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Hycroft Project

Technical Report Summary
Heap Leaching Feasibility Study
Winnemucca, Nevada, USA

Authors:

M3 Engineering and Technology
Steven Newman, Registered Member SME
Richard F. DeLong, P. Geo
Brooke Miller Clarkson, CPG
Tim Carew, P. Geo
Matt Hartmann, MAusIMM, RM-SME

Prepared For:

HYCROFT MINING CORPORATION

DATE AND SIGNATURES PAGE

This report is effective as of July 31, 2019.

M3 Engineering and Technology	**signed** *"M3 Engineering"*
Steven Newman	**signed** *"Steven Newman"*
Richard F. DeLong	**signed** *"Richard F. DeLong"*
Brooke Miller Clarkson	**signed** *"Brooke Miller Clarkson"*
Tim Carew	**signed** *"Tim Carew"*
Matt Hartmann	**signed** *"Matt Hartmann"*

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1 EXECUTIVE SUMMARY

1.1 PRINCIPAL FINDINGS

This Technical Report Summary has been prepared by M3 Engineering and Technology ("M3"), in association with SRK Consulting (U.S.), Inc. ("SRK") and Hycroft Mining Corporation ("Hycroft Mining" or "HMC" or "the Company"), following the reporting requirements of following the reporting requirements of the United States Securities and Exchange Commission's ("SEC") new mining rules under subpart 1300 and item 601 (96)(B)(iii) of Regulation S-K (the "New Mining Rules").

This Technical Report Summary provides results of the Hycroft Heap Leach feasibility study that evaluated the possibility of oxidizing and leaching transitional and sulfidic material in a heap leach application. HMC is seeking patent protection for these processes and applications. Presented herein are an updated mineral reserve and resource estimate, a supporting life-of-mine plan and the results of metallurgical testing to determine the applicability of oxidizing and leaching transition and sulfide ore in a heap leach process. The metallurgical testing includes three phases of the ongoing test program using extensive column and bottle roll test work.

Up to July 2015, Hycroft was mining and conducting heap leach operations, including the operations of two Merrill-Crowe plants. Through the first quarter of 2017, operations were limited to treatment of existing material on the leach pads using conventional cyanide heap leaching methods and recovery of a small fraction of its gold and silver production initially with the Merrill-Crowe plants and then later by carbon adsorption, from which loaded carbon was sold offsite for processing. At the end of the first quarter of 2017 and through December 31, 2018, Hycroft entered a care and maintenance mode whereby gold and silver produced were a byproduct of the maintenance activities. Mining operations resumed in the second quarter of 2019 and gold and silver production is expected to begin in the third quarter of 2019.

As part of a restart, the Company began construction of nine individual heap leach test pads to demonstrate on a commercial scale the oxidation and leaching process it has been developing over that last four years. To date, five of those test pads have been loaded with ore and are in various stages of the process.

This feasibility study includes updated mineral resources and the associated mine plan, updated operating parameters determined through ongoing testwork and updated financial metrics. The feasibility analyzes a full-scale operation including construction of new leach pads and expanded mining activities. Key components of the process that currently exist onsite include heap leach pads, a crushing facility consisting of primary, secondary, and tertiary crushing, two Merrill-Crowe plants having a total capacity of 26,000 gpm and associated support facilities.

HMC intends to implement the full-scale operation at Hycroft in the manner described in this Technical Report Summary, subject to financing and acquisition of the required permits. Table 1-1 summarizes the key statistics for the project.

Table 1-1: Hycroft Technical Report Summary Relevant Statistics

Mine Life (years)	34 years
Mine Type	Open Pit
Process Description	Heap Leach – Run of mine and ¾" crushing (oxide/transition), ½" crushing and oxidation (transition/sulfide), dedicated leach pad
Total Heap Leach Ore Mined, short tons (000s)	1,133,060
Strip Ratio	1.17
LOM Gold Ore Grade	0.011
LOM Silver Ore Grade	0.425
Initial Capital Costs ($US Millions)	$230.8
Sustaining Capital Costs ($US Millions)	$537.6
Adjustment for Escalation	None – Assumed Q2 2019 dollars
Payable Metals	
Gold (Million ounces)	7.8
Silver (Million ounces)	344.1

Unit Operating Cost:	**First 5 Years Full Operation (2020-2024)[1]**	**First 10 Years Full Operation (2020-2029)[1]**	**All Years (2019-2052)**
Mining Cost /ton mined	$1.75	$1.57	$1.61
Heap Leach Processing Cost /ton processed	$3.85	$3.98	$4.01
G&A Cost (includes Treatment & Refining, Transport, Royalties, Net Proceeds Tax) /ton processed	$0.84	$0.82	$0.86
Total Cost per Ton Processed	**$8.46**	**$8.47**	**$8.52**
By-Product Credits (Silver) /ton processed	$3.56	$4.53	$5.26
Net Operating Cost per Ton Processed	**$4.90**	**$3.94**	**$3.25**

Financial Indicators	**Base Case**	**High Metal Price**	**Moderate Metal Price**	**Low Metal Price[2]**
Gold Price (per troy ounce)	**$1,300**	$1,500	$1,400	$1,200
Silver Price (per troy ounce)	**$17.33**	$20.00	$18.67	$16.50
After Tax Project Internal Rate of Return (IRR)	**148.6%**	N/A[3]	307.9%	80.2%
After Tax NPV at 5% Discount Rate ($ Billions)	**$2.1**	$3.0	$2.6	$1.7
After Tax Payback (years)	**2.5**	<1.0	2.2	3.1

Major Permit Status	
Plan of Operations for EIS (AAO process and mining below the water table)	Expected to be received by year end 2019
Plan of Operations Amendment for Stage 1 HLF	Approval Received July 2019
Water Pollution Control Permit Modification for Stage 1 HLF	Expected to be received by year end 2019

1. 2019 has been excluded from the calculation as it is only a partial ramp-up year.
2. Low Metal Price is the basis for Mineral Reserve.
3. IRR is N/A as this price scenario generates cash flow in the first year.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Hycroft Mine is located 54 miles west of Winnemucca in Humboldt and Pershing Counties, Nevada, USA. The Hycroft property consists of 30 private parcels that comprise approximately 1,912 acres and 3,247 unpatented mining claims that encompass approximately 68,759 acres.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Access to the Hycroft Mine from Winnemucca is by means of State Route 49 (Jungo Road). A major east-west railway passes through the Hycroft claim position. There are no streams, rivers, or major lakes in the project area.

The climate for the region is arid, with precipitation averaging 7.7 inches per year. Temperatures during the summer are generally 50°F at night and near 90°F during the day. Winter temperatures are usually 20°F at night and 40°F during the day.

The current mine site has a truck shop, crushing facilities, ore processing facilities, administrative buildings, as well as other service-related structures. Electricity is currently supplied by NV Energy via overhead transmission lines. A modern communications system exists at the site.

1.4 HISTORY

Mining at Hycroft began in 1983 with a small heap leach operation known as the Lewis Mine. Lewis Mine production was followed by production from the Crofoot property in the Bay, South Central, Boneyard, Gap and Cut-4 pits along the Central Zone. Production from the north end of the Brimstone pit continued until December 1998. Due to gold prices averaging below $300/oz, the mine was placed on a care and maintenance program though processing continued through 2004 when mining ceased in December 1998.

Vista acquired the Lewis Mine in early 1987 from F. W. Lewis, Inc., and the Crofoot Mine in April 1988. The remaining leasehold interest in the Lewis property was purchased by Vista in December 2005.

The Hycroft Mine produced approximately 1.2 million ounces of gold and 2.5 million ounces of silver from 1983 to 1998 when the operations were suspended. An additional 58,700 ounces of gold was produced from the leaching and rinsing of the heap leach pads from 1999 through 2004, after the mine was placed on care and maintenance.

In May 2007, the Nevada-based holdings of Vista were spun out into Allied Nevada Gold Corp. The Hycroft Mine was included as part of the transfer of ownership allowing Allied Nevada to explore, expand, and develop the resources at Hycroft. The Hycroft Mine was reactivated in September 2007 and produced its first doré in December 2008, and achieved planned ore production by the end of 2009.

With the construction of the North leach pad in 2013, the total leach pad space for the Brimstone, Lewis and North leach pads was increased to more than 20 million square feet. In 2010, the mine began an expansion program that included construction of a 21,000 gallon per minute Merrill-Crowe processing plant and a three-stage crushing facility as well solution pumping capacity upgrades.

Active mining was stopped at Hycroft in June 2015 due to low metal prices, but active leaching of previously mined ore continued through 2018. At the end of the first quarter of 2017, Hycroft began a care and maintenance mode, while producing gold and silver as a byproduct of the maintenance activities.

On October 22, 2015, Allied Nevada emerged from its financial restructuring and changed its name to Hycroft Mining Corporation.

In late 2018, Hycroft began construction of new leach pads to demonstrate its recently developed heap oxidation and leach process at a commercial scale. Additionally, in January 2019 Hycroft began preparing the mine, including its facilities and mining equipment, for a restart. Active mining began in April 2019, with a focus on transition and sulfide material. Ore has been placed on the new leach pads and is in the active oxidation phase. Production of gold and silver is expected in the third quarter of 2019.

Since the Hycroft mine was reactivated by HMC in September 2007 through July 2019, metal sales have totaled approximately 900,000 ounces of gold and 5.0 million ounces of silver.

1.4.1 Mining History

Mining in the Sulfur District, where the Hycroft Mine is located, began in the late 1800's for native sulfur (Couch and Carpenter, 1943; Wilden, 1964). Mining of native sulfur was sporadic from 1900 to 1950 with over 181,488 tons of sulfur ore, grading approximately 20-35% sulfur (McLean, 1991).

High-grade silver, consisting of nearly pure seams of cerargyrite ($AgCl$), was also mined in 1908 at Camel Hill (Vandenburg, 1938) until 1912. Minor silver mining also occurred along the East Fault at the Snyder Adit (Friberg, 1980; Bates, 2001).

During the First World War, veins of nearly pure alunite were mined in the southern part of the Sulfur District (Clark, 1918). In 1931, several hundred tons of alunite were mined as a soil additive (Fulton and Smith, 1932). Vandenburg estimated that 454 tons of alunite was shipped to the west coast to be used as fertilizer (Vandenburg, 1938). From 1941 to 1943, cinnabar was mined from small pits in the exposed acid leach zone (Bailey, 1944). Total mercury production during this period is estimated at 1,900 lbs (McLean, 1991).

1.4.2 Exploration History

In 1966, the Great American Minerals Company began extensive exploration for native sulfur in the area of the Hycroft Mine. Approximately 200 shallow holes were drilled, and numerous trenches dug (Friberg, 1980). In 1974, the Duval Corporation (Duval) drilled 20 holes on the Hycroft property in search of a Frasch-type sulfur deposit (Wallace, 1980). Duval found no evidence of a sulfur deposit at depth, but did report elevated gold and silver values. Duval drilled two core holes (DC-1 and DC-2) and 18 rotary holes (DR-3 through 20) (Ware, 1989).

In 1977, the Cordex Syndicate mapped and rock chip sampled the Hycroft property, recognizing the potential for a bulk tonnage, low-grade precious metal deposit. In 1978, Homestake became interested in the property, recognizing similarities with the McLaughlin hot springs deposit in California. Homestake completed surface sampling and exploration drilling during 1981-1982, and although successful in defining an oxide gold/silver ore body, they dropped the property in 1982.

HRDI gained control of the district in 1985 and drilled 3,212 exploration holes, totaling 965,552 feet, between 1985 and 1999. The bulk of this drilling was shallow and focused on oxide gold mineralization at Central, Bay and Brimstone. In 2005, Canyon Resources completed 33 drill holes totaling 13,275 feet of reverse circulation (RC) drilling. These were completed primarily in the Brimstone pit area.

1.5 GEOLOGICAL SETTING AND MINERALIZATION

The Hycroft Mine is located on the western flank of the Kamma Mountains in the Basin and Range physiographic province of northwestern Nevada. The Kamma Mountains were formed during the Miocene to Quaternary Epoch from the uplift of Jurassic basement rock and emplacement of Tertiary volcanic and sedimentary rocks. The stratigraphy along the western flank of the range is down dropped to the west, along a series of north to northeast striking normal faults. These faults served as conduits for hydrothermal fluids that deposited the Hycroft mineralization.

Hycroft is a large, epithermal, low sulfidation, hot springs deposit. Gold and silver mineralization occurs as both disseminated and vein-controlled, with gold values ranging from detection to 8.8 ounces per ton (opt), and silver ranging from detection to 647.5 opt.

1.6 DEPOSIT TYPES

The deposit is typically broken into six major zones based on geology, mineralization, and alteration. These zones include Brimstone, Vortex, Central, Bay, Boneyard, and Camel. Breaks between the zones are major faults.

Mineralization at Hycroft has been deposited through multiple phases. An early silica sulfide flooding event deposited relatively low-grade gold and silver mineralization, generally along bedding. This mineralization is cross cut by later, steeply dipping quartz alunite veins. Late stage silver bearing veins are found in the Vortex zone and at depth in the Central area. Late to present supergene oxidation along faults has liberated precious metals from sulfides and further enriched gold and silver mineralization, along water table levels.

1.7 EXPLORATION

In addition to drilling activity, Hycroft Mining has also conducted geophysical surveys, soil and rock chip sampling programs, field mapping, historical data compilation, and regional reconnaissance at Hycroft. These efforts are designed to improve the understanding of the known mineralization, as well as provide data for further exploration of the greater property position. Through these activities, Hycroft Mining has identified and estimated mineral resources and mineral reserves on the property as well as a number of targets, mostly concentrated in the southern area of the claim block.

Regional exploration data from Homestake, LAC Minerals, USMX, HRDI, and others have been compiled from both in-house and public data sources. Approximately 250 drill holes, various soil and rock chip locations and results, and various field maps have been identified at present.

1.8 DRILLING

The Hycroft exploration model includes data from 1981 to December 2018 and includes 5,501 holes, representing 2,482,722 feet of drilling. There have been 5,576 drill holes completed in the Hycroft Project area; some are water wells or are outside the resource model domain and were not applied to estimation. Exploration drilling was started in 1974 by Duval Corporation, and continued through various owners including Homestake, HRDI and Canyon Resources. This historic drilling was conducted prior to the New Mining Rules reporting requirements. In the QP's opinion, no significant issues have been identified with the historic data and therefore the historic drilling and assay results are incorporated into the Hycroft model.

HMC commenced systematic exploration and resource development drilling starting in 2006. Drilling has been focused on oxide reserve delineation, sulfide resource definition, sulfide exploration, condemnation drilling for facilities, silver data and both geotechnical and metallurgical core samples. A combination of rotary, reverse circulation and core drilling techniques has been utilized to evaluate the nature and extent of mineralization. From late 2006 to August 31, 2016, HMC completed 1,970 exploration holes, totaling approximately 1.45 million feet.

1.9 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Hycroft drill hole samples were shipped to accredited, independent laboratories in Reno or Elko, Nevada, for sample preparation and analysis. Sample security and handling procedures were not investigated in detail by the QP, because the programs were completed prior the QP's involvement, except for the 2018 sonic drilling program. However, it is the QP's opinion that sample handling, preparation, and analysis methods meet current industry standards for quality.

Industry standard sampling of reverse circulation and core drill holes is utilized by HMC. The HMC QA/QC program includes analysis of standard reference materials, inert blanks, and duplicate pulps, as well as check assays by umpire laboratories. The program has been designed to ensure that at least one standard and one blank are inserted into the drill sample stream for every 40 drill samples (200 ft), which are the number of HMC samples in each ALS Chemex analytical batch. In practice, the insertion rates for the QA/QC samples are somewhat higher, based on drill-hole depth.

A transmittal sheet for both the bagged core and RC samples by drill hole was prepared for submission to the laboratory. Once at the laboratory, samples were prepared from a split of 70% passing minus 3 mesh if pieces were too large to fit in the pulverizer, and further crushing of 70% passing minus 10 mesh. A 2.2-pound split is taken and pulverized to 85% passing minus 200 mesh.

No officers, directors, or associates of the issuer were operationally involved with the routine sample preparation.

Following analysis, a complete digital file including corresponding unique self-identifying sample numbers for each sample is provided to HMC. These results are uploaded into an acQuire database and further checked using electronic methods.

1.10 DATA VERIFICATION

The HMC drill hole database has been validated by the HMC exploration group for previous technical reports. A review and validation of the HMC collar coordinate, down-hole survey, and geology data was completed in Q3 2014 by HMC geologists.

SRK completed data verification and validation in advance of geological modeling and resource estimation, first between May and July 2017, for gold, silver, sulfide sulfur, and total sulfur analytical results, and for logged geological data. During this review, the analytical databases were found to be incomplete. SRK worked with Hycroft to extract all available analytical data from the acQuire database. This resulted in a 58% increase in the sulfide sulfur dataset. The compilation of gold and silver assay values in parts per million (PPM) units resulted in more intervals with valid cyanide soluble gold to fire assay gold (Au CN:FA) values for oxide modeling, and greater precision for grade estimation. SRK completed data verification for the new analytical database in September 2017. Verification of the sulfide stockpile drilling data was completed in 2019 for the resource estimation update.

1.11 MINERAL PROCESSING AND METALLURGICAL TESTING

Hycroft Mining has been operating the Hycroft open pit heap leach facility to produce gold and silver since 2008. Prior to that, Hycroft was operated in a similar manner by Vista Gold. The cumulative performance statistics and metallurgical test data gathered are extensive.

Previous testing and feasibility analysis indicated that transition and sulfide ore can be oxidized in a heap leach operation prior to irrigation with cyanide solution. The objective of this study is to update the previous study with recent testwork and assumptions. This process, which is the subject of a pending patent application, will accomplish two goals, namely, the liberation of gold in the sulfides by oxidation using soda ash to manage pH and alkalinity, thereby increasing its recovery, and the reduction of the heap's potential to turn acidic during cyanide leaching.

Ore is classified as "oxide," "transitional," or "sulfide," depending on the solubility of its gold content in cyanide solution (refractoriness). Ores having cyanide soluble gold contents of 70% or higher are classified as oxide ore. Those with cyanide-soluble gold contents below 30% are considered sulfide. The remainder, with cyanide-soluble contents between 30 to 70% are considered transition ores.

The classification has been shown to have no correlation with sulfide sulfur content. The mining schedule developed considers the recovery of gold and silver from these ores plus the cost of treatment.

1.11.1 Historical Test Work

Beginning in 2007, Hycroft Mining examined milling options to expand production, including direct cyanidation of high-grade oxide ore, and production of a flotation concentrate from sulfide ore, followed by an oxidative treatment of the concentrate. The original focus was on oxidation methods primarily employed in the Nevada gold industry, including pressure oxidation (POX) and roasting. Test work on these processes showed that each of these options work well.

In 2013, the Company began testing a suite of alternative oxidation methods, including chlorination, ambient pressure alkaline oxidation, fine grinding with intense cyanidation, and a procedure similar to the Albion process. The goal was to develop an economically viable process that would be less expensive to build and operate than autoclaves and that would eliminate the need for offsite concentrate sales.

Batch test results were positive and indicated that Hycroft concentrates were amenable to oxidation under atmospheric conditions, using trona to create the appropriate alkaline environment to promote oxidation. Continuous pilot plant testing on three main domains was completed at Hazen Research to confirm these results.

In 2016, the viability of the atmospheric oxidation process using trona was demonstrated in a 10 ton-per-day integrated pilot plant at the mine site. This plant included primary grinding of 3/8” material, followed by flotation, atmospheric oxidation, cyanide leaching, counter-current decantation (CCD) and Merrill-Crowe precipitation.

The objective of the current study is to determine if soda ash, refined from trona, can be used during the oxidation of sulfides in a heap leach operation prior to irrigation with cyanide solution. This process, which is the subject of a pending patent application, will accomplish two goals, namely, the liberation of gold and silver in the sulfides by oxidation, thereby increasing its recovery, and the reduction of the heap’s potential to turn acidic during cyanide leaching.

Over a decade of research into various carbonate oxidation systems has laid the foundation for the pre-oxidation and cyanidation process. A history of processes that have contributed to the development of this technology is included to show the progression of the mechanism used for oxidation as well as the logic that led to current operating procedures.

1.11.2 Recent Heap Leach Test Work

Hycroft explored the application of trona (or soda ash) in heap leaching sulfide and transition ores. The interest was initially in the potential of faster restoration of heaps that have become acidic by utilizing the higher solubility of trona or soda ash in water compared to lime. As an extension of this logic, the interest developed in whether trona could provide enough neutralizing power to enable heap leaching of transition and sulfide ores.

Hycroft began investigating the potential of oxidizing and leaching transition and sulfide ores with preliminary column tests. Simultaneously, Hycroft built two test pads, running ore samples from the Central and Brimstone deposits. Some of the results from these tests indicate that oxidation in a heap in the presence of trona can transform sulfide ores into transition and oxide ores (increased cyanide-soluble gold) and improve gold recovery in transition ores. The results encouraged Hycroft to continue testing the process to optimize the conditions and to better understand the mechanism of oxidation.

Based on preliminary tests, oxidation and leaching were performed in sequence in order to separate cyanide from the carbonate/bicarbonate solutions. The general procedure, similar to the current study, is discussed in detail later in this section. Oxidation was estimated by the amount of total sulfate produced.

The original oxidation target was about 45%, which was chosen based on the recovery versus oxidation plot developed from the concentrate oxidation study. The goal was to attain 55 to 70% gold recovery. The results indicate gold recovery targets were achievable at lower oxidation rates than expected. Phase II column leach test have exceeded the expectations derived from the recovery versus oxidation curve.

The oxidation and cyanide leach tests were conducted in plexiglass cylindrical columns that were 1 foot in diameter and 4 feet high. Ore samples were crushed to nominal P100 = ½ inch, blended and loaded into the columns.

As established in Phase II testing based on preliminary experiments, oxidation and leaching had to be performed in sequence in order to separate cyanide from the carbonate/bicarbonate solutions.

Between the oxidation and leach stages, the columns were rinsed with water followed by lime-saturated water. The objective of the water rinse is to remove as much of the sulfate produced and excess carbonate alkalinity as practicable from the ore column. Sulfate that remains will react with calcium in the leach solution to precipitate $CaSO_4$, which could form a passivation layer over the solids that are being leached. Bicarbonate has been shown to react with cyanide resulting in high cyanide consumptions. The objective of the lime-water rinse is to neutralize residual bicarbonate after the water rinse. Depending on the efficiency of the water rinse, the lime-saturated rinse may not be required but this will have to be tested to determine the trade-off between the cost of lime-water rinse and the cyanide loss.

Oxidation was performed for different periods ranging from 60 days to 180 days, by adding soda ash to the ore column and applying just enough solution to the column to keep the ore wet. This status is maintained to ensure that the interstices in the ore column are filled with oxygen-supplying air and not flooded with solution. A small amount of solution is allowed to drain at the bottom of the column, enough to collect at least 50 ml of sample each day for pH analysis, and to create a weekly composite for sulfate analysis. Oxidation was tracked by the amount of sulfate produced.

Phase III also introduced a new step to the procedure and that is to add iron (as ferric chloride) to the oxidation solutions at the start of the tests. This was based on an inference from Phase II results that oxidation of sulfides is essentially driven by the ferrous-ferric redox couple, which can be maintained at around pH 10 in a carbonate environment.

For the testing program, the bulk of solution and solids assays were performed by McClelland Laboratories in Reno. Some chemical analyses were conducted in-house (Hycroft Laboratory) for confirmation, control samples and for time-sensitive assays. M3 reviewed the chemical analysis procedures on site and found them to be in accordance with standard analytical practice.

Based on the results available so far, the projected recoveries have not changed much from the Phase 2 testing. Table 1-2 is a summary of the operating parameters and metal recoveries proposed for heap leach modelling to develop a metal production schedule.

From the overall trend observed so far in the test results, it appears that gold recoveries of 70% are possible for all the domains if the conditions are right. It is recommended that testing be continued using optimal conditions to provide experimental support for this recovery target. These optimal conditions include soda ash dosage, crush size, oxidation time, maintaining moist conditions during oxidation and ensuring access to air. During operations, testing of ore is likewise recommended to fine tune the conditions to be used in the heap. The duration of the oxidation cycle is variable and dependent on parameters found in the head assay.

Table 1-2: Operating Parameters and Expected Recoveries for Heap Leaching

Domain	Nominal* Target Oxidation, %	CN^- Leach Time, days	Au Recovery, %	Ag Recovery, %
Northwest (Bay)	31	60	55	55
West (Central)	40	60	70	70
Southwest (Camel) Above Water Table	40	60	70	70
Southwest (Camel) Below Water Table	40	60	65	70
Brimstone	40	60	65	70
Vortex	40	60	65	70

*Oxidation targets will vary depending on Au_{CN}/Au_{FA}

Maximum recoveries can be attained if the correct operating conditions are observed, including the following:

1. It is essential that pH be maintained above 9.5 during the oxidation process but not higher than 11. This ensures that the catalytic action of the ferrous-ferric carbonate redox pair is prevailing.

2. The total carbonate alkalinity must be maintained at a minimum of 20,000 ppm, preferably up to 60,000 ppm to stabilize enough iron in solution.

3. During oxidation, the ore must be maintained wet because the catalytic oxidation reaction involves dissolved iron carbonate species in an electrochemical reaction.

4. However, the heap must not be saturated with solution to allow oxygen to migrate to the oxidation sites. Oxygen regenerates Fe(II) carbonate to Fe(III) carbonate.

5. When the desired oxidation level is attained, excess carbonate and bicarbonate must be rinsed out of the heap. This may be followed by a lime water rinse to neutralize any residual carbonate. This step is crucial to minimize cyanide consumption during the leach stage.

Maintaining permeability in the heap is important during both oxidation and leach stage.

Metallurgical testing is ongoing, with three 20-ft columns and three large-scale columns using the old carbon columns (CIC). Also, tails assays are pending for three of the columns that were not included in this report. At the conclusion of these tests and data analyses, M3 will prepare a technical memorandum, which will serve as an addendum to this Technical Report Summary.

1.12 MINERAL RESOURCE ESTIMATE

1.12.1 Measured and Indicated Mineral Resources

Hycroft Mining Corp. (HMC) retained SRK Consulting (U.S.), Inc. to complete a mineral resource estimate for the Hycroft Project. This Technical Report Summary provides a mineral resource estimate and classification of resources reported in accordance with the New Mining Rules.

The estimates of Mineral Resources may be materially affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the Hycroft Mine. Estimates of inferred mineral resources have significant geological uncertainty and it should not be assumed that all or any part of an inferred mineral resource will be converted to the measured or indicated categories. Mineral resources that are not mineral reserves do not meet the threshold for reserve modifying factors, such as estimated economic viability, that would allow for conversion to mineral reserves.

Table 1-3: Hycroft Heap Leach Mineral Resource Estimate, June 30, 2019 – SRK Consulting (U.S.), Inc.

Classification	Material	Tons	Contained Grade				Contained Metal	
		(kt)	AuFa OPT	AuCn OPT	AgFa OPT	S%	Au (koz)	Ag (koz)
Measured	Oxide	5,650	0.011	0.008	0.224	1.79	60	1,267
	Transition	21,746	0.011	0.005	0.186	1.80	232	4,038
	Sulfide	37,512	0.010	0.002	0.273	1.85	356	10,248
		64,908	**0.010**	**0.004**	**0.240**	**1.83**	**649**	**15,554**
Indicated	Oxide	2,619	0.006	0.005	0.229	1.89	17	599
	Transition	16,293	0.007	0.003	0.329	1.79	117	5,369
	Sulfide	310,102	0.009	0.002	0.282	1.81	2,916	87,470
		329,014	**0.009**	**0.002**	**0.284**	**1.81**	**3,050**	**93,438**
Measured and Indicated	Oxide	8,268	0.009	0.007	0.226	1.82	77	1,867
	Transition	38,039	0.009	0.004	0.247	1.80	349	9,407
	Sulfide	347,614	0.009	0.002	0.281	1.81	3,272	97,718
		393,922	**0.009**	**0.002**	**0.277**	**1.81**	**3,699**	**108,992**
Inferred	Oxide	6,191	0.007	0.005	0.267	1.72	44	1,651
	Transition	20,148	0.008	0.004	0.276	1.74	156	5,570
	Sulfide	568,704	0.010	0.002	0.214	1.76	5,516	121,930
	Fill	4,018	0.013	0.008	0.150	0.63	53	603
		599,062	**0.010**	**0.002**	**0.217**	**1.76**	**5,769**	**129,754**

Source: SRK, 2019

- *Mineral Resources are not Mineral Reserves and do not meet the threshold for reserve modifying factors, such as estimated economic viability, that would allow for conversion to mineral reserves. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit resources stated as contained within a potentially economically minable open pit; pit optimization was based on assumed prices for gold of US$1,400/oz, and for silver of US$18/oz, variable Au and Ag Recoveries based on geo-metallurgical domains, a mining cost of US$1.45/t, variable ore processing costs based on geo-metallurgical domains, and G&A cost of US$0.65/t, and a pit slope of 45 degrees;*
- *Open pit resources are reported based on calculated NSR block values and the cutoff therefore varies from block to block. The NSR incorporates Au and Ag sales costs of US$0.75/oz beyond the costs used for pit optimization;*
- *Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding;*
- *Mineral Resources are reported exclusive of Mineral Reserves.*

1.13 MINERAL RESERVE ESTIMATE

Proven and Probable Mineral Reserves have been calculated on operational economics and estimates for costs for Hycroft. HMC verified the economic pit limits of the mineral reserve estimate using Geovia Whittle® 4.5.5 software. The Hycroft Mineral Reserve Estimates are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political or other relevant issues.

Mineral Reserves at Hycroft have been determined by applying current economic criteria that are valid for the Hycroft mine. These criteria limitations have been applied to the resource model to determine which part of the Measured and Indicated Mineral Resource is economically extractable. The reported mineral reserves conform to estimation and classification requirements as set out by the New Mining Rules of Proven and Probable Mineral Reserves.

Table 1-4 summarizes the Hycroft reserves as of June 30th, 2019, estimated using a gold price of $1,200 per ounce and silver price of $16.50 per ounce, as well as operating costs and applicable recoveries. The gold and silver prices used in estimating reserves are lower than the trailing 3-year average price of $1,272.66 per ounce for gold and $16.53

per ounce for silver. These have been fully scheduled in a LOM plan and have been shown to demonstrate viable economic extraction. The reference point for these mineral reserves is ore delivered to the leach pad and does not include reductions attributed to anticipated leach recoveries. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce these Mineral Reserves.

Table 1-4: Proven & Probable Mineral Reserves – June 30, 2019

	Tons	Grades, oz/t		Contained Oz (000s)	
	(000s)	Au	Ag	Au	Ag
Proven (Heap Leach)					
Oxide ROM	22,476	0.009	0.230	205	5,211
Transition ROM	4,095	0.008	0.190	32	759
Oxide ¾" Crushed	15,252	0.012	0.720	184	10,926
Transition ¾" Crushed	4,399	0.005	0.310	24	1,367
Transition ½" Crushed	90,206	0.011	0.450	948	40,365
Sulfide ½" Crushed	250,906	0.012	0.470	2,940	116,818
Total Proven Heap Leach	***387,334***	***0.011***	***0.450***	***4,333***	***175,446***
Probable (Heap Leach)					
Oxide ROM	13,145	0.005	0.230	71	3,005
Transition ROM	3,660	0.005	0.140	20	505
Oxide ¾" Crushed	3,001	0.010	0.690	29	2,063
Transition ¾" Crushed	1,304	0.004	0.490	5	644
Transition ½" Crushed	52,467	0.010	0.460	504	24,043
Sulfide ½" Crushed	663,071	0.010	0.410	6,936	272,271
Total Probable Heap Leach	***736,648***	***0.010***	***0.410***	***7,565***	***302,531***
Total Probable Sulfide Stockpile ½" Crushed	***9,079***	***0.011***	***0.380***	***98***	***3,422***
TOTAL PROVEN & PROBABLE MINERAL RESERVES	**1,133,061**	**0.011**	**0.425**	**11,996**	**481,399**
Waste	1,321,853				
Total Tons	**2,454,914**				
Strip Ratio	1.17				

- *Mineral Reserves estimated according to the New Mining Rules definitions.*
- *Mineral Reserves estimated at $1,200/oz Au and $16.50/oz Ag.*
- *Cut-off grades used a Net Smelter Return (NSR) calculation.*
- *Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.*

1.14 MINING METHODS

Hycroft mining operations are currently planned for typical truck and shovel open pit mining methods. Production is scheduled to start at 5 million tons in year one, ramp up to 20 million tons in year two, 36 million in year three, 60 million tons in year four, 75 million in year five, and 85 million in year six. Another ramp-up in production occurs in year 10 to 100 million tons as the larger phases need stripping. This production remains steady until the later years before the end of mining when it starts to ramp down as stripping is no longer required. The life of mine stripping ratio (waste to ore) is 1.17:1.

Over the life of the mine, ore routing is based on optimal destination determination accounting for all applicable costs, recoveries, and limits (i.e. crushing capacity). The following ore routing is available to each block:

- Oxide Ore - ROM heap leach & ¾" crushed heap leach
- Transitional Ore - ROM heap leach, ¾" crushed heap leach, ½" crushed heap leach
- Sulfide Ore - ½" crushed heap leach

Mining will extend below the current water surface and dewatering is planned to allow mining to extend to final pit elevations.

The first ramp up in production is achieved with contract mining support and then mid-year 2 through mid-year 7 is completed with contract mining. Contract mining considers full-service contract mining with the contractor providing all equipment, operators, maintainers, and operations supervision.

In Year 6 mining will start to transition back to owner fleets consisting of larger 55-cubic-yard excavators and 320-t class trucks as production ramps up. Blasting services will be performed by a contractor for the life of the mine. Blast-hole drills will be capable of drilling up to 9-7/8" diameter holes and 40-ft benches. Track dozers, wheel dozers, front-end loaders, graders, water trucks, and service vehicles support the mining operation. By mid-year 7 all mining has transitioned to owner mining.

1.15 RECOVERY METHODS

A significant portion of gold in the Hycroft ore is refractory due to its association with pyrite, marcasite and other sulfides. About 94% of the ore contains enough refractory gold to economically justify pretreatment by pre-oxidation prior to cyanide leaching.

The heap leach operation is designed to treat three categories of ore, classified as described below. The process methods applied to Ore Category 3 are covered by a pending patent application.

- **Ore Category 1** (ROM ore) – lower grade ore with high cyanide soluble gold is routed directly to the leach pad and cyanide leached to extract gold and silver. This accounts for 4% of the ore over the life of mine. The gold contents are highly soluble and the remaining refractory gold contents are not projected to justify the time and expense of a pre-oxidation step, therefore it will be stacked as 'ROM'. The ore in this category is typically defined as 'ROM oxide' or 'ROM transition'.
- **Ore Category 2** (3/4" Crushed ore) – higher grade ore with high cyanide soluble gold is crushed to a P80 of ¾" and cyanide leached to extract gold and silver. This accounts for 2% of the ore over the life of mine. The gold contents are highly soluble, but additional size reduction is expected to increase gold and silver recovery enough to justify the additional expense. The remaining refractory gold contents are not projected to justify the time and expense of a full pre-oxidation cycle. The ore in this category is typically defined as '3/4" crushed oxide' or '3/4" crushed transition'.
- **Ore Category 3** (1/2" Crushed ore) – low cyanide soluble ratio ores are crushed to a P80 of ½". The crushed ore is mixed with soda ash to induce an alkaline 'pre-oxidation' process. After the oxidation process has been completed to the desired extent, the ore will be rinsed sequentially with water and saturated lime solution, and then leached with cyanide to extract gold and silver. This accounts for 94% of the ore over the life of the mine. The ore in this category is typically defined as '1/2" crushed sulfide' or '1/2" crushed transition'.

Pregnant solution from the heap leach will be processed by two existing Merrill-Crowe zinc-cementation facilities.

The Hycroft Mine is projected to begin producing gold and silver from low-grade oxide ore and sulfide ore by cyanide heap leaching in the third quarter of 2019. Compared to a traditional oxide heap leach, cash flow is delayed by a length of time equivalent to the length of the dedicated pre-oxidation process.

Table 1-5: Metal Recoveries Used for Mass Balance Simulation

Metal	Head Grade	Heap Leach Recovery, %
Au	0.011 oz/t	65
Ag	0.425 oz/t	71

The mine plan was based on recovery and operating cost models, which were also used in the financial analysis in this study. It yielded life-of-mine average head grades of 0.011 oz/t Au, 0.425 oz/t Ag and 1.92% sulfur. Predicted recoveries vary from ore to ore, depending on Au, Ag and sulfide-sulfur contents, as discussed in detail in Section 10 of this report.

HMC plans to ramp up production over five years to the design crushed ore tonnage of 36 million tons per year, starting with 4.5 million tons in 2019, increasing to 12.6 million tons in 2020, 23.3 million tons in 2021, and reaching the target 36 million tons per year by 2024. As discussed above, the yearly tonnage will be supplemented by a small percentage of ore that will be placed and leached as run-of-mine ore.

For the design, M3 uses an availability factor of 75% for the primary crusher, and 85% for the secondary and tertiary crushers if feed bins are used. These design availability factors are common for current and recent projects at M3 and in line with general vendor specifications. The stacking system that will be operational in Year 2024 will have an availability of 85%, which would be dictated by the crushing plant.

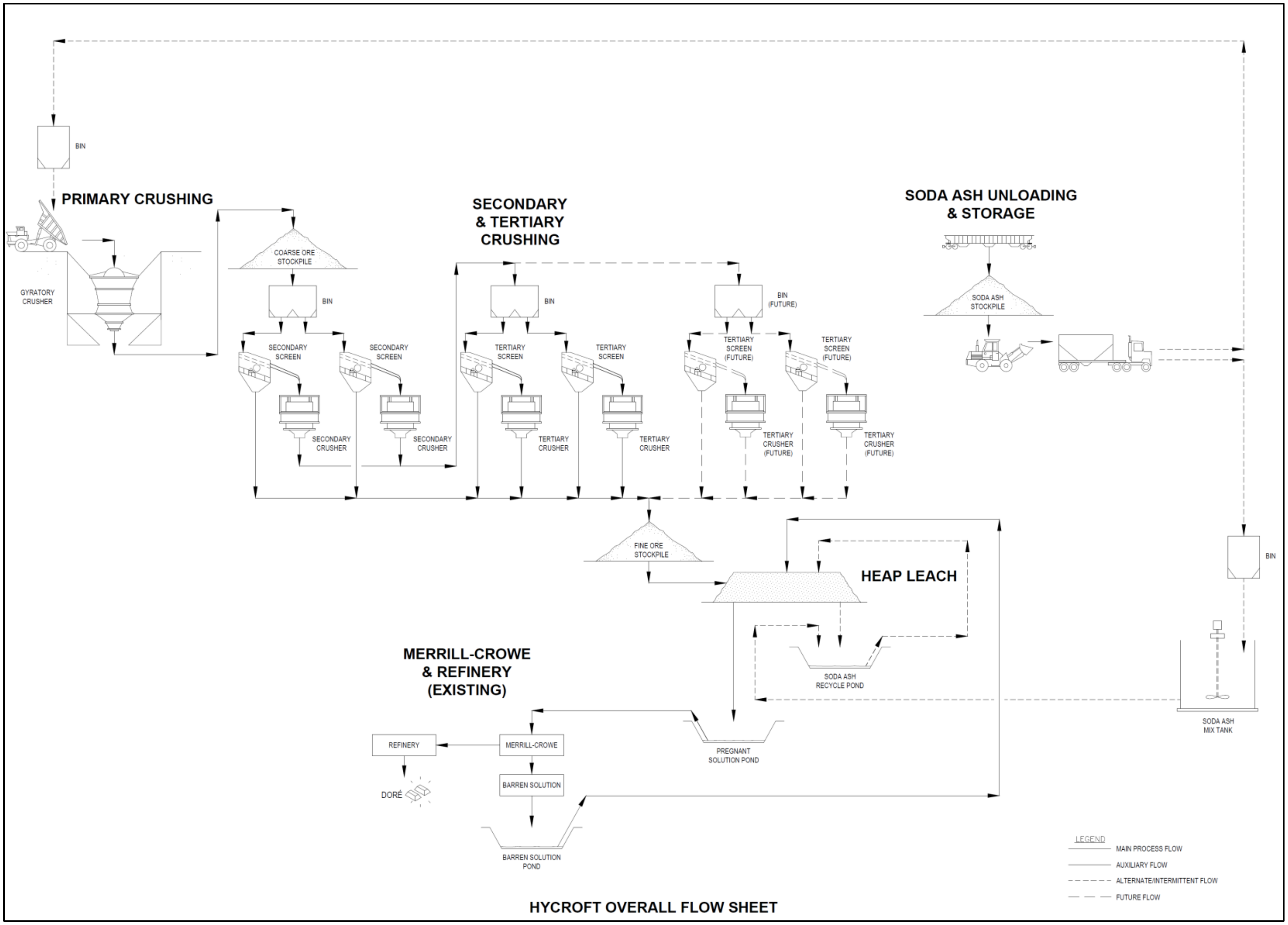


Figure 1-1: Simplified Process Flow Diagram for the Hycroft Sulfide Heap Leach Operation

1.15.1 Crushing Plant Design

The crushing system is designed to run a nominal capacity of 98,630 stpd to attain the 36 million tons per year target. The crushing system includes one primary crusher (60"x113"), two secondary crushers (XL1300 standard), and two tertiary crushers (XL1300 short head). The existing facility will be sufficient during the ramp-up period, but will require addition of two more tertiary crushers to attain the design capacity. Processing parameters in the following discussions are derived from simulations of the full plant at the design capacity. The nominal capacity was calculated at the 80th percentile hardness to allow for most of the hardness fluctuations on a day-to-day basis. Yearly average capacities were calculated at the median hardness.

Pit ore being routed as either ¾" crushed or ½" crushed ore will be transported to the primary crusher dump pocket via haul truck. Prior to the primary crusher each truck being routed as ½" crushed ore will pass under a reagent silo where a pre-determined amount of soda ash will be added to the ore. The ore will proceed through three stages of crushing to exit the tertiary crushers at a nominal P80 crush size of approximately ½". The crushed ore will then be stacked on the heap leach pads.

1.15.1 Pre-Oxidation

Pre-oxidation of sulfide and transition ore (crushed to ½") will begin at the crusher using in-situ moisture and soda ash. The soda ash requirement for the ore is relative to the sulfide-sulfur content of the ore. Regular sampling of mined material will allow reagent addition control; for the life of mine the average soda ash consumption is projected to be 14.5 lbs per ton of placed ore.

The addition of soda ash creates an alkaline environment (60,000 ppm of total Alkalinity, pH 10+) that allows some of the ferrous and ferric ions to remain in solution by complexing with carbonate ions. As discussed in Section 10, the presence of ferrous and ferric carbonate complexes for a redox pair that enhances the oxidation of iron sulfides in an heterogenous electrochemical reaction.

As the reaction proceeds, soda ash will be consumed to neutralize the resultant acid and additional soda ash will be introduced to maintain optimal reaction conditions.

Once ore has been placed on the heap, additional soda ash solution will be applied to bring the ore to field capacity (8 – 10% moisture). The solution in the heap will be replenished on a regular basis using soda ash solution in order to offset evaporation and carbonate consumption. Soda ash solution will be pumped through pipes/tubing that are separate from the lixiviant solution system.

The dissolved oxygen required for the reaction will be replenished through solution to air contact; the oxygen will be monitored inside the heap using embedded recoverable sensors. If required, air inflow can be aided by installing large perforated piping at the bottom of each panel, with ends protruding out of the heap.

Pre-oxidation duration will be determined by the characteristics of the ore and the measured extent of oxidation based upon sulfate production. The extent of oxidation will be determined by the target recoveries for each domain and the initial cyanide soluble gold, which is translated to degrees of oxidation already achieved. The number of days required to attain target oxidation is dependent upon the sulfide-sulfur content of the ore with, higher sulfide-sulfur corresponding to longer oxidation cycles. The majority of the ore is expected to take between 30 and 120 days to finish pre-oxidation. This is measured between the day that soda ash is introduced to the ore at the crusher and the day that the 'rinse' cycle begins for the panel.

1.15.2 Rinse Cycle

Ore that has undergone a pre-oxidation cycle must be rinsed, first with water, then with a saturated lime solution prior to the commencement of cyanidation. The purpose of the rinse is to wash down as much sulfate and bicarbonate, as possible.

If not removed, sulfate will precipitate as $CaSO_4$ during the leach cycle and potentially form a passivation layer against cyanidation. Bicarbonate, on the other hand, has been shown to react with cyanide to form HCN, which is not active in leaching and eventually escapes from solution, thereby increasing the cyanide consumption. The amount of rinse water required would at least be one pore volume replacement, but this should be monitored until the sulfate and alkalinity levels in the rinse water levels off at low concentrations, e.g., 2,000 ppm sulfate and 2,500 ppm total alkalinity.

Saturated lime water may be applied to scavenge the residual bicarbonate in the heap. The added complexity and cost of saturated lime water will have to be weighed against the savings in cyanide consumption that results. If the remaining bicarbonate is low enough or the leach solution has enough alkalinity that the pH can be maintained at 10.5, then most of the bicarbonate will be converted to carbonate, which does not react with cyanide. If used, the lime-saturated water will be applied to panels that have undergone pre-oxidation at a rate of 0.0025 gpm/ft^2, until one or two pore volumes have been displaced.

Rinse solutions will be supplied using the same piping that delivers lixiviant during the leach phase. Displaced solution will be sent to the soda ash recycle pond.

1.15.3 Heap Leach Cyanidation

The cyanidation conditions for all placed ore will be the same regardless of crush size or the use of pre-oxidation. The duration that these conditions are maintained is dependent on the category to which the ore belongs. For panels under active leach, a cyanide concentration of 1.0 lb/ton of solution will be maintained. The pH will be controlled using lime.

Oxide and transition material that will be leached as ROM will proceed directly from the pit to the heap and begin cyanide leach without undergoing pre-oxidation or rinse. A small percentage of oxide and transition material will be directed to the crushing plant to be reduced to a P80 of ¾" before being stacked and commencing cyanide leach. Ores from both of these categories are expected to undergo a 200-day primary leach cycle using a conservative 3:1 solution to ore ratio and an application rate of 0.0025 gpm/ft^2.

Sulfides and a portion of the transition material will be reduced to a P80 of ½" before undergoing the pre-oxidation and rinse processes on the heap. At the conclusion of the rinse, a nominal 60-day primary leach cycle will begin. A 1:1 solution to ore ratio and an application rate of 0.0025 gpm/ft^2 will be used.

1.15.4 Merrill-Crowe and Refinery

Due to the high silver content of the pregnant solution, gold and silver will be recovered by zinc cementation. Hycroft Mining has two Merrill-Crowe plants that are used to process the pregnant solution from the heap leach operation. The older plant has a capacity of 4,500 gpm. The newer plant is considerably larger, with a capacity at present of 21,500 gpm, for the total of 26,000 gpm capacity.

The wet filter cakes from the low-grade and high-grade Merrill-Crowe circuits will be transferred to retort pans, which are then put into a retort furnace to remove water and mercury. Water and then mercury are sequentially volatilized from the precipitate by heating the precipitate under a partial vacuum. The exhaust gases pass through multiple stages of condensers that drain mercury and water to a collection vessel. The last traces of mercury are removed from the retort gas by a packed bed of sulfur-impregnated carbon before being released to the atmosphere. The retorts are typically operated batch-wise, with a cycle time of approximately 18 hours.

The dried filter cake will be mixed with flux and then transferred to an electric arc furnace where it is smelted to produce doré.

1.15.5 Water Balance and Solution Management

Hycroft is currently permitted to use fresh water at a yearly average rate of 12,700 gpm. The estimated fresh water requirement is 3,189 gpm when the heap leach is operating.

Water balance and solution management for the Hycroft operation is complicated by the gradual buildup of sodium sulfate and sodium bicarbonate to a steady-state concentration in the reclaimed water. Sulfate ions were seen in some tests to slow down the sulfide oxidation reaction. Because of this, fresh water addition to the soda ash recycle pond is designed to maximize the dilution of sulfate and bicarbonate ions in the pre-oxidation circuit.

Approximately 690 gpm of the fresh water is allocated for mine dust suppression. All fresh water will be drawn from existing wells that have been operated to supply the property in the past.

1.16 PROJECT INFRASTRUCTURE

The future infrastructure for the Hycroft heap leach project considers the existing infrastructure and the requirements of the project. Currently on site are administrative buildings, mobile equipment maintenance shops, two Merrill-Crowe processing plants, a three-stage crushing system, a refinery and heap leach pads. The site also has a modern communications system provided by microwave facilities, including cellular communications. Major infrastructure categories to be constructed for the project include:

- Additional leach pad space and associated ponds, piping and other facilities
- Conveying and stacking
- Crushing system refurbishments
- Rail siding

Fresh water will be obtained from existing active and inactive production wells in a field west of the mine, and from mine dewatering. Plant water requirements are projected to fall well below the current permitted water rights.

A rail siding will be constructed that will access the nearby main east-west rail line, which is operated by Union Pacific. The rail siding will be used to receive large quantities of bulk commodities such as soda ash and lime at a reduced cost of transportation versus trucking, while reducing the potential environmental and safety hazards associated with truck transportation. M3 has provided the design for the rail unloading and materials handling facilities at the rail siding.

1.17 MARKET STUDIES AND CONTRACTS

Contracts for major consumables including fuel, lime, soda ash, cyanide and electricity are in place for the current operation. Transportation contracts are also in place for delivery of these consumable products. These contracts are renewed on an annual, biennial, triennial or quinquennial basis. The general terms and charges of these contracts are within industry standards.

Gold and silver produced at Hycroft will be sold as doré. Doré is shipped to refineries, refined and then sold at current spot prices. Marketing of doré is straightforward and arranged through continuing contractual relationships with major refineries for secure transportation of metal and refining. A contract with a refinery in Salt Lake is in place through December 2020. The cost for shipping and refining doré is in accordance with industry standards.

1.18 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACTS

Permitting for a heap leach expansion that included expanded heap leach, open pits, and waste rock facilities was completed in August 2012 with the Bureau of Land Management ("BLM") and Nevada Division of Environmental Protection ("NDEP") authorizing the proposed actions. The permits required to construct and operate the crushing system and to begin mill construction were also received from the BLM and NDEP in 2012. An Environmental Assessment ("EA") analyzing a rail spur, open pit expansion and processing complex, that includes a TMF and Heap Leach Facility, was completed and a record of decision received in January 2015.

Studies supporting permitting of a long-term TMF, and a deeper open pit, such as groundwater characterization, waste rock characterization, and archaeological and biological surveys, began late in 2009. All field work has been completed for these programs. The study information was included in a Plan of Operations that was submitted to the BLM in April 2014, requiring a supplemental EIS. Approvals are anticipated to be received for the supplemental EIS in 2019.

A Plan of Operations amendment for construction of a new leach pad was submitted to, and approved by, the BLM in July 2019. A Water Pollution Control Permit modification was submitted to NDEP in March 2019 for the leach pad expansion and is under review. It is expected a decision for the water pollution control permit will be received by the end of 2019. Future expansion activities described in this Technical Report Summary, particularly construction of additional heap leach pad space to accommodate the life of mine heap leach plan, will require multiple federal, state and local permits.

The existing Hycroft Mine workforce lives mainly in Winnemucca (Humboldt County) and Lovelock (Pershing County); this will likely remain the same for the heap leach project detailed in this report. Initial surveys indicate that the town of Winnemucca has the required infrastructure (shopping, emergency services, schools, etc.) to support the maximum workforce and dependents.

1.18.1 Mine Closure and Sustainability

Mine closure and reclamation will be performed in accordance with BLM and State of Nevada regulations and guidelines. Particular attention will be paid to leaving a post-mining land configuration that minimizes visual impact. The Company has posted surety bonds partially backed by restricted cash balances to cover its closure obligations. Future increases in reclamation bonding will either be through surety bonds supported by restricted cash balances or by letters of credit issued by banks.

The facility expansions have been and will continue to be designed and constructed to meet or exceed state and federal design criteria. Waste rock facilities are evaluated for their potential to release pollutants and monitored routinely and in accordance with an approved waste rock management plan.

All buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase.

1.19 CAPITAL AND OPERATING COSTS

The initial capital cost for the heap leach is estimated to be $230.8 million. Initial capital includes new leach pad construction, rail unloading and handling facilities for reagents, a stacking and conveying system, dewatering and crusher improvements. Hycroft began restart operations with its existing fleet as well as using a mining contractor to assist with crusher ore rehandle.

This estimate assumes all capital is on a go-forward basis. In general, M3 based this capital cost estimate on its knowledge and experience of similar facilities and work in similar locations. Resources available to M3 included recent cost data collected for a nearby mining project and for similar facilities that have been constructed, are under construction, or are being designed or studied in other locations.

The initial capital cost is broken down below by direct, indirect, and owner's costs. All capital costs are expressed in second quarter 2019 US dollars.

Table 1-6: Initial Capital Cost Breakdown

	TOTAL ($M)
Direct & Indirect	$223.2
Owner's cost	$7.6
Totals	**$230.8**

A contingency of 10% ($3,263,482 million) has been included in the M3 capital cost estimate. The accuracy of this estimate for those items identified in the scope-of work is estimated to be within the range of plus 15% to minus 15%. Accuracy is an issue separate from contingency; the latter accounts for undeveloped scope and insufficient data (e.g., geotechnical data).

HMC, through its EPCM agent, will order major material supplies (e.g., structural and mechanical steelwork) as well as bulk orders (e.g., piping and electrical). These will be issued to construction contractors on site using strict inventory control.

Operating costs were developed on a unit cost and quantity basis utilizing current labor and commodity prevailing pricing at the time of the study, first principles and similar operation comparisons. Power rates have been provided by the local electrical utility company. Data used in the analysis was derived from the internal data bases collected over a number of years. In some cases, the data was factored and/or escalated to Q2 2019 dollars.

Table 1-7: Life of Mine Operating Cost per Ton Processed

Operating Costs	
Mining cost/ton processed	$ 3.64
Process cost/ton processed	$ 4.01
G&A cost/ton processed (incl. Net Proceeds Tax, Royalties & Refining)	$ 0.86
Total operating cost/ton processed	**$ 8.52**

1.20 ECONOMIC ANALYSIS

The base case economic analysis indicates that the project has an after-tax Internal Rate of Return ("IRR") of 148.6% with a payback period of 2.5 years and with an after-tax Net Present Value ("NPV") of $2.1 billion at a 5% discount rate. The economics incorporate updated metallurgical test work and operating costs and are based on long-term prices of $1,300 per ounce of gold and $17.33 per ounce of silver. The project economics are sensitive to metal price fluctuations, as demonstrated in Table 1-8.

Table 1-8: Metal Price Sensitivity of the LOM Heap Leach Operations (after tax)

Case	Metal Prices ($/oz.)		NPV @ 0%	NPV @ 5%	After Tax IRR
	Au	Ag	*$ Billions*	*$ Billions*	
1	$1,200	$16.50	$4.2	$1.7	80.2%
2	$1,300	$17.33	$5.1	$2.1	148.6%
3	$1,400	$18.67	$6.1	$2.6	307.9%
4	$1,500	$20.00	$7.1	$3.0	N/A

1. *Downside Price (Reserve Price)*
2. *Financial Base Case*
3. *Moderate Price*
4. *Upside Price*

In addition to metal prices, the project is sensitive to capital and operating costs as shown in Table 1-9.

Table 1-9: Operating and Capital Cost Sensitivity of the LOM Heap Leach Operations (after tax), NPV @ 5%

	20% Decrease	10% Decrease	Base Case	10% Increase	20% Increase
Mining Cost	$2.41B	$2.25B	$2.1B	$1.91B	$1.75B
Processing Cost	$2.43B	$2.26B		$1.90B	$1.72B
Capital Expenditures	$2.18B	$2.13B		$2.03B	$1.98B

1.21 CONCLUSIONS AND RECOMMENDATIONS

Based on the findings of this feasibility study, it has been concluded that the Hycroft Heap Leach Project would be an economically viable project under the base case as well as reserve case financial parameters. It is recommended that HMC proceed with the restart of the heap leach operations as described in this report.

1.21.1 Prepared in Accordance with US SEC's New Mining Rules Under Subpart 1300 and Item 601 (96)(B)(iii)

The drill hole database and assaying quality for the Hycroft Mine are sufficient for the determination of Measured, Indicated and Inferred Mineral Resources. Additionally, the geological interpretations, metallurgical assumptions, and spatial drilling densities are sufficient to define and state Proven and Probable Mineral Reserves for Hycroft.

All of the aforementioned categories are prepared in accordance with the resource classification pursuant to the SEC's new mining rules under subpart 1300 and item 601 (96)(B)(iii) of Regulation S-K (the "New Mining Rules").

2 INTRODUCTION

2.1 PURPOSE AND BASIS OF REPORT

This Technical Report Summary was prepared and is issued by M3 Engineering & Technology Corp. in association with SRK Consulting (U.S.), Inc. and Hycroft Mining Corporation, a Delaware corporation with headquarters in Denver, Colorado.

This Technical Report Summary has been prepared to describe the feasibility of extracting and processing the large transition and sulfide reserve at the Hycroft property. A feasibility study has been completed with the goal of assessing the economic benefit of operating a heap leach facility capable of oxidizing and leaching the transition and sulfide reserves in addition to the traditional heap leach process recently employed at the mine. The study indicates that a heap leach process, which is the subject of a pending patent application, could be operated to economically extract and process the transition and sulfide reserves concurrently with the heap leach oxide reserve.

All material at Hycroft has been classified according to, and prepared in accordance with, the resource classification pursuant to the SEC's new mining rules under subpart 1300 and item 601 (96)(B)(iii) of Regulation S-K (the "New Mining Rules").

2.2 SOURCES OF INFORMATION

The scope of this study included a review of pertinent technical reports and data in the possession of M3, SRK and Hycroft Mining relative to metallurgical test results, the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, and Mineral Resources and Mineral Reserves. Observations and interpretations of geostatistics, geology, grade estimation, and determination of mineralized trends at Hycroft have been generated by SRK Consulting (U.S.), Inc. and Hycroft Mining. The Hycroft model has been generated and evaluated with Geovia GEMS for block modeling, Sage2001 for variography and X10-Geo for statistical analysis. Economic pit limits were determined with Geovia Whittle® 4.5.5 Strategic Planning software, the pit was designed using Vulcan 10.1 and the mine schedule was developed using Minemax Scheduler Professional Version 6.5.2.24696.

2.3 QUALIFIED PERSONS AND SITE VISITS

2.3.1 M3 Engineering & Technology

Information in this Technical Report Summary has been prepared under the supervision of employees of M3 engineering who were responsible for project management, recovery methods, process plant operating and maintenance costs, capital cost estimate and overall compilation of this report. M3 representatives visit the mine regularly with the most recent visit being on June 7, 2019.

2.3.2 Steven Newman, Registered Member SME

Information in this Technical Report Summary has been prepared under the supervision of Steven Newman, SME Registered Member, Director of Feasibility Studies of HMC. Mr. Newman is responsible for reserves, long-term mine planning and the associated feasibility studies for Hycroft. Mr. Newman works on site on a weekly basis.

2.3.3 Brooke Miller Clarkson, CPG

Information in this Technical Report Summary has been prepared under the supervision of Brooke Miller Clarkson, CPG, a SRK Senior Consultant. Ms. Clarkson is responsible for compilation of the drill hole database, review and verification of drill hole data and construction of the geologic models. Ms. Clarkson last visited site on June 5, 2017.

2.3.4 Richard F. DeLong, P. Geo

Information in this Technical Report Summary has been prepared under the supervision of Richard F. DeLong, P.Geo, President of EM Strategies, Environmental Consultants. Mr. DeLong is responsible for verification and oversight of the environmental practices and permitting as well as providing consulting assistance and advice on major environmental matters including Environmental Studies. Mr. DeLong's last visit to site was on July 27, 2018.

2.3.5 Tim Carew, P. Geo

Information in this Technical Report Summary has been prepared under the supervision of Tim Carew, P.Geo, a SRK Principal Consultant. Mr. Carew is responsible for resource estimation. Mr. Carew's last visit to site was on July 27, 2018.

2.3.6 Matt Hartmann, MScMEM, P.G., MAusIMM, Registered Member SME

Information in this Technical Report Summary has been prepared under the supervision of Matt Hartmann, Principal Consultant with SRK. Mr. Hartmann is responsible for hydrogeology and mine dewatering. Mr. Hartmann has been involved in hydrogeologic studies at the Hycroft mine since 2010, his most recent visit to site was on April 23, 2019.

2.3.7 Tabulation

Table 2-1 shows a tabulation of the qualified persons and their responsibilities.

Table 2-1: List of Qualified Persons

QP Name	Company	Qualification	Site Visit	Area of Responsibility
M3 Engineering & Technology Corporation, Tucson, AZ		PE	Several times, last visit June 7, 2019	Sections 2, 10, 14, 15, 19, 24, 25 and corresponding subsections of 1, 18, 22 and 23.
Steve Newman	Hycroft Mining Corporation, Denver, CO	Registered Member SME	On site weekly	Sections 3, 4, 5, 7.8, 12, 13, 16, 20, 21 and corresponding subsections of 1, 18, 22 and 23.
Brooke Miller Clarkson	SRK, Reno, NV	CPG	June 5, 2017	Sections 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 6, 7.1, 7.2, 7.3, 7.4, 7.5, 7.6, 7.7, 8, 9, and 23.1.
Richard F. DeLong	EM Strategies, Reno, NV	P. Geo.	July 27, 2018	Section 3.3, 3.4, 17 and corresponding subsections of 1, 22 and 23.
Tim Carew	SRK, Reno, NV	P. Geo.	July 27, 2018	Section 1.12, 11 and 22.1.
Matt Hartmann	SRK, Denver, CO	Member AusIMM, Registered Member SME	April 23, 2019	Sections 7.9, 13.6, and 18.3.1.

2.4 TERMS OF REFERENCE

Unless stated otherwise, all volumes and grades are in US customary units and currencies are expressed in constant Q2 2019 US dollars. Distances are expressed in US customary units.

This report is written specifically for the Hycroft Mine operation.

2.5 UNITS AND ABBREVIATIONS

Units and abbreviations used in this report are as shown in Table 2-2.

Table 2-2: List of Units and Abbreviations

$	United States dollar(s)
$CAD	Canadian dollar(s)
°C	Degree Celsius
°F	Degree Fahrenheit
µm	micrometer(s)
3D	Three dimensional
AA	Atomic absorption
AAO	Atmosphere Alkaline Oxidation
Ag	Silver
ALS	Auld Lang Syne
HMC	Hycroft Mining Corporation
Au	Gold
Au Eq	Gold Equivalent
Avg	Average
BAPC	Bureau of Air Pollution Control
BLM	Bureau of Land Management
BMRR	Bureau of Mining Regulation and Reclamation
BSDW	Bureau of Safe Drinking Water
BWI	Bond Work Index
BSMM	Bureau of Sustainable Materials Management
BWPC	Bureau of Water Pollution Control
CCD	Counter Current Decantation
CIC	Carbon-in-column
CNI	Call & Nicholas
CoG	Cut-off grade
Duval	Duval Corporation
EA	Environmental Assessment
EIS	Environmental Impact Statement
El	Elevation
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement and Construction Management
FCC	Federal Communication Commission
FEL	Front-end loaders
Ft	Foot
ft^2	Square foot
ft^3	Cubic foot
ft^3/day	Cubic feet per day
ft^3/sec	Cubic feet per second
G	Gram(s)
Gal	US gallon
GIS	Geographical Information Services
Gpm	US gallon per minute
Gps	Global positioning system
H	Hour(s)
h/d	Hours per day
Ha	Hectare
HDPE	High Density Polyethylene
HDR	HDR, Inc.

Hp	Horsepower
HRDI	Hycroft Resources & Development Inc.
ID3	Inverse Distance Cubed
IDS	International Directional Services
IDW	Inverse Distance Weighted
In	Inch(es)
in/yr	Inch(es) per year
IRR	Internal rate of return
ISO	International Standards Organization
kV	Kilovolt
kVa	Kilovolt x amps
kW	Kilowatt
kWh	Kilowatt hour
kWh/t	Kilowatt hour per ton
Lb	Pound
LOM	Life-of-Mine
M	Million(s)
MDA	Mine Development Associates
MDE	Maximum Design Earthquake
MDL	Method Detection Limit
MEG	Mineral Exploration and Environmental Geochemistry
MG	Million Gallons
MGT	Million Gross Tons
Mi	Mile(s)
Min	Minute(s)
Mo	month(s)
Moz	million troy ounces
Mph	miles per hour
MRDI	Mineral Resources Development Inc.
Mt	Metric tonne (2200 lb)
MVAR	Mega Volt Ampere Reactive
MW	Megawatt
MWh	Megawatt hour
MWMP	Meteoric Water Mobility Procedure
NAD	North American Datum
NDEP	Nevada Division of Environmental Protection
NDOW	Nevada Department of Wildlife
NDWR	Nevada Division of Water Resources
NN	Nearest Neighbor
NPI	Net Profit Interest
NPV	Net Present Value
NSR	Net Smelter Return
NV	Nevada
OK	Ordinary Kriging
Opt/OPT	Troy Ounce per short ton
oz	Troy ounce unless otherwise noted
(Non)PAG	(Non) Potentially acid generating
PAX	Potassium Amyl Xanthate
pcf	Pounds per cubic foot
PFDS	Precipitation Frequency Data Server

POX	Pressure Oxidation
Ppm/PPM	Parts per million
psf	Pounds per square foot
psi	Pounds per square inch
PVC	Polyvinyl Chloride
QA	Quality assurance
QC	Quality control
QP	Qualified Person
RC	Reverse Circulation
ROM	Run-of-Mine
ROW	Right-of-Way
RQD	Rock Quality Designation
SAG	Semi-Autogenous Grinding
SEC	U.S. Securities and Exchange Commission
SME	Society of Mining, Metallurgy and Exploration
SRK	SRK Consulting (U.S.), Inc.
SSDS	Small scale, direct shear
st	Short ton (US) (2000 lb)

t	Short ton (US) (2000 lb)
t/h	Short tons per hour (US)
t/y	Short tons per year (US)
TMF	Tailing Management Facility
tpd	Short tons per day (US)
TR	Technical Release
UCL	Upper control limit
µm	micrometer
USDA	United States Department of Agriculture
USFWS	United States Fish and Wildlife Service
USMX	U.S. Steel Exploration
UTM	Universal Transverse Mercator
V	Volt
VFD	Variable Frequency Drive
W	Watts
WRF	Waste Rock Fill
yd^3	Cubic yard
Yr	Year(s)

3 PROPERTY DESCRIPTION AND LOCATION

The Hycroft Mine is a gold and silver mining and processing operation located 54 miles west of Winnemucca in Humboldt and Pershing Counties, Nevada, as shown in Figure 3-1. The Hycroft property is accessible via Nevada State Route 49 (Jungo Road), an all-weather, unpaved road that is maintained by Humboldt County and HRDI. A major east-west railway runs immediately adjacent to the property.

The mine property straddles Townships 34, 35, 35½ and 36 North and Ranges 28, 29 and 30 East (MDB&M) with an approximate latitude 40°52' north and longitude 118°41' west. The mine is situated on the western flank of the Kamma Mountains on the eastern edge of the Black Rock Desert.

The use of water at Hycroft is controlled by eleven separate water right permits administered by the NDWR. These permits are held in ownership either by HRDI or by other private parties and leased to HRDI. HRDI controls a total of 21,457.95 acre feet per year (6.99 billion gallons per year) in the Black Rock Desert Hydrographic Basin.

Gold production began on the property in 1983. Through a series of permitting actions with the BLM and NDEP, HMC has incorporated all existing mining components into a current Reclamation Plan with an associated bonding instrument. As of June 30, 2019, the posted surety bond for site reclamation was $58.3 million.

The Hycroft property consists of 30 private parcels that comprise approximately 1,912 acres, and 3,247 unpatented mining claims that encompass approximately 68,759 acres. The mining claims of Hycroft are comprised of two primary properties, Crofoot and Lewis. The Crofoot and Lewis properties together include approximately 11,829 acres. The Crofoot property covers approximately 3,500 acres and is virtually surrounded by the Lewis property of 8,400 acres.

On site facilities include administration buildings, a mobile maintenance shop, light vehicle maintenance shop, warehouse, leach pads, crushing system, two Merrill-Crowe process plants and a refinery. The components for a second refinery are on-site and will be constructed as part of the expansion of mining activities. The crushing system is being refurbished as part of the restart activities and all other facilities are operational. At June 30, 2019, property, plant and equipment was valued at $65.4 million.

The Hycroft Mine operates under permit authorizations from the BLM, NDEP, NDOW, NDWR and County agencies. As of June 30, 2019, approximately 118 full-time personnel were employed at Hycroft.

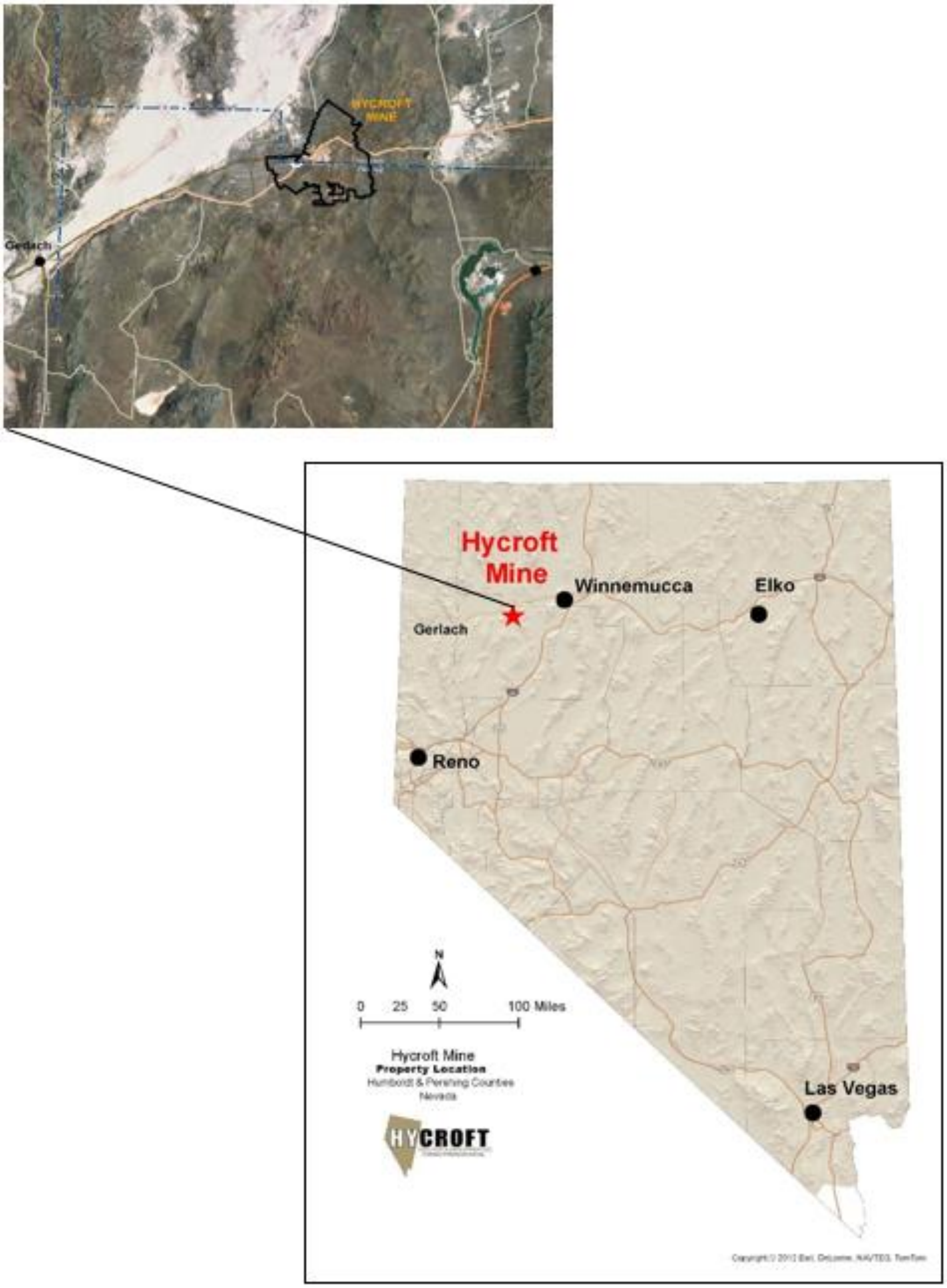


Figure 3-1: Hycroft Mine Property Location Map (June 2018)

3.1 LAND STATUS

The mine is managed and operated by HRDI, a wholly owned subsidiary of Allied VGH, Inc., which is a wholly owned subsidiary of HMC.

HRDI holds 3,247 unpatented mining claims, comprising 68,759 acres, located as follows:

- T36N, R29E, sections: 28, 32, 33
- T36N, R30E, sections: 19, 28-34
- T35 1/2N, R29E, sections: 25, 26, 35, 36
- T35N, R29E, sections: 1-3, 10-15, 21-28, 31-36
- T35N, R30E, sections: 2-10, 15-23, 25-36
- T34N, R28E, sections: 1, 2, 11, 12, 13
- T34N, R29E, sections: 1-28, 33
- T34N, R30E, sections: 2-11, 17-20, 29, 30

The company owns 30 private parcels (patented lode and placer claims) comprising 1,912 acres, located as follows:

- T35N, R29E, sections: 24, 25, 35, 36
- T35N, R30E, sections: 19, 30, 31
- T34N, R29E, sections: 1, 2

Combining the patented and unpatented claims, Hycroft claims total approximately 70,671 acres (Figure 3-2). Much of the project area is located on un-surveyed public and private land for which the sections, ranges, and townships listed above have been interpolated. Patented claims however, have been surveyed (Wilson, 2008; Prenn, 2006).

This land claim package has been assembled through a series of transactions:

- The Crofoot property and approximately 3,500 acres of claims were acquired by Vista in 1985.
- The Crofoot property, originally held under lease, is owned by HRDI subject to a 4% Net Profits Interest ("NPI") retained by the former owners, capped at total future payments of $5.1 million.
- The Lewis property and approximately 8,700 acres of claims were acquired by Vista in early 1987.
- In 2006, approximately 13,100 acres of additional claims were staked by Vista. These claims are contiguous or proximate to the original Crofoot and Lewis claims.
- From 2008 through end of October 2014, approximately 45,371 acres of additional claims were staked by HRDI contiguous to the existing Hycroft claims.

The BLM and County annual claim holding fees are paid in the third quarter of each year. Payment of annual fees is current through the 2018-2019 claim years, with $556,610 paid in 2018. Payment of annual land holding fees and taxes is required to continue to hold the Hycroft property in good standing.

HRDI controls all surface and mineral rights within the Hycroft mineral reserve and mineral resource area. No further land acquisition is required for operation of the mine and processing facilities as presently designed.

Figure 3-3 shows the property layout including site facilities, mine workings, leach pads and waste dumps.

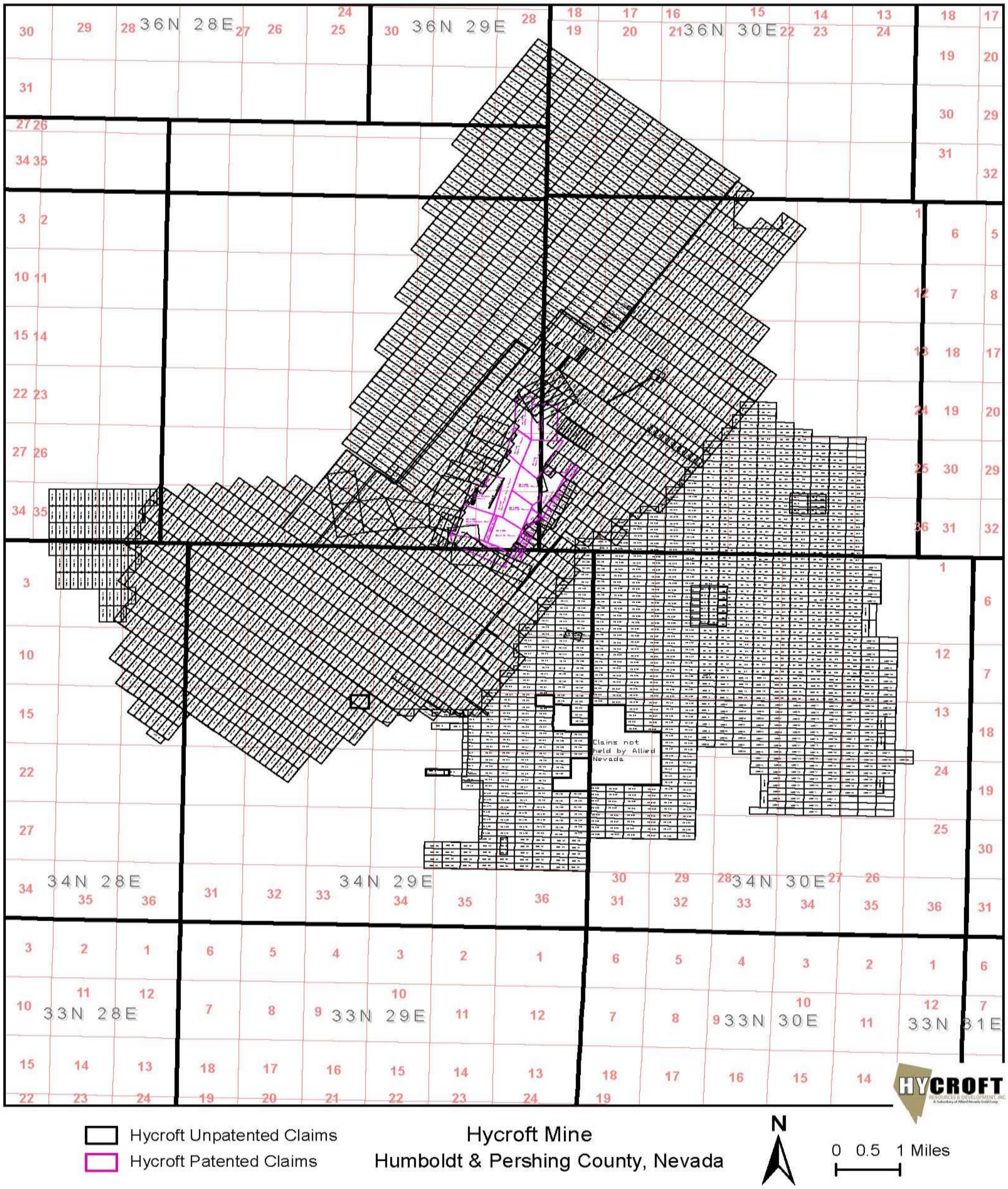
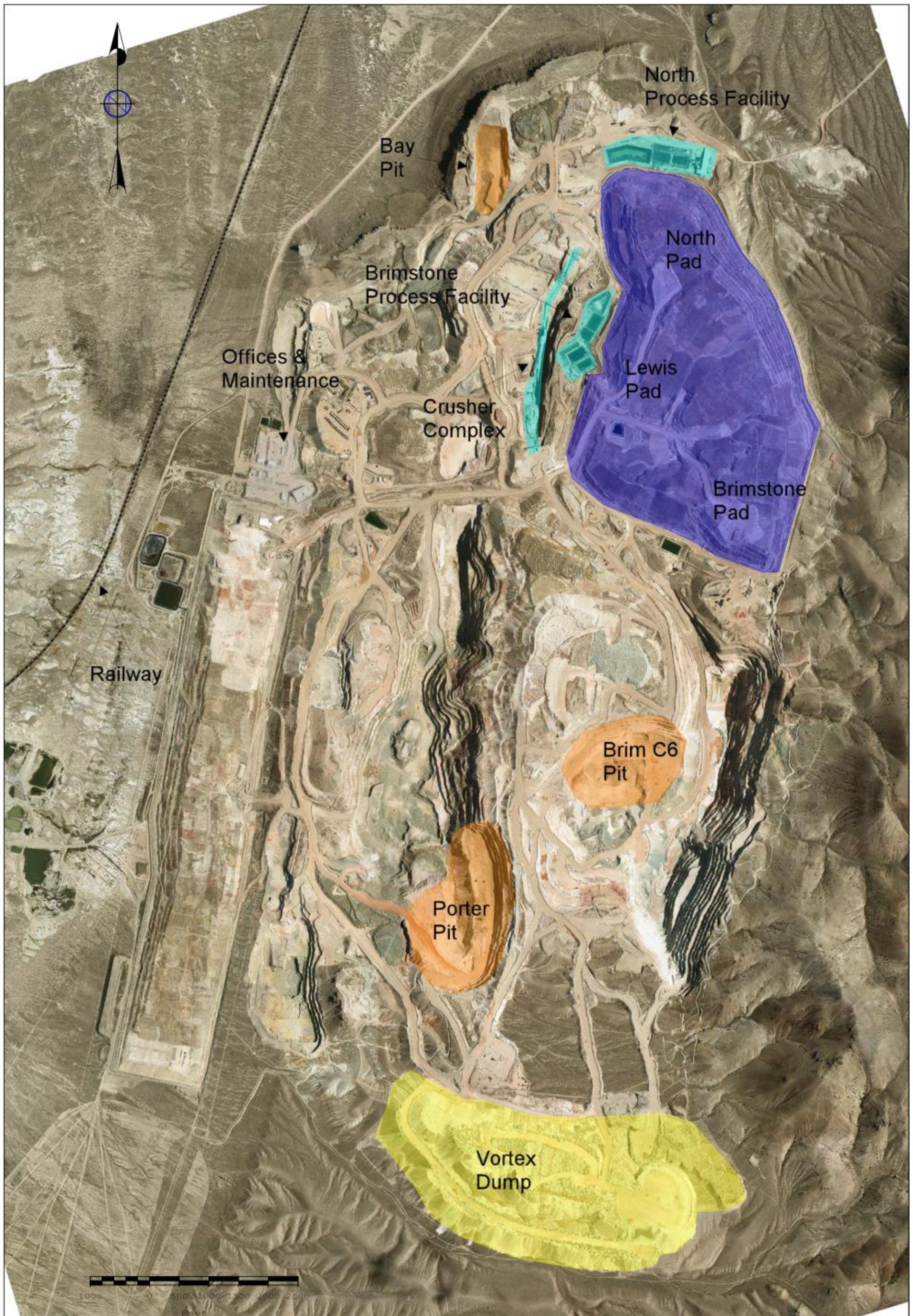


Figure 3-2: Hycroft Mine Claim Map (June 2019)



Title: Site Map	Scale: 1:17500	HYCROFT RESOURCES & DEVELOPMENT, INC.
Author: MK	Note 1:	
Date: 24-Jul-2019	Note 2:	
Bench:	Note 3:	

Figure 3-3: Current Property and Facilities Layout (June 2019)

3.2 AGREEMENTS AND ROYALTIES

The Crofoot property is held by HRDI, a wholly owned subsidiary of HMC, through Allied VGH, Inc. A 4% NPI is retained by the original Crofoot owners. In 1996, the lease/purchase agreement was amended to provide for minimum advance royalty payments of $120,000 on January 1 of each year in which mining occurs on patented and unpatented claims. The sum of payments for the Crofoot property is capped at $7.6 million, of which $2.49 million has been paid through June 2019. An additional $120,000 annually is due if ore production exceeds 5 million tons from the Crofoot property on either patented or unpatented claims in any calendar year. All advanced royalty payments are taken as a credit against the 4% NPI. Table 3-1 shows the royalty amount and other annual land holding costs.

Table 3-1: Hycroft Annual Land Holding Costs

Month Due	Lessor	Type	Amount
January-December	Crofoot[1]	Advance Royalty	$120,000
August-October	U.S. BLM, Humboldt & Pershing Counties	Claim Fees	$556,610

[1] The Crofoot royalty is only payable if mining is taking place.

3.3 ENVIRONMENTAL LIABILITIES

Gold production began on the property in 1983 and continued through 1985 when Standard Slag opened the Lewis Mine. There was a brief gap in mining until HRDI acquired the Lewis Mine and the Crofoot claims and recommenced mining in 1988. Mining operations continued until 1998 when mining was placed on standby due to low metal prices. Process operations continued until 2004 when the property was placed on care and maintenance.

Efforts began in 2003 to update the reclamation plan, associated cost estimate, and related amount of surety bond posted with the BLM. During the years ended December 31, 2011 and 2012, the Company increased collateral account balances to support additional surety bonds for the benefit of the BLM. These additional surety bonds allowed the Company to continue operations at the Hycroft Mine and to expand exploration activities outside of the Hycroft Mine. In 2011, the Company received a reimbursement of $0.5 million related to reclamation costs paid by the Company.

In January 2014, the BLM approved an updated reclamation cost estimate allowing for the phased bonding of the expansion activities. The required bond amount was lowered from $63 million to $58.3 million. The Company has entered into Surface Management Surety Bonds with insurance companies that meet the financial requirements of the BLM to comply with the total requirement of $58.3 million as detailed in the September 2013 reclamation cost estimate that requested the phasing of the mill expansion activities. Additionally, the company has posted an exploration bond with the BLM in the amount of $1.0 million and an Archaeological Resources Protection Act Surety Bond in the amount of $0.6 million.

3.4 PERMITS

The Hycroft Mine operates under permit authorizations from the BLM, NDEP, NDOW, and NDWR. All operating and environmental permits, approved by the BLM, NDEP, NDOW and NDWR, are in good standing for mining operations at Hycroft. Table 3-2 summarizes the operating permits while Table 3-3 shows the miscellaneous permits for the property.

Table 3-2: Hycroft Operating Permits

Operating Permits	Issuing Agency	Number	Status
Plan of Operations	BLM	NVN-064641	Current
Mercury Operating Permit to Construct	NDEP - BAPC	AP1041-2255	Current
Class I Air Quality Operating Permit to Construct	NDEP - BAPC	AP1041-2974	Current
Class I Air Quality Operating Permit to Construct	NDEP - BAPC	AP1041-3344	Current
Class I Air Quality Operating Permit to Construct	NDEP - BAPC	AP1041-3269	Current
Permit to Operate a Public Water System	NDEP - BSDW	HU-0864-12NTNC	Current
Class II Air Quality Permit	NDEP - BAPC	AP1041-0334.02	Current
Water Pollution Control Permit-Crofoot Project	NDEP - BMRR	NEV60013	Current
Water Pollution Control Permit-Brimstone Project	NDEP - BMRR	NEV94114	Current
Bioremediation Facility Permit	NDEP - BMRR	GNV041995-HGP15	Current
Reclamation Permit	NDEP - BMRR	134	Current
Mining General Stormwater Pollution Prevention Permit	NDEP - BWPC	R300000: MSW-177	Current
Class III Landfill Waiver	NDEP - BSMM	F-346	Current
Artificial Pond Permit (Brimstone Process Ponds)	NDOW	S34481	Current
Artificial Pond Permit (Crofoot Process Ponds)	NDOW	S36665	Current
Artificial Pond Permit (North Process Ponds)	NDOW	S36661	Current
General Onsite Sewage Disposal System	NDEP - BWPC	GNEVOSDS09	Current
Septic Onsite Disposal	NDEP - BWPC	GNEVOSD09L-0018	Current
Dam Safety Permit (Crofoot Process Ponds)	NDWR	J-273	Current
Hazardous Materials Storage Permit	NV State Fire Marshall	8250	Current
Special Use Permit	Pershing County	SUP 12-04	Current
Special Use Permit	Humboldt County	UH-12-04	Current

Table 3-3: Hycroft Miscellaneous Permits

Operating Permits	Issuing Agency	Number	Status
ROW Microwave Repeater; Sec. 29, 30	BLM	NVN46292	Current
ROW Wells/Pipeline/Power Line; Sec. 3	BLM	NVN46564	Current
ROW 2 Wells/Pipeline/Power Line	BLM	NVN46959	In renewal
ROW Road & Waterline (Old Man camp to Lewis)	BLM	NVN39119	In renewal
ROW Crofoot pipeline	BLM	NVN44999	In renewal
ROW 24 kv Aerial Powerline, Lewis/Floka	BLM	NVN54893	Current
Kamma Peak Station	FCC	WNER344	Current
Sulfur Mine Station	FCC	WNER345	Current
Winnemucca Mountain Station	FCC	WNER346	Current
Base Station & 45 Mobile Units	FCC	WNKK336	Current

Operating and miscellaneous permits that require annual maintenance fees are shown in Table 3-4. Fixed annual fees are required for storm water and public drinking water system permits based upon the current Nevada regulatory structure. The other annual fees are based on annual mining production, quantities and types of chemicals stored on site, existing and permitted surface disturbance, and the level of actual and permitted air emissions. The variable fees shown are based upon the current operational conditions.

Table 3-4: Hycroft Permits and Annual Fees

Permit and Fee Description	Annual Amount
Air Quality Operating Permit AP1041-0334.02	$3,312
Air Quality Operating Permit AP1041-2255	$14,401
Air Quality Operating Permit AP1041-2974	$22,082
Air Quality Operating Permit AP1041-3344	$14
Reclamation Permit	$30,000
Nevada Radioactive Material License	$1,100
Stormwater Permit	$200
Artificial Pond Permit	$8,750
Water Pollution Control Permit NEV94114	$20,000
Water Pollution Control Permit NEV60013	$20,000
State Fire Marshall	$150
Public Drinking Water System	$225
Septic System Permits	$600
Toxic Release Inventory Annual Fee	$3,000
Nevada LP-Gas License	$450
TOTAL	**$124,284**

Hycroft currently holds six ROW leases and two exploration notices with the BLM, as described in Table 3-5 along with fees and renewals.

Table 3-5: Right-of-Way Payment and Renewal Schedule

ROW Number	Annual Payment Amount (estimated)	Payment Date	Expiration Date
NVN46292	$125	01/01/19	12/31/48
NVN46564	$100	01/01/19	12/31/46
NVN46959	$600	01/01/19	In renewal
NVN39119	$400	01/01/19	In renewal
NVN44999	$300	01/01/19	In renewal
NVN54893	$200	01/01/19	10/10/2025
NVN96607 (notice)	N/A	N/A	05/15/2021
NVN96608 (notice)	N/A	N/A	05/15/2021

3.4.1 Hycroft Expansion Permitting

HMC submitted an amended Plan of Operations for an expansion of its heap leach facilities, open pits and waste rock facilities to the BLM in April 2010. The submittal of the Plan of Operations to the BLM initiated a National Environmental Policy Act review of the proposed action. The BLM determined that an EIS was to be performed and, in August 2012, a Record of Decision was issued authorizing the proposed action. A major modification to the State Water Pollution Control Permit was submitted in 2011 for the process components that included engineering design reports from Golder Associates. The permit modification was issued in August 2012. All other permits required for the heap leach expansion have been received.

The permit required to construct mill facilities was received in December 2012. The air quality permit for operation of a mill was submitted in December 2012 and issuance was received in late 2013.

An amended Plan of Operations that included a rail spur, open pit expansion and processing complex was submitted to the BLM in August 2012. The BLM determined that an Environmental Assessment was required, deemed the Plan of Operations complete and initiated public scoping in December 2012. NV Energy submitted a Rights-of-Way application for the power line associated with the Hycroft Mill in March 2013. The BLM determined that action should be analyzed with the Hycroft EA. A record of decision approving the EA was completed by the BLM in January 2015.

A Plan of Operations for the TMF, mining below the water table and expanded facilities was submitted to the BLM in April 2014. The BLM determined that a Supplemental Environmental Impact Statement is required. The SEIS permitting process is in progress and it is anticipated that it will be completed in 2019.

A Plan of Operations amendment for construction of a new leach pad was submitted to, and approved by, the BLM in July 2019. A Water Pollution Control Permit modification was submitted to NDEP in March 2019 for the leach pad expansion and is under review. It is expected a decision for the water pollution control permit will be received by the end of 2019. Future expansion activities described in this Technical Report Summary will require multiple federal, state and local permits.

3.4.2 Crofoot Heap Leach Facility Closure

HMC submitted an updated Final Permanent Closure Plan in November 2017, to the NDEP for the Crofoot processing facilities permitted in Water Pollution Control Permit NEV60013. Facilities to be closed include the Crofoot heap leach pad and associated processing components. The NDEP has approved the activities associated with the closure of the process plant and ponds.

The construction of the drain-down collection system was completed in 2012. Regrading of the Crofoot pad was initiated in 2017 and, once completed, growth medium will be placed on the heap leach pad.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 ACCESS

Hycroft and its related facilities are located 54 miles west of Winnemucca, Nevada. Access to the Hycroft Mine from Winnemucca or Gerlach is by means of State Road No. 49 (Jungo Road), a good quality, unpaved road. Access is also possible from Imlay or Lovelock by dirt roads intersecting Interstate 80. The majority of the mine's employees live in the Winnemucca area. Winnemucca is a commercial community on Interstate 80, 164 miles northeast of Reno, Nevada. The town is served by a transcontinental railroad and has a small public airport. There is access to adequate supplies of water and power.

4.2 CLIMATE

The climate of the region is arid, with precipitation averaging 7.7 inches per year. Most of the precipitation occurs in the winter and spring months.

Temperatures during the summer are generally 50ºF at night and 90ºF and above during the days. Winter temperatures average 20ºF at night and 40ºF during the day. There is strong surface heating during the day and rapid nighttime cooling due to the dry air, resulting in wide daily ranges in temperatures. The average range between the highest and lowest daily temperatures is 30 to 35ºF.

Winds are generally light with dust or sandstorms occurring occasionally, particularly during the spring. The mine has not experienced major delays in production due to inclement weather and operates on a year-round basis.

4.3 LOCAL RESOURCES AND INFRASTRUCTURE

The mine site has a truck shop, ore processing facilities, an administration building and other service-related structures. Power is supplied to the site from nearby power lines that are fed directly from the main power grid and there is a modern communications system including cellular connections.

The mine is in a well-known mining jurisdiction near several towns including Winnemucca, Gerlach and Lovelock. Most of the current personnel at Hycroft are from these areas. Initial surveys indicated that the town of Winnemucca has the required infrastructure (shopping, emergency services, schools, etc.) to support the maximum workforce and dependents. In addition, the mine has been successful in filling positions with qualified mining personnel from all over the country.

Water rights are shown in Table 4-1. Three production wells are located four to five miles west of the mine, and a potable well is located approximately one mile south of the Crofoot Heap. These four production wells are the main sources of water for the mine site. All of the water rights are within the Black Rock Desert Hydrographic Basin, a recently designated basin.

HMC controls a land position sufficient to support all of its planned facilities. A major east-west railway passes through the Hycroft claim position.

Table 4-1: Hycroft Water Wells and Permitted Yearly Consumption

Application No.	Permit Diversion Limit (cfs)	Annual Appropriation Limit (ac-ft)	Point of Diversion
81228	0.4	14.83	T34N R29E S3
81226	3.2	724.79	T35N R29E S31
81225	3.2	303.43	T35N R29E S31
81227	2.0	1,448	T35N R29E S31
81224	2.0	1,448	T34N R28E S1
81408	5.4	3,890	T35N R29E S31
81409	5.4	3,890	T35N R29E S31
84477	0.3	177.9	T35N R29E S31
82274	10	4,096	T35N R29E S31
82355	3.3	2,050	T35N R29E S31
82356	5.6	3,415	T34N R28E S1
Total	**40.8**	**21,457.95**	

4.4 PHYSIOGRAPHY

The mine is situated on the eastern edge of the Black Rock Desert and on the western flank of the Kamma Mountains between Winnemucca and Gerlach, Nevada.

The Black Rock Desert is a 400 square mile flat, prehistoric lakebed, completely devoid of any vegetation or animal habitat. Its name comes from a large, prominent, dark rock formation located at the north end of the desert. During the summer, the lakebed is primarily a hardpan alkaline playa. During some winters, it may become a temporary lake.

There are no streams, rivers, or major lakes in the general area. Elevations range from 4,500 to 5,500 ft above sea level.

5 HISTORY

5.1 PROPERTY HISTORY

Mining at Hycroft began in 1983 with a small heap leach operation known as the Lewis Mine. Lewis Mine production was followed by production from the Crofoot property in the Bay, South Central, Boneyard, Gap and Cut-4 pits along the Central Zone. Production from the north end of the Brimstone pit continued until December 1998. Due to gold prices averaging below $300/oz, the mine was placed on a care and maintenance program though processing continued through 2004 when mining ceased in December 1998.

Vista acquired the Lewis Mine in early 1987 from F. W. Lewis, Inc., and the Crofoot Mine in April 1988. The remaining leasehold interest in the Lewis property was purchased by Vista in December 2005, in consideration of the payment of $5.1 million resulting in the elimination of the 5% NSR royalty on gold and 7.5% NSR royalty on silver.

The Hycroft Mine produced approximately 1.2 million ounces of gold and 2.5 million ounces of silver from 1983 to 1998 when the operations were suspended. An additional 58,700 ounces of gold was produced from the leaching and rinsing of the heap leach pads from 1999 through 2004, after the mine was placed on care and maintenance.

In May 2007, the Nevada-based holdings of Vista were spun out into Allied Nevada Gold Corp. The Hycroft Mine was included as part of the transfer of ownership allowing Allied Nevada to explore, expand, and develop the resources at Hycroft.

In September 2007, Allied Nevada's Board of Directors approved the reactivation of the Hycroft Mine, and in December 2008, produced its first doré from the Hycroft mine, which was shipped to an offsite refinery for final processing, yielding gold and silver bullion. Permitting was received and construction of a new refinery was completed at the Brimstone plant site by June of 2009. The mine achieved planned ore production capacity by the end of 2009.

With the construction of the North leach pad in 2013, the total leach pad space for the Brimstone, Lewis and North leach pads was increased to more than 20 million square feet. In 2010, the mine began an expansion program that included construction of a 21,000 gallon per minute Merrill-Crowe processing plant and a three-stage crushing facility as well solution pumping capacity upgrades. All of these projects have been completed. Active mining was stopped at Hycroft in June 2015 due to low metal prices and active leaching of previously mined ore continued through 2018.

On October 22, 2015, Allied Nevada emerged from its financial restructuring and changed its name to Hycroft Mining Corporation.

In late 2018, Hycroft began construction of new leach pads to demonstrate its recently developed heap oxidation and leach process in a commercial setting. Additionally, Hycroft began preparing the mine, including its facilities and mining equipment for a restart. Active mining began in March 2019, with a focus on transition and sulfide material. Ore has been placed on the new leach pads and is in the active oxidation phase. Production of gold and silver is expected in the third quarter of 2019.

Sales since the Hycroft mine was reactivated by HMC through July 2019 has totaled approximately 900,000 ounces of gold and 5.0 million ounces of silver.

5.2 MINING HISTORY

The earliest recorded mining in the Sulfur District, where the Hycroft Mine is located, began in the late 1800's following the discovery of significant native sulfur deposits (Couch and Carpenter, 1943; Wilden, 1964). Mining of native sulfur was sporadic from 1900 to 1950 with over 181,488 tons of sulfur ore, grading approximately 20-35% sulfur, mined and milled (McLean, 1991).

In addition to sulfur, high-grade silver mineralization, consisting of nearly pure seams of cerargyrite (AgCl), was discovered in 1908 at Camel Hill (Vandenburg, 1938). Assays up to 3,439 opt silver and 0.362 opt gold were reported (Jones, 1921). Silver mining ceased in 1912 with an estimated 165,375 silver ounces produced. Minor silver mining also occurred along the East Fault at the Snyder Adit, and silver samples as high as 66 opt (Friberg, 1980) and 29 opt (Bates, 2001) were reported.

During the First World War, three 6 to 8 foot-wide veins of nearly pure alunite were mined in the southern part of the Sulfur District (Clark, 1918). In 1931, several hundred tons of alunite were mined as a soil additive (Fulton and Smith, 1932). Vandenburg estimated that 454 tons of alunite was shipped to the west coast to be used as fertilizer (Vandenburg, 1938). From 1941 to 1943, cinnabar was mined from small pits in the exposed acid leach zone (Bailey, 1944). Total mercury production during this period is estimated at 1,900 lbs. (McLean, 1991).

5.3 EXPLORATION HISTORY

In 1966, the Great American Minerals Company began extensive exploration for native sulfur in the area of the Hycroft Mine. Approximately 200 shallow holes were drilled, and numerous trenches dug (Friberg, 1980). In 1974, the Duval Corporation (Duval) drilled 20 holes on the Hycroft property in search of a Frasch-type sulfur deposit (Wallace, 1980). Duval found no evidence of a sulfur deposit at depth, but did report elevated gold and silver values. Duval drilled two core holes (DC-1 and DC-2) and 18 rotary holes (DR-3 through 20) (Ware, 1989).

In 1977, the Cordex Syndicate mapped and rock chip sampled the Hycroft property, recognizing the potential for a bulk tonnage, low-grade precious metal deposit. In 1978, Homestake became interested in the property, recognizing similarities with the McLaughlin hot springs deposit in California. Homestake completed surface sampling and exploration drilling during 1981-1982, and although successful in defining an oxide gold/silver ore body, they dropped the property in 1982.

Table 5-1 references the historical Hycroft drill database, which includes 3,358 exploration drill holes totaling 1,005,089 ft drilled from 1974 through 2005.

Table 5-1: Historical Drilling

Year	Hole Type	Company	No. of Holes	Footage	Zones Drilled0
1974	DD	Duval	2	3,341	Central
1974	RC	Duval	18	6,385	Central
1981-1982	RC	Homestake	120	23,692	Bay, Boneyard, Camel, Central
1982	Rotary	HRDI	4	650	Central
1982	RC	HRDI	65	18,450	Bay and Boneyard
1985	RC	HRDI	191	32,784	Bay, Boneyard, Central, Camel
1986	RC	HRDI	489	104,175	Bay, Boneyard, Central, Camel, Brimstone
1987	RC	HRDI	640	141,880	Brimstone, Central, Bay, Boneyard, Camel
1988	RC	HRDI	73	25,855	Brimstone, Central, Bay, Boneyard, Camel
1989	RC	HRDI	43	15,780	Central
1990	DD	HRDI	8	11,247	Brimstone, Central, Bay, Camel
1990	RC	HRDI	129	43,620	Central, Bay, Camel
1991	RC	HRDI	147	44,360	Brimstone, Bay, Central, Camel
1992	RC	HRDI	265	83,015	Brimstone, Camel
1993	DD	HRDI	6	2,320	Brimstone, Central, Bay, Camel
1993	RC	HRDI	293	103,685	Brimstone
1994	DD	HRDI	3	4,992.7	Brimstone, Central, Boneyard, Camel
1994	RC	HRDI	206	81,060	Brimstone, Central, Boneyard, Camel
1995	RC	HRDI	353	157,015	Brimstone
1996	DD	HRDI	7	3,998	Brimstone, Central, Bay, Camel
1996	RC	HRDI	163	75,090	Brimstone, Boneyard
1997	RC	HRDI	13	3,040	Brimstone
1998	Blasthole	HRDI	67	3,670	Brimstone
1999	DD	HRDI	9	4,869.7	Brimstone
1999	RC	HRDI	11	5,545	Brimstone
2005	RC	Vista	33	13,275	Brimstone

5.3.1 Bay

Bay is a flat lying zone of mineralization hosted by inter-bedded conglomeritic to sandy debris flows (Upper Camel Group) located at the northwest sector of the district. It extends for 5,000 feet in a north-south direction along the Central Fault, between northing 49,000N and 54,500N mine grid. Mineralization extends as far as 2,500 feet to the west of the Central Fault and is 20-250 feet thick. Bay was the focus of exploration drilling between 1985 and 1987, and is the western extension of the Lewis Mine, which was partially mined by Standard Slag between 1983 and 1985. Oxidation forms an 80 to 100-foot-thick blanket over the sulfide mineralization. Bay mineralization remains open to the north.

5.3.2 Central

This zone includes Central, Gap, Cut-4, and Cut-5, and occurs along a length of 10,000 feet in the immediate hanging wall of the Central Fault. The South Central zone was mined immediately after Bay, and extends from approximately northing 42,000N to 46,000N. The Gap was mined second and extends from 46,000N to 49,000N. This last historical mining of the Central zone was in Cut-4, which extends from 39,000N to 42,000N. Cut-5 is a southerly extension of the Cut-4 Zone, for which mining was initiated by HMC in June 2011. Mineralization extends from the Central fault, westward for up between 2,500 to 3,700 ft and ranges from 50 to 1,200 ft thick. Central mineralization remains open to the west, south, and at depth.

5.3.3 Boneyard

This zone strikes north-northeast and is located approximately 1,000 feet east of Bay. The zone is about 3,000 feet long and extends in a north-northeast direction from 47,800N to 50,800N and was mined concurrently with the Gap. Mineralization is 50 to 300 ft thick at Boneyard; although it remains open to the north it is currently being developed as part of the crushing platform and is not considered for mining due to existing and future infrastructure.

5.3.4 Brimstone

Brimstone is located along the east portion of the District. The system extends from at least 40,000N to 45,000N with mineralization present in the hanging wall of the west dipping, normal East Fault. Thickness of mineralization ranges from 200 to 1,100 ft thick. Vista production records show 15,500,000 tons of ore was historically mined from Brimstone, with an average cyanide soluble grade of 0.014 opt Au. Recent exploration of the shallow oxide and deep sulfide potential of the Brimstone has been conducted from 2007 to the present. Mining resumed at Brimstone in 2008. Mineralization remains open to the west.

5.3.5 Vortex

The Vortex Zone was discovered in 2008 by HMC as a result of testing a geophysical anomaly. During that year, ten holes were drilled as part of the discovery phase. The Vortex Zone, as presently explored, measures 3,000 ft x 2,500 ft, and 2,500 ft in depth. Vortex has merged with the Brimstone zone to the north and remains open to the west and at depth.

5.4 PRODUCTION HISTORY

Information on the production history of the Hycroft Mine comes from HMC's in-house documents. Production by Standard Slag commenced at the Lewis Mine in 1983 and continued until 1985. Ore from the Lewis Mine was crushed and stacked on the Lewis leach pads in the north-central part of the district. Lewis Mine production was followed by production from the Bay, South Central, Boneyard, Gap and Cut-4 pits, and finally the north end of the Brimstone pit, as outlined below in Table 5-2.

Table 5-2: Life of Mine Production from Hycroft (1983 – August 2016)

Zone	Years Mined (approximate)	Ore Tons (millions)	Gold Grade AuFA	Oz Au Produced
Lewis Mine	1983-1985	3.9	0.043	75,000
Central, Bay, Boneyard	1987-1995	53.0	0.016	688,968
Brimstone, Central	1996-1998	30.6	0.013	319,443
Residual Leaching	1999-2004	-	-	58,740
Brimstone, Cut 5, Bay	2008-Aug 2016	171.7	0.012	940,281
Total Production	**1983-Aug 2016**	**259.2**	**0.014**	**2,082,432**

The Central Zone ore was either crushed to P_{80} passing 3/4 inch or treated as ROM, depending on the blasthole grade. Central production was stacked on a series of leach pads referred to as pads 1 through 3. Pads 1 and 2 were constructed in 1987, and pad 3 in 1992. Ore was placed on pad 1 from 1988 to 1997, on pad 2 from 1989 to 1997 and on pad 3 from 1993 to 1997. Solutions from these pads were treated in the Crofoot Merrill-Crowe plant located on the northeast side of pad 1.

Detailed records are not available on historic reserve modeling in the Central and Brimstone Zones, but detailed records are available for the pad loading from these deposits. From 1988-1997, a total of 85.64 million tons of ore was placed on all pads, with an average cyanide soluble gold grade of 0.018 opt Au (1.56 million ounces of gold).

Production from the Brimstone pit was placed directly on leach pads 4 and 5 as ROM. Pad 5 consists of additional lifts placed on top of pads 1 and 2. Pad 4, constructed immediately south of the old Lewis pad, was completed in 1996. Loading of pads 4 and 5 commenced in October 1996 and July 1997, respectively. A 2,800 gallon per minute Merrill-Crowe leach solution plant (the Brimstone Plant) was completed and put into operation in February 1997. The Brimstone Merrill-Crowe plant processing capacity was increased to 4,500 gpm in 2010. The plant treated solutions from pad 4 and is located on the northwest side of the pad. Pad 5 solutions were treated in the older Crofoot plant.

No mining data exists after June 2015, when mining ceased, however production data for ongoing solution processing is current as of June 30, 2019.

6 GEOLOGICAL SETTING AND MINERALIZATION

6.1 GEOLOGICAL SETTING

The Hycroft deposit is a low sulfidation, epithermal, hot springs system that contains gold and silver mineralization. Radiometric dates of adularia (potassium feldspar) indicate that the main phase of gold and silver mineralization formed four million years ago (Ebert, 1996) when hydrothermal fluids were fed upwards along high angle, normal faults. Low grade gold and silver mineralization was co-deposited with silica and potassium feldspar throughout porous rock types.

A subsequent drop in permeability, due to sealing of the system, led to over pressuring and subsequent repeated hydrothermal brecciation. Additional precious metal mineralization was deposited during this event as breccia zones, veins, and sulfide flooding.

Gold and silver mineralization was followed 0.4 to 2.0 million years ago by an intense event of high sulfidation acid leaching of the mineralized volcanic rocks coincident with a regional water table drop. This allowed steam heated sulfur gases to condense into sulfuric acid and leach the upper portion of the mineralized rocks.

Oxidation of sulfide mineralization occurs to variable depths over the deposit, depending upon proximity to faults, extent of acid leaching, and depth to water table. Sulfide content through the deposit is variable from 0% to 20%.

6.1.1 Regional Geology

The Hycroft Mine is located on the western flank of the Kamma Mountains in the Basin and Range physiographic province of northwestern Nevada. The Kamma Mountains were formed during Miocene to Quaternary Epoch from the uplift of Jurassic basement rock and emplacement of Tertiary volcanic and sedimentary rocks. The stratigraphy along the western flank of the range is down-dropped to the west, along a series of north to northeast striking normal faults. These faults served as conduits of hydrothermal fluids that deposited the Hycroft mineralization.

The Hycroft property consists of Tertiary to Recent age, fault-controlled, low-sulfidation gold zones that occur over an area measuring approximately three miles in a north-south direction by two miles in an east-west direction. The zones are hosted in volcanic rock eruptive breccias, flows and conglomerates associated with the Tertiary Kamma Volcanics and sand to conglomeratic debris flows associated with the Tertiary Sulphur Group.

Younger rocks at the mine are Tertiary conglomerate, siltstone and fanglomerate of the Sulphur Group (locally termed “Camel Conglomerate”). These rocks are comprised of sediment eroded from the underlying Kamma Volcanics and Jurassic ALS Formation. The Sulphur Group is divided into three main units, a clast supported coarse conglomerate, a matrix supported conglomerate and an underlying tuffaceous lake sediment. This unit outcrops throughout the mine site with increasing thickness to the west.

The older Kamma Group is exposed throughout the Kamma Mountains east of the Central Fault. It underlies the Camel Conglomerate. The volcanic package is comprised of siliceous to intermediate tuffs, coarse grained volcanic clastics, fanglomerates, eruption breccias and massive to flow banded rhyolites.

The Jurassic ALS Formation underlies the Kamma volcanic package. This formation consists of a thin bedded to laminated siltstone, with calcite cementing. ALS is exposed approximately three miles east of the deposit and is encountered only at depth in drilling at Hycroft. A generalized stratigraphic column for the Hycroft deposit area is presented in Figure 6-1. This stratigraphic column illustrates the formations of volcanic origin that host the deposit with notations for lithologies in each formation. The grouping shown is the same that was used to create the lithology wireframes in the geologic model.

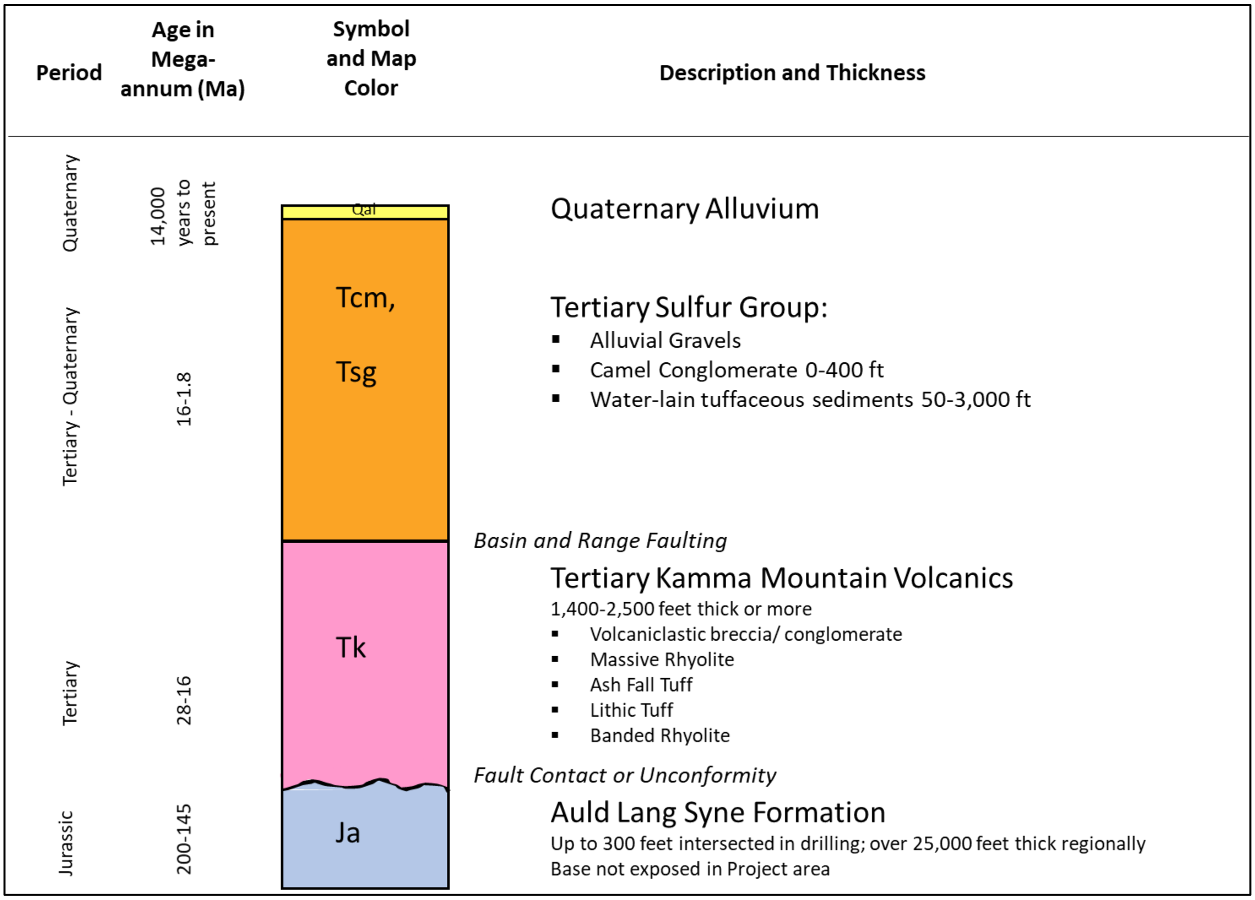


Figure 6-1: Stratigraphic Column for Hycroft Deposit Area

Source: SRK, 2019.

Seven major north-northeast trending, west dipping, normal fault zones appear to broadly control the distribution of gold and silver mineralization as shown in Figure 6-3. From west to east, these fault zones are referred to as the Range, West Splay, Central, Break, Albert, Fire, and East faults. These major structures down-drop stratigraphy and also affect the distribution of alteration and mineralization. A post-mineral basin bounding fault appears to border the Camel Conglomerate and the adjacent Pleistocene Lahontan Lake sediments in the Black Rock Desert. Based on geophysics, this structure is approximately 1 to 2 miles west of the mine site. There are several east-west trending structures that appear to provide post-mineral offset to the deposit. These form a series of horst and grabens within the deposit footprint. Going from north to south, these faults include Cliff, Ramp, Prill, Camel and Hades Faults. Figure 6-2 is a north looking section through the Hycroft Mine showing structures, volcanic rock stratigraphy, and gold/silver mineralization. There are also several other parallel fault zones that may have a significant impact on the localization of mineralization. The depth of oxide and mixed sulfide/oxide gold and silver mineralization varies considerably throughout the area. Alteration at the deposit is dominated by acid leaching, silicification, argilization, and propylitization.

6.1.2 Local Geology

The deposit is typically broken into six major zones based on geology, mineralization, and alteration. These include Brimstone, Vortex, Central, Bay, Boneyard, and Camel. The boundaries are typically major faults, namely Break, East and Ramp.

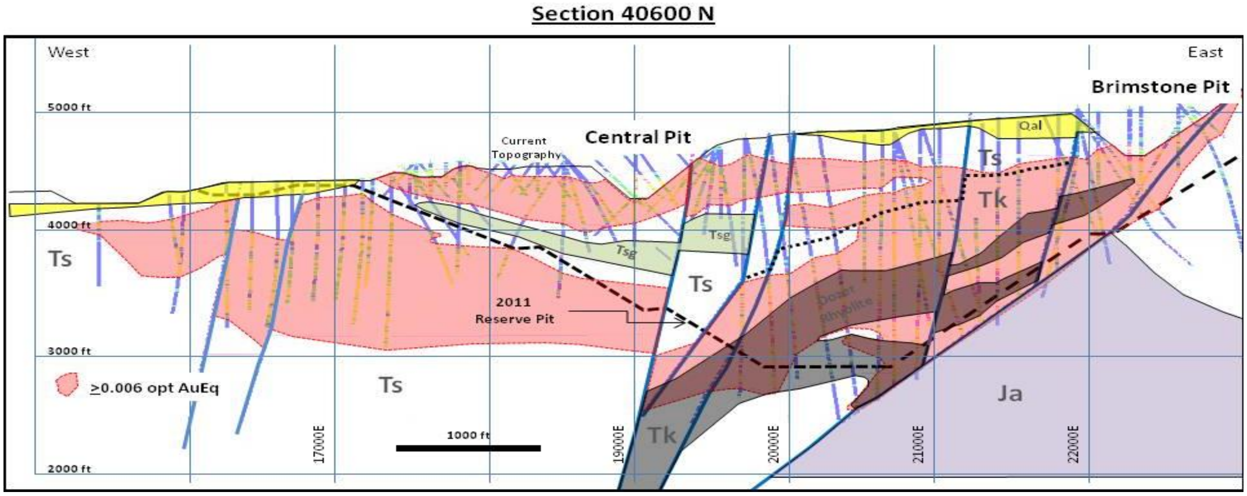


Figure 6-2: Simplified East-West Cross Sections through the Sulfur District

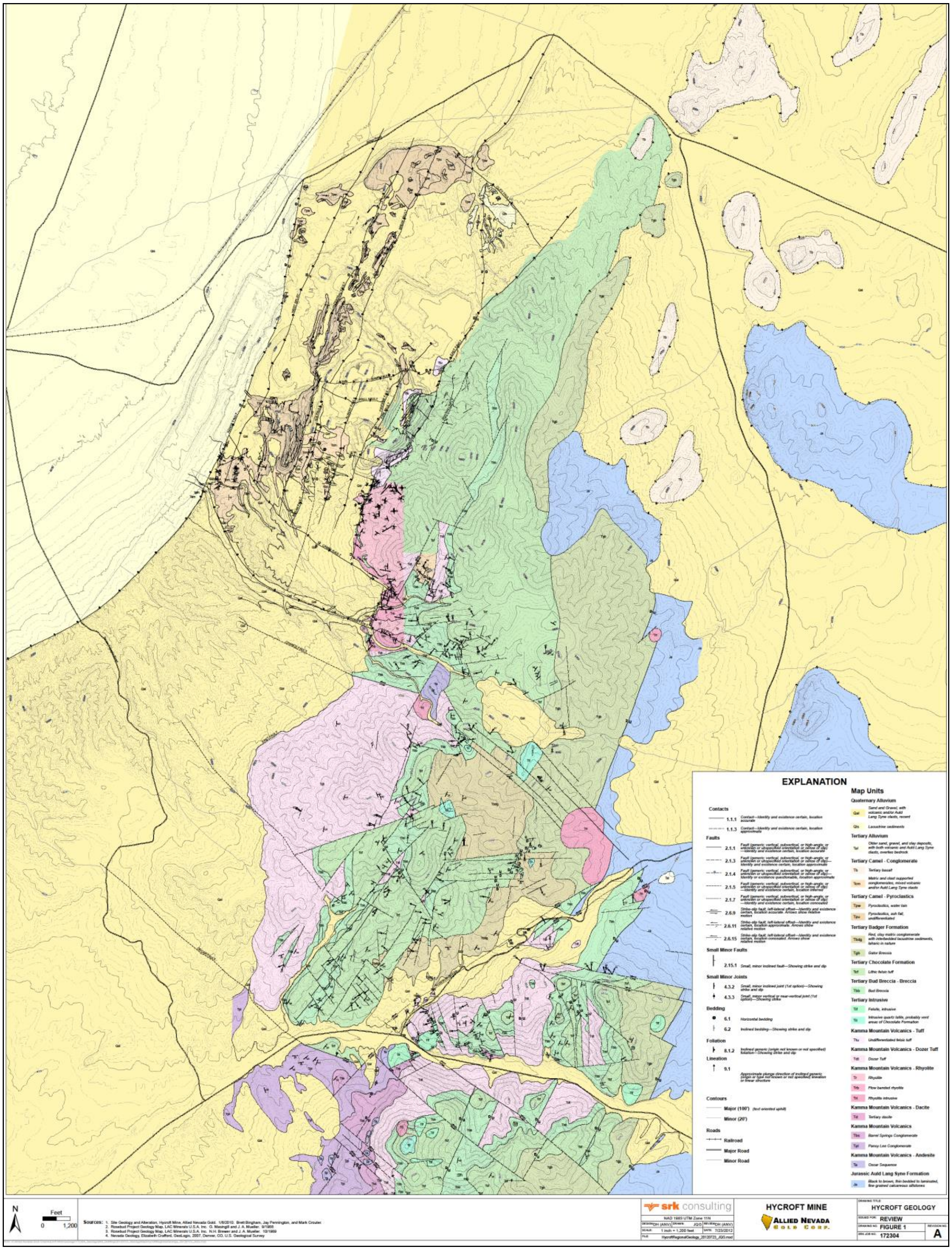


Figure 6-3: Geological Map of the Greater Hycroft Area

6.1.2.1 Brimstone

The stratigraphy at Brimstone includes up to 100 feet of alluvium, underlain by Camel Conglomerate rocks (0 feet to 400 feet), and Kamma volcanic rocks, as shown in Figure 6-4 and Figure 6-5 respectively. ALS has been drilled at depth and is in fault contact (East Fault) with the overlying Kamma Volcanics. The Brimstone ore deposit is hosted primarily by Kamma volcanic rocks in the hanging wall of the East Fault. The volcanic rocks are principally eruption breccias, tuffs, rhyolites, and volcanic rocks proximal to vents, and overlie deformed and metamorphosed shale, sandstone, and siltstone of the ALS group. Kamma Volcanics are strongly altered in the hanging wall of the East Fault, whereas the same units are weakly altered to the east in the footwall of the fault.

At Brimstone, the East Fault is a north-northeast striking, west dipping, normal fault with repeated episodes of movement, including approximately 150 feet to 200 feet of alluvial offset. Where exposed in the Brimstone Pit, the fault clearly shows steep normal movement, with slickensides that plunge 80º to 85º. At depth the fault shallows to 45° to 60º and may merge with the Central and Break Faults. The fault may have originally served as a conduit to hydrothermal fluids. Only minor mineralization is noted footwall to the fault zone.

North of the Brimstone deposit, the east-west trending Ramp and Prill faults appear to down drop favorable stratigraphy. Condemnation drilling of the leach pad to the north has shown only local zones of weak gold and silver mineralization. To the south, the Brimstone Zone transitions to the Vortex Zone, with no apparent change in stratigraphy, but changes to alteration zonation.

Host rocks were highly altered by at least four phases of alteration. The relatively porous conglomerate and breccias were preferentially acid leached by late stage steaming hydrothermal acid vapors. Acid leach alteration extends to depths of 700 feet in some areas of the Brimstone deposit as seen in Figure 6-4, indicating that the water table was present below the base of the acid leached zone. A siliceous layer (basal acid leach), up to tens of feet thick, occurs at the base of the acid leached material. Underlying the acid leaching is a layer of hydrothermal clay alteration, followed by silica potassium feldspar alteration. Pervasive silicification, veining and hydrothermal brecciation are generally found in the rhyolites and breccias.

Zones of silicification of limited thickness, oriented parallel to the East Fault, are present in the footwall zone. Alteration extends for 50 feet to 70 feet footwall to the fault, with pervasive silicification and quartz veining dominant.

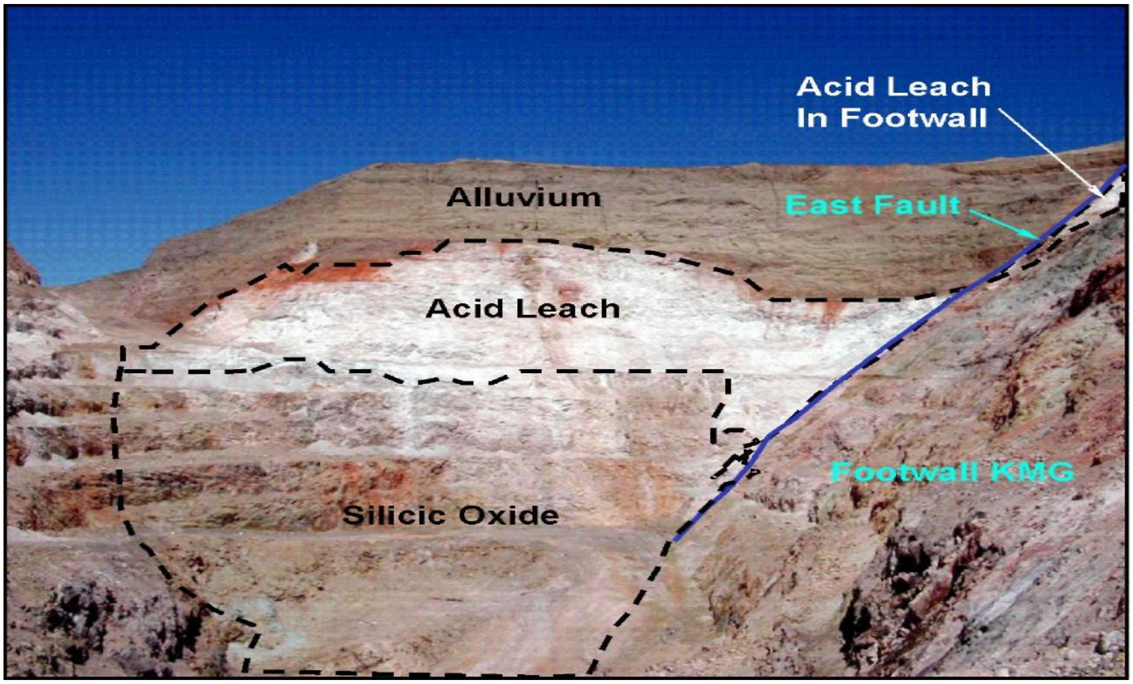


Figure 6-4: Brimstone North Pit Wall Geology

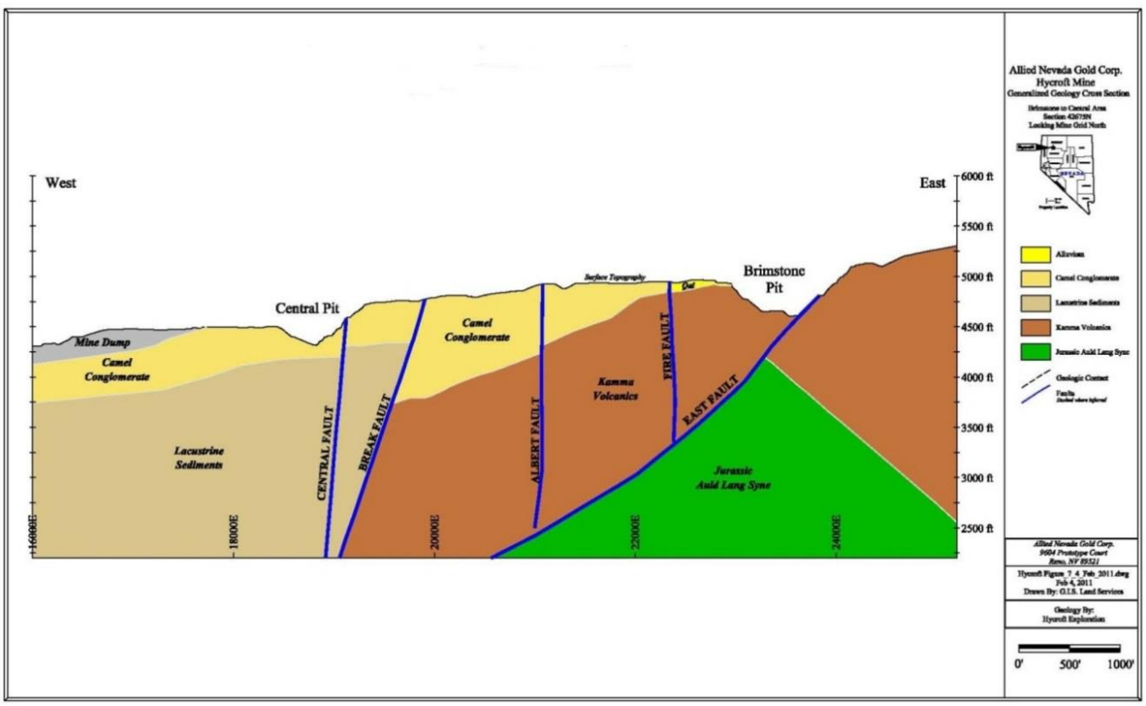


Figure 6-5: Brimstone Generalized Geology Cross Section

Gold and silver are spatially associated with fracture and breccia-controlled chalcedony sulfide mineralization. A subsequent acid alteration event produced the current distribution of oxidized and transition sulfide/oxide ore. The lower acid leach material hosts gold and silver mineralization, as does the underlying silicified and veined volcanics.

Drilling has shown that mineralization extends to a depth of over 1,200 ft in the Brimstone Zone. Mineralization thickness (true width) is 200-1,100 ft thick and remains open to the west towards the Break Fault and transitions into Vortex to the south.

6.1.2.2 Vortex

The stratigraphy in the Vortex Zone is correlative with those at the Brimstone Zone immediately to the north. Camel Conglomerate is underlain by tuffs, volcanic clastics, fanglomerates, and rhyolites of the Kamma Volcanics. The ALS is present, footwall to the East Fault, and appears to be in stratigraphic contact with the Kamma Volcanics, as seen in Figure 6-6.

The upper elevation at Vortex is hydrothermally clay (kaolinite) altered. Acid leaching is less prominent than in Brimstone and is focused primarily along the East Fault. Strong silicification to depths greater than 1,500 feet is due to veining and phreatic hydrothermal brecciation. At least four mineralizing events are present as evidenced by crosscutting vein and breccia relationships. The hydrothermal venting may have contributed to the eruption breccias overlying the Brimstone Zone. Propylitic and/or clay alteration extends outboard of the silicification.

The mineralization at Vortex is of both vein and disseminated type, with brecciated and altered rhyolite rocks and volcanic clastics acting as favorable hosts. In addition to gold mineralization, high grade silver has been encountered at Vortex; with values ranging from 10 to 647 opt. The predominant silver minerals are pyrargyrite, naumanite and miargyrite, occurring both in veins, disseminated and coarse grains along fractures.

Oxide mineralization is present at a depth of approximately 500 feet below surface, with sulfide mineralization extending to 2,500 feet below surface. Mineralization thickness (true width) is 1,000 to 1,800 ft thick. Banded quartz veins with both high-grade silver and gold have been noted in core. Drilling to date indicates that the high-grade zones are both high angle banded quartz veins and a more extensive flat lying, massive quartz zone containing visible pyrargyrite and miargyrite.

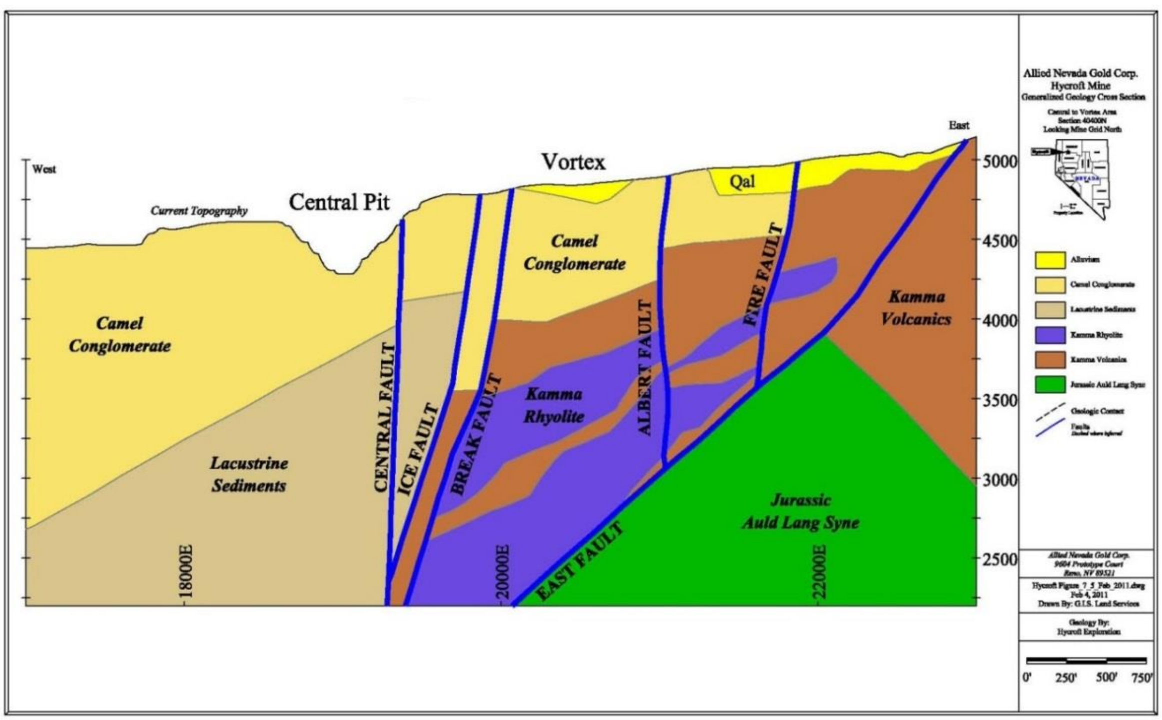


Figure 6-6: Vortex to Camel Generalized Section

6.1.2.3 Bay and Boneyard

Mineralization in the Bay and Boneyard zones is hosted by gentle, west dipping Camel Conglomerate. Both clast-supported and matrix-supported conglomerate rocks host mineralization. The basal rock type is tuffaceous lake sediments, composed of fine grained clay with minor layers of gravel and conglomerate extending to a depth greater than 1,100 feet as shown in Figure 6-7. Mineralization is primarily bedding controlled, with the Range and Central Faults as the main feeders. The Break Fault may also have zoning controls but is poorly drilled in this zone. Mineralized siliceous hot spring sinters have been historically mined indicating that this deposit represents the upper-most levels of a hot spring hydrothermal system.

The predominant alteration type at Bay is silicification. Acid leach alteration in the area is relatively minor and occurs along high angle structures (Figure 6-8). Clay alteration of the underlying lacustrine sediments is also noted in limited drill holes and is illite smectite dominated. Strong oxidation is present in the upper portion of the silicified zone.

Gold and silver mineralization is associated with flat lying Camel Conglomerate, above the lacustrine sediments of the Tsg formation. Mineralization thickness (true width) is 20 to 250 ft thick at Bay and 50-300 ft thick at Boneyard. This zone transitions into the upper zone of mineralization at Central. Bay and Boneyard remain open to the north and east.

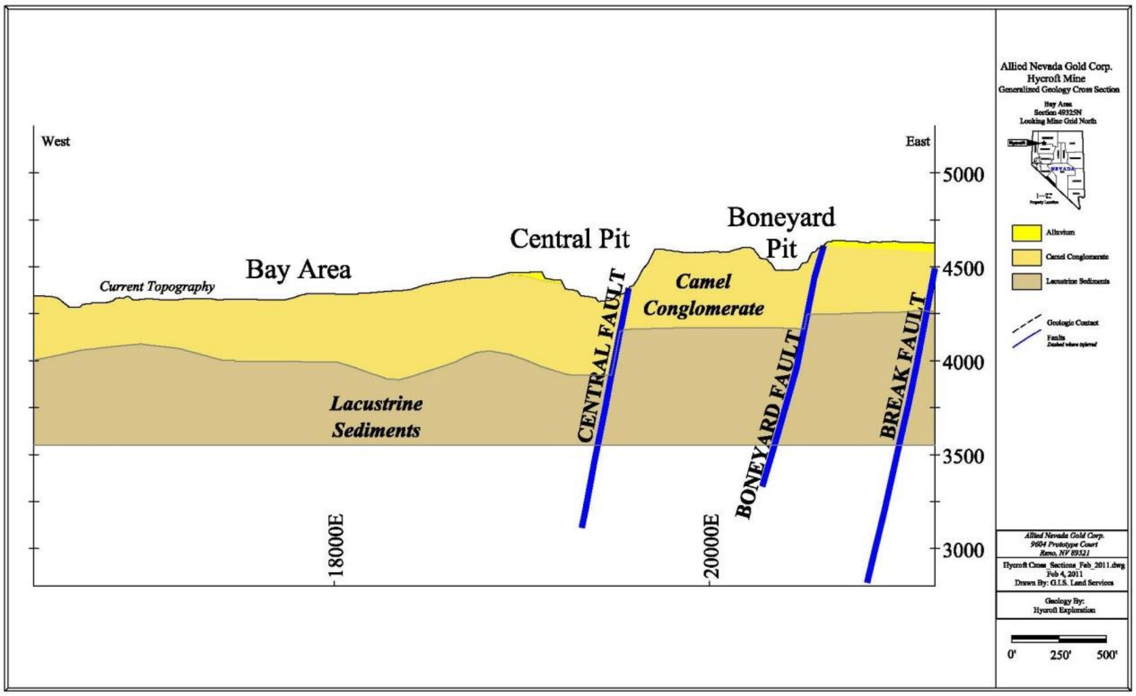


Figure 6-7: Bay Geologic Cross Section

Figure 6-8: Bay Looking North

6.1.2.4 Central

The Central Zone geology is similar in nature to that of Bay, with mineralization and alteration fed by high angle faults and fractures, with dominant lateral fluid flow through the porous conglomerate rocks of the Sulphur Group as seen in Figure 6-9. Camel Conglomerate units are underlain by lacustrine sediments. However, the lacustrine units thin dramatically to the south, with less than 50 feet of the material noted south of Cut-4.

The Central Zone is bounded to the east by the Central and Break Faults. Fault movement is unknown, but extends at least 2,000 feet, with recent reactivation in the quaternary (50 feet to 150 feet), as demonstrated by offset in the alluvium. The Range Fault to the west may provide an additional boundary, although drill data is limited at this time.

Alteration along the Central Zone is similar to that of Bay. Acid leach alteration is stronger and more widespread than at Bay and is extensive in the southern portion of the pit. The acid leaching overlies silicified conglomerate rocks, except along the immediate trace of the Central Fault where silicification dominates as the alteration type. Oxidation extends downward approximately 400 feet. Underlying the silicification and acid leaching are illite and smectite clay-altered and clay dominant lacustrine sediments. Hot spring sinter deposits have not been observed.

Gold and silver mineralization is associated with favorable stratigraphic horizons in the Camel conglomerate, with an upper and lower zone noted in drilling, separated by a north-south striking, east dipping clay layer. Mineralization remains open to the west, past the Range Fault, and at depth (>1,400 feet). Mineralization thickness (true width) in the upper zone is 50-300 ft thick, while the lower zone ranges from 300-1,200 ft thick, and remains open at depth. The zone mineralization is contiguous to the Vortex and Brimstone Zones to the east, and the Camel/Cut-5 zones to the south.

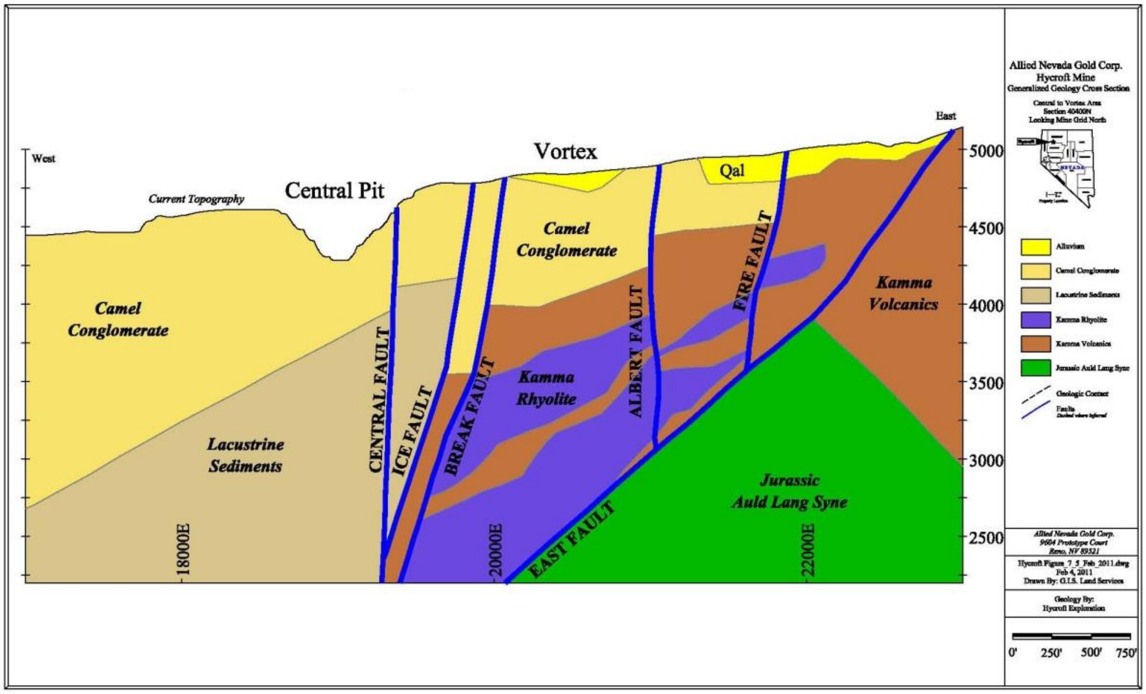


Figure 6-9: Central Pit Geologic Cross Section

6.1.2.5 Camel and Cut-5 Zones

Camel Conglomerate is the dominant lithology at Camel. The conglomerates appear to extend to depth in this zone, with only thin lake sediments drilled to date. The lack of lake sediments can be attributed to either the Camel Fault or facies changes along a shoreline. The Camel Fault is an east-west trending fault, with down-drop to the south, which is presently poorly defined by drilling.

Alteration south of the Central Pit and in the Camel Zone is predominantly comprised of silicification and clay alteration. Hydrothermal clays, overlying silicified conglomerate rocks, and basal illite-smectite clay altered rocks are present. Acid leaching in the area is relatively minor, especially with respect to the intensity and amount in the Central and Cut-4 Zones area immediately to the northeast.

Mineralization in the Camel/Cut-5 Zones is hosted by conglomerate rocks and occurs as both disseminated gold and silver associated with pyrite and marcasite, and higher-grade veins, including silver bearing pyrargyrite veins. Mineralization thickness (true width) is 200-1,100 ft thick, extends to depths greater than 1,400 feet, and remains open at depth. Oxidation extends to depths greater than 200 feet and an area of intense oxidized mordenite alteration is present between the Cut-5 and Camel Zones. Mineralization remains open to the south, west and at depth. To the north, Camel mineralization is contiguous with the lower zone of the Central Zone, while Cut-5 is contiguous with the upper zone. Mineralization is also open to the west of Camel and to the south towards Hades Fault.

6.2 ALTERATION AND MINERALIZATION

6.2.1 Alteration

The main alteration events in the Hycroft District occurred in the following sequence:

- Barren propylitic alteration of the Kamma volcanic rocks.
- Barren illite smectite clay alteration of the Camel Conglomerate and sedimentary rocks on the western portion of the Hycroft deposit and related alteration of large areas of the Kamma volcanic rocks (illite + quartz + pyrite).
- Hydrothermal activity produced a layer of kaolinite montmorillonite clay at the top of the opal chalcedony flooding and above hydrothermal breccias.
- Widespread opal chalcedony, k-spar, pyrite and marcasite (termed "silica sulfide") flooding of the sedimentary Camel Conglomerate, hydrothermal breccia ejecta, and related fragmental rocks was synchronous with the illite smectite event. Blanket acid sulfate (acid leach) alteration formed a vertically zoned layer of upper residual quartz and a lower layer of intense opalization, termed basal acid leach.
- Hypogene alteration oxidized silica and clay rich rocks at the base of acid leach alteration.
- Mordenite alteration (zeolite and clays) overprints the opal k-spar alteration, especially in the Gap and Bay areas, and reaching depths of 160 feet in places.

Most recently, supergene oxidation of acid leach, oxide and sulfide mineralization occurred along major faults, accompanied by small amounts of normal movement, displacing mineralization in the hanging wall downward.

Each alteration and type are described below in detail.

6.2.1.1 Propylitic

Propylitic alteration has only been noted in volcanic rocks of the Kamma Mountains, both in drill samples and from surface mapping in the mountains. Propylitic alteration is pervasive in the Kamma Mountains and affects the rocks in the Hycroft deposit both pervasive and as veining, especially in rhyolite flows and intrusive rocks at Vortex and southern Brimstone. Typically, the propylitic alteration gives the rocks a bright green color and minerals consist of chlorite, quartz k-spar, calcite and pyrite.

6.2.1.2 Illite Smectite Clay Alteration

Illite smectite clay alteration underlies the near surface silicification in the western portion of the district in the sedimentary rocks of the Camel Conglomerate and basal clay rocks. Rocks have been pervasively altered to a mixed layer of Illite smectite plus/minus quartz, calcite, pyrite, kaolinite and pyrrhotite assemblage. The alteration gives the rocks a gray to greenish gray color and extends to depths greater than 1,000 ft. The composition of the Illite smectite varies with both distance from faults and with depth, with increasing Illite content indicating higher temperature with depth and proximity to silicified conduits.

The contact between this alteration type and the opal k-spar alteration is transitional and suggests that the timing of the two events is roughly synchronous.

6.2.1.3 Opal K-spar Alteration

A widespread event of low-grade silica pyrite potassium feldspar alteration created a blanket of silica sulfide alteration, resulting in rocks having a glassy appearance. Fine grained, euhedral to subhedral pyrite is always associated with this alteration. Pyrite forms 2% to 5% of the rock as fairly uniform, bright yellow to brassy grains, about 0.008 in. to 0.02 in. in size, and occurring evenly distributed throughout the rock mass. Up to 50% of the rock mass is composed of microscopic potassium feldspar. The alteration gives the rocks a gray color and extends to depths greater than 2,500 feet.

6.2.1.4 Quartz Chalcedony Veining and Silica Flooding

Quartz chalcedony veins cut acid leach alteration in the Brimstone and Vortex zones. These veins and associated silica flooding of wall rock may be related to hydrothermal brecciation and phreatic eruption events. These veins and breccias may be from inches to feet thick, which show several stages of formation by their crosscutting nature and often are associated with sulfide selvages and fine-grained sulfide flooding (locally termed 'sooty sulfides'), giving the rock a dark gray to black color. Veins are commonly banded and can contain brecciated fragments of other veins. The presence of calcite as euhedral rhombs and replacement by quartz is common.

6.2.1.5 Hypogene Oxidation

This original hypogene alteration is composed of two dominant types, silicic oxide and clay oxide. Iron oxide minerals occur on fractures and in original sulfide sites. Silicic oxidation comprises about 85% of all oxide samples (Figure 6-10). Silicic oxidation underlies acid leach alteration and reaches thicknesses of up to 200 feet. In the majority of oxide mineralization, all sulfides have been converted to iron oxides. Silicic oxide is fine grained and glassy appearing, with little or no secondary porosity development. Iron oxides, sulfates and hydroxides are common accessory minerals, with hematite being the most prevalent oxide.

Oxidation of the silicic alteration can have a variety of dominant colors including white, yellow, red, and purple, depending on the relative amounts of iron oxides, hydroxides, and sulfates. Silicic oxide is composed of 65% to 85% silica, 5% to 20% clay, and 5% to 15% hematite and jarosite.

Oxidation of clay altered rock comprises about 15% of material classed as oxide and is thought to be the result of hydrothermal alteration of in-situ rock, representing formation under weak acid oxidizing conditions. Clay zones appear white to yellow to pinkish and are composed of 50% or more clay, with the usual accessory iron oxides. Clays are mixtures of montmorillonite and kaolinite with accessory alunite and occur as discontinuous layers 30 feet to 50 feet thick directly beneath basal acid leach alteration, as irregular veins, or amoeboid shaped areas scattered throughout the silica oxide alteration. At the Vortex Zone, clay oxidation may extend to depths greater than 200 feet.

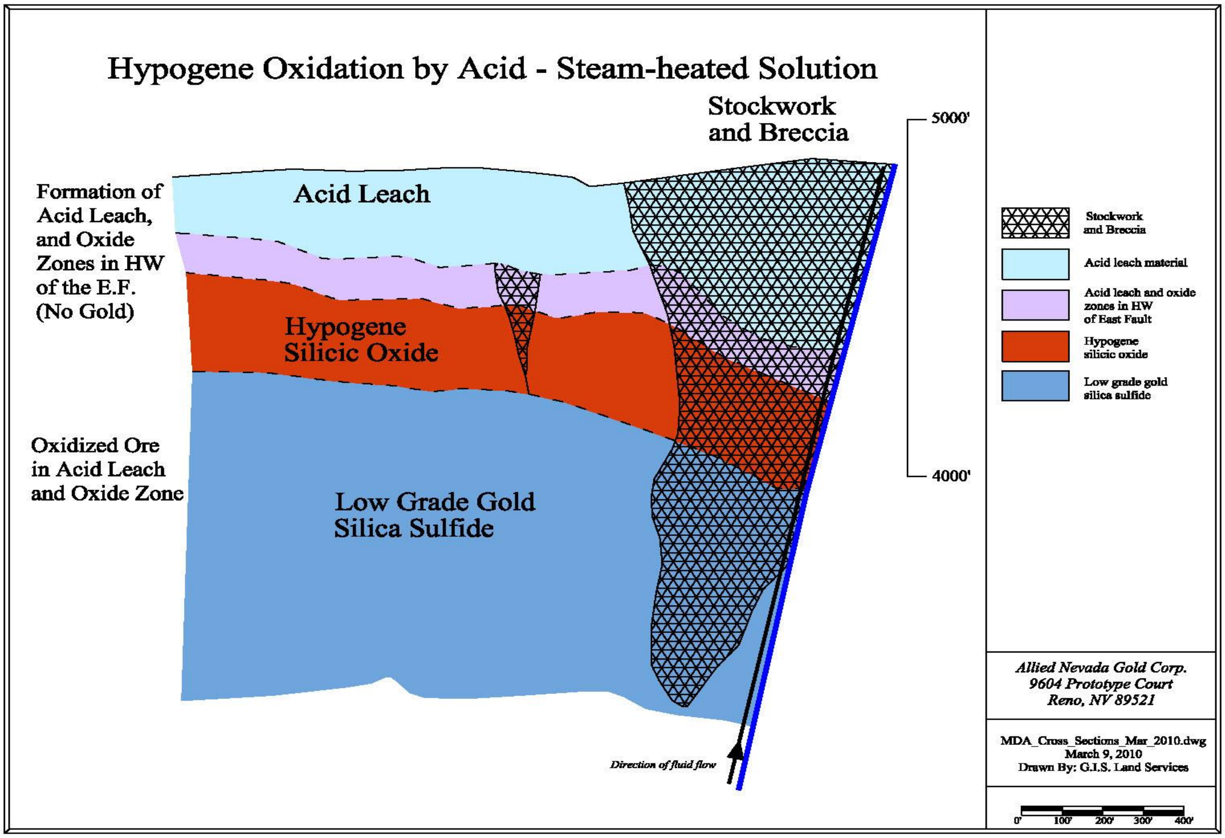


Figure 6-10: Hypogene Oxidation by Acid - Steam Heated Solution

6.2.1.6 Supergene Oxidation

Supergene oxidation extends to depth along faults, manifested as a zone of oxide stained fault gouge. Figure 6-11 shows a schematic section of the distribution of this alteration, which appear to be the final alteration event.

The zone appears very similar to oxidized/silicic and small fragments of acid leach alteration are caught up in this material. Bright red hematite most often coats all fragments in this zone. In deeper levels of the north Brimstone pit, black manganiferous oxides also occur. Supergene oxidation forms a band 20 feet to 80 feet wide in faulted contacts.

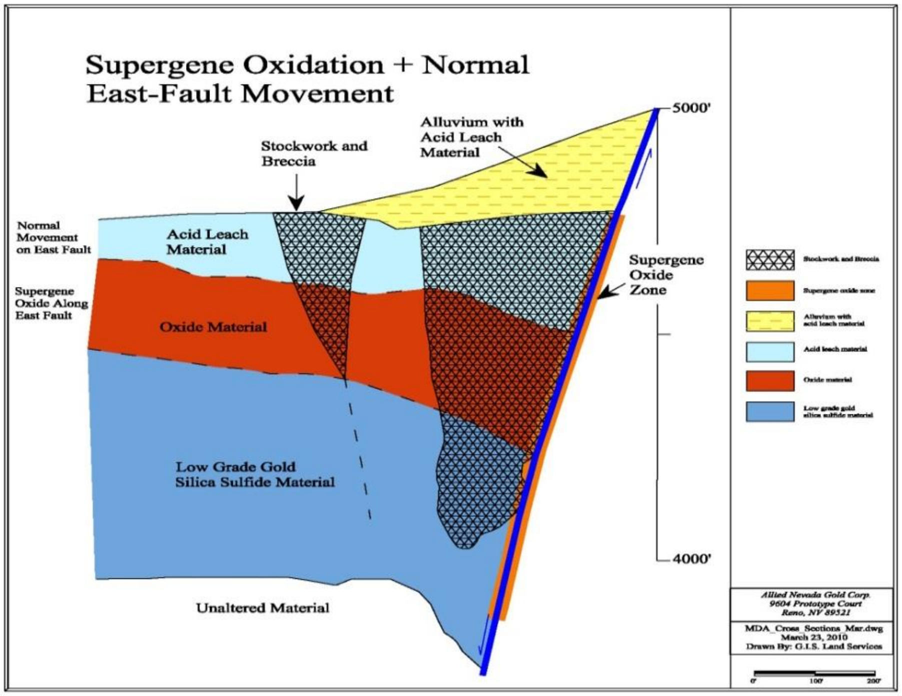


Figure 6-11: Supergene Oxidation + Normal Fault Movement

6.2.2 Mineralization

Several styles of mineralization exist at the Hycroft deposit. An early silica sulfide flooding event deposited relatively low-grade gold and silver mineralization, generally along bedding. This is crosscut by later, steeply dipping quartz alunite veins. Hypogene enrichment of gold and silver occurred at the base of the acid leach blanket. Late stage silver bearing veins are found in the Vortex zone and at depth in the Cut-5 area. Late to present supergene oxidation along faults has liberated precious metals from sulfides and enriched gold and silver, generally along water table levels. True thickness and continuity of mineralization is discussed by geologic area in Section 6.2.1.

6.2.2.1 Siliceous Sinters

Large, near-surface, silica sulfide mineralization hosts low-grade gold and silver mineralization throughout the Hycroft deposit. Bay has a blanket of this mineralization that was overprinted with mordenite alteration and the accompanying oxidation resulted in a near surface exposure of gold and silver. Gold and silver occur as very small sized grains associated with sulfides, and in the matrix of the rocks.

6.2.2.2 Quartz Alunite Veins

Steeply dipping quartz veins host gold and silver. The veins have been deposited in fractures below the basal acid leach and in fractures and voids in the main acid leach blanket. Banded veins are found in Central, Brimstone and Vortex, and are typically higher grade than the surrounding low-grade mineralization**.** Gold occurs as small sized electrum grains, averaging 30% silver, within and adjacent to opal, alunite and clay minerals. Silver also occurs as cerargyrite and iodargyrite associated with alunite, clays, or jarosite.

6.2.2.3 Quartz Chalcedony Mineralization

Fracture and breccia-controlled chalcedony pyrite marcasite mineralization is associated with primarily gold and possibly silver deposition at Brimstone and the Vortex Zone, occurring as veinlets, stockworks, in situ breccias and rotational (chaotic) breccias. This mineralization type clearly crosscuts the earlier low-grade silica pyrite alteration and acid leached material. The veinlet mineralization occurs as 0.04-inch to 0.8-inch veinlets forming 2% to 10% of the rock mass. The veinlets are composed of gray to milk white chalcedony with 5% to 10% sulfides. In situ breccias show flooding of the rock fractures with the chalcedony sulfide assemblage filling a network of fractures occupying 5% to 15% of the rock mass.

Fracturing, veining and brecciation related to phreatic explosions led to both mineral deposition and discontinuous blankets of fragmental ejecta. The ejected fragmental rocks are often acid leached, and some are cut by quartz chalcedony veining. This relationship suggests that the phreatic events were long lived and lasted throughout the acid leach event.

Sulfides are dominated by pyrite and marcasite. Pyrite occurs within veinlets as irregular anhedral masses which are sub-parallel to the veinlet edges and from 0.02 inches to 0.2 inches in size. Marcasite occurs as similar sized masses and as single crystals. Marcasite is euhedral to subhedral, with masses forming twin sheaf like groups of crystals. Gold, and possibly silver, mineralization was most likely introduced during this event. Visible gold, 0.002 inches to 0.005 inches in size, has been identified within chalcedonic veins in thin sections and is closely associated with marcasite.

6.2.2.4 Late Stage Quartz Silver Sulfide Mineralization

At Vortex and Central late stage antimony silver sulfides are associated with zones of quartz veins. Pyrargyrite occurs as discrete veins, selvages along quartz veins and as crystals deposited on both small voids and clay filled fractures. The quartz veins are both banded and massive, with the mineralized veins ranging from sub-millimeter to centimeters in size. The massive quartz veins are white quartz, often have a 'moth-eaten' appearance due to numerous vugs, and are from meters to tens of meters thick. These veins are interpreted as feeder veins, possibly along structures and cut earlier brecciation and their geometry has yet to be detailed.

This mineralization style appears to favor a horizon intercepted at roughly the same elevation at depths ranging from 900-1,375 ft within the Vortex and Central Zones from several holes spaced approximately 500 ft apart.

6.3 DEPOSIT TYPES

The Hycroft deposit is a large, epithermal, low-sulfidation hot springs deposit (Figure 6-12). Gold and silver mineralization are noted as both disseminated and vein controlled, with gold values ranging from detection to 8.8 opt, and silver ranging from detection to 647.5 opt.

Exploration drilling targets zones of hydrothermally altered rocks. Angle core holes are commonly drilled to intersect the high-angle feeder structures. See "Section 9 – Exploration" for a discussion of concepts being applied to the exploration program.

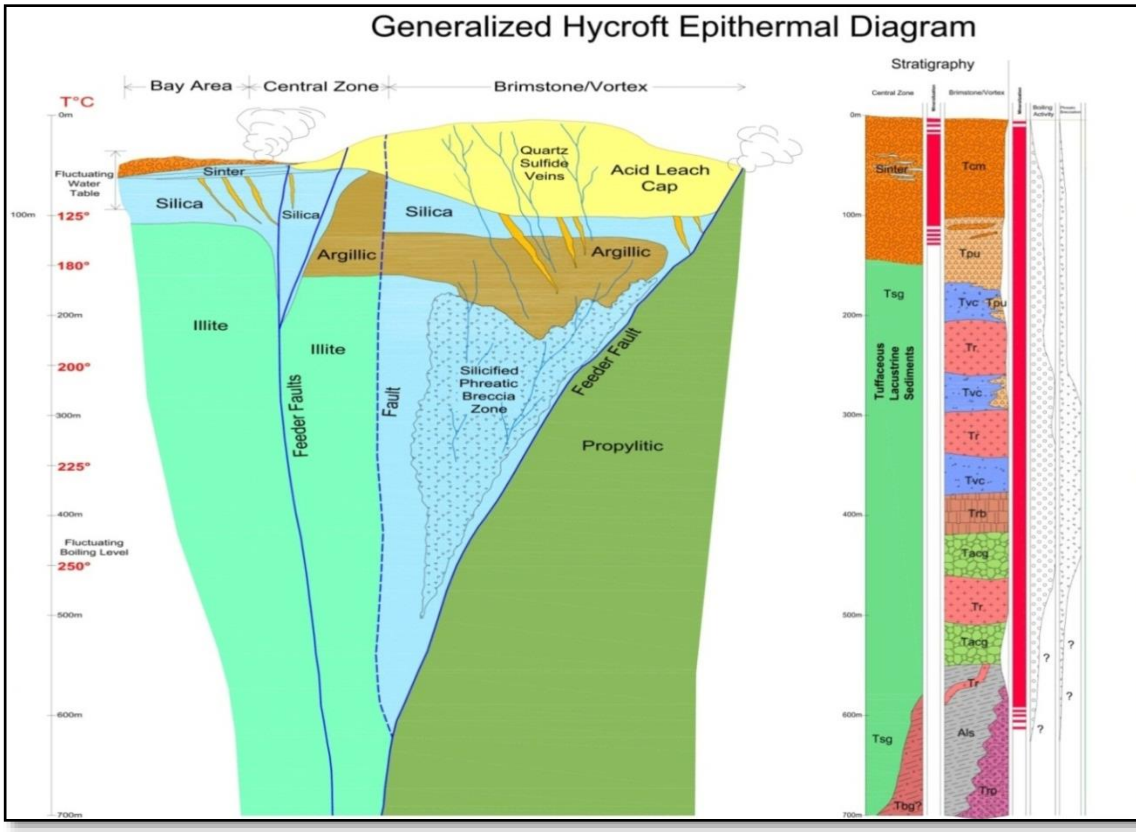


Figure 6-12: Generalized Hycroft Epithermal Diagram

7 EXPLORATION

In addition to drilling activity, Hycroft Mining has also conducted geophysical surveys, soil and rock chip sampling programs, field mapping, historical data compilation, and regional reconnaissance at Hycroft. These efforts are designed to improve the understanding of the known mineralization, as well as provide data for further exploration of the greater property position.

A soil sampling grid was conducted over the Vortex and Brimstone areas historically (1,797 samples) and was extended approximately 5,200 ft. north and 29,600 ft. south of the mine in 2011-2012 (1,834 samples). The soil sampling program was conducted primarily along the East Fault exposure, which is a primary ore controlling feature at Vortex and Brimstone. Results, using gold, silver, arsenic, and antimony, indicate potential exploration targets to the south of the Vortex area. At present these have been identified as the Wild Rose, Chance, Rabbit, Chalcedony, and Oscar target areas. Gold values range from 0 to 0.027 opt, while silver values range from 0 to 3.7 opt. Soil samples are taken on an evenly spaced grid, and screened for coarse material and wind-blown material, resulting in a fraction between 2 mm and 180 um being prepped for analysis. These samples are considered representative of local soil geochemistry and are used to guide the regional exploration effort.

Rock chip sampling has been conducted both historically in the active mine area, and on a regional basis (2007-present). A database of 2,416 samples has been compiled, covering the greater land position. Using gold, silver, arsenic, and other elements, exploration targets have been developed both north and south of the current mine. These include Wild Rose, Chance, Oscar, Rabbit, Floka, and Cliffs. Gold values range from 0 to 0.372 opt, while silver values range from 0 to 71.8 opt. Rock chip samples have been taken on most outcrops, with a focus on alteration and potential mineralization. These samples are used as a guide to exploration and are point samples only.

The land position has been surveyed with both gravity and induced polarity ("IP") geophysical techniques by HMC. The current ground-based gravity survey covers approximately 130 square miles, centered on the mine site. Gravity indicates several structural features and density changes that offer potential exploration targets. These targets include Floka, Blowout, and Oscar. Gravity has also defined the basin edge to the west, approximately 4 miles west of the Brimstone Pit.

Ground IP surveys were run over the mine site and Vortex in 2007 and extended outward in 2011 to cover approximately 24 square miles. The survey results focus on chargeability anomalies, that potentially identify sulfide material (> approximately 1.5%) at depth, and resistivity anomalies, that potentially identify silicification at depth. Results have identified additional exploration targets at Floka, Cliffs, Blowout, Wild Rose, and Chance.

Field mapping was historically and is currently carried out in all active mine areas. Mapping focuses on structure, bedding, joints, lithology, and alteration. The near mine data is incorporated into the three-dimensional geology model, while the regional work is focused on defining exploration targets for future drilling. A regional geology map covering the land position was compiled in 2012 (Figure 6-3). Regional exploration data from Homestake, LAC Minerals, USX, HRDI, and others has been compiled from both in-house and public data sources. Approximately 250 drill holes, various soil and rock chip locations and results, and various field maps have been identified at present.

7.1 DRILLING

The Hycroft exploration model includes data from 1981 to 2018 and includes 5,501 holes, representing 2,482,722 feet of drilling (Figure 7-1, Table 7-1 and Table 7-2). There have been 5,576 drill holes completed in the Hycroft Project Area; some are water wells or are outside the resource model domain, and were not applied to estimation. The drill hole collar locations are shown in Figure 7-1. Exploration drilling was started in 1974 by Duval Corporation, which was evaluating the property for a Frasch-type sulfur deposit and the copper potential. Although native sulfur appeared to be limited to the acid leach zone, gold and silver mineralization was discovered at depth, with the

deepest hole completed to 2,000 feet. Duval concluded that the property did not have large scale sulfur potential. Twenty drill holes (9,726 feet) were completed on the project.
From 1981 to 1982, Homestake, using their McLaughlin deposit as a model, completed 96 reverse circulation drill holes totaling 16,537 feet, primarily in the Bay and Boneyard areas. Shallow oxide gold mineralization was discovered, but Homestake declined the opportunity. Crofoot and American Slag then proceeded to acquire the property rights and initiated small scale oxide heap leach mining at Central and Bay in 1983. Homestake also completed 8 core holes during this timeframe, but collar location data has not been located.

HRDI gained control of the district in 1985 and drilled 3,212 exploration holes, totaling 965,552 ft, between 1985 and 1999. The bulk of this drilling was shallow and focused on oxide gold mineralization at Central, Bay and Brimstone.

In 2005, Canyon Resources completed 33 drill holes totaling 13,275 feet of RC drilling. These were completed primarily in the Brimstone pit area.

Historic drilling was conducted prior to the New Mining Rules reporting requirements. In the QP's opinion, no significant issues have been identified with this historic data and therefore the historic drilling and assay results are incorporated into the Hycroft model.

HMC commenced systematic exploration and resource development drilling starting in 2006. Drilling has been focused on oxide reserve delineation, sulfide resource definition, sulfide exploration, condemnation drilling for facilities, silver data and both geotechnical and metallurgical core samples. Between late-2006 and August 31, 2016, HMC has completed 1,970 exploration holes, totaling approximately 1.45 million feet.

A combination of rotary, reverse circulation and core drilling techniques has been utilized to verify the nature and extent of mineralization. The majority of samples have been collected using reverse circulation drilling methods on 5-foot sample intervals. Reverse circulation drilling utilizes 4.5-inch to 5.5-inch tooling. Deeper drilling is conducted with diamond drilling, using PQ, HQ and NQ tooling. This practice continued through 2013. Since 2013, a RC drilling program was completed in 2014, and a metallurgical core program with the six drill holes was completed in 2017. The metallurgical drill holes were not included in the database for mineral resource estimation and are not shown on the drill hole location map. Various protocols applied to drilling by HMC are consistent with industry standards and the resulting data is of good quality for use in the Hycroft model. Shallow drill holes to sample heap material were completed with sonic coring. The 2018 sonic drilling program was limited to 56 vertical holes in sulfide stockpiles and did not include in situ alluvium or bedrock material. While these were not used for interpolation of in situ rock, they were applied to estimate grades in fill material.

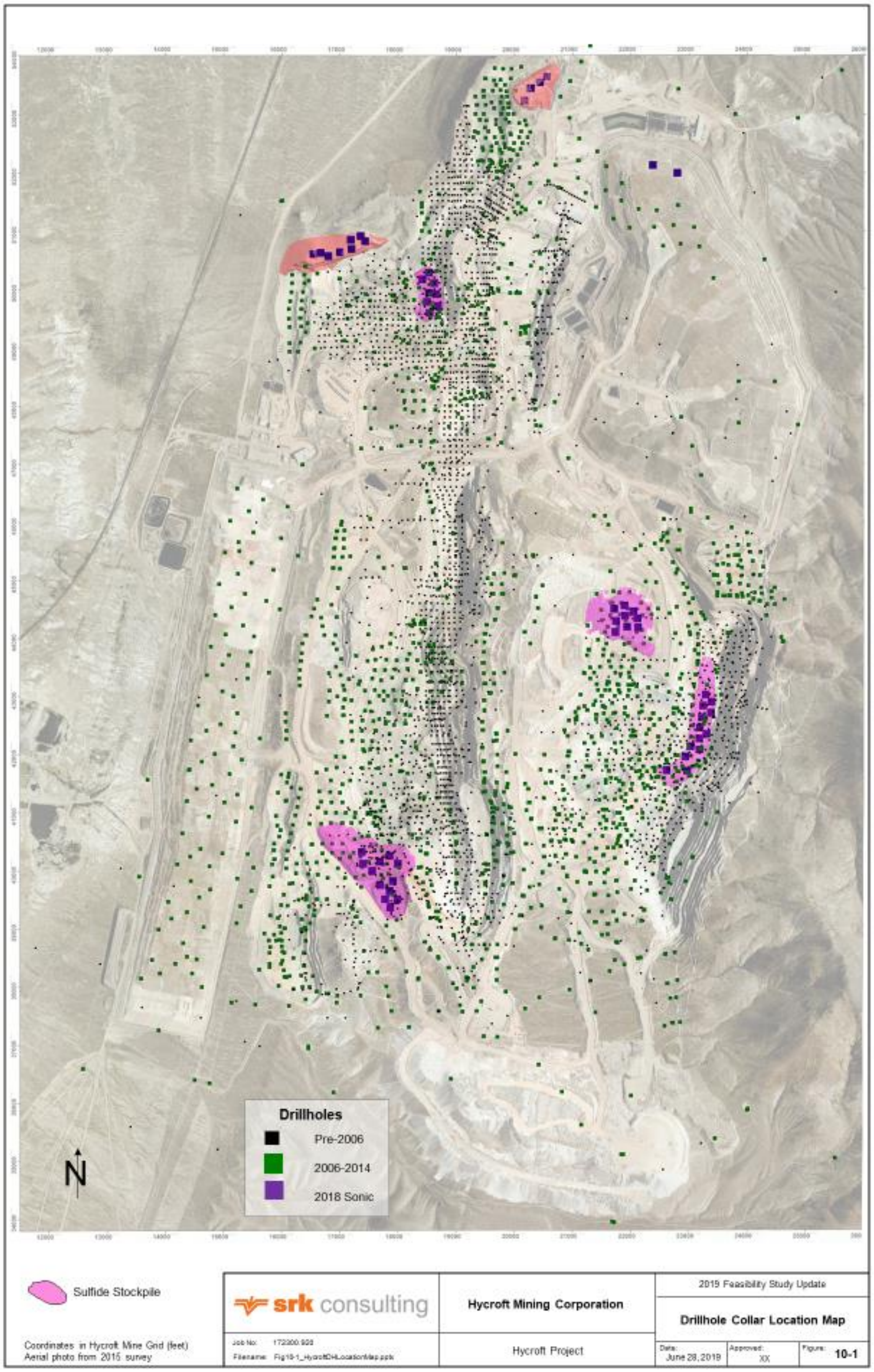


Figure 7-1: Drill Hole Collar Locations

Table 7-1: Hycroft Exploration Drill Campaigns

Year	Hole Type	Company	No. of Holes	Zones Drilled
1974	DD	Duval	2	Central
1974	RC	Duval	15	Central
1981-82	RC	Homestake	120	Bay, Boneyard, Camel
1982	Rotary	HRDI	4	Central
1982	RC	HRDI	65	Bay and Boneyard
1985	RC	HRDI	191	Bay, Boneyard, Central, Camel
1986	RC	HRDI	489	Bay, Boneyard, Central, Camel, Brimstone
1987	RC	HRDI	640	Brimstone, Central, Bay, Boneyard, Camel
1988	RC	HRDI	73	Brimstone, Central, Bay, Boneyard, Camel
1989	RC	HRDI	43	Central
1990	DD	HRDI	8	Brimstone, Central, Bay, Camel
1990	RC	HRDI	129	Central, Bay, Camel
1991	RC	HRDI	147	Brimstone, Bay, Central, Camel
1992	RC	HRDI	265	Brimstone, Camel
1993	DD	HRDI	6	Brimstone, Central, Bay, Camel
1993	RC	HRDI	293	Brimstone
1994	DD	HRDI	3	Brimstone, Central, Boneyard, Camel
1994	RC	HRDI	206	Brimstone, Central, Boneyard, Camel
1995	RC	HRDI	353	Brimstone
1996	DD	HRDI	7	Brimstone, Central, Bay, Camel
1996	RC	HRDI	163	Brimstone, Boneyard
1997	RC	HRDI	13	Brimstone
1998	Blasthole	HRDI	67	Brimstone
1999	DD	HRDI	9	Brimstone
1999	RC	HRDI	11	Brimstone
2005	RC	Vista	33	Brimstone
2006	RC	HMC	1	Brimstone
2007	RC	HMC	38	Brimstone, Bay
2007	DD	HMC	14	Brimstone, Bay
2008	RC	HMC	279	Brimstone
2008	DD	HMC	60	Brimstone, Bay
2009	RC	HMC	79	Bay, Central, Vortex, Brimstone
2009	DD	HMC	49	Bay, Vortex, Brimstone
2010	RC	HMC	279	Bay, Vortex, Brimstone, Central, Crofoot Leach
2010	DD	HMC	93	Bay, Vortex, Brimstone, Central
2011	RC	HMC	184	Brimstone, Vortex, Central
2011	DD	HMC	100	Brimstone, Vortex, Central
2012	RC	HMC	235	Brimstone, Vortex, Central, Bay
2012	DD	HMC	97	Brimstone, Vortex, Central, Bay
2013	RC	HMC	158	Brimstone, Vortex, Central, Bay
2013	DD	HMC	42	Brimstone, Vortex, Central, Bay
2013	Sonic	HMC	40	Bay, Gap, Brimstone Leach
2013	Rotary	HMC	159	Well Field, Bay and Gap Dumps
2014	RC	HMC	258	Bay, Brimstone, Central
2018	Sonic	HMC	56	Sulfide stockpiles in Bay, Brimstone, Central and Crusher
Total			**5,576**	

Table 7-2: Exploration Drill holes by Type

Drill Type	Number
Diamond Drill	490
RC	4,760
Rotary	163
Blast	67
Sonic	96
Total	**5,576**
Angle	1,996
Vertical	3,580

Exploration by Duval, Homestake, HRDI and HMC has resulted in the discovery of multiple zones of mineralization associated with the Hycroft deposit. This discovery history is shown below in Table 7-3.

Table 7-3: Discovery Years of Hycroft Mineralized Zones

Deposit	Discovery Yr.	Hole No.	Company	Present Condition
Central	1977	Duval	Duval	Oxide Mining, Remaining Reserve and Resource
Bay	1981	SR-1	Homestake	Oxide Mining, Remaining Reserve and Resource
Camel	1981	SR-27	Homestake	Oxide Mining, Remaining Reserve and Resource
Boneyard	1986	86-230	HRDI	Oxide Mining, Mined Out
Brimstone	1986	86-256	HRDI	Oxide Mining, Remaining Reserve and Resource
Vortex	2008	H08D-3170	HMC	Remaining Reserve and Resource

7.1.1 Geologic Logging

A variety of geologic logging systems have been utilized during the more than 40-year exploration history of the Hycroft deposit. Vista reviewed drill logs from holes drilled during the period from 1986 to 1998 on the Brimstone Zone, which led to the conclusion that issues existed with continuity and consistency of logging observations. Vista geologists re-logged available drill chips and core.

In 2007, HMC further refined the logging system to provide more detail on the intensity, style and distribution of the geologic attributes. All of the previous logging, where possible, has been converted to the current HMC system. The current logging system is shown in Table 7-4.

Table 7-4: HMC Logging Code Fields

Log Form Identifier	Description
Hole	Drill hole name
From	Interval start footage
To	Interval ending footage
Formation	Formation code
Lithology	Lithology code
Malt_T	Main alteration type
Malt_S	Main alteration intensity
Malt_M	Main alteration mode
2Alt_T	Secondary alteration type
2Alt_S	Secondary alteration intensity
2Alt_M	Secondary alteration mode
Vein_T	Vein type
Vein_S	Vein intensity
Vein_M	Vein mode
Min_T	Mineral type
Min_S	Mineral intensity
Min_M	Mineral mode
Sulf_T	Sulfur type
Sulf_S	Sulfur intensity
Sulf_M	Sulfur mode
Ox_T	Oxidation type
Ox-S	Oxidation intensity
Ox-M	Oxidation mode
Struct	Structure
Texture	Rock texture or structure modifier

7.1.2 Surveying

Prior to HMC drilling, drill holes were surveyed in UTM coordinates and converted to NAD 27 state plane coordinates. In 2007, drill holes were surveyed in UTM NAD 83 and converted to mine grid coordinates. From late 2008 to present, mine surveyors located drill holes using accurate GPS equipment, reporting directly in mine grid coordinates.

7.1.2.1 Drill Collar Surveys

Standard operating procedure is for the mine surveyors to lay out planned exploration drill-hole locations by GPS. After drilling is completed on a site, the actual drill hole location is surveyed by the mine surveyors using GPS and the survey data entered into the acQuire database in mine grid coordinates.

7.1.2.2 Downhole Surveys

Downhole surveying of historical exploration holes was not carried out on a routine basis. During the 1999 drilling program, downhole, multi-shot, gyroscopic surveys were completed on several of the holes. Results of this work have not shown significant deviations in shallow holes (<1,000 ft) and indicate that the lack of downhole surveys in the historical exploration holes should not pose a problem. All downhole survey data, that is available, has been entered into the database. Current HMC practice is to contract survey the holes using gyroscopic instruments, which is the accepted industry practice. These instruments record orientation, deflection and temperature. From 2009 to present, all holes, were downhole surveyed if practical. Subsurface temperatures, greater than 140°F, and ground issues have prevented some holes from being surveyed to depth.

7.2 DRILL SAMPLE RECOVERY

Ground conditions at Hycroft, especially in the upper 600 feet of the deposit, present difficult drilling conditions for both reverse circulation and core. These ground conditions include acid leach alteration, highly fractured ground, voids, variable lithology and alteration, existing dump material and faults. As a result, reverse circulation and core recovery can be low in the upper portions of the deposit.

Modern day drill techniques and muds have increased recovery to an acceptable range of 80% to 100%, which in the QP's opinion is acceptable, with losses limited to highly fractured bedrock and unconsolidated dump material. All core is geotechnically logged, and recovery is generally in the 90-100% range, with areas of low recovery restricted to acid leach, unconsolidated dump, and highly fractured zones.

7.2.1 Reverse Circulation (RC) Recovery

The RC sample recovery was generally excellent as judged by both field observations and recovered material weights. The average weight of the material collected for a 5-foot sample in 2013 was 6.36 kg for rigs using a 5 ¼ to 5 ½ inch drill bit. Field observations and estimates indicated RC recoveries of 90-95%, and sample collection for analysis is on average 11.5% of that volume. These are consistent with previous year sample mass that averaged 6.26 kg for RC samples in 2012. During the 2013 program, approximately 26% of the RC intervals have no sample recovery, as a result of voids, unconsolidated dump material, highly fractured ground, and acid leach altered material. This number is higher than previous years, as second half drilling was concentrated specifically on unconsolidated dump material at Bay and Gap. In the QP's opinion, these factors do not materially impact the accuracy and reliability of the results, as un-assayed dump material is not included in the current reserve and resource estimate. Areas or holes with consistent recovery problems were re-drilled using core techniques. Average moisture content of the RC material was 20.6%, which is significantly higher than in situ moisture around 2-4% as a result of water injection during RC drilling.

A review of grade versus recovery was completed for a 10% random sampling of RC holes completed in 2012. No statistical bias between weight and grade was noted in this evaluation.

7.2.2 Core Recovery

Core recovery is measured by the ratio of length of material returned in the tube versus the total length drilled for the run and expressed as a percent. In 2013, core sample recovery was excellent and in excess of 94% of the bedrock cored, based on geotechnical logging. The average core sample size submitted to the laboratories was 5.04 kilograms or 1.25 kg/ft; however, this is a mix of PQ, HQ, and NQ core sizes, with both ½ and ¼ sawn core submitted. Core loss was generally attributed to acid leached material, alluvium, historical dumps and void spaces. During 2013, 6% of the intervals drilled returned no measurable core, due to void spaces and lost sample. In recovered core, recovery averaged over 95%. Overall core recovery is acceptable at Hycroft and in the QP's opinion, does not present a sampling issue. Average moisture content of the core was 2.5%.

7.3 SAMPLING METHOD AND APPROACH

Industry standard sampling of reverse circulation and core was utilized by HMC. Methods for both are discussed below.

7.3.1 Reverse Circulation Sampling Methods

Reverse circulation drilling was done with RC tools utilizing a crossover sub and wet sample collection in the upper portions of the hole. A center return tri-cone drill bit was used for intervals of ground water flow. The drillers cleaned the hole between rod changes and wait for a sample return before collecting assay samples. Drills utilized in 2013 included a Schramm 685, an Explorer buggy rig, and a track mounted rig.

Rock chips were collected continuously down the hole, with individual samples taken over 5-foot intervals. Samples were submitted for assay, as collected on the rig, with standards, blanks and duplicates inserted into the sample sequence as described in the section on Quality Assurance/Quality Control. The drill crews sequentially pre-numbered the bags, by drill hole identifier and the footage interval sampled. The driller's sampler only tracked the ending footage drilled with respect to the footage marked on the bags. The drill crews were provided with 20-slot chip trays, representing 100 feet total per tray, and numbered them with whole number, start and stop footage for the 5-foot interval. The Hycroft ore deposit is considered a disseminated ore deposit; therefore, 5-foot samples are appropriate to characterize the ore deposit.

Drill water injection was regulated to minimize the fluid return while maintaining sufficient flow for drilling and sample return. HMC geologists provided drill crews with 20-inch x 24-inch bags. Cuttings were collected as a continuous fraction of the return stream from the drill rig by way of a rotary 36-inch vane splitter. The splitter had vane covers that can be added or removed to provide the desired sample weight for each interval. The cuttings were diverted to a 5-gallon, plastic bucket lined with a labeled-sample bag. When the 5-foot sample run was complete, the bucket was removed, and another clean-lined bucket was placed under the spout. The previous sample bag was sealed and placed in drill order at the site.

During drilling, a strainer was placed under the waste discharge spout to collect chips for the character chip tray. At the end of each run, the drill sampler filled the character box slot for the sample interval and discarded the rest. The contents of the strainer were not introduced into the sample bags. When freezing temperatures were expected, the bags were placed on plastic sheets to prevent them from freezing to the ground and ripping when picked up. Sample bags were either dried and drained at the drill site or in a holding area near the sample processing facility.

Filled chip trays were field-checked for numbering accuracy during visits to the drill rig and collected by an HMC geologist for logging by use of a binocular microscope.

Samples were transported down to the shipment staging area where HMC personnel inserted extra bags for certified standards and blanks. Insertion of blanks and standards was handled independently by geologists who created duplicate numbers at appropriate intervals, post scripted with "S" for standard, "Q" for certified quartz blanks, and "B" for blank bulk material.

For HMC drilling between 2006 and 2014, reverse circulation samples were retrieved from the drill rig and stored in sample bins for pickup biweekly by the analytical lab. Before release of the sample bins to the assay lab, sample identification numbers and missing samples were verified at the exploration office located at the Hycroft Mine. The intervals of "no sample recovery" were identified, tagged, and accounted for separately in the sample lists so that the lab reported them as "no sample" rather than "0" or some other arbitrary value.

7.3.2 Core Drilling Sampling Methods

Core drills used in 2013 included CT-14 and LF-100 models, utilizing a wire line retrieval system and 5-foot stroke rod advancing systems. All drills are capable of drilling PQ and HQ sized core and reducing to NQ if required.

Core drillers were responsible for obtaining a complete and representative sample of the cored interval in runs not to exceed 10 feet, with shorter increments completed in difficult conditions. Coring was generally begun with large diameter (PQ) rods in the more broken upper zones (0 ft to 600 ft) and reduced to HQ at depth. Ground conditions and drill problems could result in further reduction to NQ core. Core was recovered from the barrel by using a wire line core tube.

At the drill site, the crews placed the core in cardboard core boxes, with tops and bottoms labeled with company, property, hole identification number, box number and starting and ending depths. The bottom of the core box was laid out length wise from left to right, with the marked or labeled end to the left and the unlabeled end to the right. The first portion of core was placed in the upper left-hand tray and continuously laid in the tray from left to right, advancing down one row as each tray is completed. The bottom of the core was terminated in the lower right corner. A wooden block was inserted at the end of each run, as well as in locations deemed important by the drillers to note adverse conditions such as caving, voids, or mislatches, where the core tube failed to seat properly in the core barrel. The ending block for the run was marked with an ending footage on the thin edge and both the cut footage and recovered footage were marked on the larger surface.

Depending on ground conditions, contract drillers used either a 5-foot or 10-foot core barrel to collect samples. After the core was logged, it was the geologist's responsibility to determine appropriate sample intervals and boundaries. Sample intervals were representative of the mineralization at Hycroft and are generally less than or equal to 5 feet, unless low recovery zones prevent accurate determination of 5-foot sample lengths. Original core blocks were used by core drillers to mark the end of a cored run and ordinarily served as the primary sample boundary, subject to the rules below. Where a conflict existed between the blocks and those rules, the rules prevail, and extra blocks or metal tags labeled with the depth were inserted by the geologist to indicate sample intervals.

- A sample must not cross a lithologic boundary.
- A sample must not cross an obvious alteration boundary, including oxidation.
- A sample must not generally exceed 7 feet in length, unless combination of drill run and recovery prevent accurate determination of footage.
- Distinct vein zones are sampled separately.

Any core blocks that do not mark a sample boundary, for reason such as 'cave', 'loss', 'void', etc., must be labeled in black magic marker for photographic visibility. Cave zones and refill footage are not sampled for assay purposes.

For diamond drill core, geologists tagged sample intervals and provided sample prep technicians with a list containing the drill hole number and sample intervals. Samples were then saw cut in equal halves by HMC personnel at the Hycroft Mine core facility. Intervals with visible silver mineralization (less than 5%) were sent un-sawn to the laboratories to reduce the risk for sample bias. In 2010, some uncut metallurgical core was delivered to ALS Minerals in Reno, to provide ¾ sawn core for metallurgical testing. After cutting, the ½ sawn core was placed in bags, with sample IDs for tracking. As with the RC cuttings, intervals of "no sample recovery" were identified, tagged, and accounted for separately in the sample lists so that the lab reported them as "no sample" rather than "0" or some other arbitrary value.

The sampling operation avoided bias, to the extent possible, by cutting the core in half perpendicular to the trace of the visible bedding. When prominent veins were noted during logging, the geologist marked the trace of the cut to ensure a representative sample. The portion to be saved was placed in the core box, in its proper position, with core blocks in place. Core boxes were stacked on pallets for storage. The split portion of core was bagged and shipped in bins to the lab.

7.3.3 Sonic Drilling Sampling Methods

Rotosonic (sonic) was the best drilling method to recover representative samples in unconsolidated material with variable consistency and particle size. Sonic drilling in 2018 was completed by Major Drilling, based in Salt Lake City, Utah. Core diameter was nominally 4 inches. A minor proportion of the drilling required casing; in holes with casing, an oversized 6-inch bit was used to advance the hole. The drilling and sampling techniques used for the sonic drilling program are different from the typical procedures used for reverse circulation or wireline diamond core drilling. The procedures described below pertain only to the 2018 sonic drilling program.

Holes were advanced in 10-foot runs. At the end of each run, the drill steel was tripped to surface to collect the sample in the core barrel, at the bottom of the string of drill steel segments. A tube of polypropylene film knotted on the end was placed on the end of the core barrel. The core barrel was tilted about 20 degrees from vertical and the sample material flowed from the core barrel to the bag. When all material was in the bag, the plastic was trimmed and knotted on the top end. Each 10 ft interval was labeled and stored at the rig until the end of each shift. While at the drill pad, samples were under supervision by the drilling crew and Hycroft staff.

At the end of each shift, samples were transported to the Hycroft core shed, which was locked when not occupied. The sample tube bags were placed on cardboard PQ core boxes without dividers. Depths were measured from the tube bags and noted on the boxes. Approximate recovery was noted. The plastic film was cut away, to leave the sample material in the boxes.

A polypropylene sample bag was labeled with a serial sample ID, determined from a list of drill intervals and reference samples. Hycroft staff used a clean sample scoop to collect approximately 25% of the cuttings from the core box to the assay sample bag. Typical sample weight was between three and four pounds. The scoop allowed sampling of variable particle sizes and material consistency. Sample reduction would have been impractical with a riffle splitter, due to the variation in particle size and presence of clay. Between 10 ft sample intervals, the scoop was wiped with a clean towel to avoid cross contamination. Sample intervals matched the drill intervals and were nominally 10 ft unless the drill run was less than 10 ft, at the end of a hole. Samples were stored in the secured core shed prior to transportation to the lab.

7.4 SAMPLE QUALITY

Sample quality on RC and core rigs was assured by daily inspections of rigs during operating hours. Samples were inspected for correct labeling, size (10 lb. to 20 lb. target on reverse circulation) and condition. Core boxes were inspected both on the site and during logging for labeling, position and recovery. Zones of low recovery were noted on drill logs and on core blocks. Recovery and sample quality were also assessed using lab weights compared to expected weights. Industry standard use of muds and drill techniques were applied to ensure as good of quality sample from each drill hole interval.

7.5 SAMPLE LOCATION

Sample location is tied to both the collar and downhole survey, along with the hole ID and downhole footage. All of these items were reviewed daily by Hycroft geologists.

7.6 DOWNHOLE SURVEYS

Down hole surveying was conducted by IDS of Nevada. Gyroscopic techniques were used to locate drill hole deviations and are accepted as an industry standard of data quality. Most historic drilling was not down hole surveyed. The downhole survey results are downloaded directly into the acQuire database.

7.7 FINAL COLLAR SURVEYS

Upon drill hole completion, the Hycroft Mine surveyor located the collar coordinates of drill holes using an accurate GPS device and reports data in the mine grid coordinate system. The collar coordinates were saved to the acQuire database.

7.8 GEOTECHNICAL ROCK MASS CHARACTERIZATION

The geotechnical quality of the Hycroft rock mass has been characterized by Call & Nicholas Inc. (CNI, 2011) and Golder Associates, through analysis of surface and drill-hole information. The characterization program was designed to collect data necessary to predict the stability of planned pits. The geotechnical data collection, analysis and design were primarily conducted in 2010 and 2011. Further review of design was completed in 2013 and 2014. In 2016 the geotechnical parameters were reviewed by Golder Associates.

Geomechanical properties have been established through analysis of parameter data from both surface and drill-hole sampling programs.

Sixty-one surface structure cells were mapped in the current pit walls and rock outcrops in early June 2010. This data was used to determine statistical distributions of joint orientations, joint length and joint spacing.

Rock Quality Data ("RQD") data has also been collected by HMC on most core holes drilled in the deposit, starting in 2007. Additionally, eighteen core holes were drilled to collect geomechanical parameters and structure orientations to answer specific stability questions. These holes were drilled and logged in the 4th quarter of 2010 and 2011.

RQD zoning was accomplished near the sections used in the overall stability analyses. The RQD length-weighted average and standard deviation have been calculated for each geomechanical zone.

Laboratory testing of samples has been conducted to determine rock strength. This testing has been completed in the CNI laboratory, according to ANSI procedures.

The rock strength test program included the following:

- Twelve small-scale direct-shear ("SSDS") tests for soil like samples of argillic Tsg, Tcm and fault gouge.
- Twelve sieve analyses, hydrometers and Atterberg limit tests to determine the classification of the soil-like and fault gouge samples.
- Nine SSDS tests were conducted on rock joint surfaces for the ALS unit, and for silicified and propylitically altered rocks.
- Twenty-seven uniaxial compression tests on the various rock and alteration types recognized at Hycroft. Fourteen of these samples were instrumented with strain gauges to determine the elastic properties of the intact rock.
- Seventeen triaxial compression tests were performed on the various rock and alteration types.
- Eighteen small-scale direct-shear (SSDS) tests were conducted for the unconsolidated Camel Conglomerate (Tcm). Sieve and hydrometer analyses and Atterberg limit tests were performed to characterize the Tcm and to determine correlations between logged core parameters and the shear strength of the Tcm.

Intact shear strength has been estimated from the uniaxial compression, triaxial compression, and Brazilian Disk Tension test (Table 7-5).

Table 7-5: Hycroft Intact Shear Strength by Rock Type

Rock Type	Density (pcf)	Uniaxial Compressive Strength		Triaxial Compressive Strength				Hoek's m i	Poisson's Ratio	Young's Modulus (psi)
				Cohesion		Friction Angle				
		Mean (psi)	Standard Deviation (psi)	Mean (psi)	Standard Deviation (psi)	Mean (deg)	Standard Deviation (deg)			
Silicified Tuffs and Breccias	155	22010.60	1972.76	3526.56	316.07	54.5	4.7	27.9	0.17	4.83E+06
Propylitic Volcanics.	149	8337.00	1903.56	1722.25	393.24	45.1	10.3	15	0.19	3.90E+06
Argillic Volcanics near East Fault Zone *	136.4	1107.7	---------	232.9	---------	37	---------	---------	---------	---------

* This value was estimated from very limited testing of argillically altered volcanic rock in the footwall of the East Fault Zone during a 1997 study to evaluate failures associated with the fault zone.

Rock-mass shear strength has been established through analysis of:

- Intact block size (related to RQD, fracture frequency, and number of joint sets)
- Intact rock shear strength
- Fracture shear strength
- Fracture orientations (determines the number of degrees of freedom of movement)

The calculated rock mass strength values are shown in Table 7-6.

Table 7-6: Rock Mass Strengths Used in the Stability Analyses

Rock Type	Alteration Type	Density (pcf)	Intact Rock Strength		Fracture Strength		70% Reliable RQD (percent)	Rock Mass Strength	
			Φi (degrees)	ci (psi)	Φf (degrees)	cf (psi)		Φrm (degrees)	crm (psi)
Kamma Mtn 1	Propylitic	149	45.10	1722.25	26.51	6.80	64	34.50	162.90
Kamma Mtn 2	Anisotropic	149	45.10	1722.25	26.51	6.80	NA	27.55	92.60
Volcanic Bxa 3	Argillic	136.4	37.00	232.90	24.00	6.60	37.9	26.70	12.20
Silicified Vol. Bxa 4	Silicified	155	54.50	3526.56	30.48	7.07	69	42.50	377.50
ALS 4	Silicified	155	54.50	3526.56	30.48	7.07	69	42.50	377.50
ALS 5	Propylitic/ Argillic	149	45.10	1722.30	22.85	3.41	15	27.70	46.20
ALS 6	Bedding	149			22.85	3.41	NA	22.85	3.41

Comments:
1. crf = 0.35
2. For Steep Back Plane >50 degree, Fault Length to Spacing to come up with %intact based on TMR L=719 S=146, %intact = 5.5 Use 5%
3. Avg Tensile 204 psi, used Myer corrected phi of 37 from 1997 study of H2 volc. Bxa, and backed into uniax strength of 1107.7 psi. Seems OK. Crf = 0.5; the fracture shear strength used was the average of all Tcm tests (this is a bit different rock near the East Fault, but should be reasonable)
4. crf = 0.35, 1- shear 3901-1276, use 80% Rel. RQD to account for stress Damage
5. Oblique to Bedding in Argillic and Propylitic ALS. GT 20 degree dip. Use Propylitic Kamma Mtn, with ALS bedding fracture strength and RQD of 15 percent
6. Anisotropy for bedding dips less than 20 degrees toward pit

7.9 HYDROGEOLOGY

The hydrogeology of the Hycroft Mine and local area has been evaluated by SRK, in collaboration with HMC, through execution of a data collection program from 2010 through 2012 (SRK, 2011; SRK, 2013). Overall the Hycroft Mine area presents a complex hydrogeologic regime that includes fault associated fluid barriers, high temperature groundwater and the presence of H_2S gas. The three-year hydrogeologic data collection program utilized groundwater wells and piezometers, core hole hydraulic testing, and short and long-term aquifer testing to characterize the local groundwater system, with project specific approaches and equipment necessary at times to improve overall data quality.

Forty wells and piezometers have been installed in the Hycroft district to monitor groundwater levels and to measure hydraulic properties of the water bearing geologic formations. An additional 18 monitoring wells were installed in the basin approximately three miles west of the mine for water supply exploration. Groundwater elevation data were collected from wells within and surrounding the Hycroft Mine to define the potentiometric surface. In general, the groundwater gradient through the Hycroft Mine is primarily horizontal, east to west, flowing from the volcanic highlands in the east and discharged in the alluvial basin beneath the Black Rock Desert to the west at an average gradient of 2%. The depth to groundwater across most of the mine area is within about 700 ft of the ground surface. Range front structures associated with the Kamma Mountains define compartmentalization of the local groundwater system across the resource areas of the Hycroft Mine. This is demonstrated in part by the weak barrier imparted by the East Fault on the east side of the mine area. The potentiometric data in this area suggests an elevation difference of 360 ft across the East Fault, with higher groundwater elevations present in the east (footwall) side of the fault. Vertical flow is less prominent, however a comparison of potentiometric between shallow completions (<800 ft) and deep completions (>800 ft), indicates a small downward vertical gradient. Vertical groundwater flow also occurs, primarily within high-transmissivity faults (in particular the Albert, Break, Central, and Range faults), and primarily downward, although high heat and gas flow suggests that local upwelling may also occur locally (SRK, 2014).

Hydraulic properties of geologic units present in the Hycroft district have been estimated through testing of monitoring wells and piezometers, with faults tested through packer-isolated aquifer testing in numerous core holes. While extensive testing was completed, given the potential variability of hydrogeologic parameters within the faulted volcanic terrain, and complexity in testing a high temperature, gas-filled aquifer, the resultant characterization data should be considered indicative, but not absolute.

Hydrogeologic units for groundwater modeling have been established based on the test data. The Hycroft Mine 3-D geologic model has been utilized to apply the test results across the mine area. Seven primary hydrogeologic units and nine faults comprise the groundwater hydrogeology:

- Qal – Quaternary alluvial and colluvial material consisting of high clay content gravels and sediments displaying moderate to relatively high permeability. These sediments generally lay to the north, west and south of the mine area, and includes the coarse to fine alluvial fan facies within the Black Rock Desert.
- Qcl – Tertiary lacustrine sediments of the Black Rock Desert having low permeability.
- Tsg – High clay content lacustrine sediments of low permeability, locally inter-bedded with thin lenses of coarse gravels having moderate to high permeability.
- Tcm – Matrix supported water-lain/worked conglomerates derived from volcanics.
- Tk – Weakly to moderately altered volcanic rocks of intermediate permeability.
- Rhyolite – A deep component of the Tk sequence; probably intrusive, permeability not currently known.
- ALS – Jurassic age metasedimentary basement rocks, assumed to be of low permeability.
- Faults/Structures - Variably altered fault zones of low to high permeability cutting all other units, including Albert, Break, Camel, Central, Cliff, East, Prill, Ramp, and Range Faults.

The hydrogeologic units and structures tested, along with corresponding properties from testing and values utilized in the numeric groundwater flow model, are listed in Table 7-7.

Table 7-7: Summary of Hydraulic Conductivity (K) Values Measured in Field and Used in Model by Hydrogeologic Unit

Hydrogeologic Unit	Number of Tests	Geomean K value (ft/day)	Average K value (ft/day)	Max K value (ft/day)	Min K value (ft/day)	K value Used/Calibrated in Numerical Model (ft/day)
Alluvium	4	1.3E-01	4.7E-01	1.6E+00	1.7E-02	1.3E-01
Fan Coarse	9	2.8E+01	4.8E+02	1.4E+02	1.0E+00	4.05E+01
Fan Transition						1.44E+01
Fan Fine Facies						3.4E+00
Quaternary lakebed sediments (Qcl) (Kh/Kv)	3	1.4E-03	6.9E-03	2.0E-02	2.1E-04	0.02/0.002
Camel conglomerate (Tcm) unaltered	1	4.1E-03	4.1E-03	4.1E-03	4.1E-03	4.1E-03
Camel conglomerate (Tcm) altered in mine area	3	2.2E-02	2.5E-01	7.3E-01	8.7E-04	8.0E-02
Tuffaceous lacustrine sediments (Tsg) in North Central Valley	3	3.9E-03	8.8E-03	2.3E-02	1.3E-03	3.9E-03
Kamma Mountain volcanic rocks (Tk) altered (mine area)	5	3.2E-02	8.3E-02	1.9E-01	6.5E-04	3.2E-02
Kamma Mountain volcanic rocks (Tk) unaltered (Hades/Central Kammas)	8	4.4E-03	1.6E-02	7.0E-02	5.5E-04	5.2E-03/3.0E-03
Albert Fault	1	1.16E-01	1.16E-01	1.16E-01	1.16E-01	7E-01
East Fault	8	1.5E-01	1.0E-01	4.0E-01	1.0E-01	2.0E-03 (Kx) 3.0E-01 (Ky)
Central Fault	3	3.1E+00	5.25E+00	9.1E+00	5.5E+00	3.1E+00
Break Fault	4	8.2E-02	3.7E-01	1.3E+00	2.4E-02	2.4E-01

The Black Rock Desert basin is west of the mine and is composed of low-permeability alluvial and lacustrine sediments and the sediments in the margins of the basin are composed of higher permeability alluvium. A large alluvial fan emanates from the Rabbithole Creek and Granite Springs Wash drainages and extends approximately 15 miles north into the basin. The freshwater well field, composed of three active and seven inactive production wells, is located on this alluvial fan in coarse-grained alluvial deposits of sand and gravel that extend from the location of the existing production wells PW-2/PW-3 to the south up into the apex of the fan.

Hydrostratigraphic information from a geophysical investigation and the installed monitoring wells, combined with the aquifer testing data, suggests that the local basin aquifer is sufficiently thick to support large-scale production wells pumping up to 1,000 to 3,000 gpm. Eight production wells were drilled and cased in 2013. The production wells range between 12 and 24-inch diameter casing, have screens approximately 300 to 600 feet below land surface, and have varying pumping capacities between 300 and 3,200 gpm (HDR, 2013).

Precipitation is estimated to be 7.7 inches per year based on an average of historical records from local weather stations in Imlay, Gerlach, Lovelock, and Rye Patch. The Maxey-Eakin recharge method was used to estimate recharge from precipitation with a relationship derived between elevation and precipitation. Groundwater recharge has been estimated to be 0.088 to 0.924 inches per year, depending upon elevation (1-7% of measured precipitation). The maximum possible evapotranspiration rate has been assumed to be 59 inches per year, based on average data from the nearby Rye Patch and Imlay weather stations, with an extinction depth of 10 feet below ground surface.

SRK developed a preliminary 3-D numerical groundwater model of the project area to evaluate inflow to the proposed ultimate pit and potential dewatering requirements for mine expansion (SRK, 2011). This model is based on all available geologic, hydrologic and hydrogeological data. The groundwater model is a geologically based, fully representational model utilizing the commercially available finite-difference code Visual MODFLOW-SURFACT (SWS, 2010 and Hydrogeologic, 2006). The groundwater flow model of the Hycroft project area was constructed by:

- Incorporating 3-D stratigraphical and structural geological models.
- Assigning estimated hydraulic parameters of major hydrogeologic units, faults and outer boundary conditions.
- Simulation of recharge from precipitation, plus evapotranspiration.
- Calibrating the model to measured water levels during steady state (with mining above water table) conditions and evaluating the groundwater budget.

This 3-D numerical groundwater model was twice updated by SRK in 2014 and 2019 (SRK, 2014 and 2019). The groundwater model was expanded and re-calibrated in 2014 and only dewatering predictions were updated in 2019. The latest version of the model was transferred into Groundwater Vistas (ESI, 2017).

This model provided the basis for estimation of passive inflow to the proposed pit and design of a potential active dewatering system. The model is based on the current available hydrogeologic data collected through the various field investigation programs at the Hycroft Mine. Ongoing evaluation of the hydrogeologic system will be required as the mine advances to better educate the groundwater model, as well as refine inflow predictions to allow for dewatering system optimization and target/strategy adjustment.

8 SAMPLE PREPARATION, ANALYSES AND SECURITY

Hycroft drill hole samples were shipped to accredited, independent laboratories in Reno or Elko, Nevada, for sample preparation and analysis. Sample security and handling procedures were not investigated in detail by the QP, because they have been reported in previous technical documents. In the QP's opinion, the sample handling, preparation, and analysis methods meet current industry standards for quality, and pose little risk to the quality of Hycroft's analytical data.

Sample preparation and analysis techniques are listed below, by laboratory. These techniques were used at different times during HMC development of Hycroft. Sample preparation and analysis methods used for Hycroft samples at ALS and INS are comparable.

- **ALS Chemex (ALS)**
  - Preparation codes CRU-21, CRU-31, SPL-22Y, PUL-32: Two-phase crushing, rotary split 1000 g, pulverize to 85% < 75 µm (200 mesh).
  - Fire Assay with Atomic Absorption finish, determine total gold and silver
  - Hot Cyanide Leach to determine cyanide-soluble gold and silver
- **BVI Inspectorate/ Acme (INS)**
  - Preparation code SP-RX-2K: Crush, split, pulverize 250g to 200 mesh (< 74 µm).
  - Fire Assay with Atomic Absorption finish, determine total gold and silver
  - Hot Cyanide Leach to determine cyanide-soluble gold and silver
  - Composite pulp samples for sulfur analysis, method TC009, total sulfur minus sulfate sulfur
- **McClelland Laboratories, Inc. (MLI)**
  - Preparation code STD-PREP: Crush, split, pulverize 250g to >90% passing 150 mesh (0.1 mm).
  - Fire Assay with Atomic Absorption finish, determine total gold and silver
  - Cyanide Shake Leach to determine cyanide-soluble gold and silver
  - Mercury from cold vapor Atomic Absorption finish
  - Total sulfur and sulfide sulfur determination with LECO-type furnace
- **American Assay Labs (AAL)**
  - Generated composite samples from prepared pulps, generally 25-foot lengths
  - Sulfur and carbon analysis, method ELTRA-S
  - Multi-Element ICP with aqua regia digest

Method detection limits vary between labs for similar determinations, but are comparable for all but total silver. Analytical methods and detection limits for total silver varied through the history of the Project, and total silver was not commonly reported until recently. The total and cyanide-soluble gold, cyanide-soluble silver, and sulfur results from different laboratories are appropriate to consider together for grade estimation.

8.1 SAMPLE PREPARATION

The sample preparation procedure prior to 1999 was not documented. Sample preparation in 1999 consisted of drying, crushing, splitting and pulverizing the split.

A transmittal sheet for both the bagged core and RC samples by drill hole was prepared for submission to the laboratory. Once at the laboratory, samples were prepared from a split of 70% passing minus 3 mesh if pieces were too large to fit in the pulverizer, and further crushing of 70% passing minus 10 mesh. A 2.2-pound split is taken and pulverized to 85% passing minus 200 mesh.

No officers, directors, or associates of the issuer were operationally involved with the routine sample preparation.

8.2 ASSAY METHOD

Prior to 1992, most samples were sent to Barringer Laboratories Inc. in Golden, Colorado, for fire assay and selected intervals were cyanide soluble analyzed. From 1992 to 1999, samples were processed at the Hycroft laboratory. After 1999, samples were sent to outside laboratories for processing.

From 1992 to 1999, all of the samples were analyzed for cyanide soluble gold and silver at the Hycroft laboratory. The method employed at Hycroft was a non-standard procedure that has been developed to provide a semi-quantitative measurement of recoverable gold and silver.

HMC used four independent laboratories for assay analysis. The companies are ALS Minerals, American Assay Laboratories, Inspectorate, and McClelland Laboratories, Inc., all located in Sparks/Reno, Nevada. ALS Minerals is ISO9001:2000 compliant and has ISO17025 accreditation. American Assay Laboratories participates in the following accreditations: certificate of ISO/IEC17025, certificate of laboratory proficiency PTP-MAL, accredited by standards council of Canada, geostats of Australia certificate, Society of Mineral Analysts (USA) – round robin testing. Inspectorate has ISO9001:2008 certification. McClelland has ISO/IEC17025 certification. During 2012, the Hycroft Mine Laboratory completed gold and silver fire and gold and silver cyanide assays on 10 drill holes. The Hycroft lab was not certified at this time, but a comprehensive testing of the lab results was completed to verify reported analysis. This testing was completed by ALS Minerals, based in Sparks, NV, and verified the HRDI lab results during this period. There was no further use of the HRDI lab for exploration samples in 2013 or later programs. Assay methods used at the various laboratories are summarized in Table 8-1.

Table 8-1: Analytical Methods

Element	Method Name	Description	Level of Detection	Laboratory
Gold	AuAA23	Fire assay with AA finish	0.005ppm	ALS
Gold	AuGRA21	Fire with gravimetric finish	0.05ppm	ALS
Silver	AgGRA21	Fire with gravimetric finish	5.00ppm	ALS
Gold	AuAA13	Hot cyanide, AA finish	0.03ppm	ALS
Silver	AgAA13	Hot cyanide, AA finish	0.03ppm	ALS
Gold	FA1AT	1 assay-ton fire, AA finish	0.100ppm / 0.103ppm	Inspectorate
Gold	Au30CN	Cold Cyanide, AA finish	0.03ppm	Inspectorate
Gold	2-FA-11	Fire with gravimetric finish	0.003opt	Inspectorate
Silver	Ag-AR-TR	Aqua Regia digestion, AA finish	0.1 ppm	Inspectorate
Silver	FA/GRAV	Fire with gravimetric finish	3.40ppm / 5.00ppm	Inspectorate
Silver	Ag30CN	Cold Cyanide, AA finish	0.03ppm	Inspectorate
Gold	FA30	Fire Assay, 30-gram charge	0.003ppm	American Assay
Gold	AuCN	Cyanide with AA finish	0.03ppm	American Assay
Silver	GRAV	Gravimetric finish	7.00ppm	American Assay
Silver	D2A	2 acid (aqua regia) digest	0.200ppm	American Assay
Silver	AgCN	Hot Cyanide with AA finish	0.03ppm	American Assay
Gold	FA-AAAu	Fire assay with AA Finish	0.001 opt	McClelland
Silver	4 ACID-AA-Ag	Four acid digestion, AA finish	0.010 opt	McClelland
Gold	FA-30-Au	1 assay-ton fire, AA finish	0.0003 opt	McClelland
Silver	FA-30-Ag	30g fire assay for Ag with AA finish	0.01 opt	McClelland
Gold	CN.SHAKE-Au	10g sample, 1-hour ambient temp agitated leach with AA finish	0.0003 opt	McClelland
Silver	CN.SHAKE-Ag		0.0003 opt	McClelland
ICP	ICP-24	24 element ICP	Variable by element	Florin
Mercury	Hg-FIMS	Mercury direct by FIMS	0.010 ppm	McClelland
ICP	MEMS41	41 element ICP	Variable by element	ALS
ICP	ICP-2D	36 element ICP	Variable by element	American Assay
Sulfur	SR-I07	Sulfide Values by LECO	0.01%	ALS, McClelland
Carbon, Sulfur	Eltra	Carbon and Sulfur values by LECO	0.01%	American Assay

8.2.1 Precious Metal Fire Assay Analysis

All HMC drill samples were fire assayed using either a gravimetric or Atomic Absorption (AA) finish for gold and silver. Earlier operators, however, primarily fire assayed only for gold.

In the second half of 2013 and after, industry standard aqua regia digestion was used for total silver assays. This method replaced the gravimetric silver analysis completed previously. Rationale is a lower detection limit on the aqua regia results to 0.1 ppm, compared with 5 ppm for gravimetric analysis.

8.2.2 Cyanide Soluble Precious Metal Analysis

Industry standard cyanide soluble gold and silver analysis was completed on all HMC drill samples that returned greater than 0.100 ppm Au. The hot cyanide analytical procedure, originally developed by the Hycroft lab and utilized on most pre-2013 drill intervals, was discontinued.

8.2.3 ICP Multi-Element and LECO Sulfur Analysis

All drilling by HMC in 2007 and 2008 was assayed using a 35-element, total digestion, multi-element method, and generating 56,327 individual ICP assays. As this program provided a broad distribution of trace element data, only a limited number of samples were assayed by ICP in 2010. In addition, a limited number of sample intervals were assayed for trace elements during historical drill campaigns. In 2011, 93 holes were selected for ICP and LECO analysis at American Assay. These holes were selected to provide an approximate 500 by 200-foot spaced grid over the current reserve pit, for both ICP and LECO analysis. A sub-set of drill holes from 2012 and 2014 were also selected for multi-element ICP and LECO sulfur analysis. Sulfide sulfur analysis results that tie into production at Hycroft have been used to estimate values in the block model.

8.3 SAMPLE SECURITY

Samples were delivered to the analytical laboratory in the numbered bags, along with a transmittal sheet with the list of sample numbers, the total sample count, and codes for sample type, either RC cuttings or drill core. The lab has no knowledge of the spatial reference of the individual samples, beyond knowing the footage of a particular hole.

A variety of certified standards were submitted with each drill hole, ranging from 0.2 ppm to 2.2 ppm gold. Generally, one blank and one standard per 40 samples was submitted to the lab and checked against known values prior to database loading. In addition, a sample of unmineralized rock (marble, granite, or scoriaceous lava) was submitted as a blank and is the first sample in each drill hole group of samples, to identify potential laboratory contamination.

Core samples sawn on site were picked up by ALS Minerals or Inspectorate drivers for delivery to their facilities in Reno and Elko. American Assay Lab had been contracted by HMC to do independent assay verification on existing pulps and LECO/ICP analysis on prepared pulps. These pulps are stored at the Hycroft Mine site, and randomly selected sets were picked up on site by American Assay Lab for analysis.

Chain of custody was established by transmittal sheets, sample receipt documents from the lab and by work orders and certificates.

An HMC copy of the transmittal sheet was stored at the Hycroft Mine, along with a digital copy on the server. Once assays have been received, a copy of the assay certificate sheet is stored with the drill logs, and the original with the transmittal sheets and a digital copy is kept on the server. The transmittal sheets are indexed by job number.

Copies of the sample sequence list, the lithology log and assays are stored in paper format at the Hycroft Mine. Digital copies of all material are kept on a dedicated, backed-up server and all data is ultimately stored in the acQuire database, located on an independent backed up server. Originals of all logs and assays are stored in file cabinets on a per hole basis, indexed by hole number. HMC personnel contact the lab to obtain a job number assignment for whole or partial hole shipment and arrange for sample pickup by the lab's driver.

Coarse reject material and sample pulps were returned to the Hycroft Mine by laboratory staff and stored on site. Access to the sample preparation and storage area is limited to geologic staff.

8.4 ANALYTICAL RESULTS

Following analysis, results were posted to a digital laboratory database on which HMC had secure permission privileges. Managers downloaded the data where the sample results were cross referenced to sample numbers. Each drill hole carries a unique self-identifying sample number, simplifying the cross-referencing. The completed digital file for each drill hole was emailed to HMC by the lab and a follow-up, hard copy certificate is mailed to company offices.

Results were checked by geologists visually and loaded into the secure acQuire database. The acQuire database is further checked using electronic methods and then calculated into ounce per ton (OPT) values from values reported in parts per million and loaded to the modeling database for display and further visual QA/QC checking. During SRK's data verification, the group considered the impact of using assay values in ounce per ton units with lower precision than the reported values in parts per million. Assays in PPM were compiled to use for estimation, and historical assays reported in OPT were converted to PPM. This dataset allowed lower values to apply to estimation, to match the calculated cutoff grade that was close to the lower detection limit of pre-2006 gold fire assay results.

8.5 QUALITY ASSURANCE (QA) & QUALITY CONTROL (QC) CHECK SAMPLE AND CHECK ASSAYS

8.5.1 Historical QA/QC Program

After 1991, exploration samples were assayed for cyanide soluble gold and cyanide soluble silver at the Hycroft laboratory. Fire assays were also performed, however, no decipherable QA/QC data exists for these assays.

8.5.2 HMC QA/QC Program

The HMC QA/QC program included analysis of standard reference materials (standards), blanks, and duplicate pulps, as well as check assays by umpire laboratories. The program was designed with the intent that at least one standard and one blank were inserted into the drill sample stream for every 40 drill samples (200 ft), which is the number of HMC samples in each ALS Chemex analytical batch. In practice, the insertion rates for the QA/QC samples were somewhat higher, based on drill hole depth.

8.5.2.1 Certified Standards

Reference samples were used to evaluate the analytical accuracy and precision of the assay laboratory during the time the samples were analyzed.

HMC utilized certified reference standards from Minerals Exploration and Environmental Geochemistry (MEG) of Reno, Nevada, in the 2015-16 Demonstration Plant program. Table 8-2 below lists these standards. The Standard Certified Value is the MEAN value from all test labs. The lower and upper cutoff limits represent a 95% confidence limit applied to the upper and lower limits of the range of values from all test labs.

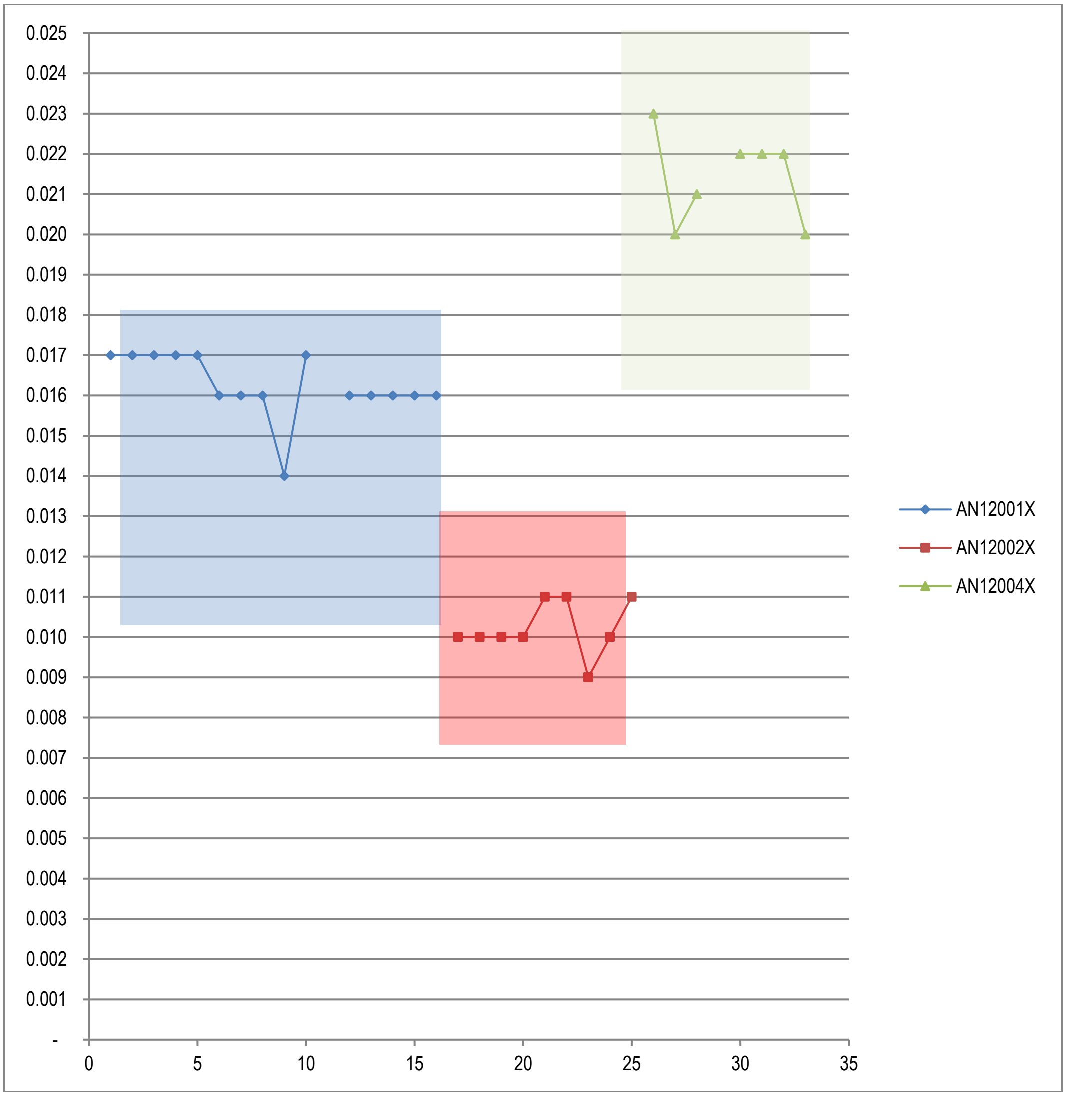
Table 8-2: Certified Standards

Standard	Standard Source	Certified Value (ppm)	Standard Deviation	Lower Cut-off Limit (LCL)	Upper Cut-off Limit (UCL)
AN12001X	MEG	0.014	0.002	0.010	0.018
AN12002X	MEG	0.010	0.001	0.007	0.013
AN12004X	MEG	0.020	0.002	0.016	0.025
S107009X	MEG	0.137	0.016	0.104	0.170
S107010X	MEG	0.187	0.009	0.169	0.204
S107011X	MEG	0.271	0.013	0.245	0.295

Source: Hycroft, 2016

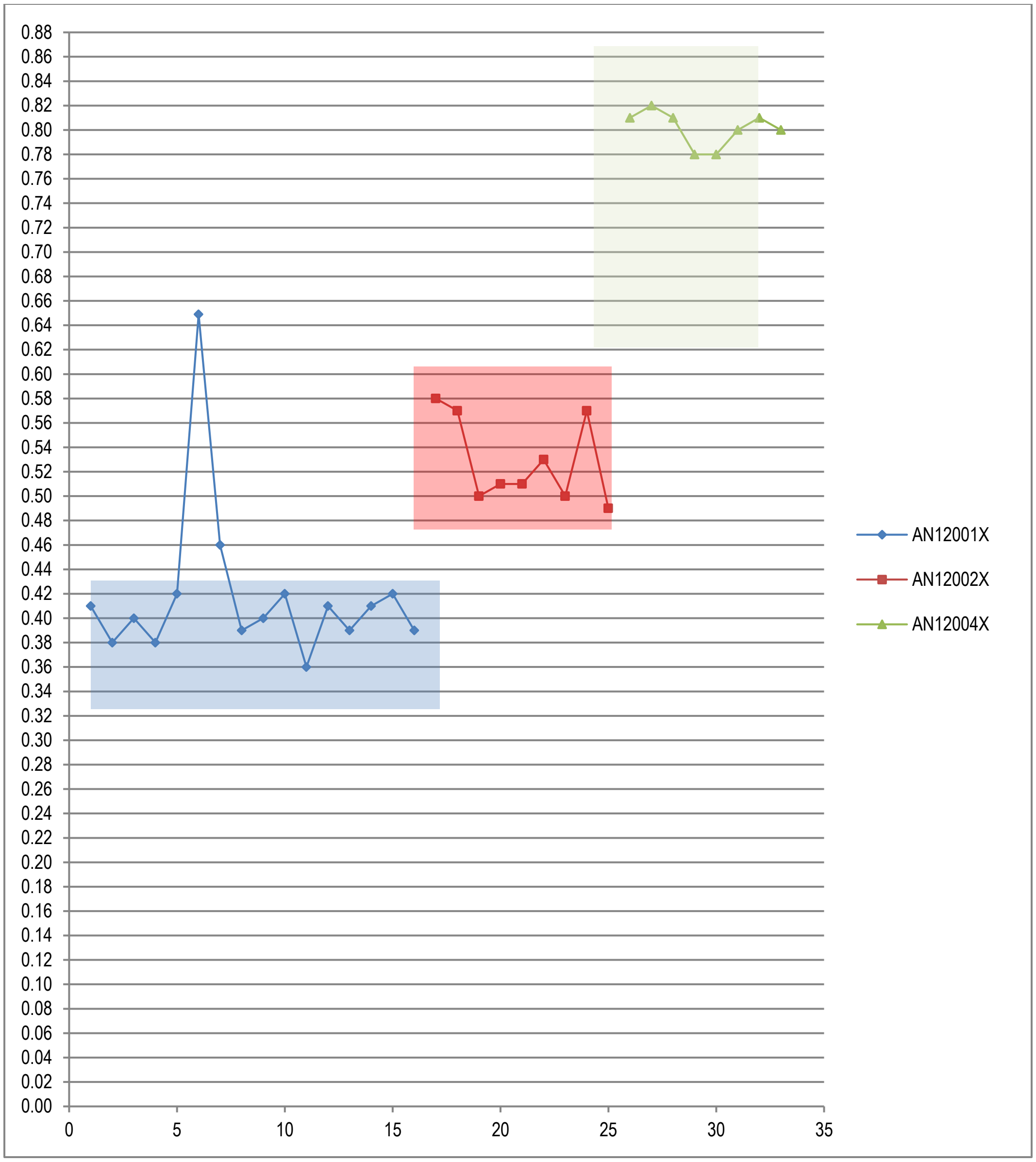
The standards were assigned sample numbers at random and inserted into the demo plant sample stream. Hycroft compiled 52 analyses of these standards and 6 analyses of bulk blanks. This equates to an insertion rate of approximately one standard or blank for day of plant operation. McClelland Laboratories completed all analysis during 2016.

HMC used six standards with known gold values, four standards with known silver values, one blank standard, and one coarse blank in QA/QC. Performance of the standards was tracked for 2016 by standards and blanks, both over time and against lower and upper cut off limits. Evaluation over time does not show any significant decay or increase for the standards utilized. For gold reporting all samples reported within the Upper and Lower Cutoff Limit. For silver reporting two samples from AN12001X and three from AN12002X reported over limits. The following discussion of the standard results includes graphical representations of the data over time and within the upper and lower cutoff limits (Figure 8-1 to Figure 8-4).



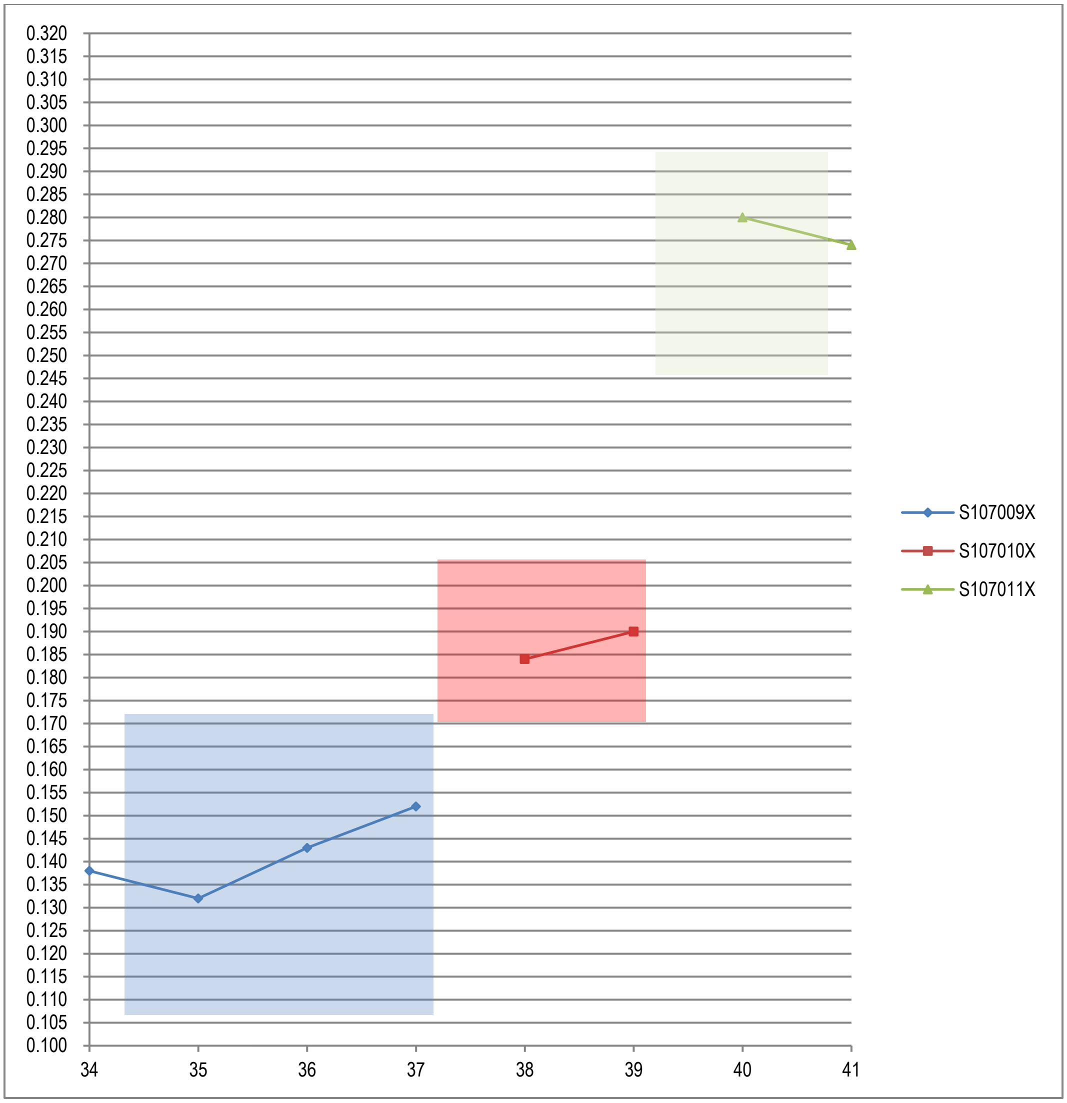
**Shaded areas reflect upper and lower cutoff range

Figure 8-1: MEG AN12001, 2, and 4X AuFA Standards Results



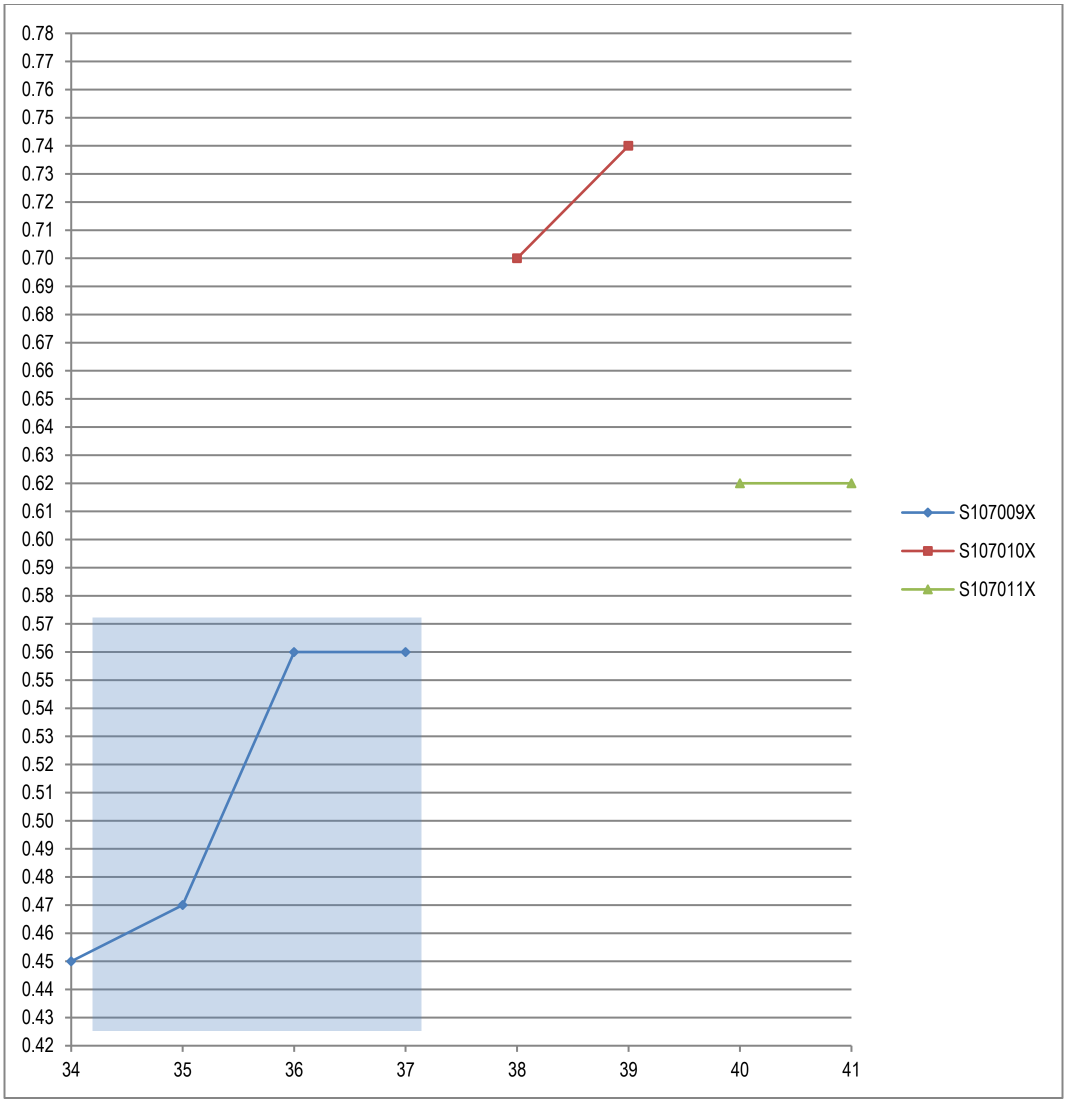
**Shaded areas reflect upper and lower cutoff range

Figure 8-2: MEG AN12001, 2, and 4X AgFA Standards Results



**Shaded areas reflect upper and lower cutoff range

Figure 8-3: MEG S107009, 10, and 11X AuFA Standards Results



**Shaded areas reflect upper and lower cutoff range
Note: 10 and 11 are gold only standards

Figure 8-4: MEG S107009, 10, and 11X AuFA Standards Results

The drilling assay certificates for Hycroft were loaded directly into the acQuire database. The program compared the standard and blanks against known upper and lower cutoff limits, and required review of any results outside acceptable ranges.

8.5.2.2 Check Assay Program

A total of 2,109 drill intervals from the 2011-2012 drill programs were selected for assay checks as part of the QA/QC program. This represents approximately 5% of the total drill intervals during the selected timeframe and were randomly selected using acQuire. Original pulped samples were collected by exploration technicians from the Hycroft pulp storage yard and shipped to American Assay Lab in Reno for analysis. Check assay analysis has been completed on a range of fire assay values from below detection to 3.340 g/t Au and 570 g/t Ag. Samples that assayed below the detection level for the analysis method used were set to 0.0 g/t for this analysis. Figure 8-5 illustrates the overall distribution of the original assay AuFA lab data (ALS Minerals and Inspectorate) against the American Assay Lab reassayed results. The R-squared value for both AuFA and AgFA indicates a high linear distribution between the two datasets (Figure 8-5 and Figure 8-6). Average gold fire values differed by 2.6%, while silver fire values differed by 3.2%

No additional drilling was completed in 2015-2016; however, HMC will continue to select a 5-10% sample set for check assays at regular intervals in future drilling programs.

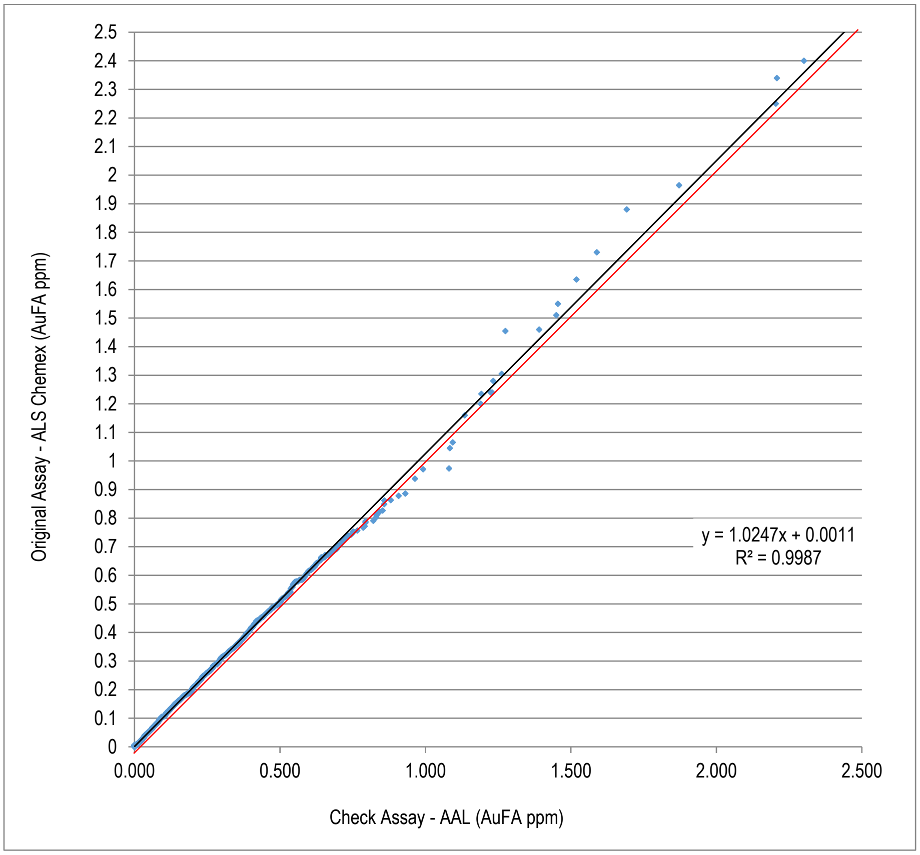


Figure 8-5: Check Assay - AuFA

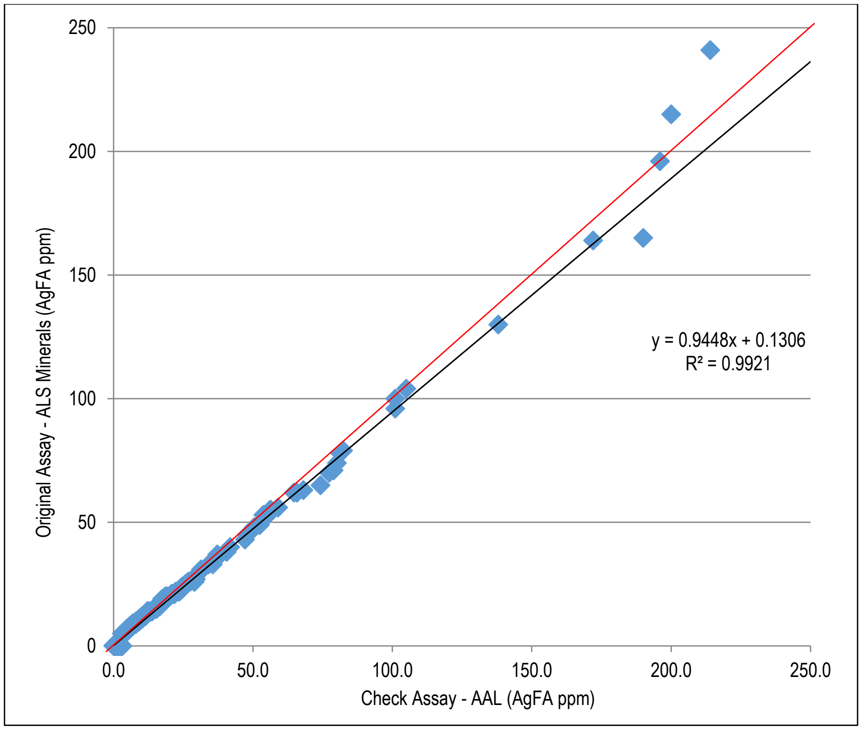


Figure 8-6: Check Assay - AgFA

During 2012 assay work at ALS, the laboratory performed a series of independent checks on duplicates from the Hycroft drilling program. Two different duplicate assay protocols were conducted as part of the internal lab QC protocol. The first was a preparation duplicate, where a split is taken after initial crushing, and then processed through pulverizing and all analytical procedures. The second duplicate was taken after pulverizing (pulp) and then processed through all analytical procedures. The first split provides quality control on laboratory precision from preparation and analysis, while the second provides quality control on analytical precision. During 2012, 5,899 pulp duplicates and 1,337 preparation duplicates were evaluated by ALS. Results are summarized in the Table 8-3 and illustrated in Figure 8-7 through Figure 8-10.

Table 8-3: ALS Duplicates

Prep Method	Assay Method	Average Original Assay (ppm)	Average Duplicate Assay (ppm)	Percent Difference	Number of Sample Pairs
Pulp Duplicate	Au-AA23	0.421	0.398	5.6	2,404
Pulp Duplicate	Ag-GRA21	12.25	10.96	10.6	2,133
Pulp Duplicate	Au-AA13hc	0.182	0.186	-1.8	661
Pulp Duplicate	Ag-AA13hc	5.69	5.51	3.1	701
Prep Duplicate	Au-AA23	0.244	0.237	2.7	429
Prep Duplicate	Ag-GRA21	11.06	10.94	1.0	562
Prep Duplicate	Au-AA13hc	0.095	0.090	5.6	170
Prep Duplicate	Ag-AA13hc	6.14	5.88	4.3	176

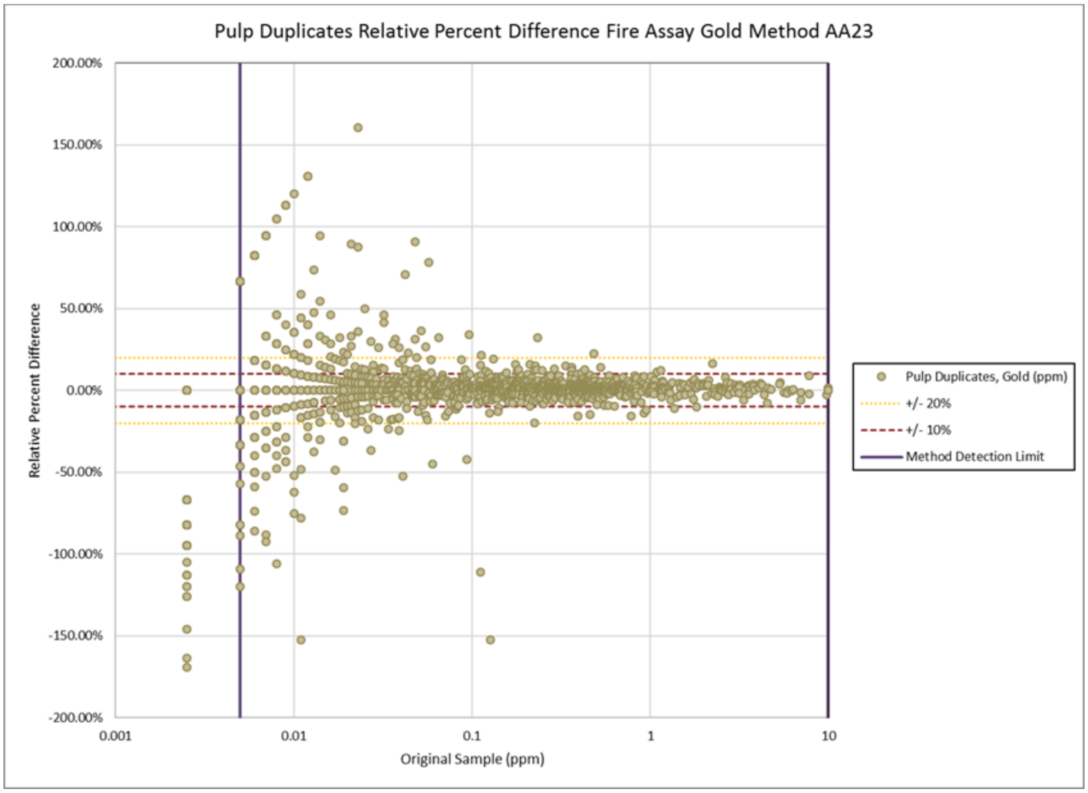


Figure 8-7: 2012 Pulp Duplicates Relative Percent Difference, Total Gold

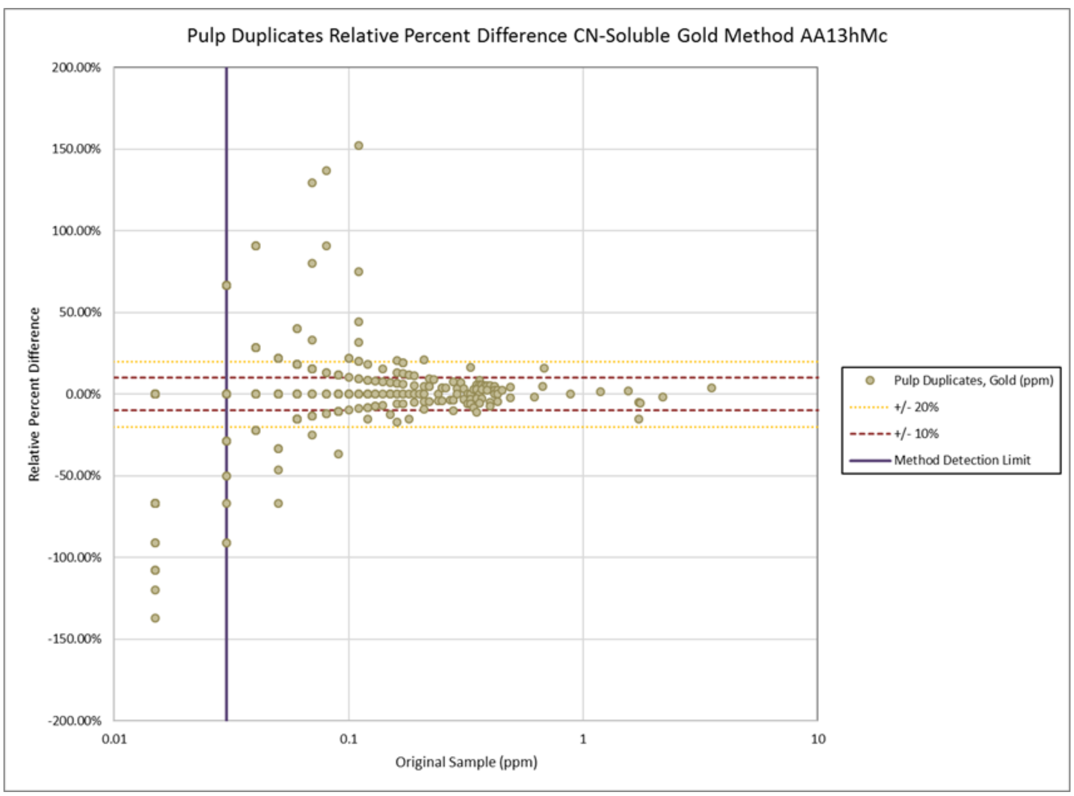


Figure 8-8: 2012 Pulp Duplicates Relative Percent Difference, CN-Soluble Gold

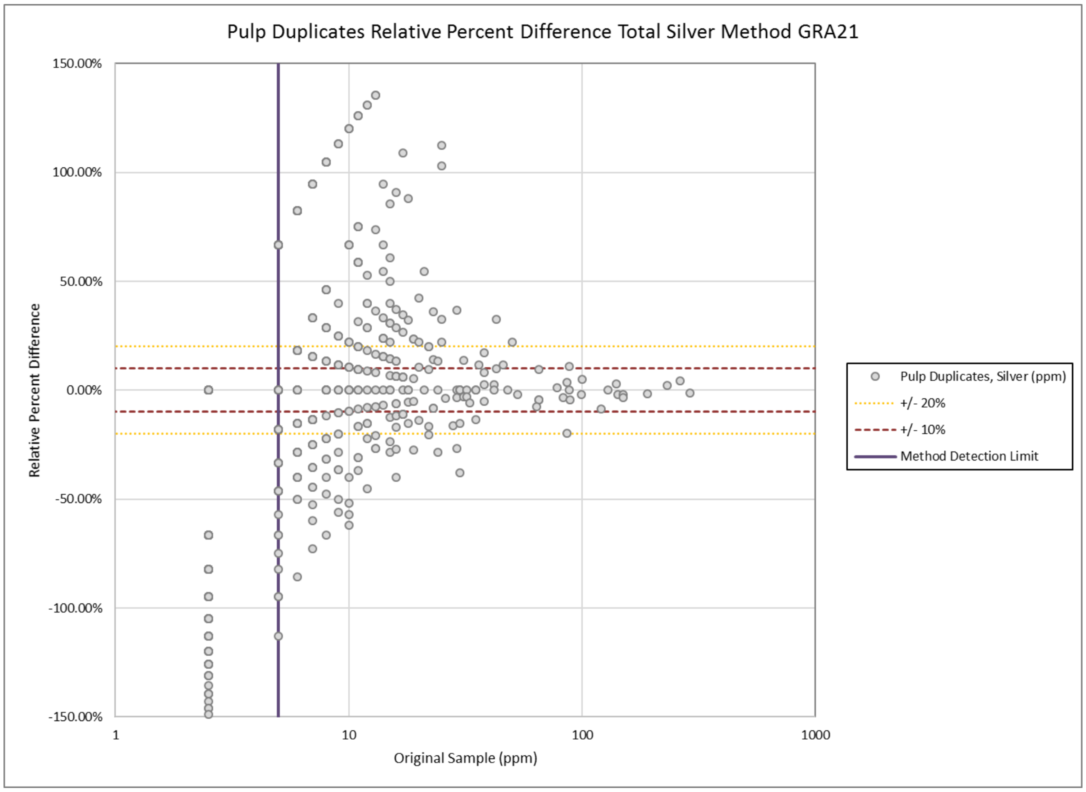


Figure 8-9: 2012 Pulp Duplicates Relative Percent Difference, Total Silver

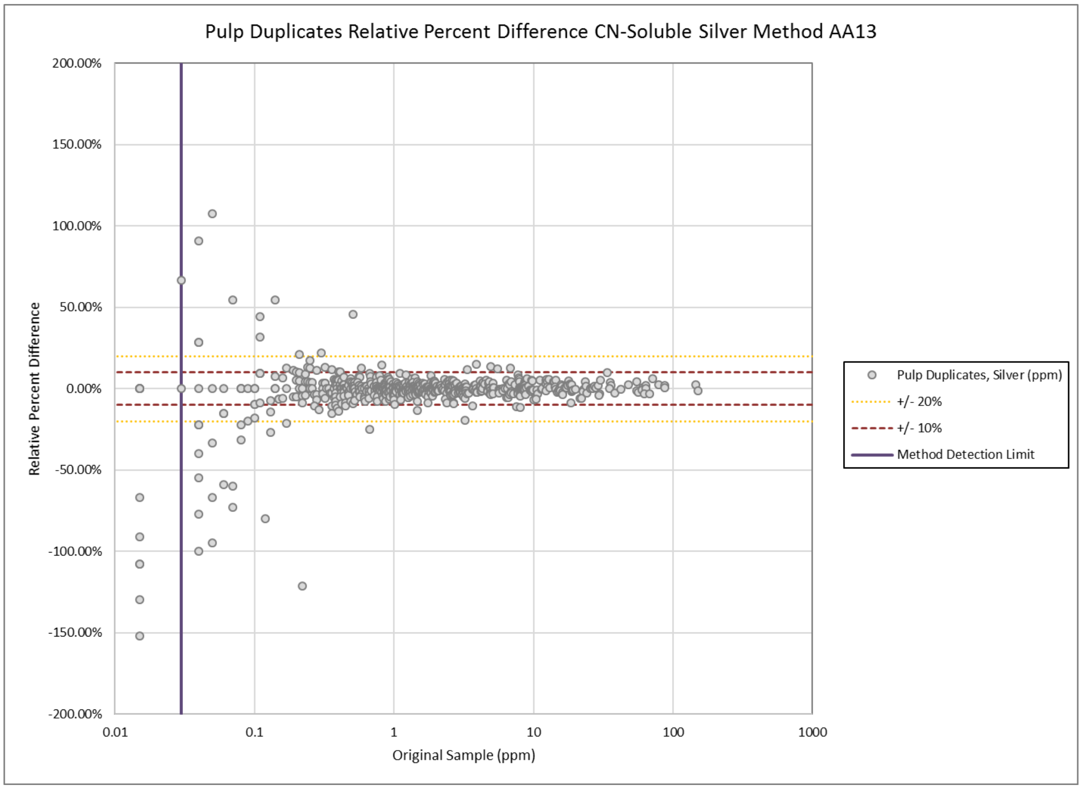


Figure 8-10: 2012 Pulp Duplicates Relative Percent Difference, CN-Soluble Silver

The results indicate good correlation in all the duplicate samples, with less than 6% variance in all methods, except for pulp duplicates with silver fire analysis. The 10.6% variance encountered in this subgroup is a result of a detection limit of 5 ppm, which is higher than the difference between the original and duplicate assay result.

The targeted resource cutoff grade of 0.003 opt gold (0.10 ppm) is five times the lower method detection limit (MDL) for total gold from fire assay. The lower MDL for cyanide-soluble gold is 0.03 ppm (0.00087 opt). Although CN-sol gold is not used directly for resource estimation, the Au CN:FA is used to define metallurgical material types. Therefore, analytical uncertainty limits the validity of ratio values for samples with low Au CN and/ or Au FA values. Samples with 0.001 or 0.002 opt Au values in the database were assigned to a material category separate from samples with significant mineralization.

The data analysis and charts show the performance of the available duplicate pairs, and the variation of values as they approach the lower MDL, which is +/- 100% of the reported value. The apparent scatter in low grade sample pairs reflects analytical uncertainty near the lower MDL. At about 0.1 ppm gold, the relative difference generally decreases and most sample pairs fall into the targeted range for variation.

Silver grades at Hycroft are variable, and silver mineralization is not genetically related to gold. Another complication in total silver results is variable analytical methods and detection limits. SRK reviewed the relative percent difference charts for silver results to evaluate confidence in the analytical data, and a minimum threshold for silver grade shells to constrain estimation. A resource cutoff grade for silver is not straightforward like gold, so SRK deferred to the limits of analytical confidence to constrain the silver estimation to a minimum of 0.1 ppm cyanide-soluble silver. Total silver was not commonly reported historically, and the CN-soluble silver dataset is more complete. Cyanide-soluble results are also more analytically consistent through time than total silver results.

8.6 OPINION ON ADEQUACY

Part of the economic mineral resource at Hycroft is near the threshold of analytical validity for older drilling. Discrepancies between modeled and mined grades have been an issue in the past in low grade zones. The gold fire analysis analytical method detection limit ranges for pre-HMC drill samples yield results that are invalid below 0.1 ppm (0.003 opt). However, areas with older drilling and no recent infill drilling have mostly been mined out. The lower precision of legacy assay data will likely have a minor impact on the resource estimation.

In the QP's opinion, the sample preparation and analysis procedures used for Hycroft drill hole samples meet current industry standards for quality and the assay results are suitable to use for mineral resource estimation and related geological modeling.

9 DATA VERIFICATION

The HMC drill hole database has been validated by the HMC exploration group. A review and validation of the HMC collar coordinate, down-hole survey, and geology data was completed in Q3 2014 by HMC geologists.

SRK completed data verification and validation in advance of geological modeling and resource estimation, first between May and July 2017, for gold, silver, sulfide sulfur, and total sulfur analytical results, and for logged geological data. During this review, the analytical databases were found to be incomplete. SRK worked with Hycroft to extract all available analytical data from the acQuire database. This resulted in a 58% increase in the sulfide sulfur dataset. The compilation of gold and silver assay values in parts per million (PPM) units resulted in more intervals with valid Au CN:FA values for oxide modeling, and greater precision for grade estimation. SRK completed data verification for the new analytical database in September 2017.

9.1 VERIFICATION OF HMC DRILL DATA

9.1.1 Data Selection

A check against laboratory certificates of all electronic assay data used in the resource model was completed in 2014 by HMC geologists. SRK verified data tables received between May and July 2017, and appended data to some, as described below.

The 2018 sonic drilling program was used for grade estimation in sulfide stockpiles only. Available data, limited to collar locations and assay results, were verified in preparation for estimation. Database values matched source documents, and no corrective actions were required. The data verification summary below pertains to the drillholes that inform the in-situ material in the grade estimation model.

9.1.2 Collar Survey Checks

Collar surveys were range-checked by HMC geologists for minimum and maximum northing, easting and elevation. The coordinates were also checked against the planned coordinates to detect errors in either set of numbers and for reversal (swapped coordinates). Drill hole plots were visually checked in Maptek Vulcan® and on topographic photo-based maps to confirm that holes are on the correct drill pads and map coordinates.

Drill hole collar locations were provided in **<hycroft_auag_lith_042915_dhd_collar.csv>**, which has 5,270 drill holes. The file **<DrillingByMethod.xlsx>** included the type of drill used, and length of reverse circulation vs. diamond core for holes pre-collared through barren material. SRK appended the drill type and lengths to a copy of the main collar location table.

SRK verified collar location coordinates and total depths for sixteen drill holes. These were selected to resolve issues noted during the initial data review, and to verify the most recent drilling. All drill hole collars verified were for Hycroft drill holes, and about half were drilled in 2014.

- AT-8 is 200 feet below the pre-mining topographic surface, and it may have the incorrect XYZ location. This drill hole was omitted from the model database.
- The collar elevation for H14R-5375 was the same as Max Depth, 160 feet. XYZ coordinates were corrected from the collar survey document provided by the client.
- H10D-3374 and H10R-3855 had identical collar locations and are both vertical. Hycroft provided the surveyed collar location for 3374. Sample and geology sheets for 3855 have conflicting total depths, and no collar or downhole survey documents were provided for this hole. 3855 was omitted from the modeling database.

- 88-1388 and 90-1447 have identical collar locations and downhole survey trajectories. According to former HMC mine geologists, 90-1447 should have priority, and 88-1388 should be ignored. 88-1388 was deleted from the collar table.

- Drill holes with collar coordinates and downhole surveys were found in the ACCDB tables, and appended to the model files if the drill holes also had assay data. After the compilation of assay data in parts per million units was completed, 178 drill holes were appended to the collar table. Most of these are water wells, sonic core holes on the leach pad, or are located southeast of the model boundary. The additional holes are likely to have a minor impact on the resource, but allow all assay data to be represented for this and other future modeling purposes.

SRK's collar table for modeling includes 5,501 drill holes, with a total of 2,482,722 drilled feet, and average depth of 455 feet. Of these, 5,257 drill holes are in the model domain and will be considered for resource estimation. The drill holes in the model domain have 2,351,634.1 total feet drilled, and average depth of 451 feet.

9.1.3 Downhole Survey Checks

All downhole surveys were checked electronically for minimum and maximum azimuth, inclination and depths. Surveys are checked against the planned azimuth and inclination to detect errors. Surveys were allowed to be taken within 200 feet of the expected hole total depth to ensure that the survey is completed before the drill is finished. Surveys were projected by the downhole surveyor to the expected total depth. Downhole surveys for drill holes completed since 2008 were completed by IDS, with north-seeking gyroscopic instruments.

In cases where the water temperature was too hot (above 140°F) to continue surveying, the deepest data was projected to the hole bottom. Total depths for projected downhole surveys, where different than actual depths, were extended or truncated to the actual depths using the projected data.

Downhole survey geometry was provided to SRK in **<hycroft_auag_lith_042915_dhd_survey.csv>**, for 5,263 drill holes. The file **<DownHoleSurvey.xlsx>** was an inventory of drill holes and indicated if each had a gyroscopic downhole survey completed.

The drill holes missing surveys in the data table are listed below.

- H13D-4562- TD 34ft
- H13R-4772- TD 120ft
- H13R-4786- TD 100ft, survey completed per inventory
- H13R-4806- TD 380ft
- H13R-4879- TD 100ft
- H14R-5396- verified, TD 680ft, survey document in verification package.
- H14R-5399- TD 645ft

According to the downhole survey inventory sheet, these were not surveyed except for H13R-4786. However, the downhole survey document from IDS for H14R-5396 was provided by Hycroft in the data verification package. Downhole survey information only for H14R-5396 was added to the survey table to include it in the model. Other recent drill holes have complete downhole surveys that were not imported to the master database, and have placeholder values. The number of placeholder values in the survey table was not quantified for this review.

SRK verified downhole survey data for 13 drill holes and most database values matched source documents. The exceptions were recent drill holes without final gyroscopic survey values loaded in the database, for H14R-5321 and

H14R-5396, as noted above. SRK appended the actual survey data to the table for modeling. These changes have minor impact on the model, but make the drill hole geometry consistent with the rest of the dataset.

SRK's downhole survey dataset for modeling includes all 5,501 drill holes in the collar table. There are six recent drill holes missing surveys. These also have incomplete geological and analytical data, and are expected to have minor impact on the model.

9.1.4 Gold and Silver Assay Verification

Laboratory source data files for all pre-2012 holes had been electronically checked. In March 2014, Hycroft exploration staff randomly checked 2,301 assay intervals, approximately 7% of all 2013 drill intervals included in the model update. The database values were compared directly with hardcopy laboratory certificate assay results. There were no errors or discrepancies related to the gold and silver assay values listed on the certificates and saved to the database.

SRK extracted the relevant results from the main acQuire database that was converted to a Microsoft Access database. All analytical results, including gold, silver, sulfur and carbon, and multi-element ICP values, are stored in the database table "dbo_CORPSAMPLEASSAY" and are identified by the sample ID. A summary of the gold and silver data in the master database table is presented in Table 9-1.

Recent drilling by HMC comprises 1,970 of the 5,501 (36.1%) drill holes in the database, and 1,426,739 of 2,482,722 (57.4%) drilled feet. Areas that lack recent drilling by HMC are generally mined out. Although different generations of assay data are merged in the new table, most of the in-situ material is informed by recent assay results reported in PPM. Older assay data reported in OPT will has a relatively low impact on estimated grades.

Table 9-1: Summary of Gold and Silver Values in Master Database Table

Units	Category	Records	Lab
PPM (ANV)	Gold Fire Assay	235,108	CMX
	Gold Cyanide	15,1247	CMX
	Silver Fire Assay	232,218	CMX
	Silver Cyanide	138,272	CMX
	Total ALS Chemex	**756,845**	
	Gold Fire Assay	30,978	INS
	Gold Cyanide	15,195	INS
	Silver Fire Assay	31,265	INS
	Silver Cyanide	15,073	INS
	Total Inspectorate	**92,511**	
	Gold Fire Assay	5,303	AAL
	Gold Cyanide	2,483	AAL
	Silver Fire Assay	1	AAL
	Silver Cyanide	2,483	AAL
	Total American Assay	**10,270**	
	Gold Fire Assay	235	UNK
	Gold Cyanide	235	UNK
	Silver Fire Assay	5	UNK
	Silver Cyanide	235	UNK
	Total Unknown Lab	**710**	
	Gold Fire Assay	965	SGS
	Silver Fire Assay	844	SGS
	Total SGS Lab	**1,809**	
	Gold Fire Assay	1,970	HYC
	Gold Cyanide	1,970	HYC

Units	Category	Records	Lab
	Silver Fire Assay	1,970	HYC
	Silver Cyanide	1,970	HYC
	Total Hycroft Lab	**7,880**	
	Total PPM values	***870,025***	
OPT (Legacy)	Gold Fire Assay	180,830	UNK
	Gold Cyanide	161,612	UNK
	Silver Fire Assay	18,077	UNK
	Silver Cyanide	125,146	UNK
	Total OPT values	***485,665***	

Source: SRK, 2017.

Most drill samples completed by HMC were sent to ALS Chemex/Minerals (CMX) in Reno, Nevada for preparation and analysis for gold and silver. The latest HMC drill holes were sent to BVI Inspectorate (INS) in Reno for preparation and analysis. A sub-set of samples was analyzed at American Assay Labs (AAL) in Reno for check assays to validate the primary results. Results from these labs comprise most of the PPM dataset.

Reported values were stored in the database with Sample ID as the identifier. SRK worked with the Hycroft master database, but was not able to locate a master sample list with Drill hole ID, From depth, and To depth, matched to Sample ID. Most of the Sample IDs were the serial number of the drill hole and the ending depth of the sample, and the Drill hole ID and depth interval could be reproduced from the Sample ID. SRK generated a master sample list and with the VLOOKUP function in Excel, populated the drill hole ID and interval values in the assay tables. The drill hole IDs and From-To depths were verified against the assay table provided initially, to honor gaps between samples. The sample list for PPM assay results has 274,555 records and assigns Drill hole ID and depth to 268,251 unique sample intervals. Some drill hole IDs were not in the collar table, and therefore, the Drill hole ID is unpopulated in the assay data table.

Many intervals had multiple values for fire assay gold and the other parameters. This was due to multiple analysis techniques and check analysis. The rank of analytical methods is listed in the database table “dbo_ASSAYTYPE” and was used to determine which value should be applied to the main dataset for modeling. While cleaning the data, the pattern of check assays from AAL emerged, and was consistent for the entire dataset. High values from conventional fire assay typically had gravimetric analysis also. If available, the gravimetric fire assay value was used instead of the conventional fire assay value.

Results from earlier drilling, mostly by Vista Gold Corporation, were reported in ounces per ton. This legacy data is in the database table with the more recent data, and the units are clearly denoted for all values. Because these data also appear in the initial assay table, SRK adopted the OPT values as received, and converted them to PPM by multiplying by 34.286. By using the original data table that contained Drill hole ID and depth interval, the data processing effort for legacy data was minimized.

SRK’s assay data table has 456,910 records, about 8,200 fewer than the original table. The difference may be attributed to gaps between samples in HMC drill holes that are not populated in SRK’s table but are populated with -9 values in Hycroft’s table. Excluding drill holes outside of the model domain, and sonic core holes in the leach pad areas, about 97.5% of the SRK assay table will be considered for resource estimation.

9.1.5 Total and Sulfide Sulfur Verification

The assay data table for total sulfur and sulfide sulfur is from an extraction of the acQuire database completed on July 21, 2017, by a technical support representative at acQuire Technology Solutions. This is the complete dataset from Hycroft’s main database, and SRK assumed this represents the most current set of sulfide results that were validated and imported. When the possibility of a complete data extraction from the Hycroft database was uncertain, Hycroft requested all results from sulfur analysis certificates from the lab and made them available to SRK.

The sulfide table includes fields for total and sulfide sulfur results by lab, as well as total and organic carbon by lab. The carbon results were not applicable to SRK's work scope. There were some overlapping intervals with results from CMX and AAL. The CMX results were for assay samples, and were superseded by composite sample results from AAL. Over 90% of the sulfur data are reported by AAL between 2011 and 2012; the rest, by INS, in Q4 2014.

Results from H12D-3606 are in the dataset but the From-To values are zero. Intervals were not available in the source documents, and these results are not currently usable for modeling. There were four drill holes with duplicated intervals. After verifying the intervals and results reported by Inspectorate, SRK corrected the data table to match source documents.

The dataset is summarized below:

- 5,408 records in 149 drill holes.
- INS reported ONLY sulfide sulfur, no total sulfur or carbon, in 23 drill holes, mostly in the Bay deposit, analyzed in Q4 2014.
- Total and sulfide sulfur results from AAL for 126 drill holes.
- 5,271 records with sulfide sulfur values; 137 samples from several drill holes are missing sulfide values.
- 4,889 records with total sulfur values, from AAL only.

Future database work should include resolving the zero values for composite interval depths, and validating all 150 sulfur certificates from AAL for drill hole samples. There may be additional sulfur results from INS, the complete set of certificates should be obtained from the lab to validate. All valid sulfide data should be imported to the HMC master database for future sulfide sulfur estimation. In the QP's opinion, the sulfur data is sufficient for modeling.

9.1.6 Geological Data Check

HMC geologists loaded geologic logs directly to the acQuire database, preventing the use of invalid logging codes and format. Visual inspection of logging data was completed to detect data entry errors. The dataset was checked electronically using scripts to compare data against source files, and to find discrepancies. Logs were visually checked against electronic data and by examining core photos where necessary. Start and ending log footages are checked for gaps and overlapping values. Holes were checked that the encoding of RC and Core (R or D) in the hole name is correct.

Total depths in the database were checked against total depths as drilled and against maximum assay depths. Several hole maximum depths as logged were found to be short by a few feet as these are the result of a missing sample for those intervals not recorded in the geologic logs. The differences are minor and are not likely to affect lithology modeling.

Logged geological data were provided to SRK in **<hycroft_auag_lith_042915_dhd_geology.csv>**, for 4,899 drill holes. There are 96 recent holes without logged geology. These are H*R- drill holes, all RC, that were completed in 2014 and not logged. There are 275 older holes without geology data. Lithology and alteration shapes provided by the client were validated without this missing data. Of the 21 drill holes selected for data verification, only six had available geologic logs. About half of the drill holes selected for verification were completed recently, and may not have hard copies of the electronic logs.

SRK selected five drill holes for verification of logged lithology and formation, after noting discrepancies between logged and modeled lithology in the wireframes. The database codes matched the source logs for the drill holes with available geology logs. There is a discrepancy between modeled lithology and logged lithology in H11D-4289, where the logged

Tcm and Tk contact was not honored in the model. This has limited impact on the model, and the modeled lithology elsewhere generally honors drill hole data. Generally, modeled alteration solids honor logged alteration. The drill hole H12D-4436 was selected to verify logged argillic and silicic alteration, which both occur in the area modeled as a boundary between the two. The alteration model solids are exclusive, meaning that they do not overlap, so defining a boundary between two alteration types that occur together is a judgement call made by the geologists.

SRK targeted about seven drill holes to verify oxidation and mineralogy logging. Logs for these were not available. Logged mineralogy and oxidation are applicable for defining metallurgical material types in material without gold data, in non-mineralized zones or intervals that lack valid cyanide-leachable to total gold ratios.

SRK used the geology table as-is for alteration and metallurgical material modeling.

9.1.6.1 Hardness

Hycroft provided logged hardness R values in the file **<Hardness_HYCROFT_052212.csv>** on July 12, 2017. The R hardness scale is qualitative, and ranges from 0 to 6, as integers. Real numbers are in the data table, from average values from logged sub-intervals in drilling runs. One drill hole had a geotechnical log available for verification. Logged and average values closely matched the database for this drill hole. Because the hardness scale is qualitative and imprecise by nature, and because values seem reasonable compared to the lithology and alteration they represent, in the QP's opinion, this dataset is suitable for estimating rock hardness as a component of mining and processing cost.

9.1.6.2 Density

Hycroft provided the available density data from specific gravity determinations on drill core and pit wall samples in **<Master Density 2010.xlsx>**. The file includes one tab with raw density data for 884 samples by Sample ID, data source, lithology, and alteration, and a tab with average density values summarized by material type. Laboratory values reported are Specific Gravity, in grams per cubic centimeter. Over 90% of the density values are from core samples collected by Allied Nevada and tested by ALS Minerals in 2010-2011. Calculated tonnage factors, as cubic feet per ton, are also included in the raw data table. SRK saved a copy of the raw data as a CSV file to import modeling software and added a field for drill hole ID. Thirty-five samples are grab samples from the Brimstone pit and do not have location coordinates. Two drill holes with twenty samples, 94-2394 and 94-2395, are not in the resource model domain and were excluded from the summary information below. Density sample locations are shown by data source in Figure 9-1, with traces of core holes and the 2018 optimized pit surface. Density values by lithology and alteration are summarized in Table 9-2. The available data were analyzed by SRK and Hycroft, to estimate tonnage factors and density values to apply for mineral resource estimation. Available data included bulk density values collected during mining, in addition to data from core sample testing.

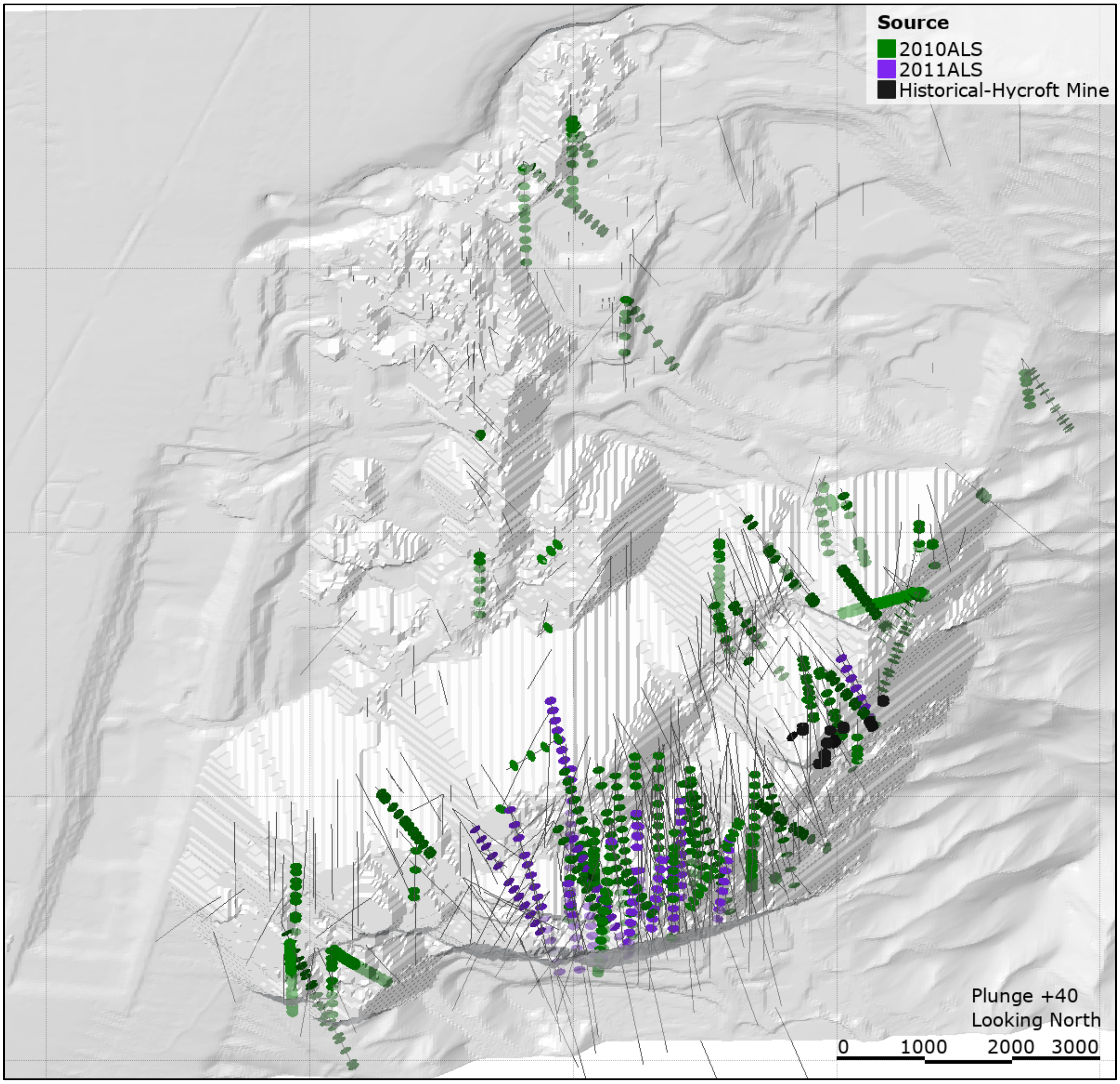


Figure 9-1: Density Samples with Optimized Resource Pit, Looking North and Down
Source: SRK, 2019

Table 9-2: Density Data Summary

Material Type		Samples	Density (grams per cubic cm)	Tonnage Factor (cubic ft per ton)
Alluvium		13	1.97	16.26
Camel Formation	Acid Leach	17	2.05	15.63
	Argillic	107	2.11	15.18
	Propylitic	6	2.14	14.97
	Silicic	225	2.53	12.66
	Unaltered	17	2.31	13.87
Kamma Formation	Acid Leach	33	2.51	12.76
	Argillic	93	2.24	14.30
	Propylitic	46	2.26	14.18
	Silicic	233	2.49	12.87
	Unaltered	39	2.41	13.29
Auld Lang Syne Formation	Silicic	10	2.65	12.09
	Unaltered	7	2.63	12.18
No Logged Lithology	Acid Leach	9	2.78	11.52
	Silicic	3	2.52	12.71
Total Samples		**858**		

Source: SRK, 2019

9.2 VERIFICATION OF HISTORICAL DRILL HOLE DATA

SRK completed a data check of the historical database in February 2008 for HMC. The electronic database provided to SRK, by HMC, contained approximately 3,183 drill holes including 186,123 records. SRK was able to locate and check original hard copy assay certificates for 175,002 records or 94%. In the process, the drill collar file was supplemented with additional details regarding laboratories and analytical detection limits. The data verification program was carried out from October 2007 through January 2008.

The Hycroft electronic database was provided by HMC in Microsoft Access format. SRK examined the contents of available historic data sets before selecting the most complete set. The database used for verification and development was called "hyc2000.db1.mdb." The data from the 2005 Canyon Resources drilling program was added to this database.

SRK was able to check 94% of the assay database against paper copies and logs. Minimal errors were noted (<1%) and errors were corrected in the database. The database consists of gold and silver assays, both fire and cyanide.

From 1983 to 1992, some full hole sample sets and other partial hole sets (selected mineralized ranges) were analyzed by Barringer. SRK was unable to locate the assay methodology or QA/QC procedures from Barringer. From 1991 to 1999, all exploration samples were analyzed on site by the Hycroft laboratory. No QA/QC records are available for this period of testing. On the occasion when Barringer check assays were available, in addition to Hycroft results, the Barringer check assay results were considered most reliable (as Barringer was an accredited facility). From 1999 to 2006, only minor analytical work was done; all by off-site laboratories such as AAL and CMX Minerals.

Geologic data was checked and validated by MRDI in 2000. As part of the MRDI program, 1,740 drill logs were selected for checking against the electronic files. A 0.8% average error was identified so they concluded the data was accurate. SRK followed up on the previous work, randomly selecting 150 drill logs for confirmatory checking. Although localized errors were observed in some of the six fields of geological data, no systematic errors were identified, such as large ranges of intervals with mismatched data. In the QP's opinion, the data is free of significant error and is appropriate for use in the determination of resources and reserves.

Several historical survey record books were preserved in the files of the Hycroft engineering office. The books contain collar coordinates of drill holes. Approximately 100 holes listed in the survey record books were checked in the electronic database, with no errors found. All of the drill data was imported into a 3-D modeling program. The collar elevations were successfully checked against the topographic surface appropriate for the time in which they were drilled.

Downhole surveys are uncommon in the historical database. There were no historic records to which the electronic data could be compared. An examination of the drill hole traces in 3-D, using the modeling program, indicates projections for the surveyed holes that appear reasonable in the opinion of the QP.

Following rigorous, record-by-record checks of the analytical database, the temporary electronic worksheets were reassembled into a single database which serves as the "assay" file for the project. The revised database contains original and updated fields for the four main analyses as follows:

ORIGINAL	NEW	DESCRIPTION
FAu	NFAu	Fire assay gold
FAg	NFAg	Fire assay silver (rarely assayed/reported)
CNAu	NCNAu	Cyanide soluble gold
CNAg	NCNAg	Cyanide soluble silver

The population of historical assay intervals was 186,123. SRK checked 175,002 intervals (94%). They identified total errors at 13%, of which 7% were related to missing data or data below detection limit. A total of 6% of the database contained substantive numerical errors, which were replaced by the reported values from the assay certificates. The most common errors were single shifts, where all records of an assay certificate were shifted by one interval (up or down). Next, there were examples of missing grades in the original electronic database for which certificate values existed. The certificate values were entered into the appropriate fields. Finally, there were occasional decimal errors made during input, which were corrected.

Drill hole coordinates were compiled into a new collar file for the database. In addition to collar coordinate information, the collar file has also been used to track the laboratory used for each drill hole, as well as the detection limits for the major elements tested.

In the assay database, records with no sample, no data, or missing data have been coded as -9. For all intervals whose value is below the detection limit for that element, the intervals have been coded as -8.

9.3 OPINION ON DATA ADEQUACY

The location, analytical, and geological data in the Hycroft database were verified against available source documents for selected drill holes. Verification focused on recent drilling, and on the sulfur dataset, to address concerns from the initial data review. In the opinion of the QP, enough is now known about the source of the sulfur data to apply it to resource estimation. The compiled assay data in PPM units is also applicable to resource estimation in the opinion of the QP.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

HMC has been operating the Hycroft open pit mine and run-of-mine heap leach facility, to produce gold and silver since 2008. Prior to that, Hycroft was operated in a similar manner by Vista Gold. The cumulative performance statistics and metallurgical test data gathered are extensive.

Beginning in 2007, Hycroft Mining examined milling options to expand production, including direct cyanidation of high-grade oxide ore, and production of a flotation concentrate from sulfide ore, followed by an oxidative treatment of the concentrate. The original focus was on oxidation methods primarily employed in the Nevada gold industry, including pressure oxidation (POX) and roasting. Test work on these processes showed that each of these options work well.

In 2013, the Company began testing a suite of alternative oxidation methods, including chlorination, ambient pressure alkaline oxidation, fine-grinding with intense cyanidation, and a procedure similar to the Albion process. The goal was to develop an economically viable process that would be less expensive to build and operate than autoclaves and that would eliminate the need for offsite concentrate sales.

Batch test results were positive and indicated that Hycroft concentrates were amenable to oxidation under atmospheric conditions, using trona as the acid neutralizing agent. Continuous pilot plant testing on three main domains was completed at Hazen Research to confirm these results.

In 2016, the viability of the atmospheric oxidation process using trona was demonstrated in a 10 ton-per-day integrated pilot plant at the mine site. This plant included primary grinding of 3/8" material, followed by flotation, atmospheric oxidation, cyanide leaching, counter-current decantation (CCD) and Merrill-Crowe precipitation.

The objective of the current study is to determine if soda ash, a refined form of trona, can be used to oxidize sulfides in a heap leach operation prior to irrigation with cyanide solution. This process, which is the subject of a pending patent application, will accomplish two goals, namely, the liberation of gold and silver in the sulfides by oxidation, thereby increasing its recovery, and the reduction of the heap's potential to turn acidic during cyanide leaching.

Over a decade of research into various carbonate oxidation systems has laid the foundation for the pre-oxidation and cyanidation process. A history of processes that have contributed to the development of this technology is included to show the progression of the mechanism used for oxidation as well as the logic that led to current operating procedures.

10.1 METALLURGICAL TESTING HISTORY

The metallurgical test programs conducted on the Hycroft deposit consisted of a series of comminution, flotation, concentrate oxidation, and cyanide leaching tests on whole ore, flotation tailing, and oxidized sulfide concentrate. The samples were mostly derived from drill cores. The bulk of the flotation tests were conducted at G&T Kamloops Laboratories ("G&T") and SGS Lakefield ("SGS"), both in Canada, and by Hazen Research Inc. ("Hazen") in Colorado.

Core samples for metallurgical testing were selected to represent the orebody, taking samples from five ore domains, as they were classified at the time. The main sources were Central, Brimstone and Vortex domains.

Ore was also classified as "oxide," "transition," or "sulfide," depending on the solubility of its gold content in cyanide solution (refractoriness). Ores having cyanide-soluble gold contents of 70% or higher are classified as oxide ore. Those with cyanide-soluble gold contents below 30% are considered sulfide. The remainder, with cyanide-soluble contents between 30 to 70% are considered transition ores. The classification has been shown to have no strong correlation with sulfide sulfur content.

10.1.1 Direct Cyanidation

Direct cyanidation leach results of bulk samples taken from all zones on the deposit were conducted early in the test program in 2010, yielding poor results, as expected. Concentrate was ground to a P80 of 325 mesh for this testing. Recoveries from Brimstone and Vortex were in the mid-20% range for gold and 80% range for silver, while other components of the deposit yielded recoveries ranging from 45 to 50% for gold and 55 to 83% for silver. In general, all samples being tested were direct cyanide leached for baseline comparison.

A good measure of recovery by direct cyanidation is the ratio of cyanide soluble metal to the total assay of the metal, that is, AuCN/AuFA and AgCN/AgFA. These ratios have been determined for a large number of exploration samples and have been included in the resource database. The cyanide soluble ratios for gold have been utilized in reserve estimation, particularly to route certain ores with higher cyanide-soluble gold to the heap leach pad without requiring the pre-treatment step.

10.1.2 Flotation

Refractory gold in Hycroft ore is believed to be associated in iron sulfides, primarily pyrite and marcasite. The goals of these tests were to determine the floatability of the sulfides, and the recovery of gold and silver in the sulfide concentrate. The ability to recover gold and silver in the sulfide concentrate reduces the volume of material to be treated.

Initial flotation test work was performed by SGS Minerals Services, Lakefield (SGS) in March of 2009 and continued at several laboratories until April 2014. During this time frame, the testing program began with bench scale tests and moved into pilot plant scale flotation tests at G&T and Hazen:

SGS Minerals Services (Lakefield) Batch Tests – March 2009
SGS Minerals Services (Lakefield) Batch Tests – Nov 2010
Kappes, Cassiday & Associates (KCA) Batch Tests – Jan 2011
Kappes, Cassiday & Associates (KCA) Locked Cycle Tests – May 2011
G&T Metallurgical Services Ltd. – Feb 2011
Hazen Research, Inc. – August 2011
Hazen Reaserch, Inc, - April 2014

A previous technical report [*M3, 2016*] provided a detailed review of the flotation results, which can be summarized as follows.

- The general trend indicated that flotation can achieve good recoveries at grinds ranging from 100 to 150 microns. Recoveries tended to decrease with grinds finer than 100 microns or coarser than 150 microns.
- Tests on the use of sodium hydrosulfide (NaHS) were mixed, but generally resulted in poor recoveries.
- Flotation in neutral pH, in general, performed better than tests at basic pH.
- The reagents used were strong, non-selective sulfide collectors, particularly potassium amyl xanthate (PAX) at 0.21 to 0.55 lb/t.
- Several tests indicate Cytec's AEROPHINE 3418A Promoter (sodium diisobutyldithiophosphinate) may improve gold and silver recovery.
- Variability flotation tests conducted by G&T [*G&T Metallurgical Services, 2011*] yielded an average mass pull of 13.8 percent.

- The same set of tests indicated a flotation time of 19 minutes for gold and 17 minutes for silver to achieve target recoveries.

10.1.3 Concentrate Oxidation

Oxidation tests on Hycroft concentrates included pressure oxidation (POX), roasting, atmospheric oxidation and other oxidation methods. The results indicated that these processes will work, with varying degrees of recovery. The following is a summary of the results of these tests. Atmospheric oxidation will be discussed separately in more detail.

10.1.3.1 Pressure Oxidation

Previous test work on POX had been conducted on pilot plant concentrates to determine operating criteria. The results indicate that: 1) an operating temperature range of 383°F to 401°F; 2) 100 psi oxygen overpressure; and 3) 60 minutes' residence time produce the highest cyanide amenability for gold and silver recovery. The POX tests also indicate that the concentrates may be prone to form jarosites, which inhibits silver recovery. The evidence for jarosite formation is:

- Color of the acidic autoclaved pulp is yellow on discharge and reddish brown when conditioned with a lime boil.
- Silver recovery is higher when the pulp is treated with a lime boil, a procedure which subjects the hot pulp for several hours to alkaline conditions.

The gold and silver recoveries from rougher concentrate POX discharge material that has been lime boiled and then leached with cyanide was in the mid-90s and 80s, respectively.

10.1.3.2 Roasting

Roaster test work was conducted on the Brimstone concentrate from a pilot plant to determine optimum conditions for processing. The results indicate that the optimum roast temperatures are between 797°F and 842°F (425°C and 450°C).

During the tests, average recoveries of 89% Au and 74% Ag were achievable from the concentrates by varying the leach and roast conditions slightly for the majority of the concentrate produced.

10.1.3.3 Other Oxidation Methods

Alternative methods were later tested, including chlorination, bio-oxidation, fine-grinding followed by intense cyanidation, and a procedure similar to the Albion process. In all cases, gold and silver recoveries from the oxidized ore were a function of the degree of oxidation.

10.1.3.4 Atmospheric Oxidation – Batch Tests

The focus of testing over the years 2013 through 2016 was to develop a process to oxidize sulfide concentrates under atmospheric conditions. The process was envisioned to be conducted in an agitated slurry at elevated temperatures, using oxygen as the oxidant and trona as the neutralizing agent for the acid produced. Several batch oxidation tests using trona were done at Hazen under various conditions on concentrates from Central, Brimstone and Vortex composites.

Batch tests using trona showed that full oxidation is not required to attain high recoveries in subsequent cyanide leaching, consistent with the findings of earlier oxidation studies. About 85% of the gold and 92% of the silver can be recovered by cyanidation if 60% of the sulfide-sulfur content in the concentrate is oxidized. The results for gold are shown in Figure 10-1.

The reaction kinetics were also found to be improved by higher temperatures up to 75ºC. Higher reaction temperatures (around 90ºC) were tested but returned slower oxidation kinetics, perhaps due to the decreased oxygen solubility in the laboratory bench-scale setting.

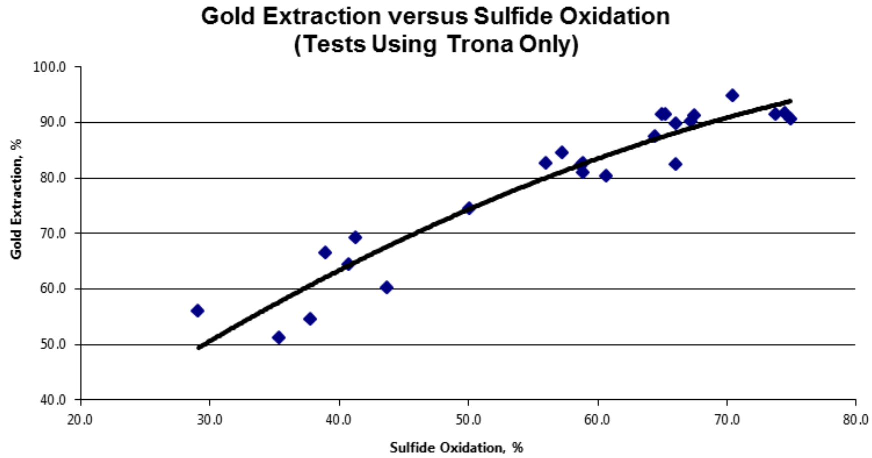


Figure 10-1: Gold and Silver Extractions Vs. Sulfide Oxidation

10.1.3.5 Pilot Plant Oxidation Tests

Continuous pilot tests in 10-liter vessels were completed at Hazen for the three domains. The results confirmed the findings of the batch tests. The pilot plant tests were run using 600 lb of trona per ton of concentrate, at 75ºC, 25-micron grind size, 20% solids and 48 hours' total residence time. Different material types oxidized at varying rates, with Vortex materials oxidizing the fastest followed by Central and then Brimstone. The Master Composite oxidation rate was comparable to Brimstone.

- Gold recovery versus sulfide oxidation was better than anticipated from bench scale tests;
- 80% gold recovery achieved at 50% sulfide oxidation for all material types
- 87% gold recovery achieved at 60% sulfide oxidation for all material types

10.1.3.6 Hycroft Mill Demonstration Plant

Hycroft Mining built a demonstration plant with nominal capacity of 10 tpd at the Hycroft mine site. The plant consisted of a ball mill, a rougher flotation bank, concentrate and tailing thickeners, a regrind mill, oxidation tanks, neutralization tanks, an oxidized concentrate thickener, cyanide leach tanks, counter-current decantation (CCD) thickeners, and a Merrill-Crowe precipitation package. It was operated continuously as an integrated plant, with concentrate surge capacity before oxidation and a pregnant solution storage before Merrill-Crowe. A report on the results of conclusions from the demonstration plant has been prepared by M3 and Blue Coast (Ibrado et al., Hycroft Mine Mill Demonstration Plant Initial Report, 31 October 2016) and presented in the National Instrument 43-101 feasibility study report [*Ibrado, A., et al, 2016*].

The demonstration plant was operated with Central and Brimstone ore that were mined from exposed mineralization at the surface of the current open pit.

Highlights of the demonstration plant test results are shown in Figure 10-2 for Central ore. For clarity, only results from Tank 1 (TK1) and Tank 5 (TK5) are shown. Oxidation levels of 60% or better were achieved when the correct steady-state testing conditions were maintained.

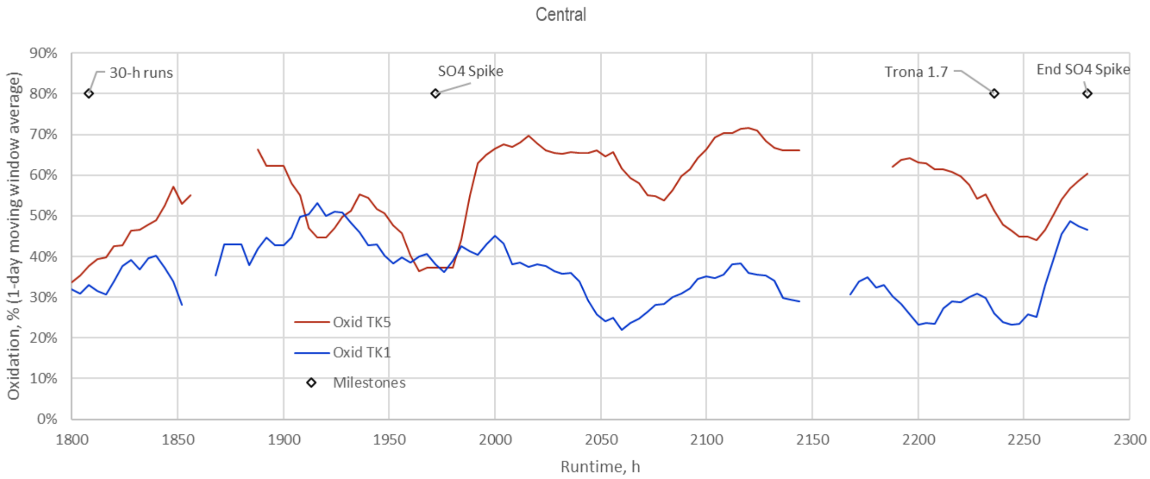


Figure 10-2: Oxidation of Central Flotation Concentrate: Sulfate Spike Test

Once the concentrates were oxidized, gold and silver recoveries significantly improved over the direct cyanidation recoveries. The results of cyanide leaching of oxidized concentrate are shown on Figure 10-3 as recovery of gold and silver during the demonstration plant operation. The graph starts with Central concentrate and then switches to Brimstone concentrates on 6/11/2016. Recovery of gold and silver from Central concentrate peak at around 85%. Gold recovery from Brimstone reaches 80 percent while silver recoveries from Brimstone peaked at 90%. The general shape of the lines roughly follows the degree of oxidation of the concentrate.

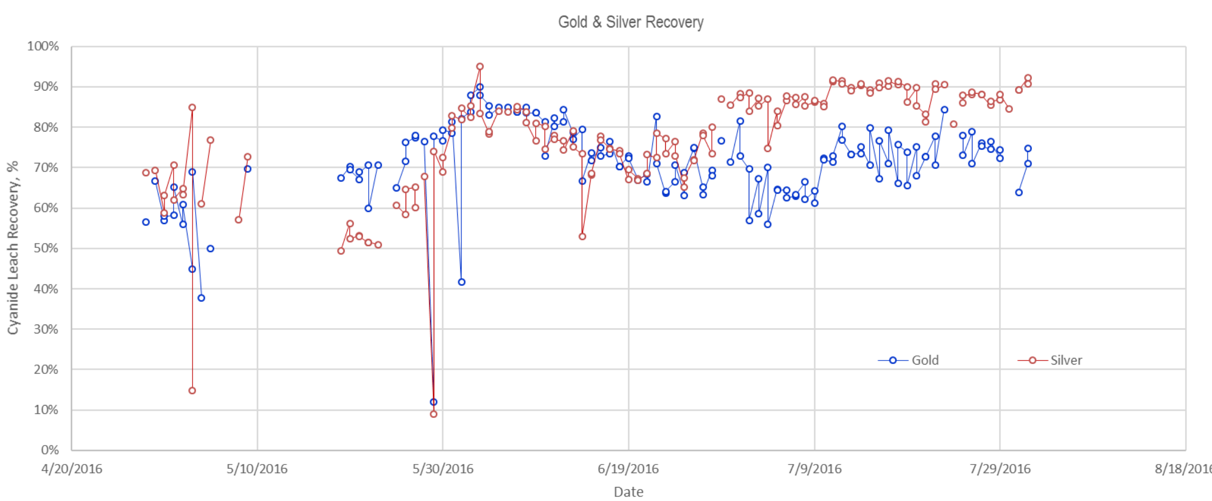


Figure 10-3: Demonstration Plant Cyanide Leach Recovery of Au and Ag

10.2 APPLICATION OF CARBONATE ASSISTED OXIDATION IN HEAP LEACHING

Hycroft explored the application of soda ash in heap leaching sulfide and transition ores. The interest was initially driven by the potential of faster restoration of heaps that have become acidic because of sulfide oxidation in the heap. The solubility of soda ash in water is much higher compared to lime. As an extension of this logic, the interest developed in whether soda ash could provide enough neutralizing power to enable heap leaching of transition and sulfide ores.

Preliminary column leach tests were performed during the time the demonstration plant was being operated. Hycroft also build two test pads, running ore samples from the Central and Brimstone deposits. Some of the results from these tests indicate that oxidation in a heap in the presence of soda ash can transform sulfide ores into transition and oxide ores (increased cyanide-soluble gold) and improve gold recovery in transition ores. The results encouraged Hycroft to continue testing the process to optimize the conditions and to better understand the mechanism of oxidation.

As in the atmospheric oxidation of concentrates, the oxidation of exposed sulfides in the heap could be accelerated by the catalytic action of the ferrous-ferric carbonate redox pair, without producing $CaSO_4$ precipitates.

10.2.1 Phase II Column Leach Tests

Based on preliminary experiments, oxidation and leaching tests were performed in sequence in order to separate cyanide from the carbonate/bicarbonate solutions. The general procedure, similar to the current study, is discussed in detail later in this section. Oxidation was estimated by the amount of total sulfate produced. Table 10-1 below shows some of the column leach tests for Central Sulfide, Brimstone Transition, Bay Transition and Vortex Sulfide.

Table 10-1: Oxidation and Metal Recoveries Attained in Phase 2 Column Leach Tests

Column	Domain	Redox	Days Ox Nominal	Oxidation* %	Au Rec %	Ag Rec %
32	Camel	Sulfide	120	33	65	60
43	Central	Sulfide	60	27	85	85
44	Central	Sulfide	60	25	75	71
40	Central	Sulfide	60	7	63	67
51	Brimstone	Transition	90	21	62	50
55	Bay	Transition	90	25	69	49
58	Vortex	Sulfide	60	9	53	26

The original oxidation target was about 45%, which was chosen based on the recovery v oxidation plot developed from the concentrate oxidation study. The goal was to attain 60 to 70% gold recovery. The results above clearly did not attain the desired oxidation levels but generally achieved the gold recovery targets. Phase II column leach test have therefore exceeded the expectations derived from recovery v oxidation curve. The test results also showed some scatter in the data that was difficult to explain.

Further analysis of the data from Phase II showed that the discrepancy stemmed from the fact that the recovery v oxidation curve was derived from sulfide concentrate oxidation tests, which essentially started from negligible cyanide soluble gold contents. In contrast, the ore samples tested in Phase II had varying levels of cyanide-soluble gold that contributed additional recoveries above those released by the oxidation process. This also explains the scatter in the data as being caused by the variability of the initial cyanide-soluble gold content of the samples.

The recovery versus oxidation plot has been corrected for initial cyanide soluble gold and redrawn in Figure 10-4.

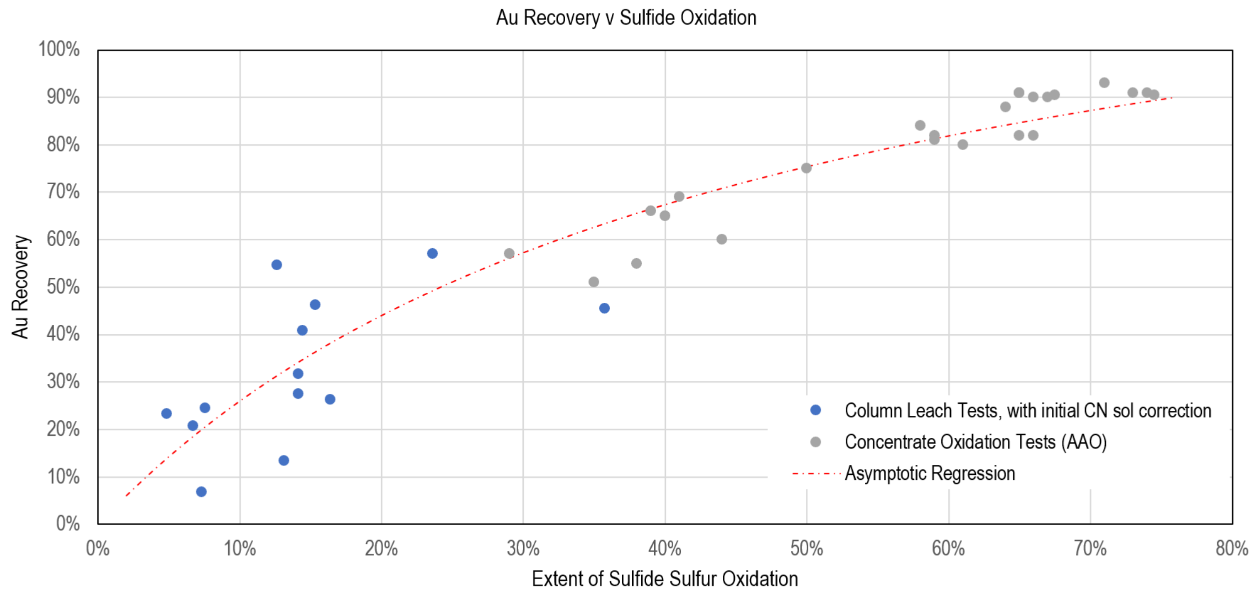


Figure 10-4: Gold Recovery v Sulfide Oxidation Plot Corrected for Initial CN-Soluble Au

Recognition of the effect of initial cyanide-soluble gold has beneficial impacts to operations. It lowers the oxidation targets to achieve the recoveries, thereby lowering soda ash consumption and potentially shortening the oxidation cycle. The same applies to column leach tests that will be performed in parallel with the progression of the heap leaching operations. Cyanide soluble gold will also become an important parameter in optimizing future mine plans.

10.3 ORE TYPES AND SAMPLING

10.3.1 Hycroft Ore Domains and Ore Types

The Hycroft orebody has been reported in the past as consisting of five ore domains; Brimstone, Central, Bay, Vortex and Camel.

Hycroft ore has also been classified as oxide, transition and sulfide. The basis of the classification is the degree of solubility of gold in cyanide solutions, defined as the ratio of cyanide soluble gold, Au_{CN}, to total gold by fire assay, Au_{FA}, or Au_{CN}/Au_{FA}. Ores are classified as oxides if $Au_{CN}/Au_{FA} \geqslant 70\%$, transition if Au_{CN}/Au_{FA} is between 30% and 70%, and as sulfides if $Au_{CN}/Au_{FA} \leqslant 30\%$. This classification is based on the refractoriness of the ore and has no correlation with sulfide-sulfur content, nor with the degree of oxidation of the total sulfur content.

Ore that is mined will be routed as follows:

1. All oxide ore will be leached directly as run-of-mine ore or crushed to minus ¾ inch,
2. All sulfide ore will be crushed to ½ inch, pre-oxidized with soda ash then leached.
3. Transition ore may be leached directly as ROM ore or crushed to minus ¾ inch, or crushed to minus ½ inch, pre-oxidized then leached (as a sulfide).

The choice of process treatment is determined by optimization driven by economics and the capacity of the crushing plant. This includes decisions on whether to crush oxide ore, or to route transition ore to direct leaching or pre-oxidation.

Table 10-2 is a list of oxide and transition ores from the five domains that will be routed to the leach pad as ROM ore or crushed ore for direct leaching. Historical recoveries, leach rates and processing costs are available for these ores.

Table 10-2: List of Domain Ores Routed to Direct Leaching

ROM ID	Heap Leach Method	Pit
Brimstone Oxide	ROM or ¾" Crush	Brimstone
Brimstone Transition	ROM or ¾" Crush	Brimstone
Central Oxide	ROM or ¾" Crush	Center
Central Transition	ROM or ¾" Crush	Center
Bay Oxide	ROM or ¾" Crush	Bay
Bay Transition	ROM or ¾" Crush	Bay
Vortex Oxide	ROM or ¾" Crush	Vortex
Vortex Transition	ROM or ¾" Crush	Vortex
Camel Oxide	ROM or ¾" Crush	Camel
Camel Transition	ROM or ¾" Crush	Camel

Table 10-3 below is a list of ores that are routed by mine optimization to pre-oxidation in the presence of soda ash prior to leaching. These ore domains were sampled for testing as described in the next subsection.

Table 10-3: List of Domain Ores Routed to Pre-Oxidation and Leaching

ROM ID	Heap Leach Method	Pit
Brimstone Sulfide	Soda Ash Pre-Ox then Leach	Brimstone
Brimstone Transition	Soda Ash Pre-Ox then Leach	Brimstone
Central Sulfide	Soda Ash Pre-Ox then Leach	Center
Central Transition	Soda Ash Pre-Ox then Leach	Center
Bay Sulfide	Soda Ash Pre-Ox then Leach	Bay
Bay Transition	Soda Ash Pre-Ox then Leach	Bay
Vortex Sulfide	Soda Ash Pre-Ox then Leach	Vortex
Vortex Transition	Soda Ash Pre-Ox then Leach	Vortex
Camel Sulfide	Soda Ash Pre-Ox then Leach	Camel
Camel Transition	Soda Ash Pre-Ox then Leach	Camel

10.3.2 Ore Samples for Metallurgical Testing

Hycroft drilled six metallurgical cores that intersected the main ore types, namely Central, Brimstone, Vortex, and Camel, to intercept sulfide and transition materials in these ore types, above and below a historical water table. In addition, tests were conducted to surface samples.

The locations of the drill holes and surface samples are shown in Figure 10-5. North-looking cross-sections of each drill core, with respect to the pit limits, water table and ore types, are shown in Figures 10-6 through 10-11. The ore body that is within the pit limits was well represented in the testing program.

Not all the test work was completed at the completion of the summary technical report. The collection of data available at the time was deemed sufficient to project recoveries because: (1) The main ore types were represented in the previous and current test results; (2) A single recovery versus oxidation trendline was developed for all transition and sulfide ores, including results from agitated leach tests and pilot plant tests on sulfide concentrates in addition to the column leach tests. This shows consistency in process behavior of the ores; (3) Some ore samples oxidized better or faster than others, resulting in better gold recoveries. This was taken into consideration on recovery projections for each ore type; and (4) The general observation was that the oxidation-rinse-and-leach procedure sequence was the most critical factor in the testing program. Essentially, the test program was an exercise to optimize this procedure for all transition and sulfide ore types.

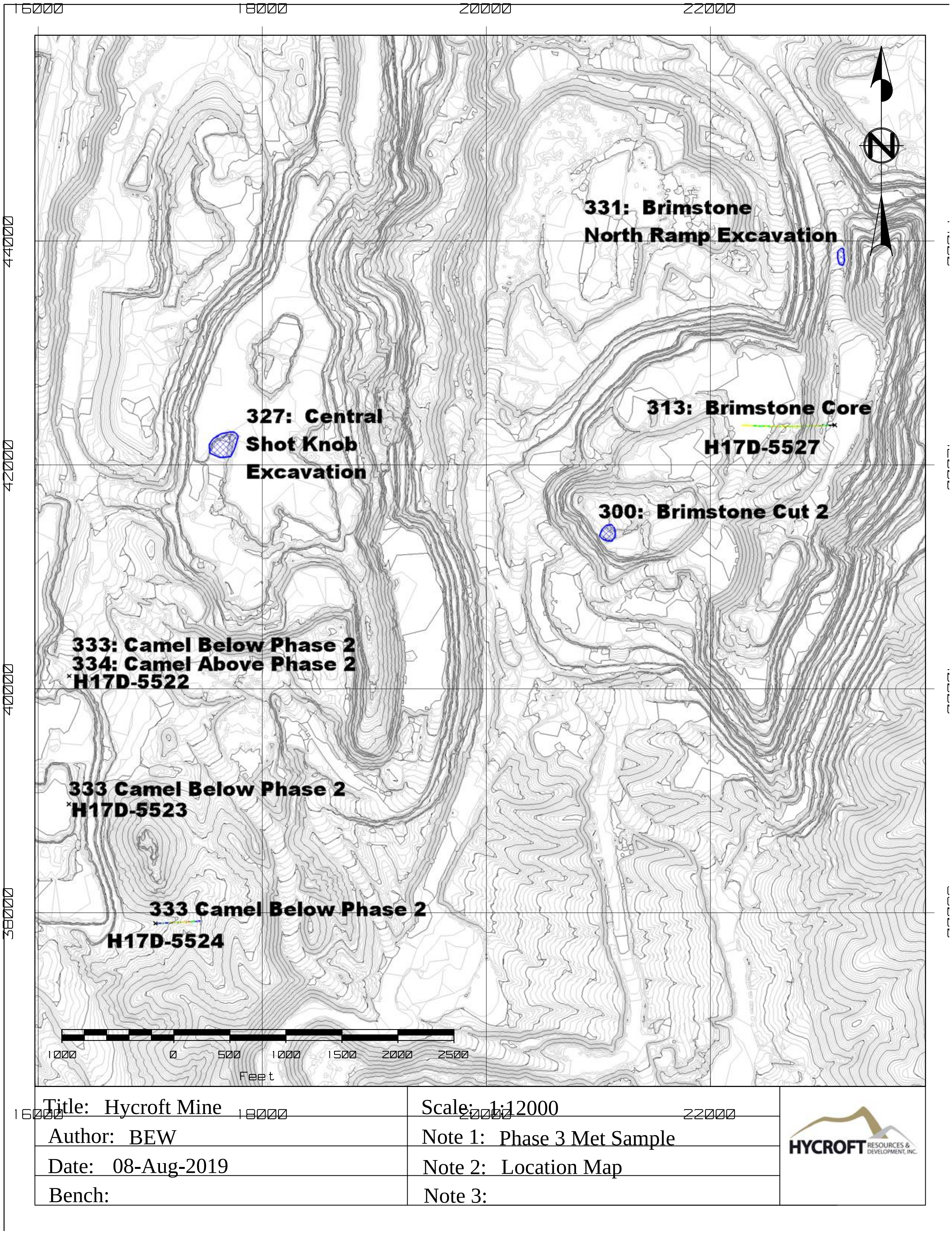


Figure 10-5: Location of Sample Sources for Phase 3 Heap Leaching Tests

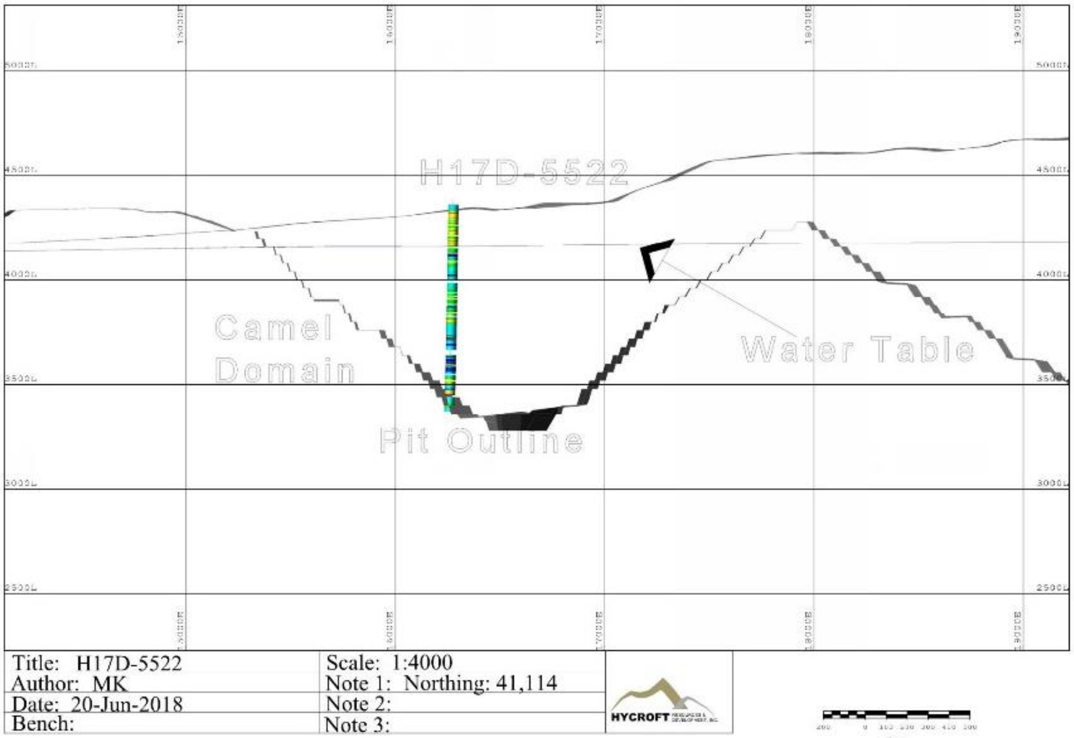


Figure 10-6: Section Showing Diamond Drill Hole H17D-5522

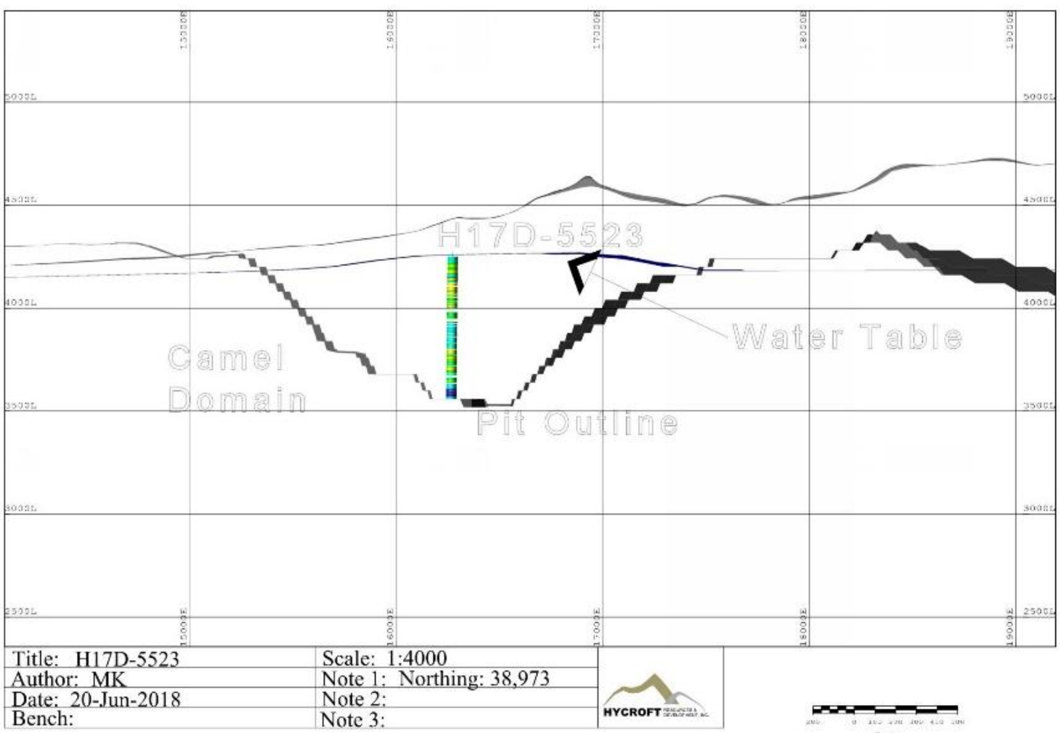


Figure 10-7: Section Showing Diamond Drill Hole H17D-5523

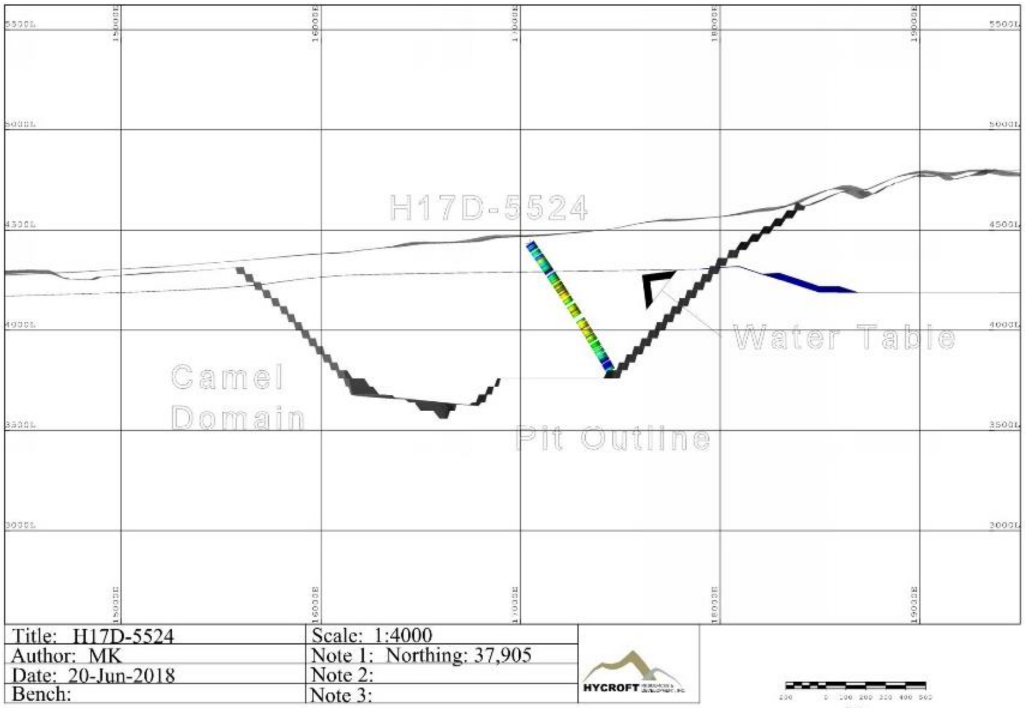


Figure 10-8: Section Showing Diamond Drill Hole H17D-5524

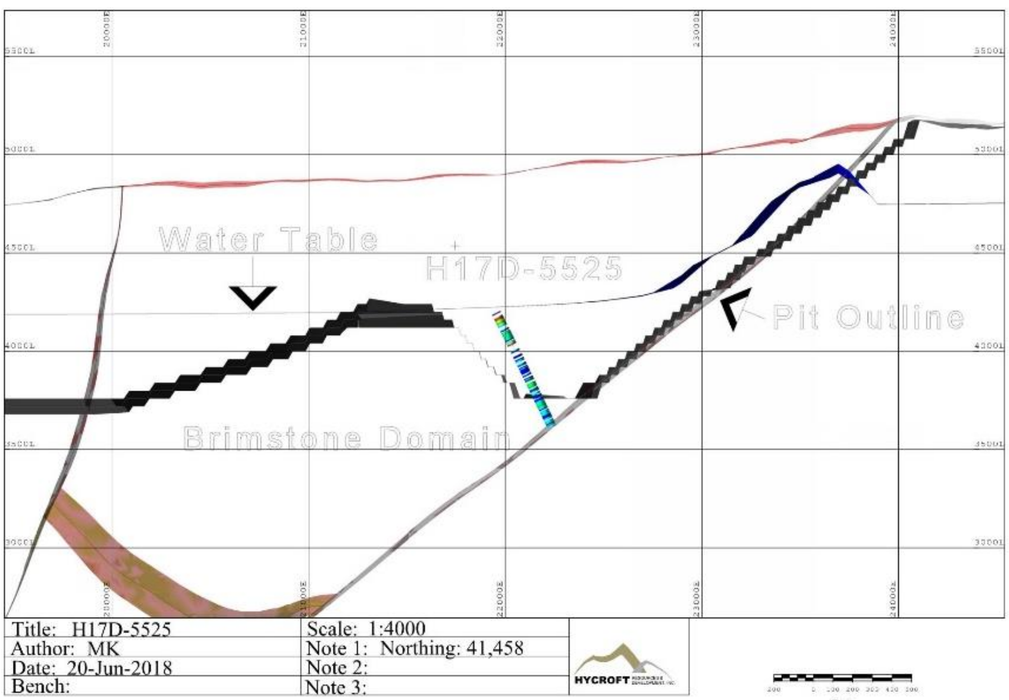


Figure 10-9: Section Showing Diamond Drill Hole H17D-5525

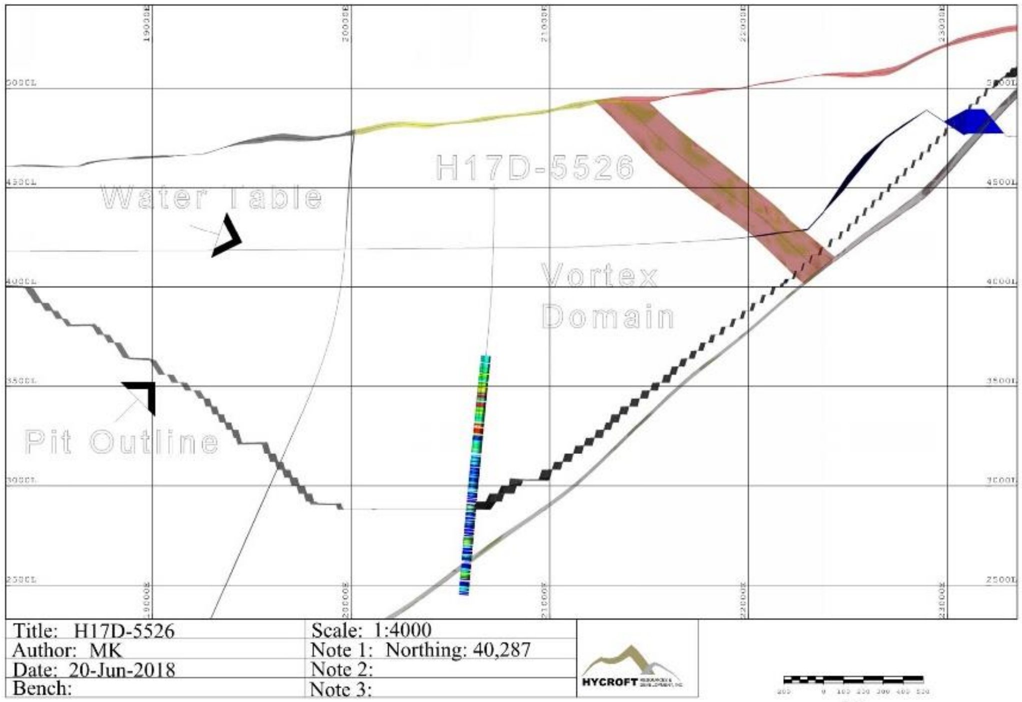


Figure 10-10: Section Showing Diamond Drill Hole H17D-5526

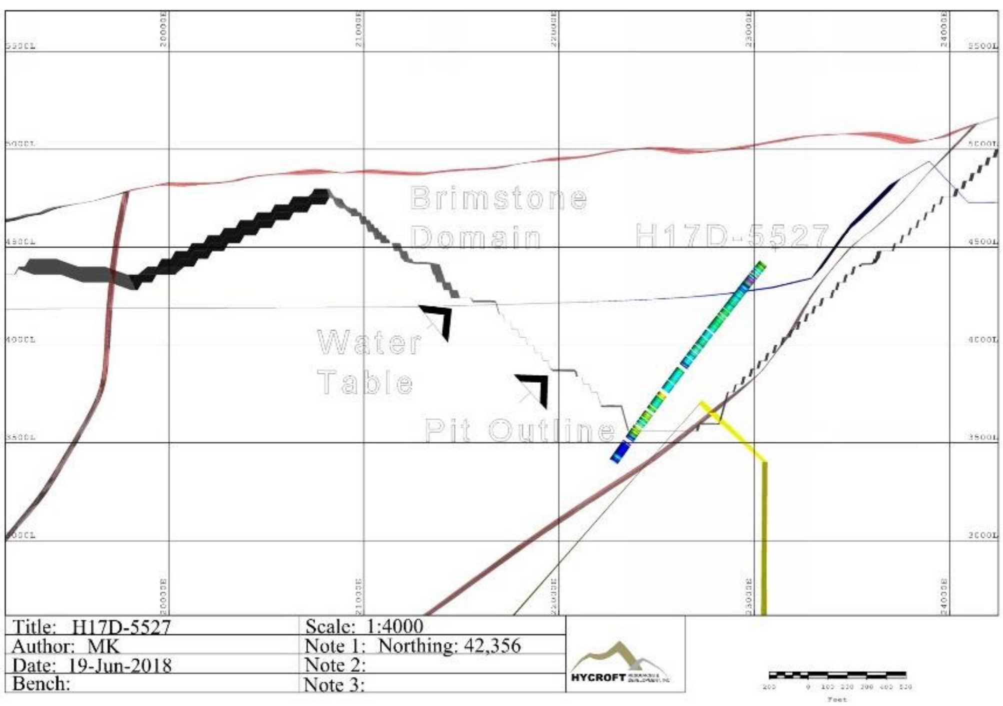


Figure 10-11: Section Showing Diamond Drill Hole H17D-5527

Table 10-4 is a list of the column leach tests that were started in Phase 3, and the source of the material tested therein. The last few columns in this series are restarts of columns from Phase 2 that did not perform well due to poor oxidation test conditions. Unfortunately, in this round of tests, a couple of deviations from the established testing procedures invalidated the majority of the tests before the deviations were corrected, as will be discussed in more detail later in this section.

Table 10-4: Phase 3 Column Oxidation and Leach Tests

Column	Source
300	Brimstone Sulfide Cut 2 Excavation
301	Brimstone Sulfide Cut 2 Excavation
302	Brimstone Sulfide Cut 2 Excavation
303	Central Sulfide Cut 5 6/18/18 Excavation
304	Central Sulfide Cut 5 6/18/18 Excavation
305	Central Sulfide Cut 5 6/18/18 Excavation
306	Brimstone Sulfide Cut 2 6/18/18 Excavation
307	Central Sulfide Cut 5 6/18/18 Excavation
308	Camel Sulfide Core from Phase 2
309	Camel Sulfide Core from Phase 2
310	Camel Sulfide from DDH 5522-5524
311	Camel Sulfide Core from DDH 5522-5524
312	Brimstone Sulfide from DDH 5527
313	Brimstone Sulfide from DDH 5527
314	Brimstone Sulfide from DDH 5527
315	Brimstone Sulfide from DDH 5527
316	Bay Sulfide Stockpile
317	Bay Sulfide Stockpile
318	Bay Sulfide Stockpile
319	Bay Sulfide Stockpile
320	Brimstone Sulfide Cut 2 Excavation
321	Brimstone Sulfide Cut 2 Excavation
322	Brimstone Sulfide Cut 2 Excavation
323	Brimstone Sulfide Cut 2 Excavation
324	Central Sulfide from Shot Knob Excavation
325	Central Sulfide from Shot Knob Excavation
326	Central Sulfide from Shot Knob Excavation
327	Central Sulfide from Shot Knob Excavation
328	Brimstone Sulfide North Ramp Excavation
329	Brimstone Sulfide North Ramp Excavation Aver
330	Brimstone Sulfide North Ramp Excavation Aver
331	Brimstone Sulfide North Ramp Excavation Aver
332	Brimstone Sulfide Excavation from Phase 2, Restart
333	Camel Sulfide Below the Water Table from Phase 2, Restart
334	Camel Sulfide Above the Water Table from Phase 2, Restart
335	Bay Sulfide from Phase 2, Restart
336	Vortex Sulfide Drill Core from Phase 2, Restart
337	Vortex Sulfide from DDH 5526
338	Vortex Sulfide from DDH 5526
339	Vortex Sulfide from DDH 5526

10.4 COMMINUTION TESTS

The Hycroft orebody has been thoroughly characterized for its comminution properties in the previous studies. These include crushing and grinding work indices, JK SimMet parameters A, b and t_a, and abrasion indices, which were reported in a previous study [*M3, 2016*]. Only the crushing work index is relevant in the heap leaching context of this study.

10.4.1 Crushing Work Index

Bond's crushing work index, CWi, was measured for seven samples – five from Vortex, one from Central, and one from Bay. The variability within this dataset is small, with a range of 6.18 to 9.76 kWh/st, standard deviation of 1.25, and a coefficient of variation of 15 %. For the design, the 80^{th} percentile value of 9.3 kWh/st was used.

10.5 COLUMN OXIDATION AND LEACH TESTS

The oxidation and cyanide leach tests were conducted in plexiglass cylindrical columns that were 1 foot in diameter and 4 feet high. Ore samples were crushed to nominal P100 = 1/2 inch, blended and loaded into the columns.

As established in Phase 2 testing based on preliminary experiments, oxidation and leaching had to be performed in sequence in order to separate cyanide from the carbonate/bicarbonate solutions.

Between the oxidation and leach stages, the columns were rinsed with water followed by lime-saturated water. The objective of the water rinse is to remove as much of sulfate produced and excess carbonate alkalinity as possible from the ore column. Sulfate that remains will react with calcium in the leach solution to precipitate $CaSO_4$, which could form a passivation layer over the solids that are being leached. Bicarbonate has been shown to react with cyanide resulting in high cyanide consumptions. The objective of the lime-water rinse is to neutralize residual bicarbonate after the water rinse. Depending on the efficiency of the water rinse, the lime-saturated rinse may not be required but this will have to be tested to determine the trade-off between the cost of lime-water rinse and cyanide loss.

Oxidation was performed for different periods ranging from 60 days to 180 days, by adding soda ash to the ore column and applying just enough solution to the column to keep the ore wet. This status is maintained to ensure that the interstices in the ore column are filled with oxygen-supplying air and not flooded with solution. A small amount of solution is allowed to drain at the bottom of the column, enough to collect at least 50 ml of sample each day for pH analysis, and to create a weekly composite for sulfate analysis. Oxidation was tracked by the amount of sulfate produced.

Phase 3 also introduced a new step to the procedure and that is to add iron (as ferric chloride) to the oxidation solutions at the start of the tests. This was based on an inference from Phase 2 results that oxidation of sulfides is essentially driven by the ferrous-ferric redox couple, which can be maintained at around pH 10 in a carbonate environment.

For the testing program, the bulk of solution and solids assays were performed by McClelland Laboratories in Reno. Some chemical analyses were conducted in-house (Hycroft Laboratory) for confirmation, control samples and for time-sensitive assays. M3 reviewed the chemical analysis procedures on site and found them to be in accordance with standard analytical practice.

During Phase 3 testing, some of the columns were operated with two deviations from the established procedures: (a) the water rinse was skipped to go directly to the lime-saturate water rinse, and (b) addition of iron, which was supposed to be only done at the start of the test, wash continued throughout the oxidation phase. Consequently, as predicted, the leach recoveries obtained were low, presumably because of the formation of passivating calcium sulfate precipitates. These tests, unfortunately, had to be rejected.

The effect of the excess iron is still being assessed. The intent was only to kickstart the reaction, which will then continue to produce enough iron ions to maintain same level in solution. The oxidation reaction produces much more iron ions than can be maintained in solutions such that most of it precipitates as hydroxides and goethite. There is a concern that the continued addition of iron may have introduced too much iron that could increase the formation of passivating iron hydroxides instead of goethite. Samples of the oxidized ore, as well as of the feed and leach tails are being submitted to Hazen Mineralogy for analysis.

Table 10-5 is a list of tests that employed the prescribed rinse procedure. Ten of them have been completed, however three of them, which are restarts of Brimstone, Bay and Vortex from Phase 2, did not have leach tails assays or size distributions. Six tests, three 20-foot columns and three large columns (in the old CIC columns) are still in progress or on hold.

Table 10-5: List of Column Leach Tests with the Prescribed Rinse Procedure

Column	Source	Status
300	Brimstone Sulfide Cut 2 Excavation	Complete
331	Brimstone Sulfide North Ramp Excavation Aver	Complete
313	Brimstone Sulfide from DDH 5527	Complete
327	Central Sulfide from Shot Knob Excavation	Complete
333	Camel Sulfide Below the Water Table from Phase 2, Restart	Complete
334	Camel Sulfide Above the Water Table from Phase 2, Restart	Complete
308	Camel Sulfide Core from Phase 2, 20' Columns	In progress
312	Brimstone Sulfide from DDH 5527, 20' Columns	In progress
328	Brimstone Sulfide North Ramp Excavation, 20' Columns	In progress
316	Bay Sulfide Stockpile, Large CIC Columns	In progress
320	Brimstone Sulfide Cut 2 Excavation, Large CIC Columns	In progress
324	Central Sulfide from Shot Knob Excavation, Large CIC Column	In progress
332	Brimstone Sulfide Excavation from Phase 2, Restart	No leach tails
335	Bay Sulfide from Phase 2, Restart	No leach tails
336	Vortex Sulfide Drill Core from Phase 2, Restart	No leach tails

Figure 10-12 through Figure 10-17 are the oxidation and leach curves for the six column leach tests that have been completed.

Figure 10-12 is the oxidation curve for a sample of Brimstone Cut 2 excavation (Column 300). The oxidation step only achieved a rate of oxidation of 5.5% based on sulfate formation. This was probably due to a lower than desired initial pH being maintained. Despite this, gold and silver recoveries were decent at 47% and 62%, respectively.

Another Brimstone sample excavated from the North Ramp area (Column 331) had better pH control and was able to attain over 12% oxidation. Gold recovery was better at 57%, while silver recovery is lower at 23%. See Figure 10-13.

Column 313 is another Brimstone sample, taken from drill core samples from Phase 2 (Figure 10-14). This column reached 12% oxidation and a gold recovery of 62%.

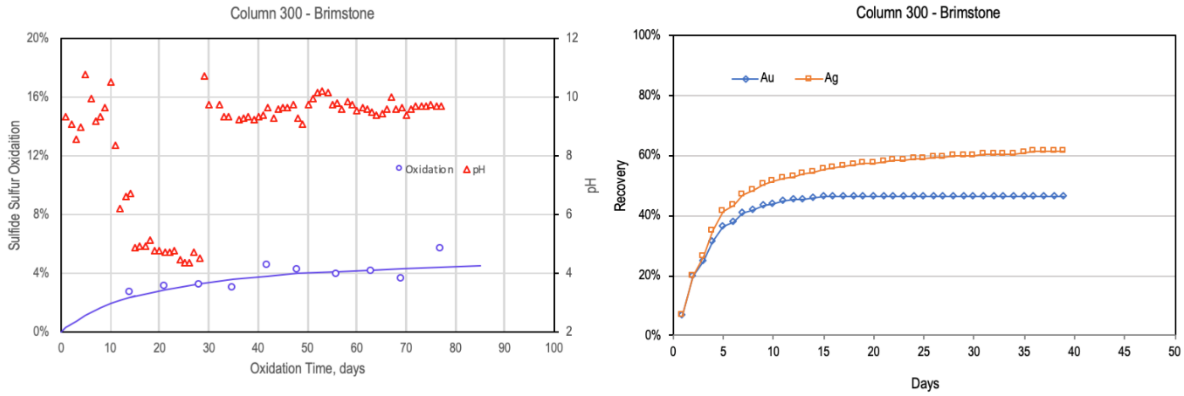


Figure 10-12: Column 300 Brimstone

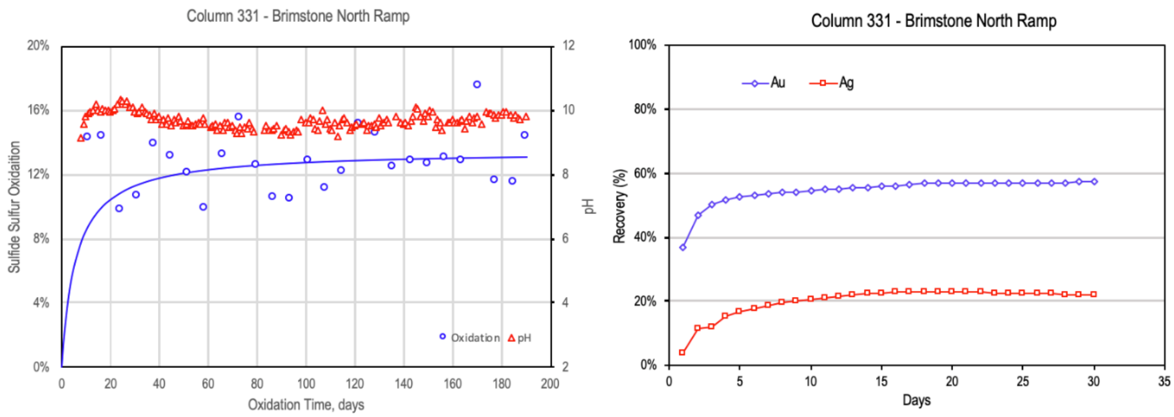


Figure 10-13: Column 331 Brimstone North Ramp

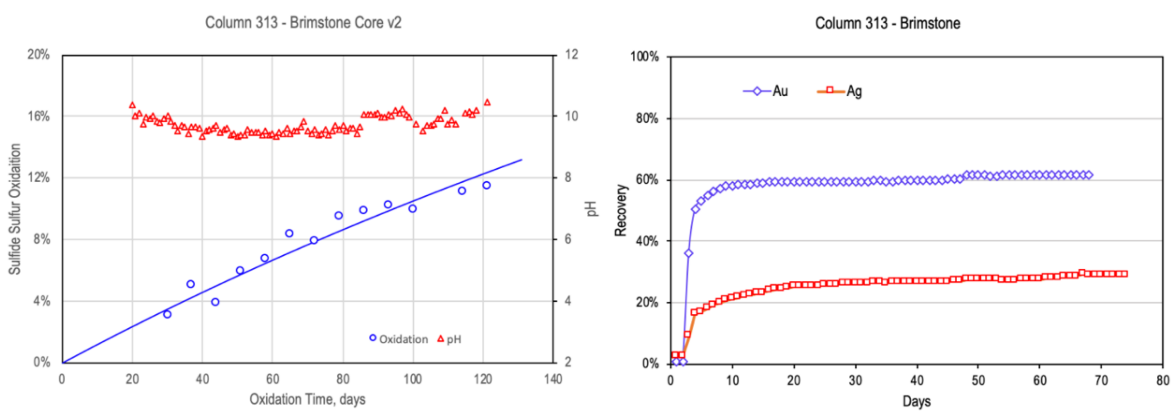


Figure 10-14: Brimstone Sulfide Core

Column 327 results, shown in Figure 10-15 below, is a test on another surface excavation sample from the Central domain. Based only on solution assays and feed fire assay, gold recovery is in excess of 80%. However, the back calculated heads deviated too far from the assayed heads. Consequently, gold recovery based on calculated heads was 20 percentage points lower. The size distribution of the tailing sample did not resemble the feed size distribution. Clearly, the tailing sample was not representative of the column.

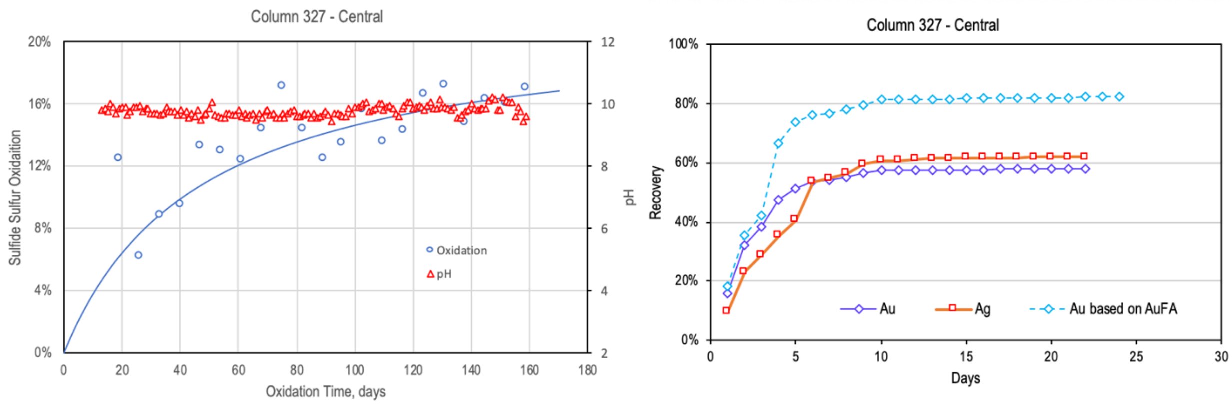


Figure 10-15: Column 327 Central Sulfide

Columns 333 and 334 (Figure 10-16 and Figure 10-17) were restarts of Camel columns that have been oxidized and leached in Phase 2. Both columns oxidized well and resulted in additional recoveries for both gold and silver. Total gold recoveries obtained were 51 % for Camel below the water table and 72% for Camel above the water table. Table 10-6 is a summary of the test results from the columns covered by the plots above.

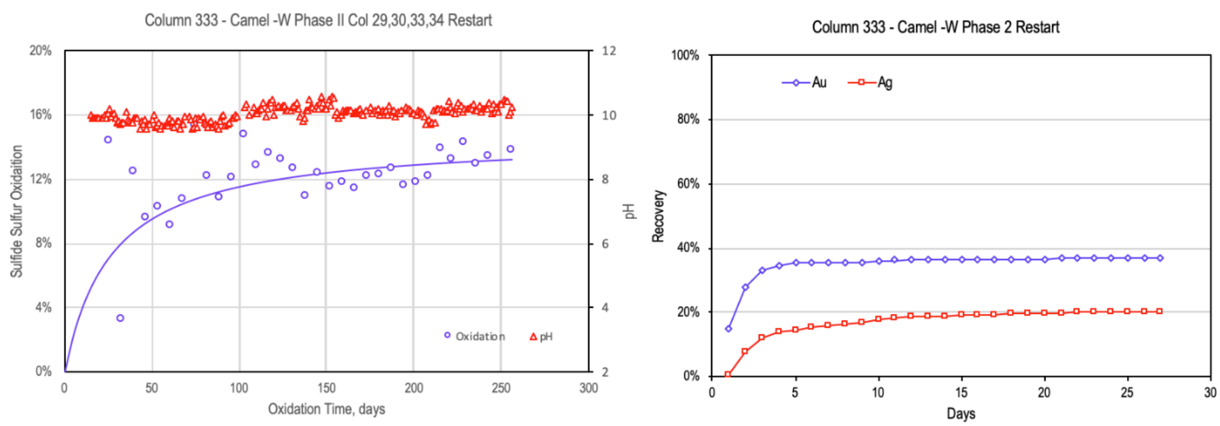


Figure 10-16: Column 333 Camel Below the Water Table - Restart

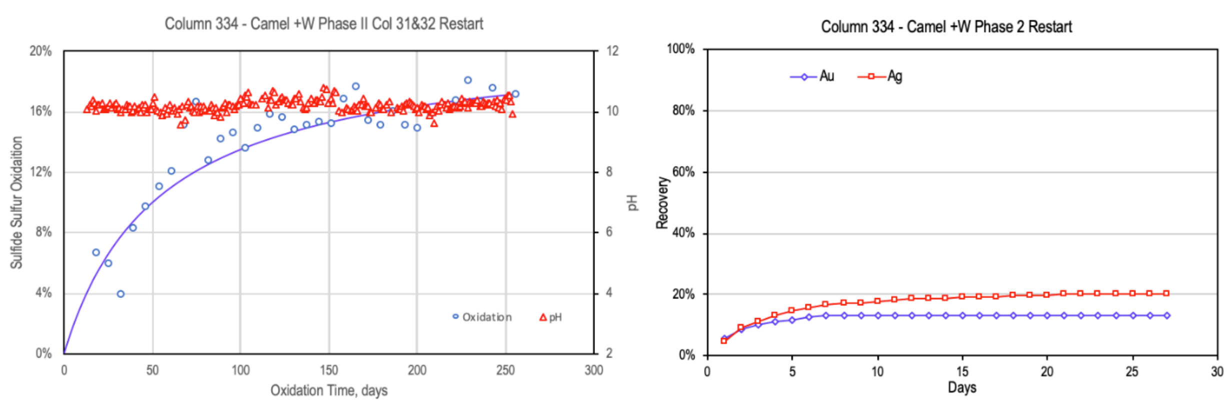


Figure 10-17: Column 334 Camel Above the Water Table Restart

Table 10-6: Summary of Test Results

Column	Domain	Au Heads, opt		Au_{CN}, opt	Oxidation, %	Recovery, %		Total Recovery, %	
		Assayed	Calc.			Au	Ag	Au	Ag
300	Brimstone	0.013	0.016	0.003	5.5	46.6	61.6		
331	Brimstone	0.014	0.010	0.005	14.7	57.4	23.1		
313	Brimstone	0.016	0.014	0.003	11.5	61.9	29.3		
327	Central	0.011	0.018	0.002	17.0	58-82	62.0		
333	Camel -W	0.011	0.007	0.000	13.8	32.4	56.4	51.2	70.5
334	Camel +W	0.008	0.015	0.001	17.1	62.4	55.3	72.0	69.2

Notes: -W = below the water table; +W = above the water table;
Total Recovery applies to Columns 333 & 334 to include recoveries from Phase 2 and Phase 3.

10.5.1 Leach Recovery as a Function of Oxidation

The results of the column oxidation followed by leach tests in general support the hypothesis that higher oxidation levels produce better gold and silver recoveries in the subsequent cyanide leach process.

10.5.2 Measurement of Oxidation

Determining the degree of oxidation was a challenge during the tests and would be a challenge as well during operations. The 50-ml sample taken every day was not an accurate sample as it represented the bottom portion of the column. The sulfate levels from these samples fluctuated particularly towards the end of the tests, instead of steadily increasing as one might expect if these were representative samples.

Sulfate concentrations probably underestimated the degree of oxidation as sulfate is only one of the oxidation products of sulfur. Thiosulfates and polythionates have been shown to be produced. A portion of the sulfates produced may also have precipitated and therefore remained with the solids even after the rinse.

During operations, the extent of sulfate production can be monitored by sampling at two or more levels in the heap leach lift under pre-oxidation. Instrumentation for oxygen concentration in the gas phase inside the lift are available and can be used, at least initially to ensure heap permeability.

10.6 CHEMISTRY OF OXIDATION

Trona is a naturally occurring evaporite mineral with the chemical formula $Na_2CO_3{\cdot}NaHCO_3{\cdot}2H_2O$. The largest known deposit of trona in the world is found in the Green River formation of Wyoming and Utah. Soda ash is manufactured from trona.

During the atmospheric oxidation process developed by Hycroft, soda ash or trona provides neutralizing capacity for the acid produced when sulfides are oxidized in a slurry. Both the carbonate and bicarbonate species can react with acid, depending on availability and pH. The oxidation and acid neutralization can be represented by the following reactions:

$$FeS_2 + 4\ Na_2CO_3 + 2.5\ H_2O + 3.75\ O_{2(g)} = FeO{*}OH + 2\ Na_2SO_4 + 4\ NaHCO_3 \qquad \Delta G° = -357.453\ kcal/mol$$

$$FeS_2 + 4\ NaHCO_3 + 3.75\ O_{2(g)} = FeO{*}OH + 2\ Na_2SO_4 + 1.5\ H_2CO_{3(a)} + 2.5\ CO_{2(g)} \qquad \Delta G° = -329.434\ kcal/mol$$

The oxidation process for sulfide concentrates is conducted at elevated temperatures, but below boiling, to maximize the reaction rate. The reaction may be carried out to neutral pH to minimize lime neutralization requirement prior to cyanidation, or to the extinction of carbonate and bicarbonate in solution to optimize trona consumption. It is possible to carry out the reaction to very acidic pH but this may lead to the formation of jarosites.

One of the most important features of the oxidation reaction is the absence of a passivating product layer. Sulfur is oxidized through a series of sulfur oxide compound through thiosulfate, polythionates, and others, finally to sulfate. Being in a sodium-based system, the sulfates produced remain in solution instead of being precipitated as calcium sulfate, as may be the case in the Albion process. Consequently, ultra-fine grinding is not required.

Since heap leach operations are conducted at ambient temperatures and with much lower sulfide sulfur concentration, the oxidation reactions are expected to be slower and, consequently, require only air to provide oxygen.

10.6.1 Role of Fe^{3+}/Fe^{2+} Couple

Initially, it was thought that soda ash or trona served purely a neutralizing duty. However, results of exploratory experiments suggested that these reagents may be speeding up the oxidation reaction. Results of the current set of tests indicate that this is the case. Oxidation tests conducted in columns of crushed ore show that the presence of trona or soda ash accelerated the oxidation process at ambient temperatures.

The mechanism proposed for this process involves the catalytic effect of the ferric and ferrous redox couple, where ferric and ferrous ions are stabilized in solution by carbonate. Table 10-7 below is a list of carbonate complexes that have been identified as stable in non-acidic solutions in the presences of high concentrations of carbonate or bicarbonate.

Table 10-7: List of Carbonate Complexes of Iron (Caldeira et al., 2009)

Ferrous Complexes	Ferric Complexes
$FeHCO_3^+$	$Fe(CO_3)_2^-$
$FeCO_{3(aq)}$	$FeCO_3^+$
$Fe(CO_3)_2^{2-}$	
$Fe(OH)CO_3^-$	

Figure 10-18 shows the stability regions of iron species in the presence of carbonate and bicarbonate. A possible redox pair could be $Fe(CO_3)_2^-$ (oxidized species) and $Fe(CO_3)_2^{2-}$ (reduced species) between pH 10 and 11, or between. $Fe(CO_3)_2^-$ (oxidized species) and $FeCO_{3(aq)}$ (reduced species) between pH 7 and 10, as shown by the reactions below:

$$Fe(CO_3)_2^- + e^- = Fe(CO_3)_2^{2-} \qquad \text{(pH 10 - 11)}$$

$$Fe(CO_3)_2^- + e^- = FeCO_{3(aq)} + CO_3^{2-} \qquad \text{(pH 7 – 10)}$$

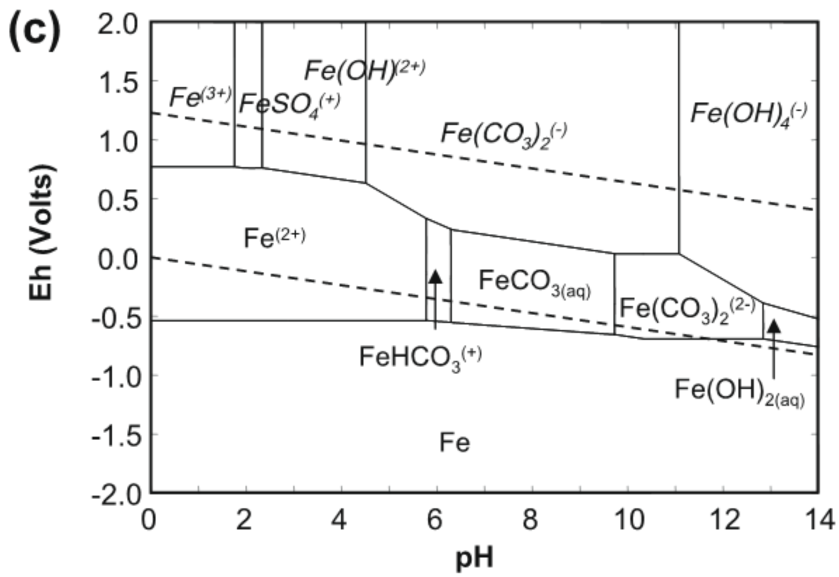


Figure 10-18: Eh-pH diagram showing dissolved iron species in the Fe-CO_3^{2-} system (from Caldeira et al., 2009)

At pHs 11 and higher, iron hydroxides predominate over the carbonates. It was reported that the redox pair of ferrous and ferric hydroxide does not catalyze the oxidation as well as the carbonate forms (Caldeira *et al.*, 2009).

The more commonly known redox pair is Fe(II)/Fe(III) at very acidic pH. Because of the high silver content of the Hycroft ore, working in these acidic regions may result in the formation of jarosites that may tie up silver.

Clearly, the best conditions for the oxidation of Hycroft sulfide and transition ores are at around pH 10 but not to exceed pH 11, in the presence of about 0.1 to 0.2 molar of carbonate or more. These translate to a carbonate alkalinity of 10,000 to 20,000 ppm. Because soda ash is being consumed by the reaction, a target of 60,000 ppm at the initial stages of oxidation was found to work well.

The basic model for Hycroft carbonate assisted pyrite oxidation solution was proposed by Caldeira et al. (2009) and involves a redox system driven at the pyrite face by the ferric/ferrous couple system. The reaction rate would be limited by one of three core factors: 1) ferrous iron solubility in alkaline solution; 2) the carbonate concentration; and 3) the available dissolved oxygen to regenerate ferrous to ferric. Figure 10-19 below is a schematic representation of this mechanism:

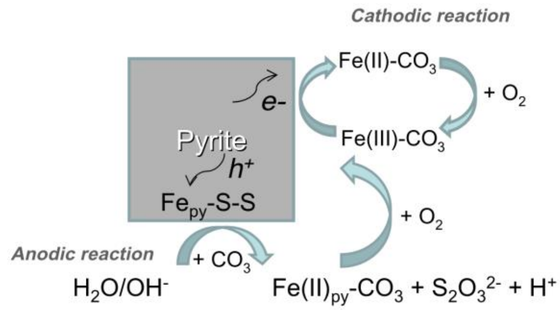


Figure 10-19: Mechanism of Pyrite Oxidation Assisted by the Fe3+/Fe2+ Couple (by Caldeira et al., 2009)

The above mechanism supports the findings of trona-assisted oxidation tests, which led to the development of the pre-oxidation process for the heap leaching of gold and silver from sulfide and transition ores (patent pending).

10.7 REAGENT CONSUMPTION

Typical cyanide leach operations require the addition of two chemical agents to produce gold and silver; the pre-oxidation modified leach process in use at Hycroft is dependent on a third reagent that supports sulfide oxidation. In addition to Sodium cyanide and lime, the proposed process must include a carbonate source. Throughout the test program, either trona or soda ash were used as carbonate sources during the pre-oxidation cycle of each test.

10.7.1 Carbonate Source

Both trona and soda ash create dual alkaline systems in solution that allow carbonate concentrations to reach over 60,000 ppm. In the Phase 2 column tests trona was used during pre-oxidation to neutralize acid and maintain carbonate concentrations high enough to facilitate oxidation by maintaining iron solubility. During Phase 3 soda ash was used in place of trona.

Soda ash addition was recorded daily to establish consumption. Solution leaving a column was sampled and alkalinity was measured so that the residual soda ash concentration could be calculated and subtracted from the original addition amount in order to calculate actual reagent consumption.

The relationship between trona addition (g) and total alkalinity (ppm) was established in the laboratory such that alkalinity measurements could be converted into trona concentration by the following equation:

$$[\text{Trona}] = \text{Total Alkalinity} / 602.59$$

Where 'Total Alkalinity' is the measured value in ppm and is the resultant concentration in grams per liter. The data used to establish this relationship can be seen in Figure 10-20 below. This relationship is consistent for soda ash, although only 67% of the mass of soda ash is required to attain the trona equivalent alkalinity.

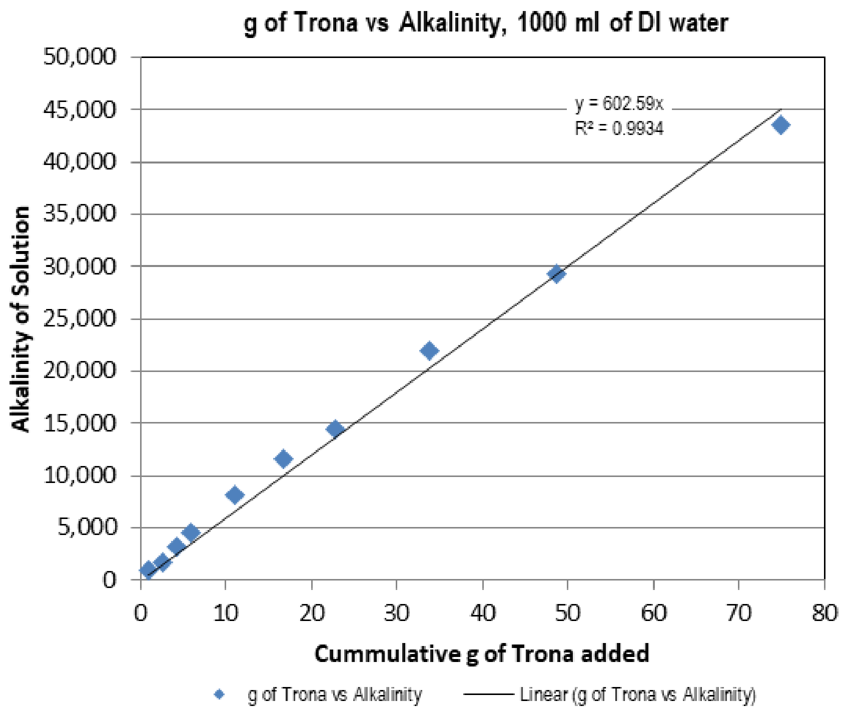


Figure 10-20: Trona Addition vs Total Alkalinity

To complete the mass balance for the alkaline species, crystallized reagent must be dissolved from the top of the column as the addition of reagents in the column setting sometimes results in reagent precipitation before the surface of the ore is penetrated. This crystallization is viewed as false consumption that is a byproduct of the solution application methods required to wet a 0.35 ft^2 instead of spraying solution over a larger heap leach area (+125,000 ft^2) at a greater rate due to depth of material.

After reducing the consumption by recovered reagent it was found that the resultant data set correlated with the following consumption equation:

Soda Ash Consumption = %Sulfide * Extent of Oxidation * 2,000 * 1.57

Trona Consumption = %Sulfide * Extent of Oxidation * 2,000 * 2.34

'%Sulfide' is the starting sulfide-sulfur content of the material to be oxidized, 'Extent of Oxidation' is the percentage of the sulfide-sulfur that is being oxidized and 'Soda Ash Consumption' is in units of pounds per ton of ore. '2,000' is the pounds per short ton conversion and '1.57' and '2.34' are the relationship coefficients developed from completed tests.

After expanding the data set in Phase 3, trona/soda ash consumption equation predictions were found to be slightly greater than the equations developed by past research (AAO). This is because the reagent application is slightly less efficient when the ore is not submerged in a reagent bath and some reagents remain inside the ore column either precipitated or within trapped solution.

Phase 3 and a retrospective on Phase 2 yielded the knowledge that the relationship between oxidation and recovery is dependent on the original conditions of the ore. For ore that begins with a large percentage of cyanide-soluble gold, less oxidation is required to reach the ultimate recovery targets. In Phases 2 and 3, the starting cyanide-soluble gold ratio ranged between 5% - 50% while the starting ratios for AAO were all at or near 0%. To account for this variety in material, the oxidation vs recovery curve was replotted using a 'discounted recovery'. The discounted recovery is the total gold recovered minus the gold originally recoverable in the head as defined by the cyanide-soluble gold ratio (Au_{CN}/Au_{FA}). The resultant plot confirmed a relationship between extent of oxidation and increase in cyanide-soluble gold (as defined by actual recovery via cyanidation) as much of the scatter from the original plot was eliminated.

The discovery of this relationship revealed that 'extent of oxidation' in the consumption equation is not a static number, rather, it is defined by the starting-cyanide solubility of gold and by the total recovery that is being targeted. These two measured inputs are then converted to an extent of oxidation target, defined by a liberation rate that is derived from the oxidation versus recovery curve.

In general, the LOM average sulfide-sulfur content is 1.92% and the nominal oxidation target is 40% for Camel, Central, Brimstone and Vortex, 31% for Bay. After adjusting for cyanide-soluble data available from the model of the ore body a projected reagent consumption was determined: 14.48 lbs/ton soda ash are required per ton of pre-oxidized ore. This figure accounts for 20% extra consumption in year 1 and 15% extra consumption in years 2 and 3 as the process operating procedure is refined.

An example of how trona consumption was tracked during testing is provided below:

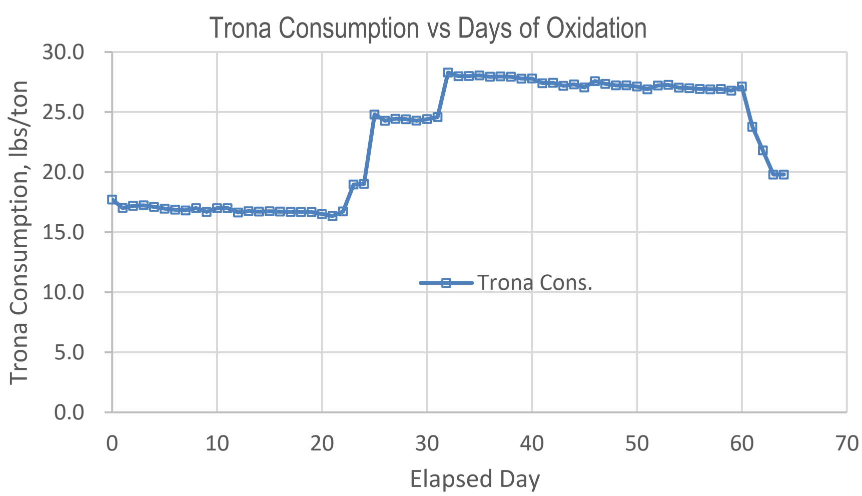


Figure 10-21: Trona Consumption of a 60-day Pre-oxidation Test (Brimstone Drill Composite)

The figure above shows trona consumption tracked for the entirety of column 62's pre-oxidation cycle. Addition is represented by steep jumps from one day to the next, small decreases in 'consumption' over time represent total alkalinity leaving the system as part of regular 50 ml sampling, and the steep decrease in consumption after 60 days of oxidation represents back-calculation of residual trona during rinsing.

As acid is generated by the oxidation reaction, trona/soda ash will be 'consumed'. This consumption occurs when the carbonate or bicarbonate of trona is converted to bicarbonate or carbon dioxide in order to neutralize the produced acid. Over time carbonate concentrations will need to be replenished, either by the addition of more carbonate containing reagents (soda ash), or by the addition of a hydroxide source (Caustic/Lime) that can convert bicarbonate to carbonate while raising the pH of the solution.

At the conclusion of the Phase 2 test work it was determined that soda ash would serve as a more efficient source of carbonate as it can deliver higher carbonate concentrations than trona and requires less mass to be moved and stored in order to provide the same total alkalinity. During Phase 3 an identical procedure was performed for each column with soda ash consumption being measured rather than trona. An example from Phase 3 is provided below:

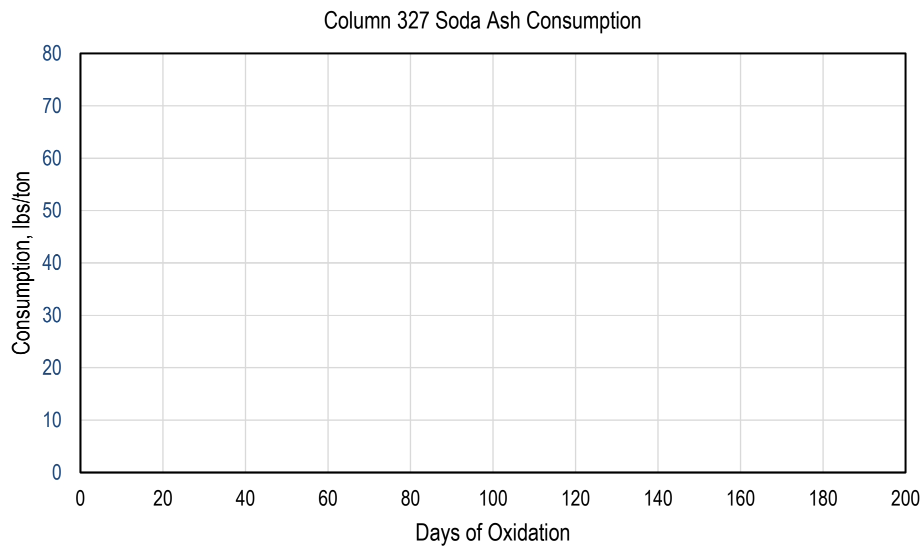


Figure 10-22: Soda Ash Consumption of a 180-day Pre-oxidation Test (Central Excavation)

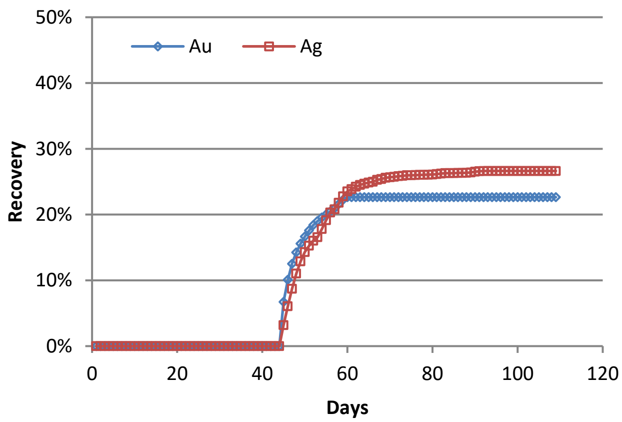
10.7.2 Cyanide

The utilization of sodium cyanide solution to leach pre-oxidized ore is no different than its utilization when leaching ore that has not been pre-treated. The projected consumption for sodium cyanide was generated by an average of the actual consumption across properly executed tests in the program. Tests that were executed before the invention of the 2-step oxidation and leach process were not included, nor were tests where pH control was not maintained for the duration of cyanidation. Residual cyanide in the pregnant solution is not considered as part of consumption as it can be recycled.

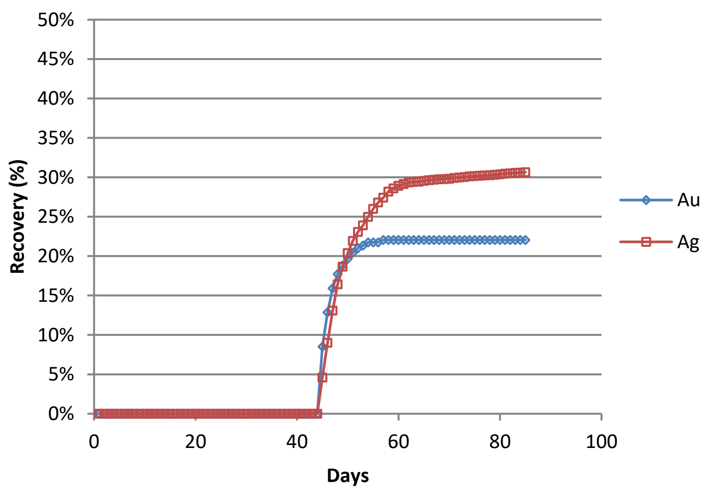
For the life of the mine, the average cyanide consumption is projected to be 1.0 lbs/ton according to the test work and the assumption that all cyanide applied to the sulfide heaps will be lost or consumed. This consumption was determined using the 1:1 solution to ore ratio for sulfide leaching and a 1.0 lbs/ton concentration of cyanide solution that was determined through a series of cyanide kinetics tests. Cyanide consumption for oxide heaps is conservatively noted as 1 lbs/ton though historical oxide heap leaching on the site has recorded consumptions of 0.6 lbs/ton or less.

The cyanide kinetics tests were performed on central ore of average grade. Several columns were created from one common bulk sample, which was crushed to P100 = ½" and blended. The columns were operated in parallel and oxidized for 30 days before being rinsed and leached by the same procedure. The intention was to note any difference in leach curves that may have resulted from insufficient cyanide concentration, a brief oxidation cycle was thought to be appropriate for this exercise as lengthy oxidations can yield short recovery curves which are more difficult to compare and more vulnerable to outliers. Cyanide concentrations between 0.5 lbs/ton and 2.0 lbs/ton were compared across the series. After 1 lbs/ton, increasing the concentration of cyanide yielded no significant increases in recovery while concentrations below 1 lbs/ton started to show a decline in silver recovery. The results are displayed below:

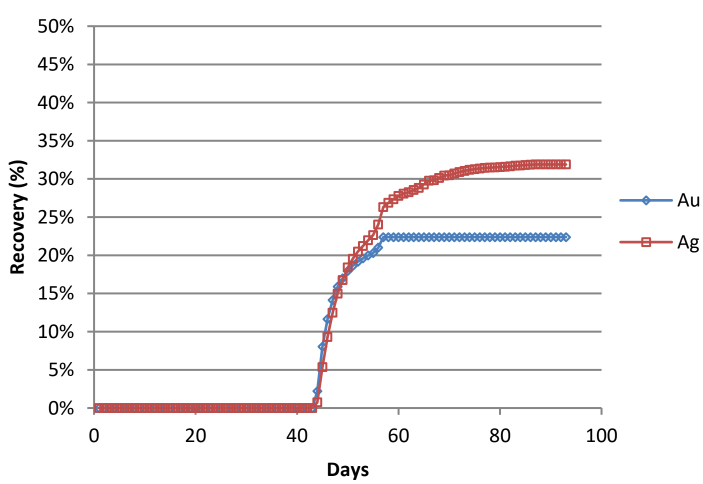
Central 0.5 lbs/ton CN



Central 1.0 lb/ton CN



Central 2.0 lbs/ton CN



Sodium cyanide loss has been observed for solution systems that contain high amounts of bicarbonate; while the mechanism is unclear repeatable experiments have consistently shown the incompatibility of trona and sodium cyanide in solution. As a result, process controls have been established to separate carbonate containing solutions from cyanide containing ones. These controls offer the upside that some cyanide may be conserved, and overall consumption reduced; for the purposes of this study the expected effects of these controls is ignored.

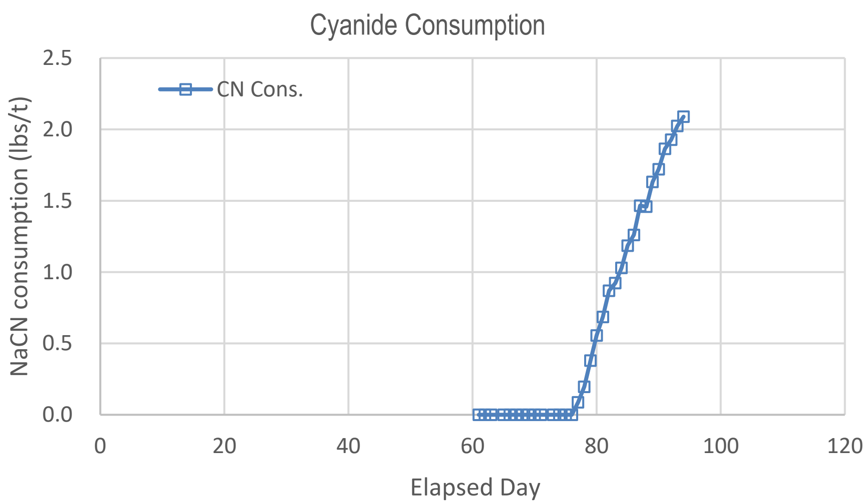


Figure 10-23: Cyanide Consumption During Rinse and Leach (Brimstone Drill Composite)

The figure above shows the consumption of cyanide for column test 61. Cyanide is only added to the column at the conclusion of the rinse. The consumption is calculated by taking the difference of the concentration in the barren solution and the pregnant solution.

Sodium cyanide is stabilized by manufacturers through the addition of sodium hydroxide. A common composition for this reagent is a 30% solution of NaCN which will also contain 3% NaOH. Sodium hydroxide is not evaluated as a consumed reagent as it is included in the cost of sodium cyanide. In the occasion that lixiviant with low cyanide concentration is diverted to the pre-oxidation ponds, the residual hydroxide concentration will serve to regenerate carbonate concentrations and thus reduce consumption of the carbonate source reagent.

10.7.3 Lime

Lime will be coupled with Sodium cyanide to form the lixiviant solution that will drive metal recovery during cyanidation. Lime acts as a hydroxide source in solution that maintains a high enough solution pH to prevent the loss of cyanide to HCN gassing. Lime will offset any additional acid generated during the leach cycle.

In addition to its role in the lixiviant solution, saturated lime solution will be used as a rinsing agent upon completion of the pre-oxidation cycle. Lime solution will be used to push out and dilute carbonate solutions prior to the addition of cyanide to a panel. This lime solution will be diverted to the carbonate solution ponds where it will serve to regenerate carbonate concentration from bicarbonate that has built up.

The consumption of lime when used for the cyanidation of pre-oxidized ore is considerably lower than when it is used to leach un-pretreated ore. For the life of the mine, the average lime consumption is projected to be 1.26 lbs/ton. This was calculated by a weighted average between the test work values for pre-oxidized ore and historical values for the oxide and transition ores that will not be pre-oxidized.

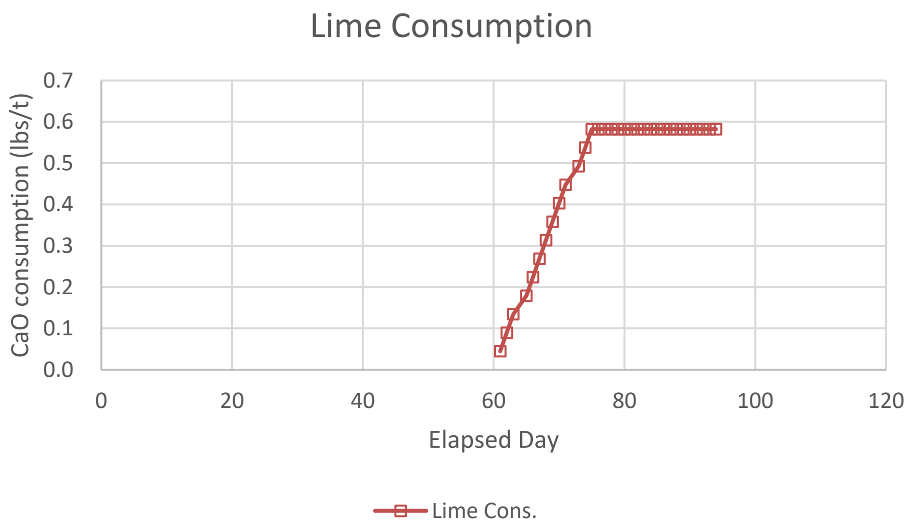


Figure 10-24: Lime Consumption During Rinse and Leach (Brimstone Drill Composite)

The figure above shows the consumption of lime for column test 61. The majority of lime addition/consumption is done during the rinse stage of the process. After cyanidation has commenced, additional lime was rarely required for any test as the NaOH provided by cyanide solution was able to neutralize residual acid generation and maintain pH. The complete reagent consumption of column 61 is illustrated below in Figure 10-25.

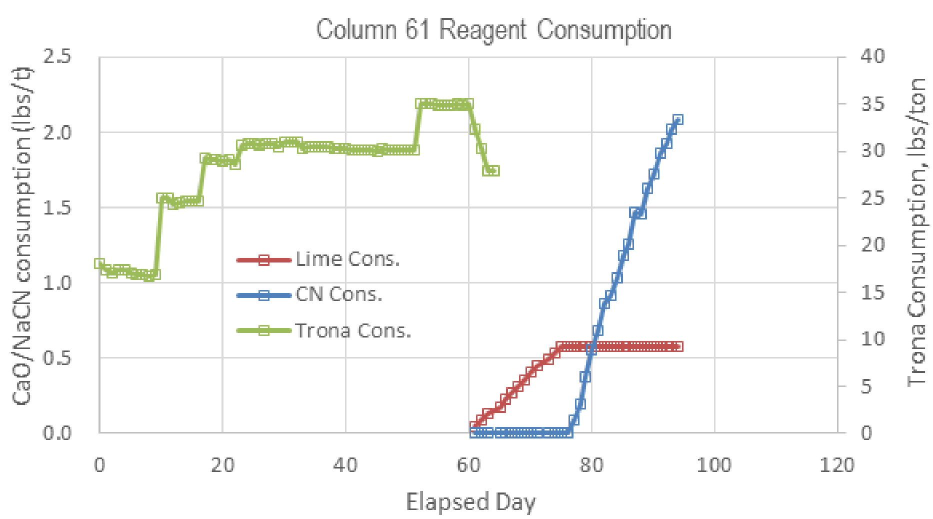


Figure 10-25: Column 61 Reagent Consumption (Brimstone Drill Composite)

10.8 METALLURGICAL PARAMETERS FOR PROCESS DESIGN CRITERIA AND FINANCIAL ANALYSIS

Metallurgical testing is ongoing. Based on the results available so far, the projected recoveries have not changed much from the Phase 2 testing. Table 10-8 is a summary of the operating parameters and metal recoveries proposed for heap leach modelling to develop a metal production schedule.

From the overall trend observed so far in the test results, it appears that gold recoveries of 70% are possible for all the domains if the conditions are right. It is recommended that testing be continued using optimal conditions to provide experimental support for this recovery target. These optimal conditions include soda ash dosage, crush size, oxidation time, maintaining moist conditions during oxidation and ensuring access to air. During operations, testing of ore is likewise recommended to fine tune the conditions to be used in the heap. The duration of the oxidation cycle is variable and dependent on parameters found in the head assay.

Table 10-8: Operating Parameters and Expected Recoveries for Heap Leaching

Domain	Nominal* Target Oxidation, %	CN^- Leach Time, days	Au Recovery, %	Ag Recovery, %
Northwest (Bay)	31	60	55	55
West (Central)	40	60	70	70
Southwest (Camel) Above Water Table	40	60	70	70
Southwest (Camel) Below Water Table	40	60	65	70
Brimstone	40	60	65	70
Vortex	40	60	65	70

*Oxidation targets will vary depending on Au_{CN}/Au_{FA}

Maximum recoveries can be attained if the correct operating conditions are observed, including the following:

1. It is essential that pH be maintained above 9.5 during the oxidation process but not higher than 11. This ensures that the catalytic action of the ferrous-ferric carbonate redox pair is prevailing.

2. The total carbonate alkalinity must be maintained at a minimum of 20,000 ppm, preferably up to 60,000 ppm to stabilize enough iron in solution.

3. During oxidation, the ore must be maintained wet because the catalytic oxidation reaction involves dissolved iron species.

4. However, the heap must not be saturated with solution to allow oxygen to migrate to the oxidation sites. Oxygen regenerates Fe(II) carbonate to Fe(III) carbonate.

5. When the desired oxidation level is attained, excess carbonate and bicarbonate must be rinsed out of the heap. This may be followed by a lime water rinse to neutralize any residual carbonate. This step is crucial to minimize cyanide consumption during the leach stage.

Maintaining permeability in the heap is important during both oxidation and leach stage.

10.9 METALLURGICAL TESTING IN PROGRESS

As mentioned earlier, three 20-ft columns and three large-scale columns using the old carbon columns (CIC) are in progress. Also, tails assays are pending for three of the columns that were not included in this report. At the conclusion of these tests and data analyses, M3 will prepare a technical memorandum, which will serve as an addendum to this Technical Report Summary.

11 MINERAL RESOURCE ESTIMATES

HMC retained SRK Consulting (U.S.), Inc. to complete a mineral resource estimate for the Hycroft Project (the Project, or Hycroft). This Technical Report Summary provides a mineral resource estimate and classification of resources reported in accordance with the SEC New Mining Rules. SRK worked with Hycroft to construct updated 3-D wireframes for alteration and oxidation zones using Leapfrog Geo software. Estimation of gold, silver, sulfide sulfur, and rock hardness in a 3-D block model was completed by Tim Carew, P. Geo., of SRK, with GEMS software.

The methods and results of resource estimation are reported below and correspond to the final version of the SRK 3-D block model, <HY20190617.bmf>. Updated Measured and Indicated classification was provided to append to the existing block model, in < Hycroft_Classification_ToHycroft_011020.csv>.

The estimates of Mineral Resources may be materially affected if mining, metallurgical, or infrastructure factors change from those currently assumed at Hycroft. Estimates of inferred mineral resources have significant geological uncertainty and it should not be assumed that all or any part of an inferred mineral resource will be converted to the measured or indicated categories. Mineral resources that are not mineral reserves do not meet the threshold for reserve modifying factors, such as estimated economic viability, that would allow for conversion to mineral reserves.

11.1 BLOCK MODEL DIMENSIONS

A regular block model was defined in GEMS to cover the volume of interest, with no rotation applied. A regular block model in GEMS has a single material type per block, and sub-blocking is not utilized. Relative to the previous model, the framework was extended to the west to include the Crofoot leach pad. A block size of 40x40x40 ft was selected based on drill hole spacing and proposed bench height.

11.1.1 Block Model Geometry

The block model geometry is summarized in Table 11-1.

Table 11-1: Block Model Geometry – Coordinates in Hycroft Mine Grid, feet

	Minimum	Maximum	# Blocks	Block Size
Easting	13000	26000	325	40
Northing	35440	54800	484	40
Elevation	2200	6600	110	40

Source: SRK, 2018

11.2 DATA COLLECTION

Data sets used in the Mineral Resource estimation include RC, core, and Sonic drill hole data, topographic surface data, and material density data.

11.2.1 Drilling Data

SRK's collar table for modeling includes 5,501 drill holes, with a total of 2,482,722 drilled feet, and average depth of 455 feet. Of these, 5,257 drill holes are in the model domain and were considered for resource estimation. The drill holes in the model domain have 2,351,634.1 total feet drilled, and average depth of 451 feet. About 1,970 of these drill holes were completed by HMC. The average total depth of this group is 720 feet. SRK's downhole survey dataset for modeling includes all 5,501 drill holes in the collar table. SRK appended available data to the collar and survey tables initially provided by HMC, during the process of assay data compilation.

11.2.2 Topographic Data

The most recent aerial survey was completed by Aero-graphics Geospatial Services of Salt Lake City, Utah in August 2015, and has been used as the base topography. The wire-frame triangulation surface has been updated by HRDI survey staff to reflect mining through the end of July 2015, based on actual mining surveys. The surface, "2015_topo.00t", has been used to define the air/rock interface in the block model.

Note that the tonnages and grades reported in the Resource Statement in Section 14.10 have been adjusted for material mined up to the end of June, 2019, based on mining surveys provided by HMC.

11.2.3 Tonnage Factor Density

HMC provided the available density data in <Master Density 2010.xlsx>. The file includes one tab with raw density data for 884 samples by Sample ID, lithology, and alteration, and a tab with average density values summarized by material type. Available data included bulk density values collected during mining, in addition to data from core sample testing. Laboratory values reported are Specific Gravity, in grams per cubic centimeter. Calculated tonnage factors, as cubic feet per ton, were also included in the dataset. SRK saved a copy of the raw data as a CSV file to import to modeling software, and added a field for drill hole ID.

Backfill, alluvium and the sulfide stockpiles have lower density than undisturbed rock. In the volcanic bedrock, alteration overprint is the main control on density. Tonnage factors were inverted to generate density values in tons per cubic foot. These density values were assigned to the model blocks by material type. Table 11-2 lists all density values used for estimation.

Table 11-2: Hycroft Tonnage Factors

Geologic Zone	Tonnage Factor (ft^3/ton)	Density (ton/ft^3)
Alluvium	18.00	0.0556
Backfill or Dump	20.00	0.0500
Acid Leach Alteration	14.90	0.0671
Sulfide Stockpiles	18.80	0.0550
Silicic Alteration	12.74	0.0785
Propylitic Alteration	14.43	0.0693
Argillic Alteration	15.34	0.0652
Unaltered/Undefined/All Other Geologic Zones	14.25	0.0702

11.3 GEOLOGIC MODELS

The geological framework for SRK's resource estimate was generated by Hycroft and SRK geologists. Modeled faults and lithology were adopted from HMC by SRK. Alteration and extent of oxidation were modeled by SRK, and include interpretations provided by HMC.

11.3.1 Structural Model

There are 27 modeled faults in the Hycroft model area. Some of them offset lithology or act as barriers, and some control mineralization as conduits for hydrothermal fluids. SRK adopted the Hycroft-modeled faults to use as the structural framework for the geological model and resource estimate.

11.3.2 Lithology and Formation Model

The Hycroft deposit is hosted in Tertiary-age volcanic units that are in fault contact at depth with Jurassic metasediments of the Auld Lang Syne (ALS) Formation. Quaternary alluvium and fill conceal a large portion of the deposit. The logged and modeled formations in the Hycroft deposit areas are listed in Table 11-3.

Table 11-3: Logged and Modeled Formation Codes

forma	Formation	Model
Dmp	Dump	Fill
Qal	Alluvium	Qal
Qls	Lacustrine Sediments	
Tal	Older alluvium	
Tc	Camel conglomerate	Tcm
Tch	Camel	
Tcp	Camel	
Tsg	Sulfur Group	Tsg
Td	Dacite (Kamma age)	--
Tk	Kamma volcanics	Tk
Vein		--
Ja	Auld Lang Syne Formation	ALS

Source: SRK, 2017

Hycroft's lithology and formation model includes the units in the last column of Table 11-3 except for Fill, which was coded separately. Formation solids are bound and offset by the following faults:

- Break
- Camel
- Central
- Cliff
- East
- Hades
- Ramp
- Range

SRK adopted the modeled formation and lithology solids for resource estimation. Generally, the fault and stratigraphic contacts compare well to drill hole data. SRK noted an east-dipping trend in the Central and Camel deposit areas, and a barren zone between upper and lower deposits in this area. The barren assay intervals coincided with logged lacustrine sediments rich in clay of the Tsg unit. SRK added a mineralization domain boundary along the barren, clay-rich zone, and adopted the sedimentary bedding geometry to guide interpolation.

11.3.3 Alteration Model

Hycroft provided the following alteration solids:

- Acid Leach
- Argillic
- Propylitic
- Silicic

Hycroft alteration solids generally honored drill hole contacts, and were based on east-west cross sections on 175-foot spacing. Consequently, the boundaries were coarse between sections, and some of the interpreted contacts on the margins of the model area were inconsistent. SRK re-interpreted alteration and rebuilt alteration wireframes. SRK used Leapfrog 3-D software to model alteration solids based on available drill hole data. The resulting solids were verified against the original solids, where available, and compared to logged drill hole lithology. The new modeled alteration solids generally honor the drill hole data and include consistent anisotropy in the interpreted boundaries. The resulting meshes are valid solids that were used for coding and ultimately density, metal grade and sulfur estimation. The new SRK alteration model did not depart dramatically from the Hycroft alteration model, and will therefore result in similar material quantities compared to the previous model. The areas with different modeled alteration have low drill hole density, and therefore higher uncertainty in material type. Pit slope parameters may differ in future pit designs if there are differences in the extent of silicic and argillic alteration.

11.3.4 Sulfide and Oxidation Model

Hycroft provided an interpreted upper extent of sulfide as a surface generated from east-west cross sections on 100-foot spaced northings, <sulf_ew100_final.00t>. Rather than geological parameters, the interpretation for this boundary was based on the ratio of cyanide-soluble and fire assay gold (Au CN:FA) in assay samples. Sulfide material has Au CN:FA equal to or less than 0.3. In mineralized areas, the ratio approach is valid to define metallurgical material types. However, where gold values are near or below method detection limits and ratio values are invalid, Hycroft's previous interpreted surface was not constrained by any material properties or logged geologic data.

SRK considered this modeled boundary when refining the interpretation to define metallurgical material types. However, SRK's metallurgical material boundary differs locally from that generated by Hycroft. SRK considered logged mineralogy and intensity of sulfide mineralization and later oxidation when interpreting the metallurgical material boundary in areas without Au CN:FA values. SRK's ppm assay database provided higher resolution in low grade areas. Samples with at least 0.010 ppm (0.000292 opt) fire assay gold had valid Au CN:FA values applied to modeling.

SRK added a calculated field to the assay table for Au CN:FA values, to use for material type modeling based on gold leachability. The samples with CN:FA > 1 can be attributed to analytical uncertainty in most cases. Based on the assay data and calculated ratio values, a material type category was assigned to each sample interval, according to Table 11-4. The ppm assay dataset has valid ratio values for 196,109 of 441,380 samples (44.4%) with paired gold values. The samples that have fire assay gold values below or near the method detection limit constitute 38.4% of the intervals with paired values. The majority of these are from values reported at <0.001, 0.001, or 0.002 opt; those reported in ppm have gold fire assay values less than the lower MDL. The ppm dataset maximized the number of samples with valid ratio values.

Table 11-4: SRK Metallurgical Material Types

AuRecoveryType	Explanation
BDL	One or both Au values below method detection limit
NoData	One or both Au values missing data, null or noted in database as -9
Zero	One or both Au values = 0
1xMDL	Fire assay = 0.001, analytical uncertainty too high for meaningful ratio
2xMDL	Fire assay = 0.002, analytical uncertainty too high for meaningful ratio
HighLeach	Au CN:FA >= 0.70
Transition	Au CN:FA between 0.30 and 0.70
Refractory	Au CN:FA <= 0.30

Source: SRK, 2017

Using implicit modeling software, boundaries between high-leach, transition, and refractory material were generated. Hycroft's top of sulfide surface was applied to assign high-leach above and refractory below as defaults. The interpreted sulfide surface was especially helpful in areas of low gold grades that lacked valid ratio values. Then, material types in drill holes were used to generate boundaries of high-leach, transition, and refractory zones with greater resolution than the initial assignment. With this approach, the interpreted sulfide boundary was reflected in the model, and drill hole data was honored locally. The modeled high-leach and transition material is mostly shallow, and the modeled volumes are mostly above or coincident with Hycroft's interpreted top of sulfide surface. Some deep material in the East Fault zone has Au ratio values in the high leach or transition ranges; however, most of the material at depth is unoxidized, as defined by ratio values and logged sulfides.

11.3.5 Grade Domains

Grade shells were used to separate populations of grade values and spatially constrain estimated values. All blocks received estimated values for gold, total silver and cyanide silver, if the drill hole data were sufficient. Blocks inside the respective grade shells were estimated with composites inside; likewise, blocks outside the grade shells were estimated with composites outside. This approach was used for total gold, total silver, and cyanide-soluble silver estimation. For each element, assay values were capped with generalized values, then composited to 20 ft lengths. The composited values were used to generate meshes around intervals that exceed the respective grade threshold. Selected faults were applied as hard domain boundaries to segregate composites. Anisotropy parallel to bedding or the East Fault was applied by domain. Except for the mineralization in the East Fault Zone, the anisotropy of the grade shells was generally flat, and approximately equal in the X and Y directions. Anisotropy in the East Fault Zone follows the dip of the fault and rake of mineralization and has a steeper dip than the other estimation domains.

Gold grade shells at 0.070 ppm were built for all areas of the model. The gold threshold of 0.070 ppm, equivalent to 0.002 opt, is the lowest value that is supported by the older assay data reported to 0.001 ounces per ton. This gold value is essentially equivalent to the current economic cut-off grade for Hycroft leach ore, 0.073 ppm (0.002 opt). Although the grade shell threshold is limited by analytical data, some areas are defined by data with higher resolution.

In the Central deposit, within the SW Upper domain, there is a group of samples with high gold values that were initially targeted for capping. However, in the opinion of the QP, the gold values in these samples appear geologically supported, and capping these grades would be overly conservative. SRK built grade shells at a 1ppm threshold in this domain to apply the reported values while limiting their influence on nearby interpolated values.

Compared to gold grade shells, the extent of silver grade shells is small. This is due to a lack of available data in older drill holes. For all domains, silver grade shells were built at a 10-ppm threshold using total silver (AgFA) data. Some domains have sparse silver data, and a combination of cyanide-soluble and total values. AgCN grade shells were built for the model areas west of the Break Fault, where the most cyanide-soluble silver values occur without total silver values.

The economic cut-off grade for silver in heap leach ore is 19.429 ppm (0.567 opt). Relatively little of the resource is carried by silver alone. However, the gold equivalent grade includes the contribution of silver to the block value. Silver values are highest in the Vortex deposit at depth, and can make blocks economically viable despite gold grades below cut-off.

Hycroft and SRK identified an opportunity in areas with assays reported in ppm to apply grade shells at a lower threshold in an additional estimation step. This approach yielded less dilution in estimated grades immediately outside of the 0.070 ppm grade shells. The assay data reported in ppm units has more resolution at low values than the opt data. To maximize the value of blocks with ppm data, additional grade shells at 0.050 ppm (0.001 opt) were built using only data reported in ppm. These additional grade shells were limited to areas with relatively recent drilling, and were relevant to the south deposit areas, particularly at depth. Generally, these grade shells form a rind on the other set.

Blocks inside the 0.050 ppm grade shell and outside the 0.070 ppm grade shell were estimated for gold, only with assays reported in ppm. The additional grade shells separate the assay populations further, and constrain estimated values in space.

Without the secondary shell, estimated grades would be diluted by low values and/ or lack of drilling support outside the main grade shell. The secondary shell threshold is ten times the method detection limit of 0.005 ppm. For reported values lower than ten times MDL, the inherent analytical uncertainty becomes proportionally larger, and results in less accuracy of reported values. The threshold of 0.050 ppm is the lowest value supported by the existing dataset reported to 0.005 ppm.

11.3.6 Geo-metallurgical Domains

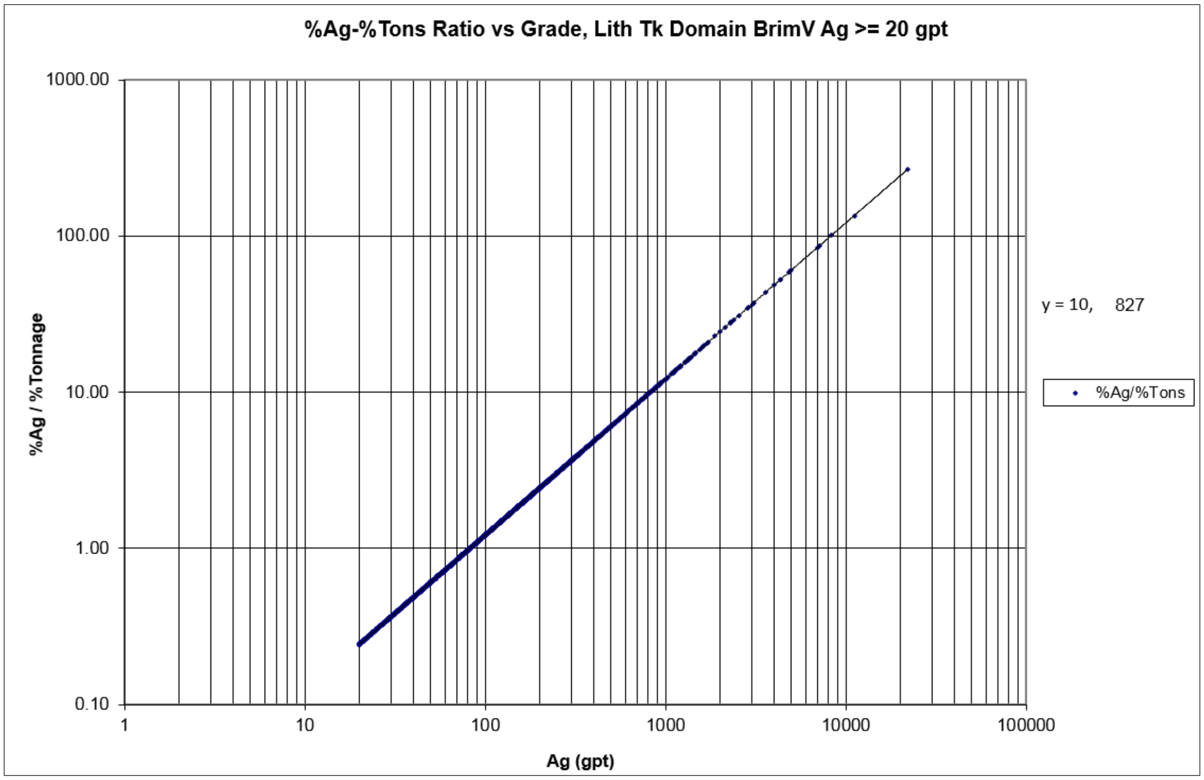
A geo-metallurgical model was generated to support mine planning and operations. Considerations for defining geo-metallurgical domains included gold to silver ratio, alteration, oxidation, depth to water table, and sulfide mineralogy and abundance. The domains are based on the known alteration and mineralogy in the deposit areas and defined by major faults.

11.4 ASSAY CAPPING AND COMPOSITING

Assay values were capped prior to compositing, as described below.

11.4.1 Capping

A capping analysis was conducted on gold and silver assay values by domain prior to compositing to determine suitable capping values to minimize the effect of outlier values. The analysis was based on a ‘metal-at-risk’ approach, in conjunction with examination of log probability plots of the domain distributions that identifies breaks in the distribution corresponding to high-grade outlier populations. The metal-at-risk approach compares the gold metal contribution of each sample (ratio of length*grade of sample to the sum of those products for all samples) to its tonnage contribution (ratio of sample length to total length of sample), starting from the highest-grade sample, as illustrated in Figure 11-1. The graph plots the ratio of %contribution to metal and %contribution to tonnage of samples as a function of their grade. In this case (silver samples in the Kamma Volcanics of the Brim Vortex domain) it shows that the highest-grade sample (22200 PPM) contributes $\approx$ 270 times more to the metal than tonnage. In terms of risk, it is generally accepted that the metal contribution of a sample should not be more than 10 times its tonnage contribution. At the 10:1 ratio threshold, the analysis suggests using a capping value of $\approx$ 830 PPM for silver in this domain/lithology.



Source: SRK, 2018

Figure 11-1: Metal%-Ton% Ratio - Brim Vortex Kamma Lithology, Ag Fire Assay

Based on the capping analysis, the capping thresholds for gold and silver are summarized in Table 11-5 and Table 11-6.

Table 11-5: Gold Assay Capping Analysis Results

Lith	Count	Min	Max	Mean	Cap PPM	Cap OPT	Capped	%Samp	% Metal Lost
Break North Tcm	25679	0.0001	12.72	0.29	5.0	0.146	41	0.16%	1.5%
Break North Tsg	4212	0.0001	1.67	0.04					
Break South Tcm	20923	0.0001	7.05	0.13	3.0	0.087	15	0.07%	0.6%
Break South Tk	4760	0.0001	17.65	0.16	3.0	0.087	4	0.08%	2.7%
Brim Vortex Tcm	39773	0.0001	31.10	0.14	3.4	0.099	36	0.09%	2.2%
Brim Vortex Tk	114161	0	108.00	0.29	4.3	0.125	241	0.21%	2.9%
Camel Tbg	837	0.0001	0.17	0.01					
Central North Tcm	56779	0.0001	20.33	0.42	6.0	0.175	43	0.08%	0.6%
Central North Tsg	4047	0.0001	3.70	0.02	3.0	0.087	1	0.02%	0.8%
Central South Tcm	104717	0.0001	48.30	0.26	4.3	0.125	242	0.23%	2.8%
Cliff Tsg	48	0.0001	0.07	0.01					
Foothills Als	1932	0.0001	20.00	0.12	3.0	0.087	3	0.16%	6.2%
Foothills Tk	12055	0.0001	5.98	0.08	3.0	0.087	9	0.07%	0.9%
Hades Tcm	304	0.0025	0.19	0.01					
Leach Pad Undiff	14622	0.0001	3.75	0.07	3.0	0.087	5	0.03%	0.2%
Range North Tcm	19	0.0025	0.10	0.02					
Range North Tsg	911	0.0001	0.19	0.01					
Range South Tcm	8009	0.0001	8.23	0.14	3.3	0.096	1	0.01%	0.4%

Source: SRK, 2018

Table 11-6: Silver Assay Capping Analysis Summary

Lith/Flt	Count	Min	Max	Mean	Cap PPM	Cap OPT	Capped	%Samp	% Metal Lost
Break North Tcm	25679	0.001	3220.55	2.32	820	24	3	0.01%	11.5%
Break North Tsg	4212	0.001	1739.99	2.58	820	24	1	0.02%	6.9%
Break South Tcm	20923	0.001	3160.00	3.96	770	22	2	0.01%	3.0%
Break South Tk	4760	0.001	1260.00	6.33	510	15	1	0.02%	2.5%
Brim Vortex Tcm	39773	0.001	4310.00	4.05	640	19	11	0.03%	4.9%
Brim Vortex Tk	114161	0.001	22200.00	9.54	830	24	84	0.07%	8.4%
Camel Tbg	837	0.001	13.02	2.56					
Central North Tcm	56779	0.001	2270.00	1.19	608	18	5	0.01%	3.5%
Central North Tsg	4047	0.001	183.00	1.99	406	12	0	0.00%	0.0%
Central South Tcm	104717	0.001	4069.99	3.04	720	21	17	0.02%	4.3%
Cliff Tsg	48	0.001	2.50	0.05					
Foothills Als	1932	0.001	613.00	5.87	856	25	0	0.00%	0.0%
Foothills Tk	12055	0.001	661.00	2.74	550	16	1	0.01%	0.3%
Hades Tcm	304	2.5	15.00	2.86					
Leach Pad Undiff	14622	0.001	740.23	3.01	565	16	1	0.01%	0.4%
Range North Tcm	19	0.05	5.00	2.01					
Range North Tsg	911	0.001	17.00	3.03					
Range South Tcm	8009	0.001	1860.00	6.29	935	27	4	0.05%	4.4%

Source: SRK, 2018

11.4.2 Compositing

The capped assay data for fire assay gold and silver was composited as 20 ft equal length composites starting at the drill hole collar, and broken at the corresponding gold and silver grade shell contacts. Any short residual intervals created in this process were merged into the previous interval. Composite intervals external to the grade shells were assigned a background rock type for grade estimation purposes.

11.5 VARIOGRAM ANALYSIS AND MODELING

The spatial continuity, or autocorrelation, of composites within the grade shell domains (and external to the grade shells) was investigated through variographic analysis using the SAGE 2001 variography package.

Down-the-hole correlograms were calculated to estimate appropriate nugget values, in addition to experimental 3D directional correlograms for use in variogram modeling. The nugget value is a representation of the inherent variability within any given domain. SRK has utilized a correlogram which measures the correlation coefficient between two sets of data, comprising values at the heads and values at the tails of vectors with similar direction and magnitude, and has been found to provide a stable estimate of spatial continuity. For ease of modelling, the correlogram value is subtracted from one and is presented in a similar graphical form to the variogram. In this report the correlograms presented this way are referred to as variograms.

Variogram modeling refers to the fitting of smooth mathematical models (curves) to the experimental variograms generated in the analysis process. Variogram models form the mathematical basis for the estimation process via the grade interpolation methods, e.g. kriging.

The nugget effect is modeled as a vertical offset at the origin of the fitted model. In the modelling process the fitted models for gold have been typically modelled using a stacked variogram with two components, namely an exponential component and a spherical component, whereas the models for silver consisted of a nugget effect and a single exponential component. The exponential and spherical components are defined by their 3D orientation, range and sill value. The range is defined by the distance where the model first levels out. Sample locations separated by distances closer than the range are spatially autocorrelated, whereas locations farther apart than the range are not. The sill is defined by the value that the variogram model attains at the range (the value on the y-axis). In this case each of these stacked components has a partial sill, which together with the nugget effect make up the total sill, which by definition is equal to one for a correlogram. The variation in 3D orientation reflects differences in spatial continuity at different ranges.

The rotation convention used in Table 11-7 is the GEMS rotation convention, with reference to the cartesian coordinate system. The order of the rotation axes is denoted by the order of the letters, i.e. ZYZ. Rotations around the axes follow the right-hand (RH) rule. In Table 11-7 component 1 refers to the first range of continuity and component 2 represents the final ranges and sill component.

11.5.1 Variography Parameters

Variogram parameters for gold and silver composites within grade shells are tabulated in Table 11-7.

Table 11-7: Variogram Parameters by Grade Shell

		Component 1								Component 2							
Gradeshell	Nugget	Type	Sill	Rotation* (Deg)			Range (Ft)			Type	Sill	Rotation* (Deg)			Range (Ft)		
				Z	Y	Z	X	Y	Z			Z	Y	Z	X	Y	Z
Gold																	
SW Upper	0.249	EXP	0.497	88	-3.1	-37	310	130	100	SPH	0.209	17.3	13.2	-16.9	56	2150	220
SW Lower	0.240	EXP	0.760	0.6	22.7	8.9	299	740	205	-	-	-	-	-	-	-	-
Lewis	0.284	EXP	0.288	-52	4	-48	370	135	105	SPH	0.128	-19	-1	-9	3960	1510	170
East Fault	0.300	EXP	0.591	-12	43	-68	110	140	260	SPH	0.109	8	-27	-45	850	3550	850
Brim Vortex	0.260	EXP	0.563	-23	-16	17	270	290	130	SPH	0.177	9	-14	37	1870	1220	490
Bay	0.284	EXP	0.288	-52	4	-48	370	135	105	SPH	0.128	-19	-1	-9	3960	1510	170
Background	0.240	EXP	0.760	-4	-90	-20	290	1530	1120	-	-	-	-	-	-	-	-
Silver																	
Brim Vortex	0.440	EXP	0.560	11	-38	65	150	500	140	-	-	-	-	-	-	-	-
Background	0.480	EXP	0.520	7.3	-10.4	78.1	1040	640	430	-	-	-	-	-	-	-	-

Source: SRK, 2018

11.6 ESTIMATION METHODOLOGY

Block grades were estimated by domain (grade shell) for fire assay gold (AuFaPPM) and silver (AgFaPPM), cyanide soluble silver (AgCnPPM), cyanide soluble gold to fire assay gold ratio (AuCnFAR), sulfide sulfur percentage (S), and hardness, using either Ordinary Kriging (OK) or Inverse Distance Cubed (ID3). The interpolation process utilized 20ft composites tagged with corresponding material types to enable the use of either hard boundaries, to prevent smearing across boundaries, or soft boundaries that allow the influence of some composites external to the grade shell to be used within a certain tolerance. The interpolations were done in three passes, with progressively larger search distances and progressively relaxed requirements in terms of minimum number of samples and maximum number of samples per drill hole, and with protection of blocks estimated in earlier passes.

11.6.1 In Situ Material

Gold fire assay (AuFA) estimates were interpolated by grade domain using OK and with matching of block domain coding with composite domain coding, i.e. utilizing hard boundaries. As described in Section 11.3.5, grade shells at a threshold of 0.050 ppm were developed in addition to the primary 0.070 ppm grade shells. These shells generally form a rind on the other set and were limited to areas with relatively recent drilling in the south deposit areas, particularly at depth. Composites used for estimation of these blocks were restricted to those with data assayed in ppm.

Cyanide soluble gold (AuCN) estimates were developed by first estimating the gold cyanide/fire assay ratio (AuCNFAR) block values and then calculating the corresponding AuCN block values by manipulation. Ratio estimates by grade domain were interpolated using the AuFA interpolation schema.

Silver fire assay (AgFA) and silver cyanide soluble (AgCN) estimates were interpolated using OK for the Brim-Vortex and Background (external to grade shells) domains, and ID3 for other domains. Hard boundaries were used between individual domains defined by the grade shells, but a 'soft' boundary approach was used between the grade shell domains and the domain defined by the background (external to the grade shells) data. The approach was based on comparison of modeled grades with available production (blasthole) data, particularly in areas where drill holes had not been sampled for silver, which restricted the continuity of the primary silver grade shells. When interpolating blocks in the background domain the interpolation allows the use of composites from adjacent primary grade domains, but adjusts the apparent distance to the sample (and hence the influence of the sample) by a factor between 0 (no influence) and 1 (full influence). A suitable factor for this purpose was selected by comparing estimates based on iterative variation of the factor, to block estimates based on production data.

Gold equivalent (AuEQ) block values were calculated from AuFA and AgFA block estimates by block manipulation using the ratio of corresponding metal prices:

$$AuEQ = AuFA + AgFA \times (Gold\ Price/Silver\ Price)$$

Sulfide sulfur (LECO%) block estimates were interpolated by alteration and grade shell domain, using ID3 and 20ft composites generated by intersection with the grade domain grade shells and back-tagged by alteration domain to allow hard boundaries to be used between alteration domains.

Hardness block estimates were interpolated using nearest Neighbor (NN) interpolation and 20ft composites based on drill hole intersections with the 3D alteration solids used to code the Alteration model. Composites falling inside these solids were tagged with the corresponding alteration code and hard boundaries between alteration domains was used, with an un-rotated anisotropic search ellipsoid of 400ft x 400ft x 100ft.

The main interpolation parameters are for gold, silver and sulfide sulfur are tabulated in Table 11-8 to Table 11-10, and the gold search parameters (Pass 1) are illustrated in Figure 11-2.

Table 11-8: Gold Interpolation Parameters by Grade Shell

Pass	Shell	Method	Sample Support			Search Neighborhood					
			Min	Max	Max/Hole	Az/Dip (X)	Az/Dip (Y)	Az/Dip (Z)	Range (X)	Range (Y)	Range (Z)
1	SW Upper	OK	3	12	2	39/2	309/2	181/87	210	90	70
	SW Lower	OK	3	12	2	80/-22	351/3	89/67	100	200	60
	Lewis	OK	3	12	2	190/-2	100/-3	142/86	240	90	70
	East Flt.	OK	3	12	2	175/-15	73/-39	102/47	90	70	170
	Brim Vortex	OK	3	12	2	96/15	7/-5	293/74	180	190	90
	Bay	OK	3	12	2	190/-2	100/-3	142/86	240	90	70
	External	OK	3	12	2	256/85	4/2	274/-5	90	350	230
2	SW Upper	OK	2	12	1	39/2	309/2	181/87	420	180	140
	SW Lower	OK	2	12	1	80/-22	351/3	89/67	200	400	120
	Lewis	OK	2	12	1	190/-2	100/-3	142/86	480	180	140
	East Flt.	OK	2	12	1	175/-15	73/-39	102/47	180	140	340
	Brim Vortex	OK	2	12	1	96/15	7/-5	293/74	360	380	180
	Bay	OK	2	12	1	190/-2	100/-3	142/86	480	180	140
	External	OK	2	12	1	256/85	4/2	274/-5	100	400	260
3	SW Upper	OK	1	12	-	39/2	309/2	181/87	630	270	210
	SW Lower	OK	1	12	-	80/-22	351/3	89/67	300	600	180
	Lewis	OK	1	12	-	190/-2	100/-3	142/86	720	270	210
	East Flt.	OK	1	12	-	175/-15	73/-39	102/47	270	210	510
	Brim Vortex	OK	1	12	-	96/15	7/-5	293/74	540	570	360
	Bay	OK	1	12	-	190/-2	100/-3	142/86	720	270	210
	External	OK	1	12	-	256/85	4/2	274/-5	150	600	390

Source: SRK, 2018

Table 11-9: Silver Interpolation Parameters by Grade Shell

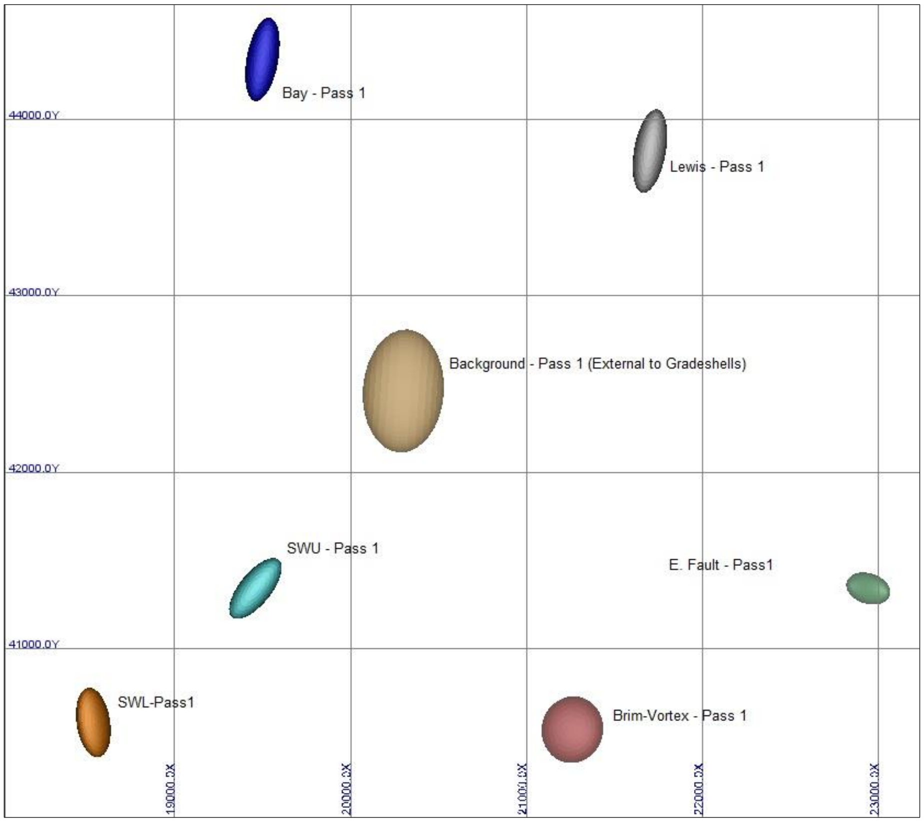
Pass	Shell	Method	Sample Support			Search Neighborhood					
			Min	Max	Max/Hole	Az/Dip (X)	Az/Dip (Y)	Az/Dip (Z)	Range (X)	Range (Y)	Range (Z)
1	SW Upper	OK	3	12	2	39/2	309/2	181/87	210	90	70
	SW Lower	OK	3	12	2	80/-22	351/3	89/67	100	200	60
	Lewis	OK	3	12	2	190/-2	100/-3	142/86	240	90	70
	East Flt.	OK	3	12	2	175/-15	73/-39	102/47	90	70	170
	Brim Vortex	OK	3	12	2	96/15	7/-5	293/74	180	190	90
	Bay	OK	3	12	2	190/-2	100/-3	142/86	240	90	70
	External	OK	3	12	2	256/85	4/2	274/-5	90	350	230
2	SW Upper	OK	2	12	1	39/2	309/2	181/87	420	180	140
	SW Lower	OK	2	12	1	80/-22	351/3	89/67	200	400	120
	Lewis	OK	2	12	1	190/-2	100/-3	142/86	480	180	140
	East Flt.	OK	2	12	1	175/-15	73/-39	102/47	180	140	340
	Brim Vortex	OK	2	12	1	96/15	7/-5	293/74	360	380	180
	Bay	OK	2	12	1	190/-2	100/-3	142/86	480	180	140
	External	OK	2	12	1	256/85	4/2	274/-5	100	400	260
3	SW Upper	OK	1	12	-	39/2	309/2	181/87	630	270	210
	SW Lower	OK	1	12	-	80/-22	351/3	89/67	300	600	180
	Lewis	OK	1	12	-	190/-2	100/-3	142/86	720	270	210
	East Flt.	OK	1	12	-	175/-15	73/-39	102/47	270	210	510
	Brim Vortex	OK	1	12	-	96/15	7/-5	293/74	540	570	360
	Bay	OK	1	12	-	190/-2	100/-3	142/86	720	270	210
	External	OK	1	12	-	256/85	4/2	274/-5	150	600	390

Source: SRK, 2018

Table 11-10: Sulfide Interpolation Parameters by Grade Shell

Pass	Shell	Method	Sample Support			Search Neighborhood					
			Min	Max	Max/Hole	Dip Az	Dip	Pitch	Range (X)	Range (Y)	Range (Z)
1	BrimVortex Silicic	ID3	1	12	2	285	-9	165	175	175	50
	BrimVortex Argilic	ID3	1	12	2	285	-9	165	175	175	50
	Central Argilic	ID3	1	12	2	95	-5	90	150	150	50
	Central Silicic	ID3	1	12	2	95	-2	90	40	100	20
	E Fault Propilitic	ID3	1	12	2	278	-45	90	80	80	30
	Other Propilitic	ID3	1	12	2	324	-3.6	127	100	100	30
	None	ID3	1	12	2	324	-3.6	127	100	100	30
	Acid Leach	ID3	1	12	2	324	-3.6	127	120	120	40
	North Argilic	ID3	1	12	2	239	-3	110	150	150	40
	North Silicic	ID3	1	12	2	270	-3	90	150	150	30
2	BrimVortex Silicic	ID3	1	12	2	285	-9	165	350	350	100
	BrimVortex Argilic	ID3	1	12	2	285	-9	165	350	350	100
	Central Argilic	ID3	1	12	2	95	-5	90	300	300	100
	Central Silicic	ID3	1	12	2	95	-2	90	80	200	40
	E Fault Propilitic	ID3	1	12	2	278	-45	90	160	160	60
	Other Propilitic	ID3	1	12	2	324	-3.6	127	200	200	60
	None	ID3	1	12	2	324	-3.6	127	200	200	60
	Acid Leach	ID3	1	12	2	324	-3.6	127	240	240	80
	North Argilic	ID3	1	12	2	239	-3	110	300	300	80
	North Silicic	ID3	1	12	2	270	-3	90	300	300	60
3	BrimVortex Silicic	ID3	1	12	2	285	-9	165	525	525	150
	BrimVortex Argilic	ID3	1	12	2	285	-9	165	525	525	150
	Central Argilic	ID3	1	12	2	95	-5	90	450	450	150
	Central Silicic	ID3	1	12	2	95	-2	90	120	300	60
	E Fault Propilitic	ID3	1	12	2	278	-45	90	240	240	90
	Other Propilitic	ID3	1	12	2	324	-3.6	127	300	300	90
	None	ID3	1	12	2	324	-3.6	127	300	300	90
	Acid Leach	ID3	1	12	2	324	-3.6	127	360	360	120
	North Argilic	ID3	1	12	2	239	-3	110	450	450	120
	North Silicic	ID3	1	12	2	270	-3	90	450	450	90

Source: SRK, 2018



Source: SRK, 2018

Figure 11-2: Gold Search Ellipses (Pass 1)

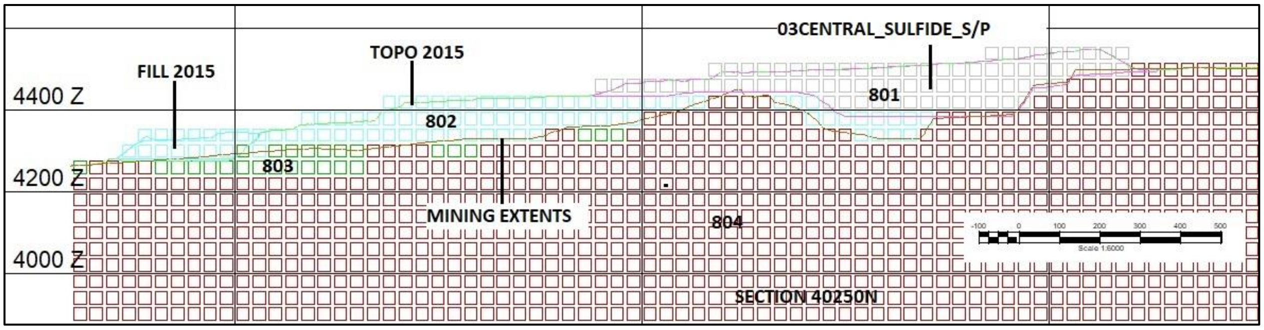
11.6.2 Fill Material

Block grades in fill material were estimated using the available drill hole data from holes drilled through volumes of fill material, either as the upper section of deeper holes into in-situ material or as specific fill sampling holes. These holes were sampled and assayed for AuFA, AgFA and AgCN, and were identified in the drill hole database. Assay data was composited as 40ft bench composites, and grades were assigned as NN estimates, using an un-rotated anisotropic search ellipse of 120ft x 120ft x 20ft (single bench in the vertical axis).

11.6.3 Sulfide Stockpile Material

Stockpile (fill) volumes were modeled as 3D shapes by Hycroft staff from surveyed mining extents and bedrock contacts in sonic drillholes, below current topography. SRK verified the fill volumes relative to the 2015 (current) topographic surface and the mining extents surface, both used for the 2018 resource estimate. Visual examination in section confirmed that the Hycroft fill volumes were consistent with the 2015 topographic and mining extents surfaces, as illustrated in Figure 11-3, which illustrates a sectional view on Section 40250N.

All material identified as fill in the 2018 resource model was modeled as generic fill (rock type 802). At the request of Hycroft, fill volumes identified as sulfide stockpiles were modeled with a unique rock type (801) for this update. In some areas the modeled stockpiles rest on underlying generic fill, as further illustrated in Figure 11-3, with blocks on the contacts being coded on a majority percentage basis. Note that rock types 803 and 804 as illustrated refer to alluvium and in-situ material respectively. Fill blocks were assigned a default density of 0.055 ton/ft3 (18.18 ft3/ton).



Source: SRK, 2019

Figure 11-3: Fill and Stockpile Coding – Section 40250N (Looking North)

The sulfide stockpile volumes have been defined since the 2018 block model estimation. Only these volumes have updated grade estimates, while the generic fill and in-situ material grade estimates are not changed from the previous model.

11.6.3.1 Database

Data used to update the mineral resource in 2019 were provided by Hycroft as CSV files (collar, survey and assay data for drill holes) and DXF files (sulfide stockpile and 2014 and 2015 generic fill volumes). The drill hole database contains 56 Sonic drillholes drilled in 2018 to sample sulfide stockpiles, ranging in depth from 11 ft to 143 ft, with a total length of 4,865 ft. All holes were drilled vertically.

11.6.3.2 Exploratory Data Analysis

Bearing in mind the nature of drillhole samples from stockpiles that have been built up over time from several locations, the EDA is restricted to a summary by stockpile area of the assay statistics for total gold (AUFA) and silver AGFA), and cyanide-soluble gold (AUCN) and silver (AGCN), as tabulated in Table 11-11 through Table 11-14.

Table 11-11: Summary Assay Statistics – AUFA (OPT)

Area	Count	Min	Max	Mean	Total	StDev	CV
Bay (06, 01)	98	0.000	0.043	0.016	1.561	0.011	0.69
Brimstone (02)	139	0.000	0.024	0.005	0.672	0.005	0.93
Central (03)	122	0.000	0.018	0.009	1.125	0.003	0.30
Crusher (04)	70	0.001	0.067	0.013	0.887	0.008	0.66
Leach Pad	22	0.002	0.016	0.005	0.120	0.003	0.52
North Bay (05)	37	0.002	0.025	0.014	0.501	0.006	0.46

Source: SRK, 2019

Table 11-12: Summary Assay Statistics – AUCN (OPT)

Area	Count	Min	Max	Mean	Total	StDev	CV
Bay (06, 01)	98	0.000	0.027	0.008	0.778	0.005	0.68
Brimstone (02)	139	0.000	0.019	0.003	0.369	0.003	1.04
Central (03)	122	0.000	0.009	0.003	0.367	0.002	0.58
Crusher (04)	70	0.000	0.013	0.004	0.293	0.002	0.55
Leach Pad	22	0.000	0.010	0.002	0.053	0.003	1.02
North Bay (05)	37	0.001	0.016	0.007	0.269	0.003	0.42

Source: SRK, 2019

Table 11-13: Summary Assay Statistics – AGFA (OPT)

Area	Count	Min	Max	Mean	Total	StDev	CV
Bay (06, 01)	98	0.005	0.270	0.062	6.075	0.060	0.97
Brimstone (02)	139	0.005	1.320	0.174	24.190	0.195	1.12
Central (03)	122	0.020	1.010	0.186	22.750	0.103	0.55
Crusher (04)	70	0.040	5.830	1.211	84.760	1.232	1.02
Leach Pad	22	0.070	0.370	0.135	2.960	0.073	0.54
North Bay (05)	37	0.005	0.780	0.135	4.995	0.175	1.30

Source: SRK, 2019

Table 11-14: Summary Assay Statistics – AGCN (OPT)

Area	Count	Min	Max	Mean	Total	StDev	CV
Bay (06, 01)	98	0.000	0.121	0.032	3.158	0.031	0.97
Brimstone (02)	139	0.001	0.980	0.093	12.940	0.140	1.51
Central (03)	122	0.003	0.863	0.091	11.090	0.085	0.94
Crusher (04)	70	0.022	3.249	0.671	46.940	0.581	0.87
Leach Pad	22	0.019	0.201	0.068	1.485	0.046	0.69
North Bay (05)	37	0.004	0.219	0.053	1.974	0.041	0.76

Source: SRK, 2019

11.6.3.3 Capping Analysis and Capping

Given that the Sonic drillhole sample assays do not represent in-situ grade distributions/domains, the erratic distribution of materials in the stockpiles, and the overall low coefficients of variation (CV), capping of values was not considered to be appropriate prior to compositing.

11.6.3.4 Compositing

As implemented in the 2018 resource model, un-capped assay values in PPM were composited into 40 ft bench composites for stockpile fill grade estimation purposes. The benches were defined to correspond to the block model levels.

11.6.3.5 Stockpile Block Grade Estimation

Updated block grade estimation was restricted to the stockpile volumes provided by Hycroft (rock code 801), with fill block grade values for the generic fill blocks (rock code 802) being retained from the 2018 resource model, where estimated. Bench composites falling within the stockpile volumes were back-tagged with the stockpile code (801) and code-matching was used in estimation to implement a hard boundary.

Stockpile fill block grades for AUFA, AUCN, AGFA, AGCN and Sulfide Sulfur % were estimated by nearest-neighbor estimation, using a search radius of 120 by 120 by 20 ft, following the procedure used for the 2018 resource modelling. The vertical range of 20 ft restricted the extension of bench composite grades to the corresponding bench. Un-estimated blocks for AUFA, AUCN, AGFA and AGCN were assigned the average grade of the estimated blocks by area as a default grade. This approach provides a declustered mean grade for the input sample data (bench composites), which is considered appropriate for un-estimated blocks, and is largely supported by available ore control data, as tabulated in Table 7. In the opinion of the QP, the Sonic drilling data, comprising sampling of the actual material in place, is more representative compared to ore control data which is compiled from many locations, and subject to possible misclassification/routing of material. Table 7 table shows the block model mean grades (BM) used, and a comparison to the corresponding ore control grades (O/C). Un-estimated blocks for sulfide sulfur (%) were similarly assigned the average grade of the estimated blocks (BM) by area as a default grade, as also tabulated in Table 14-15.

AUFA, AUCN, AGFA and AGCN block grade estimates in PPM were converted to OPT, as separate block model items, by division with a conversion factor of 34.2857.

Table 11-15: Default Grades for Un-estimated Blocks by Area

Area	O/C AuFA (OPT)	BM AuFA (OPT)	O/C AuCN (OPT)	BM AuCN (OPT)	O/C AgFA (OPT)	BM AgFA (OPT)	O/C AgCN (OPT)	BM AgCN (OPT)	BM Sulfide (%)
Bay 06	0.031	0.011	0.007	0.006	0.015	0.050	0.080	0.024	0.40
Bay01	0.015	0.027	0.004	0.013	0.100	0.098	0.050	0.056	0.94
Bay 05	0.023	0.013	0.006	0.007	0.110	0.146	0.050	0.056	1.12
Central	0.009	0.010	0.002	0.003	0.210	0.196	0.070	0.094	1.51
Crusher	0.012	0.012	0.004	0.004	0.640	1.082	0.240	0.614	3.40
Brim	0.012	0.005	0.002	0.003	0.880	0.177	0.200	0.091	2.75

Source: SRK, 2019

11.6.3.6 Classification

Based on confidence in the volume surveys for the Crusher, Brimstone and Central sulfide stockpiles, and the availability of ore control (blasthole) data that supports the use of the declustered block grade averages for un-estimated blocks, in the QP's opinion, these stockpiles are classified as Indicated. In the QP's opinion, the survey and ore control data for the Bay area stockpiles is not as representative as the Crusher, Brimstone and Central data and these stockpiles are therefore classified as Inferred.

11.6.4 Missing AuCN values

Modeled high-leach, transition, and refractory zones were coded to drill holes. Statistics on the distribution of Au CN:FA ratio values were considered, and the mean ratio values were applied to intervals missing valid ratios. The full ratio dataset was composited, and ratios were estimated with fire assay gold to the model blocks. Using the estimated total gold and gold CN:FA ratio values, the block Au CN values were calculated. This approach allows the oxide-sulfide material type in the geological model to be honored in the block ratio values.

11.7 MODEL VALIDATION

Model validation was approached through visual and statistical methods. Visual comparison was done on sections and in plan for each area of the deposit. Statistical comparison was achieved using comparative population statistics and swath plots.

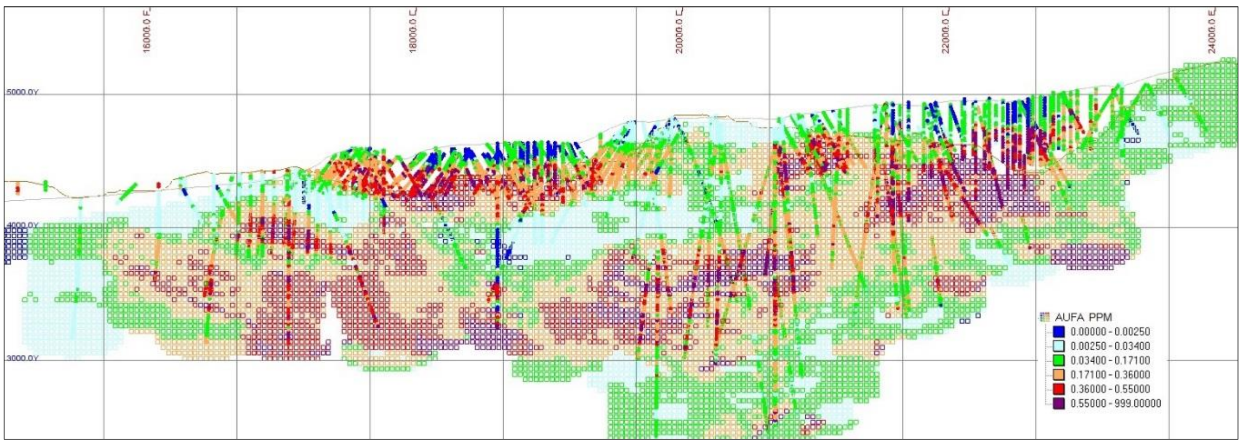
Reconciliation of the model, excluding fill, to available production data was completed. Material mined by Allied Nevada between 2008 and 2015 was compared to blocks in the mined volume. Model and production data are summarized in Table 11-16. The model compared well to historical production records for total gold ounces. The model has about 5% more tonnage, and about 4% lower gold grade, than the reported production. Reported silver grade was about 7% lower than what was predicted by the model, and resulted in silver ounces produced about 12% less than what was predicted from the block model. The QP's opinion of the Life-of-Mine reconciliation is that the model agrees well with reported production results.

Table 11-16: Reconciliation of Block Model to 2008-2015 Production

Data Source	Tonnage	Gold (opt)	Silver (opt)	Gold Ounces	Silver Ounces
SRK Block Model	153,263,016	0.0115	0.3522	1,762,525	53,979,234
Hycroft Ore Control Data	146,106,426	0.0120	0.3262	1,750,741	47,654,393
Difference from Model	**-4.67%**	**4.20%**	**-7.39%**	**-0.67%**	**-11.72%**

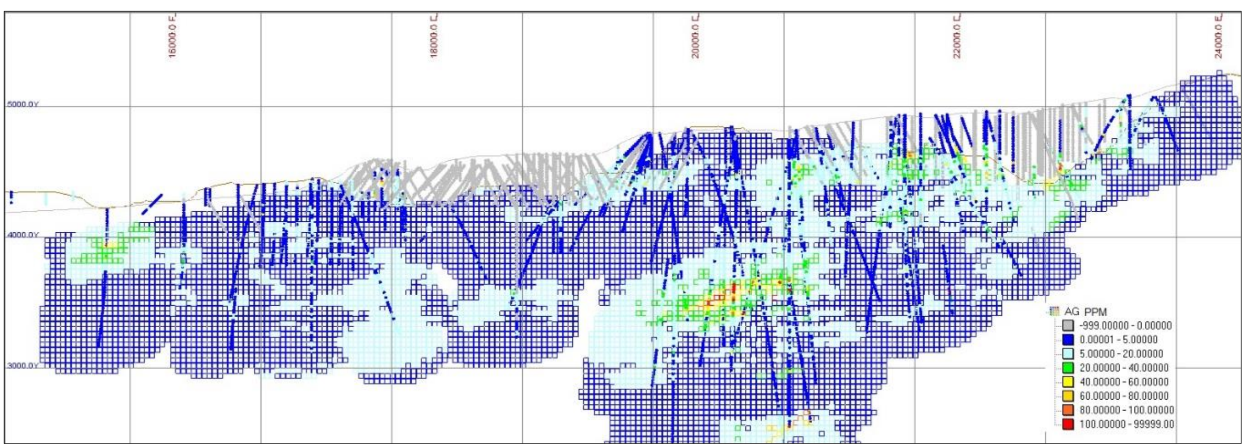
11.7.1 Visual Comparison

A visual inspection of the model in plan and section confirmed that grades were well correlated between the blocks and the composite data in each area. Example images showing block grades vs composite grades in section are provided below in Figure 11-4 through Figure 11-6.



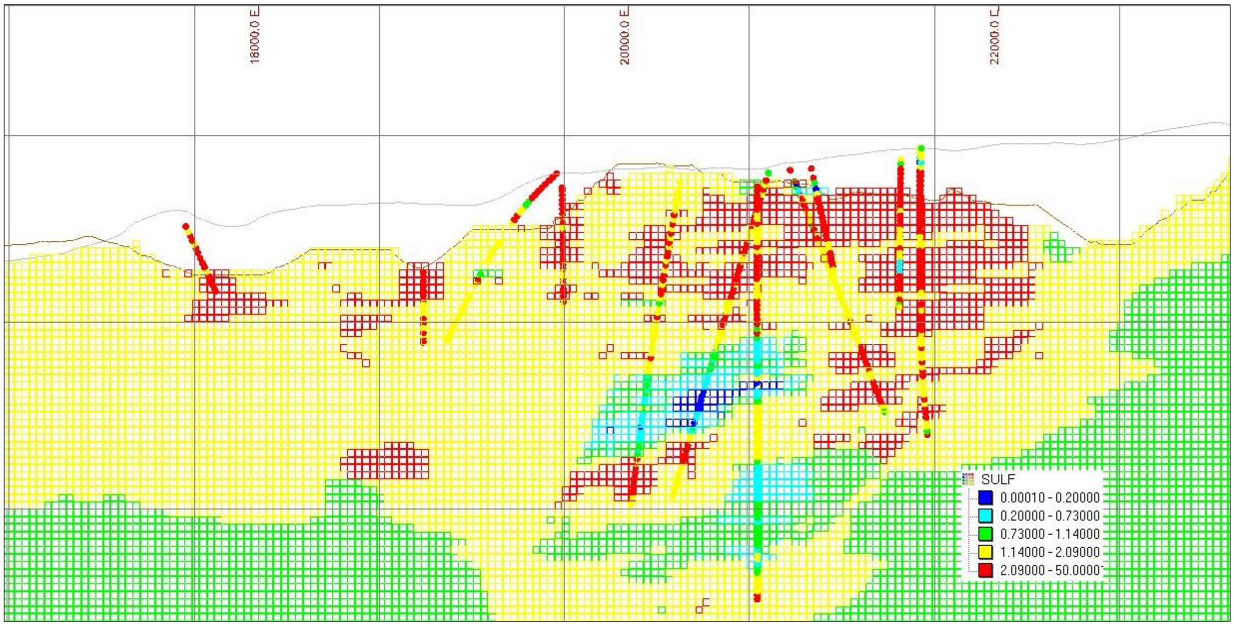
Source: SRK, 2018

Figure 11-4: 41000N AuFa Block Grades and Composites (Looking North)



Source: SRK, 2018

Figure 11-5: 41000N AgFa Block Grades and Composites (Looking North)



Source: SRK, 2018

Figure 11-6: 41000N Sulfide% Block Grades and Composites (Looking North)

11.7.2 Comparative Statistics

Statistics by interpolation domain (grade shell) were used to compare the Au and Ag NN (polygonal) and OK and IDW, where applicable, grades against each other. The NN interpolation method provides a declustered representation of the sample grades and therefore, the resulting mean grades of any other method should be similar to the mean grade of the NN estimate at a zero-cutoff grade. For Au, the OK estimates were within acceptable tolerances of the NN; approximately ±3% for each domain. The global mean estimated OK grade at zero cut-off was within ~1% of the NN estimate. For Ag, the OK and IDW estimates were within acceptable tolerances of the NN; approximately ±5% for each domain, with the higher variances corresponding to the poorly sampled Bay and Lewis domains. The global mean estimated grade at zero cut-off was within ~1.2% of the NN estimate. The domain and global comparison between OK, IDW and NN models is shown in Table 11-17 and Table 11-18.

Table 11-17: Model Validation: Comparison of Estimation Methods – Gold

Grade Shell	Mean (PPM)		% Difference
	AuFA (NN)	AuFa (OK)	(Absolute)
SW Upper	0.234	0.241	3.0%
SE Lower	0.302	0.301	0.3%
Lewis	0.173	0.172	0.6%
East Fault	0.302	0.297	1.7%
Brim Vortex	0.279	0.279	0.0%
Bay	0.218	0.222	1.8%
Background	0.047	0.048	2.1%
Global	0.107	0.106	0.9%

Source: SRK 2018

Table 11-18: Model Validation: Comparison of Estimation Methods – Silver

Grade Shell	Mean (PPM)		% Difference
	AgFA (NN)	AgFa (OK)	(Absolute)
SW Upper	30.261	30.098	0.5%
SE Lower	41.663	40.634	2.5%
Lewis	17.688	16.772	5.2%
East Fault	33.724	34.958	3.7%
Brim Vortex	26.489	26.208	1.1%
Bay	31.83	29.92	6.0%
Background	3.607	3.566	1.1%
Global	4.845	4.789	1.2%

Source: SRK 2018

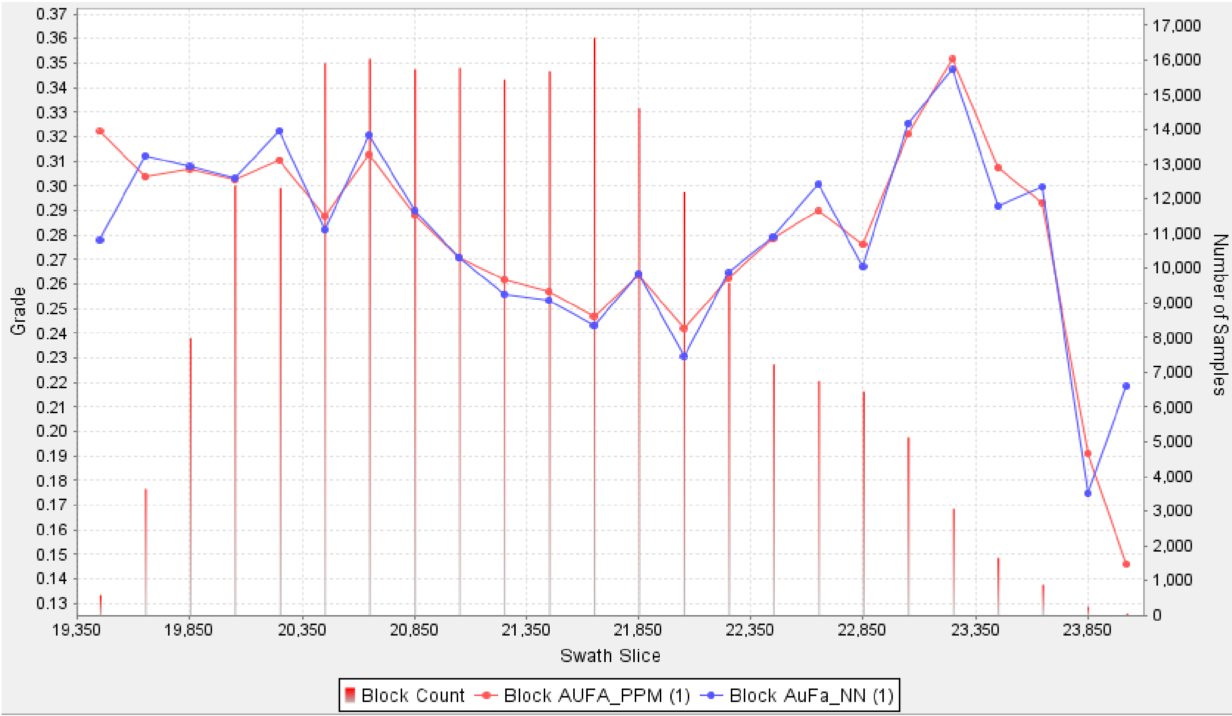
11.7.3 Swath Plots

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Using the swath plot, grade variations from the OK and IDW (where applicable) model are compared to the distribution derived from the NN grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but on a much larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the OK/IDW model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend of the OK/IDW data should be similar to the NN distribution of grade.

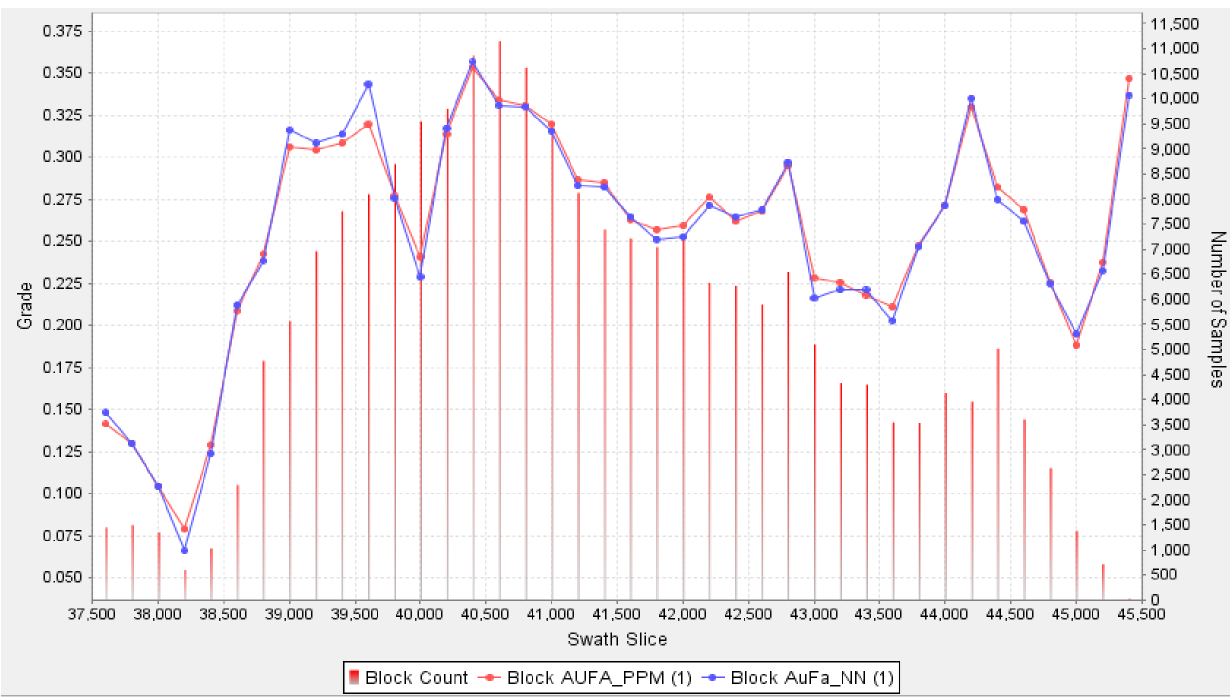
Swath plots were generated along east-west and north-south directions, and also for elevation. Swath widths were 200 feet wide for both east-west and north-south orientations, and 80 feet wide in the vertical. Au grades were plotted by OK/IDW (red traces) and NN (blue traces) for all estimated blocks. Example swath plots for the Brimstone Vortex AuFa estimates are shown in Figure 11-7 through Figure 11-9.

Based on the swath plots, it is the QP's opinion that there is a reasonable correlation between the modeling methods. The degree of smoothing in the OK/IDW model is evident in the peaks and valleys shown in some swath plots; however, this comparison shows close agreement between the OK/IDW and NN models in terms of overall grade distribution as a function of easting, northing, and elevation; especially where there are high tonnages (as shown by the vertical bars on the plots).



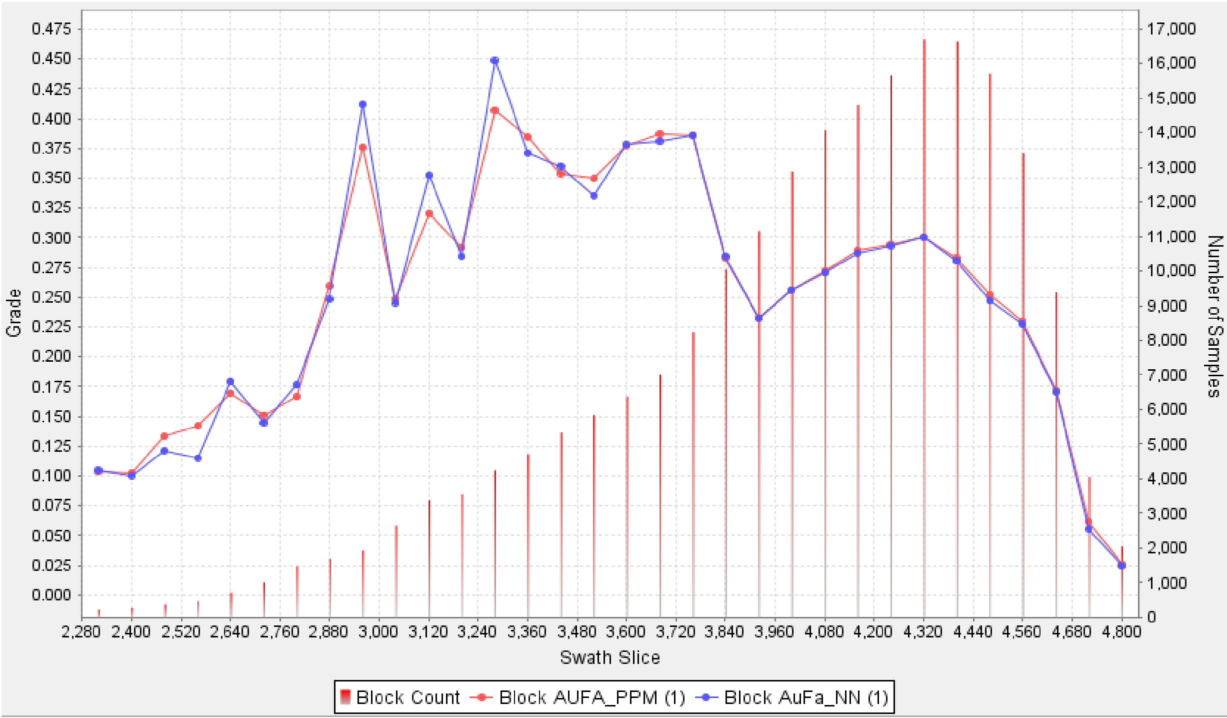
Source: SRK, 2018

Figure 11-7: AuFa Grades by Easting - Brimstone Vortex Domain



Source: SRK, 2018

Figure 11-8: AuFa Grades by Northing - Brimstone Vortex Domain



Source: SRK, 2018

Figure 11-9: AuFa Grades by Elevation - Brimstone Vortex Domain

11.8 RESOURCE CLASSIFICATION

Mineral Resources were classified into Measured, Indicated, and Inferred categories based on the reporting requirements of the New Mining Rules. Initial classification criteria considered a combination of geometric criteria and estimation quality attributes:

- Measured: blocks require a minimum of three holes located within 145 foot radii, nominally corresponding to a maximum drillhole spacing of 205 feet. For those blocks that satisfied this geometric criteria, the mean distance of the nearest three holes is 123 feet, and the mean number of holes is five.
- Indicated: minimum of two drill holes, but within a drill data spacing of 120 to 450 feet depending on domain (see Table 11-19)
- Inferred: minimum of one drill hole at a distance greater than 120 to 450 feet from source data, but within the gold grade domain (see Table 11-19)

The threshold distances for Indicated and Inferred are based on variogram ranges and therefore vary by domain. Classification criteria for Indicated and Inferred in each domain are detailed in Table 11-19.

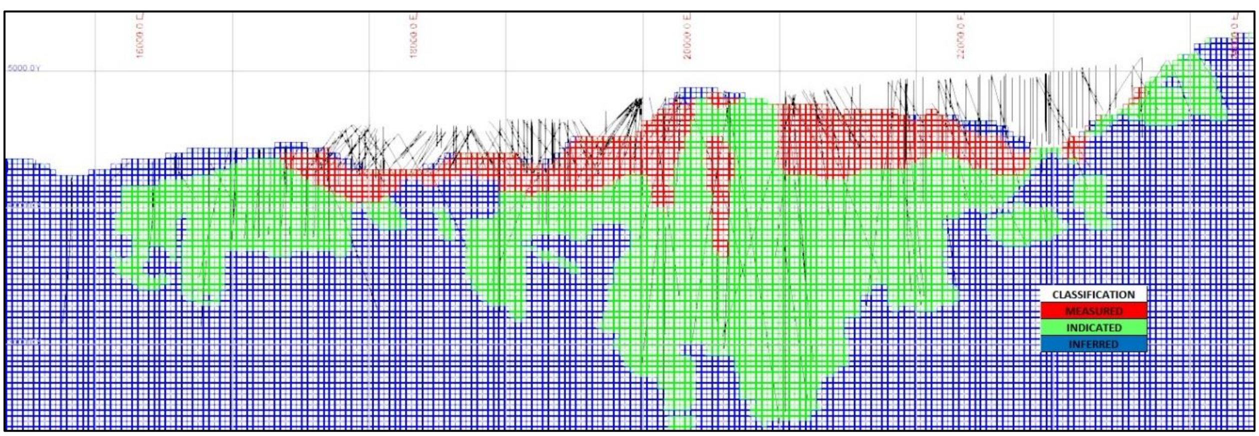
Table 11-19: Resource Classification Criteria

		Distance to Nearest Hole (Feet)	
Grade Shell	**# Drill holes**	99 - 180	> 180
SW Upper, Lewis, E Fault, Bay, Lewis proximate to E Fault	2	Indicated	Inferred
	1	Inferred	Inferred
SW Lower	**# Drill holes**	110 - 200	> 200
	2	Indicated	Inferred
	1	Inferred	Inferred
Brim Vortex	**# Drill holes**	198 - 360	> 360
	2	Indicated	Inferred
	1	Inferred	Inferred
Background, external to grade shells	**# Drill holes**	250 - 450	> 450
	2	Indicated	Inferred
	1	Inferred	Inferred
SW Upper High Grade	**# Drill holes**	66 - 120	> 120
	2	Indicated	Inferred
	1	Inferred	Inferred

Source: SRK, 2018

The block model centroids with the Class item were exported, and imported to Leapfrog 3-D to use as guide points for building solids. The raw classification coding produced small volumes of each material in some areas. These discontinuities were inspected for drilling support and continuity in the surrounding material. Many of the small volumes were omitted from the set of solids generated for model coding. This approach lends geological continuity to each resource class, and considers the mathematical support for estimation in each block.

Solids for Measured and Inferred material were generated for block model coding. Blocks were defaulted to Indicated, then coded with the geological classification solids. The smoothed geological classification is applied to report Mineral Resources, and illustrated on a typical section in Figure 11-10. Search distances, and therefore, classification criteria, vary by domain.



Source: SRK, 2018

Figure 11-10: Classification: 41000N (Looking North) 200 Ft Corridor

SRK has classified the mineral resources in accordance with Item 1302(d)(1)(iii)(A) of Regulation S-K. With reference to Table 11-20, which summarizes the sources and degree of uncertainty considered, the categories are defined as follows:

- Measured Resources – SRK limited the Measured Mineral Resources to areas where in SRK opinion the quality and quantity of the grade and tonnage are based on low levels of uncertainty. Only areas within the model where the criteria and uncertainty correspond to the Low Degree of Uncertainty column in Table 11-20 have been used to classify this category of resource.

- Indicated Resources – SRK has limited the Indicated part of a mineral resource for which in SRK's opinion the quantity and grade are estimated on the basis of adequate geological evidence and sampling. This includes all the sources of data used in under the Measured category, but also includes additional legacy data which had lower precision assays completed. The main controlling feature on the uncertainty is considered to be the drill spacing with a minimum of 2 holes within 120' – 450'ft (a function of the variogram ranges) used within different domains, to determine an adequate level of grade uncertainty. The criteria and uncertainty correspond to the Medium Degree of Uncertainty column in Table 11-20.

- Inferred Resources – SRK has limited the Inferred part of a mineral resource for areas in the geological model where the quantity and grade are estimated on the basis of limited geological evidence and sampling. SRK considers this to have the highest levels on uncertainty with limited drilling information resulting in grades not being able to be estimated with adequate confidence between drill holes for the application of modifying factors for mining. These areas of the model represent the lowest drilling density (wide spaced), which are beyond the ranges where statistically valid estimates can be assigned with confidence. SRK considers these areas of the Mineral Resource will require additional exploration drilling prior to mining. The criteria and uncertainty correspond to the High Degree of Uncertainty column in Table 11-20.

Table 11-20: Sources and Degree of Uncertainty

	Degree of Uncertainty		
Source	**Low**	**Medium**	**High**
Drilling	Exploration - no significant issues identified with historic data. Protocols consistent with industry standards.		Production (Blastholes) - uncertainty related to assay procedures. Reconciliation results based on data has less weighting in model verification. Production data were not used for resource estimation.
Sampling	Industry standard sampling utilized for core and RC drilling		
Sample Preparation/ Assay	Sample handling, preparation and analysis methods at accredited, independent laboratories meet current industry standards.		
QA-QC	Sample preparation and analysis procedures for HMC meet current industry standards, assay results suitable for use in resource estimation.	Lower precision of legacy assay data recognized, but areas with older drilling and no recent infill drilling largely mined out - minor impact on resource estimation because these have been mined out and are excluded from reporting.	
Data Verification	Collar and survey checks corrected minor inconsistencies - holes with unresolved inconsistencies removed.		
Database	Location, analytical and geological data in the database were verified to the QP's satisfaction.		

	Degree of Uncertainty		
Source	**Low**	**Medium**	**High**
Geologic Modeling	Some models based on HMC interpretations that reconciled well with drillhole logging. Other model developed by SRL using implicit modeling tools to ensure consistency.		
Bulk Density	Density data based on 860 samples distributed through resource volume, largely core samples, and tested by ALS Minerals. Overall COV and COV by domain is < 1.0, and data is considered adequate for resource estimation.		
Variography	Grade continuity as quantified with variography is variable by domain, largely based on variable drill density in some domains. Low uncertainty in domains inside the resource pit are as variograms are well informed and robust.		
Grade Estimation	Grade model verification in terms of visual verification, comparative statistics and swath plots indicate that estimated grades are largely visually representative, unbiased and within acceptable tolerances relative to the declustered mean grades.		
Prices, NSR values -pit optimization	Prices based on HMC reserve price (not exceeding 36-month trailing avg.), with industry typical premiums applied for resources.		
Drill Spacing and Estimation Criteria	Geometric criterion of minimum of 3 holes located within 145' radii of blocks. Range based on SRK experience with similar gold mineralization. Mean distance of nearest 3 holes = 123'. Blocks that satisfy these criteria were estimated using data from an average of 5 holes.	Minimum of 2 holes within a drill spacing of 120 - 450', depending on domain.	Minimum of 1 hole at distance > 120 - 450' (dependent on domain) but within gold grade domains.
Continuity of Classification Volumes	Smoothing applied to re-classify isolated volumes (groups of blocks) to ensure reasonable continuity of Measured and Indicated categories, using implicit modeling techniques. Also adjusted for depth consistent with high confidence drill density near surface.		

11.9 QUALIFIED PERSONS OPINION – FURTHER WORK

The qualified person is of the opinion that, with consideration of the SRK recommendations and opportunities outlined in Section 23.1 (Recommendations – Geology, Exploration and Drilling), that any issues relating to all applicable technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

11.10 MEASURED AND INDICATED MINERAL RESOURCE SUMMARY

Given that process recoveries and costs in the resource model are grade and/or domain dependent, the application of standard cut-off grades for resource reporting purposes is not feasible. The resources are, therefore, reported with respect to a block Net Smelter Return (NSR) value which is calculated on a block-by-block basis. The resource is also constrained by an optimized (Whittle) resource pit, in order to demonstrate that the defined resources have reasonable prospects of eventual economic extraction, a part of the New Mining Rules criteria. All classification categories were considered in the resource pit optimization. The estimation of the NSR values and development of the Whittle resource pit requires assumptions around technical and economic parameters such as process recoveries, mining methods and operating costs that are the responsibility of QP's for Sections 10, 13, 14 and 18. These parameters are outside the expertise of the QP for this section.

11.10.1 Resource Pit Optimization (Whittle) Parameters

The resource pit optimization parameters are tabulated in Table 11-21. These parameters were also used in the calculation of block NSR values for reporting purposes. Backup for cost assumptions can be found in Section 12.

Table 11-21: Resource Pit Optimization (Whittle)/NSR Parameters

	Oxide	Transitional	Sulfide
Over-all Pit Slope			
Slope Sectors	CNI w/ Golder Review	CNI w/ Golder Review	CNI w/ Golder Review
Mining Cost	**$/t**	**$/t**	**$/t**
Fill	$1.00	$1.00	$1.00
Alluvium	$1.45	$1.45	$1.45
Rock	$1.45	$1.45	$1.45
Process Cost - ROM	**$/t**	**$/t**	**$/t**
NW (Bay)	$3.27	$3.75	NA
West (Center)	$2.43	$2.91	NA
SW Upper (Camel) – Above water table	$2.49	$2.84	NA
SW Lower (Camel) – Below water table	$3.03	$3.61	NA
Brimstone North	$2.65	$3.14	NA
Brimstone Main	$2.65	$3.14	NA
LeachPad	$2.65	$3.14	NA
Foothills East	$2.65	$3.14	NA
Vortex	$2.79	$3.15	NA
Process Cost - 3/4" Crush	**$/t**	**$/t**	**$/t**
NW (Bay)	$4.70	$5.19	NA
West (Center)	$3.87	$4.23	NA
SW Upper (Camel) – Above water table	$3.79	$4.04	NA
SW Lower (Camel) – Below water table	$4.41	$4.77	
Brimstone North	$3.90	$4.50	NA
Brimstone Main	$3.90	$4.50	NA
LeachPad	$3.90	$4.50	NA
Foothills East	$3.90	$4.50	NA
Vortex	$4.23	$4.71	NA
Process Cost - 1/2" Crush	**$/t**	**$/t**	**$/t**
NW (Bay)	NA	$3.46	$3.46
West (Center)	NA	$3.46	$3.46
SW Upper (Camel) – Above water table	NA	$3.46	$3.46
SW Lower (Camel) – Below water table	NA	$3.46	$3.46
Brimstone North	NA	$3.46	$3.46
Brimstone Main	NA	$3.46	$3.46
LeachPad	NA	$3.46	$3.46
Foothills East	NA	$3.46	$3.46
Vortex	NA	$3.46	$3.46
Soda Ash Cost - 1/2" Crush	**$/t**	**$/t**	**$/t**
NW (Bay)	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
West (Center)	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
SW Upper (Camel) – Above water table	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
SW Lower (Camel) – Below water table	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
Brimstone North	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
Brimstone Main	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
LeachPad	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
Foothills East	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
Vortex	NA	See Section 12.1 – Table 12-4	See Section 12.1 – Table 12-4
G&A	**$/t**	**$/t**	**$/t**
ROM / ¾" Crushed / ½" Crushed	$0.65	$0.65	$0.65
Sustaining Capital	**$/t**	**$/t**	**$/t**

	Oxide	Transitional	Sulfide
ROM / ¾" Crushed / ½" Crushed	$0.24	$0.24	$0.24
ROM Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	ratio_au*69.8%	ratio_au*69.8%	NA
West (Center)	ratio_au*69.8%	ratio_au*69.8%	NA
SW Upper (Camel) – Above water table	ratio_au*69.8%	ratio_au*69.8%	NA
SW Lower (Camel) – Below water table	ratio_au*56.4%	ratio_au*56.4%	NA
Brimstone North	ratio_au*76.5%	ratio_au*76.5%	NA
Brimstone Main	ratio_au*76.5%	ratio_au*76.5%	NA
LeachPad	ratio_au*76.5%	ratio_au*76.5%	NA
Foothills East	ratio_au*76.5%	ratio_au*76.5%	NA
Vortex	ratio_au*76.5%	ratio_au*76.5%	NA
Ag recovery by domain (% of Fire)			
NW (Bay)	20.00%	20.00%	NA
West (Center)	15.00%	15.00%	NA
SW Upper (Camel) – Above water table	20.00%	20.00%	NA
SW Lower (Camel) – Below water table	20.00%	20.00%	NA
Brimstone North	22.00%	22.00%	NA
Brimstone Main	22.00%	22.00%	NA
LeachPad	22.00%	22.00%	NA
Foothills East	22.00%	22.00%	NA
Vortex	21.00%	21.00%	NA
3/4" Crushed Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	ratio_au*83.2%	ratio_au*83.2%	NA
West (Center)	ratio_au*81.9%	ratio_au*81.9%	NA
SW Upper (Camel) – Above water table	ratio_au*81.9%	ratio_au*81.9%	NA
SW Lower (Camel) – Below water table	ratio_au*79.2%	ratio_au*79.2%	NA
Brimstone North	ratio_au*98.0%	ratio_au*98.0%	NA
Brimstone Main	ratio_au*98.0%	ratio_au*98.0%	NA
LeachPad	ratio_au*98.0%	ratio_au*98.0%	NA
Foothills East	ratio_au*98.0%	ratio_au*98.0%	NA
Vortex	ratio_au*98.0%	ratio_au*98.0%	NA
Ag recovery by domain (% of Fire)			
NW (Bay)	25.00%	25.00%	NA
West (Center)	18.00%	18.00%	NA
SW Upper (Camel) – Above water table	37.00%	37.00%	NA
SW Lower (Camel) – Below water table	37.00%	37.00%	NA
Brimstone North	37.00%	37.00%	NA
Brimstone Main	37.00%	37.00%	NA
LeachPad	37.00%	37.00%	NA
Foothills East	37.00%	37.00%	NA
Vortex	37.00%	37.00%	NA
1/2" Crushed Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	NA	55.00%	55.00%
West (Center)	NA	70.00%	70.00%
SW Upper (Camel) – Above water table	NA	70.00%	70.00%
SW Lower (Camel) – Below water table	NA	65.00%	65.00%
Brimstone North	NA	65.00%	65.00%
Brimstone Main	NA	65.00%	65.00%
LeachPad	NA	65.00%	65.00%
Foothills East	NA	65.00%	65.00%
Vortex	NA	65.00%	65.00%
Ag recovery by domain (% of Fire)			
NW (Bay)	NA	55.00%	55.00%
West (Center)	NA	70.00%	70.00%
SW Upper (Camel) – Above water table	NA	70.00%	70.00%
SW Lower (Camel) – Below water table	NA	70.00%	70.00%
Brimstone North	NA	70.00%	70.00%
Brimstone Main	NA	70.00%	70.00%
LeachPad	NA	70.00%	70.00%
Foothills East	NA	70.00%	70.00%

	Oxide	Transitional	Sulfide
Vortex	NA	70.00%	70.00%
Payable Metal Adjustments	%	%	%
Melt Loss - Au	0.20%	0.20%	0.20%
Melt Loss - Ag	0.20%	0.20%	0.20%
Payable Au	99.90%	99.90%	99.90%
Payable Ag	99.00%	99.00%	99.00%
Metal Price	$/oz	$/oz	$/oz
Resource - Au	$1,400	$1,400	$1,400
Resource - Ag	$18.00	$18.00	$18.00
Selling Costs	$/oz	$/oz	$/oz
Bullion Treatment & Refining ($/oz)	$0.75	$0.75	$0.75
Royalties	Not Included	Not Included	Not Included

Source: Hycroft/SRK, 2019

The gold and silver prices ($1,400/oz and $18/oz respectively) selected for the calculation of NSR values for resource pit optimization and resource reporting are based on, and tie back to, the prices selected for the reserve estimation by the QP for reserves reporting. As noted in the sections on Mineral Reserve Estimates (Section 12) and Market Studies and Contracts (Section 16), the selected reserve prices of $1,200/oz and $16.5/oz for gold and silver respectively are below the three-year trailing average price as of June 30, 2019. These trailing average prices were $1,273/oz and $16.53/oz for gold and silver respectively. In accordance with common industry practice, SRK applied a premium of approximately 15% and 10% to the reserve gold and silver prices (after rounding) respectively, ensuring generation of a larger optimized pit as deemed appropriate for resource reporting. The premiums selected were consistent with consensus premiums noted in a review of several current resource estimates generated by both SRK and other consulting/operating companies.

11.11 MINERAL RESOURCE STATEMENT

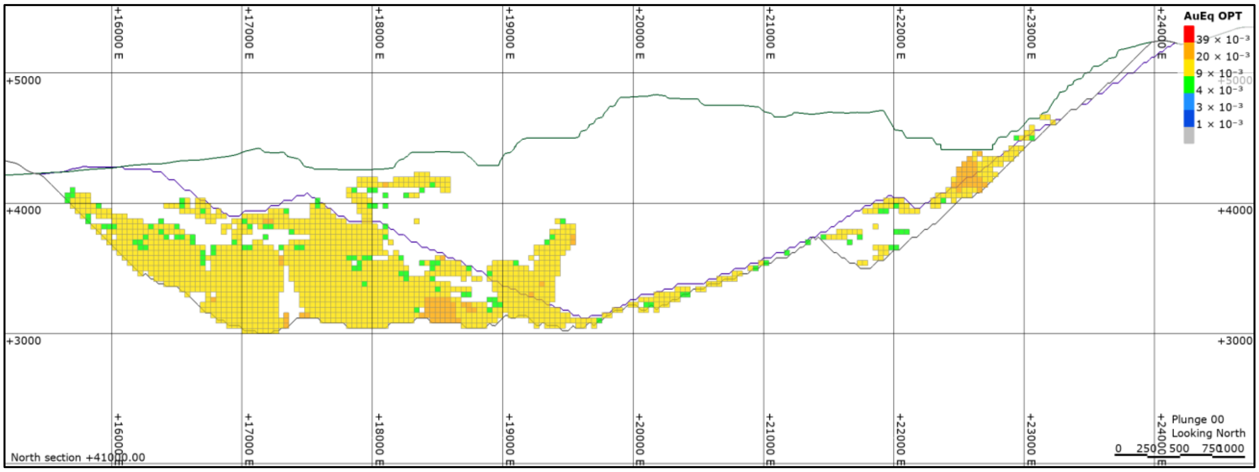
The mineral resource statement is presented in accordance with the New Mining Rules, exclusive of Mineral Reserves. A sectional view of blocks above the NSR cut-off is illustrated in Figure 11-11, color coded by gold equivalent grades (OPT).

Table 11-22: Hycroft Heap Leach Mineral Resource Estimate, June 30, 2019 – SRK Consulting (U.S.), Inc.

Classification	Material	Tons (kt)	Contained Grade				Contained Metal	
			AuFa OPT	AuCn OPT	AgFa OPT	S%	Au (koz)	Ag (koz)
Measured	Oxide	5,650	0.011	0.008	0.224	1.79	60	1,267
	Transition	21,746	0.011	0.005	0.186	1.80	232	4,038
	Sulfide	37,512	0.010	0.002	0.273	1.85	356	10,248
		64,908	**0.010**	**0.004**	**0.240**	**1.83**	**649**	**15,554**
Indicated	Oxide	2,619	0.006	0.005	0.229	1.89	17	599
	Transition	16,293	0.007	0.003	0.329	1.79	117	5,369
	Sulfide	310,102	0.009	0.002	0.282	1.81	2,916	87,470
		329,014	**0.009**	**0.002**	**0.284**	**1.81**	**3,050**	**93,438**
Measured and Indicated	Oxide	8,268	0.009	0.007	0.226	1.82	77	1,867
	Transition	38,039	0.009	0.004	0.247	1.80	349	9,407
	Sulfide	347,614	0.009	0.002	0.281	1.81	3,272	97,718
		393,922	**0.009**	**0.002**	**0.277**	**1.81**	**3,699**	**108,992**
Inferred	Oxide	6,191	0.007	0.005	0.267	1.72	44	1,651
	Transition	20,148	0.008	0.004	0.276	1.74	156	5,570
	Sulfide	568,704	0.010	0.002	0.214	1.76	5,516	121,930
	Fill	4,018	0.013	0.008	0.150	0.63	53	603
		599,062	**0.010**	**0.002**	**0.217**	**1.76**	**5,769**	**129,754**

Source: SRK, 2019

- *Mineral Resources are not Mineral Reserves and do not meet the threshold for reserve modifying factors, such as estimated economic viability, that would allow for conversion to mineral reserves. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves;*
- *Open pit resources stated as contained within a potentially economically minable open pit; pit optimization was based on assumed prices for gold of US$1,400/oz, and for silver of US$18/oz, variable Au and Ag Recoveries based on geo-metallurgical domains, a mining cost of US$1.45/t, variable ore processing costs based on geo-metallurgical domains, and G&A cost of US$0.65/t, and a pit slope of 45 degrees;*
- *Open pit resources are reported based on calculated NSR block values and the cutoff therefore varies from block to block. The NSR incorporates Au and Ag sales costs of US$0.75/oz beyond the costs used for pit optimization;*
- *Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.*
- *Mineral Resources are reported exclusive of Mineral Reserves*



Source: SRK, 2019

Figure 11-11: 41000N - Resource Blocks by AuEq Block Grades (OPT) - Looking North

11.11.1 Mineral Resource Sensitivity

Given the grade and/or domain dependent nature of the recovery and cost parameters in the resource model, the sensitivity of the resource to these parameters is difficult to quantify. Resource estimates are, however, typically most sensitive to price and this sensitivity can be analyzed in terms of the results of the resource pit for a range of prices. The variation in tonnage and grade for gold price increments of $50/oz is tabulated in Table 11-23. Quantities are reported for material outside the Reserves pit. The resource is relatively insensitive to metal price, in terms of the processing paths defined for this study, and current assumptions on price.

Table 11-23: Whittle Resource Pit - Price Sensitivity

Price ($/Oz Au)	Tons (kt)	Au (OPT)	AUFA (koz)	Change (%)	AGFA (OPT)	AGFA (koz)	Change (%)
1200	767,448	0.0100	7,674	-5.8%	0.254	194,548	-4.5%
1250	791,366	0.0099	7,835	-3.8%	0.249	197,367	-3.1%
1300	812,332	0.0098	7,961	-2.3%	0.246	199,753	-2.0%
1350	831,545	0.0097	8,066	-1.0%	0.243	201,899	-0.9%
1400	**848,540**	**0.0096**	**8,146**	**0.0%**	**0.240**	**203,734**	**0.0%**
1450	863,728	0.0095	8,205	0.7%	0.238	205,308	0.8%
1500	878,087	0.0094	8,254	1.3%	0.236	206,877	1.5%
1550	890,232	0.0094	8,368	2.7%	0.234	208,136	2.2%
1600	902,071	0.0093	8,389	3.0%	0.232	209,280	2.7%

Source: SRK, 2019

12 MINERAL RESERVE ESTIMATES

The Hycroft Mine is a restart of the historic Hycroft Mine which was profitably mined from 1983 through 2015. Mineral Reserves were estimated by Hycroft. The significant mineralized domains include Bay, Central, Camel, Brimstone, and Vortex. Based on current operating costs, process recoveries, and metal prices, all these domains contribute to the Mineral Reserves as of June 30th, 2019.

Metal prices used for Mineral Reserves are based on consensus, long-term forecasts from banks, financial institutions, and other sources and are below the trailing three-year average selling prices. Hycroft has considered the reported Mineral Resources, dilution factors, production schedules, and economic analysis for conversion of Mineral Resources to Mineral Reserves and considers it appropriate. Based on the contribution of silver to the cashflow, Hycroft has elected to include the silver content as Mineral Reserves.

Proven and Probable Mineral Reserves have been calculated on operational economics and estimates for costs for Hycroft. HMC verified the economic pit limits of the mineral reserve estimate using Geovia Whittle® 4.5.5 software. The Hycroft Mineral Reserve Estimates are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political or other relevant issues.

Mineral Reserves at Hycroft have been determined by applying current economic criteria that are valid for the Hycroft mine. These criteria limitations have been applied to the resource model to determine which part of the Measured and Indicated Mineral Resource is economically extractable. The reported mineral reserves conform to the New Mining Rules definitions of Proven and Probable Mineral Reserves.

Modifying factors are all discussed in detail in various sections of this summary technical report. It is the opinion of the Qualified Person that Hycroft has appropriately mitigated risks associated with the modifying factors noted in the summary technical report. The Qualified Person is not aware of any legal, political, or other risks that could materially affect the development of the mineral reserve.

Table 12-1 summarizes the Hycroft reserves as of June 30th, 2019, estimated using a gold price of $1,200 per ounce and silver price of $16.50 per ounce, as well as operating costs and applicable recoveries. These have been fully scheduled in a LOM plan and have been shown to demonstrate viable economic extraction. The reference point for these mineral reserves is ore delivered to the leach pad and does not include reductions attributed to anticipated leach recoveries. The Measured and Indicated Mineral Resources are exclusive of those Mineral Resources modified to produce these Mineral Reserves.

This is the initial Summary Technical Report reported under the New Mining Rules, and there is no previous year estimate of Proven and Probable Mineral Reserves to compare with the June 30, 2019 estimate.

Table 12-1: Proven & Probable Mineral Reserves – 6/30/2019

	Tons	Grades, oz/t		Contained Oz (000s)	
	(000s)	Au	Ag	Au	Ag
Proven (Heap Leach)					
Oxide ROM	22,476	0.009	0.230	205	5,211
Transition ROM	4,095	0.008	0.190	32	759
Oxide ¾" Crushed	15,252	0.012	0.720	184	10,926
Transition ¾" Crushed	4,399	0.005	0.310	24	1,367
Transition ½" Crushed	90,206	0.011	0.450	948	40,365
Sulfide ½" Crushed	250,906	0.012	0.470	2,940	116,818
Total Proven Heap Leach	***387,334***	***0.011***	***0.450***	***4,333***	***175,446***
Probable (Heap Leach)					
Oxide ROM	13,145	0.005	0.230	71	3,005
Transition ROM	3,660	0.005	0.140	20	505
Oxide ¾" Crushed	3,001	0.010	0.690	29	2,063
Transition ¾" Crushed	1,304	0.004	0.490	5	644
Transition ½" Crushed	52,467	0.010	0.460	504	24,043
Sulfide ½" Crushed	663,071	0.010	0.410	6,936	272,271
Total Probable Heap Leach	***736,648***	***0.010***	***0.410***	***7,565***	***302,531***
Total Probable Sulfide Stockpile ½" Crushed	***9,079***	***0.011***	***0.380***	***98***	***3,422***
TOTAL PROVEN & PROBABLE MINERAL RESERVES	**1,133,061**	**0.011**	**0.425**	**11,996**	**481,399**
Waste	1,321,853				
Total Tons	**2,454,914**				
Strip Ratio	1.17				

- *Mineral Reserves estimated according to the New Mining Rules definitions.*
- *Mineral Reserves estimated at $1,200/oz Au and $16.50/oz Ag.*
- *Cut-off grades used a Net Smelter Return (NSR) calculation.*
- *Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.*

Economic pit limits have been determined with Geovia Whittle® 4.5.5 strategic planning software. Geovia Whittle® 4.5.5 uses the Lerchs-Grossman economic algorithm, which is an industry standard method for optimizing open pit resources. Economic phases (pushbacks) have been designed based on the Geovia Whittle® output utilizing Maptek Vulcan® software. Minemax Scheduler Professional 6.2.4® has been used to schedule the phases to develop annual and life of mine plans.

12.1 WHITTLE ANALYSIS AND BLOCK VALUE INPUTS

Whittle, a Lerchs-Grossmann algorithm commercial software package, was used to evaluate the Hycroft Mine. Table 12-2 and Table 12-3 define the Whittle parameters and block value inputs used to generate the ultimate pit shell. Costs were generated by Hycroft personnel, metallurgical recoveries were developed by M3 Engineering, and slope inputs supplied by Call and Nicholas and Golder Associates.

A Net Smelter Return (NSR) was generated for each 40 ft x 40 ft x 40 ft block for each of the processing methods available at Hycroft, which are the following:

- Run-of-Mine (ROM) Heap Leaching of oxide and transitional material;

- ¾" Crushed Heap Leaching of oxide and transitional material; and
- ½" Crushed Heap Leaching of transitional and sulfide material.

The processing method that returned the highest net value was selected. If all processing methods returned a negative value, the block was classified as waste.

Table 12-2: Whittle Input Parameters – Heap Leach

	Oxide	Transitional	Sulfide
Mining Cost	**$/t**	**$/t**	**$/t**
Fill	$1.00	$1.00	$1.00
Alluvium	$1.45	$1.45	$1.45
Rock	$1.45	$1.45	$1.45
Incremental mining below 4,660 ft elevation per 40 ft bench	$0.016	$0.016	$0.016
Process Cost - ROM	**$/t**	**$/t**	**$/t**
NW (Bay)	$3.27	$3.75	NA
West (Center)	$2.43	$2.91	NA
SW Upper (Camel) - Above water table	$2.49	$2.84	NA
SW Lower (Camel) - Below water table.	$3.03	$3.61	NA
Brimstone North	$2.65	$3.14	NA
Brimstone Main	$2.65	$3.14	NA
LeachPad	$2.65	$3.14	NA
Foothills East	$2.65	$3.14	NA
Vortex	$2.79	$3.15	NA
Process Cost - 3/4" Crush	**$/t**	**$/t**	**$/t**
NW (Bay)	$4.70	$5.19	NA
West (Center)	$3.87	$4.23	NA
SW Upper (Camel) - Above water table	$3.79	$4.04	NA
SW Lower (Camel) - Below water table.	$4.41	$4.77	NA
Brimstone North	$3.90	$4.50	NA
Brimstone Main	$3.90	$4.50	NA
LeachPad	$3.90	$4.50	NA
Foothills East	$3.90	$4.50	NA
Vortex	$4.23	$4.71	NA
Process Cost - 1/2" Crush	**$/t**	**$/t**	**$/t**
All Domains	NA	$3.46	$3.46
Soda Ash Cost (add to Process Cost) - 1/2" Crush	$/t	$/t	$/t
All Domains	NA	Table 12-4	Table 12-4
G&A	**$/t**	**$/t**	**$/t**
ROM	$0.65	$0.65	$0.65
3/4" Crushed	$0.65	$0.65	$0.65
1/2" Crushed	$0.65	$0.65	$0.65
Sustaining Capital	**$/t**	**$/t**	**$/t**
ROM	$0.24	$0.24	$0.24
3/4" Crushed	$0.24	$0.24	$0.24
1/2" Crushed	$0.24	$0.24	$0.24
Payable Metal Adjustments	**%**	**%**	**%**
Melt Loss – Au	0.2%	0.2%	0.2%
Melt Loss – Ag	0.2%	0.2%	0.2%
Payable Au	99.9%	99.9%	99.9%
Payable Ag	99.0%	99.0%	99.0%
Metal Price	**$/oz**	**$/oz**	**$/oz**
Reserve – Au	$1,200	$1,200	$1,200
Reserve – Ag	$16.50	$16.50	$16.50
Selling Costs	**$/oz**	**$/oz**	**$/oz**
Bullion Treatment & Refining ($/oz)	$0.75	$0.75	$0.75
Royalties	NI	NI	NI

Heap leach recoveries utilized in mineral reserve determination vary by redox, domain, and process method. The recoveries utilized in Geovia Whittle® optimization are shown in Table 12-3.

Table 12-3: Recoveries Utilized in Whittle Optimization

	Oxide	Transitional	Sulfide
ROM Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	ratio_au*0.698	ratio_au*0.698	NA
West (Center)	ratio_au*0.698	ratio_au*0.698	NA
SW Upper (Camel) - Above water table	ratio_au*0.698	ratio_au*0.698	NA
SW Lower (Camel) - Below water table	ratio_au*0.564	ratio_au*0.564	NA
Brimstone North	ratio_au*0.765	ratio_au*0.765	NA
Brimstone Main	ratio_au*0.765	ratio_au*0.765	NA
LeachPad	ratio_au*0.765	ratio_au*0.765	NA
Foothills East	ratio_au*0.765	ratio_au*0.765	NA
Vortex	ratio_au*0.765	ratio_au*0.765	NA
Ag recovery by domain (% of Fire)			
NW (Bay)	20.00%	20.00%	NA
West (Center)	15.00%	15.00%	NA
SW Upper (Camel) - Above water table	20.00%	20.00%	NA
SW Lower (Camel) - Below water table	20.00%	20.00%	NA
Brimstone North	22.00%	22.00%	NA
Brimstone Main	22.00%	22.00%	NA
LeachPad	22.00%	22.00%	NA
Foothills East	22.00%	22.00%	NA
Vortex	21.00%	21.00%	NA
3/4" Crushed Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	ratio_au*0.832	ratio_au*0.832	NA
West (Center)	ratio_au*0.819	ratio_au*0.819	NA
SW Upper (Camel) - Above water table	ratio_au*0.819	ratio_au*0.819	NA
SW Lower (Camel) - Below water table	ratio_au*0.792	ratio_au*0.792	NA
Brimstone North	ratio_au*0.980	ratio_au*0.980	NA
Brimstone Main	ratio_au*0.980	ratio_au*0.980	NA
LeachPad	ratio_au*0.980	ratio_au*0.980	NA
Foothills East	ratio_au*0.980	ratio_au*0.980	NA
Vortex	ratio_au*0.980	ratio_au*0.980	NA
Ag recovery by domain (% of Fire)			
NW (Bay)	25.00%	25.00%	NA
West (Center)	18.00%	18.00%	NA
SW Upper (Camel) - Above water table	37.00%	37.00%	NA
SW Lower (Camel) - Below water table	37.00%	37.00%	NA
Brimstone North	37.00%	37.00%	NA
Brimstone Main	37.00%	37.00%	NA
LeachPad	37.00%	37.00%	NA
Foothills East	37.00%	37.00%	NA
Vortex	37.00%	37.00%	NA
1/2" Crushed Heap Leach Recovery	**%**	**%**	**%**
Au recovery by domain (% of Fire)			
NW (Bay)	NA	55.00%	55.00%
West (Center)	NA	70.00%	70.00%
SW Upper (Camel) - Above water table	NA	70.00%	70.00%

	Oxide	Transitional	Sulfide
SW Lower (Camel) - Below water table	NA	65.00%	65.00%
Brimstone North	NA	65.00%	65.00%
Brimstone Main	NA	65.00%	65.00%
LeachPad	NA	65.00%	65.00%
Foothills East	NA	65.00%	65.00%
Vortex	NA	65.00%	65.00%
Ag recovery by domain (% of Fire)			
NW (Bay)	NA	55.00%	55.00%
West (Center)	NA	70.00%	70.00%
SW Upper (Camel) - Above water table	NA	70.00%	70.00%
SW Lower (Camel) - Below water table	NA	70.00%	70.00%
Brimstone North	NA	70.00%	70.00%
Brimstone Main	NA	70.00%	70.00%
LeachPad	NA	70.00%	70.00%
Foothills East	NA	70.00%	70.00%
Vortex	NA	70.00%	70.00%

- Ratio_au is the ratio of the cyanide soluble au grade to the fire au grade

Table 12-4: Soda Ash Costs

Soda Ash Cost	=	Cost of Soda Ash x Soda Ash Required
Cost of Soda Ash	=	$0.11 per pound
Soda Ash Required	=	% Oxidation x 2000 x %Sulfide Sulfur x 1.57
% Oxidation	=	(Target Oxidation - ratio_au) / Liberation Rate
Target Oxidation	:	Bay = 55%; All Others = 70%
ratio_au	=	aucn block grade / aufa block grade
Liberation Rate		if (ratio_au le 0.05) then = 1.77
		if (ratio_au le 0.10) then = 1.89
		if (ratio_au le 0.15) then = 1.99
		if (ratio_au le 0.20) then = 2.09
		if (ratio_au le 0.25) then = 2.18
		if (ratio_au le 0.30) then = 2.27
		if (ratio_au le 0.35) then = 2.36
		if (ratio_au le 0.40) then = 2.44
		if (ratio_au le 0.45) then = 2.53
		if (ratio_au le 0.50) then = 2.60
		if (ratio_au le 0.55) then = 2.68
		if (ratio_au le 0.60) then = 2.70
		if (ratio_au le 0.70) then = 2.78

The NSR calculation covers all fixed and variable costs including mining, processing, sustaining capital deemed to be directly proportional to ore tonnage, general and administration, gross royalties, transport and shipping costs, smelting and refining costs, limits to payable metals, and refining penalties for deleterious metals. The following is an example of the method used to calculate the NSR expressed in US dollars per ton (US$/t):

NSR (US$/t) is calculated from the following equation:

> NSR = (((Au Price – Au Selling) * Au Grade * Recovery Au * Au Refine) + ((Ag Price - Ag Selling) * Ag Grade * Recovery Ag * Ag Refine)) * (1 - Royalty) - Mine Cost - Process Cost - Soda Ash Cost - Sustaining Cost - G&A Cost

Where:

NSR	= Net Smelter Return
Au Price	= Au selling price in $ per troy ounce
Au Selling	= Bullion treatment and refining cost in $ per troy ounce for gold
Au Grade	= Au fire grade in troy ounces per ton
Recovery Au	= % metallurgical recovery of Au by process route & domain
Au Refine	= % payable for Au refining losses and deductions
Ag Price	= Ag selling price in $ per troy ounce
Ag Selling	= Bullion treatment and refining cost in $ per troy ounce for silver
Ag Grade	= Ag fire grade in troy ounces per ton
Recovery Ag	= % metallurgical recovery of Ag by process route & domain
Ag Refine	= % payable for Ag refining losses and deductions
Royalty	= % royalty (note due to very limited royalty remaining, no royalty has been included)
Mine Cost	= mining cost per ton by material type
Process Cost	= process cost per ton by process type & domain
Soda Ash Cost	= soda ash cost per ton
Sustaining Cost	= sustaining cost per ton
G&A Costs	= general and administrative cost per ton

The over-all slope design parameters for the whittle pits varied between 30° and 50°. Detailed Whittle pit slope angles by zone are listed in Table 12-5. These are over-all slope angles and are the result of an iterative process that takes into consideration approximate ramp access in pit design.

Table 12-5: Whittle Pit Slope Profiles

Slope Profiles	
Zone	**Angle (deg.)**
1	45
2	38
3	41
4	50
5	30
6	32
7	32
8	32
9	45
10	38
11	45
12	50
13	38
14	38
15	50
16	50
17	43
18	37
19	40
20	45
21	40
22	35
23	45

Reserve estimates are typically most sensitive to metal price and this sensitivity can be analyzed in terms of the results of the reserve whittle pit optimization for a range of metal prices. The reserve is relatively insensitive to metal price, in terms defined for this study, and current assumptions on metal price.

Table 12-6 presents the changes in economic pit limits based on metal price sensitivity as estimated by the Lerchs-Grossman algorithm.

Additional sensitivities to financial results on costs and other factors are discussed Section 19.

Table 12-6: Geovia Whittle® Economic Pit Limits

Metal Prices		Total Tons	Waste Tons	Tons Processed	Total Contained Ounces	
Au	Ag	(000s)	(000s)	(000s)	Au (koz)	Ag (koz)
$900	$12.37	1,359,815	628,436	731,380	7,826	364,227
$1,000	$13.75	1,634,441	762,079	872,362	9,334	403,642
$1,100	$15.12	1,950,791	940,460	1,010,331	10,710	445,253
$1,150	$15.81	2,144,743	1,063,046	1,081,697	11,358	468,483
$1,200	**$16.50**	**2,515,052**	**1,295,002**	**1,220,050**	**12,689**	**508,395**
$1,250	$17.19	2,697,805	1,423,511	1,274,294	13,253	524,117
$1,300	$17.87	2,778,775	1,473,640	1,305,135	13,573	531,059
$1,400	$19.25	2,908,366	1,557,511	1,350,855	14,049	541,288
$1,500	$20.62	3,043,539	1,648,651	1,394,887	14,367	551,538

- Based on Whittle optimized shells and does not account for final mine design, nor does it include stockpiled material.

12.2 DILUTION

Hycroft determined that the dilution was accounted for in the generation of the block model for the Mineral Reserves tons and grade incorporated into the mining schedule. In Hycroft's opinion, given the type of gold and silver mineralization, the large selective mining unit, the historical reconciliations at Hycroft, and the experience of adjacent mines in the region in addressing dilution factors, the appropriate measures have been employed for dilution in the Mineral Reserves.

12.3 CUT-OFF GRADES

In the case of Hycroft's open pit, all costs listed above are accounted for during the optimization phase of pit limit planning. Once the optimum pit extents have been determined, the decision to mine the material has been made and the cost incurred; the only task remaining then is to determine the optimal routing of the material. G&A as applied at Hycroft, is a fixed cost and does not vary by the tons mined or processed. As such, G&A costs are applied as an annual cost in the mine planning and not applied as a $/t to ore processed. All material routing is based on optimal destination determination accounting for all applicable costs, recoveries, and limits (i.e. crushing capacity).

NSR's are used as the basis of mineral reserve estimations and for decisions influencing operating strategy, mine planning and design. In practice, Hycroft requires the use of the NSR calculation due to differing mining and processing costs, recoveries, and the influence of both gold and silver. Factors including the variable ore types and mineralogy, different process streams and metallurgical recoveries, and related haulage distance can all cause variability in mining and processing costs and block value.

It is important to note that calculation of the breakeven NSR contains no profit assumptions; therefore, the breakeven NSR cut-off will always be US$0/t. Typical break-even cut-off grades for individual single metal gold and silver are shown in Table 12-7. It must be noted that the NSR calculation incorporates more than the cut-off grades shown below in Table 12-7; however, the break-even cut-off grades shown are typical for Hycroft.

Table 12-7: Typical Break-even Individual Single Metal Cut-off Grade Summary

Process Method	Au (opt)	Ag (opt)
ROM Oxide Leach Recovery	0.006	0.938
ROM Transitional Leach Recovery	0.008	1.115
3/4" Crushed Oxide Leach Recovery	0.005	0.793
3/4" Crushed Transitional Leach Recovery	0.007	0.835
1/2" Crushed Transitional Leach Recovery	0.006	0.420
1/2" Crushed Sulfide Leach Recovery	0.007	0.519

- These typical break-even cut-off grades are listed for informational reference only and should not be considered actual cut-off grades used.

13 MINING METHODS

13.1 OPEN PIT DESIGN

Open Pits were designed by generating Whittle pit shells based on net block values (net smelter returns) and pit slopes recommended by CNI and Golder Associates. Using the Whittle shells as guides, Hycroft designed the final pit with haul ramps, appropriate catch benches, and mining widths. The Hycroft open pit is a large pit, covering an area nearly 3.25 miles long by 1.75 miles wide and reaching a maximum depth of approximately 2,700 feet. The reserve price Whittle shell based on $1,200/oz au and $16.50/oz ag was used guiding the design.

Pit phasing has been designed internal to the final pit limit. Phasing was based on lower revenue whittle shells, access, and minimum mining widths. The purpose of the phasing is to improve over-all economics by mining higher economic margin phases first. In total, 22 individual phases have been designed.

Haul ramps are design to be 120 ft wide, including the safety berm for double lane traffic accommodating 320-t class trucks. A 10% maximum grade has been considered in the final design. Some internal pit phase designs considered single-lane travel and 12.5% maximum grade for a very limited number of benches near the bottom of the phase. All pits are designed to be mined on 40-ft high benches with catch benches every bench. Catch bench widths varied from a maximum of 46.20 ft to a minimum of 22.85 ft.

13.2 OPEN PIT OPERATIONS

Hycroft mining operations are currently planned for typical truck and shovel open pit mining methods. The mine plan developed for the Hycroft feasibility requires an average of approximately 85 – 100 million tons per year to be mined throughout the 34-year mine life. Production is scheduled to start at 5 million tons in year one, ramp up to 20 million tons in year two, 36 million in year three, 60 million tons in year four, 75 million in year five, and 85 million in year 6. Another ramp-up in production occurs in year 10 to 100 million tons as the larger phases need stripping. This production remains steady until the later years before the end of mining when it starts to ramp down as stripping is no longer required. The life of mine stripping ratio (waste to ore) is 1.17:1.

Over the life of the mine, ore routing is based on optimal destination determination accounting for all applicable costs, recoveries, and limits (i.e. crushing capacity). The following ore routing is available to each block:

- Oxide Ore - ROM heap leach & ¾" crushed heap leach
- Transitional Ore - ROM heap leach, ¾" crushed heap leach, ½" crushed heap leach
- Sulfide Ore - ½" crushed heap leach

During the life of the mine, crusher capacity ramps up from 25 million to 36 million tons per year of crushing capacity. The crusher is always kept at full capacity. Lower-grade ROM is processed as it is encountered in mining. Due to crusher capacity limits, scheduling results in routing some higher-grade oxide and transitional material to ROM and lower-grade sulfide to waste in over-all mine net present value (NPV) optimization. Mining is planned to be carried out initially with a fleet of Hycroft owned mining equipment. The initial fleet consists of 30-cubic-yard hydraulic excavators and 200-t class trucks.

The first ramp up in production is achieved with Contract mining support. Mid-Year 2 through Mid-Year 7 is completed with Contract mining. Contract mining considers full-service contract mining with the Contractor providing all equipment, operators, maintainers, and operations supervision.

In Year 6 mining will start to transition back to owner fleets consisting of larger 55-cubic-yard excavators and 320-t class trucks as production ramps up. Blast-hole drills will be capable of drilling up to 9-7/8" diameter holes and 40-ft benches. Track dozers, wheel dozers, front-end loaders, graders, water trucks, and service vehicles support the mining operation. By Mid-year 7 all mining has transitioned to owner mining.

Overburden is hauled from the open pit to adjacent Waste Rock Fill (WRF) storage areas. Some partial pit backfilling using mined overburden from adjacent pit phases is considered to minimize haulage distances and maximize mine value. ROM ore is hauled directly to the leach pads from the open pit, while crushed ore is hauled to the primary crusher and dumped directly into the crusher hopper. Allowances have been made for some short-term ore stockpiling and re-handling near the crusher. Long-term ore stockpiling is considered in optimizing the operations. Economic stockpiles are re-handled and processed at several times during the LOM, but the majority of stockpile is processed at the end of mine life.

Vertical blast holes are loaded with ammonium nitrate and fuel oil or emulsion. Powder factors vary due to changing geology. An average powder factor of 0.69 lbs/ton is typical. All material except fill from previous mining activities require drill and blast. Blasting is performed only during daylight hours and under strict safety procedures and scheduled times. Explosives and blasting agents are handled by licensed handlers and transported and stored on site in compliance with all federal, state, and county regulations.

13.3 ORE CONTROL

Block model block dimensions are defined to incorporate the SMU of the mining equipment selected. Mining dilution and ore recovery are addressed within the block model estimation and no further adjustments have been made in the reserves or mine planning. This has been shown to be valid in historic reconciliations between ore control and block modeling.

Ore control procedures previously developed and utilized at Hycroft will be used. Cuttings from production drill holes are collected following standardized procedures and delivered to the on-site laboratory for analysis. Results will be downloaded into an acQuire database. The ore control geologist then imports the assay results into mine planning software and merges them with survey information. Interpolation of the blast hole data will be performed in Vulcan software, producing an ore control block model. The ore and waste boundaries are then delineated using predefined modeling and ore routing parameters.

Ore will be routed to the appropriate process method based on geologic, metallurgic, and economic criteria. Ore polygons and bench maps are generated as required (typically daily) and provided to mine planners, operations supervisors, and all loading operators.

13.4 WASTE ROCK STORAGE

WRF's are typically constructed by end dumping waste rock from mine haul trucks over existing waste rock facilities, onto native alluvial material, or into existing pits. Design features include irregular shapes that blend the proposed and existing WRFs with natural topography, rounded bench crests, and abutments with undisturbed lands, concurrent reclamation where practicable, and varying slope angles on side slopes. As an WRF is constructed, the slopes of individual benches will be allowed to stand at the natural angle of repose. Generally, WRF will be placed using a lift/bench approach that is designed with bench setbacks sufficient to approximate the post-reclamation configuration employing 2.5H:1V slopes. This provides both operational stability and reduction of required reclamation effort. The WRF tops will be constructed without depressions and positive slopes to promote run-off from the tops and prevent ponding of meteoric water. The tops of the slopes will be rounded into the side slopes and the bottom of the slopes will be rounded out to blend into the surrounding topography. This design will limit erosion on the slopes and approximates a natural mature slope configuration found in nature.

In accordance with the approved Waste Rock Management Plan (HRDI, 2011), material identified as potentially acid generating (PAG) and non-potentially acid generating (non-PAG) will be mixed in WRFs during operation. Preliminary geochemical modeling to predict the chemistry of seepage from WRFs indicates that seepage through the facilities would be unlikely to impact waters of the State. Additionally, preliminary modeling results showed that meteoric runoff from non-PAG materials located on the surface of the WRF will be circum-neutral with all chemical constituents below required reference values.

Storm water from undisturbed areas upgradient will be diverted around the WRFs and will be returned to natural drainages during operations. The diversions will be designed to handle the 100-year, 24-hour storm event.

Prior to use, the proposed WRF footprints will be cleared of vegetation, and growth media will be salvaged and placed in proposed growth media stockpiles. Growth media includes salvaged material to be used for covering facilities during reclamation. To facilitate concurrent reclamation, salvageable growth media will be stockpiled as close to the place of use as possible, including direct placement on top of WRFs.

Pit backfill is also planned and will be sequenced with the mining operations. Waste rock will be placed in certain pits once mining is no longer economic. Currently, Hycroft is authorized to backfill the Bay Pit and portions of the Brimstone Pit. Backfill material will not be placed in areas of anticipated post-mining groundwater inflow (at or below 3,630 feet).

13.5 GEOMECHANICS

CNI and Golder have previously assessed the stability of slopes for pits at Hycroft. Geology models of alteration, rock type and major faults, along with hydrologic models were used to develop stability sections for the ultimate pit design. Those recommendations have been considered for the current pit design.

CNI separated the Hycroft pit design into sectors based on rock quality (alteration) and location with respect to the major faults. Golder Associates continued the use these sectors to update the geotechnical slope design recommendations in a 2016 review.

In current application, if a specific slope zone is not defined, the slopes are assigned by alteration. Slopes in Argilic and Propylitic alterations have been designed at a 38° inter-ramp slope angle. Acid Leach alteration is completely mined out, not having any significant final pit slopes. Any pit slopes which may remain in acid leach alteration is designed at 41° inter-ramp slope angle. Slopes in Silicic alteration are designed at 50° inter-ramp slope angle.

West of the Break Fault, slope stability is controlled by the strength of the unconsolidated Camel Formation. Overall slope angles in the upper portion of the west wall sectors of the Vortex domain vary from 30° to 38°. Below the 3600 elevation, the inter-ramp slope angle is 50°, as competent silicified rock is present in the wall (Golder, 2016). There are four slope sectors that define pit slopes west of the Break Fault: WBF-U, WBF-M, WBF-L, WBF-2U.

To the east of the Break Fault, the rock strength improves significantly due to the pervasive silica alteration of the Camel and Kamma Mountain Formations. Pit slopes for these sectors have been designed to an inter-ramp angle of 38 - 50°. There are three slope sectors that define the pit slopes east of the Break Fault: Vortex-1U, Vortex-1L and Vortex-3.

Thinly bedded rhyolite of the ALS is located in the footwall of the East Fault at depth. A 38° inter-ramp slope angle has been utilized for this sector: Vortex-2. A minimum 220-foot buttress (which is mined as one of the final mining phases) in the silicified rock in the hanging wall has been designed to manage the weak ALS unit that is present at the toe of the slope.

The hanging wall of the East Fault is composed primarily of silicified rock of the Camel and Kamma Mountain Formations and is defined by 2 slope sectors: FWEF-1, FWEF-2. Pit slopes for these sectors have been designed to an inter-ramp angle of 45°.

North of the deeper Vortex portion of the final pit are sectors associated with the Brimstone Pit. The highest slope in this area of the pit is the 1,200-foot-high east wall. The slope has been designed at an inter-ramp angle of 45° for this sector, identified as WBRIM. The East Fault will be mined out in this sector, thereby avoiding potential failure associated with the weak fault zone.

The west wall of Brimstone is less than 1,000 feet high. The upper 150 to 200 feet of the wall will be excavated in mine waste dumps. This stacked waste rock has been laid back to 35° degrees, roughly the angle of repose. Alluvium is present below the dumped waste. Inter-ramp angles of 42° have been utilized for this material. Silicified and argillically altered rocks of the Camel Conglomerate occur at depth. North Ramp Fault slopes will be constructed mainly in argillically altered Camel Conglomerate. An inter-ramp angle of 42° has been used for slopes in this sector. Both these wall sections are defined by the slope sector NRF.

Table 13-1 and Figure 13-1 summarize and illustrate the geotechnical slope sectors.

Table 13-1: Geotechnical Slope Design

Zone	Vertical distance between Catch Benches (ft)	Face Angle (deg.)	Catch Bench Width (ft)	Inter-Ramp Angle (deg.)	Whittle Profile	Whittle Slope (deg.)
Default	40	75	29.3	45	1	45
Argillic Alteration	40	75	40.5	38	2	38
Propylitic Alteration	40	75	40.5	38	2	38
Acid Leach Alteration	40	75	35.3	41	3	41
Silicic Alteration	40	75	22.8	50	4	50
WBF-U (above 4025 ft elev)	40	60	46.2	30	5	30
WBF-2U	40	60	28.1	38	6	32
WBF-M (4025 - 3600 ft elev)	40	60	40.9	32	7	32
WBF-2L	40	60	40.9	38	8	32
WBF-L (below 3600 ft elev)	40	75	22.8	50	9	45
Vortex-1U Argillic	40	75	40.5	38	10	38
Vortex-1U Silicic	40	75	22.8	50	11	45
Vortex-1L	40	75	22.8	50	12	50
Vortex-2	40	75	40.5	38	13	38
Vortex-3 Argillic	40	75	40.5	38	14	38
Vortex-3 Silicic	40	75	22.8	50	15	50
Vortex-3L	40	75	22.8	50	16	50
WBRIM	40	75	29.3	45	17	43
NRF	40	75	33.7	42	18	37
FWEF-1	40	75	29.3	45	19	40
FWEF-2	40	75	29.3	45	20	45
Camel-1	40	75	22.8	50	21	40
Fill(1)	40	60	34	35	22	35
Alluvium	40	75	29.3	45	23	45

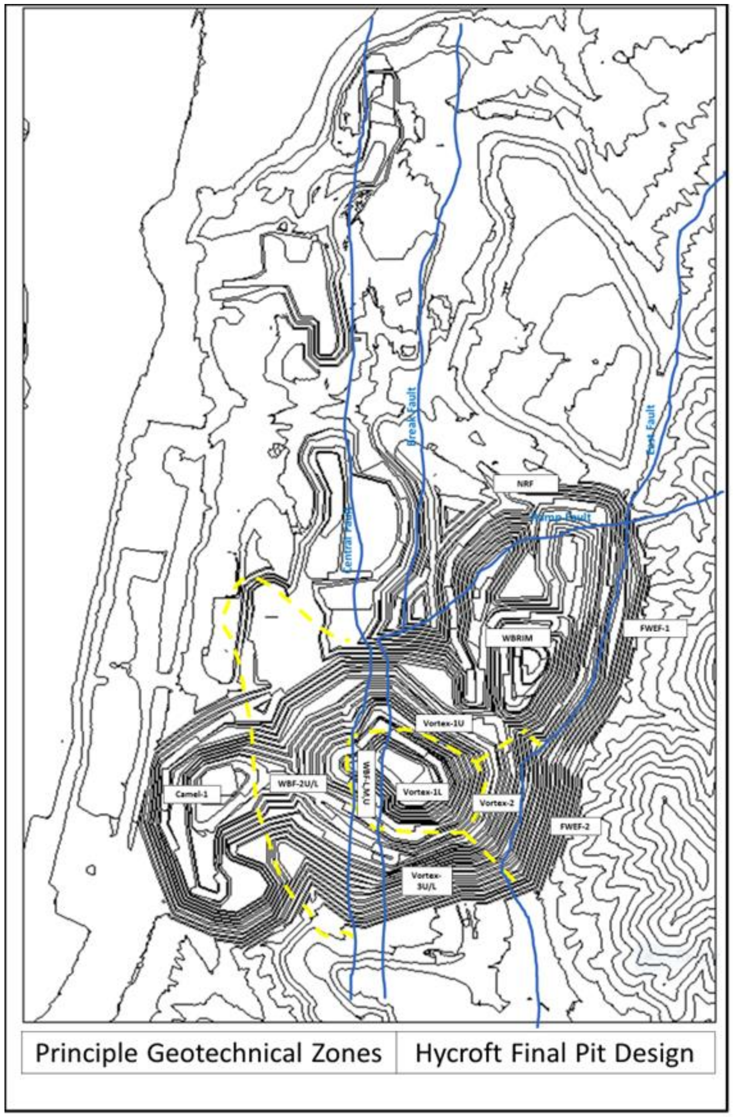


Figure 13-1: Geotechnical Sectors

13.6 MINE DEWATERING

Current assumptions are for the open pit to be mined to an ultimate elevation of 2,880 feet amsl. The potentiometric surface within the proposed pit boundary has been measured at approximately 4,200 feet amsl. Prior to excavation activities extending below the water table, dewatering will be required to ensure access to ore and to maintain a safe working environment. A combination of active dewatering by the wells with handling residual passive inflow at the bottom of the pit is proposed.

In 2011 and 2014, SRK evaluated the hydrogeologic setting of the Hycroft district, and preliminary estimated dewatering requirements for the mining plan designed for a mill feasibility study. The dewatering system estimates were reviewed by SRK in 2019 with the new pit design and slower pit sinking rates scheduled in this new feasibility study. The reviewed model predicts that dewatering rates varying between 316 gpm to nearly 1,600 gpm from seven pumping well centers over the 31-year dewatering period will draw the water table below the excavated pit as possible. Residual passive inflow to the pit is expected to be less than 200 gpm, which can be managed by an in-pit sump dewatering system.

In 2012 SRK and HRC constructed a prototype dewatering well and completed a long-term pumping test evaluating the conceptual dewatering strategy that focused on the highly transmissive faults identified during the characterization program. The prototype well was cased as a 10-inch well, completed within the Central Fault near its intersection with the Camel Fault, and designed for a target pumping rate between 300 and 500 gpm. Over the 7-day pumping test the average discharge rate was approximately 120 gpm. This lower than expected pumping rate was attributed to significant formation damage from multiple test holes completed in advance of the final well bore. Although the discharge rate was less than expected, the test did confirm the efficacy of a fault-centric dewatering strategy with the following findings:

- Data indicates that the Central Fault was highly transmissive (~3,500 ft^2/day) at the test location;
- Drawdown within the test monitoring network did not see evidence of a strong barrier to flow at the adjoining Camel Fault;
- The test did not provide evidence of a deep, geothermal upward flux beneath the mine area; and
- Static pressures in the monitoring instrumentation show a downward gradient in both the fault and in the footwall rocks.

The overall well design and siting strategy for the prototype dewatering well, forms the basis for the planning and implementation of active dewatering via groundwater well for the expansion of the Hycroft Mine.

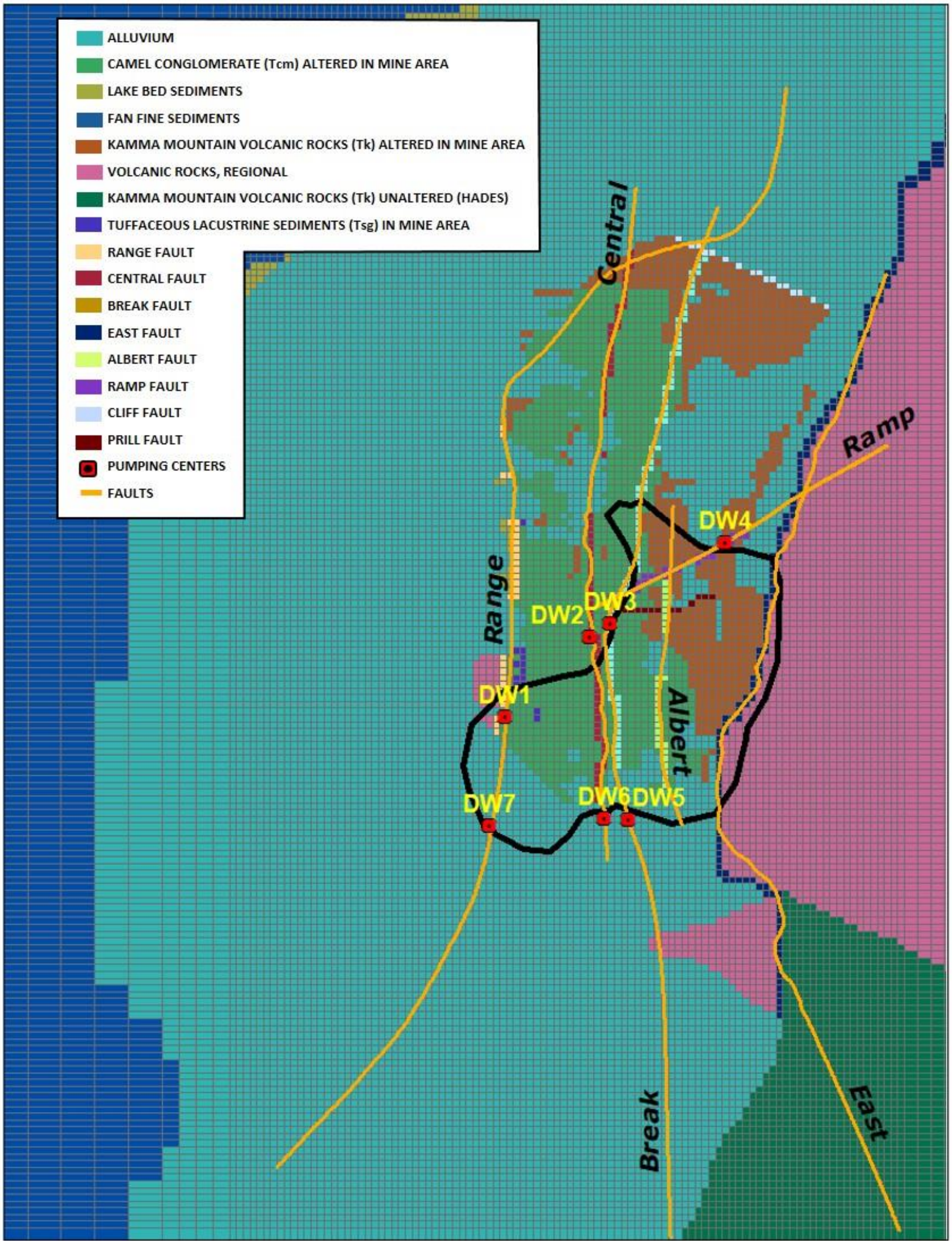
Active dewatering will be conducted through the installation of wells in approximately seven pumping centers around the perimeter of the open pit, targeting specific faults – Range, Central, Break, and Ramp. Wells will be installed to a maximum depth of 330 feet below the proposed lowest elevation of the open pit, though pumping depths may vary as the pit progresses wider and deeper. The location of seven pumping centers within identified transmissive faults around the open pit is shown in Figure 13-2. Identification of additional transmissive structures, or localized groundwater upwelling may be identified later in mine development as the pits connect and deepen beyond 4,000 ft amsl (Year 10 and beyond), necessitating modification of the dewatering strategy and/or construction of additional infrastructure.

Residual passive inflow to four individual lobes (Silver Camel, Brimstone North, Brimstone South, and Vortex) will be managed by in-pit sumps. If necessary, horizontal drains will be used in the pit walls, to reduce the pore pressures (particularly along the East Fault). Passive inflow water will be collected by the sumps and then pumped to the process facility for consumptive use.

LOM capital costs to construct and maintain the dewatering system are estimated to be $15.8 million. The system consists of ten pumping wells drilled from seven pumping centers, an in-pit sump with pump, piping to the pit rim, a pit rim water tank and piping to the process facilities. Additionally, 135 lateral drainholes were assumed for depressurization of the rock to the east of East Fault along the highwall of Brimstone Pit. The capital cost estimate was informed by the completed prototype well, and recent experience with projects of similar size and scope. The capital costs include a 20% contingency as presented to address technical uncertainty and cost estimation confidence. However, should significant changes in the conceptual hydrogeology of the local groundwater system be identified, costs associated with mine dewatering may be underestimated.

LOM operating costs for the de-watering program average $0.75 million dollars per year, primarily consisting of pump power costs and dewatering system maintenance. Early years are slightly lower, and costs rise as pit development advances. These operating costs include a 20% contingency as presented.

Mining to-date has not penetrated the water table. The feasibility mine plan remains above the water table for 3 years after mining begins, with consistent residual passive inflow not expected until Year 12.



Note: Map shows geology simulated in uppermost layer of numerical groundwater flow model (SRK, 2014). Black lines show extent of proposed pit below the water table and red dots indicate the location of identified dewatering pumping centers.

Figure 13-2: Location of Dewatering Wells Simulated by Groundwater Model (SRK, 2019)

Current design of dewatering system is based on understanding that bedrock in the mine area is generally low in hydraulic conductivity in peripheral areas where it is weakly altered to unaltered, but more permeable in the central mining area where alteration is strong, and the rocks are cut by subvertical transmissive faults. However, there are some uncertainties within the characterization of the hydrogeologic system including:

- Variability of hydraulic parameters within the various fault zones of the mining area. Limited testing along the lengthy strike of these structures has defined both high permeability and high variability across the strike of the faults. Hydraulic parameters of Range and Ramp faults, targeted for dewatering, are unknown but believed to be similar to values derived from testing in other structures; and

- The potential presence of a deep thermal groundwater system that could contribute additional inflows to the pit through faults. High water temperature and hot-spring gases suggest that an active geothermal system; however, there is no evidence that a prolific, hot-water aquifer remains beneath the site.

These uncertainties suggest the following risks in the dewatering strategy:

- Poor efficiency of dewatering via highly permeable structures would require:
    - Installation of additional dewatering wells; and/or,
    - Management of additional groundwater through passive collection within the pits.
- Increased vertical groundwater flux from a deep thermal groundwater source would require:
    - Increase pumping rate from dewatering wells; and or,
    - Installation of additional dewatering wells.

Under both risk scenarios, capital and operating costs associated with dewatering would increase. However, these risks have limited exposure early in the development and are more relevant as the pits deepen and merge after Year 10. Data collection on the dewatering system performance will inform dewatering strategies as the mine progresses, including any potential adjustments for these risks, allowing for planning and measured capital expenditure should a remedy be required. Furthermore, mine dewatering capital and operating expenditures have a 20% contingency included in part to address this risk. However, should significant changes in the conceptual hydrogeology of the local groundwater system be identified, active dewatering requirements and/or residual passive inflow may be underestimated, along with associated capital and operating costs.

13.7 LIFE OF MINE PLAN

The June 30, 2019 Life of Mine Plan (LOM) for Hycroft schedules the operation for 34 years of mining followed by 1 year of production leaching. A total of approximately 2.6 billion tons of ore and waste is scheduled to be mined. Mine production is scheduled to ramp-up from approximately 5 million tons the first year to 85 million tons per year by year 6, and ultimately to 100 million tons per year. Mining is completed using a mix of existing Hycroft fleet of mining equipment, contract mining, and new equipment purchases. Mining ramps up as new leach pad construction is completed and as additional crushing capacity and fine ore convey and stack is completed. The low initial strip ratios are a result of the historic mining that has stripped several pits down to sustainable ore.

The ore will be processed in three possible routes: ROM heap leach, ¾" crushed heap leach, and ½" crushed heap leach. The crushing plant capacity of (initially 25 million) 36 million tons per year is a main limit to both ore and total production. Due to increased margins on crushed material, the crusher is always kept at full capacity. ROM and ¾" crushed ores are not main drivers of the mine plan and it is processed as encountered in mining. Of the total 1.13

billion tons of ore, 43 million tons (4%) is routed as ROM heap leach, 24 million tons (2%) is routed as ¾" crushed heap leach, and 1.1 billion tons (94%) is routed as ½" crushed heap leach. Table 13-2 shows the LOM production schedule.

Table 13-2: Annual Production Schedule

Period	Waste Tons (000)	Ore Tons Processed (000)	Ore Grade (oz/t Au)	Ore Grade (oz/t Ag)	Contained Gold (000 oz Au)	Contained Silver (000 oz Au)	Total Tons Moved (000)	Strip Ratio
Year 1	0	4,488	0.018	0.113	80	506	4,488	0.00
Year 2	6,682	12,562	0.015	0.293	187	3,685	19,244	0.53
Year 3	12,268	23,278	0.011	0.383	255	8,918	35,546	0.53
Year 4	32,393	27,607	0.010	0.428	280	11,807	60,000	1.17
Year 5	45,912	28,689	0.012	0.340	353	9,759	74,602	1.60
Year 6	49,000	36,000	0.011	0.496	398	17,849	86,367	1.36
Year 7	44,085	40,915	0.010	0.480	407	19,628	86,182	1.08
Year 8	48,868	36,132	0.011	0.207	415	7,486	92,325	1.35
Year 9	46,975	38,025	0.009	0.570	357	21,661	85,000	1.24
Year 10	63,742	36,258	0.013	0.601	482	21,791	100,000	1.76
Year 11	63,689	36,311	0.011	0.664	398	24,095	100,000	1.75
Year 12	59,422	40,578	0.010	0.444	424	18,014	100,000	1.46
Year 13	63,173	36,827	0.009	0.606	344	22,311	100,000	1.72
Year 14	62,624	37,376	0.012	0.290	446	10,851	100,000	1.68
Year 15	63,910	36,090	0.009	0.324	313	11,707	108,388	1.77
Year 16	63,247	36,753	0.009	0.324	329	11,901	100,000	1.72
Year 17	63,250	36,750	0.009	0.330	342	12,136	100,000	1.72
Year 18	62,064	37,936	0.012	0.405	453	15,363	100,000	1.64
Year 19	48,873	36,127	0.014	0.650	507	23,476	85,000	1.35
Year 20	48,920	36,080	0.009	0.640	312	23,093	102,522	1.36
Year 21	47,910	37,090	0.009	0.356	319	13,188	96,353	1.29
Year 22	48,372	36,628	0.009	0.334	317	12,230	94,240	1.32
Year 23	48,972	36,028	0.013	0.541	455	19,489	88,305	1.36
Year 24	38,870	36,054	0.011	0.360	383	12,987	74,925	1.08
Year 25	26,356	36,009	0.011	0.253	379	9,094	67,350	0.73
Year 26	26,413	36,009	0.010	0.292	374	10,515	67,348	0.73
Year 27	25,061	36,000	0.010	0.290	378	10,425	66,039	0.70
Year 28	24,000	36,000	0.011	0.204	407	7,326	63,393	0.67
Year 29	14,000	36,000	0.010	0.180	352	6,488	59,774	0.39
Year 30	27,816	36,000	0.011	0.273	388	9,837	63,816	0.77
Year 31	24,164	36,000	0.011	0.429	394	15,439	60,164	0.67
Year 32	15,297	36,000	0.011	0.770	401	27,727	51,297	0.42
Year 33	5,526	36,000	0.009	0.818	328	29,462	56,775	0.15
Year 34	0	8,457	0.005	0.137	39	1,155	8,457	0.00
Total	**1,321,853**	**1,133,060**	**0.011**	**0.425**	**11,997**	**481,400**	**2,557,900**	**1.17**

13.8 MINE EQUIPMENT

Hycroft has a small existing fleet of mine equipment remaining on-site which is utilized in initial mining. First ramp-ups in production will utilize Contract mining with the contractor bringing in fleets capable of meeting mine production requirements. During Year 6 Hycroft will begin to re-capitalize with a new mining fleet and take over all mining activities by mid-year 7.

Table 13-3 lists the existing fleet as well as the major pieces of equipment planned by the Contract and new equipment to be purchased by Hycroft to achieve the scheduled production.

Table 13-3: Mining Fleet

Hycroft Existing Fleet	#
Hitachi EX3500 - 32 cu yd	2
Caterpillar 994 wheel loader	1
Komatsu 730E Trucks (200-t)	6
Caterpillar 16G Grader	2
Komatsu 475 Dozer	1
Caterpillar D10T Dozer	1
Caterpillar D11T Dozer	2
Caterpillar 834 RTD	1
Atlas Copco DML	1
Volvo A40D Water Truck	1
Komatsu 20k Water Truck	1
Fuel Truck A40D	1
Contractor Fleet	**#**
PC4000 - 29 cu yd	2
PC8000 - 55 cu yd	2
Komatsu 930E Trucks (320-t)	20
Blasthole Drills	7
Support Equipment	As Needed
Hycroft Purchased Fleet	**#**
Hydraulic Shovel - 47 cu yd	1
Hydraulic Shovel - 55 cu yd	2
Wheel Loader - 40 cu yd	1
Haul Trucks - 320-t	26
D11-Sized Dozer	6
18M-Sized Grader	3
RTD's	2
Water Trucks – 40k Gal.	3
Blasthole Drills	8
Service & Support Equipment	As Needed

The fleet is replaced at least one time over the mine life, with drills being replaced twice. Table 13-4 lists the hours utilized in the replacement schedule.

Table 13-4: Mine Equipment Useful Life

Useful Life	Hours
Hydraulic Shovel - 47 cu yd	80,000
Hydraulic Shovel - 55 cu yd	80,000
Wheel Loader - 40 cu yd	80,000
Haul Trucks - 320-t	120,000
D11-Sized Dozer	70,000
18M-Sized Grader	70,000
RTD's	70,000
Water Trucks - 40k Gal.	120,000
Blasthole Drills	60,000
Service & Support Equipment	As Needed

The following maps (Figure 13-3 through Figure 13-7) show the mine advancement starting at 2020 through end of mine life.

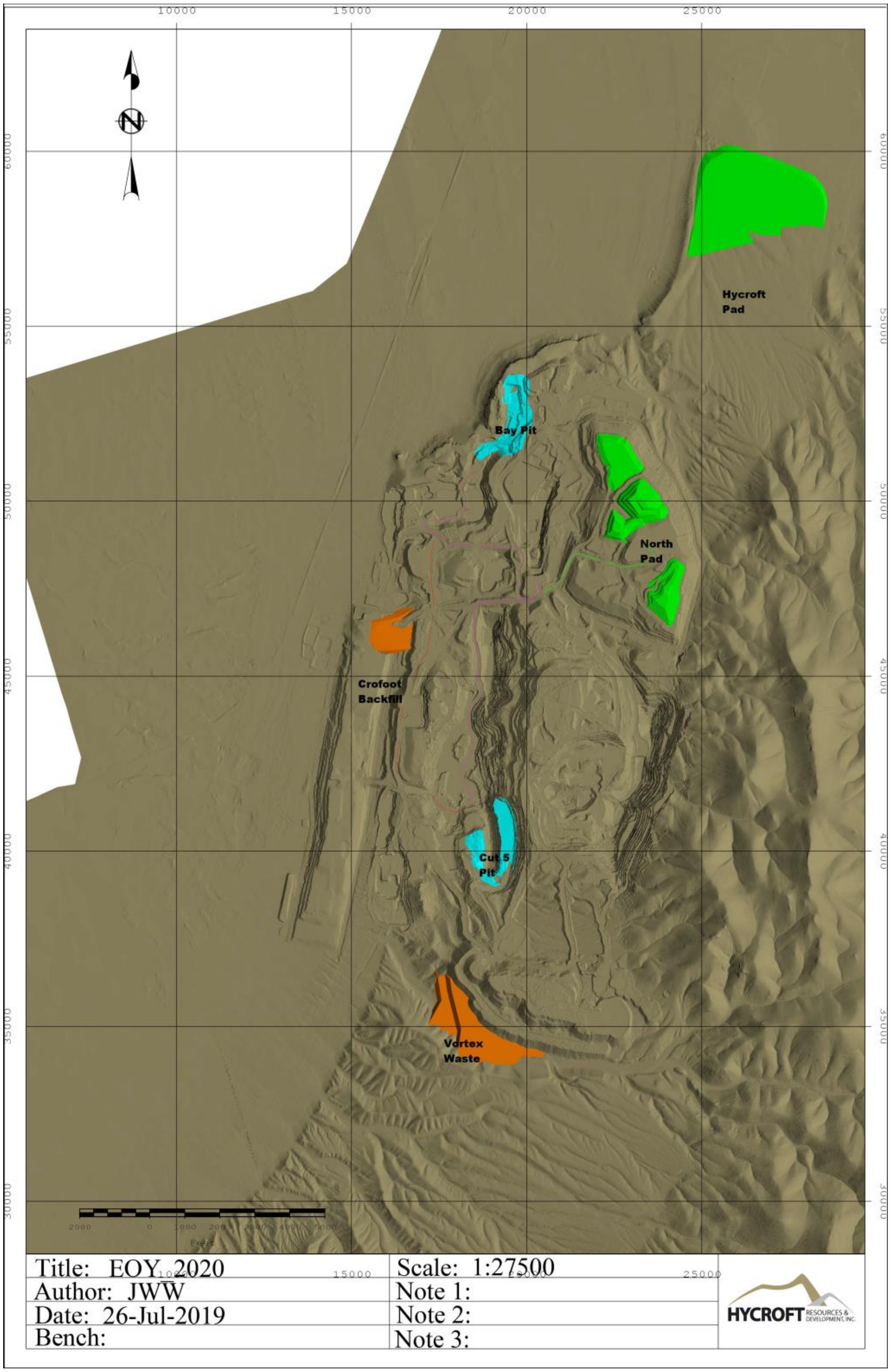


Figure 13-3: Pits, Dumps and Heap Leach End of 2020

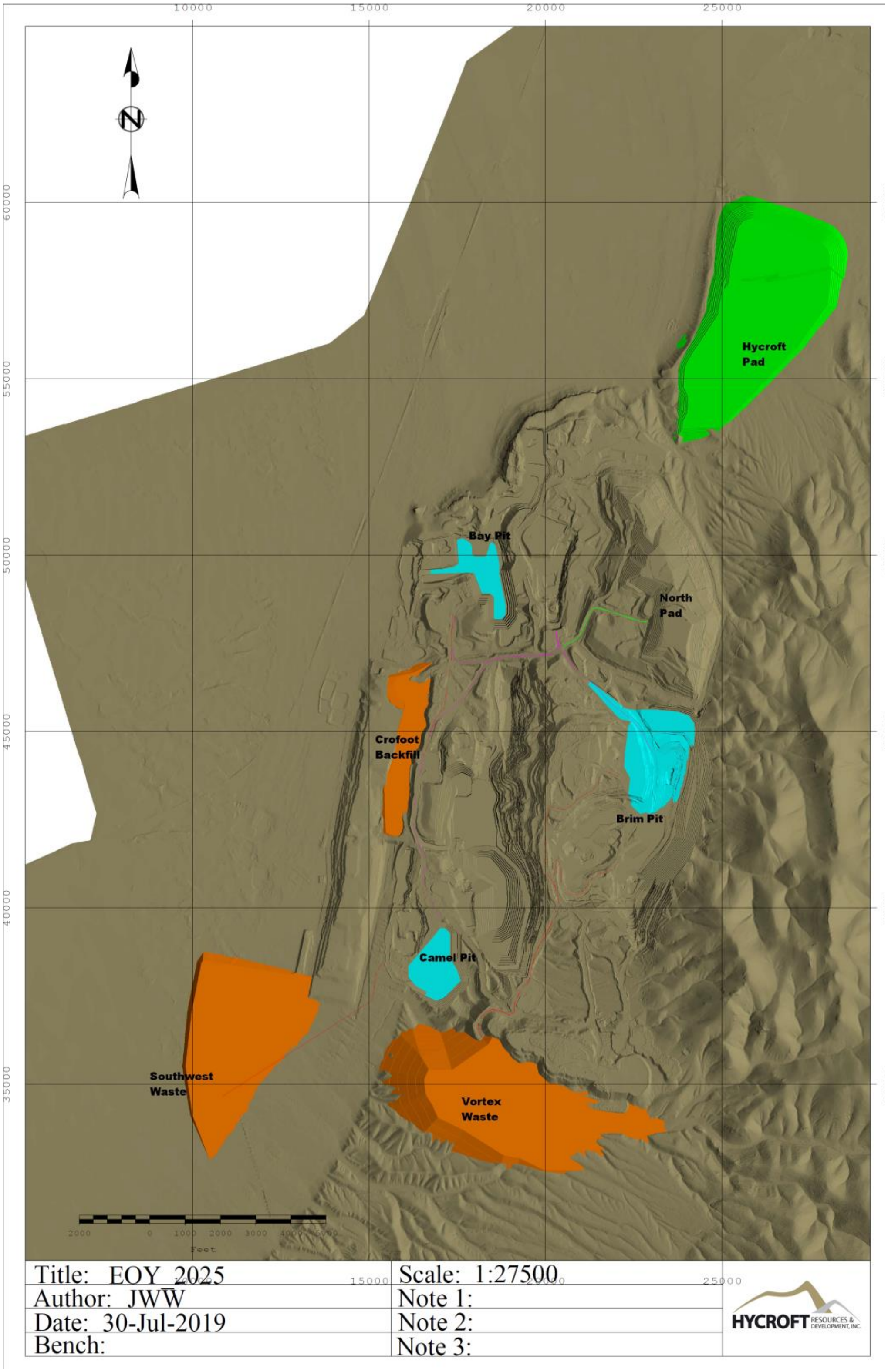


Figure 13-4: Pits, Dumps and Heap Leach End of 2025

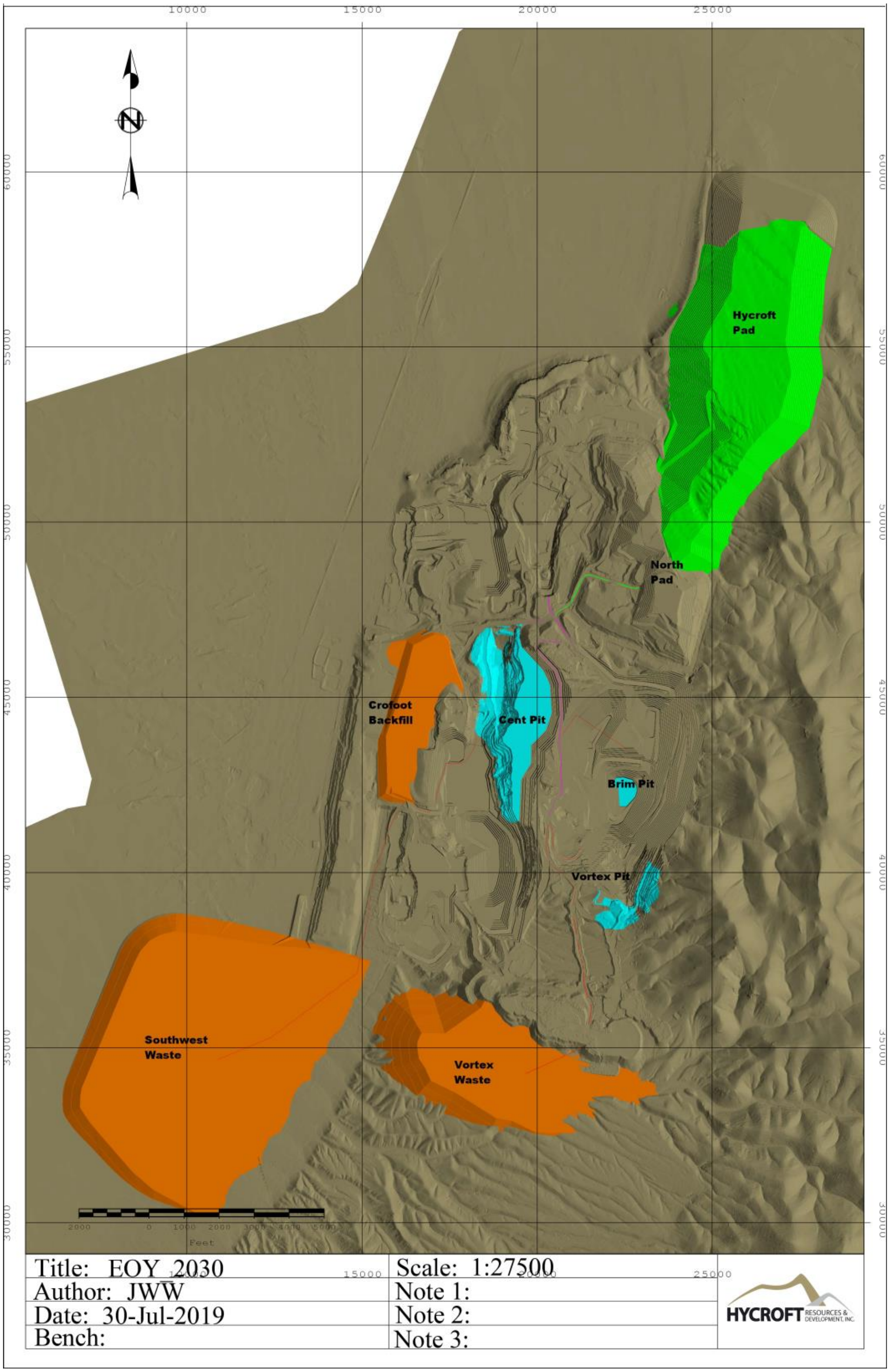


Figure 13-5: Pits, Dumps and Heap Leach End of 2030

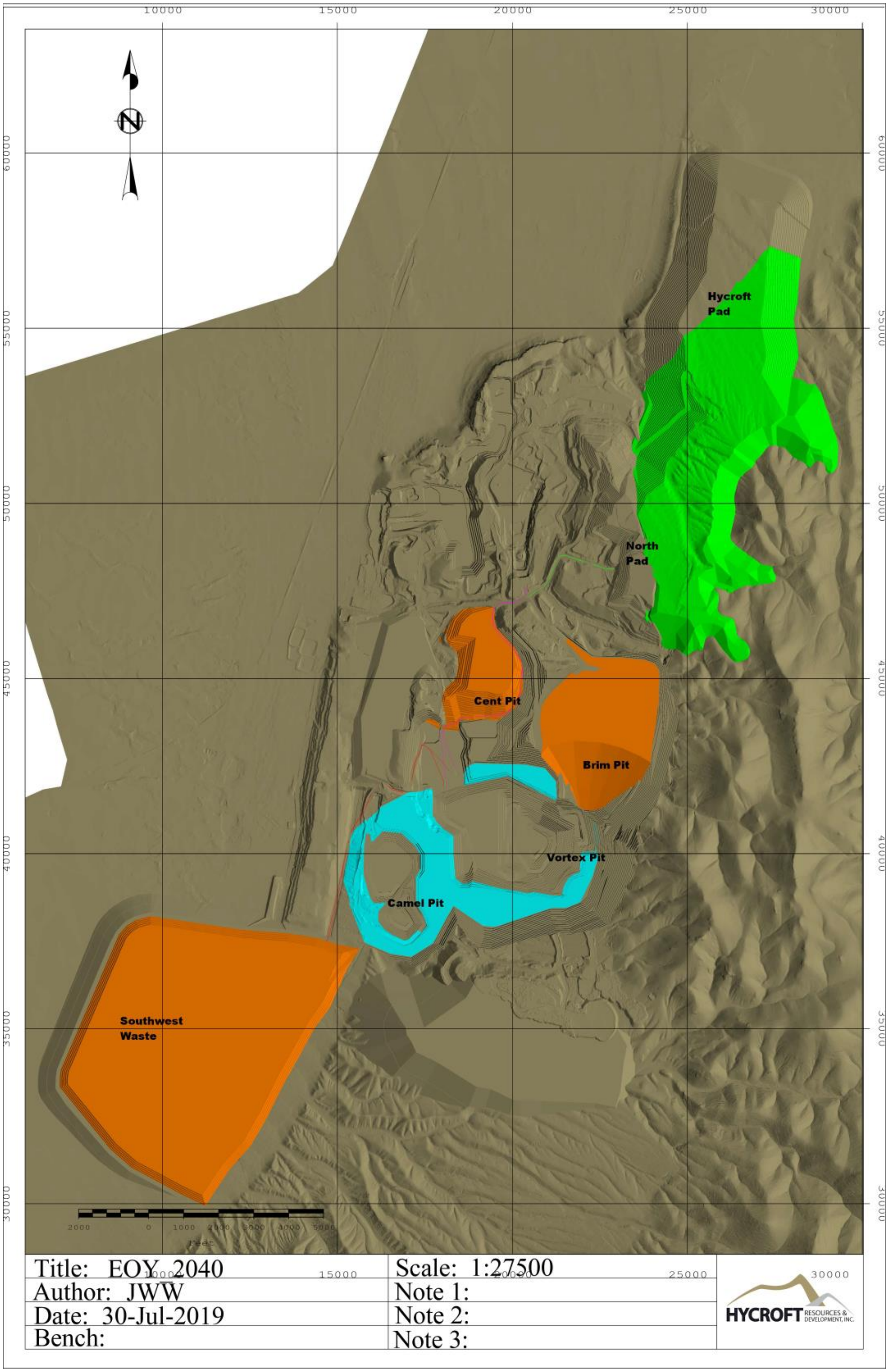


Figure 13-6: Pits, Dumps and Heap Leach End of 2040

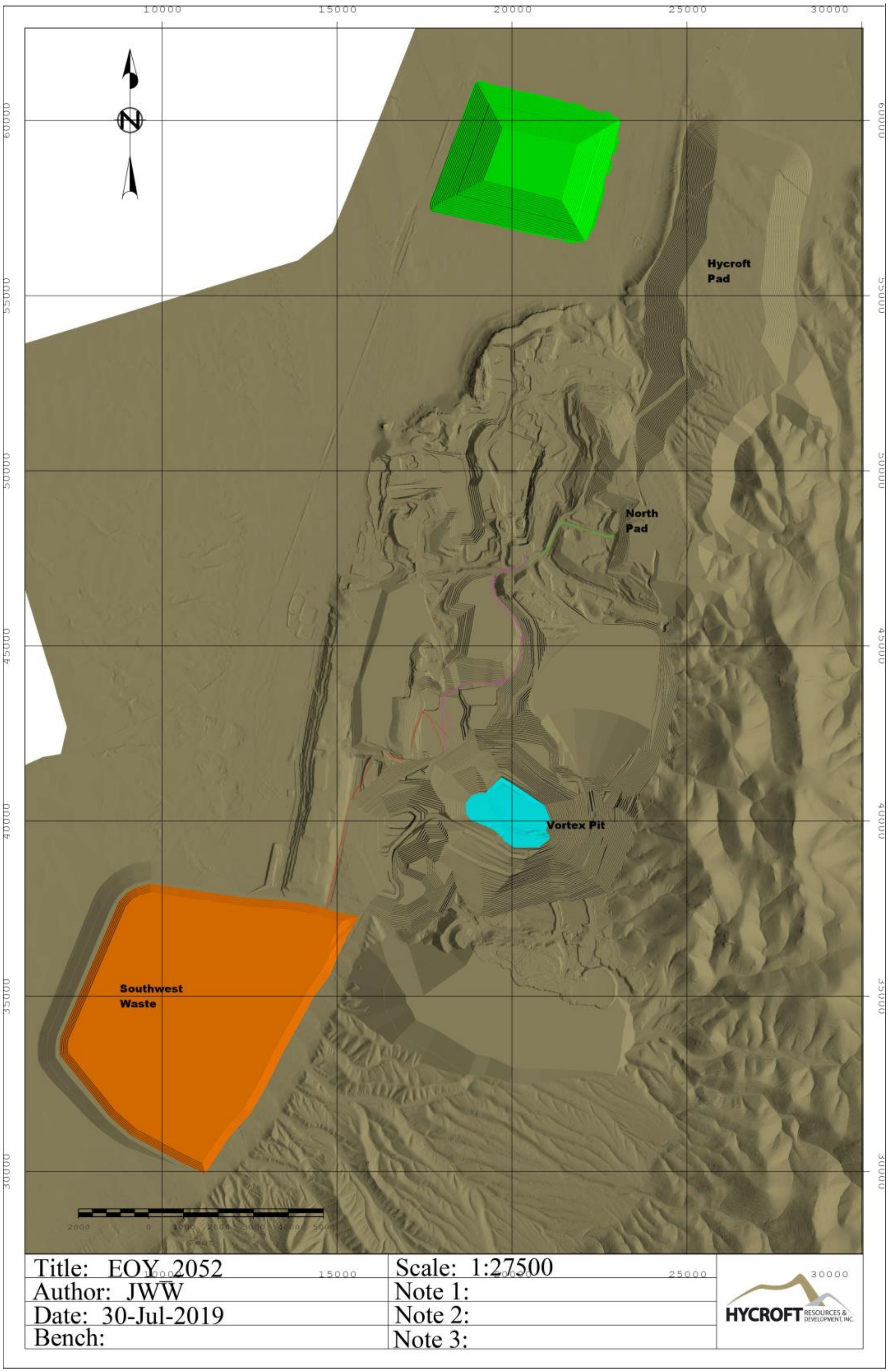


Figure 13-7: Pits, Dumps and Heap Leach End of 2052

14 RECOVERY METHODS

14.1 PROCESS DESCRIPTION

As discussed in Section 10, a significant portion of gold in the Hycroft ore is refractory due to its association with pyrite, marcasite and other sulfides. About 94% of the ore contains enough refractory gold to economically justify pretreatment by pre-oxidation prior to cyanide leaching.

The heap leach operation is designed to treat three categories of ore, classified as described below.Ore Category 1 (run-of-mine ore) and Ore Category 2 (3/4" crushed ore) both refer to oxide material that will be processed in a traditional heap leach scenario as has been applied at Hycroft in the past. These two categories combined make up only 6% of the total processed material in the life-of-mine model, so the emphasis has been placed on the discussion of oxidation and leaching of sulfide ore in this report (Ore Category 3), which are covered by a pending patent application.

The ability to oxidize and leach transition and sulfide ores in a heap process for gold cyanidation is novel in the industry. The chemistry is not new, but the implementation of the technique is critical to the success in operations. The testing program has shown that this is technically feasible by using a sodium-based neutralizer (trona or soda ash), which did not cause passivation and heap blinding due to calcium sulfate precipitation. Furthermore, the higher solubility of trona or soda ash also enables rapid neutralization in case a heap turns acidic. The level of metallurgical bench and pilot plant scale testing completed over the last four years has proven the new process to be successful. The restart of operations has included construction of three test pads (50,000 tons each, 25 ft lift height) to determine the commercial viability of the sulfide oxidation and leach process that has been developed. The development of the process identified several process components, three of which are critical, that are covered in Hycroft's patent application. M3, being part of this development process, is confident that, given the right conditions, the approach would be technically successful.

- **Ore Category 1** (ROM ore) – lower grade ore with high cyanide soluble gold is routed directly to the leach pad and cyanide leached to extract gold and silver. This accounts for 4% of the ore over the life of mine. The gold contents are highly soluble and the remaining refractory gold contents are not projected to justify the time and expense of a pre-oxidation step; therefore it will be stacked as 'ROM'. The ore in this category is typically defined as 'ROM oxide' or 'ROM transition'.
- **Ore Category 2** (3/4" Crushed ore) – higher grade ore with high cyanide soluble gold is crushed to a P80 of ¾" and cyanide leached to extract gold and silver. This accounts for 2% of the ore over the life of mine. The gold contents are highly soluble, but additional size reduction is expected to increase gold and silver recovery enough to justify the additional expense. The remaining refractory gold contents are not projected to justify the time and expense of a full pre-oxidation cycle. The ore in this category is typically defined as '3/4" crushed oxide' or '3/4" crushed transition'.
- **Ore Category 3** (1/2" Crushed ore) – low cyanide soluble ratio ores are crushed to a P80 of ½". The crushed ore is mixed with soda ash to induce an alkaline 'pre-oxidation' process. After the oxidation process has been completed to the desired extent, the ore will be rinsed sequentially with water and saturated lime solution, and then leached with cyanide to extract gold and silver. This accounts for 94% of the ore over the life of the mine. The ore in this category is typically defined as '1/2" crushed sulfide' or '1/2" crushed transition'.

Pregnant solution from the heap leach will be processed by two existing Merrill-Crowe zinc-cementation facilities.

The Hycroft Mine is projected to begin producing gold and silver from low-grade oxide ore and sulfide ore by cyanide heap leaching in the third quarter of 2019. Compared to a traditional oxide heap leach, cash flow is delayed by a length of time equivalent to the length of the dedicated pre-oxidation process.

Figure 14-1 is a simplified schematic of the process for the sulfide plant. This provides the basis for the process description that follows.

14.2 PROCESS DESIGN CRITERIA

HMC plans to ramp up production over five years to the design crushed ore tonnage of 36 million tons per year, starting with 4.5 million tons in 2019, increasing to 12.6 million tons in 2020, 23.3 million tons in 2021, and reaching the target 36 million tons per year by 2024. As discussed above, the yearly tonnage will be supplemented by a small percentage of ore that will be placed and leached as run-of-mine ore.

For the design, M3 uses an availability factor of 75% for the primary crusher, and 85% for the secondary and tertiary crushers if feed bins are used. These design availability factors are common for current and recent projects at M3 and in line with general vendor specifications. The stacking system that will be operational in Year 2024 will have an availability of 85%, which would be dictated by the crushing plant.

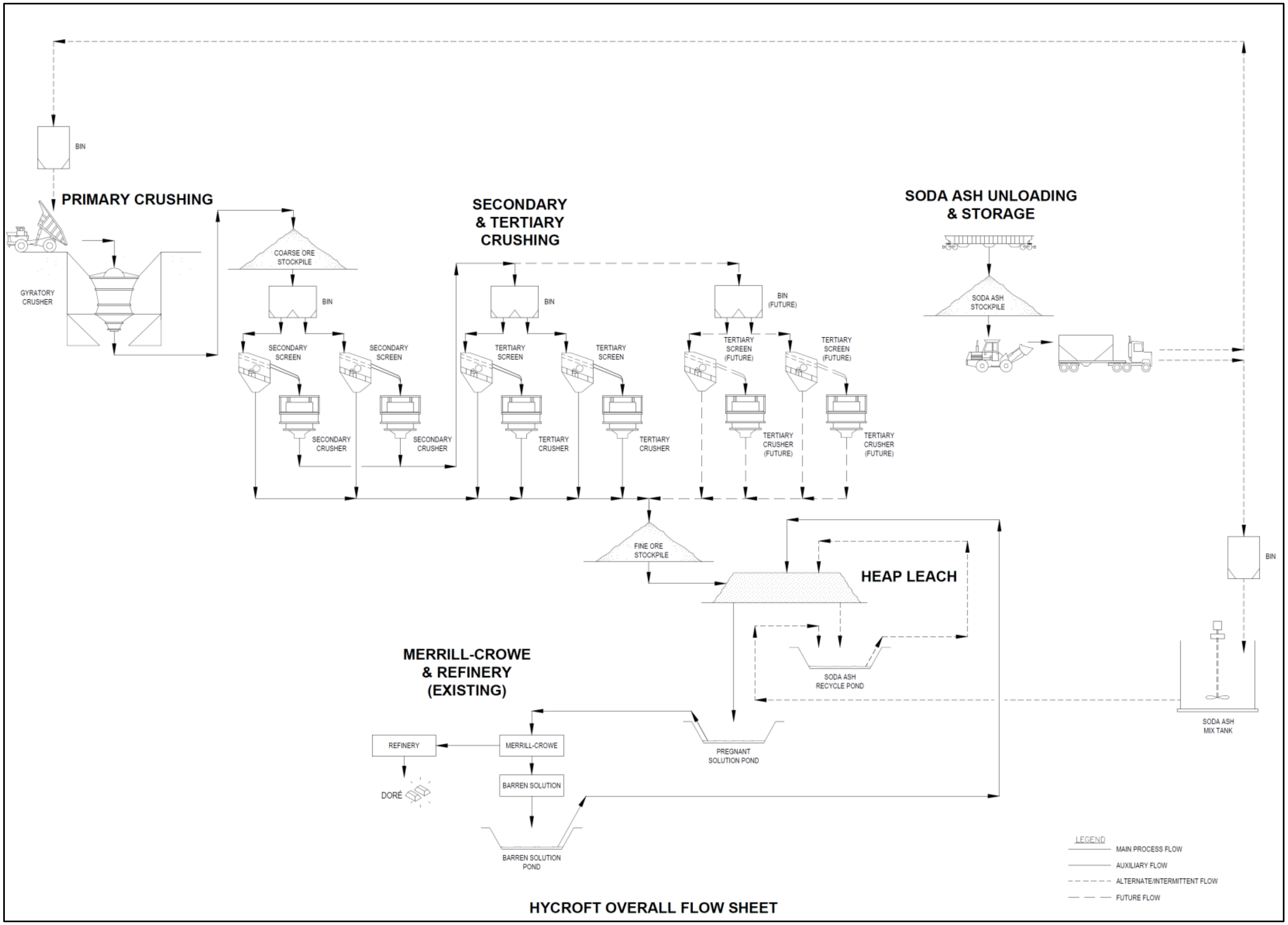


Figure 14-1: Simplified Process Flow Diagram for the Hycroft Sulfide Heap Leach Operation

Nomenclature and tracking of availability vary from operation to operation. For simplicity, M3 defines "availability" as the estimated actual run time of equipment. This would, therefore, include both "mechanical availability" and "use of mechanical availability" factors in an operating plant.

The mass balance was developed for the Hycroft process using MetSim software. The process simulation assumed overall grades and recoveries for gold, silver and sulfide-sulfur as shown in Table 14-1.

Table 14-1: Head Grades and Recoveries Used for Mass Balance Simulation

Metal	Head Grade	Heap Leach Recovery, %
Au	0.011 oz/t	65
Ag	0.425 oz/t	71

The mine plan was based on recovery and operating cost models, which were also used in the financial analysis in this study. It yielded life-of-mine average head grades of 0.011 oz/t Au, 0.425 oz/t Ag and 1.92% sulfur. Predicted recoveries vary from ore to ore, depending on Au, Ag and sulfide-sulfur contents, as discussed in detail in Section 13 of this report.

The MetSim balance forms the basis for water balance and equipment sizing, including pipes and pumps, as well as sumps or pump boxes, and defines the parameters used in the process design. Table 14-2 is a summary of the main components of the process design criteria used for the study.

14.3 CRUSHING PLANT DESIGN

The crushing plant is designed to run a nominal capacity of 98,630 stpd to attain the 36 million ore tons per year target. Hycroft has installed one primary crusher, two secondary crushers and two tertiary crushers. The existing facility will be sufficient during the ramp-up period but will require addition of two more tertiary crushers to attain the design capacity. Processing parameters in the following discussions are derived from simulations of the full plant at the design capacity.

14.3.1 Primary Crushing and Crushed Ore Stockpile

Ore will be transported by haul trucks from the mine to the existing primary crusher via a dump pocket with a 960-ton live capacity (3 truckloads). The primary crusher is a 60" x 113" gyratory crusher, with an open side setting of 7 inches and a feed opening of 60 inches. It is powered by a 1000-hp motor.

The crushed ore is discharged via a surge bin to an apron feeder. The ore is then transferred by a stacker conveyor to a coarse-ore stockpile. A belt scale is provided on the stacker conveyor to measure the amount of crushed ore delivered. A self-cleaning magnet is provided to remove any tramp steel before stockpiling.

The coarse ore stockpile has a live capacity of 27,400 tons and a total capacity of 160,000 tons. The live capacity is nominally equivalent to about 6.7 hours of heap leach feed at peak production.

The crushed ore is reclaimed via two reclaim tunnels beneath the stockpile. In the tunnels are identical reclaim lines, each comprising three reclaim feeders (two operating and one standby) and one reclaim transfer conveyor. Each reclaim feeder has a design capacity of 1,359 tph. The crushed ore is reclaimed from the stockpile at a design rate of 2,485 tph per line. Each reclaim conveyor discharges to the secondary crusher feed bins.

Dust suppression units and bag houses are installed to suppress or remove dust generated by dump trucks, crushers and other material handling equipment.

Table 14-2: Process Design Criteria Highlights

DESCRIPTION	DESIGN
Capacity	
Tons per year, tpy	36,000,000
Tons per day, tpd, nominal	98,630
Primary crusher, tph	5,480
Secondary & Tertiary crushers & stacker, tph	4,835
Availability/Use of Availability	
General	85%
Primary Crusher	75%
Secondary Crusher	85%
Tertiary Crusher	85%
Stacker	85%
Primary Crusher	
Feed F80, inches	8.2
Product P80, inches	5.2
Crushing work index, kWh/st, 80th Percentile	9.3
Secondary Screening	
Type	2-deck, multi-slope
Screen opening, top deck, inches	3
Screen opening, bottom deck, inch	0.75
Secondary Crushing	
Type/cavity	Standard/Medium
Closed-side setting, inches (mm)	1.125 (29)
Feed F80, inches	6.6
Product P80, inches	1.7
Tertiary Screening	
Type	1-deck, multi-slope
Screen opening, inches	0.75
Tertiary Crushing	
Type/cavity	Short head/Fine
Closed-side setting, inches (mm)	0.55 (14)
Feed F80, inches, Sec/Tert	1.8
Product P80, inches, Sec/Tert	0.47
Pre-Oxidation	
Moisture Content, %	8 - 12
Temperature, °F	Ambient
Oxidation Time, days	30 - 180
Sulfide-Sulfur Oxidation, % (Ore dependent)	40% Max
Soda Ash Consumption, lb/t	14.5
Cyanidation	
Leach Time, days	
ROM	200
¾” Crushed	200
½” Crushed	100
Application Rate, gpm/ft^2	0.0025

14.3.2 Secondary and Tertiary Crushing

Hycroft Mining currently operates four Raptor XL1300 cone crushers – two standard and two short heads, on secondary and tertiary duties, respectively, for the heap leach operation. The crushers are driven by 1,300-hp motors. Each crusher is fed from a bin through a vibrating screen. The two additional tertiary crushers required by Year 6 would preferentially be Raptor XL 1300 cone crushers for commonality of spares. However, equivalent crushers for other vendors may be considered in the future.

All four existing cone crushers failed on commissioning due to original mechanical design flaws. Recently, the manufacturer has redesigned the mechanisms and replaced the internals of all four crushers.

Coarse ore from the secondary crusher feed bin (500 st capacity) is fed to the secondary screens. The secondary screens are double-deck Ludowici banana screens, 10 ft by 24 ft. The crushing/screening simulations call for 3-inch (75 mm) and ¾-inch (19 mm) apertures for the top and bottom decks, respectively. Oversize materials from the two decks proceed to the secondary crushers while the undersize of the lower deck goes to the final crushing plant product.

The secondary crusher is fitted with a standard medium cavity and operated at a closed side setting of 1-1/8 inches. Coarse ore will be crushed to about 80% finer than 1.57 inches (40 mm). Product of the secondary crushers is then be conveyed to the tertiary crusher feed bin (500 st capacity).

From the tertiary crusher feed bins, the ore is fed to the tertiary screens, which are single-deck Ludowici banana screens, 12 ft by 28 ft, with 3/4-inch (19 mm) aperture screens. Oversize materials comprise the feed to the tertiary crushers. The undersize of the screen goes to final crushing-plant product.

The tertiary crushers are fitted with short-head fine cavities and operated at a closed side setting of 0.55 inch (14 mm). The material will be crushed to about 80% finer than 0.466 inch and become the final installment into the crushing-plant final product.

Overall, the product of the crushing plant will have a P80 of approximately 0.5 inch (12.7 mm). This size distribution will be characteristic of the material that is stacked on the heap.

The final crushed ore product will be conveyed towards the heap leach facility, either to a stockpile or directly loaded to trucks, which will transport the ore to the heap. A conveyor stacking system is planned to operate in Year 6 (2024) of operation.

14.4 CONVEYING AND STACKING

The final crushed ore product will be conveyed towards the heap leach facility, either to a stockpile or directly loaded to trucks, which will transport the ore to the heap.

A conveyor stacking system is planned to operate in Year 6 (2024) of operation. This will include the existing stockpile conveyor modified to discharge to the first of three new overland conveyors in series. These overland conveyors will take the crushed ore to the stacker. One or more grasshopper conveyors will connect the overland conveyors to the stacker, over the side slope of the heap, as required.

14.5 PRE-OXIDATION

Pre-oxidation of sulfide and transition ore (crushed to ½”) will begin at the crusher using in-situ moisture and solid soda ash. The soda ash requirement for the ore is relative to the %sulfide-sulfur content of the ore, starting cyanide soluble Au and the target oxidation rate. Regular sampling of mined material will allow reagent addition control; for the life of mine the average soda ash consumption is projected to be 14.5 lbs per ton of placed ore.

The addition of soda ash creates an alkaline environment (60,000 ppm of total Alkalinity, pH 10+) that allows some of the ferrous and ferric ions to remain in solution by complexing with carbonate ions. As discussed in Section 10, the presence of ferrous and ferric carbonate complexes for a redox pair that enhances the oxidation of iron sulfides in an heterogenous electrochemical reaction.

As the reaction proceeds, soda ash will be consumed to neutralize the resultant acid and additional soda ash will be introduced to maintain optimal reaction conditions.

Once ore has been placed on the heap, additional soda ash solution will be applied to bring the ore to field capacity (8 – 10% moisture). The solution in the heap will be replenished on a regular basis using soda ash solution in order to offset evaporation and carbonate consumption. Soda ash solution will be pumped through pipes/tubing that are separate from the lixiviant solution system.

The dissolved oxygen required for the reaction will be replenished through solution to air contact; the dissolved oxygen will be monitored inside the heap using embedded recoverable sensors. If required, air inflow can be aided by installing large perforated piping at the bottom of each panel, with ends protruding out of the heap.

Pre-oxidation duration will be determined by the characteristics of the ore and the measured extent of oxidation based upon sulfate production. The extent of oxidation will be determined by the target recoveries for each domain and the initial cyanide soluble gold, which is translated to degrees of oxidation already achieved. The number of days required to attain target oxidation is dependent upon the sulfide-sulfur content of the ore with, higher sulfide-sulfur corresponding to longer oxidation cycles. The majority of the ore is expected to take between 30 and 120 days to finish pre-oxidation. This is measured between the day that soda ash is introduced to the ore at the crusher and the day that the 'rinse' cycle begins for the panel.

14.6 RINSE CYCLE

Ore that has undergone a pre-oxidation cycle must be rinsed, first with water, then with a saturated lime solution prior to the commencement of cyanidation. The purpose of the rinse is to wash down as much sulfate and bicarbonate, as possible.

If not removed, sulfate will precipitate as $CaSO_4$ during the leach cycle and potentially form a passivation layer against cyanidation. Bicarbonate, on the other hand, has been shown to react with cyanide to form HCN, which is not active in leaching and eventually escapes from solution, thereby increasing the cyanide consumption. The amount of rinse water required would at least be one pore volume replacement, but this should be monitored until the sulfate and alkalinity levels in the rinse water reach a threshold concentration, e.g., 2,000 ppm sulfate and 2,500 ppm total alkalinity.

Saturated lime water may be applied to scavenge the residual bicarbonate in the heap. The added complexity and cost of saturated lime water will have to be weighed against the resulting savings in cyanide consumption. If the remaining bicarbonate is low enough, or the leach solution has enough alkalinity that the pH can be maintained at 10.5, then most of the bicarbonate will be converted to carbonate, which does not react with cyanide. If used, the lime-saturated water will be applied to panels that have undergone pre-oxidation at a rate of 0.0025 gpm/ft^2, until one or two pore volumes have been displaced.

Rinse solutions will be supplied using the same piping that delivers lixiviant during the leach phase. Displaced solution will be sent to the soda ash recycle pond.

14.7 HEAP LEACH CYANIDATION

The cyanidation conditions for all placed ore will be the same regardless of crush size or the use of pre-oxidation. The duration that these conditions are maintained is dependent on the category to which the ore belongs. For panels under active leach, a cyanide concentration of 1.0 lbs/ton of solution will be maintained. The pH will be controlled using lime.

Oxide and transition material that will be leached as ROM will proceed directly from the pit to the heap and begin cyanide leach without undergoing pre-oxidation or rinse. A small percentage of oxide and transition material will be directed to the crushing plant to be reduced to a P80 of ¾" before being stacked and commencing cyanide leach. Ores from both of these categories are expected to undergo a 200-day primary leach cycle using a conservative 3:1 solution to ore ratio and an application rate of 0.0025 gpm/ft^2.

Sulfides and a portion of the transition material will be reduced to a P80 of ½" before undergoing the pre-oxidation and rinse processes on the heap. At the conclusion of the rinse a 100-day primary leach cycle will begin. A 1:1 solution to ore ratio and an application rate between 0.0015 gpm/ft^2 and 0.0025 gpm/ft^2 will be used.

14.8 MERRILL-CROWE PRECIPITATION AND REFINERY

Due to the high silver content of the pregnant solution, gold and silver will be recovered by zinc cementation. Hycroft Mining has two Merrill-Crowe plants that are used to process the pregnant solution from the heap leach operation. The older plant has a capacity of 4,500 gpm. The newer plant is considerably larger, with a capacity at present of 21,500 gpm, for the total of 26,000 gpm capacity.

The wet filter cakes from the low-grade and high-grade Merrill-Crowe circuits will be transferred to retort pans, which are then put into a retort furnace to remove water and mercury. Water and then mercury are sequentially volatilized from the precipitate by heating the precipitate under a partial vacuum. The exhaust gases pass through multiple stages of condensers that drain mercury and water to a collection vessel. The last traces of mercury are removed from the retort gas by a packed bed of sulfur-impregnated carbon before being released to the atmosphere. The retorts are typically operated batch-wise, with a cycle time of approximately 18 hours.

The dried filter cake will be mixed with flux and then transferred to an electric arc furnace where it is smelted to produce doré.

14.9 WATER BALANCE AND SOLUTION MANAGEMENT

Hycroft is currently permitted to use fresh water at a yearly average rate of 12,700 gpm. The estimated fresh water requirement is 3,189 gpm when the heap leach is operating.

Water balance and solution management for the Hycroft operation is complicated by the gradual buildup of sodium sulfate and sodium bicarbonate to a steady-state concentration in the reclaimed water. Sulfate ions were seen in some tests to slow down the sulfide oxidation reaction. Because of this, fresh water addition to the soda ash recycle pond is designed to maximize the dilution of sulfate and bicarbonate ions in the pre-oxidation circuit.

Approximately 690 gpm of the fresh water is allocated for mine dust suppression. All fresh water will be drawn from existing wells that have been operated to supply the property in the past.

A water balance was developed for the Hycroft project as part of the mass balance model using MetSim modeling software. The water and solution management scheme is illustrated in Figure 14-2 below. The acronyms used in Figure 14-2 are: FW = fresh water, SW = seal water, PW = process water, BS = barren solution.

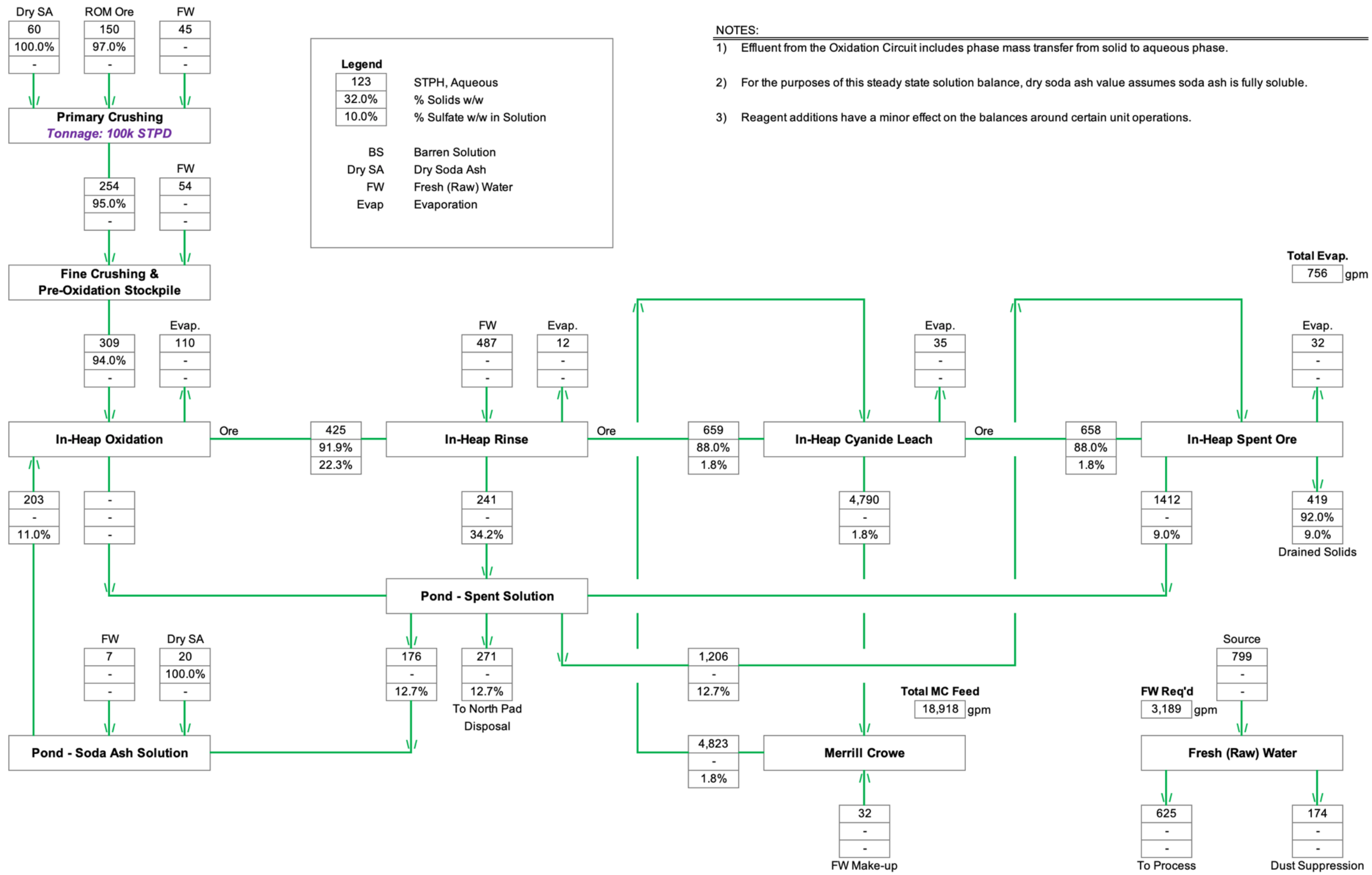


Figure 14-2: Water Balance Model

14.10 REAGENTS AND CONSUMABLES

14.10.1 Consumption Rates

Reagent storage, mixing and pumping facilities will be provided for all of the reagents used in the processing circuits. Table 14-3 below is a summary of reagents used in the process plant.

Table 14-3: Process Reagents and Consumption Rates

Reagent & Consumables	Units	Consumption
Soda Ash	lb/t	14.5
Lime	lb/t	1.4
Sodium cyanide	lb/t	1.0
Zinc dust	lb/koz Au	34.15
Zinc dust	lb/koz Ag	62.35
Zinc dust	lb/koz Hg	67.06
Primary Crusher - Liners	lb/t	0.005
Secondary Crusher - Liners	lb/t	0.004
Tertiary Crusher - Liners	lb/t	0.002

14.10.2 Soda Ash Handling

Soda ash will be delivered to the site by rail and dumped to a stockpile, which is housed in a building for dust control and to protect the product from weather. A front-end loader will move material from the stockpile to a 500-ton silo outside of the building. The silo will directly load 50-ton trucks that will deliver soda ash to its points of use across the mine. There are 2 points of use; a soda ash mix tank near the ponds, and the drive through silos that will comingle soda ash with ore in trucks that are headed to the primary crusher. The drive through silos will be where the majority of soda ash is introduced to the process. Trucks will be pneumatically unloaded into the silos at each point of use.

14.11 POWER CONSUMPTION

The power consumption in the process plant for a typical year (Year 7) is tabulated in Table 14-4 with a total consumption of 119 million kWh. This translates to about 3.3 kWh/ton or US$0.18/ton of ore processed.

Table 14-4: Summary of Power Consumption in a Typical Year (Year 7)

Area No	Mill Area	Annual kWh	Annual Cost
100	Primary Crusher	6,143,37	$ 344,029
150	Secondary & Tertiary Crushing	45,836,031	$ 2,566,818
300/350/550	Heap Leach/Stacking/Merrill-Crowe	57,824,931	$ 3,238,196
650	Water Systems	4,666,283	$ 261,312
800	Reagents	3,272,965	$ 183,286
810	Rail Unloading & Storage	986,396	$ 55,238
	Total	**118,729,982**	**$ 6,648,879**

14.12 CONTROL SYSTEMS

A crusher control room, located in the primary crusher area at the mine, will be the operating and control center for the crushers and coarse ore transport conveyors, as well as their ancillary support facilities. Solution management and stacking conveyor monitoring will be added to this control room.

14.13 PLANT SERVICES

14.13.1 Mobile Equipment

Table 14-5 lists the mobile equipment that is provided in the project capital cost estimate.

Table 14-5: Mobile Equipment List

Description	Qty	Duty
D65	1	Grading, general maintenance
D11 Dozer	2	Pad ripping, Pipelayer, special projects
D10 Dozer	1	Reclamation, general maintenance
4WD Articulated Wheel Loader	1	Soda ash handling, Clean-up
50 Ton Off-Highway Truck	2	Soda ash distribution

Mobile equipment that will be used to build leach pads is not included in the equipment list. All equipment used for the construction of leach pads will be provided by the contractors awarded the work.

14.13.2 Assay and Metallurgical Laboratories

The existing laboratory facility will be used to support mining operations. Facilities include sample receiving and storage, sample drying, sample preparation, metallurgical laboratory, wet laboratory, and fire assay for mine and process plant samples.

Forecasting and reagent dosing will be based upon analytical processes which determine metal grades, sulfide sulfur content and paste pH. Average sulfide sulfur content and paste pH will be established from drilled samples to ensure that soda ash requirements are determined prior to run of mine ore delivery to the crusher.

14.14 PRODUCTION ESTIMATE

Production by project year is tabulated in Table 14-6, showing production from the heap leach process.

Table 14-6: Hycroft Metal Production

Year	Gold Sales (oz)	Silver Sales (oz)
2019	44	399
2020	70	1,858
2021	164	2,641
2022	190	7,855
2023	196	5,795
2024	238	8,326
2025	202	8,375
2026	203	9,242
2027	361	12,071
2028	231	12,844
2029	266	14,040
2030	269	15,070
2031	227	10,195
2032	313	16,207
2033	267	8,356
2034	196	7,709
2035	206	8,010
2036	219	7,926
2037	259	9,740
2038	300	14,891
2039	214	16,188
2040	328	13,650
2041	209	8,968
2042	249	11,026
2043	229	7,812
2044	225	6,075
2045	219	6,669
2046	225	6,703
2047	382	7,661
2048	216	5,042
2049	269	8,671
2050	268	12,904
2051	214	35,947
2052	200	17,653
Total	**7,868**	**346,519**

15 PROJECT INFRASTRUCTURE

The infrastructure for the Hycroft project has been developed considering the existing facilities and the requirements of the project. This section describes the additional infrastructure required to support the Hycroft heap leach expansion project described in this report.

15.1 POWER SUPPLY

For the Hycroft Heap Leach Feasibility Study, approximately 11.5 MVA of new load will be added to the existing Hycroft substation. It was determined that the existing 37.5 MVA transformer and overhead power lines have enough capacity to accommodate this load. The 60 kV transmission voltage will be stepped down by various transformers around site to 4160V and 480V distribution voltages. All new loads will be fed from new switchgear and motor control centers. New power poles, overhead conductors, and miscellaneous equipment will need to be installed, extending from the existing overhead lines, and run to the soda ash unloading area and the heap leach electrical building.

Eventually, the new Heap Leach Pad will overtake the utility power lines coming into site. Because of this, during year eight of operations, the existing utility switchyard will be replaced and relocated from Lewis Camp and Jungo Roads closer to the soda ash area to the west on Jungo Road. New power lines will be run to the existing substation from the soda ash area switchyard. The 60 kV power lines can piggyback on the new poles and lines that are being installed and run to the soda ash area during project construction, or alternatively, new poles and lines can be installed during year eight. The older power lines and poles coming into site will be abandoned or demolished during that time.

An upgraded 60 kV transmission line and new substation was completed in Q2 2013 to provide additional power for the crushing system, Merrill-Crowe facility, electric rope shovels and existing mine site operations.

15.2 WATER SUPPLY

Fresh water will be obtained from existing active and inactive production wells in a field west of the mine, and from mine dewatering. Plant water requirements are projected to fall well below the 12,700 gpm that the site is permitted for.

15.3 COMMUNICATION

The site has new data and telephone communications provided by microwave facilities serviced from two different locations. A new 140-foot tower has been installed near the administration building and is the demarcation point for distributing bandwidth around the mine site. Voice and data are distributed around the mine site using fiber optic cable where possible and data radios where the use of fiber is not possible. Cellular communications are now available throughout the site.

15.4 RAILROAD

In order to receive soda ash and potentially fuel, reagents and other supplies, HMC will construct a rail siding off the existing rail line located north of the plant. M3 has provided a design for the rail unloading and soda ash storage facility which will be constructed in parallel with the first throughput ramp-up of the crushing plant. The permit to begin construction was received in January 2015.

15.5 TOPOGRAPHY AND DRAINAGE

The proposed site improvements are located within the existing mine plant site and offsite improvements include the rail loadout, soda ash stockpile and access haul road. In general, the site and surrounding area has rolling topography with bedrock exposed at or near the ground surface in upland and hill areas, and alluvial soils in lowlands and valleys.

Several prominent rock outcrops are visible along the ridge that borders the site to the northwest of the existing plant site. Surface soil conditions throughout the site consist primarily of sandy silt and silty sand alluvial material with fractions of gravel and cobbles. The soils are rated as well drained soils per the USDA Natural Resources Conservation Service Drainage Class Map. The geotechnical investigation has classified the area with Camel Conglomerate. Vegetation across the site comprises sagebrush and grass typical of the region. The geotechnical investigation did not encounter groundwater during the borehole drilling. The topography increases in elevation from the rail loadout to the crushed ore stockpile location and drainage flows are generally directed to the west by sheet flow drainage.

15.6 SURFACE WATER MANAGEMENT

The access road will maintain existing drainage patterns. An increase in surface flow is not anticipated since road surface treatments will be the same as existing conditions. Drainage swales along the access haul road will convey flows to maintain existing drainage patterns that flow to the west. The rail loadout will provide positive drainage away from the buildings and drainage will be controlled via surface drainage. Channels will be sized to handle the 50-year, 24-hour event.

15.7 FACILITIES LAYOUT

M3 designed the processing facilities to tie into the existing crushing facility, which includes the primary crusher, secondary crushers and tertiary crushers, and taking advantage of the grading developed previously.

Figure 15-1 shows the overall project area map showing the mine, the heap leach area, the rail unloading facilities, and other infrastructure for the project. Figure 15-2 is a more detailed view of the overall process area.

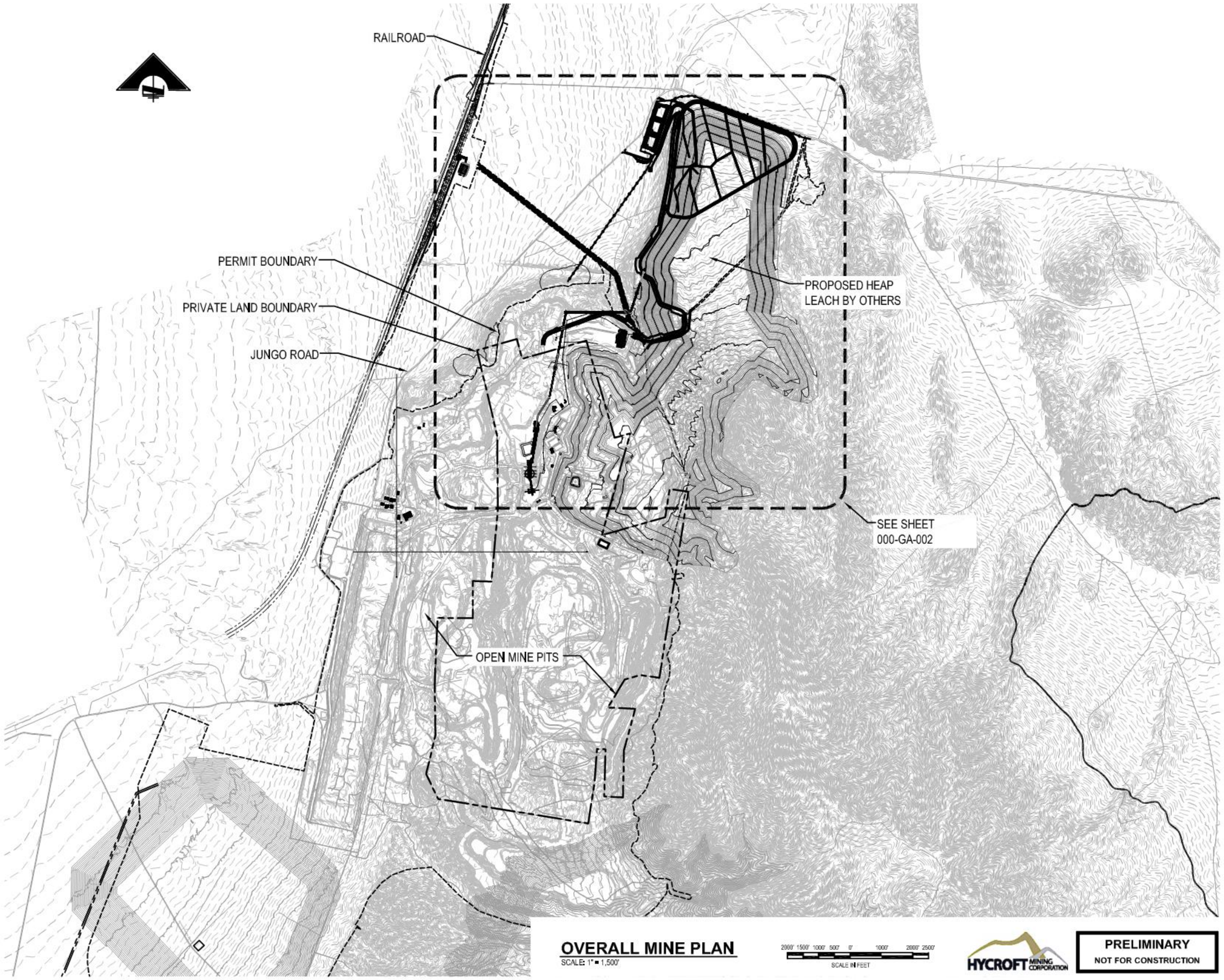


Figure 15-1: Overall Project Area Map

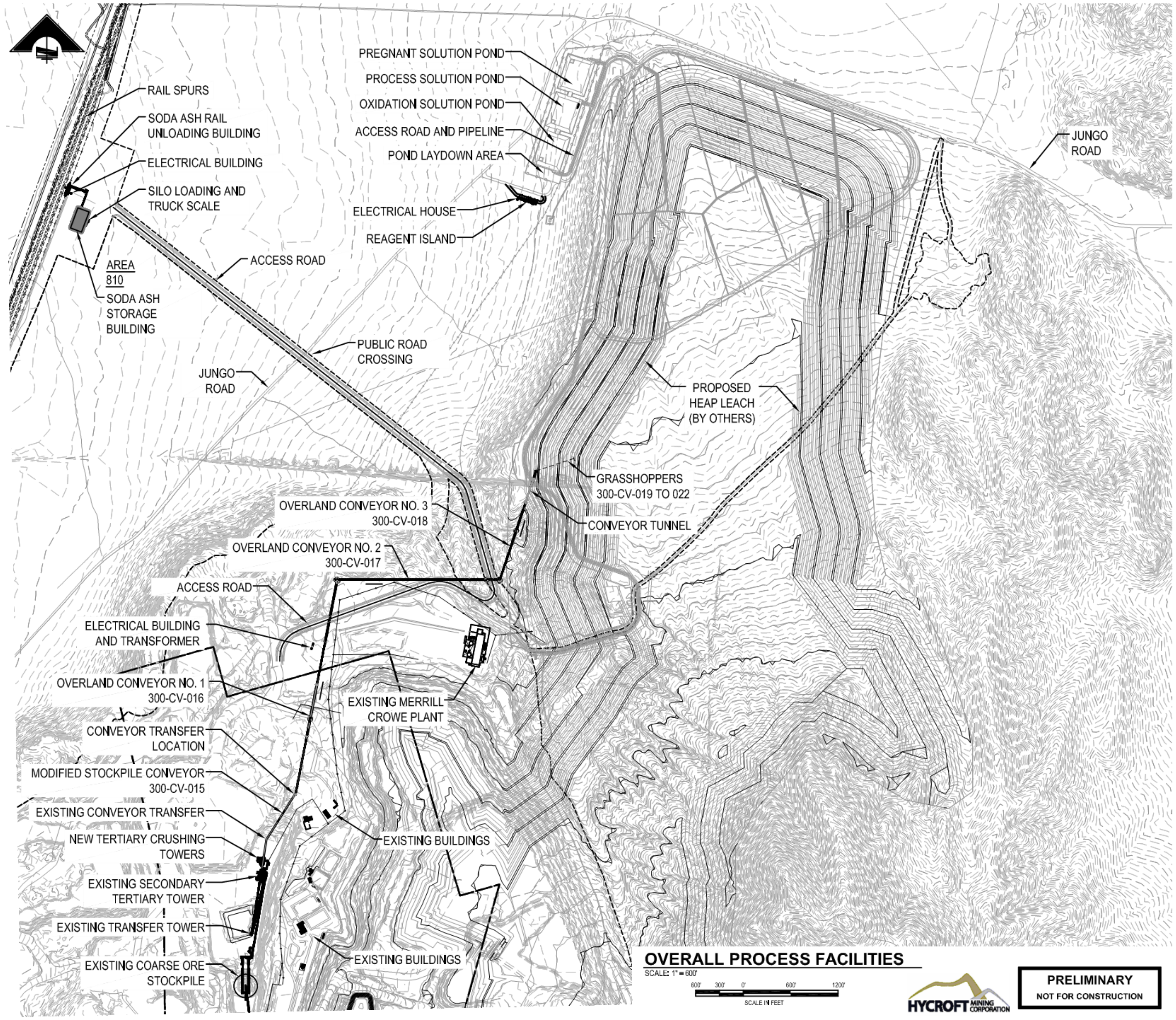


Figure 15-2: Overall Process Area Plan

16 MARKET STUDIES AND CONTRACTS

HMC has contracted M3 to complete the Feasibility Study for the heap leach expansion project.

Contracts for major consumables including fuel, lime, soda ash, cyanide and electricity are in place for the current operation. Transportation contracts are also in place for delivery of these consumable products. These contracts are renewed on an annual, biennial, triennial or quinquennial basis. The general terms and charges of these contracts are within industry standards.

Operating costs utilized in the feasibility study are based on recent costs or indications of long-term pricing from major suppliers during discussions with HMC and have been reviewed for consistency with past experience and industry pricing for these commodities, including any transportation or supply costs.

Gold and silver produced at Hycroft is expected to be sold as doré. Doré is shipped to refineries, refined and then sold at current spot prices.

Gold and silver markets are mature with reputable smelters and refiners located throughout the world. On June 30, 2019, the three-year trailing average gold and silver price per ounce was $1,273 and $16.53, respectively.

Gold is a principal metal traded at spot prices for immediate delivery and silver trading follows a pattern that is similar to that of gold. The market for gold trading typically spans 24 hours a day within multiple locations around the world (such as New York, London, Zurich, Sydney, Tokyo, Hong Kong, and Dubai). Daily prices are quoted on the London market and New York spot market and can be found on www.kitco.com.

16.1 DORÉ MARKETING

Gold and silver produced at Hycroft will be sold as doré. Doré is shipped to refineries, refined and then sold at current spot prices. Marketing of doré is expected to be arranged through continuing contractual relationships with major refineries for secure transportation of metal and refining. A contract with a refinery in Salt Lake City, Utah is in place through December 2020. The cost for shipping and refining doré included in the economic evaluation is in accordance with industry standards.

16.1.1 Doré Sales

Existing contracts formed the basis of refining and selling costs applied in the economic evaluation for gold and silver production sales. The principal commodities of gold and silver are freely traded at market prices, assuring the sale of any production.

16.1.2 Doré Shipping and Treatment Charges

Transportation and treatment contracts are currently in place and are reassessed on an annual or biennial basis. These arrangements are within industry standards and formed the basis of our economic evaluation.

17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

17.1 PERMITTING

The Environmental Impact Statement ("EIS") for the heap leach expansion that includes expanded heap leach, open pits, and waste rock facilities was completed in August 2012 with the BLM issuance of the Record of Decision authorizing the proposed action. State permits required for the operation of the heap leach expansion facilities were also received in 2012. The permits required to construct and operate the crushing system and to begin mill construction were received in 2012. The Plan of Operations for a rail spur, open pit expansion and processing complex, that includes a TMF and expanded Heap Leach Facility, was completed in December 2014, with the BLM issuance of the Record of Decision authorizing the proposed action received in January 2015. NV Energy submitted a Rights of Way application for the power line associated with the Hycroft Mill in March 2013. The BLM determined that action should be analyzed with the Hycroft Environmental Assessment ("EA"). Approval was received in December 2014.

Preliminary baseline work required for permitting of the long-term TMF and a deeper open pit, such as groundwater characterization, waste rock characterization, pit lake study and archaeological and biological surveys, began late in 2009. Field work has been completed for these programs. The study information was included in a Plan of Operations that was submitted to the BLM on April 30, 2014, requiring a supplemental EIS. Approvals are anticipated to be received in 2019.

A Plan of Operations amendment for construction of a new leach pad was submitted to, and approved by, the BLM in July 2019. A Water Pollution Control Permit modification was submitted to NDEP in March 2019 for the leach pad expansion and is under review. It is expected a decision for the water pollution control permit will be received by the end of 2019. Further expansion activities described in this Technical Report Summary, specifically the construction of additional heap leach pad space to accommodate the plans detailed in this report, will require multiple federal, state and local permits.

17.2 SOCIO-ECONOMIC IMPACTS

The existing Hycroft mine workforce lives mainly in Winnemucca (Humboldt County) and Lovelock (Pershing County); this will likely remain the same for the heap leach project detailed in this report.

Pershing and Humboldt counties are sparsely populated rural counties, with no large urban centers. Historically the development of the community in and around the City of Winnemucca has been based on the ranching, transportation, and mining industries. In particular, the last 30 years have seen a dramatic growth in population and the community as a result of precious metal exploration and development (Humboldt County, 2002). The populations in Humboldt County, Pershing County, and the State of Nevada all grew at relatively small and slow increments between 2000 and 2010.

An important part of the income of predominantly rural counties in Nevada is produced by sales tax and the net proceeds tax on mining activity within the county. Sales tax revenues are collected by the county in which delivery of the goods are taken. For the project, this would be Humboldt County.

The Hycroft personnel requirements will require close coordination with local government and businesses due to the needs of an increased regular workforce as well as the temporary staffing increases during construction periods. Dialogue with the local entities has revealed a positive response to the expansion of the mine due to current economic conditions as well as an overall acceptance of the mining industry. No agreements or negotiations are in effect between Hycroft Mining and any local interested party at this time.

Initial surveys indicate that the town of Winnemucca has the required infrastructure (shopping, emergency services, schools, etc.) to support the maximum workforce and dependents.

17.3 MINE CLOSURE & RECLAMATION

Mine closure and reclamation will be performed in accordance with BLM and State of Nevada regulations and guidelines. The mining activities occur near a National Conservation Area and associated pioneer trails. Particular attention will be paid to leaving a post-mining land configuration that minimizes visual impact. The Company has posted surety bonds partially backed by restricted cash balances to cover its closure obligations. Future increases in reclamation bonding will either be through surety bonds supported by restricted cash balances or by letters of credit issued by banks.

The facility expansions have been and will continue to be designed and constructed to meet or exceed state and federal design criteria. Waste rock facilities are evaluated for their potential to release pollutants and monitored routinely, and in accordance with an approved waste rock management plan.

After operations cease, solution in heap leach facilities will be allowed to drain until the rate of flow can be passively managed through evaporation or a combination of evaporation and infiltration. Current studies are gathering the additional hydrology and geochemistry data for use in the development of final closure plans that meet the regulatory standards.

All buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase. HMC has included reclamation and closure costs of $57.6 million in the economic evaluation.

18 CAPITAL AND OPERATING COSTS

18.1 MINE OPERATING AND MAINTENANCE COSTS

Operating costs, including capital development costs, were developed on a unit cost and quantity basis utilizing historic cost data, first principles, vendor/contractor quotations, and similar operation comparisons.

Since 2011, more than $400 million has already been spent in provision of an expansion plan. Mine and maintenance infrastructure is in place to support the required level of planned mining, including:

- Administration buildings (including mine operations and line-out offices);
- Truck Shops;
- Warehousing and laydown yards;
- Large Equipment Wash Facilities; and
- Fuel Islands.

18.1.1 Development Cost

Past mining has developed several pits down to sustainable ore and full ore deliveries are obtainable from initial start of mining. There are 4 million tons of blasted inventory within the initial pit phases. All pit access required for mining is in-place. There are also 9 million tons of stockpiles from previous mining that are considered in the mine planning and are important to early production. There are no additional mine development costs required to meet the mine plan.

18.1.2 Mining Cost

Contract Mining costs were developed using budgetary quotes received for contract mining, which considered, labor requirements, quantity of equipment being used by period and calculated fuel, explosive, lubricant, and maintenance costs.

Owner Mining costs were developed using first principle estimation, which considered, labor requirements, quantity of equipment being used by period and calculated fuel, explosive, lubricant, and maintenance costs. Historical mining costs at Hycroft as well as benchmarking with other similar sized operations was considered.

Mining costs include Operations Management, Technical Services, Pit Operations and Maintenance, Geotechnical, and Pit de-watering costs. Life-of-mine unit costs are shown in Table 18-1.

Table 18-1: Average Mining Costs for the Life of Mine[1]

Period	OPS MGMT ($/t mined)	DRILLING ($/t mined)	BLASTING ($/ton mined)	LOADING ($/t mined)	HAULING ($/t mined)	SUPPORT ($/t mined)	MAINT SHOP ($/t mined)	MINE TECH SVCS[2] ($/ton mined)	CONTRACT MINING ($/ton mined)	TOTAL ($/ton mined)
2019	$0.07	$0.00	$0.00	$0.26	$0.58	$1.06	$0.46	$0.14	$0.00	$2.57
2020	$0.02	$0.00	$0.00	$0.12	$0.22	$0.28	$0.11	$0.10	$1.77	$2.62
2021	$0.01	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.07	$2.07	$2.14
2022	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.05	$1.73	$1.78
2023	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.04	$1.50	$1.54
2024	$0.00	$0.01	$0.03	$0.01	$0.03	$0.00	$0.00	$0.04	$1.44	$1.57
2025	$0.02	$0.07	$0.12	$0.10	$0.16	$0.09	$0.05	$0.04	$0.84	$1.50
2026	$0.02	$0.11	$0.18	$0.18	$0.46	$0.17	$0.09	$0.04	$0.00	$1.25
2027	$0.02	$0.18	$0.26	$0.27	$0.40	$0.28	$0.10	$0.04	$0.00	$1.55
2028	$0.02	$0.16	$0.21	$0.22	$0.53	$0.18	$0.09	$0.04	$0.00	$1.44
2029	$0.02	$0.16	$0.26	$0.22	$0.59	$0.18	$0.09	$0.04	$0.00	$1.54

Period	OPS MGMT ($/t mined)	DRILLING ($/t mined)	BLASTING ($/ton mined)	LOADING ($/t mined)	HAULING ($/t mined)	SUPPORT ($/t mined)	MAINT SHOP ($/t mined)	MINE TECH SVCS[2] ($/ton mined)	CONTRACT MINING ($/ton mined)	TOTAL ($/ton mined)
2030	$0.02	$0.18	$0.26	$0.23	$0.58	$0.23	$0.09	$0.04	$0.00	$1.63
2031	$0.02	$0.17	$0.26	$0.23	$0.44	$0.19	$0.09	$0.04	$0.00	$1.43
2032	$0.02	$0.16	$0.25	$0.19	$0.43	$0.18	$0.09	$0.04	$0.00	$1.35
2033	$0.02	$0.15	$0.24	$0.21	$0.38	$0.20	$0.08	$0.03	$0.00	$1.32
2034	$0.02	$0.15	$0.25	$0.23	$0.59	$0.19	$0.09	$0.04	$0.00	$1.55
2035	$0.02	$0.15	$0.26	$0.22	$0.53	$0.23	$0.09	$0.04	$0.00	$1.53
2036	$0.02	$0.16	$0.26	$0.21	$0.57	$0.22	$0.09	$0.04	$0.00	$1.57
2037	$0.02	$0.19	$0.26	$0.19	$0.64	$0.21	$0.10	$0.04	$0.00	$1.67
2038	$0.02	$0.16	$0.22	$0.18	$0.77	$0.19	$0.08	$0.04	$0.00	$1.66
2039	$0.02	$0.15	$0.23	$0.18	$0.56	$0.23	$0.09	$0.04	$0.00	$1.49
2040	$0.02	$0.17	$0.24	$0.22	$0.50	$0.19	$0.09	$0.04	$0.00	$1.46
2041	$0.02	$0.17	$0.27	$0.17	$0.59	$0.22	$0.10	$0.04	$0.00	$1.59
2042	$0.03	$0.18	$0.26	$0.24	$0.85	$0.27	$0.12	$0.05	$0.00	$1.99
2043	$0.03	$0.16	$0.25	$0.27	$0.64	$0.32	$0.13	$0.05	$0.00	$1.85
2044	$0.03	$0.14	$0.25	$0.23	$0.57	$0.32	$0.13	$0.05	$0.00	$1.72
2045	$0.03	$0.14	$0.25	$0.24	$0.58	$0.26	$0.13	$0.05	$0.00	$1.68
2046	$0.03	$0.16	$0.26	$0.30	$0.76	$0.32	$0.14	$0.06	$0.00	$2.01
2047	$0.03	$0.16	$0.23	$0.25	$0.68	$0.34	$0.15	$0.06	$0.00	$1.90
2048	$0.03	$0.18	$0.27	$0.23	$0.69	$0.29	$0.14	$0.06	$0.00	$1.88
2049	$0.03	$0.15	$0.26	$0.23	$0.75	$0.31	$0.14	$0.06	$0.00	$1.94
2050	$0.02	$0.14	$0.27	$0.19	$0.85	$0.19	$0.07	$0.03	$0.00	$1.75
2051	$0.02	$0.13	$0.21	$0.17	$0.91	$0.16	$0.06	$0.01	$0.00	$1.68
2052	$0.02	$0.00	$0.00	$0.11	$0.82	$0.14	$0.06	$0.01	$0.00	$1.16
LOM Avg	**$0.02**	**$0.14**	**$0.22**	**$0.19**	**$0.52**	**$0.20**	**$0.09**	**$0.04**	**$0.20**	**$1.61**

1. Zeroes do not indicate that there is no cost, rounding may be a factor.
2. Pit dewatering and geotechnical costs are included in Technical Services.

Mine operations manpower is listed below in Table 18-2.

Table 18-2: Mine Operations Manpower

Mine Operations Staffing	2019	2020	2021	2022	2023	2024	2025 Onward (Avg)
Open Pit Management	6	6	1	1	1	1	19
Technical Services	6	11	11	14	14	14	16
Mine Operations	42	42	-	-	-	-	168
Mine Maintenance	34	34	-	-	-	-	74
Contract Miner[1]	-	76	119	181	273	273	-
Total	**88**	**169**	**131**	**196**	**288**	**288**	**277**

1. Years 2020 and 2025 each include half a year of contract mining.

18.2 PROCESS PLANT OPERATING & MAINTENANCE COSTS

The process plant operating costs are summarized by areas of the plant. Table 18-3 shows the life-of-mine cost for the operation.

Table 18-3: Process Operating Cost Summary

Process Area	LOM (000s)	$/ore ton
Crushing	$ 546,774	$ 0.48
Rehandle	$ 36,553	$ 0.03
Heap Leach	$ 3,324,021	$ 2.93
Merrill-Crowe/Refinery	$ 558,552	$ 0.49
Lab & Met Services	$ 47,980	$ 0.04
Ancillaries	$ 29,477	$ 0.03
Total Process	**$ 4,566,624**	**$ 4.00**
Heap Leach Ore Tons	1,133,060	

18.2.1 Crushed Ore Re-handle

Costs to re-handle crushed ore from the crushed ore stockpile to the leach pads have been estimated at $0.45/t, which includes all loading, hauling, and support equipment costs to move crushed ore from the stockpile and stack on the leach pad. This cost is only incurred until the fine ore convey & stack system is constructed and operational at the beginning of Year 6.

18.2.2 Process Labor & Fringes

Process labor costs were derived from a staffing plan and applying prevailing daily or annual labor rates in the area. Labor rates and fringe benefits for employees include retirement plans, insurance and all applicable social security benefits as well as all applicable payroll taxes. The staffing plan summary and gross annual labor costs are shown in Table 18-4 below.

Table 18-4: Average Annual Process Plant Labor (Typical Year – Year 7)

Staff		Annual Cost
Process Administration	16	$1,989,105
Process Operations	64	$5,572,644
Process Maintenance	33	$3,306,846
Total Process Operations	**113**	**$10,868,595**

18.2.3 Electrical Power

Power consumption was based on the equipment list connected kW, adjusted for operating time per day and anticipated operating load level. The overall power rate as provided by HMC is estimated at $0.056 per kWh. A summary of the annual power consumption and cost are shown in Table 18-5 below.

Table 18-5: Average Annual Power Cost Summary* (Typical Year – Year 7)

Area No	Area	Annual kWh	Annual Cost
100	Primary Crusher	6,143,377	$344,029
150	Coarse Ore Stacking & Reclaim	45,836,031	$2,566,818
300	Heap Leach Ore Stacking	37,537,420	$2,102,096
350	Heap Leach	4,250,568	$238,032
550	Merrill-Crowe	16,036,942	$898,069
650	Water Systems	4,666,283	$261,312
800	Reagents	3,272,965	$183,286
810	Soda Ash Rail Unloading & Storage	986,396	$55,238
	Total	**118,729,982**	**$ 6,648,879**

18.2.4 Reagents

Consumption rates were determined from the metallurgical test data. Budgetary quotations for reagents were received from local sources where available with an allowance for freight to site, or based on actual purchases.

A summary of process reagent consumption and costs are included in Table 18-6.

Table 18-6: Reagents Consumption Summary

Process Area	Consumption Rates		Unit Price, $/lb
Pre-Oxidation			
Soda Ash	lb/ore ton	14.5	$ 0.10 (rail) / $0.115 (truck)
Heap Leach			
Sodium Cyanide	lb/ton processed	1.0	$ 1.20
Lime	lb/ ton processed	1.4	$ 0.11
Antiscalant	lb/ton processed	0.009	$ 1.70
Merrill-Crowe/Refinery			
Zinc Dust	lb/koz Au / Ag	34.15 / 62.35	$ 1.80
Diatomaceous earth	lb/ton processed	0.03	$ 0.25

18.2.5 Maintenance Wear Parts and Consumables

Wear items (liners and conveyor belts) were based on industry practice for the crusher and grinding operations. These consumptions rates and unit prices are shown in Table 18-7 below.

Table 18-7: Wear Items

Area	Consumption Rates		Unit Price, $/ton crushed
Primary Crusher – Liners	lb/ton crushed	1.0	$ 0.03
Secondary Crusher – Liners	lb/ton crushed	1.0	$ 0.06
Tertiary Crusher - Liners	lb/ton crushed	1.0	$ 0.06

An allowance was made to cover the cost of maintenance of all items not specifically identified and the cost of maintenance of the facilities. The allowance was calculated for each project area as a percentage of the tangible equipment cost.

18.2.6 Process Supplies & Services

Allowances were provided in the process for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using M3's information from other operations and projects.

18.3 CAPITAL COST ESTIMATE

18.3.1 Mine Initial Capital Cost

All mining related infrastructure is already in place. All mine development for access and waste stripping was done as part of the historic mining completed prior to 2015. Initial mining through Year 5 is completed using either existing mine fleets at site, or contract mining, and minimal initial capital is required for mine equipment. Other mine related capital costs are related to de-watering infrastructure. These costs are shown in Table 21-8.

Table 18-8: Dewatering Capital

De-watering	2019	2020	2021	2022	2023
Pilot Hole / Hydraulic Testing	-	$827,781	-	$560,188	$1,095,375
Dewatering Well Construction	-	$1,149,045	-	$781,030	$1,517,060
Dewatering Well Pumping Equip./Installation	-	$390,603	-	$260,402	$520,804
Pumping Equipment Spares	-	$130,201	-	-	-
Dewatering Well Replacement	-	-	-	-	-
Pipeline & Booster Pump Construction	-	$710,493	-	$72,114	$25,631
Lateral Depressurization Drainholes	-	-	$313,483	$313,483	-
Contingency	-	$641,625	$62,697	$397,443	$631,774
Total	**$0**	**$3,849,748**	**$376,180**	**$2,384,660**	**$3,790,644**

SRK completed the capital cost estimate for mine dewatering. The costs presented are based on previous experience in the construction and operation of the prototype dewatering well, as well as knowledge and experience with similar facilities and work in similar locations. Dewatering capital costs are subject to a contingency to address uncertainties and risk as described in Section 13.6. However, should significant changes in the conceptual hydrogeology of the local groundwater system be identified, active dewatering requirements and/or residual passive inflow may be underestimated, along with the associated capital costs.

18.3.2 Initial Capital Costs

Table 18-9 shows a summary of the estimated initial capital expressed in US dollars.

Table 18-9: Estimated Initial Capital Cost (Year 1-5; 000s)

Description	Estimated by	2019	2020	2021	2022	2023
Direct & Indirect Costs						
Site General	M3	-	-	$4,396.9	-	-
Crushing & Conveying	M3	-	-	$23,285.2	$22,450.1	$60,991.5
Leach Pad & Pond Construction	Newfields	$31,950.9	$18,425.1	-	$455.0	-
Reagents	M3	-	-	-	-	$18,145.6
Rail Unloading & Storage	M3	-	-	-	$10,792.2	$21,548.2
Dewatering	SRK	-	$3,849.7	$376.2	$2,384.7	$3,790.6
Mobile Equipment	Hycroft	-	$312.0	-	-	-
Subtotal Direct & Indirect Costs		**$31,950.9**	**$22,586.8**	**$28,058.3**	**$36,082.0**	**$104,475.9**
Owner's Cost						
Total Owner's Cost[1]	Hycroft	**$5,048.3**	**$1,100.0**	**$500.0**	**$500.0**	**$500.0**
TOTAL		**$36,999.2**	**$23,686.8**	**$28,558.3**	**$35,582.0**	**$104,975.9**

1. Includes $3.1 million for final crusher payments to vendor to restart the system during the first half of 2019.

Newfields completed the Phase 1 leach pad, pond, and related solution management infrastructure capital estimate. Phase 1 pad expansion includes 8.9 million square feet of pad plus process ponds and solution piping. The Phase 1 leach pad design and associated cost estimate was at Issue-for-Construction (IFC) level to support permitting activities.

The capital cost estimates prepared by M3 are based on cost quotations on major equipment, including secondary and tertiary crushers, grasshopper conveyors and stacker. Material takeoffs were performed for concrete, steel, electrical, piping and instrumentation, for all installations designed by M3. Unit rates for labor, equipment rental, and materials of construction were developed using recent construction projects managed by M3, as well as on published industry rates.

Owner's costs were provided by Hycroft Mining.

18.3.3 Assumptions

The capital projects are assumed to be constructed in a conventional EPCM format. HMC will retain a qualified contractor to design projects and act as its agent to bid and procure materials and equipment, bid and award construction contracts, and manage the construction of the facilities.

HMC, through its EPCM agent, will order major material supplies (e.g., structural and mechanical steelwork) as well as bulk orders (e.g., piping and electrical). These will be issued to construction contractors on site using strict inventory control.

"Initial Capital" is defined as all capital required to acquire and put into operation all facilities and equipment to achieve a steady-state mining and processing rate as detailed in this feasibility study. The initial capital costs include owner's costs, new heap leach pad construction, crushing and conveying system improvements, leach pad conveying and stacking system, new reagent island, mobile equipment, rail spur unloading and storage facilities and mine dewatering costs through to the end of Year 5. Capital costs predicted for later years are carried as sustaining capital in the financial model.

18.3.4 Sunk Costs

Hycroft Mining has been increasing their mining and processing capacity since 2010. A new 21,500 gpm Merrill-Crowe plant, three-stage crushing plant, additional carbon columns, pumping and piping infrastructure and increased power capacity have been implemented. Due to the ongoing symbiotic nature of the operation, the feasibility study considers all of those costs spent before June 30, 2019 on the items noted above to be sunk costs. Therefore, the capital cost and economic returns presented in the financial analysis is on a go-forward basis only. Sunk costs are not included in the economic return of the project presented herein.

18.3.5 Estimate Accuracy

The accuracy of this estimate for those items identified in the scope-of work is estimated to be within the range of plus 15% to minus 15%; i.e., the cost could be 15% higher than the estimate or it could be 15% lower. Accuracy is an issue separate from contingency, the latter accounts for undeveloped scope and insufficient data (e.g., geotechnical data).

18.3.6 Contingency

Contingency for this estimate is 10% and is intended to cover unallocated costs from lack of detailing in scope items. It is a compilation of aggregate risk from estimated cost areas. Contingency is not simply a "buffer" to cover estimate inaccuracy. Properly calculated contingency will be spent.

18.3.7 Documents

Documents available to the estimators include the following:

Table 18-10: Documents Available to Estimators

Document	Yes/No
Design Criteria	No
Equipment List	Yes
Equipment Specifications	No
Construction Specifications	No
Flowsheets	Yes
P&IDs	Yes
General Arrangements	Yes
Architectural Drawings	No
Civil Drawings	No
Concrete Drawings	No
Structural Steel Drawings	No
Mechanical Drawings	No
Electrical One-Lines	No
Instrumentation Schematics	No
Instrument Log	No
Pipeline List	Yes
Valve List	Yes
Cable and Conduit Schedule	No

19 ECONOMIC ANALYSIS

19.1 INTRODUCTION

The financial evaluation of the project comprises the determination of the net present value (NPV) at a discount rate of 5%, the internal rate of return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of gold and silver bullion, using the recently executed refinery contract. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

19.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore from the mining operation. The production figures were obtained from the mine plan as presented earlier in this report.

The life of mine ore and waste quantities and ore grade are summarized in Table 19-1 below.

Table 19-1: Life of Mine Ore, Waste Quantities, and Ore Grade

	Tons (000')	Gold opt	Silver opt
Sulfide Heap Leach Ore	1,065,729	0.011	0.429
ROM Heap Leach Ore	43,375	0.008	0.219
Crushed Oxide Heap Leach Ore	23,956	0.010	0.626
Total Heap Leach Ore	**1,133,060**	**0.011**	**0.425**
Waste	1,228,686		
Stockpile material	196,155		
Total	**2,557,900**		

19.3 PLANT PRODUCTION STATISTICS

The major components of the process facility are crushing, heap leach and Merrill-Crowe. The product will be gold and silver doré.

The estimated metal recoveries are presented in Table 19-2.

Table 19-2: Average Life of Mine Metal Recovery Factors

	Gold %	Silver %
Heap Leach Ore	65	71

Estimated life of mine gold and silver production is presented in the table below.

Table 19-3: Life of Mine Metal Production

	Gold, kOz	Silver, kOz
Heap Leach Ore	7,868	346,519

19.3.1 Refinery Factor

The refining and shipping charges calculated in the financial evaluation are presented in the table below.

Table 19-4: Refining Factors

Gold and Silver Bullion	
Payable Gold	99.9 %
Payable Silver	99.5 %
Refining Charge ($/oz.)	$0.50
Deleterious Elements Charge ($/oz.)	$0.25
Melt Loss %	0.2%
Transportation Charges ($/shipment)	$1,800

19.4 CAPITAL EXPENDITURE

19.4.1 Initial Capital

The financial indicators have been determined with 100% equity financing of the initial capital, that is, no borrowing costs have been included in the analysis. Any acquisition cost or expenditures prior to 2019 have been treated as sunk cost and have not been included in the analysis.

The total initial capital in the financial model is $230.8 million.

19.4.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis. The total life of mine sustaining capital is estimated to be $537.6 million. This capital will be expended during a 26-year period at an average rate of $17.9 million per year.

19.4.3 Working Capital

A working capital allowance of $2.2 million for warehouse inventories is assumed in the first five years.

19.4.4 Salvage Value

No salvage value allowance has been considered in the cash flow analysis.

19.5 REVENUE

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals before refining charges and transportation charges. Metal sales prices used in the evaluation are $1,300.00/ounce for gold and $17.33/ounce for silver.

19.6 OPERATING COST

The average cash operating cost over the life of the mine is estimated to be $8.49 per ton of heap leach ore processed. Cash operating cost includes mine operations, process plant operations, general administrative cost, refining charges, shipping charges, royalties and net proceeds tax.

Table 19-5: Operating Cost

Operating Cost	Cost
Mining /ton mined	$1.61
Mining /ton processed	$3.64
Heap Leach Process /ton processed	$4.01
General Administration /ton processed	$0.45
Treatment Charges/ ton processed	$0.23
Net Proceeds Tax/ton processed	$0.17
Transportation/ton processed	$0.01
Crofoot Royalty	$0.00
Total Operating Cost /ton processed	$8.52

1. Zeroes do not indicate that no cost exists, however rounding may be a factor. Similarly, the total may not foot due to rounding.

19.7 RECLAMATION & CLOSURE

An allowance for the cost of reclamation and closure of the property has been included in the cash flow projection of $57.6 million to be expended at the end of the mine life over a two-year period.

19.8 TAXATION (FEDERAL INCOME TAX)

The cash income tax projections have been provided by HMC at a 21% statutory rate and are estimated to be $615.2 million for the life of the mine, based on depreciation of assets, recovery factors, metal prices indicated in this report and existing tax account balances of the Company.

On December 22, 2017, the Tax Cuts and Jobs Act of 2017 (the "Act") was enacted. One component to the Act was to amend how a net operating loss ("NOL") may be utilized. NOL's generated prior to January 1, 2018 may be carried back 2 years, carried forward 20 years and utilized to offset up to 100% of pretax income. Because these NOL's can be used to offset 100% of pretax income we have labeled them as "Super NOL" in the model. The Act limited the amount of pretax income that NOL's generated in periods beginning after December 31, 2017 may be used to offset. NOL's generated after December 31, 2017 may only offset 80% of pretax income and can no longer be carried back, but they can be carried forward indefinitely. NOL's generated after December 31, 2017 are labeled as "New NOL" in the model. Based on the amendments in the Act to NOL's, the model uses all Super NOL's first and then uses the New NOL's to offset up to 80% of pretax income.

19.9 PROJECT FINANCING

HMC has engaged an investment banker to advise and execute on financing and/or investment options for the project capital requirements.

19.10 NET CASH FLOW AFTER TAX

Net Cash Flow after Tax amounts to $5.1 billion for the life of the mine.

19.11 NPV AND IRR

The economic analysis indicates that the project has an internal rate of return (IRR) of 148.6%, a payback period of 2.5 years, and an NPV of $2.1 billion at a 5% discount rate.

19.12 FINANCIAL MODEL

The detailed financial model is shown in **Error! Not a valid bookmark self-reference.**.

Table 19-6: Financial Model

Hycroft Mining
Hycroft Sulfide Heap Leach Oxidation

		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
	LOM	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Mine Plan																	
Sulfide Heap Leach Ore (ktons)	**970,741**	4,488	10,385	16,132	23,332	24,000	20,088	33,546	28,572	34,426	35,960	36,000	35,106	35,198	35,739	27,552	35,975
ROM Heap Leach Ore (ktons)	**35,375**	-	1,822	5,278	3,607	4,689	-	4,915	132	2,025	258	311	4,578	827	1,376	90	753
Crushed Oxide Heap Leach Ore (ktons)	**23,956**	-	355	1,868	668	-	14,544	1,272	104	1,574	40	-	894	802	261	60	25
Stockpile Rehandle (ktons)	**102,988**	-	-	-	-	-	1,367	1,182	7,325	-	-	-	-	-	-	8,388	-
Total Ore tons mined (ktons)	**1,133,060**	4,488	12,562	23,278	27,607	28,689	36,000	40,915	36,132	38,025	36,258	36,311	40,578	36,827	37,376	36,090	36,753
Waste (ktons)	**1,228,686**	-	4,142	5,934	25,283	31,534	40,324	36,959	54,597	37,564	50,316	47,671	50,600	55,646	47,993	65,322	48,545
Stockpile (ktons)	**196,155**	-	2,539	6,334	7,110	14,379	10,043	8,308	1,596	9,411	13,426	16,018	8,822	7,527	14,631	6,976	14,703
Total tons mined (ktons)	**2,557,900**	4,488	19,244	35,546	60,000	74,602	86,367	86,182	92,325	85,000	100,000	100,000	100,000	100,000	100,000	108,388	100,000
Strip Ratio	**1.17**	-	0.5	0.5	1.2	1.6	1.4	1.1	1.4	1.2	1.8	1.8	1.5	1.7	1.7	1.8	1.7
Process Plan																	
Heap Leach Ore ROM (ktons)	**43,375**	-	1,822	5,278	3,607	4,689	-	4,915	132	2,025	258	311	4,578	827	1,376	90	753
Gold Grade (oz/t)	**0.008**	-	0.017	0.010	0.008	0.009	-	0.007	0.011	0.006	0.006	0.006	0.008	0.007	0.007	0.005	0.008
Silver Grade (oz/t)	**0.219**	-	0.210	0.241	0.232	0.278	-	0.230	0.164	0.163	0.406	0.116	0.282	0.272	0.216	0.214	0.130
Contained Gold (kozs)	**327**	-	32	50	29	41	-	32	2	13	2	2	37	5	9	0	6
Contained Silver (kozs)	**9,480**	-	383	1,271	837	1,302	-	1,130	22	329	105	36	1,290	224	297	19	98
Crushed Oxide Ore (ktons)	**23,956**	-	355	1,868	668	-	14,544	1,272	104	1,574	40	-	894	802	261	60	25
Gold Grade (oz/t)	**0.010**	-	0.031	0.012	0.013	-	0.008	0.015	0.014	0.012	0.007	-	0.020	0.010	0.007	0.003	0.008
Silver Grade (oz/t)	**0.626**	-	0.606	0.660	0.828	-	0.506	0.961	0.686	0.683	2.364	-	1.210	1.001	1.544	2.017	1.149
Contained Gold (kozs)	**243**	-	11	23	9	-	119	19	1	18	0	-	18	8	2	0	0
Contained Silver (kozs)	**15,000**	-	215	1,233	553	-	7,366	1,223	71	1,075	95	-	1,082	803	403	121	29
Sulfide Heap Leach Ore (ktons)	**1,065,729**	4,488	10,385	16,132	23,332	24,000	21,456	34,728	35,896	34,426	35,960	36,000	35,106	35,198	35,739	35,940	35,975
Gold Grade (oz/t)	**0.011**	0.018	0.014	0.011	0.010	0.013	0.013	0.010	0.011	0.009	0.013	0.011	0.011	0.009	0.012	0.009	0.009
Silver Grade (oz/t)	**0.429**	0.113	0.297	0.398	0.446	0.352	0.489	0.497	0.206	0.588	0.600	0.668	0.446	0.605	0.284	0.322	0.327
Contained Gold (kozs)	**11,427**	80	145	182	243	312	279	355	412	327	480	396	369	331	435	312	323
Contained Silver (kozs)	**456,920**	506	3,086	6,415	10,417	8,456	10,482	17,275	7,393	20,257	21,591	24,059	15,642	21,284	10,151	11,566	11,774
Total Heap Leach Ore (ktons)	**1,133,060**	4,488	12,562	23,278	27,607	28,689	36,000	40,915	36,132	38,025	36,258	36,311	40,578	36,827	37,376	36,090	36,753
Gold Grade (oz/t)	**0.011**	0.018	0.015	0.011	0.010	0.012	0.011	0.010	0.011	0.009	0.013	0.011	0.010	0.009	0.012	0.009	0.009
Silver Grade (oz/t)	**0.425**	0.113	0.293	0.383	0.428	0.340	0.496	0.480	0.207	0.570	0.601	0.664	0.444	0.606	0.290	0.324	0.324
Contained Gold (kozs)	**11,997**	80	187	255	280	353	398	407	415	357	482	398	424	344	446	313	329
Contained Silver (kozs)	**481,400**	506	3,685	8,918	11,807	9,759	17,849	19,628	7,486	21,661	21,791	24,095	18,014	22,311	10,851	11,707	11,901
Gold Recovery %	**65%**	**55%**	**37%**	**64%**	**68%**	**55%**	**60%**	**49%**	**49%**	**101%**	**48%**	**67%**	**63%**	**66%**	**70%**	**85%**	**59%**
Silver Recovery %	**71%**	**78%**	**50%**	**29%**	**66%**	**59%**	**46%**	**42%**	**123%**	**55%**	**59%**	**58%**	**83%**	**45%**	**148%**	**71%**	**64%**
Recovered Gold (kozs) - From Leach Model	**7,868**	44	70	164	190	196	238	202	203	361	231	266	269	227	313	267	196
Recovered Silver (kozs) - From Leach Model	**346,519**	399	1,858	2,641	7,855	5,795	8,326	8,375	9,242	12,071	12,844	14,040	15,070	10,195	16,207	8,356	7,709
Payable Metal (kozs)																	
Gold	**7,845**	44	70	163	189	196	238	201	202	359	230	265	268	227	312	267	195
Silver	**344,097**	396	1,845	2,623	7,800	5,754	8,268	8,316	9,178	11,986	12,754	13,942	14,965	10,124	16,094	8,298	7,655
Metal Prices																	
Gold ($/oz)	**$1,300.00**	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00
Silver ($/oz)	**$17.33**	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33
Cash Inflows ($000)																	
Gold	**$10,198,352**	$57,347	$91,056	$212,052	$246,224	$254,257	$309,107	$261,440	$262,644	$467,317	$299,521	$344,553	$348,811	$294,699	$405,065	$346,623	$253,405
Silver	**$5,963,198**	$6,859	$31,968	$45,448	$135,170	$99,720	$143,284	$144,117	$159,050	$207,723	$221,023	$241,615	$259,345	$175,451	$278,906	$143,798	$132,669
Total Cash Inflows	**$16,161,550**	$64,206	$123,024	$257,500	$381,394	$353,976	$452,390	$405,557	$421,694	$675,041	$520,544	$586,168	$608,156	$470,150	$683,971	$490,421	$386,074

	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35
Mine Plan																			
Sulfide Heap Leach Ore (ktons)	35,980	35,990	35,926	18,358	24,278	26,403	32,695	35,969	30,673	30,946	31,022	32,607	26,226	36,000	36,000	35,965	29,207	-	-
ROM Heap Leach Ore (ktons)	750	1,936	127	80	1,090	628	28	54	9	9	-	-	-	-	-	-	-	-	-
Crushed Oxide Heap Leach Ore (ktons)	20	10	74	120	370	357	-	31	343	127	-	-	-	-	-	35	-	-	-
Stockpile Rehandle (ktons)	-	-	-	17,522	11,353	9,240	3,305	-	4,984	4,926	4,978	3,393	9,774	-	-	-	6,793	8,457	-
Total Ore tons mined (ktons)	36,750	37,936	36,127	36,080	37,090	36,628	36,028	36,054	36,009	36,009	36,000	36,000	36,000	36,000	36,000	36,000	36,000	8,457	-
Waste (ktons)	47,122	46,245	44,811	63,225	56,554	55,644	50,963	37,677	30,872	31,181	29,811	27,388	23,758	26,324	19,088	15,117	20,475	-	-
Stockpile (ktons)	16,128	15,819	4,062	3,216	2,709	1,968	1,314	1,193	468	158	228	5	16	1,492	5,076	181	300	-	-
Total tons mined (ktons)	100,000	100,000	85,000	102,522	96,353	94,240	88,305	74,925	67,350	67,348	66,039	63,393	59,774	63,816	60,164	51,297	56,775	8,457	-
Strip Ratio	1.7	1.6	1.4	1.4	1.3	1.3	1.4	1.1	0.7	0.7	0.7	0.7	0.4	0.8	0.7	0.4	0.4	-	-
Process Plan																			
Heap Leach Ore ROM (ktons)	750	1,936	127	80	1,090	628	28	54	9	9	-	-	-	-	-	-	-	8,000	-
Gold Grade (oz/t)	0.007	0.007	0.004	0.005	0.006	0.005	0.007	0.009	0.007	0.006	-	-	-	-	-	-	-	0.005	-
Silver Grade (oz/t)	0.288	0.202	0.413	0.307	0.189	0.223	0.080	0.069	0.068	0.081	-	-	-	-	-	-	-	0.137	-
Contained Gold (kozs)	6	13	1	0	6	3	0	0	0	0	-	-	-	-	-	-	-	37	-
Contained Silver (kozs)	216	391	53	25	206	140	2	4	1	1	-	-	-	-	-	-	-	1,099	-
Crushed Oxide Ore (ktons)	20	10	74	120	370	357	-	31	343	127	-	-	-	-	-	35	-	-	-
Gold Grade (oz/t)	0.015	0.022	0.005	0.007	0.009	0.009	-	0.016	0.013	0.010	-	-	-	-	-	0.012	-	-	-
Silver Grade (oz/t)	1.507	2.050	1.213	0.962	0.649	0.480	-	0.080	0.084	0.090	-	-	-	-	-	0.598	-	-	-
Contained Gold (kozs)	0	0	0	1	3	3	-	1	4	1	-	-	-	-	-	0	-	-	-
Contained Silver (kozs)	30	21	90	115	240	171	-	3	29	12	-	-	-	-	-	21	-	-	-
Sulfide Heap Leach Ore (ktons)	35,980	35,990	35,926	35,880	35,630	35,643	36,000	35,969	35,657	35,873	36,000	36,000	36,000	36,000	36,000	35,965	36,000	457	-
Gold Grade (oz/t)	0.009	0.012	0.014	0.009	0.009	0.009	0.013	0.011	0.011	0.010	0.010	0.011	0.010	0.011	0.011	0.011	0.009	0.005	-
Silver Grade (oz/t)	0.330	0.415	0.649	0.640	0.358	0.334	0.541	0.361	0.254	0.293	0.290	0.204	0.180	0.273	0.429	0.770	0.818	0.123	-
Contained Gold (kozs)	336	439	506	311	309	311	455	382	375	373	378	407	352	388	394	400	328	2	-
Contained Silver (kozs)	11,890	14,951	23,334	22,953	12,742	11,919	19,487	12,981	9,065	10,503	10,425	7,326	6,488	9,837	15,439	27,706	29,462	56	-
Total Heap Leach Ore (ktons)	36,750	37,936	36,127	36,080	37,090	36,628	36,028	36,054	36,009	36,009	36,000	36,000	36,000	36,000	36,000	36,000	36,000	8,457	-
Gold Grade (oz/t)	0.009	0.012	0.014	0.009	0.009	0.009	0.013	0.011	0.011	0.010	0.010	0.011	0.010	0.011	0.011	0.011	0.009	0.005	-
Silver Grade (oz/t)	0.330	0.405	0.650	0.640	0.356	0.334	0.541	0.360	0.253	0.292	0.290	0.204	0.180	0.273	0.429	0.770	0.818	0.137	-
Contained Gold (kozs)	342	453	507	312	319	317	455	383	379	374	378	407	352	388	394	401	328	39	-
Contained Silver (kozs)	12,136	15,363	23,476	23,093	13,188	12,230	19,489	12,987	9,094	10,515	10,425	7,326	6,488	9,837	15,439	27,727	29,462	1,155	-
Gold Recovery %	**60%**	**48%**	**51%**	**96%**	**67%**	**103%**	**46%**	**65%**	**60%**	**60%**	**58%**	**55%**	**108%**	**56%**	**68%**	**67%**	**65%**	**511%**	**0%**
Silver Recovery %	**66%**	**51%**	**41%**	**64%**	**122%**	**111%**	**46%**	**84%**	**85%**	**57%**	**64%**	**91%**	**117%**	**51%**	**56%**	**46%**	**121%**	**1517%**	**0%**
Recovered Gold (kozs) - From Leach Model	206	219	259	300	214	328	209	249	229	225	219	225	382	216	269	268	214	200	-
Recovered Silver (kozs) - From Leach Model	8,010	7,926	9,740	14,891	16,188	13,650	8,968	11,026	7,812	6,075	6,669	6,703	7,661	5,042	8,671	12,904	35,947	17,653	-
Payable Metal (kozs)																			
Gold	206	218	258	299	213	327	209	248	229	224	218	225	381	215	268	267	213	200	-
Silver	7,954	7,871	9,672	14,787	16,075	13,555	8,905	10,949	7,757	6,033	6,622	6,656	7,607	5,007	8,611	12,814	35,696	17,529	-
Metal Prices																			
Gold ($/oz)	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00	$1,300.00
Silver ($/oz)	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33	$17.33
Cash Inflows ($000)																			
Gold	$267,506	$283,863	$335,574	$389,145	$276,811	$424,573	$271,398	$322,850	$297,287	$291,578	$283,548	$292,077	$495,331	$279,919	$348,791	$347,435	$276,767	$259,778	$0
Silver	$137,845	$136,399	$167,618	$256,263	$278,573	$234,906	$154,327	$189,742	$134,429	$104,552	$114,760	$115,349	$131,830	$86,771	$149,224	$222,071	$618,607	$303,785	$0
Total Cash Inflows	$405,351	$420,262	$503,192	$645,408	$555,384	$659,479	$425,725	$512,592	$431,717	$396,131	$398,308	$407,426	$627,161	$366,690	$498,015	$569,506	$895,374	$563,562	$0

Hycroft Mining
Hycroft Sulfide Heap Leach Oxidation

	LOM	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Cash Outflows ($000)													
Operating Cost													
Mining	**$4,128,218**	$11,556	$50,371	$76,080	$106,625	$114,835	$135,339	$129,328	$115,798	$132,008	$143,939	$154,328	$163,268
Heap Leach	**$3,324,021**	$9,967	$32,949	$52,949	$59,089	$85,968	$92,984	$113,920	$110,105	$116,546	$116,451	$109,639	$98,585
Crushing Costs	**$546,774**	$3,726	$9,610	$11,292	$12,712	$12,712	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992
Ore Rehandle (Crusher Feed & Crushed Rehandle)	**$36,553**	$2,020	$4,833	$8,100	$10,800	$10,800	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe/Refinery	**$558,552**	$3,155	$8,781	$9,772	$15,456	$13,530	$15,603	$15,615	$16,445	$19,289	$19,909	$21,083	$22,070
Lab & Met Services	**$47,980**	$160	$497	$646	$1,182	$1,543	$1,662	$1,526	$1,817	$1,571	$2,018	$2,017	$1,903
Ancillaries	**$29,477**	$33	$79	$132	$176	$176	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031
Site G&A	**$507,184**	$5,000	$10,000	$10,000	$12,500	$14,684	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000
Treatment & Refining Charges	**$265,791**	$332	$1,446	$2,103	$6,033	$4,493	$6,423	$6,432	$7,084	$9,323	$9,806	$10,729	$11,505
Transportation	**$6,271**	$94	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187
Warehouse Inventory	**$0**	$52	$223	$412	$695	$864	$0	$0	$0	$0	$0	$0	$0
Crofoot Royalty	**$5,110**	$17	$0	$1,417	$3,240	$436	$0	$0	$0	$0	$0	$0	$0
Net Proceeds Tax	**$192,288**	$20	$0	$2,310	$4,919	$2,732	$4,247	$1,756	$1,585	$9,809	$3,525	$6,040	$7,205
Income Taxes	**$615,247**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$4,463	$19,711
Total Operating Cost	$10,263,466	$36,132	$118,976	$175,400	$233,614	$262,961	$292,467	$304,786	$289,043	$324,756	$331,859	$344,510	$360,455
Other Costs													
Reclamation & Closure Costs	**$57,602**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Other Costs	$57,602	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Expenditures													
Geotechnical/Metallurgical Drilling	**$1,238**	$0	$0	$0	$0	$0	$0	$1,238	$0	$0	$0	$0	$0
Leach Pad & Pond Construction	**$167,396**	$31,951	$18,425	$0	$455	$0	$22,500	$0	$0	$0	$0	$455	$22,500
Mobile Equipment	**$393,958**	$0	$312	$0	$0	$0	$48,198	$110,212	$57,500	$1,200	$17,400	$0	$0
M3 Capital:													
General	**$4,397**	$0	$0	$4,397	$0	$0	$0	$0	$0	$0	$0	$0	$0
Primary Crushing	**$383**	$0	$0	$383	$0	$0	$0	$0	$0	$0	$0	$0	$0
Secondary & Tertiary Crushing	**$40,419**	$0	$0	$9,911	$9,459	$21,050	$0	$0	$0	$0	$0	$0	$0
Conveying & Stacking	**$70,560**	$0	$0	$12,991	$12,991	$39,941	$1,159	$1,159	$1,159	$1,159	$0	$0	$0
Heap Leach	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Overhead Power Line	**$1,450**	$0	$0	$0	$0	$0	$0	$0	$1,450	$0	$0	$0	$0
Reagents	**$19,767**	$0	$0	$0	$0	$18,146	$1,121	$500	$0	$0	$0	$0	$0
Rail Unloading & Storage	**$32,340**	$0	$0	$0	$10,792	$21,548	$0	$0	$0	$0	$0	$0	$0
De-watering	**$15,811**	$0	$3,850	$376	$2,385	$3,791	$1,197	$0	$376	$0	$652	$376	$0
Owner's Team	**$5,148**	$4,548	$600	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Other Annual Sustaining	**$15,500**	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500
Total Capital Expenditures	$768,369	$36,999	$23,687	$28,558	$36,582	$104,976	$74,676	$113,609	$60,985	$2,859	$18,552	$1,331	$23,000
Total Cash Outflows	$11,089,437	$73,131	$142,663	$203,958	$270,196	$367,937	$367,142	$418,396	$350,028	$327,615	$350,411	$345,841	$383,455
Net Cash Flow	$5,072,113	-$8,925	-$19,639	$53,542	$111,197	-$13,961	$85,248	-$12,838	$71,666	$347,425	$170,133	$240,327	$224,701
Cumulative Net Cash Flow		-$8,925	-$28,565	$24,977	$136,175	$122,214	$207,462	$194,624	$266,289	$613,715	$783,848	$1,024,175	$1,248,877
Mining ($/t mined)	**$1.61**	$2.57	$2.62	$2.14	$1.78	$1.54	$1.57	$1.50	$1.25	$1.55	$1.44	$1.54	$1.63
Heap Leach ($/ore t)	**$2.93**	$2.22	$2.62	$2.27	$2.14	$3.00	$2.58	$2.78	$3.05	$3.06	$3.21	$3.02	$2.43
Crushing Costs ($/crushed ore ton)	**$0.50**	$0.83	$0.89	$0.63	$0.53	$0.53	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50
Ore Rehandle ($/crushed ore ton)	**$0.03**	$0.45	$0.45	$0.45	$0.45	$0.45	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Process Plant ($/ore t)	**$0.49**	$0.70	$0.70	$0.42	$0.56	$0.47	$0.43	$0.38	$0.46	$0.51	$0.55	$0.58	$0.54
Lab & Met Services ($/ore t)	**$0.04**	$0.04	$0.04	$0.03	$0.04	$0.05	$0.05	$0.04	$0.05	$0.04	$0.06	$0.06	$0.05
G&A ($/ore t)	**$0.43**	$1.11	$0.70	$0.35	$0.40	$0.44	$0.47	$0.37	$0.47	$0.42	$0.47	$0.46	$0.38
Treatment Charges/Transportation ($/ore t)	**$0.24**	$0.09	$0.13	$0.10	$0.23	$0.16	$0.18	$0.16	$0.20	$0.25	$0.28	$0.30	$0.29
Net Proceeds Tax ($/ore t)	**$0.17**	$0.00	$0.00	$0.10	$0.18	$0.10	$0.12	$0.04	$0.04	$0.26	$0.10	$0.17	$0.18
Total Operating Cost ($/ore t)	**$8.52**	$8.05	$9.47	$7.53	$8.46	$9.17	$8.12	$7.45	$8.00	$8.54	$9.15	$9.36	$8.40

	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24
Cash Outflows ($000)												
Operating Cost												
Mining	$143,051	$135,418	$143,188	$155,266	$152,984	$156,567	$141,667	$170,598	$143,752	$137,971	$140,478	$149,336
Heap Leach	$126,333	$122,162	$104,947	$111,421	$107,707	$104,975	$95,608	$105,175	$111,422	$110,400	$98,685	$111,328
Crushing Costs	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992	$17,992
Ore Rehandle (Crusher Feed & Crushed Rehandle)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe/Refinery	$17,377	$23,195	$15,657	$14,975	$15,272	$15,203	$16,971	$21,927	$23,087	$20,767	$16,189	$18,190
Lab & Met Services	$2,003	$1,988	$2,246	$2,005	$2,005	$1,973	$1,622	$2,090	$1,899	$1,855	$1,712	$1,355
Ancillaries	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031
Site G&A	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000
Treatment & Refining Charges	$7,817	$12,390	$6,468	$5,929	$6,162	$6,109	$7,499	$11,394	$12,301	$10,483	$6,883	$8,456
Transportation	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187
Warehouse Inventory	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Crofoot Royalty	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net Proceeds Tax	$2,546	$11,237	$4,741	$1,348	$1,989	$2,257	$5,485	$8,556	$5,468	$10,706	$2,583	$4,658
Income Taxes	$10,158	$44,836	$18,916	$5,378	$7,934	$9,005	$21,887	$34,138	$21,815	$42,717	$10,305	$18,586
Total Operating Cost	$345,495	$387,436	$332,373	$332,530	$330,262	$332,298	$326,949	$390,087	$355,952	$371,108	$313,045	$348,119
Other Costs												
Reclamation & Closure Costs	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Other Costs	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Expenditures												
Geotechnical/Metallurgical Drilling	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Leach Pad & Pond Construction	$0	$0	$0	$0	$0	$455	$35,100	$0	$0	$0	$0	$0
Mobile Equipment	$0	$2,300	$0	$0	$9,200	$25,112	$28,824	$57,400	$22,000	$5,100	$0	$0
M3 Capital:												
General	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Primary Crushing	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Secondary & Tertiary Crushing	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Conveying & Stacking	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Heap Leach	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Overhead Power Line	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reagents	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Rail Unloading & Storage	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
De-watering	$0	$376	$0	$0	$376	$652	$0	$376	$0	$0	$0	$0
Owner's Team	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Other Annual Sustaining	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500
Total Capital Expenditures	$500	$3,176	$500	$500	$10,076	$26,719	$64,424	$58,276	$22,500	$5,600	$500	$500
Total Cash Outflows	$345,995	$390,612	$332,873	$333,030	$340,338	$359,017	$391,373	$448,363	$378,452	$376,708	$313,545	$348,619
Net Cash Flow	$124,155	$293,359	$157,549	$53,044	$65,013	$61,245	$111,818	$197,044	$176,932	$282,771	$112,180	$163,973
Cumulative Net Cash Flow	$1,373,032	$1,666,390	$1,823,939	$1,876,983	$1,941,996	$2,003,241	$2,115,059	$2,312,104	$2,489,035	$2,771,806	$2,883,986	$3,047,959
Mining ($/t mined)	$1.43	$1.35	$1.32	$1.55	$1.53	$1.57	$1.67	$1.66	$1.49	$1.46	$1.59	$1.99
Heap Leach ($/ore t)	$3.43	$3.27	$2.91	$3.03	$2.93	$2.77	$2.65	$2.92	$3.00	$3.01	$2.74	$3.09
Crushing Costs ($/crushed ore ton)	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50
Ore Rehandle ($/crushed ore ton)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Process Plant ($/ore t)	$0.47	$0.62	$0.43	$0.41	$0.42	$0.40	$0.47	$0.61	$0.62	$0.57	$0.45	$0.50
Lab & Met Services ($/ore t)	$0.05	$0.05	$0.06	$0.05	$0.05	$0.05	$0.04	$0.06	$0.05	$0.05	$0.05	$0.04
G&A ($/ore t)	$0.45	$0.44	$0.47	$0.45	$0.45	$0.43	$0.47	$0.47	$0.45	$0.46	$0.47	$0.47
Treatment Charges/Transportation ($/ore t)	$0.22	$0.34	$0.18	$0.17	$0.17	$0.17	$0.21	$0.32	$0.34	$0.29	$0.20	$0.24
Net Proceeds Tax ($/ore t)	$0.07	$0.30	$0.13	$0.04	$0.05	$0.06	$0.15	$0.24	$0.15	$0.29	$0.07	$0.13
Total Operating Cost ($/ore t)	$9.11	$9.17	$8.69	$8.90	$8.77	$8.52	$8.44	$9.87	$9.01	$8.97	$8.40	$9.14

	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35
Cash Outflows ($000)											
Operating Cost											
Mining	$124,923	$115,528	$110,798	$127,733	$113,820	$119,909	$117,008	$89,621	$95,290	$9,837	$1
Heap Leach	$115,955	$115,074	$115,721	$115,807	$106,446	$110,692	$106,812	$98,988	$112,315	$16,898	$0
Crushing Costs	$17,992	$17,992	$17,992	$17,992	$17,992	$17,345	$17,183	$17,183	$12,716	$497	$0
Ore Rehandle (Crusher Feed & Crushed Rehandle)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe/Refinery	$15,103	$13,441	$14,002	$14,041	$14,989	$12,338	$15,712	$19,079	$39,704	$625	$0
Lab & Met Services	$1,154	$1,154	$1,119	$1,049	$952	$1,060	$916	$567	$713	$0	$0
Ancillaries	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$1,031	$924	$129	$0
Site G&A	$17,000	$17,000	$17,000	$17,000	$17,000	$17,000	$10,000	$10,000	$5,000	$5,000	$0
Treatment & Refining Charges	$6,031	$4,725	$5,166	$5,196	$6,032	$3,944	$6,705	$9,879	$27,120	$13,390	$0
Transportation	$187	$187	$187	$187	$187	$187	$187	$187	$187	$187	$0
Warehouse Inventory	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$2,246	$0
Crofoot Royalty	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net Proceeds Tax	$2,937	$2,352	$2,477	$2,379	$12,324	$1,944	$7,167	$11,679	$23,168	$20,139	$0
Income Taxes	$11,718	$9,385	$9,884	$9,492	$49,174	$7,755	$28,597	$46,601	$92,439	$80,353	$0
Total Operating Cost	$314,030	$297,868	$295,377	$311,905	$339,947	$293,203	$311,319	$304,815	$409,577	$144,809	$1
Other Costs											
Reclamation & Closure Costs	$0	$0	$0	$0	$0	$0	$0	$0	$0	$28,801	$28,801
Salvage Value	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Other Costs	$0	$0	$0	$0	$0	$0	$0	$0	$0	$28,801	$28,801
Capital Expenditures											
Geotechnical/Metallurgical Drilling	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Leach Pad & Pond Construction	$455	$35,100	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mobile Equipment	$0	$0	$9,200	$0	$0	$0	$0	$0	$0	$0	$0
M3 Capital:											
General	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Primary Crushing	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Secondary & Tertiary Crushing	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Conveying & Stacking	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Heap Leach	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Merrill Crowe	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Overhead Power Line	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reagents	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Rail Unloading & Storage	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
De-watering	$376	$652	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Team	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Other Annual Sustaining	$500	$500	$500	$500	$500	$500	$500	$0	$0	$0	$0
Total Capital Expenditures	$1,331	$36,252	$9,700	$500	$500	$500	$500	$0	$0	$0	$0
Total Cash Outflows	$315,361	$334,120	$305,077	$312,405	$340,447	$293,703	$311,819	$304,815	$409,577	$173,610	$28,802
Net Cash Flow	$116,355	$62,011	$93,231	$95,021	$286,714	$72,987	$186,196	$264,691	$485,797	$389,952	-$28,802
Cumulative Net Cash Flow	$3,164,314	$3,226,325	$3,319,556	$3,414,577	$3,701,291	$3,774,278	$3,960,474	$4,225,165	$4,710,963	$5,100,915	$5,072,113
Mining ($/t mined)	$1.85	$1.72	$1.68	$2.01	$1.90	$1.88	$1.94	$1.75	$1.68	$1.16	$0.00
Heap Leach ($/ore t)	$3.22	$3.20	$3.21	$3.22	$2.96	$3.07	$2.97	$2.75	$3.12	$2.00	$0.00
Crushing Costs ($/crushed ore ton)	$0.50	$0.50	$0.50	$0.50	$0.50	$0.48	$0.48	$0.48	$0.35	$1.09	$0.00
Ore Rehandle ($/crushed ore ton)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Process Plant ($/ore t)	$0.42	$0.37	$0.39	$0.39	$0.42	$0.34	$0.44	$0.53	$1.10	$0.07	$0.00
Lab & Met Services ($/ore t)	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.02	$0.02	$0.00	$0.00
G&A ($/ore t)	$0.47	$0.47	$0.47	$0.47	$0.47	$0.47	$0.28	$0.28	$0.14	$0.30	$0.00
Treatment Charges/Transportation ($/ore t)	$0.17	$0.14	$0.15	$0.15	$0.17	$0.11	$0.19	$0.28	$0.76	$1.61	$0.00
Net Proceeds Tax ($/ore t)	$0.08	$0.07	$0.07	$0.07	$0.34	$0.05	$0.20	$0.32	$0.64	$2.38	$0.00
Total Operating Cost ($/ore t)	$8.40	$8.01	$7.93	$8.40	$8.08	$7.93	$7.85	$7.17	$8.81	$7.62	$0.00

Financial Indicators	
NPV @ 0%	$5,072,113
NPV @ 5%	**$2,080,798**
NPV @ 10%	$1,053,725
IRR	**148.6%**
Payback (years)	**2.5**

19.13 SENSITIVITIES

Table 19-7: Hycroft Metal Price Sensitivity

Case	Metal Prices ($/oz.)		NPV @ 0%	NPV @ 5%	After Tax IRR
	Au	Ag	*$ Billions*	*$ Billions*	
1	$1,200	$16.50	$4.2	$1.7	80.2%
2	$1,300	$17.33	$5.1	$2.1	148.6%
3	$1,400	$18.67	$6.1	$2.6	307.9%
4	$1,500	$20.00	$7.1	$3.0	N/A

1. *Downside Case (Reserve Price)*
2. *Financial Base Case*
3. *Moderate Case*
4. *Upside Case (After Tax IRR does not calculate due to positive cash flow in year one)*

The following table assumes the base case metal prices of gold ($1,300.00/oz) and silver ($17.33/oz).

Table 19-8: Hycroft Mining/Processing/Capital Cost Sensitivity

Cost Sensitivities		After tax NPV (0%)	After tax NPV (5%)	After tax NPV (10%)	After tax IRR
Area	*Scenario*	*Billions*	*Billions*	*Billions*	*%*
Mining Cost	20% Increase	$4.40	$1.75	$0.85	89%
	10% Increase	$4.74	$1.91	$0.95	115%
	10% Decrease	$5.41	$2.25	$1.15	190%
	20% Decrease	$5.74	$2.41	$1.25	242%
Processing Cost	20% Increase	$4.33	$1.72	$0.84	82%
	10% Increase	$4.70	$1.90	$0.95	111%
	10% Decrease	$5.44	$2.26	$1.16	201%
	20% Decrease	$5.80	$2.43	$1.26	280%
Capital Expenditures	20% Increase	$4.92	$1.98	$0.98	96%
	10% Increase	$5.00	$2.03	$1.02	118%
	10% Decrease	$5.15	$2.13	$1.09	200%
	20% Decrease	$5.23	$2.18	$1.13	320%

20 ADJACENT PROPERTIES

There are no properties adjacent to the Hycroft project with recent Mineral Resource or Mineral Reserve estimates.

21 OTHER RELEVANT DATA AND INFORMATION

All data relevant to the feasibility study and associated mineral reserves and mineral resources have been included in the sections of this Technical Report Summary.

22 INTERPRETATION AND CONCLUSIONS

Hycroft has a long history in heap leaching of oxide and transition ores. This history includes a wealth of knowledge on how the ore behaves during heap leaching, including operational data on reagent consumptions and other operating costs. The crushing plant that is already in place requires only minor modifications to deliver crushed ore to the leach pad. The secondary and tertiary crushers have been refurbished by the vendor.

22.1 MINERAL RESOURCES

The Hycroft resource model was reviewed and recalculated independently by SRK, who also made refinements to the geologic model. The mineral resources stated in this Technical Report are based upon currently available exploration information. This data includes historical information that was collected prior to current standards. However, the uncertainty and risk associated with this historic data has been mitigated through the addition of modern drilling that has been subjected to strict QA/QC protocols that met or exceeded the industry best practices at the time.

22.2 MINERAL RESERVES

The Hycroft deposit supports continued successful exploitation, given the size, grade, metallurgical characteristics, developed infrastructure, and the knowledge and experience of the individuals engaged in the project. The uncertainty and risk associated with the historic exploration data was mitigated where possible, through the following:

- Monthly reconciliation between the exploration model and actual production to verify the mineral reserve estimates and modeling methodologies; and
- Long-term production (1982-present) records to verify mineral reserve estimates for gold and silver.

22.3 SOCIAL IMPACT, PERMITS AND UTILITIES

The contemplated operations would provide additional direct employment for the area and provide a long-term revenue source to the local and state economy.

HMC has all the necessary permits to mine above the water table and process ore by heap leaching. The construction of additional heap leach pad space will be required to accommodate the plans detailed in this report for which multiple federal, state and local permits will need to be obtained. To date, HMC has received support of Pershing and Humboldt counties, neighboring towns including Winnemucca, Gerlach and Lovelock as well as regulatory bodies such as the BLM and EPA among others. The plan of operations was submitted for the supplemental EIS, which includes mining below the water table. Continued support is expected from local parties. HMC sees permitting as low-risk given the past success, support levels, new regulatory timelines imposed on permitting activities and the ability to mitigate delays with supplemental plans.

HMC has received power supply quotes from NV Energy, the only power supplier in Northern Nevada. HMC believes that it will achieve the GS-4 rate, established for large consumers, based on power requirements. The long-term rate used in the feasibility study is $0.056/kWh.

22.4 PROJECT FINANCIALS

Sensitivity analysis indicates that this is a robust project that can withstand 20% increases in the key cash flow components:

- If mining operating costs were to increase 20% from those currently estimated, the project would still remain viable by interpolation of the sensitivities shown in Table 19-8.

- If processing costs were to increase 20% from those currently estimated, the project would still remain viable by interpolation of the sensitivities shown in Table 19-8.
- If capital construction costs were to increase 20% from those currently estimated, the project would still remain viable by interpolation of the sensitivities shown in Table 19-8.

22.5 METALLURGICAL PROCESSING

The processing of the run-of-mine oxide and transition ores will continue with the new mine plan. Added to this are oxide and transition ores that will be crushed before stacking on the heap leach pad, as dictated by economics. Processing of these ores will implement the historical procedure, applying the same reagent dosage and leach times that were established in the past.

All sulfide ore in the mine plan will be crushed, oxidized and leached. Some transition ores will use the sulfide protocol as dictated by economics.

Tests results show that oxidation of sulfide ores is accelerated in the presence of carbonate, which may be supplied by soda ash or trona. For the ore samples that were tested with the prescribed oxidation procedure, the resulting cyanide leach recovery are expected to average at about 70%, with Central ores showing a maximum gold recovery of 85%. In essence, the oxidation of sulfide and transition ores converts them into oxide ores.

The most important control parameters in the oxidation process are pH and oxygen availability. The oxidation must be conducted in the presence of sufficient soda ash to keep the pH near the buffer region between carbonate and bicarbonate (pH 10.3). At this pH regime, ferrous and ferric carbonate complexes become stable and provides a carbonate complex version of the Fe(II)/Fe(III) couple. During operations, iron ions will already be present in the recycled carbonate solutions which should initiate the reaction sooner than the laboratory tests.

Oxygen is the ultimate oxidizer in the process. The tests results show that natural air pockets need to be formed during stacking of the ore and maintained during the oxidation phase and the leach phase. This is consistent with the procedure to keep the ore just wet enough to promote the reactions that occur in the aqueous phase, while keeping the interstices in the stack open for air to occupy.

Laboratory oxidation tests were performed for 60, 90 and 120 days. These time periods are most probably the right range of oxidation times in actual operations. However, the required times may be shorter if the presence of iron in recycled carbonate solutions is exploited, assuming that permeability of the ore stack is tightly maintained.

23 RECOMMENDATIONS

Based on financial and technical measures, exploration success, and positive economic benefits, and project developments to date, it is recommended that HMC move the project into production.

23.1 GEOLOGY, EXPLORATION AND DRILLING:

SRK recommends the following:

- The estimated costs and current metal prices have defined Hycroft's gold equivalent cutoff grade at low values compared to those typically reported by analytical labs. Future gold and silver assay methods should have method detection limits that are sufficiently low to yield repeatable results, with proportionally low analytical uncertainty, near the cutoff grade. Assay results should be reported with sufficient precision for calculation of valid ratios of cyanide-soluble to total gold, to assign gold recovery material types. Results may be reported in parts per million and converted to ounces per ton, as long as precision is maintained.
- There is an opportunity to refine the interpreted East Fault surface with new metallurgical drill hole data and consideration of assay values where geological data are incomplete. This would improve material type designations at depth.
- Drill hole density in the known deposit areas is sufficient for estimation of a mineral resource, as classified herein. If additional drilling is done before mining resumes, it should target areas that may be inaccessible during mining. Other targets should include shallow, Inferred material with economic grades in the Resource pit. This could define additional high-recovery material to include in the short-term mining plan.

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25 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

M3, SRK and HMC have relied upon written reports and statements of other individuals and companies with whom it does business. It is believed that the basic assumptions are factual and accurate, and that the interpretations are reasonable. This data has been relied upon in the feasibility study and there is no reason to believe that any material facts have been withheld or misstated. The Qualified Persons have taken all appropriate steps, in their professional judgment, to ensure that the work, information, or advice from the below noted individuals and companies is sound and the Qualified Persons do not disclaim any responsibility for the Technical Report Summary.

HMC's technical and financial professionals and consultants provided input for the following sections: mine planning, environmental, metallurgy, geology and financial models. HMC personnel who have been instrumental in preparing this study are as follows:

- Randy Buffington – President & Chief Executive Officer
- Stephen Jones – Executive Vice President & Chief Financial Officer
- Tracey Thom – Vice President, Environmental and Corporate Affairs
- Nigel Bain – Vice President, Operations
- Jeff Stieber – Vice President, Treasurer
- Tom Rice – Engineering Consultant
- Michael Kubel – Mining Engineer, Hycroft Mine
- Bill Dafoe – Process Manager, Hycroft Mine

Each of these individuals visits the Hycroft mine on a regular basis as part of the normal course of business.