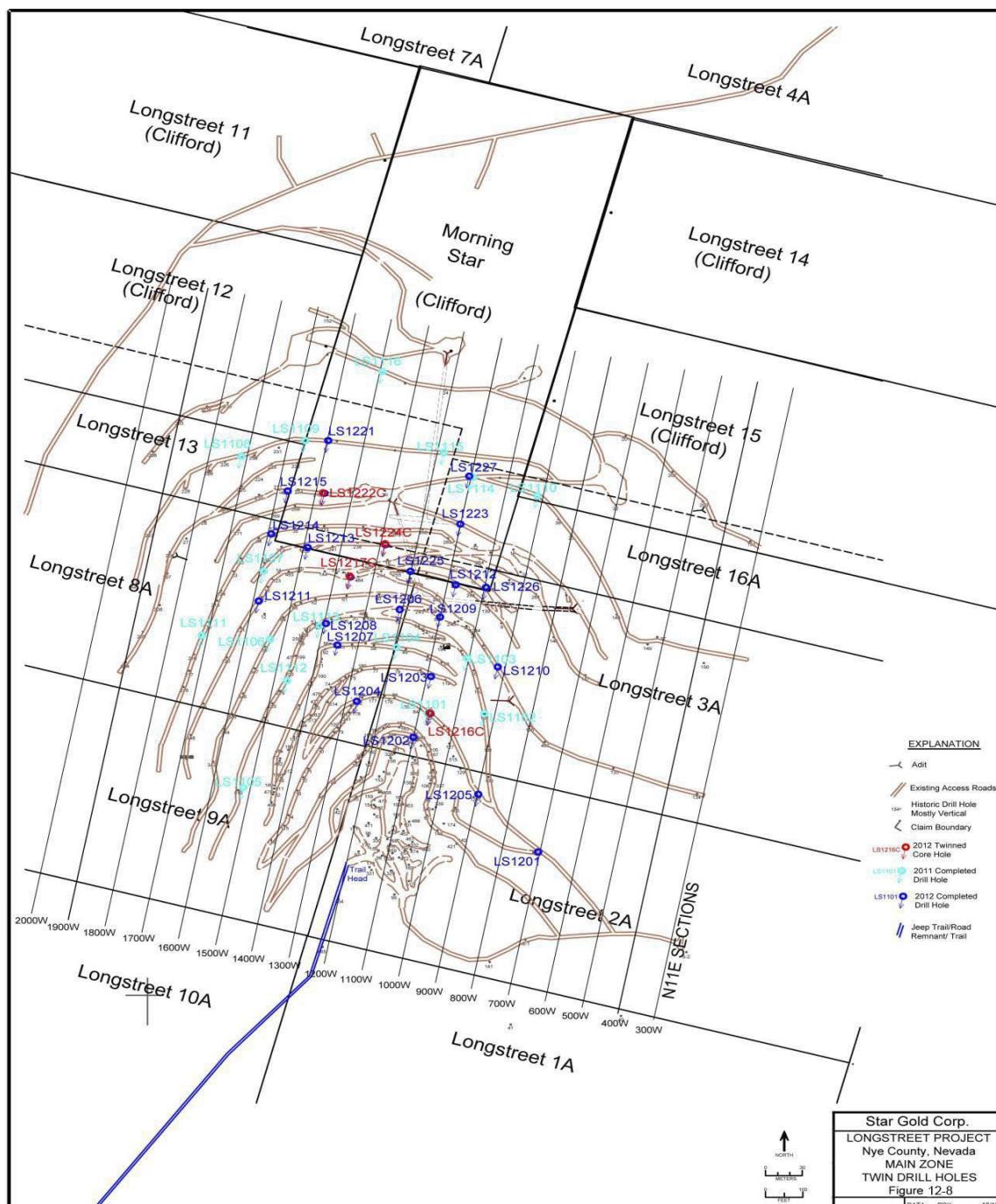
Table 9-3 shows that the average grades of the twin holes are in the same order-of-magnitude as the RC holes.

Table 9-3
Main Zone Twin Drilling Results

Historic RC Holes					2012 Twin Holes				
Hole ID	From (ft)	To (ft.)	Interval (ft)	oz/ton AuEq	Hole ID	From (ft)	To (ft.)	Interval (ft)	oz/ton AuEq
LRH-242	0	5	5	0.008	LS-1224C	105	120	15	0.029
LRH-242	170	175	5	0.006	LS-1224C	150	160	10	0.021
LRH-242	185	195	10	0.009	LS-1224C	170	175	5	0.006
LRH-242	215	300	85	0.072	LS-1224C	195	245	50	0.089
					LS-1224C	250	260	10	0.009
					LS-1224C	290	295	5	0.024
Average LRH-242			105	0.060	Average LS-1224C			95	0.056
PR-5	5	10	5	0.012	LS-1217C	0	15	15	0.016
PR-5	35	85	50	0.029	LS-1217C	30	45	15	0.039
PR-5	95	170	75	0.023	LS-1217C	50	55	5	0.019
PR-5	175	205	30	0.014	LS-1217C	60	80	20	0.059
PR-5	230	240	10	0.077	LS-1217C	85	190	105	0.024
					LS-1217C	200	205	5	0.019
					LS-1217C	220	225	5	0.012
					LS-1217C	230	235	5	0.016
					LS-1217C	250	260	10	0.017
Average PR-5			170	0.026	Average LS-1217C			185	0.027
PR-9	0	15	15	0.014	LS-1222C	5	35	30	0.013
PR-9	20	30	10	0.018	LS-1222C	60	65	5	0.012
PR-9	115	180	65	0.059	LS-1222C	80	85	5	0.009
PR-9	185	220	35	0.021	LS-1222C	125	140	15	0.014
					LS-1222C	145	220	75	0.025
Average PR-9			125	0.035	Average LS-1222C			130	0.020
LS-1101	0	10	10	0.093	LS-1216C	5	10	5	0.031
LS-1101	25	40	15	0.017	LS-1216C	20	55	35	0.013
LS-1101	45	50	5	0.008	LS-1216C	60	85	25	0.013
LS-1101	55	60	5	0.009	LS-1216C	95	110	15	0.026
LS-1101	65	70	5	0.013	LS-1216C	115	195	80	0.018
LS-1101	80	95	15	0.026	LS-1216C	200	235	35	0.021
LS-1101	100	110	10	0.007	LS-1216C	240	245	5	0.009
LS-1101	115	125	10	0.019	LS-1216C	250	255	5	0.007
LS-1101	130	255	125	0.021	LS-1216C	265	270	5	0.005
LS-1101	265	300	35	0.010	LS-1216C	275	280	5	0.006
LS-1101	320	375	55	0.025	LS-1216C	310	315	5	0.010
LS-1101	380	395	15	0.009	LS-1216C	325	330	5	0.006
					LS-1216C	340	345	5	0.005

					LS-1216C	355	360	5	0.006
					LS-1216C	385	390	5	0.007
Average LS-1101			305	0.021	Average LS-1216C			240	0.016

Source: Kern, 2012b.



Source: Kern, 2012a.

Figure 9-6
Twin Drill Hole Locations

9.4 Data Verification by Agnerian

9.4.1 Database Verification

As part of its due diligence on the Longstreet Project, Agnerian reviewed the assay database from the historic as well as the 2011, 2012, and 2013 RC drilling, and the 2012 diamond drilling completed on the property. This included comparison of laboratory certificates and assay data entered by MinQuest staff. Since digital assay data from ALS is downloaded directly from the ALS website into the MinQuest digital logging forms, data entry or transfer errors are eliminated. Hence, the 2011, 2012, and 2013 assay database is free of data entry errors.

Agnerian notes that assay certificates from the historic drilling on the property are not available. Additional discussion on the assay compatibility of the old RC database and the new 2011-2013 RC database is provided in Item 14, Mineral Resources of the Agnerian Technical Report on the Longstreet Gold-Silver Property dated December 15, 2013.

During the second site visit, Agnerian also reviewed the Star Gold exploration results and the methodology of lithologic logging of drill holes by Star Gold contract personnel. Agnerian is of the opinion that, in general, the field practices used by Star Gold are in keeping with industry standards.

9.4.2 Independent Sampling by Agnerian

As a check of previous results, Agnerian collected 15 independent samples; 13 samples from three diamond drill holes (LS-1217C, LS-1222C, and LS-1224C) as well as two grab samples from the portals of two adits, of the Main Zone target area, and sent them to SGS Laboratories, Don Mills, Ontario (SGS), for Au and Ag assays. The gold and silver determinations were done by the fire assay method, with detection limits of 1 ppb Au and 0.3 ppm Ag, respectively.

Results of independent sampling indicate that although there are some minor differences between individual samples, in general, the gold and silver values in the Agnerian samples compare well with the Star Gold assays, with a positive bias with respect to the Agnerian samples, as shown in Table 9-4 and Figure 9-7. This positive bias is caused mostly by one sample (303030), i.e., 5.659 ppm Au (Star Gold) vs. 14.400 ppm Au (Agnerian). The average grade for silver compare well; 14.4 ppm Ag (Star Gold) vs. 15.9 ppm Ag (Agnerian). Results of Agnerian sampling also indicate that there is significant gold mineralization on the Longstreet property. Agnerian, nevertheless, recommends that in future drilling programs, Star Gold send a set of samples (10% of the original samples) to another commercial laboratory for independent check assays.

Table 9-4
Agnerian Independent Sampling Results

Diamond Drill Hole	Star Gold Sample No.	Agnerian Sample No.	From (ft.)	To (ft.)	Interval (ft.)	Star Gold Results		Agnerian Results	
						ppm Au	ppm Ag	ppm Au	ppm Ag
LS-1217C	LS1217C 30-35	303028	30	35	5	0.257	6.2	0.061	3.1
LS-1217C	LS1217C 35-40	303029	35	40	5	2.640	32.4	2.130	33.2
LS-1217C	LS1217C 40-45	303030	40	45	5	0.228	14.8	0.795	19.8
LS-1217C	LS1217C 65-70	303031	65	70	5	5.695	23.3	14.400	24.3
LS-1217C	LS1217C 70-75	303032	70	75	5	1.337	21.4	1.690	33.3
LS-1222C	LS1222C 145-	303037	145	150	5	0.972	3.4	0.955	4.3
LS-1222C	LS1222C 150-	303038	150	155	5	0.469	6.0	0.234	6.3
LS-1222C	LS1222C 155-	303039	155	160	5	0.336	8.4	1.060	8.2
LS-1222C	LS1222C 160-	303040	160	165	5	1.153	7.1	0.630	7.0
LS-1224C	LS1224C 105-	303033	105	110	5	0.883	6.6	0.720	7.6
LS-1224C	LS1224C 110-	303034	110	115	5	0.536	28.8	0.554	18.3
LS-1224C	LS1224C 115-	303035	115	120	5	0.788	13.9	0.804	25.3
LS-1224C	LS1224C 120-	303036	120	125	5	0.150	<0.5	0.121	<0.3
Average						1.188	14.4	1.858	15.9
Middle adit		303026						0.117	3.7
Upper adit		303027						0.017	9.4

Notes:

1. Samples are from drill holes testing the Main Zone.
2. Samples commonly comprise of light grey, strongly bleached lapilli tuff with coarse (1 cm to >5 cm) hydrothermally altered feldspathic fragments in fine-grained groundmass.
3. 1 ppb Au and 0.3 ppm Ag are the detection limits.
4. Agnerian Sample Nos. 303026 and 303027 are grab samples from the portals of the middle and upper adits, respectively.

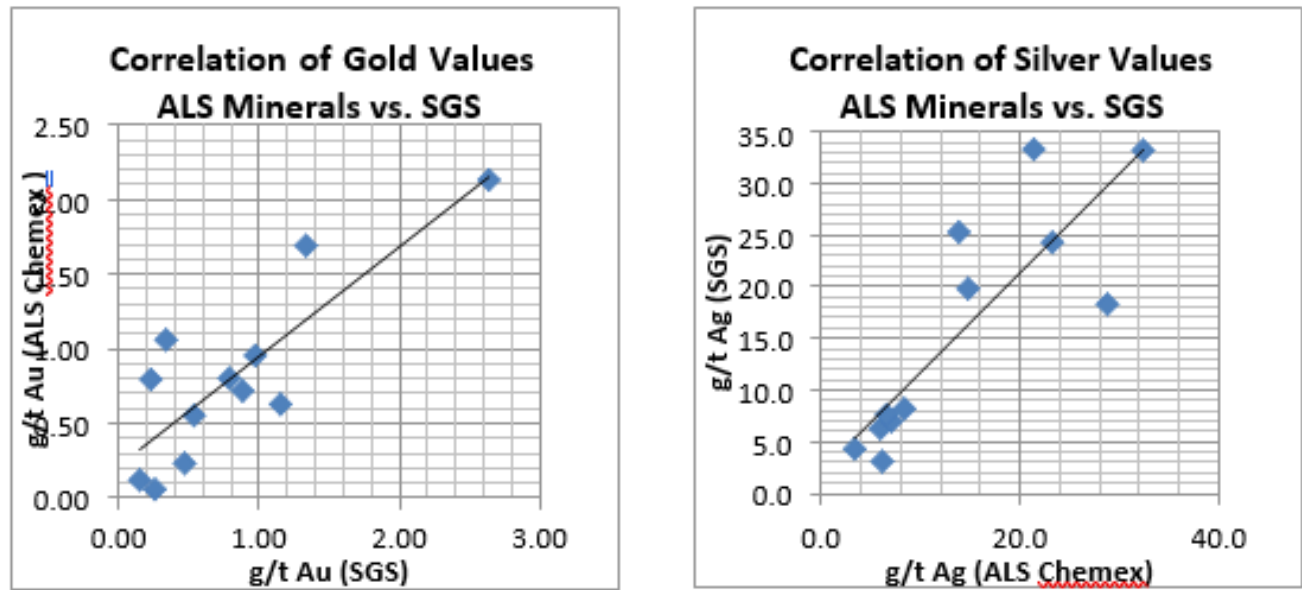


Figure 9-7
Independent Sampling Results

10.0 Mineral Processing and Metallurgical Testing

Metallurgical test work review and process design for the process plant were completed by KCA in Reno, Nevada.

KCA reviewed and verified metallurgical test procedures to ensure the data and conclusions derived from the data are in line with industry standards. It is the QP's opinion that the metallurgical test data available are sufficient to support the selected processes and conclusions regarding recoveries and costs for the Initial Assessment (IA) level.

10.1 Introduction and Summary

The Longstreet deposit has undergone two main metallurgical test programs: One in 1984 & 1987 by Kappes, Cassiday & Associates (KCA) and the second in 2014 by McClelland Laboratories Inc. These studies evaluated the performance of cyanidation and heap leaching. All test work was conducted on samples that were considered representative at the time.

A summary of all available test work programs is presented in Table 10-1

Table 10-1
Test Work Programs Longstreet Project

Report Date	Provider	Cyanide Shaking Test	Bottle Roll Quantity	Column Test Quantity
June 25, 1984	Kappes, Cassiday & Associates (KCA)	2	2	2
September 10, 1987	Kappes, Cassiday & Associates (KCA)	-	2	6
April 16, 2014	McClelland Laboratories Inc.	-	9	3
Total		2	13	11

10.2 Kappes, Cassiday & Associates (KCA) 1984 Summary

Four bulk samples of material were taken from the upper and lower adits of the Longstreet, Nevada, property by Minerva Explorations Ltd. and Kappes Cassiday & Associates (KCA). The samples were delivered to KCA on October 13, 1983. Two of the samples were classified as “low-grade oxide” material, and the other two were described as “sulfide ore.” The similar type samples were combined and tested in bucket leach tests for preliminary assessment of the amenability of the material to cyanide heap leach processing.

Gold recovery from a heap leach type test (using a 5-gallon bucket) on the “oxide” material, which had calculated head assay of 0.021 ounces Au per ton of ore, was 76% recovery in 60 days of leaching at a crushed size of 1.5”. Gold recovery in a similar test on the “sulfide” material, with a

calculated head assay of 0.017 ounces Au per ton of ore, was only 29% in 75 days of leaching at a nominal 1.5" crush size.

This leaching behavior suggested that the gold is partly locked or encapsulated in pyrite minerals (sulfide material), and is then released upon oxidation of the ore zone. The results these tests are shown in Table 10-3

Silver values of 0.5 to 1.0 ounce per ton were contained in the samples. Only 9% silver extraction was obtained from the oxide ore while 54% silver extraction was obtained from sulfide material. Additional leaching time would be expected to improve silver extraction from the sulfide material. The behavior of silver leaching in the bucket tests indicated that: Silver is contained in very fine mineralization that requires fine grinding for liberation and silver is not associated with the gold mineralization.

In addition to the cyanide bucket leach tests, other cyanide testing included 24 hour Bottle Roll Tests on pulverized portions of the two ore types as well as 2 and 24 hour cyanide shake tests on pulverized material. Recoveries in the bottle roll leach tests were 89% gold and 47% silver for the oxide material and 76% gold and 83% silver for the sulfide material. These are the approximate recoveries that could be expected in a conventional cyanide milling operation.

Cyanide consumptions in the bucket leach tests were 3.7 lb/NaCN per ton of oxidized ore and 11.3 lb NaCN/ton of sulfide ore. Lime consumptions in the tests were 2.2 lb/ton of oxide ore and 20.6 lb/ton of sulfide ore.

Table 10-2 Shows the Bottle Roll Tests

Table 10-3 Shows the Bucket Leach Tests

Table 10-4 Shows the Cyanide Shake Tests

Table 10-2
KCA 1984 Bottle Rolls Tests

KCA Test No.	KCA Sample No.	Description	Time, hours	pH	Tail Fire Assay		Calculated Head		Recovery	
					Au opt	Ag opt	Au opt	Ag opt	Au	Ag
3976A	3975A	Oxide Material	24	12	0.003	0.20	0.027	0.43	88.9	46.5
3976B	3975B	Sulfide Material	24	10.9	0.006	0.27	0.025	1.47	76.0	83.0

Table 10-3
KCA 1984 5-gallon Bucket Leach Test

KCA Test No.	KCA Sample No.	Description	Crush Size, inches	Leach Days	Metals Recovered		Recovery		Reagents, lb/ton	
					Au opt	Ag opt	Au	Ag	NaCN	Ca(OH) ₂
4002	3975A	Oxide Material	1.5	59	0.016	0.05	76.19	8.76	3.68	2.22
4168	3975B	Sulfide Material	1.5	75	0.005	0.52	29.42	53.61	11.25	20.60

Table 10-4
KCA 1984 Cyanide Shaking Tests

KCA Test No.	KCA Sample No.	Description	Time, hours	pH	Head Fire Assay		Cyanide Soluble		Recovery		Cu, ppm
					Au opt	Ag opt	Au opt	Ag opt	Au	Ag	
3976C	3975A	Oxide Material	2	10	0.022	0.24	0.015	0.31	133.3	77.4	1.0
3976D	3975A	Oxide Material	24	10.2	0.023	0.26	0.015	0.31	153.3	83.9	1.1
3985A	3975B	Sulfide Material	2	10.3	0.016	1.12	0.058	1.11	27.6	100.9	14.3
3985B	3975B	Sulfide Material	24	10.3	0.023	1.3	0.058	1.11	39.7	117.1	18.3

A tailings screen analysis was also made on the sulfide material (Sample 3975B). Unfortunately, no records we found of head screen analysis of the same sample to calculate the recovery by size fraction. No screen analysis was made on the oxide material.

Table 10-5 Shows the tailings screen analysis of the sulfide material (Sample 3975B)

Table 10-5
Tailings Screen Analysis KCA test 4168 KCA Sample 3975B

Size Fraction	Wt. lb	Wt. %	Au opt	Ag opt
+2"	2.25	3.8	-	-
-2" +1/2"	36.76	62.9	0.008	0.22
-1/2" +6M	8.38	14.3	0.016	0.74
-6M +28M	5.64	9.7	0.026	0.89
-28M +65M	2.73	4.7	0.03	1.28
-65M	2.71	4.6	0.018	1.06
Total	58.47	100	0.012	0.45

10.3 Kappes, Cassiday & Associates (KCA) 1987 Summary

Two bulk ore samples from the Longstreet, Nevada, property were delivered to KCA in Nevada for heap leach amenability testing. The bulk surface sample, KCA Sample No.8152, consisted of three 55-gallon drums and five 30-gallon drums of material. The bulk underground sample, KCA Sample No. 8153, consisted of twelve 30-gallon drums.

Column leach tests were conducted on 3 to 14 inch rocks selected from the bulk samples and on the remaining minus 3-inch fraction of each bulk sample. Tests were also conducted on portions of each bulk sample crushed to 100% passing ¼-inch and agglomerated with cement.

Recoveries on the tests on the 3-14 inch material averaged 57% gold and 3% silver in 44 days of leaching. Recoveries in the column tests on portions of the minus 3" fractions averaged 78% gold and 13% silver in 45 days of leaching. Recoveries in the column tests on material crushed to minus 1/4" and agglomerated with cement averaged 88% gold and 24% silver in 42 days of leaching.

For the run-of-mine material (+3-inch), reagents consumptions averaged 1.42 lb NaCN/ton and 0.76 lb/ton of hydrated lime. For the agglomerated material, reagent consumptions averaged 2.53 lb NaCN/ton, 0.55 lb/ton hydrated lime and 8 lb/ton of cement.

Percolation tests conducted on non agglomerated portions of the minus 1/4" crushed ore from each sample showed that agglomeration was necessary to obtain adequate percolation and prevent fines migration. If fine crushing of the ore is selected for production, agglomeration will be necessary.

In addition to the column leach tests, agitated cyanide bottle roll tests were also conducted on pulverized portions of each bulk sample to determine the optimum gold and silver recoveries which might be obtained in a mill. Gold recovery in these tests averaged 92% in 24 hours of leaching. Silver recoveries in the same tests averaged 64%.

Table 10-6 Shows the column tests for all composites.

Table 10-6
KCA 1987 Column Tests

KCA Test No.	KCA Sample ID	Feed Size	Recovery Au	Recovery Ag	NaCN lb/t	Lime lb/t	NaCN ppm	Leach Time days
8171	8152A	+3"	64%	1%	1.46	0.65	1,000	44
8173	8153A	+3"	50%	5%	1.21	0.56	1,000	44
8183	8152D	-3"	68%	15%	1.55	0.91	1,000	46
8185	8153D	-3"	88%	11%	1.49	0.93	1,000	45
8333	8152	-1/4"	86%	25%	2.62	0.50	1,000	42
8335	8153	-1/4"	91%	24%	2.45	0.60	1,000	42

Table 10-7 and Table 10-8 Shows the screen analysis (Head vs Tails) for the -3" material for both: surface and underground samples.

Table 10-7
KCA Test 8183 (-3" Surface Sample) – Screen Analysis

Size Fraction	Weight Distribution	Head oz Au/t	Tails oz Au/t	Recovery	Recovery Cum.
+2"	23%	0.0605	0.0090	77%	19%
-2" + 1"	28%	0.0380	0.0085	65%	20%
-1" + 1/2"	17%	0.0320	0.0100	51%	10%
-1/2" + 1/4"	10%	0.0185	0.0085	28%	3%
-1/4" + 10 M	10%	0.0265	0.0055	68%	8%
-10 M + 28 M	5%	0.0210	0.0065	52%	3%
-28 M + 65 M	4%	0.0315	0.0060	70%	3%
-65 M	3%	0.0365	0.0062	74%	3%

Table 10-8
KCA Test 8185 (-3" Underground Sample) – Screen Analysis

Size Fraction	Weight Distribution	Head oz Au/t	Tails oz Au/t	Recovery	Recovery Cum.
+2"	14%	0.0315	0.0085	73%	11%
-2" + 1"	26%	0.0495	0.0050	90%	24%
-1" + 1/2"	20%	0.0440	0.0055	88%	18%
-1/2" + 1/4"	14%	0.0395	0.0095	76%	11%
-1/4" + 10 M	13%	0.0460	0.0045	90%	12%
-10 M + 28 M	7%	0.0520	0.0055	89%	6%
-28 M + 65 M	4%	0.0570	0.0065	89%	4%
-65 M	3%	0.1020	0.0067	94%	2%

10.4 McClelland Laboratories 2014 Summary

The 2014 metallurgical test work program was completed by McClelland Laboratories under the supervision of a QP metallurgical engineer retained by Star Gold. The program comprised bottle-roll leach testing, column leach testing, comminution testing, and mineralogical investigations. The metallurgical test program was reviewed and accepted in 2014 by A-Z Mining Professionals.

10.4.1 Section Sample Assays

A total of 65 underground adit samples, with a combined weight of 816 lb, and three surface samples totaling 903 lb were collected for metallurgical testing. All samples were crushed to 100% passing 2 inches and analyzed in duplicate for gold and silver. The assay results are presented in Table 10-9.

The individual samples were subsequently combined to produce underground and surface composites, as well as a blended master composite. Triplicate direct head assays were performed on each composite sample. The resulting standard deviations among the triplicate assays were relatively high, particularly for the surface master composite. Although the agreement between

individual triplicate splits was poor, the average assay values were consistent with those anticipated from the sectional assay data.

Quality control samples yielded acceptable results, indicating that the analytical procedures were reliable. The observed variability is therefore interpreted to reflect the heterogeneous or locally “spotty” distribution of gold within the potentially economic mineralization rather than deficiencies in the assay process.

Table 10-9
Gold Head Assays and Head Grade Comparisons, Longstreet Composites

Determination	SMC, opt		UMC, opt		BMC, opt	
	Au	Ag	Au	Ag	Au	Ag
Direct Assay, Initial	0.00613	0.496	0.02042	1.955	0.01663	1.167
Direct Assay, Duplicate	0.01955	0.992	0.02392	1.838	0.01926	1.196
Direct Assay, Triplicate	0.01079	0.613	0.03180	1.546	0.02246	1.459
Average	0.01225	0.700	0.02538	1.780	0.01955	1.284
Standard Deviation	0.00671	0.263	0.00583	0.204	0.00292	0.175

Twenty rock specimens collected from both underground workings and surface exposures were selected for comminution testing. The remaining samples were processed separately and stage-crushed to 100% passing 2”. The underground and surface samples were then individually blended and subsequently combined to produce a master composite representative of the overall sample population. This blended material was split to create a third master composite. Portions of the composite samples were collected for bottle-roll testing.

Following composite preparation, all samples were further crushed to 80% passing ¾”, thoroughly blended, and divided into approximately 165 lb aliquots for column leach testing

10.4.2 Bottle Roll Testing

In 2014, bottle roll testing was conducted on each of the three composite samples at a feed size of 80% passing 10 mesh (0.07”) to establish lime consumption requirements and evaluate metallurgical performance prior to column leach testing. Gold and silver recoveries were similar among the three composites. Gold recoveries ranged from 80.6% to 81.9%, while silver recoveries ranged from 17.5% to 20.0%.

Additional bottle roll tests were performed on the blended master composite using a cyanide concentration of 1,000 ppm NaCN at feed sizes of 100% passing 2”, 80% passing ¾”, and 80% passing ¼” to assess the effect of particle size on metal recovery. Results indicate a moderate sensitivity of both gold and silver recovery to feed size. Compared with material crushed to 100% passing 2”, recoveries at a feed size of 80% passing 0.07” increased by 18.4% for gold and 13.9%

for silver. Despite these improvements, silver recoveries remained relatively low in all bottle roll tests.

To investigate the factors limiting silver recovery, three additional bottle roll tests were completed on the blended master composite using an elevated cyanide concentration of 5,000 ppm NaCN. Testing was conducted at particle sizes representative of heap-leach conditions (80% passing $\frac{3}{4}$ " and 80% passing $\frac{1}{4}$ ") and conventional milling conditions (80% passing 200 mesh).

Results demonstrated that increasing cyanide concentration had little effect on silver recovery at heap-leach feed sizes. However, silver recovery increased substantially when the material was finely ground. The bottle roll test conducted on material ground to 80% passing 200 mesh achieved a silver recovery of 60.6%. Gold recovery also improved moderately under finer grind conditions.

Mineralogical examination of both head and tail samples from the blended master composite confirmed that the primary factor limiting silver recovery is the extremely fine-grained nature of the silver sulfide mineralization. Where silver sulfide grains are exposed or liberated through grinding, they are readily amenable to leaching. Leach kinetics for silver at a grind size of 200 mesh were exceptionally rapid, with essentially complete silver extraction occurring within the first two hours of testing. This behavior indicates that the silver-bearing sulfide minerals are highly leachable once liberated.

In contrast, silver associated with jarosite-group minerals appears to be refractory and largely resistant to cyanide leaching, regardless of particle size reduction. This mineralogical association is considered a significant factor contributing to the lower overall silver recoveries observed during testing.

A summary of the bottle roll test results is presented in Table 10-10.

Table 10-10
3 Bottle Roll Test Results – 2013

Composite	Feed Size	NaCN (ppm)	Au Recovery (%)	ppm Au				Ag Recovery (%)	ppm Ag				Reagents (lb/ton)	
				Extracted	Tail	Calc. Head	Head Assay		Extracted	Tail	Calc. Head	Head Assay	NaCN	Lime
SMC	80% - 0.07"	1,000	80.6	0.25	0.06	0.31	0.42	20	5	20	25	24	0.16	4.2
UMC	80% - 0.07"	1,000	81.9	0.68	0.15	0.83	0.87	18.9	10	43	53	61	0.26	6.8
BMC	100% - 2"	1,000	62.9	0.44	0.26	0.7	0.67	3.6	2	54	56	44	0.14	2.6
BMC	80% - 3/4"	1,000	67.1	0.51	0.25	0.76	0.67	12.8	5	34	39	44	0.14	4.2
BMC	80% - 1/4"	1,000	77.9	0.53	0.15	0.68	0.67	13.6	6	38	44	44	<0.14	6
BMC	80%- 0.07"	1,000	81.3	0.52	0.12	0.64	0.67	17.5	7	33	40	44	0.26	5
BMC	80% - 3/4"	5,000	76.4	0.55	0.17	0.72	0.67	14.6	6	35	41	44	0.96	2
BMC	80% - 1/4"	5,000	77.6	0.45	0.13	0.58	0.67	14	6	37	43	44	1.34	2
BMC	80% - 200 M	5,000	88.7	0.47	0.06	0.53	0.67	60.6	20	13	33	44	1.82	2.6

Both gold and silver recoveries improved with decreasing particle size; however, the improvement was more pronounced for silver than for gold when fine grinding was applied. The relationship between particle size and metal recovery is illustrated in Figure 10-1

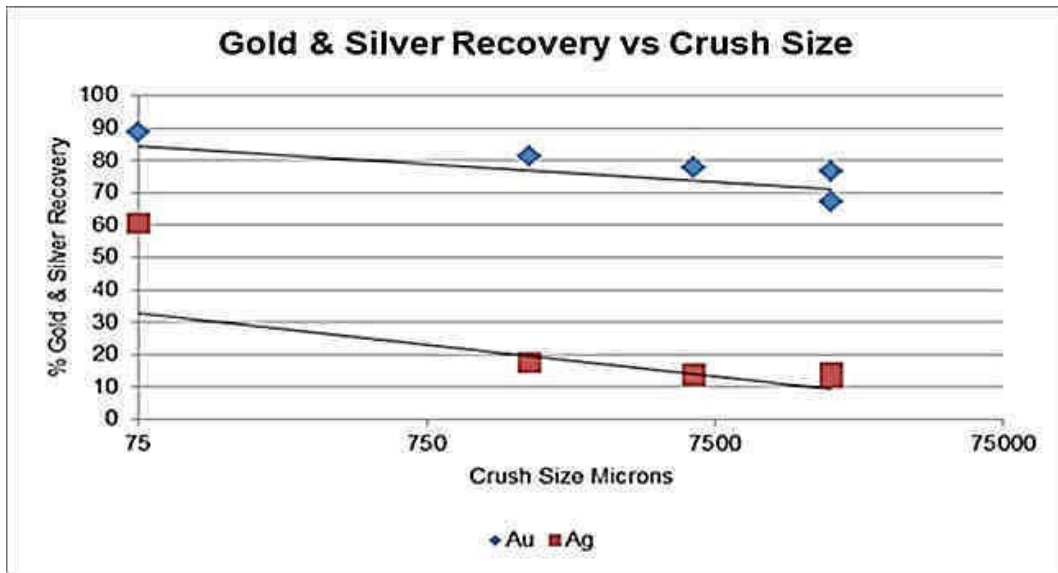


Figure 10-1 Crush Size Versus Metal Recovery

10.4.3 Column Leach Testing

Column leach testing was conducted on each of the master composite samples to evaluate gold and silver recovery, leach kinetics, and reagent consumption under conditions representative of a heap-leach operation. Test material was crushed to a particle size of 80% passing $\frac{3}{4}$ ", and lime addition rates were established based on the results of the preceding bottle roll tests.

The leach columns consisted of PVC pipes measuring 5.9" in diameter and 9.84' in height. Material was loaded carefully into the columns to minimize particle segregation and compaction. Cyanide leaching was carried out using a solution containing 1,000 ppm sodium cyanide (NaCN), applied at a rate of 0.004908 gpm/ft² of column cross-sectional area. Upon completion of the leaching cycle, the columns were rinsed with fresh water to remove residual cyanide and recover dissolved precious metals.

All three composite samples were leached for a total period of 190 days. The Surface Master Composite (SMC) achieved gold and silver recoveries of 88.9% and 20.0%, respectively. The Underground Master Composite (UMC) returned recoveries of 84.6% for gold and 15.4% for silver. The Master Blend Composite (MBC), representing a mixture of surface and underground material, yielded gold recovery of 86.3% and silver recovery of 16.7%.

The column leach results indicate consistently high gold recoveries across all composite types, while silver recoveries remained comparatively low. These results are consistent with mineralogical studies that identified a portion of the silver as occurring in refractory mineral phases. A summary of the column leach test results is presented in Table 10-11

Table 10-11
Summary Columns Test Results (80%-3/4" feed size)

Sample I.D.	Test No.	Leach Time (days)	Au Rec. %	Calc. Head Au ppm	Ag Rec. %	Calc. Head Ag ppm	NaCN Consumed lb/ton	Lime Added lb/ton
SMC	P-1	190	88.9	0.36	20	25	3.12	3.4
UMC	P-2	190	84.6	0.78	15.4	52	3.86	5.4
BMC	P-3	190	86.3	0.73	16.7	48	3.72	4.0

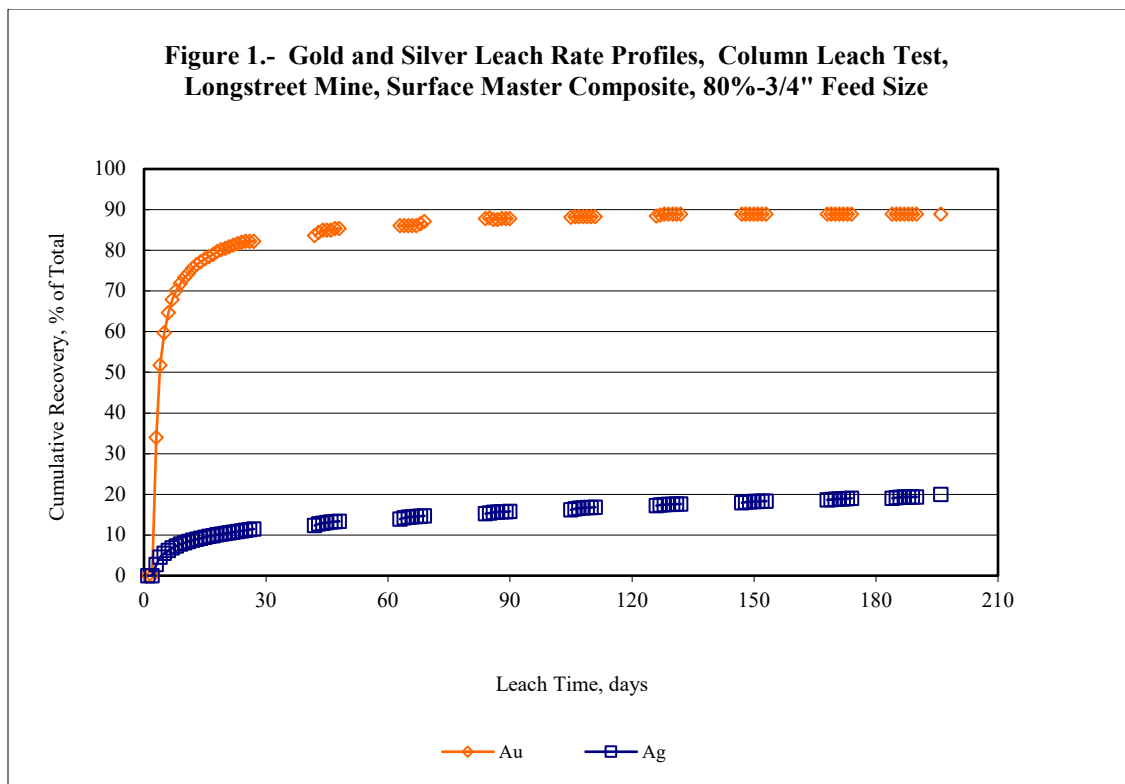
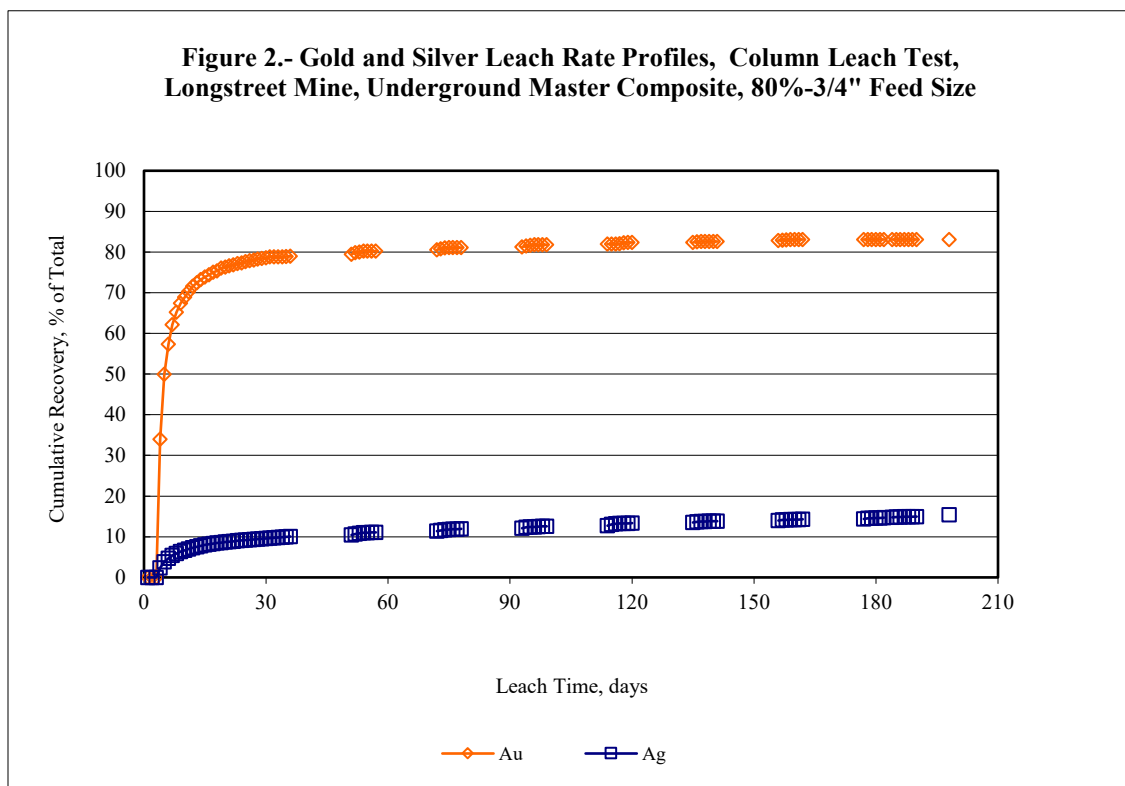
Recovery data by particle-size fraction for all three master composites indicate that additional crushing would not significantly improve gold recovery. Gold extraction was relatively consistent across the various size fractions, with only a slight increase observed in the finest fraction (-75 μ m). In contrast, silver recovery showed a stronger dependence on particle size, suggesting that fine grinding would be required to achieve substantially higher silver recoveries.

Overall metallurgical testing demonstrates that the Longstreet composite samples are well suited to heap-leach processing at a feed size of 80% passing $\frac{3}{4}$ ". Gold recoveries were consistent among the three composites, ranging from 84.6% to 88.9% following 190 days of leaching and rinsing. Silver recoveries were also similar among the composites but remained comparatively low, ranging from 15.4% to 20.0%.

Although the column leach tests were conducted over a 190-day period, gold extraction was essentially complete within the first 30 to 40 days of leaching. Silver extraction proceeded at a much slower rate, and the results suggest that extending the leach cycle beyond 190 days would not result in significant additional silver recovery.

Cyanide consumption was relatively high, ranging from 3.12 to 3.86 lb NaCN per ton of mineralized material. The elevated consumption rates are attributed, in part, to the extended duration of the leach tests. Cyanide consumption in a commercial-scale operation would typically be expected to be lower than the values observed during testing.

Leach kinetics and recovery performance for both gold and silver are illustrated in Figure 10-2 through Figure 10-4.

**Figure 10-2 Surface Master Composite Leach Kinetics****Figure 10-3 Underground Master Composite Leach Kinetics**

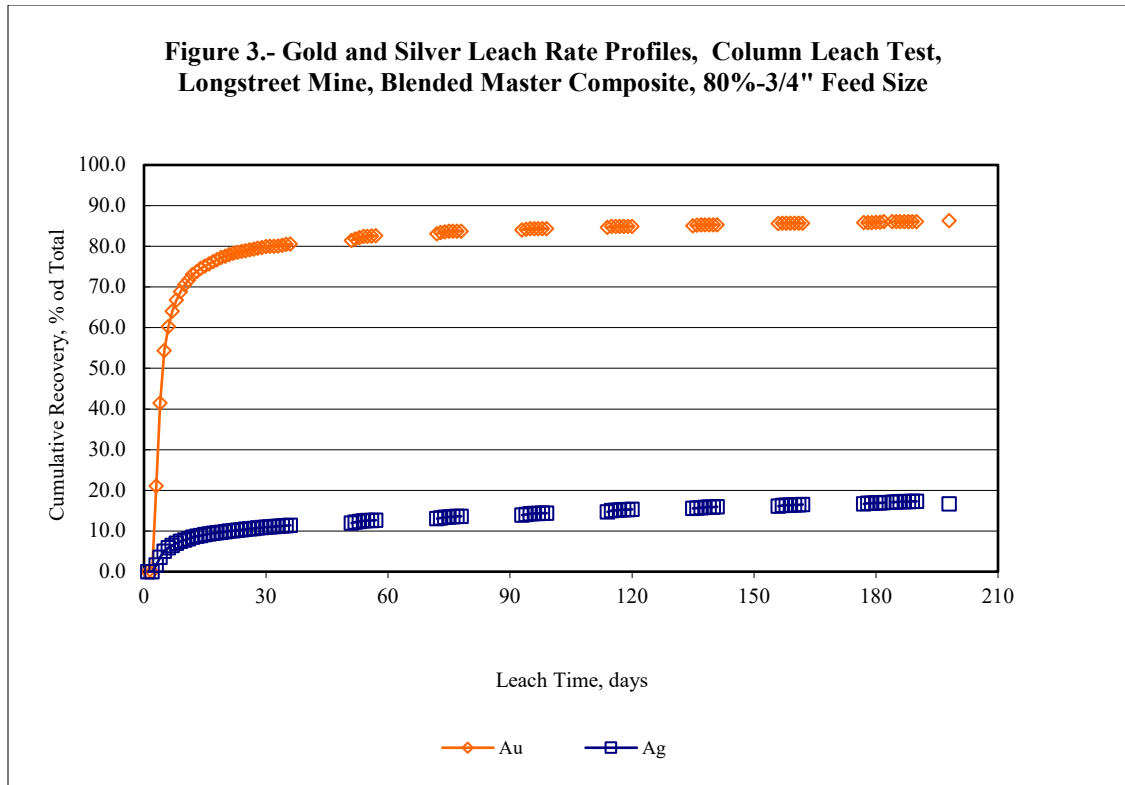


Figure 10-4 Master Blend Composite Leach Kinetics

10.4.4 Comminution Tests

10.4.4.1 Sample Preparation

A total of 20 competent rock fragments were selected from the sample population for comminution testing. Ten of the specimens were obtained from underground adit samples, while the remaining ten were collected from surface samples. The selected rock fragments were combined to form a representative test sample and submitted for crusher work index and abrasion index determinations.

No sample preparation was required for the crusher work index testing, as intact rock pieces were used directly. For abrasion index testing, the sample material was crushed and screened to obtain a particle size fraction ranging from 0.51" to 0.75". This prepared fraction was then used for abrasion resistance evaluation.

10.4.4.2 Crusher Work Index Test

The crusher work index test was performed on unprepared natural rock fragments in accordance with established testing procedures and standard laboratory protocols. The objective of the test

was to evaluate the resistance of the material to crushing and to provide data for assessing comminution requirements and equipment design considerations. Results of the mentioned tests are mentioned in Table 10-12

Table 10-12
Crusher Work Index

Sample	CWi (kW-hr/st)	CWi (kW-hr/mt)
Crusher Work Index	10.08	11.11

10.4.4.3 Abrasion Index Test

An abrasion index test was performed on a sample fraction ranging from 0.51" to 0.75" in size in accordance with standard testing procedures. The test returned a Sample Abrasion Index of 0.2431, indicating the relative abrasiveness of the material and providing information relevant to equipment wear and comminution circuit design.

10.4.4.4 Au/Ag Mineralogy

An extensive search using SEM/EDS indicates Ag phases are present in the blended master sample (Underground and Surface); however, Au was not identified. Silver sulfide is the main Ag phase and occurs as irregularly shaped inclusions in quartz, pyrite and goethite pseudomorphs after pyrite. Cube-like grains are also seen in quartz and likely represent pseudomorphs of acanthite after argentite. Grain size of the Ag sulfide is very fine with measurements that range from 0.5 μm up to approximately 5 μm . Ag sulfide is also seen as thin rinds around pyrite and as small inclusions in jarosite.

These areas are Ag rich and likely represent argentojarosite intimately mixed with the more abundant potassium variety. This phase may represent capgaronnite or possibly itlisite. The primary reason for low Ag recovery in this material appears to be due to the very fine grained nature of the Ag sulfide, which should leach easily if liberated or exposed. In contrast, Ag bearing jarosites tend to be refractory and are usually unaffected by leaching.

10.5 Recommendation and Conclusions

In conclusion, it is the QP's opinion that the current test work available is sufficient for the evaluation of the Project at a IA level with recommended metallurgical design parameters presented in this section.

The metallurgical test program yielded the following key findings:

- Gold dissolution during column leach testing was rapid, with minimal additional recovery achieved beyond approximately 30 days of leaching. In contrast, silver exhibited slower leach kinetics, with recoveries continuing to increase slightly after 120 days of leaching.
- Gold recovery demonstrated limited sensitivity to feed particle size, provided that an adequate leach cycle time was maintained.
- All composite samples at a particle size of 80% passing $\frac{3}{4}$ " responded favorably to simulated heap leaching. Gold recoveries for this size fraction ranged from 84.6% to 88.9%. Agglomeration is not required for this material.
- Silver recoveries obtained from the column leach tests were comparatively low, ranging from 15.4% to 20.0%.
- Crusher work index testing indicated that the potentially economic mineralization is characterized by relatively low hardness and only slight abrasiveness.

As part of future programs, the QP recommends:

- Additional column leach tests with $\frac{3}{4}$ " crushed ore.
- Additional column leach tests, both on conventionally and HPGR crushed material at variable crush sizes from 80% passing $\frac{1}{2}$ " to $\frac{1}{4}$ " to better understand the recovery versus size relationship and if HPGR crushing is economically beneficial.
- Compacted permeability tests should also be conducted to define potential cement agglomeration requirements (and agglomeration costs) based on the planned ultimate heap heights.
- Conduct further bottle roll and column leach testing on representative oxide material, preferably derived from drill core, to evaluate metallurgical variability across the deposit.

11.0 Mineral Resource Estimates

This Technical Report details the mineral resource estimate (MRE) reported for the Longstreet Project and follows the guidelines of Canadian National Instrument 43-101. The modeling and estimate of gold and silver resources at the Longstreet deposit, which were completed in May 2026, were done by James Ashton, P.E. who is considered a Qualified Person by the definitions and criteria set forth in NI 43-101. This resource estimate makes use of 12 drillholes completed in 2014 that have not been included in prior resource estimates. There is no affiliation between Mr. Ashton and Star Gold Corp. except that of an independent consultant/client relationship. The effective date of this mineral resource is May 29, 2026. No mineral reserves were estimated for the Longstreet project.

The MRE was calculated using a block model size of 20 ft (X) by 20 ft (Y) by 20 ft (Z). The Author estimated the gold and silver grade for each block using Inverse Distance Weighted (IDW) with locally varying anisotropy to ensure grade continuity in various directions is reproduced in the block model. The final MRE is the total resource within the mineralized domains as defined by the drilling and is constrained by an optimized pit. The final MRE is reported at various economic cutoff grades and includes estimated dilution. Details regarding the methodology used to calculate the mineral resource are documented in this section.

11.1 Data

Star Gold and other operators of the property have spent much time and resources compiling the data used in this mineral resource estimate (MRE). This data has subsequently and continually been updated, added to, and refined. The database used in the estimation of gold and silver mineral resources for Longstreet contains assay and geological information for 437 drill holes with 18,278 assay intervals. New digital topography, completed in 2017 by PhotoSat Information Ltd., with 3 ft contour interval using a datum of NAD83, Nevada State Plane West, of the current surface was used in the resource model. Modelling and Project coordinates have been converted to UTM NAD27, Zone 11, Ground converted to imperial units. The MRE was completed using the mine planning software Minsight® (v3.50).

11.2 Deposit Geology Pertinent to Resource Estimation

The current level of geological understanding of the gold and silver mineralization at the Longstreet property is understood quite well given the relatively high concentration of drilling and similarities to nearby gold mines. Gold and silver mineralization on the Longstreet property is typical of low-sulphidation epithermal Au-Ag systems associated with hydrothermal alteration assemblages within felsic volcanic rocks. These deposits are formed at relatively shallow depth,

typically within 300 feet of the surface, from hydrothermal fluids. Mineralization at Longstreet is contained in altered rocks, which are localized by geological structures, and range in size from 15 feet to more than 300 feet wide and up to 2500 feet long. Two dominant sets of mineralized structures are observed: one trending east and the second one trending north-northwest. A third (less common) structure trends east-southeast. Mineralization is comprised of altered zones, quartz stockworks, and hydrothermal breccia zones that contain disseminated pyrite. In addition, occasional quartz veins are associated with high grade gold mineralization, mainly as fracture coating material. (Source, Noland 2025)

Domains of mineralization were broken out to control the grade estimate for both gold and silver. The mineralized body strikes 11° NE with a dip of -45°. A broad low-grade halo was defined at Longstreet around a grade of ~0.003 oz. Au/ton (0.103 ppm). This main mineralized body as presently defined is roughly 1,700 ft. (518 m) in length along strike and 725 ft. (220 m) in down dip extent.

Internal to this low-grade domain is an irregular domain defined at approximately ~0.030 oz. Au/ton (0.97 ppm). The high-grade domain most likely is a result of increased quartz veining and fracturing, and results in zones with irregular shapes that are generally similar to the main low-grade domain but much thinner. The high-grade domain boundaries confine the higher gold grades during the interpolation process to within these boundaries. Figure 11-1 presents a representative section of the gold domains.

Silver mineralization is positively related to the gold mineralization, but the silver grade is only partially correlated with the gold grade having a correlation coefficient of 0.35 (Figure 11-5). The silver mineralization for the most part is concurrent with the gold mineralization and is only slightly larger in size than the gold mineralization. The zones of higher-grade silver mineralization generally occur in the same location as the higher-grade gold zones. A separate silver model was created to fully capture all the silver during grade interpolation. The broad low-grade silver zone is about 1,700 feet (520 meters) long along strike, 1,350 feet (410 meters) (wide perpendicular to the strike) and 1,980 feet (600 meters) long parallel to the dip. Two grade domains 0.200 opt Ag and 2.0 opt Ag (6.85 ppm Ag and 68.5 ppm Ag) were used to model the silver mineralization. Figure 11-2 presents a representative section of the silver domains.

Figure 11-3 and Figure 11-4 illustrate the mineralization solids created from the lower grade outlines for gold and silver respectively. The higher-grade domains are obscured by the low-grade domain for both gold and silver.



Figure 11-1
Gold Domains 1300NW



Figure 11-2
Silver Domains 1300NW

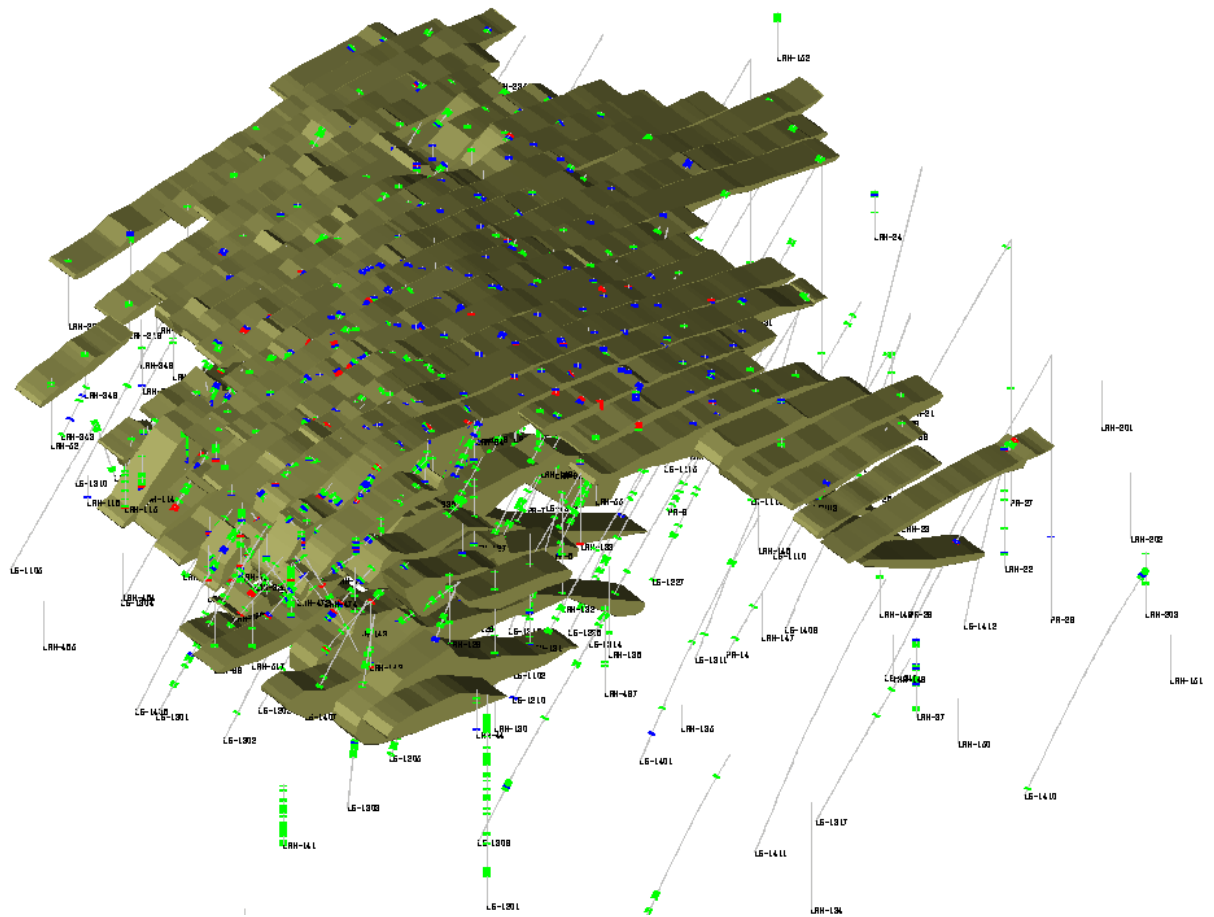
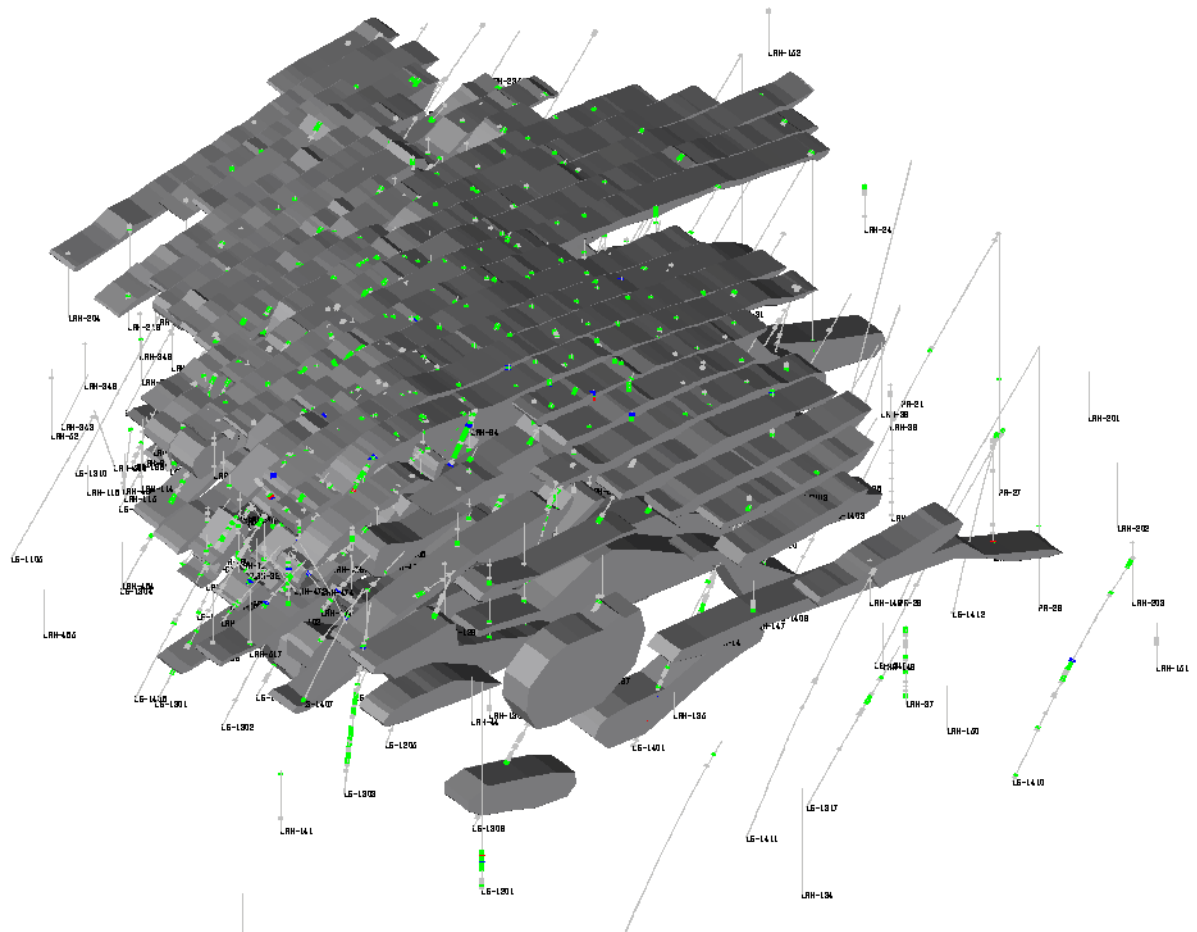
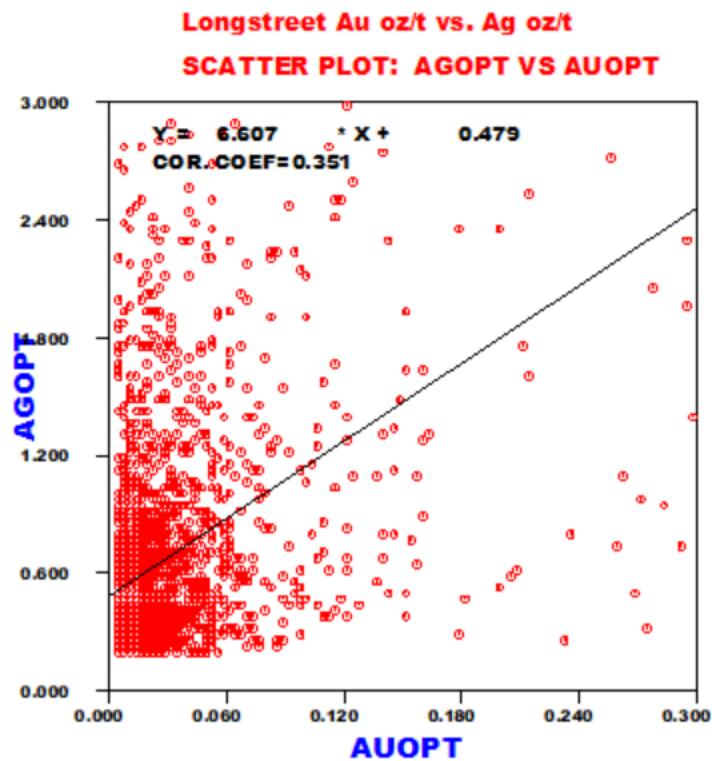


Figure 11-3
Gold Low-Grade Solid and Drilling



Source: Ashton, 2026

Figure 11-4
Silver Low-Grade Solid and Drilling



Source: Ashton 2026

Figure 11-5
Gold Assays vs. Silver Assays (0.003 Au oz/t Cutoff)

11.3 Density

In 2012, MinQuest carried out specific gravity determinations on eight drill core samples from the four diamond drill holes completed on the Longstreet property for Star Gold, using the Archimedes Principle, Specific gravity of rock = weight of rock/volume of rock (amount of water in graduated cylinder), are summarized below and listed in Table 11-1.

- Maximum: 2.51
- Minimum: 2.19
- Average: 2.37

Table 11-1
Statistics of Specific Gravity Determinations, Main Zone Drilling 2012

DDH No.	From (ft.)	To (ft.)	Interval (ft.)	Specific Gravity
LS 1216C	85.0	89.0	4.0	2.48
LS 1216C	213.2	213.6	0.4	2.51
LS 1217C	65.3	65.8	0.5	2.44
LS 1217C	175.0	175.3	0.3	2.48
LS 1222C	74.2	76.0	1.8	2.22
LS 1222C	191.0	191.5	0.5	2.32
LS 1224C	109.0	109.3	0.3	2.19
LS 1224C	200.8	201.0	0.2	2.34

Source: Kern, 2012a

The average specific gravity of 2.37 was converted to 13.518 cubic feet/ton. All blocks in the model were assigned a tonnage factor of 13.518 cubic feet/ton.

11.4 Resource Model

The Longstreet resource model was constructed on cross sections oriented N11°E using the validated drillhole assay data. The lower and higher-grade mineral domains for both gold and silver were drawn on 24 cross sections spaced 50 feet apart. These domain outlines were then used to make mineral domain solids for both gold and silver. All modeling of the Longstreet deposit resource was performed using Minesight®.

Summary statistics of the Longstreet assay database for each mineral domain are shown in Table 11-2.

Table 11-2
Summary Statistics of Assays within Mineral Domains - Gold & Silver

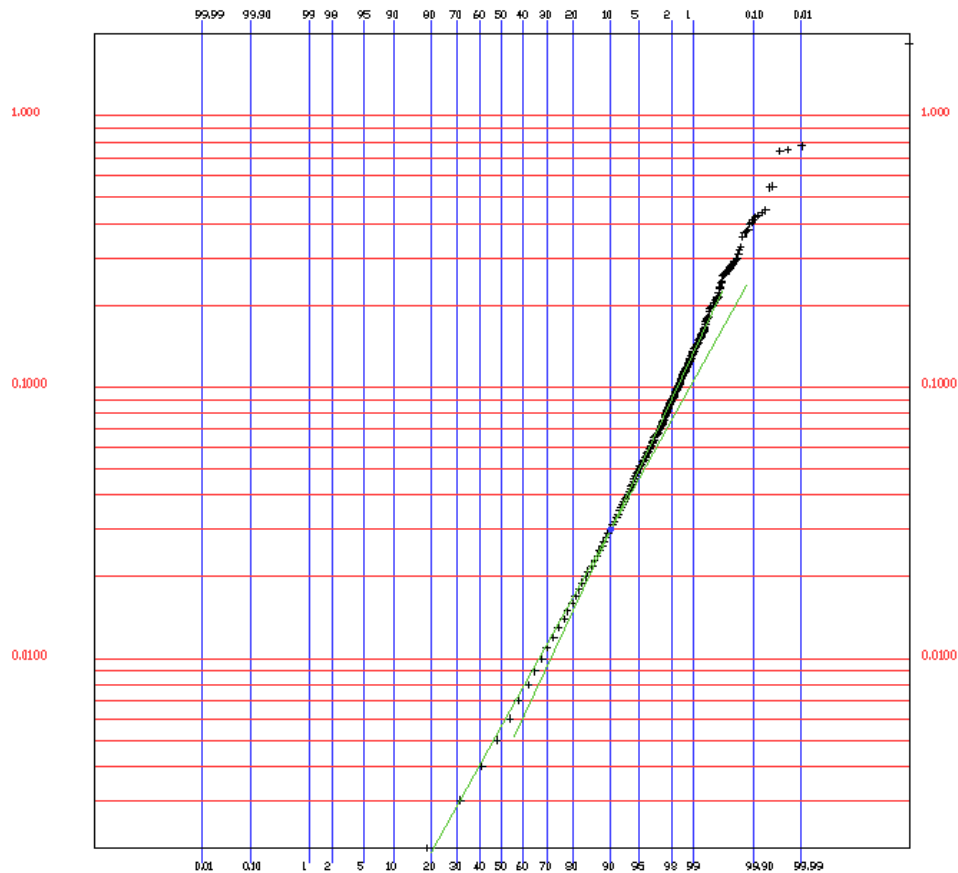
Description	Gold Low-Grade (opt)	Gold High-Grade (opt)	Silver Low-Grade (opt)	Silver High-Grade (opt)
# Samples	7,248	760	7,653	358
Min.	0.000	0.000	0.00	0.00
Max.	0.751	1.834	9.19	40.93
Mean	0.011	0.078	0.379	2.916
Std. Dev.	0.0168	0.109	0.439	3.576
C.V.	1.517	1.302	1.159	1.226

Source: Ashton, 2026

The grade distributions for all assays in the Longstreet resource area were examined to try and identify population breaks for the mineralization (Figure 11-6 (Au), & Figure 11-7 (Ag)). The gold distribution shows a very subtle break at about 0.030 opt Au. Mineralization higher than 0.030 opt Au appear to be within the same population. The silver distribution shows breaks at about 0.20, 0.55, and 3.00 opt Ag. The 0.20 opt Ag break was used as the guide for making the lower-grade mineral domain. The Author chose to use a grade of 2.0 opt Ag as the guide to construct the higher-grade mineral domain.

Project cross-sections oriented N11⁰E (looking northwest) were drawn along the strike of the deposit at regular intervals of fifty feet to best fit the drilling. The sections show topography, current and historic drill holes with assays. Geology throughout the resource area is consistent and thus geology was not used to help define the mineral domains. Generally, the mineralization dips to the northeast at about 45 degrees.

The few underground historic workings consist of two adits and a decline shaft. No information is available as to how much material was mined from these working and thus no accounting of this material has been made in this report.

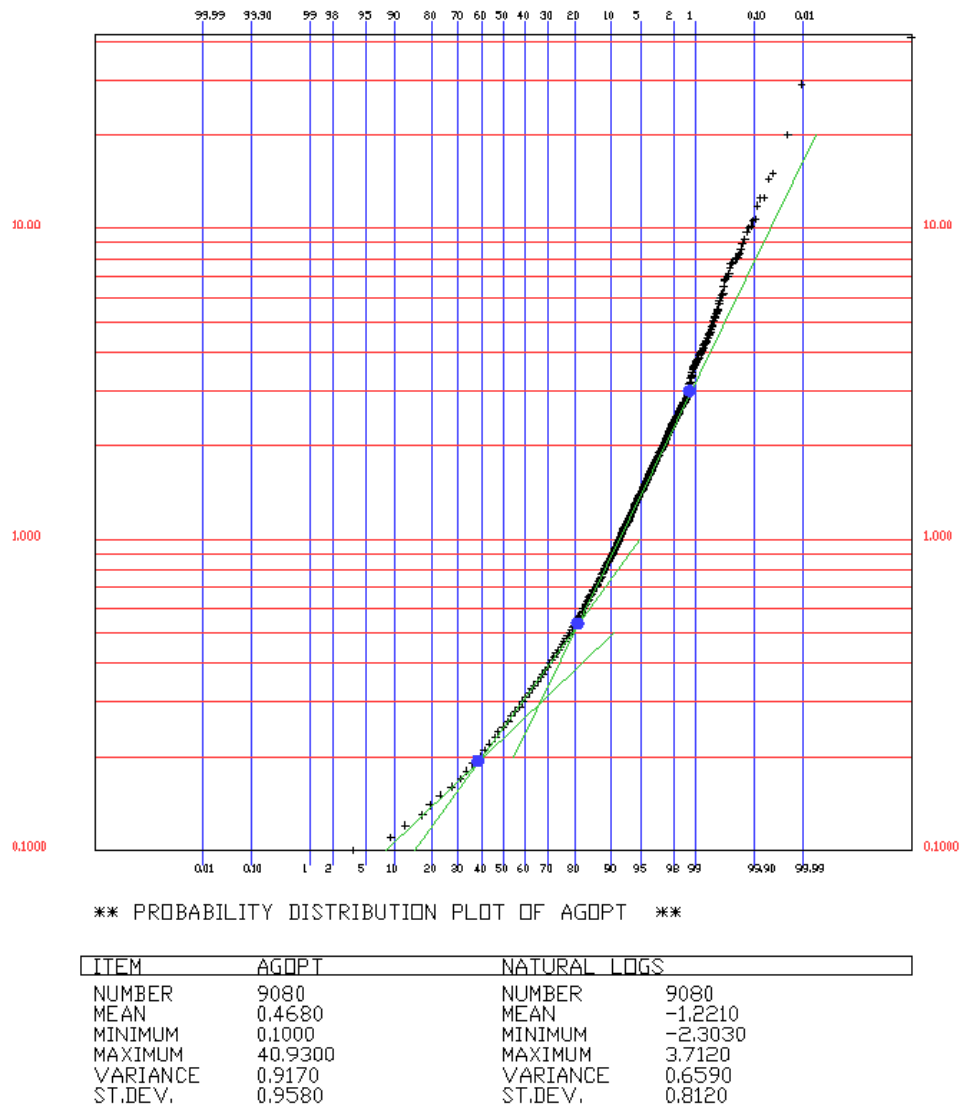


*** PROBABILITY DISTRIBUTION PLOT OF AUOPT ***

ITEM	AUOPT	NATURAL LOGS	
NUMBER	10363	NUMBER	10363
MEAN	0.0140	MEAN	-4.9600
MINIMUM	0.0020	MINIMUM	-6.2150
MAXIMUM	1.8340	MAXIMUM	0.6060
VARIANCE	0.0010	VARIANCE	1.1070
ST.DEV.	0.0360	ST.DEV.	1.0520

Longstreet Project

Figure 11-6
Log Probability Plot Gold Sample Data



Longstreet Project

Figure 11-7
Log Probability Plot Silver Sample Data

The project sections were reviewed to determine if any of the gold grade and silver grade populations identified in the grade distribution plots (Figure 11-6 and Figure 11-7) represented continuous zones of mineralization. It was determined that gold grade domains of 0.003 to 0.030 opt and greater than 0.030 opt Au showed the best continuity, and these grade ranges were assigned to gold mineral domains 1 and 2 respectively. For the silver mineralization, grade domains at 0.20 to 2.00 opt and greater than 2.0 opt Ag were chosen. These grade ranges were assigned to silver mineral domains 1 and 2 respectively. The higher-grade domains served to limit the extrapolation of these higher grades into the surrounding lower-grade mineralization. The geometry of the high-grade domains generally fit well with the overall mineralization model interpretation. The cross-sectional grade domain outlines were then used to create a solid for each grade domain.

The grade domain solids were “speared” by each drill hole in order to code each drill hole assay interval as to the appropriate grade domain for both gold and silver. Descriptive statistics and grade distributions of the assays were prepared for each grade domain and were examined and based on this review and a grade decile analysis (Parish, 1997) it was determined that grade capping was not warranted.

11.5 Longstreet Mineralization Block Model

Each block of the block model contains numerous items such as gold, silver, percent of a mineral domain within the block, tons for each block, waste grade values for material outside the mineral domains etc. Each item value in the block model is calculated using methodologies specific to each item. Values for gold and silver within the mineral domains were combined with values for gold waste and silver waste outside the mineral domains along with gold and silver recoveries to calculate a gold equivalent value for each block. The gold equivalent value will be used in reporting the mineral resource and for future economic studies.

The block model was created using 20 ft. (X) x 20 ft. (Y) x 20 ft. (Z) (6 m x 6 m x 6 m) blocks. The model dimensions are 150 columns (East), 145 rows (North), and 70 benches (Elev.). The mineral domain solids were used to calculate the percent of each domain in each block. The percentage area of each grade domain within each block was stored and these were used to weight average the grades of each domain into a zone (undiluted) grade value and block-diluted grade value. Block-diluted refers to those grades that are fully block-diluted; in other words, all zones including the un-mineralized waste “outside” the domains are weight-averaged. Zone-diluted refers to that grade that is the weight average of the zones only and does not take into account external waste dilution. The pit optimization process incorporates the waste grade value along with the mineral domain grade value to create a block-diluted grade value. This report presents the resource as block-diluted grades and tons.

The tons each block contains is calculated based on the tonnage factor and percent of the block that lies beneath the surface topography. The percentage of each block that lies below the topographic surface is stored for use in the calculation of block tonnages.

Fields stored in the block model include percent topography, percent of each domain, grade for each domain, whole block and zone-diluted block grades, resource classification, tons per block, distance to the nearest composite, number of composites and drillholes used in each block estimate, economic value of each block, and resource classification.

11.6 Composites

The assays were composited at 10 foot (3 meter) down-hole intervals respecting the mineral domain boundaries. For gold over 99.83% of the composites had a length equal to or greater than 5 feet and 89.7% of the composites had a length equal to 10 feet. For silver 99.86% of the composites had a length equal to or greater than 5 feet and 91.3% of the composites had a length equal to 10 feet. Descriptive statistics for both gold and silver composites are shown in Table 11-3.

Table 11-3
Summary Composite Statistics by Domain – Gold and Silver

Description	Gold (0.003 opt) Domain	Gold (0.030 opt) Domain	Silver (0.20 opt) Domain	Silver (2.0 opt) Domain
# Samples	3,818	447	3,984	202
Min.	0.000	0.000	0.00	0.00
Max.	0.389	0.965	5.42	10.21
Mean	0.0111	0.0746	0.3803	2.7437
Std. Dev.	0.0133	0.0702	0.3651	2.0687
C.V.	1.2012	0.9411	0.96	0.754

11.7 Variogram Analysis

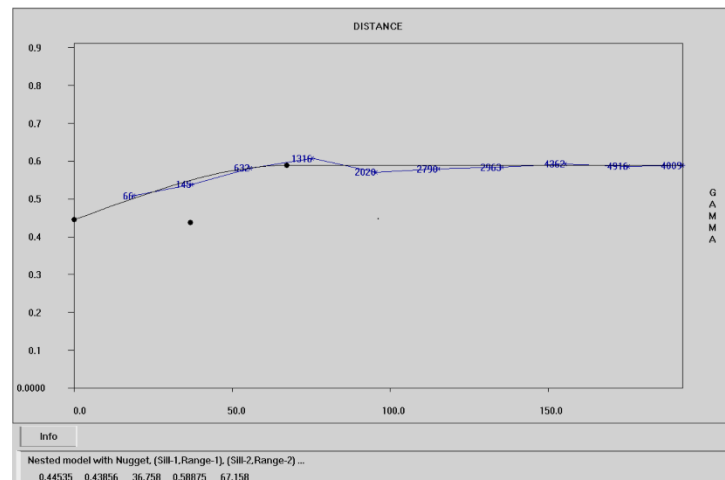
Experimental semi-variograms for each domain were calculated along the major, minor, and vertical principal directions of continuity that are defined by the domain solids. The Author calculated and modelled semi-variograms for gold and silver using the 10 ft (3.0 m) composites within each of the mineralization domains. Only the lower-grade domains for both gold and silver had enough composite data to yield any useable variograms. The higher-grade domains only yielded down-the-hole and global variograms. The principal axis of mineralization is roughly N11°E with a plunge of -30° and a dip to the west of 6°. Modelling parameters from the variogram

analysis are listed in Table 11-4, and the calculated experimental semi-variogram and models used for resource estimation for the Longstreet resource are illustrated in Figure 11-8 through Figure 11-11. Variogram calculation and modeling was done in Minesight®. Variable lag lengths, directions, and variogram types were used.

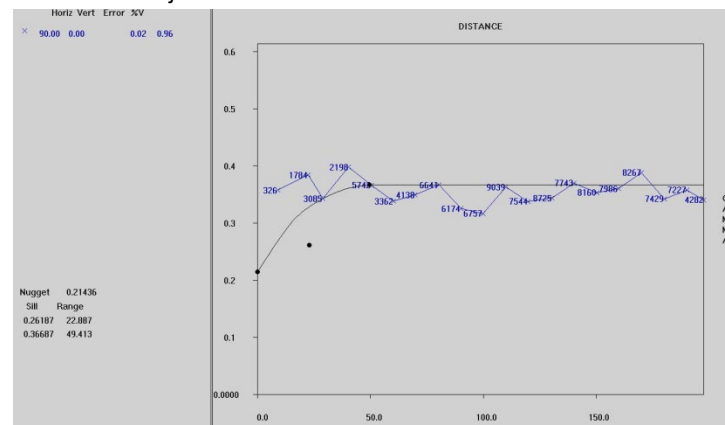
Table 11-4
Gold and Silver variogram model parameters used for resource estimation

Area	C0	Sill	Azi	Plung e	Dip e	Structure 1					Structure 2				
						Type	C1	Ranges (ft)			Type	C2	Ranges (ft)		
								Majo r	Mino r	Vert .			Majo r	Mino r	Vert .
Au_Low-Grade	0.445	0.589	11.0	-30	6	Sph	0.096	37	35	43	Sph	0.048	67	75	95
Au_High-Grade	0.292	0.423	0.0	0.0	0	Sph	0.012	30	30	30	Sph	0.119	89	89	89
Ag_Low-Grade	0.224	0.443	11.0	-35	0	Sph	0.081	62	74	45	Sph	0.138	98	114	77
Ag_High-Grade	0.233	0.368	0.0	0.0	0	Sph	0.145	65	65	65	Sph				

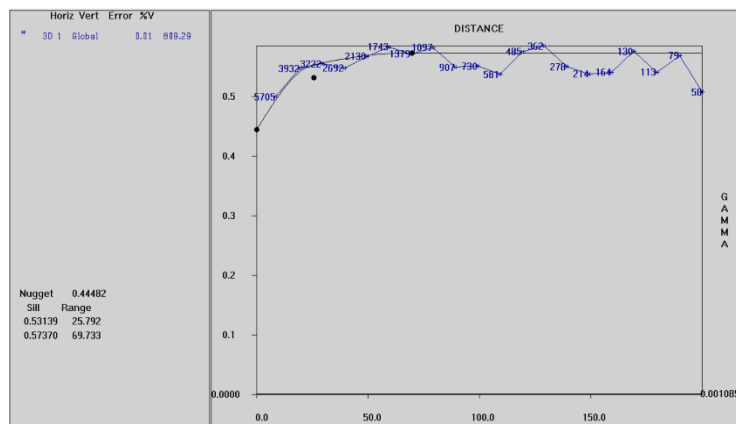
(Azi: Azimuth, Sph: spherical, C0: nugget effect, C1: covariance contribution of structure 1, C2: covariance contribution of structure 2, Plunge: dip of main axis, Dipe: easterly dip of main axis)



Major Direction



Minor Direction



Down Hole (Nugget)

Figure 11-8
Modelled Semi-variograms for Gold Composites within the Lower-Grade Gold Domains
of the Longstreet Deposit

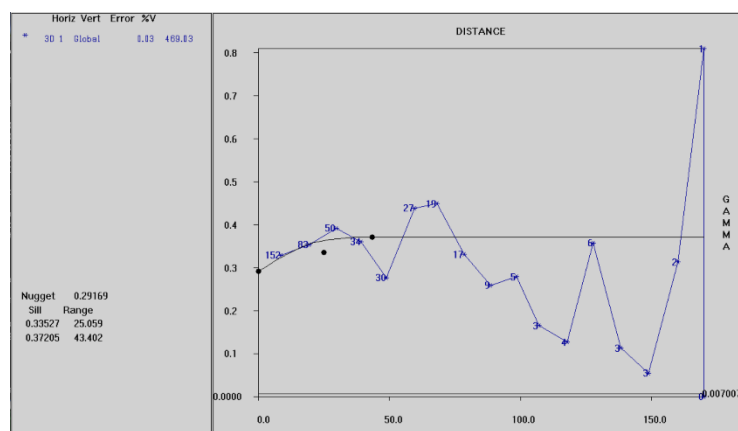
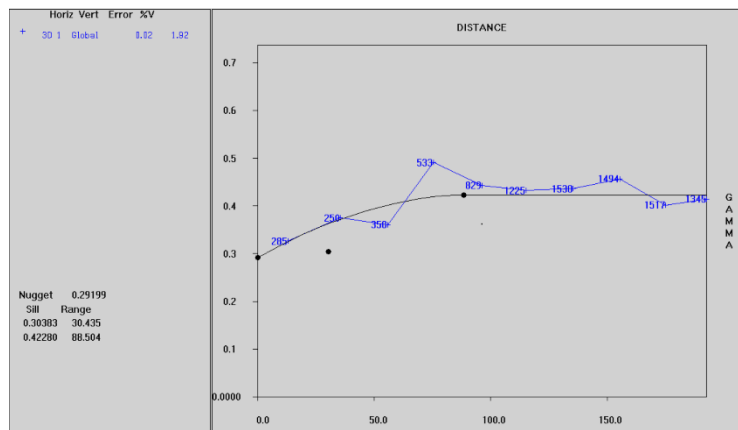
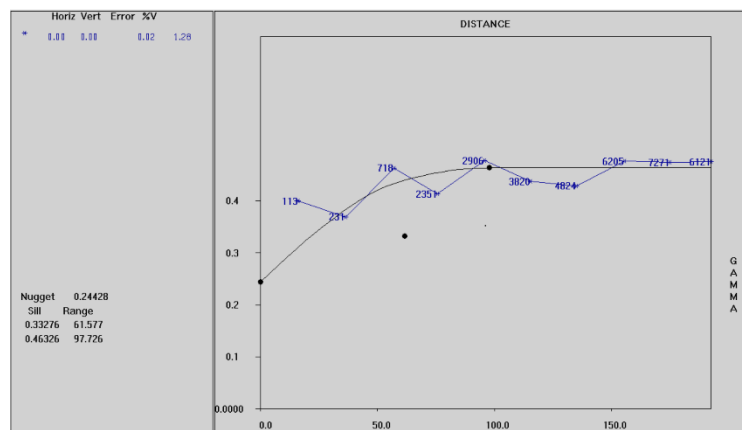
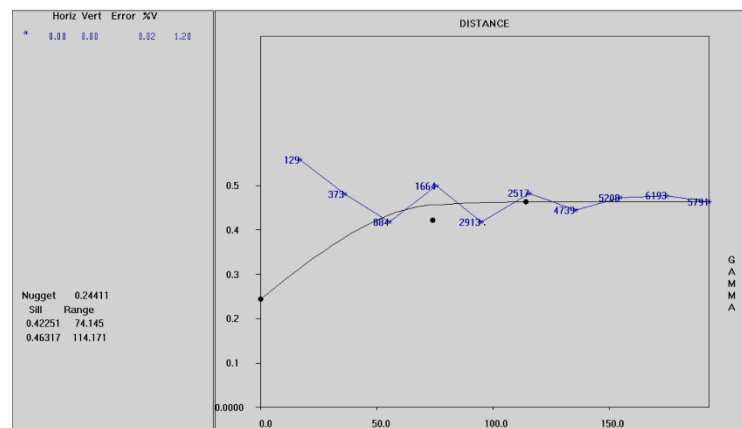


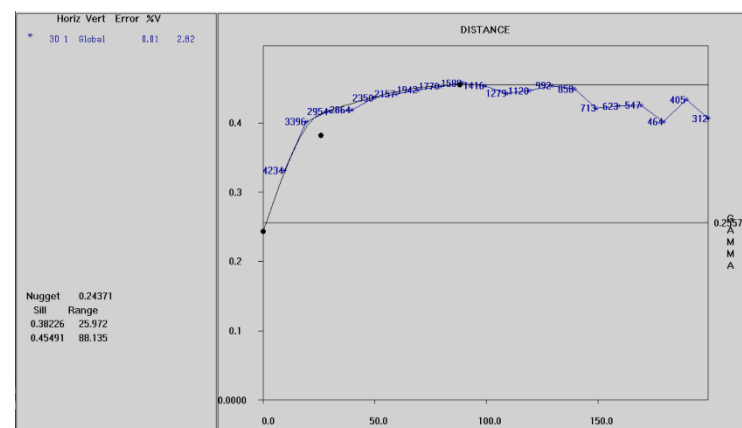
Figure 11-9
Modelled Semi-variograms for Gold Composites within the Higher-Grade Gold Domains of the Longstreet Deposit



Major Direction

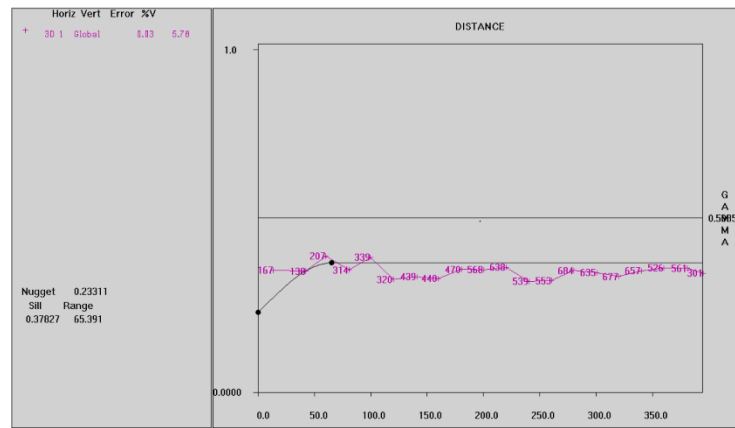


Minor Direction

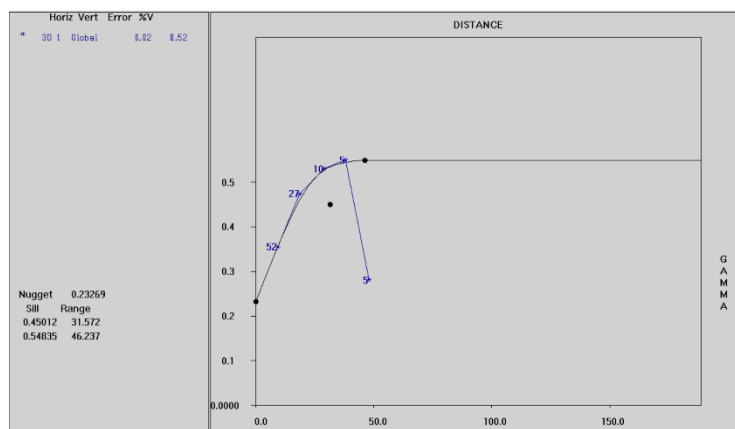


Down Hole (Nugget)

Figure 11-10
Modelled Semi-variograms for Silver Composites within the Lower-Grade Silver Domains
of the Longstreet Deposit



Omni-Directional



Down Hole (Nugget)

Figure 11-11

Modelled Semi-variograms for Silver Composites within the Higher-Grade Silver Domains of the Longstreet Deposit

11.8 Estimation Parameters for the Longstreet Mineral Resource

The gold and silver grades were estimated using three different methods: ordinary kriging, inverse distance weighted and the nearest neighbor method. Resource reporting uses the inverse distance weighted grades while the checking and validation of the estimation results used the results for nearest neighbor and kriging. Estimation parameters are given in Table 11-5 for gold and Table 11-6 for silver. The estimation of grade into each block for both the gold model and the silver model was completed in two passes. The first pass used search parameters roughly 2-times greater than that of the respective variogram. This was done to interpolate a grade value into every block within the defined mineral domain. The second over-writing pass was run with a search distance approximately equal to the variogram. Both passes used the same variogram parameters. Only those composites from each domain for both gold and silver were used to estimate into blocks from the same domain. Parameters obtained from the variogram analysis were used in the ordinary kriging interpolation and also provided the modelling parameters used in the inverse distance weighted interpolation. Cross sections of the block model grades are shown in Figure 11-12 and Figure 11-13 for gold and silver respectively.

Table 11-5
Longstreet Deposit Gold Estimation Parameters

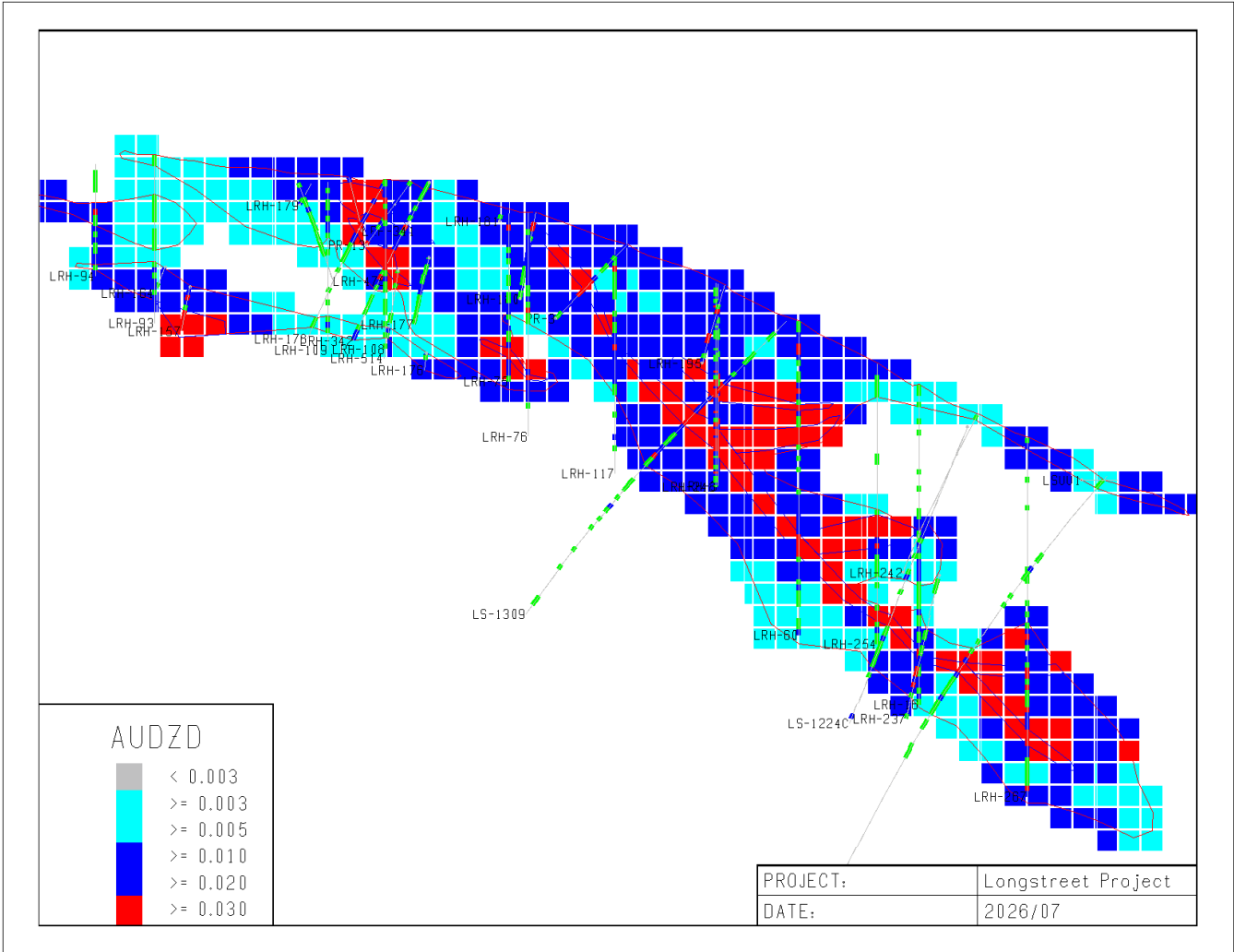
Longstreet Deposit Low-Grade Gold Mineralized Domain	
Minimum/Maximum composites – Pass 1, LG domain	1 / 12
Minimum/Maximum composites – Pass 2, LG domain	2 / 12
Maximum composites per hole Pass 1, LG domain	4
Maximum composites per hole Pass 2, LG domain	3
Primary Estimation method (power)	IDW (3)
Cutoff Grade for Changing Search Distance Pass 1	0.100
Search Distance at Specified Cutoff Pass 1	100
Cutoff Grade for Changing Search Distance Pass 2	0.100
Search Distance at Specified Cutoff Pass 2	50
Search distances – Pass 1	200 / 150 / 100
Search distances – Pass 2	100 / 75 / 50
Search directions (°) (Bearing, Plunge, Dipe)	11 / -30 / 6
Length-weighting of composites	Yes

Longstreet Deposit High-Grade Gold Mineralized Domain	
Minimum/Maximum composites – Pass 1, HG domain	1 / 12
Minimum/Maximum composites – Pass 2, HG domain	2 / 12
Maximum composites per hole Pass 1, HG domain	3
Maximum composites per hole Pass 2, HG domain	3
Primary Estimation method (power)	IDW (3)
Cutoff Grade for Changing Search Distance Pass 1	0.400
Search Distance at Specified Cutoff Pass 1	100
Cutoff Grade for Changing Search Distance Pass 2	0.400
Search Distance at Specified Cutoff Pass 2	50
Search distances – Pass 1	200 / 150 / 100
Search distances – Pass 2	100 / 100 / 50
Search directions (°) (Bearing, Plunge, Dipe)	0 / 0 / 0
Length-weighting of composites	Yes

Table 11-6
Longstreet Deposit Silver Estimation Parameters

Longstreet Deposit Low-Grade Silver Mineralized Domain	
Minimum/Maximum composites – Pass 1, LG domain	1 / 12
Minimum/Maximum composites – Pass 2, LG domain	2 / 12
Maximum composites per hole Pass 1, LG domain	4
Maximum composites per hole Pass 2, LG domain	3
Primary Estimation method (power)	IDW (3)
Cutoff Grade for Changing Search Distance Pass 1	1.0
Search Distance at Specified Cutoff Pass 1	100
Cutoff Grade for Changing Search Distance Pass 2	1.0
Search Distance at Specified Cutoff Pass 2	75
Search distances – Pass 1	225 / 150 / 100
Search distances – Pass 2	100 / 100 / 75
Search directions (°) (Bearing, Plunge, Dip)	11 / -35 / 0
Length-weighting of composites	Yes

Longstreet Deposit HG-Grade Silver Mineralized Domain	
Minimum/Maximum composites – Pass 1, LG domain	1 / 8
Minimum/Maximum composites – Pass 2, LG domain	2 / 8
Maximum composites per hole Pass 1, LG domain	3
Maximum composites per hole Pass 2, LG domain	3
Primary Estimation method (power)	IDW (3)
Cutoff Grade for Changing Search Distance Pass 1	4.0
Search Distance at Specified Cutoff Pass 1	100
Cutoff Grade for Changing Search Distance Pass 2	4.0
Search Distance at Specified Cutoff Pass 2	50
Search distances – Pass 1	200 / 200 / 100
Search distances – Pass 2	125 / 100 / 50
Search directions (°) (Bearing, Plunge, Dip)	0 / 0 / 0
Length-weighting of composites	Yes



Source: Ashton 2026

Figure 11-12
Longstreet Deposit Cross-section along 1300NW illustrating the gold estimated block model (undiluted)



Figure 11-13

11.9 Definitions

The Mineral Resources stated in this report for the Longstreet project conform to the definitions and categories set out in the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM Council on May 10, 2014. The CIM definitions are provided below for reference:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

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

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

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected

that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Mineral Resource

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

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to Probable Mineral Reserves.

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

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

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

This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

11.10 Classification

The Longstreet mineral resources are classified on the basis of the distance of the model block to the nearest composite, a minimum number of composites, and minimum number of drill holes. Two isotropic estimation passes were used to classify the resources (Table 11-7). All blocks that 'found' at least six composites with the nearest being less than 50 feet away and a minimum of three drill holes (Pass 1) are classified as indicated. The measured category (Pass 2) required a minimum of ten composites with the nearest being less than 25 feet away and a minimum of four drill holes. All remaining blocks are classified as Inferred. It is probable that with additional infill drilling the inferred resources will be upgraded to the indicated or even measured classification.

Table 11-7
Longstreet Classification Parameters

CLASS	Within Mineral Domain	Min. No. Composites	Min. No. Drill Holes	Max Dist. (ft) To Nearest Composite	Additional Restrictions
Measured	YES	10	4	25	None
Indicated	YES	6	3	50	None
Inferred	YES	1	1	Remaining Modeled Mineralization	

11.11 Longstreet Project Mineral Resources

The Longstreet project mineral resources have been estimated to reflect potential open-pit extraction and potential processing by heap leaching and carbon-in-leach extraction. To meet the requirement of the in-pit resources having reasonable prospects for eventual economic extraction, open pit optimizations for the Longstreet project were developed from the resource block model estimate using Minesight's Lerchs-Grossman pit optimization algorithm. Reasonable mining assumptions were applied to evaluate the portions of the block model (Measured, Indicated, and Inferred blocks) that could be "reasonably expected" to be mined from an open pit. The optimization parameters presented in Table 11-8 were selected based on experience, engineering principals and benchmarking against similar projects. The resulting optimized pit is used to prepare of a mineral resource statement and as a guide for designing a mineable pit with safety benches and haul road.

Table 11-8
Pit Optimization Parameters for Mineral Resources

Parameter	Longstreet	Unit
Mining Cost - Ore	\$ 2.75	\$/ton mined
Mine Haulage Cost - Ore	\$ 2.50	\$/ton mined
Mining Cost - Waste	\$ 2.25	\$/ton mined
Heap Leach Processing (Oxide)	\$9.34	\$/Ton processed
G&A Cost	3,634	\$1,000s/year
Tons per Day (years 2-4)	6,803	tons/day processed
Tons per Year (years 2-4)	1,701,000	tons/year processed
G&A per Ton	\$2.34	\$/tons processed
Au Refining Cost	\$ 5.00	\$/oz produced
NSR Royalty	3%	
Au Price	\$ 3,300	\$/oz produced
Ag price	\$ 40	\$/oz produced
Au Leach Recovery - Oxide	84%	
Ag Leach Recovery - Oxide	13%	

The optimized pit shell created using these economic parameters was used to constrain the project mineral resources to those that were potentially economic. The in-pit resources were further constrained with the application of a gold-equivalent cutoff grade of 0.005 opt. Evidence suggests that all the reported resources are oxide and amenable to heap leaching. Gold equivalency, as used in the application of the resource cutoffs, is a function of metal prices and metal recoveries. These variables, combined with the estimated gold and silver grades, are used

to calculate a gold equivalent grade for every block in the model. An example of the calculation of the gold-equivalent grade ("oz AuEq/t") is as follows:

$$\text{oz AuEq/t} = \text{oz Au/t} + (\text{oz Ag/t} \div ((3,300 \times \text{Au Recovery}) \div (40 \times \text{Ag Recovery})))$$

where "oz Au/t" and "oz Ag/t" are the estimated gold and silver block-diluted grades, respectively, and the other parameters are the metal prices and recoveries.

The total Longstreet project resources are summarized in Table 11-9. The Longstreet mineral resource estimate is the fully block diluted Inverse Distance Weighted estimate and is reported at a cutoff grade of 0.005 opt for open-pit mining. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

It is reasonably expected that the majority of Inferred mineral resources could be upgraded to the Indicated or Measured classification with continued exploration.

Mineral resources at various gold equivalent cutoff grades are shown in Table 11-10. These block-diluted resources are tabulated at various cutoffs in order to provide grade-distribution information, as well as to provide for economic conditions other than those envisioned by the reported resource cutoffs. It should be noted that the estimated resources include the IA mine schedule tons discussed in Sections 13 and 19 of this report.

Table 11-9
Longstreet Gold and Silver Mineral Resources

Measured & Indicated Resources							
Classification	Tons	Grade (oz/t)			Ounces	Ounces	Ounces
(oz. Au/ton)		Gold	Silver	Gold Eq.	Gold	Silver	Gold Eq.
Measured (.005)	1,050,336	0.019	0.382	0.020	20,000	401,200	20,700
Indicated (.005)	5,246,679	0.016	0.427	0.017	83,900	2,240,300	88,200
Total Measured & Indicated Resources							
Total M&I	6,297,015	0.016	0.419	0.017	103,900	2,641,500	108,900
Inferred Resources							
Classification	Tons	Grade (oz/t)			Ounces	Ounces	Ounces
(oz. Au/ton)		Gold	Silver	Gold Eq.	Gold	Silver	Gold Eq.
Inferred (.005)	481,517	0.014	0.400	0.015	6,700	192,600	7,100
Total MI&I	6,778,532	0.016	0.418	0.017	110,600	2,834,100	116,000

Notes to Mineral Resource Estimate:

1. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, changes in global gold markets or other relevant issues. The CIM definitions (2014) were followed for classification of Mineral Resources. The quantity and grade of reported inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as indicated Mineral Resource. It is probable that further exploration drilling will result in upgrading them to the indicated or measured Mineral Resource category.
2. The Mineral Resource Estimate incorporates over 93,071 feet of reverse circulation and core drilling in 437 holes, and outlines a near surface resource.
3. The resource was prepared by James Ashton, P.E., an independent QP, with an effective date of May 29, 2026.
4. The mineral resources are constrained by an optimized pit and presented at variable diluted gold equivalent cutoff grades, which represents mineralization that is potentially available for open-pit mining and heap-leach processing.
5. The undiluted Deep Skarn mineralization resources were quantified based on deep tabular solids representing potentially underground mineable lenses.
6. Gold equivalent values are a function of metal price and metal recoveries.
7. Rounding may result in apparent discrepancies between tons, grade, and contained metal content.

Table 11-10
Longstreet Mineralized Material

Measured Resources							
Cutoff (oz. Au/ton)	Tons	Grade (oz/t)			Ounces	Ounces	Ounces
		Gold	Silver	Gold Eq.	Gold	Silver	Gold Eq.
0.003	1,198,725	0.017	0.353	0.018	20,400	423,100	21,200
0.004	1,129,304	0.018	0.366	0.019	20,300	413,300	21,100
0.005	1,050,336	0.019	0.382	0.020	20,000	401,200	20,700
0.008	816,332	0.022	0.433	0.023	18,000	353,500	18,600
0.010	702,953	0.025	0.460	0.026	17,600	323,400	18,200
0.012	596,589	0.027	0.487	0.043	16,100	290,500	25,400
0.015	475,780	0.031	0.527	0.032	14,700	250,700	15,200
0.020	329,497	0.038	0.579	0.039	12,500	190,800	12,900
0.025	235,922	0.045	0.645	0.046	10,600	152,200	10,900
Indicated Resources							
0.003	6,039,648	0.014	0.391	0.015	84,600	2,361,500	89,000
0.004	5,640,858	0.015	0.409	0.016	84,600	2,307,100	88,900
0.005	5,246,679	0.016	0.427	0.017	83,900	2,240,300	88,200
0.008	4,036,276	0.019	0.489	0.020	76,700	1,973,700	80,400
0.010	3,205,584	0.021	0.541	0.022	67,300	1,734,200	70,600
0.012	2,554,770	0.024	0.594	0.025	61,300	1,517,500	64,200
0.015	1,883,956	0.029	0.662	0.030	54,600	1,247,200	57,000
0.020	1,226,823	0.035	0.747	0.036	42,900	916,400	44,700
0.025	884,089	0.041	0.818	0.043	36,200	723,200	37,600
Measured & Indicated Resources							
0.003	7,238,373	0.015	0.385	0.015	105,000	2,784,600	110,200
0.004	6,770,162	0.015	0.402	0.016	104,900	2,720,400	110,000
0.005	6,297,015	0.016	0.419	0.017	103,900	2,641,500	108,900
0.008	4,852,608	0.020	0.480	0.020	94,700	2,327,200	99,000
0.010	3,908,537	0.022	0.526	0.023	84,900	2,057,600	88,800
0.012	3,151,359	0.025	0.574	0.028	77,400	1,808,000	89,600
0.015	2,359,736	0.029	0.635	0.031	69,300	1,497,900	72,200
0.020	1,556,320	0.036	0.711	0.037	55,400	1,107,200	57,600
0.025	1,120,011	0.042	0.782	0.043	46,800	875,400	48,500
Inferred Resources							
0.003	880,028	0.009	0.281	0.010	7,900	247,300	8,400
0.004	578,598	0.012	0.362	0.013	6,900	209,500	7,300
0.005	481,517	0.014	0.400	0.015	6,700	192,600	7,100
0.008	329,215	0.018	0.473	0.019	5,900	155,700	6,200
0.010	239,772	0.022	0.545	0.023	5,300	130,700	5,500
0.012	175,194	0.026	0.619	0.027	4,600	108,400	4,800
0.015	117,055	0.034	0.740	0.035	4,000	86,600	4,100
0.020	76,331	0.043	0.915	0.045	3,300	69,800	3,400
0.025	60,355	0.049	0.991	0.051	3,000	59,800	3,100

In addition to the mineral resources reported herein, there is mineralization that continues beyond, and is contiguous with the reported mineral resources. The reported mineral resource estimate is pit-constrained and therefore some of the estimated mineralization (tons, grade and ounces) is

Figure 11-14
Longstreet Optimized Pit and Additional Mineralization (Looking West)

The Authors are unaware of any unusual title, taxation, marketing, or other such factors that may impact the potential development of the Longstreet project. The Author recommends that diamond-core drilling methods be used in the future as part of an infill drilling program at Longstreet with several of the holes drilled as twins to existing holes.

11.12 Validation

Validation of this model was done by comparing:

- Cross sectional interpreted domain volumes with calculated domain solid volumes,
- The IDW model to the nearest neighbor and ordinary kriging models,
- Grade distributions of composites and the model,
- Swath plots along the northing for gold and silver, and
- Visually reviewing the block grades to the composite grades.

11.12.1 Model Volume Check

As a check that the model has not over stated the volume of mineralized material a comparison was made to the mineral domains for both gold and silver. Table 11-11 shows the results of this comparison. Generally, the blocks had a lower volume than the solids but all were very close.

Table 11-11
Model-Solid Volume Comparison

	Low-Grade Gold Domain	High-Grade Gold Domain	Low-Grade Silver Domain	High-Grade Silver domain
Volume of Solid (yd ³)	4,111,314	277,732	4,885,298	183,342
Volume of Blocks (yd ³)	4,132,444	277,333	4,879,407	181,037
% Difference	5.14%	-0.14	-0.12%	-1.26%

11.12.2 Comparison of Interpolation Methods

The Inverse Distance Weighted, Ordinary Kriging and Nearest Neighbor models are tabulated for comparison purposed in Table 11-12 for the Longstreet deposit.

Table 11-12
Comparison of Interpolated Methods (Global Mineralization)

Cutoff	IDW Model		Krige Model		NN Model	
Au (oz/t)	Tons	Au (oz/t)	Tons	Au (oz/t)	Tons	Au (oz/t)
0.000	7,922,379	0.016	7,922,379	0.015	7,922,379	0.015
0.003	7,877,510	0.016	7,916,350	0.015	7,384,357	0.016
0.005	7,430,909	0.016	7,728,614	0.016	6,135,505	0.019
0.010	4,555,686	0.022	4,819,215	0.021	3,779,486	0.026
0.015	2,591,307	0.030	2,353,299	0.030	2,621,127	0.033
0.025	1,165,691	0.045	1,036,342	0.045	1,323,992	0.047
0.050	292,286	0.076	261,570	0.073	364,510	0.082
Ag (oz/t)	Tons	Ag (oz/t)	Tons	Ag (oz/t)	Tons	Ag (oz/t)
0.00	6,921,036	0.54	6,921,036	0.53	6,921,036	0.53
0.15	6,818,117	0.55	6,895,963	0.54	6,137,619	0.59
0.25	5,753,259	0.61	6,104,478	0.58	4,480,752	0.73
0.50	2,327,601	0.99	2,284,728	0.95	2,261,326	1.11
1.00	667,777	1.80	609,401	1.77	816,285	1.83
2.00	160,910	3.13	151,134	2.93	212,953	3.22
4.00	27,708	5.54	19,050	5.57	46,835	5.35

11.12.3 Grade Distribution of Composites Versus Models

The relative degree of smoothing in the block model estimates were evaluated by comparing the model grade distribution to the distribution of the underlying composites. The comparison is made using grade/volume curves. Comparisons between the IDW and Krige models to the composites are shown for the low-grade domains and the high-grade domains for both gold and silver in Figure 11-15 through Figure 11-18. In general, the curves for the low-grade resource domains for both gold and silver have a good comparison. The high-grade domain curves show that for the highest grades the models may have underestimated both the grade and volume in these areas.

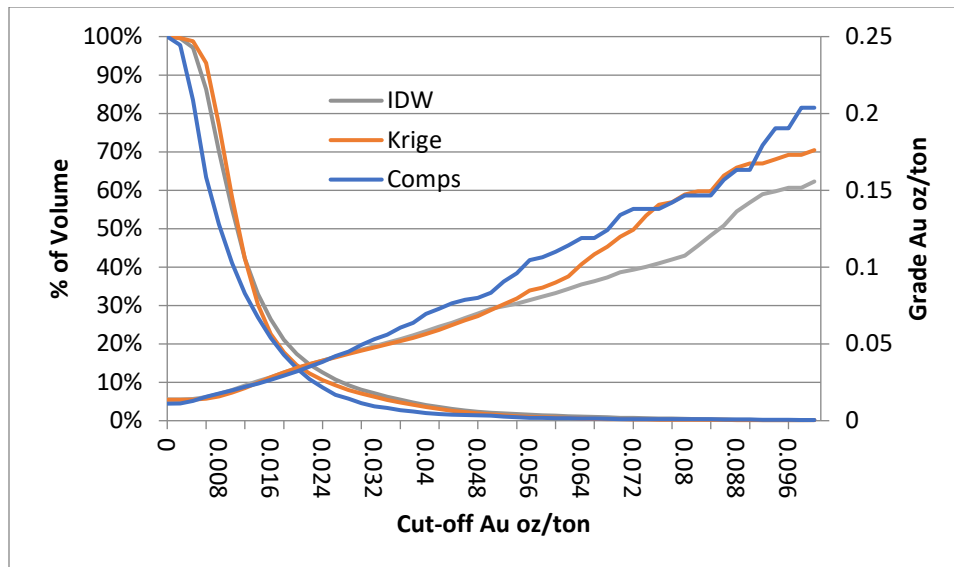


Figure 11-15
Grade/Volume Curves for Low-Grade Gold Domain

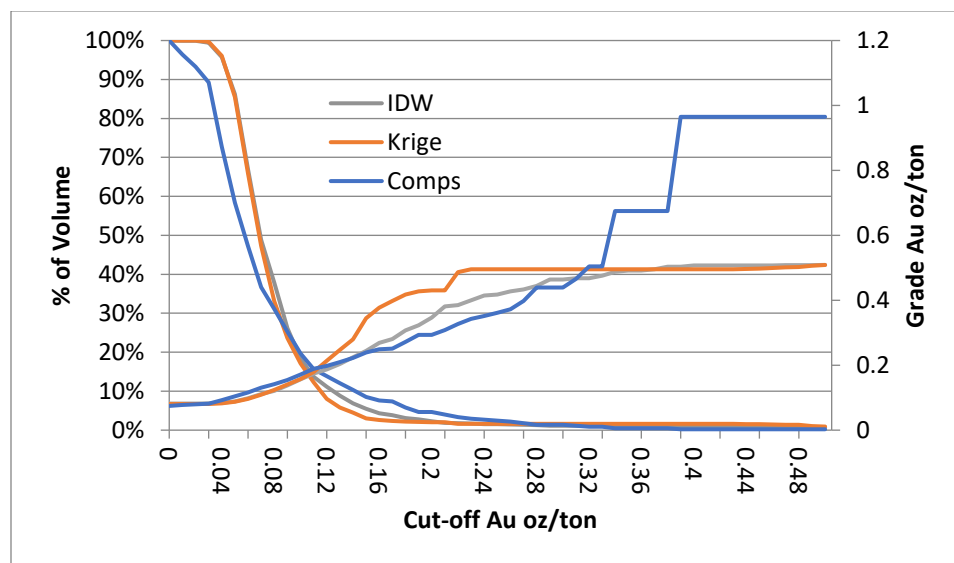


Figure 11-16
Grade/Volume Curves for High-Grade Gold Domain

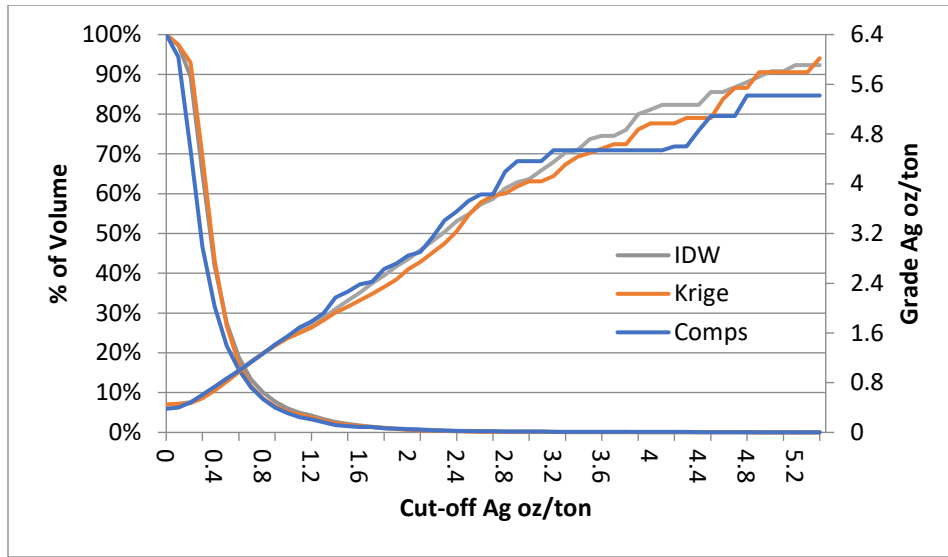


Figure 11-17
Grade/Volume Curves for Low-Grade Silver Domain

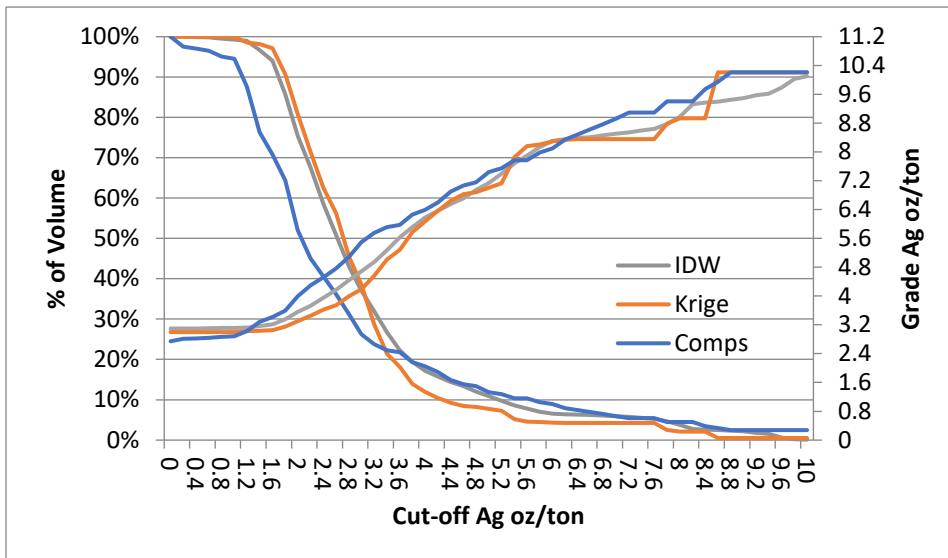


Figure 11-18
Grade/Volume Curves for High-Grade Silver Domain

11.12.4 Swath Plot Analysis

Swath plots are an important validation tool for providing comparisons between sample points (composited or un-composited) and estimated values to identify any bias towards under-estimation or over-estimation or any smoothing in the results. The swath plot is a one-dimensional graph in a specific direction of interest. A swath is a sectional slice through the block model with a specified thickness. The swath plot shows the average grade for the blocks in the swath, along with the averaged sample values in the swath.

They are generated by calculating the average gold and silver grades of composites and the estimated block model gold and silver grades within directional slices through the length of mineralization. For the Longstreet deposit swaths were generated along the eastings with a swath window of 100 feet. Swath plots for the Longstreet deposit for gold and silver estimates are illustrated in Figure 11-19 and Figure 11-20, respectively. There are minor instances of localized over- and under-estimation within the block models; overall, the block model adequately reproduces the trends observed in both gold and silver composites.



Figure 11-19

Swath Plot Comparing Composites Versus Estimated Block Model Gold Grade



Figure 11-20

Swath Plot Comparing Composites Versus Estimated Block Model Silver Grade

11.12.5 Visual Review

Detailed visual review of the block models has been conducted in both cross section and plan view. The review included checking that blocks were coded properly and calculations were made correctly. Of particular importance was the checking that domain percentages were coded properly to each block. The visual review of the model block grades compared to surrounding composite values was a major factor in selecting the inverse distance weighted method over other methods.

11.13 Comments on the Mineral Resource Estimate

During a review of the Longstreet Project data and mineral resource estimation, the Author noted several issues that should be addressed in all future estimates and economic studies. It is believed that any changes that might occur from improving these parameters and data will not have a significant impact on the total resources but will affect the ability to better predict mineralization and make the model more precise. The issues that require additional study and review include:

Density: Additional density testing should be done in order to support the existing density being used. The results from the testing thus far show a fairly consistent density for the various rock types and the Author does not envision any major changes to the resource from additional testing.

Controls on mineralization: Presently the controls on high-grade mineralization are not fully understood and are represented by domains based principally on mineral grade and not on geologic interpretation. While tons, grade and ounces might vary with a more detailed model, significant changes to the resource are not expected.

Review oxide/sulfide boundary: Currently the mineral resource is expected to be completely oxidized. This assumption needs to be investigated more thoroughly by reviewing drillhole logs. Any new drillholes need to be logged for evidence of sulfides.

Additional drilling: Drilling in the future will primarily be infill drilling. This additional drilling should incorporate as many core holes as feasible. Several of the higher-grade areas should be investigated to try and determine the geologic controls. Down hole surveys should be recorded for all future RC and core drill holes especially if the hole is an angle hole. With any new core drilling, the core should be logged for geotechnical properties.

12.0 Mineral Reserve Estimates

No mineral reserves are estimated for this Report.

13.0 Mining Methods

The Longstreet Project contains one mining area: the Main Pit. The pit will be mined top down as the size of the pit and steepness of the terrain does not allow for a phased approach. This PEA for the Longstreet Project is based on standard open-pit mining methods, with drill and blast rock breakage, and truck and loader material handling all performed by a mining contractor. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used to create a mine design, site layout, and scheduling to determine the economics for the PEA includes:

- Define geometric parameters and constraints.
- Define assumptions for the economic parameters.
- Run pit optimizations.
- Define road and ramp parameters.
- Create pit designs.
- Create waste dump designs.
- Produce mine and process production schedules.
- Define personnel and equipment requirements.
- Estimate mining costs.
- Perform an economic analysis.

Section 13 summarizes the above topics, except for the economic analysis discussed in Section 19.0.

The global resource model described in Section 11.0 was the basis for developing the final mineable pit using the optimized mineral resource pit as a guide for designing the mineable pit. The mine production schedule was based on a maximum daily production rate of 13,000 tons/day for both ore and waste rock. The production schedule was designed to provide a constant feed of mineralized material to the crusher and onto the heap leach pad via conveyor loading.

The term “ore” generally implies that sufficient technical feasibility and economic viability studies have been completed to classify the material as mineral reserve. A Qualified Person has not done sufficient work to classify the mineral resource at the Longstreet Project as current mineral reserve and the issuer is not treating the mineral resource as mineral reserve. The term “ore” is used in this report section to describe mining methods in standard mining terms.

13.1 Geometric Design Parameters

Geometric parameters include pit design parameters and land constraints. The author is not aware of any geotechnical studies that have been completed to date for the Longstreet open pit. Generally, the stability of open pit slopes developed to the same height and at the same slope angles in the same geologic units is expected to be similar to past open pits providing the material properties, groundwater, and structural geology conditions are also similar. There are two open pits within a short distance to the proposed Longstreet pit which have similar material, structural, and groundwater properties, the Manhattan pit and the large Round Mountain pit (currently in production). Using geometric measurements from these two pits, the pit design parameters used for the Longstreet pit were developed and are shown in Table 13-1. A 45° overall pit slope was used for pit optimization prior to final pit design.

Table 13-1
Longstreet Pit Design Parameters

Pit Geometric Parameters:	Units	Design Values
Rock Face (triple Bench)	feet	60
Nominal Bench Height	feet	20
Safety Bench width	feet	22
Bench Face Angle	degrees	0.51:1.0 (H:V), 63.°
Interramp Slope Angle	degrees	0.87:1 (H:V), 49°
Haul Road Width - Two Way	feet	60
Haul Road Grade - Two Way	percent	10%
Haul Road Width - Single Lane	feet	45
Haul Road Grade - Single Lane	percent	12%

All mining is anticipated to be above the water table, so no dewatering wells will be required. No significant amounts of groundwater have been intersected in the Near Surface drilling and a consistent water table has not been identified in the exploration drilling at Longstreet deposit to date. All project resources therefore appear to be above the water table. Storm water that enters the pit will be handled using in-pit sumps as needed. Any excess water that doesn't naturally infiltrate into the ground will be placed in water trucks using a portable pump and then used for dust control on haul roads.

13.2 Economic Parameters

Economic parameters were used to generate optimized pits using a Lerchs-Grossman algorithm within MineSight software. The economic parameters include mining costs, processing costs, general and administrative costs (“G&A”), refining costs, royalties, and metal recoveries. Mine planning is an iterative process, and initial costs and recoveries were assumed to determine how large the pit would be. The economic parameters were refined as concepts were developed on how material would be processed based on the oxidation state of the material. The methods for processing that were determined include:

- Use of three-stage crushing and cyanide heap leaching for oxide material; and
- Use of the Carbon-in-Leach process for gold recovery.

The economic parameters used are shown in Table 13-2. The project process rate is scheduled to be 5,500 tons per day with an average yearly (years 2-4) production of 1,701,000 tons. This process rate is used here to convert the fixed G&A component to a cost per ton for the purpose of pit optimization. The G&A cost applied as a fixed cost in the cash-flow model.

Table 13-2
Longstreet Economic Parameters

Pit Optimization Economic Parameters		
Parameter	Longstreet	Unit
Mining Cost - Ore	\$ 2.75	\$/ton mined
Mine Haulage Cost - Ore	\$ 2.50	\$/ton mined
Mining Cost - Waste	\$ 2.25	\$/ton mined
Heap Leach Processing (Oxide)	\$ 9.32	\$/Ton processed
G&A Cost	3,634	\$1,000s/year
Tons per Day (years 2-4)	5,500	tons/day processed
Tons per Year (years 2-4)	1,701,000	tons/year processed
G&A per Ton	\$ 2.34	\$/tons processed
Au Refining Cost	\$ 5.00	\$/oz produced
NSR Royalty	3%	
Au Price	\$ 3,300	\$/oz produced
Ag price	\$ 40	\$/oz produced
Au Leach Recovery - Oxide	84%	
Ag Leach Recovery - Oxide	13%	

Recoveries were applied based on recommendations by Kappes Cassidy & Associates (KCA), as summarized in Section 10.

13.3 Cutoff Grades

Cutoff grades were calculated based on the economic parameters shown in Table 13-2. These were calculated for the different material types and crushing method. Cutoff grades were calculated for both break-even and internal cutoffs. The internal cutoff grade calculation eliminates the mining cost in the calculation. The pit designs are based on economical pits and the materials inside of the pits are assumed to be mined whether the material is waste or ore. Thus, the decision on whether to process the material is made at the point where the truck needs to turn either to the waste dump or the crushing facility. The mining cost is therefore a sunk cost. The basic equation for the cutoff grade calculation is shown in Equation 1.

Equation 1: Breakeven Cutoff Grade Calculation (oz Au/t)

$$\frac{\text{Costs}}{(\text{Au}\$/\text{oz} - \text{RefCst}) * (1 - \text{Roy}\%) * \text{Rec}\%}$$

Where costs are all processing costs plus G&A costs in \$/t, RefCst is the refining cost in \$/oz gold produced, Roy% is the NSR royalty, and Rec% is the recommended recovery.

Cutoff grades are calculated in terms of oz Au/t, and are applied to a gold equivalent grade. The calculation for the gold equivalent grade for the PEA is shown in Equation 2.

Equation 2: Gold Equivalent Calculation

$$Au_{\text{Equivalent}} = Au \text{ oz/t} + Ag \text{ oz/t} / AuEq_{\text{Fact}}$$

Where:

$$AuEq_{\text{Fact}} = (Au_{\text{Price}} / Ag_{\text{Price}}) * (Au_{\text{Recovery}} / Ag_{\text{Recovery}})$$

Table 13-3 shows the cutoff grades for Longstreet at various gold prices and using operating costs as shown in Table 13-2. The economic analysis in this updated PEA has been done based on a gold price of \$3,600 per ounce Au and \$48 per ounce Ag and using the updated capital and operating costs described in Section 18.0. Because these cutoff grades are fairly low with respect to assay detection levels, a minimum cutoff grade for leaching oxide material of 0.005 oz Au/t has been applied.

Table 13-3
Longstreet Cutoff Grades (oz Au/t)

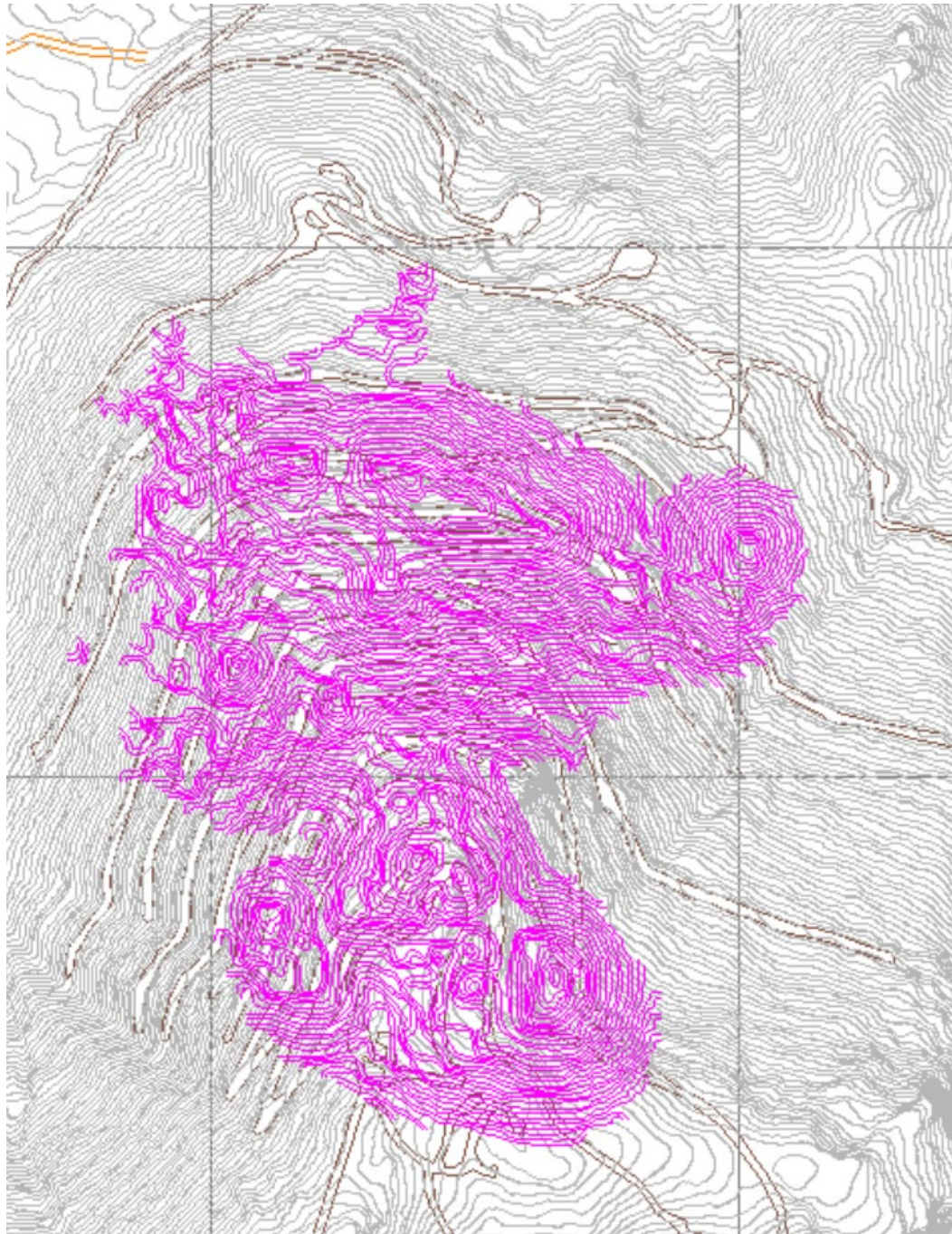
		Breakeven	Internal
\$ / oz Au		Oxide	Oxide
\$	2,700	0.008	0.007

\$	2,900	0.007	0.006
\$	3,100	0.007	0.006
\$	3,300	0.006	0.005
\$	3,500	0.006	0.005
\$	3,700	0.006	0.005
\$	3,900	0.005	0.005

13.4 Pit Optimization

Pit optimizations were run using MineSight. Inputs into the pit optimization include the resource block model along with the economic and geometric parameters previously discussed. Ultimate pit shells were selected from the optimization results to be used as a guide for designing the mineable pits used in the economic analysis.

The selection of ultimate pits is based on a simple economic analysis of the results of a set of optimized pit shells based on varying gold price. A set of seven optimized pit shells, using a Lerchs-Grossman ("LG") algorithm, were created starting with a gold price of \$2700/oz. gold and proceeding at \$200/oz. increments to \$3900/oz. gold. The \$3300/oz gold equivalent optimization was used to report the mineral resource as well as a guide for designing the mineable pit. A plan view of the final pit optimization shell for the Longstreet Project is shown on Figure 13-1.



Source: Ashton 2026

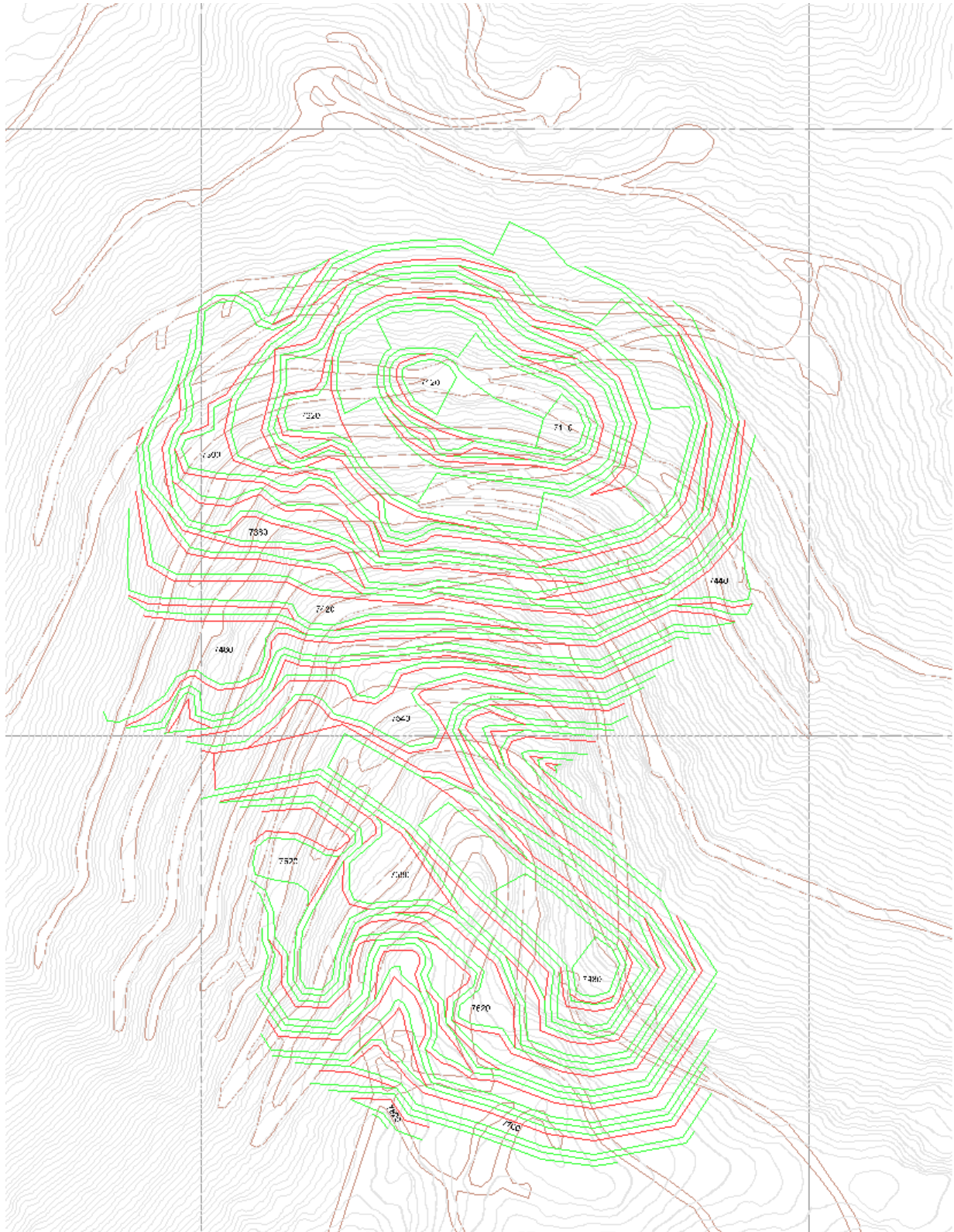
Figure 13-1
Final Optimized Pit (\$3300/Au oz.)

13.5 Open Pit Mine Design

Mine design and planning for the Longstreet pit was carried out based on the economic pit optimization shells. The bench volumes of mined resources and waste rock were generated using the MineSight block model. Dilution has been accounted for in the resource block model and included in the In-Pit Resource.

Open pit design parameters used in the design of the pit is shown in Table 13-1 and the final mineable pit used in the economic analysis is shown in Figure 13-2.

Road designs have been completed for the PEA to allow primary access for people, equipment, and consumables to the site. This includes haul roads between the designed pits, dumps, and proposed leach facility. Within the pit designs, ramps have been established for haul truck and equipment access. The initial pit access roads for the top benches will eventually be mined out and only the final ramps to the bottom of the pit will remain. The in-pit ramps will only require a single berm. Ramps outside of the pit will require two safety berms. One-lane traffic ramps are also utilized near the bottom of the pit where the strip ratio is minimal, and the traffic requirements are low.



Source: Ashton 2026

Figure 13-2
Final Mineable Pit (Red is Crests, Green is Toes)

13.6 In-Pit Resources

In-pit mineable mineral resources was estimated for the Longstreet pit. The total in-pit mineable resources, including dilution, are shown in Table 13-4. There are a total of 5.3 million tons of waste associated with the 6.5 million tons of material to be processed, and thus have an overall strip ratio of 0.81 tons of waste per ton of processed ore. The in-pit resources are reported using cutoff grade of 0.005 oz Au/t.

Table 13-4
In-Pit Mineable Resource for Longstreet Pit

	Units	Measured	Indicated	M&I	Inferred
Total Mineable Pit Resource	Tons	1,084,509	5,110,266	6,194,775	357,275
	oz Au/t	0.018	0.016	0.016	0.012
	Au Ozs	19,521	81,560	101,081	4,394
	oz Ag/t	0.370	0.429	0.419	0.378
	Ag Ozs	401,268	2,192,304	2,593,573	134,907
Total Waste	Tons	5,305,059			
Total In-Pit Tonnes	Tons	11,857,109			
Strip Ratio	W:O	0.81			

13.7 Leach Pad and Waste Dump Design

Waste dump and leach pad designs were created for the PEA to contain the waste and heap-leach material mined. A 32% swell factor and 3% in-situ moisture were assumed for the waste material which provides for both swell when mined and compaction when placed into the dump facility. The total volume requirements for containment of waste with a wet loose density of 18.38 cuft/ton and crushed leach material with a crushed density of 19.24 cuft/ton is 3.6 million cubic yards for waste material and 4.7 million cubic yards for leach material.

Both the waste dumps and Leach heap were designed with angle of repose of 35.5° and ultimate reclaimed slopes of 21.8° (2.5H to 1V).

Mineable pit, waste dumps, roads and mine specific facilities are shown in the following three figures. Figure 13-3 show the initial haul road layout, Figure 13-4 shows the start of mining, and Figure 13-5 shows the mine at the end of mining. The four waste dumps (West and East dumps) have a combined capacity of 4.1 million cubic yards. The leach pad, located 3 miles from the mine, has a capacity of 5.3 million cubic yards.

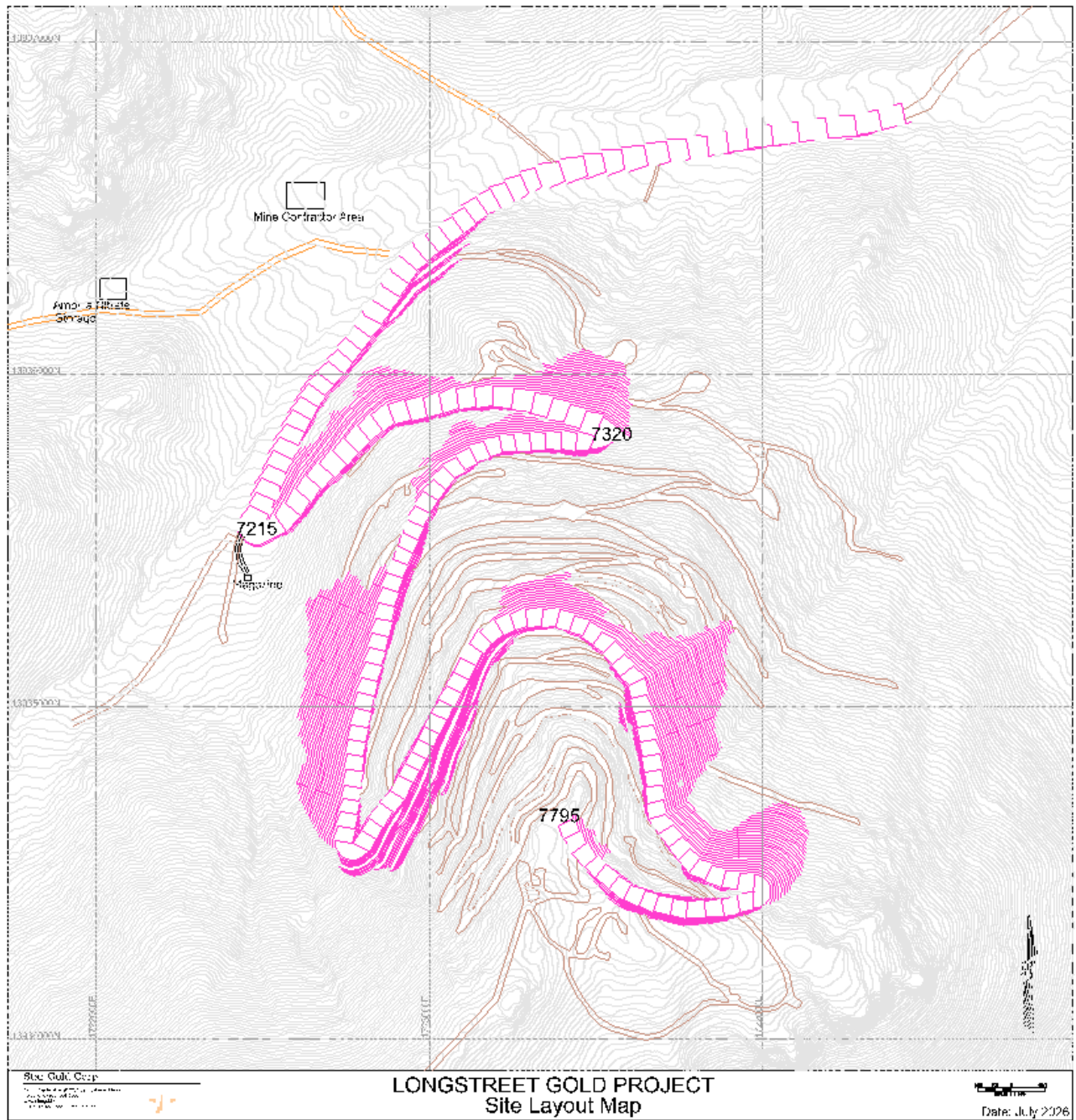


Figure 13-3
Initial Mine Haul Road Layout

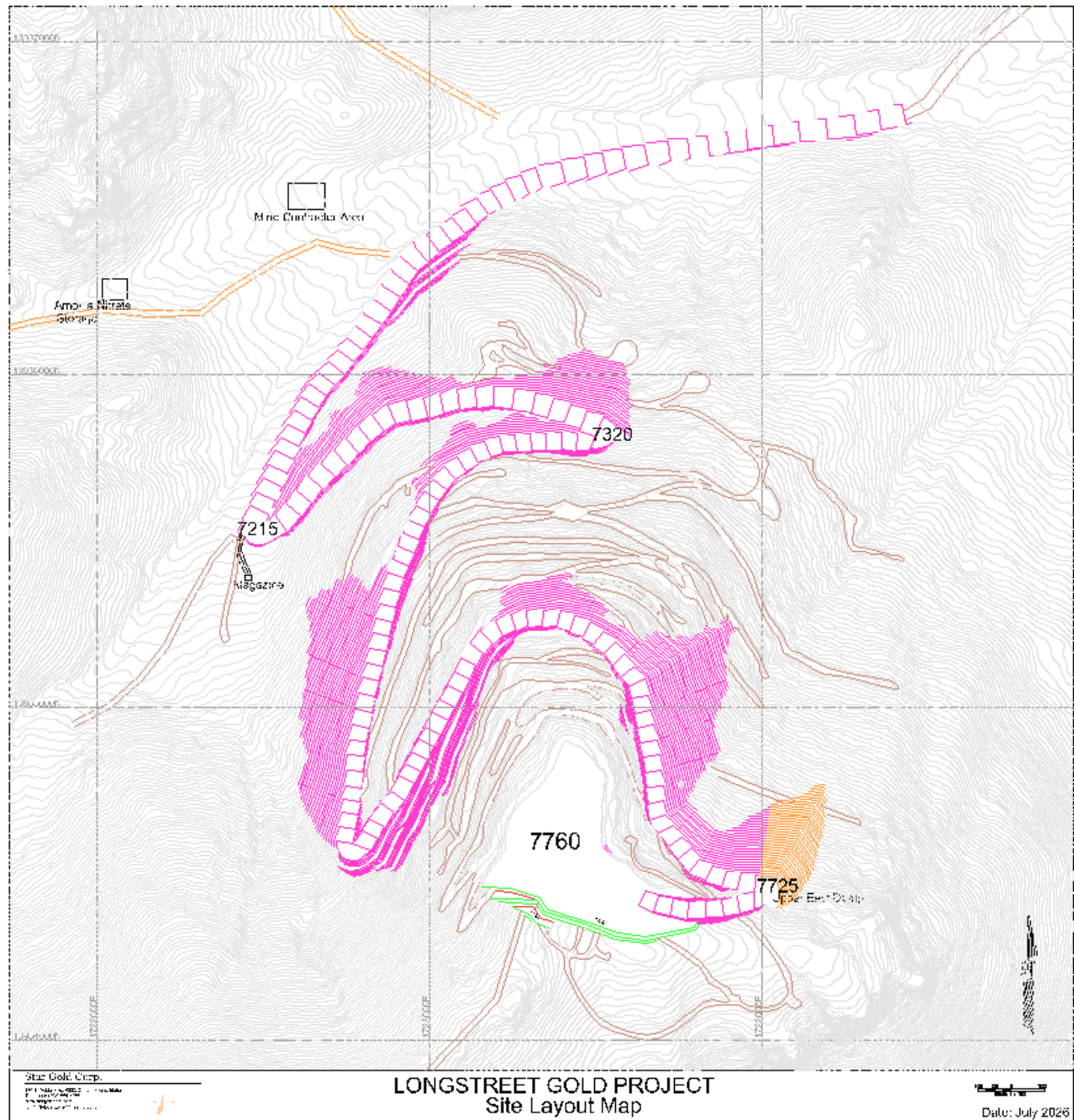


Figure 13-4
Start of Mining

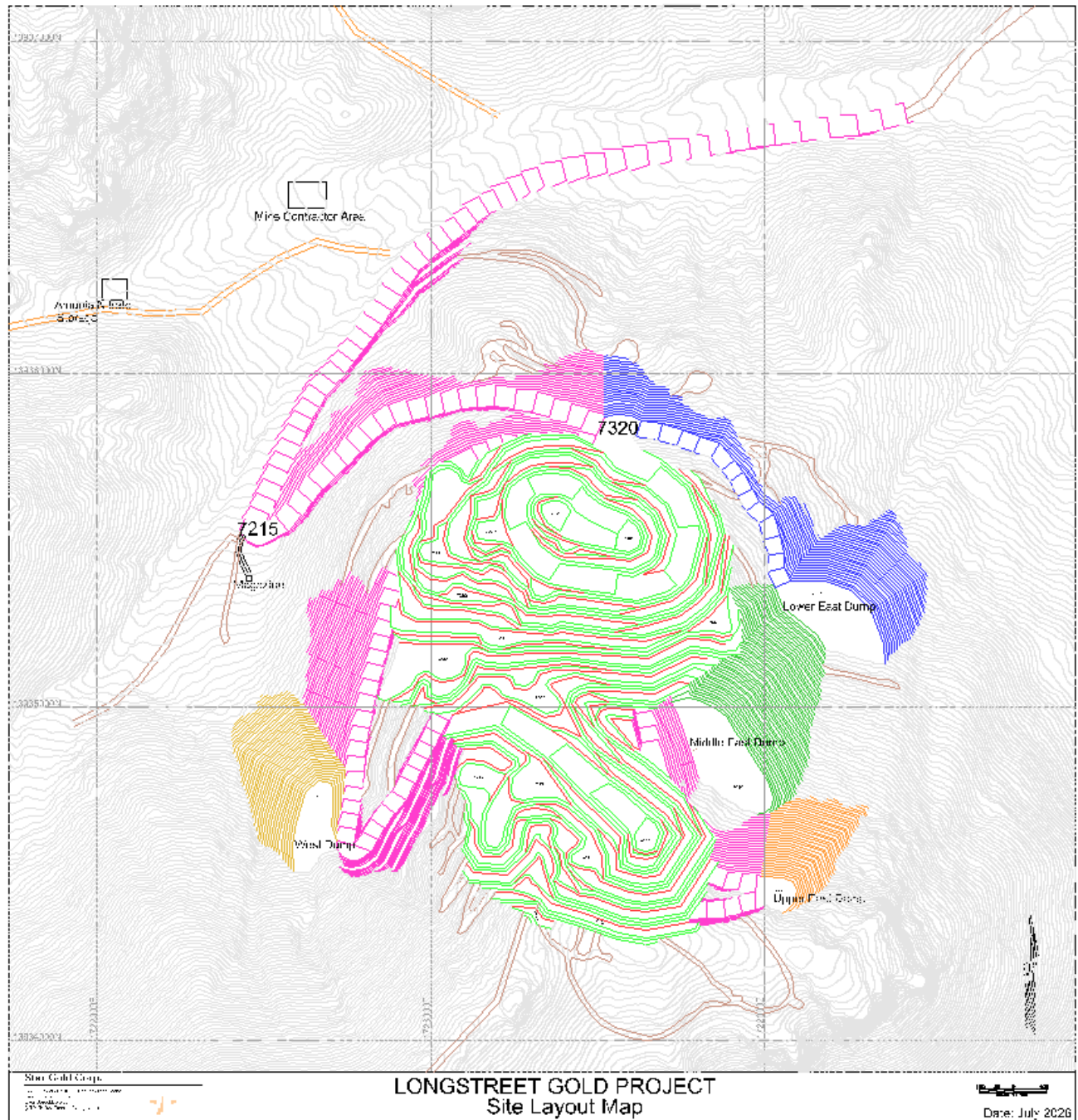


Figure 13-5
End of Mining

13.8 Mine Production Schedule

Measured, Indicated, and Inferred resources inside of the Longstreet mineable pit previously discussed were used to schedule mine production. The production schedule considers the processing of Longstreet ore material by 3-stage crushing and heap leaching.

Quarter-year periods were used to create the production schedule for the first year of production as the production rate increases as the pit opens up as mining advances. The ore material is near the surface and thus no pre-production period for waste mining was needed. Mining will occur in a top-down schedule with mining to the final pit limits as each bench is mined. Start of ore leaching and processing will begin concurrently with mining. Initial ore material will be crushed and used as over liner to protect the leach pad liner. The maximum rate for crushing and leach processing will be 5,500 tons per day. Full scale mining and processing will occur in the second half of year 1. Overall production in year 2 is lower due to the low strip ratio. Ore production is the highest in year 2. Ore production rates decrease in years 3 and 4 as mining comes to an end.

The daily mining rate will ramp up from an average of 9,500 tons per day during the first quarter of year-1 to a maximum of 13,000 tons per day in year 3. The yearly mining production schedule for Longstreet is summarized in Figure 13-5.

Table 13-5
Longstreet Mine Production Schedule

		Yr1-QTR1	Yr1-QTR2	Yr1-QTR3	Yr1-QTR4	Yr-2	Yr-3	Yr-4	Total
Leach Ore	Tons	263,190	335,010	404,030	447,670	2,168,720	1,839,870	1,093,560	6,552,050
Gold Grade	Oz Au/t	0.017	0.014	0.017	0.015	0.018	0.014	0.017	0.016
Contained Au	Ozs	4,345	4,851	6,733	6,664	39,467	25,326	18,074	105,461
Silver Grade	Oz Ag/t	0.317	0.307	0.354	0.346	0.393	0.387	0.622	0.416
Contained Ag	Ozs	83,312	102,826	142,854	154,749	851,420	712,763	680,548	2,728,471
Waste	Tons	334,862	389,602	402,526	359,316	619,083	1,406,132	1,793,538	5,305,059
Total Mined	Tons	598,052	724,612	806,556	806,986	2,787,803	3,246,002	2,887,098	11,857,109
Strip Ratio	W:O	1.27	1.16	1.00	0.80	0.29	0.76	1.64	0.81
Ave Prod/day	Tons	9,493	11,502	13,009	13,016	11,151	12,984	11,548	
Ave Ore Prod/day	Tons	4,178	5,318	6,517	7,220	8,675	7,359	4,374	

The leach process production schedule was created by KCA based on the mine production schedule, metal recoveries and leach lag times. The recoveries used to estimate recoverable gold are those shown in Table 13-2. The lag time is generated by estimating the extraction of the recoverable ounces on a month-by-month basis after placement of material. Table 13-6 **Error! Reference source not found.** shows the yearly process production schedule for the Longstreet Project.

The PEA total life-of-mine (“LOM”) gold production is estimated to be 88,587 ounces, with a LOM average recovery of 84%. Silver production is estimated to be 354,701 ounces, with an average LOM recovery of 13%.

Table 13-6
Longstreet Process Production Schedule

Units	Year 1	Year 2	Year 3	Year 4	Year 5	Total
K Tons	1,201	2,000	2,000	1,351	-	6,552
Gold						
Au opt	0.01595	0.01775	0.01376	0.01653	-	-
K Ozs Au Contained	19.151	35.491	29.194	21.624	-	105.46
K Ozs Au Recovered	16.087	29.813	24.523	18.164	-	88.587
K Ozs Au Produced	11.965	28.097	25.559	19.410	3.557	88.587
Silver						
Ag opt	0.331	0.386	0.387	0.622	-	-
K Ozs Ag Contained	401.226	772.472	774.311	780.462	-	2,728.471
K Ozs Ag Recovered	52.159	100.421	100.660	101.460	-	354.701
K Ozs Ag Produced	35.200	93.112	100.603	101.267	24.519	354.701

13.9 Mine Operations and Equipment

This PEA assumes that mining operations at Longstreet will be performed by a mine contractor thus eliminating the large capital expense of purchasing mine equipment. These costs however are reflected in higher mine operating costs. The relatively short mine life makes contract mining an economic and lower risk choice.

A mine contractor will be used for most mining activities including site preparation, haul road construction and maintenance, ore and waste drilling and blasting, excavation and haulage of ore and waste, and management of waste dumps. The mine contractor will provide all the required open pit mining equipment. The mine contractor will also provide operator training, safety training, supervision, and maintenance and office facilities for contractor’s operations.

Star Gold will provide mine technical services including overall mine management, blasting design, blasthole layout, ore grade control, mine planning, surveying, and blasthole sampling. The mine operation will be by conventional open pit mining methods utilizing rubber-tired wheel-loaders and 60-ton haul trucks. Productivity and cost analysis assumed a fleet of Caterpillar 988 wheel-loaders or an equivalent excavator to load Caterpillar 773 haul trucks. Drilling will be by conventional rotary drills drilling 6.5-inch diameter holes on a 13-ft burden by 13-ft spacing blast pattern.

At the crusher stockpile area, Star Gold will provide a front-end loader to feed the crusher from the coarse ore stockpile when trucks are not direct dumping into the primary crusher.

The mine operating schedule assumes 1-shift/day, 12-hours per shift, 250 days per year. The operating schedule assumes 8 days for holidays and two weather delay days per year.

The Longstreet project will require a maximum of approximately 73 personnel to carry out the proposed mine plan including mine contractor personnel. At its peak, Mine Operations will require 21 people per shift. Similarly, Mine Maintenance will require a maximum of 8 people per shift. Mine management, operations supervision and technical services will require 7 people.

13.10 Mining above Underground Workings

Historic underground mining has occurred in the mineralized areas of the Pit. This mining was minimal and was generally performed manually by excavating drifts (tunnels) underneath the ore zones and selectively extracting the mineralized rock from underneath – creating an open manmade cave (stope). Sometimes, mine timbers and rock bolts were used to brace the sides of the drifts and stopes, but after several decades the support is no longer effective. The unsupported openings often have no surface expression and may cave in if mining equipment gets too close.

Experience at numerous open pit mines in Nevada has shown that mining over historic underground mines can be performed safely without significantly disrupting the mining schedule; however, the presence of underground workings requires additional safety precautions to avoid ground collapse under men or equipment. Typically, a blast hole drill is used to advance probe holes to a depth of 40 feet below a mining bench to determine the presence of a mining cavity. When a cavity is located, additional probe holes are drilled to determine the extent of the cavity. Then a blasting plan is developed to fill the void with blasted rock prior to mining over the area. If additional voids are exposed during mining, additional probing, drilling and blasting will be performed until the area is safe and normal mining activities can be resumed.

14.0 Processing and Recovery Methods

Test work has shown that the Longstreet Project mineral resource is amenable to conventional heap leaching with carbon adsorption, desorption, and recovery.

Ore will be crushed at an average rate of 5,500 tpd to 80% passing size of 3/4" using a portable three-stage crushing plant. The crushed product will be conveyor stacked on the leach pad in 32.8 ft lifts. Lime will be added to the material for pH control before being stacked and leached with a dilute cyanide solution. Pregnant solution will flow by gravity to a pregnant solution pond before being pumped to carbon adsorption columns for metal recovery.

Six process alternatives were evaluated for recovery of gold and silver from loaded carbon. The alternatives considered differences in carbon handling strategy (on-site stripping versus toll stripping), the potential incorporation of HPGR crushing to improve heap leach recoveries, contract crushing, diesel generators and portable crushers. All options were evaluated using the same mine plan, production schedule, commodity prices, and financial assumptions to provide a consistent economic comparison. Recent data from similar projects in Nevada were used for this evaluation.

The six alternatives are listed below:

Option 1 – On site Stripping

Option 1A – Gold only

Option 1B – Gold & Silver

Option 2 – Toll Stripping

Option 2A – Gold only

Option 2B – Gold & Silver

Option 3 – Toll Stripping + HPGR (1/4" crush size)

Option 4 – Toll Stripping + Contract Crushing

Option 5 – Diesel Generators + Contract Crushing

Option 6 – Diesel Generators + Portable Crushers

Option 1 and 2 considered two different scenarios each. Gold-only recovery focuses on recovering gold while accounting for silver that loads with it. Gold-only was included as an option because recovering all dissolved silver requires substantially more activated carbon to load the silver. The increased carbon requirement results in higher capital costs for the adsorption and stripping circuits, while operating costs increase due to additional carbon handling, carbon replacement, and refining of the larger quantity of recovered precious metals.

Options 1 to 3 considered permanent crushers in place. Options 1 to 5 considered high voltage power lines from Tonopah to site, approximately 50 miles.

The economic comparison indicates that all conventional processing alternatives produce positive pre-tax NPVs at discount rates of 5% and 10%. Option 6 (Diesel Generators + Portable Crushers) generated the highest economic return, with a pre-tax NPV of approximately \$87 million at an 5% discount rate and \$70 million at a 10% discount rate, together with the highest pre-tax IRR of 48%.

Therefore, Option 6 was selected for the preparation of this report. Further discussion on the economics of the other alternatives is presented in Section 18.0, Capital and Operating Costs.

14.1 Process Summary

Run-of-Mine ore (ROM) will be delivered to the crushing plant feed stockpile. A front-end loader will reclaim the ROM ore and feed it to the dump hopper of the Primary Crusher. The ore will be crushed at an average rate of 5,500 tons per day to a final product size of 80% passing 3/4" using a three-stage portable crushing plant. The crushing plant will operate seven days/week, 24 hours/day with an overall estimated availability of 60%.

The crushed product will be stockpiled using a stacking conveyor and reclaimed by vibrating pan feeders. Pebble lime will be added to the reclaim material for a pH control. Test work has shown that agglomeration with cement is not required.

Ore will be stacked on the leach pad by retreat stacking uphill from the toe of the heap. Stacked ore will be leached using a drip irrigation system for solution application. After percolating through the ore, gold and silver bearing pregnant leach solution drains by gravity to a pregnant solution pond where it will be collected and pumped to a set of carbon-in-columns (CICs) where gold and silver will be removed by activated carbon.

Barren leach solution leaving the CICs will flow to a barren solution tank and then pumped to the heap leach pad for further leaching. Cyanide solution will be injected into the barren solution to maintain the desired cyanide concentration. Single-stage leaching is assumed with a 140-day leach cycle.

The adsorption circuit will consist of one train of five CICs. Each column will contain 3.3 tons of carbon. Pregnant solution will flow up through the first column and exit from the top of the open tank into the next column. Once the carbon in the first column of a train reaches a loading of 67 ounces of gold per ton (opt) and approximately 58 opt Ag, will be transferred to supersacks.

Loaded carbon from the CIC will be periodically shipped to an off-site refinery for either stripping or ashing treatment to get a final doré product. Based on other operations in Nevada, stripped carbon will not be returned to the CIC circuit at Longstreet due to the impact in carbon adsorption efficiency. Replacing all carbon is more cost effective than re use it.

All of the selected processes and equipment are established and widely used in gold processing plants. Key process design criteria are presented in Table 14-1.

Table 14-1
Longstreet Project Summary of Design Criteria

Description	Qty	Units
Gold Price	3,300	USD/oz
Silver Price	40	USD/oz
Mine		
Ore	6,552,050	st
Waste	5,305,059	st
Total Mined	11,857,109	st
Strip Ratio	0.81	
Crushed Ore Production Rate, Annual	2,000,000	st
Product Size p80	0.75	in
Grades		
Au	0.0160	opt
Ag	0.416	opt
Contained Metal		
Au	105,461	oz
Ag	2,728,471	oz
Heap Operation		
Material Feed Rate	5,500	tpd
Leaching Cycle	140	days
Leaching Schedule	365	d/a
Solution Application Rate	0.004	gpm/ft2
Recoveries		
Au	84%	
Ag	13%	
Metals in solution		
Au	79	oz/day
Ag	273	oz/day
Carbon Adsorption Columns		
Type	Open Top	
Quantity of Trains	1	
Quantity of Columns per Trains	5	

Nominal Flow Rate per Train	2,279	gpm
Design Flow Rate per Train	2,724	gpm
Column Diameter	11.8	ft
Column Height	12.7	ft
Loaded Carbon		
Au	67	opt
Ag	58	opt
Reagents		
Lime	3.58	lb/t
Cyanide	1.31	lb/t

Figure 14-1 shows the overall process flowsheet and Figure 14-2 shows the general arrangement of the mine site



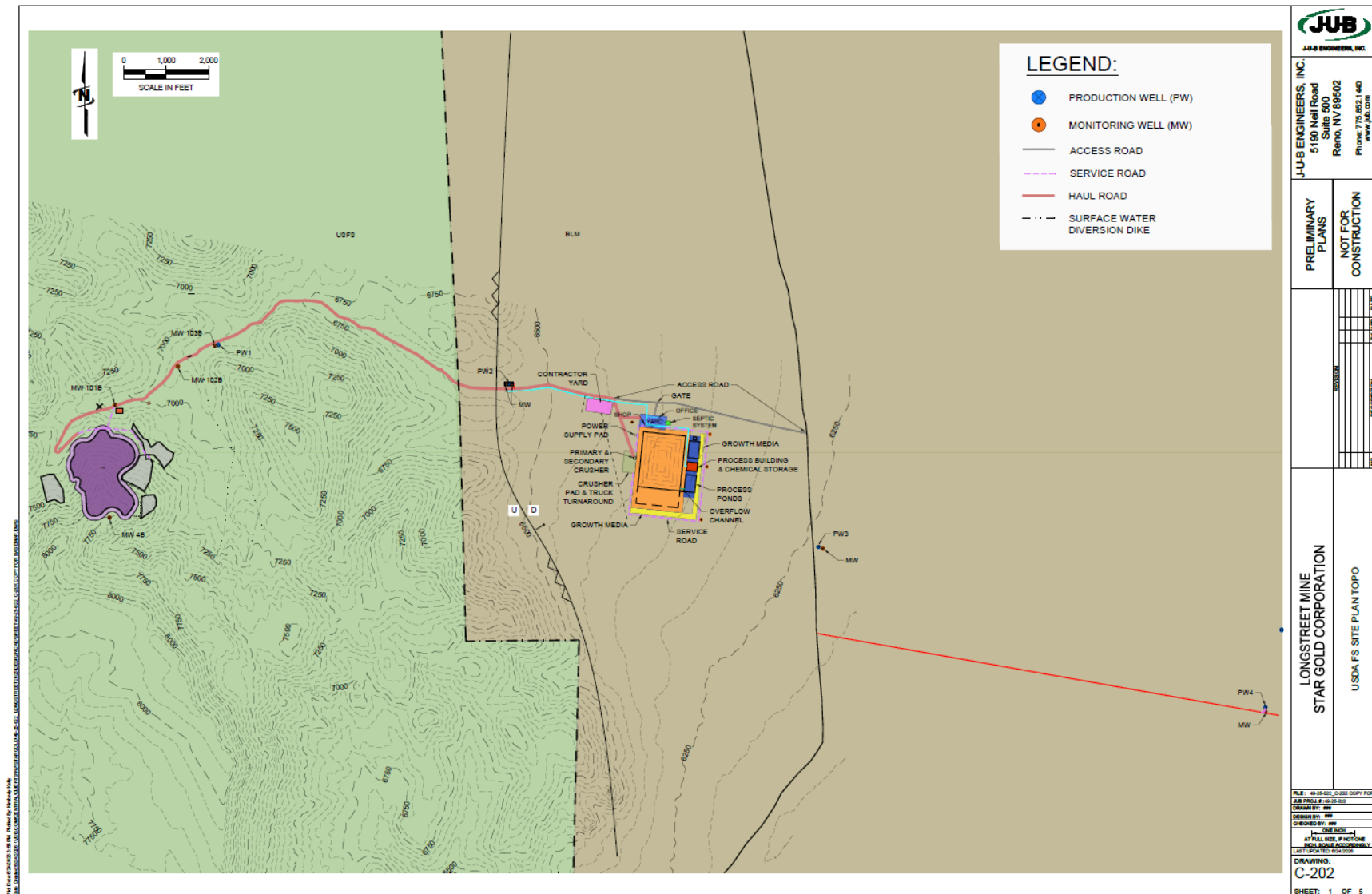


Figure 14-2
Longstreet Project General Arrangement

14.2 Crushing and Screening

The following major components are included in the crushing facility:

- Retaining wall
- Dump hopper with stationary grizzly
- Open circuit Portable Primary Crusher complete with vibrating grizzly feeder, 150 HP jaw crusher and discharge conveyor;
- Closed circuit Portable Secondary Cone Crusher complete with two deck horizontal screens, 300 HP cone crusher, oil lubrication tank, conveyors system, metal detector, spray bars and ACE6 Advance Cone Controls;
- Closed circuit Portable Tertiary Cone Crusher complete with two deck horizontal screens, 300 HP cone crusher, oil lubrication tank, conveyors system, metal detector, spray bars and ACE6 Advance Cone Controls;
- Crushed product stacker:
- Crushed product stockpile.

The crushing plant will process an average of 5,479 dry tons per day. ROM ore will be delivered to the crushing plant by 100-ton haul trucks and stockpiled adjacent to the primary crusher. Oversized rocks will be sorted and broken by excavator mounted rock breaker. A front-end loader will reclaim ROM ore from the stockpile and feed it onto a stationary grizzly located above a dump hopper. Oversized material retained on the grizzly will be broken with the mobile rock breaker.

A vibrating grizzly feeder, with parallel bars spaced 3.5 in apart, will draw ore from the dump hopper. Grizzly oversize is fed to the 150-hp primary jaw crusher. Grizzly undersize will be recombined with the jaw crusher product on the primary crushing discharge conveyor and transferred to the portable secondary 300-hp cone crusher. The primary jaw crusher will operate with a 4.25" closed side setting.

The portable secondary crusher close circuit includes a system of conveyors, double-deck vibrating screen with 2" and 1" top and bottom openings, respectively. Ore is fed onto the screen by the secondary screen feed conveyor. Oversized material from the top fractions (+1") will be conveyed to the secondary 300-hp cone crusher. Undersize (-1") from the secondary screen will be transferred to the crushed product conveyor.

The portable tertiary crusher circuit is identical to the secondary crusher with different screen openings, 1.25" and 1" top and bottom respectively. Ore is fed onto the screen by the tertiary screen feed conveyor. Oversized material from the top fractions (+1") will be conveyed to the tertiary 300-hp cone crusher. Undersize (-1") from the tertiary screen will be transferred to the crushed product conveyor.

Tertiary crushed product will discharge onto the crushed product conveyor. The crushed product conveyor will discharge material to a radial stacker at the crushed product stockpile.

Water sprays will be located at all material transfer points to reduce dust generation by the crushing circuit.

14.3 Reclamation and Conveyor Stacking

The following major components are included in the reclamation and conveyor stacking system:

- Three electromechanical reclaim feeders;
- One 30-inch x 150 ft long reclaim tunnel conveyor
- 2,800 ft³ lime silo with associated dust control and feeding equipment;
- Four 24-inch x 100 ft long ramp conveyors;
- Sixteen 24-inch x 100 ft long grasshopper conveyors;
- One 24-inch x 100 ft index feed conveyor;
- One 24-inch x 100 ft horizontal index conveyor;
- One 24-inch-wide x 150 ft long TeleStacker® Conveyor (or equivalent).

The crushed product stockpile is sized to accommodate a total capacity of approximately 5,500 tons. Crushed ore will be reclaimed from the stockpile by three electromechanical feeders to a Reclaim Conveyor in a tunnel below the stockpile. Pebble lime will be added at a rate of 3.58 lb/ton for pH control to the Reclaim Tunnel Conveyor. The reclaim conveyor discharges to the heap stacking equipment.

The heap stacking equipment includes ramp conveyors to transport crushed ore up ramps cut into the side of the heap, grasshopper conveyors that transport crushed ore across approximately horizontal areas to the specialized stacking equipment. The specialized stacking equipment includes an index feed conveyor (24-inch x 100 ft), a horizontal index conveyor (24-inch x 100 ft), and a radial stacker. The radial stacker can rotate to stack crushed ore in rows about 280 feet wide. The stacker/horizontal index conveyor combination retreat away from the face of the crushed ore while continuing to stack.

The heap will be constructed in 32.8 ft high lifts, in ore "prisms" approximately 250 ft wide. The first lift will be stacked so that the toe of the heap will be inside of the perimeter berm at closure. The effective overall slope of the heap will be approximately 2.5H:1V.

Once a lift of ore has finished leaching and is sufficiently drained, a new lift can be stacked over the top of the old lift. The old lift will be ripped prior to stacking new material on top of any old heap area or access road/ramp to break up any compacted or cemented sections.

Stacked lifts will progress in a stair-step manner. The maximum planned heap height is five lifts over the composite leach pad liner system.

14.4 Leach Pad Design

Dyer Engineering Consultants designed the leach pad (HLP) in 2018. The average elevation in the area proposed is 7,050 ft. The HLP is designed to store 6.5 million short tons of ore.

The leach pad will be a single-use, multi-lift type leach pad and has been designed with a lining system approved by the state of Nevada. The leach pad area will be constructed by clearing the pad area and stripping vegetation and growth medium. The area will need to be graded for drainage and heap stability.

The leach pad liner will be composed of the following components from top to bottom:

- Overliner consisting of two feet of crushed and screened material over a network of solution collection piping;
- 60 mil double sided, textured Linear Low-Density Polyethylene (LLDPE) geomembrane;
- 1-foot Low Hydraulic Conductivity Soil Layer consisting of screened, native soil blended with clay with a minimum permeability of 1×10^{-6} cm/sec;
- Leak detection system under the primary solution collection pipes which route solution to a monitoring sump tank;
- Prepared subgrade.

A clay source, has not yet been identified. The heap leach pad includes 2.38 million ft² of lined area and will be sized to contain the ultimate cumulative ore capacity.

Gravity solution collection pipes will be installed on top of the geomembrane liner and covered with overliner material. The pipes are sized to operate at 50% full to contain the design production flows from the upgradient tributary area, allowing additional capacity to accommodate excess solution from storm events and reduced flow capacity from pipe squeezing during loading. The gravity solution collection pipes will consist perforated corrugated polyethylene (PCPE) pipes. The pipes are typically arranged in a branching network where smaller pipes feed larger pipes.

The flow from the individual cells drain to flumes for flow measurement of the solution and sampling to determine solution concentrations. Solid HDPE pipes will carry the solution from the flumes to the pregnant pond. Should solution flows exceed the capacity of the heap outlet pipes, solution will flow over the outlet pipe berms into the solution conveyance channel and to the Event Pond.

The overliner material will act as a protective layer that resides above the LLDPE geomembrane. The main purpose of this material is to protect the composite liner system and solution collection piping from damage during stacking.

14.5 Solution Application & Storage

The Longstreet Project will use a Pregnant Solution Pond, an Event Pond, and a Barren Solution Tank for solution management.

Cyanide solution will be added to the barren tank and the solution will be utilized to irrigate the leach pad for the 140 day leach cycle. The resulting pregnant is directed by gravity to the Pregnant Pond.

Barren leach solution will be applied to the surface of the heap by a system of drip emitters at a nominal rate of 0.004 gpm/ft². The combined nominal flow to the heap will be approximately 2,279 gpm. A total leach cycle of 140 days has been selected based on metallurgical test results. PCPE pipes placed on the pad liner will facilitate solution flow to the process pond.

The barren solution tank is sized to accommodate 45,000 gal of solution, or approximately 20 minutes of residence time in the CIC. Two horizontal centrifugal pumps (one operating, one standby) at the barren tank will be used to pump barren solution to the irrigation system. Concentrated cyanide solution and antiscalant polymer will be metered into the suction side of the barren solution pump as needed.

Excess solution from the pregnant solution pond will flow by spillway to the event pond. The process pond has a design capacity of 1,279,456 ft³ (~9.58Mgal) and has been sized with the following criteria:

- Working volume equivalent for 24 hours at 2,279 gpm of solution
- Accumulation of solution resulting from a 24-hour precipitation event of 0.8 inches (30% diversion) over the entire lined area.
- Dead storage volume assuming 3 ft of slimes at the bottom of the pond
- Three feet of freeboard below the top of the containment berm

The pregnant solution pond will be constructed using the following composite liner system from top to bottom:

- 80-mil HDPE geomembrane primary liner
- 200-mil geonet leak detection layer
- 60-mil HDPE secondary liner

- 6 in low-permeability liner bedding

The event pond will collect seasonal accumulation of leach solution and/or operational upsets that can't be managed during normal operations. The event pond has a designed capacity of 925,663 ft³ (~6.9Mgal) and has been sized to contain the maximum accumulation during an average year, assumed to be:

- Water accumulated from a 100-year, 24-hour precipitation event (2.6-inch rain storm, ~2.9Mgal)
- 24 hours of pad drain-down (~3.2Mgal, assumed to be the same as total barren plus intermediate leach flow)

The event pond will be constructed using the following composite liner system from top to bottom:

- 80-mil HDPE geomembrane primary liner
- 200-mil geonet leak detection layer
- 60-mil HDPE secondary liner
- 6 in low-permeability liner bedding

Solution collected by the event pond will be returned to the barren solution tank as make-up solution

14.6 Reagents

Heap leach and process plant reagents and consumables will include sodium cyanide, pebble lime, hydrochloric acid, antiscalant and activated carbon. Propane will serve as fuel to thermal equipment in the plant.

14.6.1 Sodium Cyanide

Cyanide will be delivered by tanker truck as a 30% solution and stored in a 45,000 gallon capacity tank at the cyanide handling area adjacent to the Adsorption facilities. The tank will provide storage for approximately 6 days of solution. Concentrated cyanide solution will be metered into the barren tank.

14.6.2 Lime (CaO)

Pebble lime will be used for pH control in the heap leach process. Lime will be delivered in bulk and stored in the 105-ton silo. Pebble lime will be screw fed to the material on the crushed product reclaim conveyor at a rate of 3.58 lb per dry ton of ore. The silo will store approximately 6 days of pebble lime inventory.

14.6.3 Antiscalant

An antiscalant agent will be used in the heap leach to prevent the buildup of scale within the system. Antiscalant will be delivered in solution form in 240 gal totes. Metering pumps will dose antiscalant to the barren tank. Antiscalant will be consumed at a rate of approximately 5 gallons per day.

14.6.4 Activated Carbon

Activated carbon will be used for gold adsorption in the CIC circuit. Carbon will be delivered as granules in half-ton bulk bags. Fresh carbon being introduced to the system will be loaded into the carbon attritioning tank where it will be slurried and agitated to remove loose edges from the granules. Attritioned carbon will be pumped to the carbon dewatering screen for removal of fine carbon particles from the system. Coarse, attritioned carbon will be transferred to the barren carbon storage tank before being used in the CIC circuit for gold loading.

Fine carbon will be stored in permeable bags where water will be removed before being removed from the system. Loaded carbon will be stripped off-site and will be replaced with fresh carbon every time.

14.6.5 Propane

Propane will serve as fuel to the heating system of process building and lab facilities. Propane will be delivered as a liquid and stored in an 500-gal tank with evaporator. The process will consume an estimated 300 lbs./week.

14.6.6 Air

High-pressure plant and instrument air required by the ADR facilities will be supplied by an air compressor. Instrument air will be dried and filtered before delivery to the various equipment in the plant.

14.6.7 Water

Wells will supply fresh groundwater for make-up in the heap leach process, road dust suppression at the mine, emergency fire water, and potable water uses in the plant. Well water will be stored in a 15,000-gal tank adjacent to the well. Horizontal centrifugal pumps will distribute raw water to the potable water tank and raw/fire water tank. Hypochlorite will be metered into the potable water tank.

14.6.8 Laboratory Facilities

Analytical support, including fire assays and metallurgical testing required to support production operations, will be conducted on-site at dedicated laboratory facilities. Lab facilities will be equipped to fire assay samples coming from the mine, as well as, a small number of fire, solution, and carbon assays required for metallurgical control.

14.7 Process Water Balance

Environmental data from the National Weather Service website was used to estimate the site wide water balance.

The year 1951 was the closest to the average annual precipitation, 2020 was the driest and 1978 was the wettest on record. Evaporation data was limited so an estimation was calculated based on historical data from the Western Regional Climate Center (monthly evaporation distribution).

This is presented in Table 14-2

Table 14-2
Precipitation and Evaporation Data

Month	Rainfall (1951) inches	Pan Evaporation, (estimate) inches
January	0.6	0.0
February	0.3	3.8
March	0.3	7.3
April	0.8	10.1
May	0.0	13.6
June	0.3	16.3
July	1.0	18.0
August	0.1	15.9
September	0.0	11.3
October	0.6	6.9

November	0.4	2.9
December	0.6	0.0
Total	5.0	106.2

Based on the preceding rainfall and pan evaporation data water balances were calculated based on the tonnage of 5,500 tpd.

The water balance models for an average year, max wet season, and max dry season are presented in Table 14-3, Table 14-4 and Table 14-5 respectively.

The diagram for an average year, max wet season, and max dry season are presented in Figure 14-3, Figure 14-4 and Figure 14-5 respectively.

Assumptions for the water balance were as follows:

- Pond evaporation equals 60% of pan evaporation over 50% pond area;
- Idle heap evapotranspiration equals 75% of pan evaporation;
- Maximum evapotranspiration equals rainfall over idle area

For an average year, it was determined that the Longstreet Project will be in a water deficit and makeup water will be required. The average make-up water requirement is 126 gpm

Table 14-3
Average Year Water Balance

	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Annual
Days in Month	31	31	28	31	30	31	30	31	31	30	31	30	365
Precipitation (in)	0.6	0.6	0.3	0.3	0.8	0.0	0.3	1.0	0.1	0.0	0.6	0.4	5.0
Pan Evaporation (in)	0.0	0.0	3.8	7.3	10.1	13.6	16.3	18.0	15.9	11.3	6.9	2.9	106.2
Emitter Evap. (%)	0.0	0.0	0.9	1.6	2.3	3.1	3.7	4.1	3.6	2.6	1.6	0.7	2.0
Sprinkler Evap. (%)													
Idle Heap Evapotrans. Area (sq. ft.)	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508
Idle Heap Evapotrans. (ft)	0.0	0.0	2.9	5.4	7.6	10.2	12.2	13.5	11.9	8.5	5.2	2.2	79.6
Ore Placed on Pad (tons)	169,863	169,863	153,425	169,863	164,384	169,863	164,384	169,863	169,863	164,384	169,863	164,384	2,000,000
Precip. Collected (cu.ft.)	137,218	137,218	55,745	70,753	162,946	0	64,321	214,403	15,008	0	122,210	90,049	1,069,872
Ore Absorption (cu. ft.)	453,680	453,680	409,776	453,680	439,045	453,680	439,045	453,680	453,680	439,045	453,680	439,045	5,341,718
Emitter Evap. (cu. ft.)	0	0	136,552	285,829	385,957	535,437	621,417	707,880	626,777	431,296	270,868	112,015	4,114,027
Sprinkler Evap. (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evapotrans. (cu. ft.)	0	0	36,086	45,801	105,482	0	41,638	138,792	9,715	0	79,112	58,293	514,920
Pond Evaporation (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evaporation System (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Precip. Gain(+)/Loss(-)	(303,405)	(303,405)	(526,669)	(714,558)	(767,538)	(989,117)	(1,037,779)	(1,085,949)	(1,075,164)	(870,342)	(681,450)	(519,304)	(8,874,679)
Excess Solution Pond (gal)													
Allowable Accum. in Excess	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	
Accum. into Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Recycled from Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Quantity in Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Makeup Solution Required (gal)	2,269,631	2,269,631	3,939,754	5,345,262	5,741,581	7,399,112	7,763,127	8,123,461	8,042,785	6,510,607	5,097,601	3,884,662	66,387,212
Solution to Treat/Discharge	0	0	0	0	0	0	0	0	0	0	0	0	0

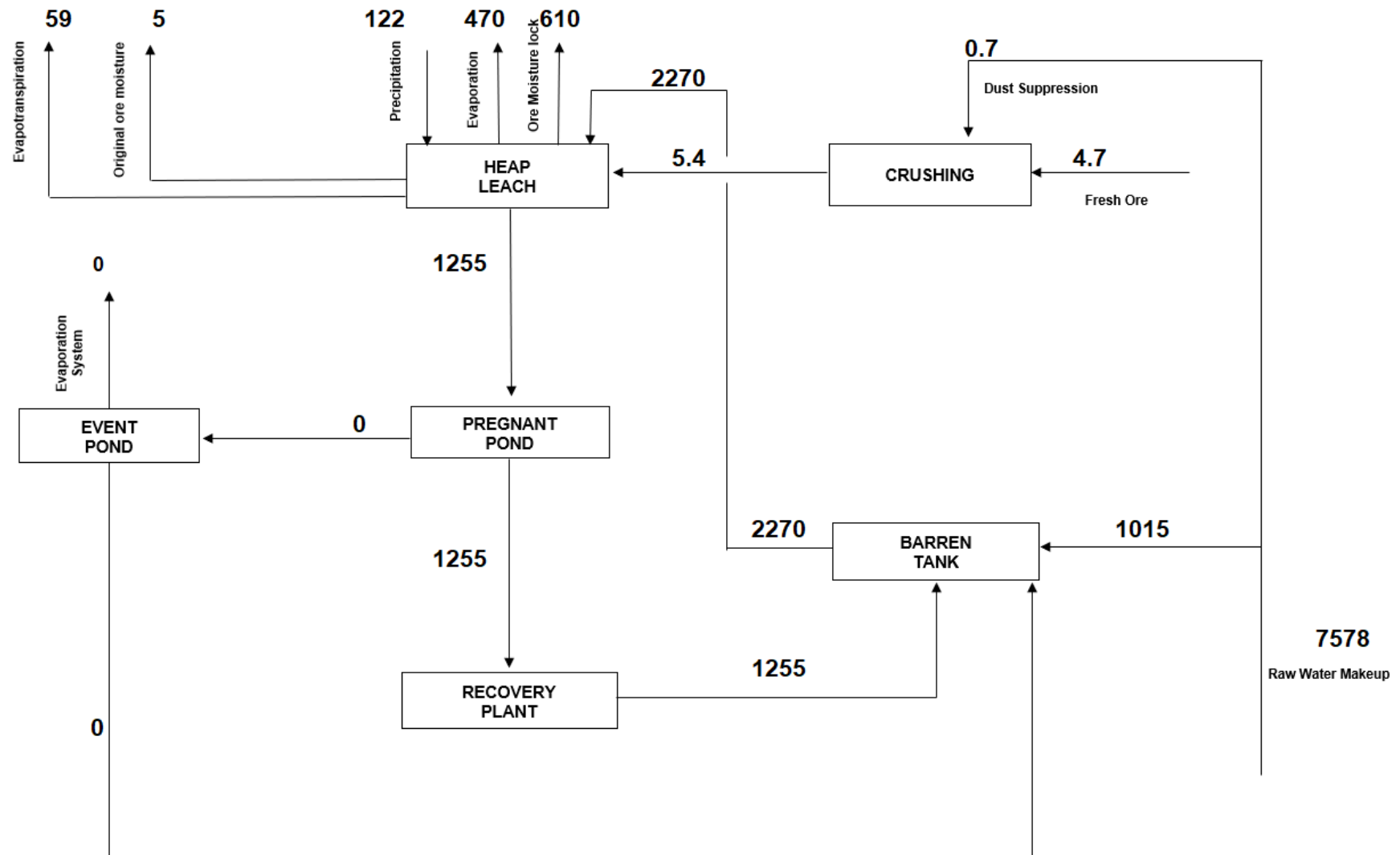


Figure 14-3 Average Year Water Balance Diagram in Gallons per Minute

Table 14-4
Wet Year Water Balance

	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Annual
Days in Month	31	31	28	31	30	31	30	31	31	30	31	30	365
Precipitation (in)	0.5	0.7	1.6	2.4	1.7	0.0	0.0	0.2	0.3	2.1	0.8	0.5	10.6
Pan Evaporation (in)	0.0	0.0	3.8	7.3	10.1	13.6	16.3	18.0	15.9	11.3	6.9	2.9	106.2
Emitter Evap. (%)	0.0	0.0	0.9	1.6	2.3	3.1	3.7	4.1	3.6	2.6	1.6	0.7	2.0
Sprinkler Evap. (%)													
Idle Heap Evapotrans. Area (sq. ft.)	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508
Idle Heap Evapotrans. (ft)	0.0	0.0	2.9	5.4	7.6	10.2	12.2	13.5	11.9	8.5	5.2	2.2	79.6
Ore Placed on Pad (tons)	169,863	169,863	153,425	169,863	164,384	169,863	164,384	169,863	169,863	164,384	169,863	164,384	2,000,000
Precip. Collected (cu.ft.)	98,626	152,226	332,325	510,280	373,062	0	0	32,160	66,465	439,527	162,946	113,634	2,281,251
Ore Absorption (cu. ft.)	453,680	453,680	409,776	453,680	439,045	453,680	439,045	453,680	453,680	439,045	453,680	439,045	5,341,718
Emitter Evap. (cu. ft.)	0	0	136,552	285,829	385,957	535,437	621,417	707,880	626,777	431,296	270,868	112,015	4,114,027
Sprinkler Evap. (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evapotrans. (cu. ft.)	0	0	215,128	330,326	241,499	0	0	20,819	43,026	284,524	105,482	73,560	1,314,364
Pond Evaporation (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evaporation System (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Precip. Gain(+)/Loss(-)	(345,670)	(286,969)	(429,130)	(559,555)	(693,439)	(989,117)	(1,060,462)	(1,150,218)	(1,057,017)	(715,339)	(667,084)	(510,987)	(8,464,989)
Excess Solution Pond (gal)													
Allowable Accum. in Excess	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	0
Accum. into Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Recycled from Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Quantity in Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Makeup Solution Required (gal)	2,585,793	2,146,679	3,210,118	4,185,763	5,187,284	7,399,112	7,932,809	8,604,228	7,907,038	5,351,108	4,990,136	3,822,445	63,322,514
Solution to Treat/Discharge	0	0	0	0	0	0	0	0	0	0	0	0	0

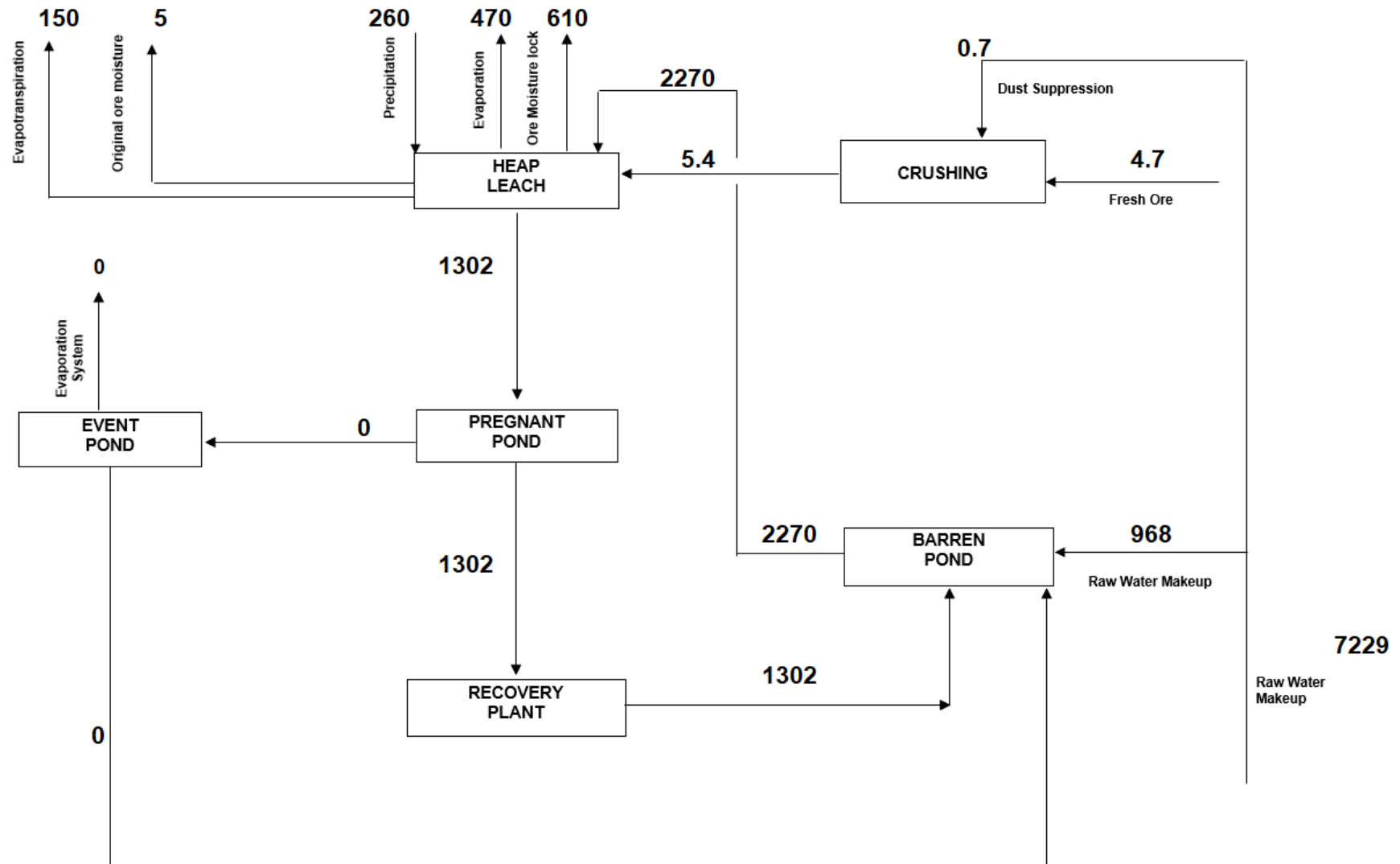


Figure 14-4 Wet Year Water Balance Diagram in Gallons per Minute

Table 14-5
Dry Year Water Balance

	Dec	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Annual
Days in Month	31	31	28	31	30	31	30	31	31	30	31	30	365
Precipitation (in)	0.2	0.0	0.3	0.2	0.6	0.3	0.2	0.1	0.0	0.0	0.0	0.1	2.0
Pan Evaporation (in)	0.0	0.0	3.8	7.3	10.1	13.6	16.3	18.0	15.9	11.3	6.9	2.9	106.2
Emitter Evap. (%)	0.0	0.0	0.9	1.6	2.3	3.1	3.7	4.1	3.6	2.6	1.6	0.7	2.0
Sprinkler Evap. (%)													
Idle Heap Evapotrans. Area (sq. ft.)	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508	1,665,508
Idle Heap Evapotrans. (ft)	0.0	0.0	2.9	5.4	7.6	10.2	12.2	13.5	11.9	8.5	5.2	2.2	79.6
Ore Placed on Pad (tons)	169,863	169,863	153,425	169,863	164,384	169,863	164,384	169,863	169,863	164,384	169,863	164,384	2,000,000
Precip. Collected (cu.ft.)	34,305	0	57,889	47,169	120,066	72,897	47,169	10,720	0	0	0	27,872	418,086
Ore Absorption (cu. ft.)	453,680	453,680	409,776	453,680	439,045	453,680	439,045	453,680	453,680	439,045	453,680	439,045	5,341,718
Emitter Evap. (cu. ft.)	0	0	136,552	285,829	385,957	535,437	621,417	707,880	626,777	431,296	270,868	112,015	4,114,027
Sprinkler Evap. (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evapotrans. (cu. ft.)	0	0	37,474	30,534	77,724	47,189	30,534	6,940	0	0	0	18,043	248,438
Pond Evaporation (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Evaporation System (cu. ft.)	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Precip. Gain(+)/Loss(-)	(416,111)	(453,680)	(525,912)	(722,875)	(782,660)	(963,410)	(1,043,828)	(1,157,779)	(1,080,457)	(870,342)	(724,548)	(541,231)	(9,282,833)
Excess Solution Pond (gal)													
Allowable Accum. in Excess	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	79,251,616	0
Accum. into Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Recycled from Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Quantity in Excess	0	0	0	0	0	0	0	0	0	0	0	0	0
Makeup Solution Required (gal)	3,112,730	3,393,763	3,934,098	5,407,479	5,854,703	7,206,805	7,808,375	8,660,789	8,082,377	6,510,607	5,419,998	4,048,689	69,440,414
Solution to Treat/Discharge	0	0	0	0	0	0	0	0	0	0	0	0	0

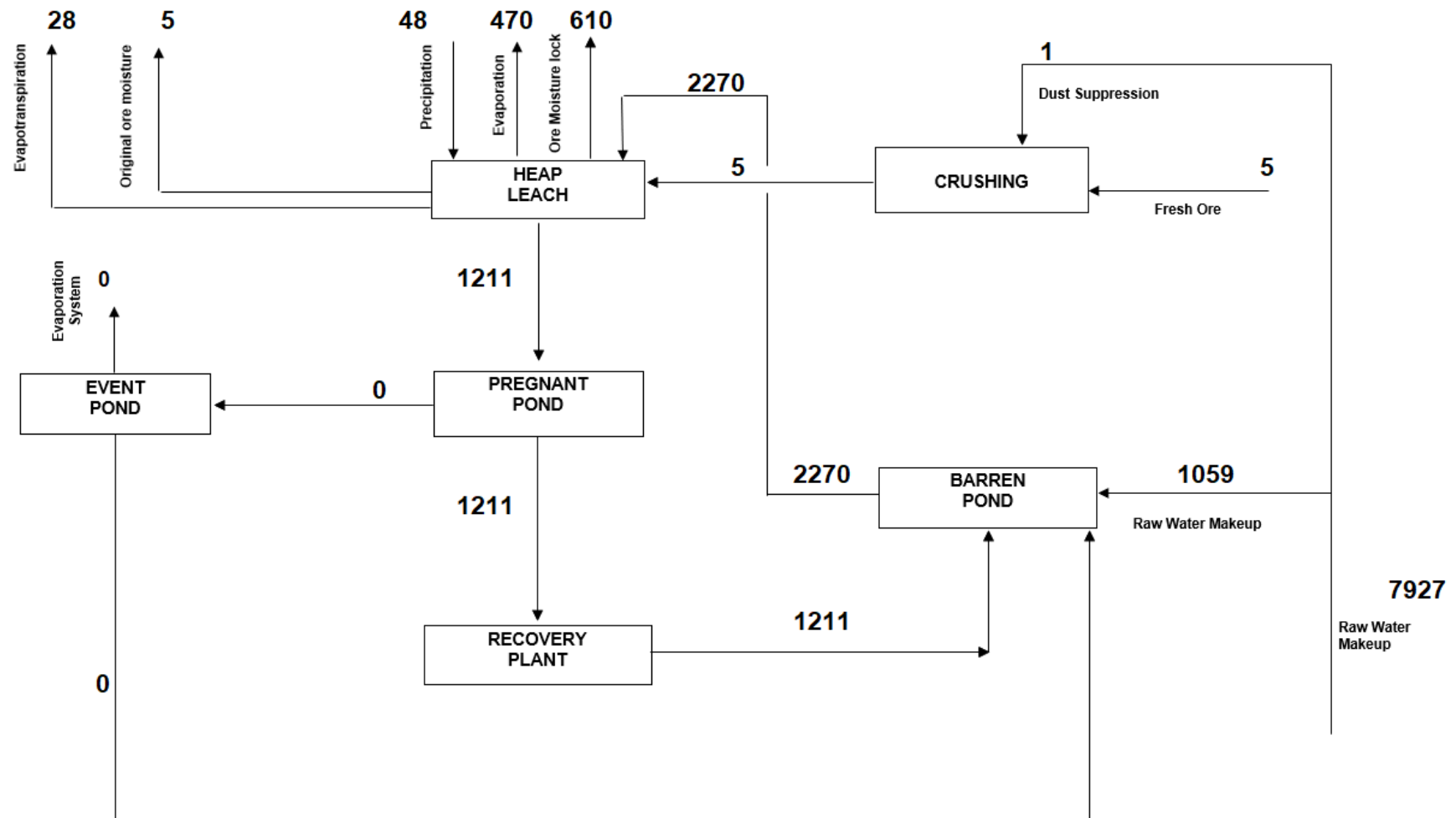


Figure 14-5
Dry Year Water Balance Diagram in Gallons per Minute

15.0 Project Infrastructure

15.1 Site Layout

Figure 15-1 Shows the proposed site layout

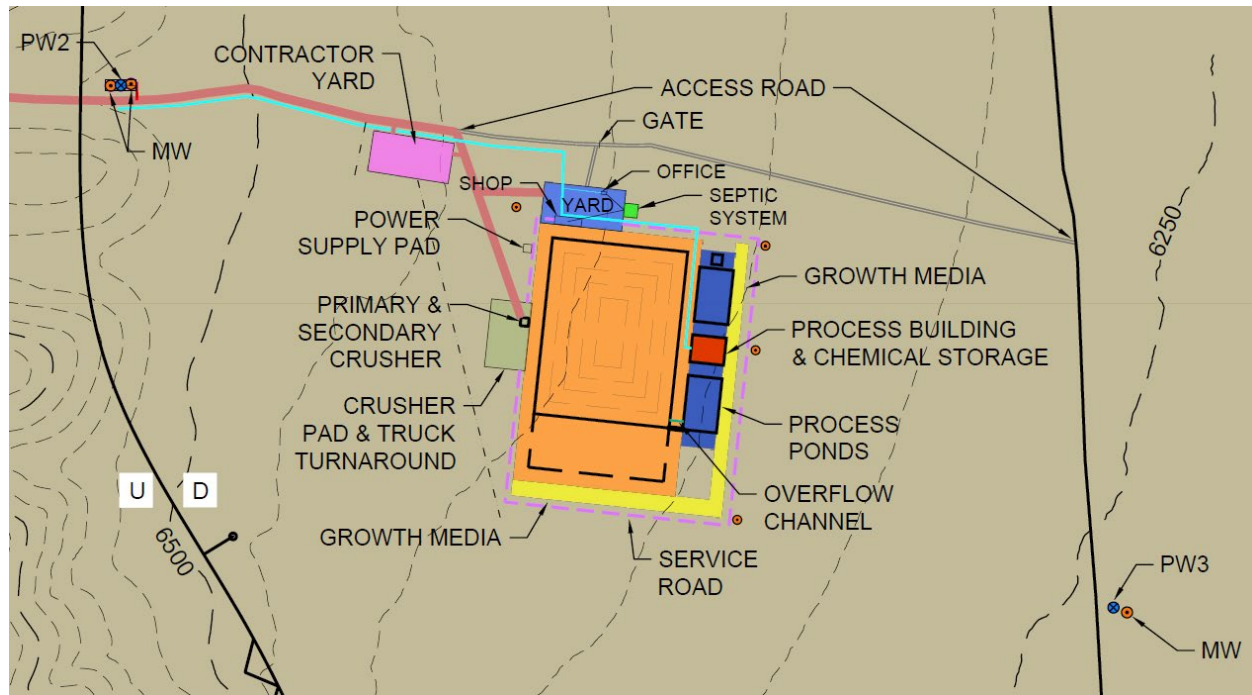


Figure 15-1
Site Layout

15.2 Site Roads

The Project is located approximately 170 miles northwest of Las Vegas, Nevada and approximately 50 miles northeast of Tonopah. A paved county road runs east-west, approximately 27 miles south of the project, connecting the site to Tonopah.

Internal roads will provide access between the crushing plant, heap leach pads, ADR process plant, bulk fuel storage and mine facilities. These roads will generally be 20 ft wide and will be constructed flush with bulk earthworks pads to ensure that water flow is achieved across the site whereby avoiding the need for deep surface drains and culvert crossings.

15.3 Mine Haulage Road

A network of mine haul roads will be constructed and maintained by the mining department and used to access the open pit, waste rock dumps and for the transport of ore to the ROM stockpile adjacent to the crushing plant.

15.4 Project Buildings

Site buildings will be 'fit-for-purpose' industrial modular-type structures

15.5 Administrative Offices

The administrative offices will be housed in a 2,160 ft² modular building located north of the heap-leach pad. The mining contractor will be responsible for providing the mine offices. Two 320 ft² modular buildings will provide office space for the crusher circuit and site security.

15.6 Mine Camp Facilities

No on-site accommodations are planned. The workforce is expected to be sourced from Nye county and the Las Vegas area and will commute daily to the Longstreet site daily.

15.7 Carbon Adsorption Facility

The Longstreet Project will utilize a carbon adsorption process for the recovery of gold from pregnant heap leach solutions. The carbon in column (CIC) circuit will consist of five columns arranged in cascade. Loaded carbon from the CIC circuit will be stored and shipped to an off-site processor for elution and refining of doré bullion.

15.8 Laboratory

A laboratory facility will be constructed near the carbon columns for analysis of samples from the mine and process. Solution and fire assays will be carried out by the owner. The module provided for the laboratory will be 53-foot, high cube shipping container. The interior of the container will be framed for walls with a drop ceiling. The interior wall finish will be a textured/FRP composite sheet with trim that is chemical and moisture resistant. The flooring will be finished with a poured in place epoxy resin; sand will be added to the epoxy resin for the main walkways

The laboratory will include all necessary eyewash/safety shower and firewater distribution.

15.9 Process Maintenance Building

The maintenance building will be 2,280 ft² pre-engineered steel structure.

15.10 Reagent Storage

Cyanide will be stored in dedicated areas of the process facilities. There is no specific area for storing virgin carbon, which can be stored on the ground.

15.11 Mine Truck Shop and Warehouse

Maintenance facilities for mining equipment will be provided by the mining contractor.

15.12 Fuel Supply

An on-site bulk diesel fuel storage tank will be supplied by the mining contractor to fuel the onsite mobile equipment. Diesel fuel will be sourced locally. A concrete pad will be constructed for the diesel tank and refueling area. There will be no gasoline storage or dispensing facilities.

15.13 Fenced Laydown Yard

15.14 Magazine Site

This area is to facilitate the ANFO storage bins, explosive and detonator magazines, and the movement of delivery and site vehicles.

15.15 Site Security

Employees and visitors will sign in and out of the site, and an active log of all on-site individuals will be maintained.

15.16 Power Supply, Communication Systems & IT

15.16.1 Power Supply

Power will be supplied by two diesel generators. One diesel generator will be dedicated to the crushing circuit and the second to the rest of the project: stacking, carbon columns, laboratory, offices and other facilities.

Diesel generators were selected as the best option versus high voltage lines from Tonopah. Quotes of similar size generators from other KCA projects in the area were used. Diesel generator for crushing and stacking circuit was sized at 2MW and diesel generator for the rest of infrastructure is 1MW.

Electrical enclosures and modular buildings will house the 480V motor control centers (“MCCs”), variable frequency drives (“VFDs”), process plant control system cabinets, plant lighting transformers and other electrical gear.

The average power demand is summarized in Table 15-1. The total peak demand load is estimated at 1,696 kW while the average power draw is anticipated to be 1,286 kW. The total installed load is forecast at 2,934 kW.

Table 15-1
Power Summary

Area	Attached Power (kW)	Demand (kW)	Peak Demand (kW)
Area 113 - Crushing	835	377	609
Area 115 - Crushed Ore Stockpile, Reclaim & Stacking	1,018	337	456
Area 122 - Heap Leach Pad & Ponds	249	183	187
Area 128 - Carbon Adsorption	135	88	90
Area 134 - Reagents	37	27	28
Area 338 - Laboratory	95	67	71
Area 362 - Water Supply, Storage & Distribution	470	180	184
Area 368 - Compressed Air & Fuel	56	10	42
Area 366 - Facilities	40	15	30
Total	2,934	1,286	1,696

Power costs are based on fuel consumption and maintenance of diesel generators and is estimated at \$0.38/kWh

15.16.2 Communication

A local utility will provide high speed internet access onsite. The internet connection will be used to provide Voice over Internet Protocol (“VoIP”) phone service. A handheld radio system will also be supplied for process and mining personnel.

15.17 Water

The water supply and distribution system is described in Section 14.6.7.

15.17.1 Sewage

Waste from the onsite restrooms is assumed to be collected and disposed of by a service.

15.17.2 Solid Waste

Hazardous wastes will be collected and stored in the hazardous waste storage facility near the mine shop. Non-hazardous solid waste will be buried in an onsite Class III landfill facility.

16.0 Market Studies

Star Gold has not completed any formal market studies for Longstreet. This section is not applicable

17.0 Environmental Studies, Permitting and Plans, Negotiations or Agreements with Local Individuals or Groups

Star Gold has staked and maintains 139 unpatented mineral exploration claims on United States Forest Service (USFS) and Bureau of Land Management (BLM) lands. It also has the right to mine on an additional 5 claims held by Clifford, et al. The Company has an active Plan of Operations with USFS and BLM to drill several water supply and monitor wells as well as additional core drilling. Star Gold also has begun work to permit a large initial baseline core drilling program for the “Red Cyprus Ridge” prospect located on the Company’s USFS claims. Updates to the Project’s baseline studies have been ongoing since June of 2025. Star Gold is planning on submitting a complete Plan of Operations Application to USFS and BLM early in 2027 to facilitate full mining operations.

The heap leach pads and most of the required support infrastructure will be located on BLM claims with mining operations occurring on USFS claims. Star Gold had previously decided to locate the heap leach pads in a small canyon immediately adjacent to the proposed mining area on USFS claims but decided to relocate the heap leach pads to BLM claims due to engineering and water management considerations with the canyon location. This location is also near additional potentially mineralized zones and may be utilized in the future.

17.1 Permitting Process

The Star Gold Project is of modest size and the area where it is located is largely undisturbed or has naturally reclaimed itself from past man-made disturbances.

To assist the permitting process for mining and exploration activities in Nevada, a Memorandum of Understanding (MOU) exists between the Nevada Division of Environmental Protection (NDEP), the USFS Forest Service (USFS) and the US Bureau of Land Management (BLM). It has been in place since 2008 and is periodically renewed with the latest renewal being in June of 2019. This agreement helps to coordinate the responsibilities of the Agencies pertaining to the administration and reclamation of lands disturbed by exploration or mining operations.

The following permits will be required from the USFS, BLM and NDEP for the mine to go into production.

17.1.1 U.S. Forest Service

- Approval of a Plan of Operations;
- Approval for upgrading access roads;

- Approval of a reclamation plan for USFS lands with notice to NDEP (the reclamation plan is part of the Plan of Operations);
- Approval of a reclamation cost estimate for USFS lands for bonding purposes; and
- An Environmental Assessment or an Environmental Impact Statement (EIS) (triggered by a request for the above approvals). It is presumed that USFS would be a Cooperating Agency or a joint Lead Agency, as much of the proposed permanent disturbance is on USFS lands.

17.1.2 Bureau of Land Management

- Approval of a Plan of Operations for any required disturbances, such as the heap leach pads, upgrading existing roads or granting rights-of way for new roads, mine office, employee locker room, equipment maintenance facilities, parts and supplies warehouse, fuel tanks, etc.
- Approval of a reclamation plan (part of Plan of Operations) in a format that has been developed jointly with NDEP;
- An Environmental Assessment or Environmental Impact Statement (EIS) (triggered by a request for the above approvals). The EA or EIS would be prepared in cooperation with the USFS, as noted above;
- Approval of a reclamation cost estimate for bond purposes (the cost estimate is separately reviewed by NDEP); and
- If a single bond is to be issued, the MOU, noted above, states that "...an interagency agreement may be executed as necessary."

17.1.3 Nevada Division of Environmental Protection (and Other Agencies, as noted)

- **Water Pollution Control Permit.** This is a major permit in Nevada, required whether or not there is any water discharge contemplated. Much of the information required for Federal EA/EIS will serve as input for the application for this permit. Analysis of the acid generating potential of all types of rock to be disturbed is an important part of this permit. Also required for the permit application are descriptions of the geological and hydrogeological conditions, proposed operating plans, proposed monitoring plans, detailed descriptions of leach pads and ponds, etc.;
- **Reclamation Permit.** This permit application must utilize guidelines prepared by NDEP and BLM (the USFS has its own guidelines). Cost estimates for carrying out the plan by a contractor would be used to determine bond amounts. As noted above, this is usually done in conjunction with the BLM and USFS. Bonding must be obtained before construction can begin;
- **Storm Water Permit.** This permit is a general permit requiring only an application to obtain coverage, but requires preparation of a Storm Water Pollution Prevention Plan;

- **Air Quality Operating Permit.** As pertaining to the Longstreet Gold Project, this permit covers emissions from diesel generators, rock crushing and mining operations;
- **Approval by the Nevada Division of Water Resources** to change the point of diversion for the water rights leased by Star Gold that are currently allocated elsewhere in the watershed to future water supply wells close to the mine; and
- **An Industrial Artificial Pond Permit** must be obtained from the Nevada Department of Wildlife.

17.1.4 Other Permits

There are several other permits, which would be expected to be issued rather routinely with minimal input from the applicant, as opposed to the above-listed permits, most of which would require significant scientific and engineering input.

Other required permits include:

- Permits to store explosives and cyanide;
- Permits to treat sanitary waste and dispose of plant and office trash on site;
- Permit if a drinking water system is to be installed;
- County permits, such as business license and building permits;
- Registration with various agencies; and
- Petroleum spill prevention plan.

Once mining commences, a Toxic Release Inventory must be filed annually with the U.S. EPA and the Nevada State Emergency Response Commission.

17.2 Timing of Approvals

Star Gold is planning on submitting a complete mine Plan of Operations Application to USFS and BLM early in 2027 to facilitate full mining operations. Based on recent permitting in Nevada and the current permitting climate being promoted by the Federal Government, the time required to secure permits, prior to the construction of facilities and mine pre-stripping, is estimated to be up to 12-14 months. This schedule is bolstered by the fact that all baseline studies are completed and are currently being updated (where required), hydrogeologic and geochemical study work has commenced and will be completed by the end of 2026. No significant objections have been raised thus far by members of the public, including indigenous peoples, environmental groups or other government agencies (note that the U.S. EPA conducts a review of all Environmental Assessment/Environmental impact statements).

Star Gold is fully funded to complete the required baseline environmental study updates, the additional technical studies underway as all engineering work and Plan of Operations preparation on an accelerated schedule as much of the required work is being completed simultaneously.

17.3 Inventoried Roadless Area

According to the USFS Decision Memo of August 2011, most of the Longstreet Exploration Project is located within the Georges Canyon Inventoried Roadless Area (IRA). While noting that the Project area is open to entry under the mining laws, the USFS states that effects to the IRA and its potential wilderness values are protected because no new roads are to be built and minimal overland travel would occur. Star Gold is working to minimize new road construction and USFS has indicated that necessary new roads or road improvements will be permitted if all required design and environmental criteria are followed

The current Project plan has been engineered to largely utilize existing access roads but the main access road will need significant upgrades to provide efficient access to the heap leach pads and to provide 24-hour access to the mine site. Additionally, new road construction and existing road improvements have been approved in the past to facilitate exploration drilling. The currently approved USFS and BLM Plans of Operations for well and exploration drilling use only existing roads for access.

17.4 Project Biology and Greater Sage-Grouse Considerations

Baseline biological studies completed on the entire claim block in 2015 have determined that there are no active or historic sage-grouse leks in or near the Project area. Updates to this study work involving additional species of interest and botany were begun in 2025 and additional work is scheduled for 2026 and spring of 2027. Additionally, Star Gold is preparing its full mine Plan of Operations with the assumption of presence with approved mitigation conditions being “built into” the approved mine Plan of Operations.

The Nevada Department of Wildlife (NDOW) has produced a map (see Figure 17-1) entitled “Annual Composite Habitat Suitability Index”, which as nearly as can be determined on a map of this scale (approximately 1 inch = 50 miles on the copy below) places the Longstreet Project near habitat areas classified as Habitat of Moderate or Low Value.

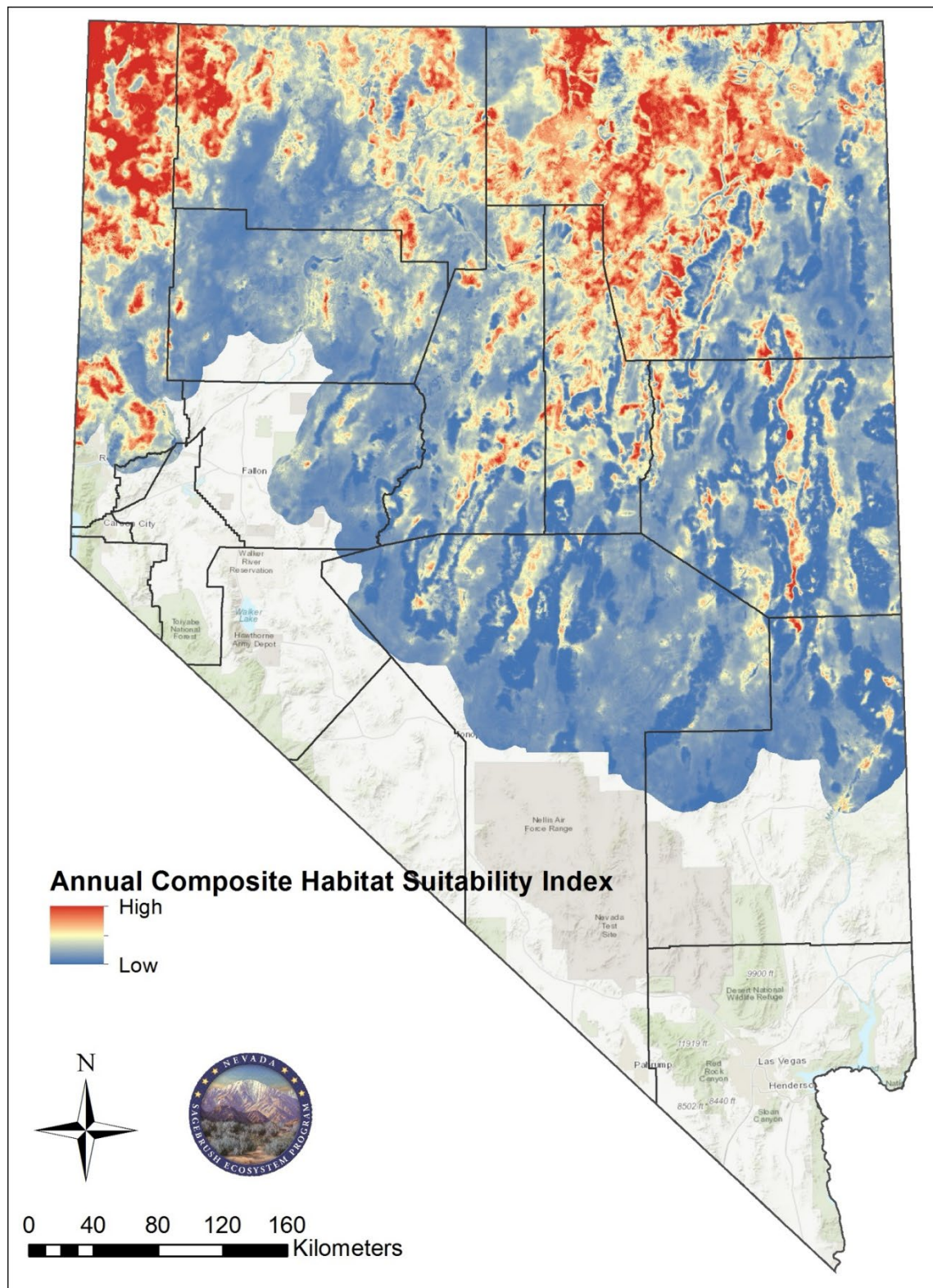


Figure 17-1
Annual Composite Habitat Suitability Index

No warranty is made by the Nevada Department of Wildlife as to the accuracy, reliability, or completeness of these data for individual use or aggregate use with other data.

This map is available for download at <https://sagebrusheco.nv.gov/maps-science>

A winter bat study was completed in January 2016 (and reconfirmed in January 2020), which indicated the presence of the Townsend's Long Eared Bat in the old underground workings of the Longstreet Mine. The Townsend's Long Eared Bat is not listed as an endangered species either federally or by the State of Nevada. Star Gold has initiated preliminary work to develop a remediation plan for this bat colony and habitat. There are many suitable sites not far from the Project where bat habitat can be created or enhanced to compensate for the loss of this habitat due to mining activities. However, the former underground Longstreet Mine workings access points have largely collapsed and are limited to two ventilation shafts/drifts. It is possible these two openings have collapsed since the last site visit in December of 2019. Verification work is currently underway as to the status of the relevant mine workings. If the workings are found to be open, a winter bat study will be scheduled for December 2026 to confirm presence of the bats or confirm that they are no longer present.

Some calendar/time of year restrictions have been placed on exploration drilling operations approved in the current USFS Plan of Operations that might be conducted in the proposed mining area. The drilling of water wells, that is approved in the current USFS Plan of Operations, is not affected by any restrictions as these wells are far from the bat habitat. The restrictions are not considered a material impact to the Project.

The first of two consecutive year raptor studies was completed in the spring of 2026 and no active nests were noted in the required study area. Star Gold had also completed two earlier raptor studies with no active nest sites being noted in the study area.

No threatened or endangered flora or fauna were identified previously on or near the Project area. If during the study update work endangered flora and/or fauna are identified the mine Plan of Operations will be adjusted accordingly to facilitate mitigation of the impact on these species.

17.5 Cultural Resources

The site of the Star Gold claims has seen previous mining operations dating back to approximately 1904 and concluding in 1929. Small-scale milling operations were conducted on the site and a small community is known to have existed near the old mine. Little remains of the past operations or community but there are the ruins of a cabin thought to have belonged to the original prospector, Mr. Longstreet, within the Project boundaries.

The Baseline Cultural Resources Study was completed on the entire claim block (both USFS and BLM claims) in 2015 and identified many Cultural Resources sites. Most of these are non-significant Native American camp locations and artifacts along with remnants of the Longstreet settlement within the Project boundaries. The most significant Native American sites are located far from the mine, leach pads and other Project impacts. Remediation plans will be developed for the sites that would be impacted, as required. Mining and leaching activities will not be impacted by any sites deemed worthy of preservation. Updates to the Cultural Resources Study are completed on an as-needed basis during the current exploration and well drilling activities. Star Gold is utilizing on-site monitoring by a cultural resources' expert, on as needed basis, during the current drilling program to minimize any cultural resource impacts.

17.6 Hydrogeology

17.6.1 Regional Hydrogeology

The Longstreet Mine is in the Monitor Range of the Great Basin section of the Basin and Range physiographic region. The Basin and Range is largely an arid region encompassing a majority of the western United States; its topography is characterized by alternating narrow faulted mountain ranges and flat fault bound valleys (basins). The Great Basin, as a section of the Basin and Range region, follows the same topographic characteristics with notable internal surface and subsurface hydrologic drainage.

The aquifer system within the Great Basin generally comprises aquifers in unconsolidated alluvial fill, sedimentary and volcanic deposits in fault bounded basins, and in various bedrock lithologies of the mountain ranges that drain into the separate basins. The mountain range bedrock units often underlie the basins. The basic hydrogeological model is illustrated in Figure 17-2. The mountain range consists of consolidated bedrock with limited unconsolidated alluvial fill. The bedrock in the mountain ranges generally is less porous and permeable rocks compared to the basin's bedrock. These rocks are characterized by fractured flow conditions. The resulting lower permeability impedes groundwater flow and the fractured flow conditions, in many cases, limit the groundwater volume available as a resource.

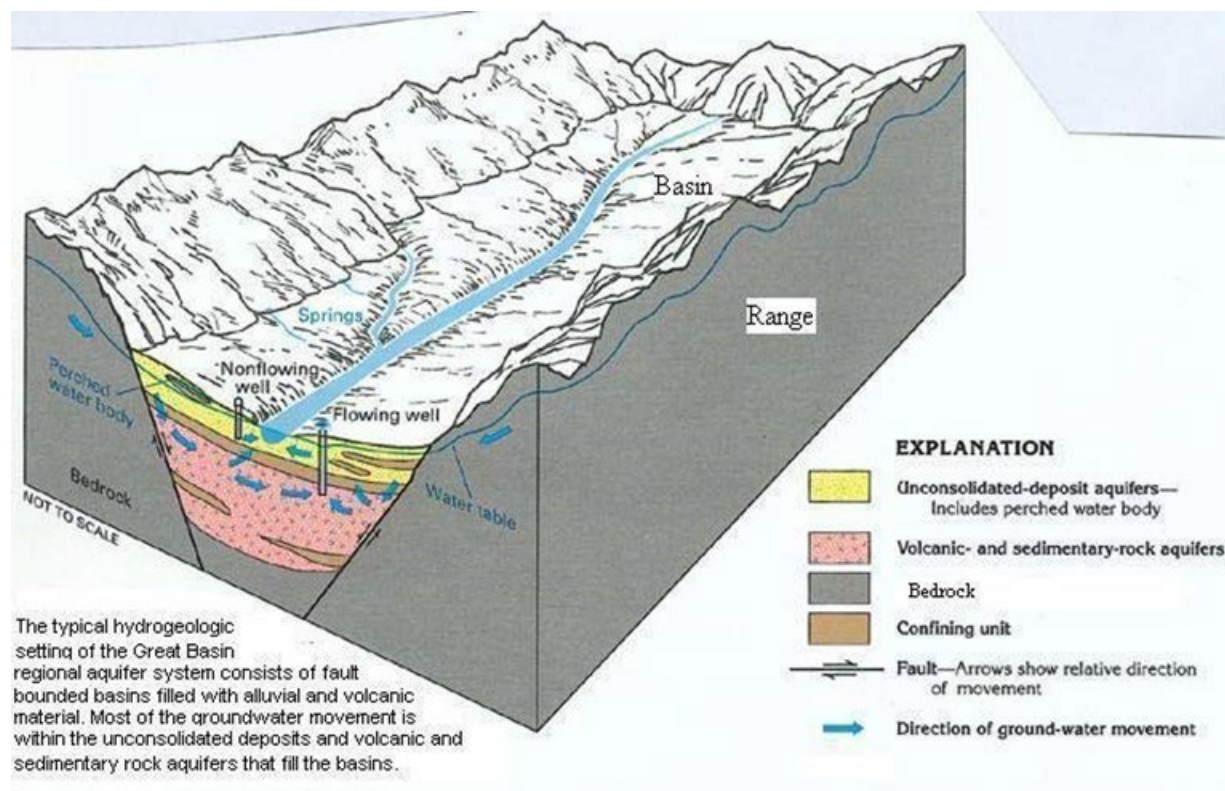


Figure 17-2
Great Basin Hydrogeology

17.6.2 Local Hydrogeology

Limited information is available regarding the actual water resources that exist within the Monitor Range where the Longstreet Project is located. Three springs are mapped on the eastern edge of the Monitor Range: Painted Rock Spring, Side Hill Spring and Four Mile Spring. All the springs are relatively close to the Longstreet deposit. They are located at or near the topographic transition between the Monitor Range and the Stone Cabin Valley. The existence of springs indicates potential exploitable groundwater in the bedrock, and/or the alluvium of the ephemeral streams that flow east out of the Monitor Range. The volume of water flowing from the springs, and any seasonal variation, is not known. The Side Hill Spring is the closest; it is located approximately 2.5 km east of the Longstreet Mine.

The greatest potential source of groundwater is Stone Cabin Valley. The valley has a drainage area of 961 square miles with a net recharge to the basin of 16,000 acre-feet of water per year.

In 1962, the cumulative 'loss/use' via evapotranspiration and reclamation was estimated to be 2,000 acre-feet per year. Since 1962, no significant development has occurred to alter this 'loss/use' estimate. This difference between recharge and discharge rates indicate this

groundwater resource could supply a substantial amount of water without significantly lowering the groundwater levels or negatively affecting existing groundwater use within the valley.

Stone Cabin Valley is principally utilized as livestock range. The Clifford Ranch is the only identified active ranch in Stone Cabin Valley. The ranch is located approximately 19 km south-southeast of the mine site. Five Mile Spring is located at this ranch.

At Mud Lake, Stone Cabin Valley via surface drainages drains into Ralston Valley. A well field exists in the Ralston Valley supplying water to the City of Tonopah. The water from this field is transported the 24 km to Tonopah by pipe.

17.6.3 Mine Site Hydrogeology

Limited information is currently available regarding the presence of water at the mineral project site. Preliminary interpretation regarding the potential availability of water can be predicated with the existing geological maps created by previous mine owners. The maps indicate that the Longstreet property is underlain predominantly by Oligocene Epoch moderately to poorly-welded tuffs with common lithic and pumice fragments. Four lithologic units have been described at the Site:

“Welded Ash Flow Tuff (Tat) – This rock is buff to grey, and contains <10% fine-to medium-grained quartz phenocrysts, 15% fine-to medium-grained feldspar phenocrysts, 5% to 15% medium to coarse-grained pumice, and 5% to 20% other “exotic” fragments in an aphanitic groundmass. The rock displays horizontal bedding and may be up to 3,000 feet thick. It exhibits pervasive hydrothermal alteration consisting of argillic alteration (bleaching and clay mineral development), silicification (quartz flooding and/or networks of numerous quartz veinlets), and potassic alteration (adularia in quartz veinlets). Supergene limonitic and goethite alteration overprint the hydrothermal alteration.

Rhyolitic Porphyry Dike (Trp) -- Rhyolitic porphyry dikes of various orientations intrude the Tat unit and may be associated with the heat source of the mineralizing fluids at Longstreet.

Siliceous Sedimentary Rock (Ts) – A thin unit of white, yellowish and grey, volcanoclastic and siliceous rock (including sinter) intermittently overlies the Tat unit. Silicic alteration is evidenced by sheeted quartz veins.

Welded Tuff (Trt) – Black to brown, strongly welded tuff occurs along ridges and overlies the Tat and Ts units. This unit is 330 feet to 400 feet thick and has a distinctive thin (approximately 3 to 10 foot thick) vitrophyre zone near its base.”

This indicates the welded tuffs have a limited capacity to store water (porosity) or allow water to flow (permeability). The welded strength of the tuff affects porosity and permeability: the greater the welding, the lower the porosity and permeability. Therefore, the strongly Welded Tuffs, by definition, have low porosity and permeability. The Welded Ash Flow Tuff and the Welded Tuff are interpreted to be dense and relatively impervious rocks. The Siliceous Sedimentary Rock offers a potential porous media for groundwater; however, its thickness may limit the volume of water that it can store. No information is available regarding the presence of water in these lithologic units.

Suggested locations for potential well sites are identified in Figure 17-3.

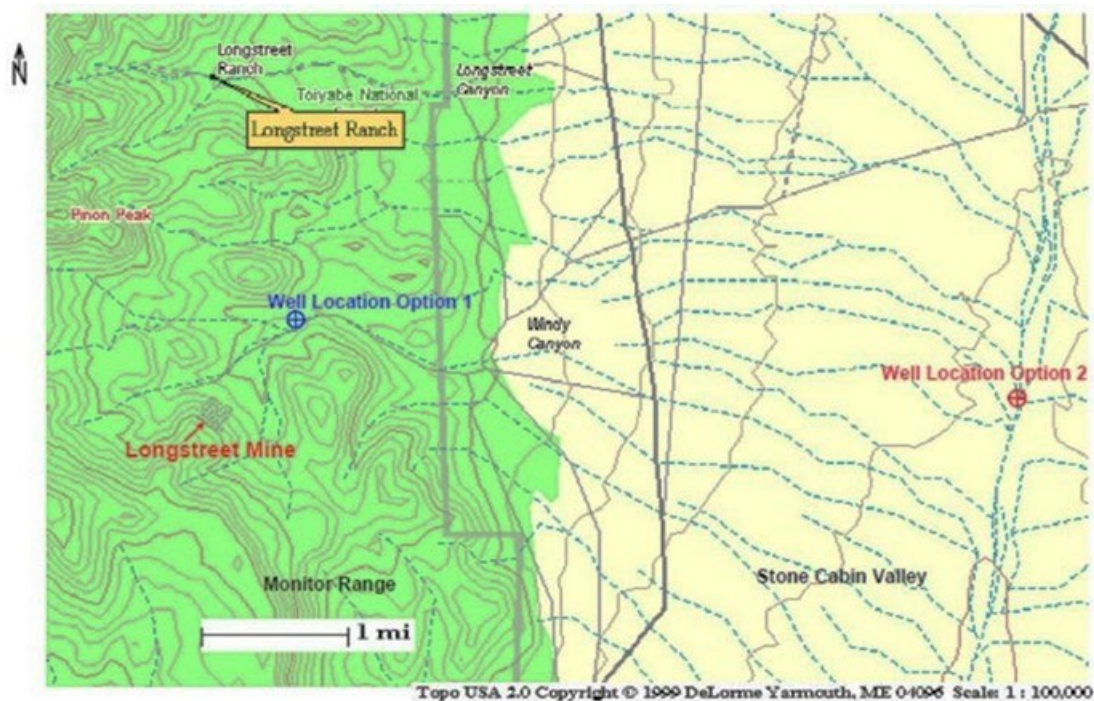


Figure 17-3

Water Well Search Location Recommendations

One of the limiting factors for water availability in the Great Basin is the low water recharge to the local aquifers due to the limited precipitation in the area. The Great Basin resides in the rain shadow of the Sierra Nevada Mountains; thus, precipitation is limited and irregular; the least precipitation occurs in the valleys and the greatest in the mountains. Winter precipitation generally consists of snow and summer precipitation is characterized by localized high intensity rain. Geologic evidence and recorded history indicate the intense rainstorms may result in flooding of the major rivers and the “dry” washes, and due to the arid conditions, the evaporation rate is high. Precipitation that does not evaporate either percolates into the subsurface or moves as surface runoff into the valley basins thereby the basin aquifers are recharged.

As illustrated in Figure 17-2, both the surface runoff, and the groundwater in the ranges flow into the valley basins; therefore, the valleys are the best sources of water.

Well Location Option 1 is approximately 1.1 km from the Longstreet Project. Water is expected but volume and sustainability is unknown. The location is chosen because the surface drainage is at the intersection of three sub-watersheds.

Well Location Option 2 is approximately 8 km from the Longstreet Project. Water is expected with the necessary volume and sustainability. This location is at the valley center and assumed to be the best location for water production. Other locations can be identified between this site and the entrance to Windy Canyon that could potentially supply the volume and long-term sustainability requirements of the Project.

Star Gold has secured, through two long term leases, 2,471 acre/feet of water rights from current owners of these water rights in Stone Cabin Valley. The acre/feet of water leased is at least 150% larger than what is anticipated to be required for mining and ore leaching applications.

Star Gold has an approved Plan of Operations with the USFS and BLM to conduct water supply and monitor well drilling in a favorable location near the Project site. The well drilling began in late August 2026 and will continue until November or December of 2026. Once water has been located, a point of diversion application will be applied for with the Nevada Division of Water Resources. Based on the results of the well drilling program and pump testing, a baseline hydrogeologic study will be developed.

The required seeps and springs location study and sampling work commenced at the Longstreet site in July 2026 and will continue a quarterly basis until June 2027. No adverse conditions have been noted thus far during the study work.

The locations of the proposed wells are shown in the following map (Figure 17-4).

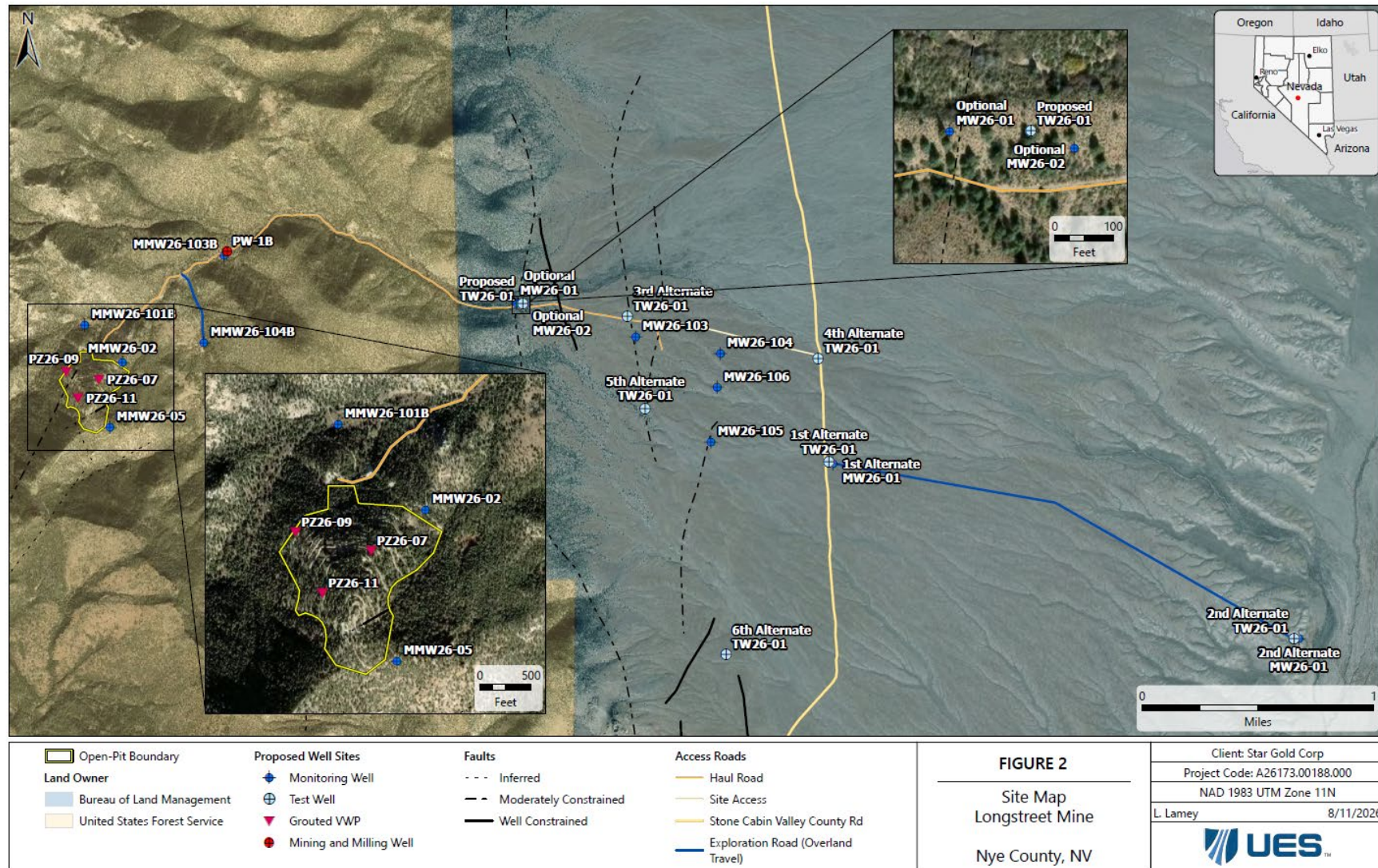


Figure 17-4
Site Map Longstreet Mine

17.7 Environmental and Permitting Conclusions

The permitting of mining operations within the United States is never a simple process and is always time consuming and expensive. Nevada is considered one of the most favorable jurisdictions to permit a heap leach gold operation due to the long history of operations in the state. Given the current information regarding this Project, it is estimated that the permitting timeline could be up to 12-14 months once the complete mine operating Plan of Operations is submitted. This timeline will likely be reduced somewhat by the modest size of the proposed operation and its relatively small impacts.

Baseline studies for flora and fauna and cultural resources have been completed with hydrogeologic, geotechnical and geochemical studies are scheduled for completion in 2026. These studies will be accomplished during the continued advanced engineering phase of the Project.

Kappes Cassiday and Associates concludes that there are no recognized potential environmental or permitting fatal or material flaws regarding this Project.

18.0 Capital and Operating Costs

Capital and operating costs for the process and administration components of the Longstreet Project were estimated by KCA. Costs for the mining components were provided by Jim Ashton. The estimated costs are considered to have an accuracy of +/-40% and are discussed in greater detail in this Section

The total Life of Mine (LOM) capital cost for the Project is \$84.2 million and is summarized in Table 18-1 below.

Table 18-1
Capital Cost Summary

Description	Cost (US\$ M)
Pre-Production Process Capital	\$57.6
Mining Capital	\$4.2
Subtotal Capital	\$61.8
Working Capital & Initial Fills	\$7.8
Total Initial Capital	\$69.6
Sustaining Capital – Mine & Process	\$4.6
Reclamation & Closure	\$10.0
Total	\$84.2

The average LOM operating cost for the Project is US\$22.98 per ton of ore processed. Table 18-2 below presents the LOM operating cost requirements for the Longstreet Project.

Table 18-2
Operating Cost Summary

Description	Cost (US\$ M)
Mine	\$7.49
Process & Support Services	\$13.27
Site G&A	\$2.22
Total	\$22.98

18.1 Capital Expenditures

The required capital cost estimates have been based on the design outlined in this report. The scope of these costs includes all expenditures for process facilities, infrastructure, construction indirect costs, mine contractor mobilization and owner mining capital costs for the Longstreet Project.

The costs presented have primarily been estimated by KCA, Dyer Consultant Engineering and James Ashton with input from Stargold. Material take-offs for earthworks, concrete and major piping have been estimated by KCA. All equipment and material requirements are based on design information described in previous sections of this report. Capital costs estimates have been made primarily using budgetary supplier quotes for contract mining and major process equipment. For smaller equipment, supplier quotes of similar KCA projects were used.

All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer or estimated to be fabricated new.

The total pre-production capital cost estimate for the Longstreet Project is estimated at US\$71 million, including all mining, process equipment and infrastructure, construction indirect costs, mine contractor costs before \$8 million working capital and initial fills. All costs are presented in third-quarter 2026 US dollars.

Pre-production capital costs for mining, processing, and infrastructure required for the Longstreet Project by area are presented in Table 18-3

Table 18-3

Summary of Mining, Process, and Infrastructure Pre-Production Capital Costs by Area

Process & Infrastructure Direct Costs	Total Supply Cost	Freight & Sales Tax	Install	Grand Total
	US\$	US\$	US\$	US\$
Area 113 - Crushing	\$3,411,406	\$219,711	\$4,282,230	\$7,913,348
Area 115 - Crushed Ore Stockpile, Reclaim & Stacking	\$3,827,745	\$478,059	\$985,220	\$5,291,024
Area 122 - Heap Leach Pad & Ponds	\$1,636,589	\$158,113	\$5,801,052	\$7,595,753
Area 128 - Carbon Adsorption	\$3,589,972	\$104,499	\$836,167	\$4,530,638
Area 129 - Carbon Handling	\$0	\$0	\$0	\$0
Area 131 - Strip & Refinery	\$0	\$0	\$0	\$0
Area 134 - Reagents	\$350,895	\$32,259	\$530,443	\$913,596
Area 338 - Laboratory	\$2,099,790	\$0	\$368,850	\$2,468,640
Area 360 - Power	\$4,923,104	\$620,311	\$750,527	\$6,293,942
Area 362 - Water Supply, Storage & Distribution	\$899,539	\$93,417	\$580,898	\$1,573,854
Area 368 - Compressed Air & Fuel	\$17,996	\$2,310	\$38,881	\$59,187
Area 366 - Facilities	\$504,391	\$13,148	\$119,514	\$637,053
Area 008 - Plant Mobile Equipment	\$2,442,441	\$307,748	\$4,103	\$2,754,291
Process & Infrastructure Total Direct Costs	\$23,703,867	\$2,029,573	\$14,297,885	\$40,031,325
Spare Parts	\$427,451			\$427,451
Sub Total with Spare Parts				\$40,458,776
Contingency	\$6,748,043			\$6,748,043
Process & Infrastructure Total Direct Costs with Contingency				\$47,206,819

Indirect Costs	Grand Total
Indirect Field Costs	\$1,669,039

Indirect Field Costs Contingency				incl
Total Indirect Costs				\$1,669,039

Other Owner's Costs				Grand Total
Other Owner's Costs				\$3,807,000
Other Owner's Costs Contingency				\$190,350
Total Other Owner's Costs				\$3,997,350

Initial Fills				\$269,949
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Sub Total Cost Before EPCM				\$53,143,158
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EPCM	10%			\$4,720,682
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Process, Mining & Infrastructure Working Capital (60 days)				\$7,583,938
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Mining Capital Costs				\$4,197,500
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Sub Total Mining, Process & Infrastructure Pre-Production Capital Cost				\$69,645,278
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18.2 Mining Capital Costs

Mine capital costs have been estimated by James Ashton based on an assumption that contract mining will be used. The mine capital costs have been prepared for both contractor and owner capital and are used in the economic analysis in Section 19.0. The mine capital cost estimate is shown in Table 18-4.

Table 18-4
Mining Capital

Mining Capital Costs	Units	Initial	Sustaining	Total LOM
Capitalized Mine Haulage Road	K USD	\$600		\$600
Site, Waste Dump/Pit Preparation	K USD	\$100		\$100
Mob/Demob	K USD	\$300	\$150	\$450
Mine Access Road	K USD	\$2,200		\$2,200
Storm Water Control	K USD	\$50		\$50
Mining Support	K USD	\$400		\$400
Subtotal	K USD	\$3,650	\$150	\$3,800
Contingency (15%)	K USD	\$548	\$23	\$570
Total Costs	K USD	\$4,198	\$173	\$4,370

18.3 Process and Site Infrastructure Capital Costs

18.3.1 Process and Site Infrastructure Capital Cost Basis

Process and infrastructure costs have been estimated by KCA. All equipment and material requirements are based on the design information described in previous sections of this Report. Capital costs are based on budgetary quotes for all major and most minor equipment of KCA similar projects. Where Project specific quotes were not available a reasonable estimate or allowance was made based on recent quotes in KCA's files. All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer or to be fabricated new.

Each area in the process cost build-up has been separated into the following disciplines, as applicable:

- Major earthworks & liner;
- Civil (concrete);
- Structural steel;
- Platework;
- Mechanical equipment;
- Piping;
- Electrical;
- Instrumentation;
- Infrastructure & Buildings;
- Supplier Engineering; and
- Commissioning & Supervision.

Pre-production process and infrastructure costs by discipline are presented in Table 18-5.

Table 18-5
Summary of Pre-Production Capital Costs by Discipline

Discipline	Cost @ Source	Freight	Sales Tax	Total Supply Cost	Install	Grand Total
	US\$	US\$	US\$	US\$	US\$	US\$
Major Earthworks & Liner	\$258,470	incl.	\$19,644	\$278,114	\$5,601,106	\$5,879,220
Civils (Supply & Install)	\$3,355,690	incl.	incl.	\$3,355,690	incl.	\$3,355,690
Structural Steelwork (Supply & Install)	\$181,007	incl.	incl.	\$181,007	incl.	\$181,007
Platework (Supply & Install)	\$0	incl.	incl.	\$0	\$0	\$0
Mechanical Equipment	\$17,098,026	\$740,796	\$1,178,086	\$19,016,908	\$7,857,329	\$26,874,237
Piping	\$525,358	\$52,536	\$39,927	\$617,821	\$365,826	\$983,647
Electrical	\$1,000,001	\$14,443	\$15,301	\$1,029,744	\$400,600	\$1,430,344
Instrumentation	\$335,419	\$16,771	\$25,492	\$377,681	\$207,600	\$585,281
Infrastructure	\$1,515,994	\$25	\$11,608	\$1,527,627	\$108,000	\$1,635,627
Spare Parts	\$427,451			\$427,451		\$427,451

Contingency	\$7,027,570			\$7,027,570		\$7,027,570
Plant Total Direct Costs	\$31,724,984	\$824,570	\$1,290,058	\$33,839,612	\$14,540,461	\$48,380,074

Freight, sales taxes, and installation costs are considered for each discipline. Freight costs, when quoted, were used in the study. When freight was not quoted, freight cost was based on loads as bulk freight and have been estimated at 5% of the equipment cost.

Installation costs, when quoted, were used in the study. Where not directly quoted, installation costs are estimated from the equipment cost and an hourly installation rate of US\$114.

18.3.2 Major Earthworks and Liner

Earthworks and liner quantities for the heap have been estimated by Dyer Engineering Consultants and by KCA for all other Project areas. Earthworks and liner supply and installation will be performed by contractors with imported fill being supplied by the mining contractor.

Unit rates for site earthworks and liner supply and installation are based on rates from a similar project in 2025.

Earthworks also include cost for the crushing retaining wall and the earthworks associated with the reclaim tunnels. Total preproduction earthworks costs are estimated at US\$5.9 million.

18.3.3 Civils

Civils is the cost of concrete. Concrete quantities have been estimated by KCA. Where available, quoted prices were used. When necessary, a concrete unit cost of US\$1,700 per cubic yard was used.

The total costs for concrete are estimated at US\$3.4 million.

18.3.4 Structural Steel

Costs for structural steel, including steel grating, structural steel, and handrails. The structural steel costs for the crushing plant are estimated at \$0.18M.

The steel associated with the rest of the plant were included, but not itemized, in the carbon columns quote used for this study.

18.3.5 Platework

The platework discipline includes costs for the supply and installation of steel tanks, bins, and chutes. Platework costs have been included, but not itemized, in the quotes used for this study.

18.3.6 Mechanical Equipment

The majority of mechanical equipment costs are from vendor packages.

Mechanical equipment costs, not included in vendor packages, are based on the mechanical equipment list and vendor quotes. Where quotes were not available, reasonable allowances were made based on KCA's data base. All costs assume equipment purchased new from the manufacturer or to be fabricated new. The total installed mechanical equipment cost is estimated at US\$26.9 million.

18.3.7 Piping

Heap leach solution collection piping quantities and unit rates were estimated by KCA. Installation hours were estimated by supply price with a unit rate of US\$114 per hour.

Piping was estimated by a percentage of Mechanical Equipment and compared to other KCA similar projects. No installation was included for drip irrigation; it is assumed this will be installed by operators.

The total installed piping cost is estimated at US\$0.525 million.

18.3.8 Electrical

Electrical equipment, within equipment packages, was part of the quoted price but not itemized. The total installed electrical equipment cost is estimated at \$1.4 million.

Diesel generators for main power were selected as the best option versus high voltage lines from Tonopah. Quotes of similar size generators from other KCA projects in the area were used. The total cost for diesel generators is estimated at \$6.7 million.

18.3.9 Instrumentation

Instrumentation costs are primarily included as part of turn-key or complete vendor supply packages and are not itemized.

Minor miscellaneous instrumentation costs have been estimated as percentages of the mechanical equipment supply cost for each process area. The total installed instrumentation cost is estimated at US\$0.209 million.

18.3.10 Infrastructure

The infrastructure costs are miscellaneous costs including fencing, flumes for the heap leach, laboratory and process maintenance facilities and the installation costs for modular buildings.

The total infrastructure costs are estimated at US\$1.6 million

18.3.11 Process Mobile Equipment

Mobile equipment included in the capital cost estimate are detailed in Table 18-6. Process Mobile Equipment.

Table 18-6
Process Mobile Equipment

Description	Quantity
Track Hoe W/Rock Hammer	1
Front Loader	1
Telehandler	1
Dozer (Heap)	1
Mechanic Service Truck	1
Flatbed Truck	1
Skid Steer Loader	1
Pickup Truck	4
Light Plant	2

Costs for process mobile equipment are based on both quotes and on costs from a 2025 cost guide adjusted for inflation. Mobile equipment costs are located in the mechanical equipment cost estimate.

The total Plant Mobile Equipment costs are estimated at US\$2.75 million

18.3.12 Spare Parts

Spare parts costs are estimated at 2.5% of the mechanical equipment supply costs. Total spare parts costs are estimated at US\$0.427 million.

18.3.13 Contingency

Contingency for the process and infrastructure has been applied to the total direct costs by discipline. Contingency has been applied ranging from 10% to 20% as detailed in Table 18-7. The overall contingency for process and infrastructure is estimated at 16.9% of the direct costs.

Table 18-7
Process & Infrastructure Contingency

Direct Costs Contingency	%	Total (US\$)
Major Earthworks	20.0%	\$1,175,844
Civils (Supply & Install)	20.0%	\$671,138
Structural Steelwork (Supply & Install)	20.0%	\$36,201
Platework (Supply & Install)	20.0%	\$0
Mechanical Equipment	15.0%	\$4,031,136
Piping	20.0%	\$105,164
Electrical	20.0%	\$274,228
Instrumentation	20.0%	\$41,717
Commissioning and Supervision	20.0%	\$0
Infrastructure	20.0%	\$327,125
Spare Parts	20.0%	\$85,490
Total Direct Costs Contingency	16.9%	\$7,027,570

18.3.14 Construction Indirect Costs

Indirect field costs include temporary construction facilities, construction services, quality control, survey support, warehouse and fenced yards, support equipment, etc. Construction indirect costs are summarized in Table 18-8.

Table 18-8
Construction Indirect Costs

Description	Total (US\$)
Misc. Hotels, etc.	\$75,000
QA/QC Earthworks, Liner, and Concrete	\$350,000
Surveying	\$200,926
Construction Equipment Rentals & Operating Costs	\$150,000
Office Equipment (Copiers, Printers, Computers, Plotter)	\$42,000
Clinic	\$26,250
Construction Vehicle O&M (2 Pickups + Flatbed)	\$120,000

Description	Total (US\$)
Construction Tools	\$26,250
Construction Phone / Internet	\$25,000
Construction Power Opex and Rental	\$205,440
Portable Toilet Service	\$50,000
Outside Consultants / Vendor Reps	\$40,000
Construction Warehouse (Core Shed)	\$30,000
Construction Office Trailers / Containers (Rental & set-up)	\$50,000
Sub-Total Indirect Costs	\$1,390,866
Indirect Contingency (20%)	\$278,173
Total	\$1,669,039

18.3.15 Other Owners Costs

Other Owner's construction costs are intended to cover the following items:

- Owner's costs for labor, home office support, vehicles, travel and consultants during construction;
- Subscriptions, licence fees, etc.;
- Taxes and Permits;
- Work place health and safety costs during construction.

Other Owner's construction costs are estimated based on 16 months of site construction and are summarized in Table 18-9.

Table 18-9
Other Owner's Costs

Description	Total (US\$)
Operator Training	\$10,000
MSHA Training and fit testing	\$33,000
Relocation, pre-employment physicals, hearing, pulmonary, etc.	\$150,000
Access Roads and Maintenance	\$15,000
Traffic Study	\$50,000
Surveying (not including construction needs)	\$10,000
First Aid and Medical during Construction	\$10,000
Construction Water	\$30,000
Safety and Road Signage, Traffic controls during construction	\$50,000
Employee Housing Assistance	\$250,000
Owner's Insurance	\$250,000
Support and Consultants	\$100,000
Communications and Computer Equipment	\$100,000
Early Staffing	\$1,000,000
Metallurgical Testing	\$25,000
Outside Lab Services (until on-site lab ready)	\$135,000

Furniture	\$50,000
Description	Total (US\$)
Land Lease	\$21,000
BLM Fees	\$150,000
State and County Fees/Taxes	\$90,000
Royalties	\$0
ERP System (enterprise, resource, planning) and work order system	\$100,000
Community Relations / Charity	\$20,000
Utility Rights-of-Way (Power line, Water line, Access Roads, etc.)	\$45,000
Personnel Safety Equipment, incl. AED, gas monitors, etc.	\$13,000
Office Supplies, Copier/Scanner, other office expenses	\$45,000
Builder's All Risk Insurance	\$255,000
Baseline Studies (Biological, Desert T, Raptor, Burrowing Owl, Etc.)	\$100,000
Tortoise Fence	\$0
Cactus Garden and Harvesting	\$0
Environmental Testing (Phase 2 WRC, etc.)	\$175,000
Development Impact Fees (EA Ph. 2)	\$100,000
Permits and Fees (WTP, landfill, AQ, WPCP, SPCC, SWP3, Dam ...)	\$150,000
Shop Tools and Furnishings	\$100,000
General Supplies, Operations and Maintenance	\$75,000
Light Vehicle Operating Costs	\$100,000
Local Office Rental	\$0
Sub-Total Other Owner's Costs	\$3,807,000
Owner's Costs Contingency (5%)	\$190,350
Total Owner's Costs	\$3,997,350

18.3.16 Initial Fills

The initial fills consist of consumable items stored on site at the outset of operations, which includes sodium cyanide (NaCN), antiscalant, activated carbon and acid. Initial fills are summarized below in Table 18-10 below

Table 18-10
Initial Fills

Item	Basis	Needed Weight lb or gal	Quantity to Order lb or gal	Unit Price US\$	Total Cost US\$
NaCN (lbs)	Full Tank	45,000	90,000	\$1.45	\$130,636.00
Carbon (lbs)	Full Circuit & Inventory	50,000	50,000	\$2.43	\$121,500.00
Antiscalant (gals)	1 month	300	550	\$2.10	\$1,174.00
Sodium Hypochlorite	Full Tank	30	30	\$9.33	\$2,827.00

Lab Chemicals		Included with Lab Costs			
Carbon Dewatering Bag/Liner	3 months	100	100	\$108.28	\$10,828.00
Pallets	3 months	100	100	\$29.84	\$2,984.10
Total					\$269,949.00

18.3.17 Engineering, Procurement, & Construction Management

The estimated costs for engineering, procurement and construction management (EPCM) for the development, construction, and commissioning are based on a percentage of the direct capital cost. The total EPCM cost is estimated at US\$8.3 million, or 10% of the process and infrastructure direct costs.

The EPCM costs cover services and expenses for the following areas:

- Project Management.
- Detailed Engineering.
- Engineering Support.
- Procurement.
- Construction Management.
- Commissioning.
- Vendors Reps.

For some major equipment packages, costs associated with detailed engineering, commissioning, and installation supervision have been included in the vendor's quotes; these costs are reflected in the supplier engineering estimate of the capital costs and have been considered when estimating the EPCM costs and are not included in this estimate.

18.3.18 Working Capital

Working capital is money that is used to cover operating costs from start-up until a positive cash flow is achieved. Once a positive cash flow is attained, Project expenses will be paid from earnings. Working capital for the Project is estimated to be US\$7.6 million based on 60 days of operation and includes all mine, process and G&A operating costs, process pre-production costs and the initial fill of reagents.

18.3.19 Operating Cost Summary

Process operating costs for the Longstreet Project have been estimated based on information presented in earlier sections of this Report. LOM Mining costs were provided by James Ashton at US\$7.49 per ton of ore and are based on quotes for contract mining with estimated owner's mining costs.

Process operating costs have been estimated by KCA from first principles. Labor costs and staffing were sourced from KCAs files of a recent project in Nevada. Unit consumptions of materials, supplies, power, water and delivered supply costs were estimated. LOM average processing costs are estimated at US\$13.85 per ton ore.

General administrative costs (G&A) have been estimated by KCA. G&A costs include project specific labor and salary requirements and operating expenses. G&A costs are estimated at US\$2.22 per ton ore.

Operating costs were estimated based on third-quarter 2026 US dollars and are presented with no added contingency based upon the design and operating criteria present in this report. Nevada sales taxes have not been added to the process operating costs.

The operating costs presented are based upon the ownership of all process production equipment and office buildings. The owner will employ and direct all operating maintenance and support personnel for all site activities.

Operating costs estimates have been based upon information obtained from the following sources:

- Contractor mining quotes and owner mining costs from James Ashton and N.A. Degerstrom;
- G&A costs estimated by KCA;
- Project metallurgical testwork and process engineering;
- Supplier quotes for reagents and fuel;
- Recent KCA project file data; and
- Experience of KCA staff with other similar operations.

Where specific data do not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exist. Freight costs have been estimated where delivered prices were not available.

18.3.20 Mining Operating Costs

Mining operating costs are based on a quote from N.A. Degerstrom. Unit rates and cost per year are shown in Table 18-12

Table 18-11
Contract Mining Unit Rates

Description	Cost (US\$/ton)
Ore Haul to Crusher	\$3.06 (ton Ore)
Waste Haul	\$2.09 (ton Waste)
Drill and Blast	\$1.05 (ton Mined)

Fuel consumption was quoted separately. A diesel price of \$3.34/gal was used from KCA recent projects in the area to calculate the cost. The LOM fuel cost works out to \$1.29/ton of ore. The fuel consumption per year and total cost is shown in Table 18-12.

Table 18-12
Fuel consumption and Fuel Cost Contract Mining per Year

Description	Year 1		Year 2		Year 3		Year 4		Total
	Gallons	k US\$	Gallons	k US\$	Gallons	k US\$	Gallons	k US\$	
Ore Haul to Crusher	196,806	\$658	253,286	\$847	228,847	\$765	136,019	\$455	\$2,725
Waste Haul	145,543	\$487	53,767	\$180	133,114	\$445	169,788	\$568	\$1,680
Drill and Blast	83,295	\$279	69,534	\$233	86,667	\$290	77,085	\$258	\$1,059

18.3.21 Process Operating Costs

Average annual process and G&A operating costs are presented in Table 18-13 below.

Table 18-13
Process and G&A Operating Cost

COST	COST TYPE	US\$/TON ORE
Labor - All Process Areas		
Process	Fixed	\$3.117
Laboratory	Fixed	\$0.507
SUBTOTAL		\$3.624
G&A		
G&A Labor	Fixed	\$1.093
Fixed Costs	Fixed	\$1.132
SUBTOTAL		\$2.224
Area 0113 - Crushing		
Power (All Crushing)	Variable	\$0.000
Diesel Generator	Variable	\$1.204
Lube Generator	Variable	\$0.031

COST	COST TYPE	US\$/TON ORE
980 Loader - Operating Cost	Variable	\$0.363
Wear & Maintenance (Primary)	Variable	\$0.030
Wear & Maintenance (Secondary & Tertiary)	Variable	\$0.230
Overhaul / Maintenance (Screen/Misc.)	Variable	\$0.440
SUBTOTAL		\$2.298
Area 0114 - Stacking		
Power	Variable	\$0.000
Cat D6T Dozer at heap - Operating Cost	Variable	\$0.027
Maintenance Supplies	Variable	\$0.143
SUBTOTAL		\$0.170
Area 0122 - Heap Leach & Solution Handling		
Power	Variable	\$0.000
Diesel Generator	Variable	\$0.490
Lube Generator	Variable	\$0.041
Piping/Drip tubing	Fixed	\$0.032
Maintenance Supplies	Fixed	\$0.034
SUBTOTAL		\$0.598
Area 0128 - ADR Recovery Plant		
Power	Variable	\$0.000
Carbon	Variable	\$0.000
Misc. Operating Supplies	Variable	\$0.009
Maintenance Supplies	Variable	\$0.029
Carbon Bags	Variable	\$0.027
Toll Stripping	Variable	\$3.248
SUBTOTAL		\$3.313
Area 0134 - Reagents (Included in ADR)		
Power	Variable	\$0.000
Lime	Variable	\$0.731
Cyanide (Ore)	Variable	\$1.991
Cyanide (Elution)	Variable	\$0.000
Caustic	Variable	\$0.000
Hydrochloric Acid	Variable	\$0.000
Antiscalant	Variable	\$0.000
Fluxes	Variable	\$0.000
Maintenance Supplies	Fixed	\$0.000
SUBTOTAL		\$2.722
Area 0362 - Water Supply & Distribution		
Power	Variable	\$0.000
Pump Maintenance / Overhaul	Variable	\$0.006
Hypochlorite	Fixed	\$0.002
SUBTOTAL		\$0.009
Area 0152 - Laboratory		
Power	Fixed	\$0.000
Building Heating	Fixed	\$0.000
Assays, Solids	Variable	\$0.045
Assays, Solutions	Variable	\$0.078
Miscellaneous Supplies	Fixed	\$0.037
SUBTOTAL		\$0.161

COST	COST TYPE	US\$/TON ORE
Facilities & Support Services		
Power - Buildings/Misc.	Fixed	\$0.000
Building Heating	Fixed	\$0.004
Plant Administration Building	Fixed	\$0.038
Process Office/ADR	Fixed	\$0.009
Process Office/Crusher	Fixed	\$0.009
Mining Administration Building	Fixed	\$0.026
Lunch Area	Fixed	\$0.026
Guard Office Gate	Fixed	\$0.009
Restrooms	Fixed	\$0.044
Restroom Pumping	Fixed	\$0.010
Light Vehicles	Fixed	\$0.049
Carbon Transport (Included in Toll-Stripping)	Fixed	\$0.000
Skid Steer Loader	Fixed	\$0.014
Light Plant	Fixed	\$0.009
Mechanics Service Truck	Fixed	\$0.024
Telehandler (CAT TL943C)	Fixed	\$0.013
Flatbed Truck	Fixed	\$0.024
Crane (65-ton)	Fixed	\$0.067
SUBTOTAL		\$0.378
Total G&A Costs		\$2.224
Total Processing Costs		\$13.271
Fixed Costs		\$6.332
Variable Costs		\$9.164
TOTAL OPERATING COST		\$15.495

18.3.22 Personnel and Staffing

Staffing requirements for process and administration personnel have been estimated by KCA based on recent projects. Total process personnel are estimated at 44 persons. G&A labor is estimated at 10 people.

Personnel requirements and costs are summarized in Table 18-14

Table 18-14
Process Personnel Summary

Descriptiopn	People	US\$/year
Process Supervision	2	\$ 505,700
Crushing and Reclaim	12	\$ 1,463,092
Leach	8	\$ 922,670
Recovery	4	\$ 514,059
Maintenance	10	\$ 1,219,243
Other	1	\$ 204,600
Laboratory	7	\$ 790,149
Total	44	\$ 5,619,513

18.3.23 Power

Power usage for the process and process-related infrastructure was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned operating times to determine the average energy usage and cost. Power requirements for the Project are presented in Table 15-1 in Section 15.0 of this report.

The total attached power for the process and infrastructure is estimated at 2.9 MW. The average power draw is 1.3 MW.

Diesel generators for main power was selected as the best option versus high voltage lines from Tonopah. Quotes of similar size generators from other KCA projects in the area were used. The total cost for diesel generators is estimated at \$6.3 million.

The total consumed power for these areas is approximately 4.63 kWh/t ore. Power will be supplied by diesel generators. The power cost is estimated at US\$0.38/kWh.

18.3.24 Consumable Items

Operating supplies have been estimated based upon unit costs. Freight costs are included in all operating supply and reagent estimates. Reagent consumptions have been derived from test work and from design criteria considerations. Other consumable items have been estimated by KCA based on KCA's experience with other similar operations.

18.3.24.1 Heap Leach Consumables

Pipes, Fittings and Emitters – The heap pipe costs are estimated to be US\$0.032/ton ore and are based on a complete change of drip tubing and an allowance for valves fitting and pipes

Sodium Cyanide (NaCN) – Delivered sodium cyanide is estimated at US\$1.45/lb, this is the average of quoted cyanide prices. Cyanide is consumed in the heap leach at 1.31 lb/ton ore.

Pebble Lime (CaO) – Pebble lime is consumed at an average rate of 3.58 lb/ton ore for pH control of the heap. A delivered price of US\$0.204/lb was estimated. The pebble lime was quoted, the delivery cost was estimated based on the quoted price in 2025 escalated for inflation.

Antiscalant (Scale Inhibitor) – Antiscalant consumption is based on a dosage 5 ppm to the suction of the barren and preg pumps. A delivered price of US\$2.10/lb based on a quote from a local supplier.

18.3.24.2 *Carbon Adsorption Columns*

Activated Carbon – Loaded carbon will be stripped in an outside facility. Based on experience from other projects in Nevada, due to lack of carbon reactivation, activated carbon will be fully replaced. A delivered price of \$2.43/lb was estimated based on a quote from 2025 escalated for inflation. Total cost of activated carbon is estimated at \$2.63/ton and is included in the toll-stripping cost shown in Table 18-13.

18.3.24.3 *Laboratory*

Fire assaying and solution assaying of samples will be conducted in the on-site laboratory. The fire assays are based on:

- One blasthole assay per 360 ton mined
- 20% excess assays
- 9 process fire assays per day

The laboratory will assay approximately 56 solutions per day.

18.3.24.4 *Wear, Miscellaneous Operating & Maintenance Supplies*

Wear, overhaul and maintenance of equipment along with miscellaneous operating supplies for each area have been estimated as allowances based on the tons of ore processed. The allowances for each area were developed based on published data as well as KCA's experience with similar operations.

Wear steel is estimated at \$0.26 per ton. Maintenance and operating supplies costs (excluding G&A) are estimated at US\$4,161,538 per year or US\$1.694 per ton ore.

18.3.24.5 *Mobile / Support Equipment*

Mobile and support equipment are required for the process and include one telehandler, one skid steer loader, two portable light plants, one service truck, one flat bed truck and four pickup trucks. An allowance of \$100,000 per year was added for crane rental.

Support equipment annual operating costs are estimated at US\$319,335 or US\$0.201 per ton of ore. Support equipment operating costs are presented in Table 18-15.

Table 18-15
Support Equipment Operating Costs

Description	Unit	Qty.	Unit Cost	Annual Cost, US\$
Light Vehicles	h/y	2,288	\$26.73	\$80,273
Skid Steer Loader	h/y	1,095	\$16.18	\$23,250
Light Plant	h/y	5,840	\$2.01	\$15,369
Mechanics Service Truck	h/y	1,100	\$27.62	\$39,871
Telehandler (CAT TL943C)	h/y	1,095	\$15.03	\$21,597
Flatbed Truck	h/y	1,100	\$27.00	\$38,975
Crane (65-ton)	\$/yr	1	\$100,000.00	\$100,000
Total				\$319,335

18.3.24.6 General and Administrative Costs

General and administrative expenses are expected to average US\$3.3 million per year and include costs for insurance, communications, environmental management, health and safety supplies, security, and travel. For the cost estimate G&A expenses are represented as fixed costs.

G&A labor expenses are presented in Table 18-16, G&A expenses are presented in Table 18-17.

Table 18-16
G & A Labor

Job Title	Qty.	Salary	Hourly	Overtime	Bonus	Burdens	Ea.	Total
Mine Manager	1	\$225,750			\$112,875	\$79,013	\$417,638	\$417,638
Purchasing Agent	1	\$94,500			\$18,900	\$33,075	\$146,475	\$146,475
HSE Manager	1	\$162,750			\$56,963	\$56,963	\$276,675	\$276,675
Warehouse Tech	1		\$75,616	\$7,562	\$4,537	\$26,466	\$114,180	\$114,180
AP Clerk	1		\$67,704		\$4,062	\$23,696	\$95,463	\$95,463
IT Tech	1		\$80,808		\$4,848	\$28,283	\$113,939	\$113,939
HSE Tech	1		\$80,808		\$4,848	\$28,283	\$113,939	\$113,939
Security Tech	3		\$75,616	\$7,562	\$4,537	\$26,466	\$114,180	\$342,541
TOTAL	10							\$1,620,851

Table 18-17
G & A Expenses

Description	Note	Annual Cost
General Maintenance Supplies	Allowance	\$25,000
Office Furniture and Supplies	Estimate	\$25,000
Phone/Internet/Data	Allowance	\$20,000
Courier/Postage	Allowance	\$15,000
Light Vehicle Operating Costs	Estimate	\$75,000
Recruiting and On-Boarding	Allowance	\$100,000
Communications & Public Relations	Allowance	\$75,000
Insurance (Auto, Liability, W/Comp)	Estimate	\$200,000
BLM Fees	2023 Cost	\$150,000
Land Lease	2023 Cost	\$21,000
State and County Fees/Taxes	2023 Cost	\$90,000
Safety Supplies	Allowance	\$50,000
Environmental (Compliance Testing, Etc.)	Allowance	\$100,000
Training and Training Supplies	Allowance	\$30,000
Professional Services (HR, IT, Payroll)	Estimate/Allowance	\$180,000
Consultants	Allowance	\$100,000
Business Meetings and Travel	Allowance	\$60,000
Legal and Accounting Fees	Allowance	\$100,000
Dues/Memberships/Subscriptions	Allowance	\$15,000
Access Road Maintenance	From Ledcor Quote	\$75,000
Janitorial Services	Allowance	\$20,000
Other	10%	\$152,600
TOTAL		\$1,678,600

18.3.24.7 Reclamation and Closure Costs

A cost estimate for reclamation and closure was made by KCA. This includes work to be conducted from the closure of the mine, end of operation activities and concurrent rehabilitation work. These costs exclude G&A costs during closure.

The main objectives of the reclamation and closure plan include:

- Proper abandonment of all groundwater wells;
- Closure of the heap leach pad through process solution recirculation for inventory reduction and stabilization, cover/growth media placement and revegetation, and

construction of an evapotranspiration (ET) cell to collect and management long-term drain down;

- Removal or abandonment of pipelines;
- Surface reclamation of roads and other surface disturbances;
- Demolition of process facilities and salvage/removal of equipment and residual reagents for proper disposal;
- Establishment of appropriate post-closure stormwater management and control.

Reclamation and closure costs is estimated at US\$10M or US\$1.53 per ton of ore processed.

19.0 ECONOMIC ANALYSIS

19.1 Summary

Based on the estimated production schedule, capital costs and operating costs, a cash flow model was prepared by KCA for the economic analysis of the Longstreet Project. All of the information used in this economic evaluation has been taken from work completed by KCA and other consultants working on this project as described in previous sections of this study.

The project economics were evaluated using a discounted cash flow (DCF) method, which measures the Net Present Value (NPV) of future cash flow streams. Using a gold price of \$3,600/oz, a period of six years including one year of investment and pre-production and four years of production, reclamation and closure, a processing rate of 5,500 tons per day, gold recovery of 84%, silver recovery of 13%, capital and operating costs of this report, the proposed operation shows promising economics.

The after-tax NPV for the Longstreet Project is US\$67M at 5% with an IRR of 40%. The life of mine (LOM) all in sustaining cost is US\$1,729.

19.2 Methodology

The economic model was developed by KCA based on the following:

- The cash flow model is based on the preliminary mine production schedule from James W. Ashton.
- The period of analysis is five years including four years of production and one year for reclamation and closure.
- All cash flow amounts are in US dollars (US\$). All costs are considered to be third-quarter 2026 costs. Inflation is not considered in this model.
- The Internal Rate of Return (IRR) is calculated as the discount rate that yields a zero Net Present Value (NPV).
- The NPV is calculated by discounting the annual cash back to Year -1 and different discount rates. All annual cash flows are assumed to occur at the end of each respective year.
- The payback period is the amount of time, in years, required to recover the initial construction capital cost.
- Working capital is considered in this model and includes initial fills, mining, processing and general administrative operating costs for 60 days. The model assumes working capital is recovered during the final two years of operation, 50% on year four and 50% on year five.

- There are no royalties associated with the Longstreet Project.
- Government taxes are included in the model.
- 100% equity financing is assumed.
- Salvage value for process equipment is considered and is applied at the end of the project.
- Reclamation and closure costs are included.

19.3 Assumptions

General assumptions for the model, including cost inputs, parameters, royalties and taxes are as follows:

- Gold price of US\$3,600/oz.
- Silver price of US\$48/oz.
- Gold and silver production and revenue in the model are delayed from the time material is stacked to account for time required for gold to be recovered from the heap. This results in an estimated 18% of the recoverable values being delayed until the following year.
- Pre-production capital costs for the project are spent entirely in Year -1. Sustaining capital for the heap leach pad expansion is spent in Year 2. Capital cost estimates are presented in greater detail in Section 18.0 of this report.
- Cash operating costs per payable ounce represent the mine site operating costs including mining, processing, administration costs with a credit for silver produced. Operating costs are presented in greater detail in Section 18.3.19 of this report.
- All in sustaining costs per payable ounce represent the mine site operating costs including mining, processing, metal transport, refining, administration costs with a credit for silver produced as well as the LOM sustaining capital and reclamation and closure costs.
- The cash flow analysis evaluates the project on a stand-alone basis. No withholding taxes or dividends are included. No head office or overheads for the parent company are included.
- Tariffs and inflation were not taken into consideration.
- A portion of the NSR royalty is assumed to be purchased in the first year of operation. A 1.5% NSR was applied in the economic model.

A summary of key economic parameters is shown in Table 19-1

Table 19-1
Key Economic Parameters

Item	Value	Units
Au Price	3,600	US\$/oz
Ag Price	48	US\$/oz
Au Avg. Recovery	84	%
Ag Avg. Recovery	13	%
Treatment Rate	5,500	ton/d
Annual Produced eqAu, Avg.	21.1	koz
Royalties	1.5	%
Mine Life	4	years
Payback	1.8	years

A summary of the economic model is presented in Table 19-2

Table 19-2
Economic Analysis Summary

Economic Analysis		
Internal Rate of Return (IRR), Pre-Tax	48.30%	
Internal Rate of Return (IRR), After-Tax	39.57%	
Average Annual Cashflow (Pre-Tax)	\$43	M
NPV @ 5% (Pre-Tax)	\$87	M
Average Annual Cashflow (After-Tax)	\$38	M
NPV @ 5% (After-Tax)	\$67	M
Gold Price Assumption	\$3,600	/Ounce
Silver Price Assumption	\$48	/Ounce
Pay-Back Period (Years based on After-Tax)	1.8	Years
Capital Costs		
Initial Capital	\$62	M
Working Capital & Initial Fills	\$8	M
LOM Sustaining Capital	\$5	M
Operating Costs (Average LOM)		
Mining	\$7.49	/Ton processed
Processing & Support	\$13.27	/Ton processed
G&A	\$2.22	/Ton processed
Total Operating Cost	\$22.98	/Ton processed
Total Cash Cost	\$1,617	/Ounce Au
All-in Sustaining Cost	\$1,729	/Ounce Au
Production Data		
Life of Mine	4	Years

Total Tons to Crusher	6,552,050	Ton
Grade Au (Avg.)	0.016	ounces per ton
Grade Ag (Avg.)	0.416	ounces per ton
Contained Au oz	105,461	Ounces
Contained Ag oz	2,728,471	Ounces
Mine Throughput per day	13,000	Tons/day
Metallurgical Recovery Au (Overall)	84%	
Metallurgical Recovery Ag (Overall)	13%	
Average Annual Silver Production	80,310	Ounces
Average Annual Gold Production	20,057	Ounces
Total Silver Produced	354,701	Ounces
Total Gold Produced	88,587	Ounces
LOM Strip Ratio	0.81:1	

19.4 Capital Expenditures

Capital expenditures include initial capital (pre-production or construction costs), sustaining capital and working capital. The capital expenditures are presented in detail in Section 18.0 of this report. The pre-production capital expenditures for the project are summarized in Table 18-3.

The economic model assumes working capital and initial fills will be recovered at the end of the operation and are applied as credits against the capital cost. Working capital and initial fills are assumed to be recovered during years 4 and 5. Salvage value for equipment is considered as taxable income and is applied during Year 5.

19.5 Gold Production

Total gold produced for the Longstreet deposit is estimated at 88,587 ounces of payable ounces gold. Annual production profiles for gold are presented in Figure 19-4 with 20,057 payable ounces of gold recovered annually on average.

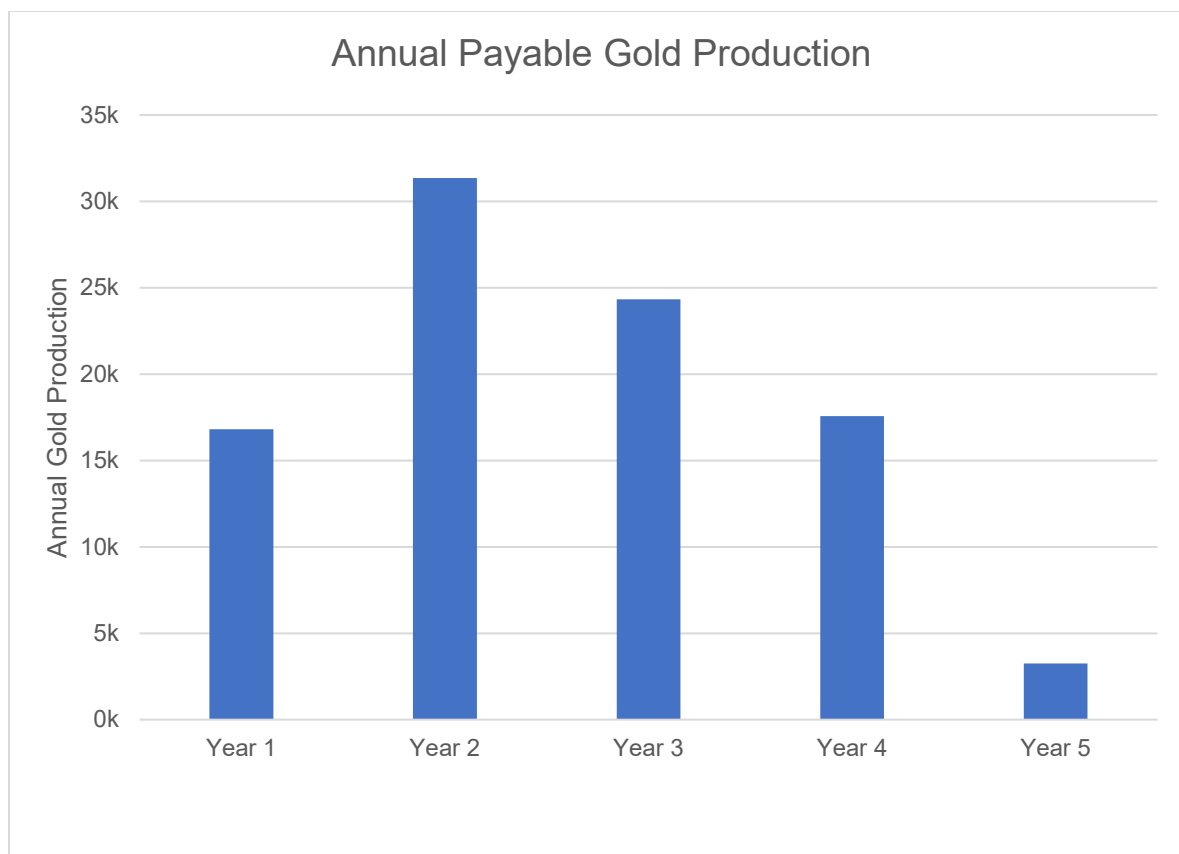


Figure 19-1
Annual Gold Production

19.6 Operating Costs

Operating costs were estimated by KCA for all process and support services. G&A operating costs were estimated by KCA with input from Star Gold. Mining costs were estimated by James W. Ashton. LOM operating costs for the Longstreet deposit are summarized in Table 19-3. A detailed description of the operating cost build-up is included in Section 18.0 of this report.

Table 19-3
Operating Costs

Description	Cost (US\$/ton)
Mining Cost	\$7.49
Process Cost	\$13.27
G&A	\$2.22
Total	\$22.98

19.7 Royalties

A portion of the NSR royalty is assumed to be purchased in the first year of operation. A 1.5% NSR was applied in the economic model.

19.8 Taxes

The following taxes are included in the Cash Flow:

- Nevada Excise Tax
- Income Tax
- Nevada Net Proceeds Tax

The Nevada Excise Tax is 0.765% of the Net Revenue.

19.9 Economic Cash Flow

The Base Case After-tax NPV for the Longstreet Project is US\$67M at 5% with an IRR of 40% using a gold price of US\$3,600/oz. The base case life of mine (LOM) all in sustaining cost US\$1,729.

Table 19-4
Cash Flow Model

Assumptions		
Au Price	3,600	\$/oz
Ag Price	48	\$/oz
Au Recovery	84%	
Ag Recovery	13%	
Treatment Rate	5,500	stpd
Gold Refining Charge	0.00	\$/oz
Silver Refining Charge	0.00	\$/oz
Gold Pay Factor	100.0%	
Silver Pay Factor	100.0%	
Royalties	1.50%	
Nevada Au & Ag Mine Royalty (Excise Tax)	0.77%	
Salvage Value Percentage (Infrastructure)	10.0%	Assumed
Salvage Value Percentage (Process Eq.)	20.0%	Assumed
Salvage Value Percentage (Electrical Eq.)	15.0%	Assumed

Output							
	<i>Pre-Tax NPV</i>	<i>i, %</i>	<i>After-Tax NPV</i>				
	\$108,057,522	0%	\$85,473,975	Mine Life	4.0	years	
	\$87,280,921	5%	\$67,326,315				
	\$76,573,348	8%	\$57,980,137	Payback	1.8	years	
	\$70,055,930	10%	\$52,293,607				
	\$55,622,644	15%	\$39,706,552				
	48%	<i>IRR</i>	40%				
Total Au Recovered	88,587	Ounces	Stripping Ratio	0.81	t/t		
Payable Ounces	88,587	Ounces					
Annual Au oz (avg payable oz)	23,329		LOM ore grade	0.016	gpt Au		
Max Annual Au oz	31,360		LOM ore grade	0.416	gpt Ag		
Cost per ounce (- Reclaim.), \$	\$1,617						
All-in Sustaining Cost per ounce, \$	\$1,729		LOM Tons	6,552,050			

				Year 1				Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Item	UNITS	TOTAL	Year -1	Q1	Q2	Q3	Q4						
Total Mined													
Total Ore, tonnes		6,552,050		263,190	335,010	484,068	500,000	2,036,352	1,839,870	1,093,560			
Au, opt		0.016		0.017	0.014	0.017	0.017	0.018	0.014	0.017			
Ag, opt		0.416		0.3165	0.3069	0.3600	0.3873	0.3836	0.3874	0.6223			
Contained Au, oz		105,461		4,345	4,851	8,190	8,467	36,208	25,326	18,074			
Open Pit		105,461		4,345	4,851	8,190	8,467	36,208	25,326	18,074			
Contained Ag, oz		2,728,471		83,312	102,826	174,276	193,634	781,112	712,763	680,548			
Open Pit		2,728,471		83,312	102,826	174,276	193,634	781,112	712,763	680,548			
Waste Mined		5,305,059		334,862	389,602	425,374	387,591	567,961	1,406,132	1,793,538			
Total mined		11,857,109		598,052	724,612	909,441	887,591	2,604,313	3,246,002	2,887,098			
Strip Ratio (W:O)		0.81		1.27	1.16	0.88	0.78	0.28	0.76	1.64			

Ore Processed	Total	Year -1	Q1	Q2	Q3	Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Ore Processed												
Ore Processed to Heap Leach	6,552,050		263,190	335,010	484,068	500,000	2,036,352	1,839,870	1,093,560			
Au grade, opt	0.016		0.017	0.014	0.017	0.017	0.018	0.014	0.017			
Contained Au, oz	105,461		4,345	4,851	8,190	8,467	36,208	25,326	18,074			
Ag grade, opt	0.416		0.317	0.307	0.360	0.387	0.384	0.387	0.622			
Contained Ag, oz	2,728,471		83,312	102,826	174,276	193,634	781,112	712,763	680,548			
Eq. Au oz	138,533		5,355	6,098	10,302	10,814	45,676	33,965	26,323			
Recoverable Gold by Period, oz	88,587		3,650	4,075	6,880	7,112	30,415	21,273	15,182			
Ultimate Recovery, Au	84%		84%	84%	84%	84%	84%	84%	84%			
Recoverable Silver by Period, oz	354,701		10,831	13,367	22,656	25,172	101,545	92,659	88,471			
Ultimate Recovery, Ag	13%		13%	13%	13%	13%	13%	13%	13%			
Total Gold Produced Profile, oz	88,587		1,034	3,527	4,841	6,759	30,014	23,064	16,375	2,973		
Total Silver Produced Profile, oz	354,701		2,166	9,533	14,802	21,611	100,919	94,807	89,483	21,381		
Eq. Gold, oz	93,316		1,063	3,654	5,039	7,047	31,360	24,328	17,568	3,258		
Silver payable, oz	354,701		2,166	9,533	14,802	21,611	100,919	94,807	89,483	21,381		
Gold payable, oz	88,587		1,034	3,527	4,841	6,759	30,014	23,064	16,375	2,973		
Eq. Gold payable, oz	93,316		1,063	3,654	5,039	7,047	31,360	24,328	17,568	3,258		
Gold Subject to Royalty	1,400		16	55	76	106	470	365	264	49		
Refining & Transportation Charge	\$0		\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$0		
NET REVENUE	\$335,938,997		\$3,826,747	\$13,154,733	\$18,139,550	\$25,368,004	\$112,894,620	\$87,579,626	\$63,245,893	\$11,729,824	\$0	\$0

		Year 1										
OPERATING COSTS	Total	Year -1	Q1	Q2	Q3	Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Operating Costs \$/t ore												
Mining Cost \$7.49	\$49,050,486		\$2,121,482	\$2,700,398	\$3,901,898	\$4,030,325	\$11,412,209	\$13,477,445	\$11,406,729			
Processing Cost \$13.27	\$86,953,739		\$3,102,697	\$3,949,369	\$5,706,580	\$5,894,405	\$24,073,122	\$22,415,694	\$17,927,065	\$3,884,807		
G&A Cost \$2.22	\$14,572,573		\$548,821	\$698,585	\$1,009,410	\$1,042,634	\$3,299,451	\$3,299,451	\$3,299,451	\$1,374,771		
TOTAL OPERATING COSTS \$22.98	\$150,576,799	\$0	\$5,773,001	\$7,348,353	\$10,617,888	\$10,967,363	\$38,784,781	\$39,192,590	\$32,633,244	\$5,259,578	\$0	\$0
\$19,392,391												
OPERATNG CASH FLOW	\$185,362,198	\$0	-\$1,946,253	\$5,806,381	\$7,521,662	\$14,400,641	\$74,109,839	\$48,387,036	\$30,612,649	\$6,470,246	\$0	\$0

TAXES	Total	Year -1	Q1	Q2	Q3	Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Taxes												
Income Tax Payable	\$20,052,163	\$0	\$0	\$0	\$0	\$1,196,713	\$9,760,716	\$5,889,359	\$3,205,374	\$0	\$0	\$0
TOTAL TAXES	\$20,052,163	\$0	\$0	\$0	\$0	\$1,196,713	\$9,760,716	\$5,889,359	\$3,205,374	\$0	\$0	\$0

CASH FLOW BEFORE CAPITAL	\$165,310,036	\$0	-\$1,946,253	\$5,806,381	\$7,521,662	\$13,203,927	\$64,349,123	\$42,497,676	\$27,407,275	\$6,470,246	\$0	\$0
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CAPITAL COSTS	Total	Year -1	Year 1				Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
			Q1	Q2	Q3	Q4						
Capital Costs												
Mine												
Capitalized Mine Haulage Road	\$750,000	\$600,000								\$150,000		
Site, Waste Dump/Pit Preparation	\$100,000	\$100,000										
Mob/Demob	\$300,000	\$300,000										
Mine Access Road	\$2,200,000	\$2,200,000										
Storm Water Control	\$50,000	\$50,000										
Mining Support	\$400,000	\$400,000										
Contingency	\$570,000	\$547,500								\$22,500		
Mine Subtotal	\$4,370,000	\$4,197,500								\$172,500		
Major Earthworks & Liner	\$9,583,815	\$5,879,220					\$3,704,595					
Civils (Supply & Install)	\$3,355,690	\$3,355,690										
Structural Steelwork (Supply & Install)	\$181,007	\$181,007										
Platework (Supply & Install)	\$0	\$0										
Mechanical Equipment Supply	\$19,016,908	\$19,016,908										
Mechanical Equipment Install	\$7,857,329	\$7,857,329										
Piping	\$525,819	\$525,819										
Electrical Supply	\$970,542	\$970,542										
Electrical Install	\$400,600	\$400,600										
Instrumentation	\$208,584	\$208,584										
Infrastructure	\$1,635,627	\$1,635,627										
Spare Parts	\$427,451	\$427,451										
Mercury Storage	\$0											
EPCM & Commissioning	\$4,720,682	\$4,720,682										
Contingency	\$7,488,962	\$6,748,043					\$740,919					
Indirect Costs (incl. contingency)	\$1,669,039	\$1,669,039										
Owner's Costs (incl. contingency)	\$3,997,350	\$3,997,350										
Sub-Total Capital Costs	\$66,409,405	\$61,791,391	\$0	\$0	\$0	\$0	\$4,445,514	\$0	\$0	\$172,500	\$0	\$0
Working Capital (Initial Fills)	\$269,949	\$269,949										
Working Capital Process, Mining, G&A	\$7,583,938	\$7,583,938										

Less: Working Capital Recovery	\$7,853,887								\$3,926,944	\$3,926,944		
Net Working Capital	\$0	\$7,853,887	\$0	\$0	\$0	\$0	\$0	\$0	-\$3,926,944	-\$3,926,944	\$0	\$0
Subtotal	\$66,409,405	\$69,645,278	\$0	\$0	\$0	\$0	\$4,445,514	\$0	-\$3,926,944	-\$3,754,444	\$0	\$0
Reclamation & Closure	\$10,000,000									\$10,000,000		
Less: Salvage Value	\$4,143,813									\$4,143,813		
TOTAL CAPITAL	\$72,265,592	\$69,645,278	\$0	\$0	\$0	\$0	\$4,445,514	\$0	(\$3,926,944)	\$2,101,743	\$0	\$0

PRE-TAX NET CASH FLOW	Total	Year -1	Q1	Q2	Q3	Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Pre-Tax Net Cash Flow												
Pre-tax net cash flow -pre Royalties	\$113,096,607	-\$69,645,278	-\$1,946,253	\$5,806,381	\$7,521,662	\$14,400,641	\$69,664,325	\$48,387,036	\$34,539,593	\$4,368,502	\$0	\$0
Royalty Payable	\$5,039,085		\$57,401	\$197,321	\$272,093	\$380,520	\$1,693,419	\$1,313,694	\$948,688	\$175,947	\$0	\$0
Nevada Excise Tax 0.77%	\$2,531,384		\$28,835	\$99,124	\$136,686	\$191,154	\$850,689	\$659,934	\$476,574	\$88,387	\$0	\$0
Pre-tax net cash flow - After Royalties	\$108,057,522	-\$69,645,278	-\$2,003,654	\$5,609,060	\$7,249,568	\$14,020,121	\$67,970,905	\$47,073,341	\$33,590,904	\$4,192,555	\$0	\$0
		-\$69,645,278		\$24,875,094			\$67,970,905	\$47,073,341	\$33,590,904	\$4,192,555	\$0	\$0
Cumulative		-\$69,645,278	-\$71,648,932	-\$66,039,873	-\$58,790,304	-\$44,770,184	\$23,200,722	\$70,274,063	\$103,864,967	\$108,057,522	\$108,057,522	\$108,057,522

			Year 1									
After-TAX NET CASH FLOW		Year -1	Q1	Q2	Q3	Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
After-Tax Net Cash Flow												
Income & Other Taxes	\$20,052,163	\$0	\$0	\$0	\$0	\$1,196,713	\$9,760,716	\$5,889,359	\$3,205,374	\$0	\$0	\$0
After-Tax net annual Cash Flow, \$	\$85,473,975	-\$69,645,278	-\$2,032,490	\$5,509,935	\$7,112,882	\$12,632,253	\$57,359,500	\$40,524,047	\$29,908,957	\$4,104,168	\$0	\$0
	\$85,473,975	-\$69,645,278		\$23,222,581			\$57,359,500	\$40,524,047	\$29,908,957	\$4,104,168	\$0	\$0
Cumulative		-\$69,645,278	-\$71,677,768	-\$66,167,833	-\$59,054,950	-\$46,422,697	\$10,936,803	\$51,460,851	\$81,369,807	\$85,473,975	\$85,473,975	\$85,473,975

Income Taxes		\$11,583,231	\$0	\$0	\$0	\$0	\$0	\$6,195,361	\$3,579,992	\$1,807,878	\$0	\$0	\$0
Net Proceeds of Mineral Tax		\$8,468,932	\$0	\$0	\$0	\$0	\$1,196,713	\$3,565,356	\$2,309,367	\$1,397,496	\$0	\$0	\$0
Total Taxes		\$20,052,163	\$0	\$0	\$0	\$0	\$1,196,713	\$9,760,716	\$5,889,359	\$3,205,374	\$0	\$0	\$0
					1		2	3	4	5			
Payback Period after tax					1.0		0.8	0.0	0.000	0.0	0.0	0.0	
Payback Period before tax					1.0		0.7	0.0	0.0	0.0	0.0	0.0	

19.10 Sensitivity Analysis

To estimate the relative strength of the project, base case sensitivity analyses have been completed analyzing the economic sensitivity to several parameters including changes in gold price, capital costs and average operating cash cost per ton of material processed. The sensitivities are based on +/- 25% of the base case.

The after-tax analysis is presented in Table 19-5. Figure 19-2 through Figure 19-4 present graphical representations of the after-tax sensitivities. Under the downside scenario, a 25% reduction in the base-case gold price to \$3,600 per ounce, has the greatest adverse effect on project economics.

The economic indicators chosen for sensitivity evaluation are the internal rate of return (IRR) and NPV at 0% and 5% discount rates.

Table 19-5
Sensitivity Analysis Results

			NPV		
	Variation	IRR	0%	5%	10%
Gold Price					
75%	\$2,700	10.1%	\$19,419,833	\$8,915,397	\$200,663
90%	\$3,240	28.5%	\$59,226,374	\$44,112,630	\$31,587,748
100%	\$3,600	39.6%	\$85,473,975	\$67,326,315	\$52,293,607
110%	\$3,960	49.9%	\$111,098,631	\$89,989,513	\$72,508,441
125%	\$4,500	64.3%	\$149,470,937	\$123,877,723	\$102,689,908
Capital Costs					
75%	\$41,747,430	57.2%	\$102,076,326	\$83,816,005	\$68,686,726
90%	\$59,062,186	45.8%	\$92,114,915	\$73,922,191	\$58,850,854
100%	\$72,265,592	39.6%	\$85,473,975	\$67,326,315	\$52,293,607
110%	\$86,797,185	34.2%	\$78,833,035	\$60,730,438	\$45,736,359
125%	\$111,084,928	27.4%	\$68,871,624	\$50,836,624	\$35,900,487
Operating Costs					
75%	\$84,699,449	54.5%	\$123,118,175	\$100,590,652	\$81,945,031
90%	\$121,967,207	45.7%	\$100,531,655	\$80,632,050	\$64,154,176
100%	\$150,576,799	39.6%	\$85,473,975	\$67,326,315	\$52,293,607
110%	\$182,197,926	33.3%	\$70,416,295	\$54,020,580	\$40,433,037
125%	\$235,276,248	23.5%	\$47,829,775	\$34,061,977	\$22,642,182

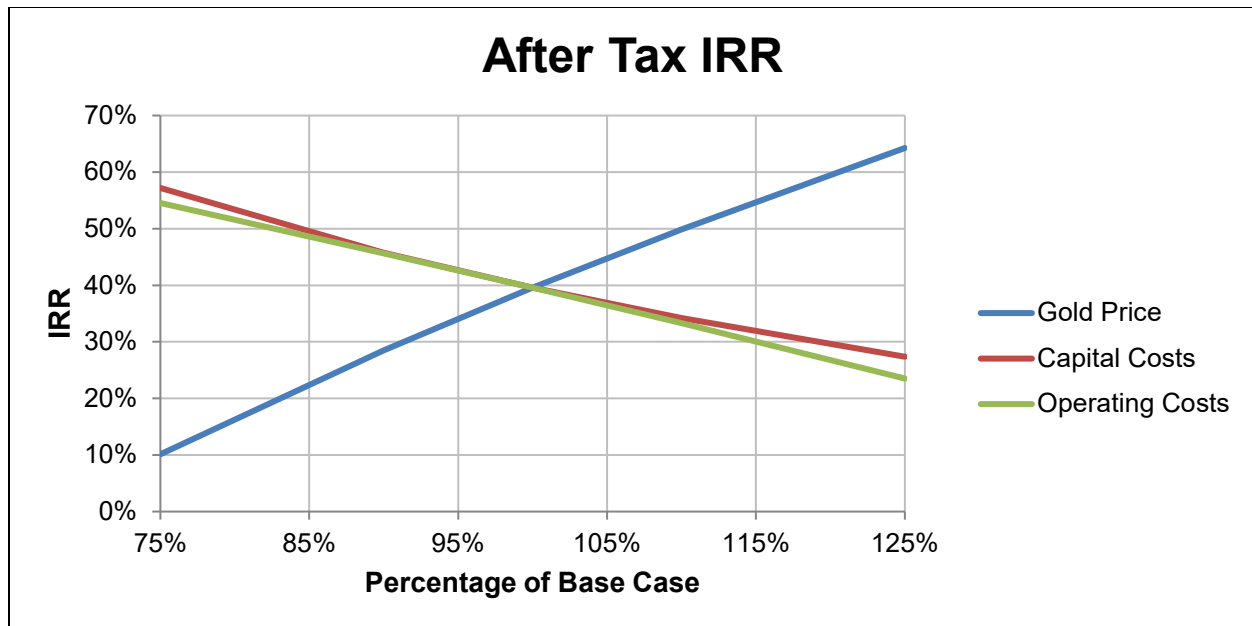


Figure 19-2
After Tax Sensitivity – IRR

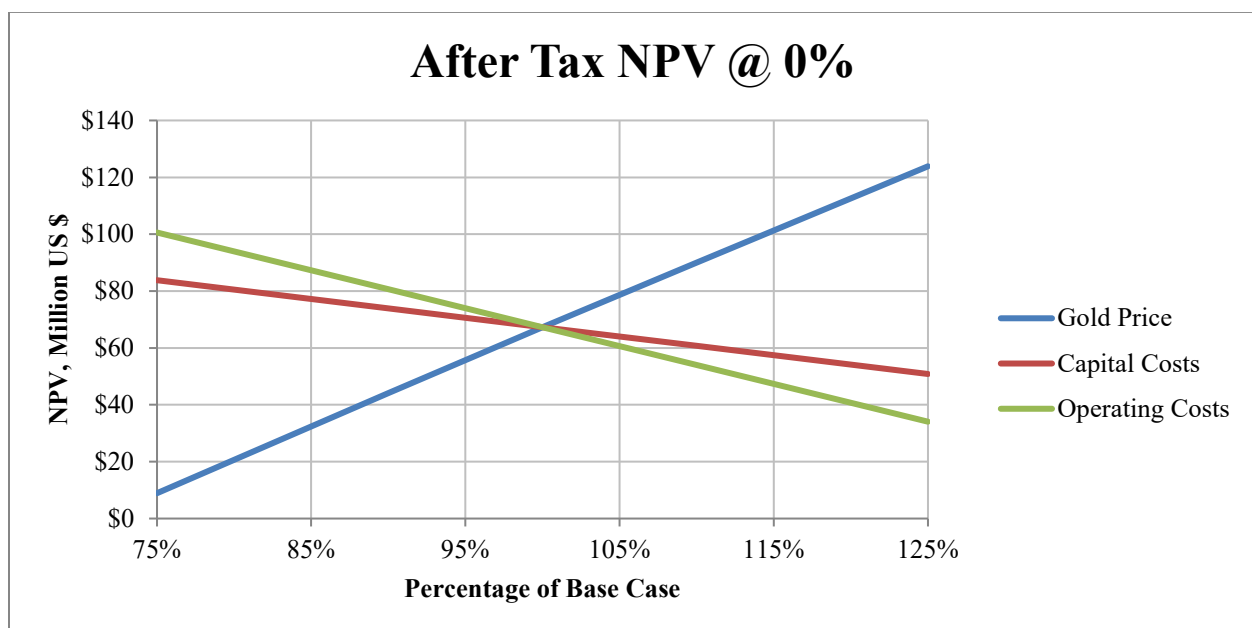


Figure 19-3
After Tax Sensitivity – NPV @ 0%

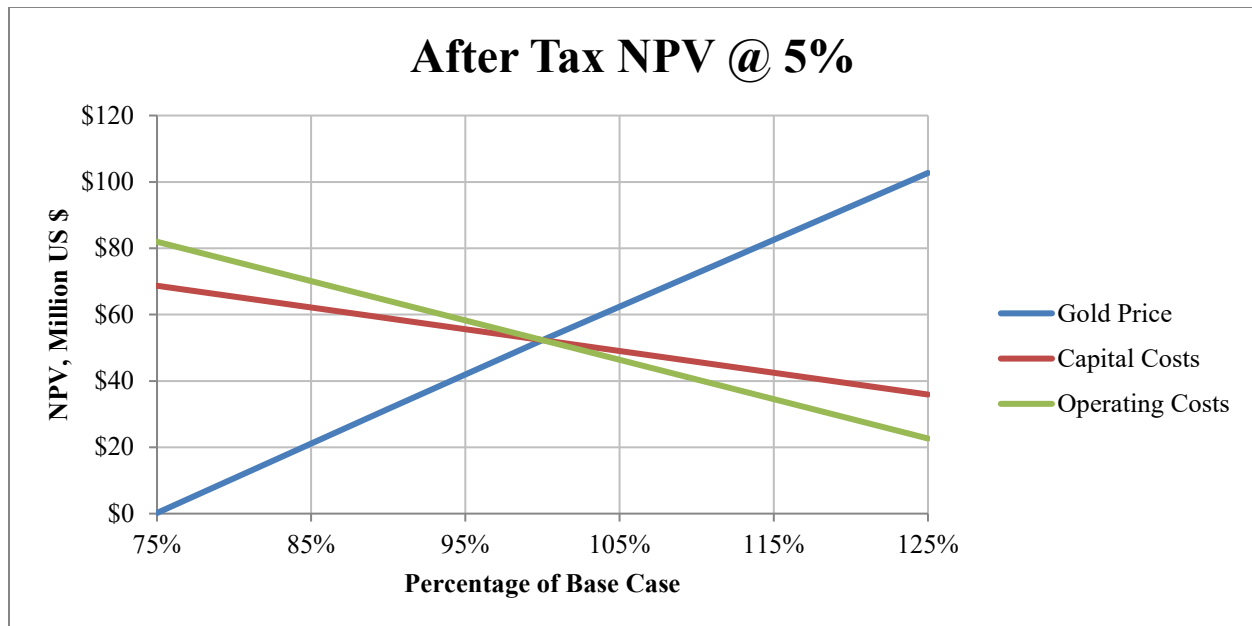


Figure 19-4
After Tax Sensitivity – NPV @ 5%

20.0 Adjacent Properties

The author is not aware of any adjacent active mining properties within a 15 mile radius of the property. The nearest producing mine is the Round Mountain Mine, some 25 miles to the northwest. Nearby (less than 30 miles) active exploration projects include the Midway (Midway Gold Corp) in Ralston Valley. There are several recently announced exploration projects in the historic Manhattan and Belmont districts, including Scorpio Gold's programs at Gold Wedge in Manhattan.

21.0 Other Relevant Data and Information

The author is not aware of any other data or information which would add materially to the understanding or authenticity of this report.

22.0 Interpretation and Conclusions

22.1 Interpretation

22.1.1 Property Description and Location

The Longstreet Project is favorably located in an established Nevada mining jurisdiction and is accessible by existing paved and gravel roads. The 142 unpatented mining claims cover approximately 2,500 acres on lands administered by the United States Forest Service (USFS) and Bureau of Land Management (BLM). The additional claims east of the deposit and along the connecting corridor provide a potential location and access route for the proposed heap-leach facilities.

Project development will require coordination with both the USFS and BLM because the Mineral Resource and principal exploration areas occur on USFS-administered land, while the proposed leach-pad area occurs on BLM-administered land. The remote location and lack of permanent site infrastructure will require development of access, water, power, communications, and support facilities. Contract mining and the regional sourcing of labor and services are reasonable assumptions for a project of this scale and reduce initial capital requirements.

22.1.2 Geology and Mineralization

The Longstreet deposit is interpreted as a volcanic-hosted, low-sulfidation epithermal gold-silver system developed principally within rhyolitic ash-flow tuffs. Mineralization is spatially associated with quartz-adularia veins, silicification, brecciation, limonitic fractures, alteration, and steeply dipping structural zones. The interpreted deposit model is consistent with the observed geological and alteration relationships and provides an appropriate basis for continued exploration and Mineral Resource estimation.

The Main Zone is controlled by multiple vein sets, including an important east-trending structure associated with the highest grades identified to date. The continuity and local distribution of high-grade mineralization remain important uncertainties because gold occurs irregularly within structurally controlled veins and breccias. Red Knob, Cyprus Ridge, Opal Ridge, Northeast Main, North Zone, Spire, North Slope, and West Main represent additional exploration targets, several of which are incompletely tested or undrilled.

22.1.3 Exploration and Drilling

The combined geological mapping, lithogeochemical sampling, reverse-circulation drilling, and diamond drilling programs have adequately defined the Main Zone for an Initial Assessment-level evaluation. The large proportion of drill holes containing mineralized intervals supports the interpretation of a laterally and vertically extensive mineralized system. However, drilling has been concentrated in the Main Zone, and significant exploration potential remains elsewhere on the property.

Historic and recent exploration results indicate that several peripheral targets contain favorable structures, alteration, and anomalous gold and silver values. These targets may provide opportunities to expand the Mineral Resource or extend the Project life but cannot be assigned economic significance without systematic drilling. Additional diamond drilling would also improve the geological interpretation and provide information for structural, density, geotechnical, and metallurgical evaluations.

22.1.4 Data Verification

The available verification work, including duplicate assays, standards, blanks, laboratory checks, database comparisons, twin drilling, and independent sampling, generally supports the use of the database for the current Mineral Resource Estimate. The more recent Star Gold analytical data are considered reliable and appear to be free of material transcription or data-transfer errors.

Confidence in portions of the historic drilling database is reduced by missing assay certificates, incomplete documentation, uncertain collar locations, and poor gold-grade correlation in certain twin-hole comparisons. These limitations are partly mitigated by the quantity and quality of the more recent Star Gold drilling. However, further verification is warranted where historic drill holes materially influence the Mineral Resource classification or proposed mine plan.

22.1.5 Mineral Processing and Metallurgical Testing

The available metallurgical test work is sufficient to support an Initial Assessment (IA) level heap-leach process concept. Column-leach tests indicate that material crushed to 80% passing $\frac{3}{4}$ " inch amenable to conventional cyanide heap leaching, with relatively rapid gold dissolution and limited additional gold extraction after approximately 30 to 40 days. The selected design recovery of 84% for gold is supported by the available column-test results and is conservative relative to the reported recovery range of approximately 87%.

Silver recovery is substantially lower and slower than gold recovery. Fine-grained silver sulfides and silver associated with jarosite limit heap-leach extraction. Consequently, the selected design

recovery of 13% for silver discounts four points from the reported recovery of the column test results. Historical testing demonstrates that finer crushing can improve gold and silver recovery, although material crushed to minus ¼ inch may require cement agglomeration.

The mineralized material is characterized as relatively soft and only slightly abrasive based on the available comminution testing. These characteristics support the proposed portable three-stage crushing circuit. Additional variability, permeability, and crush-size testing will be required before establishing final commercial design criteria.

22.1.6 Mineral Resource Estimate

The Mineral Resource Estimate, effective May 29, 2026, provides an adequate basis for the Initial Assessment (IA). At a diluted cutoff grade of 0.005 oz AuEq/t, the estimate includes approximately 6.30 million tons of Measured and Indicated Mineral Resources grading 0.016 opt Au/t and 0.419 oz Ag/t, containing approximately 103,900 oz Au and 2.64 million oz Ag. An additional 0.48 million tons of Inferred Mineral Resources grades 0.014 oz Au/t and 0.400 oz Ag/t and contains approximately 6,700 oz Au and 192,600 oz Ag.

The Mineral Resources are constrained within an optimized pit and demonstrate reasonable prospects for economic extraction under the assumptions applied in the report. Model validation indicates that the block model generally reproduces the observed gold and silver grade trends. Nevertheless, the limited understanding of high-grade structural controls and incomplete historic documentation create uncertainty in local grade and tonnage estimates. The use of a single average specific gravity and the limited diamond-core database also warrant additional investigation.

Inferred Mineral Resources are considered too geologically speculative to have economic considerations applied that would enable classification as Mineral Reserves.

22.1.7 Mining Methods

Conventional open-pit mining by a mining contractor is an appropriate IA level mining method for the near-surface deposit. The designed pit contains approximately 6.55 million tons of leach material and 5.31 million tons of waste, resulting in a life-of-mine strip ratio of approximately 0.81:1. The shallow nature of the mineralization allows mining to begin without a separate major preproduction stripping phase, which benefits the Project economics.

The mine plan incorporates Measured, Indicated, and Inferred Mineral Resources and therefore does not constitute a Mineral Reserve. Pit-slope parameters are based on comparisons with other Nevada operations rather than project-specific geotechnical investigations. In addition, the

locations and conditions of historic underground workings have not been fully defined. These matters represent important design and operating uncertainties that must be investigated before advancing the Project to a higher-confidence study.

The proposed four-year mining and heap-placement schedule is consistent with the estimated inventory and nominal processing capacity. Continued metal production after mining ceases is reasonable because of the modeled heap-leach recovery lag.

22.1.8 Processing and Recovery Methods

The selected process flowsheet, consisting of three-stage crushing, conveyor stacking, cyanide heap leaching, carbon-in-column adsorption, and off-site treatment of loaded carbon, is technically conventional and appropriate for the deposit and Project scale. The selected configuration of portable crushing, diesel-generated power, and off-site carbon treatment reduces initial infrastructure and capital requirements.

The process design is based on approximately 5,500 tons per day of heap placement, a product size of 80% passing $\frac{3}{4}$ inch, a 140-day leach cycle, 84% gold recovery, 13% silver recovery, and a nominal solution flow of 2,279 gpm. The proposed 6.5 million-ton leach pad and associated solution ponds are conceptually sized for the planned production.

Reliance on diesel generation and complete replacement of shipped loaded carbon simplifies the site configuration but increases exposure to fuel, transportation, carbon, and toll-treatment costs.

22.1.9 Capital and Operating Costs

The capital and operating cost estimates are appropriate for an Initial Assessment and were developed using vendor quotations, contractor pricing, Nevada project data, internal databases, and first-principles estimates in third-quarter 2026 U.S. dollars. The selected development strategy concentrates capital on crushing, stacking, heap-leach facilities, CIC recovery, diesel power generation, water supply, and essential site infrastructure.

The average life-of-mine operating cost is estimated at \$22.98 per ton processed, comprising approximately \$7.49/t for mining, \$13.27/t for processing and support, and \$2.22/t for general and administrative costs. Project economics are particularly exposed to diesel-generated power, toll treatment, carbon replacement, contract-mining rates, and the relatively short operating life over which fixed costs must be recovered.

22.1.10 Economic Analysis

The Initial Assessment indicates positive after-tax economics under the base-case assumptions. The cash-flow model reports an after-tax NPV at a 5% discount rate of approximately \$67.3 million, an after-tax IRR of approximately 40%, and an after-tax payback period of approximately 1.8 years. These results support continued evaluation of the Project but do not demonstrate economic viability at a Mineral Reserve level.

Project value is most sensitive to gold price, followed by operating and capital costs. The short mine life and inclusion of Inferred Mineral Resources indicate that the economic results could change materially as a result of changes in Mineral Resource classification, metallurgical recovery, operating performance, capital requirements, development schedule, or operating costs.

22.2 Conclusions

22.2.1 Property Description and Location

The Project location, access, claim position, and availability of regional mining services are adequate to support continued development evaluation. No location-related issue identified in the report precludes advancement of the Project. Federal land administration, permitting, surface-use arrangements, water availability, and access to the proposed leach-pad area remain important development considerations.

22.2.2 Geology and Mineralization

The available geological evidence supports the interpretation of Longstreet as a structurally controlled, low-sulfidation epithermal gold-silver deposit. The geological model is appropriate for the current Mineral Resource Estimate. Multiple incompletely tested targets provide meaningful opportunities to expand the known mineralized system.

22.2.3 Exploration and Drilling

Existing drilling adequately supports the Main Zone Initial Assessment. Additional drilling is required to improve the understanding of local grade continuity, convert Inferred Mineral Resources, test extensions of the Main Zone, collect technical data, and evaluate high-priority exploration targets outside the current resource area.

22.2.4 Data Verification

The QP considers the data used for the Mineral Resource Estimate adequate and reliable for the current study. Missing historic assay certificates and incomplete early-program records reduce confidence locally but do not invalidate the overall database. Future verification should focus on portions of the resource model materially supported by historic drilling.

22.2.5 Mineral Processing and Metallurgical Testing

Available testwork supports conventional heap leaching of material crushed to 80% passing $\frac{3}{4}$ inch using design recoveries of 84% for gold and 13% for silver. Gold extraction is favorable, whereas silver extraction is limited by mineralogy and liberation. Additional variability, permeability, and crush-size testing is required to support the next study stage and determine whether HPGR or finer crushing would provide an economic benefit.

22.2.6 Mineral Resource Estimate

The Mineral Resource Estimate is suitable for use in the Initial Assessment and supports the proposed scale of development. No Mineral Reserves have been estimated. Additional density data, structural interpretation, verification drilling, and infill diamond drilling are required to increase confidence in the estimate and support future conversion of Mineral Resources to Mineral Reserves.

22.2.7 Mining Methods

The proposed contractor-operated open pit is a reasonable Initial Assessment-level mining approach. The low strip ratio and near-surface mineralization are favorable characteristics. Project-specific geotechnical investigations, detailed characterization of historic underground workings, and higher-confidence resource classification are required before completing a final pit design and Mineral Reserve estimate.

22.2.8 Processing and Recovery Methods

The portable crushing, heap-leach, CIC, and off-site carbon-treatment flowsheet is technically feasible at the Initial Assessment level and is appropriate for the Project scale. The process concept remains subject to confirmation of site conditions, water supply, commercial toll-treatment arrangements, and final equipment design criteria.

22.2.9 Environmental and Permitting

Given the current information regarding this Project, it is estimated that the permitting timeline could be up to 12-14 months once the complete mine operating Plan of Operations is submitted. This timeline will likely be reduced somewhat by the modest size of the proposed operation and its relatively small impacts.

Baseline studies for flora and fauna and cultural resources have been completed with hydrogeologic, geotechnical and geochemical studies are scheduled for completion in 2026. These studies will be accomplished during the continued advanced engineering phase of the Project.

Kappes Cassiday and Associates concludes that there are no recognized potential environmental or permitting fatal or material flaws regarding this Project.

22.2.10 Capital and Operating Costs

The capital and operating cost estimates are adequate for an Initial Assessment and support comparison of the development alternatives considered. Project costs are particularly exposed to contract mining, diesel fuel, power generation, cyanide, toll treatment, transportation, and fixed G&A requirements.

22.2.11 Economic Analysis

The base-case economic analysis demonstrates potentially favorable Project economics, with an after-tax NPV5 of approximately \$67.3 million, an after-tax IRR of approximately 40%, and an after-tax payback period of approximately 1.8 years.

23.0 Recommendations

23.1 Property Description and Location

- Advance coordination with the USFS and BLM and prepare a comprehensive permitting register and schedule identifying the required federal, state, and county approvals, baseline studies, critical-path activities, and responsible parties.
- Complete updated evaluations of site access, utilities, communications, labor availability, accommodations, and construction logistics. Confirm that the eastern claims and connecting corridor provide adequate legal and physical access for construction and operation of the leach facility.

23.2 Geology and Mineralization

- Refine the three-dimensional geological and structural model, with particular emphasis on the east-trending high-grade structure, crosscutting vein sets, alteration domains, oxide-sulfide boundaries, and controls on locally high gold grades.
- Complete detailed mapping and systematic prioritization of Red Knob, Cyprus Ridge, Opal Ridge, Northeast Main, North Zone, Spire, North Slope, and West Main using consistent geological, geochemical, and geophysical criteria.

23.3 Exploration and Drilling

- Complete a phased diamond-core and RC drilling program to:
- Infill the Main Zone and convert Inferred Mineral Resources that affect the proposed mine plan.
 - Twin selected historic drill holes.
 - Test extensions of the known mineralization.
 - Evaluate the highest-priority exploration targets.
 - Obtain oriented core for structural, geotechnical, density, and metallurgical investigations.
- Survey all new drill holes using downhole survey equipment and accurately resurvey drill collars and accessible historic workings. Prioritize near-pit resource conversion and technical data collection before undertaking lower-priority regional exploration.
- Update the geological model and Mineral Resource Estimate following completion of the drilling program.

23.4 Data Verification

- Implement a written QA/QC protocol that includes certified reference materials, blanks, field duplicates, coarse duplicates, pulp duplicates, and independent laboratory checks inserted at defined frequencies in every analytical batch.
- Submit approximately 10% of original samples to an independent laboratory and conduct check assays on approximately 5% of previous assays, focusing on historic data that materially influence Mineral Resource classification or the proposed mine plan.
- Digitize, index, and preserve all available assay certificates, geological logs, survey records, and chain-of-custody documentation. Establish documented QA/QC acceptance criteria, failure investigations, and corrective-action procedures before beginning additional drilling.

23.5 Mineral Processing and Metallurgical Testing

- Conduct additional column-leach tests on spatially and geologically representative drill-core composites crushed to 80% passing $\frac{3}{4}$ inch. The test program should confirm recovery, leach kinetics, cyanide and lime consumption, and metallurgical variability.
- Complete comparative column tests using conventional and HPGR crushing at product sizes ranging from 80% passing $\frac{1}{2}$ inch to $\frac{1}{4}$ inch. Use the results to determine whether additional recovery from finer crushing would justify the corresponding capital and operating costs.
- Perform compacted permeability and load-percolation tests at the planned ultimate heap height.

23.6 Mineral Resource Estimate

- Collect additional specific-gravity measurements distributed by lithology, alteration, oxidation state, depth, and grade. Replace or validate the use of a single average specific gravity in future Mineral Resource estimates.
- Improve the interpretation of high-grade structural controls and verify the oxide-sulfide boundary. Review the influence of historic drill holes on Measured and Indicated classification and adjust the classification where verification is insufficient.
- Update the Mineral Resource model following the recommended drilling. Re-evaluate grade capping, geological domaining, search parameters, and classification criteria using the expanded database. Continue validating the model through comparison with alternative interpolation methods, swath plots, composite distributions, and visual review.

23.7 Mining Methods

- Complete project-specific geotechnical drilling, oriented-core logging, laboratory rock testing, structural analysis, hydrogeological characterization, and slope-stability analysis to establish appropriate pit-design criteria.
- Survey and investigate historic underground workings. Incorporate the results into the mine design and develop appropriate exclusion zones, probe-drilling requirements, ground-control procedures, and remediation allowances.
- Prepare an updated mine plan using the revised Mineral Resource model, detailed pit phases, haul profiles, dilution and ore-loss estimates, equipment productivities, contractor quotations, stockpiling strategy, and quarterly production requirements.

23.8 Processing and Recovery Methods

- Advance the process design through preparation of updated equipment sizing calculations; site layouts; piping and instrumentation diagrams; electrical load lists; and a process-control philosophy.
- Confirm the design storm, pond capacities, leak-detection system, liner configuration, foundation conditions, and the availability and suitability of low-permeability soil and overliner material.
- Complete water-supply drilling and pump testing. Evaluate water quality for potable use and environmental management.
- Obtain firm commercial proposals for off-site carbon treatment, refining, transportation, and security.

23.9 Capital and Operating Costs

- Update vendor and contractor quotations and develop a consistent estimate basis, scope matrix, battery limits, escalation date, contingency methodology, construction schedule, and owner-cost basis.
- Update operating costs using current quotations for contract mining, diesel fuel, power generation, toll treatment, transportation, labor, maintenance, and laboratory services.
- Evaluate opportunities to reduce operating costs through carbon reuse or regeneration, alternative crushing strategies, and optimization of fixed staffing requirements over the relatively short mine life.

23.10 Economic Analysis

Advance the Project to a prefeasibility-level study after completing the recommended drilling, resource estimation, geotechnical, metallurgical, hydrogeological, environmental, permitting, engineering, and cost-estimation work.

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25.0 Reliance on Information Provided by the Registrant

The authors relied upon a data packet provided by Star Gold and by the previous owner, MinQuest for much of the historical information in this report. Sample results and drilling data were confirmed by examining original assay certificates or reports whenever available. Drill logs, drill assays and all other parameters utilized for the resource estimations were verified independently by examination of original data whenever possible. The author believes that all information available and utilized for this report is reliable and valid.

26.0 Date and Signature Page

This technical report summary ("TRS") entitled "Initial Assessment, SK 1300 Technical Report on the Longstreet Gold & Silver Project, Nye County, Nevada, USA" is current as of July 31, 2026 and has been prepared by:

Qualified Person	Responsible for the Following Sections	Signature	Date
Travis J. Manning.	Responsible for Sections 2, 3, 4, 10, 14, 15, 16, 19 and relevant portions of Sections, 1, 21, 22, 23, 24 and 25.		
James W. Ashton	Responsible for Sections 7, 8, 9, 11, 13 and relevant portions of Sections 1, 2, 18, 22 and 23.		
Reinis N. Sipols	Responsible for Section 17 and relevant portions of Sections 1 and 22		
Richard Kern	Responsible for Sections 5, 6, 20 and relevant portions of Sections 1, 22 and 23		

27.0 Appendices

27.1 Appendix 1 – Claims List

27.2 Appendix 2 – Drill Holes Collars and Assays Highlights

27.3 Appendix 3 – McClelland Report 2014

Appendix 1. Claims List

TonopahOffice
101RadarRoad
POBox1111
Tonopah,NV89049
Phone(775)482-8116
Fax(775)482-8111

Office Of Nye County Recorder
Deborah Beatty - Recorder

PahrumpOffice
170 NFloyd St Ste 1
Pahrump,NV89060
Phone(775)751-6340
Fax(775)751-6341

Receipt2025-07845

Product	Name	Extended
NOI	Notice of Intent to Hold	\$1,716.00
Document # 1056349, Document Info: GREAT BASIN RESOURCES, Document Info: GREAT BASIN RESOURCES, Claims 142		
Total		\$1,716.00
Tender (Check)		\$1,716.00
Name GREAT BASIN RESOURCES, Check # 1496, Name GREAT BASIN RESOURCES		

I hereby affirm that this document submitted for recording does not contain a social security number.

Signed

Richard R. Kern Agent

DOC # 1056349

Official Records Nye County Nevada
Deborah Beatty - Recorder
08/26/2025 01:53:37 PM
Requested By: GREAT BASIN RESOURCES
Recorded By: vu RPTT:\$0.00
Recording Fee: \$1,716.00
Page 1 of 5



AFTER FILING/RECORDING,
PLEASE RETURN TO:
Great Basin Resources Inc.
5560 Rue St Tropez
Reno, NV 89511

Nye County Recorder
101 Radar Road
P.O. Box 1111
Tonopah, NV 89049

**AFFIDAVIT OF PAYMENT OF CLAIM MAINTENANCE AND NOTICE OF INTENTION TO
HOLD UNPATENTED MINING CLAIMS**

The undersigned being first duly sworn deposes and says:

Great Basin Resources Inc., a Nevada corporation, with offices located at 5560 Rue St Tropez, Reno, NV, 89511, is the owner or authorized agent on behalf of the owner of each of the claims described on Exhibit A attached hereto and incorporated herein ("the Claims").

Great Basin Resources Inc., as owner or as the authorized agent on behalf of the owner of the Claims, intends to hold said Claims from September 2, 2025 until September 2, 2026.

Great Basin Resources Inc., as owner or as the authorized agent on behalf of the owner of the Claims, has paid to the U.S. Department of the Interior - Bureau of Land Management, on or before September 1st, 2025, the annual maintenance fees for the assessment year beginning September 2, 2025 as prescribed in the 1993 Omnibus Budget Reconciliation Act of 1993, Public Law 103-66, for the Claims.

This Affidavit of Payment of Claim Maintenance Fees and Notice of Intention to Hold Unpatented Mining Claims is made and recorded pursuant to NRS 517.230(3).

The recording fee due the Nye County Recorder has been calculated by multiplying each of the 142 claims by \$12.00 plus \$12.00 per document for a total amount due the Nye County Recorder, enclosed herewith, of \$1,716.00.

EXHIBIT A

NYE COUNTY

<u>CLAIM NAME</u>		<u>CLAIMANT'S NAME</u>	<u>NMC NUMBER</u>
Morning Star		Roy Clifford et al	96719
Longstreet	11	Roy Clifford et al	164002
Longstreet	12	Roy Clifford et al	164003
Longstreet	14	Roy Clifford et al	164005
Longstreet	15	Roy Clifford et al	164006
Longstreet	1 A	MinQuest Inc.	799562
Longstreet	2 A	MinQuest Inc.	799563
Longstreet	3 A	MinQuest Inc.	799564
Longstreet	4 A	MinQuest Inc.	836168
Longstreet	5 A	MinQuest Inc.	836169
Longstreet	6 A	MinQuest Inc.	799565
Longstreet	7 A	MinQuest Inc.	799566
Longstreet	8 A	MinQuest Inc.	799567
Longstreet	8	MinQuest Inc.	836170
Longstreet	9 A	MinQuest Inc.	799568
Longstreet	10	MinQuest Inc.	836171
Longstreet	10 A	MinQuest Inc.	836172
Longstreet	12	MinQuest Inc.	843867
Longstreet	13	MinQuest Inc.	799570
Longstreet	14	MinQuest Inc.	843868
Longstreet	16 A	MinQuest Inc.	799569
Longstreet	16	MinQuest Inc.	843869
Longstreet	18	MinQuest Inc.	843870
Longstreet	20	MinQuest Inc.	843871
Longstreet	26	MinQuest Inc.	843872
Longstreet	28	MinQuest Inc.	836173
Longstreet	30	MinQuest Inc.	836174
Longstreet	32	MinQuest Inc.	799571
Longstreet	34	MinQuest Inc.	799572
Longstreet	36	MinQuest Inc.	836175
Longstreet	37	MinQuest Inc.	836176
Longstreet	39	MinQuest Inc.	836177
Longstreet	40	MinQuest Inc.	851568
Longstreet	41	MinQuest Inc.	836178
Longstreet	42	MinQuest Inc.	843873
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Longstreet	45	MinQuest Inc.	836180

Longstreet	46	MinQuest Inc.	843875
Longstreet	47	MinQuest Inc.	836181
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Longstreet	50	MinQuest Inc.	843877
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Longstreet	107	MinQuest Inc.	836189
Longstreet	108	MinQuest Inc.	836190
Longstreet	109	MinQuest Inc.	855021
Longstreet	110	MinQuest Inc.	855022
Longstreet	111	MinQuest Inc.	855023
Longstreet	112	MinQuest Inc.	855024
Longstreet	113	MinQuest Inc.	855025
Longstreet	114	MinQuest Inc.	855026
Longstreet	115	MinQuest Inc.	855027
Longstreet	118	MinQuest Inc.	851569
Longstreet	119	MinQuest Inc.	851570
Longstreet	120	MinQuest Inc.	851571
Longstreet	121	MinQuest Inc.	851572
Longstreet	122	MinQuest Inc.	851573
Longstreet	123	MinQuest Inc.	851574
Longstreet	124	MinQuest Inc.	851575
Longstreet	200	MinQuest Inc.	1073640
Longstreet	201	MinQuest Inc.	1073641
Longstreet	202	MinQuest Inc.	1073642
Longstreet	203	MinQuest Inc.	1073643
Longstreet	204	MinQuest Inc.	1073644
Longstreet	205	MinQuest Inc.	1073645

Longstreet	206	MinQuest Inc.	1073646
Longstreet	207	MinQuest Inc.	1073647
Longstreet	208	MinQuest Inc.	1073648
Longstreet	209	MinQuest Inc.	1073649
Longstreet	210	MinQuest Inc.	1073650
Longstreet	211	MinQuest Inc.	1073651
Longstreet	212	MinQuest Inc.	1073652
Longstreet	213	MinQuest Inc.	1073653
Longstreet	214	MinQuest Inc.	1073654
Longstreet	215	MinQuest Inc.	1073655
Longstreet	216	MinQuest Inc.	1073656
Longstreet	217	MinQuest Inc.	1073657
Longstreet	218	MinQuest Inc.	1073658
Longstreet	219	MinQuest Inc.	1073659
Longstreet	220	MinQuest Inc.	1073660
Longstreet	221	MinQuest Inc.	1073661
Longstreet	222	MinQuest Inc.	1073662
Longstreet	223	MinQuest Inc.	1073663
Longstreet	224	MinQuest Inc.	1073664
Longstreet	225	MinQuest Inc.	1073665
Longstreet	226	MinQuest Inc.	1073666
Longstreet	227	MinQuest Inc.	1073667
Longstreet	228	MinQuest Inc.	1073668
Longstreet	229	MinQuest Inc.	1073669
Longstreet	230	MinQuest Inc.	1073670
Longstreet	231	MinQuest Inc.	1073671
Longstreet	232	MinQuest Inc.	1073672
Longstreet	233	MinQuest Inc.	1073673
Longstreet	234	MinQuest Inc.	1073674
Longstreet	235	MinQuest Inc.	1073675
Longstreet	236	MinQuest Inc.	1073676
Longstreet	237	MinQuest Inc.	1073677
Longstreet	66	MinQuest Inc.	1080730
Longstreet	238	MinQuest Inc.	1080731
Longstreet	239	MinQuest Inc.	1080732
Longstreet	240	MinQuest Inc.	1080733
Longstreet	241	MinQuest Inc.	1080734
Longstreet	242	MinQuest Inc.	1080735
Longstreet	243	MinQuest Inc.	1080736
Longstreet	244	MinQuest Inc.	1080737
Longstreet	245	MinQuest Inc.	1080738
Longstreet	246	MinQuest Inc.	1080739
Longstreet	247	MinQuest Inc.	1080740

Longstreet	248	MinQuest Inc.	1080741
Longstreet	301	MinQuest Inc.	1116062
Longstreet	302	MinQuest Inc.	1116063
Longstreet	303	MinQuest Inc.	1116064
Longstreet	304	MinQuest Inc.	1116065
Longstreet	305	MinQuest Inc.	1116066
Longstreet	306	MinQuest Inc.	1116067
Longstreet	307	MinQuest Inc.	1116068
Longstreet	308	MinQuest Inc.	1116069
Longstreet	309	MinQuest Inc.	1116070
Longstreet	310	MinQuest Inc.	1116071
Longstreet	311	MinQuest Inc.	1116072
Longstreet	312	MinQuest Inc.	1116073
Longstreet	313	MinQuest Inc.	1116074
Longstreet	314	MinQuest Inc.	1116075
Longstreet	315	MinQuest Inc.	1116076
Longstreet	316	MinQuest Inc.	1116077
Longstreet	317	MinQuest Inc.	1116078

Total Claims

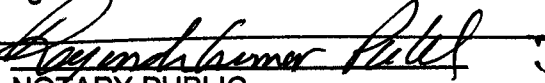
142

Dated and signed this 4 day of August, 2025
For Great Basin Resources Inc.

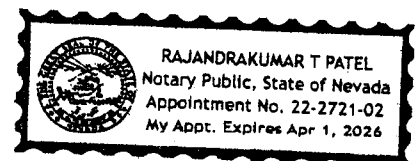

Richard R. Kern, Agent

STATE OF NEVADA)
) SS.
COUNTY OF WASHOE)

Signed and sworn to before me this 4 day of August, 2025, by Richard R. Kern, as
agent for Great Basin Resources Inc.


NOTARY PUBLIC

My Commission Expires: 04/01/2026 [SEAL]
Notary Public in and for Washoe Country, Nevada



Appendix 2. Drill Hole Collars and Assay Highlights



Summary of Significant Mineralized Intersections in Reverse Circulation Drill Holes

**Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada**

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton Au	oz/ton Ag
								From	To			
LRH-1	1984	525225E	4247433N	7,613	0	-90	115	0	115	115	0.055	0.62
LRH-2	1984	525249E	4247449N	7,620	0	-90	170	10	170	160	0.058	0.81
LRH-3	1984	525278E	4247452N	7,626	0	-90	180	0	180	180	0.027	0.54
LRH-4	1984	525355E	4247420E	7,638	0	-90	75	0	75	75	0.035	0.90
LRH-5	1984	525323E	4247440N	7,637	0	-90	200	0	200	200	0.035	0.46
LRH-6	1984	525373E	4247425N	7,612	0	-90	40	0	40	40	0.032	0.62
LRH-7	1984	525353E	4247447N	7,609	0	-90	140	0	140	140	0.019	0.35
LRH-8	1984	525328E	4247458N	7,608	0	-90	215	0	215	215	0.016	0.52
LRH-9	1984	525388E	4247446N	7,575	0	-90	70	0	70	70	0.024	0.75
LRH-10	1984	525367E	4247487N	7,545	0	-90	200	0	200	200	0.017	0.64
LRH-11	1984	525333E	4247502N	7,535	0	-90	245	0	30	30	0.010	0.10
LRH-11	1984	525333E	4247502N	7,535	0	-90	245	160	245	85	0.013	0.42
LRH-12	1984	525195E	4247454N	7,543	0	-90	100	0	100	100	0.020	0.15
LRH-13	1984	525197E	4247479N	7,526	0	-90	105	15	105	90	0.019	0.54
LRH-14	1984	525206E	4247492N	7,526	0	-90	180	10	70	60	0.015	0.31
LRH-14	1984	525206E	4247492N	7,526	0	-90	180	110	165	55	0.019	0.34
LRH-15	1984	525259E	4247493E	7,554	0	-90	235	0	225	225	0.020	0.33
LRH-16	1984	525295E	4247504N	7,736	0	-90	285	0	10	10	0.006	0.07
LRH-16	1984	525295E	4247504N	7,736	0	-90	285	135	285	150	0.012	0.52
LRH-17	1984	525398E	4247514N	7,484	0	-90	240	5	50	45	0.005	0.20
LRH-17	1984	525398E	4247514N	7,484	0	-90	204	120	190	70	0.006	0.48
LRH-17	1984	525398E	4247514N	7,484	0	-90	240	205	225	20	0.010	0.81
LRH-18	1984	525337E	4247431N	7,642	0	-90	115	0	115	115	0.011	0.34
LRH-19	1984	525304E	4247449N	7,630	0	-90	115	0	115	115	0.033	0.77

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-20	1984	525265E	4247452N	7,623	0	-90	200	0	200	200	0.024	0.41
LRH-21	1984	525364E	4247436N	7,612	0	-90	100	0	55	55	0.028	0.62
LRH-22	1984	525481E	4247555N	7,359	0	-90	280	0	35	35	0.014	0.09
LRH-22	1984	525481E	4247555N	7,359	0	-90	280	240	250	10	0.011	1.74
LRH-23	1984	525445E	4247528N	7,427	0	-90	245	No significant values				
LRH-24	1984	525341E	4247663N	7,274	0	-90	105	0	15	15	0.011	0.38
LRH-25	1984	525287E	4247378N	7,721	0	-90	145	0	145	145	0.007	0.18
LRH-26	1984	525315E	4247373N	7,722	0	-90	175	0	15	15	0.009	0.30
LRH-26	1984	525315E	4247373N	7,722	0	-90	175	50	175	125	0.008	0.58
LRH-27	1984	525131E	4247524N	7,395	0	-90	185	0	155	155	0.008	0.09
LRH-28	1984	525146E	4247541N	7,403	0	-90	145	0	105	105	0.009	0.25
LRH-28	1984	525146E	4247541N	7,403	0	-90	145	115	130	15	0.006	0.11
LRH-29	1984	525266E	4247565N	7,446	0	-90	365	0	85	85	0.006	0.14
LRH-29	1984	525266E	4247565N	7,446	0	-90	365	210	315	105	0.020	0.49
LRH-30	1984	525293E	4247567N	7,439	0	-90	165	0	30	30	0.013	0.25
LRH-31	1984	525404E	4247493N	7,504	0	-90	180	100	165	65	0.012	0.45
LRH-32	1984	525367E	4247518N	7,497	0	-90	300	0	20	20	0.007	0.18
LRH-32	1984	525367E	4247518N	7,497	0	-90	300	185	195	10	0.011	0.17
LRH-33	1984	525170E	4247494N	7,474	0	-90	200	0	130	130	0.011	0.24
LRH-33	1984	525170E	4247494N	7,474	0	-90	200	180	200	20	0.007	0.07
LRH-34	1984	525190E	4247525N	7,471	0	-90	245	0	215	215	0.016	0.27
LRH-35	1984	525220E	4247534N	7,483	0	-90	245	20	95	75	0.006	0.20
LRH-35	1984	525220E	4247534N	7,483	0	-90	245	110	120	10	0.006	0.04
LRH-35	1984	525220E	4247534N	7,483	0	-90	245	135	245	110	0.023	0.37
LRH-36	1984	525275E	4247535N	7,489	0	-90	165	0	20	20	0.019	0.43
LRH-37	1984	525515E	4247433N	7,401	0	-90	180	0	20	20	0.008	0.26
LRH-37	1984	525515E	4247433N	7,401	0	-90	180	50	110	55	0.007	0.19
LRH-38	1984	525435E	4247542N	7,422	0	-90	100	No significant values				
LRH-39	1984	525417E	4247562N	7,410	0	-90	165	No significant values				

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton Au	oz/ton Ag
LRH-40	1984	525486E	4247081N	7,710	0	-90	250	No significant values				
LRH-41	1984	525396E	4247073N	7,718	0	-90	250	20	80	60	0.005	0.01
LRH-41	1984	525396E	4247073N	7,718	0	-90	250	130	160	30	0.005	0.07
LRH-41	1984	525396E	4247073N	7,718	0	-90	250	190	220	30	0.005	0.03
LRH-42	1984	525254E	4247273N	7,765	0	-90	160	60	80	20	0.005	0.06
LRH-42	1984	525254E	4247273N	7,765	0	-90	160	130	155	25	0.006	0.10
LRH-43	1984	525330E	4247318N	7,763	0	-90	160	25	140	115	0.010	0.21
LRH-44	1984	525414E	4247232N	7,727	0	-90	100	15	30	15	0.004	0.007
LRH-44	1984	525414E	4247232N	7,727	0	-90	100	85	95	10	0.007	0.01
LRH-45	1984	525328E	4247415N	7,668	0	-90	60	0	60	60	0.023	0.71
LRH-46	1984	525208E	4247302N	7,681	0	-90	155	0	20	20	0.007	0.20
LRH-46	1984	525208E	4247302N	7,681	0	-90	155	90	140	50	0.023	0.17
LRH-47	1984	525218E	4247415N	7,618	0	-90	250	0	115	115	0.011	0.17
LRH-47	1984	525218E	4247415N	7,618	0	-90	250	195	230	35	0.007	0.36
LRH-48	1984	525241E	4247412N	7,651	0	-90	235	0	95	95	0.020	0.36
LRH-48	1984	525241E	4247412N	7,651	0	-90	235	155	200	45	0.009	0.19
LRH-49	1984	525310E	4247421N	7,667	0	-90	255	0	175	175	0.016	0.52
LRH-49	1984	525310E	4247421N	7,667	0	-90	255	240	255	15	0.005	0.19
LRH-50	1984	525362E	4247439N	7,610	0	-90	150	0	65	65	0.009	2.64
LRH-51	1984	525236E	4247444N	7,615	0	-90	300	0	250	250	0.018	0.29
LRH-52	1984	525131E	4247319N	7,531	0	-90	150	No significant values				
LRH-53	1984	525226E	4247433N	7,614	0	-90	200	0	120	120	0.027	0.44
LRH-53	1984	525226E	4247433N	7,614	0	-90	200	175	185	10	0.005	0.31
LRH-54	1984	525321E	4247441N	7,637	0	-90	180	0	125	125	0.011	0.58
LRH-55	1984	525366E	4247402N	7,641	0	-90	150	0	20	20	0.007	0.34
LRH-56	1984	525248E	4247427N	7,644	0	-90	200	5	30	25	0.010	0.18
LRH-56	1984	525248E	4247427N	7,644	0	-90	200	65	120	55	0.029	0.41
LRH-57	1984	525233E	4247391N	7,660	0	-90	150	0	105	105	0.012	0.15
LRH-57	1984	525233E	4247391N	7,660	0	-90	150	135	150	15	0.013	0.24

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-83	1984	525339E	4247409N	7,666	0	-90	200	0	180	180	0.010	0.41
LRH-84	1984	525328E	4247362N	7,726	0	-90	50	0	40	40	0.009	0.16
LRH-85	1984	525328E	4247360N	7,729	0	-90	150	0	20	20	0.012	0.28
LRH-85	1984	525328E	4247360N	7,729	0	-90	150	50	60	10	0.007	0.35
LRH-85	1984	525328E	4247360N	7,729	0	-90	150	85	100	15	0.008	0.49
LRH-86	1984	525301E	4247376N	7,725	0	-90	200	0	35	35	0.011	0.68
LRH-86	1984	525301E	4247376N	7,725	0	-90	200	90	100	10	0.005	0.41
LRH-86	1984	525301E	4247376N	7,725	0	-90	200	165	200	35	0.006	0.46
LRH-87	1984	525327E	4247267N	7,771	0	-90	100	10	55	45	0.013	0.28
LRH-88	1984	525329E	4247283N	7,766	0	-90	100	20	100	80	0.028	0.54
LRH-89	1984	525320E	4247338N	7,765	0	-90	150	0	90	80	0.004	0.18
LRH-89	1984	525320E	4247338N	7,765	0	-90	150	140	150	10	0.004	0.20
LRH-90	1984	525292E	4247348N	7,758	0	-90	180	0	105	105	0.019	0.29
LRH-90	1984	525292E	4247348N	7,758	0	-90	180	140	170	30	0.052	0.85
LRH-91	1984	525281E	4247336N	7,754	0	-90	150	0	150	150	0.025	0.75
LRH-92	1984	525272E	4247320N	7,751	0	-90	150	0	110	110	0.010	0.31
LRH-93	1984	525252E	4247301N	7,741	0	-90	150	40	55	15	0.006	0.09
LRH-93	1984	525252E	4247301N	7,741	0	-90	150	100	130	30	0.015	0.24
LRH-94	1984	525240E	4247287N	7,733	0	-90	100	35	50	15	0.024	0.07
LRH-95	1984	525270E	4247284N	7,780	0	-90	140	0	15	15	0.004	0.11
LRH-95	1984	525270E	4247284N	7,780	0	-90	140	55	70	15	0.010	0.15
LRH-95	1984	525270E	4247284N	7,780	0	-90	140	100	115	15	0.005	0.47
LRH-96	1984	525279E	4247248N	7,806	0	-90	150	10	40	30	0.006	0.09
LRH-97	1984	525271E	4247233N	7,808	0	-90	145	30	60	30	0.02	0.32
LRH-98	1984	525291E	4247230N	7,804	0	-90	160	50	70	20	0.043	0.42
LRH-98	1984	525291E	4247230N	7,804	0	-90	160	110	125	15	0.013	0.04
LRH-99	1984	525301E	4247194N	7,806	0	-90	150	90	100	10	0.006	0.04
LRH-99	1984	525301E	4247194N	7,806	0	-90	150	140	150	10	0.005	0.02
LRH-100	1984	525307E	4247278N	7,793	0	-90	150	0	150	150	0.009	0.18

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-101	1984	525306E	4247259N	7,795	0	-90	150	0	150	150	0.006	0.19
LRH-102	1984	525314E	4247235N	7,788	0	-90	160	0	160	160	0.008	0.29
LRH-103	1984	525273E	4247303N	7,766	0	-90	150	0	150	150	0.005	0.28
LRH-104	1984	525306E	4247350N	7,759	0	-90	150	0	25	25	0.010	0.31
LRH-104	1984	525306E	4247350N	7,759	0	-90	150	70	80	10	0.006	0.67
LRH-104	1984	525306E	4247350N	7,759	0	-90	150	90	105	15	0.005	0.29
LRH-104	1984	525306E	4247350N	7,759	0	-90	150	130	150	20	0.006	0.70
LRH-105	1984	525329E	4247326N	7,762	0	-90	150	85	150	65	0.003	0.15
LRH-106	1984	525330E	4247297N	7,766	0	-90	150	15	150	135	0.024	0.57
LRH-107	1984	525274E	4247376N	7,716	0	-90	200	0	200	200	0.009	0.32
LRH-108	1984	525263E	4247363N	7,718	0	-90	150	0	130	130	0.033	0.25
LRH-109	1984	525254E	4247348N	7,717	0	-90	150	0	45	45	0.013	0.34
LRH-110	1984	525241E	4247330N	7,710	0	-90	160	0	80	80	0.008	0.22
LRH-111	1984	525230E	4247310N	7,706	0	-90	150	No significant values				
LRH-112	1984	525226E	4247302N	7,705	0	-90	60	0	35	35	0.004	0.67
LRH-113	1984	525221E	4247290N	7,706	0	-90	100	0	75	75	0.008	0.20
LRH-114	1984	525214E	4247267N	7,711	0	-90	100	65	85	20	0.018	0.06
LRH-115	1984	525211E	4247258N	7,714	0	-90	100	75	100	25	0.012	0.06
LRH-116	1984	525200E	4247244N	7,709	0	-90	60	45	55	10	0.006	0.06
LRH-117	1984	525266E	4247426N	7,657	0	-90	200	0	130	130	0.012	0.35
LRH-118	1984	525322E	4247402N	7,686	0	-90	150	0	55	55	0.017	0.71
LRH-119	1984	525342E	4247393N	7,680	0	-90	150	45	65	20	0.007	0.38
LRH-119	1984	525342E	4247393N	7,680	0	-90	150	100	120	20	0.005	0.85
LRH-120	1984	525164E	4247414N	7,524	0	-90	150	0	15	15	0.010	0.17
LRH-120	1984	525164E	4247414N	7,524	0	-90	150	60	70	10	0.007	0.13
LRH-121	1984	525175E	4247442N	7,521	0	-90	150	0	80	80	0.008	0.12
LRH-121	1984	525175E	4247442N	7,521	0	-90	150	130	150	20	0.009	0.78
LRH-122	1984	525193E	4247470N	7,526	0	-90	150	0	150	150	0.011	0.33
LRH-123	1984	525201E	4247487N	7,524	0	-90	200	0	200	200	0.018	0.23

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-124	1984	525159E	4247476N	7,472	0	-90	200	0	160	160	0.008	0.61
LRH-125	1984	525175E	4247506N	7,472	0	-90	260	0	155	155	0.007	0.65
LRH-126	1984	525202E	4247530N	7,476	0	-90	260	60	205	145	0.053	0.70
LRH-127	1984	525346E	4247334N	7,725	0	-90	160	145	160	15	0.007	0.26
LRH-128	1984	525365E	4247284N	7,715	0	-90	160	No significant values				
LRH-129	1984	525355E	4247310N	7,720	0	-90	240	65	100	35	0.007	0.17
LRH-129	1984	525355E	4247310N	7,720	0	-90	240	170	210	40	0.009	0.55
LRH-130	1984	525377E	4247303N	7,688	0	-90	360	185	240	55	0.006	1.03
LRH-131	1984	525377E	4247331N	7,674	0	-90	260	165	180	15	0.008	0.19
LRH-131	1984	525377E	4247331N	7,674	0	-90	260	245	260	15	0.011	2.29
LRH-132	1984	525374E	4247361N	7,657	0	-90	260	No significant values				
LRH-133	1984	525369E	4247385N	7,649	0	-90	200	0	20	20	0.006	0.24
LRH-133	1984	525369E	4247385N	7,649	0	-90	200	190	200	10	0.013	0.24
LRH-134	1984	525550E	4247295N	7,560	0	-90	125	No significant values				
LRH-135	1984	525476E	4247303N	7,592	0	-90	50	No significant values				
LRH-136	1984	525407E	4247349N	7,605	0	-90	200	No significant values				
LRH-137	1984	525349E	4247498N	7,537	0	-90	240	95	215	120	0.008	0.46
LRH-138	1984	525375E	4247481N	7,548	0	-90	180	0	15	15	0.006	0.12
LRH-138	1984	525375E	4247481N	7,548	0	-90	180	95	160	55	0.012	0.93
LRH-139	1984	525377E	4247460N	7,573	0	-90	150	0	85	85	0.011	0.37
LRH-140	1984	525316E	4247503N	7,536	0	-90	280	0	30	30	0.025	0.85
LRH-140	1984	525316E	4247503N	7,536	0	-90	280	110	165	55	0.005	0.80
LRH-140	1984	525316E	4247503N	7,536	0	-90	280	190	270	80	0.027	2.00
LRH-141	1984	525378E	4247131N	7,774	0	-90	140	60	75	15	0.004	0.03
LRH-141	1984	525378E	4247131N	7,774	0	-90	140	90	115	25	0.009	0.09
LRH-141	1984	525378E	4247131N	7,774	0	-90	140	130	140	10	0.005	0.06
LRH-142	1984	525352E	4247241N	7,764	0	-90	150	135	150	15	0.029	0.17
LRH-143	1984	525336E	4247253N	7,767	0	-90	150	115	145	30	0.020	0.05
LRH-144	1984	525244E	4247494N	7,550	0	-90	270	0	25	25	0.009	0.18

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-144	1984	525244E	4247494N	7,550	0	-90	270	115	210	95	0.019	0.30
LRH-145	1984	525223E	4247493N	7,540	0	-90	220	0	185	185	0.022	0.42
LRH-145	1984	525223E	4247493N	7,540	0	-90	220	210	220	10	0.016	0.16
LRH-146	1984	525431E	4247434N	7,514	0	-90	100				No significant values	
LRH-147	1984	525457E	4247397N	7,494	0	-90	100				No significant values	
LRH-148	1984	525482E	4247454N	7,415	0	-90	100				No significant values	
LRH-149	1984	525507E	4247426N	7,414	0	-90	100				No significant values	
LRH-150	1984	525553E	4247409N	7,426	0	-90	100				No significant values	
LRH-151	1984	525591E	4247522N	7,276	0	-90	100				No significant values	
LRH-152	1984	525246E	4247730N	7,250	0	-90	100				No significant values	
LRH-153	1984	525288E	4247303N	7,784	195	-60	200	0	95	95	0.014	0.64
LRH-153	1984	525288E	4247303N	7,784	195	-60	200	160	200	40	0.009	0.11
LRH-154	1984	525284E	4247277N	7,797	0	-90	160	0	60	60	0.014	0.74
LRH-155	1984	525288E	4247234N	7,806	0	-90	100	5	35	30	0.020	0.24
LRH-156	1984	525298E	4247320N	7,782	225	-60	200	0	200	200	0.030	0.44
LRH-157	1984	525280E	4247316N	7,762	250	-60	200	0	200	200	0.028	0.49
LRH-158	1984	525316E	4247297N	7,790	202	-60	160	0	145	145	0.013	0.23
LRH-159	1984	525284E	4247282N	7,797	225	-60	150	0	40	40	0.011	0.68
LRH-159	1984	525284E	4247282N	7,797	225	-60	150	60	75	15	0.004	0.07
LRH-160	1984	525277E	4247327N	7,754	207	-60	150	0	35	35	0.014	0.40
LRH-160	1984	525277E	4247327N	7,754	207	-60	150	75	140	65	0.006	0.32
LRH-161	1984	525283E	4247340N	7,754	220	-60	140	0	120	120	0.025	0.31
LRH-162	1984	525289E	4247348N	7,755	255	-60	135	0	135	135	0.040	0.45
LRH-163	1984	525305E	4247349N	7,761	296	-60	150	0	30	30	0.019	0.27
LRH-163	1984	525305E	4247349N	7,761	296	-60	150	65	135	70	0.016	0.18
LRH-164	1984	525269E	4247316N	7,750	220	-60	150	0	55	55	0.008	0.49
LRH-164	1984	525269E	4247316N	7,750	220	-60	150	95	130	35	0.004	0.16
LRH-165	1984	525305E	4247281N	7,797	228	-60	150	15	115	100	0.015	0.30
LRH-166	1984	525302E	4247265N	7,797	222	-60	150	0	55	55	0.011	0.38

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-166	1984	525302E	4247265N	7,797	222	-60	150	95	130	35	0.008	0.15
LRH-167	1984	525309E	4247235N	7,792	233	-60	150	80	110	30	0.116	0.85
LRH-168	1984	525295E	4247245N	7,803	213	-60	150	50	65	15	0.006	0.10
LRH-168	1984	525295E	4247245N	7,803	213	-60	150	115	130	15	0.018	0.05
LRH-169	1984	525312E	4247221N	7,792	242	-60	150	90	100	10	0.052	0.56
LRH-170	1984	525277E	4247242N	7,807	222	-60	150	45	65	20	0.008	0.15
LRH-170	1984	525277E	4247242N	7,807	222	-60	150	140	150	10	0.006	0.25
LRH-171	1984	525268E	4247257N	7,793	0	-90	100	60	90	30	0.009	0.06
LRH-172	1984	525318E	4247340N	7,765	245	-60	140	0	30	30	0.014	0.37
LRH-172	1984	525318E	4247340N	7,765	245	-60	140	60	120	60	0.010	0.35
LRH-173	1984	525280E	4247219N	7,810	0	-90	120	70	85	15	0.046	0.08
LRH-174	1984	525341E	4247259N	7,759	237	-60	150	35	150	115	0.013	0.15
LRH-175	1984	525322E	4247366N	7,728	259	-60	150	0	35	35	0.008	0.15
LRH-176	1984	525296E	4247375N	7,727	260	-60	200	75	200	125	0.012	0.29
LRH-177	1984	525283E	4247376N	7,722	254	-60	150	0	150	150	0.009	0.21
LRH-178	1984	525264E	4247363N	7,719	210	-60	160	0	75	75	0.025	0.30
LRH-178	1984	525264E	4247363N	7,719	210	-60	160	150	160	10	0.006	0.34
LRH-179	1984	525251E	4247344N	7,716	214	-60	160	0	60	60	0.018	0.34
LRH-179	1984	525251E	4247344N	7,716	214	-60	160	75	85	10	0.008	0.24
LRH-179	1984	525251E	4247344N	7,716	214	-60	160	135	145	10	0.009	0.19
LRH-180	1984	525274E	4247402N	7,690	261	-60	160	0	160	160	0.015	0.39
LRH-181	1984	525261E	4247395N	7,688	238	-60	140	0	100	100	0.007	0.26
LRH-182	1984	525246E	4247384N	7,683	228	-60	160	0	160	160	0.008	0.14
LRH-183	1984	525237E	4247369N	7,682	217	-60	160	0	105	105	0.014	0.15
LRH-184	1984	525226E	4247354N	7,677	203	-60	160	0	15	15	0.006	0.12
LRH-184	1984	525226E	4247354N	7,677	203	-60	160	85	100	15	0.013	0.11
LRH-184	1984	525226E	4247354N	7,677	203	-60	160	150	160	10	0.010	0.17
LRH-185	1984	525219E	4247340N	7,673	205	-60	160	15	65	50	0.007	0.26
LRH-185	1984	525219E	4247340N	7,673	205	-60	160	140	160	20	0.009	0.06

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-186	1984	525214E	4247328N	7,674	199	-60	100	0	60	60	0.007	0.18
LRH-187	1984	525232E	4247360N	7,681	207	-60	100	0	50	50	0.006	0.09
LRH-188	1984	525210E	4247305N	7,682	199	-60	100	0	25	25	0.007	0.44
LRH-188	1984	525210E	4247305N	2,682	199	-60	100	60	80	20	0.005	0.07
LRH-189	1984	525203E	4247295N	7,677	190	-60	100	20	100	80	0.024	0.11
LRH-190	1984	525236E	4247395N	7,660	200	-60	100	0	60	60	0.010	0.19
LRH-191	1984	525241E	4247410N	7,653	197	-60	160	0	105	105	0.018	0.42
LRH-191	1984	525241E	4247410N	7,653	197	-60	160	130	140	10	0.006	0.03
LRH-192	1984	525244E	4247421N	7,646	241	-60	150	0	150	150	0.014	0.34
LRH-193	1984	525253E	4247425N	7,649	270	-60	100	0	90	90	0.008	0.17
LRH-194	1984	525339E	4247424N	7,649	268	-60	160	0	95	95	0.049	1.23
LRH-195	1984	525282E	4247453N	7,625	250	-60	240	0	240	240	0.025	0.33
LRH-196	1984	525257E	4247450N	7,622	258	-60	200	0	200	200	0.020	0.31
LRH-197	1984	525229E	4247438N	7,613	227	-60	160	0	100	100	0.043	0.60
LRH-197	1984	525229E	4247438N	7,613	227	-60	160	135	150	15	0.008	0.19
LRH-198	1984	525224E	4247429N	7,616	205	-60	160	0	115	115	0.011	0.27
LRH-199	1984	525220E	4247415N	7,621	195	-60	100	0	90	90	0.012	0.14
LRH-200	1984	525216E	4247401N	7,626	200	-60	100	0	100	100	0.009	0.17
LRH-201	1984	525489E	4247621N	7,304	0	-90	105			No significant values		
LRH-202	1984	525528E	4247586N	7,295	0	-90	150			No significant values		
LRH-203	1984	525554E	4247558N	7,268	0	-90	140			No significant values		
LRH-204	1984	525081E	4247409N	7,394	0	-90	150			No significant values		
LRH-205	1984	525095E	4247438N	7,395	0	-90	105	0	25	25	0.009	0.17
LRH-206	1984	525109E	4247462N	7,403	0	-90	105			No significant values		
LRH-207	1984	525115E	4247492N	7,392	0	-90	105	0	10	10	0.007	0.21
LRH-208	1984	525121E	4247507N	7,391	0	-90	105			No significant values		
LRH-209	1984	525163E	4247552N	7,411	0	-90	305	0	10	10	0.006	0.16
LRH-209	1984	525163E	4247552N	7,411	0	-90	305	20	30	10	0.007	0.33
LRH-209	1984	525163E	4247552N	7,411	0	-90	305	60	70	10	0.009	0.13

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)		From	To		Au	Ag
LRH-209	1984	525163E	4247552N	7,411	0	-90	305	95	115	20	0.007	0.07
LRH-209	1984	525163E	4247552N	7,411	0	-90	305	155	185	30	0.011	0.21
LRH-210	1984	525190E	4247565N	7,423	0	-90	345	0	110	110	0.009	0.22
LRH-210	1984	525190E	4247565N	7,427	0	-90	345	200	220	20	0.012	0.12
LRH-210	1984	525190E	4247565N	7,423	0	-90	345	240	345	105	0.007	0.09
LRH-211	1984	525207E	4247570N	7,429	0	-90	355	0	175	175	0.025	0.26
LRH-211	1984	525207E	4247570N	7,429	0	-90	355	215	295	80	0.012	0.15
LRH-211	1984	525207E	4247570N	7,429	0	-90	355	340	355	15	0.007	0.09
LRH-212	1984	525237E	4247568N	7,441	0	-90	215	0	95	95	0.010	0.16
LRH-213	1984	525243E	4247536N	7,486	0	-90	205	0	205	205	0.008	0.24
LRH-214	1984	525156E	4247467N	7,473	0	-90	165	0	15	15	0.011	0.20
LRH-214	1984	525156E	4247467N	7,473	0	-90	165	40	60	20	0.019	0.50
LRH-214	1984	525156E	4247467N	7,473	0	-90	165	145	165	20	0.011	0.53
LRH-215	1984	525149E	4247447N	7,475	0	-90	105	55	65	10	0.031	0.69
LRH-216	1984	525137E	4247417N	7,476	0	-90	105	0	35	35	0.007	0.19
LRH-217	1984	525133E	4247401N	7,479	0	-90	105	0	30	30	0.023	0.28
LRH-218	1984	525128E	4247386N	7,480	0	-90	80	0	15	15	0.006	0.20
LRH-219	1984	525207E	4247383N	7,630	210	-60	100	0	55	55	0.011	0.18
LRH-220	1984	525201E	4247371N	7,630	210	-60	100	0	45	45	0.036	0.19
LRH-220	1984	525201E	4247371N	7,630	210	-60	100	85	95	10	0.018	0.22
LRH-221	1984	525195E	4247358N	7,628	210	-60	100	0	15	15	0.008	0.11
LRH-221	1984	525195E	4247358N	7,628	210	-60	100	65	100	35	0.008	0.19
LRH-222	1984	525221E	4247594N	7,398	255	-60	160	0	35	35	0.009	0.11
LRH-222	1984	525221E	4247594N	7,398	255	-60	160	120	140	20	0.007	0.20
LRH-223	1984	525203E	4247590N	7,398	246	-60	215	0	25	25	0.011	0.22
LRH-223	1984	525203E	4247590N	7,398	246	-60	215	90	215	125	0.010	0.09
LRH-224	1984	525189E	4247582N	7,399	235	-60	160	0	110	110	0.009	0.18
LRH-225	1984	525176E	4247572N	7,401	230	-60	155	0	120	120	0.006	0.15
LRH-226	1984	525163E	4247596N	7,358	246	-60	135	10	30	20	0.008	0.11

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-226	1984	525163E	4247596N	7,358	246	-60	135	70	135	65	0.009	0.08
LRH-227	1984	525101E	4247530N	7,337	205	-60	120	0	10	10	0.005	0.09
LRH-228	1984	525130E	4247571N	7,348	214	-60	140	No significant values				
LRH-229	1984	525150E	4247587N	7,355	242	-60	130	0	95	95	0.007	0.19
LRH-230	1984	525187E	4247602N	7,368	241	-60	200	0	130	130	0.012	0.17
LRH-230	1984	525187E	4247602N	7,368	241	-60	200	170	180	10	0.006	0.05
LRH-231	1984	525206E	4247609N	7,372	255	-60	200	140	200	60	0.016	0.10
LRH-232	1984	525112E	4247617N	7,274	218	-60	90	0	10	10	0.009	0.18
LRH-233	1984	525125E	4247634N	7,271	210	-60	155	5	15	10	0.005	0.12
LRH-234	1984	525148E	4247647N	7,277	218	-60	200	0	15	15	0.008	0.12
LRH-235	1984	525165E	4247650N	7,286	210	-60	100	No significant values				
LRH-236	1984	525103E	525103N	7,277	240	-60	160	0	20	20	0.007	0.20
LRH-237	1984	525318E	4247520N	7,509	241	-60	300	0	30	30	0.016	0.51
LRH-237	1984	525318E	4247520N	7,509	241	-60	300	130	175	45	0.014	0.34
LRH-237	1984	525318E	4247520N	7,509	241	-60	300	215	280	65	0.018	1.29
LRH-238	1984	525271E	4247520N	7,510	0	-90	300	95	120	25	0.040	0.90
LRH-238	1984	525271E	4247520N	7,510	0	-90	300	185	270	85	0.009	0.51
LRH-239	1984	525228E	4247516N	7,512	0	-90	260	0	50	50	0.007	0.08
LRH-239	1984	525228E	4247516N	7,512	0	-90	260	90	200	110	0.021	0.47
LRH-240	1984	525194E	4247504N	7,498	0	-90	140	45	130	85	0.012	1.51
LRH-241	1984	525250E	4247517N	7,514	254	-60	240	0	50	50	0.005	0.13
LRH-241	1984	525250E	4247517N	7,514	254	-60	240	125	235	110	0.017	0.49
LRH-242	1984	525295E	4247521N	7,509	216	-60	300	185	300	115	0.032	1.36
LRH-243	1984	525325E	4247458N	7,610	0	-90	232	22	176	152	0.015	0.54
LRH-243	1984	525325E	4247458N	7,610	0	-90	232	190	228	38	0.006	0.36
LRH-244	1984	525363E	4247438N	7,611	0	-90	78	0	64	64	0.029	0.84
LRH-245	1984	525336E	4247431	7,642	0	-90	110	0	110	110	0.008	0.32
LRH-246	1984	525350E	4247447N	7,612	0	-90	115	0	100	100	0.030	0.19
LRH-247	1984	525324E	4247439N	7,638	0	-90	162	0	42	42	0.013	0.14

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-247	1984	525324E	4247439N	7,638	0	-90	162	62	136	74	0.017	0.57
LRH-248	1984	525278E	4247451N	7,627	0	-90	182	0	182	182	0.022	0.54
LRH-249	1984	525236E	4247444N	7,615	0	-90	162	0	132	132	0.018	0.26
LRH-250	1984	525225E	4247431N	7,615	0	-90	120	0	112	112	0.072	0.59
LRH-251	1984	525335E	4247471N	7,586	0	-90	200	0	85	85	0.018	0.51
LRH-251	1984	525335E	4247471N	7,586	0	-90	200	125	175	50	0.014	0.39
LRH-252	1984	525254E	4247492N	7,555	0	-90	200	0	190	190	0.017	0.33
LRH-253	1984	525314E	4247487N	7,566	0	-90	250	0	235	235	0.007	0.63
LRH-254	1984	525290E	4247494N	7,555	0	-90	250	0	30	30	0.009	0.30
LRH-254	1984	525290E	4247494N	7,555	0	-90	250	130	250	120	0.01	0.36
LRH-255	1984	525304E	4247494N	7,554	0	-90	260	0	55	55	0.007	0.12
LRH-255	1984	525304E	4247494N	7,554	0	-90	260	100	260	160	0.012	0.74
LRH-256	1984	525327E	4247492N	7,555	0	-90	230	0	55	55	0.008	0.41
LRH-256	1984	525327E	4247492N	7,555	0	-90	230	90	230	140	0.022	0.97
LRH-257	1984	525350E	4247483N	7,560	0	-90	200	0	20	20	0.009	0.15
LRH-257	1984	525350E	4247483N	7,560	0	-90	200	95	190	95	0.014	0.86
LRH-258	1984	525364E	4247475N	7,564	0	-90	160	0	10	10	0.008	0.14
LRH-258	1984	525364E	4247475N	7,564	0	-90	160	75	150	75	0.008	0.61
LRH-259	1984	525278E	4247501N	7,542	0	-90	280	5	60	55	0.007	0.08
LRH-259	1984	525278E	4247501N	7,542	0	-90	280	175	235	60	0.033	1.00
LRH-259	1984	525278E	4247501N	7,542	0	-90	280	270	280	10	0.033	2.00
LRH-260	1984	525384E	4247498N	7,516	0	-90	260	75	125	50	0.004	0.17
LRH-260	1984	525384E	4247498N	7,516	0	-90	260	150	180	30	0.008	0.81
LRH-261	1984	525416E	4247474N	7,506	0	-90	120	0	65	65	0.010	0.38
LRH-262	1984	525386E	4247464N	7,558	0	-90	140	0	100	100	0.020	0.66
LRH-263	1984	525393E	4247488N	7,521	0	-90	220	0	40	40	0.004	0.25
LRH-263	1984	525393E	4247488N	7,521	0	-90	220	70	120	50	0.006	0.65
LRH-264	1984	525370E	4247505N	7,514	0	-90	200	0	10	10	0.006	0.12
LRH-264	1984	525370E	4247505N	7,514	0	-90	200	170	190	20	0.012	1.41

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(⁰)	(⁰)		From	To		Au	Ag
LRH-265	1984	525345E	4247526N	7,493	0	-90	300	0	20	20	0.048	0.48
LRH-265	1984	525345E	4247526N	7,493	0	-90	300	150	160	10	0.017	0.04
LRH-265	1984	525345E	4247526N	7,493	0	-90	300	195	205	10	0.011	0.05
LRH-265	1984	525345E	4247526N	7,493	0	-90	300	225	285	60	0.027	1.96
LRH-266	1984	525324E	4247533N	7,487	0	-90	290	80	120	40	0.020	0.21
LRH-266	1984	525324E	4247533N	7,487	0	-90	290	185	290	105	0.020	1.40
LRH-267	1984	525299E	4247534N	7,489	0	-90	320	0	10	10	0.015	0.29
LRH-267	1984	525299E	4247534N	7,489	0	-90	320	115	320	105	0.020	0.81
LRH-268	1984	525220E	4247552Nn	7,459	0	-90	320	0	10	10	0.022	0.45
LRH-268	1984	525220E	4247552N	7,459	0	-90	320	105	165	60	0.014	0.19
LRH-268	1984	525220E	4247552N	7,459	0	-90	320	220	240	20	0.024	0.57
LRH-269	1984	525203E	4247548N	7,454	0	-90	320	45	175	130	0.037	0.44
LRH-269	1984	525203E	4247548N	7,454	0	-90	320	200	225	25	0.007	0.13
LRH-270	1984	525187E	4247542N	7,449	0	-90	300	0	80	80	0.013	0.13
LRH-270	1984	525187E	4247542N	7,449	0	-90	300	130	195	65	0.015	0.23
LRH-271	1984	525174E	4247532N	7,444	0	-90	200	0	40	40	0.016	0.35
LRH-271	1984	525174E	4247532N	7,444	0	-90	200	90	160	70	0.013	0.22
LRH-272	1984	525208E	4247516N	7,499	0	-90	280	0	20	20	0.007	0.14
LRH-272	1984	525208E	4247516N	7,499	0	-90	280	75	200	125	0.037	0.43
LRH-274	1984	525222E	4247570E	7,436	0	-90	280	0	85	85	0.008	0.10
LRH-274	1984	525222E	4247570E	7,436	0	-90	280	245	200	55	0.020	0.28
LRH-274	1984	525222E	4247570E	7,436	0	-90	280	255	280	25	0.037	0.33
LRH-325	1984	525301E	4247324N	7,782	0	-90	60	0	60	60	0.026	0.22
LRH-326	1984	525316E	4247302N	7,794	240	-60	200	0	200	200	0.026	0.37
LRH-327	1984	525285E	4247275N	7,798	230	-60	150	0	150	150	0.006	0.68
LRH-328	1984	525293E	4247229N	7,803	337	-60	150	60	120	60	0.009	0.13
LRH-328	1984	525293E	4247229N	7,803	337	-60	150	130	140	10	0.005	0.10
LRH-329	1984	525298E	4247216N	7,801	357	-60	100	25	35	10	0.008	N/A
LRH-330	1984	525286E	4247246N	7,805	345	-60	100	45	70	25	0.017	0.10

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-331	1984	525282E	4247216N	7,810	340	-60	150	40	90	50	0.008	0.17
LRH-332	1984	525313E	4247239N	7,789	340	-60	150	40	150	110	0.034	0.28
LRH-333	1984	525287E	4247335N	7,761	240	-60	150	35	120	85	0.010	0.25
LRH-334	1984	525296E	4247341N	7,766	240	-60	150	25	150	125	0.060	1.26
LRH-335	1984	525319E	4247336N	7,767	260	-60	200	0	10	10	0.006	0.17
LRH-335	1984	525319E	4247336N	7,767	260	-60	200	30	200	170	0.015	0.34
LRH-336	1984	525322E	4247332N	7,767	154	-60	100	25	45	20	0.009	0.11
LRH-337	1984	525332E	4247297N	7,763	250	-58	180	0	160	160	0.014	0.21
LRH-338	1984	525330E	4247300N	7,765	315	-58	150	0	10	10	0.005	0.19
LRH-338	1984	525330E	4247300N	7,765	315	-58	150	50	150	100	0.045	0.68
LRH-339	1984	525332E	4247275N	7,763	320	-57	150	0	105	105	0.011	0.17
LRH-340	1984	525332E	4247273N	7,764	234	-60	150	0	25	25	0.012	0.19
LRH-340	1984	525332E	4247273N	7,764	234	-60	150	65	80	15	0.005	0.16
LRH-340	1984	525332E	4247273N	7,764	234	-60	150	115	150	35	0.009	0.19
LRH-341	1984	525260E	4247353N	7,721	075	-60	160	0	160	160	0.039	0.73
LRH-342	1984	525271E	4247373N	7,717	214	-60	160	15	160	145	0.025	0.25
LRH-343	1984	525153E	4247317N	7,576	200	-60	100	15	25	10	0.006	0.01
LRH-343	1984	525153E	4247317N	7,576	200	-60	100	65	75	10	0.008	N/A
LRH-344	1984	525161E	4247348N	7,576	0	-90	100	80	90	10	0.006	N/A
LRH-345	1984	525168E	4247362N	7,577	0	-90	100	No significant values				
LRH-346	1984	525177E	4247380N	7,579	0	-90	100	0	40	40	0.015	0.39
LRH-347	1984	525187E	4247404N	7,576	0	-90	100	0	100	100	0.010	0.19
LRH-348	1984	525134E	4247342N	7,527	0	-90	100	0	25	25	0.005	0.08
LRH-349	1984	525145E	4247371N	7,521	0	-90	100	No significant values				
LRH-421	1984	525351E	4247241N	7,764	242	-60	200	125	200	75	0.008	0.32
LRH-459	1984	525299E	4247232N	7,798	0	-90	150	110	125	15	0.010	0.46
LRH-460	1984	525322E	4247236N	7,783	243	-60	200	40	70	30	0.008	0.15
LRH-460	1984	525322E	4247236N	7,783	243	-60	200	120	135	15	0.022	0.77
LRH-460	1984	525322E	4247236N	7,783	243	-60	200	180	195	15	0.010	0.27

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
				(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LRH-461	1984	525315E	4247242N	7,787	081	-60	160	60	90	30	0.009	0.06
LRH-462	1984	525308E	4247253N	7,793	0	-90	200	10	120	110	0.044	0.64
LRH-463	1984	525308E	4247277N	7,792	060	-60	200	45	200	155	0.029	0.17
LRH-464	1984	525255E	4247190N	7,834	0	-90	90	No significant values				
LRH-465	1984	525242E	4247145N	7,844	0	-90	100	No significant values				
LRH-466	1984	525350E	4247274N	7,739	245	-60	200	10	65	55	0.009	0.07
LRH-466	1984	525350E	4247274N	7,739	245	-60	200	110	200	90	0.014	0.32
LRH-467	1984	525328E	4247323N	7,764	243	-60	200	0	200	200	0.026	0.46
LRH-468	1984	525332E	4247309N	7,761	250	-60	200	0	175	175	0.020	0.28
LRH-469	1984	525290E	4247289N	7,798	060	-60	50	5	50	45	0.006	0.13
LRH-470	1984	525284E	4247282N	7,797	060	-60	50	0	50	50	0.007	0.30
LRH-471	1984	525280E	4247260N	7,800	060	-60	150	0	150	150	0.021	0.52
LRH-472	1984	525305E	4247228N	7,795	060	-60	100	70	85	15	0.005	0.11
LRH-473	1984	525300E	4247254N	7,800	060	-60	150	0	55	55	0.007	0.16
LRH-474	1984	525314E	4247236N	7,788	060	-60	100	20	35	15	0.007	0.12
LRH-475	1984	525250E	4247341N	7,716	060	-60	100	0	45	45	0.009	0.28
LRH-476	1984	525241E	4247377N	7,682	060	-60	50	0	50	50	0.042	0.40
LRH-477	1984	525236E	4247370N	7,680	0	-90	50	0	35	35	0.011	0.19
LRH-479	1984	525202E	4247289N	7,679	120	-60	30	20	30	10	0.218	0.56
LRH-480	1984	525200E	4247279N	7,681	160	-60	100	0	20	20	0.005	0.10
LRH-480	1984	525200E	4247279N	7,681	160	-60	100	60	90	30	0.005	0.09
LRH-481	1984	525209E	4247326N	7,667	130	-60	100	20	45	25	0.017	0.40
LRH-481	1984	525209E	4247326N	7,661	130	-60	100	85	100	15	0.004	0.30
LRH-482	1984	525207E	4247318N	7,670	020	-60	100	25	80	55	0.065	0.37
LRH-483	1984	525214E	4247493N	7,533	240	-60	140	0	140	140	0.018	0.40
LRH-484	1984	525271E	4247489N	7,563	240	-60	120	10	65	55	0.010	0.27
LRH-484	1984	525271E	4247489N	7,563	240	-60	120	105	120	15	0.004	0.09
LRH-485	1984	525202E	4247530N	7,477	225	-60	100	0	20	20	0.010	0.14
LRH-485	1984	525202E	4247530N	7,477	225	-60	100	40	90	50	0.010	0.24

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LRH-486	1984	525211E	4247569N	7,432	210	-60	100	0	15	15	0.026	0.27
LRH-486	1984	525211E	4247569N	7,432	210	-60	100	45	100	55	0.005	0.09
LRH-487	1984	525417E	4247331N	7,601	0	-90	200	0	10	10	0.005	0.16
LRH-488	1984	525313E	4247261N	7,688	260	-60	100	15	85	70	0.007	0.24
LRH-510	1984	525173E	4247290N	7,624	106	-60	100	5	20	15	0.005	0.08
LRH-510	1984	525173E	4247290N	7,624	106	-60	100	80	95	15	0.004	0.06
LRH-511	1984	525202E	4247292N	7,677	108	-60	100	40	95	55	0.006	0.10
LRH-512	1984	525211E	4247306N	7,683	108	-60	100	0	30	30	0.011	0.77
LRH-512	1984	525211E	4247306N	7,683	108	-60	100	70	85	15	0.011	0.19
LRH-513	1984	525232E	4247358N	7,682	120	-60	100	60	70	10	0.004	0.10
LRH-513	1984	525232E	4247358N	7,682	120	-60	100	90	100	10	0.004	0.08
LRH-514	1984	525246E	4247372N	7,692	117	-60	150	0	80	80	0.024	0.21
LRH-514	1984	525246E	4247372N	7,692	117	-60	150	110	130	20	0.009	0.11
LRH-515	1984	525349E	4247322N	7,725	247	-60	200	0	20	20	0.004	0.12
LRH-515	1984	525349E	4247322N	7,725	247	-60	200	45	55	10	0.007	0.17
LRH-515	1984	525349E	4247322N	7,725	247	-60	200	75	145	70	0.018	0.48
LRH-516	1984	525297E	4247240N	7,800	0	-90	150	45	90	45	0.011	0.11
LRH-516	1984	525297E	4247240N	7,800	0	-90	150	105	130	25	0.004	0.14
LRH-517	1984	525313E	4247206N	7,799	0	-90	150	No significant values				
V-1	1987	525301E	4247325N	7,782	0	-90	195	0	130	130	0.026	0.36
V-1	1987	525301E	4247325N	7,782	0	-90	195	150	195	45	0.017	0.29
V-2	1987	525288E	4247288N	7,798	0	-90	105	0	100	100	0.016	0.47
PR-1	1987	525293E	4247378N	7,723	180	-45	300	0	15	15	0.003	0.07
PR-1	1987	525293E	4247378N	7,723	180	-45	300	50	70	20	0.005	0.04
PR-1	1987	525293E	4247378N	7,723	180	-45	300	115	215	100	0.012	0.37
PR-1	1987	525293E	4247378N	7,723	180	-45	300	245	295	50	0.013	1.24
PR-2	1987	525342E	4247346N	7,722	191	-45	300	15	25	10	0.004	0.39
PR-2	1987	525342E	4247346N	7,722	191	-45	300	65	75	10	0.005	0.51
PR-2	1987	525342E	4247346N	7,722	191	-45	300	95	145	50	0.013	0.30

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
PR-2	1987	525342E	4247346N	7,722	191	-45	300	165	200	35	0.014	0.48
PR-3	1987	525278E	4247427N	7,661	180	-45	300	0	120	120	0.018	0.53
PR-3	1987	525278E	4247427N	7,661	180	-45	300	140	180	40	0.028	1.99
PR-3	1987	525278E	4247427N	7,661	180	-45	300	195	205	10	0.005	0.09
PR-3	1987	525278E	4247427N	7,661	180	-45	300	220	290	70	0.004	0.08
PR-4	1987	525225E	4247465N	7,577	191	-50	300	0	120	120	0.019	0.22
PR-4	1987	525225E	4247465N	7,577	191	-50	300	150	165	15	0.004	N/A
PR-4	1987	525225E	4247465N	7,577	191	-50	300	195	235	40	0.006	0.31
PR-5	1987	525264E	4247492N	7,563	191	-50	300	30	205	175	0.017	0.24
PR-5	1987	525264E	4247492N	7,563	191	-50	300	230	245	15	0.024	1.67
PR-6	1987	525361E	4247445N	7,605	191	-50	300	0	60	60	0.023	0.38
PR-6	1987	525361E	4247445N	7,605	191	-50	300	120	130	10	0.006	N/A
PR-6	1987	525361E	4247445N	7,605	191	-50	300	165	190	25	0.005	N/A
PR-6	1987	525361E	4247445N	7,605	191	-50	300	285	300	15	0.008	0.89
PR-7	1987	525298E	4247556N	7,455	191	-50	500	5	15	10	0.006	N/A
PR-7	1987	525298E	4247556N	7,455	191	-50	500	95	120	25	0.006	0.19
PR-7	1987	525298E	4247556N	7,455	191	-50	500	205	265	60	0.030	1.90
PR-7	1987	525298E	4247556N	7,455	191	-50	500	295	305	10	0.005	0.23
PR-8	1987	525380E	4247515N	7,494	198	-50	300	70	145	75	0.014	0.41
PR-8	1987	525380E	4247515N	7,494	198	-50	300	145	170	25	0.310	1.73
PR-9	1987	525245E	4247566N	7,445	196	-50	300	0	30	30	0.009	0.06
PR-9	1987	525245E	4247566N	7,445	196	-50	300	115	220	105	0.039	0.28
PR-10	1987	525316E	4247321N	7,782	234	-45	300	0	235	235	0.021	0.35
PR-10	1987	525316E	4247321N	7,782	235	-45	300	265	285	20	0.004	0.18
PR-11	1987	525332E	4247258N	7,768	235	-45	300	25	45	20	0.007	0.11
PR-11	1987	525332E	4247258N	7,768	235	-45	300	65	150	85	0.011	0.28
PR-12	1987	525371E	4247284N	7,707	235	-45	300	55	75	20	0.005	0.18
PR-12	1987	525371E	4247284N	7,707	235	-45	300	85	95	10	0.006	0.49
PR-12	1987	525371E	4247284N	7,707	235	-45	300	105	150	45	0.007	0.17

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
PR-12	1987	525371E	4247284N	7,707	235	-45	300	290	300	10	0.009	1.81
PR-13	1987	525267E	4247369N	7,717	235	-45	195	0	120	120	0.015	0.41
PR-13	1987	525267E	4247369N	7,717	235	-45	195	150	195	45	0.005	0.18
PR-14	1987	525442E	4247472N	7,469	191	-45	300	0	45	45	0.010	0.58
PR-14	1987	525442E	4247472N	7,469	191	-45	300	215	225	10	0.004	1.51
PR-15	1987	525155E	4247504N	7,444	191	-45	300	0	10	10	0.007	0.46
PR-15	1987	525155E	4247504N	7,444	191	-45	300	75	85	10	0.007	0.12
PR-16	1987	525172E	4247555N	7,418	191	-45	300	10	20	10	0.005	0.05
PR-16	1987	525172E	4247555N	7,418	191	-45	300	40	190	150	0.013	0.22
PR-17	1987	525388E	4247581N	7,396	191	-45	300	20	30	10	0.005	0.03
PR-17	1987	525388E	4247581N	7,396	191	-45	300	220	255	35	0.034	4.85
PR-18	1987	525301E	4247606N	7,376	191	-45	350	0	15	15	0.005	0.15
PR-18	1987	525301E	4247606N	7,376	191	-45	350	80	90	10	0.005	0.45
PR-18	1987	525301E	4247606N	7,376	191	-45	350	180	195	15	0.007	0.25
PR-18	1987	525301E	4247606N	7,376	191	-45	350	240	255	15	0.004	0.17
PR-18	1987	525301E	4247606N	7,376	191	-45	350	280	295	15	0.091	1.20
PR-18	1987	525301E	4247606N	7,376	191	-45	350	325	335	10	0.011	0.05
PR-19	1987	525254E	4247615N	7,371	191	-45	240	0	65	65	0.007	0.20
PR-21	1987	525407E	4247673N	7,274	191	-45	300			No significant values		
PR-22	1987	525310E	4247668N	7,269	191	-45	400	0	10	10	0.007	0.24
PR-22	1987	525310E	4247668N	7,269	191	-45	400	340	390	50	0.008	3.37
PR-23	1987	525253E	4247697N	7,269	191	-45	350			No significant values		
PR-24	1987	525193E	4247670N	7,283	191	-45	400			No significant values		
PR-25	1987	525310E	4247668N	7,269	0	-90	600	0	20	20	0.005	0.21
PR-26	1987	525407E	4247673	7,274	191	-45	450			No significant values		
PR-27	1987	525407E	4247673N	7,274	0	-90	600			No significant values		
PR-28	1987	525464E	4247619N	7,314	191	-45	450			No significant values		
PR-29	1987	525464E	4247619N	7,314	0	-90	600	410	420	10	0.006	0.15
PR-30	1987	525253E	4247697N	7,269	191	-45	450			No significant values		

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton Au	oz/ton Ag
PR-31	1987	525253E	4247697N	7,269	0	-90	600	No significant values				
LS-1101	2011	525327E	4247358N	7,733	191	-50	400	0	40	40	0.024	0.34
LS-1101	2011	525327E	4247358N	7,733	191	-50	400	80	245	165	0.009	0.61
LS-1101	2011	525327E	4247358N	7,733	191	-50	400	290	325	35	0.007	0.32
LS-1101	2011	525327E	4247358N	7,733	191	-50	400	360	375	15	0.013	2.77
LS-1102	2011	525375E	4247361N	7,669	191	-70	350	210	225	15	0.008	0.99
LS-1103	2011	525350E	4247415N	7,648	191	-45	300	0	50	50	0.013	0.47
LS-1103	2011	525350E	4247415N	7,648	191	-45	300	180	195	15	0.004	0.57
LS-1103	2011	525350E	4247415N	7,648	191	-45	300	255	285	30	0.007	1.06
LS-1104	2011	525303E	4247423N	7,667	191	-45	380	0	95	95	0.014	0.92
LS-1104	2011	525303E	4247423N	7,667	191	-45	380	130	200	70	0.005	0.34
LS-1104	2011	525303E	4247423N	7,667	191	-45	380	235	250	15	0.008	0.39
LS-1104	2011	525303E	4247423N	7,667	191	-45	380	325	335	10	0.005	0.17
LS-1105	2011	525176E	4247286N	7,663	191	-45	250	No significant values				
LS-1106	2011	525199E	4247422N	7,581	191	-45	250	0	90	90	0.011	0.32
LS-1106	2011	525199E	4247422N	7,581	191	-45	250	145	160	15	0.007	0.90
LS-1107	2011	525194E	4247494N	7,509	191	-45	300	0	135	135	0.006	0.31
LS-1107	2011	525194E	4247494N	7,509	191	-45	300	155	165	10	0.005	0.16
LS-1107	2011	525194E	4247494N	7,509	191	-45	300	200	270	70	0.014	0.95
LS-1108	2011	525176E	4247599N	7,356	191	-45	300	0	55	55	0.011	0.21
LS-1108	2011	525176E	4247599N	7,356	191	-45	300	85	120	35	0.021	0.24
LS-1108	2011	525176E	4247599N	7,356	191	-45	300	165	180	15	0.016	0.18
LS-1109	2011	525229E	4247614N	7,371	191	-70	300	0	40	40	0.008	0.26
LS-1110	2011	525417E	4247559N	7,415	191	-45	350	205	215	10	0.004	0.88
LS-1111	2011	525148E	4247434N	7,482	191	-45	300	0	50	50	0.015	0.34
LS-1112	2011	525213E	4247392N	7,631	191	-45	300	0	35	35	0.013	0.22
LS-1112	2011	525213E	4247392N	7,631	191	-45	300	170	180	10	0.004	0.07
LS-1113	2011	525240E	4247443N	7,620	191	-45	290	0	145	145	0.023	0.39
LS-1113	2011	525240E	4247443N	7,620	191	-45	290	185	195	10	0.003	0.30

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LS-1113	2011	525240E	4247443N	7,620	191	-45	290	240	285	45	0.006	0.79
LS-1114	2011	525365E	4247571N	7,402	191	-70	400	155	170	15	0.009	0.70
LS-1114	2011	525365E	4247571N	7,402	191	-70	400	265	310	45	0.008	1.88
LS-1115	2011	525341E	4247598N	7,380	191	-45	400	235	295	60	0.016	1.26
LS-1116	2011	525290E	4247677N	7,265	191	-45	400	0	30	30	0.008	0.20
LS-1116	2011	525290E	4247677N	7,265	191	-45	400	250	265	15	0.006	0.12
LS-1116	2011	525290E	4247677N	7,265	191	-45	400	310	325	15	0.027	0.11
LS-1201	2012	525419E	4247233N	7,725	0	-90	480	50	70	20	0.005	0.02
LS-1201	2012	525419E	4247233N	7,725	0	-90	480	225	240	15	0.004	0.04
LS-1201	2012	525419E	4247233N	7,725	0	-90	480	390	405	15	0.007	1.13
LS-1202	2012	525317E	4247340N	7,765	191	-45	500	0	255	255	0.018	0.53
LS-1202	2012	525317E	4247340N	7,765	191	-45	500	355	365	10	0.005	0.20
LS-1203	2012	525331E	4247395N	7,680	187	-45	550	0	40	40	0.008	0.43
LS-1203	2012	525331E	4247395N	7,680	187	-45	550	190	230	40	0.004	0.79
LS-1203	2012	525331E	4247395N	7,680	187	-45	550	285	325	40	0.006	0.62
LS-1203	2012	525331E	4247395N	7,680	187	-45	550	430	445	15	0.008	0.19
LS-1204	2012	525275E	4247371N	7,705	191	-45	400	15	100	85	0.028	0.51
LS-1204	2012	525275E	4247371N	7,705	191	-45	400	170	180	10	0.005	1.48
LS-1204	2012	525275E	4247371N	7,705	191	-45	400	220	240	20	0.011	0.29
LS-1204	2012	525275E	4247371N	7,705	191	-45	400	260	270	10	0.004	0.21
LS-1205	2012	525369E	4247285N	7,700	191	-65	320	95	285	190	0.065	0.37
LS-1206	2012	525306E	4247455N	7,605	191	-45	500	0	140	140	0.026	0.70
LS-1206	2012	525306E	4247455N	7,605	191	-45	500	210	235	25	0.015	1.00
LS-1206	2012	525306E	4247455N	7,605	191	-45	500	335	355	20	0.004	0.14
LS-1206	2012	525306E	4247455N	7,605	191	-45	500	380	395	15	0.005	0.49
LS-1207	2012	525252E	4247420N	7,635	191	-45	400	0	125	125	0.010	0.42
LS-1207	2012	525252E	4247420N	7,635	191	-45	400	200	215	15	0.021	0.33
LS-1208	2012	525245E	4247445N	7,615	191	-70	320	0	135	135	0.018	0.38
LS-1208	2012	525245E	4247445N	7,615	191	-70	320	180	195	15	0.034	0.45

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton Au	oz/ton Ag
								From	To			
LS-1209	2012	525339E	4247451N	7,605	191	-45	450	0	120	120	0.015	0.50
LS-1209	2012	525339E	4247451N	7,605	191	-45	450	180	190	10	0.005	0.13
LS-1209	2012	525339E	4247451N	7,605	191	-45	450	285	315	30	0.018	2.39
LS-1210	2012	525388E	4247405N	7,600	191	-45	400	235	245	10	0.006	1.48
LS-1210	2012	525388E	4247405N	7,600	191	-45	400	275	295	20	0.004	1.19
LS-1210	2012	525388E	4247405N	7,600	191	-45	400	390	400	10	0.006	0.34
LS-1211	2012	525187E	4247455N	7,528	191	-45	330	0	110	110	0.009	0.25
LS-1211	2012	525187E	4247455N	7,528	191	-45	330	130	180	50	0.004	0.44
LS-1212	2012	525351E	4247480N	7,555	191	-45	500	0	15	15	0.010	0.25
LS-1212	2012	525351E	4247480N	7,555	191	-45	500	210	235	25	0.005	0.47
LS-1212	2012	525351E	4247480N	7,555	191	-45	500	320	345	25	0.010	1.69
LS-1213	2012	525230E	4247517N	7,505	191	-45	430	0	220	220	0.015	0.29
LS-1213	2012	525230E	4247517N	7,505	191	-45	430	380	390	10	0.034	0.03
LS-1214	2012	525200E	4247525N	7,470	191	-45	450	20	165	145	0.013	0.35
LS-1215	2012	525214E	4247567N	7,425	191	-45	470	0	225	225	0.022	0.31
LS-1218N	2012	525025E	4248411N	7,296	191	-45	400	0	40	40	0.011	0.11
LS-1218N	2012	525025E	4248411N	7,296	191	-45	400	175	190	15	0.010	0.05
LS-1218N	2012	525025E	4248411N	7,296	191	-45	400	225	245	20	0.005	0.09
LS-1218N	2012	525025E	4248411N	7,296	191	-45	400	270	330	60	0.007	0.25
LS-1219N	2012	525080E	4248341N	7,316	191	-45	400	50	85	35	0.004	0.11
LS-1219N	2012	525080E	4248341N	7,316	191	-45	400	110	175	65	0.006	0.09
LS-1219N	2012	525080E	4248341N	7,316	191	-45	400	290	310	20	0.004	0.05
LS-1220N	2012	525143E	4248281N	7,349	191	-45	400	40	70	30	0.014	0.16
LS-1220N	2012	525143E	4248281N	7,349	191	-45	400	120	135	15	0.009	0.16
LS-1220N	2012	525143E	4248281N	7,349	191	-45	400	265	290	25	0.005	0.25
LS-1221	2012	525252E	4247610N	7,365	191	-70	600	0	50	50	0.008	0.23
LS-1223	2012	525354E	4247539N	7,470	191	-45	510	0	20	20	0.006	0.19
LS-1223	2012	525354E	4247539N	7,470	191	-45	510	155	205	50	0.008	0.51
LS-1223	2012	525354E	4247539N	7,470	191	-45	510	370	385	15	0.006	0.68

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton	
								From	To		Au	Ag
LS-1223	2012	525354E	4247539N	7,470	191	-45	510	395	410	15	0.005	0.64
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	0	50	50	0.010	0.29
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	100	110	10	0.006	0.22
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	130	165	35	0.022	1.31
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	190	210	20	0.006	1.45
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	255	340	85	0.008	0.94
LS-1225	2012	525311E	4247491N	7,560	191	-45	480	360	370	10	0.005	0.07
LS-1226	2012	525377E	4247478N	7,550	191	-45	450	0	75	75	0.008	0.42
LS-1226	2012	525377E	4247478N	7,550	191	-45	450	85	105	20	0.005	0.58
LS-1226	2012	525377E	4247478N	7,550	191	-45	450	320	355	35	0.004	1.39
LS-1226	2012	525377E	4247478N	7,550	191	-45	450	400	410	10	0.004	0.24
LS-1227	2012	525367E	4247571N	7,402	191	-45	500	0	10	10	0.004	0.11
LS-1227	2012	525367E	4247571N	7,402	191	-45	500	230	255	35	0.018	1.03
LS-1227	2012	525367E	4247571N	7,402	191	-45	500	335	350	15	0.004	0.41
LS-1301	2013	525295E	4247236N	7,792	191	-45	300	0	50	50	0.008	0.27
LS-1301	2013	525295E	4247236N	7,792	191	-45	300	90	105	15	0.004	0.13
LS-1301	2013	525295E	4247236N	7,792	191	-45	300	150	160	10	0.016	0.06
LS-1301	2013	525295E	4247236N	7,792	191	-45	300	190	215	25	0.009	0.14
LS-1302	2013	525316E	4247293N	7,788	191	-45	450	0	40	40	0.015	0.89
LS-1302	2013	525316E	4247293N	7,788	191	-45	450	75	150	75	0.011	0.56
LS-1302	2013	525316E	4247293N	7,788	191	-45	450	210	265	55	0.013	0.46
LS-1303	2013	525355E	4247239N	7,750	191	-80	400	90	110	20	0.003	1.12
LS-1303	2013	525355E	4247239N	7,750	191	-80	400	165	205	40	0.006	0.26
LS-1303	2013	525355E	4247239N	7,750	191	-80	400	220	270	50	0.004	0.27
LS-1304	2013	525234E	4247300N	7,705	191	-45	300	45	85	40	0.006	0.36
LS-1304	2013	525234E	4247300N	7,705	191	-45	300	120	155	35	0.047	0.36
LS-1305	2013	525246E	4247384N	7,683	191	-45	400	0	125	125	0.008	0.27

Table 30-1 Reverse Circulation Drilling Record (1984-1997, and 2011-2013)
Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	UTM Coordinates		Elevation (ft.)	Azimuth (°)	Inclination (°)	Length (ft.)	Intersection (ft.)		Interval (ft.)	oz/ton Au	oz/ton Ag
								From	To			
LS-1305	2013	525246E	4247384N	7,683	191	-45	400	205	220	15	0.005	0.27
LS-1306	2013	525373E	4247378N	7,649	191	-45	400	205	230	25	0.005	0.97
LS-1307	2013	525197E	4247356N	7,628	191	-45	300	10	40	30	0.004	0.11
LS-1307	2013	525197E	4247356N	7,628	191	-45	300	125	140	15	0.010	1.16
LS-1308	2013	525420E	4247328N	7,601	191	-45	400	85	100	15	0.004	0.16
LS-1308	2013	525420E	4247328N	7,601	191	-45	400	285	310	25	0.005	1.56
LS-1309	2013	525289E	4247469N	7,593	191	-45	350	0	230	230	0.022	0.64
LS-1309	2013	525289E	4247469N	7,593	191	-45	350	325	340	15	0.003	0.33
LS-1310	2013	525150E	4247385N	7,150	188	-45	295	No significant values				
LS-1311	2013	525411E	4247491N	7,480	191	-55	400	20	115	95	0.008	0.66
LS-1312	2013	525324E	4247529N	7,487	191	-45	400	45	70	25	0.009	0.74
LS-1312	2013	525324E	4247529N	7,487	191	-45	400	120	220	100	0.013	1.04
LS-1312	2013	525324E	4247529N	7,487	191	-45	400	295	310	15	0.010	0.61
LS-1313	2013	525264E	4247531N	7,480	191	-45	400	0	15	15	0.009	0.29
LS-1313	2013	525264E	4247531N	7,480	191	-45	400	50	105	55	0.019	0.74
LS-1313	2013	525264E	4247531N	7,480	191	-45	400	160	200	40	0.038	0.81
LS-1313	2013	525264E	4247531N	7,480	191	-45	400	235	250	15	0.005	0.15
LS-1314	2013	525392E	4247461N	7,550	191	-45	400	0	60	60	0.018	0.63
LS-1314	2013	525392E	4247461N	7,550	191	-45	400	310	370	60	0.006	2.06
LS-1315	2013	525138E	4247529N	7,420	191	-45	270	0	15	15	0.005	0.32
LS-1316	2013	525278E	4247611N	7,380	191	-45	395	0	30	30	0.014	0.22
LS-1316	2013	525278E	4247611N	7,380	191	-45	395	175	280	105	0.010	0.29
LS-1317	2013	525519E	4247421N	7,401	191	-45	280	No significant values				
LS-1318	2013	525152E	4247588N	7,355	191	-45	280	0	75	75	0.006	0.32
LS-1319	2013	525483E	4247561N	7,359	191	-45	410	175	195	20	0.047	15.43
LS-1320	2013	525272E	4247685N	7,260	191	-45	100	0	20	20	0.004	0.20

Source: Star Gold, 2012, Kern, 2012, 2013.

Notes:

1. Average assay values >0.005 oz/ton Au (0.18 g/t Au) are considered significant.
2. Detection limits: 2 g/t Ag, 0.01 g/t Au; Results below these detection limits are half of the detection limits.
3. Gold and silver values are rounded.
4. NSV means no significant values.
5. N/A means not available.
6. Reported assay values are uncut.
7. Elevations are drill hole collar elevations in metres.
8. Drill holes LS-1218N, 1219N, and 1220N were drilled on the North Zone.
9. The exact date of the historic holes from 1984 to 1987 are uncertain.

Summary of Significant Mineralized Intersections in Diamond Drill Holes (1987 and 2012)

Table 30-2 Diamond Drilling Record (1987 and 2012)

Star Gold Corp. – Longstreet Au-Ag Project, Nevada

Hole ID	Year	Zone	UTM Coordinates		Elevation	Azimuth	Inclination	Length	Intersection (ft.)		Interval	oz/ton	oz/ton
					(ft.)	(°)	(°)	(ft.)	From	To	(ft.)	Au	Ag
LS-1	1987	Cyprus Ridge	N/A	N/A	N/A	216	-75	285	260	270	10	0.011	N/A
LS-2	1987	Cyprus Ridge	N/A	N/A	N/A	215	-83	345	310	320	10	0.013	N/A
LS-3	1987	Cyprus Ridge	N/A	N/A	N/A	225	-75	340	240	260	20	0.010	N/A
LS-4	1987	Cyprus Ridge	N/A	N/A	N/A	225	-70	290	100	120	20	0.012	N/A
LS-4	1987	Cyprus Ridge	N/A	N/A	N/A	225	-70	290	230	260	30	0.010	N/A
LS-5	1987	Cyprus Ridge	N/A	N/A	N/A	185	-70	360	300	310	10	0.034	N/A
LS-6	1987	Cyprus Ridge	N/A	N/A	N/A	225	-70	280				No significant values	
LS-7	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	275				No significant values	
LS-8	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	150				No significant values	
LS-9	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	330	220	230	30	0.009	N/A
LS-10	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	80				No significant values	
LS-11	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	100				No significant values	
LS-12	1987	Cyprus Ridge	N/A	N/A	N/A	0	-90	185				No significant values	
LS-1216C	2012	Main	525331E	4247362N	7,733	191	-55	397	25	235	210	0.008	0.47
LS-1217C	2012	Main	525265E	4247488N	7,563	191	-55	298	0	205	205	0.017	0.40
LS-1222C	2012	Main	525252E	4247566N	7,440	191	-50	299	5	65	60	0.006	0.17
LS-1222C	2012	Main	525252E	4247566N	7,440	191	-50	299	125	220	95	0.017	0.30
LS-1224C	2012	Main	525300E	4247518N	7,509	216	-65	299	105	120	15	0.021	0.48
LS-1224C	2012	Main	525300E	4247518N	7,509	216	-65	299	155	175	20	0.005	0.06
LS-1224C	2012	Main	525300E	4247518N	7,509	216	-65	299	195	255	60	0.034	2.47

Source: Noland, 2012.

Notes:

1. Diamond drilling in 1987 carried out by Cyprus Minerals Company.
2. Diamond drilling in 2012 carried out by Star Gold Corp.

Appendix 3. McClelland Metallurgical Report 2014



McCLELLAND LABORATORIES, INC.

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E-MAIL mli@mettest.com

April 16, 2014

Mr. David Segelov
Star Gold Corp.
611 E. Sherman Ave.
Coeur d'Alene, ID 83814

Dear David:

Enclosed is our report concerning results from heap leach cyanidation testing conducted on three composites from your Longstreet project.

Our invoice for the completed work will be sent under separate cover.

Thank you for allowing us to serve you.

Sincerely,

Jared R. Olson
Metallurgist / Project Manager

JRO:mh
Enclosure



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**Report
on
Heap Leach Cyanidation Testing -
Longstreet Project
MLI Job No. 3829
April 16, 2014**

for

**Mr. David Segelov
Star Gold Corp.
611 E. Sherman Ave.
Coeur d'Alene, ID 83814**

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April 16, 2014**

for

**Mr. David Segelov
Star Gold Corp.
611 E. Sherman Ave.
Coeur d'Alene, ID 83814**

EXECUTIVE SUMMARY

A heap leach cyanidation testing program was undertaken on three composites from the Longstreet project. The composites were prepared from 19 underground adit samples and 3 surface samples taken from the project site. Composites were identified as the surface, underground, and master composites. Average gold head grades of the composites ranged from 0.38 to 0.84 gAu/mt ore. Average silver head grades of the three composites 25 to 59 gAg/mt ore.

Column leach tests were conducted on each composite at an 80%-19mm feed size to determine gold recovery, recovery rate, and reagent requirements, under simulated heap leaching conditions. Bottle roll tests were conducted on each of the three composites at an 80%-1.7mm feed size. Additional bottle roll tests were conducted on the master composite at feed sizes ranging from 100%-50mm to 80%-75µm, and at cyanide concentrations of 1.0 and 5.0 gNaCN/L, to determine sensitivity to feed size and cyanide concentration.

Summary results from the column leach and bottle roll tests are presented in Table 1.

Table 1. - Summary Metallurgical Results, Heap Leach Testing, Longstreet Composites

Composite	Test Type	Test No.	Leach Time, days	Feed Size	NaCN Conc. g/L	Recovery, %		Calc'd. Head, g/mt ore		NaCN Consumed, kg/mt ore	Lime Added, kg/mt ore
						Au	Ag	Au	Ag		
Surface	CLT	P-1	190	80%-19mm	1.0	88.9	20.0	0.36	25	1.56	1.7
Surface	BRT	CY-2	4	80%-1.7mm	1.0	80.6	20.0	0.31	25	0.08	2.1
Underground	CLT	P-2	190	80%-19mm	1.0	84.6	15.4	0.78	52	1.93	2.7
Underground	BRT	CY-1	4	80%-1.7mm	1.0	81.9	18.9	0.83	53	0.13	3.4
Master	CLT	P-3	190	80%-19mm	1.0	86.3	16.7	0.73	48	1.86	2.0
Master	BRT	CY-4	4	100%-50mm	1.0	62.9	3.6	0.70	56	0.07	1.3
Master	BRT	CY-5	4	80%-19mm	1.0	67.1	12.8	0.76	39	0.07	2.1
Master	BRT	CY-6	4	80%-6.3mm	1.0	77.9	13.6	0.68	44	<0.07	3.0
Master	BRT	CY-3	4	80%-1.7mm	1.0	81.3	17.5	0.64	40	0.13	2.5
Master	BRT	CY-7	4	80%-19mm	5.0	76.4	14.6	0.72	41	0.48	1.0
Master	BRT	CY-8	4	80%-6.3mm	5.0	77.6	14.0	0.58	43	0.67	1.0
Master	BRT	CY-9	4	80%-75µm	5.0	88.7	60.6	0.53	33	0.91	1.3

Column test results show that the three composites were readily amenable to heap leach cyanidation treatment at an 80%-19mm feed size. Gold recoveries were similar for each composite, and ranged from 84.6% to 88.9%, in 190 days of leaching and rinsing.

Column leach test silver recoveries were low and ranged from 15.4% to 20.0%.

Gold recovery rates, during column leach testing were rapid. Silver recovery rates were slow.

Several bottle roll tests were conducted on the master composite at varying feed sizes and cyanide concentrations, in an attempt to improve silver recovery. Tests conducted at crushed feed sizes (-50mm to 80%-1.7mm), using cyanide concentrations of 1.0 and 5.0 gNaCN/L, indicated that the composite was not particularly sensitive to feed size or cyanide concentration, with respect to silver recovery. Silver recoveries, at these conditions, ranged from 3.6% to 17.5%. A test conducted at 80%-75µm (5.0 gNaCN/L) showed that fine grinding was effective in substantially increasing silver recovery by cyanidation (to 60.6%). Column leach test recovery by size fraction data also suggest that fine grinding would likely be required to maximize silver recovery.

The master composite was not particularly sensitive to feed size with respect to gold recovery, given a sufficiently long leach cycle. Gold recovery achieved by column leach testing of the master composite at an 80%-19mm feed size (86.3%) was nearly the same as the bottle roll test recovery achieved at the 80%-75µm feed size (88.7%).

Cyanide consumption during column leach testing was high, in part because of the long leach cycles employed, and ranged from 1.56 to 1.93 kgNaCN/mt ore. Cyanide consumption for column leach testing typically is significantly higher than observed during commercial heap leaching. Cyanide consumption was much lower during bottle roll testing, and ranged from <0.07 to 0.13 kgNaCN/mt ore, when a cyanide concentration of 1.0 gNaCN/L was maintained.

The column test lime additions of 1.7 to 2.7 kg/mt ore were sufficient for maintaining protective alkalinity throughout the leach cycle.

SAMPLE PREPARATION AND HEAD ANALYSIS

On June 30, 2013, a total of 22 “section” bulk samples were received for preparation and analysis. Of those samples, 19 were designated as underground adit samples, and 3 were designated as surface samples. Underground adit samples weighed 9 to 34 kg each. Surface samples weighed 129 to 145 kg each.

A total of 20 competent pieces of rock were taken from the 22 samples for comminution testing. Half of the 20 rock pieces were selected from the underground adit samples, and half were taken from the surface samples. The rock pieces were combined and then submitted to Phillips Enterprises for crusher work index and abrasion index testing.

After comminution testing samples were taken, each of the 22 samples was stage crushed in entirety to 100%-50mm. Each of the underground adit samples was then blended and split to generate duplicate 1 kg head assay samples. Surface samples were each blended and split to generate duplicate 5 kg splits. Each of the assay splits was then assayed directly using conventional fire assay procedures, to determine gold and silver content. In many cases (low grade samples), the silver grades required that assays be repeated using a four acid digest procedure in order to obtain a more reliable result.

Assay results were reviewed by Dan Peldiak of Coffey Mining, and instructions were issued to blend and split each of the 22 samples and combine splits in order to generate a surface composite, an underground composite, and a master composite. The surface and underground composites were then crushed to 80%-19mm. The 19mm material was blended and split to generate 75 kg for a column leach test, 25 kg for a head screen analysis, triplicate 1 kg splits for head analysis, and 1 kg for further crushing. That 1 kg split was crushed to 80%-1.7mm for bottle roll testing.

Prior to crushing the master composite, a 5 kg split was taken for bottle roll testing (100%-50mm feed size). Remaining material was then crushed to 80%-19mm. The 19mm material was blended and split to generate 75 kg for a column leach test, 25 kg for a head screen analysis, 2 kg for bottle roll testing, triplicate 1 kg splits for head analysis, and 2 kg for further crushing. That 2 kg split was crushed to 80%-6.3mm and then split in half. One half split was further crushed to 80%-1.7mm. Both half splits were then used for bottle roll testing. The triplicate head assay splits from each of the three composites were assayed for gold (fire assay) and silver content (four acid digest).

After initial bottle roll testing was complete, additional sample preparation of the master composite was required. Rejects from the master composite at the 80%-19mm feed size were blended and split to generate an additional 2 kg sample for bottle roll testing and 3 kg for further crushing. That 3 kg split was sequentially crushed, blended, and split to generate 1 kg samples for bottle roll tests (80%-6.3mm and 100%-850µm feed sizes) and for mineralogical analysis (nominal 1.7mm feed size). The sample taken for mineralogical analysis, labeled as “3829 BMC”, as well as a sample of bottle roll test tailings, were submitted to DCM Science Laboratories for evaluation.

Composite head assay results and head grade comparisons are presented in Tables 2 and 3. Section sample assay results are given in Section 1 of the Appendix. Composite make-up information, including a description of the samples submitted for comminution testing, is provided in Section 2 of the Appendix. Comminution testing results from Phillips Enterprises are provided in Section 3 of the Appendix. The mineralogical analysis report from DCM Science Laboratories is provided in Section 4 of the Appendix.

**Table 2. - Gold and Silver Head Assay Results and Head Grade Comparisons,
Longstreet Surface and Underground Composites**

Determination	Surface Composite		Underground Composite	
	gAu/mt ore	gAg/mt ore	gAu/mt ore	gAg/mt ore
Direct Assay, Init.	0.21	17	0.70	67
Direct Assay, Dup.	0.67	34	0.82	63
Direct Assay, Trip.	0.37	21	1.09	53
Calc'd., Bottle Roll, 80%-1.7mm	0.31	25	0.83	53
Calc'd., Head Screen	0.34	25	0.82	65
Calc'd., Column	0.36	25	0.78	52
Average	0.38	25	0.84	59
Std. Deviation	0.15	6	0.13	7

**Table 3. - Gold and Silver Head Assay Results and Head Grade Comparisons,
Longstreet Master Composite**

Determination	Head Grade, g/mt ore	
	Au	Ag
Direct Assay, Init.	0.57	40
Direct Assay, Dup.	0.66	41
Direct Assay, Trip.	0.77	50
Calc'd., Bottle Roll, 1.0 gNaCN/L		
100%-50mm	0.70	56
80%-19mm	0.76	39
80%-6.3mm	0.68	44
80%-1.7mm	0.64	40
Calc'd., Bottle Roll, 5.0 gNaCN/L		
80%-19mm	0.72	41
80%-6.3mm	0.58	43
80%-75µm	0.53	33
Calc'd., Head Screen	0.79	47
Calc'd., Column	0.73	48
Average	0.68	44
Std. Deviation	0.08	6

Head assay results and calculated head grades show that the Longstreet composites contained between 0.38 and 0.84 gAu/mt ore. Gold head grade standard deviation ranged from 0.08 to 0.15 gAu/mt ore.

Average silver head grades ranged from 25 to 59 gAg/mt ore. Silver head grade standard deviation ranged from 6 to 7 gAg/mt ore.

In the case of the surface composite, head grade comparisons showed that gold and silver head grade standard deviation was high relative to head grade, and that gold occurrence was somewhat “spotty.” The high standard deviation for this composite was due to variability in the direct head assays. Because of the high variability, head assays for all three composites were repeated, and the original results were confirmed.

DIRECT AGITATED CYANIDATION TEST PROCEDURES AND RESULTS

Direct agitated cyanidation (bottle roll) tests were conducted on each composite at a feed size of 80%-1.7mm to determine gold and silver recovery, recovery rate, and reagent requirements. Tests were also conducted on the master composite at feed sizes ranging from 100%-50mm to 80%-1.7mm, and at a cyanide concentration of 1.0 gNaCN/L to determine sensitivity to feed size. Due to the very low silver recoveries observed during initial testing, additional bottle roll tests were conducted on the master composite at feed sizes ranging from 80%-19mm to 80%-75µm, and at a cyanide concentration of 5.0 gNaCN/L to determine response to milling/cyanidation treatment and sensitivity to cyanide concentration.

Ore charges were mixed with water to achieve 40 weight percent solids. In the case of the test conducted at 80%-75µm, a crushed ore sample (100%-850µm) was ground using a laboratory steel ball mill, and then settled and decanted to achieve 40% solids. Natural pulp pH was measured. Lime was added to adjust the pH of the pulps to 11.0 before adding cyanide. Sodium cyanide (either 1.0 or 5.0 gNaCN/L), was then added to the alkaline pulps.

Leaching was conducted by rolling the pulps in bottles on laboratory rolls for 96 hours. Rolling was suspended briefly after 2, 6, 24, 48 and 72 hours to allow the pulps to settle so samples of pregnant solution could be taken for gold and silver analysis by A.A. and ICP methods. Pregnant solution volumes were measured and sampled. Make-up water, equivalent to that withdrawn, was added to the pulps. Cyanide concentrations were restored to initial levels. Lime was added when necessary, to maintain the leaching pH at between 10.8 and 11.2. Rolling was then resumed.

After 96 hours, the pulps were settled to separate liquids and solids. Final pregnant solution volumes were measured and sampled for gold and silver analysis. Final pH and cyanide concentrations were determined. Leached residues were filtered, dried, and assayed in triplicate to determine residual precious metal content.

Overall metallurgical results from bottle roll tests are provided in Tables 4 through 6. Corresponding gold leach rate profiles are shown graphically in Figures 1 through 3. Summary BRT results are presented graphically in Figure 4. Detailed bottle roll test data are provided in Section 5 of the Appendix to this report.

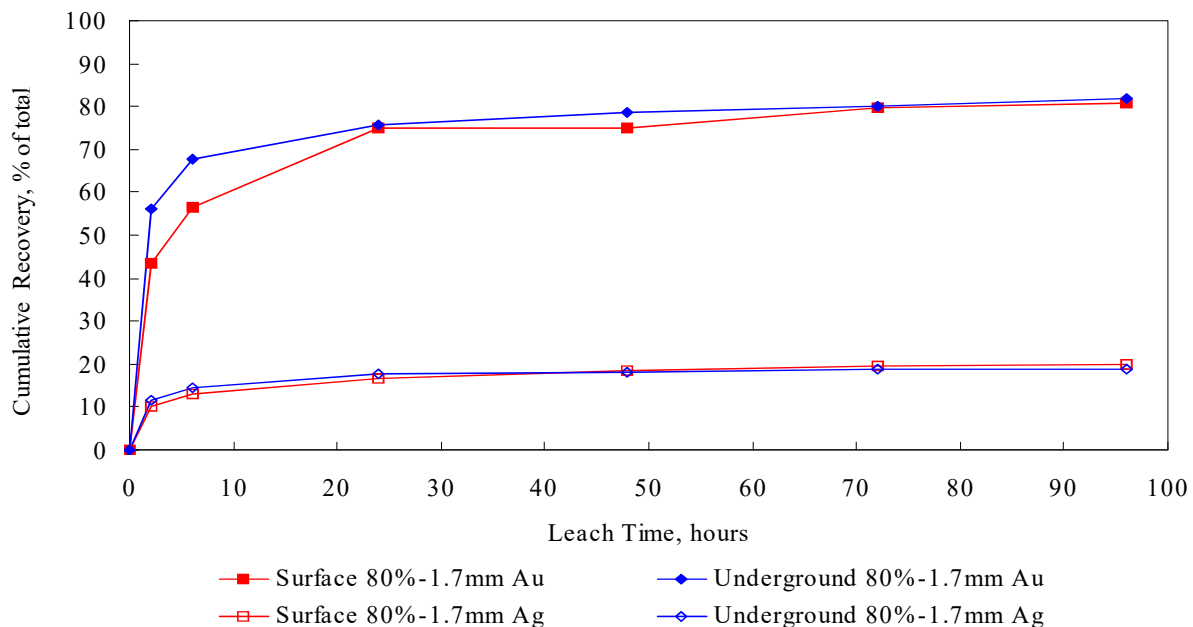
**Table 4. - Overall Metallurgical Results, Bottle Roll Tests,
Longstreet Surface and Underground Composites, 1.0 gNaCN/L**

Composite:	Surface		Underground	
Feed Size:	80%-1.7mm		80%-1.7mm	
Metallurgical Results	(CY-2)		(CY-1)	
Extraction: % of total	Au	Ag	Au	Ag
in 2 hours	43.5	10.3	56.0	11.4
in 6 hours	56.6	12.9	67.7	14.5
in 24 hours	75.0	16.6	75.7	17.6
in 48 hours	74.9	18.3	78.6	18.2
in 72 hours	79.7	19.6	79.9	18.9
in 96 hours	80.6	20.0	81.9	18.9
Extracted, gAu/mt ore	0.25	5	0.68	10
Tail Assay, gAu/mt ore ¹⁾	0.06	20	0.15	43
Calc'd. Head, gAu/mt ore	0.31	25	0.83	53
Average Head, gAu/mt ore ²⁾	0.38	25	0.84	59
NaCN Consumed, kg/mt ore	0.08		0.13	
Lime Added, kg/mt ore	2.1		3.4	
Final pH	11.1		11.3	
Natural pH (40% solids)	7.5		5.5	

1) Average of triplicate tail assays.

2) Average of all head grade determinations.

**Figure 1. - Gold and Silver Leach Rate Profiles, Bottle Roll Tests,
Longstreet Surface and Underground Composites, 1.0 gNaCN/L**



Overall metallurgical results show that the surface and underground composites were amenable to direct agitated cyanidation treatment at the 80%-1.7mm feed size. Gold recoveries from the surface and underground composites were 80.6% and 81.9%, respectively in 96 hours of leaching.

Silver recoveries from the surface and underground composites were 20.0% and 18.9%, respectively.

Gold recovery rates were moderate. Gold extraction from the underground composite decreased slightly between 24 and 48 hours. This minor decrease is likely due to analytical variability. The effects of analytical variability are more apparent in low grade solutions such as those generated during leaching of this composite. Silver recovery rates were slow. It does not appear that extending the leach cycle would significantly improve silver recovery.

Cyanide consumption for the surface and underground composites was low (0.08 and 0.13 kgNaCN/mt ore, respectively). Lime requirement was low for the surface composite (2.1 kg/mt ore) and moderate for the underground composite (3.4 kg/mt ore).

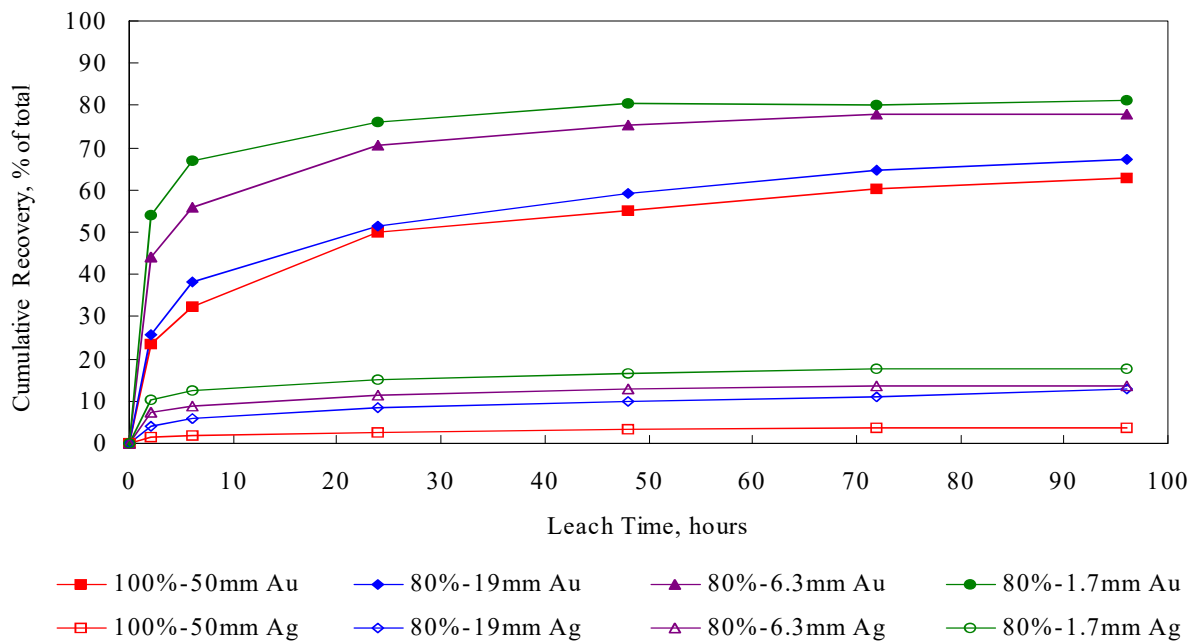
**Table 5. - Overall Metallurgical Results, Bottle Roll Tests,
Longstreet Master Composite, 1.0 gNaCN/L**

Composite:	Master		Master		Master		Master	
Feed Size:	100%-50mm		80%-19mm		80%-6.3mm		80%-1.7mm	
Metallurgical Results	(CY-4)		(CY-5)		(CY-6)		(CY-3)	
Extraction: % of total	Au	Ag	Au	Ag	Au	Ag	Au	Ag
in 2 hours	23.6	1.4	25.7	4.2	44.1	7.2	53.9	10.4
in 6 hours	32.5	2.0	38.4	6.0	56.0	8.8	67.0	12.4
in 24 hours	50.1	2.7	51.5	8.5	70.7	11.6	76.0	14.9
in 48 hours	55.1	3.3	59.0	10.1	75.2	13.0	80.5	16.6
in 72 hours	60.1	3.6	64.9	11.1	77.9	13.6	80.1	17.5
in 96 hours	62.9	3.6	67.1	12.8	77.9	13.6	81.3	17.5
Extracted, gAu/mt ore	0.44	2	0.51	5	0.53	6	0.52	7
Tail Assay, gAu/mt ore ¹⁾	0.26	54	0.25	34	0.15	38	0.12	33
Calc'd. Head, gAu/mt ore	0.70	56	0.76	39	0.68	44	0.64	40
Average Head, gAu/mt ore ²⁾	0.68	44	0.68	44	0.68	44	0.68	44
NaCN Consumed, kg/mt ore	0.07		0.07		<0.07		0.13	
Lime Added, kg/mt ore	1.3		2.1		3.0		2.5	
Final pH	11.1		11.2		11.0		10.9	
Natural pH (40% solids)	6.6		6.5		6.4		6.3	

1) Average of triplicate tail assays.

2) Average of all head grade determinations.

**Figure 2. - Gold and Silver Leach Rate Profiles, Bottle Roll Tests,
Longstreet Master Composite, 1.0 gNaCN/L**



Overall metallurgical results show that the master composite was amenable to direct agitated cyanidation treatment at feed sizes ranging from 100%-50mm to 80%-1.7mm. Gold recoveries ranged from 62.9% (-50mm) to 81.3% (1.7mm), in 96 hours of leaching at a cyanide concentration of 1.0 gNaCN/L.

Silver recoveries ranged from 12.8% to 17.5% at feed sizes ranging from 80%-19mm to 80%-1.7mm. Silver recovery was significantly lower at the 100%-50mm feed size (3.6%).

Gold recovery rates were slow for the 50mm feed, and increased slightly at each incrementally finer feed size. Gold extraction from the 50mm and 19mm feeds was progressing at a slow, but significant rate when leaching was terminated after 96 hours. It is expected that if leach cycles for these tests were extended, gold recoveries would have approached the recoveries achieved from the 6.3mm and 1.7mm feeds. Silver recovery rates were also slow, however it does not appear that extending the leach cycle would significantly improve silver recovery.

Cyanide consumption for the master composite was low at all feed sizes tested and ranged from <0.07 to 0.13 kgNaCN/mt ore. Lime requirements were low to moderate and ranged from 1.3 to 3.0 kg/mt ore.

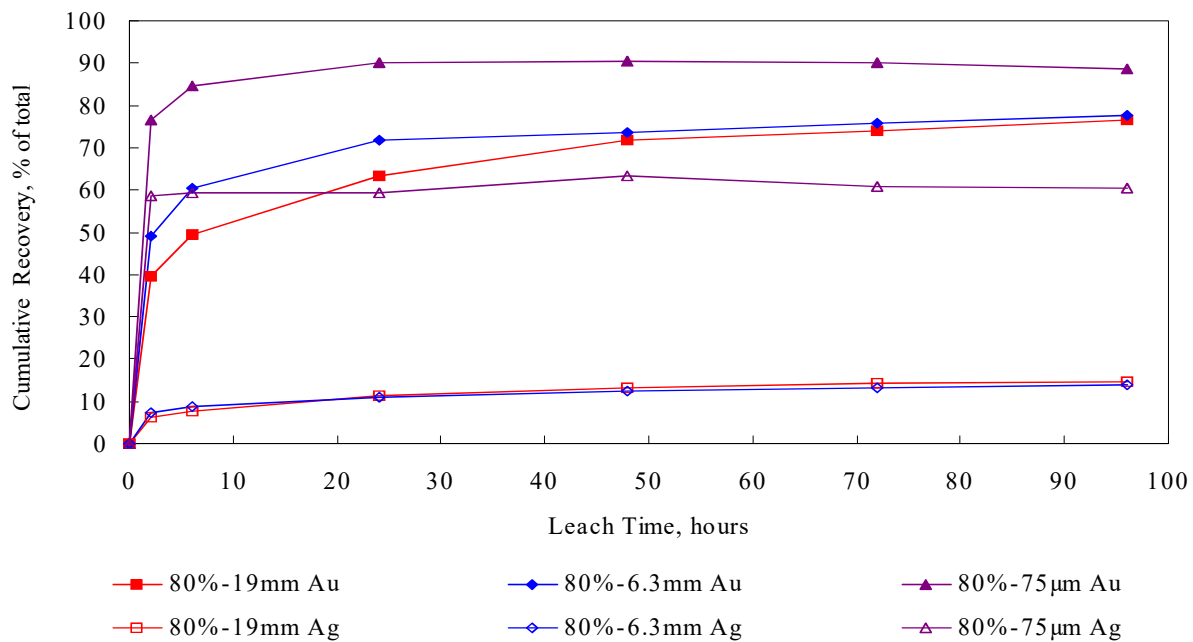
**Table 6. - Overall Metallurgical Results, Bottle Roll Tests,
Longstreet Master Composite, 5.0 gNaCN/L**

Composite: Feed Size:	Master 80%-19mm (CY-7)		Master 80%-6.3mm (CY-8)		Master 80%-75µm (CY-9)	
Metallurgical Results	Au	Ag	Au	Ag	Au	Ag
Extraction: % of total						
in 2 hours	39.6	6.4	49.1	7.2	76.4	58.7
in 6 hours	49.3	7.8	60.3	8.7	84.6	59.5
in 24 hours	63.4	11.3	71.8	10.9	89.9	59.5
in 48 hours	71.7	13.2	73.7	12.3	90.3	63.3
in 72 hours	74.0	14.4	75.8	13.2	90.2	60.9
in 96 hours	76.4	14.6	77.6	14.0	88.7	60.6
Extracted, gAu/mt ore	0.55	6	0.45	6	0.47	20
Tail Assay, gAu/mt ore ¹⁾	0.17	35	0.13	37	0.06	13
Calc'd. Head, gAu/mt ore	0.72	41	0.58	43	0.53	33
Average Head, gAu/mt ore ²⁾	0.68	44	0.68	44	0.68	44
NaCN Consumed, kg/mt ore	0.48		0.67		0.91	
Lime Added, kg/mt ore	1.0		1.0		1.3	
Final pH	11.0		11.0		11.2	
Natural pH (40% solids)	6.3		6.3		7.0	

1) Average of triplicate tail assays.

2) Average of all head grade determinations.

**Figure 3. - Gold and Silver Leach Rate Profiles, Bottle Roll Tests,
Longstreet Master Composite, 5.0 gNaCN/L**



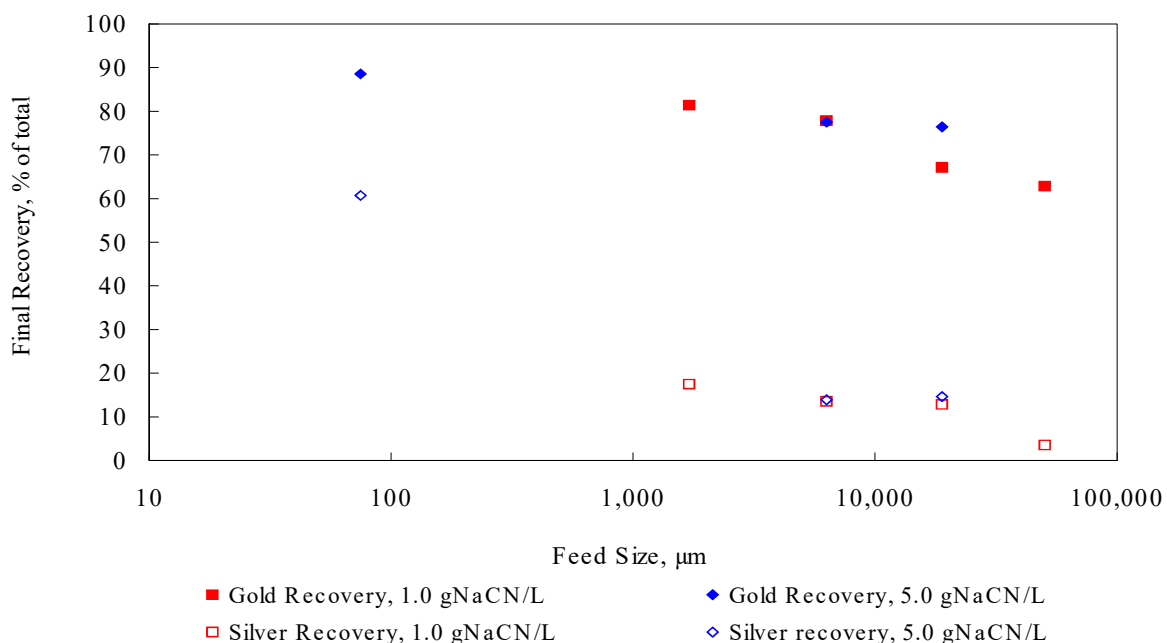
Overall metallurgical results from show that the master composite was readily amenable to direct agitated cyanidation treatment at the 80%-75 μ m feed size. Gold recovery at this feed size was 88.7%, in 96 hours of leaching, and at a cyanide concentration of 5.0 gNaCN/L. Gold recoveries at the 19mm and 6.3mm feed sizes were 76.4% and 77.6%, respectively.

Silver recoveries at the 19mm and 6.3mm feed sizes were 14.6% and 14.0%, respectively. Silver recovery at the 80%-75 μ m feed size was significantly higher (60.6%).

Gold and silver recovery rates were moderate at the 19mm and 6.3mm feed sizes. Gold extraction was still progressing at a slow, but significant rate when leach cycles were terminated after 96 hours. It is expected that gold recoveries may be higher if leach cycles were extended beyond 96 hours. Gold and silver recovery rates at the 75 μ m feed size were rapid, and extraction was essentially complete in the first 2 to 6 hours of leaching. Both gold and silver recovery decreased slightly at times during leaching of the 75 μ m feed. It is believed that these slight decreases were due to analytical variability.

Cyanide consumption was moderate and ranged from 0.48 to 0.91 kgNaCN/mt ore. Lime requirements were low and ranged from 1.0 to 1.3 kg/mt.

Figure 4. - Gold and Silver Recovery vs. Feed Size, Bottle Roll Tests, Longstreet Master Composite, 1.0 and 5.0 gNaCN/L



Low silver recoveries were observed at feed sizes as fine as 80%-1.7mm, and at a cyanide concentration of 1.0 gNaCN/L. In an effort to improve silver recovery, tests were conducted at feed sizes as fine as 80%-75µm and at an elevated cyanide concentration (5.0 gNaCN/L). Comparing these results shows that decreasing the feed size to 80%-75µm dramatically improved silver recovery. Results also show that increasing the cyanide concentration to 5.0 gNaCN/L likely would not improve gold or silver recovery during heap leaching.

The data show that the master composite was not particularly sensitive to feed size, with respect to silver recovery, in the range of crushed feed sizes tested (-50mm to 1.7mm). Within this range of sizes, silver recovery ranged from 3.6% at the 100%-50mm feed size to 17.5% at the 80%-1.7mm feed size. When fine grinding was employed (80%-75µm), the silver recovery increased substantially to 60.6%.

The overall trend of the bottle roll test recovery vs. feed size data seems to indicate that gold recovery was moderately sensitive to feed size. However this would not likely be the case in a heap leach setting. Gold recovery achieved by column leach testing of the master composite at an 80%-19mm feed size (86.3%) was nearly the same as the bottle roll test recovery achieved at the 80%-75µm feed size (88.7%). Bottle roll test leach rate data (see Figures 2 and 3) show that gold recovery rate was slow, but increased with decreasing feed size and potentially with increasing cyanide concentration. Because bottle roll tests were terminated after 96 hours, gold recovery was limited by the slow leach rates of the tests conducted at coarse feed sizes.

It is believed that increasing the cyanide concentration did not significantly impact the gold or silver recovery. At feed sizes of 80%-6.3mm and 19mm, bottle roll tests were conducted at both 1.0 and 5.0 gNaCN/L concentrations. At the 19mm feed size, gold recovery was slightly higher at the elevated cyanide concentration. This difference is potentially due to an increased leach rate at the higher concentration. Gold recoveries at the 6.3mm feed size, and silver recoveries at both feed sizes were essentially the same, regardless of cyanide concentration.

COLUMN PERCOLATION LEACH TEST PROCEDURES AND RESULTS

Column percolation leach tests were conducted on each composite at an 80%-19mm feed size to determine gold and silver recoveries, recovery rates, and reagent requirements, under simulated heap leaching conditions.

Hydrated lime was mixed with the dry ore charges before column loading procedures. Lime additions were based on bottle roll test lime requirements. Ore charges were placed into the 15 cm diameter by 3 m high PVC leaching columns in a manner to minimize particle segregation and compaction.

Leaching was conducted by applying cyanide solution (1.0 gNaCN/L) over the ore charges at a rate of 12 Lph/m² of column cross-sectional area. Pregnant effluent solutions were collected each 24 hour period. Pregnant solution volumes were measured by weighing, and samples were taken for gold and silver analysis using A.A. methods. Cyanide concentration and pH were determined for each pregnant solution. Pregnant solutions were then pumped through a three

stage carbon circuit for adsorption of dissolved gold values. Barren solution volumes were measured and sampled for the same analyses. Barren solution, with appropriate make-up reagent, was applied to the ore charges daily.

After leaching, fresh water rinsing was conducted to remove residual cyanide (county requirement) and to recover dissolved gold and silver values. Moisture required to saturate the ore charges (in process solution inventory) and retained moisture were determined. Apparent ore bulk densities were measured before and after leaching.

Drain down rate tests were conducted after rinsing was complete. Tests were conducted by terminating solution application, and at that time, measuring drain volume. Drain volumes were collected and measured periodically by weighing until drain down was complete.

After leaching, rinsing, and draining, residues were removed from the columns and moisture samples were taken immediately. Leached residues were air dried, blended and split to obtain samples for a tail screen analysis. Head and tail screen analyses were conducted, using the same size fractions, to determine residual gold and silver content and distribution, and to obtain recovery by size fraction data.

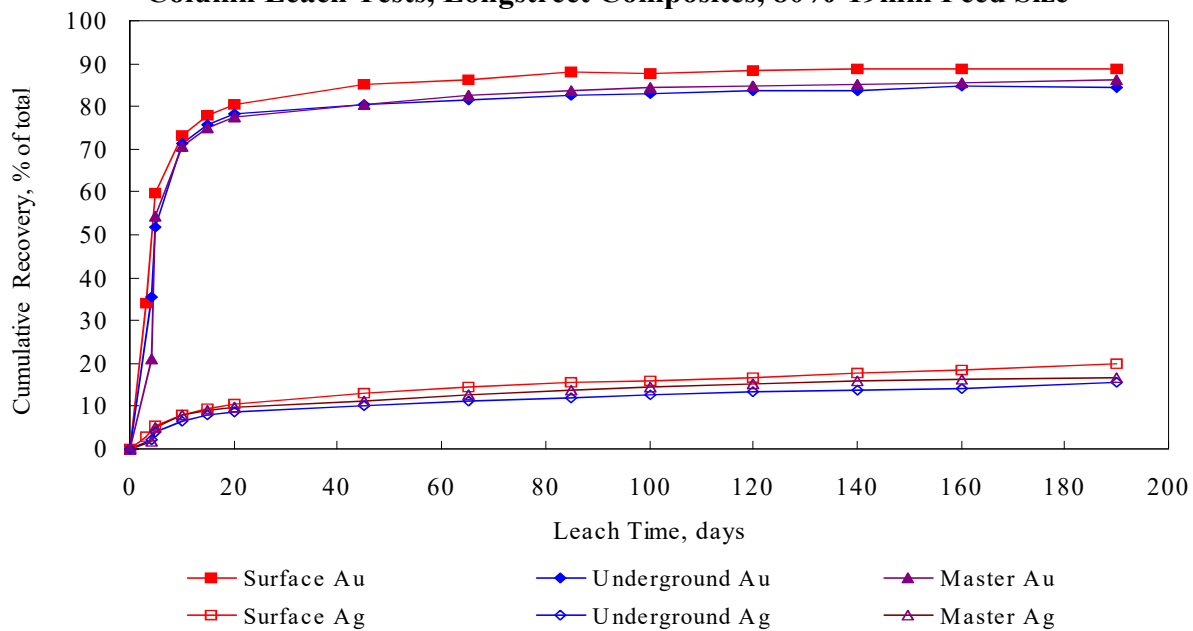
Overall metallurgical results from column tests are shown in Table 7. Gold leach rate profiles are shown graphically in Figure 5. Head and tail screen analysis results, and recovery by size fraction data are provided in Tables 8 through 10. Metallurgical balance data are provided in Table 11. Physical ore characteristics data are provided in Table 12. Detailed column leach test data, including results from drain down rate tests, are provided in Section 6 of the Appendix to this report.

**Table 7. - Overall Metallurgical Results, Column Leach Tests,
Longstreet Composites, 80%-19mm Feed Size**

Composite: Metallurgical Results	Surface (P-1)		Underground (P-2)		Master (P-3)	
Extraction: % of total	Au	Ag	Au	Ag	Au	Ag
1st Effluent	34.0	2.8	35.5	2.3	21.1	1.6
in 5 days	59.8	5.5	51.8	3.8	54.4	5.0
in 10 days	73.4	8.1	71.4	6.6	70.5	7.8
in 15 days	77.9	9.4	75.9	8.0	75.1	9.1
in 20 days	80.5	10.4	78.1	8.7	77.6	9.8
in 45 days	85.0	13.0	80.5	10.0	80.5	11.4
in 65 days	86.1	14.4	81.7	11.1	82.6	12.7
in 85 days	88.0	15.4	82.5	11.9	83.7	13.6
in 100 days	87.8	15.8	83.1	12.6	84.3	14.4
in 120 days	88.3	16.8	83.6	13.3	84.9	15.3
in 140 days	88.9	17.7	83.9	13.8	85.3	15.9
in 160 days	88.9	18.3	84.6	14.2	85.7	16.4
End of Leach/Rinse	88.9	20.0	84.6	15.4	86.3	16.7
Extracted, gAu/mt ore	0.32	5	0.66	8	0.63	8
Tail Screen, gAu/mt	0.04	20	0.12	44	0.10	40
Calculated Head, gAu/mt ore	0.36	25	0.78	52	0.73	48
Average Head, gAu/mt ore ¹⁾	0.38	25	0.84	59	0.68	44
NaCN Consumed, kg/mt ore	1.56		1.93		1.86	
Lime Added, kg/mt ore	1.7		2.7		2.0	
Final Solution pH	10.3		10.4		10.3	
pH After Rinse	10.2		10.5		10.2	
Leach/Rinse Cycle, Days	190		190		190	

1) Average of all head grade determinations.

**Figure 5. - Gold and Silver Leach Rate Profiles,
Column Leach Tests, Longstreet Composites, 80%-19mm Feed Size**



Overall metallurgical results show that the Longstreet composites were readily amenable to simulated heap leaching treatment, at the 80%-19mm feed size. Gold recoveries were similar for all three composites, and ranged from 84.6% to 88.9% in 190 days of leaching and rinsing.

Silver recoveries were also similar for all three composites, and ranged from 15.4% to 20.0%.

Gold leach rates were rapid and gold extraction was essentially complete in the first 30 days of leaching. Silver leach rates were slow, however, it is not expected that extending the leach cycles beyond 190 days would significantly improve recovery.

Cyanide consumption was high for each composite and ranged from 1.56 to 1.93 kgNaCN/mt ore. Cyanide consumption for column leach testing typically is significantly higher than observed during commercial heap leaching. Cyanide consumption was likely driven higher during column testing due to the long leach cycles employed.

The lime additions of 1.7 to 2.7 kg/mt ore were sufficient for maintaining protective alkalinity throughout the leaching cycle.

Table 8. - Head and Tail Screen Analysis Results, and Recovery by Size Fraction Data, Longstreet Surface Composite, 80%-19mm Feed Size

Size Fraction	Weight, %	Cum. Wt., %	Distribution					
			Assay, g/mt		Au		Ag	
			Au	Ag	%	Cum. %	%	Cum. %
HEAD SCREEN ANALYSIS RESULTS								
+19mm	20.5	20.5	0.27	27	16.5	16.5	22.1	22.1
-19mm+12.5mm	31.3	51.8	0.35	28	32.6	49.1	35.1	57.2
-12.5mm+6.3mm	23.2	75.0	0.27	24	18.6	67.7	22.3	79.5
-6.3mm+1.7mm	15.1	90.1	0.36	22	16.2	83.9	13.3	92.8
-1.7mm+850µm	4.0	94.1	0.26	21	3.1	87.0	3.4	96.2
-850µm+425µm	1.3	95.4	0.30	19	1.2	88.2	1.0	97.2
-425µm+150µm	1.5	96.9	0.31	17	1.4	89.6	1.0	98.2
-150µm+75µm	0.7	97.6	0.50	16	1.0	90.6	0.4	98.6
-75µm	2.4	100.0	1.32	15	9.4	100.0	1.4	100.0
Composite	100.0		0.34	25	100.0		100.0	
TAIL SCREEN, COLUMN LEACHED RESIDUES (P-1)								
+19mm	19.8	19.8	0.03	18	15.3	15.3	17.8	17.8
-19mm+12.5mm	37.0	56.8	0.04	25	38.0	53.3	46.2	64.0
-12.5mm+6.3mm	21.6	78.4	0.03	20	16.7	70.0	21.6	85.6
-6.3mm+1.7mm	12.1	90.5	0.07	17	21.8	91.8	10.3	95.9
-1.7mm+850µm	2.7	93.2	0.04	16	2.8	94.6	2.1	98.0
-850µm+425µm	1.5	94.7	0.03	11	1.2	95.8	0.8	98.8
-425µm+150µm	1.5	96.2	0.03	9	1.1	96.9	0.7	99.5
-150µm+75µm	0.8	97.0	0.04	6	0.8	97.7	0.2	99.7
-75µm	3.0	100.0	0.03	2	2.3	100.0	0.3	100.0
Composite	100.0		0.04	20	100.0		100.0	
RECOVERY BY SIZE FRACTION DATA, COLUMN LEACH TEST (P-1)								
Size Fraction	Weight, %		Assay, g/mt				Recovery, %	
	Head	Tail	Au		Ag		Au	Ag
			Head	Tail	Head	Tail		
+19mm	20.5	19.8	0.27	0.03	27	18	88.9	33.3
-19mm+12.5mm	31.3	37.0	0.35	0.04	28	25	88.6	10.7
-12.5mm+6.3mm	23.2	21.6	0.27	0.03	24	20	88.9	16.7
-6.3mm+1.7mm	15.1	12.1	0.36	0.07	22	17	80.6	22.7
-1.7mm+850µm	4.0	2.7	0.26	0.04	21	16	84.6	23.8
-850µm+425µm	1.3	1.5	0.30	0.03	19	11	90.0	42.1
-425µm+150µm	1.5	1.5	0.31	0.03	17	9	90.3	47.1
-150µm+75µm	0.7	0.8	0.50	0.04	16	6	92.0	62.5
-75µm	2.4	3.0	1.32	0.03	15	2	97.7	86.7
Composite	100.0	100.0	0.34	0.04	25	20	88.2	19.9

**Table 9. - Head and Tail Screen Analysis Results, and Recovery by Size Fraction Data,
Longstreet Underground Composite, 80%-19mm Feed Size**

Size Fraction	Weight, %	Cum. Wt., %	Assay, g/mt		Distribution			
					Au		Ag	
			Au	Ag	%	Cum. %	%	Cum. %
HEAD SCREEN ANALYSIS RESULTS								
+19mm	19.8	19.8	0.85	64	20.5	20.5	19.5	19.5
-19mm+12.5mm	30.8	50.6	0.67	70	25.1	45.6	33.3	52.8
-12.5mm+6.3mm	20.2	70.8	0.65	62	16.0	61.6	19.3	72.1
-6.3mm+1.7mm	14.0	84.8	0.86	74	14.7	76.3	16.0	88.1
-1.7mm+850µm	3.4	88.2	0.83	65	3.4	79.7	3.4	91.5
-850µm+425µm	2.1	90.3	0.79	59	2.0	81.7	1.9	93.4
-425µm+150µm	2.5	92.8	0.78	47	2.4	84.1	1.8	95.2
-150µm+75µm	1.3	94.1	1.04	54	1.6	85.7	1.1	96.3
-75µm	5.9	100.0	1.99	41	14.3	100.0	3.7	100.0
Composite	100.0		0.82	65	100.0		100.0	
TAIL SCREEN, COLUMN LEACHED RESIDUES (P-2)								
+19mm	21.9	21.9	0.11	39	20.9	20.9	19.2	19.2
-19mm+12.5mm	27.4	49.3	0.10	41	23.8	44.7	25.3	44.5
-12.5mm+6.3mm	19.9	69.2	0.13	49	22.5	67.2	22.0	66.5
-6.3mm+1.7mm	14.2	83.4	0.12	64	14.8	82.0	20.5	87.0
-1.7mm+850µm	3.3	86.7	0.12	53	3.4	85.4	3.9	90.9
-850µm+425µm	2.3	89.0	0.12	48	2.4	87.8	2.5	93.4
-425µm+150µm	2.2	91.2	0.12	36	2.3	90.1	1.8	95.2
-150µm+75µm	1.3	92.5	0.12	25	1.4	91.5	0.7	95.9
-75µm	7.5	100.0	0.13	24	8.5	100.0	4.1	100.0
Composite	100.0		0.12	44	100.0		100.0	
RECOVERY BY SIZE FRACTION DATA, COLUMN LEACH TEST (P-2)								
Size Fraction	Weight, %		Assay, g/mt				Recovery, %	
	Head	Tail	Au		Ag		Au	Ag
			Head	Tail	Head	Tail		
+19mm	19.8	21.9	0.85	0.11	64	39	87.1	39.1
-19mm+12.5mm	30.8	27.4	0.67	0.10	70	41	85.1	41.4
-12.5mm+6.3mm	20.2	19.9	0.65	0.13	62	49	80.0	21.0
-6.3mm+1.7mm	14.0	14.2	0.86	0.12	74	64	86.0	13.5
-1.7mm+850µm	3.4	3.3	0.83	0.12	65	53	85.5	18.5
-850µm+425µm	2.1	2.3	0.79	0.12	59	48	84.8	18.6
-425µm+150µm	2.5	2.2	0.78	0.12	47	36	84.6	23.4
-150µm+75µm	1.3	1.3	1.04	0.12	54	25	88.5	53.7
-75µm	5.9	7.5	1.99	0.13	41	24	93.5	41.5
Composite	100.0	100.0	0.82	0.12	65	44	85.4	31.6

Table 10. - Head and Tail Screen Analysis Results, and Recovery by Size Fraction Data, Longstreet Master Composite, 80%-19mm Feed Size

Size Fraction	Weight, %	Cum. Wt., %	Assay, g/mt		Distribution			
					Au		Ag	
			Au	Ag	%	Cum. %	%	Cum. %
HEAD SCREEN ANALYSIS RESULTS								
+19mm	16.8	16.8	1.02	43	21.7	21.7	15.3	15.3
-19mm+12.5mm	28.3	45.1	0.42	45	15.0	36.7	27.0	42.3
-12.5mm+6.3mm	20.2	65.3	0.90	47	23.0	59.7	20.1	62.4
-6.3mm+1.7mm	16.4	81.7	0.66	57	13.7	73.4	19.9	82.3
-1.7mm+850µm	4.8	86.5	0.71	53	4.3	77.7	5.4	87.7
-850µm+425µm	3.0	89.5	0.64	48	2.4	80.1	3.1	90.8
-425µm+150µm	2.9	92.4	0.64	44	2.4	82.5	2.7	93.5
-150µm+75µm	1.5	93.9	1.08	37	2.0	84.5	1.2	94.7
-75µm	6.1	100.0	2.01	41	15.5	100.0	5.3	100.0
Composite	100.0		0.79	47	100.0		100.0	
TAIL SCREEN, COLUMN LEACHED RESIDUES (P-3)								
+19mm	21.0	21.0	0.06	41	12.9	12.9	21.3	21.3
-19mm+12.5mm	35.2	56.2	0.13	43	47.0	59.9	37.4	58.7
-12.5mm+6.3mm	20.8	77.0	0.07	35	15.0	74.9	18.0	76.7
-6.3mm+1.7mm	12.0	89.0	0.08	50	9.9	84.8	14.8	91.5
-1.7mm+850µm	2.8	91.8	0.10	46	2.9	87.7	3.2	94.7
-850µm+425µm	1.3	93.1	0.11	37	1.5	89.2	1.2	95.9
-425µm+150µm	1.6	94.7	0.10	29	1.6	90.8	1.2	97.1
-150µm+75µm	1.0	95.7	0.12	22	1.2	92.0	0.6	97.7
-75µm	4.3	100.0	0.18	22	8.0	100.0	2.3	100.0
Composite	100.0		0.10	40	100.0		100.0	
RECOVERY BY SIZE FRACTION DATA, COLUMN LEACH TEST (P-3)								
Size Fraction	Weight, %		Assay, g/mt				Recovery, %	
	Head	Tail	Au		Ag		Au	Ag
			Head	Tail	Head	Tail		
+19mm	16.8	21.0	1.02	0.06	43	41	94.1	4.7
-19mm+12.5mm	28.3	35.2	0.42	0.13	45	43	69.0	4.4
-12.5mm+6.3mm	20.2	20.8	0.90	0.07	47	35	92.2	25.5
-6.3mm+1.7mm	16.4	12.0	0.66	0.08	57	50	87.9	12.3
-1.7mm+850µm	4.8	2.8	0.71	0.10	53	46	85.9	13.2
-850µm+425µm	3.0	1.3	0.64	0.11	48	37	82.8	22.9
-425µm+150µm	2.9	1.6	0.64	0.10	44	29	84.4	34.1
-150µm+75µm	1.5	1.0	1.08	0.12	37	22	88.9	40.5
-75µm	6.1	4.3	2.01	0.18	41	22	91.0	46.3
Composite	100.0	100.0	0.79	0.10	47	40	87.3	14.2

Head screen analysis results showed that the Longstreet composites contained 0.34 to 0.82 gAu/mt ore and 25 to 65 gAg/mt ore. Contained gold and silver values generally were fairly evenly distributed throughout the various size fractions, with a moderate enrichment of gold values in the fines (-150µm). Size fraction assays for the master composite were somewhat “spotty”.

Tail screen analysis results showed that the leached residues contained 0.04 to 0.12 gAu/mt ore and 20 to 44 gAg/mt ore. Residual gold values generally were fairly evenly distributed throughout the various size fractions. Silver values in the leach residues were not evenly distributed and a minor to moderate depletion of silver values was noted in the finer size fractions.

Recovery by size fraction data indicate that finer crushing or grinding would not substantially improve gold recovery. Gold recovery generally was similar throughout the various size fractions with only a slightly elevated recovery noted in the finest size fraction (-75µm).

Recovery by size fraction data indicate that fine grinding would likely be required to maximize silver recovery from the surface and master composites. Silver recovery by size fraction data for the underground composite show that silver recovery was similar throughout the various size fractions, and that fine grinding may not significantly improve silver recovery from this composite.

**Table 11. - Metallurgical Balances, Column Leach Tests,
Longstreet Composites, 80%-19mm Feed Size**

	Metallurgical Balance		
	Sol. vs. Tail	Carbon vs. Tail	Head vs. Tail ²⁾
Surface Composite (P-1)			
Extracted, gAu/mt ore	0.32	0.33	0.34
Tail Assay, gAu/mt	0.04	0.04	0.04
Calculated, Head, gAu/mt	0.36	0.37	0.38
Recovery, %	88.9	89.2	89.5
Deviation, g/mt ¹⁾	N/A	0.01	0.02
Precision, %	100.0	97.2	94.4
Underground Composite (P-2)			
Extracted, gAu/mt ore	0.66	0.69	0.72
Tail Assay, gAu/mt	0.12	0.12	0.12
Calculated, Head, gAu/mt	0.78	0.81	0.84
Recovery, %	84.6	85.2	85.7
Deviation, g/mt ¹⁾	N/A	0.03	0.06
Precision, %	100.0	96.2	92.3
Master Composite (P-3)			
Extracted, gAu/mt ore	0.63	0.63	0.58
Tail Assay, gAu/mt	0.10	0.10	0.10
Calculated, Head, gAu/mt	0.73	0.73	0.68
Recovery, %	86.3	86.3	85.3
Deviation, g/mt ¹⁾	N/A	<0.01	0.05
Precision, %	100.0	>98.6	93.2

1) Deviation from solution versus tail balance.

2) Calculated, based on average of all head grades and tail screen results.

Solution versus tail and loaded carbon versus tail metallurgical balances agreed closely. Deviation between the two balances was 0.03 gAu/mt ore or less. Head versus tail metallurgical balances agreed closely as well. Deviation between these and the solution versus tail balances ranged from 0.02 to 0.06 gAu/mt ore. These deviations are all within normally expected experimental and analytical precision limits for column leach tests. The solution versus tail balance is considered the most reliable because of the number of check analyses performed on column test pregnant solutions. That balance was used for all percent recovery calculations.

**Table 12. - Physical Ore Characteristics Data,
Column Leach Tests, Longstreet Composites**

Composite	Feed Size	Test No.	Ore Charge, kg	Moisture, wt. %			Apparent Bulk Density, mt ore/m ³	
				As Rec'd.	To Saturate*	Retained	Before	After
Surface	80%-19mm	P-1	71.97	0.1	14.3	6.8	1.35	1.36
Underground	80%-19mm	P-2	71.75	0.1	21.5	11.6	1.35	1.35
Master	80%-19mm	P-3	71.95	0.1	14.5	9.8	1.36	1.36

* Calculated on a dry ore weight basis.

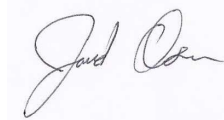
Physical ore characteristics data show that little or no “slumping” of charges occurred during leaching. Moisture requirements were not unusual for 19mm feeds. No solution percolation, fines migration or solution channeling problems were encountered during leaching.

CONCLUSIONS

- Each of the three Longstreet composites was readily amenable to simulated heap leach cyanidation treatment at an 80%-19mm feed size. Gold recoveries of 84.6% to 88.9% were achieved.
- Column test silver recoveries were low and ranged from 15.4% to 20.0%.
- Gold recovery rates, during column leach testing, were rapid. Silver recovery rates were slow.
- Bottle roll test results and column test recovery by size fraction data indicate that fine grinding would likely be required to maximize silver recovery. A silver recovery of 60.6% was achieved by milling/cyanidation treatment (80%-75µm feed size).
- Results indicate that, within the range of feed sizes tested, the master composite was not particularly sensitive to feed size, with respect to gold recovery, given a sufficiently long leach cycle.
- Increasing the cyanide concentration from 1.0 to 5.0 gNaCN/L did not significantly improve silver bottle roll test recoveries at the 19mm or 6.3mm feed sizes.
- Cyanide consumption during column leach testing was high and ranged from 1.56 to 1.93 kgNaCN/mt ore. Cyanide consumption was significantly lower during bottle roll testing, and ranged from <0.07 to 0.13 kgNaCN/mt ore when a cyanide concentration of 1.0 gNaCN/L was used.
- Lime requirements generally were low.

RECOMMENDATIONS

We recommend that load/permeability testing be conducted on column leached residue samples to confirm permeability under simulated commercial heap leach stack height compressive loadings.

A handwritten signature in black ink, appearing to read "Jared Olson", is centered on the page.

Jared R. Olson
Metallurgist / Project Manager

JRO:mh

APPENDIX

Section 1 - Section Sample Assay Results

Section 2 - Composite Make-up Information

Section 3 - Comminution Testing Results

Section 4 - Mineralogical Analysis Report

Section 5 - Detailed Bottle Roll Test Results

Section 6 - Detailed Column Leach Test Results

APPENDIX

Section 1 - Section Sample Assay Results

Table A1-1. - Gold and Silver Head Assay Results, Longstreet Mine, Section Samples

Sample	Au, g/mt ore			Ag, g/mt ore		
	Init.	Dup.	Average	Init.	Dup.	Average
LUA 10'	0.93	0.83	0.88	66	110	88
LUA 20'	2.10	1.89	2.00	173	125	149
LUA 30'	1.91	2.22	2.07	141	177	159
LUA 40'	2.71	3.28	2.99	182	214	198
LUA 50'	3.09	3.18	3.14	202	281	242
LUA 60'	2.65	3.86	3.25	117	89	103
LUA 180' to 190'	0.10	0.06	0.08	11	9	10
LUA 190' to 200'	0.09	0.07	0.08	8	10	9
LUA 200' to 210'	0.39	0.36	0.37	16	14	15
LUA 210' to 220'	0.24	0.17	0.20	7	5	6
LUA 220' to 230'	0.15	0.21	0.18	4	4	4
LUA 230' to 240'	0.29	0.23	0.26	17	13	15
LUA 240' to 250'	0.33	0.38	0.36	20	21	21
LUA 250' to 260'	0.24	0.56	0.40	8	11	10
LUA 260' to 270'	0.28	0.21	0.24	11	8	10
LUA 270' to 280'	0.16	0.20	0.18	7	8	8
LUA 280' to 290'	0.24	0.29	0.27	6	7	7
LUA 290' to 300'	0.10	0.15	0.12	5	6	6
LUA 300' to 310'	0.28	0.27	0.27	6	7	7
LS1	0.16	0.10	0.13	12	11	12
LS2	0.25	0.35	0.30	14	15	15
LS3	0.61	0.48	0.54	34	32	33

APPENDIX

Section 2 - Composite Make-up Information

**Table A2-1. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 10' (LUA 10')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	10	-	*6.05	30.8
LUA 2 OF 2	10	-	6.51	33.1
LUA 3 OF 3	10	-	7.10	36.1
Composite Total			19.66	100.0

*Comminution sample taken

**Table A2-2. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 20' (LUA 20')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 2	20	-	4.29	51.0
LUA 2 OF 2	20	-	*4.12	49.0
Composite Total			8.41	100.0

*Comminution sample taken

**Table A2-3. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 30' (LUA 30')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 2	30	-	*4.30	39.4
LUA 2 OF 2	30	-	6.61	60.6
Composite Total			10.91	100.0

*Comminution sample taken

**Table A2-4. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 40' (LUA 40')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	40	-	6.31	35.9
LUA 2 OF 3	40	-	*5.71	32.5
LUA 3 OF 3	40	-	5.55	31.6
Composite Total			17.57	100.0

*Comminution sample taken

**Table A2-5. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 50' (LUA 50')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	50	-	*4.85	35.1
LUA 2 OF 3	50	-	4.99	36.1
LUA 3 OF 3	50	-	3.98	28.8
Composite Total			13.82	100.0

*Comminution sample taken

**Table A2-6. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 60' (LUA 60')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	60	-	*5.88	33.0
LUA 2 OF 3	60	-	5.68	31.8
LUA 3 OF 3	60	-	6.28	35.2
Composite Total			17.84	100.0

*Comminution sample taken

**Table A2-7. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 180' to 190' (LUA 180' to 190')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 4	180	190	5.36	26.6
LUA 2 OF 4	180	190	7.43	36.8
LUA 3 OF 4	180	190	*4.43	22.0
LUA 4 OF 4	180	190	2.95	14.6
Composite Total			20.17	100.0

*Comminution sample taken

**Table A2-8. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 190' to 200' (LUA 190' to 200')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	190	200	5.04	31.6
LUA 2 OF 3	190	200	4.94	31.0
LUA 3 OF 3	190	200	5.95	37.4
Composite Total			15.93	100.0

**Table A2-9. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 200' to 210' (LUA 200' to 210')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 4	200	210	*4.80	25.5
LUA 2 OF 4	200	210	5.17	27.5
LUA 3 OF 4	200	210	5.13	27.3
LUA 4 OF 4	200	210	3.70	19.7
Composite Total			18.8	100.0

*Comminution sample taken

**Table A2-10. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 210' to 220' (LUA 210' to 220')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 4	210	220	*5.23	25.3
LUA 2 OF 4	210	220	6.56	31.7
LUA 3 OF 4	210	220	5.09	24.6
LUA 4 OF 4	210	220	3.82	18.5
Composite Total			20.7	100.0

*Comminution sample taken

**Table A2-11. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 220' to 230' (LUA 220' to 230')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	220	230	6.04	42.4
LUA 2 OF 3	220	230	5.41	38.0
LUA 3 OF 3	220	230	2.78	19.5
Composite Total			14.23	100.0

**Table A2-12. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 230' to 240' (LUA 230' to 240')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 6	230	240	4.60	13.4
LUA 2 OF 6	230	240	5.74	16.7
LUA 3 OF 6	230	240	5.55	16.1
LUA 4 OF 6	230	240	5.42	15.7
LUA 5 OF 6	230	240	7.37	21.4
LUA 6 OF 6	230	240	5.75	16.7
Composite Total			34.43	100.0

**Table A2-13. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 240' to 250' (LUA 240' to 250')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	240	250	*5.16	26.2
LUA 2 OF 3	240	250	7.25	36.8
LUA 3 OF 3	240	250	7.30	37.0
Composite Total			19.71	100.0

*Comminution sample taken

**Table A2-14. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 250' to 260' (LUA 250' to 260')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	250	260	6.10	37.5
LUA 2 OF 3	250	260	5.45	33.5
LUA 3 OF 3	250	260	4.72	29.0
Composite Total			16.27	100.0

**Table A2-15. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 260' to 270' (LUA 260' to 270')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 4	260	270	5.80	25.7
LUA 2 OF 4	260	270	5.57	24.6
LUA 3 OF 4	260	270	6.00	26.5
LUA 4 OF 4	260	270	5.23	23.1
Composite Total			22.6	100.0

**Table A2-16. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 270' to 280' (LUA 270' to 280')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	270	280	6.33	42.1
LUA 2 OF 3	270	280	5.61	37.3
LUA 3 OF 3	270	280	3.11	20.7
Composite Total			15.05	100.0

**Table A2-17. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 280' to 290' (LUA 280' to 290')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 3	280	290	5.53	34.1
LUA 2 OF 3	280	290	6.35	39.1
LUA 3 OF 3	280	290	4.35	26.8
Composite Total			16.23	100.0

**Table A2-18. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 290' to 300' (LUA 290' to 300')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 4	290	300	5.69	26.4
LUA 2 OF 4	290	300	6.49	30.1
LUA 3 OF 4	290	300	7.03	32.6
LUA 4 OF 4	290	300	2.38	11.0
Composite Total			21.59	100.0

**Table A2-19. - Composite Make-up Information, Longstreet Project,
Underground Adit Sample at 300' to 310' (LUA 300' to 310')**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LUA 1 OF 5	300	310	5.62	20.3
LUA 2 OF 5	300	310	6.63	23.9
LUA 3 OF 5	300	310	5.49	19.8
LUA 4 OF 5	300	310	5.00	18.0
LUA 5 OF 5	300	310	4.99	18.0
Composite Total			27.73	100.0

**Table A2-20. - Composite Make-up Information, Longstreet Project,
Surface Sample 1 (LS1)**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LS-#1 1 OF 6	-	-	*19.26	15.2
LS-#1 2 OF 6	-	-	*20.87	16.5
LS-#1 3 OF 6	-	-	24.61	19.4
LS-#1 4 OF 6	-	-	*19.81	15.6
LS-#1 5 OF 6	-	-	19.88	15.7
LS-#1 6 OF 6	-	-	22.34	17.6
Composite Total			126.77	100.0

*Comminution sample taken

**Table A2-21. - Composite Make-up Information, Longstreet Project,
Surface Sample 2 (LS2)**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LS-#2 1 OF 6	-	-	*20.74	14.5
LS-#2 2 OF 6	-	-	*22.49	15.7
LS-#2 3 OF 6	-	-	*26.44	18.5
LS-#2 4 OF 6	-	-	*23.96	16.8
LS-#2 5 OF 6	-	-	24.38	17.1
LS-#2 6 OF 6	-	-	24.85	17.4
Composite Total			142.86	100.0

*Comminution sample taken

**Table A2-22. - Composite Make-up Information, Longstreet Project,
Surface Sample 3 (LS3)**

Sample ID	Intervals, ft		Weight to Comp.	
	From	To	kg	%
LS-#3 1 OF 6	-	-	*20.88	15.2
LS-#3 2 OF 6	-	-	*22.77	16.5
LS-#3 3 OF 6	-	-	25.18	18.3
LS-#3 4 OF 6	-	-	*20.24	14.7
LS-#3 5 OF 6	-	-	24.72	18.0
LS-#3 6 OF 6	-	-	23.83	17.3
Composite Total			137.62	100.0

*Comminution sample taken

**Table A2-23. - Composite Make-up Information, Longstreet Project,
Underground Master Composite (UMC)**

Sample ID	Weight to Comp.	
	kg	%
LUA 10'	11.18	5.5
LUA 20'	6.36	3.1
LUA 30'	8.78	4.3
LUA 40'	11.07	5.5
LUA 50'	11.07	5.5
LUA 60'	11.09	5.5
LUA 180' to 190'	10.89	5.4
LUA 190' to 200'	10.99	5.4
LUA 200' to 210'	10.93	5.4
LUA 210' to 220'	10.94	5.4
LUA 220' to 230'	11.06	5.5
LUA 230' to 240'	11.01	5.4
LUA 240' to 250'	11.00	5.4
LUA 250' to 260'	11.01	5.4
LUA 260' to 270'	11.00	5.4
LUA 270' to 280'	10.90	5.4
LUA 280' to 290'	10.94	5.4
LUA 290' to 300'	10.96	5.4
LUA 300' to 310'	11.03	5.5
Composite Total	202.21	100.0

**Table A2-24. - Composite Make-up Information, Longstreet Project,
Surface Master Composite (SMC)**

Sample ID	Weight to Comp.	
	kg	%
LS1	64.94	33.5
LS2	64.59	33.3
LS3	64.56	33.3
Composite Total	194.09	100.0

**Table A2-25. - Composite Make-up Information, Longstreet Project,
Blended Master Composite (BMC)**

Sample ID	Weight to Comp.	
	kg	%
UMC	73.70	54.4
SMC	61.80	45.6
Composite Total	135.50	100.0

**Table A2-26. - Composite Make-up Information, Longstreet Project,
Comminution Testing Composite, 50mm to 75mm Rock Pieces**

Sample	Weight to Comp.	
	kg	%
LUA 10'	0.45	3.5
LUA 20'	0.64	5.0
LUA 30'	0.49	3.8
LUA 40'	0.56	4.4
LUA 50'	0.67	5.3
LUA 60'	0.77	6.0
LUA 180' - 190'	0.63	4.9
LUA 200' - 210'	0.56	4.4
LUA 210' - 220'	0.36	2.8
LUA 240' - 250'	0.96	7.5
LS1	0.59	4.6
LS1	0.71	5.6
LS1	0.78	6.1
LS2	0.58	4.6
LS2	0.43	3.4
LS2	0.80	6.3
LS2	0.56	4.4
LS3	0.92	7.2
LS3	0.78	6.1
LS3	0.49	3.8
Composite Total	12.73	100.0

APPENDIX

Section 3 - Comminution Testing Results

PHILLIPS ENTERPRISES, LLC

Metallurgical Testing and Consulting Services

Mailing Address:
2501 Braun Dr.
Golden, CO 80401
(303) 279-0443

Laboratory and Shipping:
5946 McIntyre Street
Golden, CO 80403
Phone: (303) 854-2037
Fax: (303) 216-0258
E-mail: phillips81@att.net

August 16, 2013

Jared Olson
Project Engineer
MLI
1016 Greg St.
Sparks, NV 89431

Subject: Comminution Tests
Reference: Project 3829; Longstreet Mine
Phillips Project No.: 133017

Dear Mr. Olson:

Phillips Enterprises, LLC has completed a crusher work index test and an abrasion index test for your Project 3829. The sample was delivered on July 31, 2013.

Sample Preparation

No preparation was required for the crusher test sample. Pieces were natural rock. Fragments were used for the abrasion test.

The abrasion test sample was crushed and screened to extract a $\frac{3}{4}$ " x $\frac{1}{2}$ " size fraction.

Crusher Work Index Test

The crusher work index test was conducted on a natural rock pieces according to test protocol.

Sample	CWi (kW-hr/st)	CWi (kW-hr/mt)
Longstreet Mine	10.08	11.11

≡ PHILLIPS ENTERPRISES LLC ≡

Abrasion Index Test

An abrasion index test was conducted on a -3/4"+1/2" fraction of the sample according to test protocol.

Sample	Abr. Index
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Longstreet Mine	0.2431
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Data pages are attached for all tests. Please feel welcome to contact us if you have questions.

The sample rejects and products will be returned to MLI in the near future.

Sincerely,

Robert J. Phillips
Owner/manager

APPENDIX

Section 4 - Mineralogical Analysis Report



December 23, 2013

Mr. Jared Olson
McClelland Laboratories, Inc.
1016 Greg Street
Sparks, NV 89431

Dear Mr. Olson:

We have performed analyses of two leach samples (client samples **3829 BMC Mineralogy** and **3829 CY-7**). The results are outlined in the following report.

Thank you for the opportunity to provide this service. If you have any questions, please call.

Sincerely,

A handwritten signature in black ink that reads "Ron Schott". The signature is fluid and cursive, with the first name "Ron" and last name "Schott" clearly distinguishable.

Ron Schott
Laboratory Director



12421 W. 49th Avenue, Unit #6
Wheat Ridge, CO 80033 (303) 463-8270

Petrographic, Scanning Electron Microscopy and X-Ray Diffraction Analysis
Page 1 of 19

Client:	Analysis Date:	12-20-13
McClelland Laboratories, Inc.	Reporting Date:	12-23-13
1016 Greg Street	Receipt Date:	12-13-13
Sparks, NV 89431	Client Job No.:	3829
	Project Title:	None Given
	DCMSL Project:	MCCL17,18,20

The purpose of this study is to determine the bulk mineralogy of two leach samples (client samples **3829 BMC Mineralogy** and **3829 CY-7**) with an emphasis on Au/Ag mineralogy. Each sample was prepared as a standard polished thin section for study by transmitted/reflected light microscopy and scanning electron microscopy (SEM) equipped with an energy dispersive system (EDS). In addition both samples were prepared as standard powder packs for analysis by x-ray diffraction (XRD). Color photomicrographs, SEM images and spectra are included for documentation of relevant features.

Mineral Assemblage

Petrographic, SEM and XRD analyses indicate the bulk mineralogy in both samples is essentially the same, therefore the mineralogy will be discussed together under individual category headings.

Silicate Mineralogy

In thin section silica mineralogy dominates both samples and is contained in fragments of porphyritic rhyolite. Quartz is the dominant phase and occurs in several habits. Secondary quartz seen filling fractures generally has a granular, mosaic texture with a grain size up to 100 μm . In the rhyolite matrix quartz takes on bipyramidal outlines that generally show deeply embayed margins. These grains float in a very fine grained matrix of quartz/feldspar and measure up to 1 mm in size. In the presence of vugs quartz takes on euhedral and prismatic outlines. Many of the rhyolite fragments are in part glassy and show vesicular glass structures preserved by divitrification and replacement by fine grained quartz. Sanidine feldspar occurs as subhedral to euhedral water clear phenocrysts. Grain size varies greatly from 20 μm laths to larger phenocrysts over 1.5 mm. Feldspar spherulites are also a common feature. Along some fractures intermingled with granular quartz, small grains of secondary adularia are also present. Clay mineralogy is present in low amounts and is primarily vermicular kaolinite that is seen filling small vugs. Some of the sanidine and matrix glass show mild replacement by a fine grained, brown colored clay with a sinuous habit.

Oxide Mineralogy

Both samples contain low amounts of oxide with hematite and goethite as the main types. Hematite occurs as small rosettes, thin strings and small pockets. Goethite is generally seen as euhedral pseudomorphs after pyrite. Yellow limonitic iron oxide is in the form of irregularly shaped masses or intermixed with kaolinite. Secondary rutile forms small aggregates and honey colored prisms in quartz.

Sulfate Mineralogy

Sulfates are well represented with jarosite as the main type. Jarosite occurs as granular masses mixed with iron oxide or as pseudomorphs after pyrite. Jarosite is also seen filling fractures that cross cut rock fragments. Barite is present as a trace and occurs as small inclusions up to 2 μm in quartz.

Phosphate Mineralogy

Phosphates are present as a trace and are mainly apatite and lesser amounts of a monazite mineral carrying typical rare earth chemistry. The majority of grains are locked in quartz/feldspar and measure up to 5 μm .

Sulfide Mineralogy

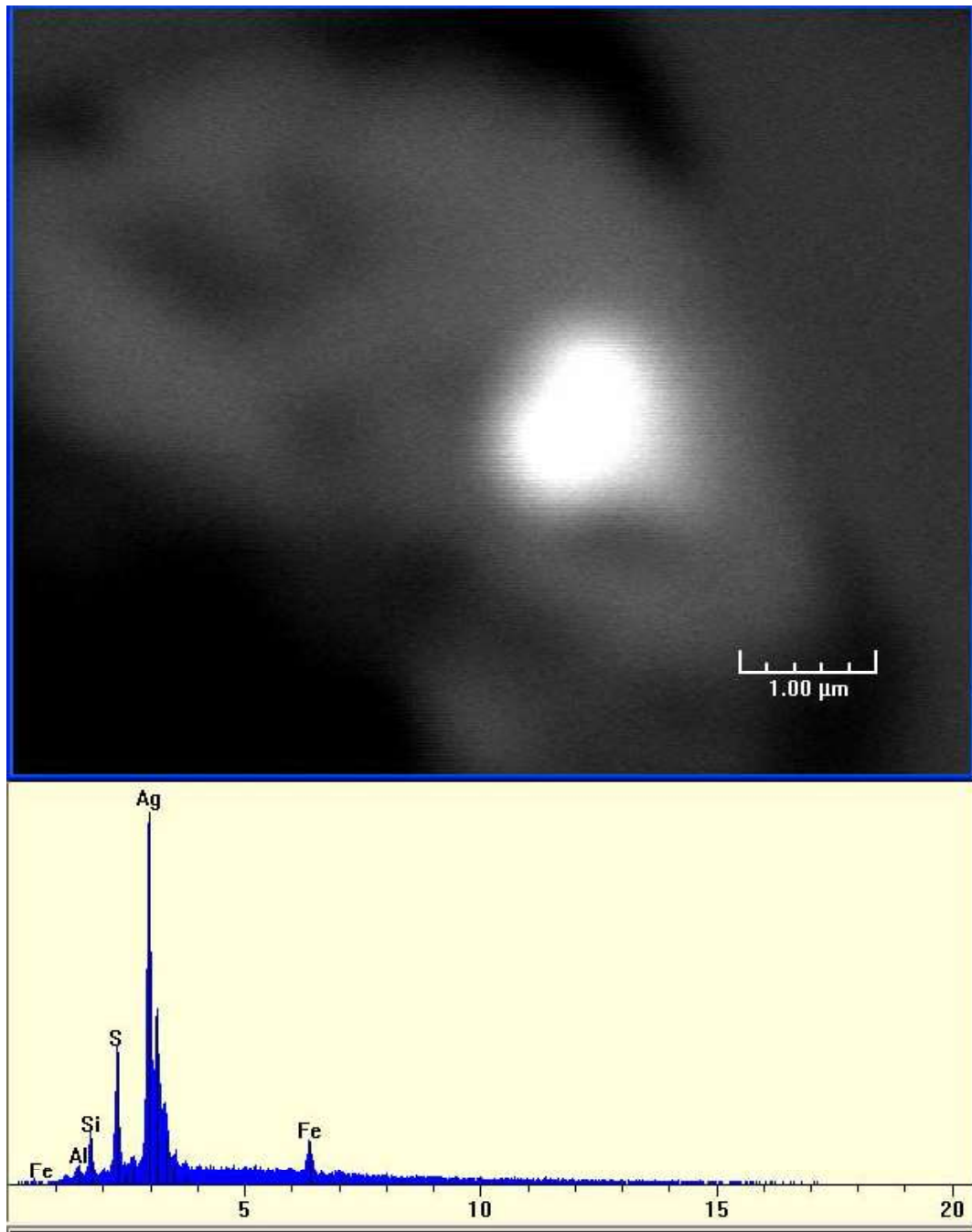
Sulfides are present as a trace with pyrite as the main type. Pyrite occurs as minute cubes and drop-like grains that vary in size from $<1\ \mu\text{m}$ up to approximately 20 μm . Most grains are unaltered but a small population wear thin goethite jackets. A trace of chalcopyrite is present and shows no apparent decay.

Au/Ag Mineralogy

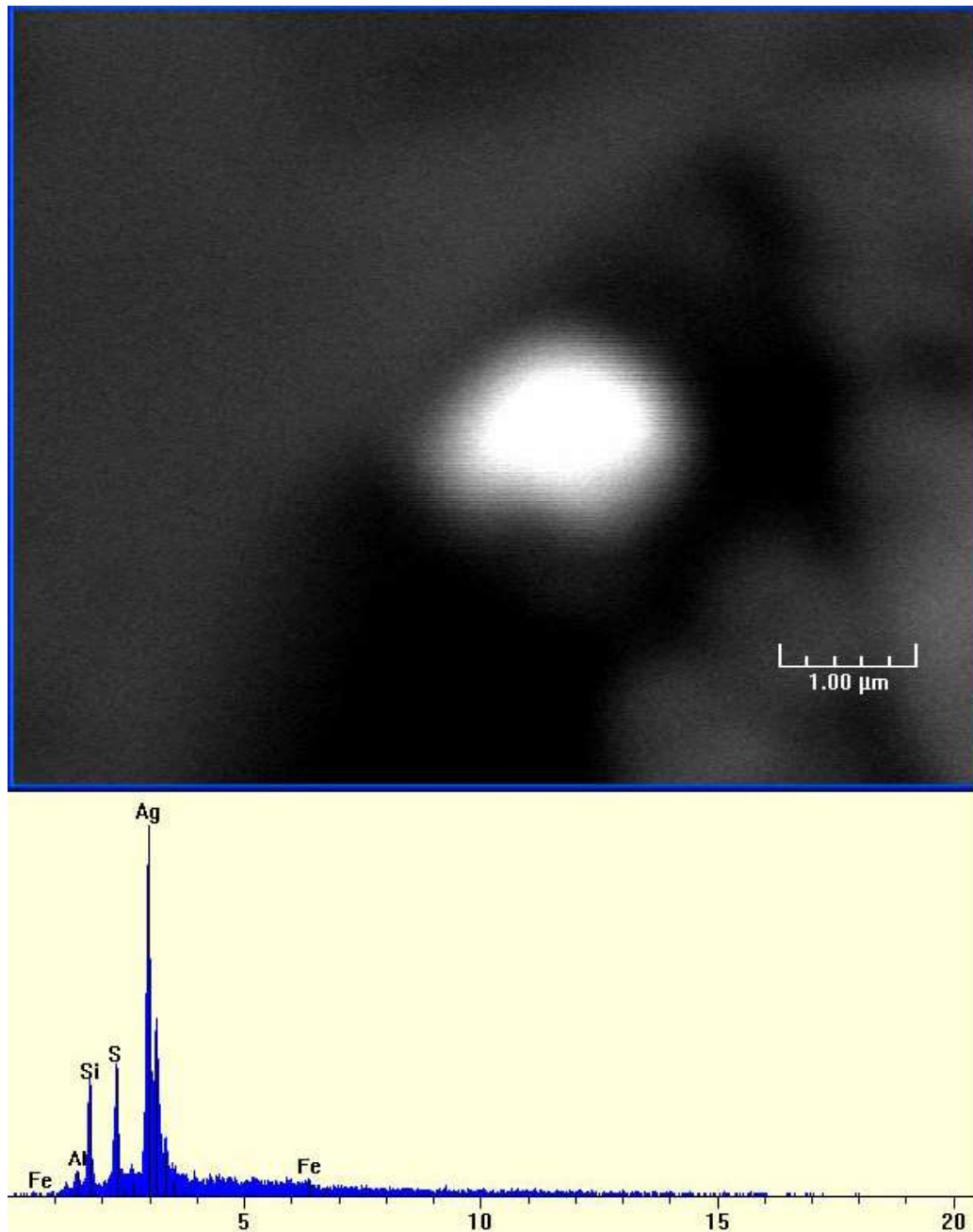
An extensive search using SEM/EDS indicates Ag phases are present in both samples, however, Au was not identified. Silver sulfide is the main Ag phase and occurs as irregularly shaped inclusions in quartz, pyrite and goethite pseudomorphs after pyrite. Cube-like grains are also seen in quartz and likely represent pseudomorphs of acanthite after argentite. Grain size of the Ag sulfide is very fine with measurements that range from 0.5 μm up to approximately 5 μm . Ag sulfide is also seen as thin rinds around pyrite and as small inclusions in jarosite. Much of the jarosite analyzed in these samples by EDS contains low but detectable Ag. The jarosite contained in the samples is potassium jarosite, however, vague bright areas in large masses are discernable using backscatter imaging. These areas are Ag rich and likely represent argentojarosite intimately mixed with the more abundant potassium variety. In sample **3829 CY-7** one small grain having a chemistry of Hg, Br, Cl and Ag was identified as an inclusion in quartz with a measurement just over 1 μm . This phase may represent capgaronnite or possibly iltsite. The primary reason for low Ag recovery in this material appears to be due to the very fine grained nature of the Ag sulfide, which should leach easily if liberated or exposed. In contrast, Ag bearing jarosites tend to be refractory and are usually unaffected by leaching.

The following table outlines individual phases based on XRD, petrography and SEM/EDS.

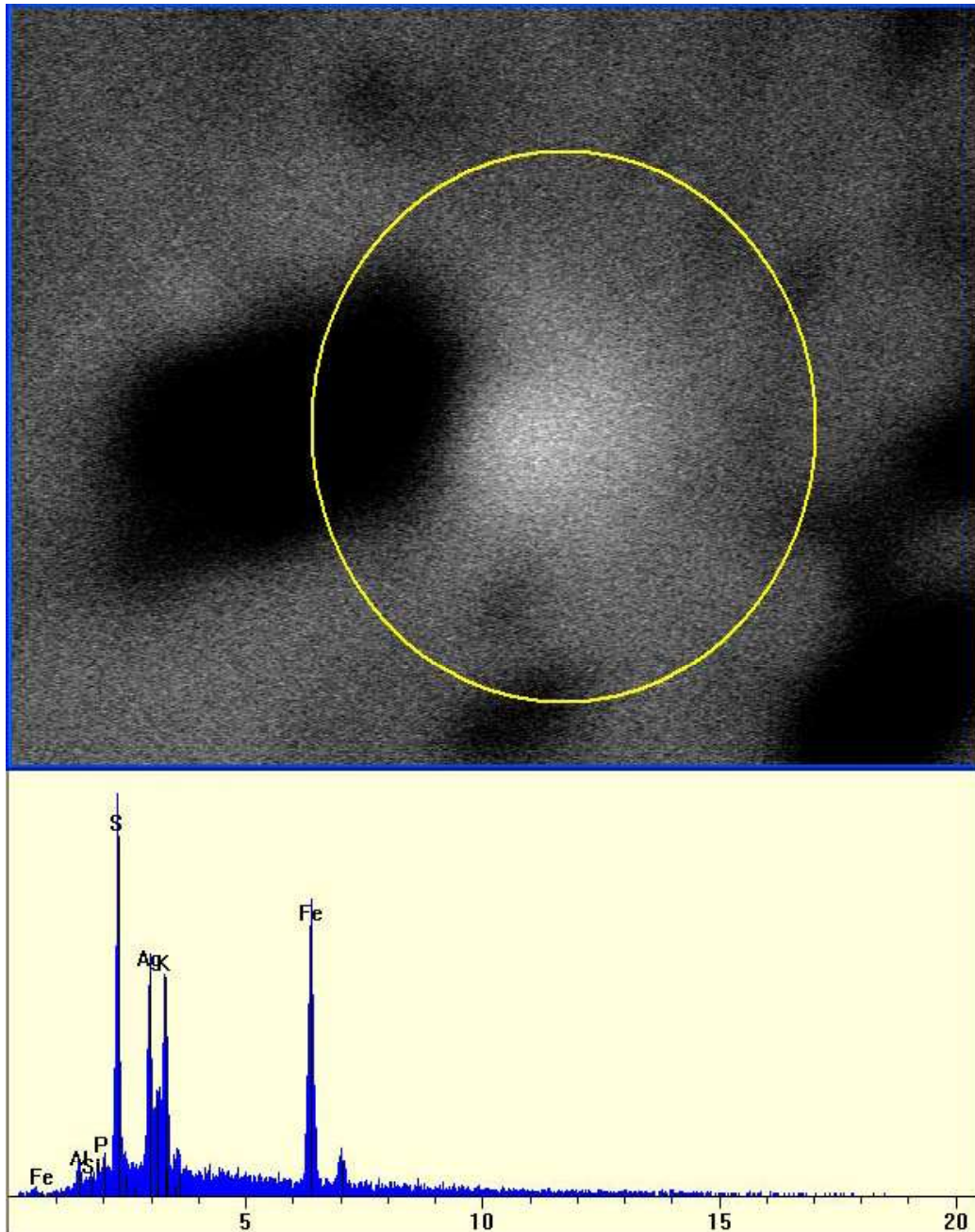
Client sample No.:	3829 BMC Mineralogy	3829 CY-7
Quartz	60	59
K-Feldspar	27	26
Jarosite	7	9
Iron Oxide	3	3
Pyrite	TR	TR
Chalcopyrite	TR	TR
Barite	TR	TR
Apatite	TR	TR
Zircon	TR	TR
Rutile	TR	TR
Ag Sulfide	TR	TR
Sericite	TR	TR
Capgaronnite (?)	TR	TR



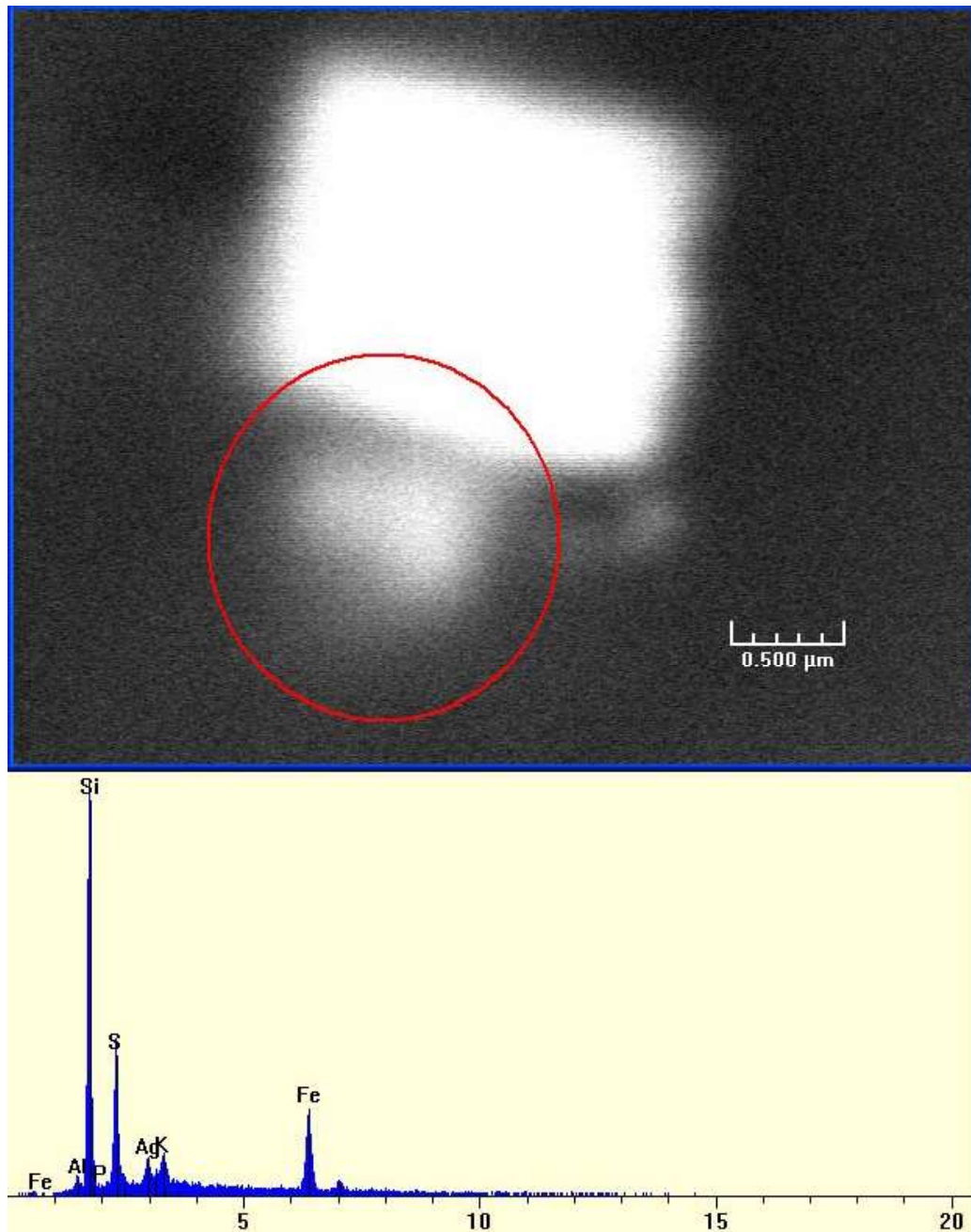
Client sample no. **3829 BMC Mineralogy**
Backscatter image of Ag sulfide surrounded by jarosite – 16,000X.



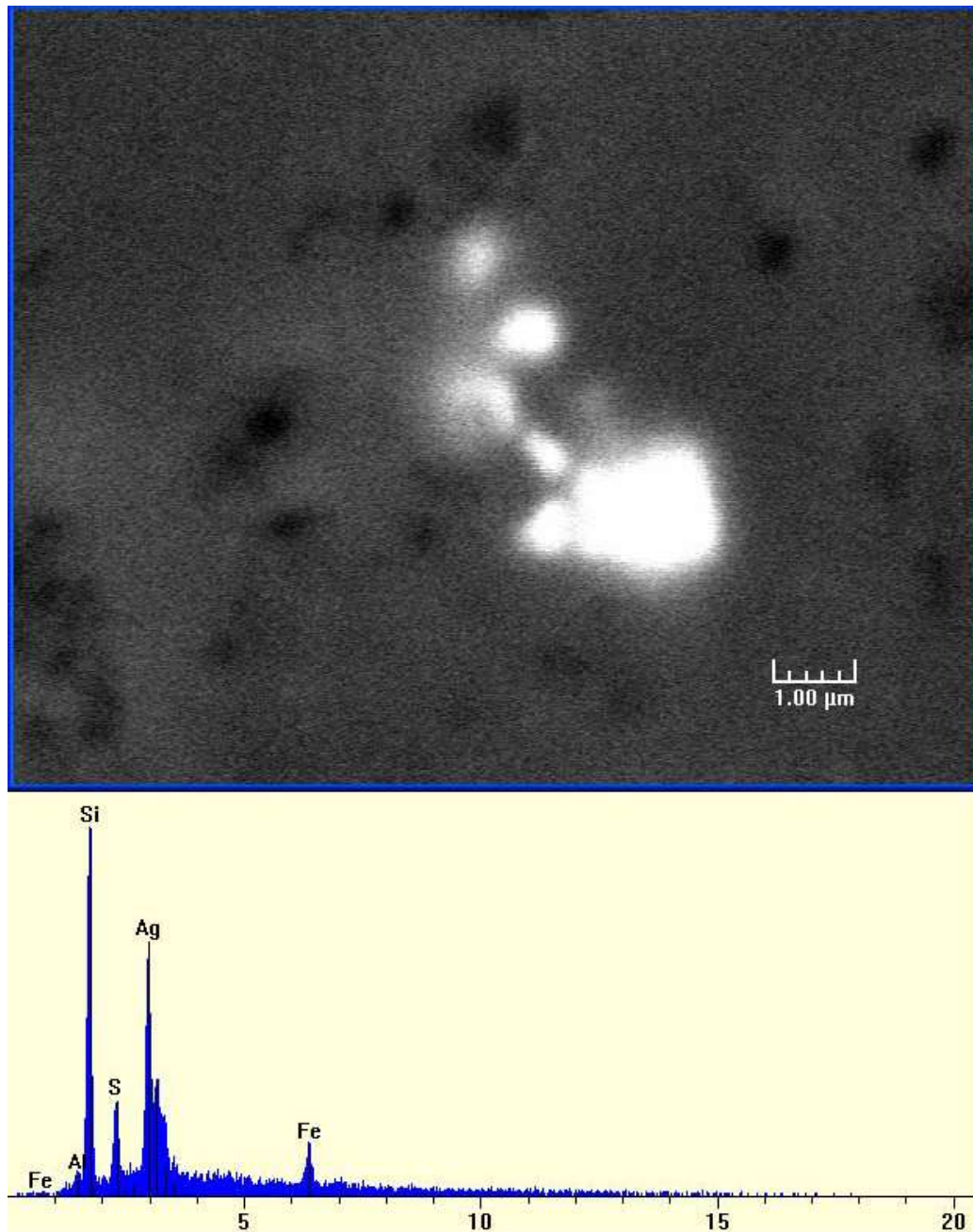
Client sample no. **3829 BMC Mineralogy**
Backscatter image of Ag sulfide in quartz – 16,000X.



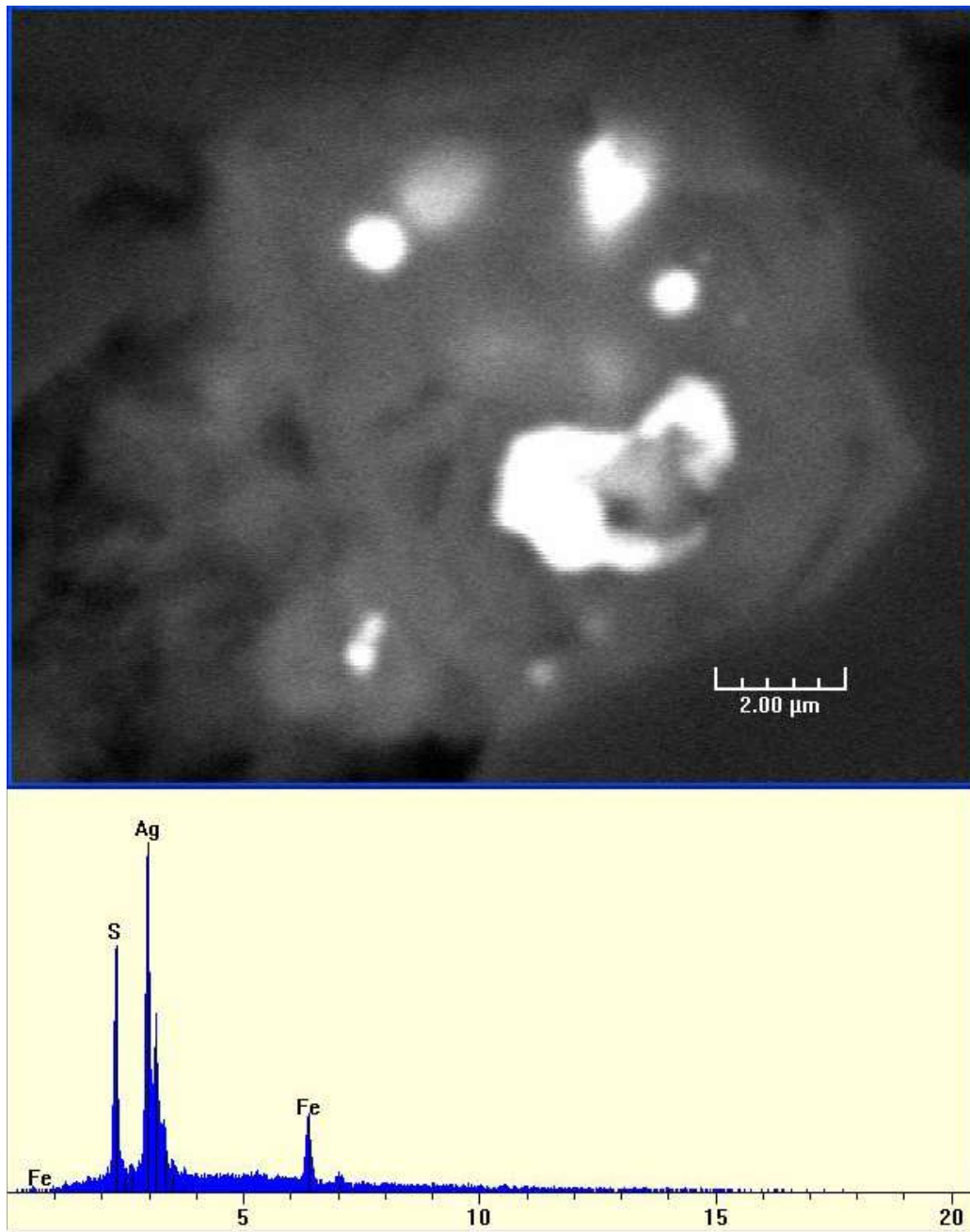
Client sample no. **3829 BMC Mineralogy**
Backscatter image of a bright Ag rich area in jarosite – 23,000X.



Client sample no. **3829 BMC Mineralogy**
Backscatter image of a pyrite cube attached to Ag bearing jarosite – 27,000X.

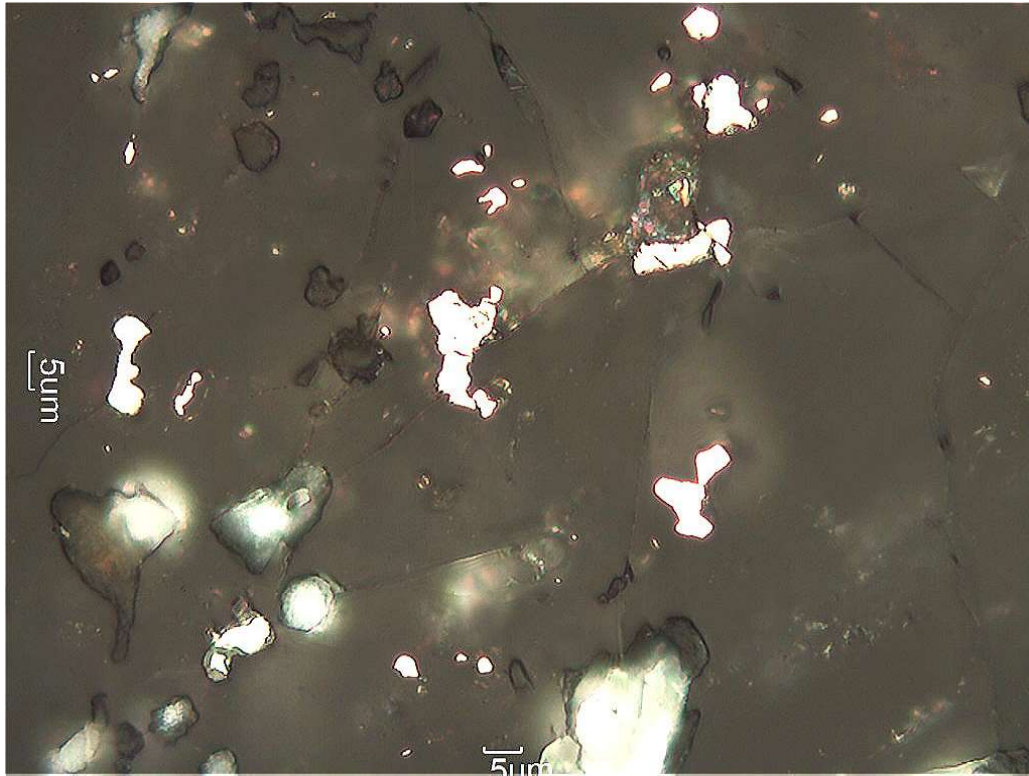


Client sample no. **3829 BMC Mineralogy**
Backscatter image of Ag sulfide grains mixed with iron oxide in quartz – 9,500X.

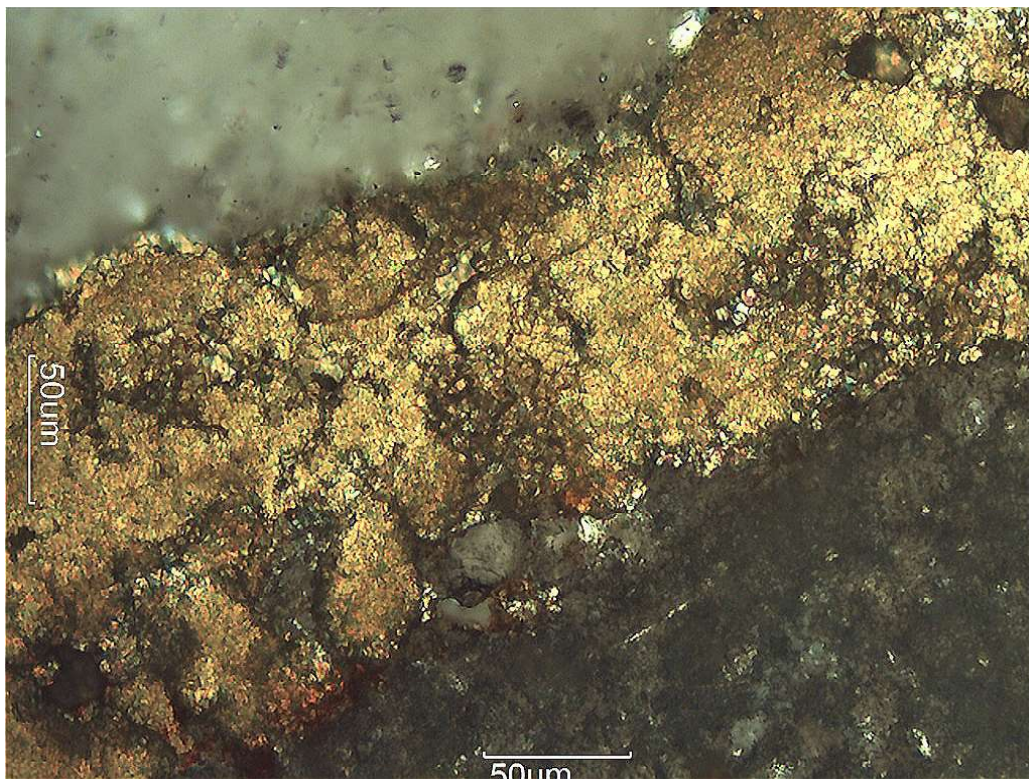


Client sample no. **3829 BMC Mineralogy**

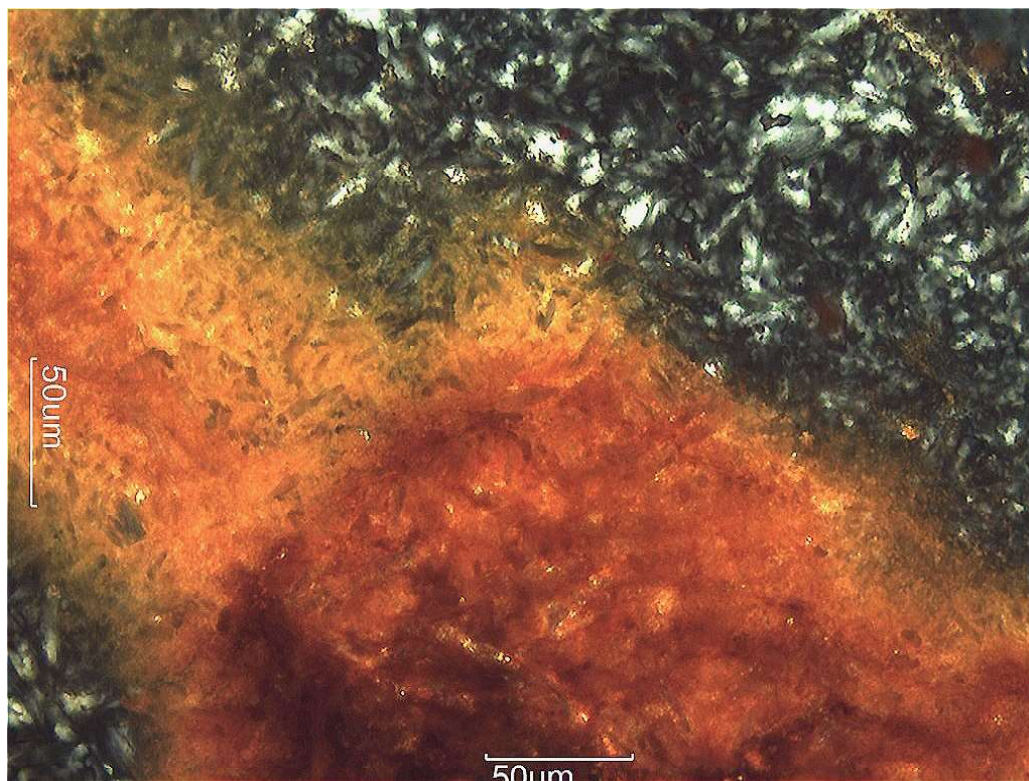
Backscatter image of several Ag sulfide grains in a goethite pseudomorph after pyrite – 7,500X.



Client sample no. **3829 BMC Mineralogy**
Photo 1 – Numerous grains of pyrite in quartz. Reflected light – 500X.

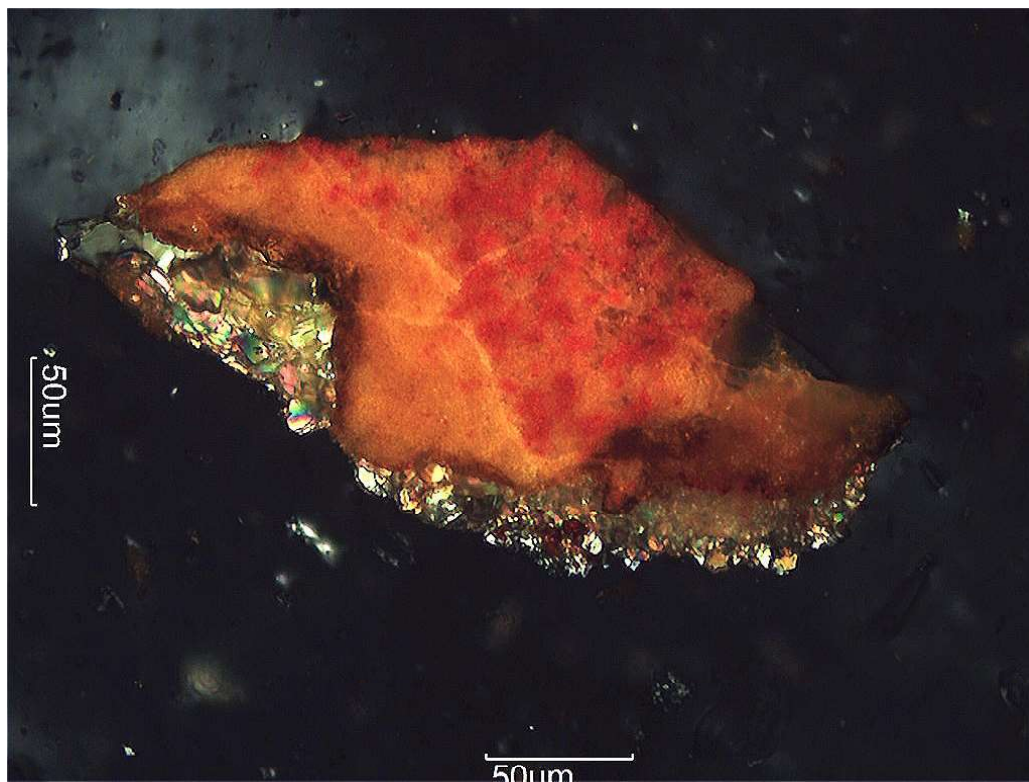


Client sample no. **3829 BMC Mineralogy**
Photo 2 – Large mass of granular jarosite. Polarized light – 200X.



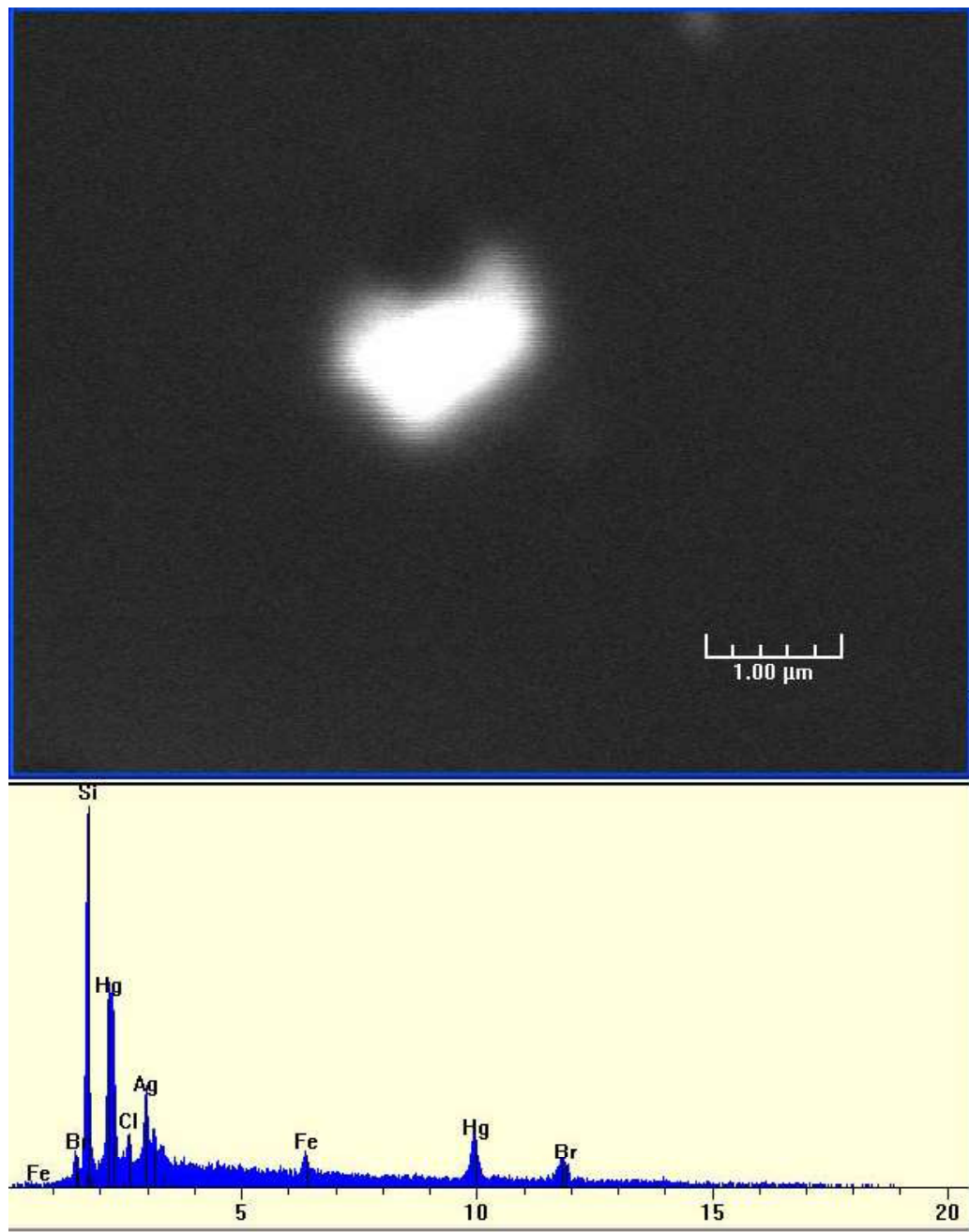
Client sample no. **3829 BMC Mineralogy**

Photo 3 – Iron oxide mixed with fine grained kaolinite. Polarized light – 200X.

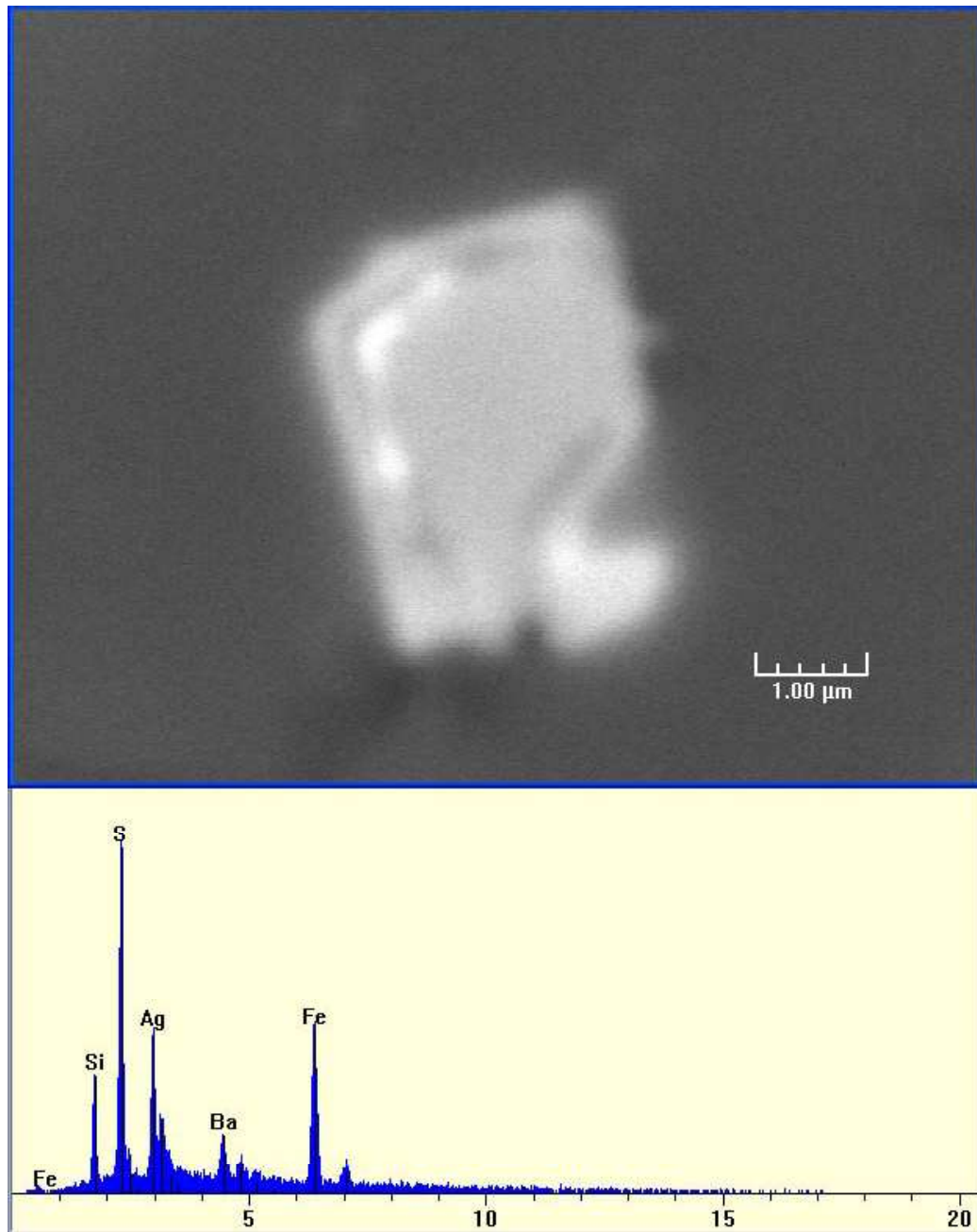


Client sample no. **3829 BMC Mineralogy**

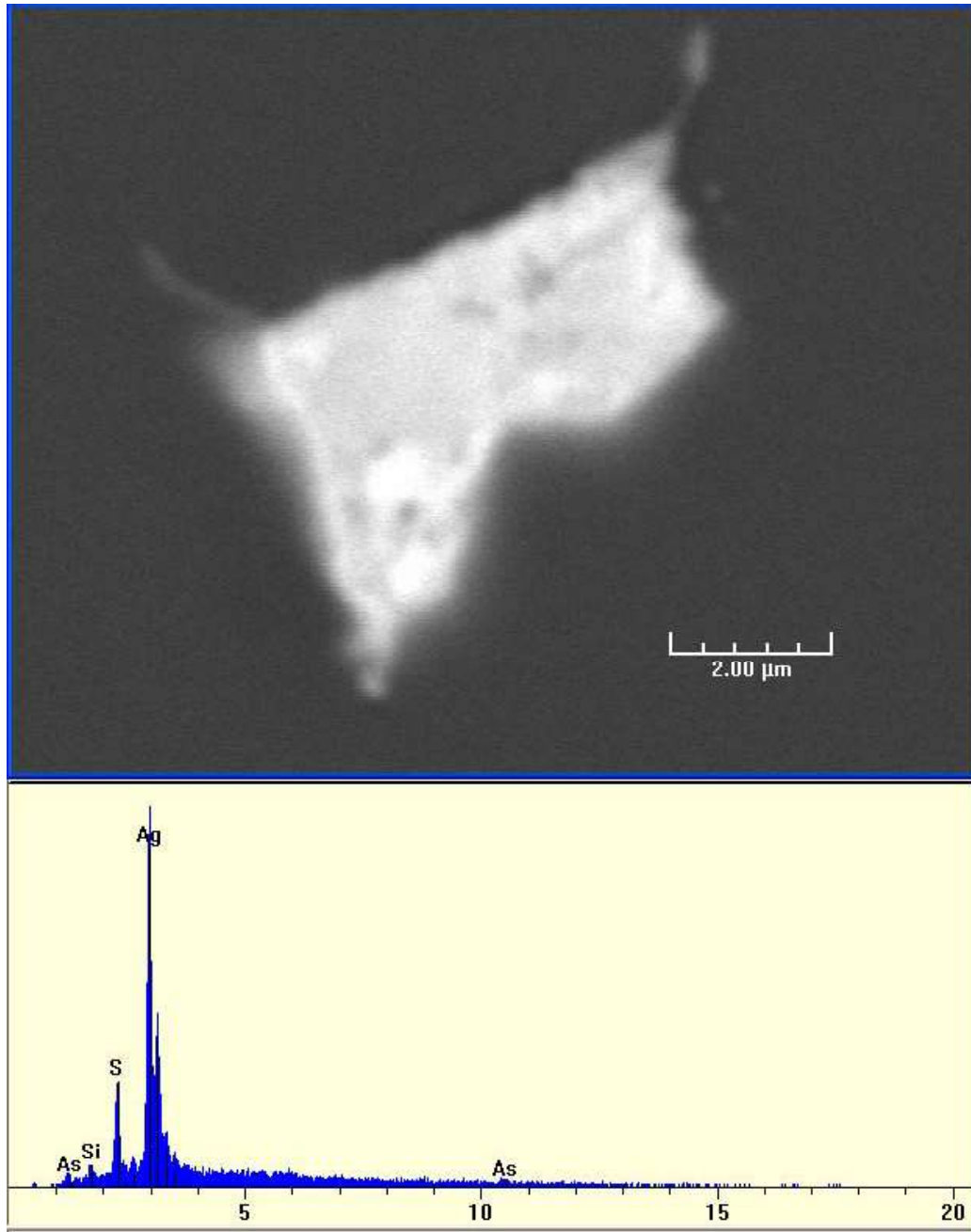
Photo 4 – Iron oxide rimmed by colorful jarosite. Reflected light – 200X. Crossed Nichols



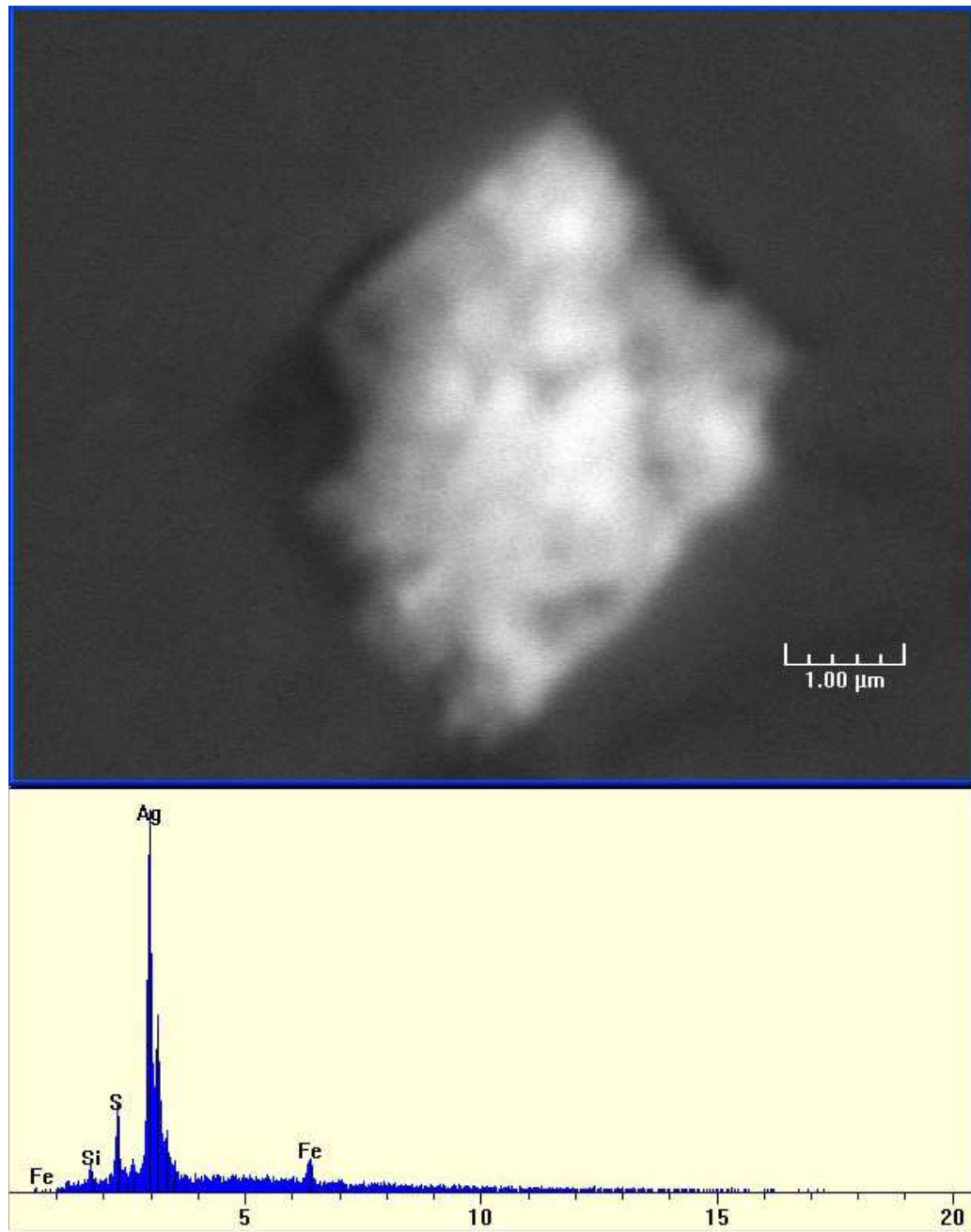
Client sample no. **3829 CY-7**
Backscatter image of capgaronnite (?) inclusion in quartz – 16,000X.



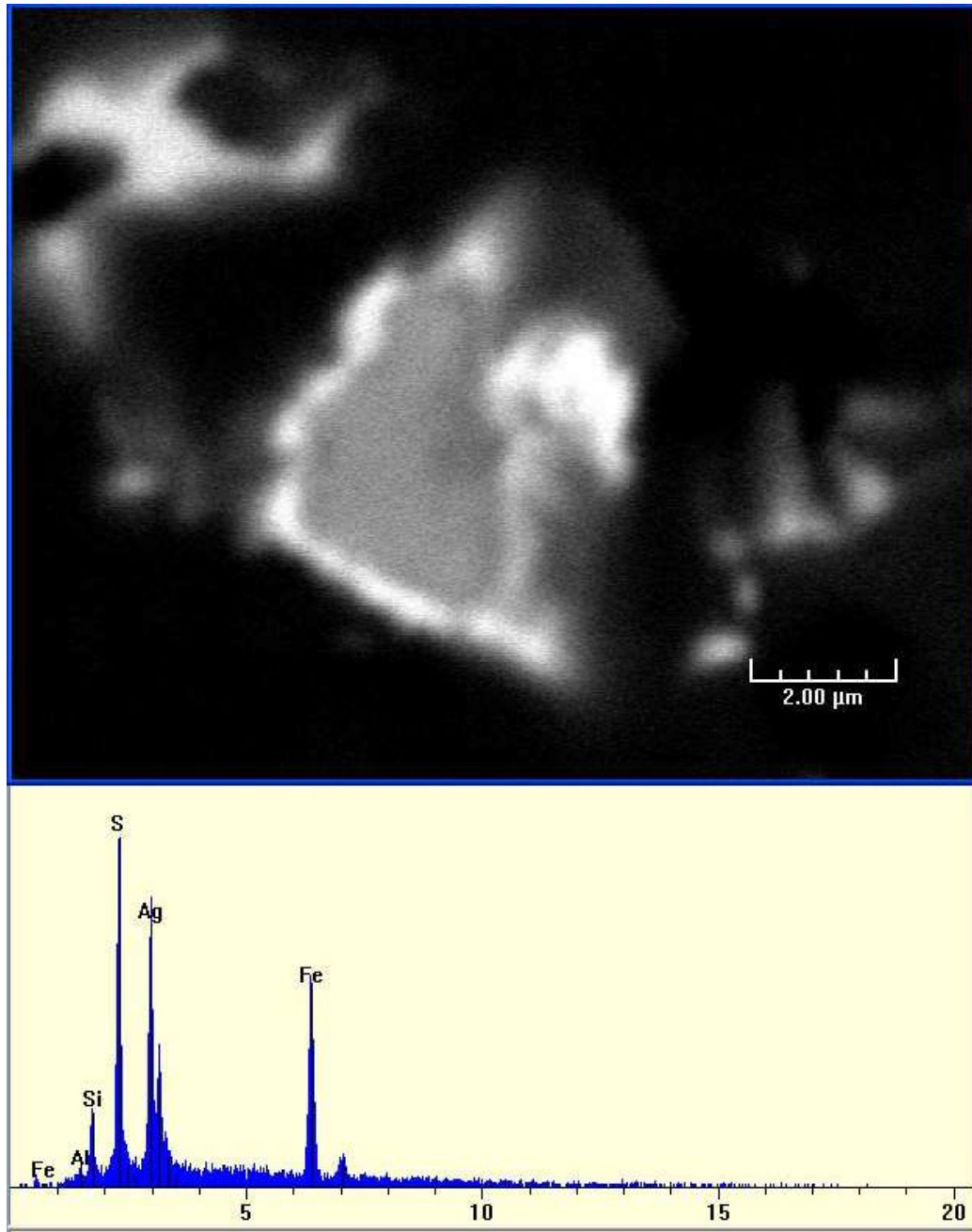
Client sample no. **3829 CY-7**
Backscatter image of pyrite rimmed with barite followed by bright Ag sulfide – 13,000X.



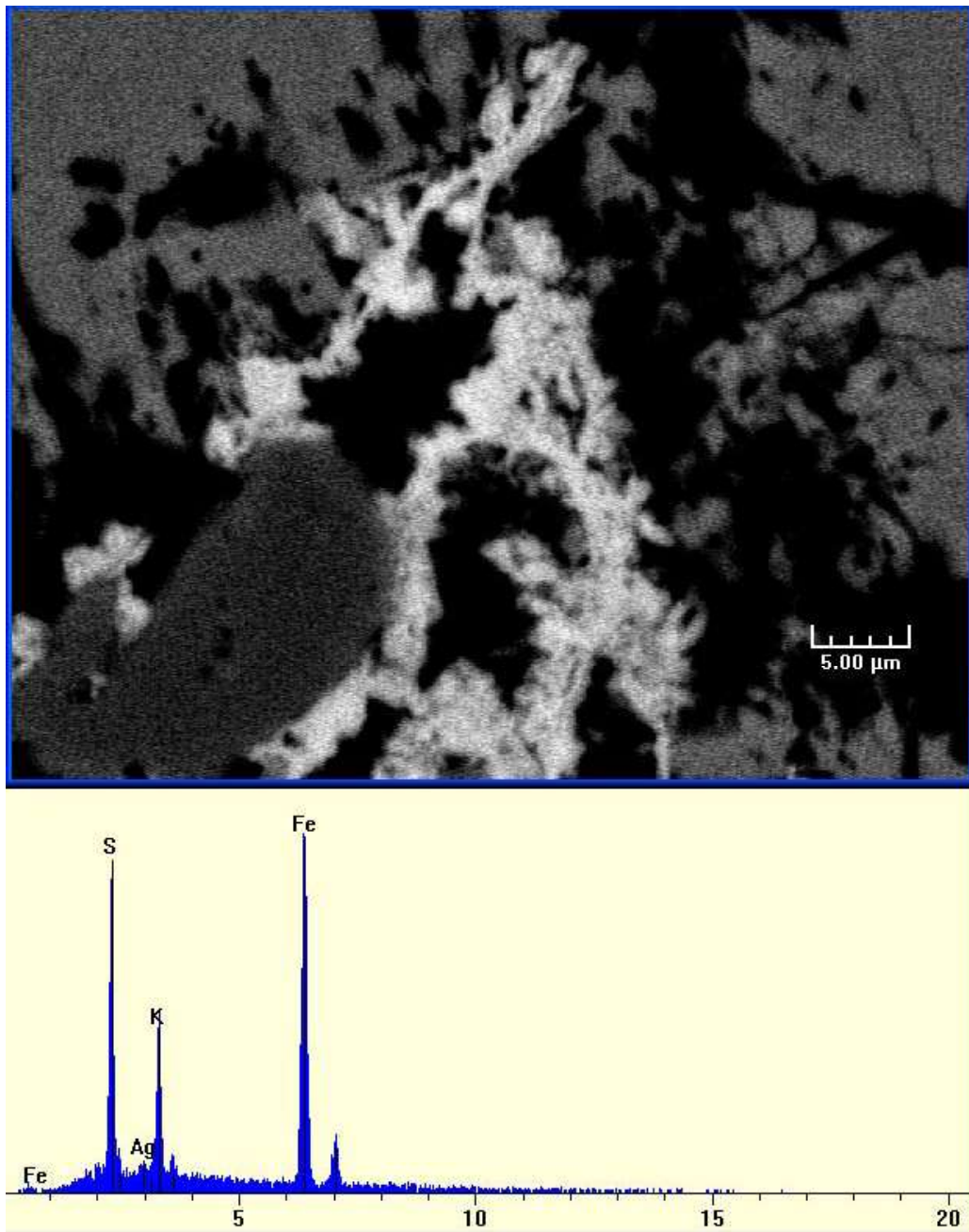
Client sample no. **3829 CY-7**
Backscatter image of Ag sulfide in quartz – 9,500X.



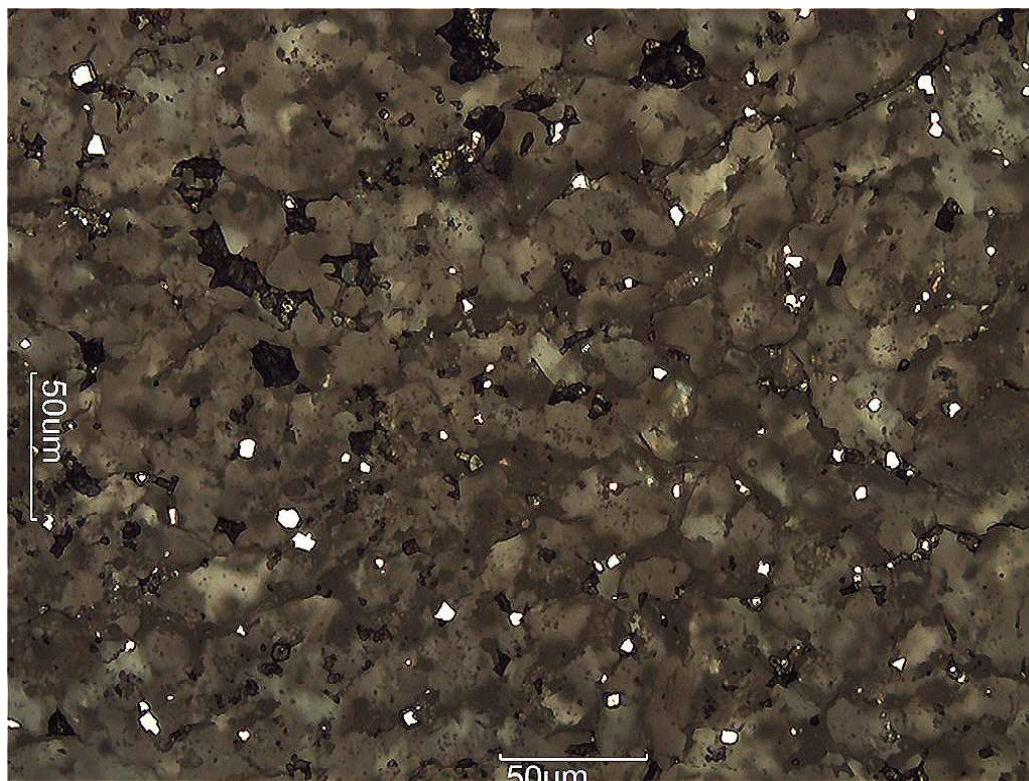
Client sample no. **3829 CY-7**
Backscatter image of cube like grain of Ag sulfide in quartz – 14,000X.



Client sample no. **3829 CY-7**
Backscatter image of pyrite rimmed with Ag sulfide – 8,500X.



Client sample no. **3829 CY-7**
Backscatter image of bright jarosite with detectable Ag content – 2,300X.



Client sample no. **3829 CY-7**

Photo 1 – Numerous small grains of pyrite with granular quartz. Reflected light – 200X.



Client sample no. **3829 CY-7**

Photo 2 – Jarosite pseudomorph after pyrite. Reflected light – 500X. Crossed Nichols



Client sample no. **3829 CY-7**
Photo 3 – Small grain of chalcopyrite with granular quartz. Reflected light – 200X.

APPENDIX

Section 5 - Detailed Bottle Roll Test Results

Bottle Roll Test

Project No. 3829
 Test No. CY-1
 Composite Underground Composite
 Feed Size 80%-1.7mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.70	67
Duplicate	0.82	63
Triplicate	1.09	53
Average	0.87	61

Ore Charge 981.6 g
 Solution Vol. 1.4724 L
 Natural pH 5.5
 Final Residue Wt 984.2 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.18	41
Duplicate	0.14	46
Triplicate	0.14	43
Average	0.15	43

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	1.47	1.30	-----	-----	-----	-----	-----
2	116	1.00	10.4	0.11	0.00	0.31	4.04	0.03596	0.46864	0.116
6	104	0.95	10.2	0.17	0.60	0.35	4.82	0.0364	0.50128	0.0988
24	102	1.00	10.5	0.10	1.00	0.37	5.57	0.03774	0.56814	0.102
48	112	1.00	11.3	0.11	0.30	0.36	5.37	0.04032	0.60144	0.112
72	138	0.95	11.3	0.20	0.10	0.34	5.34	0.04692	0.73692	0.1311
96	-----	1.00	11.3	-----	-----	0.32	5.03	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.456	0.465	56.0
6	0.551	0.562	67.7
24	0.617	0.629	75.7
48	0.640	0.652	78.6
72	0.651	0.663	79.9
96	0.669	0.68	81.9

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
5.948	6.060	11.4
7.566	7.707	14.5
9.171	9.343	17.6
9.445	9.622	18.2
10.002	10	18.9
10.283	10	18.9

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.3
0.00	1.3
0.07	1.9
0.06	3.0
0.06	3.3
0.13	3.4
0.13	3.4

	Au	% of Total
Extracted g/mt ore	0.68	81.9
Tail assay, g/mt	0.15	
Calculated Head g/mt ore	0.83	
NaCN Consumed, kg/mt ore	0.13	
Lime Added, kg/mt ore	3.4	

Ag	% of Total
10	18.9
43	
53	

Bottle Roll Test

Project No. 3829
 Test No. CY-2
 Composite Surface Composite
 Feed Size 80%-1.7mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.21	17
Duplicate	0.67	34
Triplicate	0.37	21
Average	0.42	24

Ore Charge 992.7 g
 Solution Vol. 1.4891 L
 Natural pH 7.5
 Final Residue Wt 982.2 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.05	19
Duplicate	0.07	19
Triplicate	0.05	21
Average	0.06	20

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	1.49	1.00	-----	-----	-----	-----	-----
2	117	1.00	10.9	0.12	0.40	0.09	1.72	0.01053	0.20124	0.117
6	106	1.00	11.2	0.11	0.00	0.11	2.02	0.01166	0.21412	0.106
24	106	1.00	10.7	0.11	0.30	0.14	2.48	0.01484	0.26288	0.106
48	112	1.00	11.0	0.11	0.10	0.13	2.60	0.01456	0.2912	0.112
72	103	0.95	10.8	0.17	0.30	0.13	2.61	0.01339	0.26883	0.09785
96	-----	1.00	11.1	-----	-----	0.12	2.57	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.134	0.135	43.5
6	0.174	0.176	56.6
24	0.231	0.232	75.0
48	0.231	0.232	74.9
72	0.245	0.247	79.7
96	0.244	0.25	80.6

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
2.561	2.580	10.3
3.209	3.233	12.9
4.108	4.139	16.6
4.550	4.583	18.3
4.856	4.892	19.6
5.065	5	20.0

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
0.00	1.4
0.00	1.4
0.01	1.7
0.01	1.8
0.08	2.1
0.08	2.1

	Au	% of Total
Extracted g/mt ore	0.25	80.6
Tail assay, g/mt	0.06	
Calculated Head g/mt ore	0.31	
NaCN Consumed, kg/mt ore	0.08	
Lime Added, kg/mt ore	2.1	

Ag	% of Total
5	20.0
20	
25	

Bottle Roll Test

Project No. 3829
 Test No. CY-3
 Composite Master Composite
 Feed Size 80%-1.7mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 1042.8 g
 Solution Vol. 1.5642 L
 Natural pH 6.3
 Final Residue Wt 1036.5 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.13	33
Duplicate	0.12	33
Triplicate	0.12	34
Average	0.12	33

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	1.56	1.00	-----	-----	-----	-----	-----
2	107	0.95	10.2	0.18	1.00	0.23	2.77	0.02461	0.29639	0.10165
6	107	1.00	11.3	0.10	0.00	0.27	3.11	0.02889	0.33277	0.107
24	103	1.00	10.7	0.10	0.30	0.29	3.57	0.02987	0.36771	0.103
48	153	1.00	10.9	0.15	0.10	0.29	3.79	0.04437	0.57987	0.153
72	153	1.00	10.8	0.15	0.20	0.26	3.65	0.03978	0.55845	0.153
96	-----	0.95	10.9	-----	-----	0.24	3.52	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.360	0.345	53.9
6	0.447	0.429	67.0
24	0.507	0.486	76.0
48	0.537	0.515	80.5
72	0.534	0.512	80.1
96	0.543	0.52	81.3

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
4.333	4.155	10.4
5.161	4.949	12.4
6.213	5.958	14.9
6.925	6.641	16.6
7.286	6.987	17.5
7.641	7	17.5

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
0.07	1.9
0.07	1.9
0.06	2.2
0.06	2.3
0.06	2.5
0.13	2.5

	Au	% of Total
Extracted g/mt ore	0.52	81.3
Tail assay, g/mt	0.12	
Calculated Head g/mt ore	0.64	
NaCN Consumed, kg/mt ore	0.13	
Lime Added, kg/mt ore	2.5	

Ag	% of Total
7	17.5
33	
40	

Bottle Roll Test

Project No. 3829
 Test No. CY-4
 Composite Master Composite
 Feed Size 100%-50mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 4999.5 g
 Solution Vol. 7.4993 L
 Natural pH 6.6
 Final Residue Wt 5017.1 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.23	52
Duplicate	0.28	59
Triplicate	0.26	52
Average	0.26	54

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	7.50	3.00					
2	109	1.00	10.8	0.11	0.40	0.11	0.53	0.01199	0.05777	0.109
6	109	1.00	10.7	0.11	0.60	0.15	0.74	0.01635	0.08066	0.109
24	104	1.00	10.5	0.10	1.00	0.23	1.00	0.02392	0.104	0.104
48	109	1.00	10.7	0.11	1.00	0.25	1.19	0.02725	0.12971	0.109
72	104	0.95	10.9	0.47	0.70	0.27	1.31	0.02808	0.13624	0.0988
96	-----	1.00	11.1	-----	-----	0.28	1.40	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.825	0.165	23.6
6	1.137	0.227	32.5
24	1.753	0.351	50.1
48	1.927	0.385	55.1
72	2.104	0.421	60.1
96	2.207	0.44	62.9

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
3.975	0.795	1.4
5.607	1.122	2.0
7.638	1.528	2.7
9.167	1.834	3.3
10.196	2	3.6
11.007	2	3.6

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	0.6
0.00	0.7
0.00	0.8
0.00	1.0
0.00	1.2
0.07	1.3
0.07	1.3

	Au	% of Total
Extracted g/mt ore	0.44	62.9
Tail assay, g/mt	0.26	
Calculated Head g/mt ore	0.70	
NaCN Consumed, kg/mt ore	0.07	
Lime Added, kg/mt ore	1.3	

Ag	% of Total
2	3.6
54	
56	

Bottle Roll Test

Project No. 3829
 Test No. CY-5
 Composite Master Composite
 Feed Size 80%-19mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 2040.8 g
 Solution Vol. 3.0612 L
 Natural pH 6.5
 Final Residue Wt 2014.2 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.19	31
Duplicate	0.33	38
Triplicate	0.24	34
Average	0.25	34

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	3.06	2.00	-----	-----	-----	-----	-----
2	109	1.00	11.2	0.11	0.00	0.13	1.09	0.01417	0.11881	0.109
6	102	1.00	10.8	0.10	0.30	0.19	1.51	0.01938	0.15402	0.102
24	100	0.95	10.5	0.24	0.70	0.25	2.12	0.025	0.212	0.095
48	105	1.00	10.7	0.10	0.70	0.28	2.47	0.0294	0.25935	0.105
72	108	1.00	10.9	0.11	0.50	0.30	2.64	0.0324	0.28512	0.108
96	-----	1.00	11.2	-----	-----	0.30	2.79	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.398	0.195	25.7
6	0.596	0.292	38.4
24	0.799	0.391	51.5
48	0.916	0.449	59.0
72	1.006	0.493	64.9
96	1.039	0.51	67.1

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
3.337	1.635	4.2
4.741	2.323	6.0
6.763	3.314	8.5
8.046	3.943	10.1
8.826	4.325	11.1
9.570	5	12.8

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
0.00	1.0
0.00	1.1
0.07	1.5
0.07	1.8
0.07	2.1
0.07	2.1

	Au	% of Total
Extracted g/mt ore	0.51	67.1
Tail assay, g/mt	0.25	
Calculated Head g/mt ore	0.76	
NaCN Consumed, kg/mt ore	0.07	
Lime Added, kg/mt ore	2.1	

Ag	% of Total
5	12.8
34	
39	

Bottle Roll Test

Project No. 3829
 Test No. CY-6
 Composite Master Composite
 Feed Size 80%-6.3mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 1001.6 g
 Solution Vol. 1.5024 L
 Natural pH 6.4
 Final Residue Wt 993.0 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.15	38
Duplicate	0.12	42
Triplicate	0.18	35
Average	0.15	38

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 1.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (gpL)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	1.00	-----	1.50	1.00	-----	-----	-----	-----	-----
2	105	1.00	10.5	0.10	1.00	0.20	2.12	0.021	0.2226	0.105
6	104	1.00	11.5	0.10	0.00	0.24	2.43	0.02496	0.25272	0.104
24	105	1.00	11.0	0.10	0.00	0.29	3.08	0.03045	0.3234	0.105
48	122	0.95	10.7	0.12	0.20	0.29	3.29	0.03538	0.40138	0.1159
72	114	0.95	10.7	0.18	0.80	0.28	3.26	0.03192	0.37164	0.1083
96	-----	1.00	11.0	-----	-----	0.26	3.21	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.300	0.300	44.1
6	0.382	0.381	56.0
24	0.482	0.481	70.7
48	0.512	0.511	75.2
72	0.532	0.53	77.9
96	0.534	0.53	77.9

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
3.185	3.180	7.2
3.873	3.867	8.8
5.103	5.095	11.6
5.742	5.732	13.0
6.098	6	13.6
6.394	6	13.6

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
0.00	2.0
-0.01	2.0
-0.01	2.0
0.06	2.2
0.06	3.0
0.06	3.0

	Au	% of Total
Extracted g/mt ore	0.53	77.9
Tail assay, g/mt	0.15	
Calculated Head g/mt ore	0.68	
NaCN Consumed, kg/mt ore	<0.07	
Lime Added, kg/mt ore	3.0	

Ag	% of Total
6	13.6
38	
44	

Bottle Roll Test

Project No. 3829
 Test No. CY-7
 Composite Master Composite
 Feed Size 80%-19mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 1997.5 g
 Solution Vol. 2.9963 L
 Natural pH 6.3
 Final Residue Wt 1977.5 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.13	34
Duplicate	0.16	35
Triplicate	0.22	36
Average	0.17	35

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 5.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	5.01	-----	15.00	2.00	-----	-----	-----	-----	-----
2	104	5.05	11.7	0.39	0.00	0.19	1.74	0.01976	0.18096	0.5252
6	100	4.95	11.4	0.64	0.00	0.23	2.07	0.023	0.207	0.495
24	104	5.00	11.0	0.54	0.00	0.29	2.96	0.03016	0.30784	0.52
48	100	4.80	10.9	1.09	0.00	0.32	3.38	0.032	0.338	0.48
72	100	5.00	11.0	0.51	0.00	0.32	3.59	0.032	0.359	0.5
96	-----	4.90	11.0	-----	-----	0.32	3.64	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.569	0.285	39.6
6	0.709	0.355	49.3
24	0.912	0.456	63.4
48	1.032	0.517	71.7
72	1.064	0.533	74.0
96	1.096	0.55	76.4

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
5.214	2.610	6.4
6.383	3.196	7.8
9.257	4.634	11.3
10.823	5.418	13.2
11.791	5.903	14.4
12.299	6	14.6

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
-0.07	1.0
0.02	1.0
0.01	1.0
0.32	1.0
0.33	1.0
0.48	1.0

	Au	% of Total
Extracted g/mt ore	0.55	76.4
Tail assay, g/mt	0.17	
Calculated Head g/mt ore	0.72	
NaCN Consumed, kg/mt ore	0.48	
Lime Added, kg/mt ore	1.0	

Ag	% of Total
6	14.6
35	
41	

Bottle Roll Test

Project No. 3829
 Test No. CY-8
 Composite Master Composite
 Feed Size 80%-6.3mm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 1007.4 g
 Solution Vol. 1.5111 L
 Natural pH 6.3
 Final Residue Wt 988.0 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.11	35
Duplicate	0.12	38
Triplicate	0.15	39
Average	0.13	37

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 5.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	5.00	-----	7.56	1.00	-----	-----	-----	-----	-----
2	104	5.00	11.4	0.53	0.00	0.19	2.06	0.01976	0.21424	0.52
6	100	5.00	11.2	0.47	0.00	0.22	2.36	0.022	0.236	0.5
24	105	4.90	11.0	0.67	0.00	0.25	2.82	0.02625	0.2961	0.5145
48	113	4.85	10.9	0.78	0.00	0.24	3.04	0.02712	0.34352	0.54805
72	100	4.95	11.0	0.57	0.00	0.23	3.05	0.023	0.305	0.495
96	-----	4.85	11.0	-----	-----	0.22	3.06	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.287	0.285	49.1
6	0.352	0.350	60.3
24	0.420	0.416	71.8
48	0.431	0.428	73.7
72	0.443	0.439	75.8
96	0.451	0.45	77.6

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
3.113	3.090	7.2
3.780	3.753	8.7
4.712	4.677	10.9
5.340	5.301	12.3
5.699	5.657	13.2
6.019	6	14.0

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.0
0.00	1.0
0.01	1.0
0.13	1.0
0.36	1.0
0.44	1.0
0.67	1.0

	Au	% of Total
Extracted g/mt ore	0.45	77.6
Tail assay, g/mt	0.13	
Calculated Head g/mt ore	0.58	
NaCN Consumed, kg/mt ore	0.67	
Lime Added, kg/mt ore	1.0	

Ag	% of Total
6	14.0
37	
43	

Bottle Roll Test

Project No. 3829
 Test No. CY-9
 Composite Master Composite
 Feed Size 80%-75µm

Head Assay	g Au/mt	g Ag/mt
Predicted		
Initial	0.57	40
Duplicate	0.66	41
Triplicate	0.77	50
Average	0.67	44

Ore Charge 999.8 g
 Solution Vol. 1.4997 L
 Natural pH 7.0
 Final Residue Wt 921.0 g

Tail Assay	g Au/mt	g Ag/mt
Initial	0.06	13
Duplicate	0.06	13
Triplicate	0.06	13
Average	0.06	13

Solid Density Wt. % 40.0
 Cyanide Conc. Maintained at: g/L 5.0

Raw Data

Leach Time Hours	Solution Withdrawn			Reagents Applied		Sol. Analysis		Removed from pulp		
	mL	NaCN (g/L)	pH	NaCN (g)	Lime (g)	Au (mg/L)	Ag (mg/L)	Au (mg)	Ag (mg)	NaCN (g)
0	-----	5.00	-----	7.50	1.30	-----	-----	-----	-----	-----
2	106	4.85	11.1	0.74	0.00	0.27	12.92	0.02862	1.36952	0.5141
6	100	5.00	11.1	0.50	0.00	0.28	12.17	0.028	1.217	0.5
24	114	4.80	11.0	0.85	0.00	0.28	11.36	0.03192	1.29504	0.5472
48	114	4.85	11.5	0.78	0.00	0.26	11.34	0.02964	1.29276	0.5529
72	109	5.00	11.1	0.55	0.00	0.24	9.94	0.02616	1.08346	0.545
96	-----	4.90	11.2	-----	-----	0.22	9.37	-----	-----	-----

Metallurgical Results

Cumulative Au Extraction			
Leach Time Hours	mg	g/mt ore	% of total
0		0.000	0.0
2	0.405	0.405	76.4
6	0.449	0.449	84.6
24	0.477	0.477	89.9
48	0.478	0.479	90.3
72	0.478	0.478	90.2
96	0.474	0.47	88.7

Cumulative Ag Extraction		
mg	g/mt ore	% of total
	0.000	0.0
19.376	19.380	58.7
19.621	19.625	59.5
19.623	19.627	59.5
20.888	20.892	63.3
20.081	20.085	60.9
20.310	20	60.6

Reagent Requirements Cumulative kg/mt ore	
Cyanide Consumed	Lime Added
	1.3
0.23	1.3
0.23	1.3
0.53	1.3
0.76	1.3
0.76	1.3
0.91	1.3

	Au	% of Total
Extracted g/mt ore	0.47	88.7
Tail assay, g/mt	0.06	
Calculated Head g/mt ore	0.53	
NaCN Consumed, kg/mt ore	0.91	
Lime Added, kg/mt ore	1.3	

Ag	% of Total
20	60.6
13	
33	

APPENDIX

Section 6 - Detailed Column Leach Test Results

3829 P-1

		NaCN added		379.72 g	NaCN		1.00 g/L solution	g/mt ore	
Kilograms	71.97	NaCN Consumption		1.56 kg/mt ore				Au	Ag
		Lime:		1.7 kg/mt ore				Head Grade	0.38 25
								Head Screen	0.34 25
								Tail Screen	0.04 20

Daily Column Leach Test Data,
Sample I.D. Surface Composite

Feed Size 80%-19mm

Date	Days Leached	Pregnant Solution Analyses					Barren Solution										
		NaCN					Analyses		Au Extraction		Ag Extraction		NaCN	Au		Ag	
		Vol. l.	Conc. g/l	pH	Au ppm	Ag ppm	Au ppm	Ag ppm	g/mt ore	Cum. %	g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg
8/30	1							0.000	0.0	0.000	0.0	0.00	0.00	0.00	0.00	0.00	0.00
8/31	2							0.000	0.0	0.000	0.0	0.07	0.00	0.00	0.00	0.00	0.00
9/1	3	3.75	0.95	11.9	2.35	13.20	0.00	0.00	0.122	34.0	0.688	2.8	0.09	8.81	8.81	49.50	49.50
9/2	4	4.74	0.85	11.8	0.97	6.77	0.00	0.03	0.186	51.8	1.134	4.5	0.11	4.60	13.41	32.09	81.59
9/3	5	5.05	0.90	11.8	0.41	3.39	0.00	0.02	0.215	59.8	1.369	5.5	0.11	2.07	15.48	16.97	98.56
9/4	6	4.92	0.95	11.8	0.26	2.81	0.00	0.02	0.233	64.7	1.560	6.2	0.12	1.28	16.76	13.72	112.28
9/5	7	4.85	0.90	11.6	0.17	2.27	0.00	0.00	0.244	67.9	1.712	6.8	0.13	0.82	17.58	10.91	123.19
9/6	8	5.01	0.90	11.0	0.12	1.75	0.00	0.02	0.253	70.2	1.833	7.3	0.14	0.60	18.19	8.77	131.95
9/7	9	4.87	0.90	11.1	0.09	1.57	0.00	0.02	0.259	71.9	1.938	7.8	0.15	0.44	18.62	7.54	139.50
9/8	10	4.79	0.90	10.9	0.08	1.43	0.00	0.03	0.264	73.4	2.032	8.1	0.16	0.38	19.01	6.75	146.25
9/9	11	4.39	0.80	10.7	0.06	1.27	0.00	0.04	0.268	74.4	2.107	8.4	0.18	0.26	19.27	5.42	151.67
9/10	12	5.43	0.70	10.8	0.06	1.08	0.00	0.05	0.272	75.6	2.186	8.7	0.20	0.33	19.60	5.66	157.33
9/11	13	4.81	0.60	10.7	0.05	0.94	0.00	0.04	0.276	76.6	2.245	9.0	0.23	0.24	19.84	4.27	161.60
9/12	14	5.04	0.60	10.8	0.03	0.87	0.00	0.05	0.278	77.1	2.303	9.2	0.26	0.15	19.99	4.18	165.78
9/13	15	4.85	0.65	10.7	0.04	0.88	0.00	0.05	0.280	77.9	2.359	9.4	0.29	0.19	20.18	4.01	169.79
9/14	16	4.65	0.70	10.7	0.03	0.86	0.00	0.04	0.282	78.4	2.411	9.6	0.31	0.14	20.32	3.74	173.53
9/15	17	4.97	0.75	10.7	0.03	0.81	0.00	0.05	0.284	79.0	2.464	9.9	0.33	0.15	20.47	3.82	177.35
9/16	18	4.58	0.75	10.3	0.04	0.76	0.00	0.05	0.287	79.7	2.509	10.0	0.35	0.18	20.65	3.23	180.58
9/17	19	5.44	0.70	10.4	0.02	0.71	0.00	0.06	0.288	80.1	2.559	10.2	0.37	0.11	20.76	3.61	184.19
9/18	20	4.89	0.70	10.4	0.02	0.67	0.00	0.05	0.290	80.5	2.601	10.4	0.40	0.10	20.86	2.97	187.16
9/19	21	4.75	0.70	10.5	0.02	0.67	0.00	0.06	0.291	80.9	2.641	10.6	0.42	0.10	20.96	2.93	190.09
9/20	22	4.86	0.70	10.3	0.02	0.65	0.00	0.07	0.293	81.3	2.681	10.7	0.44	0.10	21.05	2.85	192.94
9/21	23	4.87	0.70	10.5	0.02	0.65	0.00	0.06	0.294	81.6	2.720	10.9	0.47	0.10	21.15	2.81	195.75
9/22	24	4.55	0.80	10.6	0.02	0.65	0.00	0.06	0.295	82.0	2.757	11.0	0.49	0.09	21.24	2.65	198.40
9/23	25	5.87	0.80	10.4	0.01	0.51	0.00	0.08	0.296	82.2	2.794	11.2	0.49	0.06	21.30	2.69	201.09
9/24	26	4.62	0.75	10.3	0.00	0.59	0.00	0.09	0.296	82.2	2.826	11.3	0.52	0.00	21.30	2.32	203.40
9/25	27	4.99	0.75	10.3	0.00	0.57			0.296	82.2	2.859	11.4	0.54	0.00	21.30	2.39	205.79
9/26	28	Rest Cycle															
10/9	41						0.00	0.08					0.54	0.00	21.30	0.00	205.79
10/10	42	4.68	0.50	10.3	0.08	3.65	0.00	0.25	0.301	83.7	3.091	12.4	0.57	0.37	21.67	16.67	222.46
10/11	43	4.89	0.65	10.4	0.04	1.61	0.00	0.17	0.304	84.4	3.183	12.7	0.60	0.20	21.87	6.60	229.06
10/12	44	4.94	0.70	10.6	0.03	0.83	0.00	0.13	0.306	85.0	3.228	12.9	0.62	0.15	22.02	3.23	232.29
10/13	45	4.29	0.75	10.6	0.00	0.67	0.00	0.11	0.306	85.0	3.258	13.0	0.65	0.00	22.02	2.21	234.51
10/14	46	5.27	0.75	10.5	0.00	0.60	0.00	0.14	0.306	85.0	3.295	13.2	0.67	0.00	22.02	2.60	237.11
10/15	47	5.20	0.80	10.5	0.02	0.46	0.00	0.15	0.307	85.4	3.318	13.3	0.68	0.10	22.12	1.68	238.78
10/16	48	4.91	0.80	10.5	0.00	0.48			0.307	85.4	3.340	13.4	0.69	0.00	22.12	1.59	240.38
10/17	49	Rest Cycle															
10/30	62						0.00	0.15					0.69	0.00	22.12	0.00	240.38
10/31	63	4.68	0.55	10.0	0.04	2.38	0.00	0.26	0.310	86.1	3.484	13.9	0.73	0.19	22.31	10.37	250.75
11/1	64	5.00	0.70	10.1	0.00	1.22	0.00	0.16	0.310	86.1	3.550	14.2	0.75	0.00	22.31	4.77	255.52
11/2	65	4.92	0.70	10.3	0.00	0.72	0.00	0.20	0.310	86.1	3.588	14.4	0.77	0.00	22.31	2.73	258.25
11/3	66	5.02	0.80	10.3	0.00	0.59	0.00	0.18	0.310	86.1	3.615	14.5	0.79	0.00	22.31	1.94	260.19
11/4	67	4.54	0.85	10.2	0.00	0.54	0.00	0.17	0.310	86.1	3.637	14.5	0.81	0.00	22.31	1.53	261.73
11/5	68	5.32	0.85	10.2	0.02	0.45	0.00	0.19	0.311	86.5	3.658	14.6	0.82	0.11	22.42	1.53	263.25
11/6	69	5.02	0.80	10.2	0.03	0.44			0.314	87.1	3.675	14.7	0.83	0.15	22.57	1.24	264.49
11/7	70	Rest Cycle															
11/20	83						0.00	0.23					0.83	0.00	22.57	0.00	264.49
11/21	84	4.46	0.60	10.0	0.04	2.37	0.01	0.32	0.316	87.8	3.806	15.2	0.86	0.18	22.74	9.40	273.89
11/22	85	4.92	0.75	10.2	0.02	1.09	0.02	0.28	0.317	88.0	3.857	15.4	0.88	0.05	22.79	3.73	277.62
11/23	86	4.88	0.70	10.2	0.00	0.73	0.00	0.28	0.315	87.6	3.887	15.5	0.91	-0.10	22.69	2.13	279.76
11/24	87	4.88	0.75	10.3	0.00	0.59	0.00	0.26	0.315	87.6	3.907	15.6	0.93	0.00	22.69	1.45	281.21
11/25	88	4.92	0.70	10.2	0.01	0.51	0.00	0.25	0.316	87.8	3.924	15.7	0.95	0.05	22.74	1.18	282.39
11/26	89	5.00	0.75	10.2	0.00	0.50	0.00	0.26	0.316	87.8	3.941	15.8	0.97	0.00	22.74	1.23	283.61
11/27	90	4.93	0.75	10.3	0.00	0.51			0.316	87.8	3.957	15.8	0.99	0.00	22.74	1.19	284.80
11/28	91	Rest Cycle															
12/11	104						0.00	0.28					0.99	0.00	22.74	0.00	284.80
12/12	105	4.49	0.50	10.2	0.02	1.87	0.00	0.02	0.317	88.1	4.054	16.2	1.03	0.09	22.83	6.97	291.77
12/13	106	4.93	0.75	10.4	0.01	0.94	0.00	0.02	0.318	88.3	4.117	16.5	1.05	0.05	22.88	4.53	296.30
12/14	107	4.92	0.60	10.4	0.00	0.45	0.00	0.03	0.318	88.3	4.146	16.6	1.08	0.00	22.88	2.11	298.42
12/15	108	4.74	0.70	10.4	0.00	0.32	0.00	0.03	0.318	88.3	4.165	16.7	1.10	0.00	22.88	1.36	299.78
12/16	109	4.99	0.75	10.7	0.00	0.27	0.00	0.03	0.318	88.3	4.182	16.7	1.12	0.00	22.88	1.19	300.97
12/17	110	5.14	0.75	10.8	0.00	0.22	0.00	0.04	0.318	88.3	4.196	16.8	1.14	0.00	22.88	0.98	301.95
12/18	111	5.20	0.75	10.3	0.00	0.19			0.318	88.3	4.206	16.8	1.15	0.00	22.88	0.78	302.74
12/19	112	Rest Cycle															
1/1	125						0.00	0.05					1.15	0.00	22.88	0.00	302.74
1/2	126	4.41	0.55	10.1	0.01	1.52	0.00	0.08	0.318	88.5	4.296	17.2	1.19	0.04	22.92	6.45	309.18
1/3	127	4.86	0.65	10.3	0.01	0.68	0.00	0.08	0.319	88.7	4.336	17.3	1.22	0.05	22.97	2.90	312.08
1/4	128	4.94	0.65	10.4	0.01	0.40	0.00	0.07	0.32	88.9	4.358	17.4	1.25	0.05	23.02	1.57	313.65
1/5	129	4.93	0.75	10.4	0.00	0.31	0.00	0.06	0.32	88.9	4.374	17.5	1.26	0.00	23.02	1.17	314.82
1/6	130	4.89	0.85	10.3	0.01	0.29	0.00	0.07	0.32	88.9	4.390	17.6	1.28	0.05	23.07	1.11	315.93
1/7	131</																

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		NaCN added		379.72 g	NaCN		1.00 g/L solution	g/mt ore	
Kilograms	71.97	NaCN Consumption		1.56 kg/mt ore					Au
		Lime:		1.7 kg/mt ore					Ag
								Head Grade	0.38
								Head Screen	0.34
								Tail Screen	0.04
									25
									25
									20

Daily Column Leach Test Data,
Sample I.D. Surface Composite

Feed Size 80%-19mm

Pregnant Solution Analyses							Barren Solution													
NaCN							Analyses		Au Extraction		Ag Extraction		NaCN	Au		Ag				
Date	Days Leached	Vol. l.	Conc. g/l	pH	Au ppm	Ag ppm	Au ppm	Ag ppm	Cum. g/mt ore	Cum. %	Cum. g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg			
1/23	147	4.44	0.50	10.2	0.01	1.33	0.00	0.16	0.32	88.9	4.489	18.0	1.36	0.04	23.11	5.50	323.10			
1/24	148	5.16	0.70	10.4	0.00	0.61	0.00	0.14	0.32	88.9	4.522	18.1	1.38	0.00	23.11	2.33	325.43			
1/25	149	4.88	1.00	12.0	0.00	0.44	0.00	0.13	0.32	88.9	4.542	18.2	1.38	0.00	23.11	1.43	326.86			
1/26	150	4.82	0.75	10.3	0.00	0.38	0.00	0.12	0.32	88.9	4.558	18.2	1.40	0.00	23.11	1.17	328.03			
1/27	151	5.18	0.75	10.3	0.00	0.29	0.00	0.13	0.32	88.9	4.570	18.3	1.42	0.00	23.11	0.89	328.92			
1/28	152	1.35	0.65	10.3	0.00	0.40	0.00	0.11	0.32	88.9	4.574	18.3	1.44	0.00	23.11	0.24	329.16			
1/29	153	5.21	0.80	10.4	0.00	0.30			0.32	88.9	4.587	18.3	1.45	0.00	23.11	1.00	330.16			
1/30	154	Rest Cycle																		
2/12	167						0.00	0.12					1.45	0.00	23.11	0.00	330.16			
2/13	168	4.51	0.55	10.2	0.00	1.22	0.00	0.18	0.32	88.9	4.655	18.6	1.49	0.00	23.11	4.89	335.05			
2/14	169	4.94	0.75	10.5	0.00	0.59	0.00	0.14	0.32	88.9	4.683	18.7	1.51	0.00	23.11	2.00	337.05			
2/15	170	4.90	0.70	10.3	0.00	0.45	0.00	0.16	0.32	88.9	4.704	18.8	1.53	0.00	23.11	1.49	338.54			
2/16	171	4.99	0.75	10.3	0.01	0.33	0.00	0.14	0.32	88.9	4.715	18.9	1.55	0.05	23.16	0.83	339.37			
2/17	172	4.93	0.80	10.2	0.00	0.31	0.00	0.12	0.32	88.9	4.727	18.9	1.56	0.00	23.16	0.81	340.18			
2/18	173	5.01	0.80	10.3	0.00	0.29	0.00	0.13	0.32	88.9	4.738	19.0	1.58	0.00	23.16	0.84	341.03			
2/19	174	4.92	0.75	10.3	0.00	0.28			0.32	88.9	4.748	19.0	1.60	0.00	23.16	0.71	341.74			
2/20	175	Rest Cycle																		
Rinse Cycle																				
3/1	184	0.47	0.10	9.7	0.00	0.42			0.32	88.9	4.751	19.0	1.60	0.00	23.16	0.20	341.94			
3/2	185	4.34	0.40	10.4	0.00	0.75			0.32	88.9	4.796	19.2	1.57	0.00	23.16	3.26	345.19			
3/3	186	5.07	0.10	10.2	0.00	0.28			0.32	88.9	4.816	19.3	1.57	0.00	23.16	1.42	346.61			
3/4	187	5.01	0.10	10.3	0.00	0.16			0.32	88.9	4.827	19.3	1.56	0.00	23.16	0.80	347.41			
3/5	188	4.93	0.00	10.3	0.00	0.11			0.32	88.9	4.835	19.3	1.56	0.00	23.16	0.54	347.96			
3/6	189	5.15	0.00	10.2	0.00	0.06			0.32	88.9	4.839	19.4	1.56	0.00	23.16	0.31	348.26			
3/7	190	4.62	0.00	10.2	0.00	0.04			0.32	88.9	4.842	19.4	1.56	0.00	23.16	0.18	348.45			
Drain Down																				
3/13	196	3.29	0.00	9.3	0.00	0.03			0.32	88.9	5	20.0	1.56	0.00	23.16	0.10	348.55			
Extracted, g/mt ore									0.32	88.9	5	20.0								
Tail, g/mt ore									0.04		20									
Calculated Head, g/mt ore									0.36		25									

		NaCN added		423.30 g	NaCN		1.00 g/L solution	-----	
Kilograms	71.75	NaCN Consumption		1.93 kg/mt ore				Au	Ag
		Lime:		2.7 kg/mt ore				Head Grade	0.84
								Head Screen	59
								Tail Screen	0.82
									65
									44

Daily Column Leach Test Data,

Sample I.D. Underground Composite

Feed Size 80%-19mm

Date	Days Leached	Pregnant Solution Analyses					Barren Solution										
		NaCN					Analyses		Au Extraction		Ag Extraction		NaCN	Au		Ag	
		Vol. l.	Conc. g/l	pH	Au ppm	Ag ppm	Au ppm	Ag ppm	Cum. g/mt ore	Cum. %	Cum. g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg
8/30	1							0.000	0.0	0.000	0.0	0.00	0.00	0.00	0.00	0.00	0.00
8/31	2							0.000	0.0	0.000	0.0	0.07	0.00	0.00	0.00	0.00	0.00
9/1	3							0.000	0.0	0.000	0.0	0.14	0.00	0.00	0.00	0.00	0.00
9/2	4	4.07	0.35	9.0	4.88	21.20	0.00	0.03	0.277	35.5	1.203	2.3	0.19	19.86	19.86	86.28	86.28
9/3	5	5.32	0.70	9.9	1.72	10.60	0.00	0.03	0.404	51.8	1.986	3.8	0.21	9.15	29.01	56.24	142.52
9/4	6	4.19	0.65	10.0	1.02	7.83	0.01	0.07	0.464	59.5	2.442	4.7	0.25	4.27	33.29	32.65	175.18
9/5	7	5.11	0.60	10.2	0.59	5.05	0.00	0.03	0.505	64.8	2.796	5.4	0.27	2.96	36.25	25.45	200.63
9/6	8	5.24	0.70	10.0	0.34	3.82	0.00	0.05	0.530	68.0	3.073	5.9	0.29	1.78	38.03	19.86	220.49
9/7	9	4.85	0.80	10.3	0.23	3.17	0.00	0.05	0.546	69.9	3.284	6.3	0.31	1.12	39.15	15.12	235.61
9/8	10	3.99	0.75	10.3	0.20	3.08	0.00	0.05	0.557	71.4	3.451	6.6	0.34	0.80	39.94	12.03	247.64
9/9	11	4.73	0.70	10.2	0.14	2.69	0.00	0.08	0.566	72.6	3.625	7.0	0.37	0.66	40.61	12.47	260.11
9/10	12	5.88	0.70	10.3	0.12	2.28	0.00	0.13	0.576	73.8	3.806	7.3	0.38	0.71	41.31	13.00	273.11
9/11	13	4.66	0.70	10.2	0.09	1.97	0.00	0.10	0.582	74.6	3.925	7.5	0.40	0.42	41.73	8.52	281.63
9/12	14	4.94	0.70	10.4	0.08	1.87	0.00	0.14	0.587	75.3	4.047	7.8	0.43	0.40	42.13	8.73	290.36
9/13	15	4.73	0.65	10.2	0.07	1.72	0.00	0.13	0.592	75.9	4.150	8.0	0.46	0.33	42.46	7.42	297.78
9/14	16	4.53	0.65	10.2	0.06	1.58	0.00	0.13	0.596	76.4	4.241	8.2	0.49	0.27	42.73	6.49	304.27
9/15	17	4.86	0.60	10.3	0.07	1.42	0.01	0.14	0.600	77.0	4.328	8.3	0.52	0.34	43.07	6.24	310.51
9/16	18	4.52	0.50	10.0	0.05	1.23	0.00	0.15	0.603	77.3	4.395	8.5	0.56	0.18	43.25	4.85	315.36
9/17	19	5.42	0.45	10.0	0.06	1.10	0.00	0.17	0.607	77.9	4.468	8.6	0.59	0.33	43.57	5.20	320.55
9/18	20	4.82	0.50	10.0	0.03	0.96	0.00	0.14	0.609	78.1	4.520	8.7	0.63	0.14	43.72	3.76	324.31
9/19	21	4.78	0.55	10.1	0.03	0.96	0.00	0.15	0.611	78.4	4.574	8.8	0.66	0.14	43.86	3.87	328.19
9/20	22	4.58	0.60	10.0	0.03	0.97	0.00	0.16	0.613	78.6	4.625	8.9	0.70	0.14	44.00	3.68	331.87
9/21	23	4.80	0.65	10.2	0.03	0.98	0.00	0.16	0.615	78.9	4.679	9.0	0.73	0.14	44.14	3.89	335.75
9/22	24	4.52	0.70	10.3	0.02	0.96	0.00	0.16	0.616	79.0	4.729	9.1	0.75	0.09	44.23	3.52	339.28
9/23	25	5.89	0.75	10.1	0.03	0.84	0.00	0.20	0.619	79.3	4.786	9.2	0.76	0.18	44.41	4.13	343.41
9/24	26	4.44	0.75	10.0	0.02	0.84	0.00	0.22	0.620	79.5	4.824	9.3	0.79	0.09	44.50	2.71	346.12
9/25	27	4.91	0.75	10.1	0.02	0.89	0.00	0.21	0.622	79.7	4.869	9.4	0.81	0.10	44.59	3.25	349.37
9/26	28	4.93	0.80	10.2	0.02	0.92	0.00	0.28	0.623	79.9	4.917	9.5	0.82	0.10	44.69	3.46	352.83
9/27	29	4.92	0.75	10.4	0.02	0.82	0.00	0.23	0.624	80.0	4.954	9.5	0.84	0.10	44.79	2.61	355.44
9/28	30	4.67	0.75	10.4	0.02	0.78	0.00	0.24	0.626	80.2	4.988	9.6	0.86	0.09	44.88	2.47	357.91
9/29	31	4.93	0.75	10.4	0.02	0.80	0.00	0.25	0.627	80.4	5.026	9.7	0.88	0.10	44.98	2.72	360.63
9/30	32	5.14	0.75	10.4	0.00	0.83	0.00	0.33	0.627	80.4	5.068	9.7	0.90	0.00	44.98	2.99	363.62
10/1	33	4.71	0.70	10.4	0.00	0.86	0.00	0.33	0.627	80.4	5.101	9.8	0.93	0.00	44.98	2.37	365.98
10/2	34	4.89	0.80	10.1	0.00	0.86	0.00	0.30	0.627	80.4	5.136	9.9	0.94	0.00	44.98	2.52	368.51
10/3	35	4.29	0.75	10.2	0.01	0.91	0.00	0.31	0.628	80.5	5.169	9.9	0.97	0.04	45.03	2.37	370.88
10/4	36	4.54	0.80	10.3	0.01	0.93			0.628	80.5	5.206	10.0	0.99	0.05	45.07	2.64	373.52
10/5	37	Rest Cycle															
10/18	50						0.00	0.31					0.99	0.00	45.07	0.00	373.52
10/19	51	5.60	0.55	9.9	0.04	3.00	0.00	0.63	0.631	80.9	5.418	10.4	1.02	0.22	45.30	15.22	388.74
10/20	52	4.52	0.70	10.1	0.05	3.01	0.00	0.41	0.634	81.3	5.563	10.7	1.04	0.23	45.52	10.39	399.13
10/21	53	4.27	0.70	10.0	0.02	1.61	0.00	0.33	0.636	81.5	5.630	10.8	1.07	0.09	45.61	4.78	403.92
10/22	54	5.58	0.70	10.1	0.02	1.16	0.00	0.43	0.637	81.7	5.696	11.0	1.09	0.11	45.72	4.79	408.71
10/23	55	5.00	0.80	10.1	0.00	0.85	0.00	0.37	0.637	81.7	5.725	11.0	1.11	0.00	45.72	2.06	410.76
10/24	56	4.74	0.80	10.0	0.00	0.79	0.00	0.40	0.637	81.7	5.751	11.1	1.12	0.00	45.72	1.86	412.62
10/25	57	3.96	0.75	10.1	0.00	0.85			0.637	81.7	5.769	11.1	1.15	0.00	45.72	1.33	413.95
10/26	58	Rest Cycle															
11/8	71						0.00	0.37					1.15	0.00	45.72	0.00	413.95
11/9	72	5.39	0.55	9.9	0.03	2.45	0.00	0.82	0.639	82.0	5.927	11.4	1.18	0.16	45.88	11.32	425.27
11/10	73	4.36	0.65	10.1	0.03	2.49	0.00	0.60	0.641	82.2	6.020	11.6	1.22	0.13	46.01	6.67	431.94
11/11	74	5.40	0.70	10.1	0.02	1.48	0.00	0.64	0.643	82.4	6.089	11.7	1.23	0.11	46.12	4.93	436.87
11/12	75	4.79	0.75	10.2	0.01	1.18	0.00	0.59	0.643	82.5	6.122	11.8	1.25	0.05	46.17	2.39	439.26
11/13	76	4.89	0.75	10.1	0.00	1.03	0.00	0.54	0.643	82.5	6.150	11.8	1.27	0.00	46.17	2.03	441.29
11/14	77	4.88	0.80	10.1	0.00	0.97	0.00	0.59	0.643	82.5	6.178	11.9	1.29	0.00	46.17	1.98	443.27
11/15	78	4.84	0.75	10.2	0.00	0.93			0.643	82.5	6.199	11.9	1.31	0.00	46.17	1.49	444.76
11/16	79	Rest Cycle															
11/29	92						0.00	0.68					1.31	0.00	46.17	0.00	444.76
11/30	93	4.37	0.50	9.9	0.02	2.58	0.00	0.93	0.645	82.6	6.308	12.1	1.35	0.09	46.25	7.81	452.57
12/1	94	4.83	0.60	10.2	0.02	2.24	0.00	0.83	0.646	82.8	6.392	12.3	1.38	0.10	46.35	6.08	458.64
12/2	95	5.18	0.75	10.2	0.02	1.49	0.00	0.77	0.647	83.0	6.441	12.4	1.40	0.10	46.45	3.49	462.13
12/3	96	4.63	0.75	10.3	0.01	1.30	0.00	0.72	0.648	83.1	6.470	12.4	1.42	0.05	46.50	2.09	464.22
12/4	97	4.90	0.80	10.3	0.00	1.17	0.00	0.71	0.648	83.1	6.499	12.5	1.44	0.00	46.50	2.06	466.28
12/5	98	4.76	0.75	10.3	0.00	1.07	0.00	0.73	0.648	83.1	6.519	12.5	1.46	0.00	46.50	1.47	467.75
12/6	99	4.81	0.80	10.3	0.00	0.98			0.648	83.1	6.533	12.6	1.48	0.00	46.50	0.99	468.74
12/7	100	Rest Cycle															
12/20	113						0.00	0.71					1.48	0.00	46.50	0.00	468.74
12/21	114	4.24	0.45	10.1	0.02	2.45	0.00	0.00	0.649	83.2	6.627	12.7	1.52	0.08	46.59	6.77	475.51
12/22	115	4.52	0.65	10.3	0.00	2.28	0.00	0.04	0.649	83.2	6.771	13.0	1.55	0.00	46.59	10.31	485.82
12/23	116	5.02	0.80	10.4	0.00	1.22	0.00	0.04	0.649	83.2	6.853	13.2	1.57	0.00	46.59	5.92	491.74
12/24	117	5.11	0.75	10.2	0.01	0.53	0.00	0.06	0.650	83.3	6.888	13.2	1.59	0.05	46.64	2.50	494.24
12/25	118	5.01	0.70	10.4	0.02	0.35	0.00	0.09									

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		NaCN added		423.30 g	NaCN		1.00 g/L solution	g/mt ore		
Kilograms	71.75	NaCN Consumption		1.93 kg/mt ore					Au	Ag
		Lime:		2.7 kg/mt ore					Head Grade	59
Metric Tons	0.072								Head Screen	65
								Tail Screen		44

Daily Column Leach Test Data,
Sample I.D. [Underground Composite](#)

Feed Size [80%-19mm](#)

		Pregnant Solution Analyses					Barren Solution		Au Extraction		Ag Extraction		NaCN	Au		Ag	
Date	Days Leached	NaCN			Au ppm	Ag ppm	Analyses		Cum. g/mt ore	Cum. %	Cum. g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg
		Vol. l.	Conc. g/l	pH			Au ppm	Ag ppm									
1/11	135	4.64	0.50	10.0	0.00	1.40	0.00	0.14	0.652	83.6	7.018	13.5	1.69	0.00	46.79	6.09	503.57
1/12	136	4.76	0.70	10.3	0.02	1.32	0.00	0.15	0.653	83.8	7.096	13.6	1.71	0.10	46.88	5.57	509.14
1/13	137	4.90	0.75	10.3	0.01	0.73	0.00	0.14	0.654	83.9	7.135	13.7	1.73	0.05	46.93	2.81	511.95
1/14	138	4.85	0.75	10.3	0.00	0.46	0.00	0.13	0.654	83.9	7.156	13.8	1.75	0.00	46.93	1.52	513.47
1/15	139	5.01	0.80	10.3	0.00	0.40	0.00	0.17	0.654	83.9	7.175	13.8	1.77	0.00	46.93	1.34	514.81
1/16	140	4.71	0.80	10.3	0.00	0.36	0.00	0.16	0.654	83.9	7.187	13.8	1.78	0.00	46.93	0.83	515.64
1/17	141	4.79	0.80	10.4	0.00	0.38			0.654	83.9	7.201	13.8	1.80	0.00	46.93	1.00	516.64
1/18	142	Rest Cycle															
1/31	155						0.00	0.20					1.80	0.00	46.93	0.00	516.64
2/1	156	4.50	0.55	10.1	0.02	1.34	0.00	0.27	0.655	84.0	7.270	14.0	1.84	0.09	47.02	5.01	521.65
2/2	157	4.83	0.75	10.5	0.01	1.24	0.00	0.29	0.656	84.1	7.335	14.1	1.86	0.05	47.07	4.61	526.27
2/3	158	5.12	0.75	10.3	0.01	0.74	0.00	0.30	0.657	84.2	7.367	14.2	1.88	0.05	47.12	2.31	528.57
2/4	159	4.71	0.75	10.4	0.01	0.59	0.00	0.26	0.66	84.6	7.384	14.2	1.90	0.05	47.17	1.25	529.82
2/5	160	4.85	0.80	10.3	0.01	0.53	0.00	0.26	0.66	84.6	7.402	14.2	1.92	0.05	47.21	1.24	531.07
2/6	161	4.57	0.80	10.5	0.00	0.49	0.00	0.25	0.66	84.6	7.414	14.3	1.94	0.00	47.21	0.91	531.98
2/7	162	5.07	0.80	10.5	0.00	0.49			0.66	84.6	7.431	14.3	1.95	0.00	47.21	1.21	533.19
2/8	163	Rest Cycle															
2/21	176						0.00	0.30					1.95	0.00	47.21	0.00	533.19
2/22	177	4.41	0.50	10.1	0.00	1.39	0.00	0.40	0.66	84.6	7.495	14.4	1.99	0.00	47.21	4.60	537.79
2/23	178	5.01	0.70	10.4	0.00	1.22	0.00	0.42	0.66	84.6	7.552	14.5	2.01	0.00	47.21	4.07	541.86
2/24	179	4.92	0.75	10.3	0.01	0.77	0.00	0.35	0.66	84.6	7.575	14.6	2.03	0.05	47.26	1.65	543.51
2/25	180	4.75	0.75	10.3	0.00	0.72	0.00	0.38	0.66	84.6	7.598	14.6	2.05	0.00	47.26	1.64	545.14
2/26	181	4.84	0.80	10.3	0.01	0.62	0.00	0.37	0.66	84.6	7.613	14.6	2.07	0.05	47.31	1.06	546.21
2/27	182	4.58	0.80	10.4	0.01	0.59			0.66	84.6	7.624	14.7	2.09	0.05	47.36	0.82	547.02
		Rinse Cycle															
3/1	184	5.70	0.80	10.4	0.00	0.61			0.66	84.6	7.672	14.8	2.03	0.00	47.36	3.48	550.50
3/2	185	4.21	0.90	10.4	0.00	0.64			0.66	84.6	7.710	14.8	1.97	0.00	47.36	2.69	553.19
3/3	186	4.78	0.45	10.4	0.00	0.43			0.66	84.6	7.739	14.9	1.95	0.00	47.36	2.06	555.25
3/4	187	4.63	0.20	10.5	0.01	0.25			0.66	84.6	7.755	14.9	1.93	0.05	47.40	1.16	556.41
3/5	188	4.86	0.05	10.6	0.00	0.16			0.66	84.6	7.766	14.9	1.93	0.00	47.40	0.78	557.18
3/6	189	5.07	0.00	10.5	0.00	0.10			0.66	84.6	7.773	14.9	1.93	0.00	47.40	0.51	557.69
3/7	190	4.35	0.00	10.5	0.00	0.08			0.66	84.6	7.778	15.0	1.93	0.00	47.40	0.35	558.04
		Drain Down															
3/15	198	4.05	0.00	9.2	0.00	0.10			0.66	84.6	8	15.4	1.93	0.00	47.40	0.41	558.44
									Extracted, g/mt ore		0.66	84.6	8	15.4			
									Tail , g/mt ore		0.12			44			
									Calculated Head, g/mt ore		0.78			52			

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Kilograms	71.95	NaCN added		423.30 g	NaCN	1.00 g/L solution	-----		
		NaCN Consumption		1.86 kg/mt ore			Au	Ag	
Metric Tons	0.072	Lime:		2.00 kg/mt ore			Head Grade	0.68	44
							Head Screen	0.79	47
							Tail Screen	0.10	40

Daily Column Leach Test Data,
Sample I.D. Master Composite

Feed Size 80%-19mm

Date	Days Leached	Pregnant Solution Analyses					Barren Solution										
		NaCN					Analyses		Au Extraction		Ag Extraction		NaCN	Au		Ag	
		Vol. l.	Conc. g/l	pH	Au ppm	Ag ppm	Au ppm	Ag ppm	Cum. g/mt ore	Cum. %	Cum. g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg
8/30	1							0.000	0.0	0.000	0.0	0.00	0.00	0.00	0.00	0.00	0.00
8/31	2							0.000	0.0	0.000	0.0	0.07	0.00	0.00	0.00	0.00	0.00
9/1	3	1.94	0.55	10.2	5.70	29.30	0.01	0.04	0.154	21.1	0.790	1.6	0.13	11.06	11.06	56.84	56.84
9/2	4	4.70	0.80	11.8	2.29	13.80	0.00	0.03	0.303	41.4	1.689	3.5	0.15	10.71	21.77	64.66	121.50
9/3	5	5.11	0.80	11.8	1.33	10.30	0.00	0.04	0.397	54.4	2.418	5.0	0.16	6.80	28.57	52.48	173.98
9/4	6	4.99	0.80	11.6	0.62	5.69	0.01	0.06	0.440	60.3	2.810	5.9	0.18	3.09	31.66	28.19	202.17
9/5	7	4.92	0.85	11.2	0.41	4.35	0.00	0.05	0.467	64.0	3.103	6.5	0.19	1.97	33.63	21.10	223.26
9/6	8	5.17	0.85	10.6	0.28	3.61	0.00	0.06	0.487	66.8	3.359	7.0	0.20	1.45	35.07	18.41	241.67
9/7	9	4.97	0.75	10.6	0.22	3.09	0.00	0.05	0.503	68.9	3.568	7.4	0.22	1.09	36.17	15.05	256.72
9/8	10	4.89	0.90	10.6	0.18	2.75	0.00	0.08	0.515	70.5	3.751	7.8	0.23	0.88	37.05	13.19	269.92
9/9	11	4.58	0.90	10.4	0.13	2.46	0.00	0.08	0.523	71.7	3.902	8.1	0.24	0.60	37.64	10.86	280.77
9/10	12	5.58	0.85	10.4	0.13	2.11	0.00	0.13	0.533	73.0	4.060	8.5	0.24	0.73	38.37	11.37	292.14
9/11	13	5.02	0.75	10.3	0.08	1.72	0.00	0.10	0.539	73.8	4.171	8.7	0.26	0.40	38.77	7.97	300.11
9/12	14	5.04	0.70	10.3	0.08	1.57	0.00	0.12	0.544	74.6	4.274	8.9	0.29	0.40	39.17	7.40	307.51
9/13	15	4.91	0.60	10.3	0.06	1.45	0.00	0.12	0.549	75.1	4.364	9.1	0.32	0.29	39.47	6.51	314.02
9/14	16	4.88	0.45	10.2	0.06	1.30	0.00	0.11	0.553	75.7	4.444	9.3	0.36	0.29	39.76	5.73	319.75
9/15	17	4.98	0.45	10.4	0.06	1.18	0.00	0.10	0.557	76.3	4.518	9.4	0.40	0.30	40.06	5.32	325.07
9/16	18	4.77	0.50	10.1	0.05	1.10	0.00	0.11	0.560	76.7	4.584	9.5	0.43	0.24	40.30	4.74	329.81
9/17	19	5.22	0.60	10.2	0.05	1.07	0.00	0.13	0.564	77.2	4.654	9.7	0.46	0.26	40.56	5.02	334.83
9/18	20	4.94	0.65	10.1	0.04	1.04	0.00	0.12	0.566	77.6	4.716	9.8	0.49	0.20	40.76	4.47	339.31
9/19	21	5.05	0.65	10.1	0.04	0.99	0.00	0.15	0.569	78.0	4.777	10.0	0.51	0.20	40.96	4.39	343.69
9/20	22	4.92	0.65	10.0	0.03	0.98	0.00	0.16	0.571	78.3	4.833	10.1	0.54	0.15	41.11	4.06	347.75
9/21	23	4.98	0.70	10.2	0.03	1.02	0.00	0.16	0.573	78.5	4.892	10.2	0.56	0.15	41.26	4.26	352.01
9/22	24	4.82	0.75	10.3	0.02	1.02	0.00	0.17	0.575	78.7	4.949	10.3	0.58	0.10	41.35	4.10	356.11
9/23	25	5.64	0.75	10.2	0.02	0.87	0.00	0.20	0.576	78.9	5.006	10.4	0.59	0.11	41.46	4.04	360.15
9/24	26	4.79	0.75	10.1	0.02	0.91	0.00	0.22	0.578	79.1	5.052	10.5	0.61	0.10	41.56	3.34	363.49
9/25	27	5.06	0.75	10.1	0.02	0.92	0.00	0.20	0.579	79.3	5.101	10.6	0.63	0.10	41.66	3.53	367.03
9/26	28	5.05	0.75	10.3	0.02	0.96	0.00	0.08	0.580	79.5	5.154	10.7	0.65	0.10	41.76	3.83	370.85
9/27	29	5.11	0.75	10.4	0.02	0.84	0.00	0.22	0.582	79.7	5.208	10.9	0.67	0.10	41.86	3.88	374.74
9/28	30	4.96	0.75	10.4	0.02	0.80	0.00	0.23	0.583	79.9	5.248	10.9	0.69	0.10	41.96	2.85	377.58
9/29	31	5.11	0.75	10.4	0.01	0.83	0.00	0.25	0.584	80.0	5.291	11.0	0.71	0.05	42.02	3.07	380.65
9/30	32	5.17	0.80	10.4	0.00	0.82	0.00	0.31	0.584	80.0	5.332	11.1	0.72	0.00	42.02	2.96	383.62
10/1	33	4.86	0.75	10.4	0.01	0.89	0.00	0.28	0.585	80.1	5.370	11.2	0.74	0.05	42.06	2.74	386.36
10/2	34	4.96	0.75	10.1	0.02	0.88	0.00	0.27	0.586	80.3	5.411	11.3	0.76	0.10	42.16	2.94	389.30
10/3	35	4.52	0.70	10.2	0.02	0.89	0.00	0.29	0.587	80.4	5.447	11.3	0.79	0.09	42.25	2.65	391.94
10/4	36	4.66	0.75	10.3	0.01	0.89			0.588	80.5	5.485	11.4	0.81	0.05	42.30	2.67	394.61
10/5	37	Rest Cycle															
10/18	50						0.00	0.29					0.81	0.00	42.30	0.00	394.61
10/19	51	5.62	0.45	9.9	0.08	3.70	0.00	0.67	0.594	81.4	5.753	12.0	0.84	0.45	42.75	19.32	413.93
10/20	52	4.81	0.60	10.1	0.05	2.39	0.00	0.41	0.597	81.8	5.865	12.2	0.87	0.24	42.99	8.08	422.01
10/21	53	4.46	0.65	10.0	0.03	1.51	0.00	0.33	0.599	82.1	5.930	12.4	0.90	0.13	43.12	4.64	426.65
10/22	54	5.64	0.75	10.1	0.03	1.12	0.00	0.42	0.602	82.4	5.994	12.5	0.92	0.17	43.29	4.63	431.28
10/23	55	5.14	0.75	10.2	0.00	0.86	0.00	0.38	0.602	82.4	6.026	12.6	0.93	0.00	43.29	2.28	433.56
10/24	56	4.94	0.75	10.1	0.01	0.82	0.00	0.41	0.602	82.5	6.055	12.6	0.95	0.05	43.34	2.11	435.67
10/25	57	4.23	0.70	10.1	0.01	0.86			0.603	82.6	6.077	12.7	0.98	0.04	43.38	1.55	437.22
10/26	58	Rest Cycle															
11/8	71						0.00	0.38					0.98	0.00	43.38	0.00	437.22
11/9	72	5.44	0.45	9.9	0.05	3.01	0.00	0.88	0.607	83.1	6.277	13.1	1.02	0.27	43.66	14.44	451.66
11/10	73	4.65	0.55	10.1	0.03	2.15	0.00	0.62	0.609	83.4	6.354	13.2	1.06	0.14	43.80	5.51	457.17
11/11	74	5.44	0.65	10.1	0.02	1.48	0.00	0.62	0.610	83.6	6.422	13.4	1.08	0.11	43.91	4.89	462.06
11/12	75	4.69	0.75	10.2	0.01	1.25	0.00	0.58	0.611	83.7	6.459	13.5	1.10	0.05	43.95	2.70	464.76
11/13	76	5.21	0.80	10.1	0.00	1.06	0.00	0.57	0.611	83.7	6.495	13.5	1.11	0.00	43.95	2.56	467.32
11/14	77	5.03	0.75	10.2	0.00	0.98	0.00	0.63	0.611	83.7	6.523	13.6	1.13	0.00	43.95	2.02	469.34
11/15	78	4.99	0.75	10.2	0.00	0.98			0.611	83.7	6.547	13.6	1.15	0.00	43.95	1.68	471.02
11/16	79	Rest Cycle															
11/29	92						0.00	0.70					1.15	0.00	43.95	0.00	471.02
11/30	93	4.46	0.45	10.1	0.04	3.14	0.00	0.89	0.613	84.0	6.692	13.9	1.19	0.18	44.13	10.43	481.46
12/1	94	5.04	0.50	10.3	0.02	2.05	0.00	0.86	0.615	84.2	6.772	14.1	1.23	0.10	44.23	5.79	487.25
12/2	95	5.31	0.75	10.2	0.01	1.48	0.00	0.81	0.615	84.3	6.820	14.2	1.24	0.05	44.28	3.47	490.72
12/3	96	4.79	0.75	10.3	0.00	1.34	0.00	0.74	0.615	84.3	6.852	14.3	1.26	0.00	44.28	2.29	493.01
12/4	97	5.06	0.75	10.3	0.00	1.21	0.00	0.77	0.615	84.3	6.885	14.3	1.28	0.00	44.28	2.35	495.36
12/5	98	4.95	0.75	10.3	0.00	1.11	0.00	0.76	0.615	84.3	6.907	14.4	1.30	0.00	44.28	1.57	496.92
12/6	99	4.75	0.75	10.3	0.00	1.05			0.615	84.3	6.922	14.4	1.32	0.00	44.28	1.11	498.04
12/7	100	Rest Cycle															
12/20	113						0.00	0.69					1.32	0.00	44.28	0.00	498.04
12/21	114	4.72	0.55	10.3	0.04	2.90	0.00	0.03	0.618	84.7	7.063	14.7	1.36	0.19	44.47	10.17	508.21
12/22	115	4.94	0.65	10.4	0.02	1.99	0.00	0.05	0.619	84.9	7.198	15.0	1.38	0.10	44.57	9.68	517.88
12/23	116	4.83	0.65	10.4	0.00	0.99	0.00	0.05	0.619	84.9	7.261	15.1	1.41	0.00	44.57	4.53	522.41
12/24	117	5.21	0.75	10.2	0.00	0.53	0.00	0.07	0.619	84.9	7.296	15.2	1.43	0.00	44.57	2.51	524.92
12/25	118	5.19	0.70														

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		NaCN added	423.30 g	NaCN	1.00 g/L solution	-----	
Kilograms	71.95	NaCN Consumption	1.86 kg/mt ore			Au	Ag
						Head Grade	0.68 44
Metric Tons	0.072	Lime:	2.00 kg/mt ore			Head Screen	0.79 47
						Tail Screen	0.10 40

Daily Column Leach Test Data,
Sample I.D. Master Composite

Feed Size 80%-19mm

Pregnant Solution Analyses							Barren Solution										
Date	Days Leached	NaCN					Analyses		Au Extraction		Ag Extraction		NaCN	Au		Ag	
		Vol. l.	Conc. g/l	pH	Au ppm	Ag ppm	Au ppm	Ag ppm	Cum. g/mt ore	Cum. %	Cum. g/mt ore	Cum. %	Consumed kg/mt ore	mg	cum. mg	mg	cum. mg
1/11	135	4.76	0.40	10.1	0.02	1.74	0.00	0.17	0.621	85.0	7.460	15.5	1.54	0.10	44.67	7.82	536.76
1/12	136	4.91	0.55	10.3	0.02	1.16	0.00	0.17	0.622	85.2	7.527	15.7	1.57	0.10	44.77	4.83	541.59
1/13	137	4.95	0.70	10.2	0.01	0.72	0.00	0.16	0.623	85.3	7.565	15.8	1.59	0.05	44.81	2.70	544.29
1/14	138	5.03	0.75	10.3	0.00	0.52	0.00	0.16	0.623	85.3	7.590	15.8	1.61	0.00	44.81	1.80	546.09
1/15	139	5.18	0.75	10.2	0.00	0.46	0.00	0.18	0.623	85.3	7.612	15.9	1.63	0.00	44.81	1.57	547.65
1/16	140	4.87	0.75	10.2	0.00	0.43	0.00	0.17	0.623	85.3	7.628	15.9	1.65	0.00	44.81	1.18	548.83
1/17	141	4.94	0.75	10.3	0.00	0.44			0.623	85.3	7.646	15.9	1.67	0.00	44.81	1.31	550.14
1/18	142	Rest Cycle															
1/31	155						0.00	0.20						0.00	44.81	0.00	550.14
2/1	156	4.60	0.50	10.4	0.03	1.82	0.00	0.30	0.625	85.6	7.748	16.1	1.71	0.14	44.95	7.35	557.49
2/2	157	4.96	0.55	10.4	0.01	1.14	0.00	0.28	0.625	85.7	7.806	16.3	1.74	0.05	45.00	4.12	561.61
2/3	158	5.41	0.65	10.2	0.00	0.76	0.00	0.29	0.625	85.7	7.843	16.3	1.76	0.00	45.00	2.68	564.30
2/4	159	4.89	0.75	10.3	0.00	0.63	0.00	0.26	0.625	85.7	7.865	16.4	1.78	0.00	45.00	1.60	565.90
2/5	160	5.06	0.75	10.3	0.00	0.56	0.00	0.25	0.625	85.7	7.886	16.4	1.80	0.00	45.00	1.51	567.41
2/6	161	4.76	0.75	10.4	0.00	0.51	0.00	0.25	0.625	85.7	7.902	16.5	1.82	0.00	45.00	1.15	568.56
2/7	162	5.22	0.75	10.4	0.00	0.50			0.625	85.7	7.921	16.5	1.84	0.00	45.00	1.34	569.89
2/8	163	Rest Cycle															
2/21	176						0.00	0.30					1.84	0.00	45.00	0.00	569.89
2/22	177	4.50	0.45	10.1	0.02	1.73	0.00	0.44	0.627	85.9	8	16.7	1.88	0.09	45.09	6.26	576.15
2/23	178	5.21	0.65	10.3	0.00	1.15	0.00	0.41	0.627	85.9	8	16.7	1.90	0.00	45.09	3.75	579.90
2/24	179	5.07	0.70	10.3	0.00	0.79	0.00	0.33	0.627	85.9	8	16.7	1.92	0.00	45.09	1.91	581.81
2/25	180	4.93	0.80	10.3	0.01	0.74	0.00	0.35	0.627	85.9	8	16.7	1.94	0.05	45.14	1.97	583.78
2/26	181	5.00	0.75	10.3	0.00	0.63	0.00	0.34	0.627	85.9	8	16.7	1.96	0.00	45.14	1.37	585.14
2/27	182	4.69	0.70	10.3	0.01	0.59			0.628	86.0	8	16.7	1.99	0.05	45.19	1.03	586.17
		Rinse Cycle															
3/1	184	5.88	0.70	10.4	0.00	0.59			0.628	86.0	8	16.7	1.93	0.00	45.19	3.47	589.64
3/2	185	4.46	0.80	10.4	0.00	0.61			0.628	86.0	8	16.7	1.88	0.00	45.19	2.72	592.36
3/3	186	4.77	0.20	10.3	0.00	0.34			0.628	86.0	8	16.7	1.86	0.00	45.19	1.62	593.99
3/4	187	5.11	0.10	10.4	0.00	0.19			0.628	86.0	8	16.7	1.86	0.00	45.19	0.97	594.96
3/5	188	4.86	0.00	10.4	0.00	0.15			0.628	86.0	8	16.7	1.86	0.00	45.19	0.73	595.68
3/6	189	5.26	0.00	10.2	0.00	0.09			0.628	86.0	8	16.7	1.86	0.00	45.19	0.47	596.16
3/7	190	4.37	0.00	10.2	0.00	0.08			0.628	86.0	8	16.7	1.86	0.00	45.19	0.35	596.51
		Drain Down															
3/15	198	3.59	0.00	9.1	0.00	0.10			0.63	86.3	8	16.7	1.86	0.00	45.19	0.36	596.87
Extracted, g/mt ore									0.63	86.3	8	16.7					
Tail, g/mt ore									0.10		40						
Calculated Head, g/mt ore									0.73		48						

**Table A6-1. - Drain Down Rate Tests, Column Leached Residues,
Longstreet Mine Composites, 80%-19mm Feed Size**

Drain Time, hours	Effluent Solution								
	Surface Composite			Underground Composite			Master Composite		
	Liters	Cum. L/mt ore	Rate, L/hr/mt	Liters	Cum. L/mt ore	Rate, L/hr/mt	Liters	Cum. L/mt ore	Rate, L/hr/mt
0.08	0.017	0.24	2.951	0.015	0.21	2.611	0.015	0.21	2.604
0.25	0.032	0.68	2.614	0.030	0.63	2.458	0.218	3.24	17.810
0.50	0.051	1.39	2.833	0.047	1.28	2.618	0.046	3.88	2.556
1.0	0.104	2.83	2.889	0.097	2.63	2.702	0.094	5.18	2.611
2.0	0.179	5.32	2.486	0.780	13.50	10.864	0.176	7.63	2.444
4.0	0.191	7.97	1.326	0.220	16.56	1.532	0.208	10.51	1.444
8.0	0.160	10.19	0.556	0.190	19.21	0.662	0.188	13.13	0.653
24	0.191	12.85	0.166	0.301	23.40	0.262	0.257	16.69	0.223
48	0.085	14.03	0.049	0.176	25.85	0.102	0.129	18.49	0.075
72	0.026	14.39	0.015	0.080	26.96	0.046	0.053	19.22	0.031
96	0.006	14.47	0.003	0.035	27.45	0.020	0.024	19.56	0.014
120				0.019	27.72	0.011	0.011	19.71	0.006
144				0.011	27.87	0.006	0.004	19.76	0.002

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