# NI 43-101 Technical Report
# Feasibility Study
# Mt. Hamilton Gold and Silver Project
# Centennial Deposit and Seligman Deposit
# White Pine County, Nevada

**Report Prepared for**

## Mt. Hamilton LLC

***With***

## Solitario Exploration & Royalty Corp.

***And***

## Ely Gold Minerals Inc.

**Report Prepared by**

srk consulting

**SRK Project Number 181700.100**

**Effective Date: August 14, 2014 (Re-statement of Reserves)**
**Report Date: October 16, 2014**

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# Summary (Item 1)

## Introduction

This report was prepared as a Canadian National Instrument 43-101 (NI 43-101) Technical Report for Mt. Hamilton LLC (MH-LLC), a limited liability company owned by Solitario Exploration & Royalty Corp. (Solitario) and Ely Gold and Minerals Inc. (Ely Gold), by SRK Consulting (U.S.), Inc. (SRK). Within this report, MH-LLC may be construed as MH-LLC separately or collectively as MH-LLC, Solitario and Ely Gold.

This 2014 Technical Report supersedes the existing 2012 Technical Report (SRK, 2012) and represents a fully remodeled combination of both the original Centennial Deposit reserves reported in 2012 with the addition of new mineral reserves for the adjacent Seligman Deposit. The Centennial and Seligman Deposits, both wholly owned by MH-LLC, are now collectively referred to as "Mt. Hamilton" or the "Project".

The Seligman Deposit is located immediately north of the Centennial Deposit, sharing Centennial's mineralogical and metallurgical characteristics. The Northeast Seligman Mine was formerly operated from 1994-1997 by Rea Gold. Production at the mine was halted prematurely in June of 1997 due to operational problems and low metal prices. The Seligman contribution to the mineral reserve reported herein represents an unmined portion of that deposit which comprises approximately 25% of the new Mt. Hamilton reserve.

The 22.5 million ton (Mt) reported reserve in this Technical Report was intentionally constrained by the size of the private land parcel owned by MH-LLC on which the currently permitted leach pad is to be located. Additional in–pit Indicated and Inferred Resources are reported in the Resource Statement that may be economically processed and placed on an expanded leach pad with additional permitting. It is anticipated that a program of definition drilling will be initiated in the near future to provide adequate drill density and metallurgical/geotechnical samples so that these resources may be converted to reserves and soon thereafter, permitting of the expanded leach pad will be initiated.

The two deposits will share the same ore flow and ore processing facilities. Initiation of construction activities for the Project is dependent upon securing financing for the project, which is currently underway. A Feasibility Study document (SRK, 2014a) was produced in conjunction with this Technical Report, which contains all recent and relevant data to support the summary descriptions and conclusions made herein.

Mt. Hamilton is an advanced mineral project with a favorable economic projection based on feasibility-level capital and operating costs from detailed mining and process engineering. Mining will occur in several open pits at high elevation (8,100 to 9,480 ft) using conventional truck and shovel/loader methods to deliver 10,000 t/d ore to a primary jaw crusher at 8,650 ft elevation. Crushed ore will be dropped approximately 415 ft in a vertical ore pass to an underground chamber, where it will be reclaimed by pan-feeder to a conveyor. Ore will travel via conveyor 4,425 ft underground on a -15% decline to the conveyor adit portal and then transferred to a coarse-ore stockpile at 7,550 ft elevation. A reclaim tunnel under the stockpile will feed a secondary cone crusher, reducing the particle size to 90% passing -3/4 inch for radial stacking on a 22.5 Mt capacity HDPE-lined leach pad. Stacked ore will be leached with a cyanide solution. Pregnant solution will be collected in ponds and processed using conventional adsorption-desorption-recovery (ADR) carbon-in-column technology to produce a gold/silver doré product on site.

This report provides mineral resource and mineral reserve estimates, and a classification of resources and reserves in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM). It also meets the standards of the U.S. Securities and Exchange Commission Industry Guide 7 for estimating and reporting reserves.

## Technical Economics

The indicative economic results are shown on Table 1. The following provide the basis of the SRK LoM plan and economics:

- Production Rate: 10,000 t/d ore;
- Mine Life: 7 years;
- Payable metal of 415 koz gold and 1,690.4 koz silver;
- Unprocessed stockpile of 2.96 Mt at 0.010 oz/t gold;
- Average Recovery: 76% for gold; 39% for silver
- Life of Mine Strip Ratio: 2.5:1.0 (waste:ore, includes stockpiled ore);
- Initial Capital Cost: US$91.7 million;
- Life of Mine Capital Cost: US$121.5 million;
- Underlying NSR-Royalty: 3.4% ;
- Cash Costs per Gold-Equivalent (AuEq) Ounce Recovered: US$558;
- Average Annual Gold Production: 68,600 oz; Average Gold Grade: 0.024 oz/t;
- Average Annual Silver Production: 279,400 oz; Average Silver Grade:0.197 oz/t;
- Average Annual Gold-Equivalent Production: 73,000 oz;
- <u>After tax</u> Internal Rate of Return (IRR): 26.0%; and
- <u>After tax</u> payback Period: 2.9 years.

**Table 1: Indicative Economic Results**

| Description | Units | With Tax | Without Tax |
|---|---|---|---|
| **Market Prices** | | | Without Tax |
| Gold (LoM Avg) | /oz-Au | $1,300 | $1,300 |
| Silver (LoM Avg) | /oz-Ag | $20.00 | $20.00 |
| **Estimate of Cash Flow (all values in US$000's** | | | |
| **Payable Metal** | | | |
| Gold | koz | 415.0 | 415.0 |
| Silver | koz | 1,690.4 | 1,690.4 |
| **Gross Revenue** | - | | |
| Gold | | $539,494 | $539,494 |
| Silver | | $33,808 | $33,808 |
| **Revenue** | | **$573,302** | **$573,302** |
| Refinery & Transport | | ($3,273) | ($3,273) |
| **Gross Revenue** | | **$570,030** | **$570,030** |
| Royalty | | ($17,015) | ($17,015) |
| **Net Revenue** | | **$553,015** | **$553,015** |
| **Operating Costs** | US$/t-ore | | |
| Mining | $5.99 | $134,740 | $134,740 |
| Processing | $4.11 | $92,427 | $92,427 |
| G&A | $0.84 | $18,863 | $18,863 |
| Property & Net Proceeds Tax | $0.58 | $12,943 | $12,943 |
| **Total Operating** | **$11.51** | **$258,972** | **$258,972** |
| **Operating Margin (EBITDA)** | | **$294,042** | **$294,042** |
| LoM Capital | | $121,518 | $121,518 |
| Income Tax | | $56,643 | $0 |
| **Cash Flow Available for Debt Service** | | **$115,882** | **$184,760** |
| **NPV 5%** | | **$78,466** | **$131,835** |
| **NPV 8%** | | **$60,817** | **$106,951** |
| **IRR** | | **26.0%** | **35.4%** |

Source: SRK, 2014

# Property Description and Ownership

The Mt. Hamilton Project, which contains the Centennial and Seligman Deposits, is located in White Pine County, Nevada at 115.56° W Longitude and 39.25° N Latitude, in the northern White Pine Mountains. The terrain is high mountain desert with cold winters and warm summers. Project elevations range from 7,000 ft to 9,500 ft above mean sea level (amsl). The Project area has good connections to the infrastructure of northeastern Nevada, and is accessed from U. S. Highway 50 on gravel-surfaced public and private roads. The Project will be operated using generated power initially, then converting to line power early in the mine life.

Water rights for full production have been secured; MH-LLC has appropriated a total of 875 acre-feet per annum of water, an amount sufficient for peak water requirements for the operation and construction. Water will be supplied by an existing well in Seligman Canyon. The Seligman Canyon well is capable of producing 500 gallons per minute (gpm) and a second, backup well can produce 200 gpm. Water resource exploration is proposed to install and develop an additional well closer to the processing facility prior to operations.

## History and Agreements

Phillips Petroleum Co. (Phillips) acquired much of the area of the current Property in 1968 and, between 1968 and 1982, drilled over 100,000 ft in the exploration for tungsten-copper-molybdenum deposits. In 1984 Northern Illinois Coal, Oil and Resources Mineral Ventures, subsequently renamed Westmont Gold Inc., (Westmont) entered into a joint venture with Phillips and Queenstake Resources Ltd. to explore the property for open-pit mineable gold-silver mineralization. By early 1989, this work had defined the Seligman and Centennial Deposits. The property was transferred to Mt. Hamilton Mining Company (MHMC, a Westmont subsidiary) after November 1993. Rea Gold Corp. (Rea) acquired MHMC in June 1994 and began production of the Seligman deposit located to the north of Centennial in November 1994. Rea had planned to commence mining of the Centennial deposit in 1997, but this never occurred. Rea ceased mining in June 1997, but continued leaching until declaring bankruptcy in Canadian Bankruptcy Court in November 1997. In 2002, the US Bankruptcy Trustee abandoned all of the unpatented claims, allowing them to lapse for failure to pay the annual maintenance fees. Centennial Minerals Company LLC staked claims covering the Centennial Deposit in late 2002, and in 2003 purchased all of the patented mining claims and Fee lands from the US Bankruptcy court. Augusta Resource Corporation (Augusta), through its 100% owned subsidiary Diamond Hill Minerals Ltd (DHI), acquired a leasehold interest in the property from Centennial in late 2003. Under an agreement with Augusta dated November 15, 2007, Ivana Ventures Inc. (Ivana) acquired 100% of the shares of DHI. Ivana changed its name to Ely Gold & Minerals (Ely) in 2008. On August 26, 2010, Solitario signed a Letter of Intent with Ely to earn up to an 80% interest in Ely's Mt. Hamilton gold property. In December 2010, Solitario and Ely formed MH-LLC which now holds 100% of the Mt. Hamilton project assets, and signed an LLC Operating Agreement. In December 2013, Ely and Solitario made a final payment to Augusta eliminating all financial obligations to Augusta.

**Land Position - Claims**

The MH-LLC land position consists of both private property and unpatented mining claims on federal land. MH-LLC controls the Property through direct ownership and through lease option agreements. The Project is comprised of two parcels of fee simple land totaling 240 ac, nine surveyed Patented Mineral Claims totaling 120.57 ac, and 305 unpatented Federal mining claims totaling approximately 5,094 ac. Claims are located in Sections 8, 9, 15, 16, 17, 21, 22, 27, 28, Township 16N, Range 57E, in White Pine County, Nevada.

Ely Gold's predecessor, Ivana, acquired DHI Minerals (US) Ltd. (DHI) from Augusta in November, 2007. DHI had previously acquired, through a lease agreement with Centennial Minerals Company (Centennial), the mineral rights to the "H" series claims and patented claims. These claims cover the resources and reserves at the Centennial Deposit in the north central part of the Property. DHI has assigned 100% of its lease holding interest in the above mentioned claims to MH-LLC.

**Environmental Liabilities and Permitting**

Previous mining at the Property was conducted by Rea in the NE Seligman area, and included the construction of open pit excavations, a waste rock dump and a heap leach pad. The site of the former mine-associated facilities has been partially reclaimed by the U.S. Department of Agriculture, United States Forest Service (USFS, or Forest Service) and the U.S Department of the Interior Bureau of Land Management (BLM). All buildings have been removed and the leach pad associated with previous mining has been covered with soil, re-contoured, and seeded. MH-LLC currently has no environmental liabilities related to this previous mining activity. Various federal agencies, departments within the State of Nevada and White Pine County, and local governments are cooperating agencies in permitting mine development and process facilities at the site.

A Mine Plan of Operations (MPO) was submitted to the Forest Service for mining activities on National Forest System (NFS) lands. The MPO was determined to be complete by the USFS and scoping of the project was conducted in order to determine the issues to be evaluated to comply with the National Environmental Policy Act (NEPA). The USFS determined that an Environmental Assessment (EA) was required. Upon completion of the EA, a Finding of No Significant Impact and a draft Decision Notice were published on July 4, 2014. The Objection Period ended on August 18, 2014 with no objections filed. Phased bonding for reclamation of the mining areas will be required. The initial bonding of the first phase was submitted to the Forest Service, reviewed and accepted on September 24, 2014. The bill of collection, receipt and issue is pending.

Road access to the mine and to the administration/processing areas each requires crossing BLM land in order to enter the MPO area on Forest Service property. These two access routes are subject to a Right of Way grant by the BLM, which was issued to MH-LLC in 2013.

A Nevada Reclamation Permit (NRP) application has been submitted for the area covered by the MPO. The application for this permit is under review by the Nevada Division of Environmental Protection (NDEP), Bureau of Mining Regulation and Reclamation (BMRR).

The private land used for processing the ore and administrative functions is being permitted and bonded separately through the NDEP BMRR and will have a separate Nevada Reclamation Permit. An application for this permit has been filed and is under review. The USFS will not be involved in this permit approval although operations on private land are considered in the NEPA analysis as a connected action.

A Water Pollution Control Permit (WPCP) has been issued by the NDEP. The WPCP covers the entire project including both public and private land.

An Air Quality Permit application has been submitted to NDEP for review. A preliminary ADR plant design has been completed in order to provide the detail necessary for design of the mercury control systems to be incorporated in and reviewed under the Air Quality Permit.

Because of previously permitted mining activity at the Project, SRK has no reason to believe that the few remaining permits to mine the mineral resources of the Project could not be reasonably obtained from State and Federal regulatory agencies.

## Geology and Mineralization

The Mt. Hamilton Property is located in the White Pine Mountains, which are in the eastern sector of the Great Basin in east-central Nevada. The White Pine Mountains are one of the many mountain ranges that have been uplifted along north-striking steeply dipping normal faults formed during extension that formed the Great Basin Physiographic Province. This region was subjected to east-to-west compression during the Sevier and Laramide orogenies in the Cretaceous and early Tertiary periods. This compression resulted in the formation of broadly north-trending folds and thrust faults. Two major folds are present in the project area: the Hoppe Springs anticline (into which the Seligman stock has intruded) and the Silver Bell syncline to the west. The folded units are a package of Cambrian- to Pennsylvanian-age sedimentary rocks, but only the Cambrian age units are present in the Project area. The igneous intrusive stocks were the cause of district-wide contact metamorphism that resulted in hornfels and skarn alteration of the Cambrian-age host rock units.

The units that host gold mineralization are the Middle Cambrian Secret Canyon Shale, and to a much lesser extent, the Upper Cambrian Dunderberg Shale (Burgoyne, 1993). In general, both units consist of calcareous laminated mudstones with thin limestone interbeds. The Dunderberg disconformably overlies the Secret Canyon, and both of these units are exposed at the surface in the Project area. Together, they are up to 2000 ft thick, and host all gold and silver mineralization considered in this report. Younger Paleozoic rock units form the Pancake and White Pine Mountain Ranges, west and east of the project area.

Early metasomatic alteration converted shales and carbonaceous siltstones of the upper Secret Canyon shale to hornfels after shales and calc-silicate skarn after silty carbonates. Mineralization at Mt. Hamilton consists of skarn-hosted tungsten, molybdenum, and copper +/- zinc with later epithermal gold and silver. Gold mineralization is primarily hosted in a 200 to 300 ft thick skarn horizon, bounded by upper (200 ft thick) and lower (450 ft thick) hornfels units. The bounding hornfels had lower permeability and were therefore less receptive to late-stage mineralization. The interbedded skarn in the Centennial area was subject to late-stage, low-angle faulting. These faults were conduits to late mineralizing solutions and oxidation. The result is an oxide-hosted epithermal gold deposit overprinting a retrograde polymetallic skarn. The main Centennial precious metal mineralization is contained within a southeast dipping (15° to 20°) tabular zone that ranges from 20 to 250 ft in thickness. In the NE Seligman area, ore grade mineralization appears to be largely stratiform in shallow-dipping, bedding-parallel, structurally and chemically prepared zones with local high-angle, cross-cutting, possible "feeder" zones (Burgoyne, 1993). At Centennial, the mineralization is controlled by late low-angle structures that are discordant to bedding and oxidized to significant depth. Gold grades of samples within the retrograde alteration range from <0.001 oz/t Au (lower analytical method detection limit) to 0.995 oz/t. The occasional high grades appear to be associated with crosscutting structures and veins within the skarn as described below.

In the Centennial deposit, weathering and oxidation of original sulfide mineralization caused formation of oxide mineralization (with low sulfide mineral residuals) from which gold is recoverable by cyanide heap leaching. In general, the acid generating capacity of the surrounding carbonate rocks is low or nil, and their acid consuming capacity is high. Gold is present as free gold, residing in iron oxide minerals or quartz, and adsorbed on clay minerals. Sulfosalt-bearing veins may be associated locally with the higher grades of gold and particularly silver. These veins cut both skarn and intrusive rocks and are closely associated with zones of retrograde alteration. These veins range in thickness from about 2 to 60 cm. In the Seligman deposit the mineralization is similar to Centennial but is, on average, thinner. Widely spaced drilling between the two deposits indicates that they represent one contiguous mineralized system that could be connected by additional drilling. Locally, more sulfide minerals are preserved at Seligman in comparison to Centennial. As seen in the mine excavations of the NE Seligman deposit, veins seem to exhibit strong continuity along strike.

## Exploration Drilling and Data Quality

A total of four drilling programs have been completed by MH-LLC in the Project area since 2008. Drill holes designed to enhance the resource model, collect rock quality geotechnical data and provide material for metallurgical testing have been completed using diamond drilling techniques (core) and reverse circulation (RC) techniques. Regardless of the main application, all drill holes were sampled and analyzed for whole-rock composition and abundance of precious metals using industry-standard analytical procedures at accredited laboratories.

The most recent drilling was undertaken by MH-LLC in 2011 and 2012 and included 60 holes. The 2011 Centennial drilling program targeted sample material for metallurgical testing. There were seven holes completed including one HQ core and six RC holes totaling 4,424 ft drilled. This was followed by eight exploration or resource confirmation holes completed in the Centennial resource in 2012, including two HQ core, one PQ core, and five RC holes, totaling 5,734 ft drilled. However, the focus of the 2012 drilling program was the Seligman deposit. A total of 45 drill holes were completed in the Seligman resource, including two HQ core, six PQ core, and 37 RC holes, totaling 14,980 ft drilled. The objective of these boreholes was to upgrade the resource classification, and to provide samples for metallurgical and geotechnical evaluation.

RC samples were collected at the rig and were under the control of MH-LLC staff or consultants until they were relinquished to the analytical lab for preparation and analysis. Whole core was collected in boxes at the drill rig and transported back to the MH-LLC core shed for photographing, logging, and splitting with a diamond-blade saw. A continuous half-core was sampled, and the other half was retained in the original core box for future reference. Core samples remained under MH-LLC control until they were relinquished to the analytical lab for preparation and analysis.

Evaluation of check assay results from an outside lab was completed in 2013 and 2014. Drill hole sample sequences included QA/QC samples at a frequency equal to or greater than currently accepted industry standards, and most analytical programs included duplicate analysis on samples selected randomly to assess the quality of the analytical data. All available results are discussed in the Data Verification section of this report. Recent results support resource model estimations and confirm existing data from respective nearby drill holes. Primary assay results indicate that preparation and analytical procedures are defensible, and results are suitable for inclusion in CIM-compliant resource and reserve estimates.

## Metallurgy

The Mt. Hamilton ore reserve has historically been viewed as two ore deposits: Centennial and Seligman. However, recent metallurgical test work has confirmed that oxide mineralization in igneous and skarn rock types responds metallurgically the same in the two deposits. A significant amount of the 2012 drilling was dedicated to Seligman metallurgical characterization, as necessary to convert these resources to reserves.

The lithology of the Centennial deposit consists primarily of oxidized metasediments and some igneous rock (Seligman Granodiorite), with a small percentage of un-oxidized equivalents of the same rock types in comparison to the Seligman deposit. The projected gold recovery from column test work for the Centennial oxide deposit is 79% over the planned 210 day operational leach cycle. Actual column tests for Centennial ore were run for 160 days and scaled up for operations.

Material types for Seligman can be characterized as oxide, transition and sulfide based on their response to cyanide soluble assays. In relation to Centennial, Seligman mineralization contains a larger percentage of sulfide bearing mineralization. Oxide reserves were the target in the production plan, but transition ores, at lower recoveries, will also be mined in small quantities to access the oxide ore. The projected gold recovery from column test work for the Seligman oxides is 80% over the planned 210 day operational leach cycle. Actual column tests on Seligman ore were run for 160 days and 120 days in 2011 and 2012 respectively and scaled up for operations accordingly.

Metallurgical work on samples from the 2011-2012 Centennial and Seligman drilling was conducted to augment and refine metallurgical characterization of the deposits made in previous studies. The new Centennial column tests were composited from 10 PQ-sized core samples and the bottle roll variability tests from 19 RC holes. The Seligman column tests were composited from 7 PQ split-core samples and the bottle roll variability tests from 32 RC holes.

Bottle roll tests were conducted to provide variability data by rock type, metallurgical ore type, lithology, feed grade and spatial coverage. The selection criteria for the bottle roll samples were based on: 1) intervals above an appropriate AuEq cut-off grade (CoG); 2) the average AuEq feed grade for each hole; and 3) the average 'gold ratio' of cyanide soluble to fire assay gold analyses (CN/FA). A single interval for each RC hole most closely representing the average gold-equivalent and average gold ratio values for the hole was selected for the variability samples.

The recoveries and reagent consumptions indicated by column leach testing of oxide ore from the Centennial and Seligman deposits are virtually identical. And there is a strong correlation between cyanide soluble assays and metallurgical recovery that can be applied to grade control for operations.

The overall projected gold recovery based on metallurgical test work is 79% from oxide ores. However, recovery applied to economics was determined on a block-by-block basis calculated from interpolated grades in the block model. The database of cyanide soluble to fire assay paired data (CN/FA) informed a recovery estimate for every block in the resource/reserve model. In some parts of the mine plan, lower recovery ore will be mined to access higher recovery ore. Therefore, high and low recovery material will be blended and the average leach recovery of the current reserve is 76% for gold and 39% for silver. The determination of recoveries based on modeled analytical values is considered to be an advancement for characterizing overall projected recoveries in comparison to assigned recoveries based on observed oxidation from geologic logging.

Additional ore characterization data were used to design crushing and stacking for the heap leach operation. A conservative comminution test value of 8.0 kWh/t was utilized for crushing plant sizing at the optimum leach feed size of 90% passing ¾ inch. Strength and stability testing determined that the ore can be stacked without agglomeration to at least a height of 210 ft and the heap volume is based on a 110 $lb/ft^2$ crushed ore bulk density.

## Mineral Resource Estimate

This Technical Report represents the first tabulation of a resource estimate for the Mt. Hamilton Project as a combination of both the Centennial and Seligman Deposits.

The 2014 Mt. Hamilton resource estimate was based on 857 drill holes with an average hole depth of 370 ft for a total of 317,739 ft of drilling. The drill data were verified and validated by SRK in compliance with NI 43-101 guidance. This consolidated Mt. Hamilton resource estimate includes 60 new infill drill holes that converted earlier Inferred resources to the Indicated category, while also expanding the Seligman resource.

SRK estimated gold and silver grades using inverse distance weighted (IDW) to the second power for each one of seven geologically defined individual grade wireframes, using a three-pass search, with increasingly expanded search distances. In addition to IDW metal grades, the interpolation stored average distance to composites, number of composites and number of drill holes used to estimate each block. A second grade estimation routine was conducted to store nearest neighbor (NN) grades and distance to closest composite for use in model validation.

Mineral resources were classified under the categories of Measured, Indicated and Inferred according to standards as defined by the CIM. Classification of the resources reflects the relative confidence of the grade estimates. This is based on several factors, including: sample spacing relative to the geological and geostatistical observations regarding the continuity of mineralization; mining history; specific gravity determinations; accuracy of drill collar locations; and quality and reliability of the assay data.

The resource model was further investigated with a Lerchs-Grossmann (LG) pit optimization to ensure a reasonable stripping ratio was applied and the resource had *a "reasonable expectation for economic extraction"* as required by NI 43-101 guidelines. Mintec's MineSight® software was used to generate the LG pit optimization using operating cost inputs described in the footnotes of the Mineral Resource Statement. Table 2 is the Mineral Resource Statement for the updated Mt. Hamilton Gold-Silver Deposit.

**Table 2: Mineral Resource Statement at $1,300/oz Au, Mount Hamilton Gold-Silver Deposit, White Pine County, Nevada, March 25, 2014 (0.006 Au oz/t Cut-off)**

| Resource Category | Tons | Au Grade | Ag Grade | AuEq Grade | | Contained Ounces (thousands of oz) | | |
|---|---|---|---|---|---|---|---|---|
| | (000's) | oz/t | oz/t | oz/t | g/tonne | Au | Ag | AuEq |
| Measured | 1,427 | 0.030 | 0.209 | 0.033 | 1.125 | 42 | 299 | 47 |
| Indicated | 32,283 | 0.021 | 0.194 | 0.024 | 0.830 | 685 | 6,271 | 782 |
| **Measured and Indicated** | **33,710** | **0.022** | **0.195** | 0.025 | 0.843 | **727** | **6,569** | **828** |
| Inferred | 6,721 | 0.018 | 0.171 | 0.020 | 0.696 | 119 | 1,153 | 136 |

Source: SRK, 2014

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves estimate;
- Resources stated as contained within a potentially economically minable open pit; pit optimization was based on assumed gold and silver prices of US$1,300/oz and US$19.60/oz, respectively, block-by-block modeled recovery averaging 76% for Au and 39% for Ag, an ore mining cost of US$2.06/t for Seligman, an ore mining cost of US$1.64/t for Centennial and an ore processing cost of US$4.95/t; west pit slopes 45°, east pit slopes of 50°;
- Resources are reported using a 0.006 oz/t contained gold CoG;
- AuEq was calculated using a Ag:Au ratio of 65:1; and,
- Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.

## Mineral Reserves

Engineering completed at Mt. Hamilton includes sufficient drilling, mine planning and economic evaluation to report the Measured and Indicated Mineral Resources as Proven and Probable Mineral Reserves, with reserves intentionally limited to the currently permitted 22.5 Mt heap leach pad capacity.

The Mineral Reserves stated below for Mt. Hamilton were developed using the Lerchs-Grossman pit optimization algorithm in Mintec's Minesight® 3D mining software. Pit slopes applied were developed from dedicated geotechnical drilling supervised and analyzed by SRK in 2011 and 2012.

Pit optimization is based on preliminary economic estimations of mining, processing and selling related costs, slope angles, and metal recoveries. These pit optimization factors vary from those reported in the final economic analysis, which are based on the final pit design and production schedule. The pit optimization software considered grades and tonnages in the model along with estimated recoveries, mining and processing factors, and costs to determine what material could be economically extracted. Table 3 shows the parameters used for pit optimization. A conservative gold price was used to guide pit designs (US$840/oz).

**Table 3: Lerchs-Grossmann Pit Optimization Parameters**

| Item | Units | Cost |
|---|---|---:|
| Gold Price | US$/oz | $840.00 |
| Silver Price | US$/oz | $12.68 |
| Mining Cost Waste | US$/t mined | $1.45 |
| Mining Cost Ore | US$/t mined | $1.87 |
| Processing Cost | US$/t processed | $3.69 |
| G & A | US$/t processed | $0.71 |
| Royalty | % of recovered revenue | 3.4% |
| Recovery Gold | | 76% |
| Recovery Silver | | 39% |
| Interramp Slope Angle | | 45° to 50° |

Source: SRK, 2014

In order to report Proven and Probable Reserves an engineered pit was developed which takes into account planned mining access and extraction considerations. The reserves and production that reports to the leach pad was constrained by the size of the private parcel on which the permitted pad is to be constructed, and significant Indicated Resources that would normally have been categorized as Proven and Probable were not placed into reserves or the economic model. An expansion of the pad onto Forest Service property adjacent to the planned pad is feasible in order to accommodate additional material. However, this leach pad expansion will require additional permitting.

The statement of Proven and Probable Reserves for Mt. Hamilton is presented in Table 4.

**Table 4: Mt. Hamilton Mineral Reserve Statement, SRK Consulting (U.S.), Inc. Ultimate Designed Pit Reserves at 0.006 oz/t Au CoG, August 14, 2014**

| Reserve Category | Tons | Au Grade | Ag Grade | AuEq Grade | | Contained Ounces (thousands of oz) | |
|---|---:|---:|---:|---:|---:|---:|---:|
| | (000's) | oz/t | oz/t | oz/t | g/tonne | Au | Ag |
| Proven | 1,240 | 0.029 | 0.198 | 0.031 | 1.060 | 36.6 | 245.8 |
| Probable | 21,260 | 0.024 | 0.198 | 0.025 | 0.870 | 508.8 | 4,213.8 |
| **Proven and Probable** | **22,500** | **0.024** | **0.198** | **0.026** | **0.880** | **545.4** | **4,459.6** |
| Total Waste | 63,319 | | | | | | |

Source: SRK, 2014

- Reserves are reported using a CoG of 0.006 oz/t Au;
- The CoG was based on a gold price of US$1,300/oz and a silver price of US$20/oz;
- The CoG was calculated at an average recovery of 76% for Au and 39% for Ag;
- Average recovery for gold was calculated from a recovered grade item modeled for each model block based on cyanide soluble and total gold grades;
- Metal grades reported are diluted; and
- Some numbers may not add due to rounding.

Mineral Reserves stated above are contained within and are not additional to the Mineral Resources stated in this report.

To address dilution, SRK used software to construct an expanded dilution envelope outboard of original mineralized gradeshell to simulate predicted “over-mining.” In broad, thick areas of the deposit (e.g. Centennial), the dilution envelope was built to a fixed radius 8 ft outboard of the interpreted mineralization. The distance of 8 ft represents one half of the bucket width of the shovel/loader proposed for this operation. In areas of tabular, shallowly-dipping mineralization (e.g. Seligman), dilution was modeled with a thin vertical and larger radial envelope (2 ft vertical, 6 ft radial). These thin tabular zones are planned to be developed on smaller benches than the rest of the deposit, allowing for more selectivity. The volume between the mineralization boundary and the expanded halo was assigned a zero grade for estimating dilution.

In accordance with the CIM classification system only Measured and Indicated resource categories were converted to reserves (through inclusion within the open-pit mining limits). In this Mineral Reserve statement the mined Inferred mineral resource is reported as waste. Inferred resources, while not convertible to reserves, will be extracted during the mining of Proven and Probable reserves, and constitute “non-reserve material” that will likely add incremental ounces to the life of mine production.

# Development and Operations

**Mining**

Oxide mineralization at Mt. Hamilton is close to the surface and the resource lends itself to an open pit mining method. The mine design consists of two main pits with the approximate dimensions of 1,900 ft wide by 2,600 ft long by 800 ft deep; with a volume of 36 Myd$^3$. The pit designs were segregated into at least three phases each for production scheduling with 90 ft wide ramps (including berm) at a maximum in-pit road grade of 10%. Mining operations at Mt. Hamilton have a stripping ratio 2.5:1, waste to ore, with mining taking place on the side of a hill at an approximate elevation of 9,000 ft amsl. Ore will be hauled from the pits to a primary crusher located on the southwest rim of the Centennial pit or stockpiled near the crusher for later use. Waste rock will be placed as valley fill in Cabin Gulch, a centrally located valley between Centennial and Seligman. The final waste rock storage facility will be regraded to 2.5 H/1V per State of Nevada regulations for reclamation.

The mine life is estimated to be seven years with an additional nine months of pit pre-stripping. The life-of-Mine (LoM) average mining rate is estimated at 3.5 Mt/y ore (10,000 t/d) and approximately 8.5 Mt/y waste.

Open pit mining will be by conventional diesel-powered equipment, utilizing a combination of blasthole drills, hydraulic shovel, rubber-tired wheel loaders and off-highway 100 t trucks. Support equipment composed of graders, track dozers, and a water truck will aid in the mining of the Mineral Reserve and waste.

The mine is scheduled to initially operate on two ten-hour shifts per day, seven days per week, 350 days per year. During Year 1 the mine will shift to two 12-hour shifts per day, seven days per week. To match the slowdown in waste production, the third quarter of Year 6 is reduced to a single 10 hr shift, seven days a week and will be maintained for the remainder of production. Operating efficiency was estimated to be 83% (50 minutes/hour) and mechanical availability estimated at 85%.

Mining operations will require four crews operating on rotating shifts. There are several rotating shift schedules. The most widely used schedule in Nevada is based on a 28 day rotation. Because of the distance from the towns of Ely or Eureka, the crews will be transported to the site in company supplied vans.

Mining crew manpower during the peak production years will include 60 hourly equipment operators and 12 salaried personnel for a total of 72 full-time employees at the mine. In addition, two contract personnel will work on an as-needed basis for blasthole loading.

Tables 5 and 6 list the mining equipment planned to support the project. This equipment fleet was the basis for the mining capital cost estimate.

**Table 5: Primary Mining Equipment List**

| Equipment Type | Description | Size | Max Number Required |
|---|---|---|---|
| Atlas Copco DM45 | Blast Drill Rig | 540hp, 5 inch to 9 inch hole diameter, up to 175 ft hole depth, 45,000 ft-lb pulldown | 1 |
| Atlas Copco T45 | Blast Drill Rig | 325hp, 3½ inch to 5 inch hole diameter, up to 92 ft hole depth, with a 41 hp rock drill | 1 |
| Caterpillar 6030FS | Hydraulic Shovel | 1,530 hp, 19.6 $yd^3$ | 1 |
| Caterpillar 992K | Wheel Loader | 814 hp, 15 $yd^3$ | 1 |
| Caterpillar 777G | Haul Truck | 1,025 hp, 99.6 t payload | 7 |

Source: SRK, 2014

**Table 6: Support Mining Equipment List**

| Equipment Type | Description | Size/Comment | Max Number Required |
|---|---|---|---|
| Contractor Supplied | ANFO loading truck | | 1 |
| Caterpillar 14M | Motor Grader | 259 hp,14 ft blade | 2 |
| Cat D9T | Bulldozer | 410 hp, 107,000 lb, SEMI-U Blade | 1 |
| Cat D10T | Bulldozer | 580 hp, 155,500 lb, U-blade | 1 |
| Caterpillar 740B | Water Truck | 474 hp, 8,000 gal | 1 |
| Manufacturer TBD | Fuel/Lube Truck | 33,000 lb 6x4 | 1 |
| Manufacturer TBD | Mechanics Truck | 33,000 lb 6x4 | 2 |
| Manufacturer TBD | Light Plant | 30 ft mast | 6 |

Source: SRK, 2014

## Processing

Recovery of gold and silver from the Centennial Project will be performed by heap leaching and conventional ADR carbon-in-column processing. The dedicated heap leach pad (leach pad), process ponds and ancillary facilities were designed to accommodate a leachable reserve of approximately 22.5 Mt of crushed ore from the mine at a rate of 10,000 t/d. All of the process components of the operation are designed on private land (patented claims) controlled by MH-LLC with permit approval for operations by Forest Service and the State of Nevada, Division of Environmental Protection anticipated in the third quarter (Q3) of 2014.

Mined ore will be primary crushed near the open pit to minus 4 inch and conveyed to an ore pass. The ore pass will drop the ore vertically approximately 415 ft where it will be loaded on a conveyor in a 4,425 ft long adit. From the loading point at the base of the ore pass, the drift and conveyor have a

-15% grade to the portal. Once out of the adit, the ore will be belt transferred to a coarse ore stockpile. A reclaim tunnel under the coarse ore stockpile will feed a secondary crusher where the ore will be crushed to 90% passing 3/4 inch and conveyed and stacked on the leach pad with a radial stacker. A summary of heap leach pad design parameters is presented in Table 7.

**Table 7: Summary of Heap Leach Pad Operations Design Parameters**

| Design Parameter | Feasibility Design |
|---|---|
| Ore stacking rate | 625 t/h |
| Crushed Ore Bulk Density | 110 lb/ft$^2$ |
| Ore lift height | 30 ft |
| Solution application rate | 0.004 gpm/ft$^2$ |
| Ore leach cycle | 210 days |
| Ore leach area | 4.43 Mft$^2$ |
| Solution pumping rate | 3,240 gpm |
| HLP base slope | 17% upper (east), 13% lower pad (west) |
| HLP max design height | 210 ft above base |

Source: SRK, 2014

The proposed heap leach pad and process plant and facilities will have an approximate footprint area of 134 ac. Including the incline portal, secondary crusher pad, and heap leach pad, construction and operation will occupy virtually the entire area of the private parcel upon which it is located. The heap leach pad will be located on moderately sloping and generally uniform topography southwest of the pit in the valley. The leach pad will be roughly square in plan at an average pre-construction elevation of 7,400 ft amsl. The HDPE-lined base receiving ore will range from approximately 13% upslope from the stability berm and toe pad to 17% at the eastern boundary of the heap leach pad. The leach pad will have a total lined area of 4.43 Mft$^2$, or approximately 102 ac. Underliner for the leach pad will be bentonite-amended soil or a local low-permeability native soil sourced locally. Overliner will be crushed ore. The stacked ore height will gradually increase as it progresses from west to east until reaching its apex, with a regraded maximum vertical height of approximately 210 ft above the prepared base.

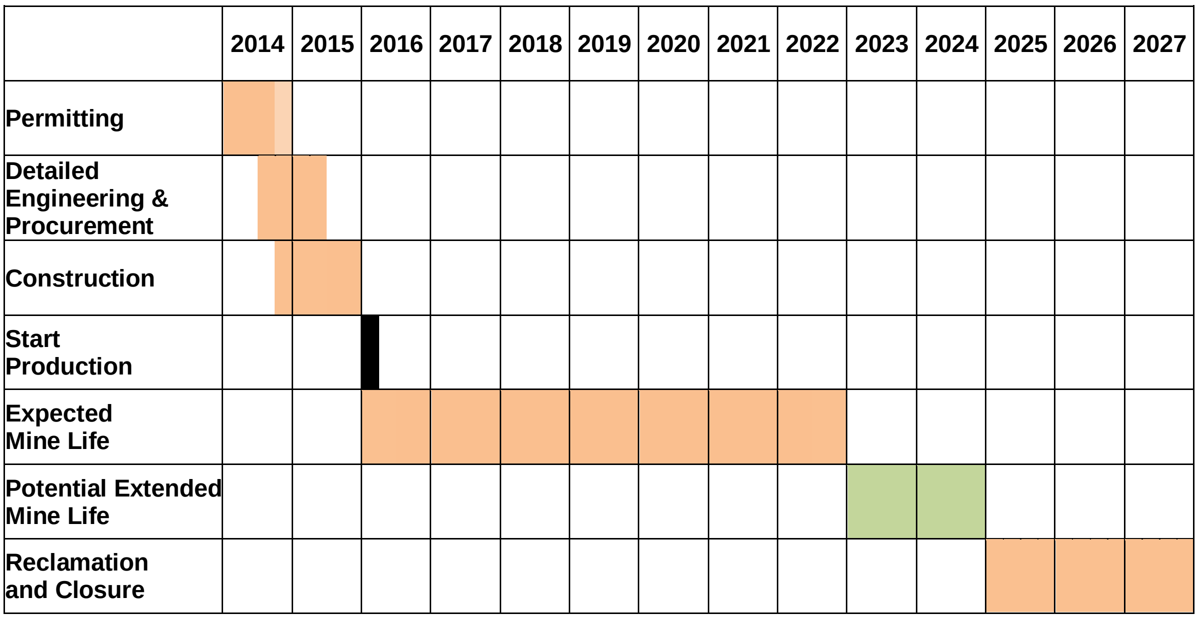
An expansion of the leach pad is planned to accommodate the Indicated resources not contained in the reserve and Indicated resources converted from Inferred by drilling to be conducted in 2015. The expansion of the leach pad will be constructed as a continuation of the pad to the south which has similar or more favorable geotechnical characteristics to the planned pad. The expansion will occur on Forest Service land and will require a permitting process to be initiated in 2015. The construction of the pad expansion is anticipated to begin in the fifth year of operations.

An ADR circuit will be used for processing. The ADR plant will be fed at the rate of 3,240 gpm by a submersible pump in the pregnant pond. The ADR plant consists of five, 12 ft diameter carbon columns, a 4.5 t strip and acid wash system, electrolytic cells, and an induction smelting furnace. The final product will be a doré bar. Electrolytic cells of the ADR plant have been sized to accommodate Ag:Au ratios of 6:1 in the final doré. The ADR plant will contain a mercury retort and all mercury control systems as currently required by the State of Nevada regulations.

Manpower for crushing, processing and analytical will include seven salaried and 50 hourly staff, for a total of 57 full-time employees supporting processing. Combined with the mining staff, the operation will require 129 full-time employees plus administration.

**Project Development Schedule**

The current schedule for project development is presented in Figure 1. Permitting for the operation is expected to be completed by the end of 2014. Detailed engineering, procurement and construction will also commence in late 2014, subject to financing, leading to commencement of gold production in Q1 2016. A projected seven year mine life will be followed by approximately three years of closure and reclamation. There is a potential, depending on future metal prices and operating costs, to process additional material that will be mined and stockpiled, but requires additional heap leach pad space. These roughly 50,000 AuEq ounces could extend the mine life by two additional years as shown in Figure 1.




Source: SRK, 2014

**Figure 1: Development Schedule for the Mt. Hamilton Project**

## Capital and Operating Costs

Capital costs used in the feasibility-level economic analysis for Mt. Hamilton were based heavily on vendor and specialist quotations. A total of 98% of mining, 94% of process, and 78% of owner and infrastructure capital costs are linked to vendor quotes. SRK has applied additional contingencies to these estimates for omissions. Similarly, operating costs, as driven by consumables or labor rates were supported by recent relevant vendor information or public domain mining services cost providers, typically InfoMine® Costmine™.

The capital cost summary for the Project is presented in Table 8. The initial capital requirement is projected at US$91.7 million, plus ongoing and closure costs for a total life-of-mine capital cost of US$121.6 million. MH-LLC intends to lease most of the major mining equipment. The lease costs were included in the mining cost calculation. A residual payment, due at the end of the lease period, and an initial payment were included in the capital costs. At present, contract mining is still being considered for the Project but is not reflected in any of the calculations presented in this report.

**Table 8: Summary of Capital Costs**

| Initial Capital Cost Item | Cost US$000's |
|---|---|
| Mining | 17,837 |
| Processing | 25,380 |
| Leach Pad | 7,401 |
| Owner and Infrastructure | 32,116 |
| Contingency | 9,011 |
| **Initial Capital Total** | **$91,745** |
| Sustaining Capital | 17,197 |
| Closure Costs | 8,815 |
| Contingency | 3,760 |
| **LoM Total Capital** | **$121,518** |

Source: SRK, 2014

The operating cost summary for the Project is presented in Table 9. These costs were built up from a zero cost basis. Mining costs were dictated by the equipment selected and the conditions of the mine environment. Infomine® CostMine™ data were used to determine equipment hourly costs and hourly wage rates. The equipment productivities were determined from published manufacturer's data and benchmarked against analog operations. These factors were treated in a conservative manner to reflect the difficulties of operating at over 9,000 ft elevation in rural Nevada.

Processing costs were developed from: 1) wage rates from similar projects in Nevada; 2) reagent consumption as determined by site-specific test programs or industry standards and current prices; and 3), wear and replacement parts by testing or manufactures recommendations. The process staffing plan allows for the climatic conditions and the separation between the laboratory and the processing plant.

The supervisory and administrative support staff was sized to efficiently handle the administrative, technical and management functions required for the proposed operation. Provisions for training, and regulatory mandated safety functions were also included.

**Table 9: Summary of Operating Costs**

| Operating Costs | (US$000) | US$/t-ore |
|---|---|---|
| Mining | $134,740 | $5.99 |
| Processing | $92,427 | $4.11 |
| G&A | $18,863 | $0.84 |
| **Total Operating** | **$246,029** | **$10.93** |

Source: SRK, 2014

Key assumptions for developing the operating costs are provided in Table 10. The operation will switch from generated power to line power at the end of the second year of operations. The diesel fuel cost of US$3.20 per gallon (gal) was based on an August, 2014 delivered-cost quotation from an Elko, Nevada supplier.

**Table 10: Operating Cost Assumptions**

| Cost Item | Amount | Unit |
|---|---|---|
| Power Cost – Energy (line) | $0.0487 (1) | US$/kWh |
| Power Cost – Demand | $8.00 | US$/kW |
| Power Cost – Energy (generated) | $0.252 | US$/kWh |
| Diesel Cost | $3.20 | US$/gal |
| Lube Cost | $10.33 | US$/gal |
| Prill Cost | $550.00 | US$/t |
| Sales tax rate | 7.725% | |

Source: SRK, 2014
(1) Does not include peak demand.

# Conclusions and Recommendations

The purpose of the 2014 FS was to collect and analyze sufficient data to reduce or eliminate risk in the technical components of the project and to refine economic projections based on current cost data. SRK offers the following conclusions for key components of the proposed mining operation at Mt. Hamilton following the addition of the adjacent Seligman reserves.

## Geology, Drilling and Exploration Data Quality

The geological and drilling database for the Mt. Hamilton property is robust, and recent drilling (2008-2012) carried out by MH-LLC, involving modern quality controls, has validated historic drilling in areas where new and old drilling overlap. Hole location and survey risk is considered very low as most drill sites can be confirmed using current and pre-mining aerial photography and topography. MH-LLC has a well-organized core storage and sample preparation facility in Ely, Nevada and follows industry standard protocols for material handling and documentation.

There is still a large dependence on historic data in parts of Seligman that were collected before current quality controls were in place. Infill drilling has improved the overall quality of the assay database by adding a higher proportion of validated samples to the total. Additional infill drilling will continue to improve data quality and also fill gaps in the cyanide-soluble data set. Installation of the proposed conveyor incline and ore pass will expose new geology, which will improve the geologic model and could define new drill targets accessible from the new conveyor incline.

**Recommendations:**

- Continue to compile and review assay results from future drilling converting resources to reserves as the results are received to improve batch quality when the analytical program is still active;
- Include a second split from a minimum of 5% of the coarse reject samples to verify the adequacy of crush size for assay repeatability;
- Randomly select roughly 5% of pulp samples from future drilling for check assay at a second independent laboratory, for all parameters used in resource estimation; and
- Map and sample the conveyor incline during development.

## Mineral Resources

From an exploration perspective, additional infill drilling could upgrade resource classifications to make more gold ounces eligible for reserves at price assumptions utilized in the current reserve statement. Inferred mineralization within the resource pit between the Seligman and Centennial deposits has strong prospect for upgrading.

The quality of the historic data used in the resource estimate has been verified by recent drilling and confirmed by an analysis of quality control data by SRK. Resources in the 2014 Mineral Resource Statement reflect a refinement of tonnage and grade estimates that used updated density and infill drilling results for Seligman and Centennial. Measured and Indicated mineral resources for the combined Mt. Hamilton gold-silver deposit are reported at 828,000 AuEq ounces with an additional Inferred mineral resource of 136,000 AuEq ounces. These resources are contained within an open pit mining configuration (resource pit) driven by US$1,300/oz gold and US$19.60/oz silver values.

There are more than 230,000 oz of in situ gold modeled outside the resource pit that are not categorized at this time, and not reportable as NI 43-101 compliant resources due to current economics. The majority of the uncategorized material is down-dip to the east and into the hill slope requiring an increasing proportion of stripping to access mineralization. Higher metal prices would convert some of this material into reportable resources where drill density is sufficient.

Exploration potential outside of the planned operational area has been demonstrated in surface soil gold anomalies located mostly east of Seligman and south of Centennial. Principal targets include Chester/Wheeler Ridge, U4, Five Way and White Pine. Sparse or historic drilling in other exploration areas may have missed additional resources, which might be sterilized by the current mine design (e.g. Five Way, Cabin Gulch). Near-term condemnation drilling should address this possibility. Future exploration should also consider sulfide-hosted gold/silver as well as other commodities (Mo, W, Cu) that may be economic in a milling scenario.

**Recommendations:**

- Targeted infill drilling to characterize material in expanded pits and to upgrade resources from Inferred to Indicated classification and confirm continuity in narrow mineralized zones;
- Exploration drilling to test the large and strong Wheeler Ridge gold-in-soil anomaly south of the Centennial resource;
- Continue to build the multi-element database to get spatial distribution of base and transition metals;
- Improve geologic logging methods to capture material properties that affect rock mechanics and metallurgy for future feasibility analysis;
- Detailed stratigraphic/structural geology modeling (from historic mapping data) to identify step-out exploration targets that could add to the resource;

**Mineral Reserves and Mining**

A conventional truck and shovel operation is proposed for operations at a mining rate of 10,000 t/d ore. Only Measured and Indicated resources were converted to reserves using US$840/oz gold and US$12.68/oz silver pricing along with conservative operating cost assumptions. Recovery and dilution were addressed in the definition of ore. The assumption of low metal prices in the reserve model mitigates substantial down-side price risk while providing high-quality ore to the engineered leach pad, which has a private-property limited capacity of 22.5 Mt of ore.

Dedicated oriented core drilling and geotechnical characterization of the rock mass has been applied to reserves. SRK's analysis of the geotechnical data supports an overall pit slope of 50°. Flatter slopes, which include ramps, were designed on the west side of the open pit.

The mining production schedule was built around detailed phase designs that include full mining equipment access. The designs contain detailed haulage profiles used to determine haulage costs.

Mining on 10 and 20 ft benches, triple benched to 60 ft using a hydraulic shovel allows for selectivity in tabular ore. Drilling and blasting on 10 ft benches in ore and the use of a wheeled loader will aid mining precision in thinner ore zones. Oxide ore is visibly distinguishable from un-oxidized waste, and in most cases this will improve grade control efficiency.

All previous drilling at Centennial and mining in the adjacent NE Seligman mine indicate that groundwater greatly exceeds the depth of proposed mining. Therefore, the proposed open pit will be dry and will require no provisions for dewatering.

SRK has proposed a design for ore delivery that accommodates winter operating conditions at high elevations. The predominantly underground ore-flow system will protect conveyors and should require less maintenance with less weather-related down-time. Although some geotechnical work has been completed regarding the adit and ore pass, there remain some uncertainties in the ore-flow system related to the geotechnical characterization of the proposed adit and ore-pass chamber. Ideally, both of these excavations would have received a complete geotechnical evaluation at feasibility level based on pilot-hole drilling; however, permitting and seasonal limitations have precluded this assessment. To mitigate the uncertainty, SRK, based on outside underground subcontractor pricing, applied heavy contingencies for ground support, which added costs to the planned underground development. This was deemed necessary in the absence of geotechnical supporting data.

Other components of the ore flow system, including the conveyor and stacker array are well understood, vendor quoted, and considered to be of low risk for consistent ore delivery. Excavation and construction for the underground ore-flow system are scheduled to begin in Q4, 2014, pending financing by the Company.

**Recommendations:**

- Additional oriented geotechnical diamond core drilling in the extreme southernmost Seligman; and
- Improve geologic and geotechnical engineering confidence for the ore pass and conveyor incline using oriented core drilling to better predict and cost ground support requirements.

**Metallurgy and Processing**

Overall, the results of the 2013 Seligman-focused metallurgical testing of oxide ores were comparable, if not more favorable than previous results for Centennial. One of the key findings from the drilling and testing of the Seligman and North Centennial ores was the favorable leach profile of Seligman igneous oxide, which had been largely overlooked by previous operators.

Metallurgical characterization is at feasibility level for all of the drilled or re-drilled parts of Centennial and North Seligman, leaving only extreme south Seligman needing further test work. Metallurgical risk for this area is considered low. Column test work on the oxide ores of both the Centennial and Seligman deposits demonstrates recovery of 79% to 80%. Sulfidic ores were also evaluated and found to be refractory in carbon-in-column processing. Testing showed that transitional ores were economically feasible to process in some cases. Cyanide soluble assay techniques have been shown to be effective to readily identify economic ore from waste in transitional ore. The projected average overall gold recovery of 76% is a result of the inclusion of some economic transitional ore in the mine plan. Modelled gold recovery based on paired cyanide soluble and fire assays provides a high degree of detail in characterization of expected operational recoveries in comparison to assigned recoveries based solely on observed oxidation of the ore.

There could be economic benefit to additional comminution and hydraulic conductivity testing on Seligman igneous material. Additional comminution testing on igneous material may show that less work is needed to crush igneous than skarn material. The current assumption is that all material will crush as skarn.

The recent 2014 detailed design work and contractual cost arrangements for the ADR plant by Kappes Cassiday and Associates (KCA) has improved confidence in cost estimates related to plant construction and operation. The strip rate of the plant was designed for 4.5 t of carbon to accommodate a throughput of 10,000 t/d. The crushing circuit planned can accommodate this tonnage, with variable belt speeds to match ore delivery rates.

Remodeling and rescheduling the reserves in 2014 largely removed concerns about overloading silver in the process circuit, but there are still phases in the production schedule when Ag:Au ratio should be monitored. In situations where the ratio is high, it can be remedied by blending stockpiled ore and/or stripping the carbon more frequently.

The current plan for leach pad underliner is to amend soils in place. There are less expensive options for underliner from known local clay borrow sources that should be investigated to reduce costs.

The selected processing methodology is considered low risk. The ADR carbon-in-column method for gold and silver recovery is proven technology and widely used in analogous operations in Nevada.

Power will be initially supplied at the mine and ADR by generators. The production water supply has been defined and water rights sufficient for project start-up have been secured by MH-LLC. This 2014 FS used the existing Seligman well as the primary source for production water, but further hydrogeologic exploration is planned to locate a source closer to the planned leach operation to reduce costs.

There is no tailings risk associated with this processing plan as no tailings will be generated. Spent ore will remain on containment (HDPE liner) after leaching and the facility will be reclaimed in place during closure.

**Recommendations:**

- Additional comminution and hydraulic conductivity testing on igneous material;
- Additional metallurgical characterization in conjunction with reserve drilling at South Seligman; and
- Further investigate local clay borrow source for leach pad underliner.

**Infrastructure**

Power and water are the key elements of the project infrastructure. Both systems are at feasibility level for design and costing. There are opportunities to upgrade both systems and these have been built into the economic evaluation. In year three of operations, MH-LLC expects to convert from generated power to line power reducing unit costs from US$0.25/kWh to US$0.05/kWh. With such a change in costs, there is both a risk and an opportunity related to power costs depending on the timing of the installation compared to plan.

Water supply costs are currently based on the existing Seligman water well as the primary source. MH-LLC plans to install a new well, about 1.5 miles closer to the process plant with lower pumping costs and piping risk. The new well(s) will likely become the primary water supply for operations.

**Recommendations:**

- Develop a second water supply well to supply up to 500 gpm during peak construction and operation. The current plan to install a new water supply at the Admin Parcel near the process plant is considered a top priority as this could become the primary water supply for the operation, securing availability during peak demand.

**Environmental Studies and Permitting**

Permits for activity on private land have been submitted to the appropriate State agencies for review. A Water Pollution Control Permit has been issued by the State of Nevada including a recently updated Waste Rock Management Plan. The approved method of waste rock placement is blending which requires no special segregation of ore by geochemical character.

Air Quality Permit applications have been submitted to the state and approval of a permit to construct is expected in Q4 2014. Separate Reclamation Permit applications for the mine and processing facilities have been submitted to the State of Nevada. These are in the final stages of review and approval.

The Mine Plan of Operations (MPO) for activity on public land (USFS) has been submitted to the Forest Service. The Decision Notice Objection Period has ended and final signing of the MPO will occur when initial bonding has been posted. A Right of Way grant has been issued by the BLM for access to Forest Service land and private land where mining and processing will occur.

Water quality sampling from existing monitoring wells is ongoing and is reported to the state. Water supply capacity was confirmed by pump testing in 2013. Water rights for the operation have been secured.

The Project has several characteristics that are favorable for permitting including: 1) No anticipated pit lake; 2) Acid neutralizing waste rock; 3) Deep groundwater beneath the proposed leach pad; and 4) Process components operated and closed on private land.

The mine closure cost without contingency as calculated by SRK using the Standard Reclamation Cost Estimator is US$8.8 million.

**Recommendations:**

- Permitting is advanced and no further recommendations apply.

**Projected Economic Outcomes**

The additional metal brought into reserves by the 2011-2012 exploration drilling, geotechnical and metallurgical test work has helped to offset fixed capital requirements and has improved project economics in the 2014 FS compared to the 2012 FS. Metal prices of US$1,300/oz gold and US$20.00/oz silver were applied to the 2014 economic evaluation. The economic model is constrained by the capacity of the engineered/permitted leach pad to 22.5 Mt. of ore. Production will have a 2.5:1 waste: ore stripping ratio (including stockpile) and a mining rate of 10,000 t/d ore, resulting in 545.4 koz contained gold and 4,459.6 koz of contained silver. Metal recoveries are projected at 76% and 39% for gold and silver, respectively.

The economic results, at a discount rate of 8%, indicate a Net Present Value of US$60.8 million with an IRR of 26.0% (after estimated taxes). Payback will be in 2.9 years from the start of production. Initial capital costs are projected at US$91.7 million with a total capital cost for the Project of US$121.5 million. The cash costs per gold-equivalent ounce recovered is US$558.

Economics of the Mt. Hamilton Project, as developed in this study, are fairly insensitive to commodity prices. This is because the pit was designed using a gold price of $840/oz. Metal prices have fallen over the last three years from near all-time highs in 2011. The current metal prices have slowed down production at neighboring Nevada mines and made available additional skilled labor to support the Mt. Hamilton operation.

## Recommendations

Work programs recommended to advance the Project include drilling, engineering designs and technical studies as follows:

Drilling:

- Resource conversion drilling (RC) (Inferred upgrade to Measured/Indicated outside of but adjacent to the ore within the current mine plan);
- Seligman south area resource/metallurgical confirmation RC and core drilling;
- Exploration drilling on proximal targets that, if successful, would utilize project infrastructure;
- Geotechnical drilling and analysis for underground development of the ore flow system; and
- Supplemental (closer to processing) water supply well drilling and piping design.

Engineering Designs:

- Staff engineer for detailed design project management;
- Detailed designs for underground reclaim chamber and infrastructure; and
- Construction-level designs on ancillary facilities.

Technical Studies:

- Seligman South metallurgical and geotechnical studies; and
- Finalize environmental permitting for the existing reserves and initiate permitting for a power line and leach pad expansion.

A total anticipated cost for advancement of the project during the Pre-Construction phase is US$2.9 million. The cost break-down for the work programs described above are presented in Table 11.

**Table 11: Recommended Pre-Construction Work Program Costs**

| Work Program | Estimated Cost US$ | Assumptions/Comments |
|---|---|---|
| Priority 1a and 1b resource/reserve conversion drilling (RC) | 400,000 | 31 holes for 9,000 ft @ US$45/ft |
| Priority 2 and 3 resource/reserve conversion drilling (RC) | 390,000 | 24 holes for 8,700 ft @ US$45/ft |
| Geotechnical drilling for underground development (DD) | 500,000 | 2,500 ft @ 200/ft incl. supervision |
| Relocate water supply well closer to processing | 350,000 | pump tests and pumps, design |
| **Total Drilling** | **1,640,000** | |
| Detailed design project management | 200,000 | salaried new hire or contract PM |
| Detailed design for underground reclaim chamber and infrastructure | 50,000 | specialist contractor/engineer |
| Detailed design for crushing, process and infrastructure and preliminary EPCM | 500,000 | specialist contractor/engineer |
| **Total Detailed Design** | **750,000** | |
| Seligman geotechnical analysis | 25,000 | consultant engineer |
| Environmental permitting | 150,000 | environmental contractor |
| **Total Technical Studies** | **175,000** | |
| **Sub Total** | **2,565,000** | |
| **Contingency @15%** | **384,750** | |
| **Total** | **2,949,750** | |

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# Appendices

# 1 Introduction (Item 2)

## 1.1 Terms of Reference and Purpose of the Report

This report was prepared as a Canadian National Instrument 43-101 (NI 43-101) Technical Report for Mt. Hamilton LLC (MH-LLC) a limited liability company owned by Solitario Exploration & Royalty Corp. (Solitario) and Ely Gold and Minerals Inc. (Ely Gold), by SRK Consulting (U.S.), Inc. (SRK). Within this report, MH-LLC may be construed as MH-LLC separately or collectively as MH-LLC, Solitario and Ely Gold. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by MH-LLC subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits MH-LLC to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with MH-LLC. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a newer Technical Report has been issued.

This report provides mineral resource and mineral reserve estimates, and a classification of resources and reserves in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM). It also meets the standards of the U.S. Securities and Exchange Commission Industry Guide 7 for estimating and reporting reserves.

The mineral properties addressed in this report are MH-LLC's wholly owned Centennial and Seligman gold and silver Projects ("Centennial" or "Seligman", respectively, or collectively referred to as "Mt. Hamilton" or the "Project"), located in the historic Mt. Hamilton mining district of central Nevada. This Technical Report represents feasibility-level mining, processing, cost estimation and economic evaluation for the Mt. Hamilton Mineral Reserves. A Feasibility Study document (SRK, 2014a) was produced in conjunction with this Technical Report and contains all recent and relevant data to support the summary descriptions and conclusions made herein.

## 1.2 Qualifications of Consultants (SRK)

The SRK Group comprises over 1,500 staff worldwide, offering expertise in a wide range of mineral resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This relationship permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated record of accomplishment in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, Technical Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs. Neither SRK nor any of its employees and associates employed in the preparation of this report has any beneficial interest in MH-LLC or in the assets of MH-LLC. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings. SRK will be paid a fee for this work in accordance with normal professional consulting practice.

This Technical Report has been prepared by a team of consultants sourced principally from SRK's Reno, Nevada office (the Consultants). These consultants are specialists in the fields of geology, hydrogeology, geochemistry, Mineral Resource and Mineral Reserve estimation and classification, open pit mining, underground mining, geotechnical, environmental, permitting, mineral processing and mineral economics disciplines.

The SRK personnel involved with the Project, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. Listed below are the QPs who have provided input to this Technical Report and the Sections for which they are responsible:

- J. Pennington, (SRK) C.P.G., MSc. is responsible for introduction, geology, and the Mineral Resource Estimate; Sections 1, 2 except for 2.5, 3, 4,12, 21, 22, 23, 24, 25, 26 and portions of the Summary, Sections 23 and 24 summarized therefrom, of this Technical Report;
- Brooke Miller (SRK) C.P.G., MSc. is responsible for geology and resources Sections 2, 3, 4, 5, 6, 7, 8, 9, 10 and portions of the Summary, Sections 23 and 24 summarized therefrom, of this Technical Report;
- Richard DeLong (Enviroscientists), MS, PG, RG, CEM is responsible for environmental, permitting and community impact Sections 2.5, 18 and portions of the Summary, Sections 23 and 24 summarized therefrom, of this Technical Report;
- Herbert Osborne (SRK), RM-SME, is responsible for crushing, conveying and stacking Section 15.2.
- Chris Sheerin, (SRK), RM-SME, MSc. is responsible for process, metallurgical testing and recovery Sections: 11, 15 except for 15.2 and 15.3, and portions of the Summary, Sections 23 and 24 summarized therefrom, of this Technical Report;
- Kent Hartley (SRK) P.E. Mining, BSc is responsible for mineral reserves, mining methods, infrastructure, market studies, capital and operating costs and economic analysis Sections 13, 14 except 14.4, 16, 17, 19, 20;
- Mike Levy (SRK), P.E, P.G. is responsible for the pit slope geotechnical Section: 14.4; and
- Evan Nikirk (SRK), P.E., MSc. is responsible for the heap leach pad design Section 15.3.

Other contributing authors:

- Walt Hunt, (Solitario) (Sections: 2.2,2.3, 2.4); and
- Justin Smith, (SRK) BSc. (Section: 13, 14).

### 1.2.1 Details of Inspection

MH-LLC has hosted several site visits to the Property over the last six years of SRK project involvement, including most recently a QP visit on April 25, 2014. The site visit was conducted to review drill core and chips, drilling, logging and sampling procedures in MH-LLC's core storage facility in Ely, Nevada, as well as a visit to the project site at Mt. Hamilton to review the proposed pit area, waste-rock storage areas the future potential leach pad site. Table 1.2.1.1 lists the site visit participants.

**Table 1.2.1.1: SRK Site Visit Participants**

| Personnel | SRK Office | Expertise | Date(s) of Visit |
|---|---|---|---|
| J. Pennington | Reno | Geology, Resources | June 29, 2011 |
| Brooke Miller | Reno | Geology, Resources | October 24, 2011 |
| Kent Hartley | Reno | Mining & Economics | April 25, 2014 |
| Evan Nikirk | Reno | Civil Geotechnical | June 29, 2011 |
| Amy Prestia | Reno | Geochemistry | September 21, 2009 |

## 1.3 Reliance on Other Experts (Item 3)

SRK's opinion contained herein is based on information provided to SRK by MH-LLC throughout the course of the investigations. SRK has relied upon the work of other consultants in the project areas in support of this Technical Report. The sources of information include data and reports supplied by MH-LLC personnel as well as documents referenced in Section 25.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

### 1.3.1 Sources of Information and Extent of Reliance

SRK relied on others for the following information in the referenced sections:

- MH-LLC for Land Tenure and Permit Status and Agreements Status: Section 2;
- Enviroscientists: Section 18;
- McClelland Laboratories (McClelland): Recent relevant test results for Section 11; and
- Kappes Cassiday and Associates (KCA): adsorption-desorption-recovery (ADR) plant design and specifications.

The items pertaining to land tenure and partnerships have not been independently reviewed by SRK and SRK did not seek an independent legal opinion of these items.

## 1.4 Effective Date

The effective date of this report is August 14, 2014. This represents the date in which Mineral Reserves were restated for the Project.

## 1.5 Units of Measure

The data described in this report are generally expressed as US units of measure: miles, feet, for the land/legal subdivision, etc., as these are the common units of measure in the United States. All currency references are US dollars (US$) unless specified otherwise. Tons are reported in short tons (t) equal to 2,000 lb unless specified as a metric tonne (tonne) equal to 1,000 kg, or 2,204.6 lb. Unless otherwise specified, values are expressed in ounces per short ton (oz/t) for drill hole assay and resource gold (Au) and silver (Ag) values. Drill hole coordinates may be listed with both truncated Nevada East State Plane coordinates (feet) and UTM coordinates (meters).

# 2 Property Description and Location (Item 4)

## 2.1 Property Description and Location

The Mt. Hamilton Property (Property), which contains the Centennial and Seligman gold and silver deposits, is located in White Pine County, Nevada at 115.558890° W Longitude and 39.250867° N Latitude. The project area is in Township 16 North, Range 57 East. Within that area, the planned mine sites are in Sections 15,16, 21 and 22), planned waste rock storage in Sections 16 and 17, and the proposed heap leach facility in Section 20. The project site is on the western flank of Mount Hamilton, which is on the north end of the White Pine Mountains. The property lies about 10 miles south of U.S. Highway 50 via White Pine County Road 5 and about 60 miles west of Ely, Nevada via U.S. Highway 50. The nearby communities, Ely and Eureka, are approximately equidistant from the project site. From either community, the project site can be accessed by car, on paved and gravel-surface roads, in about an hour. The Project location is shown in Figure 2.1.1.

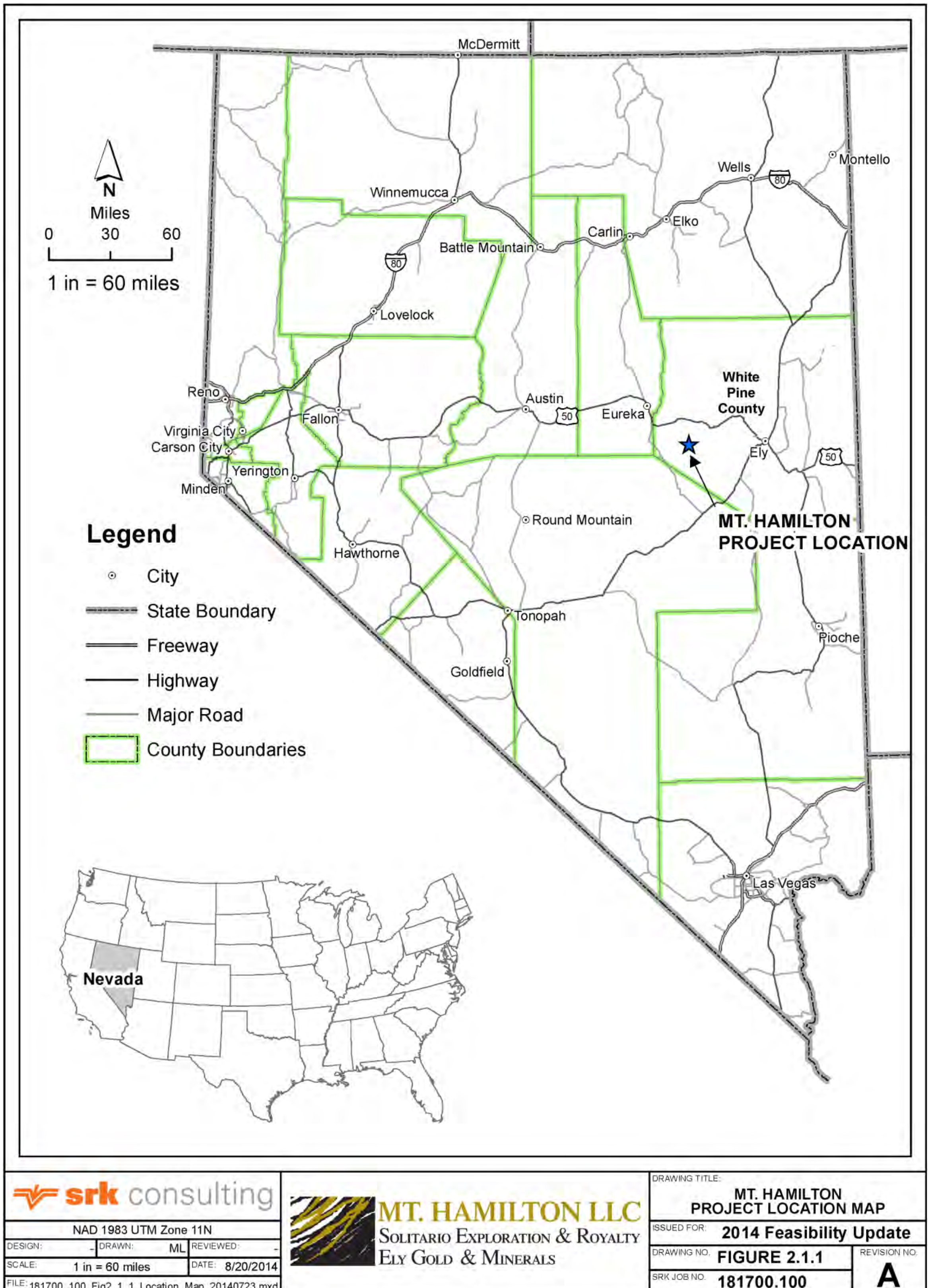
## 2.2 Mineral Titles

The MH-LLC land position includes private land and unpatented mining claims on federal land and controls the Property through direct ownership and through lease option agreements. The Property is comprised of two parcels of fee simple land totaling 240 ac, nine surveyed Patented Mineral Claims (Table 2.2.1), totaling 120.57 ac, and 255 unpatented Federal mining claims (Table 2.2.2), totaling approximately 4,530 ac. Claims are located in Sections 8, 9, 15, 16, 17, 21, 22, 27, 28, Township 16N, Range 57E, White Pine County, Nevada (Figure 2.2.1). All unpatented claims are staked on the ground in accordance with Bureau of Land Management and Nevada regulations. The lands which comprise the unpatented mining claims are controlled by the US Mining Law of 1872 and are situated on Public Lands administered by the U.S Department of Agriculture, Forest Service. The patented claims and the two fee simple parcels are private lands for which MH-LLC controls all surface and mineral rights.

**Table 2.2.1: Private Land Parcels for Ely Gold Mt. Hamilton Property**

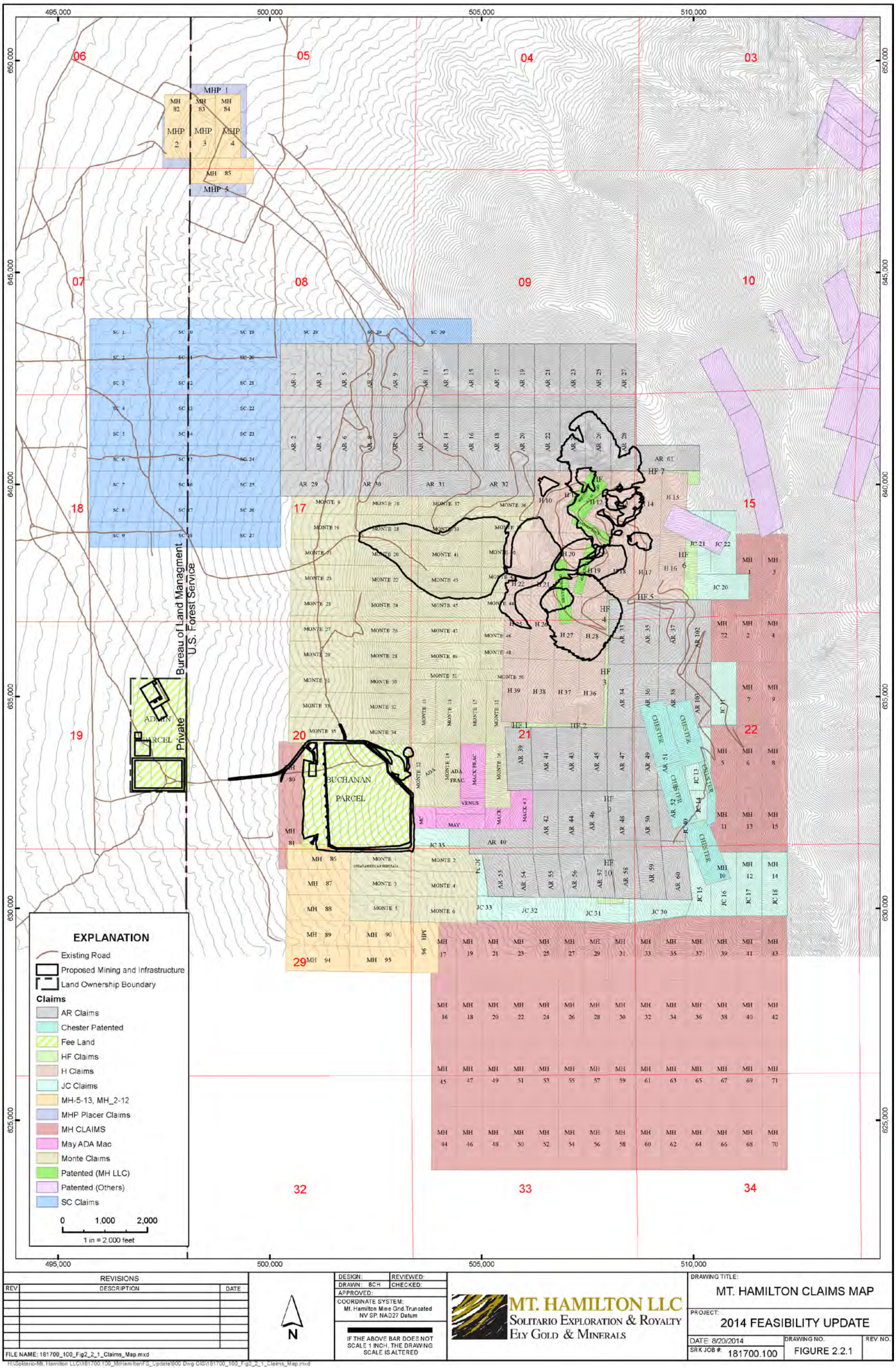
| Parcel # | US Mineral Survey # | Name | Date Issued | Acreage |
|---|---|---|---|---|
| 09-400-07 | n/a | Henkle-Buchanan | n/a | 160 |
| 09-400-06 | n/a | Admin | n/a | 80 |
| | | | **Total Fee Land** | **240** |
| 99-059-05 | 69 | Badger state | 09/15/1882 | 4.59 |
| 99-059-25 | 66 | Centennial | 05/31/1881 | 9.54 |
| 99-059-66 | 41 | Gloucester | 04/15/1874 | 5.51 |
| 99-060-81 | 68 | Woo Hop | 02/28/1882 | 11.48 |
| 99-059-27 | 42 | Chester | 12/21/1874 | 6.89 |
| 99-059-28 | 3763 | Chester #1 | 04/11/1912 | 82.56 |
| 99-059-29 | | Chester #2 | | |
| 99-059-30 | | Chester #3 | | |
| 99-059-31 | | Chester #4 | | |
| | | | **Total Patented Claims** | **120.57** |

Source: SRK, 2014




Source: SRK, 2014

**Figure 2.1.1: Mt. Hamilton Site Location Map**




Source: SRK, 2014

**Figure 2.2.1: Mt. Hamilton Claims Map**

**Table 2.2.2: Federal Mining Claim List for Mt. Hamilton LLC Property**

| Claim Name | BLM NMC # | Location Date | Claim Name | BLM NMC # | Location Date |
|---|---|---|---|---|---|
| AR 1 | 899951 | 02-Jun-05 | AR 51 | 896944 | 05-Apr-05 |
| AR 2 | 899952 | 02-Jun-05 | AR 52 | 896945 | 05-Apr-05 |
| AR 3 | 899953 | 02-Jun-05 | AR 57 | 933806 | 01-Sep-06 |
| AR 4 | 899954 | 02-Jun-05 | AR 58 | 896951 | 05-Apr-05 |
| AR 5 | 899955 | 02-Jun-05 | AR 59 | 896952 | 05-Apr-05 |
| AR 6 | 899956 | 02-Jun-05 | AR 60 | 896953 | 05-Apr-05 |
| AR 7 | 899957 | 02-Jun-05 | AR 61 | 899983 | 02-Jun-05 |
| AR 8 | 899958 | 02-Jun-05 | SC 1 | 1005079 | 23-Feb-09 |
| AR 9 | 899959 | 02-Jun-05 | SC 2 | 1005080 | 23-Feb-09 |
| AR 10 | 899960 | 02-Jun-05 | SC 3 | 1005081 | 23-Feb-09 |
| AR 11 | 899961 | 02-Jun-05 | SC 4 | 1005082 | 23-Feb-09 |
| AR 12 | 899962 | 02-Jun-05 | SC 5 | 1005083 | 23-Feb-09 |
| AR 13 | 899963 | 02-Jun-05 | SC 6 | 1005084 | 23-Feb-09 |
| AR 14 | 899964 | 02-Jun-05 | SC 7 | 1005085 | 23-Feb-09 |
| AR 15 | 899965 | 02-Jun-05 | SC 8 | 1005086 | 23-Feb-09 |
| AR 16 | 899966 | 02-Jun-05 | SC 9 | 1005087 | 23-Feb-09 |
| AR 17 | 899967 | 02-Jun-05 | SC 10 | 1005088 | 23-Feb-09 |
| AR 18 | 899968 | 02-Jun-05 | SC 11 | 1005089 | 23-Feb-09 |
| AR 19 | 899969 | 02-Jun-05 | SC 12 | 1005090 | 23-Feb-09 |
| AR 20 | 899970 | 02-Jun-05 | SC 13 | 1005091 | 23-Feb-09 |
| AR 21 | 899971 | 02-Jun-05 | SC 14 | 1005092 | 23-Feb-09 |
| AR 22 | 899972 | 02-Jun-05 | SC 15 | 1005093 | 23-Feb-09 |
| AR 23 | 899973 | 02-Jun-05 | SC 16 | 1005094 | 23-Feb-09 |
| AR 24 | 899974 | 02-Jun-05 | SC 17 | 1005095 | 23-Feb-09 |
| AR 25 | 899975 | 02-Jun-05 | SC 18 | 1005096 | 23-Feb-09 |
| AR 26 | 899976 | 02-Jun-05 | SC 19 | 1005097 | 23-Feb-09 |
| AR 27 | 899977 | 02-Jun-05 | SC 20 | 1005098 | 23-Feb-09 |
| AR 28 | 899978 | 02-Jun-05 | SC 21 | 1005099 | 23-Feb-09 |
| AR 29 | 899979 | 02-Jun-05 | SC 22 | 1005100 | 23-Feb-09 |
| AR 30 | 899980 | 02-Jun-05 | SC 23 | 1005101 | 23-Feb-09 |
| AR 31 | 899981 | 02-Jun-05 | SC 24 | 1005102 | 23-Feb-09 |
| AR 32 | 899982 | 02-Jun-05 | SC 25 | 1005103 | 23-Feb-09 |
| AR 33 | 896926 | 05-Apr-05 | SC 26 | 1005104 | 23-Feb-09 |
| AR 34 | 896927 | 05-Apr-05 | SC 27 | 1005105 | 23-Feb-09 |
| AR 35 | 896928 | 05-Apr-05 | SC 28 | 1005106 | 23-Feb-09 |
| AR 36 | 896929 | 05-Apr-05 | SC 29 | 1005107 | 23-Feb-09 |
| AR 37 | 896930 | 05-Apr-05 | SC 30 | 1005108 | 23-Feb-09 |
| AR 38 | 896931 | 05-Apr-05 | HF 1 | 1056978 | 01-Sep-11 |
| AR 41 | 933798 | 01-Sep-06 | HF 2 | 1056979 | 01-Sep-11 |
| AR 43 | 933800 | 01-Sep-06 | HF 3 | 1056980 | 01-Sep-11 |
| AR 45 | 896938 | 05-Apr-05 | HF 4 | 1056981 | 01-Sep-11 |
| AR 46 | 896939 | 05-Apr-05 | HF 5 | 1056982 | 01-Sep-11 |
| AR 47 | 896940 | 05-Apr-05 | HF 6 | 1056983 | 01-Sep-11 |
| AR 48 | 896941 | 05-Apr-05 | HF 7 | 1056984 | 01-Sep-11 |
| AR 49 | 896942 | 05-Apr-05 | HF 8 | 1056985 | 01-Sep-11 |
| AR 50 | 896943 | 05-Apr-05 | HF 9 | 1056986 | 12-Sep-11 |

| Claim Name | BLM NMC # | Location Date |
|---|---|---|
| HF 10 | 1056987 | 12-Sep-11 |
| MH 1 | 1049740 | 06-May-11 |
| MH 2 | 1049741 | 06-May-11 |
| MH 3 | 1049742 | 06-May-11 |
| MH 4 | 1049743 | 06-May-11 |
| MH 5 | 1049744 | 07-May-11 |
| MH 6 | 1049745 | 06-May-11 |
| MH 7 | 1049746 | 07-May-11 |
| MH 8 | 1049747 | 07-May-11 |
| MH 9 | 1049748 | 07-May-11 |
| MH 11 | 1049750 | 07-May-11 |
| MH 13 | 1049752 | 06-May-11 |
| MH 14 | 1049753 | 07-May-11 |
| MH 15 | 1049754 | 07-May-11 |
| MH 16 | 1049755 | 09-May-11 |
| MH 17 | 1049756 | 09-May-11 |
| MH 18 | 1049757 | 09-May-11 |
| MH 19 | 1049758 | 09-May-11 |
| MH 20 | 1049759 | 09-May-11 |
| MH 21 | 1049760 | 09-May-11 |
| MH 22 | 1049761 | 09-May-11 |
| MH 23 | 1049762 | 09-May-11 |
| MH 24 | 1049763 | 09-May-11 |
| MH 25 | 1049764 | 09-May-11 |
| MH 26 | 1049765 | 08-May-11 |
| MH 27 | 1049766 | 08-May-11 |
| MH 28 | 1049767 | 08-May-11 |
| MH 29 | 1049768 | 08-May-11 |
| MH 30 | 1049769 | 08-May-11 |
| MH 31 | 1049770 | 08-May-11 |
| MH 32 | 1049771 | 08-May-11 |
| MH 33 | 1049772 | 08-May-11 |
| MH 34 | 1049773 | 08-May-11 |
| MH 35 | 1049774 | 08-May-11 |
| MH 36 | 1049775 | 08-May-11 |
| MH 37 | 1049776 | 08-May-11 |
| MH 38 | 1049777 | 08-May-11 |
| MH 39 | 1049778 | 08-May-11 |
| MH 40 | 1049779 | 08-May-11 |
| MH 41 | 1049780 | 08-May-11 |
| MH 42 | 1049781 | 08-May-11 |
| MH 43 | 1049782 | 08-May-11 |
| MH 44 | 1049783 | 09-May-11 |
| MH 45 | 1049784 | 09-May-11 |
| MH 46 | 1049785 | 09-May-11 |
| MH 47 | 1049786 | 09-May-11 |
| MH 48 | 1049787 | 09-May-11 |
| MH 49 | 1049788 | 09-May-11 |

| Claim Name | BLM NMC # | Location Date |
|---|---|---|
| MH 50 | 1049789 | 09-May-11 |
| MH 51 | 1049790 | 09-May-11 |
| MH 52 | 1049791 | 09-May-11 |
| MH 53 | 1049792 | 09-May-11 |
| MH 54 | 1049793 | 09-May-11 |
| MH 55 | 1049794 | 09-May-11 |
| MH 56 | 1049795 | 09-May-11 |
| MH 57 | 1049796 | 09-May-11 |
| MH 58 | 1049797 | 09-May-11 |
| MH 59 | 1049798 | 09-May-11 |
| MH 60 | 1049799 | 09-May-11 |
| MH 61 | 1049800 | 09-May-11 |
| MH 62 | 1049801 | 09-May-11 |
| MH 63 | 1049802 | 09-May-11 |
| MH 64 | 1049803 | 09-May-11 |
| MH 65 | 1049804 | 09-May-11 |
| MH 66 | 1049805 | 09-May-11 |
| MH 67 | 1049806 | 09-May-11 |
| MH 68 | 1049807 | 09-May-11 |
| MH 69 | 1049808 | 09-May-11 |
| MH 70 | 1049809 | 09-May-11 |
| MH 71 | 1049810 | 09-May-11 |
| MH 72 | 1049811 | 04-Jul-11 |
| MH 80 | 1053919 | 10-Jul-11 |
| MH 81 | 1053920 | 10-Jul-11 |
| MH 82 | 1069276 | 20-Feb-12 |
| MH 83 | 1069277 | 20-Feb-12 |
| MH 84 | 1069278 | 20-Feb-12 |
| MH 85 | 1069279 | 20-Feb-12 |
| MH 86 | 1069280 | 20-Feb-12 |
| MH 87 | 1069281 | 20-Feb-12 |
| MH 88 | 1069282 | 20-Feb-12 |
| MH 89 | 1069283 | 20-Feb-12 |
| MH 90 | 1069284 | 20-Feb-12 |
| MH 91 | 1069285 | 20-Feb-12 |
| MH 92 | 1069286 | 20-Feb-12 |
| MH 93 | 1069287 | 20-Feb-12 |
| MH 94 | 1093380 | 28-May-13 |
| MH 95 | 1093381 | 28-May-13 |
| MH 96 | 1093382 | 28-May-13 |
| MHP 1 | 1069271 | 27-Feb-12 |
| MHP 2 | 1069272 | 27-Feb-12 |
| MHP 3 | 1069273 | 27-Feb-12 |
| MHP 4 | 1069274 | 27-Feb-12 |
| MHP 5 | 1069275 | 27-Feb-12 |
| AR 39 | 933796 | 01-Sep-06 |
| AR 40 | 933797 | 01-Sep-06 |
| AR 42 | 933799 | 01-Sep-06 |

| Claim Name | BLM NMC # | Location Date |
|---|---|---|
| AR 44 | 933801 | 01-Sep-06 |
| AR 53 | 933802 | 01-Sep-06 |
| AR 54 | 933803 | 01-Sep-06 |
| AR 55 | 933804 | 01-Sep-06 |
| AR 56 | 933805 | 01-Sep-06 |
| AR 102 | 1044898 | 21-May-11 |
| AR 103 | 1044899 | 21-May-11 |
| H 10 | 839910 | 26-Nov-02 |
| H 11 | 839911 | 26-Nov-02 |
| H 12 | 839912 | 26-Nov-02 |
| H 13 | 839913 | 26-Nov-02 |
| H 14 | 839914 | 26-Nov-02 |
| H 15 | 839915 | 26-Nov-02 |
| H 16 | 839916 | 26-Nov-02 |
| H 17 | 839917 | 26-Nov-02 |
| H 18 | 839918 | 26-Nov-02 |
| H 19 | 839919 | 23-Nov-02 |
| H 20 | 839920 | 26-Nov-02 |
| H 21 | 839921 | 23-Nov-02 |
| H 22 | 839922 | 23-Nov-02 |
| H 25 | 839923 | 23-Nov-02 |
| H 26 | 839924 | 23-Nov-02 |
| H 27 | 839925 | 26-Nov-02 |
| H 28 | 839926 | 23-Nov-02 |
| H 36 | 839927 | 26-Nov-02 |
| H 37 | 839928 | 26-Nov-02 |
| H 38 | 839929 | 26-Nov-02 |
| H 39 | 839930 | 26-Nov-02 |
| MC | 839931 | 23-Nov-02 |
| Ada | 839932 | 23-Nov-02 |
| Mack #3 | 839933 | 23-Nov-02 |
| Mack Fraction | 839934 | 23-Nov-02 |
| VENUS | 861421 | 18-Nov-03 |
| MAY | 861422 | 18-Nov-03 |
| MACK | 861423 | 18-Nov-03 |
| ADA FRACTION | 861424 | 18-Nov-03 |
| Monte 1 | 875113 | 20-Feb-12 |
| Monte 2 | 875114 | 07-Jun-04 |
| Monte 3 | 875115 | 20-Feb-12 |
| Monte 4 | 875116 | 07-Jun-04 |
| Monte 5 | 875117 | 20-Feb-12 |
| Monte 6 | 875118 | 07-Jun-04 |
| Monte 9 | 1061262 | 20-Feb-12 |
| Monte 10 | 1061263 | 20-Feb-12 |
| Monte 11 | 1061264 | 20-Feb-12 |
| Monte 12 | 1061265 | 20-Feb-12 |
| Monte 13 | 1061266 | 20-Feb-12 |
| Monte 14 | 1061267 | 20-Feb-12 |
| Monte 15 | 1061268 | 20-Feb-12 |
| Monte 16 | 1061269 | 20-Feb-12 |
| Monte 17 | 1061270 | 20-Feb-12 |
| Monte 18 | 1061271 | 20-Feb-12 |
| Monte 19 | 1061272 | 20-Feb-12 |
| Monte 20 | 1061273 | 20-Feb-12 |
| Monte 21 | 1061274 | 20-Feb-12 |
| Monte 22 | 1061275 | 20-Feb-12 |
| Monte 23 | 1061276 | 20-Feb-12 |
| Monte 24 | 1061277 | 20-Feb-12 |
| Monte 25 | 1061278 | 20-Feb-12 |
| Monte 26 | 1061279 | 20-Feb-12 |
| Monte 27 | 1061280 | 20-Feb-12 |
| Monte 28 | 1061281 | 20-Feb-12 |
| Monte 29 | 1061282 | 20-Feb-12 |
| Monte 30 | 1061283 | 20-Feb-12 |
| Monte 31 | 1061284 | 20-Feb-12 |
| Monte 32 | 1061285 | 20-Feb-12 |
| Monte 33 | 1061286 | 20-Feb-12 |
| Monte 34 | 1061287 | 20-Feb-12 |
| Monte 35 | 1061288 | 20-Feb-12 |
| Monte 36 | 1061289 | 21-Feb-12 |
| Monte 37 | 1061290 | 21-Feb-12 |
| Monte 38 | 1061291 | 21-Feb-12 |
| Monte 39 | 1061292 | 21-Feb-12 |
| Monte 40 | 1061293 | 21-Feb-12 |
| Monte 41 | 1061294 | 21-Feb-12 |
| Monte 42 | 1061295 | 21-Feb-12 |
| Monte 43 | 1061296 | 21-Feb-12 |
| Monte 44 | 1061297 | 20-Feb-12 |
| Monte 45 | 1061298 | 20-Feb-12 |
| Monte 46 | 1061299 | 20-Feb-12 |
| Monte 47 | 1061300 | 20-Feb-12 |
| Monte 48 | 1061301 | 20-Feb-12 |
| Monte 49 | 1061302 | 20-Feb-12 |
| Monte 50 | 1061303 | 20-Feb-12 |
| Monte 51 | 1061304 | 20-Feb-12 |
| JC 11 | 1044891 | 21-May-11 |
| JC 13 | 1044892 | 21-May-11 |
| JC 14 | 1044893 | 21-May-11 |
| JC 15 | 1044894 | 21-May-11 |
| JC 16 | 1047577 | 09-Jun-11 |
| JC 17 | 1047578 | 09-Jun-11 |
| JC 18 | 1047579 | 09-Jun-11 |
| JC 20 | 1044895 | 22-May-11 |
| JC 21 | 1044896 | 22-May-11 |
| JC 22 | 1044897 | 22-May-11 |
| JC 30 | 1047580 | 10-Jun-11 |

| Claim Name | BLM NMC # | Location Date |
|---|---|---|
| JC 31 | 1047581 | 10-Jun-11 |
| JC 32 | 1047582 | 10-Jun-11 |
| JC 33 | 1047583 | 10-Jun-11 |
| JC 34 | 1047584 | 10-Jun-11 |
| JC 35 | 1047585 | 10-Jun-11 |
| JC 40 | 1054204 | 08-Aug-11 |
| AR 102 | 1044898 | 21-May-11 |
| AR 103 | 1044899 | 21-May-11 |
| JWP 1 | 1082917 | 19-Oct-12 |
| JWP 2 | 1082918 | 19-Oct-12 |
| JWP 3 | 1082919 | 19-Oct-12 |
| JWP 4 | 1082920 | 19-Oct-12 |

| Claim Name | BLM NMC # | Location Date |
|---|---|---|
| JWP 5 | 1082921 | 19-Oct-12 |
| JWP 6 | 1082922 | 19-Oct-12 |
| JWP 12 | 1082923 | 19-Oct-12 |
| JWP 19 | 1082924 | 04-Dec-12 |
| JWP 20 | 1082925 | 04-Dec-12 |
| JWP 21 | 1082926 | 04-Dec-12 |
| JWP 22 | 1082927 | 04-Dec-12 |
| JWP 23 | 1082928 | 04-Dec-12 |
| JWP 100 | 1082929 | 04-Dec-12 |

## 2.3 Nature and Extent of Issuer's Interest

Ely Gold's predecessor, Ivana Ventures Inc., acquired DHI Minerals (US) Ltd. (DHI) from Augusta Resource Corporation (Augusta) in November, 2007. DHI had previously acquired, through a lease agreement with Centennial Minerals Company (Centennial), the mineral rights to the "H" series claims and patented claims shown in Table 2.2.2 and Figure 2.2.1. These claims cover the resources and reserves at the Centennial and Seligman Deposits in the north central part of the Property. DHI has assigned 100% of its lease holding interest in the above mentioned claims to MH-LLC.

MH-LLC also directly owns unpatented claims and controls through lease-holding interest additional unpatented claims as shown on Figure 2.2.1.

The Fee lands shown in Sections 19 and 20 on Figure 2.2.1 are titled to MH-LLC.

## 2.4 Royalties, Agreements and Encumbrances

In order to maintain the Property in good standing, MH-LLC has the following land obligations and options.

- Annual advance minimum royalty payments to Centennial of US$300,000. These payments are credited against an existing 6% future production royalty to Centennial subject to royalty buydown options (as discussed below). As of the date of this report, MH-LLC has paid US$1.7 million in advanced royalty payments that are deductible from future royalty distributions.
- The Centennial royalty may be reduced to 2.75% by Solitario making a payment of US$3.5 million at any time prior to commercial production.
- The Centennial royalty may be further reduced to 1% by Solitario making a US$1.5 million payment any time prior to one year after commencement of commercial production.
- The CMC Shell and JC Shell lease agreements pertaining to certain unpatented claims outside of the resource area require annual payments of US$80,000 and US$110,000 respectively, due in June 2015 and annually thereafter at Solitario's option, to maintain the lease agreements are in force.
- The Monte claims are subject to a lease/option agreement with option payments of US$150,000 due September and US$200,000 due September 2015. After these option

payments are completed and for so long as the agreement is in good standing an annual royalty is paid to the underlying owner consisting of cash payments equal to 33 oz of gold annually. There are no current reserves or resources on the Monte claims property.

## 2.5 Environmental Liabilities and Permitting

### 2.5.1 Environmental Liabilities

SRK is unaware of any outstanding environmental liabilities aside from minor reclamation obligations associated with existing drill roads that are still actively used.

A portion of the Mt. Hamilton Property which was mined during the 1990's by a previous operator has been extensively reclaimed by the U.S. Department of Agriculture, United States Forest Service (USFS, or Forest Service). The leach pad associated with previous mining has also been covered with soil, contoured, and revegetated. At the time of SRK's site visits, seeding was successful and the pad is now completely grass-covered. The site of the former mine-associated infrastructure has been completely reclaimed and virtually all remains of buildings have been removed. The only significant elements of the former mining operation are the haulage road from the old leach pad to the NE Seligman Mine site and the open pit mining areas from prior operations. This road remains in excellent repair and provides ready access to both of the deposit areas. MH-LLC currently has no environmental liabilities related to this previous mining activity.

### 2.5.2 Required Permits and Status

The Project is being permitted separately on National Forest System (NFS) lands and patented mining claims, where the mining and access will occur, and on private land owned by MH-LLC where the processing of the ore is planned and administrative infrastructure will be located. A Mine Plan of Operations (MPO) was submitted to the Forest Service for mining activities on NFS lands. The MPO was determined to be complete by the USFS and scoping of the project was conducted in order to determine the issues to be evaluated to comply with the National Environmental Policy Act (NEPA). The USFS determined that an Environmental Assessment (EA) was required. Upon completion of the EA, a Finding of No Significant Impact and a draft Decision Notice were published on July 4, 2014. The Objection Period ended on August 18, 2014. No objections were received by the USFS. Phased bonding for reclamation of the mining areas will be required. The initial bonding of the first phase will need to be in place prior to construction activities.

Road access to the mine and to the administration/processing areas each requires crossing BLM land in order to enter the MPO area on Forest Service property. These two access routes are subject to a Right of Way grant by the BLM, which was issued to MH-LLC in 2013.

A Nevada Reclamation Permit (NRP) application has been submitted for the area covered by the MPO. The application for this permit is under review by the Nevada Division of Environmental Protection (NDEP) Bureau of Mining Regulation and Reclamation (BMRR).

The private land used for processing the ore and administrative functions is being permitted and bonded separately through the NDEP BMRR and will have a separate Nevada Reclamation Permit. An application for this permit has been filed and is under review. The USFS will not be involved in this permit approval although operations on private land were considered in the NEPA analysis as a connected action.

A Water Pollution Control Permit (WPCP) has been issued by the NDEP. The WPCP covers the entire project including both public and private land.

An Air Quality Permit application has been submitted to NDEP for review. A preliminary ADR plant design has been completed in order to provide the detail necessary for design of the mercury control systems to be incorporated in and reviewed under the Air Quality Permit.

Because of previously permitted mining activity at the Project and permits already approved by various agencies for the MPO, SRK has no reason to believe that remaining permits necessary to mine the mineral resources of the Project could not be reasonably obtained from State and Federal regulatory agencies.

**Other State of Nevada Permits**

Permits from the Bureau of Water Pollution Control (BWPC) are associated with water-related issues (e.g., storm water discharges and sanitary septic systems).

Water appropriations are processed through the Nevada Division of Water Resources (NDWR) and the State Engineer’s Office. MH-LLC has appropriated 875 acre-feet per annum (AFA) of water, an amount sufficient for peak water requirements for the operation including construction.

**Local Permitting**

A Special Use Permit for septic and excavation/building permits will be required from White Pine County; usually a copy of the MPO provides sufficient information for the County to review and issue this permit.

To the best of SRK’s knowledge, MH-LLC is in full compliance with all contractual and regulatory obligations. Because of previously permitted mining activity at the Project, SRK currently has no reason to believe that permits to mine the mineral resources at Mt. Hamilton could not be reasonably obtained from the state and federal regulatory agencies.

Major permits for future mining operations are summarized in Table 2.5.2.1.

**Table 2.5.2.1: Summary of Major Permits Required for Mining Operations**

| Regulatory Agency | Permit Name |
|---|---|
| **Federal Permits** | |
| US Forest Service | • Approved Plan of Operations/Decision Memo |
| Bureau of Alcohol, Tobacco, Firearms, and Explosives | • Authorization to purchase, transport, or store explosives |
| Mine Safety and Health Administration | • Notification of Commencement of Operation<br>• Employee and Facility Health and Safety |
| Environmental Protection Agency | • Hazardous Waste ID No. (small quantity generator) |
| Bureau of Land Management | • Roads and Utility Rights-of-Way |
| **State Permits** | |
| ***Nevada Division of Environmental Protection*** | |
| Bureau of Mining Regulation and Reclamation | • Water Pollution Control Permit<br>• Reclamation Permit |
| Bureau of Air Pollution Control | • Class I Air Quality Operating Permit<br>• Mercury Operating Permit |
| Bureau of Water Pollution Control | • Septic Permit |
| Bureau of Waste Management | • Approval to Operate a Solid Waste System<br>• Hazardous Waste Management Permit |
| Bureau of Safe Drinking Water | • Potable Water Permit |
| ***Nevada Division of Water Resources*** | |
| | • Permit to Appropriate Water<br>• Permit to Construct a Dam<br>• Hole Plugging |
| ***Nevada Department of Wildlife*** | |
| | • Industrial Artificial Pond Permit |
| ***State Fire Marshall*** | |
| | • Hazardous Materials Permit |
| **Local Permits** | |
| ***White Pine County*** | |
| | • Special Use Permit<br>• Building Permit<br>• Business License |

Source SRK, 2014

## 2.6 Other Significant Factors and Risks

SRK is not aware of any other significant factors or risks associated with the proposed mine development at this site.

# 3 Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 5)

## 3.1 Topography, Elevation and Vegetation

The Mt. Hamilton Property lies in the Basin and Range physiographic province, which is a series of north-trending mountain ranges with typically 2,000 to 5,000 ft of topographic relief above relatively broad and flat intervening valleys. The property is situated in the rugged western flanks of the White Pine Mountains. Seligman Canyon is an ephemeral drainage and is the largest in the project area; several smaller canyons also transect the property.

Local relief is approximately 4,000 ft in the area, ranging from about 6,500 ft (above mean sea level) amsl at the base of Newark Valley to 10,745 ft amsl at the summit of Mt. Hamilton, which is located about one mile southeast of the property. The project area is on the flank of Mt. Hamilton, between 6,500 ft and 9,500 ft amsl, and most of the infrastructure will be built on private land on the gravel and silt alluvial fan downslope from exposed bedrock. This soil is well-drained, and has incised dry drainages spaced several hundred feet apart. Surface slope averages about 6%, and increases to more than 10% closer to the exposed bedrock of the range front. Terrain is rugged in higher areas with shallow soil and exposed bedrock, and slopes are very steep. The former processing plant and leach pad site used during operation at the historical Seligman Mine is located at the boundary between scrubland (dominated by sagebrush and various grasses) and forest dominated by juniper and piñon pine at an elevation of approximately 7,000 ft. At the abandoned mine site, located at 9,000 ft elevation, forest cover is less dense and pine is dominant. No agriculture exists in the area, but there are leases in effect for cattle grazing.

Dominant flora species include piñon and white pine trees at higher elevations; sagebrush, saltbrush, rabbitbrush and other low shrubs, and grasses along with juniper and piñon pine dominate at lower elevations. Cacti and perennial wildflowers are also present, but shrubs and trees are the dominant land cover. Soil is well-drained, and has poorly-developed topsoil less than 3 ft thick. Root penetration has been observed up to 6 ft below ground surface (bgs) in the planned leach pad area, and is more typically about 3 ft deep. Caliche horizons have also been observed 3 to 9 ft-bgs.

## 3.2 Climate and Length of Operating Season

Climate is typical for the high-desert regions of eastern Nevada- typically with hot, dry summers and cold snowy winters. Summer high temperatures can peak at 100° Fahrenheit (F) (38°C), with winter low temperatures typically at 0° to 15°F (-18° to -9°C), and winter high temperatures of only 30-40°F (-1° to 4°C). Most of the precipitation for the region falls as snow in the winter months, with lesser precipitation as rainfall in the spring and as thunderstorms during the late summer. Winter storms can deposit many feet of snow in the upper mountains. During years of high-snowfall, elevations above about 7,000 ft can be continually snow-covered from November through April.

In the absence of better road access and the equipment necessary to keep roads open, the typical exploration season for the Mt. Hamilton Property is from May through November. Drilling activities in the region are commonly conducted during June through October. Improved road access and road maintenance/snow removal equipment would extend the operating season through the winter months for year-round mining.

## 3.3 Sufficiency of Surface Rights

The surface rights on the Mt. Hamilton Property are owned in part by MH-LLC but are predominantly public domain administered by the USFS. Minor portions of the local access to the Property are administered by the U.S. Bureau of Land Management. All areas of proposed activities fall either on MH-LLC private land or on unpatented mining claims controlled by MH-LLC. In the latter case proposed actions will be subject to approval by the USFS of a Plan of Operations and qualified by the terms of the Decision Notice for that document.

## 3.4 Accessibility and Transportation to the Property

The property lies about 10 miles south of U.S. Highway 50 via White Pine County Road 5, and thence about 45 miles west of Ely, Nevada. The nearby communities, Ely and Eureka, are approximately equidistant from the project site. From either community, the project site can be accessed by car, on paved and unpaved roads, in about an hour. The deposit area is accessed from the Seligman haul road, and a network of narrow prospecting roads. All roads off Highway 50 are gravel-surface, one- or two-lane, and most transect land administered by the BLM or the USFS. Local roads are continuous over sub-sections of privately-owned land, all of which are owned by Mt. Hamilton LLC.

## 3.5 Infrastructure Availability and Sources

Ely has a population of about 4,000. Ely is the support community for the Robinson (Copper) Mine. Ely is also the County seat for White Pine County and all land records and related support material are located in the county offices there. The city of Elko, Nevada is located approximately a three-hour drive north of the Mt. Hamilton Property. Elko has a population base of about 36,000 and is a support community for many major gold mining operations in northern Nevada. As such, Elko has all the services available to support gold exploration and development activities in the region. Eureka, with a population of approximately 2,000, is located approximately 50 miles west of the property along Nevada State Route 50 and was the support community for the recently closed Ruby Hill (Gold) Mine.

### 3.5.1 Power

The nearest power line of sufficient capacity for mine operations is approximately 17 miles from the project site along Hwy 50. The current mine plan includes on-site diesel generated electrical power, with conversion to line power early in the mine life.

### 3.5.2 Communications

Cellular phone service is intermittently available at the proposed leach pad and truck shop facilities, but is limited in the proposed pit area due to the steep topography. As is typical of most pre-construction mine sites, landline telephones and internet services are not currently available at the site.

### 3.5.3 Water

There is a water well in Seligman Canyon capable of producing 500 gallons per minute (gpm) and a second, backup well that produces 200 gpm. Water rights sufficient for project start-up have been secured by MH-LLC. MH-LLC has appropriated a total of 875 AFA of water, an amount sufficient for peak water requirements for the operation and construction. Water resource exploration is proposed to install and develop an additional well closer to the processing facility. This 2014 FS assumes that water would be obtained from the more distant, Seligman Canyon site.

### 3.5.4 Mining Personnel

The labor force for mining at Centennial would be drawn largely from Ely and Eureka, Nevada. These local populations are part of established mining communities with producing mines nearby where a sufficient workforce of experienced open pit miners is available. All personnel would live in nearby communities and there is adequate housing available to accommodate all future personnel.

### 3.5.5 Potential Tailings Storage Areas

The mine plan is based on a cyanide heap leach gold and silver recovery system, and will not require a tailings storage area. Spent ore material will not be removed from the lined leach pad.

### 3.5.6 Potential Waste Disposal Areas

There is currently a waste rock disposal area in Cabin Gulch from the historical mining in the NE Seligman Pits. Waste rock produced during planned mining will also be placed at the Cabin Gulch site and in a smaller location directly upslope. The expansion of the Cabin Gulch dump will allow for reclamation of this historical disposal facility which was never reclaimed after mining was completed at the NE Seligman mine site.

### 3.5.7 Potential Heap Leach Pad Areas

The planned leach pad lies on private land approximately 4,500 ft southwest of the planned Centennial Pit and immediately west of the range front on pediment gravel. The ore will be transported by truck from the mine to a primary crusher and ore pass where the crushed ore will be dropped about 415 vertical feet onto a conveyor in an underground adit. The conveyor will deliver ore through the portal of the adit directly onto the private land. Near the adit opening, ore will undergo secondary crushing and then be placed on the leach pad by radial stacker.

### 3.5.8 Potential Processing Plant Sites

The ore will be secondary crushed and conveyed to the heap leach pad where it will be leached. Solutions will be treated by conventional ADR technology. The ADR processing plant will be located immediately adjacent to the leach pad and will have associated process ponds. No milling, flotation or vat leach processing is planned for the ores at the Project.

# 4 History (Item 6)

## 4.1 Prior Ownership and Ownership Changes

Phillips Petroleum Co. (Phillips) acquired much of the area of the current Property in 1968 and, between 1968 and 1982, drilled over 100,000 ft in the exploration for tungsten-copper-molybdenum deposits. A study prepared for Phillips in June 1978 quoted an “ore reserve” of 6.2 Mt at a grade of 0.37% $WO_3$ including 4.2 Mt grading 0.42% $WO_3$, 0.37% Mo and 0.6% Cu. These data are historical and have not been reviewed by a QP. The resource is not reconciled with or compliant with CIM resource classifications; and, MH-LLC is not reporting this as a current or compliant resource estimate.

In 1984 Northern Illinois Coal, Oil and Resources Mineral Ventures, subsequently renamed Westmont Gold Inc., (Westmont) entered into a joint venture with Phillips and Queenstake Resources Ltd. to explore the property for open-pit mineable gold-silver mineralization. By early 1989, this work had defined the Seligman and Centennial gold deposits. Permitting activities for the Mt. Hamilton Project were commenced in 1988. In 1991, Westmont reported a geological resource of 11.4 Mt at 0.05 oz/t Au and 0.5 oz/t Ag (Myers et al., 1991). These data are historical and have not been reviewed by a QP. The resource is not reconciled with or compliant with CIM resource classifications; and, MH-LLC is not reporting this as a current or compliant resource estimate.

The property was transferred to Mt. Hamilton Mining Company (MHMC, a Westmont subsidiary) after November 1993. In 1993, the Mt. Hamilton resources were estimated at 10.4 Mt at 0.05 oz/t Au and 0.334 oz/t Ag in the Seligman deposit (0.02 oz/t Au cut-off) and 6.187 Mt at 0.046 oz/t Au and 0.555 oz/t Ag in the Centennial deposit (0.016 oz/t Au cut-off). These data are historical and have not been reviewed by a QP. The resource is not reconciled with or compliant with CIM resource classifications; and, MH-LLC is not reporting this as a current or compliant resource estimate.

Rea Gold Corp. acquired MHMC in June 1994 and began production of the Seligman deposit in November 1994. Rea encountered a number of operational problems during the first year of production amplified by low gold price. Rea had planned to commence mining of the Centennial deposit in 1997, which contained resources as defined below. Rea ceased mining in June 1997, but continued leaching until declaring bankruptcy in Canadian Bankruptcy Court in November 1997. Subsequently the US subsidiary, Mt. Hamilton Mines Corporation was forced into US bankruptcy when the State of Nevada rescinded their permit to purchase and use cyanide.

In 2002, the US Bankruptcy Trustee abandoned all of the unpatented claims allowing them to lapse for failure to pay the annual maintenance fees. Centennial Minerals Company LLC staked claims covering the Centennial Deposit in late 2002, and in 2003 purchased all of the patented mining claims and Fee lands from the US Bankruptcy court. Augusta, through its 100% owned subsidiary Diamond Hill Minerals Ltd (DHI), acquired a leasehold interest in the property from Centennial in late 2003. Under an agreement with Augusta Resource Corporation (Augusta) dated November 15, 2007, Ivana acquired 100% of the shares of DHI. Ivana changed its name to Ely Gold & Minerals in 2008. On August 26, 2010, Solitario signed a Letter of Intent with Ely to earn up to an 80% interest in Ely’s Mt. Hamilton gold property. In December 2010, Solitario and Ely formed MH-LLC which now holds 100% of the Mt. Hamilton project assets, and signed an LLC Operating Agreement. On November 30, 2013, Ely made the final payments pursuant to the Augusta Agreement.

## 4.2 Previous Exploration and Development Results

As stated in Section 4.1 exploration was conducted by Phillips, Westmont and Queenstake on the northern core of the Property containing the Centennial, Seligman and tungsten-molybdenum mineralization arranged symmetrically around the Seligman intrusive stock. Ely Gold completed infill drilling and conducted additional metallurgical testing at the Centennial Deposit during 2008-10.

Additional exploration was conducted in the late 1980's and 1990's peripheral to the Monte Cristo stock approximately one mile to the south of the Centennial deposit. Shell Oil Company, Westmont and Augusta all drilled exploration holes in this area in search of copper, and tungsten-molybdenum deposits.

The only mine development of commercial scale was by Rea Gold Corp. at the Seligman mine as described above.

## 4.3 Historic Mineral Resource and Reserve Estimates

All of the resources mentioned in Section 4.1 for the Seligman, Centennial and tungsten molybdenum deposits calculated by Phillips, Westmont and Rea do not comply with CIM resource classifications.

**2008 Scott Wilson Roscoe Postle & Associates (NI 43-101)**

An NI 43-101 Technical Report by Scott Wilson Roscoe Postle & Associates (SWRPA) stated a CIM-compliant resource for the Project dated February 11, 2008 (SWRPA, 2008) (Table 4.3.1). SWRPA classified all resources at the Centennial deposit as Inferred due to the lack of supporting documentation and drill samples.

**Table 4.3.1: Centennial Inferred Resources (SWRPA 2008)**

| CoG | Mt | Au (oz/t) | Au (oz) | Ag (oz/t) | Ag (oz) |
|---|---|---|---|---|---|
| 0.016 | 12.3 | 0.034 | 415,200 | 0.177 | 2,175,000 |

Ag grade and contained ounces are in terms of NaCN soluble Ag

**2009 SRK (NI 43-101)**

In 2008, Ely Gold subsequently located drill core and chips and supporting data including drill logs and assay certificates. The new materials and data were catalogued and audited by SRK. A revised resource estimate was issued by Ely Gold in an NI 43-101 compliant Technical Report and Preliminary Economic Assessment dated May 8, 2009 (SRK, 2009). The resource statement from that report is provided in Table 4.3.2 at a cut-off grade (CoG) of 0.009 oz/t Au. The CoG was developed using metal prices of US$750/oz Au and US$13/oz Ag with a projected gold recovery of 73%.

**Table 4.3.2: Mineral Resource Statement for Ely Gold's Centennial Deposit 2009**

| Category | Tons | Au Grade (oz/t) | Au (oz) | Ag Grade (oz/t) | Ag (oz) |
|---|---|---|---|---|---|
| Measured | 760,000 | 0.039 | 29,640 | 0.130 | 98,800 |
| Indicated | 11,857,000 | 0.030 | 355,710 | 0.145 | 1,719,265 |
| **Measured and Indicated** | **12,617,000** | **0.031** | **385,350** | **0.144** | **1,818,065** |
| Inferred | 1,491,000 | 0.012 | 17,892 | 0.122 | 181,902 |

CoG 0.009 oz/t Au.

**2010 SRK (NI 43-101)**

In 2010 with metal prices up sharply, Ely Gold requested SRK to update the economic evaluation for Centennial from which they issued a new PEA (SRK, 2010). The underlying resource block model was unchanged from 2009 to 2010 and the Mineral Resource Statements, therefore, were identical.

However, metal prices used in the 2010 update were US$900/oz gold and US$15/oz silver. Gold recovery was increased from 73% to 75%, based on favorable metallurgical results that were received in the time period between the two reports. The combination of higher metal prices and higher recovery estimates resulted in a lower CoG calculation of 0.0065 oz/t Au for the 2010 statement. The resources were reported as contained within a potentially minable pit configuration using the aforementioned metal prices and costs of US$1.75/t for mining, US$3.50/t for processing and US$0.75/t for G&A. The 2010 Mineral Resource Statement is presented in Table 4.3.3.

**Table 4.3.3: 2010 Mineral Resource Statement, SRK, 2010 (In Pit)**

| Category | kt | Au Grade (oz/t) | Au (koz) | Ag Grade (oz/t) | Ag (koz) |
|---|---|---|---|---|---|
| Measured | 823 | 0.037 | 30 | 0.129 | 106 |
| Indicated | 13,534 | 0.028 | 379 | 0.153 | 2,071 |
| **Measured and Indicated** | **14,357** | **0.029** | **409** | **0.152** | **2,177** |
| Inferred | 3,369 | 0.010 | 34 | 0.129 | 435 |

CoG: 0.0065 oz/t Au

**2012 SRK (NI 43-101)**

The most recent NI 43-101 compliant Technical Report update, prior to this 2014 update, was issued by MH-LLC on October 25, 2012. It included a Mineral Resource Statement for the Centennial Deposit and separately the Seligman Deposit. In addition, Mineral Reserves were stated for the Centennial Deposit, which were supported by an economic mine plan and feasibility-level cost estimation.

The Mineral Resources and Mineral Reserves reported in 2012 are presented in Table 4.3.4, 4.3.5, and 4.3.6 below. Seligman resources were reported to a 0.006 oz/t Au CoG. Centennial resources and reserves were reported to a 0.006 AuEq CoG. The price and cost assumptions for CoG determinations are provided as footnotes to the respective statements.

**Table 4.3.4: Mineral Resource Statement Centennial Gold-Silver Deposit, White Pine County, Nevada, SRK Consulting (U.S.), Inc.**

| Resource Category | Tons (000's) | Au Grade (oz/t) | Contained Au (oz) | Ag Grade (oz/t) | Recoverable Ag (oz)* |
|---|---|---|---|---|---|
| Measured | 918 | 0.032 | 29,524 | 0.155 | 142,152 |
| Indicated | 22,732 | 0.022 | 497,330 | 0.132 | 3,010,471 |
| **Measured and Indicated** | **23,650** | **0.022** | **526,854** | **0.133** | **3,152,624** |
| Inferred | 3,454 | 0.018 | 60,859 | 0.079 | 273,457 |

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves estimate;
- Mineral reserves stated for Centennial in this Technical Report are inclusive of resources stated, and are not additional to resources stated;
- Resources stated as contained within a potentially economically minable open pit above a 0.006 oz/t AuEq CoG;
- Pit optimization is based on assumed gold and silver prices of US$1,600/oz and US$40.00/oz, respectively, effective heap leach recoveries of 75% and 30% for gold and silver, respectively, a mining, processing and G&A cost of US$5.81/t; Net Smelter Return 1% and pit slopes of 50°.
- Reported Au ounces are contained metal subject to process recovery which will result in a reduced number of payable ounces;
- Reported Ag ounces have already received a recovery discount during resource modeling; therefore, there will be minimal further reduction of payable Ag ounces after processing; and
- Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

**Table 4.3.5: Mineral Resource Statement of the Seligman Gold-Silver Deposit, SRK Consulting (U.S.) Inc., July 31, 2012**

| Resource Category | Tons (Millions) | Au Grade | | Ag Grade | | AuEq | Contained Ounces | | |
|---|---|---|---|---|---|---|---|---|---|
| | | oz/t | g/tonne | oz/t | g/tonne | oz/t | Au | Ag | AuEq |
| **Indicated** | **6.95** | **0.022** | **0.76** | **0.097** | **3.34** | **0.023** | **154,388** | **676,665** | **166,691** |
| Inferred | 3.77 | 0.021 | 0.71 | 0.144 | 4.94 | 0.022 | 78,044 | 543,671 | 87,929 |

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves estimate;
- Resources stated as contained within a potentially economically minable open pit stated above a 0.006 oz/t Au CoG;
- Pit optimization is based on assumed gold and silver prices of US$1,500.00/oz and US$20.00/oz, respectively and pit slopes of 50°;
- Metallurgical recoveries of 70% for gold in skarn, 65% for gold in igneous, and 35% for silver, and an ore mining and processing cost of US$6.45/t;
- A net smelter return royalty of 3.4% was applied as a selling costs in the pit optimization;
- Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate and numbers may not add due to rounding.

**Table 4.3.6: Mineral Reserves Statement, Centennial Gold-Silver Deposit, White Pine County, Nevada, SRK Consulting (U.S.), Inc.**

| Classification | Resource (kt) | Au Grade (oz/t) | Contained Au (koz) | Ag Grade (oz/t) | Contained Ag (koz) |
|---|---|---|---|---|---|
| Proven | 923 | 0.032 | 29.3 | 0.155 | 142.7 |
| Probable | 21,604 | 0.021 | 457.8 | 0.134 | 2,884.3 |
| **Total Proven and Probable** | **22,527** | **0.022** | **487.1** | **0.134** | **3,028.2** |

- Some numbers may not add properly as a function of rounding.
- A net smelter return royalty of 3.4% was applied as a selling costs in the pit optimization;
- Reserves are based upon 0.006 oz/t – AuEq CoG, using US$1,200/oz-Au gold price and US$20/oz-Ag.

A 2014 update of these resources and reserves and their associated mining and processing plan and costs are the subject of this report.

## 4.4 Historic Production

Between 1994 to1997, production by Rea from the NE Seligman mine is reported to be 124,000 oz Au and 310,250 oz Ag. The mined tonnage of ore during the period of production by Rea is unknown. The haul road was extended to the Centennial pit area and the area of the starter pit was clear-cut and grubbed of vegetation in preparation for preproduction stripping which was scheduled to begin in 1997, but was never initiated. Hence, there has been no historic production from the Centennial deposit.

# 5 Geological Setting and Mineralization (Item 7)

## 5.1 Regional Geology

The Mt. Hamilton Property is located in the White Pine Mountains, which are in the eastern sector of the Great Basin in east-central Nevada. This region was subjected to east to west compression during the Sevier and Laramide orogenies in the Cretaceous and early Tertiary. This compression resulted in the formation of broadly north-trending folds and thrust faults. Two major folds are present in the project area: the Hoppe Springs anticline (into which the Seligman stock has intruded) and the Silver Bell syncline to the west. Scattered magmatism was common during this time period, as evidenced in the Mt. Hamilton area by the Cretaceous Seligman and Monte Cristo stocks, which are dated at 104.5 to 106.6 Ma (K-Ar, biotite) and 101.2 Ma (K-Ar, biotite), respectively. Base- and precious-metal deposits related to igneous activity of this general age are widespread across western North America.

Extension beginning in the middle Tertiary has affected much of southwestern North America, resulting in the basin and range style of physiography that is present from southern Oregon to central Mexico. The White Pine Mountains are one of the many mountain ranges that have been uplifted along north-striking steeply dipping normal faults. A map of the regional geology is shown in Figure 5.1.1 (Crafford, A.E.J, 2007).

## 5.2 Local and Property Geology

The Mt. Hamilton Property is located near the southern end of the Battle Mountain Gold Trend, a northwest-oriented trend that contains several major gold mines as well as dozens of smaller mines and prospects and together with the Carlin trend to the northeast are the two largest gold belts in Nevada. The property consists of gently folded Cambrian-age sedimentary rocks intruded by the Monte Cristo and Seligman stocks. A map of local geology in Figure 5.2.1 shows the location of the igneous intrusive units relative to existing and planned open pit excavations (Crafford, A.E.J, 2007).

### 5.2.1 Stratigraphy

Burgoyne (1993, p. 2) provides a succinct summary of the sedimentary and igneous rock sequence at the Mt. Hamilton Property:

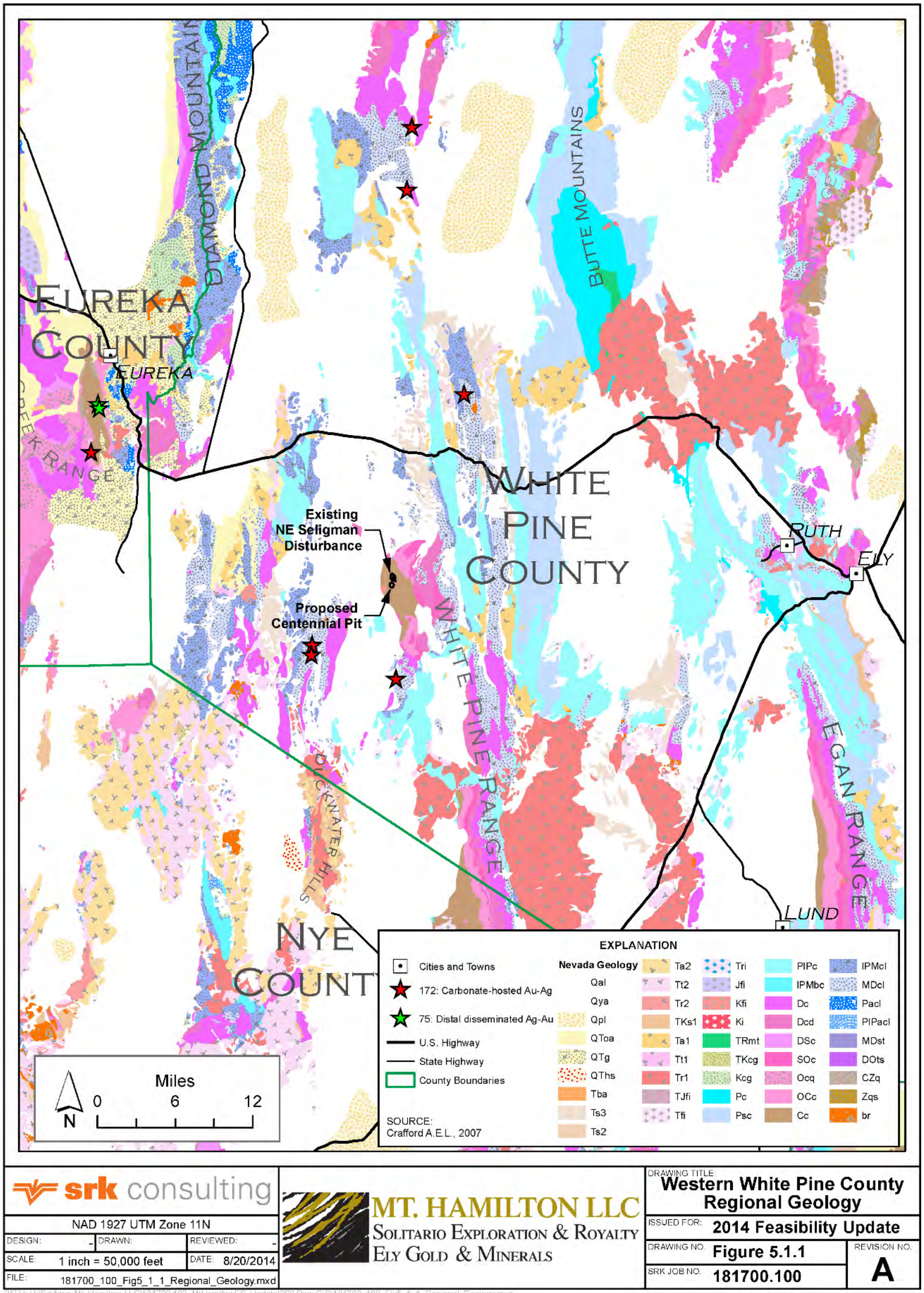
> *"Sedimentary rocks in the Mount Hamilton area range from Middle Cambrian to Pennsylvanian [age]. Stratigraphic units include the middle Cambrian Eldorado Dolomite, Geddes Limestone, and Secret Canyon Shale and the Upper Cambrian Dunderberg Shale.*
>
> *The Eldorado Dolomite, the oldest formation in the area, consists of gray to white stromatalitic dolomite up to 660 ft thick. The Geddes Limestone overlies the Eldorado Dolomite and deep drilling indicates that the Eldorado Dolomite-Geddes Limestone contact is a breccia zone. The Geddes Limestone consists of dark gray, platy limestone and has a thickness in excess of 100 ft.*
>
> *The Secret Canyon Shale accounts for the majority of the sedimentary sequence in the project area and is about 1,000 ft thick. It consists of four sub-units: a basal thin-bedded pale green shale, a thin-bedded limestone with shale partings, a thin-bedded greenish shale, and an uppermost series of interbedded limestone and shale.*

*The Dunderberg Shale disconformably overlies the Secret Canyon Shale and is 400 to 1,000 ft thick. This formation consists of a basal greenish shale and mudstone with thin limestone interbeds. A middle sequence of interbedded carbonaceous shale and limestone with shale partings forms the bulk of the formation. An uppermost sequence consists of thinly bedded, nodular limestone with shale partings.*

*The sedimentary sequence has been intruded by two stocks of Cretaceous age. The Seligman stock is a medium-grained, hornblende-biotite granodiorite. The stock is elongated in a north-south direction along the axis of the Hoppe Springs anticline. Potassium-argon age dating on the biotite gives [ages] of 104.5 to 106.6 million years.*

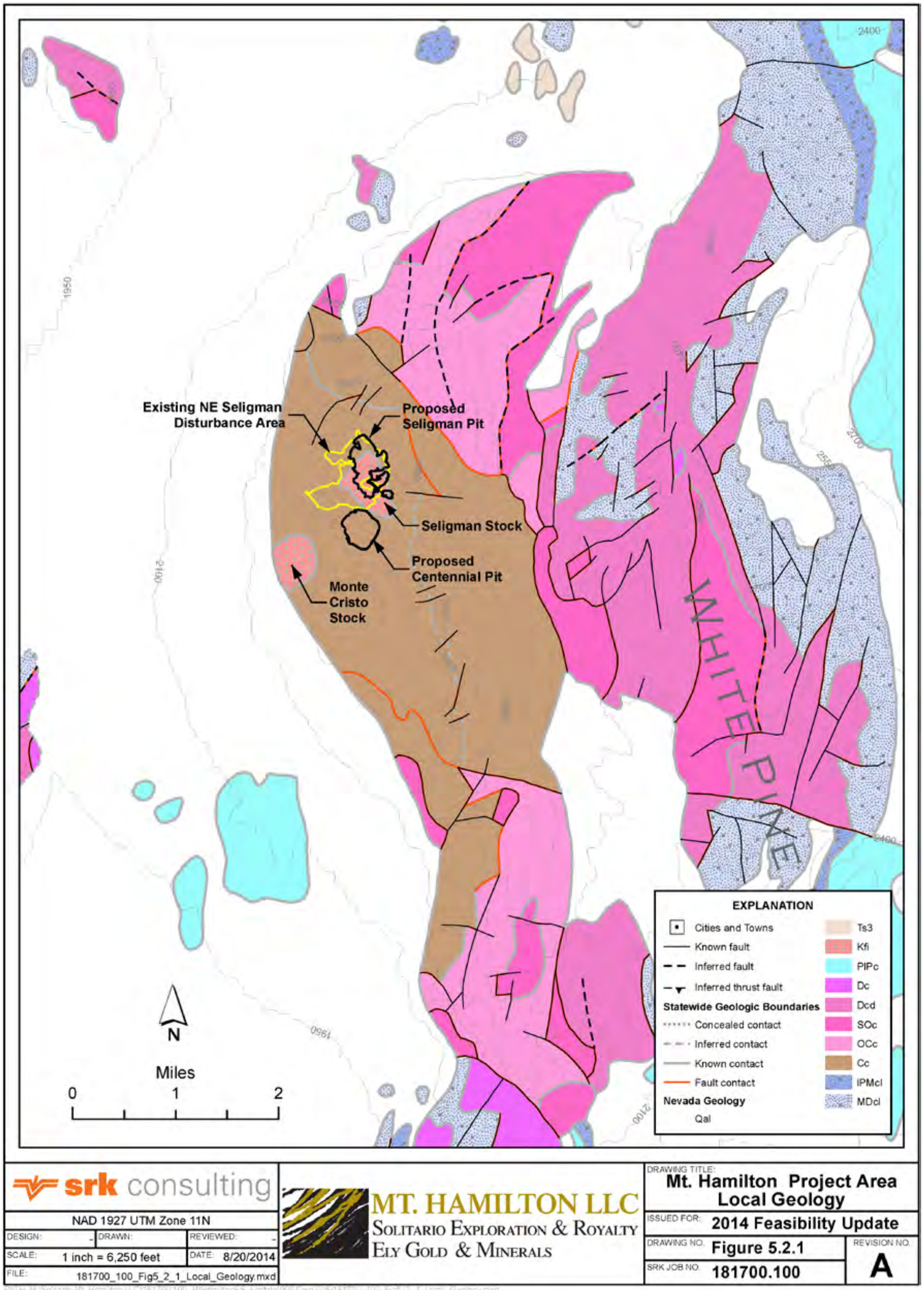
*The Monte Cristo stock, composed of biotite granite-porphyry, is located 0.5 mi southwest of the Seligman stock. The stock displays extensive quartz stockwork [veining] and quartz flooding. Potassium-argon age dating on biotite gives [an age of] 101.2 million years.*

*Several dykes and sills occur throughout the area and range from 3 to 30 ft thick and are compositionally similar to the Seligman and Monte Cristo stocks."*




Source: SRK, 2014

**Figure 5.1.1: Western White Pine County Regional Geology Map**




Source: SRK, 2014

**Figure 5.2.1: Mt. Hamilton Project Area Local Geology**

### 5.2.2 Alteration

A description of the alteration at Mt. Hamilton is provided by Burgoyne (1993):

*"Alteration within the Seligman stock is marked by secondary biotite (potassic alteration), propylitic alteration of mafic minerals and plagioclase to chlorite, epidote, and calcite. Sericitic alteration is associated with pervasive silicification and locally with extensive pyrite.*

*A hydrothermal alteration aureole is present in the sedimentary rocks concentrically about the Monte Cristo and Seligman stocks. The alteration aureole is about 3 mi long by 1.5 mi wide. Alteration is complex but an early first stage is represented by the formation of hornfels, a dominantly metamorphic stage. A later cross-cutting, metasomatic [alteration phase] resulted in the formation of skarn.*

*The hornfels stage has altered shales and calcareous shales to fine grained, pale green diopside-quartz-potassium feldspar proximal to the intrusives. This alteration grades outward to fine-grained biotite-quartz hornfels distal to the intrusives. The shales have been bleached and silicified up to several hundred feet beyond the biotite hornfels. The limestone layers within the shales have been altered to medium-grained marble with occasional fine-to-medium grained tremolite or wollastonite, often with garnet, developed at the limestone-shale contacts.*

*The transition from hornfels to skarn is marked by increasing iron content in the pyroxene and the formation of andraditic garnet.*

*Retrograde alteration or extensive oxidation and breakdown of primary mineralogy is limited in extent and consists of two periods. The earliest and most common (Type I) is garnet altered to quartz, calcite, and pyrite. The later alteration (Type 2) is represented by gold and silver mineralization and is represented by the alteration of garnet and pyroxene to quartz, epidote, iron oxides, actinolite, chlorite, and manganese enriched epidote."*

Alteration associated with epithermal, structurally controlled gold mineralization is characterized by weak silicification and argillization. Later oxidation by meteoric waters has converted pyrite to iron oxides.

### 5.2.3 Structure

A description of the structural control of mineralization at Mt. Hamilton is provided by Burgoyne (1993):

*"The main Centennial mineralization is contained within a south dipping (15°-20°) tabular zone that ranges from 20 to 250 ft thickness. It is postulated that northwest and northeast feeder faults containing gold-silver mineralization are present.*

*[At nearby Seligman] ore grade mineralization appears to be largely stratiform in shallow-dipping, bedding-parallel, structurally and chemically prepared zones with local high-angle, cross-cutting, possible "feeder" zones."*

## 5.3 Significant Mineralized Zones

Two zones of gold mineralization have been recognized at Mt. Hamilton: the Seligman and Centennial Zones. Prior to mining, the Seligman deposits were modeled as shallow-dipping zones approximately 3,300 ft by 1,000 ft, averaging 50 ft in thickness. During mining, REA Gold found that some high-grade mineralized zones at Seligman appeared to be controlled by steep, north dipping fractures and shear zones.

At Centennial the mineralization is controlled by late low-angle structures that are discordant to bedding and oxidized to significant depth. The low-angle structures dip to the SSE at approximately 10-15°, and carry the majority of the oxide mineralization. Natural weathering and oxidation of original sulfide mineralization caused formation of oxide mineralization (with low sulfide mineral residuals) from which gold is recoverable by cyanide heap leaching. The acid generating capacity of the surrounding carbonate rocks is low or nil, and their acid consuming capacity is high. Gold is present as free gold, residing in iron oxide minerals or quartz, and adsorbed on clay minerals.

Gold occurs predominantly in zones of retrograde alteration and, to a minor extent, in prograde garnet-pyroxene skarn. The retrograde alteration zones are comprised of a quartz-goethite-epidote-calcite assemblage that replaces garnet-pyroxene skarn. Gold grades of samples within the retrograde alteration range from <0.001 oz/t Au (lower analytical method detection limit) to 0.995 oz/t. The occasional high grades appear to be associated with crosscutting structures and veins within the skarn as described below. In the Centennial gold database, a total of seven values were greater than the 0.36 oz/t value used as a cap for the resource estimate.

Sulfosalt-bearing veins consisting primarily of quartz and stibnite with minor, variable amounts of sphalerite, galena, pyrite, covellite, bornite, chalcopyrite, bournonite and jamesonite typically occur within the mineralized zones and may be associated locally with the higher grades of gold. These veins cut both skarn and intrusive rocks and are closely associated with zones of retrograde alteration. These veins range in thickness from about 2 cm to 60 cm. As seen in the mine excavations of the Seligman deposit, these veins seem to exhibit strong continuity along strike.

# 6 Deposit Type (Item 8)

The mineralization associated with the Seligman stock, including the NE Seligman and Centennial deposits as well as other less-explored occurrences, is described by SWRPA as the polymetallic skarn deposit type (Myers et al., 1991). Deposits of this type have been described by numerous authors, including G.E. Ray of the British Columbia Geological Survey. Values are expressed here in grams per metric tonne (g/tonne), ounces per short ton (oz/t) and million short tons (Mt). Typically, these deposits range from 0.4 to 13 Mt and from 2 g/tonne (0.065 oz/t) Au to 15 g/tonne (0.48 oz/t) Au, with median grades and tonnage of 8.6 g/tonne (0.28 oz/t) Au, 5.0 g/tonne (0.16 oz/t) Ag and 213,000 t. Nickel Plate (Hedley District, BC) produced over 71 t of Au from 13.4 Mt of ore (grading 5.3 g/tonne [0.17 oz/t] Au). The 10.3 Mt Fortitude deposit (Battle Mountain Gold Trend, Nevada) graded 6.9 g/tonne (0.22 oz/t) Au, whereas the 13.2 Mt McCoy skarn (Nevada) graded 1.5 g/tonne (0.048 oz/t) Au (Ray, G.E, 1988).

More recent work suggests that the gold deposits at Seligman and Centennial are actually epithermal deposits that were controlled by structures that cut the skarn-altered carbonate rocks and are not directly associated with fluids related to contact metasomatism. Subsequently, low angle structures were filled by pyritic gold bearing mineralization that were oxidized by meteoric ground water.

## 6.1 Mineral Deposit

Replacement mineralization at Mt. Hamilton consists of skarn-hosted tungsten, molybdenum, and copper. Metal mineralization appears to have been emplaced in several separate events. Tungsten, as scheelite, is disseminated in thin-bedded skarn zones within diopsidic hornfels or skarn replacements of the Secret Canyon Shale, and overlying dolomite and shale of the Dunderberg Shale, and is locally associated with massive garnet-pyroxenite skarns that replace limestone beds. Tungsten grades are locally as high as 2% $WO_3$ but generally range in the tens to hundreds of parts per million (ppm).

Molybdenum is associated with prograde pyroxene-dominant skarn and grades range from tens to hundreds of ppm Mo. Silicified molybdenum-bearing breccias cut both the NE Seligman stock and adjacent pyroxene-tremolite hornfels. Molybdenum mineralization is in part contemporaneous with, and in part post-dates the tungsten mineralization.

Copper, as chalcopyrite, is disseminated within garnet-pyroxene skarn, occurs primarily southeast of the Seligman stock, and appears to be cogenetic with tungsten and molybdenum. Cu grades are usually <250 ppm. Zinc is associated with garnet-pyroxene skarn and locally grades up to 3%.

Late stage epithermal activity with associated gold and silver mineralization overprinted the older skarn alteration.

## 6.2 Geological Model

Mt. Hamilton geology is characterized by a polymetallic skarn overprinted by late-stage epithermal gold mineralization concentrated along two shallowly dipping faults that provided ingress to hydrothermal fluids likely sourced from the Seligman stock, or a related intrusion. Early metasomatic alteration converted shales and silty carbonates of the upper Secret Canyon shale (and/or Hamburg dolomite) to hornfels (after shales) and calc-silicate skarn (after silty carbonates). Gold mineralization is primarily hosted in a 200 to 300 ft thick skarn horizon, bounded by upper (200 ft thick) and lower (450 ft thick) hornfels units. The bounding hornfels had lower permeability and were therefore less receptive to late-stage mineralization. The interbedded skarn was subject to late-stage, low-angle faulting. These faults were conduits to late mineralizing solutions and oxidation. The result is an oxide-hosted epithermal gold deposit overprinting a retrograde polymetallic skarn. Gold is contained within iron oxide minerals that represent oxidized pyrite within lightly silicified fracture fillings and within sparsely distributed quartz sulfosalt veins.

# 7 Exploration (Item 9)

Most of the exploration work done by MH-LLC at the Mt Hamilton Project through 2013 was borehole drilling and sampling for resource definition. Drilling methods, sample preparation and analysis methods for gold and silver are discussed below in Sections 8 and 9 relating to resource and reserve related drill holes. The majority of drilling was in the Centennial and Seligman resource areas. Several exploration holes were drilled in the Chester Prospect area, south and slightly east of Centennial.

During the 2011 field season, a surface mapping program was conducted in the planned Centennial Pit area. A high-resolution survey of current topography in the Seligman area was completed during the 2012 field season to support geotechnical investigation and mine planning. No recent surface exploration in the form of mapping or sampling has been conducted since 2012.

## 7.1 Relevant Exploration Work

Previous property owners conducted extensive exploration programs on the property, including mapping, surface geochemical sampling, and exploratory drilling. The methods and results from these programs are elaborated in the PEA document (SRK, 2010), and were determined to be conducted according to industry standard practices. The reader is directed to the PEA for additional details.

## 7.2 Surveys and Investigations

Surface geologic mapping was done in the fall of 2011 by a MH-LLC staff geologist. The area surveyed included the vicinity north and east of the Monte Cristo stock. The intent was to identify marker beds favorable for mineralization in the Centennial area, and tie the new data to data in the Westmont surface geology map produced during the 1980's and 90's.

High-resolution topographic data was collected with a Maptek I-Site® laser scanner, by Maptek survey consultants. The existing Seligman Pit areas were the focus of the survey to evaluate structures and slope stability in the existing mined areas. The resulting data was processed by the surveyor, and provided to MH-LLC and SRK for planning and analysis.

### 7.2.1 Procedures and Parameters

Surface geologic mapping data are currently stored in a digital MapInfo database. Mapping methods include measurement of feature orientation and description of materials according to standard geologic mapping practices.

High-resolution topographic data is stored as Drawing Exchange Format (.dxf) files. The spatially-referenced photograph generated during the survey is stored as a digital image file.

## 7.3 Sampling Methods and Quality

Borehole logging and sampling methods are described in the following sections. No other materials were sampled or analyzed during recent additional exploration programs.

## 7.4 Significant Results and Interpretation

Results from the topographic survey have been used for access road construction during drilling, and will also be applied to slope stability studies, mine plans and future geologic modeling.

To date, results from the surface geologic mapping program have not been applied to additional exploration plans. Integration of the new data with previous mapping in the area was completed in 2012.

# 8 Drilling (Item 10)

Recent resource definition and related drilling discussed in this report was completed by MH-LLC in 2011 and 2012. A summary description of the drill hole database, drilling methods and sampling procedures used for these programs is presented in this section. Sample QA/QC analytical results from the 2008-2012 drilling programs are discussed in Section 9. Details presented in this section of the report are specific to the most recent 2011-2012 drilling conducted by MH-LLC since the last Technical Report was filed in 2012 (SRK, 2012).

## 8.1 Drilling History Summary

Drilling completed by previous property owners is extensive, with good coverage in the Seligman and Centennial Deposit areas, as shown in Figure 8.1.1. The details of these previous programs have been presented in previous Technical Reports (SRK, 2010, 2012). Table 8.1.1 summarizes all drilling completed in the Mt. Hamilton Complex area prior to the second quarter of 2011, and compares it to drill holes with gold values for select intervals. Some of the historic drill holes across the Mt. Hamilton complex do not have assay data for all intervals, and therefore do not provide a complete profile of mineralization.

**Table 8.1.1: Drilling Completed at the Mount Hamilton Complex, to Q2 2011**

| Company | Hole Type | Number of Holes | Total Length Drilled (ft) | Number of Holes with Gold Assays | Total Length, Holes with Gold Assays (ft) |
|---|---|---|---|---|---|
| Pre- Ely Gold | RC | 852 | 256,009 | 304 | 112,593 |
| Pre- Ely Gold | Core | 26 | 9,219 | 7 | 2,666 |
| Ely Gold | HQ Core | 5 | 2,241 | 5 | 2,241 |
| Solitario | HQ Core | 12 | 7,121 | 12 | 7,121 |
| Solitario | RC | 6 | 3,625 | 6 | 3,625 |

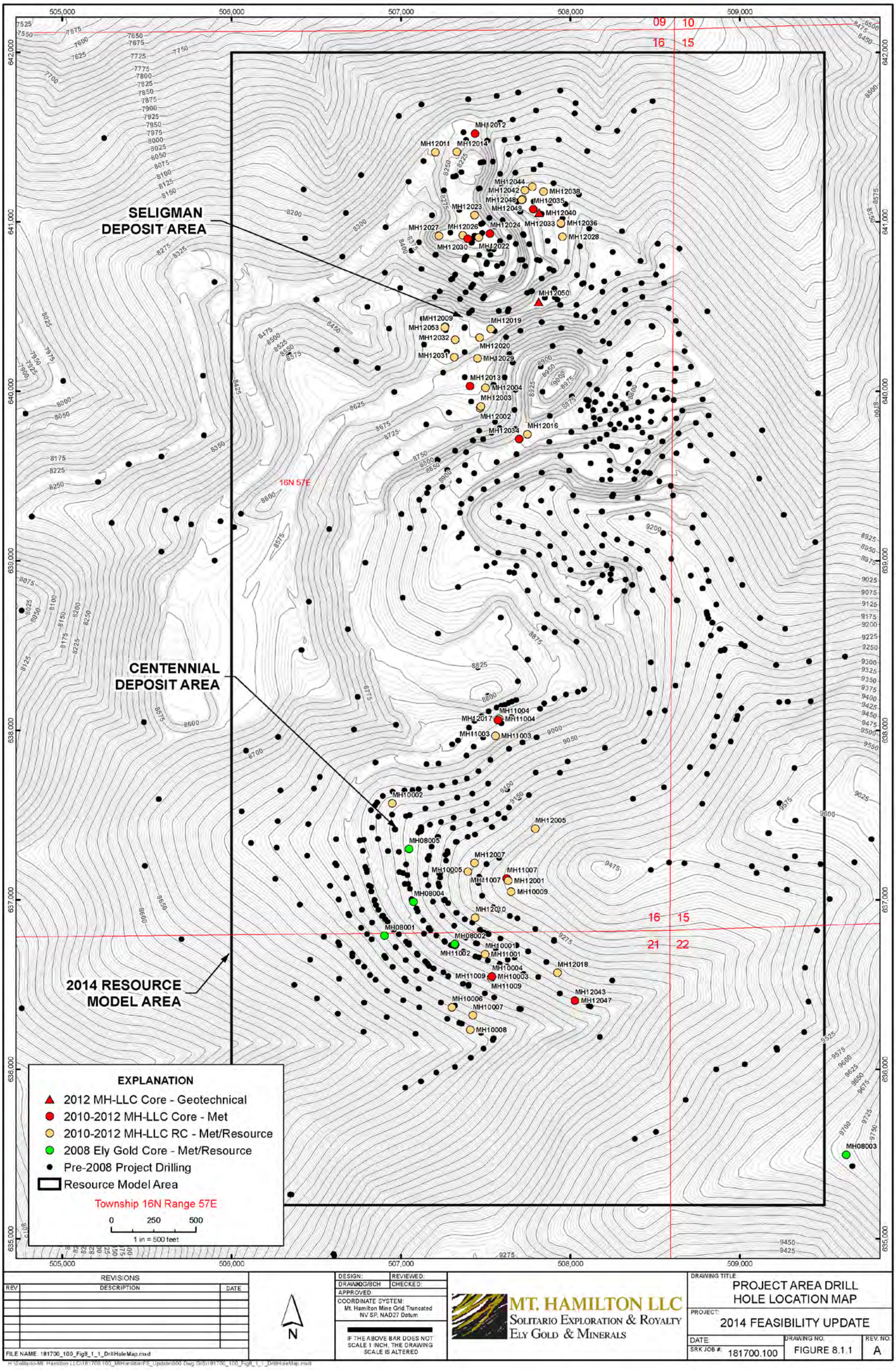
Source: SRK, 2012

## 8.2 Recent Drilling (2011-2012)

Investigation of the Seligman assay database in March 2012 indicated that assay data were available for many intervals missing from the database. SRK, on behalf of MH-LLC, added the data from paper copies of assay certificates to the updated Seligman digital database. Drill holes with data added for the 2012 Seligman resource model are not reflected in Table 8.1.1.

Reverse circulation (RC) and diamond core (DD) drilling comprise the most recent 2011 and 2012 drilling programs. The focus was to provide additional drill coverage in the Seligman Deposit to upgrade it from resource to reserve level. Objectives of this program were to increase sample density, and to obtain sufficient metallurgical and geotechnical data to support feasibility-level engineering and costing. All of these boreholes were also included in the geologic and resource database, and used for resource estimation. Nearly all intervals were sampled and analyzed for gold and silver content. Those not sampled lacked visible evidence of mineralization, and were located outside of the main deposit areas. Most of the sampled intervals were also analyzed for whole-rock composition including major and trace elements.

In late 2011, boreholes for two groundwater monitoring wells were drilled and installed with a mud rotary rig. They are located outside of the resource area, and were not sampled for geochemical testing analysis. However, a geologist logged the cuttings during drilling, and relevant geologic data from them was included in the geologic database for modeling. These boreholes are not discussed further in this document, because they were not drilled or sampled for mineral resource definition.




Source: SRK, 2014

**Figure 8.1.1: Project Area Drill Hole Location Map**

### 8.2.1 Drilling and Survey Methods

Industry-standard techniques were used to drill and collect sample material. Each technique is summarized below. Locations for the 2011 and 2012 drillholes are shown for the Seligman Deposit in Figure 8.2.1.1 and for the Centennial Deposit, Figure 8.2.1.2. Completed borehole collar locations were surveyed by Solarus, LLC, a certified Professional Land Surveyor (PLS) contractor based in Ely, Nevada. Downhole orientation of angled boreholes was measured with either multi-shot camera tools on the drill string or with gyroscopic tools operated by International Directional Services (IDS), Inc. of Elko, Nevada. Vertical boreholes were typically not surveyed, because little deviation in bearing has been observed in previous drilling.

### 8.2.2 Reverse Circulation (RC)

Reverse circulation (RC) drilling consisted of impact- and rotation-driven advance with a hammer bit on the end of the string of double-walled pipe. At Mt. Hamilton, RC drilling was done with a 5 ¼ inch hammer bit. The rig rotates the drill string between about 15 and 60 rpm, and provides pullback force to maintain consistent pressure on the face of the bit. During drilling, water was added for dust control, to comply with United States air quality regulations. Although wet RC drilling is standard practice in the U.S., it can lead to sampling bias. At the Project, the project geologist and technician oversaw the drill crews to ensure the best possible sample quality was obtained.

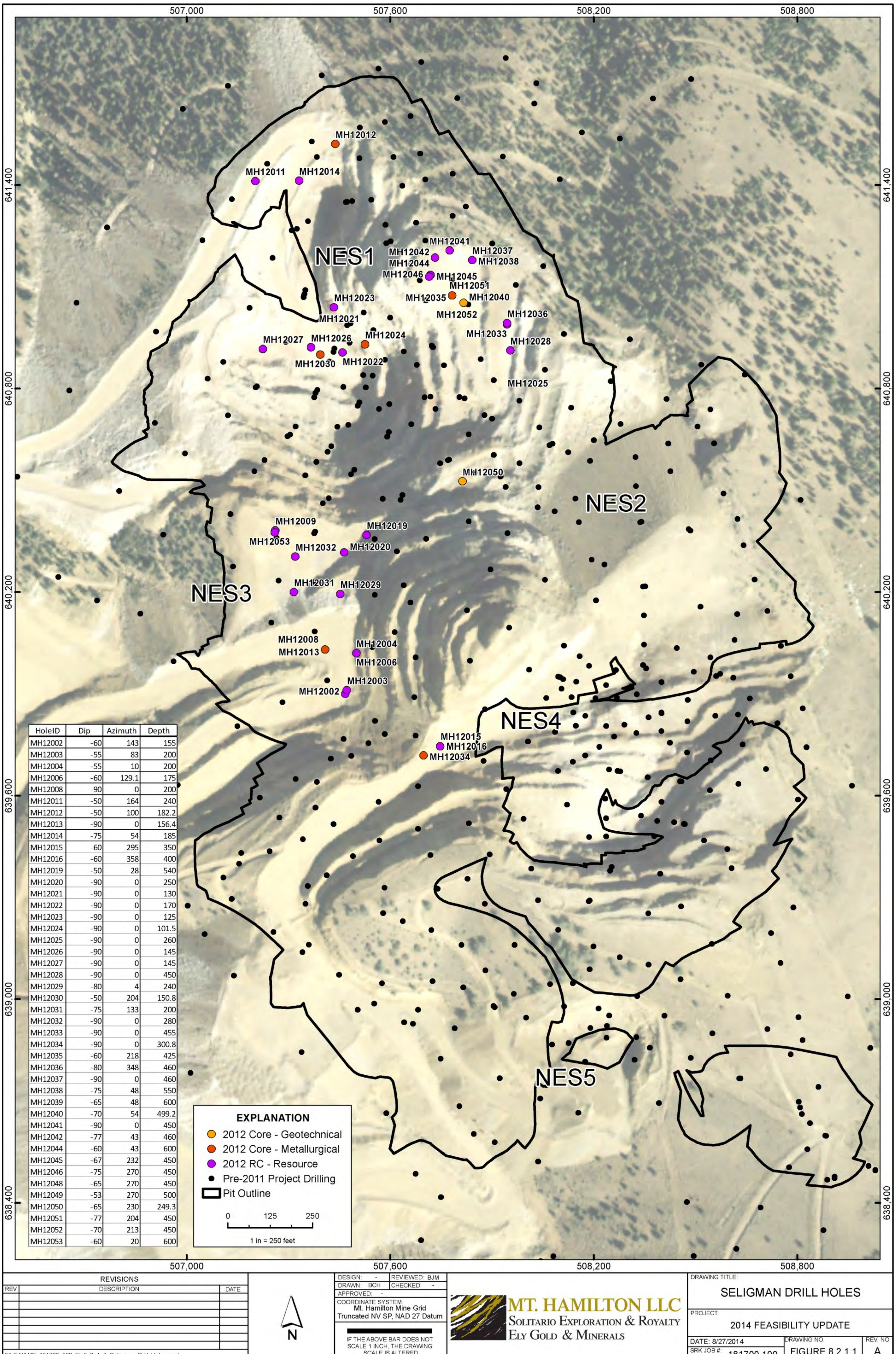
At the Mt. Hamilton Project, RC drilling was used for resource definition drilling, and several boreholes from the 2011 program were used for metallurgical testing. Particle size of RC drill cuttings was typically between approximately 5 mm and 30 mm, and was not suitable for all testing applications. However, RC boreholes cut a larger cross-sectional area than HQ or PQ-diameter diamond core boreholes, and therefore provide larger samples relative to the length of the borehole. Larger samples are often advantageous in precious metals deposits with high variability in metal grade.

The drilling apparatus and techniques used at Mt. Hamilton were typical of RC drilling. Cuttings were circulated out of the borehole during advance with positive fluid pressure. The material passed through a cyclone splitter, then discharged to either sample containers or sumps for drilling fluids.

### 8.2.3 Diamond Core: PQ-diameter

Continuous diamond-bit coring uses a rapidly rotating (350 to 1,000 rpm or greater) single-walled drill string and an annular bit to cut a solid sample. The core is retrieved by a core barrel on a wireline run inside the drill string. The core bit face may be set with diamonds, or diamonds can be impregnated in a matrix that is designed to progressively wear away, to constantly expose new cutting surfaces.

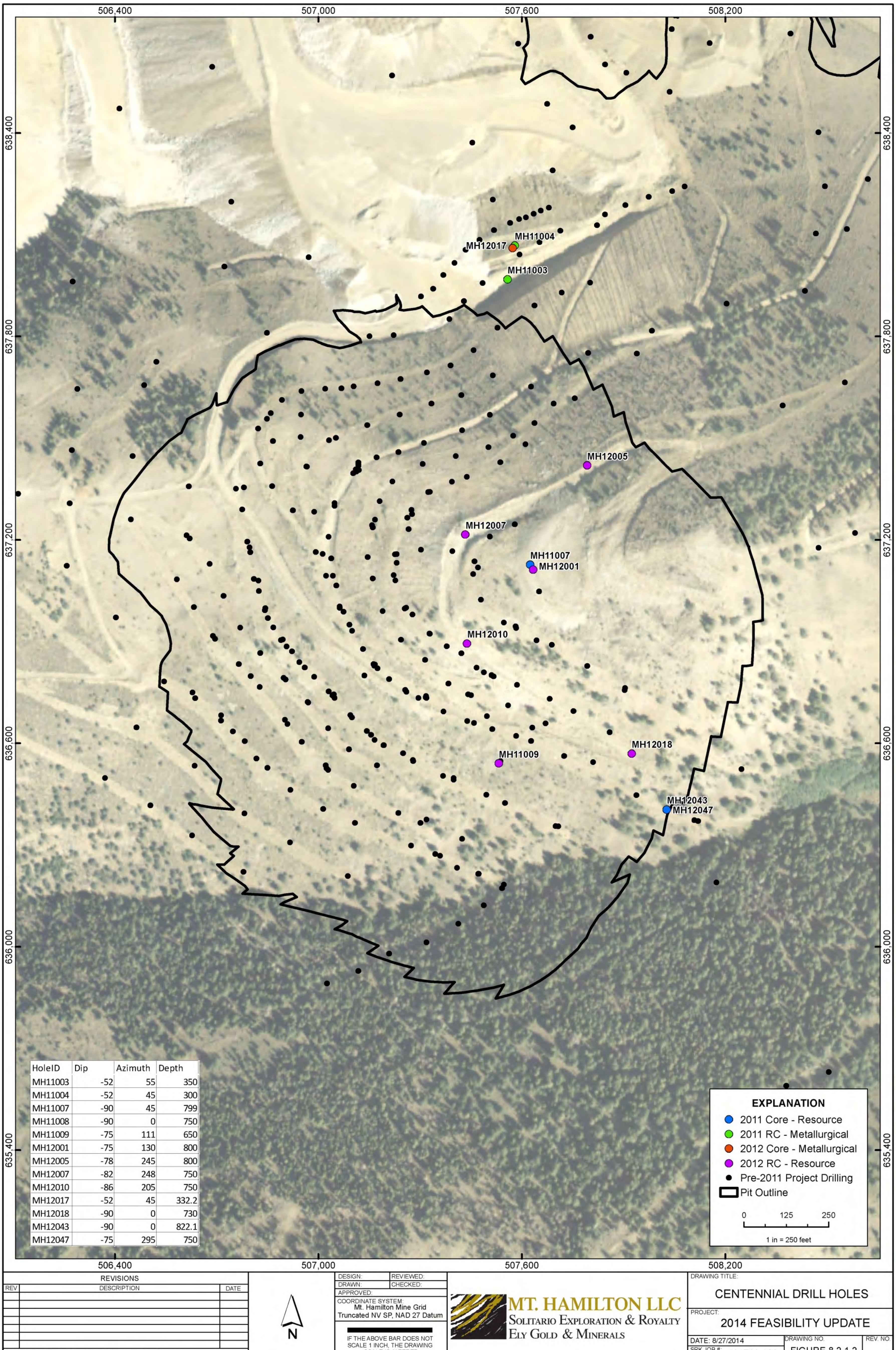
Generally, core drilling is done to obtain material with large particle size, and to quantify features of the rock mass from intact rock. In particular, large-diameter PQ core was drilled at Mt. Hamilton to obtain material for metallurgical testing. In 2012, six PQ-diameter (85 mm, 3 ¾ inch) boreholes were completed in the Seligman Deposit, and one in the northern Centennial Deposit. Bottle roll and cyanide column leach tests were completed to refine recovery estimates for the main material types.




| HoleID | Dip | Azimuth | Depth |
|---|---|---|---|
| MH12002 | -60 | 143 | 155 |
| MH12003 | -55 | 83 | 200 |
| MH12004 | -55 | 10 | 200 |
| MH12006 | -60 | 129.1 | 175 |
| MH12008 | -90 | 0 | 200 |
| MH12011 | -50 | 164 | 240 |
| MH12012 | -50 | 100 | 182.2 |
| MH12013 | -90 | 0 | 156.4 |
| MH12014 | -75 | 54 | 185 |
| MH12015 | -60 | 295 | 350 |
| MH12016 | -60 | 358 | 400 |
| MH12019 | -50 | 28 | 540 |
| MH12020 | -90 | 0 | 250 |
| MH12021 | -90 | 0 | 130 |
| MH12022 | -90 | 0 | 170 |
| MH12023 | -90 | 0 | 125 |
| MH12024 | -90 | 0 | 101.5 |
| MH12025 | -90 | 0 | 260 |
| MH12026 | -90 | 0 | 145 |
| MH12027 | -90 | 0 | 145 |
| MH12028 | -90 | 0 | 450 |
| MH12029 | -80 | 4 | 240 |
| MH12030 | -50 | 204 | 150.8 |
| MH12031 | -75 | 133 | 200 |
| MH12032 | -90 | 0 | 280 |
| MH12033 | -90 | 0 | 455 |
| MH12034 | -90 | 0 | 300.8 |
| MH12035 | -60 | 218 | 425 |
| MH12036 | -80 | 348 | 460 |
| MH12037 | -90 | 0 | 460 |
| MH12038 | -75 | 48 | 550 |
| MH12039 | -65 | 48 | 600 |
| MH12040 | -70 | 54 | 499.2 |
| MH12041 | -90 | 0 | 450 |
| MH12042 | -77 | 43 | 460 |
| MH12044 | -60 | 43 | 600 |
| MH12045 | -67 | 232 | 450 |
| MH12046 | -75 | 270 | 450 |
| MH12048 | -65 | 270 | 450 |
| MH12049 | -53 | 270 | 500 |
| MH12050 | -65 | 230 | 249.3 |
| MH12051 | -77 | 204 | 450 |
| MH12052 | -70 | 213 | 450 |
| MH12053 | -60 | 20 | 600 |

Source: SRK, 2014

**Figure 8.2.1.1: Seligman Drill Holes**




| HoleID | Dip | Azimuth | Depth |
|---|---|---|---|
| MH11003 | -52 | 55 | 350 |
| MH11004 | -52 | 45 | 300 |
| MH11007 | -90 | 45 | 799 |
| MH11008 | -90 | 0 | 750 |
| MH11009 | -75 | 111 | 650 |
| MH12001 | -75 | 130 | 800 |
| MH12005 | -78 | 245 | 800 |
| MH12007 | -82 | 248 | 750 |
| MH12010 | -86 | 205 | 750 |
| MH12017 | -52 | 45 | 332.2 |
| MH12018 | -90 | 0 | 730 |
| MH12043 | -90 | 0 | 822.1 |
| MH12047 | -75 | 295 | 750 |

Source: SRK, 2014

**Figure 8.2.1.2: Centennial Drill Holes**

### 8.2.4 Diamond Core: HQ-diameter

Between the 2011 and 2012 programs, a total of five HQ-diameter (65 mm, 2 ¼ inch) boreholes were completed. The 2011 HQ hole, MH11007, was drilled in the Centennial deposit for resource definition. Two of the four 2012 HQ holes were drilled for exploration in the southern Centennial resource, near the MPO pit wall. All three HQ holes in Centennial were completed with a conventional wireline rig utilizing a single-barrel core tube.

The other two HQ holes drilled in 2012 were in the Seligman area, to obtain geotechnical engineering data for pit slope stability studies. Triple-tube core from these boreholes was oriented, to obtain absolute orientations of structural features and to provide samples for rock strength testing. Results of the geotechnical logging and testing on these cores are presented in a separate document, but assay results are presented below.

## 8.3 Sampling Procedures

Summary statistics of drillhole sampling are presented in Table 8.3.1. Except for the top of the HQ core hole drilled in 2011, all material from the recent drilling programs was sampled and assayed for gold. All holes were analyzed for whole rock composition except for several in 2011. Sampling procedures for the 60 total RC and core drillholes are summarized below.

**Table 8.3.1: Summary of Sampling for Recent Drillholes**

| Interval Summary | 2011 RC (1) | 2011 HQ(2) | 2012 RC (3) | 2012 HQ Geotech (4) | 2012 HQ Exploration (5) | 2012 PQ Metallurgy (6) |
|---|---|---|---|---|---|---|
| Average Length | 5.0 | 3.9 | 5.0 | 4.4 | 3.7 | 4.1 |
| Minimum Length | 5.0 | 1.0 | 5.0 | 1.2 | 0.8 | 1.0 |
| Maximum Length | 5.0 | 5.0 | 5.0 | 6.7 | 8.3 | 7.4 |
| Intervals | 718 | 105 | 3445 | 172 | 431 | 421 |
| Holes | 6 | 1 | 42 | 2 | 2 | 7 |

Source: SRK, 2014

(1) All drillholes are in the Centennial resource area and within the MPO pit volume.
(2) The top 388 ft of MH11007 (in Centennial) were not sampled because the material did not appear to be mineralized.
(3) Five drillholes are in the Centennial area, and the other 37 are in the Seligman area.
(4) Both drillholes are in the Seligman area.
(5) Both drillholes are in the Centennial area.
(6) Six drillholes are in the Seligman area, one is in the northern Centennial area.

### 8.3.1 Drill Core Sampling

Before sampling, drill core was oriented (if applicable), washed, photographed and logged to capture geotechnical parameters. Typically, geologic logging was completed on half core after sampling. All core was halved with a core saw and sampled on intervals determined by the logging geologist. The sampled intervals were continuous, except for several lengths with no visible indication of mineralization.

Rock quality in low-recovery zones can be quantified with geotechnical core logging, in addition to noting circulation loss and low sample mass during drilling. As with RC samples, core samples with low recovery can be biased, and may either over- or under-represent metal abundance. The sample quality should be considered when including low-recovery interval results in the resource. Recovery in the 2011-2012 core drilling program was typically in excess of 90%.

### 8.3.2 Reverse Circulation Drill Sampling

During RC drilling, about half of the drill cuttings were collected from the cyclone discharge port for geochemical testing. The entire drilled length was sampled in continuous intervals. Five foot intervals were collected for Mt. Hamilton RC drilling. This sample length is typical for the gold industry, and suitable for the Mt. Hamilton deposits. A small sub-sample of representative material from each interval was placed in a chip tray, for geologic logging and archives. Both splits were collected separately every 20$^{th}$ sample interval as duplicates to assess the sample quality and potential sampling bias.

## 8.4 Interpretation and Relevant Results (2011-2012)

Significant mineralized intercepts from the 2011 and 2012 drilling programs are summarized below. Most of the intervals from the 2012 program have also been reported in press releases by Solitario Exploration and Royalty Corp., in November and December, 2012. Generally, the criteria for reporting significant intercepts included the following:

- Minimum drilled length 20 ft, or 15 ft in several cases;
- Minimum grade 0.2 ppm (resource CoG);
- Maximum of 15 ft of internal waste less than 0.2 ppm; and
- Length-weighted average grade greater than approximately 0.5 ppm, less for some intervals.

For completeness, not all reported interval meet all of the criteria. The intercepts reported are generally economic grade, with continuity suitable for mining. Due to constraints on surface access, many drillholes were drilled somewhat oblique to the mineralization trend. Therefore, the intercept length is not a representation of true thickness, which is typically about 80% of the mineralized interval length listed.

Not all of the holes drilled in this program were targeting mineralization. Some were for geotechnical or waste rock geochemical sampling purposes.

The results of duplicate RC interval sampling are also presented below and include a discussion of sampling bias at the drill rig, and the quality of RC samples in general.

### 8.4.1 Seligman Area Results

The focus of the 2012 drilling program was the Seligman Deposit. A total of 45 drillholes were completed in this resource- two HQ core, six PQ core, and 37 RC- with a total 14,980 ft drilled length. Mineralized intercepts for all 2012 drillholes in the Seligman resource are summarized in Table 8.4.1.1. The core holes are denoted with superscripted numerals, to distinguish them from the RC holes.

### Table 8.4.1.1: Summary of 2012 Seligman Drilling Results

| Drillhole | Start (ft) | End (ft) | Length (ft) | Length (m) | Au (ppm) | Ag (ppm) | AuEq (ppm) (1) | AuEq (oz/t) (2) |
|---|---|---|---|---|---|---|---|---|
| MH12002 | No Significant Mineralization | | | | | | | |
| MH12003 | No Significant Mineralization | | | | | | | |
| MH12004 | 130 | 200 | 70 | 21.3 | 0.501 | 11.7 | 0.714 | 0.021 |
| MH12006 | 25 | 55 | 30 | 9.1 | 0.545 | 6.7 | 0.667 | 0.019 |
| MH12008 | No Significant Mineralization | | | | | | | |
| MH12009 | 320 | 345 | 25 | 7.6 | 0.518 | 4.7 | 0.603 | 0.018 |
| MH12009 | 425 | 465 | 40 | 12.2 | 2.588 | 3.4 | 2.650 | 0.077 |
| MH12011 | 20 | 45 | 25 | 7.6 | 0.354 | 33.6 | 0.964 | 0.028 |
| MH12012 (3) | 6 | 115 | 108 | 33.0 | 0.754 | 2.0 | 0.791 | 0.023 |
| MH12012 (3) | 139 | 164 | 25 | 7.7 | 0.837 | 0.2 | 0.842 | 0.025 |
| MH12013 (3) | 6 | 42 | 36 | 11.0 | 0.603 | 10.2 | 0.788 | 0.023 |
| MH12014 | 20 | 50 | 30 | 9.1 | 0.839 | 11.0 | 1.039 | 0.030 |
| MH12015 | 0 | 30 | 30 | 9.1 | 0.448 | 4.8 | 0.534 | 0.016 |
| MH12015 | 120 | 200 | 80 | 24.4 | 0.802 | 20.7 | 1.177 | 0.034 |
| MH12016 | 0 | 35 | 35 | 10.7 | 0.311 | 16.2 | 0.605 | 0.018 |
| MH12016 | 205 | 245 | 40 | 12.2 | 0.295 | 12.4 | 0.520 | 0.015 |
| MH12016 | 265 | 315 | 50 | 15.2 | 0.822 | 25.0 | 1.277 | 0.037 |
| MH12019 | 100 | 155 | 55 | 16.8 | 0.343 | 2.9 | 0.396 | 0.012 |
| MH12019 | 225 | 250 | 25 | 7.6 | 0.577 | 7.8 | 0.719 | 0.021 |
| MH12019 | 315 | 345 | 30 | 9.1 | 0.575 | 10.9 | 0.774 | 0.023 |
| MH12020 | 0 | 20 | 20 | 6.1 | 0.358 | 2.9 | 0.411 | 0.012 |
| MH12020 | 60 | 80 | 20 | 6.1 | 0.571 | 8.1 | 0.718 | 0.021 |
| MH12020 | 115 | 145 | 30 | 9.1 | 0.462 | 13.4 | 0.707 | 0.021 |
| MH12021 | No Significant Mineralization | | | | | | | |
| MH12022 | 0 | 40 | 40 | 12.2 | 0.627 | 1.7 | 0.658 | 0.019 |
| MH12022 | 65 | 95 | 30 | 9.1 | 1.041 | 0.6 | 1.052 | 0.031 |
| MH12022 | 110 | 130 | 20 | 6.1 | 0.666 | 1.4 | 0.691 | 0.020 |
| MH12023 | 0 | 25 | 25 | 7.6 | 0.299 | 4.5 | 0.382 | 0.011 |
| MH12023 | 90 | 110 | 20 | 6.1 | 1.341 | 9.0 | 1.503 | 0.044 |
| MH12024 (3) | 66 | 85 | 19 | 5.8 | 0.759 | 3.7 | 0.826 | 0.024 |
| MH12025 | No Significant Mineralization | | | | | | | |
| MH12026 | 0 | 145 | 145 | 44.2 | 0.668 | 0.9 | 0.684 | 0.020 |
| MH12027 | 0 | 95 | 95 | 29.0 | 0.422 | 0.7 | 0.435 | 0.013 |
| MH12028 | 330 | 355 | 25 | 7.6 | 0.749 | 26.7 | 1.235 | 0.036 |
| MH12029 | 35 | 60 | 25 | 7.6 | 0.682 | 5.8 | 0.787 | 0.023 |
| MH12029 | 120 | 155 | 35 | 10.7 | 0.358 | 7.9 | 0.502 | 0.015 |
| MH12029 | 190 | 210 | 20 | 6.1 | 0.559 | 6.6 | 0.679 | 0.020 |
| MH12030 (3) | 9 | 34 | 25 | 7.7 | 1.582 | 1.6 | 1.582 | 0.046 |
| MH12031 | 70 | 95 | 25 | 7.6 | 0.521 | 8.8 | 0.680 | 0.020 |
| MH12032 | 105 | 130 | 25 | 7.6 | 0.510 | 4.0 | 0.583 | 0.017 |
| MH12033 | 335 | 360 | 25 | 7.6 | 2.271 | 7.7 | 2.412 | 0.070 |
| MH12033 | 395 | 455 | 60 | 18.3 | 0.363 | 3.3 | 0.423 | 0.012 |
| MH12034 (3) | 87 | 132 | 45 | 13.7 | 0.631 | 3.9 | 0.702 | 0.020 |
| MH12034 (3) | 206 | 236 | 30 | 9.1 | 0.571 | 8.6 | 0.729 | 0.021 |
| MH12035 (3) | 259 | 372 | 113 | 34.5 | 0.742 | 3.1 | 0.799 | 0.023 |
| | Including: | | | | | | | |
| MH12035 (3) | 319 | 372 | 54 | 16.4 | 1.081 | 5.0 | 1.173 | 0.034 |
| MH12036 | No Significant Mineralization | | | | | | | |
| MH12037 | No Significant Mineralization | | | | | | | |
| MH12038 | 385 | 415 | 30 | 9.1 | 0.492 | 3.7 | 0.559 | 0.016 |
| MH12038 | 440 | 460 | 20 | 6.1 | 0.530 | 3.4 | 0.591 | 0.017 |
| MH12038 | 520 | 535 | 15 | 4.6 | 2.078 | 14.5 | 2.342 | 0.068 |
| MH12039 | No Significant Mineralization | | | | | | | |
| MH12040 (4) | 356 | 375 | 19 | 5.9 | 1.225 | 3.1 | 1.282 | 0.037 |

| Drillhole | Start (ft) | End (ft) | Length (ft) | Length (m) | Au (ppm) | Ag (ppm) | AuEq (ppm) (1) | AuEq (oz/t) (2) |
|---|---|---|---|---|---|---|---|---|
| MH12041 | No Significant Mineralization | | | | | | | |
| MH12042 | 355 | 380 | 25 | 7.6 | 1.423 | 27.2 | 1.917 | 0.056 |
| MH12044 | 400 | 440 | 40 | 12.2 | 0.730 | 10.7 | 0.925 | 0.027 |
| MH12045 | 265 | 300 | 35 | 10.7 | 0.462 | 4.4 | 0.542 | 0.016 |
| MH12045 | 420 | 440 | 20 | 6.1 | 3.026 | 1.8 | 3.058 | 0.089 |
| MH12046 | 305 | 335 | 30 | 9.1 | 0.661 | 6.3 | 0.776 | 0.023 |
| MH12048 | 305 | 380 | 75 | 22.9 | 0.625 | 19.6 | 0.981 | 0.029 |
| MH12049 | 320 | 385 | 65 | 19.8 | 1.124 | 7.2 | 1.255 | 0.037 |
| MH12050 (4) | 102 | 140 | 37 | 11.3 | 0.363 | 9.4 | 0.534 | 0.016 |
| MH12051 | 265 | 330 | 65 | 19.8 | 1.226 | 14.5 | 1.489 | 0.043 |
| MH12052 | 285 | 330 | 45 | 13.7 | 1.202 | 5.8 | 1.308 | 0.038 |
| MH12052 | 395 | 420 | 25 | 7.6 | 0.454 | 12.5 | 0.681 | 0.020 |
| MH12053 | 45 | 90 | 45 | 13.7 | 0.243 | 6.8 | 0.367 | 0.011 |

Source: SRK, 2014
(1) AuEq = gold grade + (silver grade ÷ 55)
(2) Converted from grams per metric tonne (g/tonne), or ppm, to troy ounces per short ton (oz/t). oz/t = ppm ÷ 34.286.
(3) PQ core hole
(4) HQ core hole

## 8.4.2 Centennial Area Results

The 2011 Centennial drilling program targeted sample material for metallurgical testing. There were seven holes completed- one HQ core and six RC- with a total of 4,424 ft drilled. Mineralized intercepts are summarized in Table 8.4.2.1.

**Table 8.4.2.1: Summary of 2011 Centennial Drilling Results**

| Drillhole | Start (ft) | End (ft) | Length (ft) | Length (m) | Au (ppm) | Ag (ppm) | AuEq (ppm) (1) | AuEq (oz/t) (2) |
|---|---|---|---|---|---|---|---|---|
| MH11003 | 110 | 145 | 35 | 10.7 | 0.585 | 12.3 | 0.809 | 0.024 |
| MH11004 | 155 | 215 | 60 | 18.3 | 0.484 | 9.2 | 0.651 | 0.019 |
| MH11005 (5) | 135 | 185 | 50 | 15.2 | 0.555 | -- | -- | -- |
| MH11006 (5) | 360 | 385 | 25 | 7.6 | 0.707 | -- | -- | -- |
| MH11007 (4) | 453 | 484 | 31 | 9.5 | 0.478 | 14.3 | 0.739 | 0.022 |
| MH11007 (4) | 614 | 644 | 30 | 9.1 | 0.406 | 17.2 | 0.719 | 0.021 |
| MH11008 (5) | 475 | 510 | 35 | 10.7 | 0.511 | 13.3 | 0.753 | 0.022 |
| MH11009 | 485 | 540 | 55 | 16.8 | 0.407 | 16.6 | 0.709 | 0.021 |

Source: SRK, 2014
(1) AuEq = gold grade + (silver grade ÷ 55)
(2) Converted from grams per metric tonne (g/tonne), or ppm, to troy ounces per short ton (oz/t). oz/t = ppm ÷ 34.286.
(3) PQ core hole
(4) HQ core hole, all others are RC drillholes
(5) Outside of 2013 Resource Model Extents

Several exploration or confirmation drillholes were completed in the Centennial resource in 2012- two HQ core, one PQ core, and five RC- with a total of 5,734 ft drilled. The 2012 Centennial drilling results are summarized in Table 8.4.2.2.

**Table 8.4.2.2: Summary of 2012 Centennial Drilling Results**

| Drillhole | Start (ft) | End (ft) | Length (ft) | Length (m) | Au (ppm) | Ag (ppm) | AuEq (ppm) (1) | AuEq (oz/t) (2) |
|---|---|---|---|---|---|---|---|---|
| MH12001 | 545 | 575 | 30 | 9.1 | 0.371 | 17.5 | 0.689 | 0.020 |
| MH12005 | 545 | 565 | 20 | 6.1 | 0.367 | 22.4 | 0.774 | 0.023 |
| MH12007 | 340 | 460 | 120 | 36.6 | 1.46 | 10.5 | 1.65 | 0.048 |
| MH12010 | 425 | 550 | 125 | 38.1 | 1.816 | 18.5 | 2.153 | 0.063 |
| MH12017 (3) | 78 | 141 | 63 | 19.3 | 0.285 | 8.8 | 0.446 | 0.013 |
| MH12017 | 159 | 217 | 58 | 17.7 | 0.861 | 11.3 | 1.067 | 0.031 |
| MH12018 | 575 | 635 | 60 | 18.3 | 1.002 | 24.3 | 1.443 | 0.042 |
| MH12043 (4) | 228 | 242 | 14 | 4.1 | 1.331 | 1.5 | 1.358 | 0.040 |
| MH12043 (4) | 589 | 615 | 26 | 7.8 | 0.835 | 135.0 | 3.290 | 0.096 |
| MH12047 (4) | 575 | 601 | 27 | 8.1 | 3.607 | 56.0 | 4.626 | 0.135 |

Source: SRK, 2014
(1) AuEq = gold grade + (silver grade ÷ 55)
(2) Converted from grams per metric tonne (g/tonne), or ppm, to troy ounces per short ton (oz/t). oz/t = ppm ÷ 34.286.
(3) PQ core hole
(4) HQ core hole, all others are RC drillholes

## 8.4.3 RC Interval Duplicate Results

The focus of duplicate RC interval sampling was to identify sampling bias at the drill rig. About half of the borehole material was collected for analysis, and it is assumed that it is representative of the entire mass. In theory, the material is not fractionated in the cyclone and the sampled material is the same as the un-sampled material. If material is preferentially sorted in the cyclone, the duplicate sample pairs will show bias. Sample preparation and analysis techniques are the same for each original and duplicate sample pair.

Duplicate interval samples were collected for every 20th interval, or 100 ft drilled. There are 34 duplicate pairs from the 2011 program, and 19 of these have silver results in addition to gold results. Two of the six RC holes were analyzed for gold only. From the 2012 program, all 146 drill duplicate pairs have total gold and silver analytical values. One of these has cyanide-soluble gold and silver results, but these data are not presented in this report.

SRK used plots of percent relative difference (PRD) vs. average value to analyze the difference between the original and duplicate values as a percentage of the average value of the sample pair, according to the following equation:

*PRD = (Original Value – Duplicate Value) / (Average Value) * 100*

PRD values for 2011 duplicate sample pair plots showed highly variable gold content and less variability in silver. The industry standard target PRD is +/- 30% of the average value of the pair. Although the silver values for ore-grade samples are within 30% PRD, most of the ore-grade gold values are outside of the target range. Additional analysis is needed to show bias in the sample pairs.

PRD for 2012 duplicate sample pairs showed approximately symmetrical distribution in gold and silver values. Most of the ore-grade gold value differences are within 30% of the average value for the pair. Most of the sample pairs have average silver values below the approximate resource CoG, 4 ppm. Sample pairs with average silver values greater than 4 ppm have high variability, but most have PRD within 30% of the average value.

Quantile-Quantile (Q-Q) plots were also used to detect bias that may not be apparent in PRD plots. They were used to determine if the two data sets come from populations with a common distribution (ASL, 2012).

Q-Q plots for 2011 gold and silver sample pairs showed that low-grade duplicate samples were higher grade than the original samples, and on average, the duplicate sample values are about 68% lower grade than the original sample values. High variability illustrated in the PRD plot is evident in the Q-Q plot, as the correlation coefficient value ($R^2$) of the data set is less than 0.95. The Q-Q plot for silver values shows no bias and low variability between original and duplicate samples.

Q-Q plots were analyzed for 2012 gold and silver sample. This data set was much larger than the 2011 data set, and showed better correlation between original and duplicate values. Gold values show no significant bias and little variability. Silver values varied more for higher-grade samples than those for lower grades, and duplicate samples had an apparent low bias. If the highest-grade pair of values is excluded, the average distribution of paired values is near parity and the variability of the data set decreases significantly.

Based on the statistical analysis of original and duplicate sample values, there is no apparent systematic sampling bias in the recent RC drilling.

# 9 Sample Preparation, Analysis and Security (Item 11)

For all Ely Gold and Solitario drilling programs, independent and reputable laboratories performed all steps of the sample preparation and analysis process. Samples were delivered to either of two laboratories in Sparks or Elko, Nevada by Company staff or consultants. Samples from the 2008 drilling program were prepared and analyzed for gold, silver and bulk geochemistry at ALS Chemex (ALS); samples from the 2010 and 2011 drilling programs were prepared and analyzed with comparable methods at American Assay Laboratories (AAL).

SRK representatives have made numerous visits to the MH-LLC core storage facility in Ely, Nevada. The systems and procedures in place from previous drilling programs were adequate to maintain sample integrity in 2011-2012. Although SRK did not observe the drilling and sampling procedures during the recent programs, the same procedures and personnel were used and we are confident in the quality of the samples.

All samples were in the custody of MH-LLC until they were delivered to the American Assay Laboratories (AAL) preparation lab in Elko, Nevada. AAL maintained custody of all sample material until it was returned to the MH-LLC office in Ely, Nevada for permanent storage. The procedures and logistics of sampling are elaborated below.

## 9.1 Analytical Methods

All samples submitted were analyzed for gold using a fire assay process with atomic absorption spectroscopy (AAS), and samples from most drill holes were also analyzed for whole-rock geochemistry including silver, with a two-acid digestion, Inductively-Coupled Plasma Mass Spectroscopy (ICP-MS) process. Pulp material consumed by these processes is 30 g and 0.5 g, respectively. Select mineralized intervals were later analyzed for cyanide-soluble (CN) gold and silver with a cyanide extraction and AAS solution analysis. This CN-soluble analysis was done at AAL for all recent samples, and consumed 30 g of pulp sample.

## 9.2 Security Measures

During the drilling programs, MH-LLC controlled access to drill sites via a locked gate on the main property access road. Only MH-LLC agents and drilling contractors handled the drill samples before they were delivered to the lab. Agents of MH-LLC transported all sample material to the secure core storage facility in Ely for processing and temporary storage before delivering it to the analytical preparation lab. After the samples were delivered to the AAL preparation lab in Elko, Nevada, AAL maintained custody until the samples were assayed and released in sealed pulp form back to MH-LLC for transport to Ely and permanent storage. Remaining sample materials are stored indoors in MH-LLC's core facility, and are available for review or additional testing, if needed.

## 9.3 Sample Preparation

Sample preparation was done at ALS for 2008 drill samples, and AAL for subsequent drilling samples. Prep procedures at the two labs are comparable, and are detailed below.

### 9.3.1 Laboratories

**ALS Chemex (now ALS Minerals) Labs**

Samples from the 2008 drilling program were delivered to ALS Chemex Labs (Chemex) in Elko or Reno, Nevada for both sample preparation and analysis. Chemex held ISO 9002:1994 and ISO 9001:2000 certifications for its laboratories in North America when the Ely Gold samples were processed.

The standard preparation procedure used for Ely Gold samples included drying the samples to remove excess moisture, fine crushing samples with a jaw crusher to at least 70% of the volume less than 2 mm, split off 250 g with a riffle splitter and pulverize the 250 g split to better than 85% passing 75 microns.

**American Assay Labs**

Samples from 2010-2012 drilling programs were delivered to American Assay Labs (AAL) in Sparks, Nevada for preparation and analysis. Since incorporation in 1987, AAL has provided laboratory services in North and South America to all major mining companies and has documentation for ISO 17025 certification. AAL has participated in all CANMET-PTP MAL studies since their inception in 1998.

The standard sample preparation procedure used for Solitario samples included drying the samples to remove excess moisture, crushing with a jaw crusher to 90% passing 10 mesh (2 mm), split off 300 g with a Jones splitter (coarse reject), and pulverize an additional 300 g split to 90% passing 150 mesh (about 105 microns).

## 9.4 QA/QC Procedures and Results

Analytical quality assurance procedures included analysis of standard samples with certified gold and silver values, analysis of barren material, and duplicate analysis of randomly selected prepared pulp samples. Both ALS and AAL insert samples of standard reference material (SRM) into the sequence of samples for internal sample quality control. Results from internal standards are verified by the lab before the analytical results are finalized and released to the client.

MH-LLC also inserted SRM samples in the drill sample sequence, and received duplicate analysis of randomly selected pulp samples for analysis run at AAL. Results of duplicate analysis and the Company's SRM samples are discussed in this section and are used to assess the quality of sample preparation and analysis.

### 9.4.1 Standard Reference Materials (SRM)

Analysis of SRM samples with known metal abundance is part of the exploration industry standard practices to assess the quality of sample preparation and analytical procedures and to verify results that serve as the foundation of resource models. The SRM used for drill hole samples at Mt. Hamilton were made from natural materials, and all steps in the preparation and data analysis process were overseen by Shea Clark Smith, C.P.G., at MEG Labs in Washoe City, Nevada (MEG, 2013). Materials used in this evaluation have statistically significant mean and standard deviation values, which are shown in Table 9.4.1.1. No SRM samples for cyanide-soluble (CN) gold and silver analysis were used.

Blank samples are barren material known to be absent of the metals of interest at the applied method detection limits. Coarse blank materials were used for all drilling programs; initially, a certified rhyolite was used, until the Solitario staff began using landscape marble rock instead. Metrics to evaluate the "Marble Blank" results are based on the method detection limits because this material is assumed to be void of precious metals and it does not have certified mean values.

**Table 9.4.1.1: Certified Values of Standard Reference Materials used at Mt. Hamilton**

| Sample Type | Average Gold | Standard Deviation Gold | Gold Min. 95% Confidence Interval | Gold Max. 95% Confidence Interval | Average Silver |
|---|---|---|---|---|---|
| MEGAu.09.01 | 0.68 | 0.01 | 0.54 | 0.83 | 9.58 |
| MEGAu.09.03 | 2.09 | 0.16 | 1.75 | 2.42 | 17.22 |
| MEGAu.09.04 | 3.39 | 0.2 | 2.99 | 3.8 | 26.27 |
| Prep Blank | 0.009 | 0.006 | - | - | 0.1 |
| S104007x | 0.75 | 0.01 | 0.71 | 0.78 | 40 |
| S104011x | 7.12 | 0.3 | - | - | 0.6 |
| S105005x | 2.41 | 0.08 | 2.25 | 2.58 | 4 |

Source: SRK, 2014

**2008 Drilling**

The three types of reference materials used for 2008 drill samples were provided to Augusta Resources by MEG Labs. These are identical blanks and standards to those used during the 1996 and 1997 drill programs. The analytical values of these materials are not certified because they did not have complete round-robin analysis of at least 20 samples at least four different analytical labs. MEG Labs was contacted to verify the quality of these samples and to request the mean values of select elements. No mean values for gold were calculated, so these seven samples are not applicable for the Centennial resource verification. At that time, SRK was informed that these data were confidential and the samples were intended for internal verification of Augusta's analytical results. Thus, the results from these SRM samples are not considered in the assessment of the 2008 drilling results.

The suite of SRM samples used for 2008 drill samples included a coarse (>1 inch clast size) rhyolite Prep Blank sample. Material of this size fraction passes through all steps of the sample prep process, and is preferable to the standard silica sand material that many exploration companies use. Coarse blank sample material was used to ensure the sample preparation equipment is cleaned properly, in addition to ensuring a systematic high bias in analytical results does not exist.

**2010-2012 Drilling**

SRM samples for these drilling programs were from the certified MEGAu09XX series of materials for gold and silver. Blank materials used were coarse rhyolite for the initial 2010 drilling, and coarse marble for the balance. All results are applicable to assessing the quality of the analytical data from 2010-2012 drilling programs.

### 9.4.2 Quality Control Sample Implementation

Standard Reference Material (SRM) samples were included in the drill sample sequence at a frequency equal to or greater than the minimum industry requirement, and provide robust data verification for FA gold and ICP silver results. A blank sample and mineralized SRM sample were inserted in the drill sample sequence after every 15 drill samples for core, and after every 20 drill samples for RC.

### 9.4.3 Quality Control Sample Results

Quality control results and analyses for the 2008-2011 drilling programs were reported previously (SRK, 2012). Analytical data and graphical presentation of the 2012 quality control results are included in the 2014 FS document (SRK, 2014a) that supports this Technical Report. The following is a summary of those results.

**Standards**

Six SRM were used during the 2012 drilling program. Three of these were the remaining SRM from the 2011 program. The three new SRM have comparable gold grades to the others. Although silver values in them vary, all mean silver values are greater than 4 ppm. Generally, SRM results from the 2012 program were within the target value ranges, and appear to be symmetrically distributed about the mean values. Distribution of reported values is much different than that of the 2011 results, and suggests that American Assay Lab's (AAL) analytical quality improved. Results from the lowest-grade SRM appear to be biased slightly low, but the mid-grade SRM results are all within the target range and appear to be distributed symmetrically about the mean value. The distribution of silver results from 2012 is similar to that of 2011 results. Generally, values are high compared to the certified mean values, but about 80% of them are within the target range of values. Reported values for the high- and mid-grade SRM are generally within the targeted ranges and do not appear to be biased. However, nearly all reported values for the low-grade SRM are less than the certified mean value, and many are less than the target minimum analytical value. Gold results for the same SRM appear to have low bias also. This pattern could be caused by incomplete sample dissolution, or analytical methods that vary from those used during the process to determine the mean value.

**Blanks**

All but four of the 2012 blank samples were composed of coarse landscape marble. There are four samples of silica sand, material MEG Blank 12.01, included in the 2012 sample batch. These samples performed similarly to the coarse marble samples. Only two of the 2012 blank samples had reported gold values greater than ten times the method detection limit. Silver values for all samples were less than five times the method detection limit, and most were reported below it. These results suggest that sample preparation equipment is adequately cleaned between samples, and no apparent sample mixing occurred.

**Assay Duplicates**

Assay duplicate results include total gold and total silver, and cyanide-soluble gold and silver from the 2012 program. The target PRD range for duplicate analysis results is within 10% of the original value.

Most of the paired results were close to the target range for economic-grade samples. Sample pairs with average grades below resource cut-off values show greater variability. As values approach the method detection limit, small differences between the original and duplicate values are a greater percentage difference of the average. This trend is typical and expected for these data sets. Total gold (fire assay) duplicate results from the 2012 program showed the most variability relative to the average pair value, but statistical analysis of the data sets shows very strong correlation and near parity between the two data sets. As expected, the assay duplicate pairs have less variability than the RC sample duplicate pairs.

### 9.4.4 Sample Quality Conclusions

**Recent Data**

Standard and blank samples generally show accurate and repeatable results, and the quality of analytical data in the 2012 program appears to be good. Assay duplicate pairs generally show repeatable results within industry-standard tolerance ranges, especially for samples with mineralization of economic importance. Because assay QC results were reviewed after the analysis programs were complete, no re-analysis of questionable blank or standard samples was done. At this time, we do not recommend re-analysis of any samples. Candidates for re-analysis or verification would include brackets of sample intervals including two standard samples that greatly vary from the acceptable values range, and several blank samples with reported values well above 10 times the method detection limit.

The proportion of blank and standard samples inserted in the drill sample sequence exceeds the number required to verify both fire assay and ICP results. Although MH-LLC did not provide standard samples with certified values for cyanide-soluble gold or silver, the results were verified by comparing them to the reported total values and to the material characteristics noted in geologic logs. The quantity and quality of duplicate analysis pairs are adequate to show repeatable analytical procedures and results.

The results discussed above indicate that the analytical quality for recent drillhole samples is adequate and suitable for use in resource estimation. Due to improved sample quality and analytical technology, recent results have greater precision than historical results.

**Current Data vs. Historic Data**

In a visual comparison of recent and historical drillhole intercepts for nearby drillholes, SRK found that historical results are generally biased low. Cross sections and additional information about this comparison can be found in Appendix A. Historical samples with CN-Au values greater than 0.009 oz/t were fire assayed. For samples without fire assay results, the CN-soluble gold value is used in the database. It is reasonable to expect a low bias for reported low-grade gold values because many of them were only assayed by CN-soluble method results.

Visual comparison of historical to modern drilling results in Seligman and Centennial confirms that historical drillholes locate mineralization correctly, and their reported gold values are comparable or slightly less than fire assay gold values in nearby recent drillholes, making historical intercepts conservative for application in resource estimation. Recent drilling results have confirmed the adequacy of the historical drillhole data, which, even after recent drilling, still comprises the majority of the resource estimation database. The overall conclusion is that historical drilling is suitable for use in resource estimation.

## 9.5 Opinion on Adequacy

The proportion of control samples to drill core samples in the 2008, 2010, 2011 and 2012 drilling programs met or exceeded industry standards. Most results for control samples and, if available, duplicate analyses, indicate accurate and repeatable data were provided by both ALS Chemex and American Assay Labs.

It is the opinion of SRK that implemented controls on analytical QA/QC meet industry standard practice. Results show that the analytical data is of quality suitable to be used for mineral resource estimations.

# 10 Data Verification (Item 12)

All available assay and multi-element geochemistry data for the Mt. Hamilton Project was verified in preparation for the 2014 resource model. SRK's verification of gold assay and multi-element geochemistry data used in previous resource estimations is described in detail in SRK, 2013a. The focus of the following section is the data verification completed for the 2011 and 2012 drilling results, which had not previously been verified or used in resource estimation. The 2014 Mt. Hamilton resource estimation is the first instance these data were applied for disclosure of mineral resources. SRK's data verification procedures, and limitations of the data, are elaborated below.

## 10.1 Procedures

AAL sent analytical results directly to MH-LLC and SRK via email in .xls file format. There was no hand entry of analytical data. All results were reported in parts per million (ppm), equivalent to grams per metric tonne, for trace elements and percentage for major elements. In a copy of each file, MH-LLC or SRK paired sample IDs with drillhole intervals to include in the model database. Duplicate and standard samples were coded by SRK to extract for additional analysis, and exclude from the model database. Results below method detection limit (bdl) were reported by AAL as negative values equal to the lower method detection limit. The capabilities of the modeling software determined how SRK processed the bdl values before appending them to the modeling database.

Because the resource model database is in troy ounce per short ton (oz/t) units, reported ppm gold and silver results were converted to oz/t in new data fields. The rest of the geochemistry data set was used with the reported ppm or percentage units. When the conversion was completed, results reported below detection limit were set equal to zero and flagged with an integer code to document the reason for the zero value. Unsampled intervals with sample recovery were also assigned zero, and flagged to show they were not sampled. Intervals with no recovery were assigned null values, to allow interpolations to pass through them. The gold and silver values in oz/t appended to the model database were numeric, and no further data formatting was needed for them.

SRK used two software packages during the resource modeling process. Leapfrog Geo® was used to generate geologic solids and mineralization grade shells for gold, silver, arsenic, sulfur and antimony, but resource estimation was completed with MS3D® and MS Torque® from Mintec of Tucson, Az. Leapfrog® software can accommodate negative and non-numeric data in assay data fields and allows the user to substitute or omit non-numeric values from any interpolation or compositing tasks. MS Torque® is a database module associated with MineSight® 3D, built on a Microsoft SQL Server platform and similar to the database program, Microsoft Access. MS Torque® requires the data type and precision of numeric data to be specified before data fields can be populated. Numeric fields can be used for calculations, and therefore, assay data fields are specified as numeric. Consequently, non-numeric values are not accommodated the way they are in Leapfrog®, but because the gold and silver values were standardized for the model database, all were imported to the numeric data fields. Original reported assay values may be imported as text, or with validation criteria that allows small negative values.

Imported values were verified in Torque® data tables and in drillhole views of data in MS3D®. In Leapfrog®, values were verified in data tables and visually in the 3D workspace. SRK compiled geologic data for an updated geologic model. Comparison of drillhole geology and assay data was one of several approaches used to verify the drillhole intervals assigned by sample ID.

## 10.2 Limitations

SRK was missing the sample sheets from MH12043, 0.0-501.9 ft, which was sampled later than the rest of the drillhole. An intercept between 228.2 and 241.7 ft averages 1.331 ppm gold and 1.8 ppm silver. SRK did not realize these intervals had been sampled until after the resource model was complete, and did not know that assay results were available. Consequently, the top of this drillhole was not included in the model database by the data cut-off date. No other missing data was identified for the two most recent drilling programs.

Data integrity and security are issues that SRK has addressed throughout the project history. The assay database should have limited access to edit data, accidentally or otherwise, and should be applied to resource estimation with minimal processing. SRK has considered several database management solutions, and recently adopted the Torque®-based system in favor of Access. SRK believes the current workflow of using a Torque® database directly connected to the resource estimation software is an adequate and reliable data management solution.

Without a dedicated ODBC program and a full-time database administrator to manage it, laboratory results must be sorted and formatted before they are added to the main database. The risk of mistakes in edited spreadsheets is mitigated by minimizing the amount of data manipulation, and by editing copies of laboratory results files. The original files are saved on SRK's secure server and are also available from recipients' email or the laboratory. After the data is loaded to the Torque® database on the secure server, editing permissions are limited to the SRK database administrators.

## 10.3 Data Adequacy

Analytical methods were adequate to characterize the gold and silver content in the boreholes and the results are suitable to use for resource estimation. Multi-element analysis results were used for additional material characterization, and appear to be reasonable. Because the results for arsenic, sulfur, and other modeled elements are not reported in a resource estimate, they are not subject to stringent quality control criteria. The analysis results from the 2011 and 2012 drilling programs were verified and are adequate and suitable to use for mineral resource estimation.

# 11 Mineral Processing and Metallurgical Testing (Item 13)

## 11.1 Introduction

This summary of metallurgical results and applications was prepared to provide an overview of the metallurgical understanding of the Project. Detailed supporting metallurgical test results are provided in the FS (SRK, 2014a). This Section provides a description of Mt. Hamilton mineralization and metallurgical characterization, including:

- Mineralogy and metallurgical ore types;
- The representativeness of the ore samples to the deposit;
- Metallurgical ore characterization;
- Metal recovery and recovery rate projections; and
- The basis of the process design criteria.

Mount Hamilton has historically been viewed as two ore deposits: 1) the Centennial Deposit; and 2) the Seligman Deposit. Exploration and metallurgical work suggests that these two deposits may be interconnected and potentially mined within a single pit. The two deposits are composed of similar host rocks, including skarn, hornfels and igneous granodirorite. The material planned to be mined at Seligman has a greater proportion of igneous rock to skarn compared to Centennial. Oxide mineralization in all rock types responds metallurgically the same in the two deposits.

Since the 2012 FS, a significant amount of additional metallurgical test work was completed for the Seligman Deposit, supporting recovery estimates, and allowing this material to be classified as reserves. Most of the text and analyses in Sections 11.1 through 11.4.6 were presented in the 2012 FS but have been paraphrased here for completeness. The remainder of this Section from item 11.4.7 through 11.6 contains details of the 2013 metallurgical test work (not previously reported) and the application of those results to the current 2014 FS.

## 11.2 Mineralogy and Metallurgical Ore Types

The Mt. Hamilton gold and silver mineralization is hosted predominately by calc-silicate skarn with lesser amounts of hornfels and igneous granodiorite. Original limestones and calcareous siltstones were metamorphosed to skarn and hornfels by the Seligman Stock, a volumetrically significant intrusion of igneous rock situated beneath and between the two ore deposits. Recent test work has shown that skarn and igneous rock types respond the same metallurgically. The rock types are further subdivided into oxide, transition and sulfide ore types, each of which respond distinctly to cyanide leaching.

Gold mineralization occurs in discrete particles of fine gold or electrum in the Mt. Hamilton ores. Assay replicability is 90% for gold in the material tested.

Silver mineralization is highly variable in content and metallurgical response in the material tested. The metallurgical tests indicate that silver mineralization is not metallurgically associated with gold mineralization, though they are often spatially coincident. Silver mineralization is in the form of sulfosalts or jarosites.

It will be shown that the skarn and igneous lithologies responded metallurgically similar as well as by rock type; therefore, the metallurgical ore types were distinguished into three categories based on their extent of oxidation alone. The extent of oxidation was determined primarily through Rmax as well as through an estimate of percent pyrite from the geologic core logs. Rmax is defined as CNAu/FAAu = gold Recovery Maximum (Rmax).

- Oxide – gold Rmax values above 70% and pyrite below 0.5%.
- Transition – Rmax grade equivalent above a cut-off, 0.14 ppm, and less than 70% gold Rmax and pyrite between 0.5% and 2.5%.
- Sulfide – Rmax grade equivalent below a cut-off of 0.14 ppm and pyrite above 2.5%.

## 11.3 Sample Representation to the Deposit and Compositing

The locations for 19 metallurgical PQ diamond drill core holes were selected to represent the average ore types and feed grade of the deposit. The core hole locations were spatially selected which represent approximately 1.2 Mt of ore per drill hole. Additionally, intervals from 47 reverse circulation holes were selected to conduct the variability bottle roll test work. This represents approximately 0.3 Mt per sample.

In general, the column composite samples consisted of PQ core intervals, and the interval selection criteria were based on: 1) the deposit AuEq CoG; 2) the AuEq average deposit grade; and 3) equally weighting each hole going into the composite to achieve spatial distribution. These criteria provided the mechanism where-by the column tests were representative of the deposit’s average metallurgical ore types.

In contrast, the variability bottle roll samples (from core and reverse circulation drill hole intervals) were selected to include all material types (not intended to be representative of the deposit average) to assess the potential metallurgical ore type variability of the deposit.

Table 11.3.1 lists drilled materials used for metallurgical characterization.

**Table 11.3.1: Summary of Metallurgical Test Work for Mt. Hamilton**

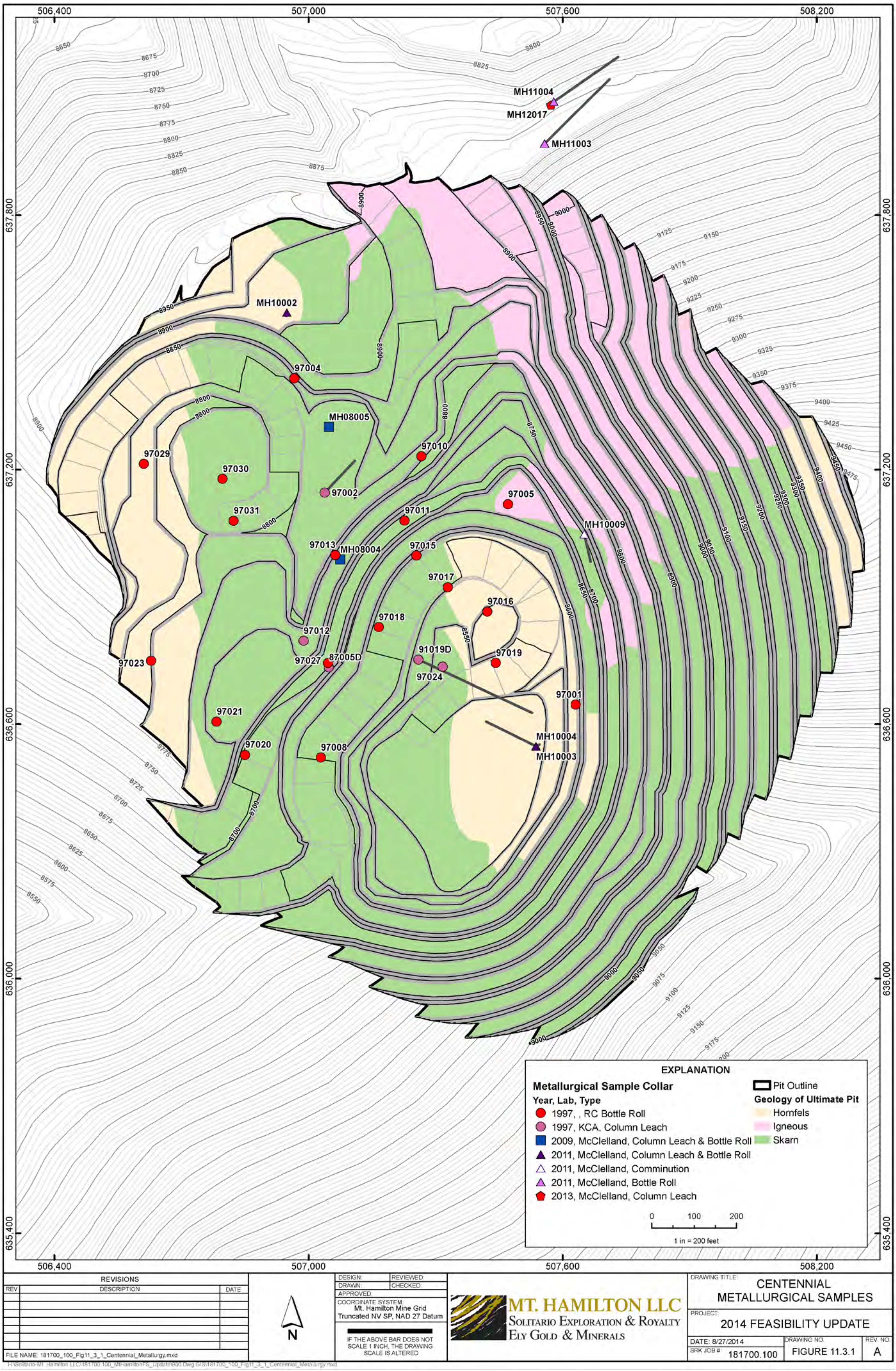
| Program | Type | Drill Holes | Area and Purpose |
|---|---|---|---|
| 2013 MLI | Skarn Oxide (Column) | MH12012,24,30,35 | Seligman Resource Recovery Model |
| | Skarn Transition (Column) | MH12012,24,30,35 | Seligman Resource Recovery Model |
| | Skarn Sulfide (Bottle Roll) | MH12012,24,30,35 | Seligman Resource Recovery Model |
| | Igneous Oxide (Column) | MH12013,34 | Seligman Resource Recovery Model |
| | Igneous Transition (Column) | MH12013,34 | Seligman Resource Recovery Model |
| | Skarn Oxide (Column) | MH12017 | Centennial N. Resource Recovery Model |
| | Ox.,Trans., Sul. | 28 RC Holes | Variability |
| 2011 MLI | Igneous Oxide (16 Bottle Rolls) | MH11003, 04 | Centennial Igneous Characterization |
| 2011 MLI | Skarn Oxide (Column) | MH10002 | Centennial Resource Recovery Kinetics |
| | Skarn Oxide (Column) | MH10003, 04 | Centennial Resource Recovery Kinetics |
| | Skarn Mixed (32 Bottle Rolls) | MH10002, 03, 04 | Centennial Resource Recovery Model |
| 2009 / 10 MLI | Skarn Oxide (Column) | MH08004 | Centennial Resource Recovery Kinetics |
| | Skarn Oxide (Column) | MH08005 | Centennial Resource Recovery Kinetics |
| | Skarn Oxide (64 Bottle Roll) | MH08004, 05 | Centennial Leach Feed Size |
| 1997 KCA | Skarn Oxide (9 Columns) (5 Bottle Roll) | MH87005D MH91019D MH96002 MH97002 MH97012 MH97024 | Centennial Resource Leach Feed Size Recovery Kinetics |
| Pre 1997 (via Carrington) | Skarn Mixed (32 Bottle Rolls) | (19 RC Holes) | Scoping, Variability |
| Pre 1997 (Other) | 4 Columns 28 Bottle Rolls | NA | Scoping |

Source: SRK, 2014

The early column composites in the 2012 FS were slightly higher in feed grade than the average for the deposit, so a feed grade vs. recovery relationship was developed to enable a recovery adjustment for 2012 FS recovery estimate. For the 2014 FS, a recovery model by cyanide soluble assay has been developed, and will be discussed further.

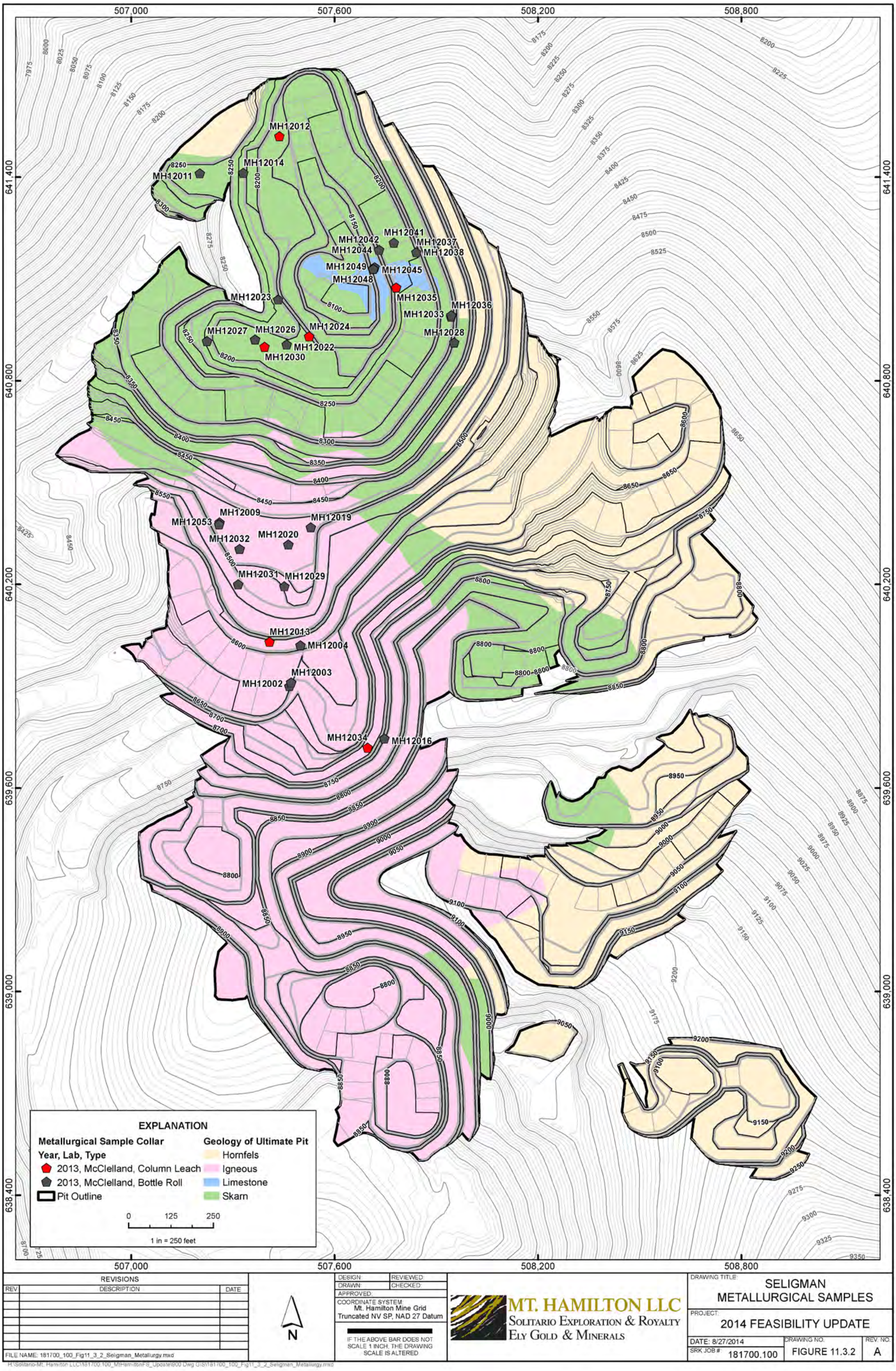
Details of the drill intervals and sample compositing are provided in the 2014 FS.

Figures 11.3.1 and 11.3.2 provide the metallurgical sample locations for the Centennial and Seligman ores.




Source: SRK, 2014

**Figure 11.3.1: Centennial Metallurgical Test Sample Locations**




Source: SRK, 2014

**Figure 11.3.2: Seligman Metallurgical Test Sample Locations**

## 11.4 Metallurgical Ore Characterization

Process selection and metal recovery estimates were determined in seven phases between 1997 and 2013 through a series of column and bottle roll leach metallurgical studies. The initial six phases characterized primarily the Centennial ores as described in the 2012 FS. Seligman ores were primarily characterized after the 2012 FS in the October 2013 Seligman and Centennial North metallurgical test program.

**Pre 2012 FS**

1) Pre – 1997 Scoping Studies; various laboratories including the Mt Hamilton laboratory
   - Four column and 28 bottle roll tests
2) 1997 Carrington (2009); Centennial Scoping Study
   - 32 bottle rolls
3) 1997 Kappes Cassiday and Associates (KCA);Centennial Characterization
   - Nine column and five boll roll tests
4) 2009 to 2010 McClelland Laboratories Inc. (MLI); Centennial Characterization
   - Two column and 64 scoping, leach feed size and variability bottle roll tests
5) 2011 MLI; Centennial Characterization
   - Two column and 38 scoping, leach feed size and variability bottle roll tests
6) 2011 MLI; Centennial Igneous Characterization
   - 18 variability bottle rolls

**Post 2012 FS**

1) 2013 MLI; Seligman Characterization
   - Five column and 87 scoping and variability bottle roll tests

The following ore characterization sections provide only the relevant results for the production heap leach recovery and recovery rate estimates and process design criteria. Ore Characterization details are provided in the 2014 FS (SRK, 2014a).

### 11.4.1 Pre-1997

The metallurgical test history of the Mt. Hamilton Project began in 1988 with Westmont as the owner of the property. The Centennial testing programs were concurrent with mining operations for the NE Seligman Mine. In some cases, the source of metallurgical test samples (i.e. NE Seligman vs. Centennial) was unclear. Approximately 28 bottle roll tests at varying crush sizes were carried out. There were four column tests performed, however the material makeup of the columns was not detailed. The test programs were done at various outsourced laboratories and at the Mt. Hamilton (Rea Gold) laboratories.

The data from these tests and a more definitive 1997 study were used in part to guide sample selection for the more recent test programs overseen by SRK. The 2009-2013 test program results moreover validated the 1997 test program results.

### 11.4.2 1997 Carrington (2006) – Centennial Mixed

A total of 32 variability bottle roll test results for some of the 1997 drilling were provided by Carrington in 2009. The average bottle roll gold recovery was 64% with a standard deviation of 14%.

These data included cyanide soluble assays and was evaluated by SRK to validate the 2014 recovery model. The Rmax was 69% gold recovery.

### 11.4.3 1997 KCA Test Program – Centennial Oxide

A metallurgical characterization test program on Centennial was performed by KCA in 1997. The purpose of this test work was to characterize the Centennial resource and determine the optimum leach feed size and heap leach recovery and recovery kinetics. The report entitled “Final metallurgical test work on core samples from the Mt. Hamilton – Centennial Zone“(KCA, 1997). The Centennial ores were identified as Main Zone and NW Upper Zone. The program consisted of five column scoping bottle roll tests, nine column tests, and 18 variability bottle roll tests.

Core holes 87005D and 91019 were received as composites by KCA. Core holes 96002D, 96003D, 97002, 97012, and 97024 were received in boxed 5 ft intervals. Each interval was fire assayed for gold and silver. The intervals were then composited by assay to approximate the ore resource head grade.

**Bottle Roll Tests**

The program included five column scoping bottle roll tests for the nine column tests. The tests were run on the column composites at 100 mesh size for 48 hours. The average recovery for the tests was 86.8 and 51% gold and silver, respectively.

**Column Tests**

The KCA Column Test Program consisted of nine tests at 1½ in and 1 in sizes. Test materials and results are shown in Table 11.4.3.1.

**Table 11.4.3.1: KCA 1997 Column Test Results**

| Sample ID | Calculated Head Au oz/t | Calculated Head Ag oz/t | Size (inches) | Days Leach | Recovery % | |
|---|---|---|---|---|---|---|
| | | | | | Au | Ag |
| 87005D | 0.043 | 0.17 | 1.00 | 48 | 74.4 | 58.8 |
| 87005D | 0.043 | 0.57 | 1.50 | 54 | 86.0 | 36.9 |
| 91019D | 0.056 | 0.43 | 1.00 | 54 | 79.3 | 46.5 |
| 91019D | 0.062 | 0.51 | 1.50 | 54 | 77.4 | 47.0 |
| 96002D (1) | 0.040 | 0.28 | 1.00 | 48 | 82.5 | 32.1 |
| 96003D (1) | 0.023 | 0.16 | 1.00 | 48 | 78.3 | 12.5 |
| 97002 | 0.112 | 0.56 | 1.00 | 44 | 80.4 | 37.3 |
| 97012 | 0.041 | 0.11 | 1.00 | 44 | 65.9 | 9.1 |
| 97024 | 0.056 | 0.44 | 1.00 | 44 | 75.0 | 15.9 |
| **Average** | **0.053** | **0.36** | | **49** | **76.5** | **32.9** |

Source: SRK, 2012
(1) Tests outside of current pit limits

The results showed that a finer leach feed size improved recovery; however, tests were still leaching at the end of the test period suggesting a longer leach period would improve recovery. A longer leach period was applied to future test work.

### 11.4.4 2009/2010 - McClelland Laboratories – Centennial Skarn Oxide

In 2009-2010, McClelland Laboratories of Reno, Nevada (MLI) conducted a metallurgical test program on core samples from drilling done by Ely Gold in 2008. The test work included 64 bottle roll and two column tests. The purpose of this test work was to further characterize the Centennial resource and determine the optimum leach feed size and heap leach recovery and recovery kinetics. The McClelland report is entitled “Report on Heap Leach Cyanidation Testing Centennial Project MLI Job Number 3354” (McClelland, 2010).

The half core samples were selected to fill in the gaps from the 1997 KCA column/bottle roll tests, geographically and at depth. The holes selected were MH08004 and MH08005. The half cores were composited into 20 ft intervals. MH08004 had a continuous ore zone of 142 to 249 ft. MH08005 had a continuous ore zone from 100 to 276 ft.

**Bottle Roll Tests**

The bottle roll testing was conducted to establish sample variability and to establish an economic leach feed size.

A total of 64 bottle roll tests were performed during the McClelland test program. A total of 48 tests were done on the half-core composites, and 16 tests were done on assay rejects from the corresponding intervals to preserve sample for the column tests.

A p100 minus 1 inch size was determined to be the economic size for Centennial and was utilized in the ensuing column tests. The average gold recovery of the bottle roll tests for the 16 assay reject samples was 77.5% with a standard deviation of 3.5%.

SRK again utilized this data to validate the recovery model for Centennial.

**Column Tests**

Two column tests were done at McClelland Laboratories. The tests were on drill holes MH08004 and MH080005 at p100 1 inch size.

The 20 ft interval composites at p100 1½ inch from the bottle roll series were reduced to p100 1 inch size by stage crushing. The 20 ft intervals assaying less than 0.005 oz/t Au were excluded from the columns.

Column testing was continued until the leach curves were “mature” so that a recovery could be projected beyond the column leach time. The results of the column leach tests are presented in Table 11.4.4.1.

**Table 11.4.4.1: 2009 McClelland Column Leach Test Results**

| Column | Calculated Head Assay oz/t | | Recovery % | | Column Days | Au Recovery Projected at 160 Days |
|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | | |
| MH08004 | 0.032 | 0.38 | 72.1 | 21.7 | 120 | 74.1 |
| MH08005 | 0.033 | 0.42 | 75.4 | 37.9 | 120 | 78.2 |

Source: SRK, 2012

The reagent requirements were calculated to be:

- Lime (CaO) 5 lb/t; and
- Sodium Cyanide 0.40 lb/t.

**32 Element ICP Scans**

Thirty-two element ICP scans on bottle roll composites and the first five day solutions from the columns show low concentrations of cyanocides (Cu-Mn-Ni) in the Centennial ores.

Mercury in solution was low (0.03 ppm), however a mercury retort and controls are included in the design of the processing plant described in this 2014 FS in accordance with State of Nevada permitting requirements.

The results from this work confirm that a conventional carbon ADR plant with a mercury retort will be sufficient for precious metal recovery without impurities in the dore'

### 11.4.5 2011 McClelland Laboratories - Centennial Skarn Mixed

McClelland Laboratories conducted a metallurgical test program directed by SRK on drill core samples in 2011. The program included 32 scoping and variability bottle roll tests and two column tests. The purpose of the tests was to further characterize Centennial and validate the Mount Hamilton heap leach design criteria. The McClelland report is entitled "Metallurgical Testing Centennial Drill Core Composites MLI Job No. 3528" (McClelland, 2011).

The drilling sites were selected by SRK to test sections in new areas of mineralization defined in the drilling.

The three holes utilized for testing were MH10002, MH10003, and MH10004. MH10002 and MH10003 were vertical holes. MH10004 was a 65º angle hole. MH10003 and MH10004 were in the same ore zone and both oxide and transition ores were combined for the column test. Half core samples were used for testing.

**<u>Bottle Roll Testing</u>**

Bottle roll tests were conducted on 32 samples prepared from the half cores and assay rejects as column scoping tests and variability tests. The nine core composites were reduced to p80 ¾ inch size for bottle roll testing and subsequent column testing. Duplicate bottle roll tests were conducted on core composites and assay rejects as cyanide assay tests tests.

The intervals were selected on the basis of cross sections and assays. Excluded from the composites were intervals of less than 0.005 oz/t Au.

The samples containing higher amounts of sulfide displayed lower recovery as expected. The average bottle roll gold recovery of the mixed skarn samples was 65% with a standard deviation of 15%.

This predictable recovery was particularly suitable to modeling recovery with cyanide soluble assays which is discussed in detail under the resource recovery model section. SRK also evaluated this data to validate the recovery model for Centennial; the Rmax was also found to be 65% gold recovery.

**<u>Column Tests</u>**

Two column tests were done at McClelland: Column C1 from MH10002, Column C2 from MH10003 and MH10004. The columns were run at a specified size of p80 ¾ inch.

Results from the 2001 column tests are shown in Table 11.4.5.1.

**Table 11.4.5.1: 2011 McClelland Column Test Results**

| Column | Calculated Head oz/t | | Recovery % | | Column Days | Projected Au Recovery at 160 Days |
|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | | |
| MH 10002 | 0.034 | 0.44 | 81.7 | 35.6 | 118 | 83.4 |
| MH 10003/4 | 0.045 | 0.61 | 79.4 | 56.6 | 118 | 81.0 |

Source: SRK, 2012

Column testing was continued until the leach curves were “mature” and a regression analysis could be made to extend the leach curve.

The reagent requirements were calculated to be:

Lime (CaO) 5 lb/t; and
Sodium Cyanide (NaCN) 0.8 lb/t.

**Comminution Testing**

Comminution tests were performed on whole core samples. The test samples were selected to represent the ores at three depths and grades. Results are presented in Table 11.4.5.2.

**Table 11.4.5.2: Comminution Results from 2011 Metallurgical Test Work**

| Hole | Depth | Work Index (kWh-ton) | Abrasion Index |
|---|---|---|---|
| MH 10009 | 493-507 | 4.97 | 0.0422 |
| MH 10009 | 569-579 | 7.85 | 0.0717 |
| MH 10009 | 641-650 | 8.03 | 0.0351 |

Source: SRK, 2012

The Work Index (Wi) is a measure of breakability of the ore and power requirement. For design purposes SRK used 8.03 KWH/t. Overall, the ores fracture easily at a low power requirement.

The Centennial ore has a relative low abrasion index.

**Recovery By Size Fraction**

Head and tails screen analysis were conducted on both column samples to verify the optimum leach feed size. The results showed similar results to the earlier 2009/2010 MLI results and again confirmed that the optimum leach feed size of 90% minus 3/4 inch is appropriate.

**32 Element ICP Scans**

Similar results were seen here as in the 2009/2010 MLI test work, low concentrations of cyanocides (Cu-Mn-Ni) in the Centennial ores. Again this supports processing by conventional ADR recovery.

**Height/Percolation Tests**

A Height/Percolation study was done on a residue sample from the C-2 (MH10003/4) column tests. The test consists of measuring percolation rates at varying heights. The heights are simulated by applying pressure by a hydraulic ram. The residues were tested at 12 heights from 0 to 220 ft.

The results indicate that Centennial ores will maintain adequate percolation rates up to a 220 ft heap height without agglomeration.

### 11.4.6 2011 McClelland Laboratories – Centennial Igneous

In the 2010 SRK PEA (SRK, 2010) resource ores were identified as skarn with minor hornfels. The 2014 FS reserve includes a significant addition of igneous ores located in part of the Seligman Stock.

In 2011 McClelland conducted 18 bottle roll tests on assay rejects from the north part of the Centennial ore body. Nine of the tests were cyanide soluble assay tests. The average gold grade of the igneous material is 0.013 oz/t Au, lower than average.

The results from this test work showed that the igneous ores also contain cyanide amenable gold at a similar recovery to the skarn-dominated ores, but that they were slightly below the average feed grade. The average gold recovery was 74% with a standard deviation of 4.6%.

SRK also evaluated these results to validate the 2014 recovery model. The Rmax was also 74% gold recovery.

### 11.4.7 McClelland Laboratories – 2013 (Seligman and Centennial North)

After the 2012 FS, an ore Characterization program was conducted to characterize the Seligman reserves and improve the characterization of the northern part of Centennial.

Dedicated 2011-2012 Seligman and Centennial North drilling programs confirmed economic gold and silver grades in grade ranges that support historic results. Specific PQ-sized drill holes were used to collect sufficient skarn, hornfels and igneous rock to conduct metallurgical tests to characterize metal recovery of these materials.

A total of five column tests and 87 bottle roll tests were conducted at McClelland. To verify that the Centennial process design criteria were also appropriate for Seligman, the test parameters were developed based on the metallurgical findings in the 2012 FS. One column test and two bottle roll tests characterized material from the northern area of the Centennial deposit, in igneous rock. The other four column tests and bottle roll tests characterized material from the Seligman deposit.

**Ore Type Definition**

The maximum gold recovery (Rmax) values based on the cyanide soluble assays and the percent pyrite from the geologic logs define the three metallurgical ore types:

- **Oxide** – Rmax values above 70% and pyrite less than 0.5%;
- **Transition** – Rmax grade equivalent greater than 0.14 ppm (0.004 oz/t), less than 70% Rmax, and pyrite between 0.5% and 2.5%; and
- **Sulfide** – Rmax grade equivalent less than 0.14 ppm (0.004 oz/t) and pyrite greater than 2.5%.

The maximum gold recovery is the quotient of the CN-sol and total gold values, a ratio between 0 and 1 (or 0% and 100%). A gold recovery of 70% was used to calculate the recovered AuEq CoG of 0.14 ppm Au to remain consistent with the Centennial FS. Maximum recovery AuEq grade is based on metal prices of US$1,500/oz gold and US$20/oz silver. CN-sol silver grades are divided by 75 and the result is added to the CN-sol gold grade to calculate maximum recovery AuEq grade.

**Column Test Compositing**

The column composite selection criteria were based on: 1) the AuEq CoG; 2) the AuEq average deposit grade; and 3) equally weighting each hole going into the composite.

The intervals were then sorted on AuEq feed grade values for each drill hole. All Intervals in each drill hole with grades between the cut-off and the highest grade providing the average grade were selected for the composites. The average grade used for the skarn and igneous ore types were 0.960 ppm AuEq (0.028 oz/t), and 0.583 ppm AuEq (0.017 oz/t), respectively. These grades match the grades reported in the July 31, 2012 Seligman resource statement prepared by SRK.

Lastly, equal amounts of material were collected from each drill hole to build column composites. For example, holes with many intervals of material meeting criteria contributed less weight per interval than those holes with fewer intervals that met criteria.

The column tests were composited from seven PQ metallurgical holes.

**Bottle Roll Test Work**

The objectives of bottle roll testing included the following:

- Six preg robbing tests to determine if there were any ore types with poisoning (‘preg robbing’) effects;
- Five column leach scoping tests to establish the column leach procedures;
- One sulfide composite to determine if a column test should also be conducted, and obtain reagent consumption data; and
- 32 variability tests for ore variability and recovery modeling by ore type.

Prior to compositing the intervals for column test work, preg robbing tests were conducted on select intervals of the column composites to identify any potential ‘preg robbing’ ore types. Intervals with low Rmax values were selected from core and RC drillholes for the tests. The six preg robbing bottle roll samples were composited from 10 sample intervals. The results showed the samples are not preg robbing. As a result, all originally proposed intervals were included in the column composites. The poisoning effect of preg robbing materials was not present in the Seligman ores.

Preliminary bottle roll leach tests were conducted on the five column composites and a skarn sulfide composite at two feed sizes. The 96 hour 19 mm feed size tests were used for column parameter determination. The 106um feed size tests (Rmax) for 48 hours was used for recovery modeling. The gold Rmax of the bottle rolls by ore type were 76% for oxide and 34% for transition. Due to the low gold recovery from the sulfide bottle roll test, a large-scale column test of this material was not necessary.

Variability bottle roll testing was conducted on 32 samples from the Seligman deposit to provide recovery data by rock type, metallurgical ore type, feed grade and location. The tests were conducted at two feed sizes 10 mesh and 106 µm (Rmax). Results show that variability is predominately by metallurgical ore type (oxide, transition and sulfide). More discussion on this is provided in the Recovery Modeling Section.

**Column Test Work**

Five column leach tests were conducted on five ore types, as discussed above. The column leach feed size was 90% passing 19 mm; the optimum feed size determined in the 2012 FS. The columns were constructed of 6 inch PVC pipe, except the Igneous Transition column was 4 inch PVC. All columns were 10 ft tall. Additionally, a single Seligman skarn sulfide bottle roll test was conducted, at a feed size of 90% passing 19 mm, to verify the predicted lower amenability of sulfide ores to cyanidation and reagent consumption.

Column test recoveries, shown in Table 11.4.7.1, for skarn oxide, igneous oxide and Centennial North oxide ores were all amenable to cyanidation, and were similar to the 2012 FS ore types. Transition and sulfide ores were less amenable to cyanidation as expected. The estimated 120 day recoveries were developed using a regression analysis on the leach period following the last rest cycle.

**Table 11.4.7.1: 2013 Column Test Results Summary**

| Column Composite | Calc. Head (g/tonne) | | Extracted (g/tonne) | | Tail Screen (g/tonne) | | 106 Day Recovery (%) | | Est. 120 Day Recovery (%) | | Ratio |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | Au | Ag | Au | Ag | Au | Ag | Ag:Au |
| Skarn Oxide | 0.71 | 1.4 | 0.52 | 0.6 | 0.19 | 0.8 | 73.20 | 42.9 | 73.40 | 44.5 | 1.2 |
| Skarn Transition | 2.17 | 8.4 | 0.62 | 2.2 | 1.55 | 6.2 | 28.60 | 26.2 | 29.60 | 26.3 | 3.5 |
| Igneous Oxide | 0.55 | 5.1 | 0.47 | 2.4 | 0.08 | 2.7 | 85.50 | 47.1 | 85.70 | 48.7 | 5.1 |
| Igneous Transition | 0.53 | 7.2 | 0.18 | 3.4 | 0.35 | 3.8 | 34.00 | 47.2 | 34.50 | 47.6 | 19.0 |
| Centennial N. Oxide | 0.73 | 13.6 | 0.57 | 4.9 | 0.16 | 8.7 | 78.10 | 36.0 | 80.00 | 37.7 | 8.6 |

Source: SRK, 2014

## Reagent Consumption

Bottle roll reagent consumption rates are typically representative of full scale production consumption rates and used to predict them. Reagent consumption results are presented in Table 11.4.7.2. The average bottle roll reagent consumption is 0.27 kg/tonne NaCN and 0.85 kg/tonne lime. A well-operated heap leach will consume 25% to 33% of the cyanide consumption of an extended column leach. The main reason for higher consumption in columns is the higher cyanide concentration in the reagent. Consumption of NaCN for many heap leach operations is between 0.2 and 0.3 kg/tonne, which suggests that the bottle roll reagent consumption test results for Seligman and Centennial North are reasonable.

**Table 11.4.7.2 Reagent Consumption for Bottle Roll and Column Tests**

| Composite | kg/tonne | | | |
|---|---|---|---|---|
| | Bottle Roll | | Column | |
| | NaCN | Lime | NaCN | Lime |
| Skarn Oxide | 0.15 | 0.7 | 1.39 | 0.7 |
| Skarn Transition | 0.13 | 0.7 | 1018 | 0.7 |
| Skarn Sulfide | 0.28 | 0.7 | NA | 0.7 |
| Igneous Oxide | 0.34 | 1.2 | 1.94 | 1.2 |
| Igneous Transition | 0.34 | 0.9 | 1.86 | 0.9 |
| Centennial North Oxide | 0.36 | 0.9 | 1.62 | 0.9 |
| **Average** | **0.27** | **0.85** | **1.60** | **0.85** |

Source: SRK, 2014

The 2013 consumption rates compare favorably to the 2012 FS estimated consumption rates of 0.4 kg/tonne NaCN and 2.5 kg/tonne Lime. Reagent consumption rates also compare favorably between oxide, transition and sulfide.

## Recovery by Size Fraction Results

Based on the head and tail screen recovery by size fraction results for the five column leach tests, the tail assays between nominal 19 mm and 12 mm are within 0.01 gpt Au for four composites and within 0.02 gpt Au for the igneous transition composite. This difference in tail assays do not support a finer nominal 12 mm feed size and confirm that the nominal 19 mm feed size prescribed in the 2012 FS is the appropriate heap leach feed size.

The results also confirm that agglomeration is not required. All five composites consisted of less than 5% passing 150 µm.

**Leach Solution Chemistry**

Samples of the first five days of column leach solution were collected and analyzed with Inductively Coupled Plasma Mass Spectrometry (ICP). Table 11.4.7.3 provides the results from the ICP analysis for the five columns.

**Table 11.4.7.3 Multi-Element ICP Analyses of Column Solutions (mg/L)**

| Analysis | MHSO P-1 101-105 | MHST P-2 201-205 | MHIO P-3 301-305 | MHIT P-5 501-505 | MHCNO P-4 401-405 |
|---|---|---|---|---|---|
| Al | 0.2 | 0.6 | 0.2 | 0.5 | 0.8 |
| As | <0.1 | <0.1 | 0.4 | 1.2 | 0.2 |
| Ba | <0.1 | <0.1 | <0.1 | <0.1 | 0.1 |
| Bi | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 |
| Ca | 55.1 | 52.7 | 55.2 | 102 | 50.5 |
| Cd | 0.3 | 0.2 | 0.4 | 0.1 | <0.1 |
| Co | <0.1 | 0.2 | 0.1 | 0.1 | <0.1 |
| Cr | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 |
| Cu | 9.4 | 15.0 | 16.9 | 19.3 | 37.4 |
| Fe | <0.1 | <0.1 | <0.1 | 0.2 | 0.3 |
| Hg | 0.007 | 0.006 | 0.040 | 0.014 | 0.052 |
| K | 4.1 | 7.1 | 4.0 | 8.5 | 8.4 |
| Mg | 0.7 | <0.1 | 1.9 | 1.3 | 0.4 |
| Mn | <0.1 | 0.1 | <0.1 | <0.1 | <0.1 |
| Mo | 0.8 | 7.8 | 2.0 | 2.0 | 2.7 |
| Ni | 0.1 | 0.1 | 0.1 | 0.1 | <0.1 |
| Pb | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 |
| Sr | 0.6 | 1.0 | 0.6 | 1.1 | 0.5 |
| Ti | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 |
| V | <0.1 | <0.1 | <0.1 | <0.1 | <0.1 |
| W | 0.6 | 0.1 | 0.3 | <0.1 | 0.9 |
| Zn | 131 | 68.0 | 42.8 | 76.5 | 3.5 |

Source: McClelland, 2014

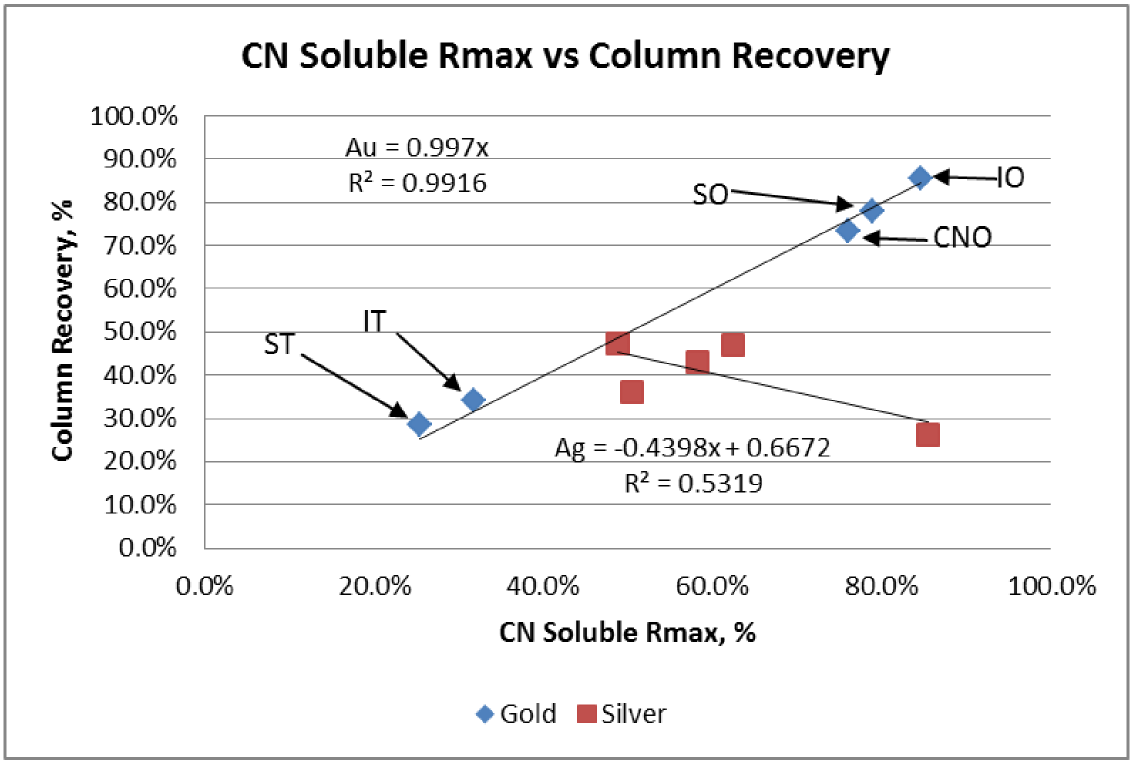
Compared to the 2012 FS metallurgy test results, these numbers are favorable and would not change the assumptions for the ADR plant sizing. For example, Cu in the 2012 FS samples was a high of 22.2 mg/L compared to a high of 37.4 mg/L for Centennial North Oxide; and Hg was a high of 0.033 mg/L in the 2012 FS compared to a high of 0.052 mg/L for Centennial North.

# 11.5 Precious Metal Recovery and Recovery Rate Predictions

## 11.5.1 Precious Metal Recovery Prediction

The 2012 FS recovery prediction took into account feed grade, leach feed size, leach time and production scale-up. The average column recoveries were then adjusted up and down for each of the above four variables, and the overall Centennial deposit gold and silver recovery was estimated at 79% and 38%, respectively.

During the 2013 Seligman test program, a recovery model was developed utilizing the cyanide soluble assays [*CNAu/FAAu = gold Recovery Maximum (Rmax)].* A relationship between CN Soluble Rmax and Column Recovery (120 day estimated) was developed for the purpose of predicting recovery in the resource block model. Recovery in the 2014 FS utilizes a block-by-block estimate of recovery, rather than a recovery assignment by rock type. Gold recovery was capped in the block model at the maximum bottle roll recovery of 92.7%. The robust relationship between CN solubility and column recovery facilitated a modeled recovery prediction, which provides higher resolution and accuracy compared to assigning recoveries based on rock type. This relationship is illustrated in Figure 11.5.1.1.

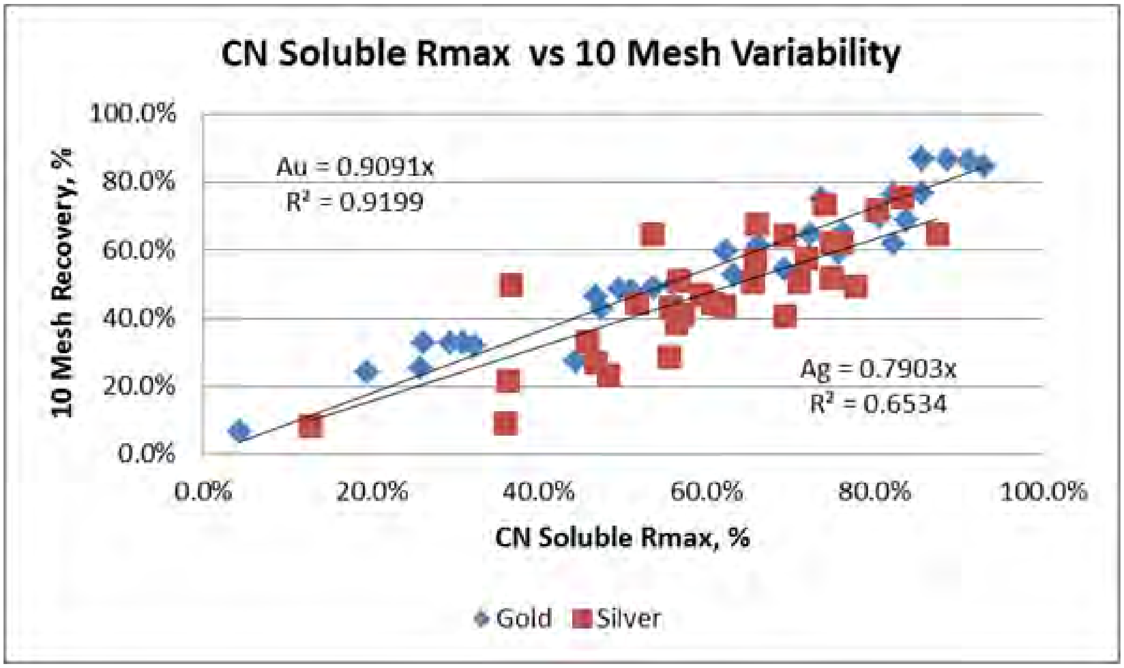



Source: SRK, 2014

**Figure 11.5.1.1: CN Soluble Rmax vs. Column Recovery**

Gold recovery is primarily a function of the extent of oxidation and can be predicted from the CN soluble gold assays. However, the correlation coefficient for silver is too low to reliably predict silver recovery from CN soluble silver assays.

A similar relationship of recovery estimation was developed using the Seligman variability samples, and is supported by the recovery relationship shown in Figure 11.5.1.2.




Source: SRK, 2014

**Figure 11.5.1.2: CN Soluble vs. 10 Mesh Variability Bottle Roll**

These CN soluble recovery relationships for gold and silver are very similar to the column recovery relationships, again suggesting that gold recovery is predictable as a function of the extent of oxidation.

To assess the veracity of the Rmax recovery model the earlier Centennial variability test results were checked against the model. Table 11.5.1.1 provides a comparison of the actual average gold recovery vs the model predicted gold recovery for the four variability bottle roll test programs.

**Table 11.5.1.1: Actual vs Predicted Recovery for Centennial Ore (Centennial Variability Tests)**

| Source | Recovery % Au | |
|---|---|---|
| | Actual | Model Predicted |
| Carrington | 63.9 | 63.1 |
| MLI 2009 / 2010 | 77.5 | 74.9 |
| MLI 2011 Skarn | 65.1 | 59.2 |
| MLI 2011 Igneous | 73.7 | 66.9 |

Source: SRK, 2014

These results would suggest that the recovery model is conservative for the higher grade Centennial ore.

The Rmax correlation for silver was poor so the average Centennial and Seligman column test recoveries were used for silver recovery in the block model.

***It is important to note that not all the ores characterized through variability bottle role test work will be processed during mine production. The oxides and some transitional ores will be***

***processed and the sulfides and a portion of the transitional ores will be sent to waste. The determination of ore and waste is based on the cyanide soluble assay and the Rmax model.***

The predicted average gold and silver recoveries for the overall Mt. Hamilton reserve are 76.3% and 39%, respectively.

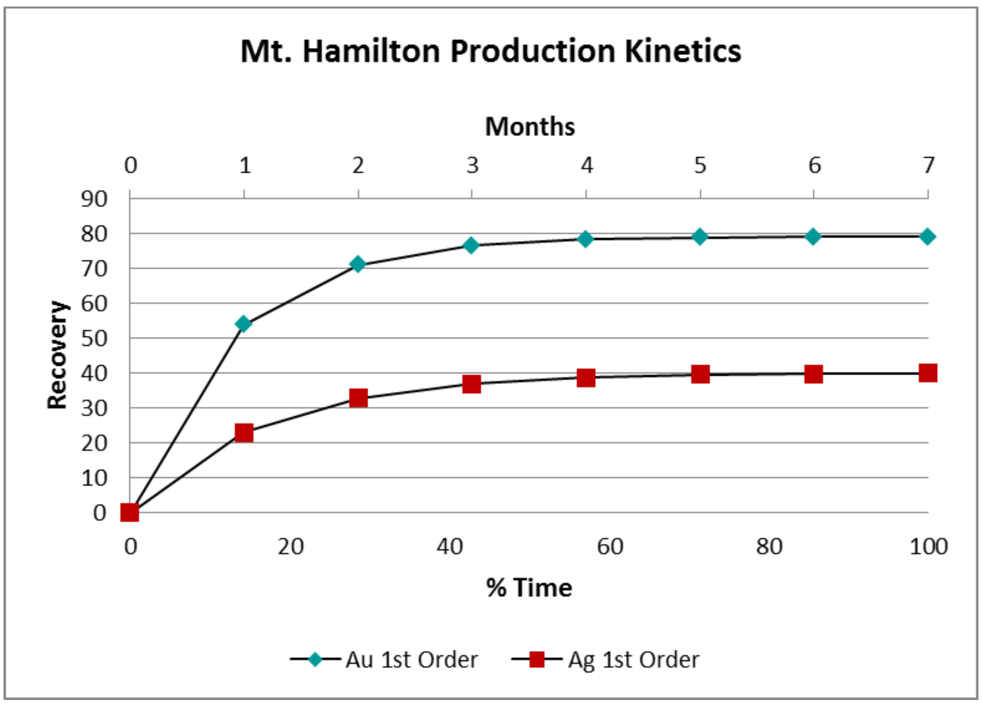
### 11.5.2 Production Leach Rate Prediction

The 2012 FS leach rate projected production leach rates using an empirical formula. The initial leach is multiplied by 3x (three times), the knee of the curve is multiplied by 2x and the "tail out" is 1x. In the Centennial ore 160 days of column leach is equivalent to 210 days of field leach.

The column leach rate for Seligman ores was slightly faster than Centennial and estimated to be complete in 120 days; therefore, the Centennial kinetics drove the design criteria for leach pad operation.

The leach solution application rates and the area under leach were therefore designed for a 210 day leach cycle.

The production leach rates are based on the column leach kinetics as provided in Figure 11.5.2.1.

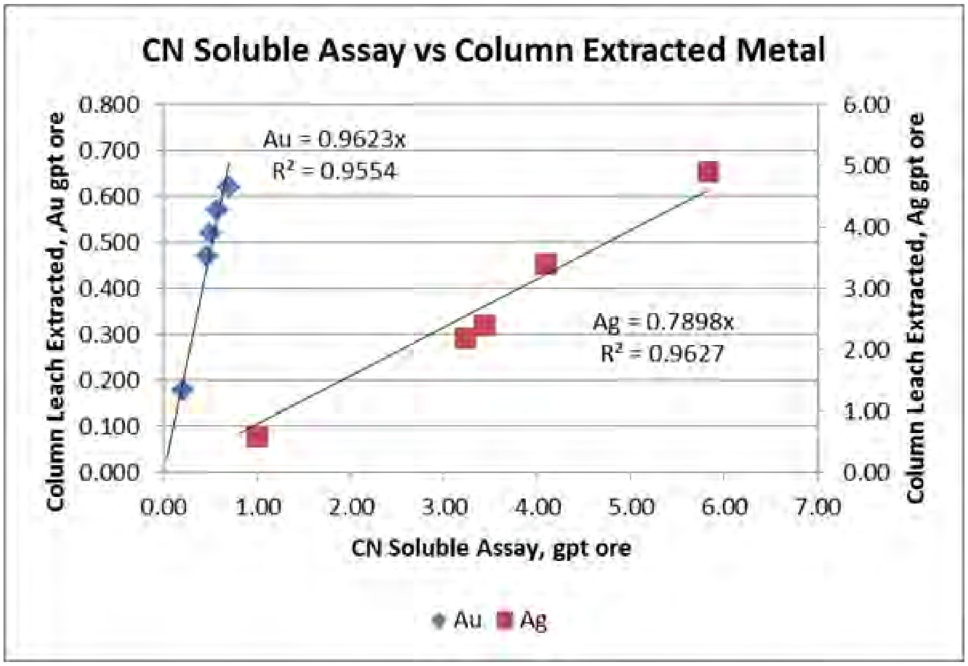



Source: SRK, 2014

**Figure 11.5.2.1: CN Soluble vs. 10 Mesh Variability Bottle Roll**

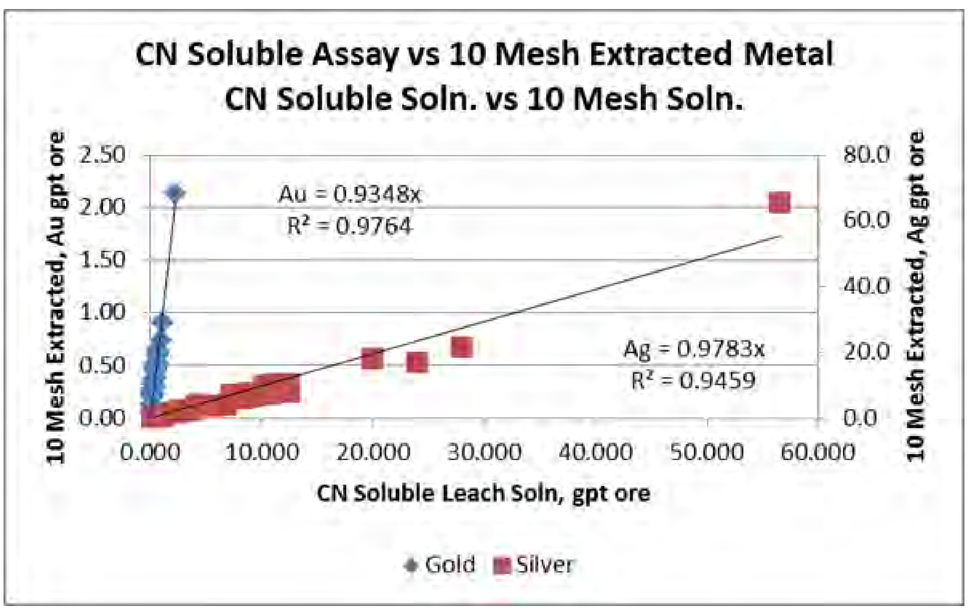
### 11.5.3 Grade Control Application

Relationships between CN soluble assays and column and variability bottle roll extracted metals were developed for the purpose of showing how grade control can be conducted. Figures 11.5.3.1 and 11.5.3.2 show the results. The column and variability sample relationships for both gold and silver show strong correlations. Therefore, CN soluble assays can be used as a method for conducting grade control for both metals. Although grade control will be primarily based on CN-soluble gold grade, CN-sol silver grade can be used to calculate AuEq, if that parameter were to be used for grade control as well as soluble gold.




Source: SRK, 2014

**Figure 11.5.3.1: CN Soluble Assay vs. Column Extracted Metal**




Source: SRK, 2014

**Figure 11.5.3.2: CN Soluble Assay vs. 10 Mesh Extracted Metal**

## 11.6 Basis of Key Process Design Criteria

### 11.6.1 Precious Metal Recovery

Precious metal recovery was based on the ore characterization for both Centennial and Seligman.

Gold recovery is based the Rmax model developed during the Seligman ore characterization program with a very strong correlation to cyanide soluble assays (Rmax).

The Rmax correlation for silver on the other hand was poor so silver recovery is based on the average column test recoveries for both Centennial and Seligman:

- Au – Rmax Model (76.3% = calculated block model average); and
- Ag – Column test average (39%).

### 11.6.2 Leach Feed Size

The leach feed size of 90% passing ¾ inch was based on the 2012 FS ore characterization studies. The Seligman column leach tests were all run at the ¾" size, and the same optimum leach feed size was confirmed by the recovery by size fraction test work.

### 11.6.3 Leach Recovery Rate

The production leach rate of 210 days is based on the 2012 FS. The Seligman recovery rate was slightly faster than Centennial ores, so the 2014 FS production leach rate was not changed.

The cyanide solution application rate of 0.004 gpm/ft$^2$ was based on 2012 FS. The application rate for the Seligman ore characterization tests was the same.

The amount of ore under leach at any one time (approximately 2.1 million tons) is based on the 10,000 t/d stacked ore production rate, the 210 day leach cycle and solution application rate.

The pregnant solution flow (3,000 gpm) and metals concentration and hence the ADR recovery plant design capacity is based on the solution application rate and the total area under leach, 750,000 ft$^2$.

### 11.6.4 Heap and Lift Height

The heap lift height is a conservative 30 ft at steady state and based on benchmarking other operations. The total heap height of 220 ft is based on Centennial and Seligman load permeability test work.

### 11.6.5 Reagent Consumption

Reagent Consumption was based on the 2012 FS. Ore Characterization of the Seligman deposit provided slightly lower reagent consumptions, so the 2014 FS assumed reagent consumption rates were not changed. Major reagent consumption specifications are provided in Table 11.6.5.1.

**Table 11.6.5.1: Major Reagent Consumption**

| Reagent | Use |
|---|---|
| Lime (CaO) | 4 lb/t |
| Sodium Cyanide | 0.6 lb/t |

# 12 Mineral Resource Estimation (Item 14)

## 12.1 Introduction

The report is intended for the use of MH-LLC for the further development and advancement of the Mt. Hamilton Project. This report provides a mineral resource estimate and classification of resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, dated May 10, 2014 (CIM).

SRK estimated mineral resources for the Centennial Deposit using lithology, structure, alteration and oxidation remodeled in 2013 and included 60 new drill holes for the Seligman Deposit. The revised geologic model was used to update block model densities for a more accurate tonnage prediction, and to better control the distribution of gold and silver grades. Previous resource models addressed the Seligman and Centennial deposits separately; but due to the proximity of the deposits, similarity of geologic setting and metallurgy and the shared proposed infrastructure, it is appropriate to include both in one model. Recent exploration and metallurgical work suggests that these two deposits may be interconnected and potentially mined within a single pit. The block model was updated in 2014 with revised classification criteria. The resulting block model was applied to refine resource and reserve estimations, and for mine development planning.

The resource estimate and related geologic modeling were conducted by, or under the supervision of, J. B. Pennington, M.Sc., Principal Resource Geologist and Mining Group Leader of SRK Consulting in Reno, Nevada. Mr. Pennington is a Certified Professional Geologist as recognized by the American Institute of Professional Geologists and a Qualified Person as defined by the Canadian Institute of Mining and Petroleum National Instrument 43-101 (NI 43-101) and is knowledgeable in all aspects of public reserve/resource disclosure and compliance. He has completed resource modeling, due diligence, acquisition and evaluations assignments for precious and base metals in Australia, Indonesia, North America, Mexico, Colombia, Peru and Russia. Geologic modeling was completed in Leapfrog Mining® software, and resource estimation/block modeling was completed using Mintec’s MineSight® 3D software. The project was built in U.S. units (feet) and all metal grades are in troy ounces per short ton (oz/t).

**Cautionary Note to U.S. Investors concerning estimates of Measured and Indicated Resources and Inferred Resources:** This report uses the terms “Measured” and “Indicated resources.” These terms are recognized and required by Canadian regulations; The SEC does not recognize them and U.S. investors are cautioned not to assume that any part or all of mineral resources in these categories will ever be converted into reserves. This section also uses the term “Inferred resources.” This term is recognized and required by Canadian regulations; the SEC does not recognize it. “Inferred resources” have a great amount of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resource will ever be upgraded to a higher category. Under Canadian rules, estimates of Inferred Mineral Resources may not form the basis of feasibility or prefeasibility studies, except in rare cases. U.S. investors are cautioned not to assume that part or all of an Inferred resource exists, or is economically or legally minable. **Reserves meeting the requirements of the Securities and Exchange Commission’s Industry Guide 7 for Mt. Hamilton project are described in the Mining section of this 2014 FS.**

## 12.2 Project Coordinates

The Project coordinate system established during mining and exploration in the 1990's is still used. It is based on the Nevada State Plane (NV SP), East Zone projection, 1927 North American Datum. Northing coordinates in the truncated mine grid system are one million less than the Nevada State Plane coordinates, but no other transformations were done to alter the Project coordinates from the NV SP coordinates. Coordinate system units are feet, and all units presented in this document are U.S. units (feet, miles, troy ounces, short tons, etc.) unless otherwise specified.

## 12.3 Drillhole Location and Topography

Topographic surfaces used for model coding are pre-mining and current. These surfaces are the same as those used in the 2012 Seligman and Centennial resource models. The current topography is from a March, 2009 survey by Intrasearch. Original topography in the Seligman pit area was available from MH-LLC. The elevation of this surface was adjusted to better match the current topography. In disturbed areas beyond the original topographic surface extent, the original topography was approximated with 3D modeling. SRK created surfaces with points and polylines under the Cabin Gulch waste dump and other areas of backfill. The resulting surfaces were merged with the adjusted original topography and used to create volumes for backfill model coding.

Drillhole collar locations, both historical and current, were validated against topography and aerial photos. Collar locations for 2012 drillholes were surveyed by Solarus, a public land surveyor. Plan views of drillhole collar locations are presented in the drilling section of this report (Section 8) Figures 8.1.1, 8.2.1.1 and 8.2.1.2.

Several historical drillholes have inconsistent collar location data that could not be reconciled to topography. These were omitted from the resource estimation.

Downhole projections of drillhole orientation are from multi-shot cameras or gyroscopic surveys for angle holes, and some vertical holes. Typically, vertical holes are assumed to not deviate in orientation and are assigned -90° dip. Several uncertainties in downhole survey results from the 2012 drilling program were identified, and reconciled. No other erroneous downhole survey data was identified in the data set.

## 12.4 Drillhole Database

Extensive geologic and assay database verification and updating was completed by SRK in advance of resource modeling. The drillhole database files used for the 2014 model have been audited and updated by SRK during recent drilling campaigns to include in the resource model all relevant data available as of August 2013. These data were then added to a MineSight® Torque database to collate gold and silver values from the previous Centennial and Seligman datasets. Calculated fields used logic-based scripts to bring values from multiple data fields to a single field for each of total gold and silver, and cyanide-soluble gold and silver. These values were then used for geostatistical analysis to determine capping values, variography and other parameters by mineral domain.

There are 1,006 unique drillholes in the entire Project database, including two monitoring wells. Of these, 903 drillholes are in the resource model area. Two of these, 88002 and 97024, were omitted from the estimation due to uncertain location or missing data. Drillhole statistics are presented in Table 12.4.1.

**Table 12.4.1: Drillhole Database Statistics**

| Data | Total Au (oz/t) | CN-Sol Au (oz/t) | Total Ag (oz/t) | CN-Sol Ag (oz/t) |
|---|---|---|---|---|
| Number of Samples | 58,573 | 58,573 | 58,573 | 58,573 |
| Mean | 0.0066 | 0.0020 | 0.0630 | 0.0203 |
| Variance | 0.0009 | 0.0002 | 0.0870 | 0.0120 |
| Standard Deviation | 0.0293 | 0.0134 | 0.2950 | 0.1095 |
| Minimum | 0.000 | 0.000 | 0.000 | 0.000 |
| Maximum | 1.962 | 1.218 | 18.250 | 5.139 |
| Coefficient of Variation | 4.4417 | 6.5410 | 4.6789 | 5.3803 |

Source: SRK, 2014

### 12.4.1 Assay Data Validation (Data Quality Assessment)

Data quality for analytical results from the 2012 and late 2011 drilling programs were analyzed by SRK prior to modeling. The methods of the analysis, a description of quality control standard reference materials (standards, blanks and duplicates) and the results of the analysis are documented in Section 9 and 10 of this report

SRK concluded the proportion of blank and standard samples inserted in the drill sample sequence exceeds the number required to verify both fire assay and ICP results. Although MH-LLC did not provide standard samples with certified values for cyanide-soluble gold or silver, the results were verified by comparing them to the reported total values and to the material characteristics noted in geologic logs. The quantity and quality of duplicate analysis pairs are adequate to show repeatable analytical procedures and results.

SRK also concluded that standard and blank samples generally show accurate and repeatable results, and the quality of analytical data was improved in the 2012 program compared to 2011. Assay duplicate pairs generally showed repeatable results within industry-standard tolerance ranges, especially for samples with mineralization of economic importance (SRK, 2013a).

### 12.4.2 Conversion of "Recoverable" to Total Silver

Prior to 2008, portions of the drilling results did not include fire assay data for silver. In July of 2013, SRK determined that the four drilling campaigns completed at Centennial from 2008-2012 generated enough high quality paired CN-soluble/Fire Assay silver data (n=1,058 pairs) to revisit their statistical relationship, and reset the missing total silver values in the Centennial database. Having total silver values for Centennial makes the data set compatible with all of the other total metal assays for gold and silver for both Centennial and the neighboring Seligman deposit. The combined Centennial-Seligman database was used in 2014 resource modeling.

A single recommended relationship provides a strong coefficient of determination of 0.90 for all paired Centennial data. Metallurgically, silver cyanide solubility behaves similarly in oxide and sulfide, by rock type (igneous and non-igneous) and by deposit (Centennial and Seligman). The slopes of all relationships evaluated were approximately 1.8 and all have a strong coefficient of determination of 0.9 or greater. Therefore, CN-soluble silver assays in the Centennial database were converted to total silver using the equation:

*Total Ag = 1.8* CNsolAg*

## 12.5 Geologic Model

The Mt. Hamilton deposit is a hydrothermal gold-silver system hosted in contact-metamorphosed Paleozoic sedimentary units, and to a lesser extent, an adjacent granodiorite intrusion. SRK has modeled the Centennial and Seligman deposits for previous resource estimations, and a similar approach to geologic modeling was used for the 2014 model. Both Leapfrog Mining® and the new Leapfrog Geo® 3D software were used to define mineral domains and geologic contacts.

Improving on the previous geologic models, the new model encompasses the entire project area, and incorporates all available surface mapping and drillhole data. In addition to defining the geologic boundaries in the resource area, SRK also interpreted bedrock lithology in planned infrastructure areas.

### 12.5.1 Lithology and Alteration Modeling

Metamorphic alteration and primary lithology were logged and coded as one combined data field. The extent of the contact metamorphic aureole around the igneous intrusions was modeled by grouping the metamorphosed lithology codes. Between the Seligman and Monte Cristo Stocks, drillhole data indicates that contact metamorphism is continuous. In areas lacking data, controls were added to create a volume that parallels the extents of the modeled igneous units. Nearly all of the material in the resource model area has been metamorphosed, except for several pockets of shale or limestone in the north end of the Seligman resource near Seligman Canyon.

After contact surfaces were generated, they were exported in a portable file format and imported to the MineSight® 3D (MS3D) project to use for block model coding. Table 12.5.1.1 and 12.5.1.2 shows the modeled lithology and alteration domains used to assign material types in the block model.

**Table 12.5.1.1.: Modeled Lithology Domains**

| Material- LITH Model Item | Code |
|---|---|
| Igneous | 1 |
| Skarn | 2 |
| Hornfels | 3 |
| SedSH- shale, modeled "hornfels" outside alteration aureole | 4 |
| SedLS- limestone, modeled "skarn" outside alteration aureole | 5 |
| Waste Rock/Fill | 6 |
| Alluvium | 7 |

Source: SRK, 2014

**Table 12.5.1.2.: Modeled Alteration Domains**

| Material- ALT Model Item | Code |
|---|---|
| Altered; inside modeled contact aureole. | 1 |
| Unaltered; outside of contact aureole. Assigned to all and over-coded. | 2 |

Source: SRK, 2014

### 12.5.2 Oxidation Modeling

Iron oxide minerals are the alteration products of sulfides exposed to oxidizing fluids. There is not a discrete supergene mineral zone at Mt. Hamilton. However, oxidation of sulfides makes the contained gold amenable to heap leaching, and is important for resource definition. Besides metallurgical considerations, oxidation intensity changes the density of the host rock. Due to these factors, SRK remodeled oxidized zones with higher resolution in the 2014 model.

Historic oxide logging protocol resulted in oxidation being under-represented where incomplete oxidation had left visible sulfides in samples. If sulfides were present, oxidation was not represented in the database. To address this, SRK standardized the oxidation/reduction (redox) parameters noted in the “Oxide” logging field, and included all available data for future modeling. Initially, the degree of oxidation and sulfide mineralization were tabulated separately, each on a 0-3 scale. In a spreadsheet, a logic function was created to populate the combined code, on a 0-6 scale. This facilitated separate characterization of oxide and sulfide in the deposit, which was used to compare to metallurgical responses and waste rock geochemistry of the different rock types.

### 12.5.3 Mineral Domain Modeling (Grade Shells)

The Mt. Hamilton model space was subdivided into seven mineral domains as shown in Table 12.5.3.1.

**Table 12.5.3.1: Mineral Domains**

| Code | Domain Name | Description |
|---|---|---|
| 1 | Centennial – Non Igneous | Skarn and Hornfels; North boundary at Cabin Gulch valley bottom |
| 2 | Centennial - Igneous | Granodiorite; North boundary at Cabin Gulch valley bottom |
| 3 | Seligman Stock - South | Granodiorite; Flat to south dipping mineralization |
| 4 | Seligman Stock - North | Granodiorite; Low-angle north dipping mineralization |
| 5 | Seligman – Skarn North | Skarn and Hornfels; Strong north-northeast dipping grade |
| 6 | Seligman – Skarn East | Skarn and Hornfels; Low-angle north dipping mineralization |
| 7 | Seligman – Skarn South | Skarn and Hornfels; Flat to south dipping mineralization |

Source: SRK, 2014

The primary criteria for the assignment of domains was host rock (igneous vs. non-igneous), followed by changes in structure (faults or changes in the orientation of mineralization). The domains were used in geostatistical analyses and in grade interpolation. They were coded into the block model item “DOM.” A different anisotropy was applied to grade interpolation in each domain in accordance with the structural trends interpreted in each domain.

Within each of the seven modeling domains listed in Table 12.5.3.1, a separate mineral domain (grade shell) was built for Au and Ag. An implicit modeling (3D contouring) approach was selected with heavy influence from the modeler on grade orientation, anisotropy and range. The grade shells were built on a 0.004 oz/t CoG. The low-grade envelope was modeled to allow resource reporting of lower grades if metal prices increase in the future. These grade shells were coded into exploration block model item “MZONE.”

Silver was initially modeled at a CoG of 0.6 oz/t. At this grade all of the relevant Ag mineralization was contained in the Au grade shell, therefore, Ag was estimated inside the Au gradeshell. Because gold is the primary economic metal, mining will be driven by Au grades and Ag production is a byproduct.

Cyanide-soluble Au (CNAu) was modeled separately and has its own set of grade shells by model domain. Modeling criteria mimic the criteria for total Au listed in Table 12.5.3.1, but due to a lower sample distribution (CNAu assays were not always present), the grade shells were typically smaller and less continuous.

## 12.6 Density

In the previous resource models, SRK applied tonnage factors by material type from 58 density determinations completed by MRDI in 1997. Materials tested were from the Centennial area, and included skarn, hornfels, quartz vein, and igneous, in order of abundance. These materials are representative of the Centennial rock mass, which is similar to the Seligman rock mass. Therefore, the same tonnage factors were applied to resource estimations for the Centennial and Seligman deposits.

In July 2013, SRK selected samples from available drill core for additional density determinations. The purpose of this program was to characterize Seligman materials, compare results to the Centennial data set, and confirm the established values for Centennial materials. A total of 22 density determinations were completed on drill core samples from the 2012 drilling program. Igneous, skarn, hornfels and quartz veins with varying degrees of oxidation were tested. These materials comprise over 90% of the total volume in the resource pits. Shale and limestone are a minor component of the total resource volume, and were not tested. Published density values for shale and limestone were used to assign tonnage factors for the Seligman and Mt. Hamilton resource models (Berkman, 1989). The new results compared well with the established values, and were integrated with previous results to produce updated tonnage factors. The 2014 updated tonnage factors are provided in Table 12.6.1.

**Table 12.6.1: Tonnage Factors by Rock Type**

| Lithology | Oxidation | 2014 TF ($ft^3/t$) |
|---|---|---:|
| Igneous | Unox | 11.6 |
| Igneous | Oxide | 12.4 |
| Skarn | Unox | 10.8 |
| Skarn | Oxide | 12.4 |
| Hornfels | Unox | 11.0 |
| Hornfels | Oxide | 12.4 |
| Shale (1) | n/a | 12.5 |
| Limestone (1) | n/a | 12.0 |
| Waste Rock/Fill (2) | n/a | 17.0 |

(1) Densities for sediments outside the alteration envelope were based on published values (Berkman, 1989), not SRK test work.

(2) Waste rock or fill tonnage factor is (All Oxide/All Unoxidized) * 110 lb/ft$^3$, from Centennial metallurgical work. Ex: 12.4/11.6 * 110 = 117 lb/ft$^3$. 2,000 lb/t / 117 lb/ft$^3$ = 17.0 ft$^3$/t.

Tonnage factors were assigned to each model block according to the combination of modeled lithology and oxidation.

There was a significant change in approach for assigning density in the 2013 model compared to 2012. After developing the new density values using the combined density database, SRK deemed it necessary to apply the oxide (lower) density to a larger volume of rock than had been applied in 2012. In 2012, all of the igneous rock was assigned a density of 11.7 ft$^3$/t with no differentiation between oxidized and un-oxidized. In 2014, SRK modeled a significant amount of oxide in the Seligman Stock from the Seligman resource area; hence, the density in much of the stock was reduced from 11.7 to 12.4 ft$^3$/t. This resulted in a significant reduction in its overall tonnage contribution to ore and, to a lesser extent, waste.

Similarly, in 2012, only strongly oxidized skarn and hornfels rock types were coded with lower density (12.34 ft$^3$/t) in the model. Weak and moderate oxide zones were assigned the typical skarn value of 10.51 $ft^3$/t. In 2014, after a detailed review of the descriptions of each of the density samples, SRK concluded that both the moderately and strongly oxidized skarn and hornfels units required a density reduction to 12.4 $ft^3$/t. As a result there was a significant loss of approximately 6% of the tonnage of skarn and hornfels in mineralized zones. The net effect is a more conservative 2014 model compared to 2012, which SRK believes is more representative of tons to be mined and facilitates more accurate mine planning.

## 12.7 Assay Capping and Compositing

Raw assays for total Au and Ag, inside the mineralized zone were capped prior to compositing. Raw assay statistics for assays in the mineralized zone are shown in Table 12.7.1. There were no cyanide silver data for domain 6.

**Table 12.7.1: Raw Assay Statistics by Mineral Domain Inside 0.004 oz/t Grade Shell**

| Assay | DOM | Valid | Minimum | Maximum | Mean | Std. Devn. | Co. of Variation |
|---|---|---|---|---|---|---|---|
| Au | 1 | 7,635 | 0 | 0.995 | 0.025 | 0.043 | 1.712 |
| | 2 | 1,095 | 0 | 0.158 | 0.012 | 0.015 | 1.215 |
| | 3 | 319 | 0 | 0.103 | 0.014 | 0.016 | 1.122 |
| | 4 | 1,303 | 0 | 0.490 | 0.015 | 0.021 | 1.425 |
| | 5 | 2,797 | 0 | 1.726 | 0.031 | 0.080 | 2.573 |
| | 6 | 430 | 0 | 0.830 | 0.044 | 0.077 | 1.747 |
| | 7 | 525 | 0 | 1.962 | 0.041 | 0.117 | 2.839 |
| AuCN | 1 | 7,635 | 0 | 0.661 | 0.011 | 0.028 | 2.643 |
| | 2 | 1,095 | 0 | 0.144 | 0.005 | 0.012 | 2.234 |
| | 3 | 319 | 0 | 0.068 | 0.001 | 0.006 | 7.484 |
| | 4 | 1,303 | 0 | 0.108 | 0.006 | 0.011 | 1.991 |
| | 5 | 2,797 | 0 | 0.492 | 0.004 | 0.019 | 4.762 |
| | 6 | 430 | 0 | 0.306 | 0.009 | 0.031 | 3.347 |
| | 7 | 525 | 0 | 1.218 | 0.009 | 0.058 | 6.655 |
| Ag | 1 | 7,635 | 0 | 13.798 | 0.229 | 0.603 | 2.631 |
| | 2 | 1,095 | 0 | 18.250 | 0.278 | 0.712 | 2.558 |
| | 3 | 319 | 0 | 3.900 | 0.170 | 0.418 | 2.467 |
| | 4 | 1,303 | 0 | 7.280 | 0.203 | 0.468 | 2.306 |
| | 5 | 2,797 | 0 | 13.500 | 0.149 | 0.490 | 3.290 |
| | 6 | 430 | 0 | 4.460 | 0.198 | 0.450 | 2.280 |
| | 7 | 525 | 0 | 2.630 | 0.091 | 0.236 | 2.596 |
| AgCN | 1 | 7,635 | 0 | 5.139 | 0.104 | 0.251 | 2.416 |
| | 2 | 1,095 | 0 | 4.965 | 0.113 | 0.243 | 2.155 |
| | 3 | 319 | 0 | 0.169 | 0.002 | 0.011 | 6.068 |
| | 4 | 1,303 | 0 | 3.590 | 0.033 | 0.148 | 4.424 |
| | 5 | 2,797 | 0 | 2.010 | 0.014 | 0.093 | 6.455 |
| | 6 | 430 | 0 | 0 | 0 | 0 | |
| | 7 | 525 | 0 | 0.794 | 0.006 | 0.048 | 8.518 |

Source: SRK, 2014

Capping thresholds were interpreted from cumulative probability plots (CPP) of each element in each domain. Capping values for total Au and Ag are presented in Table 12.7.2.

**Table 12.7.2: Grade Capping Values by Mineral Domain**

| Mineral Domain | Au Cap (oz/t) | Ag Cap (oz/t) |
|---|---|---|
| 1 | 0.36 | 8.46 |
| 2 | 0.06 | 3.28 |
| 3 | 0.06 | 1.83 |
| 4 | 0.12 | 4.00 |
| 5 | 0.56 | 2.84 |
| 6 | 0.27 | 1.40 |
| 7 | 0.30 | 1.50 |

Source: SRK, 2014

Assay intervals were typically the standard 5 ft as drilled. These were converted to 10 ft fixed-length down-hole composites for resource estimation. By comparison, the 2012 Centennial model was estimated using 20 ft fixed length composites. Higher resolution in Centennial has resulted in less edge smoothing. Statistics for the composites are presented in Table 12.7.3. Total and CN-soluble gold and silver values were used for calculated fields. There were no cyanide silver data for domain 6.

**Table 12.7.3: Composite Statistics by Model Domain Inside 0.004 oz/t Grade Shell**

| Assay | DOM | Valid | Minimum | Maximum | Mean | Std. Devn. | Variance | Co. of Variation |
|---|---|---|---|---|---|---|---|---|
| AuTotCAP | 1 | 4,084 | 0 | 0.320 | 0.022 | 0.031 | 0.0010 | 1.410 |
| | 2 | 621 | 0 | 0.059 | 0.011 | 0.009 | 0.0001 | 0.859 |
| | 3 | 196 | 0 | 0.055 | 0.012 | 0.010 | 0.0001 | 0.885 |
| | 4 | 792 | 0 | 0.105 | 0.013 | 0.013 | 0.0002 | 0.987 |
| | 5 | 1,608 | 0 | 0.560 | 0.026 | 0.051 | 0.0026 | 2.001 |
| | 6 | 265 | 0 | 0.249 | 0.035 | 0.042 | 0.0018 | 1.196 |
| | 7 | 328 | 0 | 0.294 | 0.027 | 0.043 | 0.0018 | 1.574 |
| | **Total** | **7,894** | **0** | **0.560** | **0.021** | **0.035** | **0.0012** | **1.640** |
| AuCNCAP | 1 | 4,084 | 0 | 0.223 | 0.009 | 0.021 | 0.0004 | 2.302 |
| | 2 | 621 | 0 | 0.059 | 0.005 | 0.008 | 0.0001 | 1.751 |
| | 3 | 196 | 0 | 0.048 | 0.001 | 0.004 | 0 | 6.305 |
| | 4 | 792 | 0 | 0.069 | 0.005 | 0.009 | 0.0001 | 1.828 |
| | 5 | 1,608 | 0 | 0.300 | 0.003 | 0.015 | 0.0002 | 4.308 |
| | 6 | 265 | 0 | 0.152 | 0.008 | 0.022 | 0.0005 | 2.835 |
| | 7 | 328 | 0 | 0.289 | 0.007 | 0.025 | 0.0006 | 3.638 |
| | **Total** | **7,894** | **0** | **0.300** | **0.007** | **0.018** | **0.0003** | **2.685** |
| AgTotCAP | 1 | 4,084 | 0 | 6.810 | 0.200 | 0.416 | 0.1734 | 2.086 |
| | 2 | 621 | 0 | 2.890 | 0.239 | 0.348 | 0.1209 | 1.454 |
| | 3 | 196 | 0 | 1.340 | 0.135 | 0.214 | 0.0458 | 1.586 |
| | 4 | 792 | 0 | 3.645 | 0.172 | 0.303 | 0.0918 | 1.758 |
| | 5 | 1,608 | 0 | 2.179 | 0.123 | 0.244 | 0.0594 | 1.984 |
| | 6 | 265 | 0 | 1.272 | 0.153 | 0.243 | 0.0589 | 1.588 |
| | 7 | 328 | 0 | 1.250 | 0.078 | 0.159 | 0.0254 | 2.048 |
| | **Total** | **7,894** | **0** | **6.810** | **0.176** | **0.355** | **0.1262** | **2.018** |
| AgCNCAP | 1 | 4,084 | 0 | 2.576 | 0.090 | 0.169 | 0.0284 | 1.866 |
| | 2 | 621 | 0 | 1.809 | 0.103 | 0.166 | 0.0275 | 1.617 |
| | 3 | 196 | 0 | 0.085 | 0.002 | 0.008 | 0.0001 | 4.449 |
| | 4 | 792 | 0 | 1.920 | 0.028 | 0.108 | 0.0116 | 3.848 |
| | 5 | 1,608 | 0 | 1.158 | 0.012 | 0.067 | 0.0045 | 5.497 |
| | 6 | 265 | 0 | 0 | 0 | 0 | 0 | |
| | 7 | 328 | 0 | 0.494 | 0.005 | 0.037 | 0.0014 | 7.064 |
| | **Total** | **7,894** | **0** | **2.576** | **0.060** | **0.143** | **0.0205** | **2.373** |

Source: SRK, 2014

## 12.8 Variogram Analysis and Modeling

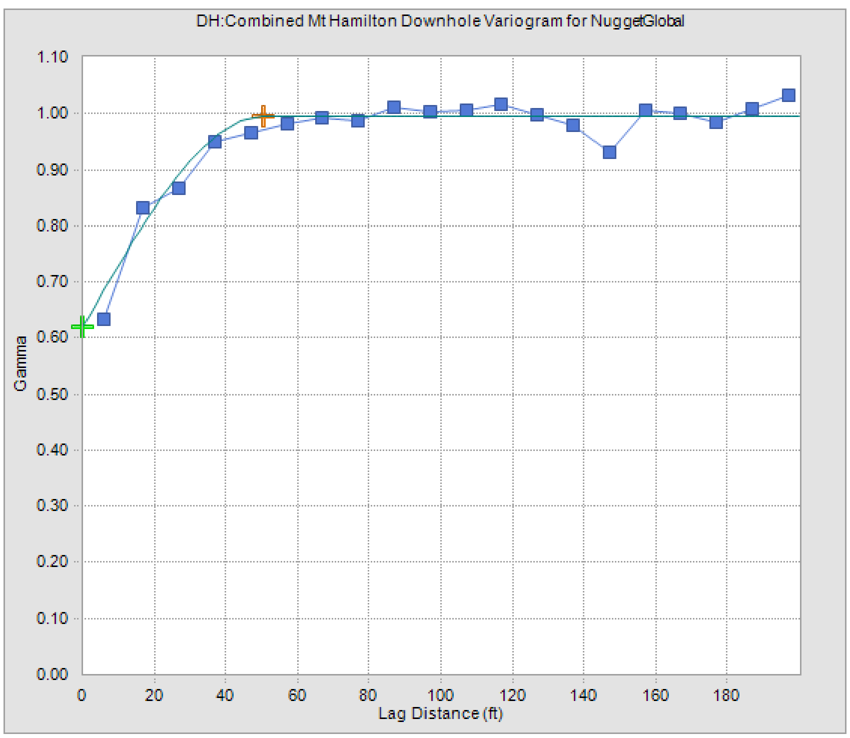
The variogram analysis was carried out on composites inside the mineralized zone by domain. SRK used the MineSight® Data Analyst tool kit to develop correlograms, which normalize the sill to 1.0. As with most gold deposits, grade variability and nugget effect result in marginal variograms. A satisfactory result was achieved when using a global omni-directional variogram for Au. From this variogram, the global nugget value was established at 0.62. Figure 12.8.1 illustrates the downhole variograms for Au.

This nugget value was fixed for the remainder of the analysis to establish ranges (lags) to facilitate grade estimation and resource classification. The global variogram for all Au composites in all domains is illustrated in Figure 12.8.2.

A range of 150 ft was modeled from this variogram. SRK used 2/3 of this range (100 ft) as the default intermediate range for grade estimation.

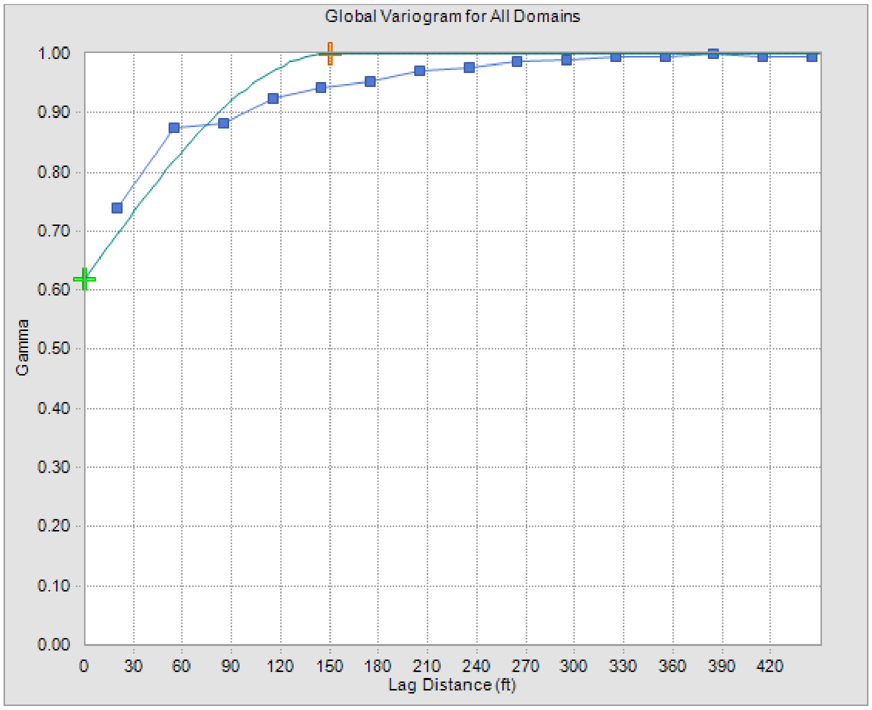
Domain 1 (Centennial Skarn) had the largest number of composites and an independent variogram analysis was performed on this domain. Using the original nugget value of 0.62, a maximum range of 170 ft was modeled from a dip direction of 140° and a dip of -15°. Using the typical 2/3 range approach for defining search criteria, the intermediate range for estimation in this domain was 115 ft. Figure 12.8.3 illustrates the variogram for Au in Domain 1.

Variograms for silver were more erratic than gold with a higher nugget value (0.70) but longer ranges. SRK was not comfortable with the variograms enough to attach detailed search parameters to the variograms. Instead, fairly generic search criteria were used for silver based loosely on the silver variograms but also on ranges defined for gold.



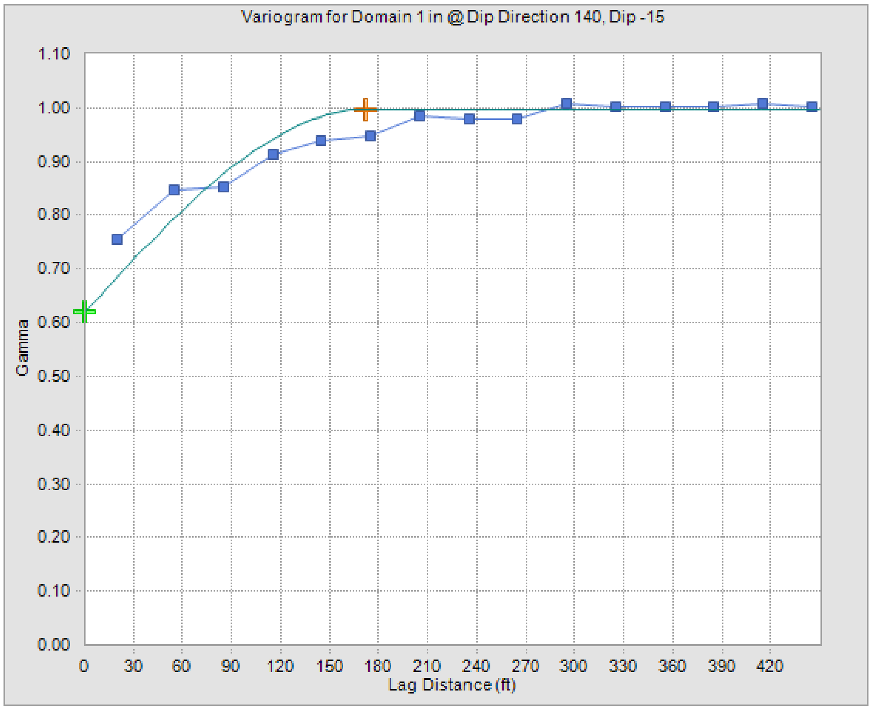

Source: SRK, 2014

**Figure 12.8.1: Downhole Variogram for Gold - All Samples in MZONE = 1**




Source: SRK, 2014

**Figure12.8.2: Global Variogram for Gold - All Samples in All Domains**




Source: SRK, 2014

**Figure 12.8.3: Variogram of Au Composites in Domain 1 (Centennial Skarn)**

## 12.9 Block Model

The Project coordinate system established during mining and exploration in the 1990's is still used. It is based on the Nevada State Plane (NV SP), East Zone projection, 1927 North American Datum. Northing coordinates in the mine grid system are one million less than the Nevada State Plane coordinates, but no other transformations were done to alter the Project coordinates from the NV SP coordinates. Coordinate system units are feet, and all units presented in this document are U.S. units (feet, miles, troy ounces, short tons, troy ounces per short ton, etc.).

The Mt. Hamilton block model has the spatial characteristics and limits shown in Table 12.9.1. The model was not rotated. It was built using full blocks and block percents that were coded to honor mineral domain wireframes. The resource block model was constructed using Mintec's MineSight® 3D mining software.

**Table 12.9.1: Mt. Hamilton 3D Block Model Origin and Extents (ft)**

| Item | Minimum | Maximum | Size | Blocks |
|---|---|---|---|---|
| Easting | 506,000 | 509,700 | 20 | 185 |
| Northing | 635,200 | 642,000 | 20 | 340 |
| Elevation | 7,650 | 9,650 | 10 | 200 |
| **Total** | | | | **12,580,000** |

Source: SRK, 2014

The 20 ft x 20 ft x 10 ft (XYZ) block size for the models was selected to approximate the selective mining unit (SMU) and a 10 ft bench height. The average sample spacing in the data set is approximately 125 ft with spacing approaching 75 ft in Centennial and the north Seligman skarn area.

Whole blocks were coded with 3D wireframes of lithology, oxidation, sulfide, and model domains using a 50% rule, while a block percent was used to code topography and the mineral domains (gradeshells) for increased accuracy in reporting. Wireframe solids were also generated for expanded "diluted" gradeshells. These were coded as a block percent and handled like the primary gradeshells to refine grade reporting.

SRK initially created an exploration block model, and used select items from it to build an engineering model. Geology and mineralization were modeled to original pre-mining topography in the Exploration Model. The Engineering model used current topography to allow for better identification and characterization of surface backfill. The Engineering model also incorporated items for calculating dilution and recovery for conversion of resources to reserves. The Engineering model will be described in Section 13 of this report.

## 12.10 Grade Estimation Methodology

Total gold and silver grades were estimated using the inverse distance squared (IDW) algorithm. As is frequently the case for gold deposits, SRK was not comfortable using the variograms for kriging, mostly because of the high nugget value. Grades estimation was repeated using polygonal methods (nearest neighbor) to facilitate model validation. The SRK polygonal method used one composite to estimate each block and applied anisotropy similar to that used the IDW estimate.

A three-pass approach was used to estimate grades in which a first short-range search was used to inform block at roughly 1/3 of the variogram range. A second intermediate search was then applied to inform blocks at roughly 2/3 the variogram range. Finally, a third pass was applied with a long range to estimate all blocks inside the mineralized envelope not previously estimated in the first two passes. In each successive estimation pass, the composites used were flagged and then omitted from use in subsequent estimation passes.

Grade estimation procedures (methods and ranges) for total Au and total Ag used in 2013 modeling were nearly identical to those used in 2012, refined only by a larger data set and more comprehensive variography.

Based on the variogram analysis of the composites in the combined 2013 Mt. Hamilton database, search criteria were established for estimating Au and Ag. These criteria are presented in Table 12.10.1 and Table 12.10.2 for Au and Ag, respectively.

**Table 12.10.1: Gold Grade IDW Estimation Criteria**

| Domain Name | Domain No. | Search Pass | Search Ellipse Range (ft) | | | Search Orientation (degrees) | | | No. Composites | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Major | Semi-Major | Minor | Z | X' | Y' | Min per block | Max per block | Max per hole |
| Cent-SK | 1 | Short | 60 | 45 | 10 | 140 | -10 | 0 | 4 | 8 | 1 |
| | | Medium | 115 | 95 | 15 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 30 | | | | 1 | 8 | 1 |
| Cent-IG | 2 | Short | 50 | 50 | 10 | 55 | -20 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 15 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 20 | | | | 1 | 8 | 1 |
| Selig-IG-S | 3 | Short | 50 | 50 | 10 | 95 | -12 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 15 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 20 | | | | 1 | 8 | 1 |
| Selig-IG-N | 4 | Short | 50 | 50 | 10 | 25 | -15 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 15 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 20 | | | | 1 | 8 | 1 |
| Selig-SK-N | 5 | Short | 50 | 50 | 20 | 20 | -25 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 30 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 40 | | | | 1 | 8 | 1 |
| Selig-SK-E | 6 | Short | 50 | 50 | 20 | 345 | -20 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 50 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 75 | | | | 1 | 8 | 1 |
| Selig-SK-SE | 7 | Short | 50 | 50 | 20 | 20 | -10 | 0 | 4 | 8 | 1 |
| | | Medium | 100 | 100 | 30 | | | | 3 | 8 | 1 |
| | | Long | 350 | 300 | 40 | | | | 1 | 8 | 1 |

Source: SRK, 2014

Silver grades were estimated by criteria very similar to gold grades. Most of the variograms for Ag demonstrated ranges (lags) of greater than 200 ft, but the variograms had a high nugget value of 0.70, so confidence in the variography was low. SRK applied generic search ranges to silver weakly supported by variography and in line with ranges determined for Au. Search orientations and composite usage protocols for Ag were identical to Au in all other aspects.

**Table 12.10.2: Silver Grade IDW Estimation Criteria**

| Domain Name | Domain No. | Search Pass | Search Ellipse Range (ft) | | |
|---|---|---|---|---|---|
| | | | Major | Semi-Major | Minor |
| All | 1-7 | Short | 50 | 50 | 10 |
| | | Medium | 100 | 100 | 30 |
| | | Long | 300 | 300 | 55 |

Source: SRK, 2014

Cyanide-soluble gold (AuCN) values were first interpolated inside the CN-soluble gold gradeshell to decluster the AuCN analytical values. The declustered values were then paired with gold fire assay (AuFA) values in each block to calculate AuCN/FA ratios. By domain, the paired data were used to develop regression equations. The domain-specific equations were used to calculate AuCN values in drillhole composites where they were absent. Once populated in all available drillhole composites, AuCN was estimated using the same parameters as those used for total Au (cf. Table 12.10.1).

Cyanide-soluble silver (AgCN) was not estimated.

## 12.11 Model Validation

Various measures were implemented to validate the Mt. Hamilton resource block model. These measures included the following:

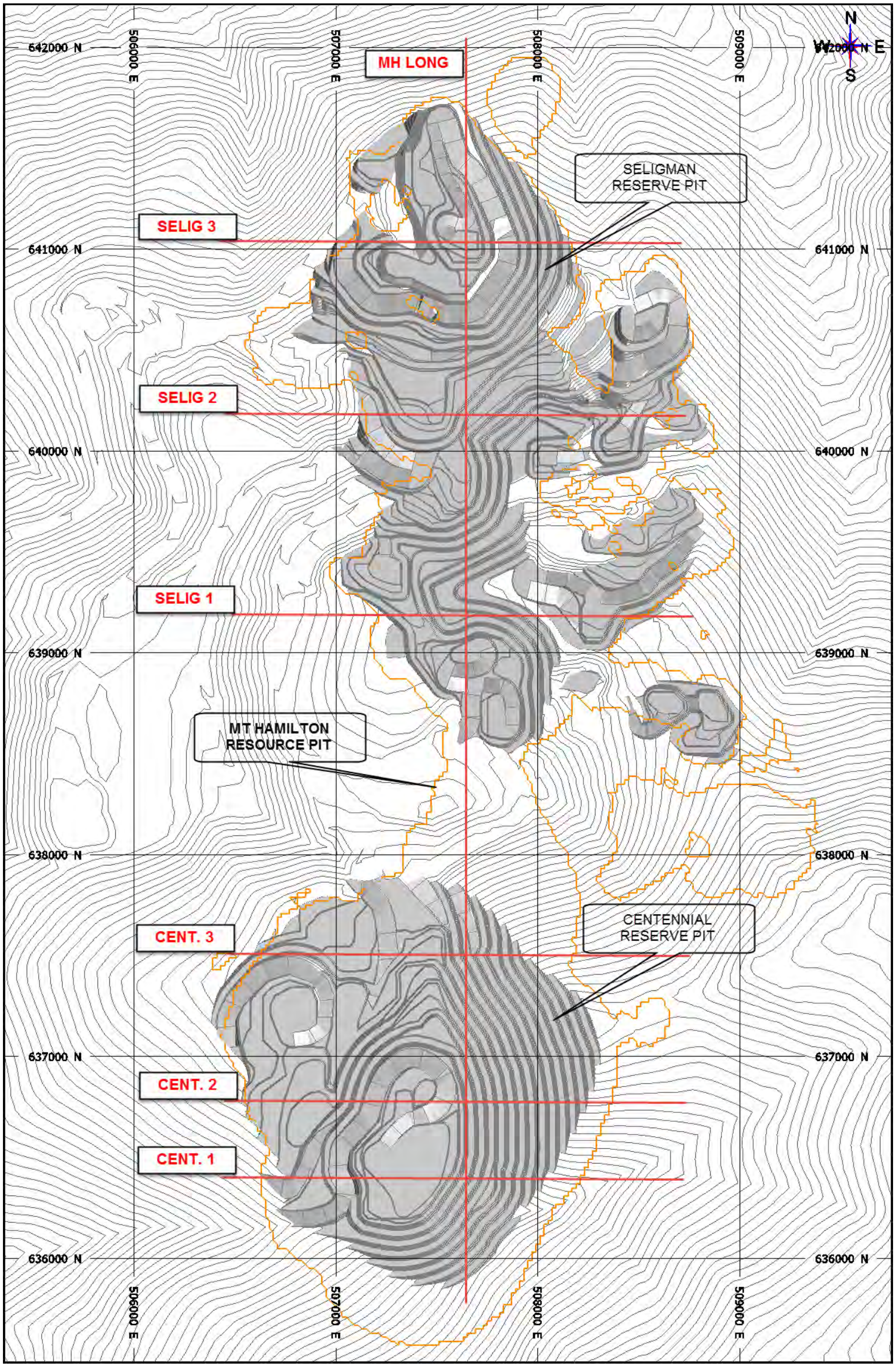
- Comparison of drillhole composites with resource block grade estimates from all zones visually in both plan and section;
- Statistical comparisons between block and composite data using distribution analyses;
- Comparison of IDW to a nearest neighbor (NN) model; and
- Swath plot analysis (drift analysis) comparing the inverse distance model with the NN model.

**<u>Visual Comparison</u>**

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement. A plan view for cross-section locations (traces) is presented in Figure 12.11.1

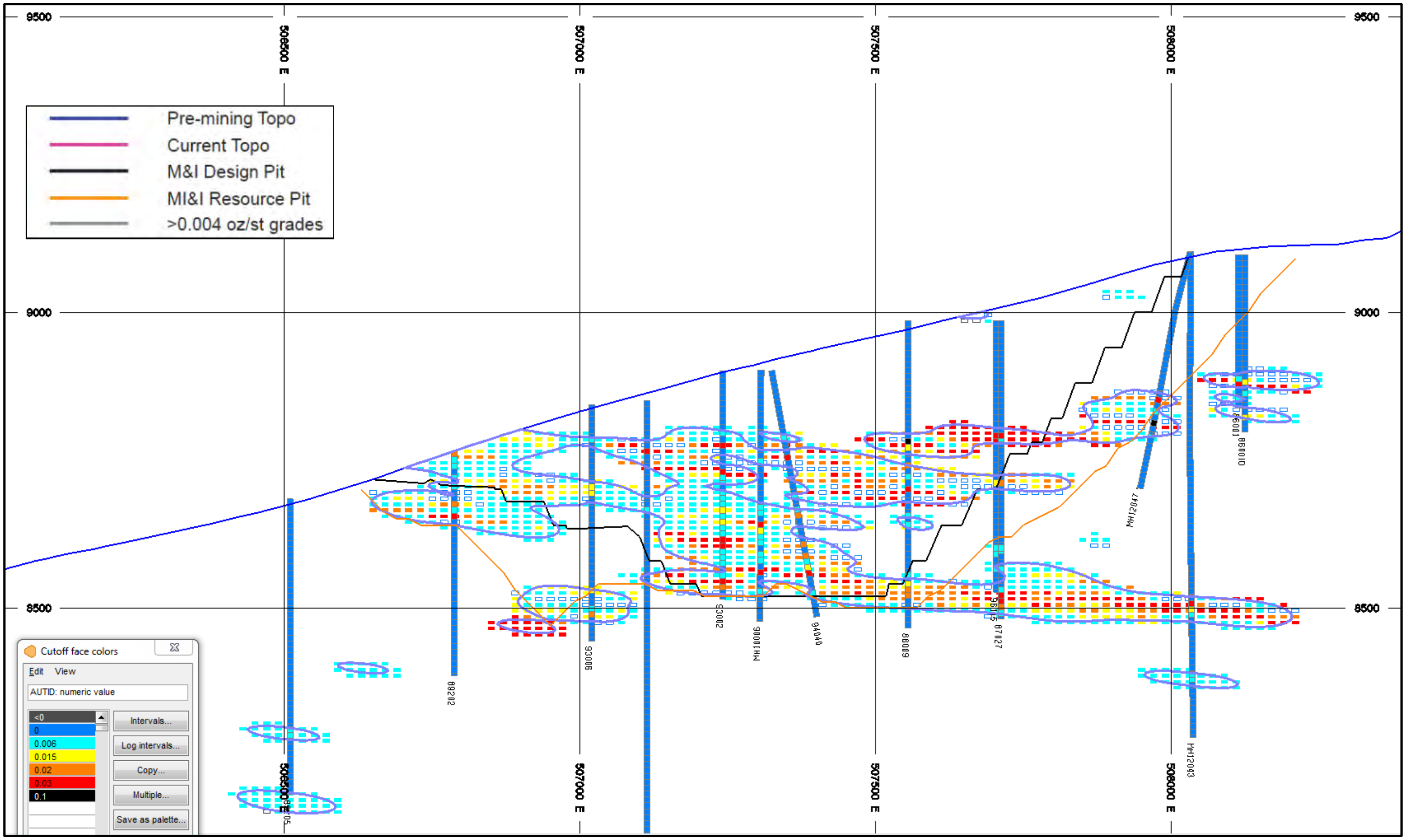
Example cross-section views showing block values and composite grades within their respective grade shells are provided in Figure 12.11.2 through Figure 12.11.8. The color code key for gold grades is captured in each image in oz/t. Block color codes match drillhole color codes. The view window for drillholes is +/- 35 ft from the section. The grey boundary around mineralization is the 0.004 oz/t grade shell. The ultimate resource-limiting LG pit is shown in orange, built on Measured, Indicated and Inferred resources. The current design pit for mine planning is shown in black, based on Measured and Indicated resources only. Original topography is shown in blue. Current topography (mined and back filled) is magenta. The two topographic surfaces diverge north of Cabin Gulch (fill) and in Seligman depicting previous mining.

The north Seligman area was re-drilled by Mt. Hamilton in 2014 as a priority to upgrade the resource classification of material from Inferred to Indicated. The south Seligman/north Centennial area (637500-639500N) included in Figure 12.11.8 has not yet been infill-drilled by Mt. Hamilton, LLC. Therefore, there exists an opportunity to add mineable resources both by upgrading the classification through infill drilling, and by expanding the mineralized zone.



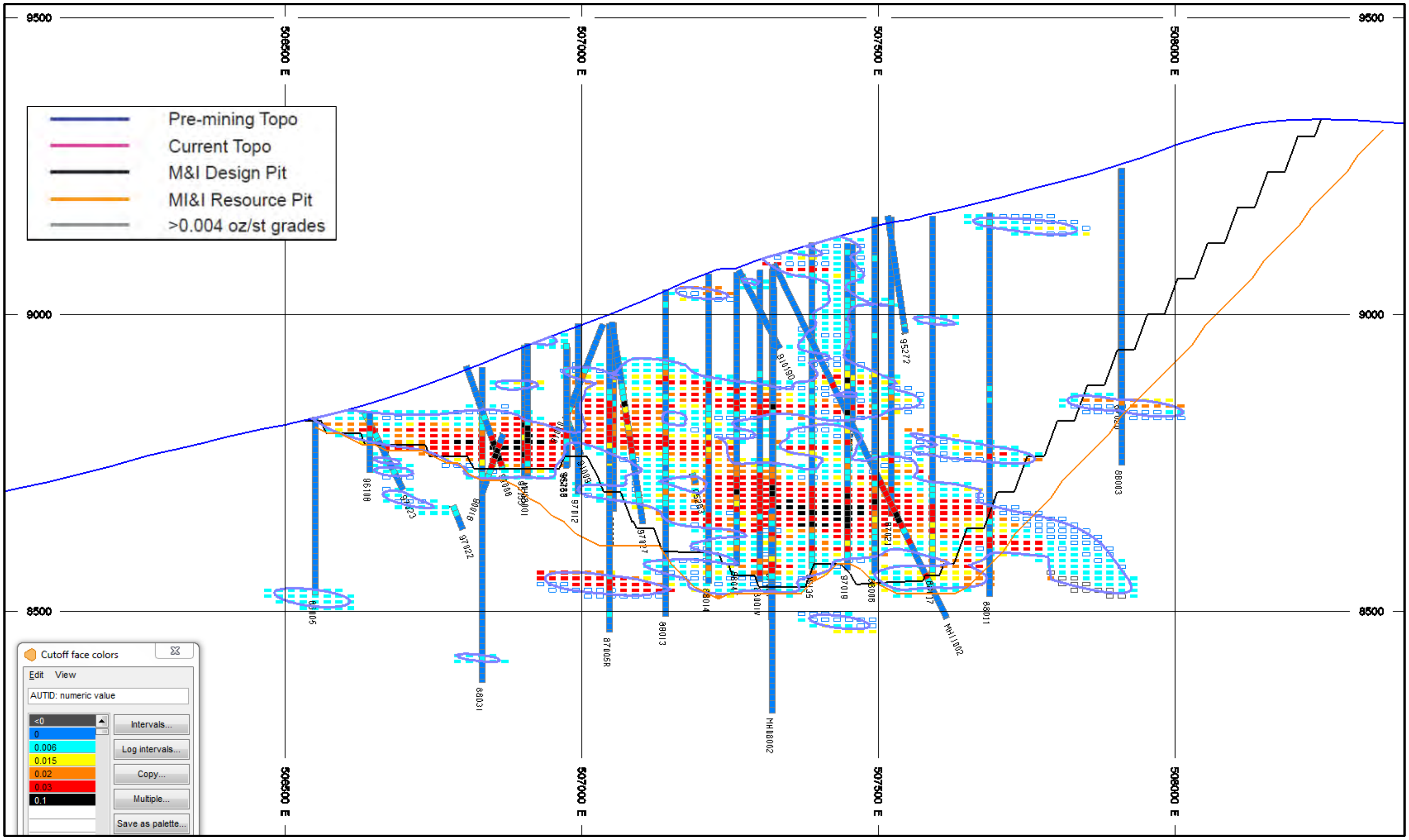

Source: SRK, 2014

**Figure 12.11.1: Plan View of Mt. Hamilton with Cross Section**



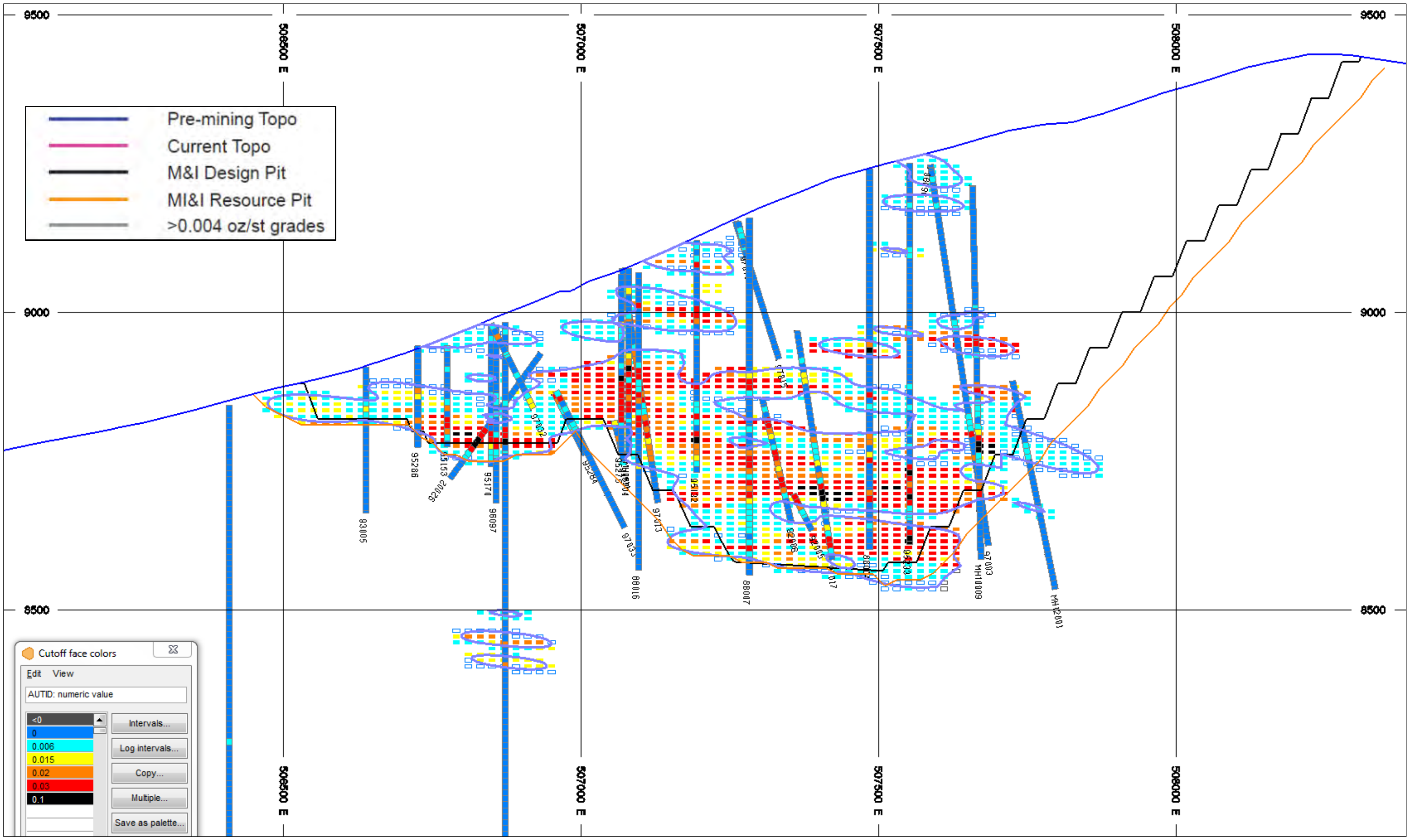

Source SRK, 2014

**Figure 12.11.2: East-West Cross Section "Cent.1" at 636390N – Drill Hole and Model Gold Grades**



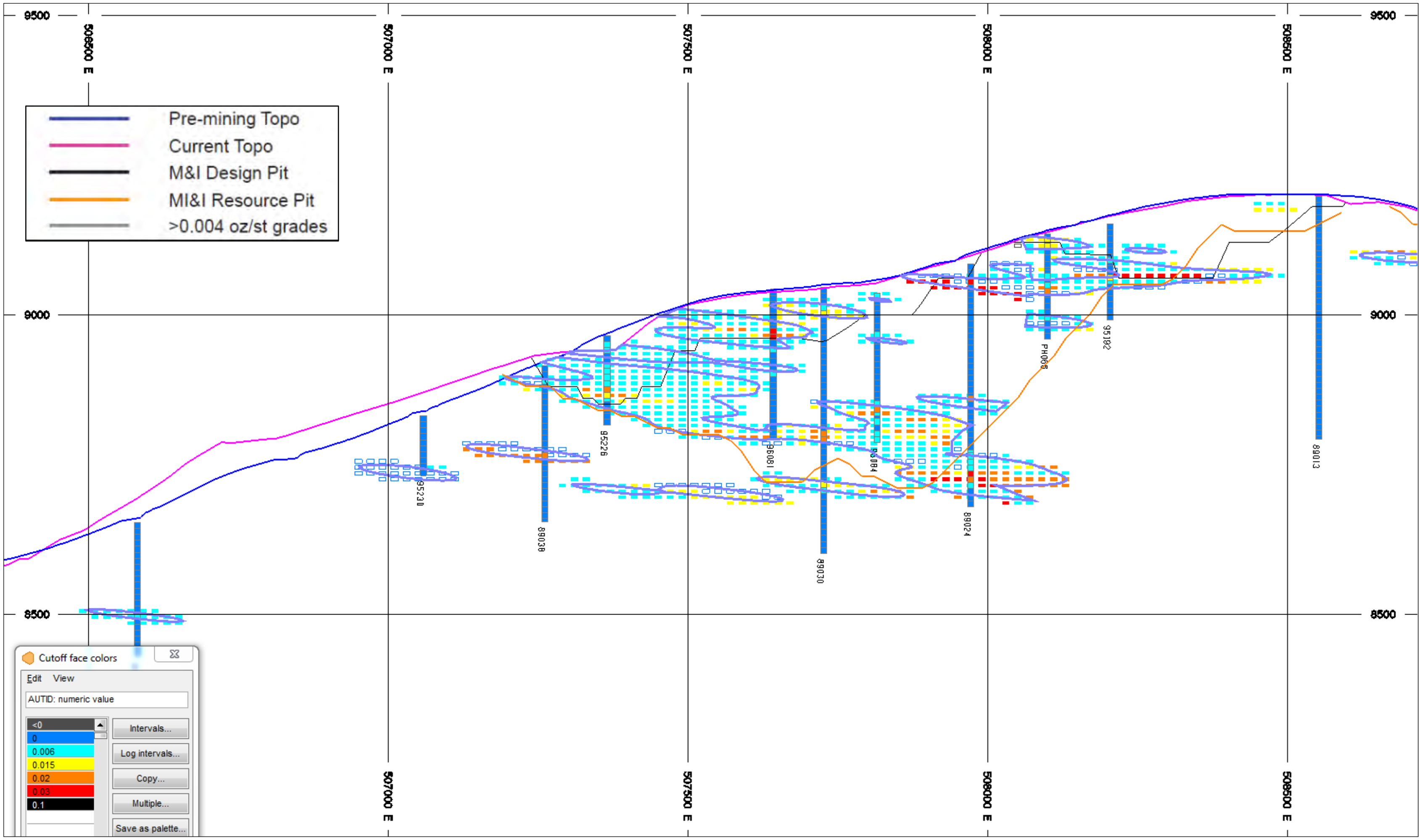

Source SRK, 2014

**Figure 12.11.3: East-West Cross Section “Cent.2” at 636770N – Drill Hole and Model Gold Grades**



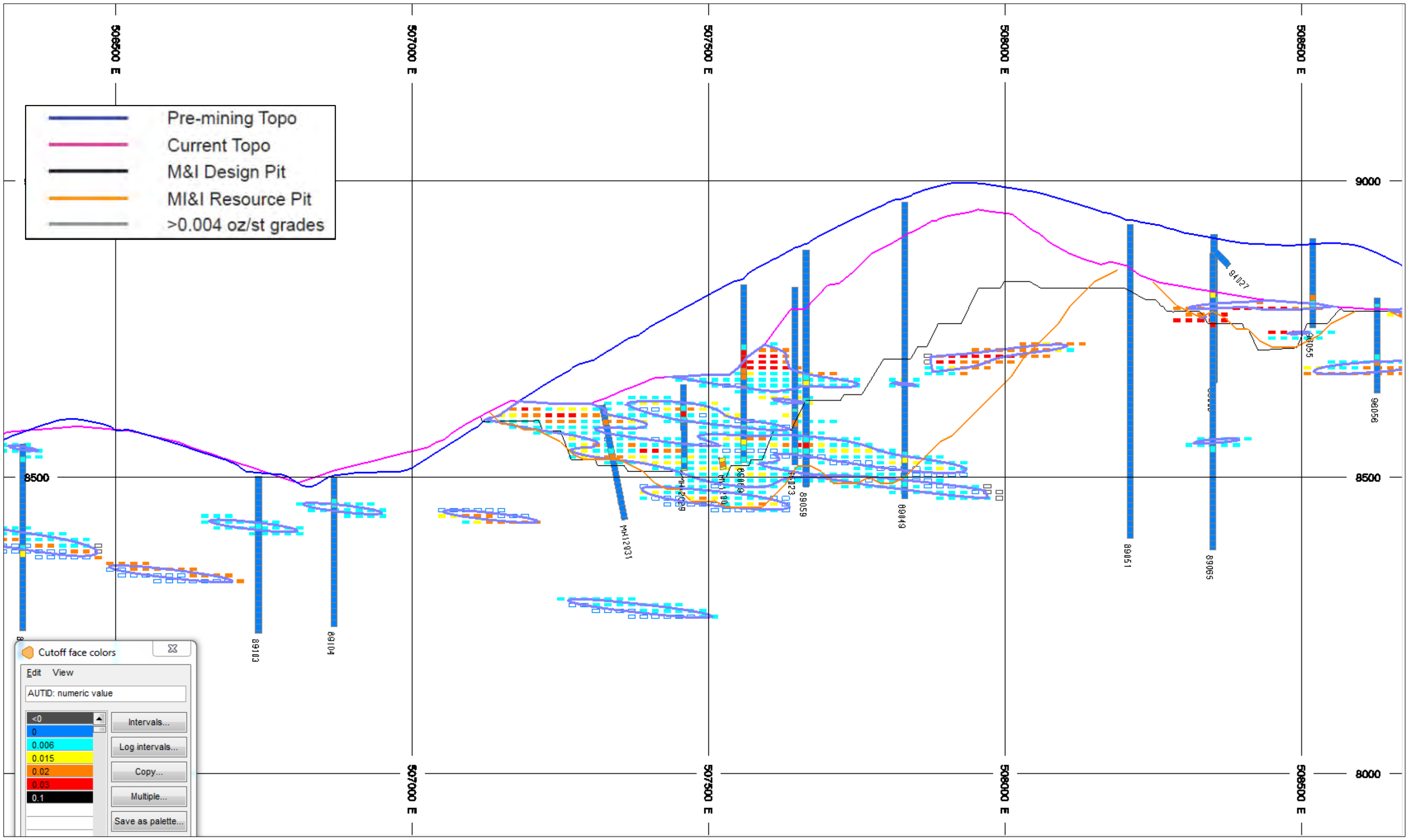

Source SRK, 2014

**Figure 12.11.4: East-West Cross Section "Cent.3" at 637490N – Drill Hole and Model Gold Grades**



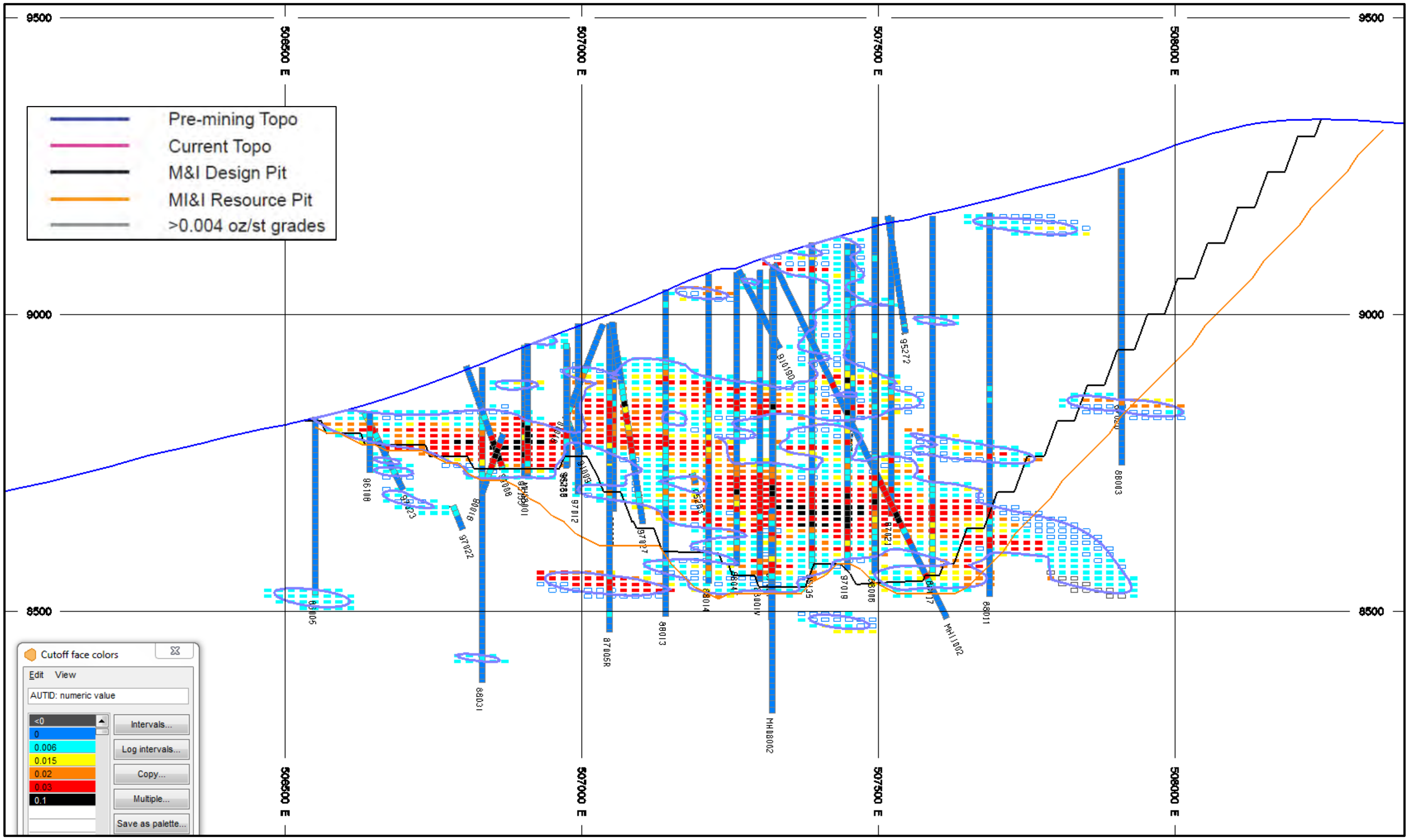

Source SRK, 2014

**Figure 12.11.5: East-West Cross Section "Selig.1" at 639190N – Drill Hole and Model Gold Grades**



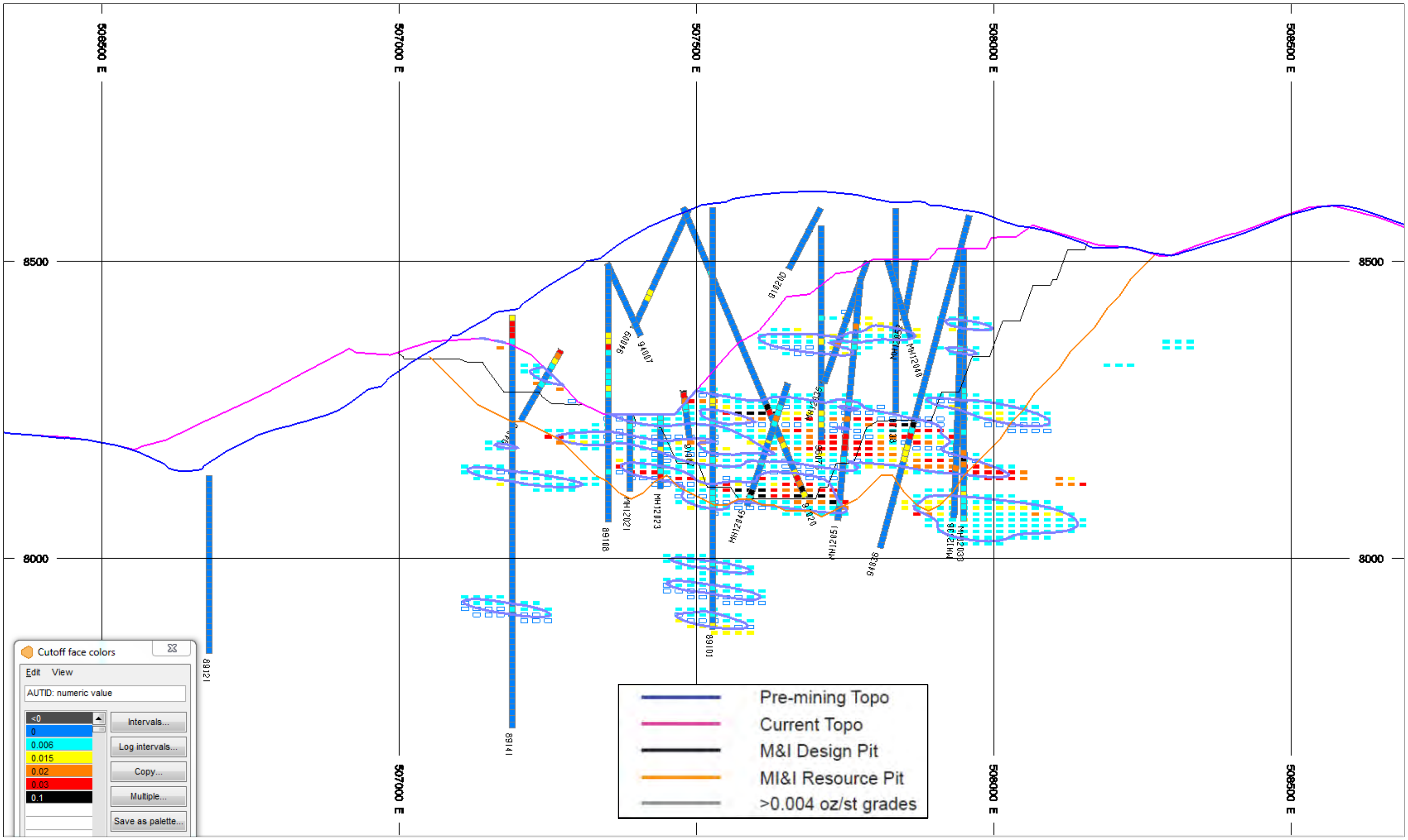

Source SRK, 2014

**Figure 12.11.6: East-West Cross Section “Selig.2” at 640170N – Drill Hole and Model Gold Grades**




Source SRK, 2014

**Figure 12.11.7: East-West Cross Section “Selig.3” at 641030N – Drill Hole and Model Gold Grades**




Source SRK, 2014

**Figure 12.11.8: North-South Longitudinal Section "MH-Long" at 507650E – Drill Hole and Model Gold Grades**

## Block-Composite Statistical Comparison

SRK conducted statistical comparisons between the IDW blocks contained within mineral domains and their underlying composite grades. Tables comparing composites to blocks are shown in Table 12.11.1 and Table 12.11.2 for Au and Ag respectively. Tons weighted block estimates for Au were 14% lower than supporting composites. Tons weighted block estimates for Ag were 3% lower. Both comparisons were made inside the mineralized envelope with no CoG applied.

**Table 12.11.1: Statistical Comparison of Blocks to Composites for Gold**

| Domain | Mass | Composites | Blocks | Cmp>Blk |
|---|---|---|---|---|
| | kt | Au oz/t | Au oz/t | % Diff (1) |
| 1 | 28,427 | 0.022 | 0.018 | 17% |
| 2 | 9,445 | 0.011 | 0.010 | 6% |
| 3 | 5,345 | 0.012 | 0.011 | 0% |
| 4 | 18,028 | 0.012 | 0.012 | -1% |
| 5 | 19,536 | 0.026 | 0.020 | 25% |
| 6 | 7,021 | 0.035 | 0.031 | 12% |
| 7 | 10,956 | 0.027 | 0.023 | 13% |
| **Total** | **98,759** | **0.021** | **0.018** | **14%** |

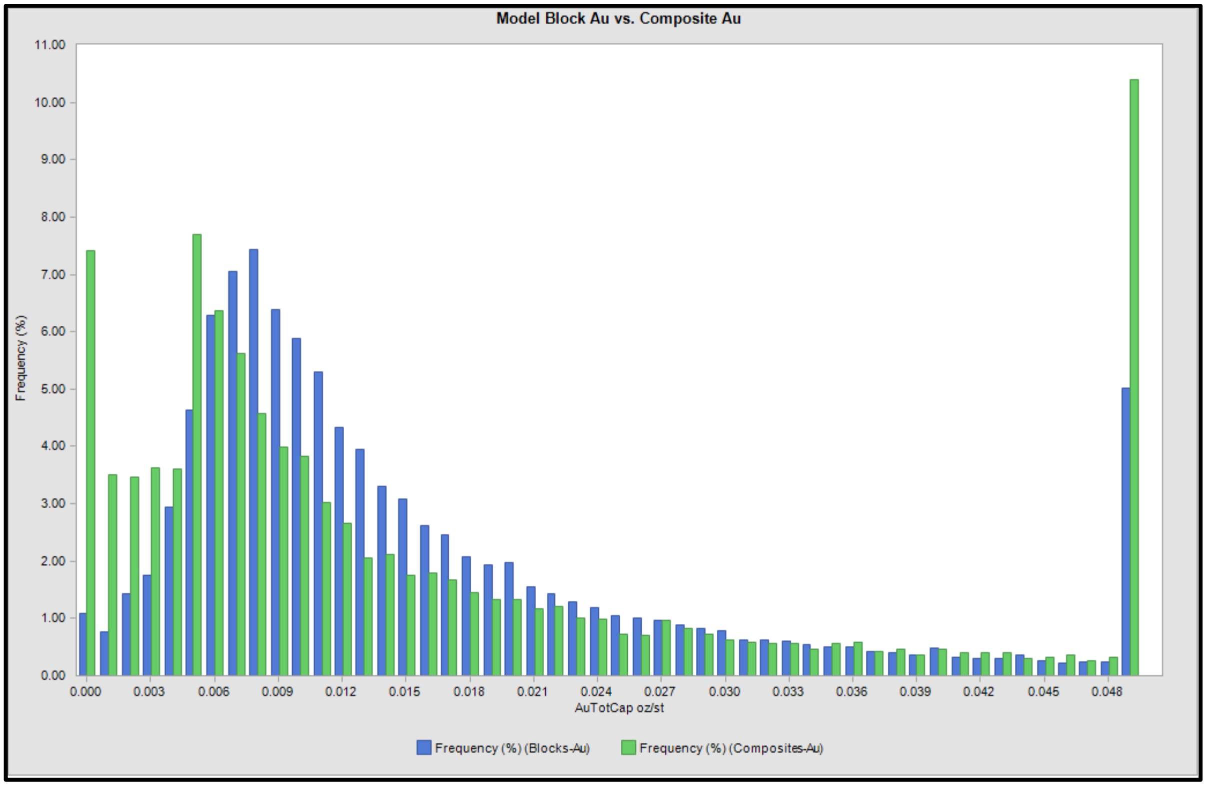
Source: SRK, 2014
(1) (Original Value – Duplicate Value) / (Average Value) * 100

**Table 12.11.2: Statistical Comparison of Blocks to Composites for Silver**

| Domain | Mass | Composites | Blocks | Cmp>Blk |
|---|---|---|---|---|
| | kt | Ag oz/t | Ag oz/t | % Diff (1) |
| 1 | 28,427 | 0.196 | 0.182 | 7% |
| 2 | 9,445 | 0.233 | 0.217 | 7% |
| 3 | 5,345 | 0.119 | 0.151 | -23% |
| 4 | 18,028 | 0.168 | 0.174 | -4% |
| 5 | 19,536 | 0.121 | 0.107 | 12% |
| 6 | 7,021 | 0.147 | 0.129 | 13% |
| 7 | 10,956 | 0.071 | 0.093 | -27% |
| **Total** | **98,759** | **0.158** | **0.154** | **3%** |

Source: SRK, 2014
(1) (Original Value – Duplicate Value) / (Average Value) * 100

A histogram comparing block and composite gold grades is provided in Figure 12.11.9. This comparison shows that the model grade distribution for gold is appropriately smoothed when compared with the underlying composite distribution.




Source: SRK, 2014

**Figure 12.11.9: Histogram Comparing Block Au oz/t to Composite Au oz/t Distribution**

## Comparison of Interpolation Methods

For comparative purposes, grades were also estimated using NN interpolation methods. The results of the NN model are compared to the IDW model at a zero CoG in Table 12.11.3 and Table 12.11.4 for Au and Ag respectively. This comparison confirms conservation of metal at a zero cut-off, and shows an overall agreement on both a grade and total metal for the two estimation methods. Block diluted grades were used in the resource statement for all metals.

**Table 12.11.3: Comparison of IDW and NN Tonnage and Grade at a Zero Cut-off for Au**

| Domain | Mass | Au NN | AU IDW | NN>IDW |
|---|---|---|---|---|
| | kt | Au oz/t | Au oz/t | % Diff[1] |
| 1 | 28,427 | 0.018 | 0.018 | -2% |
| 2 | 9,445 | 0.010 | 0.010 | -1% |
| 3 | 5,345 | 0.011 | 0.011 | -5% |
| 4 | 18,028 | 0.012 | 0.012 | -3% |
| 5 | 19,536 | 0.020 | 0.020 | 0% |
| 6 | 7,021 | 0.030 | 0.031 | -5% |
| 7 | 10,956 | 0.023 | 0.023 | -3% |
| **Total** | **98,759** | **0.018** | **0.018** | **-2%** |

Source: SRK, 2014
(1) (Original Value – Duplicate Value) / (Average Value) * 100

**Table 12.11.4: Comparison of IDW and NN Tonnage and Grade at a Zero Cut-off for Ag**

| Domain | Mass | Au NN | AU IDW | NN>IDW |
|---|---|---|---|---|
| | kt | Au oz/t | Au oz/t | % Diff (1) |
| 1 | 28,427 | 0.181 | 0.182 | -1% |
| 2 | 9,445 | 0.224 | 0.217 | 3% |
| 3 | 5,345 | 0.149 | 0.151 | -1% |
| 4 | 18,028 | 0.173 | 0.174 | -1% |
| 5 | 19,536 | 0.103 | 0.107 | -4% |
| 6 | 7,021 | 0.130 | 0.129 | 1% |
| 7 | 10,956 | 0.098 | 0.093 | 5% |
| **Total** | **98,759** | **0.154** | **0.154** | **0%** |

Source: SRK, 2014
(1) (Original Value – Duplicate Value) / (Average Value) * 100

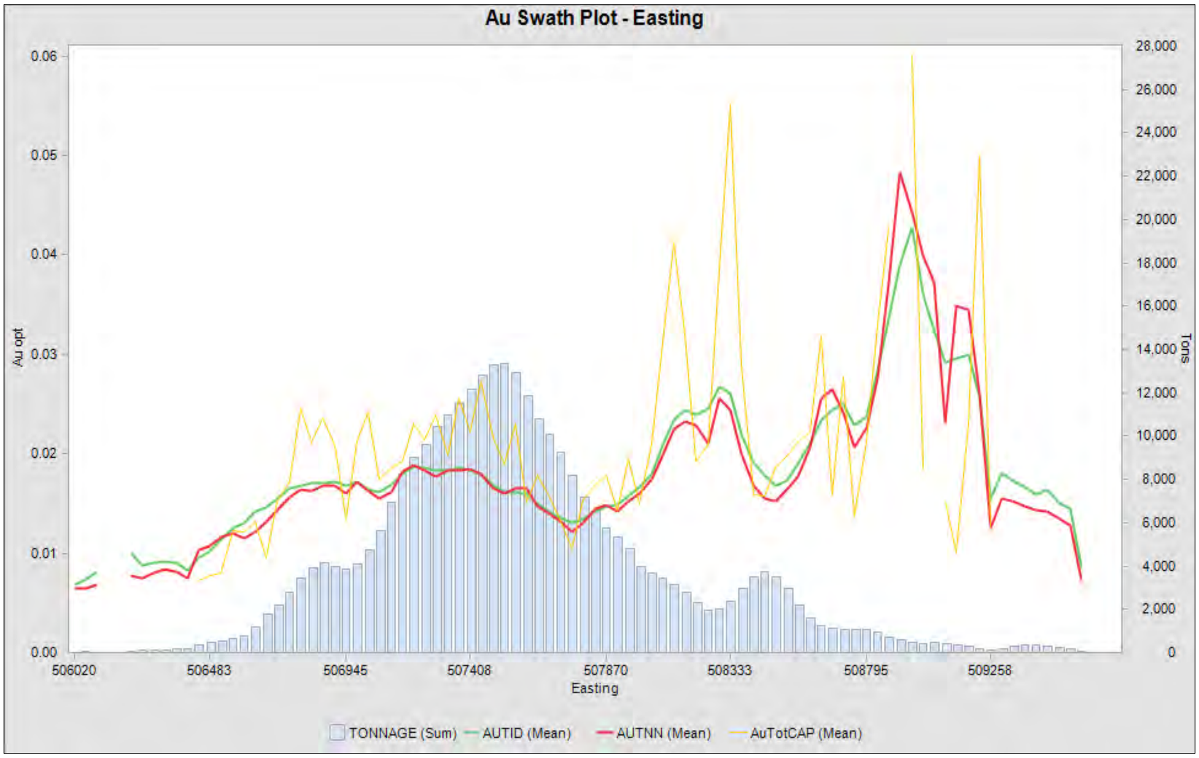
## Swath Plots (Drift Analysis)

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 IDW model are compared to the distribution derived from the NN grade model and source composites.

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 IDW model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend of the IDW should be similar to the NN distribution of grade.

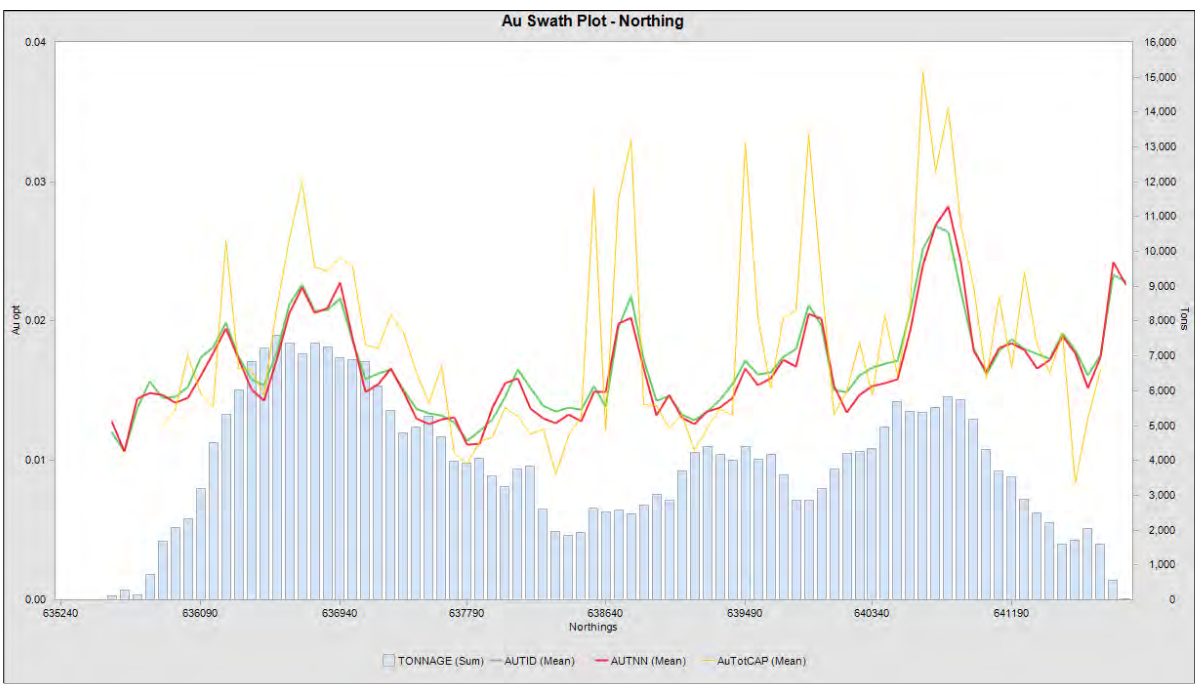
Swath plots were generated for gold and silver along east-west and north-south directions, and also for elevation. Swath widths were 40, 80, and 20 ft wide for east-west, north-south and elevation, respectively. Items plotted include total gold blocks by inverse distance (AUTID), total gold blocks by nearest neighbor (AUNN), gold composites (AuTotCap), total silver blocks by inverse distance (AGTID), total silver blocks by nearest neighbor (AGTNN) and silver composites (AGTID). The swath plots are shown in Figure 12.11.10 through Figure 12.11.15, inclusive.

According to the swath plots, there is good correspondence between the modeling methods. The degree of smoothing in the IDW model is evident in the peaks and valleys shown in some swath plots; however, this comparison shows close agreement between the IDW and NN models in terms of overall grade distribution as a function of easting, northing and elevation especially where there are high tonnages (vertical bars on the plots). The plots also demonstrate the high degree of variance of the input composites and the model smoothing of the composite grades.



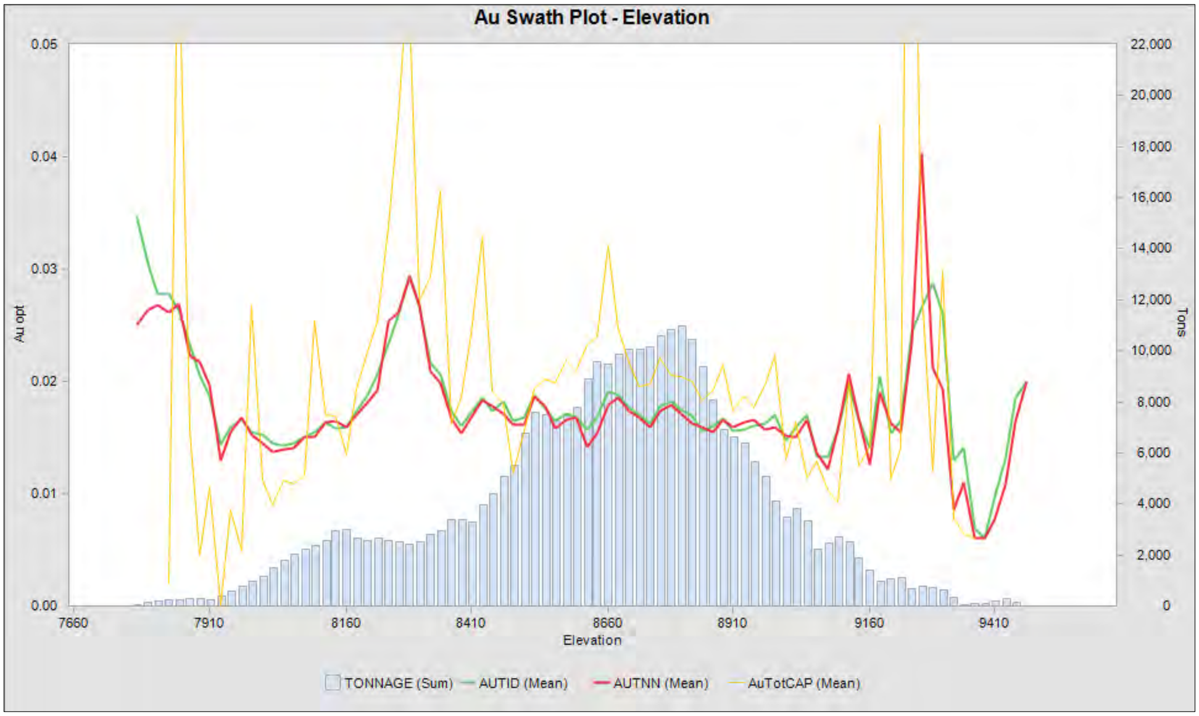

Source: SRK, 2014

**Figure 12.11.10: Swath Plot of Easting for Gold**



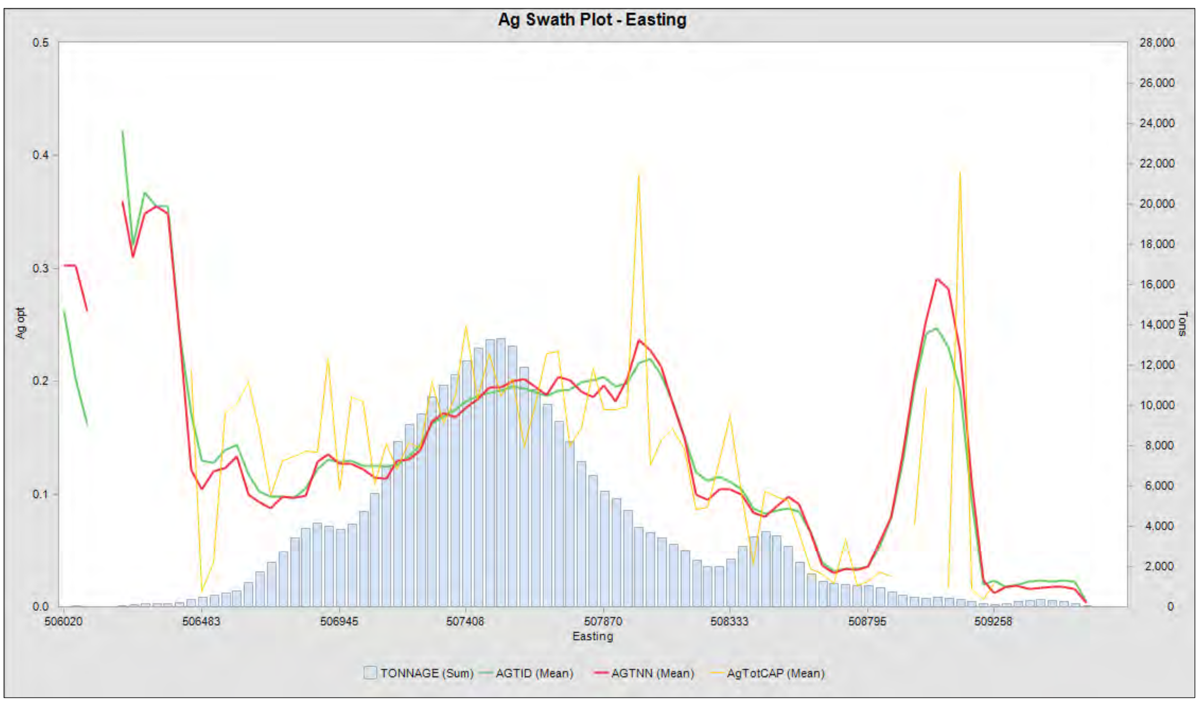

Source: SRK, 2014

**Figure 12.11.11: Swath Plot of Northing for Gold**



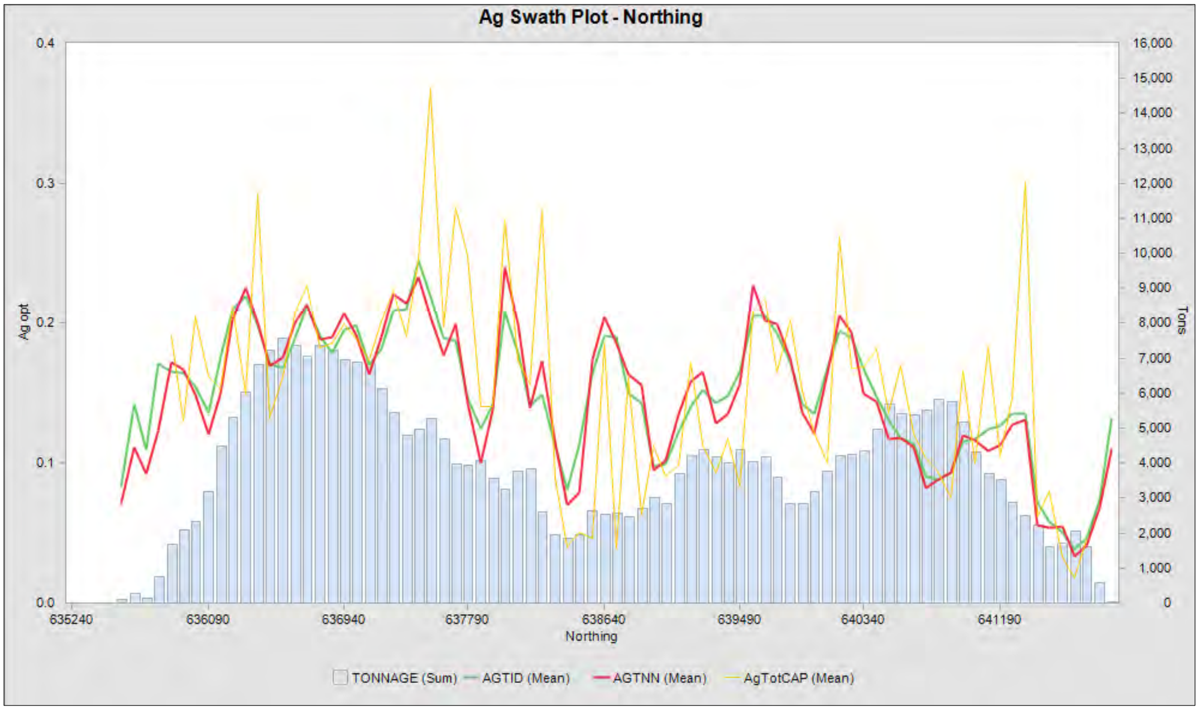

Source: SRK, 2014

**Figure 12.11.12: Swath Plot of Elevation for Gold**



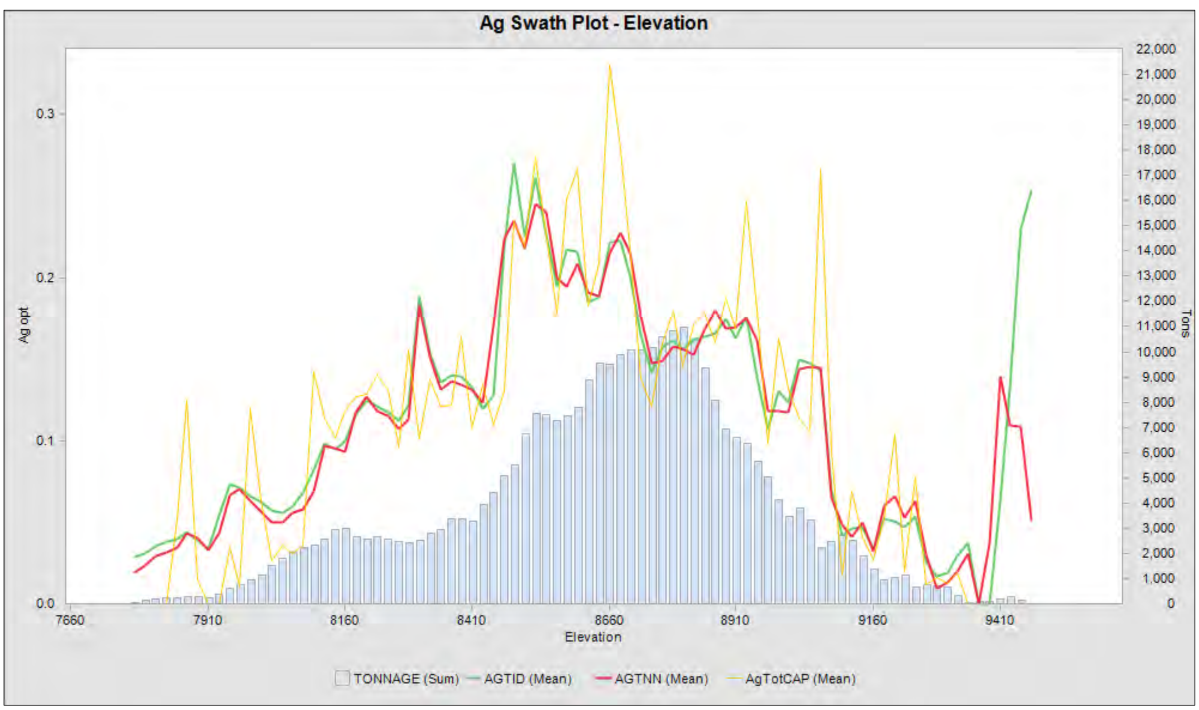

Source: SRK, 2014

**Figure 12.11.13: Swath Plot of Easting for Silver**




Source: SRK, 2014

**Figure 12.11.14: Swath Plot of Northing for Silver**




Source: SRK, 2014

**Figure 12.11.15: Swath Plot of Elevation for Silver**

## 12.12 Resource Classification

Classification of the resources for Mt. Hamilton reflects the relative confidence of the grade estimates. Confidence is dependent on several factors including: sample spacing relative to geological and geostatistical observations defining the continuity of mineralization, mining history, density determinations, accuracy of drill collar locations, and quality of the assay data.

Resources stated in this Technical Report were classified based on the criteria summarized in Table 12.12.1. These criteria were applied in a dedicated classification interpolation run using the same search orientations as grade estimation by domain. Search ranges were based on variography. The 2014 Mineral Resource Statement (Section 12 of this report) differs from the September 17, 2013 Mineral resource statement in the application of search criteria for Indicated resources. The 2014 statement utilized 120 ft as the maximum search range compared to 100 ft used in 2013. The longer search range is supported by variography and drill spacing.

As a final step in classification, SRK inspected the results in 3D and identified small clusters of Inferred blocks surrounded by broad volumes of Indicated blocks, commonly called “spotted dog” (Stephenson P R et al, 2006). The small Inferred clusters commonly called the “spotted dog” are artifacts in the interpolation process. They were flagged and manually converted from Inferred to Indicated classification, this post-process treatment was concentrated inside the optimized pit shape, i.e. Inferred blocks outside of the resource-limiting pit were not converted. There were approximately 5,000 oz of Au above a CoG of 0.006 oz/t Au converted in this process.

**Measured Mineral Resources** – Resource that was estimated with a minimum of three composites from at least three different drillholes within a maximum search radius of 50 ft.

**Indicated Mineral Resources** – Resource that was estimated with a minimum of two composites from at least two different drillholes within a maximum search radius of 120 ft.

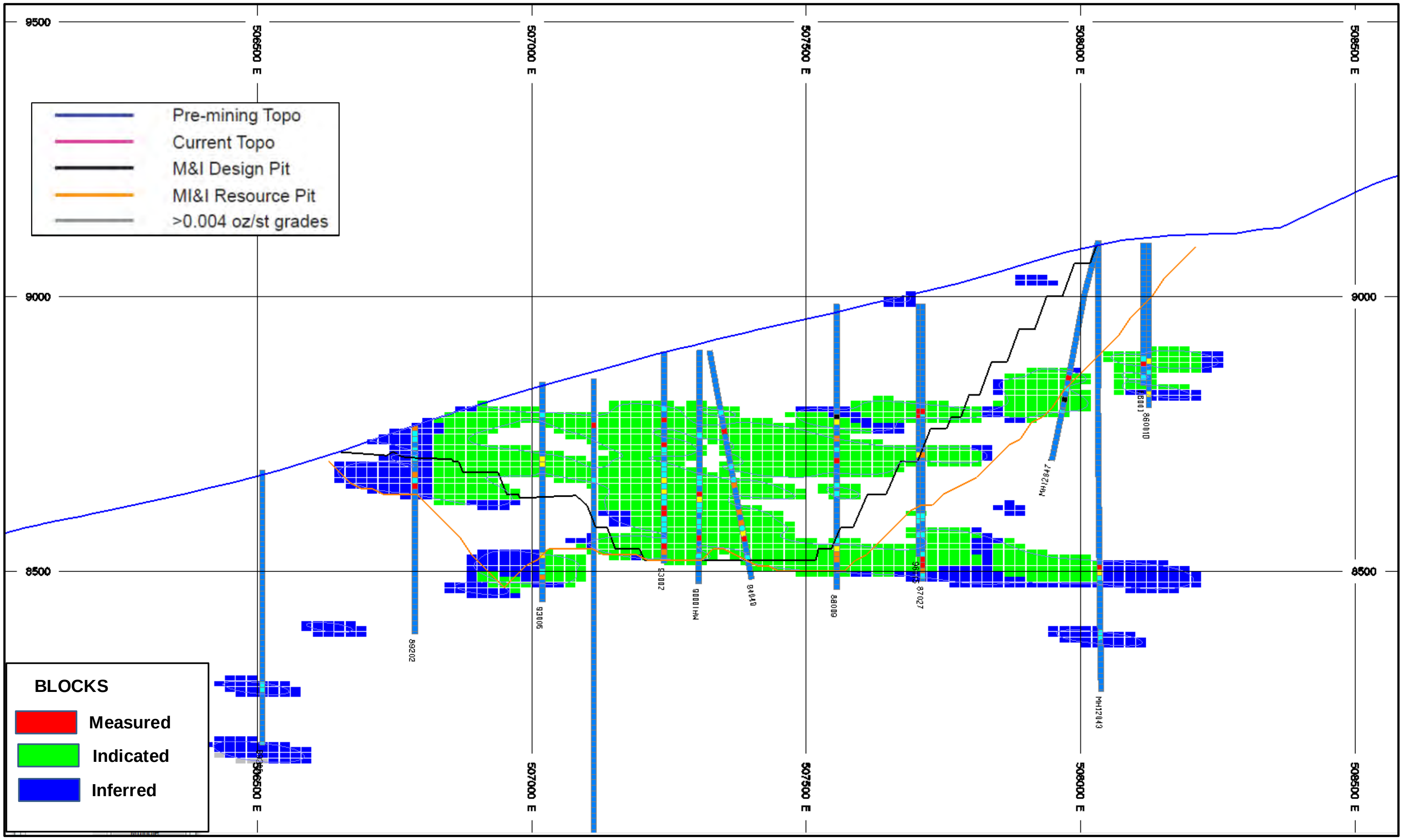
**Inferred Mineral Resources** – Resource that failed to meet criteria of Measured or Indicated but still fell within the interpreted mineral domain at a maximum search radius of 350 ft was classified as Inferred. Inferred resource could be estimated with a single composite.

**Table 12.12.1: Mt. Hamilton Resource Classification Criteria**

| Mt Hamilton Confidence Classification Scheme | | | | | | |
|---|---|---|---|---|---|---|
| Class | Minor Axis Search (ft) | Semi-Major Search (ft) | Major Axis Search (ft) | Minimum Number of Composites | Maximum Number of Composites | Maximum From One Drillhole |
| Measured | 15 | 50 | 50 | 3 | 8 | 1 |
| Indicated | 30 | 95 | 120 | 2 | 8 | 1 |
| Inferred | 30 | 350 | 350 | 1 | 8 | 1 |

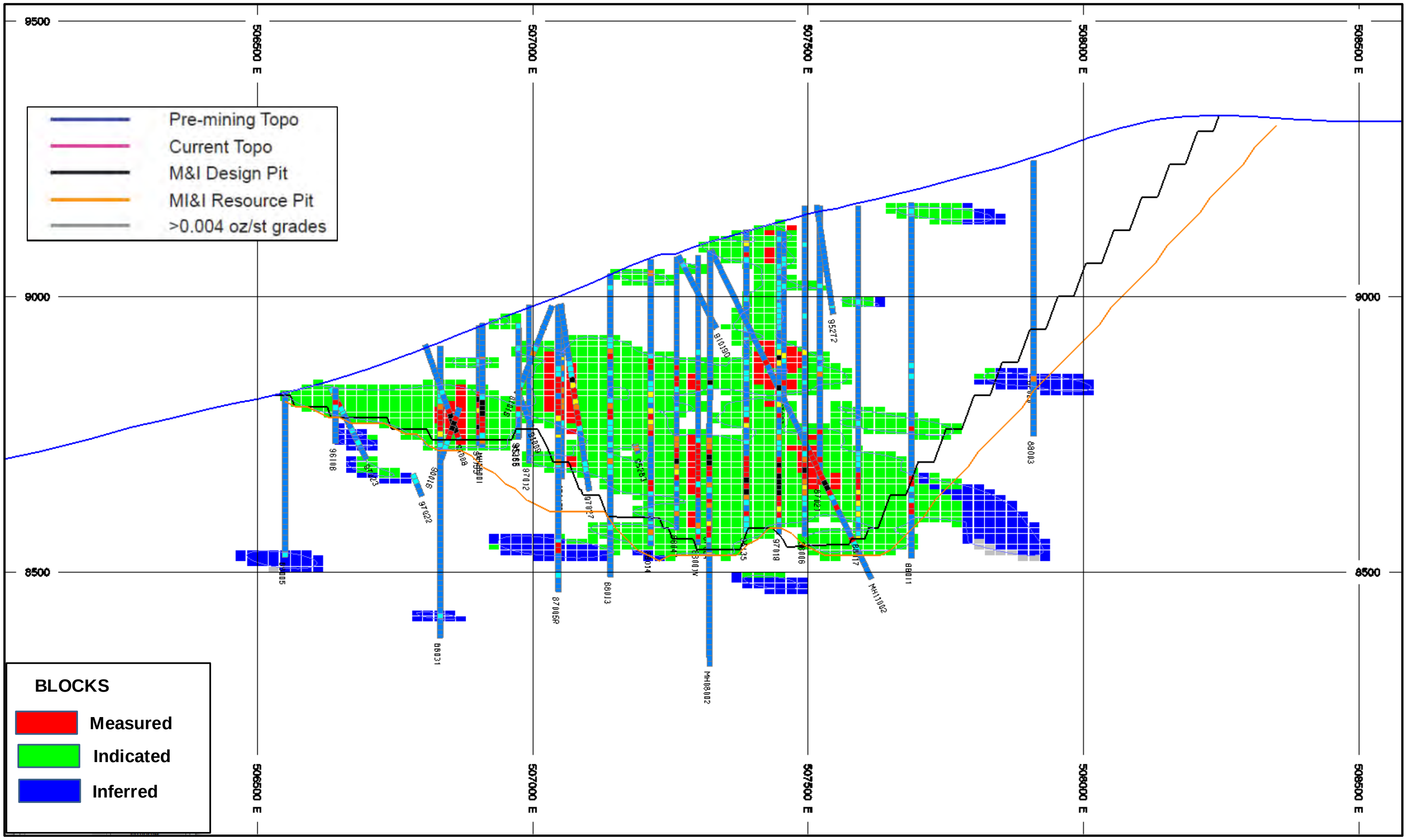
Source: SRK, 2014

Example cross sections of model blocks color coded by classification are presented in Figure 12.12.1 through Figure 12.12.7. The color code for the cross-sections is as follows: 1 = Measured (red); 2 = Indicated (green); 3 = Inferred (blue). These are the same cross-sections used to illustrate model grades in the previous section. Resources outside of the potentially mineable shape were not included in the resource statement and could be opportunities for resource expansion with additional drilling. The plan view of cross section traces was presented in Figure 12.11.1.



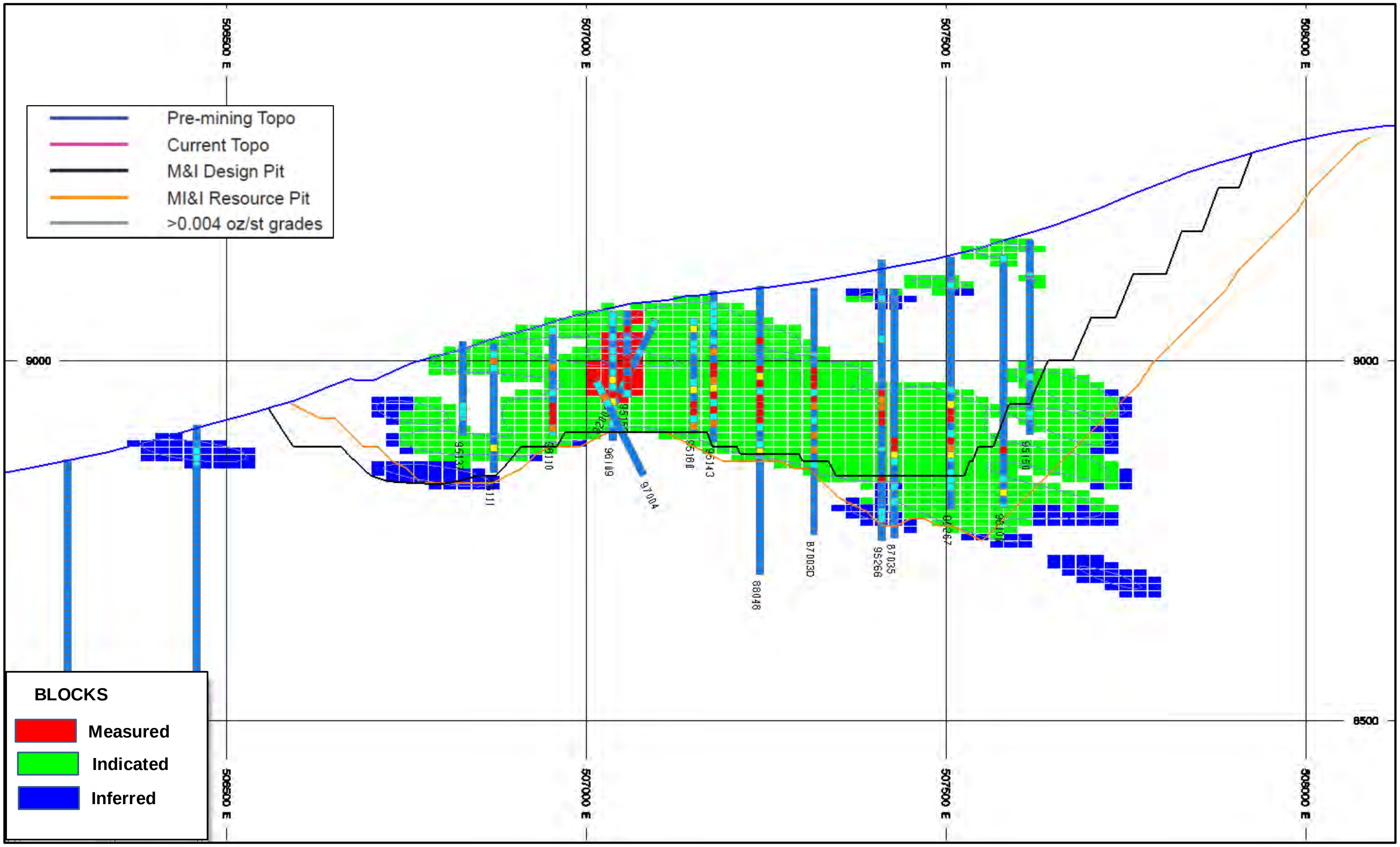

Source SRK, 2014

**Figure 12.12.1: East-West Cross Section “Cent.1” at 636390N – Drill Holes and Model Classification**



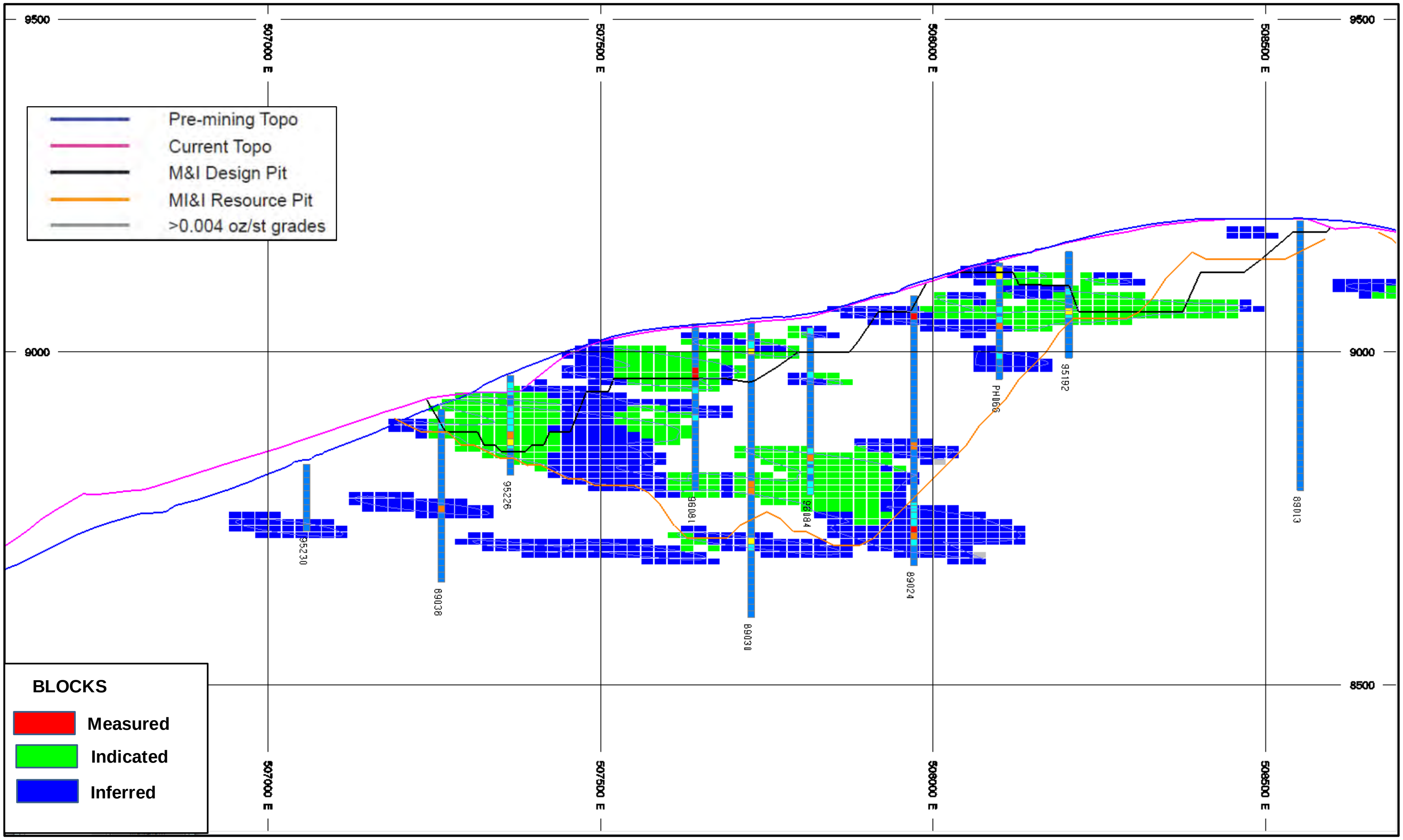

Source SRK, 2014

**Figure 12.12.2: East-West Cross Section "Cent.2" at 636770N – Drill Holes and Model Classification**



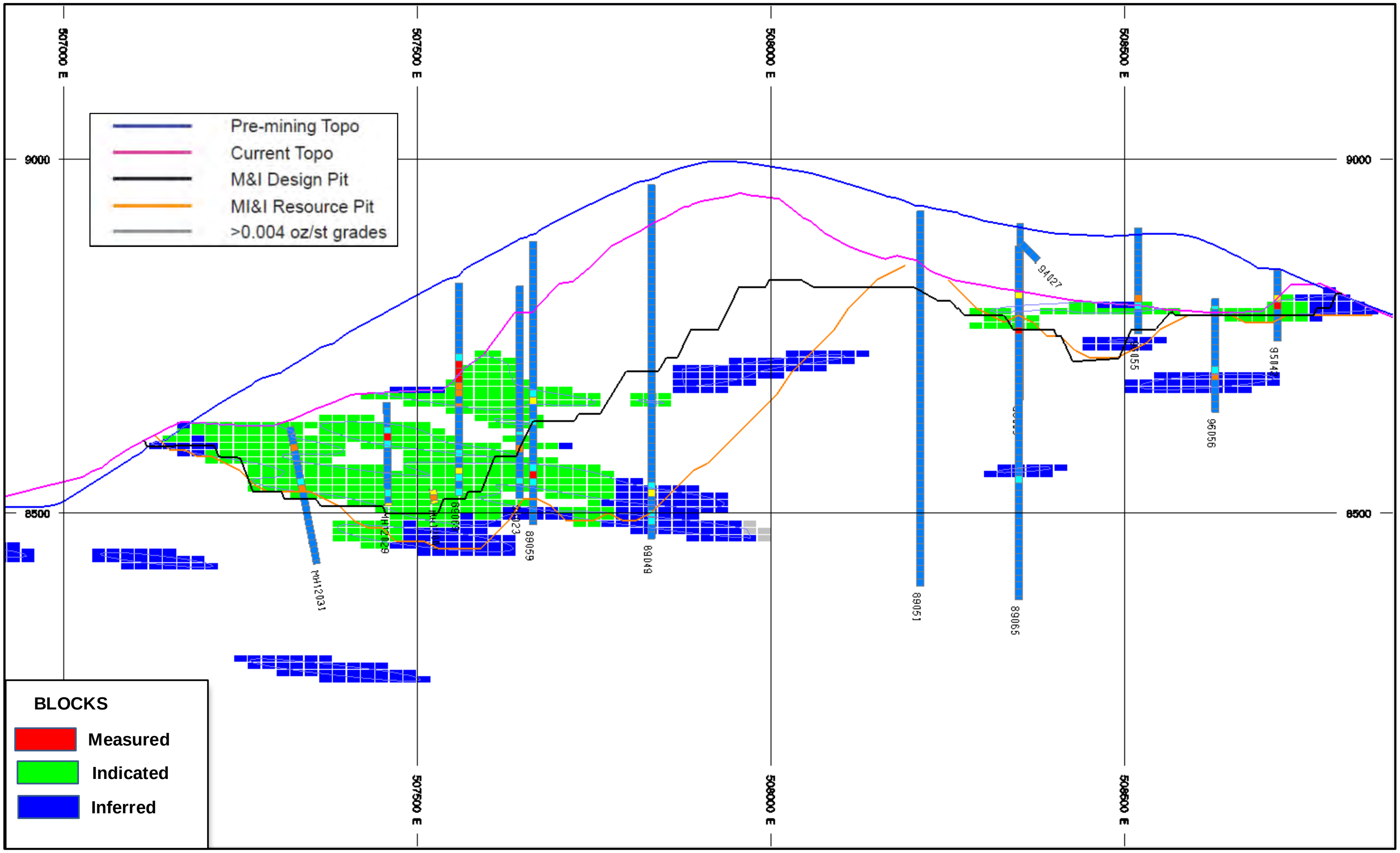

Source SRK, 2014

**Figure 12.12.3: East-West Cross Section "Cent.3" at 637490N – Drill Hole and Model Classification**



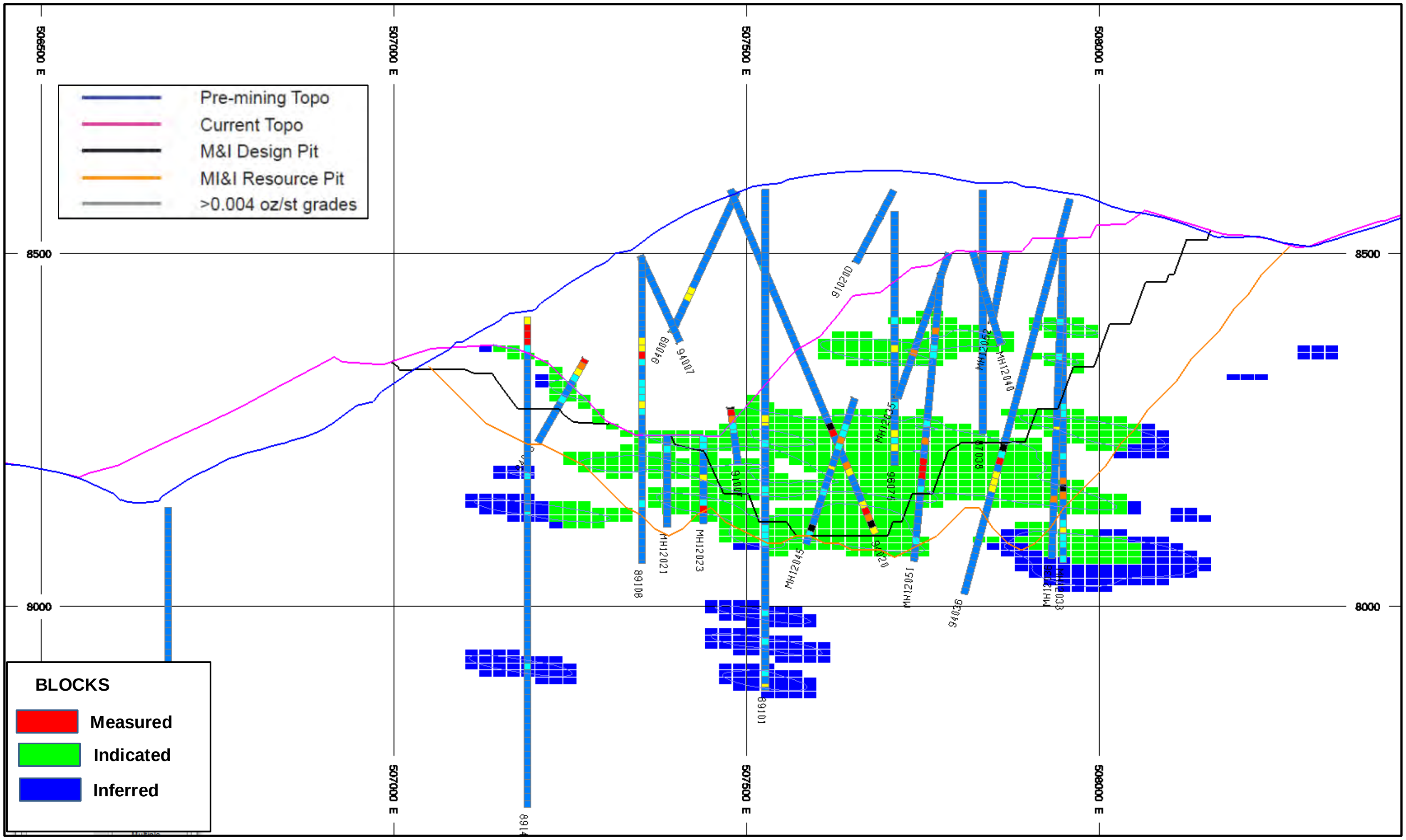

Source SRK, 2014

**Figure 12.12.4: East-West Cross Section “Selig.1” at 639190N – Drill Hole and Model Classification**



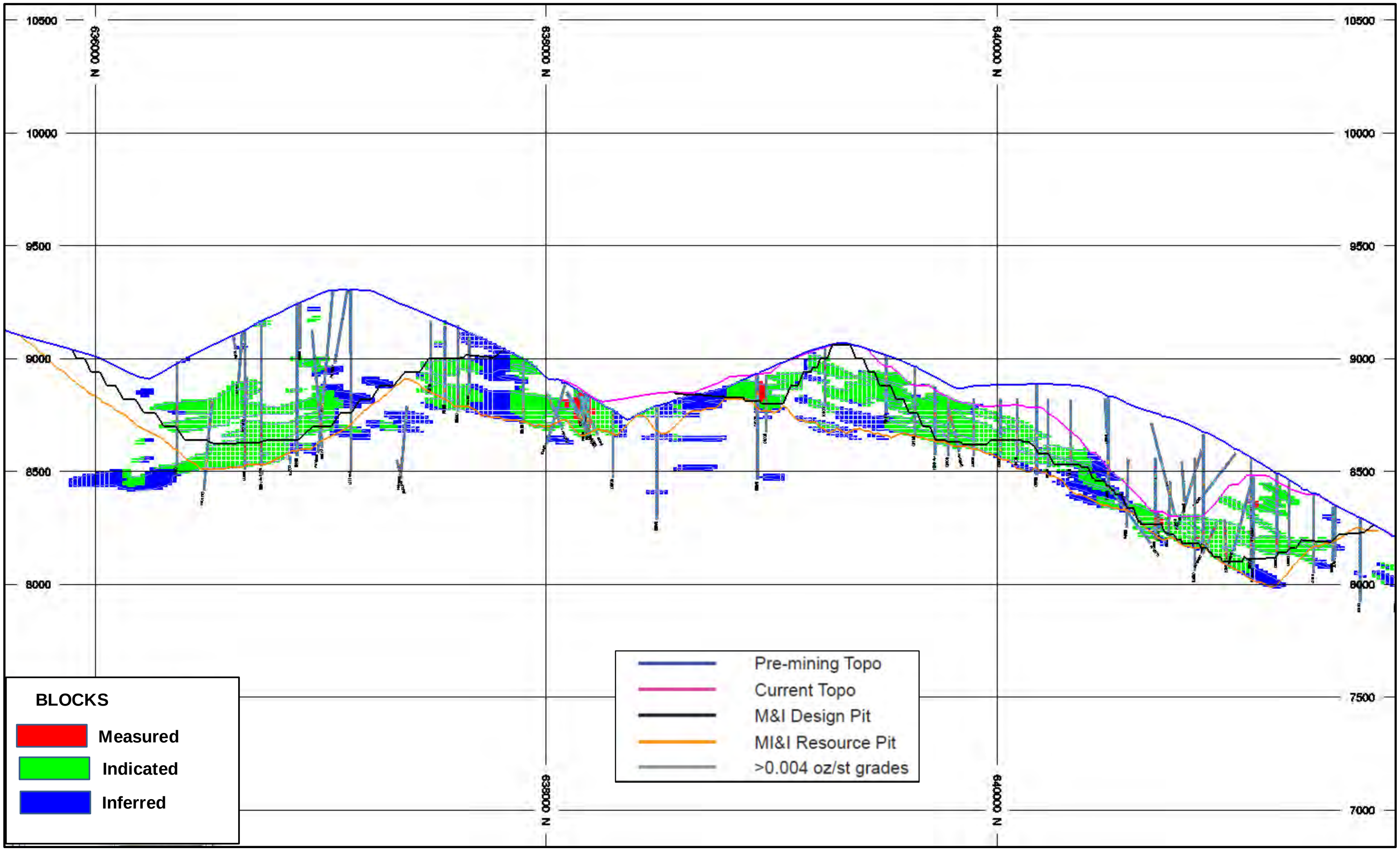

Source SRK, 2014

**Figure 12.12.5: East-West Cross Section “Selig.2” at 640170N – Drill Hole and Model Classification**




Source SRK, 2014

**Figure 12.12.6: East-West Cross Section "Selig.3" at 641030N – Drill Hole and Model Classification**




Source SRK, 2014

**Figure 12.12.7: North-South Longitudinal Section “MH-Long” at 507650E – Drill Hole and Model Classification**

## 12.13 Mineral Resource Statement

The Mineral Resource statement for the Mt. Hamilton deposit is presented in Table 12.13.1. An optimized pit was used to constrain the reportable resource. The optimized pit defining the mineral resource is shown in plan view in Figure 12.11.1.

**Table 12.13.1: Mineral Resource Statement, Mount Hamilton Gold-Silver Deposit, White Pine County, Nevada, March 25, 2014 (0.006 Au oz/t Cut-off)**

| Resource Category | Tons | Au Grade | Ag Grade | AuEq Grade | | Contained Ounces (thousands of oz) | | |
|---|---|---|---|---|---|---|---|---|
| | (000's) | oz/t | oz/t | oz/t | g/tonne | Au | Ag | AuEq |
| Measured | 1,427 | 0.030 | 0.209 | 0.033 | 1.125 | 42 | 299 | 47 |
| Indicated | 32,283 | 0.021 | 0.194 | 0.024 | 0.830 | 685 | 6,271 | 782 |
| **Measured and Indicated** | **33,710** | **0.022** | **0.195** | 0.025 | 0.843 | **727** | **6,569** | **828** |
| Inferred | 6,721 | 0.018 | 0.171 | 0.020 | 0.696 | 119 | 1,153 | 136 |

Source: SRK, 2014

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves estimate;
- Resources stated as contained within a potentially economically minable open pit; pit optimization was based on assumed gold and silver prices of US$1,300/oz and US$19.60/oz, respectively, block-by-block modeled recovery averaging 76% for Au and 39% for Ag, an ore mining cost of US$2.06/t for Seligman, an ore mining cost of US$1.64/t for Centennial and an ore processing cost of US$4.95/t; west pit slopes 45°, east pit slopes of 50°;
- Resources are reported using a 0.006 oz/t contained gold CoG;
- AuEq was calculated using a Ag:Au ratio of 65:1;
- Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.

The Mt. Hamilton resource estimate was informed by 857 drillholes with an average hole depth of 370 ft for a total of 317,739 ft of drilling. The drill data were verified and validated by SRK in compliance with NI 43-101 requirements. This consolidated Mt. Hamilton resource estimate includes 60 new infill drillholes that converted earlier Inferred resources to the Indicated category, while also expanding the Seligman resource.

A Break Even cut-off grade (BE CoG) of 0.006 oz/t gold was applied to the resource statement. The CoG for the resource was determined using a gold price of US$1,300.00/oz, a silver price of US$19.60/oz, a recovery of approximately 79% (variable by material type), combined mining and processing costs of US$6.04/t and a 3.4% NSR royalty. The calculation for determining the CoG was:

$$\textit{BE CoG} = \frac{\text{Total Unit Mining, Processing and Administration Operating Costs}}{\text{(Au Price – (Royalty + Final Refining Costs)) x Process Recovery}}$$

A pit-optimization exercise was performed on the Mt. Hamilton model using Mintec's MineSight® 3D software. MineSight® 3D employs the industry-accepted Lerchs-Grossmann algorithm, which essentially determines the maximum pit extents by defining blocks taken within a pit against the amount of waste needed to reach those blocks. Blocks classified as Measured, Indicated and Inferred were all used to define the resource pit shell. Input criteria for the pit optimization are described in the footnotes of the resource statement.

## 12.14 Mineral Resource Sensitivity

A price sensitivity analysis shown in Figure 12.14.1 was developed with multiple runs of the Lerchs-Grossmann (LG) pit optimization algorithm at incremental Au sales prices ranging from US$50.00/oz to US$2,000.00/oz at US$50 increments. Silver prices were factored to match the change in gold price for each LG run. Values below US$400 were omitted from the graph as they yielded less than 10,000 gold-equivalent ounces. Measured, Indicated, and Inferred material was considered ore for this resource sensitivity. Mining and processing costs from the detailed engineering costing work done for this feasibility study were used for calculating block values for each mining area.

The results of this sensitivity analysis indicate the majority of the potential Measured and Indicated ounces are captured by the US$1,250/oz Au sales price pit and that there is a significant amount of Inferred material that, if upgraded, could have a positive impact on economics for pits designed above US$1,000/oz Au.

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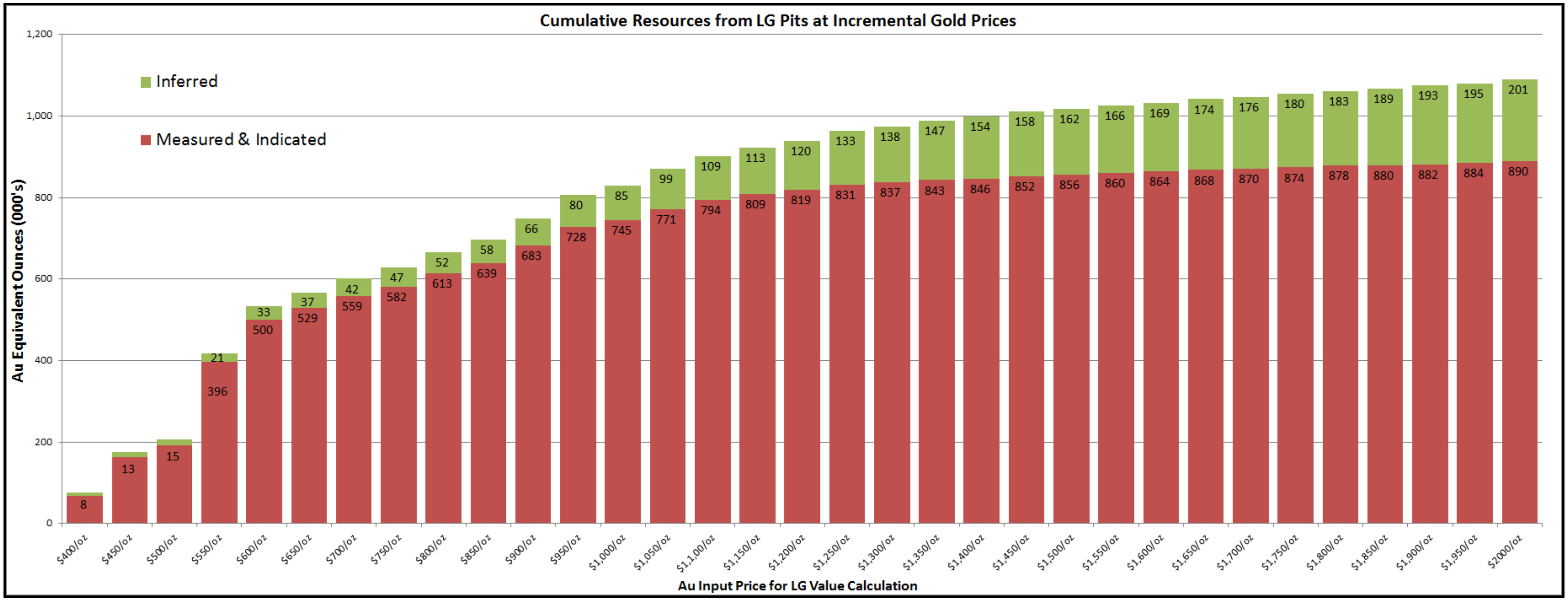



Source: SRK, 2014

**Figure 12.14.1: Cumulative Resources from LG Pits at Incremental Gold Prices**

## 12.15 Relevant Factors

There are no obvious impediments to developing the gold-silver resource at Mt. Hamilton. Environmental, permitting, legal title, taxation, product marketability, infrastructure or other factors that could affect resources are being addressed by Mt. Hamilton LLC and do not appear to present any barriers to project development.

## 12.16 Resource Potential

The area in the south end of the Seligman deposit has been under-drilled and has mineralized intercepts laterally and vertically. This was the historic NES 5 area, drilled by Westmont and Rea Gold, who were specifically targeting skarn mineralization and ignored both igneous-hosted mineralization and contact-related mineralization. SRK has recommended additional drilling in this area, believing it has a strong potential for extension. The area could generate high-grade intercepts if historic data are an indication. Addition of Mineral Reserves from this south Seligman area (after the pre-requisite geochemical, metallurgical and geotechnical drilling and testing requirements are met) would likely impact the mining sequence and improve overall economics.

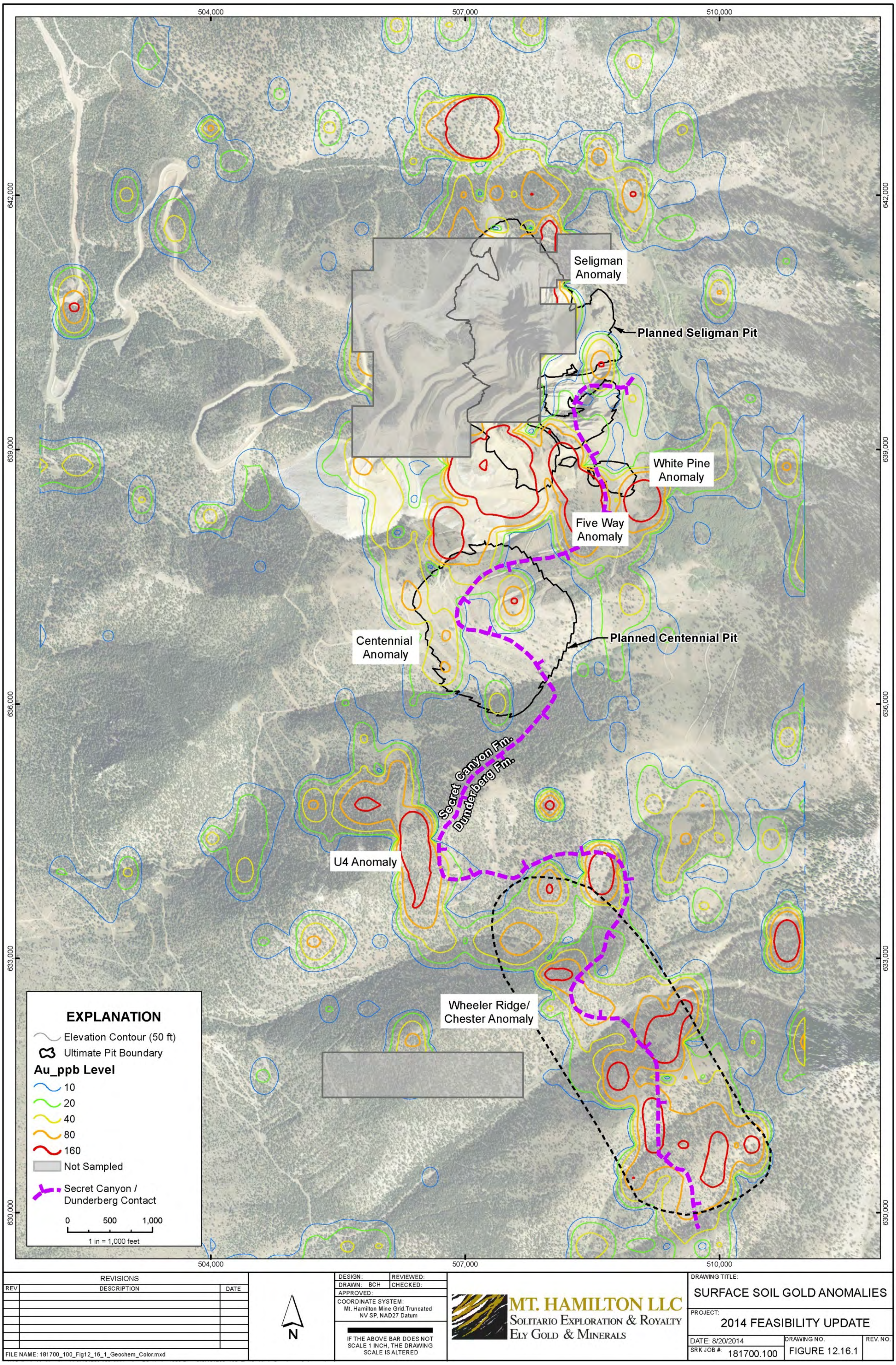
Upon completion of the planned conveyor incline, a significant exploration opportunity exists to test skarn and porphyry potential adjacent to the Monte Cristo stock and expand definition of gold and molybdenum mineralization in the "Shell" area south of the stock.

Other exploration targets exist at the site and are being developed by MH-LLC by surface drilling and geochemical sampling. Of note are the Wheeler Ridge, Chester, U4, Five Way and White Pine prospects that lie south of Centennial and are shown in Figure 12.16.1. MH-LLC has near-term plans to drill test Wheeler Ridge and have secured permits for drilling.

The Five Way exploration target lies adjacent to the southeastern limit of the Seligman pit. Previous scout drilling in this area has intersected significant thin intervals of high-grade mineralization that is located in the Dunderberg Formation overlying the prolific Secret Canyon Formation ore host. Additional drilling will be conducted to determine if economic ore in this area will connect with known resources/reserves to the west. The White Pine soil anomaly adjacent to the Five Way area has not been tested with drilling. It has a geochemical signature and stratigraphic location similar to the Centennial anomaly.

The significant large soil anomaly that exists along trend to the south at the Wheeler Ridge/Chester area is very similar to the anomalies that define the Seligman and Centennial deposits. Limited drilling has occurred on the patented Chester claims along the eastern boundary of this trend but permits were not previously in place to allow for more complete testing of this zone. Permits were received in 2013 for further testing in this area. The previous, mostly shallow drilling at Chester was located in the Dunderberg rather than the preferred target of the Secret Canyon Formation but, nevertheless, intersected thin ore grade zones of mineralization. Thinner, less continuous intercepts are characteristic of the Dunderberg and their presence is a strong indication of potential in the underlying Secret Canyon Formation.

The U4 area was drilled by Westmont in the early 1990's without significant positive results. A re-evaluation of this earlier work will be made to determine if potential remains to be tested in conjunction with the Wheeler Ridge/Chester exploration program.




Source: SRK, 2014

**Figure 12.16.1: Resource Sensitivity at Variable Gold Prices**

# 13 Mineral Reserve Estimate (Item 15)

## 13.1 Reserve Estimation

The conversion of mineral resources to ore reserves required accumulated knowledge achieved through Lerchs-Grossmann (LG) pit optimization, detailed pit design, and associated modifying parameters. Reserve estimation was achieved using Mintec's MineSight® 3D software and applies to the full Mt. Hamilton resource. Detailed access, haulage and operational cost criteria were applied in this process for each deposit (Centennial and Seligman) independently. The project was built in U.S. units and all metal grades are in troy ounces per short ton (oz/t).

The orientation, proximity to the topographic surface, and geological controls of the Mt. Hamilton mineralization support mining of the ore reserves with open pit mining techniques. To calculate the mineable reserve, pits were designed following an optimized LG pit based on US$840/oz Au and US$12.68/oz Ag sales prices. These prices were chosen to create the primary guide surface based on a price sensitivity and profitability study that showed the pit maximized profitability while targeting the currently permitted fixed ore tonnage of 22.5 Mt. The quantities of material within the designed pits were calculated using a 0.006 Au oz/t CoG, which is based on the static US$1,300/oz Au and US$20/oz Ag metal prices observed at the time of this study.

Consequently, a significant tonnage of Indicated Resource that would normally have been classified as Probable Reserves, was excluded from reserve classification and the economic model.

## 13.2 Mineral Reserve Statement

The Mt Hamilton mine open pit Mineral Reserve statement is presented in Table 13.2.1.

**Table 13.2.1: Mineral Reserve Statement Mt. Hamilton Gold-Silver Deposit, White Pine County, Nevada, SRK Consulting (U.S.), Inc. August 14, 2014**

| Reserve Category | Tons | Au Grade | Ag Grade | AuEq Grade | | Contained Ounces (thousands of oz) | |
|---|---|---|---|---|---|---|---|
| | (000's) | oz/t | oz/t | oz/t | g/tonne | Au | Ag |
| Proven | 1,240 | 0.029 | 0.198 | 0.031 | 1.060 | 36.6 | 245.8 |
| Probable | 21,260 | 0.024 | 0.198 | 0.025 | 0.870 | 508.8 | 4213.8 |
| **Proven and Probable** | **22,500** | **0.024** | **0.198** | **0.026** | **0.880** | **545.4** | **4459.6** |
| Total Waste | 63,319 | | | | | | |

Source: SRK, 2014

- Reserves are reported using a CoG of 0.006 oz/t Au;
- The CoG was based on a gold price of US$1,300/oz and a silver price of US$20/oz;
- The CoG was calculated at an average recovery of 76% for Au and 39% for Ag;
- Average recovery for gold was calculated from a recovered grade item modeled for each model block based on cyanide soluble and total gold grades;
- Metal grades reported are diluted; and
- Some numbers may not add due to rounding.

Mineral Reserves stated above are contained within and are not additional to the Mineral Resources stated in Section 12 of this report.

## 13.3 Conversion of Resources to Reserves

Conversion of resources to reserves required consideration of:

- The ore extraction method(s) used in relation to the ore body characteristics which determine mining dilution and recovery;
- Associated project operating costs and resulting CoG's; and
- Current permitted capacity of the heap leach pad.

In accordance with the CIM classification system only Measured and Indicated resource categories can be converted to reserves (through inclusion within the open-pit mining limits). In all mineral reserve statements Inferred mineral resources are reported as waste. In some mineral resource statements Inferred mineral resources are reported separately and are clearly identified.

CoG is a function of technical and economical parameters and defines the economic portion of the resource at the time of determination. Break even CoG considers the total unit operating costs, including mining, processing and administration, process recovery, metal prices and additional costs for freight, smelting and/or refining. Where applicable, royalties are included in the calculation.

Once such a CoG is defined all the ore with a gold grade above this value should be considered as economically mineable. Ore feed to plant will have an average grade higher than the CoG value, and this difference provides the profit (return on capital) for the business.

The CoG may be modified to other values during the mining operations in order to optimize business profits. These operational CoG grades may accomplish different specific purposes.

### 13.3.1 Break Even Cut-off Grade

The typical expression for a BE gold CoG is (allowing for appropriate use of units):

$$\textit{BE CoG} = \frac{\textit{Total Unit Mining, Processing and Administration Operating Costs}}{\textit{(Au Price – (Royalty + Final Refining Costs)) x Process Recovery}}$$

### 13.3.2 Internal Cut-off Grade

An alternative (operational) CoG , the internal CoG, takes into account all operating costs, but mostly excludes mining costs based on the concept that once material has been mined (for example to access ore with grades above the BE CoG) the mining cost is considered to be a sunk cost. If the material can pay for the downstream processing and other costs then it qualifies as ore. This can be adjusted to allow for differential ore and waste haulage (or other) costs.

The typical expression for an internal (Int.) gold CoG is (allowing for appropriate use of units):

$$\textit{Int. CoG} = \frac{\textit{Total Unit Processing and Administration Operating Costs}}{\textit{(Au Price – (Royalty + Final Refining Costs)) x Process Recovery}}$$

The CoG used by MineSight® 3D to determine whether a block was ore or waste was reported as 0.006 oz/t-Au. To keep consistency with what was used in the optimization, 0.006 oz/t-Au was used to define ore and waste. This value is subject to change due to actual processing cost and realized gold price during operation.

# 14 Mining Methods (Item 16)

Mt. Hamilton is gold and silver deposit, with head grades averaging approximately 0.024 oz/t gold. Silver is also present in the deposit at an average grade of 0.198 oz/t. The mineralization is close to the surface and the resource lends itself to an open pit mining method.

Mining operations at the Mt. Hamilton deposit have a stripping ratio of 2.5:1, waste to ore with mining taking place on the western flank of the White Pine Mountains at an average elevation of 8,800 ft above sea level.

The mine design consists of a Centennial Pit with approximate dimensions of 1,900 ft wide (east-west) by 2,000 ft long by 900 ft deep, with a volume of 650 $Mft^3$; and the Seligman Pit, with approximate dimensions of 1,900 ft wide, 3,200 ft long and 700 ft deep, with a volume of 325 $Mft^3$. The pit designs were segregated into multiple phases for production scheduling with 90 ft wide design ramps (including berms) at a maximum in-pit road grade of 10%.

Open pit mining will be by conventional diesel-powered equipment, utilizing a combination of blasthole drills, hydraulic shovel, wheel loaders and off-highway 100 t trucks. Support equipment composed of graders, track dozers, and a water truck will aid in the mining of the Mineral Reserve and waste. Ore grade materials will be hauled and dumped in the primary crusher or stockpiled for later processing. The ore will be crushed to minus 4 inch and conveyed to an ore pass. The ore pass will drop the ore vertically approximately 415 ft where it will be loaded on a conveyor in a 4,425 ft long drift. From the loading point at the base of the ore pass, the drift and conveyor have a -15% grade to the portal. Once out of the drift, the ore will be transferred through to a series of belts to a coarse ore stockpile. A reclaim tunnel under the coarse ore stockpile feeds a secondary crusher where the ore will be crushed to 90% passing 3/4 inch and conveyed and stacked on the leach pad with a radial stacker. A general facilities layout is provided in Figure 14.1.

## 14.1 Mining History

The NE Seligman Mine, which will be mined out by the proposed new Seligman Pit, was operated by Rea Gold from 1994-1997. The Nevada Department of Minerals and Nevada Bureau of Mines report total production of 124,000 oz of gold and 310,250 oz of silver from the NE Seligman Mine by Rea Gold over this operating period. The haul road was extended to the Centennial pit area and the area of the starter pit was clear-cut and grubbed of vegetation in preparation for preproduction stripping which was scheduled to begin in 1997, but was never initiated.

## 14.2 Pre-Production Mine Development

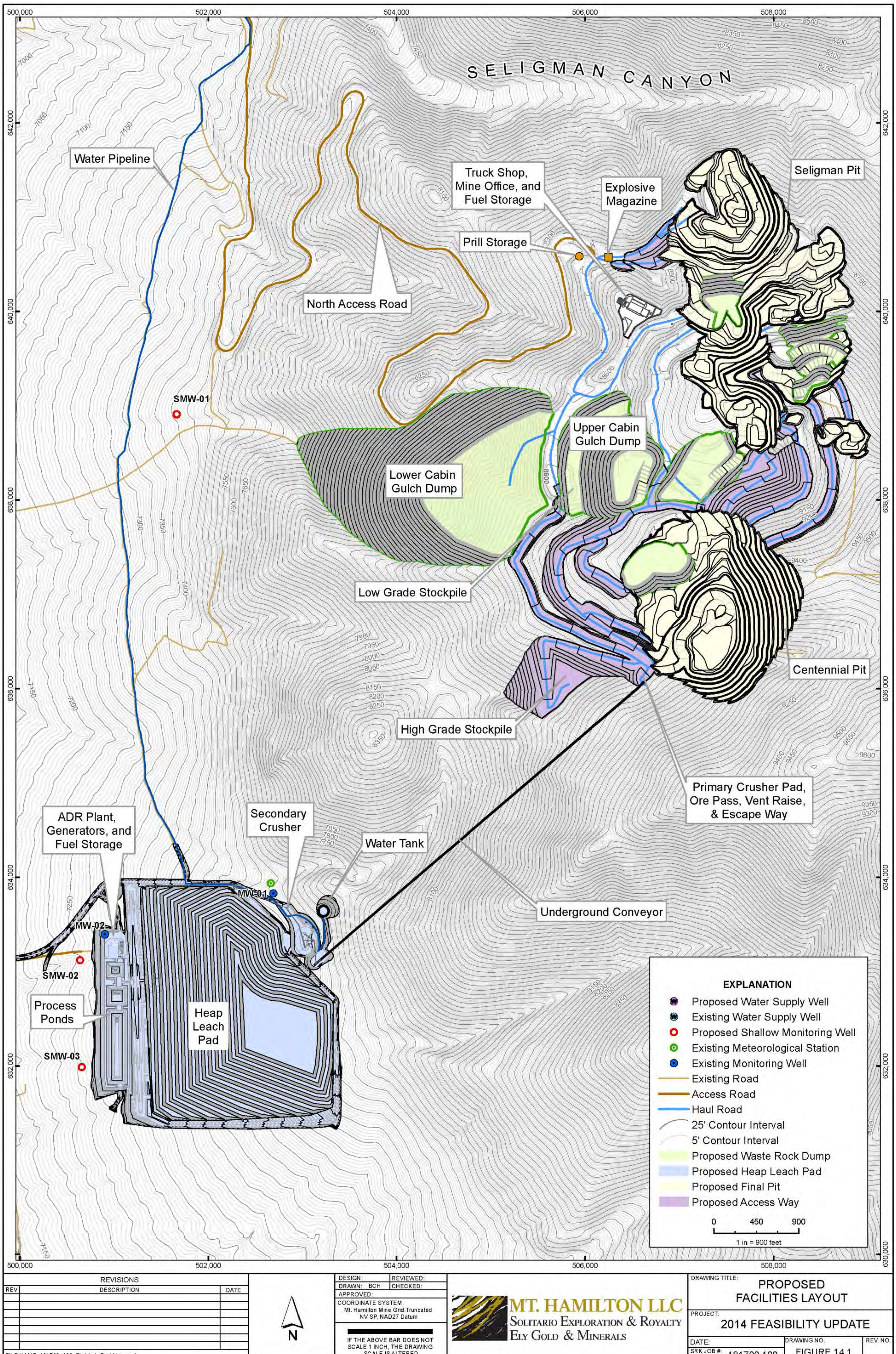
Mine development will be self-performed by MH-LLC with the mining truck fleet and loader. In addition, an analysis of contract mining is underway.

### 14.2.1 Pre-stripping and Access Road Construction

The mine is located in steep terrain. Initial access will be made from the existing haul roads that were developed by Rea Gold wherever possible.

Access to the mine will be via the North Access Road, shown in Figure 14.1. Initial access to the primary crusher location will be via the 90 ft wide (including berm) Crusher Access road shown in

Figure 14.2.1.1. This road will be constructed primarily by cut with some cut to fill and is designed with an approximate grade of 5% up to the crusher.




Source: SRK, 2014

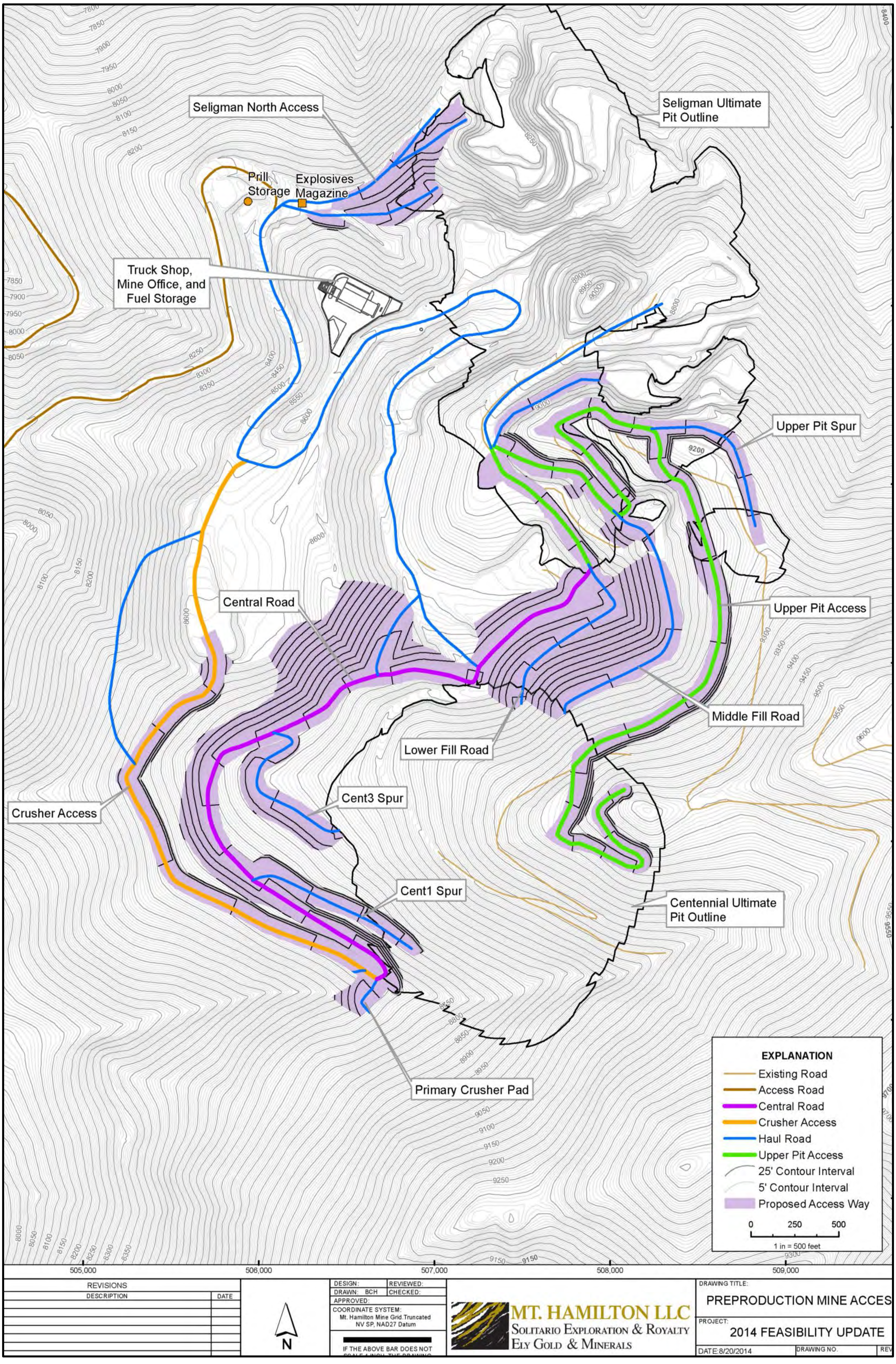
**Figure 14.1: General Facilities Layout**

Initial access to the top of the Centennial Pit will be via a 90 ft wide haul road cut from the Seligman Pit area across the upper slopes of Cabin Gulch called the Upper Pit Access road in Fig. 14.2.1.1. This road will primarily be cut in with some fill in the switchbacks to prevent excessive cut. This road begins at the existing haul road with a series of switchbacks developed on the northern slope of Cabin Gulch within the ultimate Seligman Pit design. These switchbacks utilize existing development wherever possible and will provide access to the upper benches of the Seligman mining area at a grade of approximately 9%. Once this road reaches the ridge line it will then be cut around Cabin Gulch to the south at a 6% grade to access the upper benches of the Centennial Pit.

The material generated mining the Crusher Access and Upper Pit Access roads will be used to start construction on the 90 ft wide (including berm) Central Haul road. When completed, the Central Haul road will become the main haulage route from both mining areas to the primary crusher and the primary waste dump access for Centennial. A short portion of the Central Haul road will be in cut, but the rest will be built on fill material. This road is designed with a 9% grade as illustrated in 14.2.1.

Access to the middle benches in Centennial Phases I, II, and IV will be achieved with the 90 ft wide Middle and Lower Fill roads. The Middle Fill road lies roughly 120 ft below the Upper Pit Access road in Cabin Gulch and connects the uppermost Upper Pit Access road switchback to the Centennial Pit at a 5.5% grade. The Lower Fill road lies 120 ft below the Middle Fill road at an initially steeper 11% grade, which may be adjusted down during operations as more fill material becomes available. The Lower Fill road connects from the intersection of the Upper Pit Access and Central Haul Roads to the Centennial Pit.

The lower portions of Centennial will be accessed via the 90 ft wide Cent1 and Cent3 Spurs shown in 14.2.1. These roads will be constructed at an 8% grade and will be cut down over the life of the mine as the pit exit drops in elevation.




Source: SRK, 2014

**Figure 14.2.1.1: Pre-production Mine Access**

## 14.3 Engineering Block Model

The exploration (resource) block model (XBM) was built using pre-mining topography to allow reconciliation to past mining production and to better control the model interpolation. Because the XBM did not account for the material movement during past mining operations, it was necessary to make adjustments that properly accounted for the previous mining cut and fill. The model that includes the final existing mining and backfill surfaces is the Engineering Block Model (EBM).

To create the EBM, a second block model was generated with the same dimensions as the XBM. The items relevant to engineering tasks, including grades, tonnage factors, material types, and ore percentages, were copied from the XBM into the EBM. Next, a topographic surface was created that accounted for all of the material mined, but did not include fill. This "cut topo" surface allowed for the removal of all XBM modeled ore that had been previously mined and ensured that any XBM ore blocks that had been replaced with fill would not be considered as ore. To remove the mined ore, the percentage of each block below the cut topo surface was then calculated and stored to each block. The ore percent of each block was then normalized to the cut topo percentage to remove all of modeled ore that had been mined. Following this, a fill solid was used to flag all blocks that now represent fill material and the percentage of each block below the current (as-built) topography was calculated and stored to the TOPO item in the EBM. Finally, the tonnage factor for each block was assigned based on material type including the newly assigned fill blocks and additional items for material classification, dilution, and scheduling were added.

The EBM was used for all Resource and Reserve reporting as well as mine planning. Details of model items, codes and definitions are included in the 2014 FS.

### 14.3.1 Recovery Modeling and Application

Based on recent detailed SRK metallurgy, gold recovery was calculated using the Rmax equation, which includes the ratio of in situ cyanide soluble gold grade to the in situ total gold grade, and results in a gold recovery value stored to each block within the EBM. Average gold recovery for the Mt Hamilton reserve is approximately 76.2%.

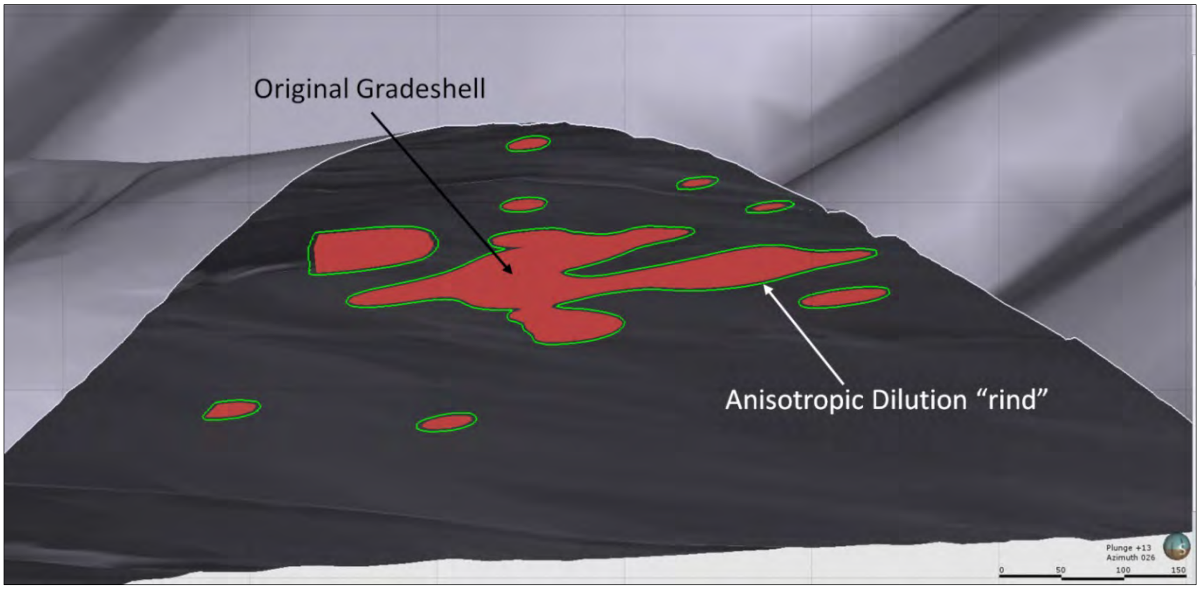
Silver recoveries were assigned to the Seligman and Centennial deposits of 43.6% and 38%, respectively.

Both in situ and diluted gold and silver grade values were then multiplied by their corresponding grade items and stored to a new diluted grade item in the block model. These grades were then used for the pit optimization and production scheduling.

### 14.3.2 Dilution Modeling and Application

The amount of dilution that will be mined with the ore in any given block varies based on the equipment used, size and orientation of the deposit structure, bench height, and mining direction. To account for these factors when considering the problem of external dilution, SRK developed a method of creating a rind of material with varying thickness around the mineralization grade shell and used this excess material at zero grade to dilute the in situ grade and volume of ore within each block.

To develop the dilution zone peripheral to the gold mineralization gradeshells, SRK used implicit modeling whereby a halo was added to the original gradeshell to simulate the predicted “over-mining". In broad, disseminated areas of the deposit the dilution envelope was built to a fixed radius 8 ft outboard of the interpreted gradeshell. In areas of tabular, shallowly-dipping mineralization dilution was modeled with a thin vertical and larger radial envelope as these are planned to be developed on smaller benches than the rest of the deposit, allowing for more control of the dilution. Structural controls and anisotropy from the original gradeshell interpretations were applied to the dilution envelope and adjusted as required to mimic mining. The material within this "dilution rind" was assigned a zero grade for estimating reserves. This dilution envelope is shown relative to the mineralization gradeshell in Figure 14.3.2.1.




Source: SRK, 2014

**Figure 14.3.2.1: Conceptual illustration of Dilution "Rind" Surrounding Mineralization Grade Shell**

The new expanded dilution shell was then used to define the volume of ore reported from each block and the interpolated grades were adjusted to ensure that with the increase in volume mined, the total ounces would remain the same. This reduction in grade also moved some ore blocks into the category of diluted waste, as they were no longer economic when including the additional waste.

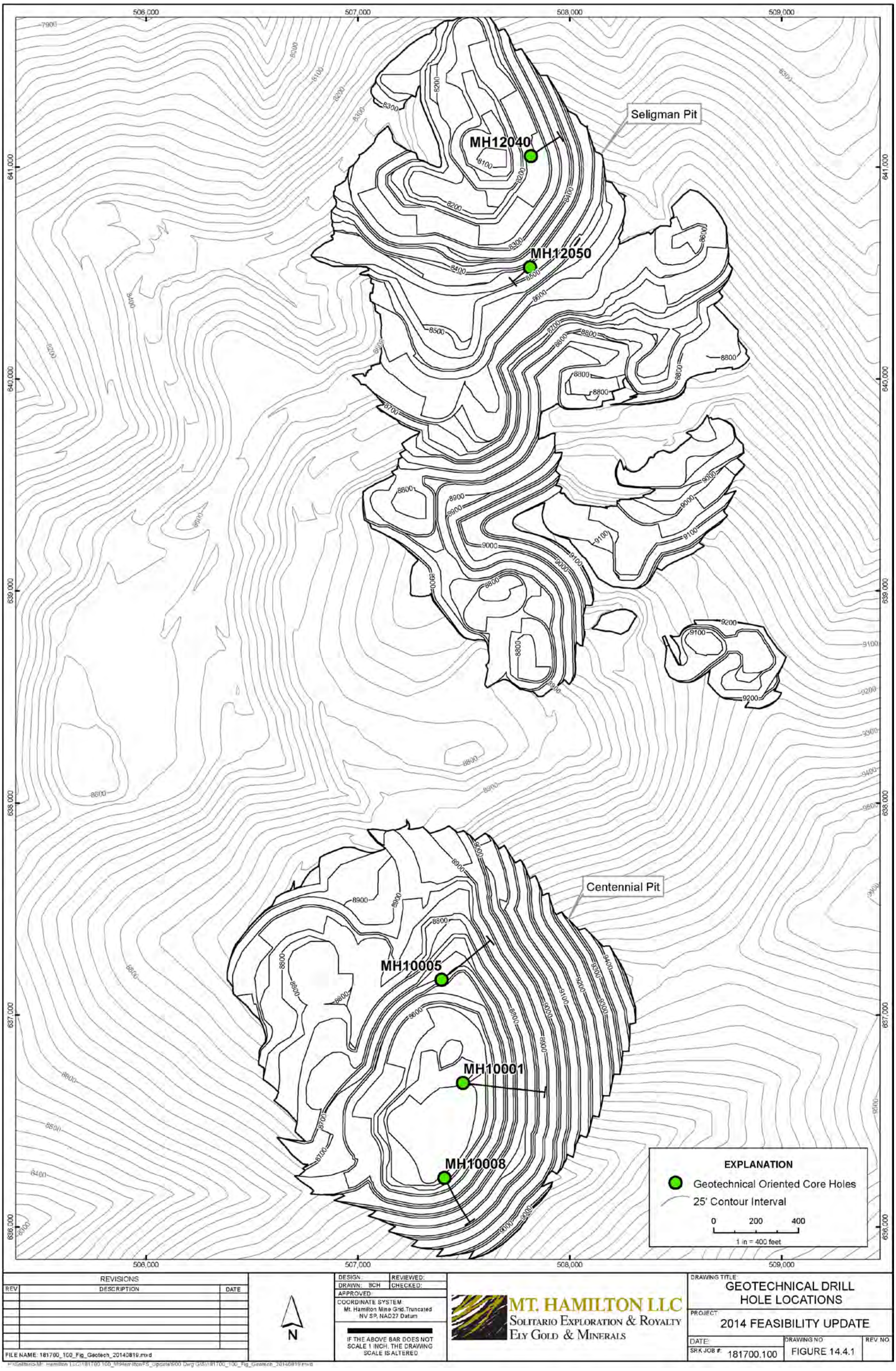
Dilution and mining losses for the Mt. Hamilton mine plan are presented in Table 14.3.2.1.

**Table 14.3.2.1: Mt Hamilton Dilution and Ore Losses**

| Mt Hamilton Dilution and Ore Losses | Difference | % |
|---|---|---|
| Dilution Tonnage | 1,411,616 | 6.0% |
| Ore Loss Tonnage | 1,653,170 | 6.6% |
| Metal Loss - Au | 13,870 | 2.4% |
| Metal Loss - Ag | 177,829 | 3.6% |

## 14.4 Pit Slope Geotechnical Evaluation

After defining objectives and completing the dedicated geotechnical drilling program described below, SRK prepared a feasibility-level geotechnical pit slope evaluation report incorporating recommendations pertaining to optimal pit slope angles and pit architecture for mine design purposes (SRK, 2011a). The significant findings of that report related to pit slope configuration are described in this Section. The locations of supporting geotechnical drill holes are illustrated in Figure 14.4.1. All of the holes shown were drilled as HQ-diameter core and oriented at the rig site using the Reflex ACT II® core orientation tool.




Source: SRK, 2014

**Figure 14.4.1: Geotechnical Drill Hole Locations**

### 14.4.1 Geotechnical Program Objectives

The primary objectives of the feasibility-level geotechnical evaluation for the Centennial project were:

- To collect geotechnical information pertaining to the in situ materials appropriate for a feasibility-level evaluation;
- To characterize geotechnical conditions in and around the area of the proposed open pits;
- To undertake laboratory testing of geomechanical properties of representative samples of the in situ materials;
- To develop a geotechnical model to serve as the basis for the geomechanical evaluation;
- To conduct geomechanical analyses; and
- To make recommendations pertaining to optimal slope angles and pit architecture for mine design purposes.

### 14.4.2 Geotechnical Work Program

The principle stages of the geotechnical evaluation work program were comprised of the following:

- Recommendation of the number, location and orientation of core holes sufficient for a feasibility-level characterization of in situ materials in the open pit area;
- Geotechnical core logging and orientation (oriented core) of discontinuities intersecting core recovered from the drill holes;
- Selection of representative drill core samples from the respective lithological units encountered in the geotechnical drill holes for laboratory testing;
- Submission of the representative samples to the University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, for geomechanical testing;
- Analyses and interpretation of the geotechnical data and laboratory test results to produce a comprehensive analytical model of in situ properties;
- Examination of the anticipated behavior of the geotechnical model relative to expected mining-induced stresses, using various analytical methods; and
- Formulation of pit slope design recommendations.

### 14.4.3 Recommended Pit Slope Configurations

For certain geologic environments, the combination of the average anticipated bench face angle and the preferred interramp angle, based on global (interramp/overall) stability considerations, alone, do not provide a sufficiently wide average catch bench width to effectively control rock fall and/or overbank slough accumulation. In such instances, recommended interramp angles are flattened sufficiently to provide adequately wide average catch benches. This is primarily determined by the analytic indications that a bench could be totally lost and the overlying bench undercut approximately 2% of the time.

Recommendations for interramp and overall slope angles are premised on the rock mass being dry, but depressurization up to approximately 10 meters to 60 meters behind slope faces can be expected should groundwater be encountered. Based on these criteria, SRK recommends that pit slopes at Centennial be designed with a 50° maximum interramp angle using 60 ft high benches with 70° bench face angles and 28 ft wide catch benches. These recommendations are based heavily on achievable bench face angles and less on overall, interramp stability due to the highly competent nature of the skarn and hornfels. Relatively conservative discontinuity lengths were used in the bench design analyses. Significant opportunity exists to steepen certain sectors of the pit depending primarily on actual joint lengths (expected to be less conservative than those assumed here) which can be obtained from mapping of existing surface outcrops or from bench excavations during operation. Existing pit slopes from previous mining at Seligman in the 1990s are stable at angles greater than 52° in many locations.

## 14.5 Pit Optimization

Pit optimization was carried out at Mt. Hamilton using Mintec Inc.'s MineSight® Economic Planner pit optimization software. Pit optimization is based on preliminary economic estimations of mining, processing and selling related costs, slope angles, and metal recoveries. These pit optimization factors are likely to vary from those reported in the final economic analysis, which is based on the pit design criteria and production schedule. The pit optimization software considered grades, tonnages, and recoveries in the model along with mining and processing factors and costs to determine what material could be economically extracted through the use of the LG algorithm.

### 14.5.1 Pit Optimization Parameters

This report was produced at a time of static metal markets with precious metal values in the range of US$1,325/oz for gold and US$20.00/oz for silver. Using prices in this range produced a pit shape with ore tonnage in excess of the currently permitted leach pad capacity of 22.5 Mt. Because of this, SRK was directed to determine the pit limit that would provide the most economical 22.5 Mt of ore. Three price sensitivity analyses were performed with varying capital considerations and a pit from each case with approximately 22.5 Mt of ore was selected for further analysis. Detailed LG phasing was performed on each case to include minimum mining widths and followed the best mining direction. A production schedule was then calculated for each set of LG pits with multiple cut-off strategies. Based on these production schedules, the economics for each case were evaluated and it was determined that an ultimate pit should be designed following an LG phase at prices of US$840/oz US$12.68/oz for gold and silver respectively as this was the most economic case. The selected Ag price is proportional to a US$20/oz spot price as US$840/oz gold is to the US$1,325 spot price for Au. Pit slopes were set to 50° with a 5° reduction (flattening) where access roads were anticipated. The ultimate LG input parameters are summarized in Table 14.5.1.1.

**Table 14.5.1.1: Lerchs-Grossmann Input Parameters**

| Description | Centennial | | | | Seligman | | | |
|---|---|---|---|---|---|---|---|---|
| **Grade Items** | **Au Diluted Grade** | | **Ag Diluted Grade** | | **Au Diluted Grade** | | **Ag Diluted Grade** | |
| Commodity Selling Price | US$840.00 | /oz | US$12.68 | /oz | US$840.00 | /oz | US$12.68 | /oz |
| Commodity Selling Cost | US$4.35 | /oz | US$0.85 | /oz | US$4.35 | /oz | US$0.85 | /oz |
| Default Process Recovery (1) | 76.4% | | 38.0% | | 76.4% | | 43.6% | |
| Royalty (on NSR) | 3.4% | | 3.4% | | 3.4% | | 3.4% | |
| **Mining Costs** | | | | | | | | |
| Ore | US$1.80 | | /t ore | | US$1.95 | | /t ore | |
| Waste | US$1.40 | | /t waste | | US$1.50 | | /t waste | |
| **Processing Costs** | | | | | | | | |
| Capital and Transportation | US$0.00 | | /t ore | | US$0.00 | | /t ore | |
| Crush/Leach Cost | US$3.69 | | /t ore | | US$3.69 | | /t ore | |
| G&A | US$0.71 | | /t ore | | US$0.71 | | /t ore | |
| Total Ore PC | US$4.40 | | /t ore | | US$4.40 | | /t ore | |

Source: SRK, 2014

(1) The default Au recovery is 76.4%. However, each block was assigned a "recovered" grade that took into account the recovery for that specific block and a break even recovered Au oz/t CoG was utilized to define ore and waste block-by-block

## 14.5.2 Pit Optimization Results

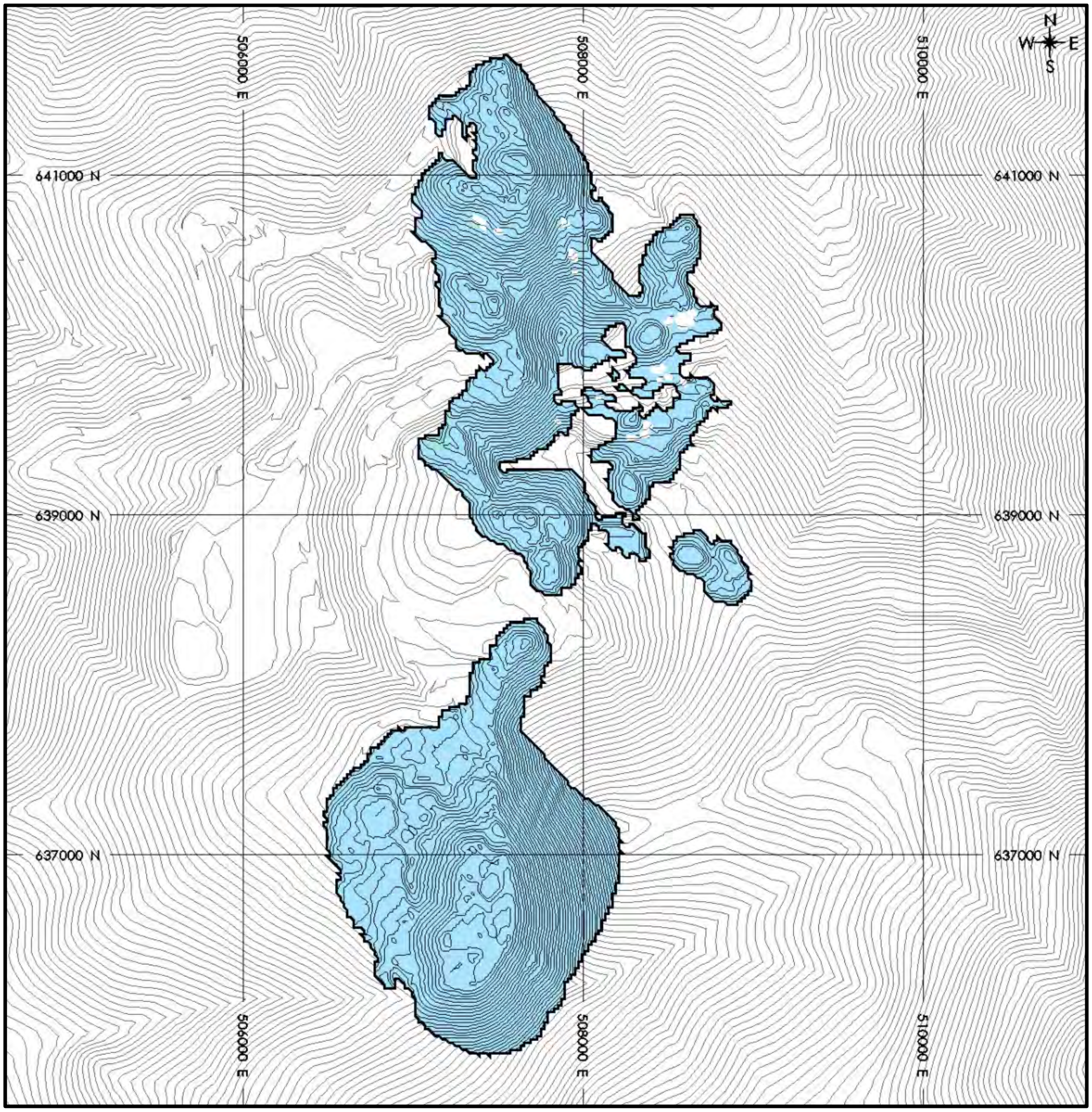
Table 14.5.2.1 shows the results of the US$840/oz LG pit.

**Table 14.5.2.1: US$840 Au Sales Price LG Quantities**

| Item | Value |
|---|---|
| Waste Tons (000's) | 54,004 |
| Ore Tons (000's) | 23,264 |
| Strip Ratio | 2.32 |
| Avg. Diluted Au Grade (oz/t) | 0.025 |
| Avg Diluted Ag Grade (oz/t) | 0.207 |
| Contained Au Ounces (000's) | 572 |
| Contained Ag Ounces (000's) | 4,823 |

Source: SRK, 2014

The ultimate LG pit configuration is shown in Figure 14.5.2.1.




Source: SRK, 2014

**Figure 14.5.2.1: Ultimate LG Pit Configuration**

## 14.6 Mine Design

Using the US$840 Au sales price LG phases described in section 14.5.1 as a guide, detailed pit designs were created for each mining area that included safety catch benches and access roads.

All pits were designed at an overall pit slope angle of 50° using a 70° face angle and a 28 ft-wide catch bench per geotechnical recommendations. A bench height of 20 ft was selected to match anticipated equipment sizing for Centennial, while the Seligman Pit was designed with 10 ft benching to allow for selective mining of the ore (waste will be mined on 20 ft benches). In the Centennial Pit triple benching was used, where a catch bench is left every 60 ft rather than on each bench. In the Seligman Pit a catch bench was left every six benches to match the 60 ft benching in Centennial. Four designed pit phases were developed for the Centennial mining area and four phases were designed for Seligman.

Access for the Seligman pits will be challenging due to high-relief terrain and previous mining activity. To ensure that access would be maintained through the life of the mine, several sub phases were created within the main Seligman phases connecting to each planned access point.

During the process of pit construction, several iterations were required to produce a final pit design that would:

- Maintain safe operating width on all phases;
- Provide higher grade mineralization early in the project life; and
- Allow for construction of surface roads to provide access to higher benches.

### 14.6.1 Designed Pit Parameters

Haul road widths were based on an expected fleet of 100 t capacity haul trucks. For this equipment a 75 ft running width will provide a truck width to running surface width ratio of about 3.5, to meet industry recommendations for safety and reduced operating cost. All road designs include an additional 15 ft width to allow adequate space for a safety berm, resulting in a 90 ft road width in the designs.

While the majority of the haul roads were designed to handle two-way traffic, in deeper areas of both pits, it was necessary to reduce road width to single-lane traffic to minimize excessive waste stripping or loss of recoverable ore. One-way traffic haul roads for pit bottoms and short jump ramps between access points are designed at a width of 68 ft including the berm.

In-pit roads were designed with an 8% to 10% gradient wherever possible with an occasional jump ramp as steep as 12% when necessary to maintain access or in the bottom benches of the pit.

Table 14.6.1.1 lists the parameters used for pit design.

**Table 14.6.1.1: Designed Pit Parameters**

| Parameters | Centennial | Seligman | Units |
|---|---|---|---|
| Interramp Pit Slope (1) | 50 | 50 | deg |
| Bench Face Angle | 70 | 70 | deg |
| Bench Height | 20 | 10 | ft |
| Benches per Catch Bench | 3 | 6 | |
| Catch Bench Width | 28 | 28 | ft |
| Road Grade (2) | 8% to 10% | 8% to 10% | |
| Bottom 2 benches | 12% | 12% | |
| Road Width (including berm) (3) | 90 | 90 | ft |
| Bottom 2 benches | 68 | 68 | ft |
| Slot Cut Road Width (3) | 75 | 75 | ft |
| Bottom 2 benches | 54 | 54 | ft |
| Minimum Mining Width (4) | 225 | 225 | ft |

Source: SRK, 2014

(1) When mining through fill a 1.5:1 (21.8°) pit slope was designed with a 28 ft wide catch bench every 60 ft.
(2) 8 to 10% grades were used where possible but steeper grades up to 12% may be used for short distances when necessary to maintain access.
(3) Assumes 100 t trucks
(4) Narrower widths may be used for short cuts and small benches or when material can be dozer pushed to reduce mining costs.

### 14.6.2 Pit Design Results

Cent 1 is designed primarily to access the shallow ore in the Centennial mining area. Secondary to this several jump ramps were left in the highwall to ensure that access would be maintained to the intermediate benches of Cent 2.

Previous scheduling exercises showed a pocket of deep ore with very good grade that has typically been included in the final phase design. Since this ore was left in one of the final phases, a spike in grade was typically observed in the final periods of mining that was significantly discounted in the project economics. It was decided that Cent 2 would mine down to include some of this deep ore to attempt to move that grade forward in the schedule. The upper most benches of Cent 2 are planned to mine to the ultimate pit limit as the mining width for those upper benches would have been too narrow for two separate pushbacks. This does increase the stripping early on. If the ultimate pit limit is increased in the future, these benches should be narrowed to allow less pre-stripping of Cent 2. Access roads for Cent 4 are left in the highwall of Cent 2.

The Cent 3 pit is planned to mine the marginal ore at shallow depth on the western edge of the Centennial deposit. This pit is accessed through Cent 1 and via an overland ramp constructed from the Central Haul road. This pit can be mined simultaneously with Cent 3 if there is any additional equipment availability.

Cent 4 accesses the deepest ore in the Centennial mine plan and mines down to the ultimate pit limit. No access roads are planned in the highwall of this pit design. Additional drilling in the southern portion of this pit has the potential to increase the reserve. If material is upgraded, the mining sequence may be improved.

Selig A is designed to access the deep Northern Seligman ore. Due to the previous mining activity, access for this pit is difficult to develop and maintain. Additionally, the LG guide pit comes very close to the existing highwall and does not include a reasonable mining width on each bench. Because of these issues, the Selig A pit was divided into several sub-phases that connected to each external access point. The ramps left in the highwall were left wide at 120 ft to allow the equipment fleet enough room to go back and pull the ramps efficiently as they mined down. For benches where the mining width is too narrow, fill was added to increase the mining width. Some benches will also be accessed via a fill road. All of the re-handling of this fill material (~1 Mt) is included in the production schedule. Additionally, some or the Eastern skarn that was included in the B phase of the LGs was incorporated into this design as it could only be access from benches in the Selig A pit.

Selig B1 and Selig B2 are both satellite pits in the eastern skarn. These pits are tied into the planned access roads when possible but will require some overland access cut to be developed. Based on the shallow nature of these pits and large quantity of inferred material, the reserves within these pits could be increased with additional drilling.

Selig C is planned to mine the southern portion of the Seligman deposit and is the last pit to be mined in the production schedule. This pit must be mined following the upper benches of the Centennial mining area as they are accessed via a road cut into topography above this pit. There is some inferred material in this pit, which if upgraded, has the potential to move the pit forward in the mining sequence.

Incremental designed pits are shown in plan view in Figure 14.6.2.1 which includes a call out for cross section A-A' through the Centennial phase designs. The A-A' cross section through the Centennial phase designs is provided as Figure 14.6.2.2.

Table 14.6.2.1 details the ore and waste tonnages within the Centennial designed pits and Table 14.6.2.2 details the ore and waste tonnages within the Seligman designed pits.

**Table 14.6.2.1: Centennial Designed Pit Tonnages**

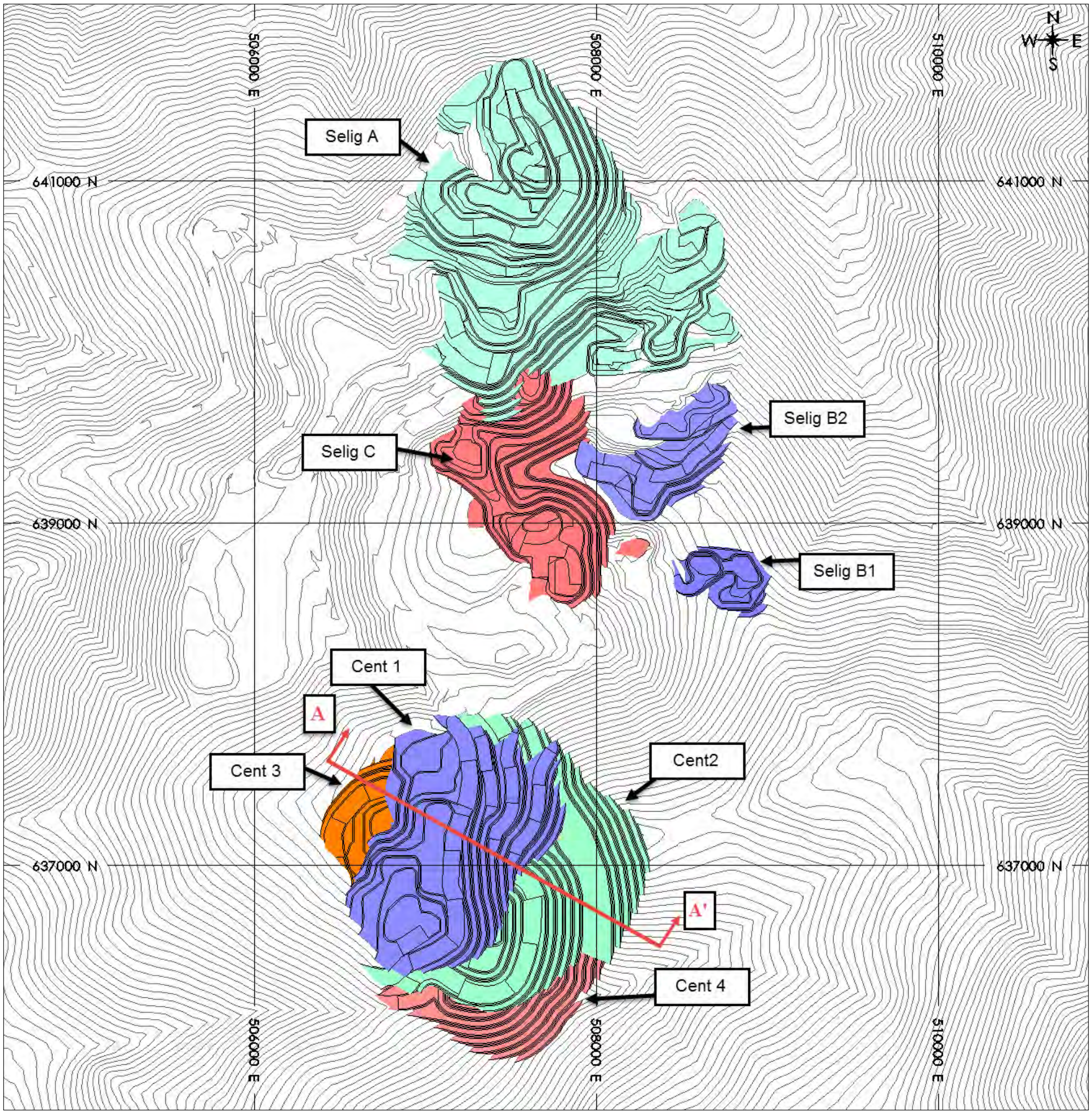
| Item | Cent 1 | Cent 2 | Cent 3 | Cent 4 | Total |
|---|---|---|---|---|---|
| Waste Tons | 11,065 | 18,249 | 1,683 | 11,980 | 42,976 |
| Ore Tons (000's) | 5,745 | 4,246 | 887 | 5,577 | 16,455 |
| Strip Ratio | 1.93 | 4.30 | 1.90 | 2.15 | 2.61 |
| Ave Au Grade (oz/t) | 0.024 | 0.024 | 0.026 | 0.018 | 0.022 |
| Ave Ag Grade (oz/t) | 0.073 | 0.189 | 0.219 | 0.267 | 0.176 |
| Contained Au Ounces (000's) | 140 | 104 | 23 | 102 | 369 |
| Contained Ag Ounces (000's) | 421 | 801 | 194 | 1,487 | 2,904 |

Source: SRK, 2014

**Table 14.6.2.2: Seligman Designed Pit Tonnages**

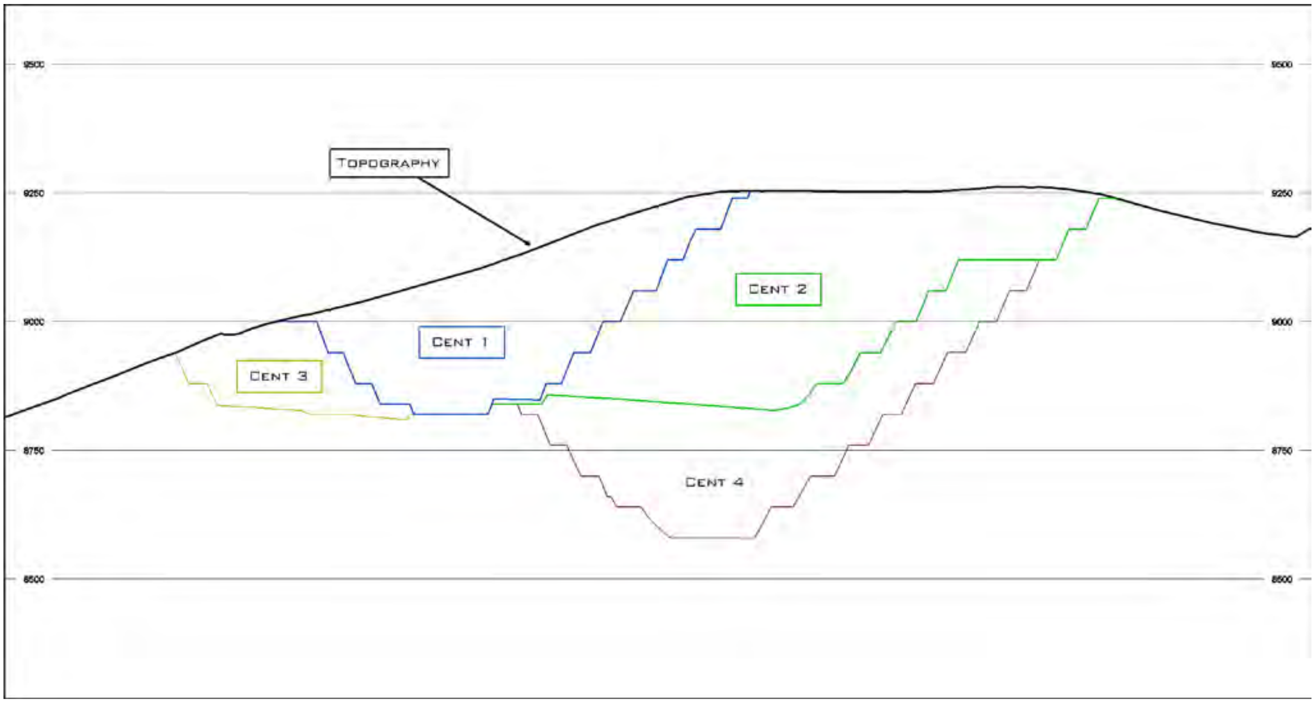
| Item | Selig A | Selig B1 | Selig B2 | Selig C | Total |
|---|---|---|---|---|---|
| Waste Tons | 15,070 | 591 | 1,708 | 4,547 | 21,917 |
| Ore Tons (000's) | 3,666 | 134 | 277 | 1,968 | 6,045 |
| Strip Ratio | 4.11 | 4.42 | 6.16 | 2.31 | 3.63 |
| Ave Au Grade (oz/t) | 0.023 | 0.042 | 0.028 | 0.015 | 0.021 |
| Ave Ag Grade (oz/t) | 0.150 | 0.019 | 0.048 | 0.154 | 0.143 |
| Contained Au Ounces (000's) | 84 | 6 | 8 | 29 | 127 |
| Contained Ag Ounces (000's) | 549 | 3 | 13 | 302 | 867 |

Source: SRK, 2014




Source: SRK, 2014

**Figure 14.6.2.1: Incremental Designed Pits – Plan View**




Source: SRK, 2014

**Figure 14.6.2.2: Centennial Pit Design – Cross Section A-A'**

## 14.6.3 Mining Losses

The inclusion of haul roads and creation of a practical pit design when compared with pit optimization results, indicate a 24.2% increase in stripping ratio, 20.2% increase in waste generation, and 3.3% decrease in ore feed tonnage. This difference is almost entirely due to the additional access requirements for the Seligman mining area. The LG surface is very narrow around the existing Seligman Pit and the design needed to be widened in those areas to allow equipment access. This widening was done with fill when possible, but some cut was required. On a tonnage basis, the Centennial pit saw only a 5% increase in waste material over the LG design.

Table 14.6.3.1 shows the mining losses associated with the larger pit size after the inclusion of in-pit ramps and minimum mining widths. The engineered pit is slightly larger than the optimized LG pit.

**Table 14.6.3.1: Mining Losses from Pit Design vs. Optimized LG Pit**

| Item | US$840 LG Phase | Designed Pit | Difference | % Difference |
|---|---|---|---|---|
| Waste Tons (000's) | 54,004 | 64,893 | 10,889 | 20.2% |
| Ore Tons (000's) | 23,264 | 22,500 | -764 | -3.3% |
| Strip Ratio | 2.32 | 2.88 | 0.56 | 24.2% |
| Ave Au Grade (oz/t) | 0.025 | 0.022 | 0 | -10.4% |
| Ave Ag Grade (oz/t) | 0.207 | 0.168 | 0 | -19.2% |
| Contained Au Ounces (000's) | 572 | 496 | -76 | -13.3% |
| Contained Ag Ounces (000's) | 4,823 | 3,771 | -1,052 | -21.8% |

Source: SRK, 2014
Strip ratio for the designed pit does not include 2.9 Mt mined as ore but retained as stockpile so as not to exceed permitted leach pad capacity.

## 14.7 Waste Rock Storage Design

Primary waste rock storage facilities (dumps) are located northwest of the Centennial Pit in Cabin Gulch, with some additional material planned as backfill assuming it proves economically viable during operations to do so and does not sterilize possible reserves. The waste rock facilities external to the mined pits have been designed for a final reclaimed slope 2.5H:1V angle consistent with Nevada State reclamation requirements (Figure 14.1). Placed waste rock is assumed to have a tonnage factor of 17 $ft^3$/t. The dumps were designed with 10% additional volume above the projected reserve volume to increase operational flexibility. In most cases, end-dump methods will be used to place the waste rock. The Cabin Gulch design is a valley fill in two lifts, a large lower lift and a smaller upper lift to facilitate high elevation stripping.

## 14.8 Stockpile Design

Two stockpiles are planned to allow for more operational flexibility when a surplus of ore is present. The first stockpile will store low grade material and is located on top of the existing dump in Cabin gulch as shown in Figure 14.1. The second stockpile will be located just below the primary crusher southwest of the Centennial pit and will store high grade material. In the current production schedule, material with a recoverable Au grade between 0.004 oz/t and 0.016 oz/t that was not shipped to the crusher was stored in the Low Grade Stockpile. Material with a recoverable Au grade of 0.016 and higher was stored in the High Grade Stockpile when not shipped directly to the crusher.

By having stockpiles, the operation can be optimized to send the best material available to the crusher each period. Segregating the high and low grade materials into separate stockpiles allows for more control during CoG optimization. There are three typical situations where stockpiling allows for optimization. First, in times when there is excess ore mining capacity, the operation will mine through additional ore and ship the highest grade to the crusher while stockpiling the lower grades. Second, when ore mining is limited due to stripping hurdles the stockpiled material can supplement the ore feed. Finally when the ore grades available in the pit are lower than the grades stored in the stockpile, the stockpiles can be shipped to the crusher while the ore in the pit is stockpiled. In each of these scenarios, higher grade material will be moved forward in the schedule and the project NPV can be increased.

## 14.9 Haulage

Haulage calculations for the production schedule were estimated using MineSight® Haulage (MSH). This tool was used to calculate average cycle times for ore and waste from each level of every pit phase to all possible destinations. These cycle times were then used during production scheduling to calculate the required truck hours each period.

Haul profiles were calculated by digitizing the entire haulage network from every phase to all possible destinations and the software was allowed to determine the fastest route. The haulage profiles were then exported to spreadsheet format for validation.

### 14.9.1 Haulage Parameters

Based on the selected 100 t truck fleet, haulage calculations were performed assuming Caterpillar 777G (tier4) trucks. Operating parameters, mainly rim pull and braking curves, for these haul units were taken from the *Caterpillar Performance Handbook - Edition 42.* To ensure the accuracy of the calculation in MineSight® Haulage, six haul profiles of varying distances and gradients were taken from each mining area and analyzed in both MSH and Caterpillar's Fleet Production and Cost Analysis (FPC) software. FPC offers a more robust assessment of the true travel time to be encountered in the field, but has limited application for Life-of-Mine (LoM) planning. The cycle times were then compared and the fleet input parameters were adjusted in MSH until an acceptable correlation was seen between the cycle times from both programs.

Due to the high altitude and anticipated weather conditions, haulage speeds were capped at 25 mph.

## 14.10 Mine Production Schedule

Production scheduling was carried out using MineSight® Schedule Optimizer v8.50-01. The schedule was constructed around a daily crusher feed of 10,000 t/d, which translates to 3.5 Mt/y. The amount of waste stripping was maximized at approximately 32,000 t/d translating to 11.25 Mt/y assuming 350 operating days per year at the mine.

As recovery was variable by block, the production schedule was driven on recovered grade where the Au and Ag grade of each block was multiplied by that block's metallurgical recovery. This allowed the value for each block to be calculated independently and that value was then used to optimize the Net Present Value (NPV) of the schedule each period. At the average gold recovery for the deposit of approximately 76%, the in situ CoG is approximately 0.006 oz/t Au, but was higher for some individual periods when higher grade material could be pushed forward to improve the NPV of the schedule. Any material above cut-off that was displaced by higher grade material was stockpiled and reclaimed from the stockpile in a later period.

The production schedule was used to estimate the quantities of waste material produced each year for dump designs and to calculate the annual total haulage requirements.

A monthly pre-production/pre-strip period was included assuming the material would be moved utilizing the MH-LLC mine equipment. Subsequent to pre-strip, time periods in the schedule are monthly for the first two years of production and then quarterly for the LoM. The total mine life in the schedule is just under 7 years.

### 14.10.1 Production Scheduling Methodology

Using software, pit design mining shapes were cut into benches and the tons and grades were calculated for each of these mining shapes along with the required access cut shapes. This information was then imported to the scheduler and a linear optimization engine was used to compare the possible mining combinations in each period and determine the most economic mining pattern. This schedule was required to meet ore and waste targets each period while considering pre-stripping requirements, the maximum number of benches that could be mined in a given period, and the development required to access subsequent benches. Through this process stockpiling was used to store excess ore when possible and to supplement ore feed during periods when additional stripping was required.

Preproduction construction and stripping was scheduled for approximately seven months prior to ore production.

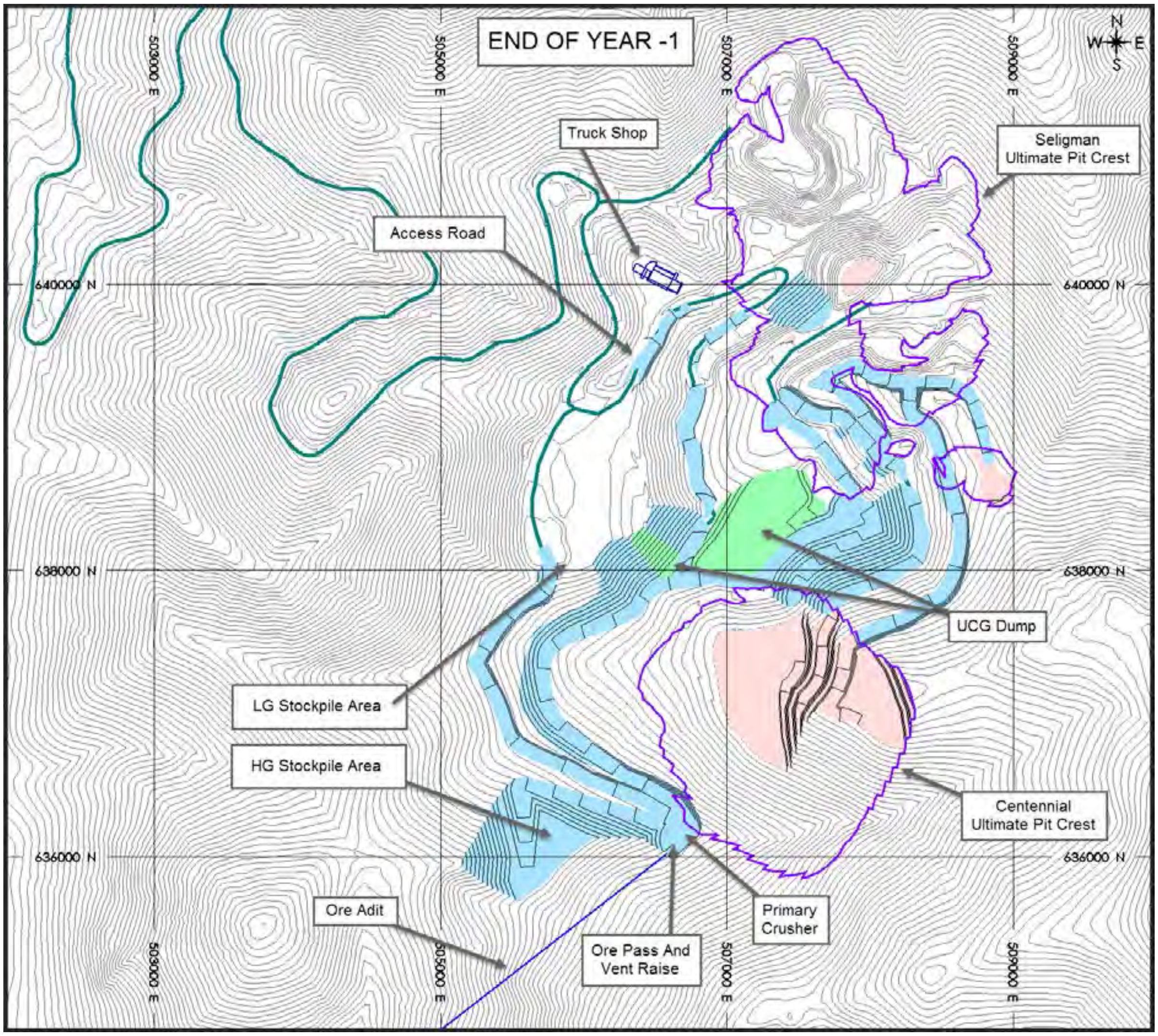
### 14.10.2 Production Schedule Results

Table 14.10.2.1 details the mine production schedule. Yearly progress maps are shown as Figures 14.10.2.1 through 14.10.2.9.

### Table 14.10.2.1: Mine Production Schedule

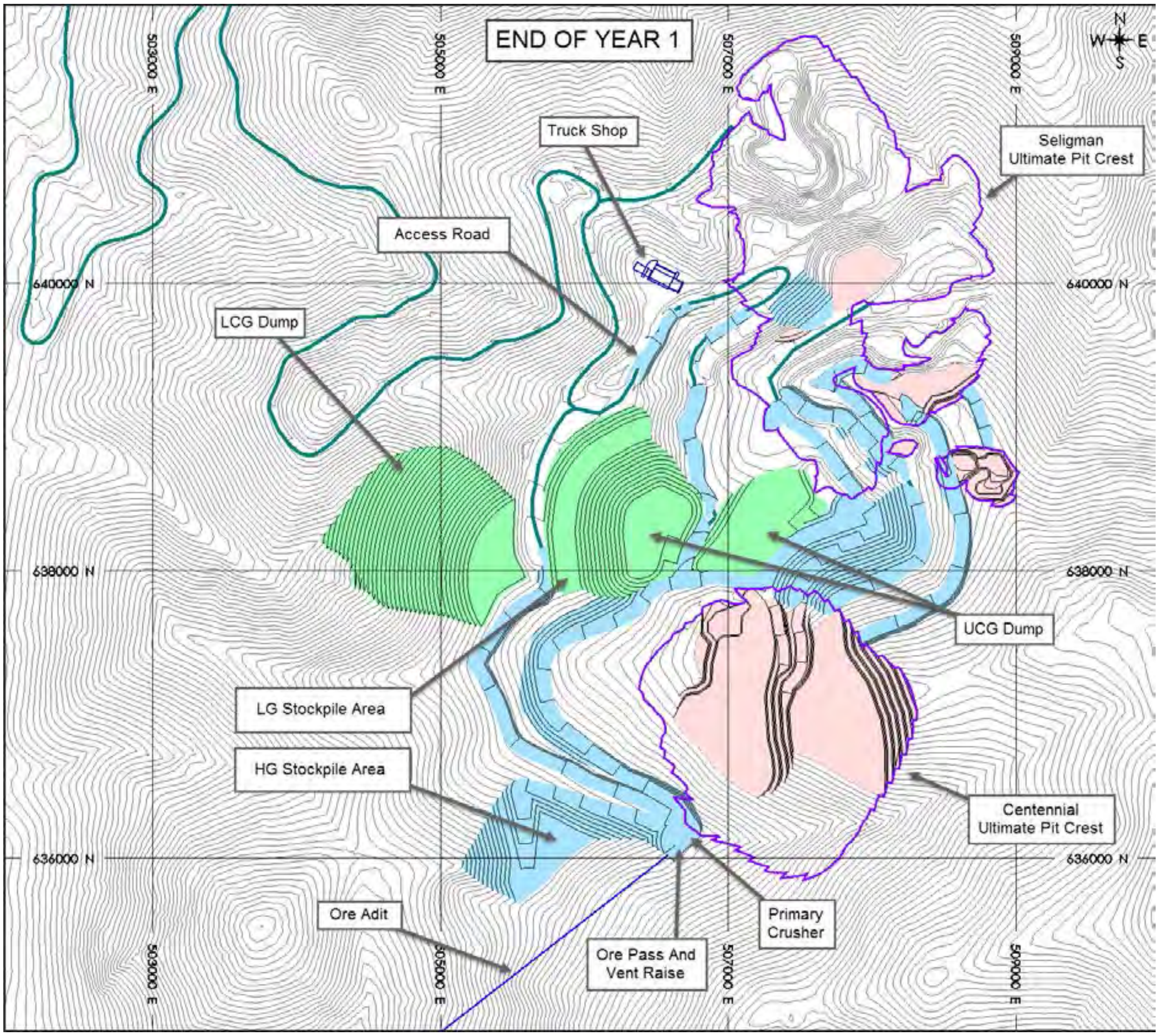
| Production Period | | Date | Period ID | Preproduction & Development Waste | Total Waste | Total Ore | Au oz Contained | Ag oz Contained | Total Material Mined (including rehandle) | Strip Ratio | Au Grade Mined | Ag Grade Mined | Ore to Crusher | Au Contained to Crusher | Ag Contained to Crusher | Au Recovery | Ag Recovery | Ore to Stockpile | Ore from Stockpile | Total Contained Au Crushed | Total Contained Ag Crushed |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | kt | kt | kt | koz | koz | kt | w/o | oz/t | oz/t | kt | oz/t | oz/t | % | % | kt | kt | koz | koz |
| LoM | | Total | | 2,532 | 60,473 | 25,463 | 607 | 5,012 | 88,468 | 2.47 | 0.024 | 0.197 | 22,500 | 0.024 | 0.198 | 76.2% | 39.1% | 8,625 | 5,662 | 545.4 | 4,459.6 |
| Preproduction | 2015 | Apr-15 | 1 | 357.1 | 0.0 | 2.9 | 0.0 | 0.1 | 360.0 | 125.2 | 0.011 | 0.045 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 2.9 | 0.0 | 0.0 | 0.0 |
| | | May-15 | 2 | 590.6 | 0.0 | 18.3 | 0.3 | 2.1 | 608.9 | 32.3 | 0.018 | 0.113 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 18.3 | 0.0 | 0.0 | 0.0 |
| | | Jun-15 | 3 | 572.9 | 0.0 | 7.1 | 0.1 | 0.3 | 580.0 | 80.2 | 0.010 | 0.036 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 7.1 | 0.0 | 0.0 | 0.0 |
| | | Jul-15 | 4 | 58.5 | 659.8 | 1.7 | 0.0 | 0.1 | 720.0 | 426.4 | 0.011 | 0.044 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 1.7 | 0.0 | 0.0 | 0.0 |
| | | Aug-15 | 5 | 86.4 | 809.8 | 3.8 | 0.0 | 0.1 | 900.0 | 235.8 | 0.011 | 0.019 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 3.8 | 0.0 | 0.0 | 0.0 |
| | | Sep-15 | 6 | 61.9 | 813.5 | 24.6 | 0.2 | 1.1 | 900.0 | 35.6 | 0.010 | 0.043 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 24.6 | 0.0 | 0.0 | 0.0 |
| | | Oct-15 | 7 | 154.9 | 755.9 | 39.3 | 0.5 | 9.5 | 950.0 | 23.2 | 0.013 | 0.243 | 0.0 | 0.000 | 0.000 | 0.0% | 0.0% | 39.3 | 0.0 | 0.0 | 0.0 |
| | | Nov-15 | 8 | 0.0 | 1,107.5 | 74.9 | 1.2 | 20.0 | 1,182.4 | 14.8 | 0.016 | 0.266 | 60.0 | 0.019 | 0.232 | 73.0% | 38.5% | 32.5 | 17.6 | 1.1 | 13.9 |
| | | Dec-15 | 9 | 0.0 | 1,061.8 | 56.5 | 1.1 | 10.7 | 1,118.3 | 18.8 | 0.020 | 0.190 | 120.0 | 0.013 | 0.138 | 71.7% | 38.8% | 18.2 | 81.7 | 1.5 | 16.6 |
| Production | 2016 | Jan-16 | 10 | 0.0 | 1,083.6 | 67.4 | 3.2 | 1.9 | 1,151.0 | 16.1 | 0.048 | 0.028 | 116.0 | 0.029 | 0.059 | 81.2% | 41.5% | 0.4 | 49.0 | 3.3 | 6.8 |
| | | Feb-16 | 11 | 0.0 | 1,063.0 | 137.0 | 4.5 | 15.7 | 1,200.0 | 7.8 | 0.033 | 0.114 | 116.0 | 0.034 | 0.097 | 80.8% | 41.0% | 21.0 | 0.0 | 3.9 | 11.2 |
| | | Mar-16 | 12 | 0.0 | 1,026.0 | 159.6 | 2.8 | 32.7 | 1,185.6 | 6.4 | 0.017 | 0.205 | 174.0 | 0.015 | 0.175 | 70.2% | 38.0% | 0.0 | 14.4 | 2.7 | 30.5 |
| | | Apr-16 | 13 | 0.0 | 1,024.5 | 175.5 | 2.9 | 36.7 | 1,200.0 | 5.8 | 0.017 | 0.209 | 174.0 | 0.015 | 0.188 | 70.7% | 38.0% | 1.5 | 0.0 | 2.7 | 32.7 |
| | | May-16 | 14 | 0.0 | 770.2 | 429.8 | 7.2 | 88.1 | 1,200.0 | 1.8 | 0.017 | 0.205 | 290.0 | 0.019 | 0.203 | 75.3% | 38.0% | 139.8 | 0.0 | 5.6 | 59.0 |
| | | Jun-16 | 15 | 0.0 | 871.6 | 252.2 | 4.8 | 48.4 | 1,123.8 | 3.5 | 0.019 | 0.192 | 290.0 | 0.017 | 0.181 | 73.8% | 38.0% | 38.4 | 76.2 | 5.1 | 52.5 |
| | | Jul-16 | 16 | 0.0 | 960.0 | 237.8 | 4.7 | 41.7 | 1,197.8 | 4.0 | 0.020 | 0.175 | 290.0 | 0.018 | 0.161 | 72.2% | 38.0% | 0.0 | 52.2 | 5.1 | 46.8 |
| | | Aug-16 | 17 | 0.0 | 960.0 | 236.9 | 4.8 | 42.6 | 1,196.9 | 4.1 | 0.020 | 0.180 | 290.0 | 0.018 | 0.165 | 72.6% | 38.0% | 0.0 | 53.1 | 5.2 | 47.8 |
| | | Sep-16 | 18 | 0.0 | 734.1 | 515.9 | 11.5 | 147.4 | 1,250.0 | 1.4 | 0.022 | 0.286 | 290.0 | 0.031 | 0.377 | 80.3% | 38.0% | 225.9 | 0.0 | 8.9 | 109.4 |
| | | Oct-16 | 19 | 0.0 | 960.0 | 275.3 | 6.4 | 73.1 | 1,235.3 | 3.5 | 0.023 | 0.266 | 290.0 | 0.023 | 0.252 | 75.7% | 38.0% | 0.0 | 14.7 | 6.6 | 73.0 |
| | | Nov-16 | 20 | 0.0 | 904.5 | 240.2 | 5.6 | 60.4 | 1,144.7 | 3.8 | 0.023 | 0.252 | 300.0 | 0.021 | 0.232 | 75.2% | 38.3% | 45.5 | 105.3 | 6.4 | 69.5 |
| | | Dec-16 | 21 | 0.0 | 892.2 | 357.8 | 9.5 | 84.2 | 1,250.0 | 2.5 | 0.027 | 0.235 | 300.0 | 0.029 | 0.249 | 78.6% | 38.0% | 57.8 | 0.0 | 8.6 | 74.8 |
| | 2017 | Jan-17 | 22 | 0.3 | 898.5 | 351.2 | 9.6 | 74.8 | 1,250.0 | 2.6 | 0.027 | 0.213 | 290.0 | 0.030 | 0.224 | 77.0% | 38.0% | 61.2 | 0.0 | 8.7 | 65.0 |
| | | Feb-17 | 23 | 0.0 | 930.5 | 251.3 | 7.0 | 52.3 | 1,181.7 | 3.7 | 0.028 | 0.208 | 290.0 | 0.024 | 0.192 | 74.3% | 38.0% | 29.5 | 68.3 | 7.1 | 55.6 |
| | | Mar-17 | 24 | 0.0 | 875.4 | 374.6 | 8.5 | 50.2 | 1,250.0 | 2.3 | 0.023 | 0.134 | 290.0 | 0.025 | 0.146 | 75.6% | 38.0% | 84.6 | 0.0 | 7.4 | 42.3 |
| | | Apr-17 | 25 | 0.0 | 700.5 | 549.5 | 14.0 | 83.6 | 1,250.0 | 1.3 | 0.025 | 0.152 | 290.0 | 0.035 | 0.191 | 77.4% | 38.0% | 259.5 | 0.0 | 10.2 | 55.5 |
| | | May-17 | 26 | 171.7 | 686.3 | 392.0 | 9.5 | 58.0 | 1,250.0 | 2.2 | 0.024 | 0.148 | 290.0 | 0.028 | 0.167 | 74.7% | 38.3% | 102.0 | 0.0 | 8.3 | 48.4 |
| | | Jun-17 | 27 | 0.0 | 842.0 | 408.0 | 11.0 | 65.7 | 1,250.0 | 2.1 | 0.027 | 0.161 | 290.0 | 0.032 | 0.184 | 76.3% | 38.6% | 118.0 | 0.0 | 9.4 | 53.4 |
| | | Jul-17 | 28 | 0.0 | 591.9 | 658.1 | 25.4 | 119.7 | 1,250.0 | 0.9 | 0.039 | 0.182 | 290.0 | 0.062 | 0.256 | 79.1% | 38.0% | 368.1 | 0.0 | 17.9 | 74.2 |
| | | Aug-17 | 29 | 102.8 | 808.2 | 151.6 | 4.2 | 8.0 | 1,062.6 | 6.0 | 0.027 | 0.052 | 290.0 | 0.026 | 0.104 | 74.3% | 39.9% | 49.0 | 187.4 | 7.5 | 30.0 |
| | | Sep-17 | 30 | 0.0 | 891.6 | 182.4 | 3.5 | 22.9 | 1,074.1 | 4.9 | 0.019 | 0.126 | 290.0 | 0.017 | 0.101 | 72.2% | 39.4% | 68.4 | 175.9 | 4.8 | 29.3 |
| | | Oct-17 | 31 | 0.0 | 897.5 | 143.9 | 1.9 | 25.1 | 1,041.4 | 6.2 | 0.013 | 0.174 | 290.0 | 0.012 | 0.124 | 68.7% | 38.3% | 62.5 | 208.6 | 3.6 | 36.1 |
| | | Nov-17 | 32 | 20.1 | 923.9 | 150.8 | 2.9 | 41.7 | 1,094.9 | 6.3 | 0.019 | 0.276 | 290.0 | 0.015 | 0.175 | 69.4% | 38.2% | 16.0 | 155.1 | 4.3 | 50.8 |
| | | Dec-17 | 33 | 0.0 | 907.1 | 342.9 | 5.9 | 83.5 | 1,250.0 | 2.6 | 0.017 | 0.243 | 290.0 | 0.018 | 0.252 | 71.5% | 38.0% | 52.9 | 0.0 | 5.1 | 73.1 |
| | 2018 | Jan-18 | 34 | 0.0 | 2,528.7 | 1,221.3 | 24.5 | 340.2 | 3,750.0 | 2.1 | 0.020 | 0.279 | 890.0 | 0.023 | 0.317 | 74.1% | 38.0% | 331.3 | 0.0 | 20.7 | 282.0 |
| | | Apr-18 | 35 | 0.0 | 2,762.5 | 987.5 | 20.7 | 131.5 | 3,750.0 | 2.8 | 0.021 | 0.133 | 870.0 | 0.022 | 0.134 | 77.6% | 38.0% | 117.5 | 0.0 | 18.8 | 116.5 |
| | | Jul-18 | 36 | 58.8 | 2,055.0 | 1,636.2 | 37.8 | 175.8 | 3,750.0 | 1.3 | 0.023 | 0.107 | 870.0 | 0.033 | 0.135 | 78.1% | 38.0% | 766.2 | 0.0 | 28.4 | 117.5 |
| | | Oct-18 | 37 | 0.0 | 2,406.8 | 1,343.2 | 43.8 | 376.9 | 3,750.0 | 1.8 | 0.033 | 0.281 | 870.0 | 0.045 | 0.365 | 76.6% | 38.0% | 473.2 | 0.0 | 39.0 | 317.4 |
| | 2019 | Jan-19 | 38 | 0.0 | 2,264.2 | 1,485.8 | 36.9 | 340.6 | 3,750.0 | 1.5 | 0.025 | 0.229 | 890.0 | 0.032 | 0.293 | 75.4% | 38.0% | 595.8 | 0.0 | 28.8 | 260.4 |
| | | Apr-19 | 39 | 0.0 | 1,948.7 | 1,801.3 | 52.6 | 449.9 | 3,750.0 | 1.1 | 0.029 | 0.250 | 870.0 | 0.045 | 0.376 | 77.7% | 38.0% | 931.3 | 0.0 | 39.0 | 327.1 |
| | | Jul-19 | 40 | 0.0 | 1,964.1 | 1,785.9 | 46.3 | 433.4 | 3,750.0 | 1.1 | 0.026 | 0.243 | 870.0 | 0.039 | 0.372 | 76.3% | 38.0% | 915.9 | 0.0 | 34.3 | 323.4 |
| | | Oct-19 | 41 | 0.0 | 2,115.0 | 1,635.0 | 43.5 | 391.3 | 3,750.0 | 1.3 | 0.027 | 0.239 | 870.0 | 0.037 | 0.326 | 75.3% | 38.9% | 765.0 | 0.0 | 32.1 | 283.3 |
| | 2020 | Jan-20 | 42 | 0.0 | 2,793.2 | 864.8 | 16.7 | 123.8 | 3,658.0 | 3.2 | 0.019 | 0.143 | 890.0 | 0.017 | 0.128 | 81.8% | 43.0% | 66.8 | 92.0 | 15.3 | 113.5 |
| | | Apr-20 | 43 | 256.9 | 2,317.2 | 777.2 | 12.4 | 143.5 | 3,351.2 | 3.3 | 0.016 | 0.185 | 870.0 | 0.015 | 0.156 | 77.8% | 41.1% | 305.9 | 398.8 | 13.2 | 135.7 |
| | | Jul-20 | 44 | 39.4 | 2,684.3 | 328.1 | 8.0 | 37.5 | 3,051.8 | 8.3 | 0.024 | 0.114 | 870.0 | 0.015 | 0.104 | 72.3% | 39.6% | 156.2 | 698.2 | 13.0 | 90.2 |
| | | Oct-20 | 45 | 0.0 | 2,871.5 | 324.5 | 8.6 | 41.8 | 3,196.0 | 8.8 | 0.027 | 0.129 | 870.0 | 0.015 | 0.107 | 73.3% | 40.4% | 8.5 | 554.0 | 12.9 | 93.0 |
| | 2021 | Jan-21 | 46 | 0.0 | 2,417.6 | 1,212.5 | 32.8 | 175.6 | 3,630.1 | 2.0 | 0.027 | 0.145 | 890.0 | 0.030 | 0.151 | 74.4% | 43.6% | 322.5 | 0.0 | 26.9 | 134.1 |
| | | Apr-21 | 47 | 0.0 | 2,070.9 | 1,580.5 | 27.9 | 217.5 | 3,651.4 | 1.3 | 0.018 | 0.138 | 870.0 | 0.022 | 0.167 | 82.1% | 43.6% | 710.5 | 0.0 | 19.3 | 145.5 |
| | | Jul-21 | 48 | 0.0 | 659.9 | 644.6 | 9.5 | 96.8 | 1,304.5 | 1.0 | 0.015 | 0.150 | 870.0 | 0.013 | 0.132 | 79.1% | 42.1% | 106.9 | 332.3 | 11.6 | 114.8 |
| | | Oct-21 | 49 | 0.0 | 190.8 | 446.7 | 7.6 | 100.6 | 637.5 | 0.4 | 0.017 | 0.225 | 870.0 | 0.013 | 0.162 | 76.2% | 41.3% | 88.0 | 511.3 | 11.7 | 141.3 |
| | 2022 | Jan-22 | 50 | 0.0 | 11.1 | 116.7 | 2.1 | 33.3 | 127.8 | 0.1 | 0.018 | 0.285 | 890.0 | 0.011 | 0.120 | 73.4% | 40.2% | 12.6 | 785.9 | 9.5 | 106.4 |
| | | Apr-22 | 51 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | - | - | - | 870.0 | 0.010 | 0.096 | 72.5% | 39.8% | 0.0 | 870.0 | 8.5 | 83.7 |
| | | Jul-22 | 52 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | - | - | - | 160.0 | 0.010 | 0.096 | 72.5% | 39.8% | 0.0 | 160.0 | 1.6 | 15.4 |

Source: SRK, 2014



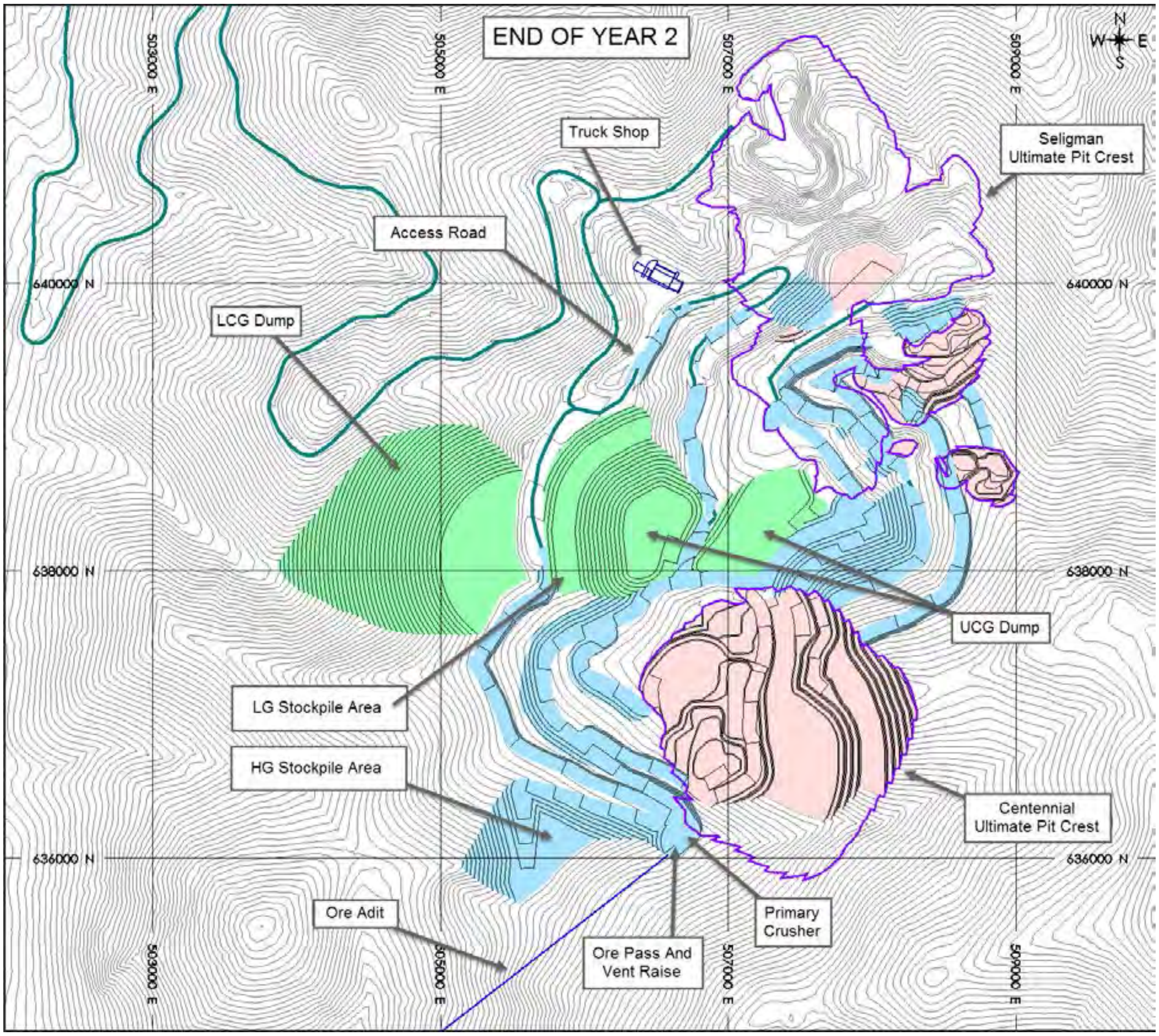

Source: SRK, 2014

**Figure 14.10.2.1: Annual Mining Progression – End of Year -1**



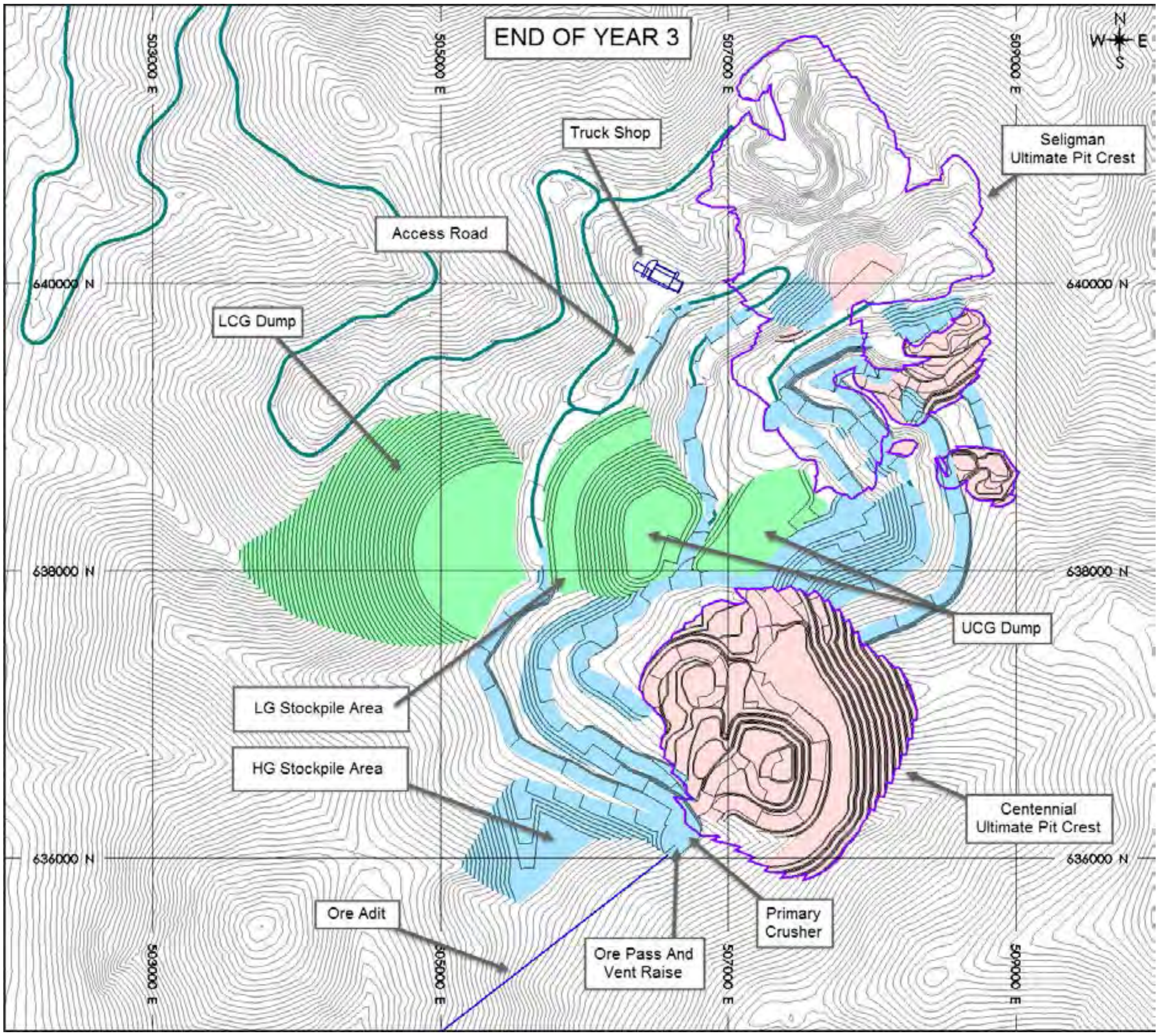

Source: SRK, 2014

**Figure 14.10.2.2: Annual Mining Progression – End of Year 1**



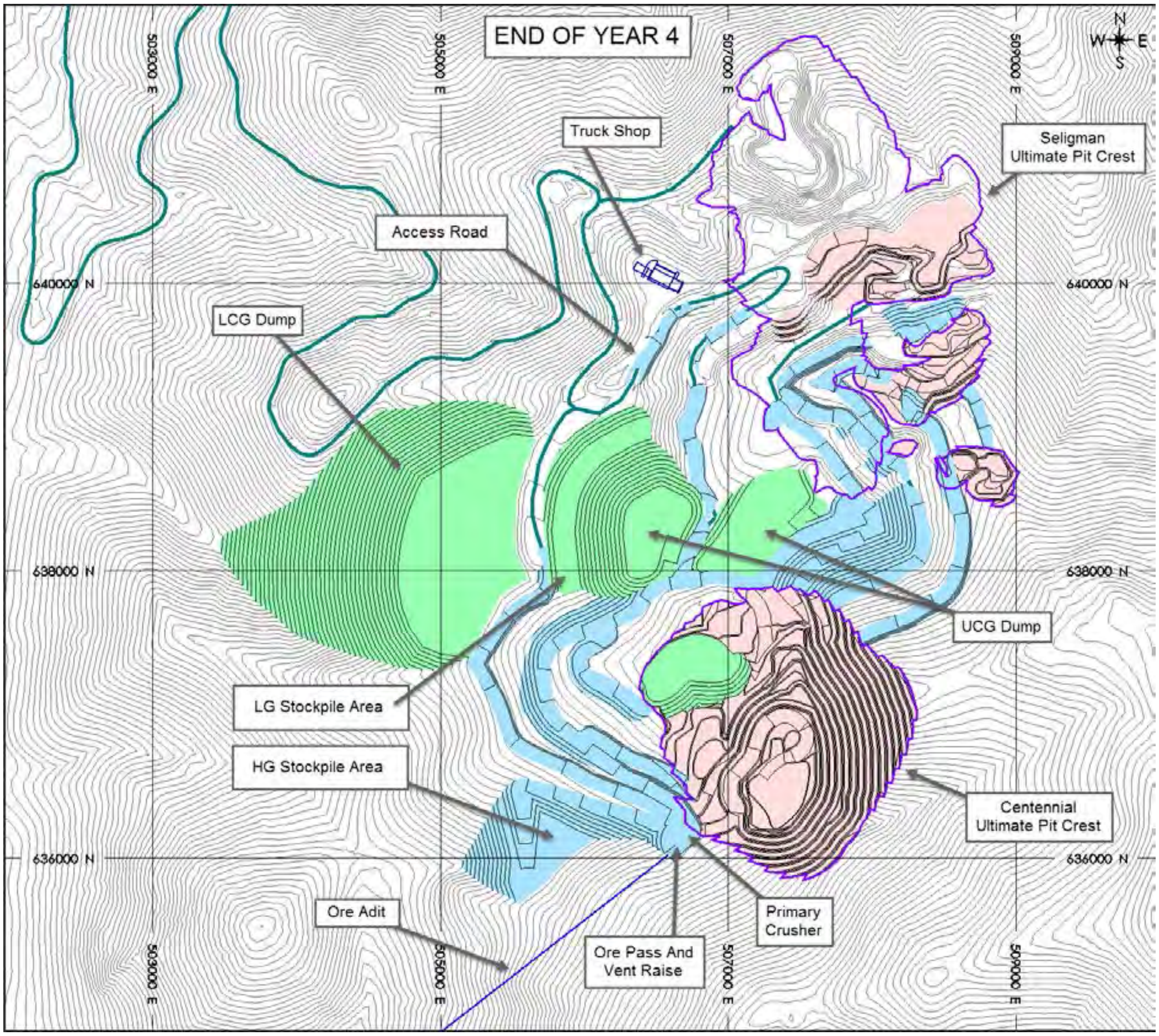

Source: SRK, 2014

**Figure 14.10.2.3: Annual Mining Progression – End of Year 2**



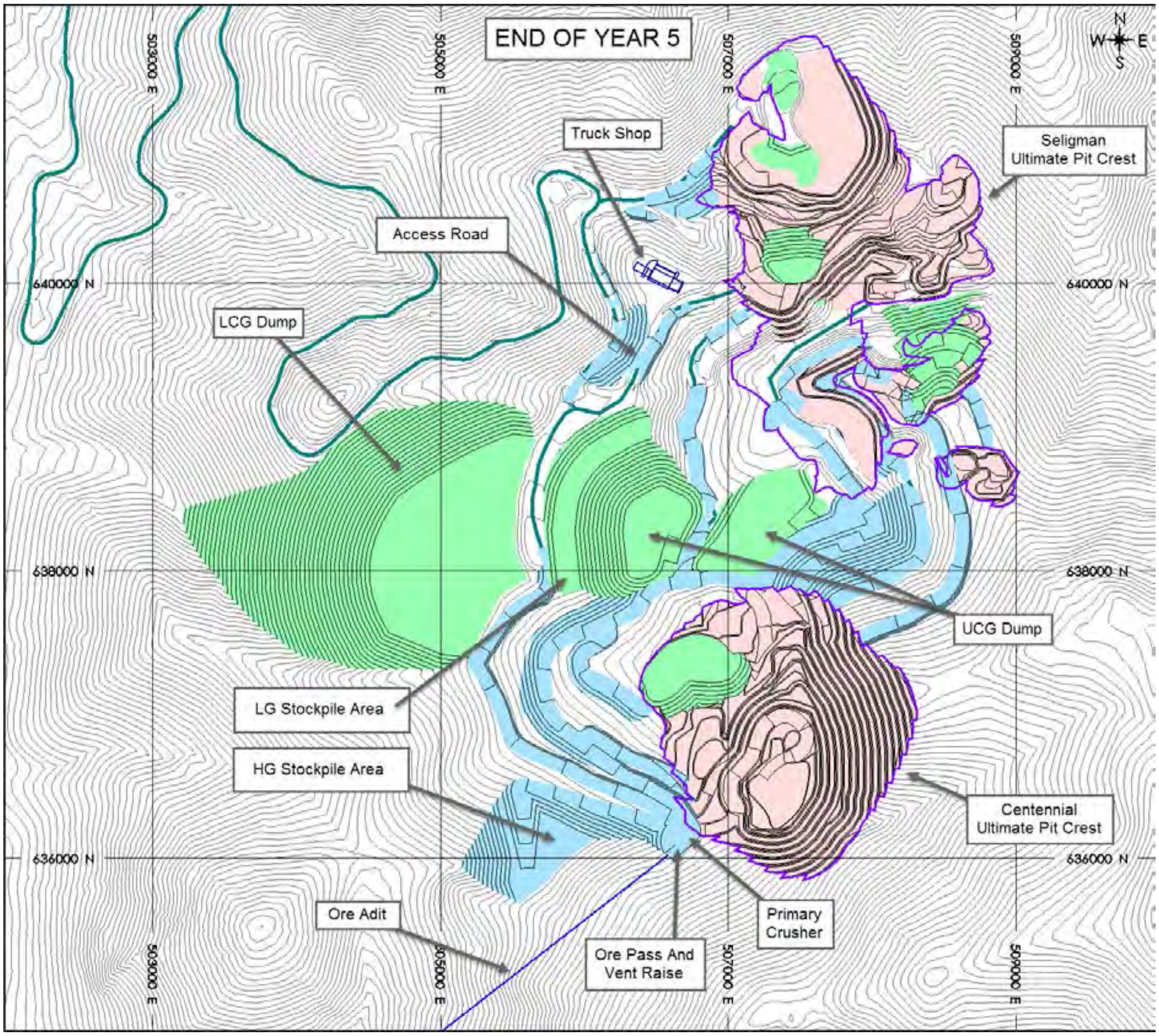

Source: SRK, 2014

**Figure 14.10.2.4: Annual Mining Progression – End of Year 3**



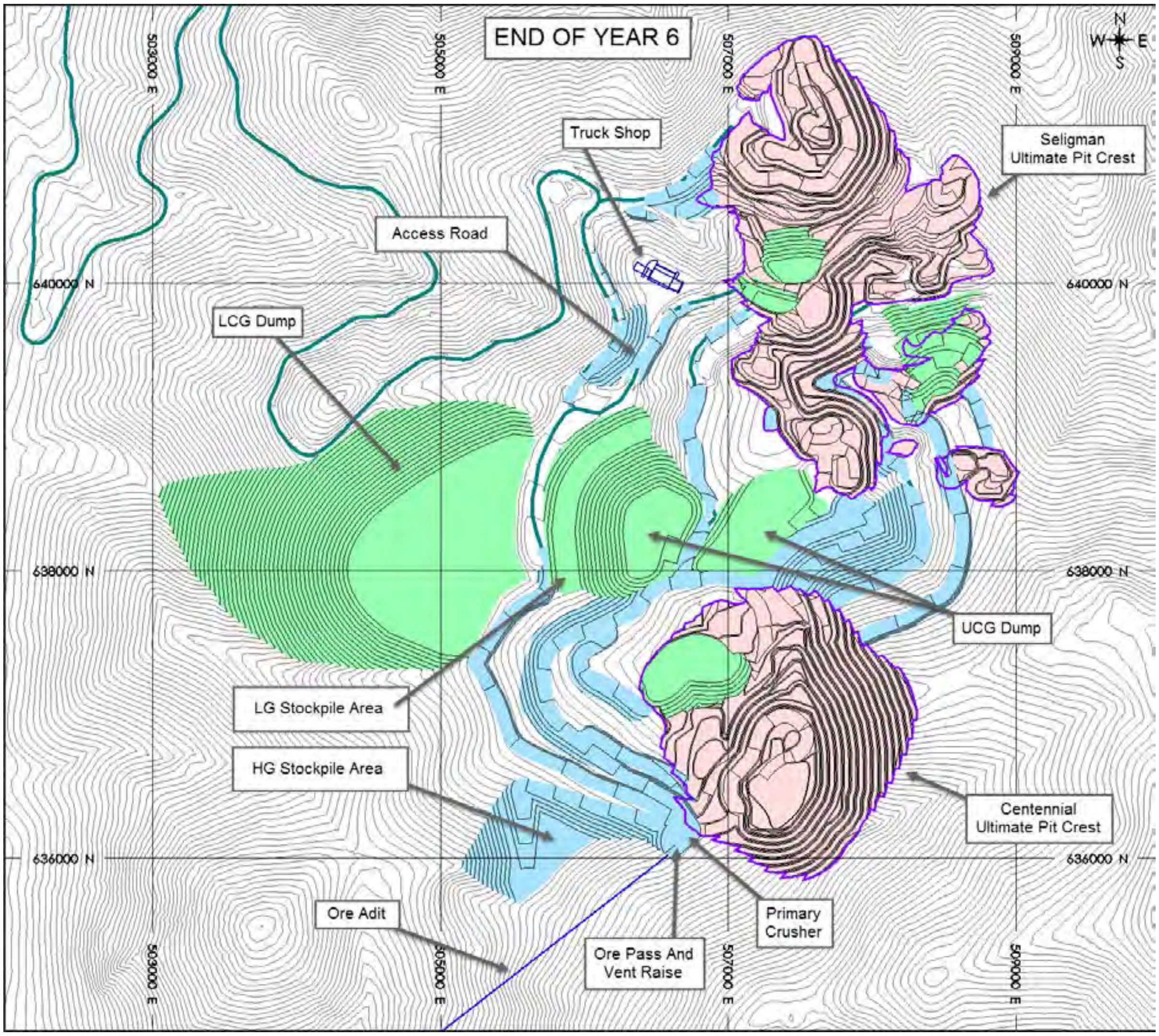

Source: SRK, 2014

**Figure 14.10.2.5: Annual Mining Progression – End of Year 4**



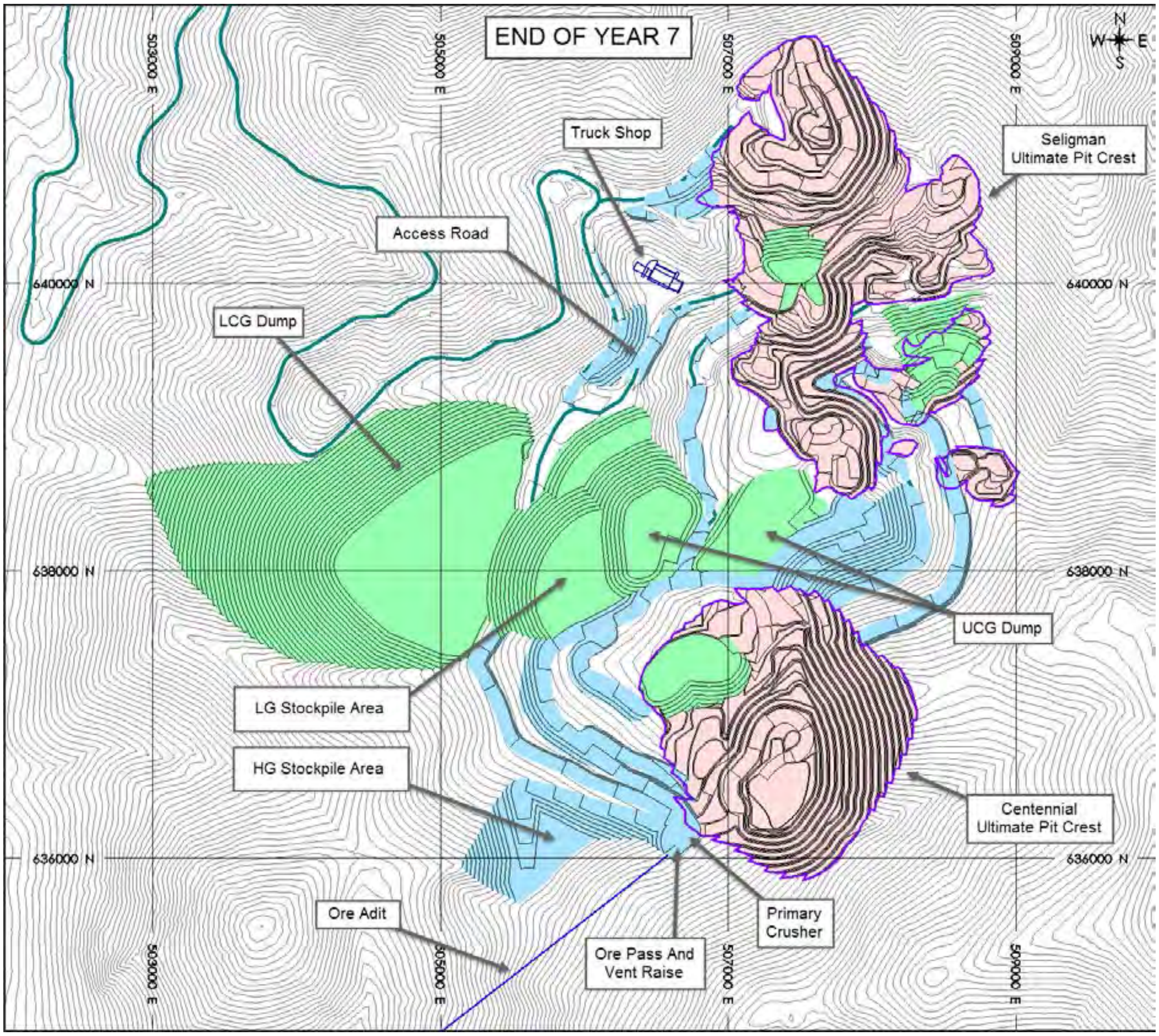

Source: SRK, 2014

**Figure 14.10.2.6: Annual Mining Progression – End of Year 5**



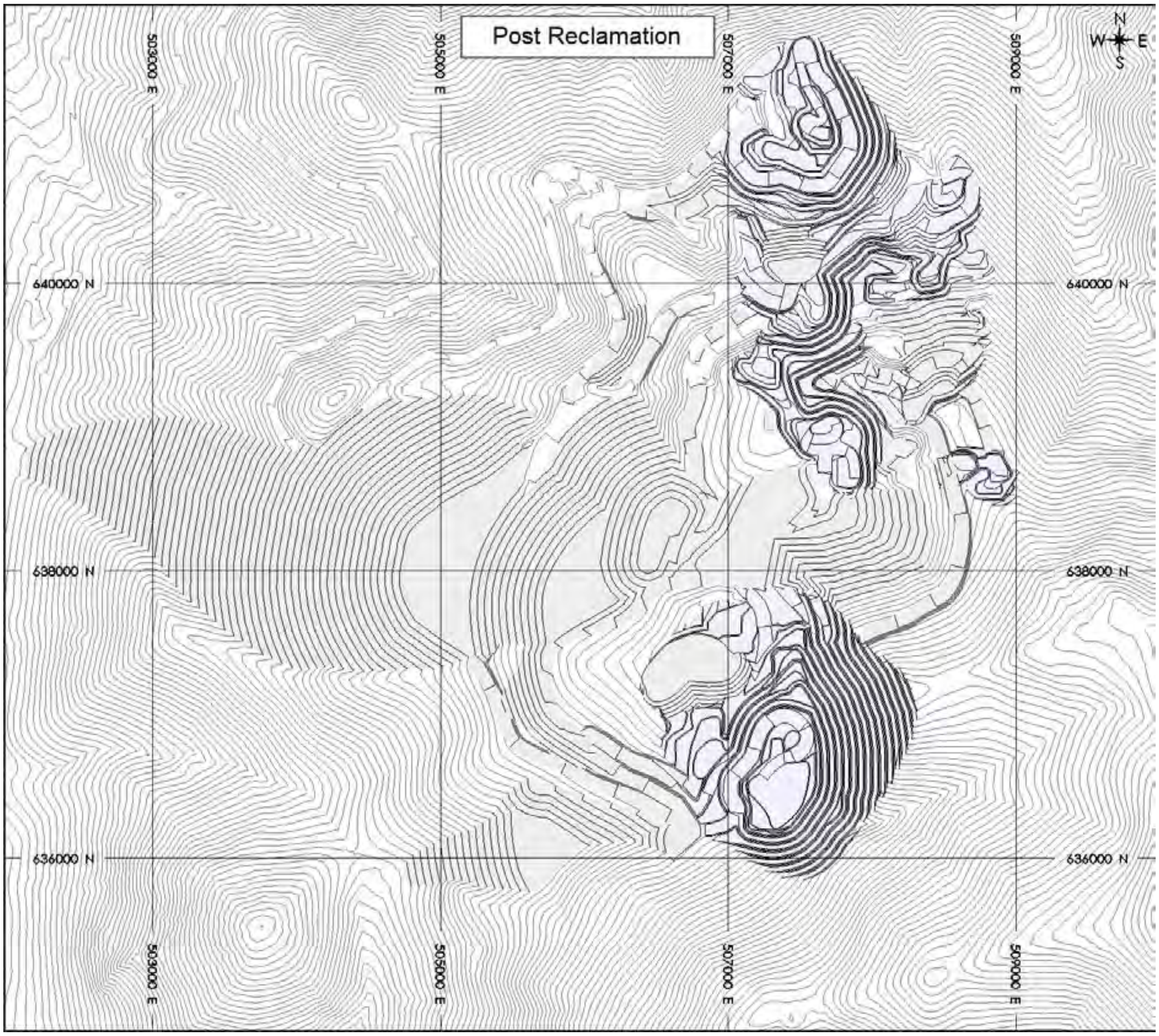

Source: SRK, 2014

**Figure 14.10.2.7: Annual Mining Progression – End of Year 6**




Source: SRK, 2014

**Figure 14.10.2.8: Annual Mining Progression – End of Year 7**

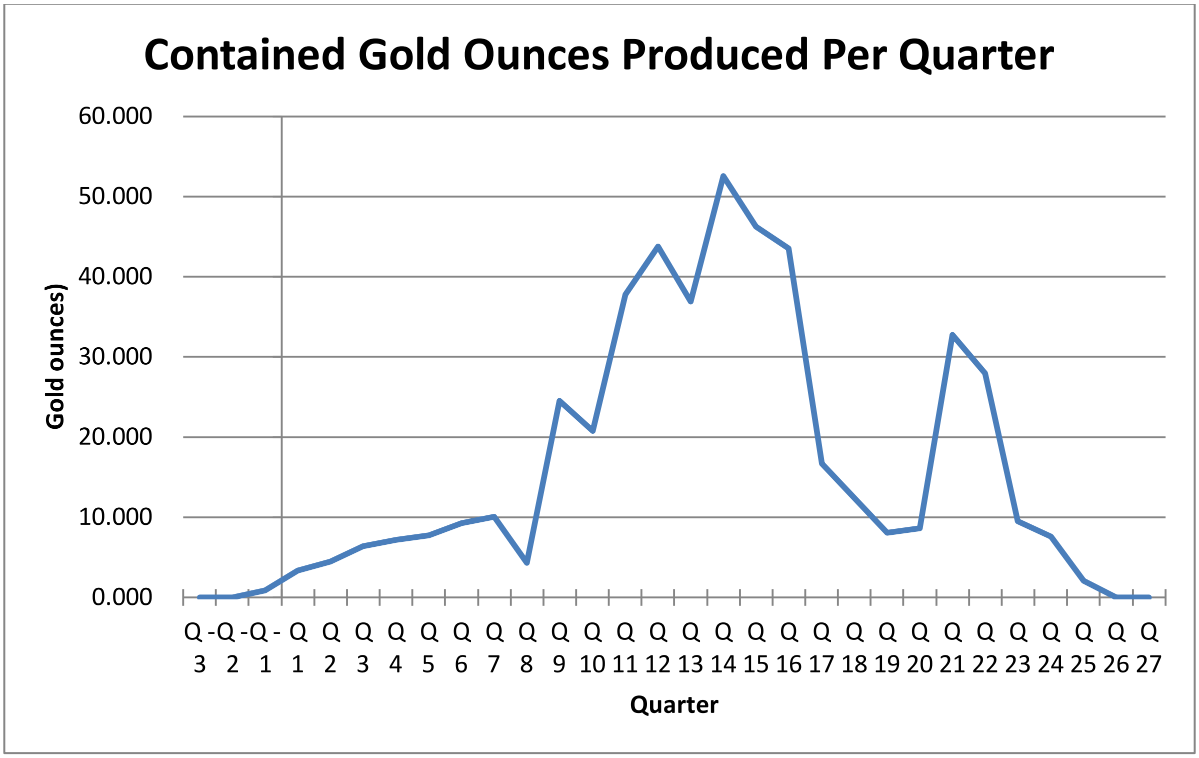


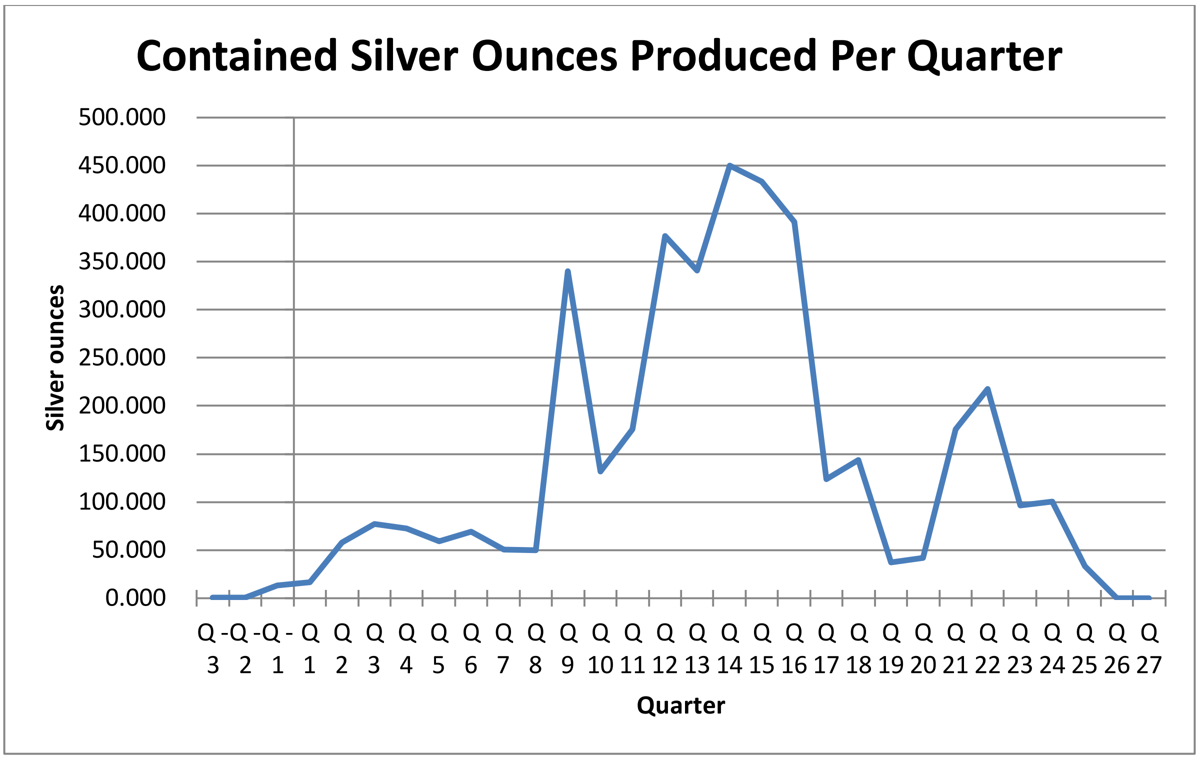

Source: SRK, 2014

**Figure 14.10.2.9: Annual Mining Progression – Post Reclamation**

### 14.10.3 Grade Distribution

Ore grade from the mine was not held constant during scheduling. Rather, the best material available in each period was processed and sub grade material was stockpiled. By allowing the cut-off for material to vary by period, the NPV of the project was improved as the ounces recovered in earlier periods were increased. Additionally, the pit pushbacks were designed to take advantage of ore near surface and localized higher grade material, giving an irregular grade distribution with lower grades at the beginning of each pushback and higher grades towards the end of each pushback. By scheduling multiple pushbacks at once this effect was minimized; however, some grade variability is still present in the production schedule. Figure 14.10.3.1 shows the production schedule grade distribution graphically.




Source: SRK, 2014

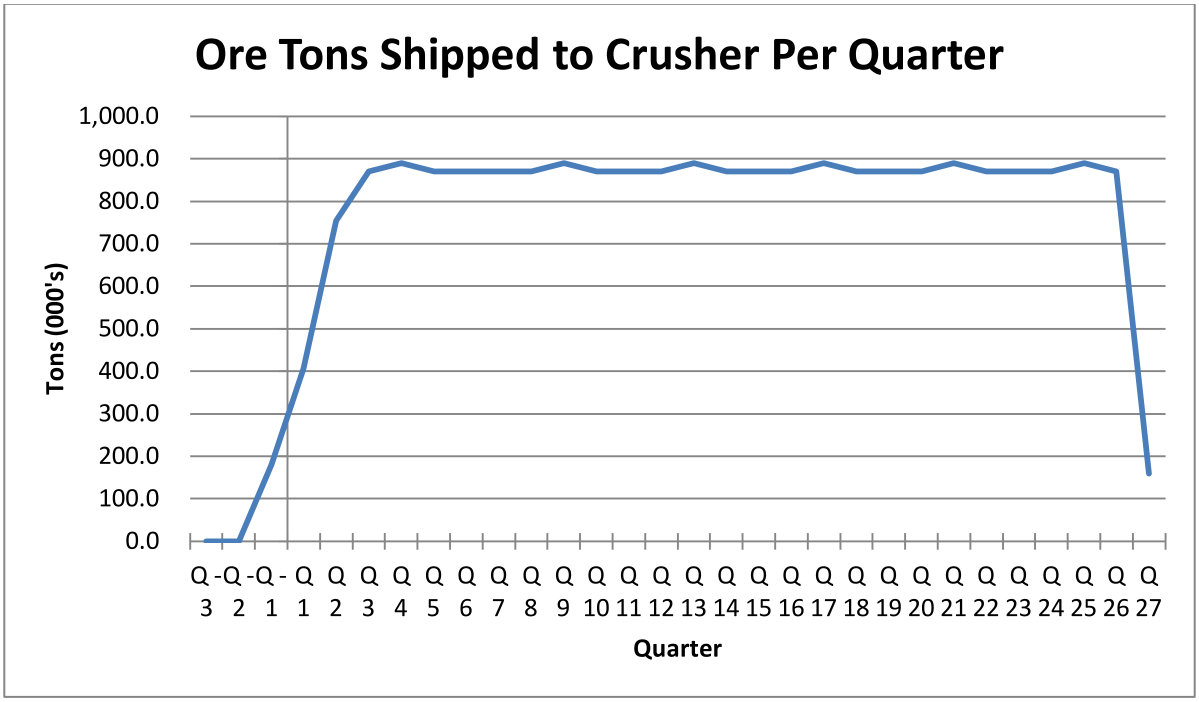
**Figure 14.10.3.1: Gold and Silver Grade Distribution over the Life-of-Mine**

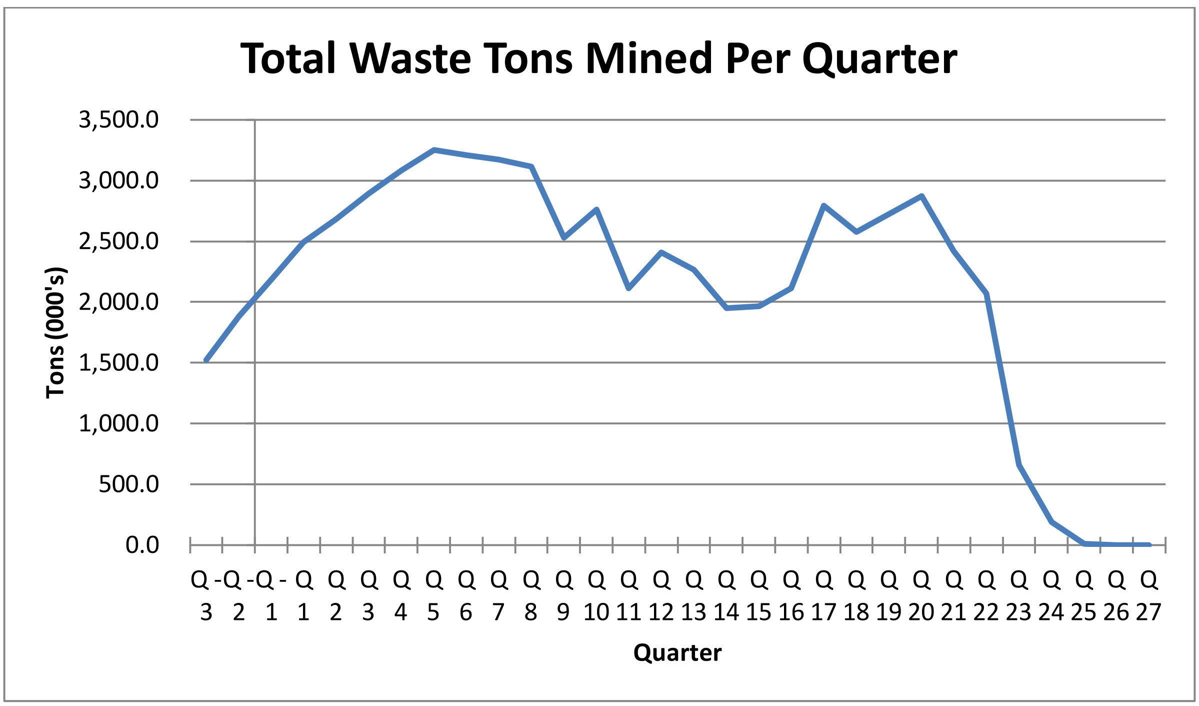
### 14.10.4 Tonnage Distribution

The ore and waste tonnage distributions in the production schedule are presented in Figure 14.10.4.1.

Ore tonnage distribution is constant at approximately 3.5 Mt/y with the best material available being shipped to the crusher. Because of this, some lower grade material is often stockpiled while higher grades will be reclaimed from the stockpile in that same period.

Waste tonnage mined averages 10 Mt/y for the first 5 years of production with a peak of 11.25 Mt in the first production period as stripping is increased to access very high grade material in the bottoms of Cent 2 as quickly as possible. Once this material is mined, the waste stripping rate decreases to roughly 8.2 Mt in Year 4 when the Centennial Pit is completed. Following this, there is a stripping spike in Year 5 to 10.9 Mt when the Seligman Pit is stripped.




Source: SRK, 2014

**Figure 14.10.4.1: Ore and Waste Production Distribution over the Life-of-Mine**

## 14.11 Mining Operations and Equipment

Mt. Hamilton reserves will be mined by conventional truck and shovel open-pit mining methods. The mine life is estimated to be 7 years with an additional nine months of pit pre-stripping. LoM mining-rate averages for the mine are estimated at 3.5 Mt/y ore and approximately 9 Mt/y waste.

### 14.11.1 Mine Operations and Equipment

The mine is scheduled to initially operate on two ten-hour shifts per day, seven days per week, 350 days per year. During Year 1 the mine will shift to two 12-hour shifts per day, seven days per week. To match the slowdown in waste production, the third quarter of Year 6 is reduced to a single 10 hr shift, seven days a week and will be maintained for the remainder of production. Table 14.11.1.1 shows how the number of shifts and hours per shift varies over the LoM.

**Table 14.11.1.1: Production Shift Schedule**

| **Production year** | **-1** | **1** | **2** | **3** | **4** | **5** | **6 (1)** | **7** |
|---|---|---|---|---|---|---|---|---|
| Working days/year | 240 | 350 | 350 | 350 | 350 | 350 | 325 | 12 |
| Hours/shift | 10 | 12 | 12 | 12 | 12 | 12 | 12 | 10 |
| Shift/day | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 |

Source: SRK, 2014
(1) Initial Shift Schedule. Reduced to one 10 hr shift in 4th quarter.

Operating efficiency was determined by assuming that a total 30 minutes would be lost at the beginning and end of each shift for crew line out and getting operators to and from equipment. An additional five minutes per hour would also be lost due to other inefficiencies. For a twelve hour shift this reduces the working time from 720 min down to 605 min and yields an operating efficiency of approximately 84%.

Mechanical availability was estimated at 90% during pre-production, assuming new equipment will be purchased. Availability gradually decreases as the hours on each piece of equipment increase. By Year 7, mechanical availability was assumed to be at 80%.

**Manpower**

Mining operations will require four crews operating on 12 hour rotating shifts. There are several rotating shift schedules that can be utilized by the operation. Because of the distance from the towns of Ely, Eureka, and Duckwater the crews will be transported to the site in company supplied vans.

Mining crew manpower during the peak production years will include a total of 57 equipment operators, 16 maintenance personnel and 12 salaried and support personnel. In addition, two contract personnel will work on an as needed basis for blasthole loading and initiation.

**Blast-Hole Drilling**

Blasthole drilling will be done with track-mounted blasthole drills. The Atlas Copco DM45 was selected for the blasthole drill for this project based on its use in similar sized projects throughout Nevada and the Western United States. A single DM45 will carry out most of the drilling on 20 ft benches, while an Atlas Copco T45 will drill the 10 ft benches in thinner ore zones and is capable of drilling a blast pattern on the 20 ft benches to back up the DM45. Two drills are required to mine variable thickness ore zones and assure that blasthole drilling will meet production requirements. Waste drilling is planned with a 13 ft x 14 ft pattern on the 20 ft bench with 4 ft of subdrilling. The hole diameter will be 6-¾ inch. Drilling will be done with a 6 inch downhole hammer on 5-½ inch drill steel. Ore zones will be drilled with the same equipment on a 14 ft x 14 ft pattern to match the rock characteristics.

All the ore, and 20% of waste drilling, in the Seligman Pit is planned with a 8 ft x 8 ft pattern on the 10 ft bench with 3 ft of subdrilling. The hole diameter will be 4-½ inch. Drilling will be done with a top hammer on 2-3/8 inch drill steel.

**Blasting**

A blasting contractor will be responsible for loading the blastholes and initiating the blasts. The hole loading sequence will start by lowering a one pound booster attached to a non-electric blasting cap down the hole. It is anticipated that the mine will be dry and that Ammonium Nitrate and Fuel Oil (ANFO) will be used as the primary blasting agent. Bulk ammonium nitrate prills will be delivered to an on-site storage silo. A blasthole loading truck will transport the prill to the shot pattern, mix the prill with fuel oil (diesel) and a measured amount of powder will be loaded into each hole. The remaining part of the hole will be filled with drill cuttings or crushed rock (stemming) to control the blast energy and minimize fly rock. Once the holes are loaded, the lead lines to the blasting caps will be tied together with a series of downhole and surface delays to control the blast.

To minimize operational delays, blasting will occur during the lunch break or between shifts.

Initially, the powder factor (pounds of explosives per ton of rock) will be 0.5 for waste and 0.4 for ore. The lower powder factor in ore is to minimize the horizontal movement in ore, for more effective ore control. Once in production, the powder factor will be modified to optimize the secondary crusher throughput and minimize drilling, blasting, loading, and crushing costs after production begins.

In addition to loading the blastholes and initiating the blast, the blasting contractor will supply prill silos, explosive magazines, an ANFO mixing and loading truck, and a skid steer loader to stem the holes. The contractor will also supply inventory control for the blasting agents and supplies, and be responsible for regulatory control of the blasting materials. Cost for these services was included in the economic analysis.

**Loading**

The primary loading unit will be a Caterpillar 6030FS hydraulic shovel or equivalent. The 6030FS is planned to be equipped with the "Heavy Rock" shovel bucket, rated at 19.6 $yd^{3.}$ A hydraulic shovel was selected as the primary loader due to its ability to selectively mine on the bench, its ability to mine harder rock with more breakout force, and its fast cycle time to load trucks.

The ore-waste contacts are flat-lying boundaries that will cross the digging face horizontally. The ore and waste have enough color difference to allow visual discrimination. The digging characteristics of a hydraulic shovel will allow the operator to segregate the ore from the waste on a truck-by-truck basis, minimizing ore loss and dilution. The 6030FS is sized to load a 100 t truck in three passes.

The bucket fill factors for ore and waste were adjusted to assure a minimum three pass loading cycle. Loading was estimated assuming a 95% bucket fill factor for waste and a 100% bucket fill factor for ore. Loading operating parameters are shown in Table 14.10.1.2.

**Table 14.10.1.2: Loader Operating Parameters**

| Caterpillar 6030FS | Capacity ($yd^3$) | Bank Dens. ($ft^3$/t) | Swell Factor | Fill Factor | Bucket Cap. (t) | Cycle Time (min) |
|---|---|---|---|---|---|---|
| Waste | 19.6 | 10.50 | 40% | 95% | 31.78 | 0.47 |
| Mine Ore | 19.6 | 12.34 | 40% | 100% | 33.07 | 0.47 |

The shovel will be supported by a Caterpillar 992K wheel loader with a 15 $yd^3$ bucket. This loader is also sized to match the 100 t haul trucks. The loader will also be used to feed the crusher from stockpiles when ore is not available in the pit. It was assumed that the loader would be used to feed the crusher 30% of the time the shovel was in operation; however, the loader will still be in operation between shifts and on non-mining days.

During times when both loaders are being utilized for pit loading it is assumed that the 6030FS will be operated in the pit with the most mineable ore.

**Hauling**

Haulage will be done with Caterpillar 777G 100 t haul trucks. The loading, hauling, dumping, delays and availability published by Caterpillar were used to determine fleet requirements.

Table 14.10.1.3 shows the fixed haulage times assumed for the loading, spotting and dumping. This table shows the estimated load per truck based on the 6030FS hydraulic shovel loading unit. Trucks are loaded with three cycles of the loading shovel.

**Table 14.10.1.3: Truck Operating Parameters**

| CAT 777G | Capacity (t) | Loaded (t) | Load Time (min) | Spot Time (min) | Dump Time (min) |
|---|---|---|---|---|---|
| Waste | 101.2 | 95.3 | 0.99 | 0.75 | 1.25 |
| Ore | 101.2 | 99.2 | 0.99 | 0.75 | 1.50 |

Source: SRK, 2014

**Major Support Equipment**

Support equipment will include a Caterpillar D9 dozer, a Caterpillar D10 dozer, two Caterpillar 14M motor graders, and a Cat 740 articulated truck with an 8,000 gallon (gal) water tank.

**Equipment Fleet Summary**

The following equipment fleet is proposed for mining. The primary mine equipment fleet is listed in Table 14.10.1.4, and the support mine equipment is listed in Table 14.10.1.5.

**Table 14.10.1.4: Primary Mining Equipment List**

| Equipment Type | Description | Size | Max Number Required |
|---|---|---|---|
| Atlas Copco DM45 | Blast Drill Rig | 540hp, 5 inch to 9 inch hole diameter, up to 175 ft hole depth, 45,000 ft-lb pulldown | 1 |
| Atlas Copco T45 | Blast Drill Rig | 325hp, 3½ inch to 5 inch hole diameter, up to 92 ft hole depth, with a 41 hp rock drill | 1 |
| Caterpillar 6030FS | Hydraulic Shovel | 1,530 hp, 19.6 $yd^3$ | 1 |
| Caterpillar 992K | Wheel Loader | 814 hp, 15 $yd^3$ | 1 |
| Caterpillar 777G | Haul Truck | 1,025 hp, 99.6 t payload | 7 |

Source: SRK, 2014

**Table 14.10.1.5: Support Mining Equipment List**

| Equipment Type | Description | Size/Comment | Max Number Required |
|---|---|---|---|
| Contractor Supplied | ANFO loading truck | | 1 |
| Caterpillar 14M | Motor Grader | 259 hp,14 ft blade | 2 |
| Cat D9T | Bulldozer | 410 hp, 107,000 lb, SEMI-U Blade | 1 |
| Cat D10T | Bulldozer | 580 hp, 155,500 lb, U-blade | 1 |
| Caterpillar 740B | Water Truck | 474 hp, 8,000 gal | 1 |
| Manufacturer TBD | Fuel/Lube Truck | 33,000 lb 6x4 | 1 |
| Manufacturer TBD | Mechanics Truck | 33,000 lb 6x4 | 2 |
| Manufacturer TBD | Light Plant | 30 ft mast | 6 |

Source: SRK, 2014

## 14.11.2 Ancillary Mining Operations

**Site Preparation**

The mine and dumps are located on steep terrain, with little or no topsoil. Where topsoil is thick enough to be recovered, and where slopes are flat enough to operate safely, topsoil will be dozed or hauled to stockpiles where it can be used for future reclamation.

**Drainage Preparation**

Storm water management will occur through the use of cut-off contour drains to control and separate mine-impacted surface water from clean water catchments. It is assumed that 1.5 ft deep V-ditches will be constructed using a bulldozer or motor grader with 1.5:1 side slopes. These will provide adequate capacity to divert water around the waste-rock storage facilities during storm events.

**Snow Removal**

Snow removal will be required on the pit access road and along the pit haul roads, loading areas, drilling bench, and dump areas. The mine support motor grader and dozers will be used for snow removal. Snow removal around the administrative area and crusher will be done by plant operations and support personnel using a motor grader dedicated to the plant area and other support equipment. The capital and operating cost of this equipment is included in the Project’s G&A costs.

# 15 Recovery Methods (Item 17)

Recovery of gold from the Mt. Hamilton Project will be accomplished by a multi-lift heap leach with a carbon ADR plant. The dedicated heap leach pad (leach pad), process ponds, ADR plant and ancillary facilities were designed to accommodate a leachable reserve of approximately 22.5 Mt of crushed ore from the Seligman and Centennial open pits. **The 22.5 Mt is the physical limitation for leach pad construction on MH-LLC's privately held property.** Recently, MH-LLC commissioned a detailed ADR plant design by KCA, of Reno Nevada (KCA, 2014). This design facilitated detailed costing and construction scheduling for the 2014 FS.

## 15.1 Processing Methods - General

Run-of-Mine (RoM) ore will be primary crushed near the southwest rim of the Centennial open pit, and transported to the secondary crushing facility adjacent to the leach pad via a vertical ore pass, and underground conveyor belt. Secondary crushed ore will be transported to the leach pad via overland and portable conveyors, and stacked on the leach pad by a radial stacker.

Table 15.1.1 provides the feasibility design parameters for the heap leach pad.

**Table 15.1.1: Summary of Heap Leach Pad Feasibility Design Parameters**

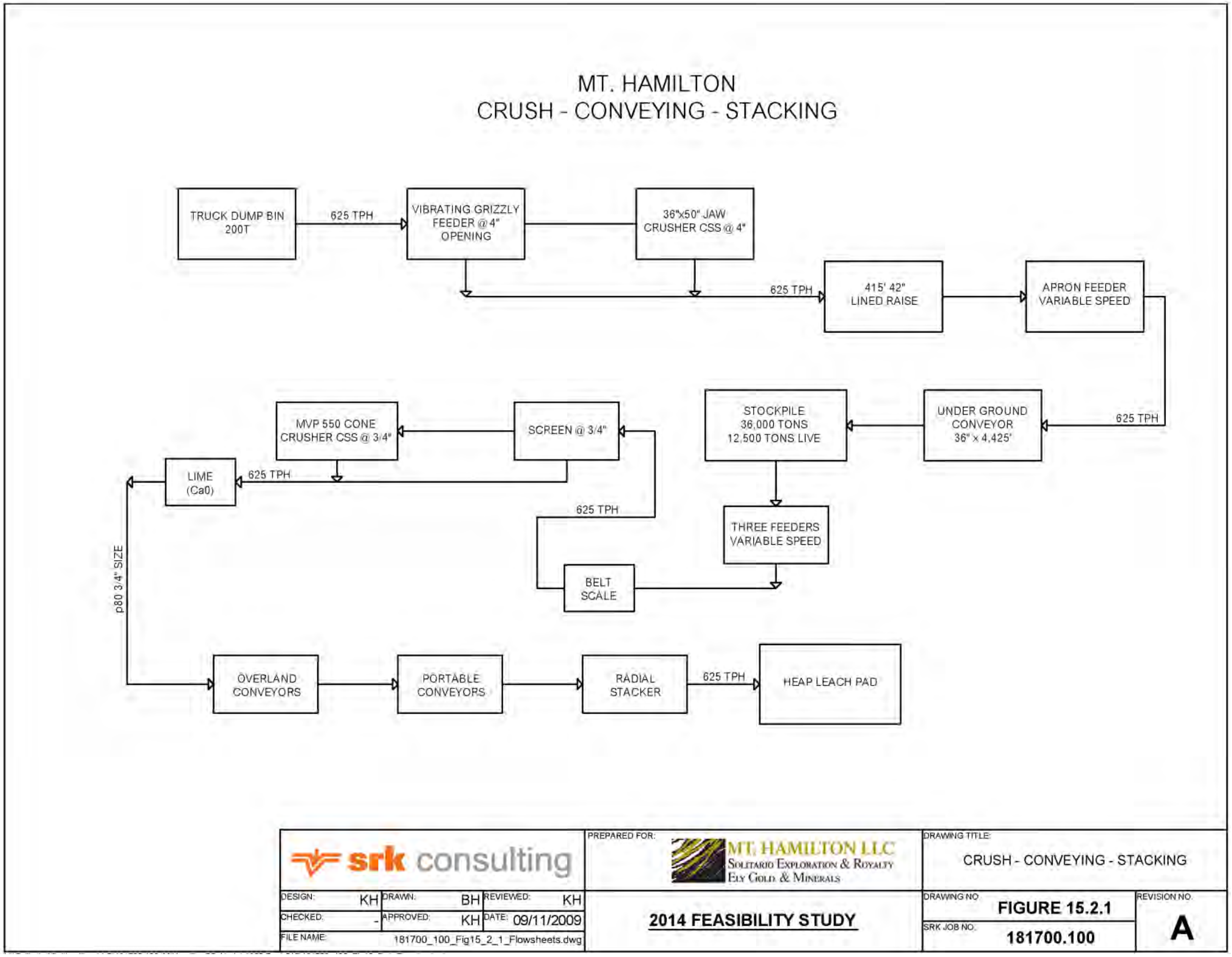
| Design Parameter | Feasibility Design |
|---|---|
| Ore stacking rate | 625 t/h (10,000 t/d) |
| Crushed Ore Bulk Density | 110 lb/ft$^2$ |
| Ore lift height | 30 ft |
| Solution application rate | 0.004 gpm/ft$^2$ |
| Ore leach cycle | 210 days |
| Ore leach area | 4.43 Mft$^2$ |
| Solution pumping rate | 3,240 gpm |
| HLP base slope | 17% upper (east), 13% lower pad (west) |
| HLP maximum height | 210 ft above base |

Source: SRK, 2014

## 15.2 Crushing and Conveying and Stacking

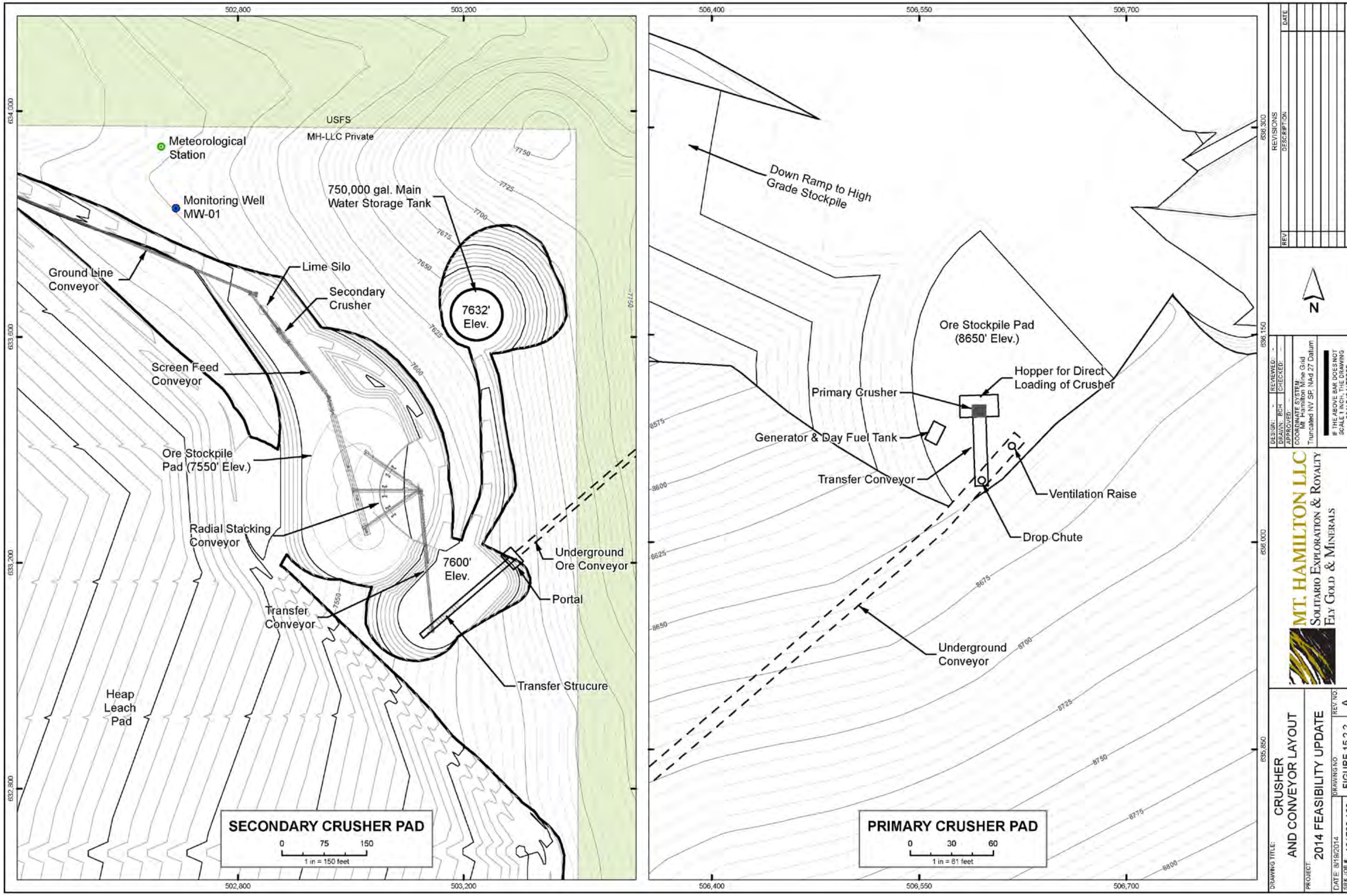
The flow sheet for crushing, conveying and stacking is presented in Figure 15.2.1 and described below. The primary and secondary crusher layout is shown in Figure 15.2.2.

Ores will be crushed in two stages to 90% passing ¾ inch size and conveyor stacked to a maximum height of 210 ft in multiple lifts. Primary crushing will occur on a crushing pad built near the open pit at an elevation of 8,650 ft (amsl). A 415 ft, steel lined, 60 inch diameter vertical raise will transport the crushed ore to a feeder and a conveyor belt. The 4,425 ft long conveyor belt will transport the ore at a grade of approximately -15% to a series of 36 inch conveyor belts, and to a stockpile located near the secondary crusher facility, adjacent to the leach pad. The stockpile will feed a secondary cone crusher plant at an elevation of 7,550 ft (amsl). The secondary cone crusher plant will feed an overland conveyor, a series of portable conveyors and radial stacker to the heap leach pad.




Source: SRK, 2014

**Figure 15.2.1: Crushing, Conveying and Stacking General Flow Diagram**




Source: SRK, 2014

**Figure 15.2.2: Primary and Secondary Crusher Layout**

### 15.2.1 Primary Crushing

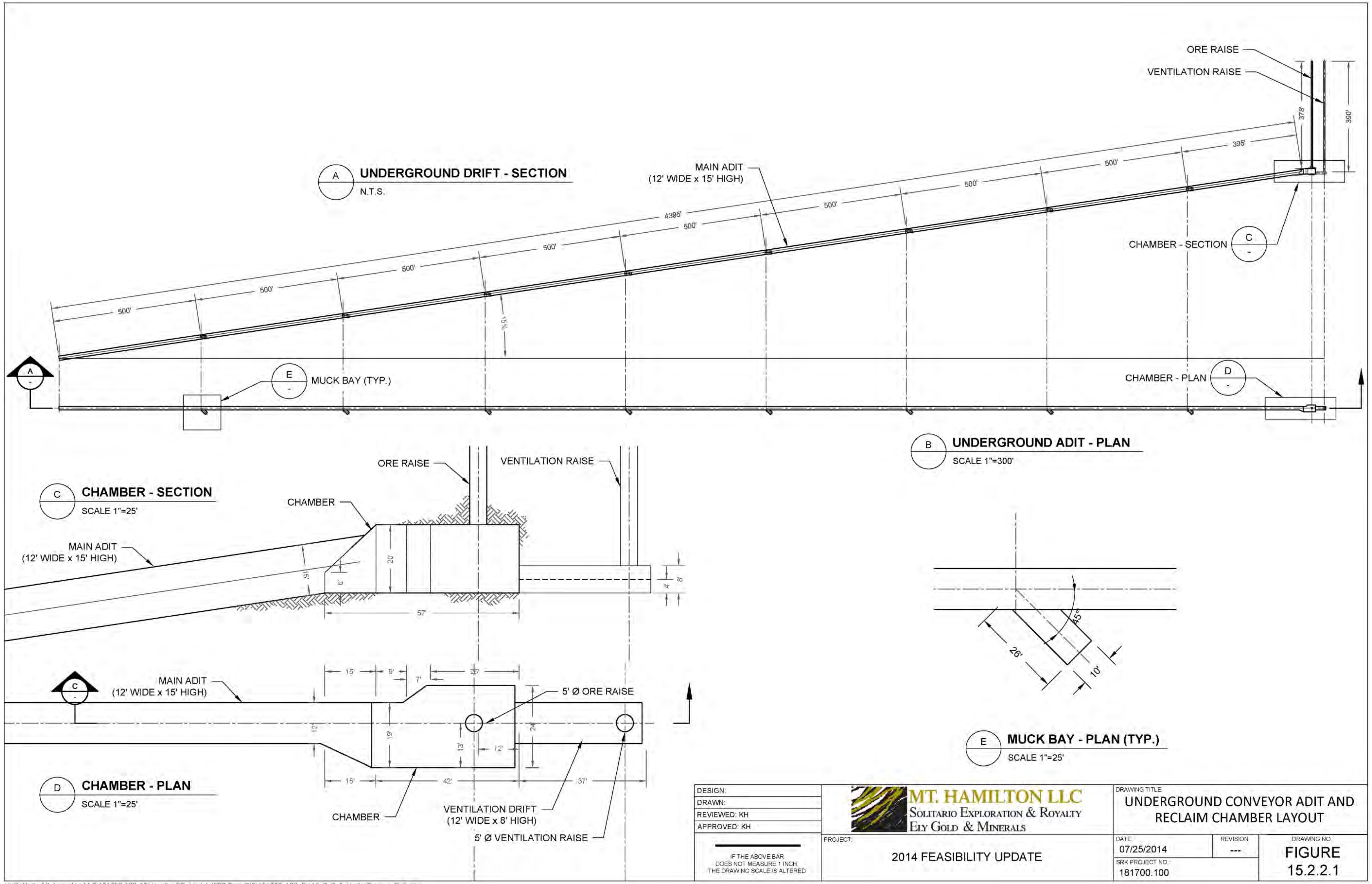
RoM ores will be fed to a 130 t dump bin by 100 t trucks, via front end loader or by direct dumping. From the dump bin, the ores will be fed to a vibrating grizzly feeder with 4 inch openings. Oversize from the grizzly feeder will feed directly to a 36 inch x 50 inch Lippman jaw crusher with a closed side setting of 4 inches. The undersize from the grizzly and the jaw crusher product will be combined and conveyed to a 60 inch steel lined raise. The dump bin will be a free standing structure. The vibrating grizzly and jaw crusher will be mounted on a portable steel frame.

### 15.2.2 Ore Pass, Ventilation Raise and Underground Conveyor

The primary crushed ore will be conveyed from the primary crusher to a 60 inch diameter x 415 ft vertical steel lined ore pass. The raise will have a rock box and a 6 ft long replaceable extension at the top. The bottom will have a hydraulic cut off valve pinned to the back of a 26 ft x 20 ft x 20 ft high underground chamber. Cross section details of the underground reclaim chamber are shown in Figure 15.2.2.1. The cut off valve will feed a replaceable 6 ft section consisting of a chute feeding a 48 inch wide heavy duty apron feeder with a variable speed drive. The apron feeder will feed a 4,425 ft long, 36 inch wide conveyor belt. The conveyor belt will be a channel frame suspended from the back of a 15 ft high x 12 ft wide adit at -15% grade. The drift portal is located at 7,600 ft elevation (amsl). The drive for the conveyor will be a 300-horsepower motor located in the underground chamber. The motor will provide regenerative braking, backed up by a standard friction brake for the decline conveyor. A gravity tower belt take-up system will be located at the portal.

A belt scale will be provided on the conveyor to regulate the variable speed apron feeder.

A second, 60 inch diameter shotcrete-lined raise will be installed a short distance from the ore raise. The second raise will be utilized for power and water lines, ventilation fan and an emergency escape-way. The escape-way will be equipped with a motorized rescue hoist.




Source: SRK, 2014

**Figure 15.2.2.1: Underground Reclaim Chamber**

### 15.2.3 Coarse Ore Stockpile

The ore from the decline conveyor will be conveyed by a 266 ft long, 36 inch wide conveyor belt to a 125 ft long, 36 inch wide radial stacker. The radial stacker will create a stockpile with a 12,500 t live capacity and 38,000 t total capacity. The coarse ore will be reclaimed via three electromechanical vibrating feeders located in a 10 ft diameter tunnel under the stockpile feeding a 248 ft long, 36 inch wide conveyor belt.

### 15.2.4 Secondary Crushing

The secondary crushing plant consists of a 6 ft x 20 ft two-deck screen and a Terex MVP 550 cone crusher. A 125 ft long, 36 inch wide conveyor will feed the screen at 625 TPH. Material from the screen greater than ¾ inch in size will gravity feed to the Terex MVP 550 cone crusher with a closed side setting (CCS) of ¾ inch. The screen undersize, at 100% passing ¾ inch, will be combined with the crusher product to produce a 90% passing ¾ inch size feed to the heap leach pad. A 98 ft long, 36 inch wide conveyor will feed the ore to the overland and heap stacking conveyors. A 150 t capacity lime silo will be placed on the crusher discharge belt to add pebble lime to the ore. The pebble lime addition will be applied by a rotary valve controlled by a belt scale.

A sampling system will be installed on the 98 ft long belt conveyor consisting of a swing arm belt sampler feeding a 1,200 lb capacity bin. The bin will be taken to the assay laboratory on a shift basis.

### 15.2.5 Overland Conveyor and Stacking

A 600 ft and a 965 ft long, 36 inch wide overland conveyor will convey the ore to a heap stacking system. The channel overland conveyor will be mounted on concrete sleepers. The head and tail pulleys will be skid mounted. The overland conveyor can be easily shortened or lengthened as necessary to accommodate the heap stacking system.

The overland conveyor will feed a series of 134 ft long jump (grasshopper) portable conveyors with a working length of 2,144 ft. The jump conveyors will feed a radial stacker. The overall length of the radial stacker is 137 ft, of which 60 ft is in the stinger (telescoping) portion. The stacking height full extended is 41 ft; at the retracted length, the stack height is 25 ft.

The heap will be stacked in 30 ft lifts by the stacker system. The initial slopes of the base of the leach pad are up to 15% grade. The ore must be stacked from the heap base upslope to prevent liner damage. In the initial construction of each phase, the stacking system will be aided by dozer pushing.

## 15.3 Heap Leach Pad Design

Detailed designs for the Mt. Hamilton Project heap leach pad were prepared for a 2013 Water Pollution Control Permit application (SRK, 2013b) and were approved by the State of Nevada NDEP-BMRR following an updated Engineering Design Report submitted in May, 2014. The following is a summary of the approved leach pad design.

The proposed heap leach pad is designed to be constructed in four phases: each phase will consist of five cells for a total of 20 cells, as illustrated by the base grading shown in Figure 15.3.1. Current planning calls for constructing Phase III and IV simultaneously. The leach pad and associated facilities will cover an area of about 135 ac and, together with the crusher pad and drainage facilities, will occupy almost all of the 160 ac of private property upon which they are located. The heap leach pad will be located on moderately sloping and generally uniform topography southwest of the pit. The leach pad is roughly square in plan and extends from an elevation of 7,264 ft amsl at the toe of the process ponds to an elevation of 7,640 ft amsl at the crest of the eastern perimeter road. The lined base receiving ore will range from approximately 13% upslope from the stability berm and toe pad to 17% at the eastern boundary of the heap leach pad. The leach pad will have a total lined area of 4.43 $Mft^2$, or approximately 102 ac. The reclaimed (regraded) final leach pad configuration and cross-sections are shown in Figures 15.3.2, 15.3.3, and 15.3.4.

An average dry density of 110 $lb/ft^3$ (or, 1.5 $t/yd^2$) for stacked ore was used to determine the proper leach pad dimensions to contain the proposed (pad limited) ore reserve of 22.5 Mt.

The topography of the leach pad slopes from east to west with a naturally-occurring drainage approximately located along the longitudinal axis of the leach pad. The stacked ore height will gradually increase as it progresses from west to east until reaching its apex, with a regraded maximum vertical separation of approximately 210 ft above the prepared base. Large column height/percolation tests performed in 2011 confirmed a maximum stacking height of 220 ft without agglomeration using a solution application rate 0.004 $gpm/ft^2$. Therefore, the proposed maximum design height is within tested limits.

## 15.3.1 Pad Construction

Pad construction will include foundation preparation, liner system installation, solution collection piping system installation, placement of overliner material, and the construction of cell and phase divider berms.

**Foundation Preparation**

Prior to developing each phase, the pad and perimeter berm footprint will be cleared and grubbed of existing vegetation and topsoil. Phase I construction will also include clearing, grubbing, and cut-to-fill grading in the areas where the process ponds and ADR plant will be constructed. Topsoil will be removed to a minimum average depth of 1.6 ft from the base of each phase and stockpiled for later use as growth media cover. It is estimated that 332,000 cubic yards of growth media will be required to complete reclamation of the final regraded heap leach pad surface at the end of the project. The growth media stockpile area for all phases will be located in the southern part of the adjacent Admin Parcel, another parcel of MH-LLC private land.

**Liner System**

The leach pad liner system will be a compacted 12 inch-thick low-permeability soil layer overlain by a single geosynthetic liner. The primary liner will be a double-textured (i.e., roughened on both sides) 80 mil high density polyethylene (HDPE) geomembrane liner. Alternately a 60 mil LLDPE liner is being considered for the primary liner. The underliner will consist of a compacted 12 inch-thick layer of either imported low-permeability soil or an admixture of bentonite and native soil to achieve a hydraulic conductivity of $1x10^{-6}$ centimeters per second (cm/sec) or less. If the latter, the low-permeability soil layer will be constructed in place by excavating to a minimum depth of 12 inches, mixing the excavated soil with bentonite at the designated ratio, moisture conditioning, then placing and compacting the mixture to a finished base grade.

The solution channel and process ponds will each be constructed with a double synthetic liner system consisting of an 80 mil HDPE primary liner over a polyethylene geonet, overlying a 60 mil HDPE secondary liner. Each pond liner system will be equipped with a leak collection and recovery system (LCRS).

**Heap Leach Pregnant Solution Recovery System**

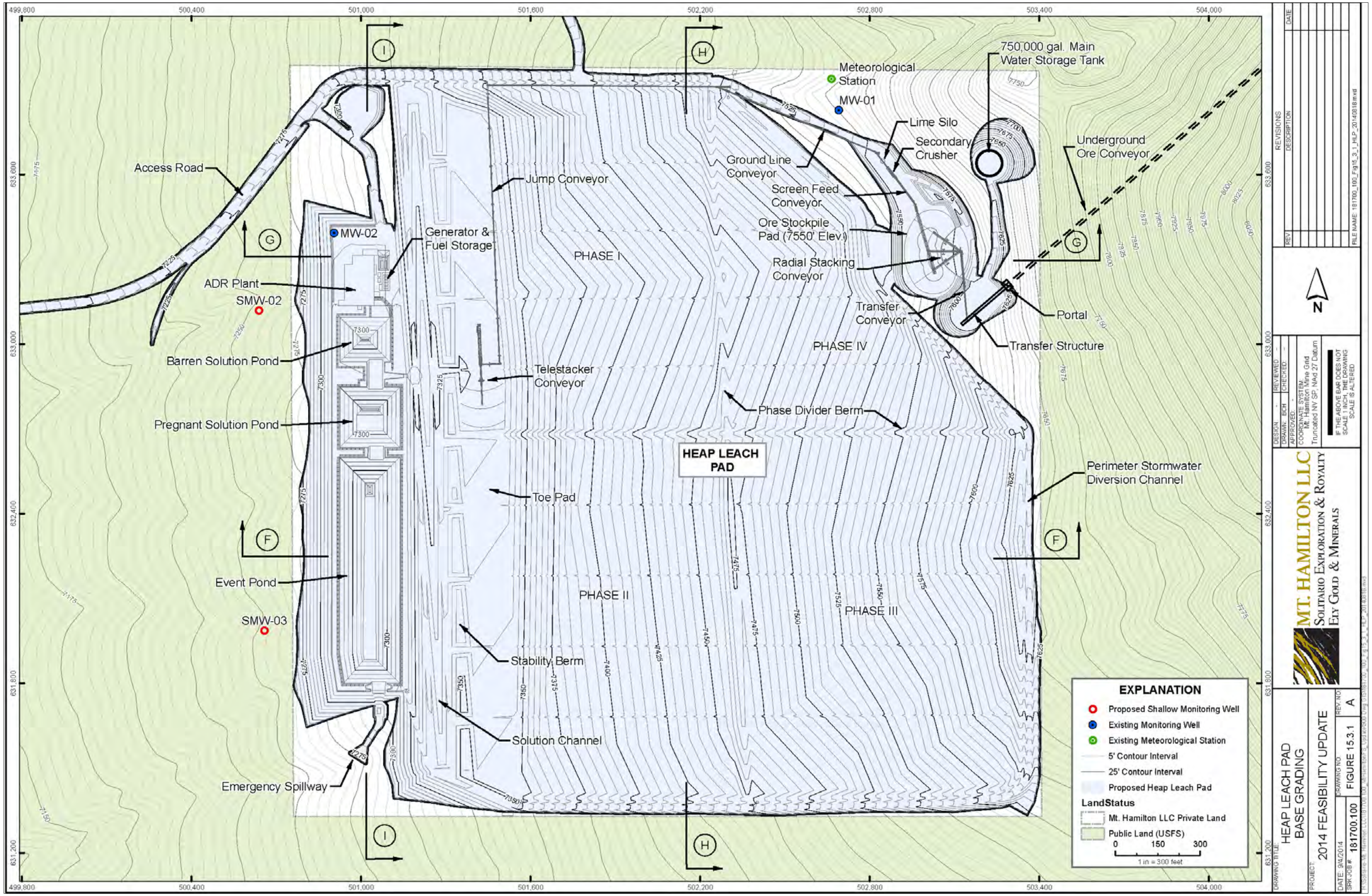
The pregnant solution collection and recovery system will consist of a network of collection pipes designed to collect leach solution and transport it to the process ponds. The pipe network will utilize three different pipe sizes and two types, consisting of 4 inch, 12 inch, and 24 inch diameters and both corrugated, smooth interior, perforated HDPE (also referred to as corrugated polyethylene tubing, or "CPT") and smooth-interior, solid-wall, corrugated HDPE pipe.

**Overliner**

To complete the solution recovery system, a 3 ft thick overliner layer comprised of crushed ore or sized local rock will be applied with the radial stacker and then redistributed with a small, low-ground-pressure dozer over the primary liner and network of collection pipes. This layer will protect the synthetic liner and pipe network during subsequent loading.

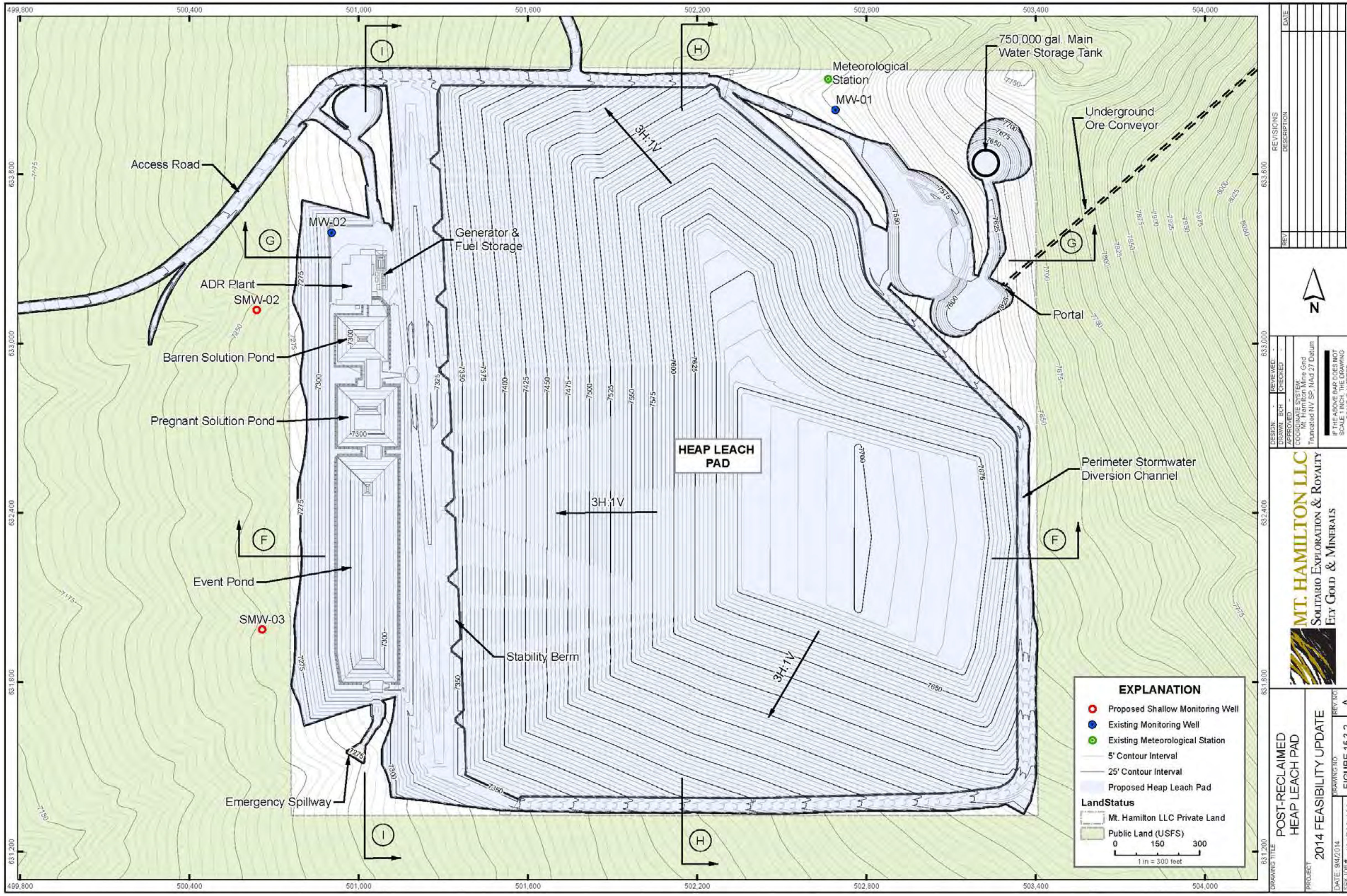
**Solution Channel Leak Collection and Recovery System (LCRS)**

A leak collection and recovery system (LCRS, or "leak detection system") will be installed under the solution channel to monitor and detect leaks if they develop in the liner system. The LCRS will consist of a 4 inch diameter corrugated, smooth-interior, perforated HDPE pipe embedded in drain rock wrapped in an 8 oz/yd$^2$ non-woven geotextile. The perforated pipe and drainage media will be installed in a 20 inch deep v-ditch constructed below the primary liner along the centerline of the solution channel. The LCRS perforated pipe and drainage media will be underlain by the secondary liner, a 60 mil HDPE geomembrane. The geonet that will be installed in between the primary and secondary liner will be extended into the LCRS v-ditch.



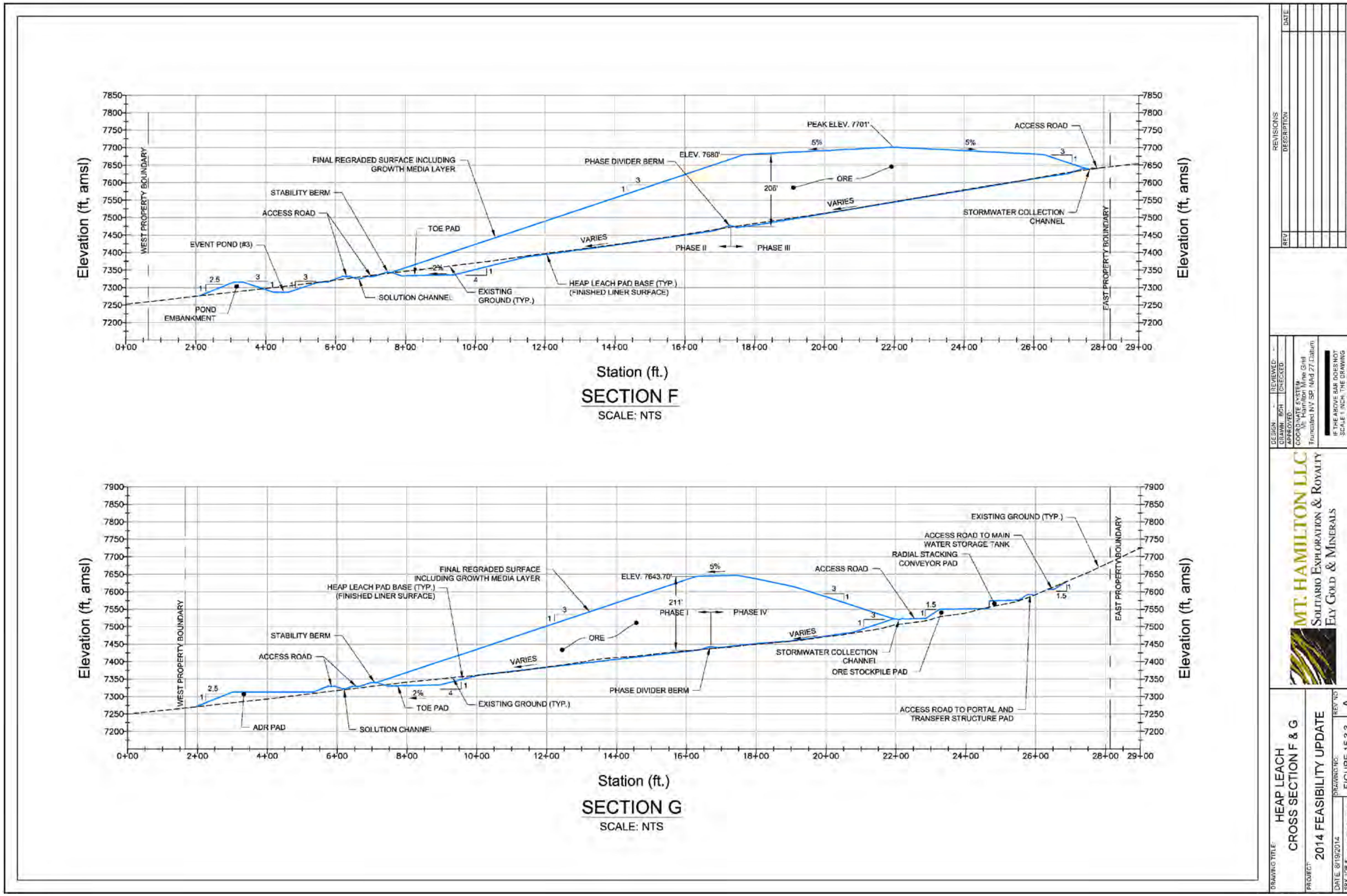

Source: SRK, 2014

**Figure 15.3.1: Heap Leach Pad Site Layout**



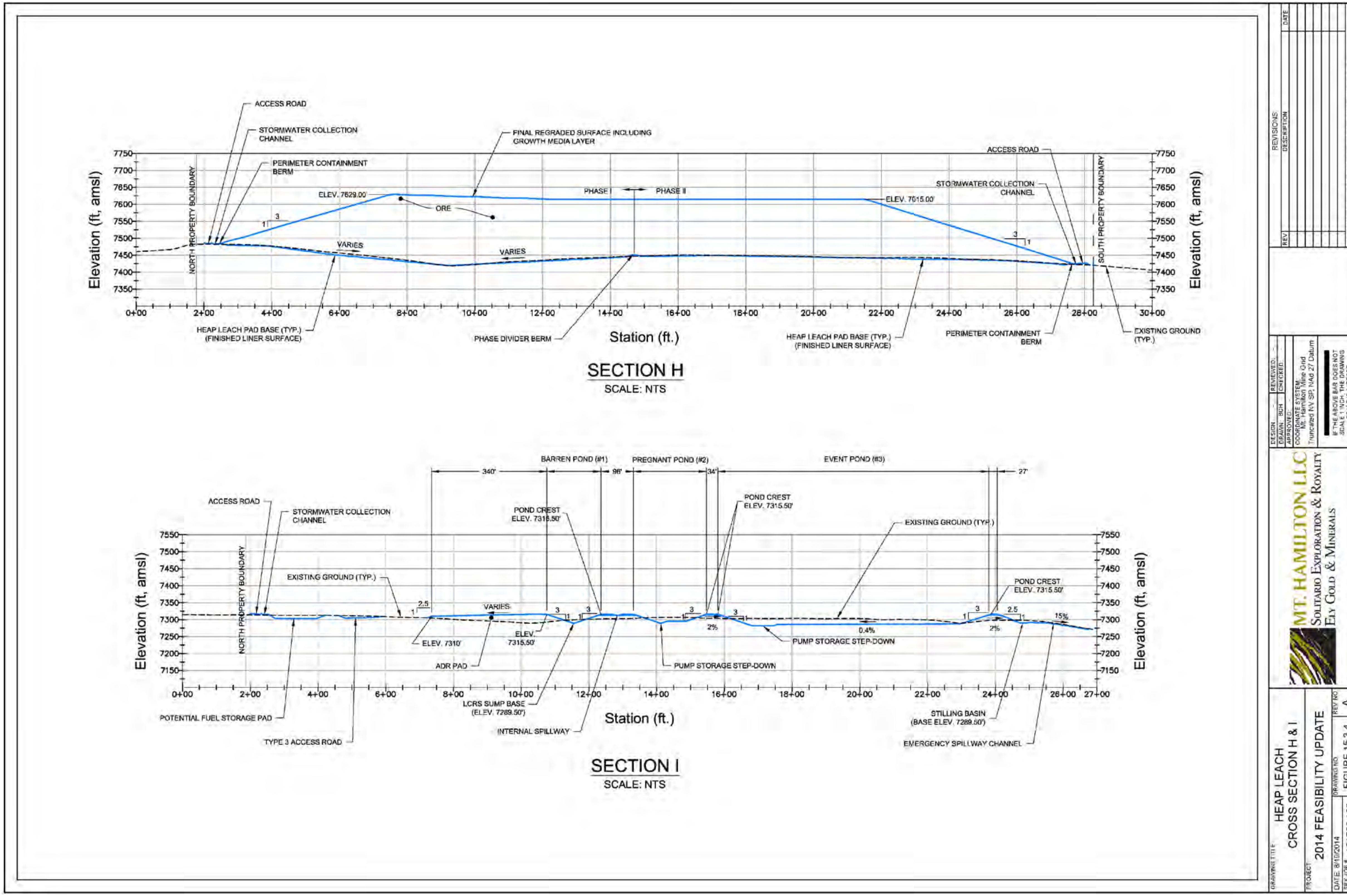

Source: SRK, 2014

**Figure 15.3.2: Heap Leach Pad Post-Reclamation**




Source: SRK, 2014

**Figure 15.3.3: Heap Leach Pad Cross Sections (1 of 2)**




Source: SRK, 2014

**Figure 15.3.4: Heap Leach Pad Cross Sections (2 of 2)**

### 15.3.2 Leach Pad Stability Analysis

**Seismicity**

A seismic hazard analysis was performed for the heap leach pad design using the Probabilistic Seismic Hazard Analysis (PSHA) method. The PSHA method uses a Poisson Probability Model to estimate ground accelerations expressed as a percent chance of exceedance for a given time period, which can also be expressed with a recurrence interval. The probabilistic seismic hazard at the site was obtained from the U.S. Geological Survey Earthquake Hazards Program website (http://earthquake.usgs.gov/hazards/apps/map/), with potential seismic ground motions expressed as a fraction of acceleration due to gravity (g). The peak ground acceleration (PGA) for a 2% probability of exceedance in 50 years was determined to be 0.19 g, which is equivalent to a recurrence interval of 2,475 years or a 0.004 annual rate of exceedance.

**SLIDE Stability Analysis**

Stability analyses were performed on critical slope surfaces of the leach pad using the computer program SLIDE (Version 5.026). SLIDE is a 2-dimensional slope stability analysis program for evaluating the factor of safety, or probability of failure, for circular and non-circular failure surfaces in a defined slope section. SLIDE analyzes the stability of slip surfaces using vertical slice limit equilibrium methods (e.g., Bishop, Janbu, Spencer, etc.). Individual slope surfaces can be analyzed, or random search methods can be applied to locate the critical slip surface for a given slope. Deterministic (safety factor) or probabilistic (probability of failure) analyses can be carried out.

The stability of the west-facing slope of the leach pad was evaluated both for the initial lift of the ore at the stability berm and toe pad during operations and for the full height of the final regraded configuration of the reclaimed heap leach pad at the end of the project. The results for both analyses are presented in Table 15.3.2.1.

**Table 15.3.2.1: Summary of Results for Heap Leach Pad Slope Stability Analyses**

| Sections | Circular Failure | | Noncircular Failure | |
|---|---|---|---|---|
| | Static | Pseudostatic | Static | Pseudostatic |
| Initial Lift | n/a | n/a | 2.19 | 1.30 |
| Final Reclaimed Surface | 1.74 | 1.10 | 1.99 | 1.47 |

Source: SRK 2014

For all analyses, the factors of safety (FoS) under static condition and pseudostatic conditions are higher than the required minimum FoS of 1.3 and 1.05, respectively. Therefore, the proposed heap leach pad will be stable under both static and pseudostatic conditions for both the initial lift and final ore grading configurations.

### 15.3.3 Stormwater Diversion Design

The heap leach pad will require the construction of an upgradient stormwater diversion channel to divert potential drainage of stormwater onto the leach pad. The proposed diversion channel will be located on the upslope side of the eastern property boundary, as shown on Figure 15.3.1.

A temporary stormwater diversion channel will be constructed on the leach pad parcel to prevent run-on from entering or impacting the process facilities during Phases I and II of leach pad development. During base construction of the Phase III heap leach pad, a permanent diversion channel will be constructed on the upgradient side of the perimeter access road to prevent run-on from impacted the final heap configuration during the later phases of leach pad development and through the closure and post-closure periods. In addition, culverts and diversion ditches may be placed in and around the process facilities as necessary for further stormwater control. During operations, stormwater runoff from the heap will be captured by the solution collection system, channeled to the process ponds, and incorporated in the process circuit.

### 15.3.4 Process Pond Design and Storage Requirements

Three solution ponds will be required for the Mt. Hamilton Project, a pregnant solution pond, a barren solution pond, and an event pond. Each pond will be double-lined and equipped with a leak collection and recovery system.

**Process Pond Design Criteria and Storage Requirements**

The design criteria for process pond operational water management are:

- To remain fully functional and fully contain all process fluids including all accumulations resulting from a 25-year, 24-hour storm event; and
- To withstand the runoff from the 100-year, 24-hour storm event falling on the HLP and other processes watersheds contributing to the pond inventory.

The design of the process ponds considers the following design parameters:

- Pond interior sideslope angle: 3H:1V
- Pond pump draft depth (dead storage): 4 ft above pond base;
- 12 hours minimum available operating inventory;
- A backup generator will be provided for use during primary power outages;
- Storm storage: contain the 25-year, 24-hour storm runoff from all process facilities in the pregnant and barren solution ponds and the event pond;
- Minimum dry freeboard is 2 ft; and,
- Liner System: 80 mil HDPE primary liner, 60 mil HDPE secondary liner, and LCRS

To accommodate the volume of rainfall that falls on the leach pad and process ponds during mine operation, it was assumed that the entire 25-year, 24-hour storm depth (2.87 inches) will report to the process ponds. Rainfall on the leach pad will enter the solution process either as infiltration through the heap, or as surface runoff into the channel formed between the perimeter berm and toe of the heap. The volume for each storage component of the pregnant solution pond, the barren solution pond, and the event pond are summarized in Table 15.3.4.1.

**Table 15.3.4.1: Process Pond Storage Characteristics**

| Storage Requirement | Pregnant Solution Pond Volume | | Barren Solution Pond Volume | | Event Pond Volume | |
|---|---|---|---|---|---|---|
| | (cf) | (gal) | (cf) | (gal) | (cf) | (gal) |
| 16-hour Operating Volume | 385,000 | 2,880,000 | 77,000 | 576,000 | n/a | n/a |
| 25-Year, 24-Hour Storm Volume | n/a | n/a | n/a | n/a | 1,189,700 | 8,899,900 |
| Flexible Volume | 33,200 | 248,500 | 146,400 | 1,095,000 | 1,160,300 | 8,680,100 |
| Volume Above Internal Spillway | 33,900 | 253,700 | 22,000 | 164,400 | n/a | n/a |
| Dry Freeboard Volume | 100,400 | 751,000 | 66,700 | 498,900 | 347,200 | 2,597,400 |
| Dead Storage (Pump Draft) | 9,900 | 73,900 | 5,900 | 44,500 | 12,400 | 93,100 |
| Sum of Component Volumes | 562,400 | 4,207,100 | 318,000 | 2,378,800 | 2,709,600 | 20,270,500 |

Source: SRK, 2014

The above table demonstrates that the total volume of each pond is greater than the sum of component volumes required for each pond, and thus the ponds are adequately sized for the design criteria described above.

### 15.3.5 Process Pond Construction

The process ponds will be constructed as part of the Phase I leach pad construction and will include foundation preparation, leak collection and recovery system (LCRS) installation, and a double-containment liner system. The process pond footprint will be cleared and grubbed of existing vegetation and topsoil will be placed in the growth media stockpile. Pond geometries are shown in Figure 15.3.1.

A double synthetic liner system is proposed for both the solution ponds and the event pond. The system will consist of an 80 mil HDPE primary liner placed over a polyethylene geonet, overlying a 60 mil HDPE secondary liner.

The pregnant and barren ponds and the event pond are each designed with a LCRS sump, which is located at the center of the pregnant pond and at the north side of the barren solution pond and the event pond. Each sump will include drainage gravel placed 2 ft deep and wrapped in an 8 oz/yd$^2$ nonwoven geotextile. Each sump will be underlain by the secondary 60 mil HDPE liner and overlain by the geonet and primary 80 mil HDPE liner. The soil beneath the secondary liner will be amended as necessary and compacted to create a 2 ft-thick layer with a maximum permeability of $1x10^{-7}$ cm/s. An HDPE riser pipe will be located between the primary and secondary liners and extend to the pond crest to allow leak detection monitoring and removal of solution collected in the sump.

## 15.4 Leach Solution Application

Solution applied to the stacked ore (heap) on the leach pad will be distributed from the barren solution tank by two centrifugal pumps operating in parallel, with make-up water provided to the circuit from the barren solution pond via a submerged pump, and a 14 inch-diameter steel distribution header at the heap base. Every 240 ft on the header, at cell dividers, there will be a reducer and valve followed by 8 inch-diameter HDPE piping to the heap. The 8 inch-diameter HDPE piping will connect to 4 inch-diameter HDPE pipe at approximate 350 ft intervals. The 4 inch diameter HDPE pipe will be drilled and tapped on both sides to accommodate solution distribution through 175 ft-long emitter lines extended over the ore surface. The pregnant leach application solution flow will be up to 3,240 gpm.

The leach application rate is 0.004 gpm/sf for 90 days and the pregnant leach solution flow collected from the base of the leach pad will be an estimated 3,000 gpm.

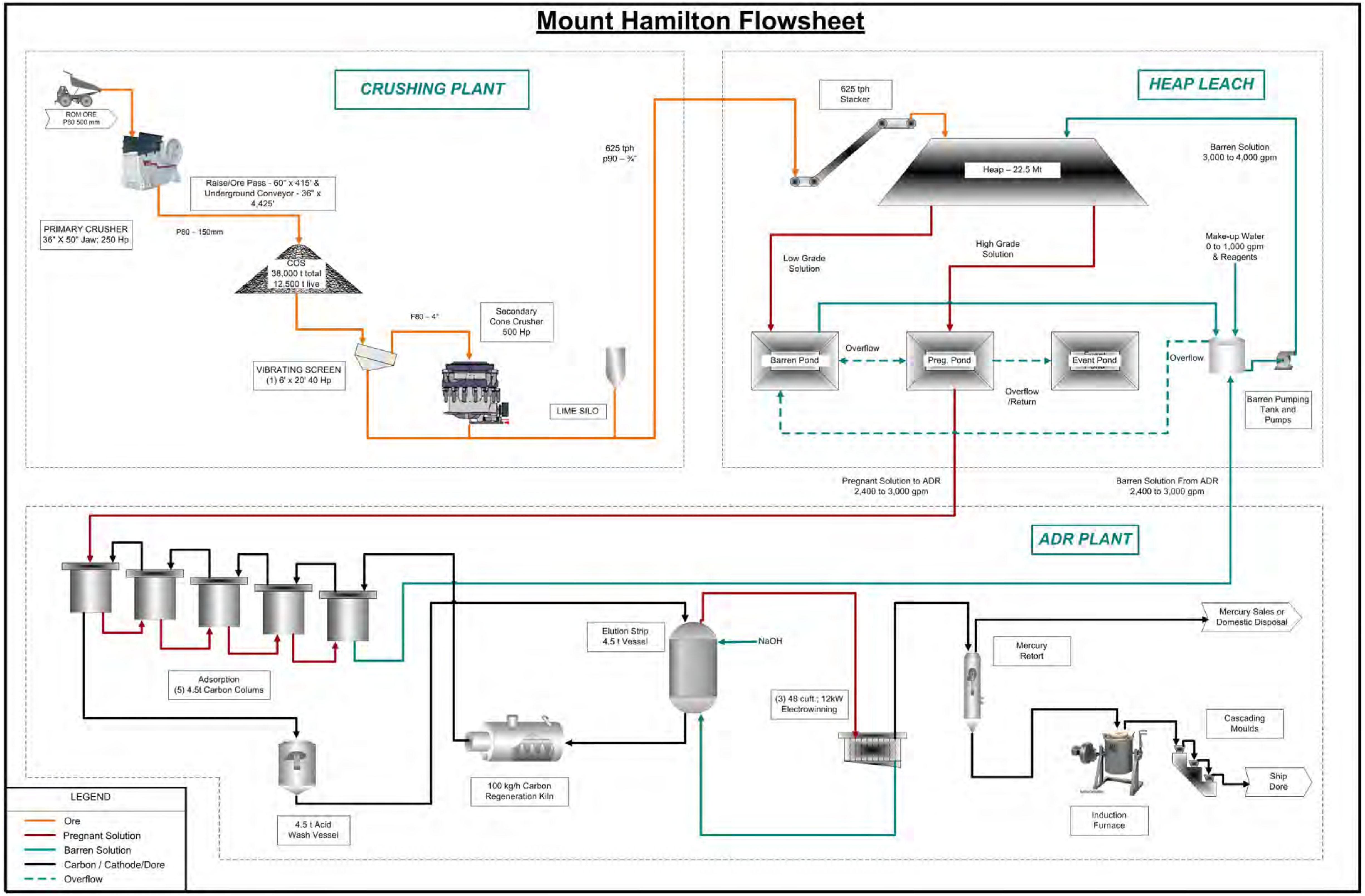
The emitter lines will be buried from October to March to prevent freezing. A filter will be installed on the barren solution piping to prevent emitter clogging. The barren solution pump will have a variable frequency drive and will be capable of providing 3,240 gpm flow to the heap for the initial two phases of pad operation. Pump replacement or the installation of supplemental booster pumps will be required to maintain flows to the ultimate heap leach height.

## 15.5 Plant Design and Operations

### 15.5.1 ADR Plant Design

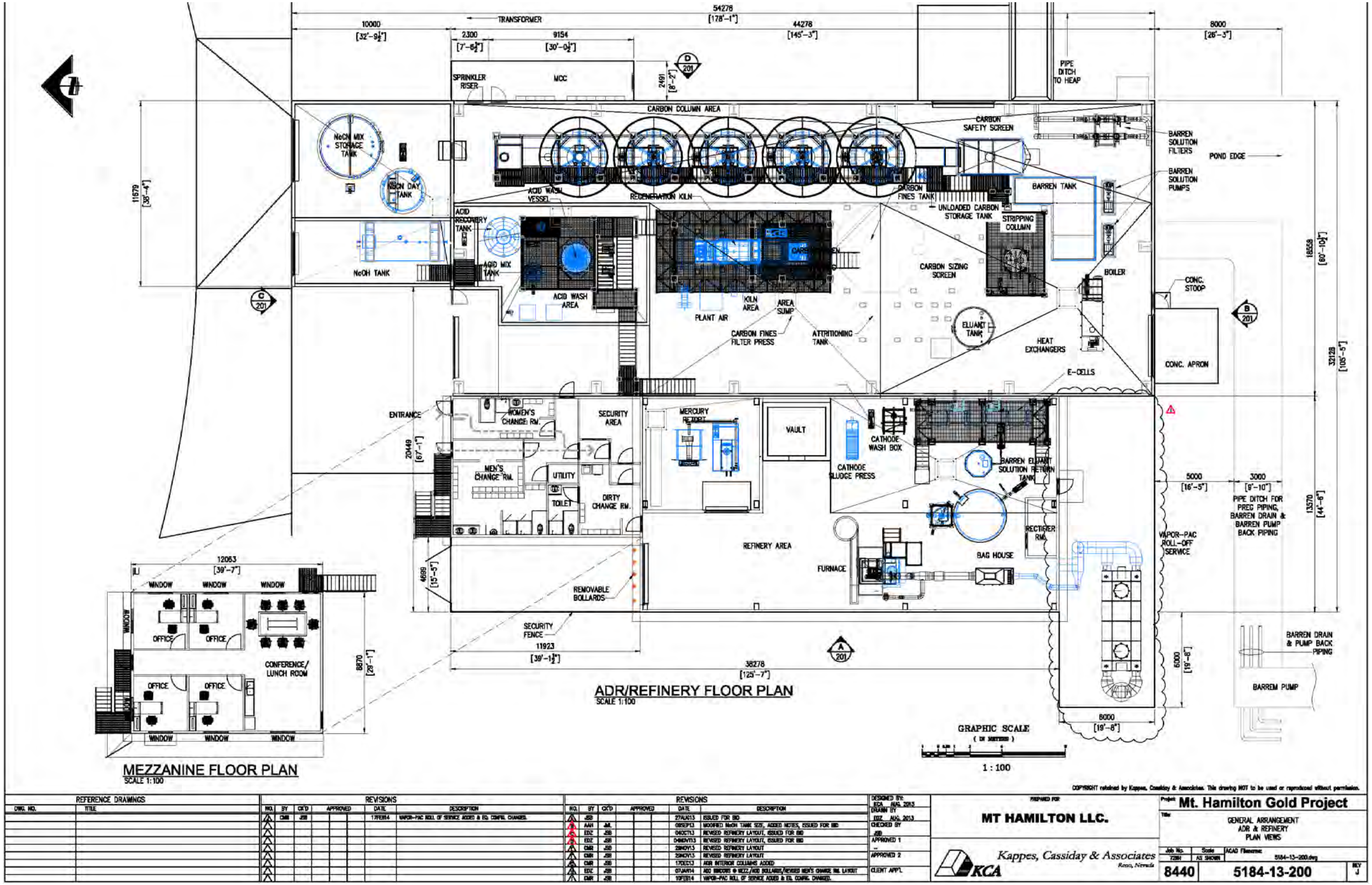
A carbon ADR circuit will be used at the Mt. Hamilton Project. The ADR plant will have all the mercury controls installed as currently required by the State of Nevada. The KCA plant design criteria and detailed design are incorporated in the 2014 FS (KCA, 2014). The following is a summary of the plant design.

The ADR plant consists of five, 12 ft diameter carbon columns, a 4.5 t strip and acid wash system, electrolytic cells, mercury retort and mercury controls and an induction smelting furnace. The final product will be a doré bar. Electrolytic cells of the ADR plant have been sized to accommodate Ag/Au ratios of 8.7/1 in the final doré. The flow sheet for the heap leach doré recovery is presented in Figure 15.5.1.1. The KCA plant layout details are presented in Figure 15.5.1.2.




Source: SRK, 2014

**Figure 15.5.1.1: Mt. Hamilton Process Flow Sheet**




Source: KCA,2014

**Figure 15.5.1.2: ADR Plant Layout**

### 15.5.2 ADR Plant Operations

The ADR plant will be fed at the rate of 3,000 gpm by submersible pumps in the pregnant pond. The pregnant solution will flow over a trash screen and then to a cascading series of 12 ft diameter carbon columns. The barren solution from the column series will flow over a safety screen, into the barren pumping tank and then pumped back to the heap. The purpose of the safety screen is to remove occasional carbon “floaters.” The activated carbon will be transferred countercurrent to the solution flow in 4.5 t lots by a recessed impeller pump. The countercurrent flow of carbon allows the carbon to become fully loaded in the initial tank of the series, while providing a barren solution discharge from the last tank.

The loaded carbon will be transferred to a 4.5 t acid wash vessel. After acid washing and neutralization, the 4.5 t lot will be stripped of doré in a pressure vessel. The stripped carbon will be regenerated by heating in a kiln to remove oil and grease. The regenerated carbon will be quenched and screened and returned to the last carbon column. The regeneration kiln is rated at 2.64 t/d; the strip circuit is rated at 4.5 t/d. The excess carbon will report directly to the screening and then to the last carbon column.

New carbon will be wetted and screened prior to being added to the last carbon column. Fines from the screening operation will be collected in a filter press.

The carbon will be “stripped” of doré values in a 4.5 t capacity pressure vessel at 340º F. Sodium hydroxide will be added to the stripping solution to aid stripping and provide electrolyte for the subsequent electro-winning. The solution will be heated to 280º F by a propane fired hot water heater and heat exchangers. The strip solution will flow to an insulated holding tank. The stripping cycle will be 6-12 hours at 83 gpm.

Solution from the insulated holding tank will be pumped to three sludging electrolytic cells. The barren solution from the electrolytic cells will be pumped back to the insulated holding tank. The sludge from the electrolytic cells will be pumped to a filter press. The damp filter cake will be manually loaded into trays. The trays will be placed in a 6 $ft^3$ mercury retort. After the 24 hour retorting process, the trays will be cooled, dumped and the sludge mixed with fluxes. The retorted sludge/flux mix will be charged to an electric induction furnace for smelting into doré bars.

The ADR building will be a multi sectional building with the main section (ADR) approximately 145 ft long x 61 ft wide x 45 ft eave height pre-engineered steel structure. An additional pre-engineered section 38 ft x 33 ft x 20 ft high for the cyanide area will be attached to the ADR section. The refinery (44.5 ft x 86 ft x 25 ft high) will share a wall with the ADR building. The refinery will be constructed of CMU walls with a lightweight concrete roof. The 45 ft eave portion will contain the cascading carbon columns and screens, the regeneration kiln and carbon handling system, the acid wash and stripping vessel, the strip heating system and insulated holding tank. The secure area will contain the electrolytic cells, mercury retort, flux mixing, slag granulation, and the induction melting furnace. The mercury retort will be contained in 23 ft x 23 ft enclosed area. An adjacent 20 ft x 64 ft curbed concrete slab will contain the dust collectors, mercury controls, exhaust fans and furnace chillers for the secure area.

The refinery area contains space for a 10 ft x 12 ft modular vault. A 15 ft x 39 ft concrete slab with a 10 ft cyclone fence and lockable gates will be constructed adjacent to the refinery main door. This area will allow materials to move in and out of the refinery area without compromising security.

Security cameras will be installed at strategic locations, connected to remote monitors and DVD recorders

A 40 ft x 39 ft dual level office/facilities complex will be connected to the refinery and ADR building. The building will contain women's and men's toilet and change room facilities, lunch/conference room, offices, and a security area

### 15.5.3 Assay Laboratory

An Assay Laboratory capable of performing 270 wet atomic adsorption analyses and 42 fire assay analyses per day will be installed at the office complex. The assay laboratory will be housed in a 60 ft x 40 ft x 14 ft eave height pre-engineered steel building.

The building will contain an office and sanitary facilities. The sample preparation will have drying ovens, crushing and pulverizing and splitting equipment for up to 204 samples per day. The sample preparation area will have a dedicated ventilation system for dust control. The fire assay section will have two large electric furnaces for fusion and one smaller furnace for cupellation. The fire assay section will have a dedicated ventilation system. The AA section will have hot plates, centrifuges and an acid fuming hood. A four-element AA machine will be installed.

The building will contain space and equipment for a metallurgical laboratory. The metallurgical laboratory will have wet and dry screen sizing equipment, bottle rolling equipment, filtering equipment and equipment for up to four column tests.

The ADR plant will have an identical four-element AA machine for routine plant and heap solution assays.

The assay laboratory work schedule is five, ten hour days. Fire Assaying will be done five days per week, AA analysis and sample preparation will work six days per week. The assay laboratory will be staffed to provide five, ten hour days for the personnel.

The heap leach feed shift sample will be sampled with a Harrison Cooper Sampling System. The sampling system will consist of a primary sampler, a jaw crusher reducing the particle size to1/4 inch size and secondary sampler. The sampling system will be located at the secondary crusher discharge conveyor. A single 40 lb sample will be delivered to the laboratory once a shift for additional preparation and analysis.

## 15.6 Consumable Requirements

### 15.6.1 Power

Power for the secondary crushing system, conveying and heap stacking, ADR plant and heap pumps, office complex will be initially provided by four, 725-Kilowatt Cat® generators, operating at 480 volts. The generators will have an automatic paralleling system to start and stop the generators according to load demand. The maximum demand will require three generators on line, leaving a spare generator for service.

The generators will be housed in a three-sided 40 ft long x 20 ft wide x 16 ft high eave, pre-engineered steel structure. The switchgear and controls will be housed in an attached 20 ft x 10 ft x 12 ft high eave, fully enclosed space.

### 15.6.2 Water Supply

The peak make-up water requirement for the Project is 500 gpm. The water source for the Project will be an existing well located at the mouth of the Seligman Canyon, a distance of 11,000 ft from an 80 ft diameter x 20 ft high water storage tank. The well will be equipped with a submersible pump, pumping to an enclosed tank and booster pump. The system is designed for a peak flow of 500 gpm, and consistent delivery of 400 gpm. The booster pump will pump to the 80 ft diameter x 20 ft high, 750,000 gal tank located above the Heap/ADR site at an elevation of 7,614 ft (amsl). Power for the well and booster pump will be provided by an overhead 4,160 volt, 13.8 kV or 14.4 kV power line. The power source will be the four, 725 kW generators.

One or two additional water wells will be constructed nearer to the ADR plant for backup or to provide primary water supply. Exploration drilling for the necessary water source and development of this source will occur in Year -1.

### 15.6.3 Major Reagents

Major reagents and usage for the leach operation are provided in Table 15.6.3.1. The reagent amounts were determined during metallurgical test work performed by McClelland Laboratories in 2011 and summarized in Section 11 of this report.

**Table 15.6.3.1: Major Reagent Consumption**

| Reagent | Use |
|---|---:|
| Lime (CaO) | 4.0 lb/t |
| Sodium Cyanide | 0.6 lb/t |

Source: SRK, 2014

### 15.6.4 Labor Requirements

Labor requirements are divided into two sets: 1) 24 hours per seven day week, and 2) 10 hours per five day week schedules. Labor in each category is listed in Table 15.6.4.1 and 15.6.4.2. The total processing plant and assay laboratory labor requirement is 50 workers.

**Table 15.6.4.1: 24 Hours per 7 Day Week Scheduled Labor**

| 24 Hours per 7 Day Schedule | Per Shift | Total |
|---|---:|---:|
| Primary Crush | 1 | 4 |
| Convey/Stockpile | 1 | 4 |
| Secondary Crush to Overland | 1 | 4 |
| Overland to Stack | 1 | 4 |
| ADR | 2 | 8 |
| Utility | 2 | 8 |
| **Totals** | **8** | **32** |

Source: SRK, 2014

**Table 15.6.4.2: 10 hours per 5 day Week Scheduled Labor**

| 10 Hours per 5 Day Schedule - Day Shift | Per Shift | Total |
|---|---:|---:|
| Laboratory | 7 | 7 |
| Leach Pad Pipers/Utility | 5 | 5 |
| Refiner | 1 | 1 |
| Maintenance | 5 | 5 |
| **Totals** | **18** | **18** |

Source: SRK, 2014

## 15.7 Process Equipment Requirements

Table 15.7.1 lists the major process equipment items identified along with the number of units required and specifications. These items form the basis for process capital cost estimation.

### Table 15.7.1: Major Process Equipment Items Specifications and Quantities

| Equipment Description | Size | Max Required |
|---|---|---|
| **Primary Crusher Area** | | |
| Rock Box | 130 t live load | 1 |
| Lipman J3650 Portable Jaw Crushing Plant | 36 inch x 50 inch jaw crusher, 250 hp, 51 inch wide x 22 ft long vibrating grizzly feeder, on steel truck frame | 1 |
| NPK Pedestal Breaker system | 2,000 ft-lb, 50 hp | 1 |
| C1-Jaw Transfer conveyor | 119 ft long, 60 inch belt, 25 hp, w/ tramp iron magnet | 1 |
| AES Control van | 8 ft x 6 ft | 1 |
| **Underground Equipment** | | |
| Universal FL4 Chain Apron Feeder | 48 inch wide x 12 ft long, 15 hp variable speed drive | 1 |
| C2-Feed Tunnel Conveyor | 4,615 ft long, 36 inch belt, 300 hp | 1 |
| **Secondary Crusher (Drift to Leach Pad )** | | |
| C3 Radial Stacker Feed Conveyor | 266 ft long, 36 inch belt, 15 hp | 1 |
| C4 Radial Stacker | 125 ft long, 36 inch belt, 40 hp | 1 |
| C5 Stockpile Reclaim Conveyor | 248 ft long, 36 inch belt, 30 hp, w/3 vibro-mechanical feeders rated at 500 t/h | 1 |
| C6 Screen Feed Conveyor | 125 ft long, 36 inch belt, 25 hp | 1 |
| Fabtec Portable MVP 550 Cone Plant | MVP 550 Cone crusher, 500 hp, 6 ft x 20 ft, 2 deck 40 hp feed screen, on steel truck frame | 1 |
| Control Van w/ Operators Module | 8 ft x 6 ft | 1 |
| Lime Storage Silo | | 1 |
| C7 Crusher Discharge Conveyor | 98 ft long, 36 inch belt, 30 hp | 1 |
| C9 Ground Line Conveyor | 600 ft long, 36 inch belt, 20 hp | 1 |
| C10 Ground Line conveyor | 965 ft long, 36 inch belt, 75 hp | 1 |
| Leach pad Conveyors | | |
| C11 Jump Conveyor | 50 ft long, 36 inch belt, 10 hp | 1 |
| C12-1 to 16 "grasshopper" Conveyors | 134 ft long, 36 inch belt, 20 hp | 16 |
| C13 Cross Feed Conveyor | 67 ft long, 36 inch belt, 55 hp | 1 |
| C14 HIC Conveyor | 137 ft long, 36 inch belt, 55 hp | 1 |
| C15 Telestacker Conveyor | 136 ft long, 36 inch belt, 55 hp | 1 |
| **ADR Plant** | | |
| CIC Circuit | 5 each - 12 ft dia. columns 4.5T carbon | 1 |
| Acid Wash System | 4.5 t acid wash vessel w/pumps, tanks and controls | 1 |
| Strip System | 4.5 t carbon strip system w/ pumps, tanks and controls | 1 |
| Solution Heater | Propane Hot Water Heater, with heat exchangers and controls | 1 |
| Electrowinning | 3 each - 48 $ft^3$ cells, 12 kW rectifier, sludge filter, w/ tanks, pumps, and controls | 1 |
| Carbon Handling System | Tanks, pumps, filter and controls | 1 |
| Carbon Regeneration | 2.6t/d kiln, electric, w/ hoppers, tanks screens and pumps | 1 |
| Refining | Electric induction furnace, flux and slag handling, molds, balances, jaw and roll crushers, screen and concentrating table | 1 |
| Mercury Removal System | Scrubbers, Mercury Retort | 1 |
| Barren Pumps to Heap | 2 @ 1620 gpm @ 350 ft TDH, 200 hp | 1 |
| Submersible Pump | 2 @ 1500 gpm @ 147 ft TDH, 75 hp | 3 |
| **Process Mobile Equipment** | | |
| Caterpillar 262D Skid Steer loader | 74 hp w/ bucket, cab, A/C | 1 |
| Bobcat S650 Skid steer | 74 hp, w/ bucket, pallet forks, cab, A/C, underground package | 1 |
| Kubota MX5100 Maintenance Tractor | 52 hp, underground package | 1 |
| Pallet Jack | Battery powered, 4,400 lb capacity | 1 |
| Caterpillar D7E Dozer | 235 hp, 56,670 lb, standard blade | 1 |
| Caterpillar 420E IT Backhoe | 102 hp, 1.3 $yd^3$ loader bucket, backhoe | 1 |
| Caterpillar TL1055 Telehandler | 142 hp, 10,000 lb capacity, 55 ft lift height | 1 |
| Trailer mounted Compressor | 78 hp, 185 cfm @ 100 psi | 1 |
| Pipe trailer | 3 axle, 43 ft bed | 1 |
| Emitter Plow | 4 gang plow | 1 |
| Flatbed truck | 2 t | 1 |
| Mechanic Service Truck | TBD | 1 |
| McElroy 412 pipe fusion machine | 18 hp, HDPE Pipe fusion from 4 inch to 12 inch pipe | 1 |

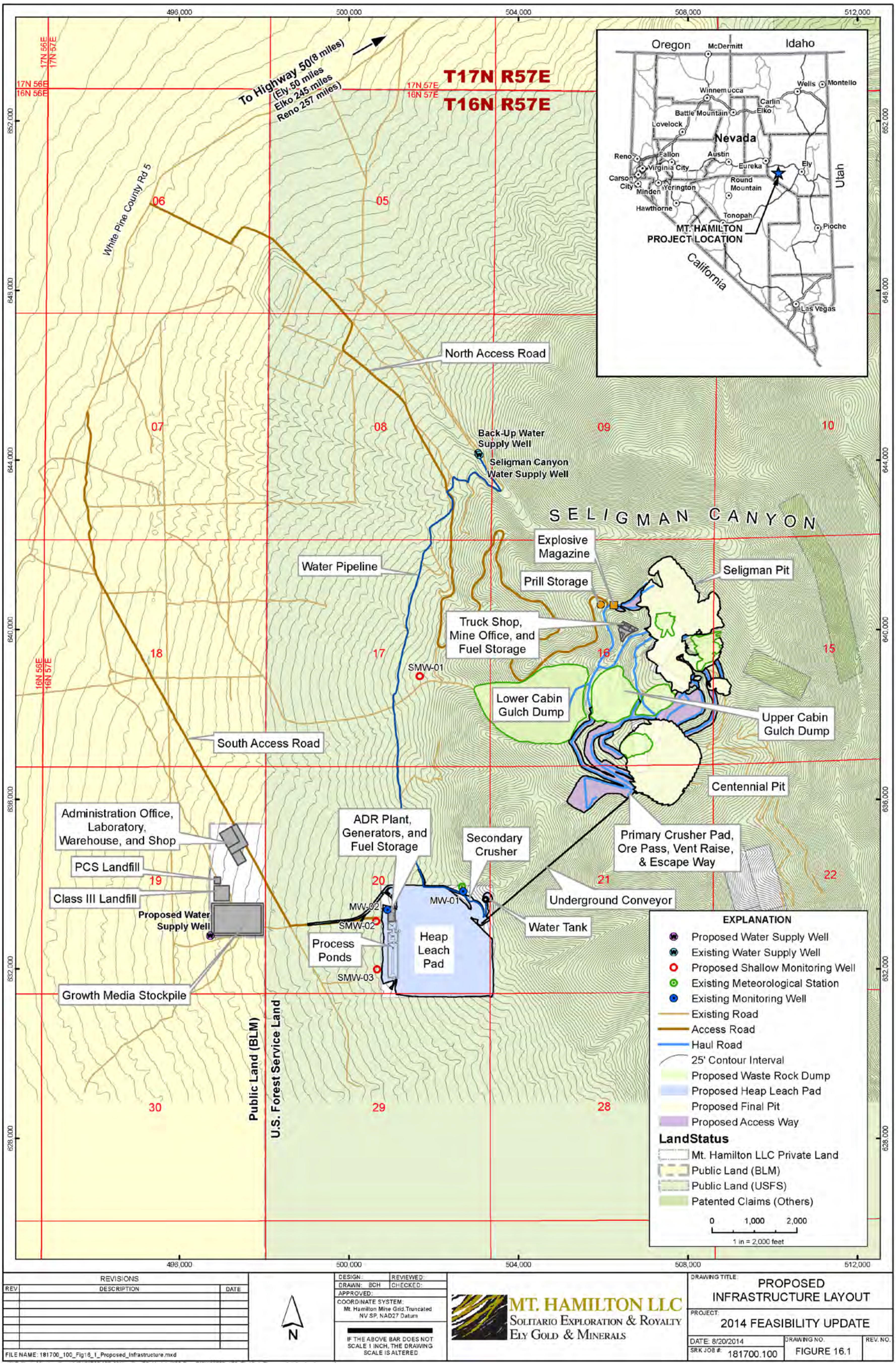
# 16 Project Infrastructure (Item 18)

Specifications for the proposed Project infrastructure are provided in Table 16.1. The general layout of these facilities is illustrated in Figure 16.1.

**Table 16.1: Infrastructure Items and Specifications**

| Feature | Description | | | |
|---|---|---|---|---|
| | Length (ft) | Width (ft) | Height (ft) | Area ($ft^2$) |
| **Facilities** | | | | |
| Warehouse | 90 | 65 | 15 | 4,000 |
| Plant Maintenance Shop | | | | 1,500 |
| Administration Office | 70 | 45 | 11 | 3,100 |
| Laboratory | 65 | 45 | 14 | 2,900 |
| PCS Landfill | 150 | 150 | N/A | 22,500 |
| Class III Landfill | 360 | 360 | N/A | 129,600 |
| Growth Media Stockpile | 1140 | 710 | N/A | 809,400 |
| Fenced Laydown Yard | 400 | 320 | N/A | 128,000 |
| Truck Shop | 220 | 100 | 50 | 22,000 |
| Mine Office in Truck Shop | 115 | 30 | 12 | 3,450 |
| ADR Plant | 145 | 60 | 40 | 8,700 |
| Refinery | 90 | 45 | 20 | 4,000 |
| Explosives Storage | Magazine downhill and 500 ft from truck shop, prill storage at base of existing haul road | | | |
| **Power** | | | | |
| Plant Generators | 4 x 725 kW | | | |
| ADR Generator Diesel Fuel Storage | 2 x 10,000 gal | | | |
| Mine Generators | 2 x 425 kW, 1 x 150 kW | | | |
| Mine Diesel Fuel Storage | 2 x 10,000 gal | | | |
| Site Gasoline Storage | 5,000 gal | | | |
| Future Line Power | 69 kV | | | |
| **Water** | | | | |
| Seligman Water Supply Well | 500 gpm | | | |
| Seligman Backup Well | 200 gpm | | | |
| Proposed New Water Supply Well | 500 gpm | | | |
| Main Water Pipeline | 1,460 ft | | 8 inch diameter | |
| Crusher Water Pipeline | 3,950 ft | | 2.5 inch diameter | |
| Potable Water Pipeline | 3,060 ft | | 3 inch diameter | |
| Water Tank | 750,000 gal capacity, location NE Leach Pad | | | |
| **Communications** | | | | |
| Telephone and Internet | Satellite or microwave-based system | | | |
| 2-way Radio | Line-of-sight repeater | | | |
| **Material** | | | | |
| Waste Rock Storage | Total 63.4 Mt | | | |
| Heap Leach Pad | Total 22.5 Mt | | | |
| **Waste** | | | | |
| Waste Disposal/Landfill | Non-hazardous waste and Petroleum-Contaminated Soil (PCS) disposal on site | | | |
| Septic | Admin area, leach pad area, and truck shop area. | | | |

Source: SRK, 2014




Source: SRK, 2014

**Figure 16.1: Existing and Proposed Infrastructure**

## 16.1 Power

Currently, the nearest power line of sufficient capacity for mine operations is approximately 17 miles from the Project site along Hwy 50. Mt Wheeler Power, Inc. (Mt. Wheeler) is planning to add a new power line that will pass within 12.5 miles of the Mt. Hamilton Mine, which will greatly reduce the required development to bring line power to the mine. The current mine plan includes on-site diesel generated electrical power for the first two years of production and line power supplied by Mt. Wheeler beginning in the third year of production or earlier depending on the time necessary to acquire permits.

During the first two years of production, power for the processing plant area will be supplied by four 725 kW paralleling generators located near the plant. These primary generators will supply electricity to the plant, administration office, laboratory and warehouse area, secondary crusher, proposed water well and all of the conveyors except conveyors C1 and C2. They will be housed in a three-sided 40 ft long x 20 ft wide x 16 ft high eave, pre-engineered steel structure. The switchgear and controls will be housed in an attached 20 ft x 10 ft x 12 ft high eave, fully enclosed space. The flow sheet for plant area power distribution is presented in Figure 16.1.1.

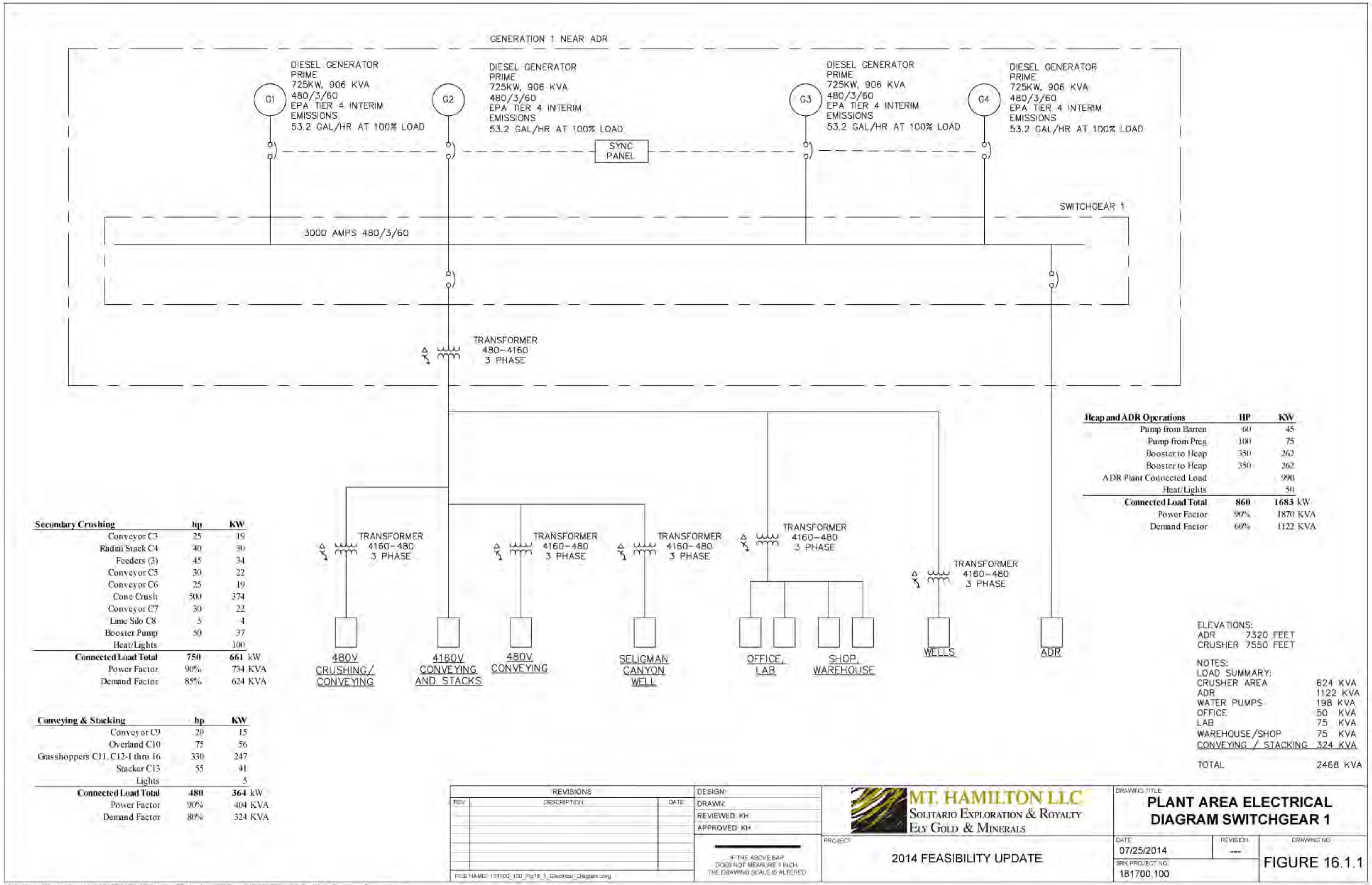
Distribution of high voltage power is to be determined at either 4160 V, 13.8 kV, or 14.4 kV 3 phase, 60 Hz. The most likely scenario is 13.8 kV or 14.4 kV. The voltage will be stepped down at the load locations to 460 volts or 110/220 volts as needed. The 4160 volt feed to the underground conveyor will be completed by the underground contractor. There will be secondary distribution to "grass hopper" Conveyors at 4,160 V, 3 phase, 60 Hz.

The mine area generators will include two 455 kW factory enclosed generators to supply electricity to the truck shop and primary jaw crusher and conveyors C1 and C2. The generators will be located near the truck shop and supply 460 volt power to the crusher. A 150 kW trailer mounted generator will provide power for maintenance projects or to tie into either the upper or lower grids for emergency duty. The 150 kW generator cannot be synchronized with the other CAT generators.

The flow sheet for mine area power distribution is presented in Figure 16.1.2.

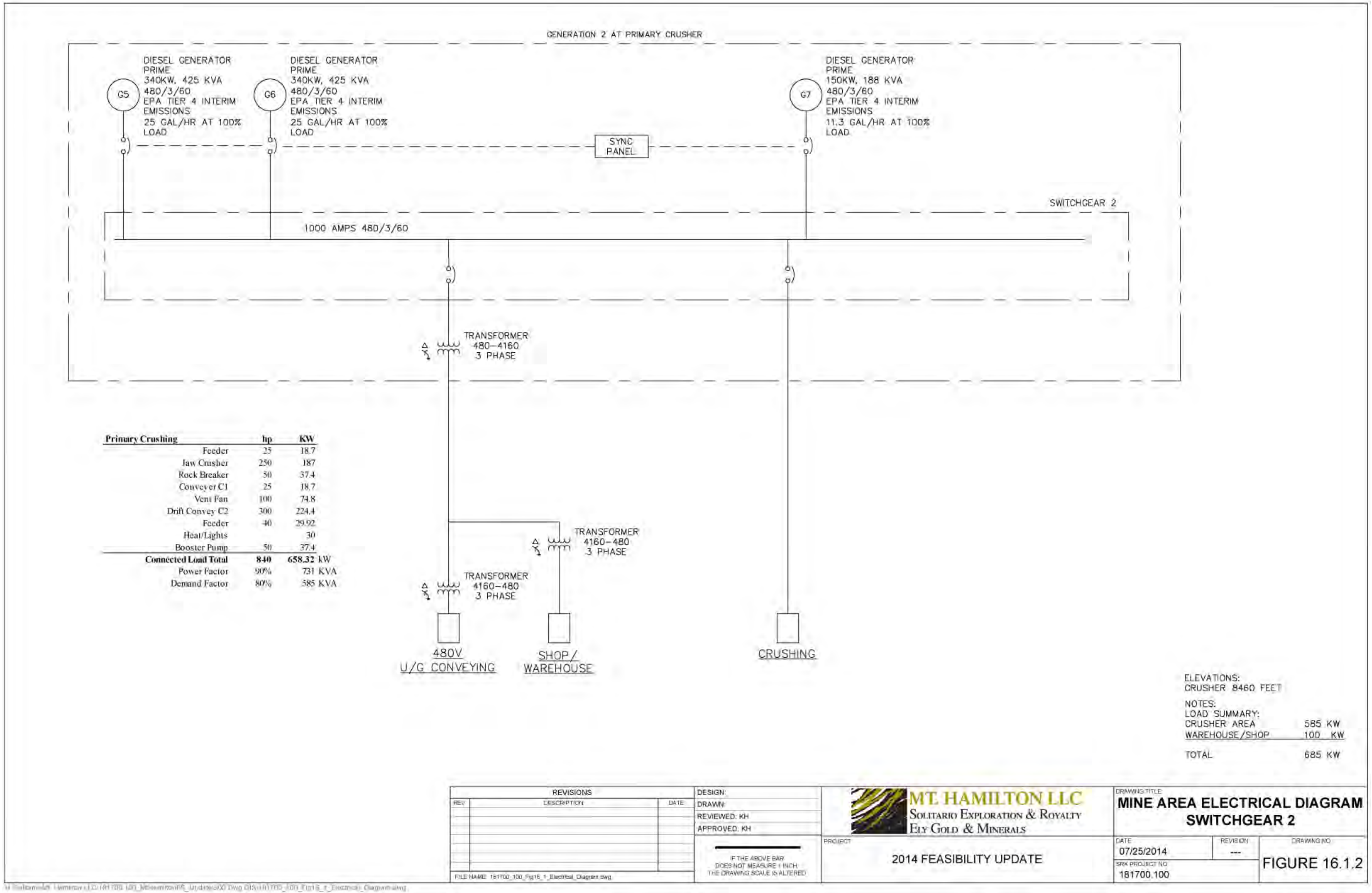
Both the ADR and Truck Shop generators will have 20,000 gal of bulk fuel storage, which will also supply other equipment in the area.

At the beginning of the third year of production or earlier, the mine will switch to line power supplied by Mt. Wheeler. A 69 kV power line is currently planned to originate from the Machacek substation in Eureka to service other mining operations in the area and will pass within 12.5 miles of the Mt. Hamilton operation. One of these mining operations is currently under construction and it is assumed that the power line will be built to service this operation. This 69 kV power line will feed into a transformer near the mine generators and will then feed the mine power distribution system. This switch to line power will provide a significant savings in operating costs over the life of the mine.




Source: SRK, 2014

**Figure 16.1.1: Process Area Power Distribution Flow Sheet**




| Primary Crushing | hp | KW |
|---|---|---|
| Feeder | 25 | 18.7 |
| Jaw Crusher | 250 | 187 |
| Rock Breaker | 50 | 37.4 |
| Conveyer C1 | 25 | 18.7 |
| Vent Fan | 100 | 74.8 |
| Drift Convey C2 | 300 | 224.4 |
| Feeder | 40 | 29.92 |
| Heat/Lights | | 30 |
| Booster Pump | 50 | 37.4 |
| **Connected Load Total** | **840** | **658.32** kW |
| Power Factor | 90% | 731 KVA |
| Demand Factor | 80% | 585 KVA |

Source: SRK, 2014

**Figure 16.1.2: Mine Area Power Distribution Flow Sheet**

## 16.2 Communications

Cellular phone service is intermittently available at the proposed leach pad and truck shop facilities, but is limited in the proposed pit area due to the steep topography. As is typical of most pre-construction mine sites, landline telephones and internet services are not currently available at the site; however, a site-specific 2G network connection was being investigated at the time of this writing.

During operation, communications will likely be through a microwave-based system. This system will support internet and telephone communications. Radio communications for mining operations will use line-of-sight repeater technology.

## 16.3 Water

There is a well in Seligman Canyon capable of producing 500 gpm and a second, backup well that produces 200 gpm (Figure 16.1). These water wells were utilized by Rea Gold for production during the mining at the NES operation. Pumping capacity for the Seligman well was verified by SRK in 2012. It was determined that this well alone likely would not supply enough water during peak demand during construction.

For the cost estimations of the 2014 FS, SRK has developed a water line layout and piping plan using the Seligman Canyon well as the source. This well is located approximately 2.5 miles north of the proposed leach pad and processing plant. A pump at the Seligman Well will supply 400 gpm of water conveyed in an 8 inch HDPE pipe to the plant site. From the plant site, water will flow 1,460 ft in an 8 inch pipe to a 750,000 gal storage tank located on the east side of the leach pad facility for subsequent gravity distribution to the secondary crusher. A second 2.5 inch steel pipe will be used to deliver water 650 ft to the adit portal and then 4,425 ft to the ore pass receiving bin. A booster pump at this location will pump 60 gpm from the 7,600 ft bottom elevation up the ore pass to the primary crusher at 8,470 ft elevation. From the plant site, a separate 8 inch HDPE pipe will deliver 25 gpm to the administration and laboratory buildings, located 3,060 ft west of the plant on the Admin Parcel. Flow in this pipe may be reversed if future water supply is located on the Admin Parcel.

To supply adequate water during peak demand, SRK recommends that another well be installed. Additional water resources are being evaluated closer to the planned leach pad site. An initial phase hydrogeologic exploration drilling program was completed in October 2011, and results suggest that one or more wells in an alluvium-hosted aquifer could supply water needed for mining and heap leach operations. For the purposes of this study, an additional well was assumed to be constructed near the administrative office prior to operations (Figure 16.1), to supply water during construction and serve as a backup to the Seligman Canyon well during mining. Depending on timing of installation and production capacity of the new well, it could become the primary water supply well for the operation. MH-LLC has water rights at the location of these wells adequate for the currently estimated water supply requirements.

## 16.4 Fuel

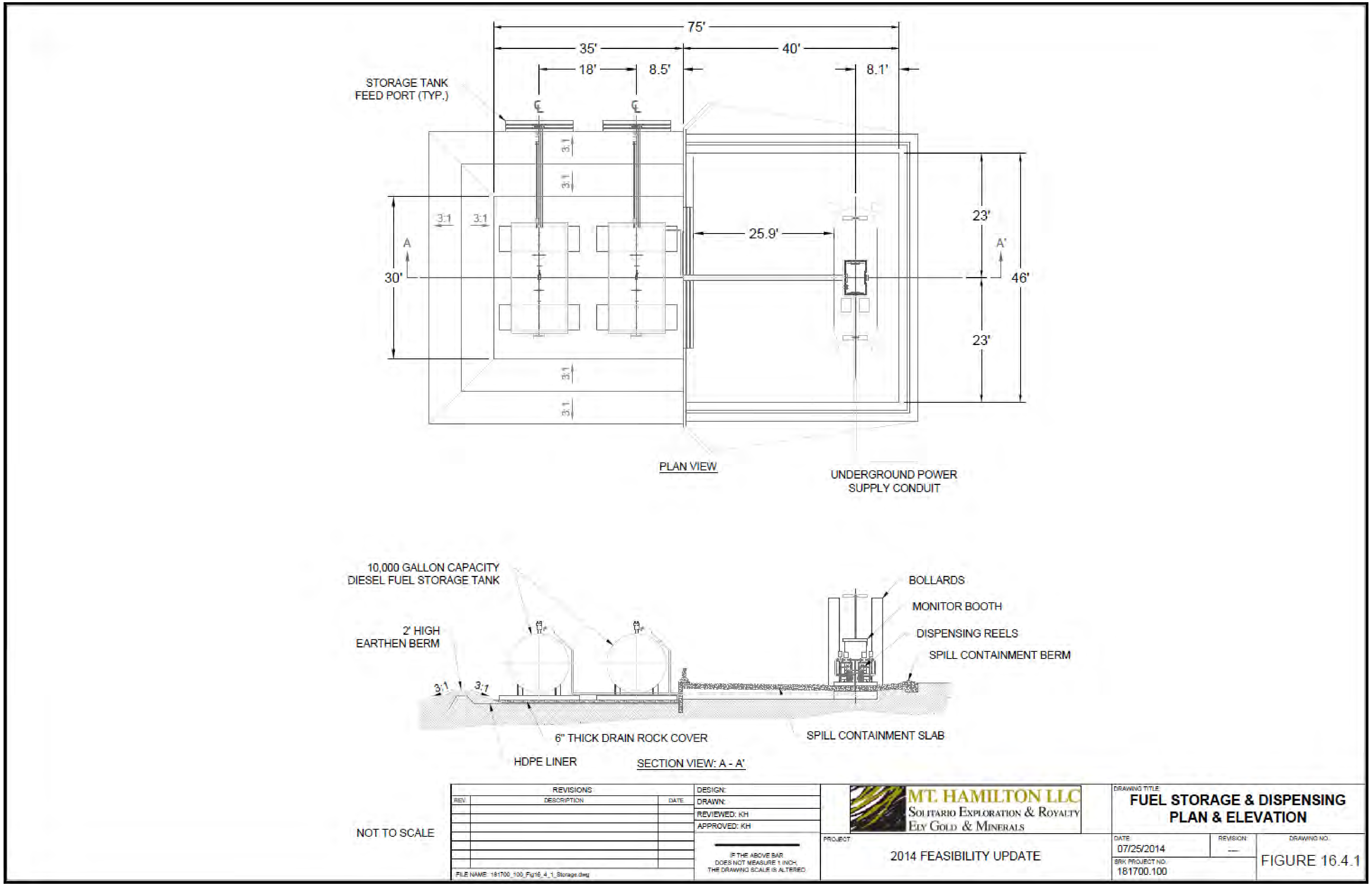
Fuel storage areas will be constructed near the processing plant and the truck shop. Fuel will be purchased in bulk and stored in 10,000 gal diesel and 5,000 gal gasoline tanks inside appropriate containment near the processing plant. Fuel will be dispensed directly to most vehicles from the storage areas. One service truck will be fitted with a fuel tank to supply fuel to the leach pad dozer.

Diesel will also be stored in a tank located near the truck shop. A secondary containment system will be implemented using earthen berms and 80 mil HDPE liner around the tanks and a spill containment slab underneath the refueling stations. The proposed design of the fuel storage area near the truck shop is shown in Figure 16.4.1.

## 16.5 Processing Plant Building

The ADR process building, to be located immediately west of and adjacent to the leach pad, will be a 145 ft long x 60 ft wide x 40 ft eave height pre-engineered steel structure. An 88 ft x 44 ft x 20 ft eave height refinery area will be attached to the ADR building. The 40 ft eave portion will contain the cascading carbon columns and screens, the regeneration kiln and carbon handling system, the acid wash and stripping vessel, the strip heating system and insulated holding tank. The secure refinery area will contain the electrolytic cells, mercury retort, flux mixing and the induction melting furnace. An adjacent 20 ft x 64 ft curbed concrete slab will contain the dust collectors, mercury controls, exhaust fans and furnace chillers for the secure area.

Also attached to the processing plant building is a 39 ft long x 33 ft wide x 20 ft eave height office area and a 38 ft long x 33 ft wide by 20 ft eave height reagent mixing and storage area. The general layout of the proposed ADR plant building in relation to the leach pad is shown in Figure 16.1.




Source: SRK, 2014

**Figure 16.4.1: Proposed Design of Fuel Storage Area**

## 16.6 Mine Administration Office

The mine administration office, 70 ft x 45 ft x 11 ft eave height, will be located on the Admin Parcel west of the mine. The Admin Parcel layout is illustrated in Figure 16.6.1. The proposed design of the 3,150 sq. ft office is shown in Figure16.6.2. The building will include space for mine management, administrative support, human resources, engineering, and information technology staff. It will also include a training room, meeting room and restrooms.

## 16.7 Warehouse & Plant Maintenance Shop

The warehouse and plant maintenance shop building will be constructed near the mine administration office and laboratory on the Admin Parcel (Figure 16.6.1). The warehouse area requirement is 4,000 $ft^2$; the proposed building is 90 ft x 65 ft x 15 ft eave height and includes additional space for a process plant maintenance shop. The warehouse area will include two offices. The attached 1,500 $ft^2$ shop area would include an office, two-ton pedestal crane, compressor and welding outlets. The warehouse and shop buildings would share restroom facilities and lunch area. The proposed design of the warehouse is shown in Figure 16.7.1.

## 16.8 Laboratory

The laboratory will be 65 ft x 45 ft x 14 ft eave height located near the administration building. Laboratory building will consist of a sample prep room, a fire assay area, and a Met/Wet lab area. The power requirement is 480/240/120V. The proposed design of the laboratory is shown in Figure 16.8.1.

## 16.9 Waste Disposal Areas

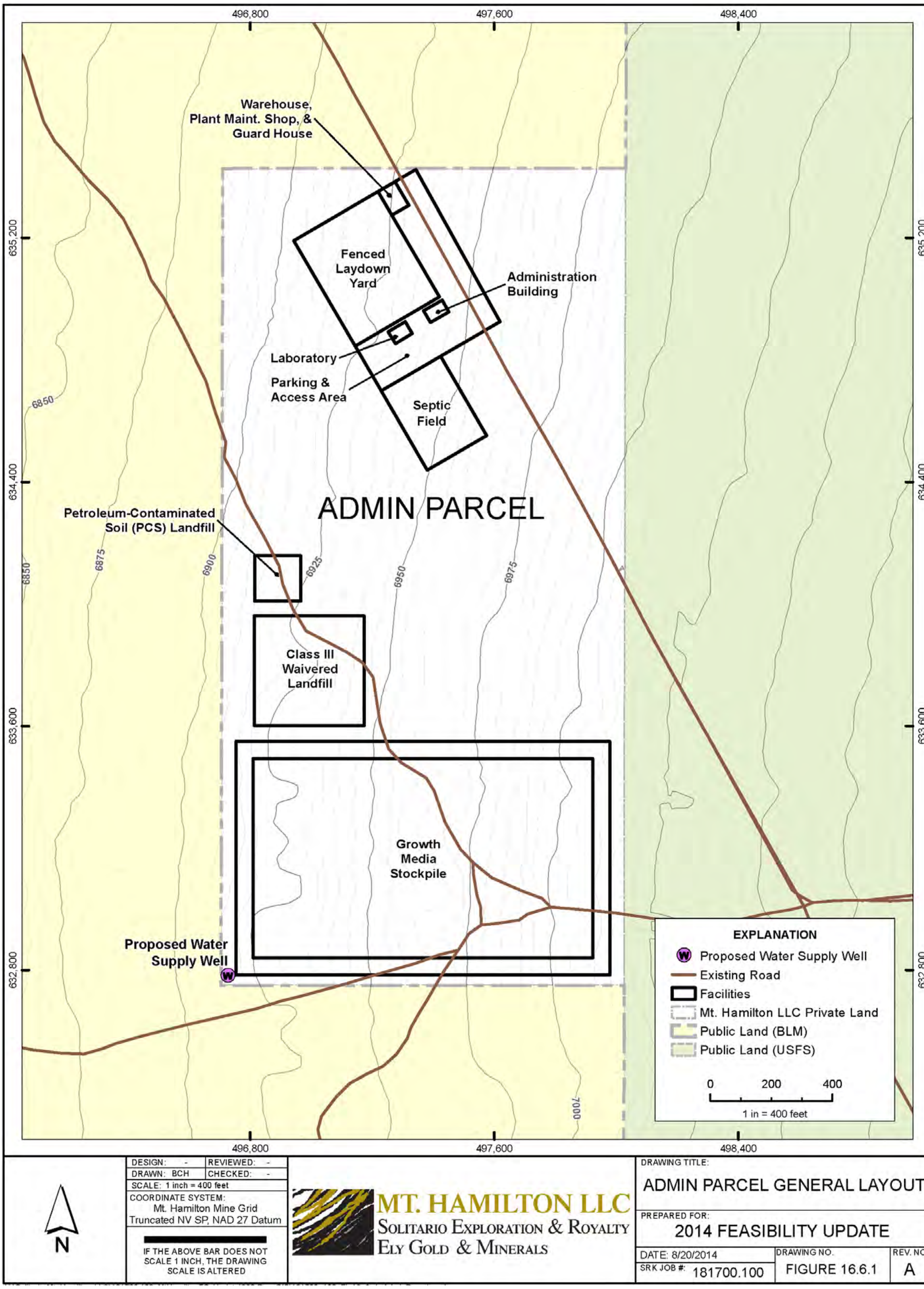
A waste disposal site for construction debris and office- and shop-generated waste will be built on the west side of the Admin Parcel. The proposed Class III Waivered Landfill is 360 ft x 360 ft and will be for non-hazardous waste disposal only. A Petroleum-Contaminated Soil (PCS) storage area with approximate dimensions of 150 ft x 150 ft is proposed to be located north of the Class III Landfill. Other hazardous or regulated wastes will be transported offsite to licensed facilities for disposal.

## 16.10 Growth Media Stockpile

Top soil (growth media) moved during construction will be stored on the south end of the Admin Parcel in the Growth Media Stockpile Area. The proposed area is 1,400 ft x 710 ft with a 25 or 50 ft buffer on all sides.

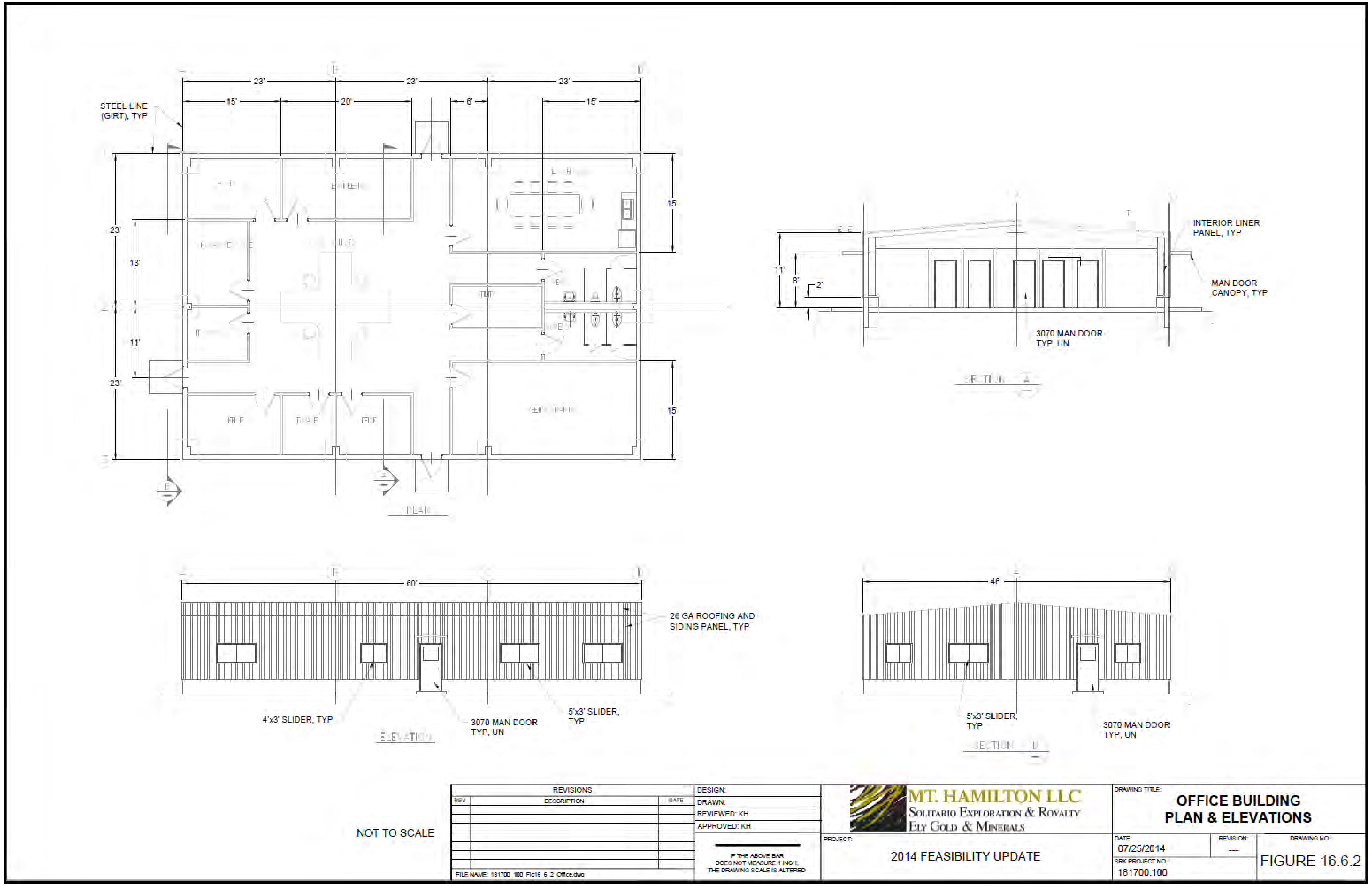
## 16.11 Septic

Three septic systems will be installed. One will service the process building. The administration building and laboratory, warehouse and plant maintenance building will be handled by a second system. The truck shop area will have a separate septic system. The crusher areas will use portable toilets.



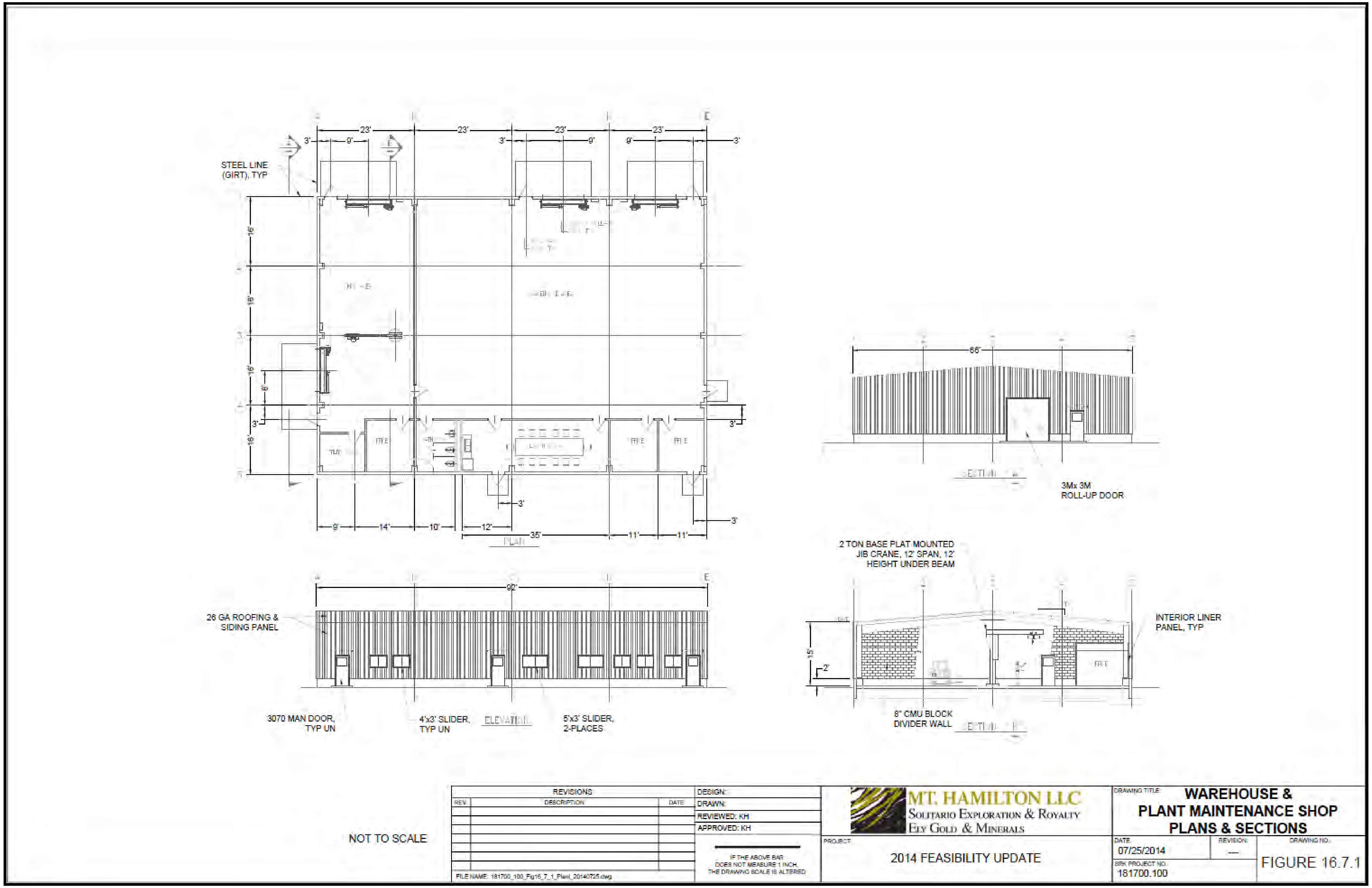

Source: SRK, 2014

**Figure 16.6.1: Admin Parcel Layout**



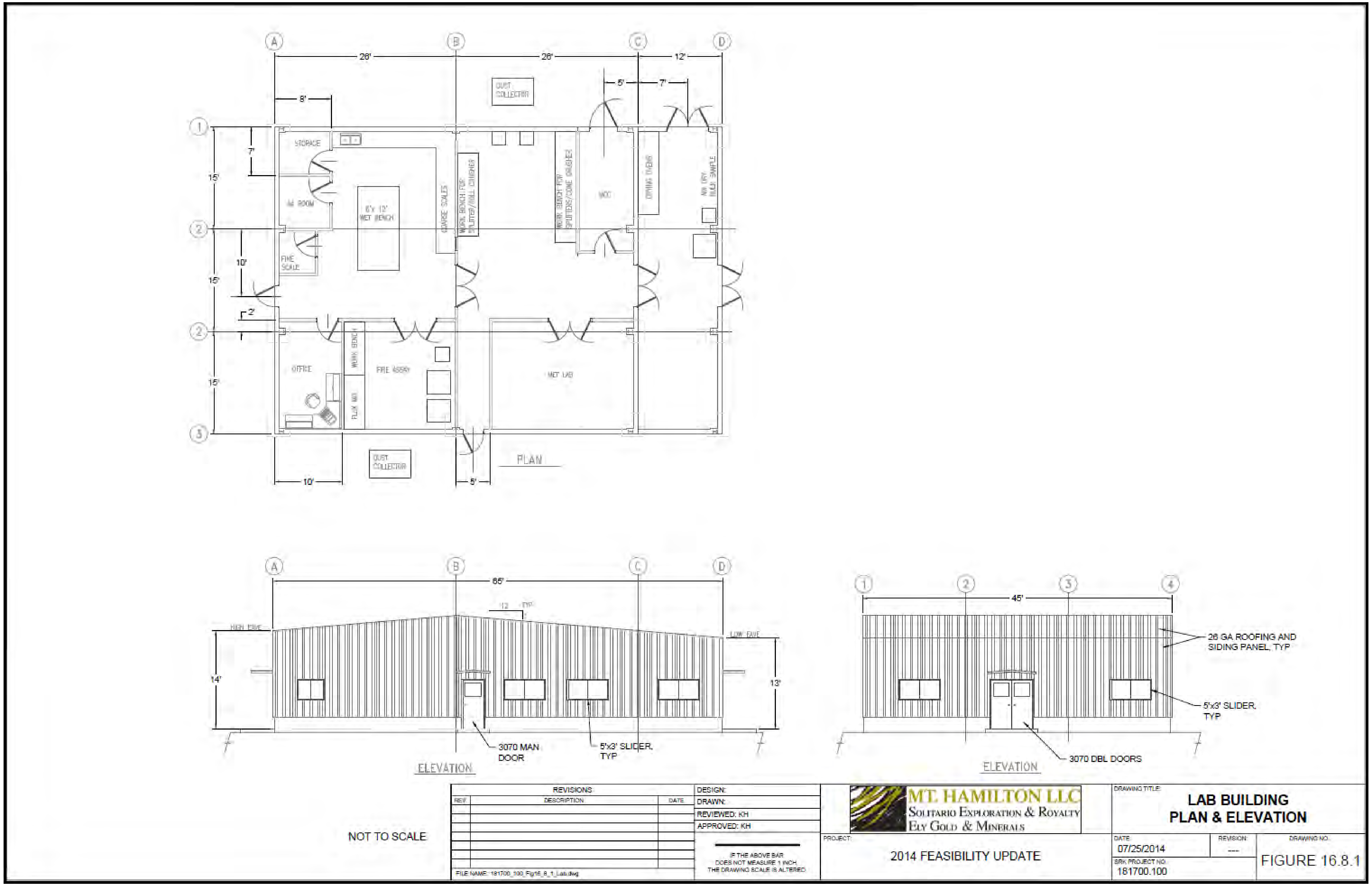

Source: SRK, 2014

**Figure 16.6.2: Proposed Design of Office Building**




Source: SRK, 2014

**Figure 16.7.1: Proposed Design of the Warehouse Building**




Source: SRK, 2014

**Figure 16.8.1: Proposed Design of Laboratory**

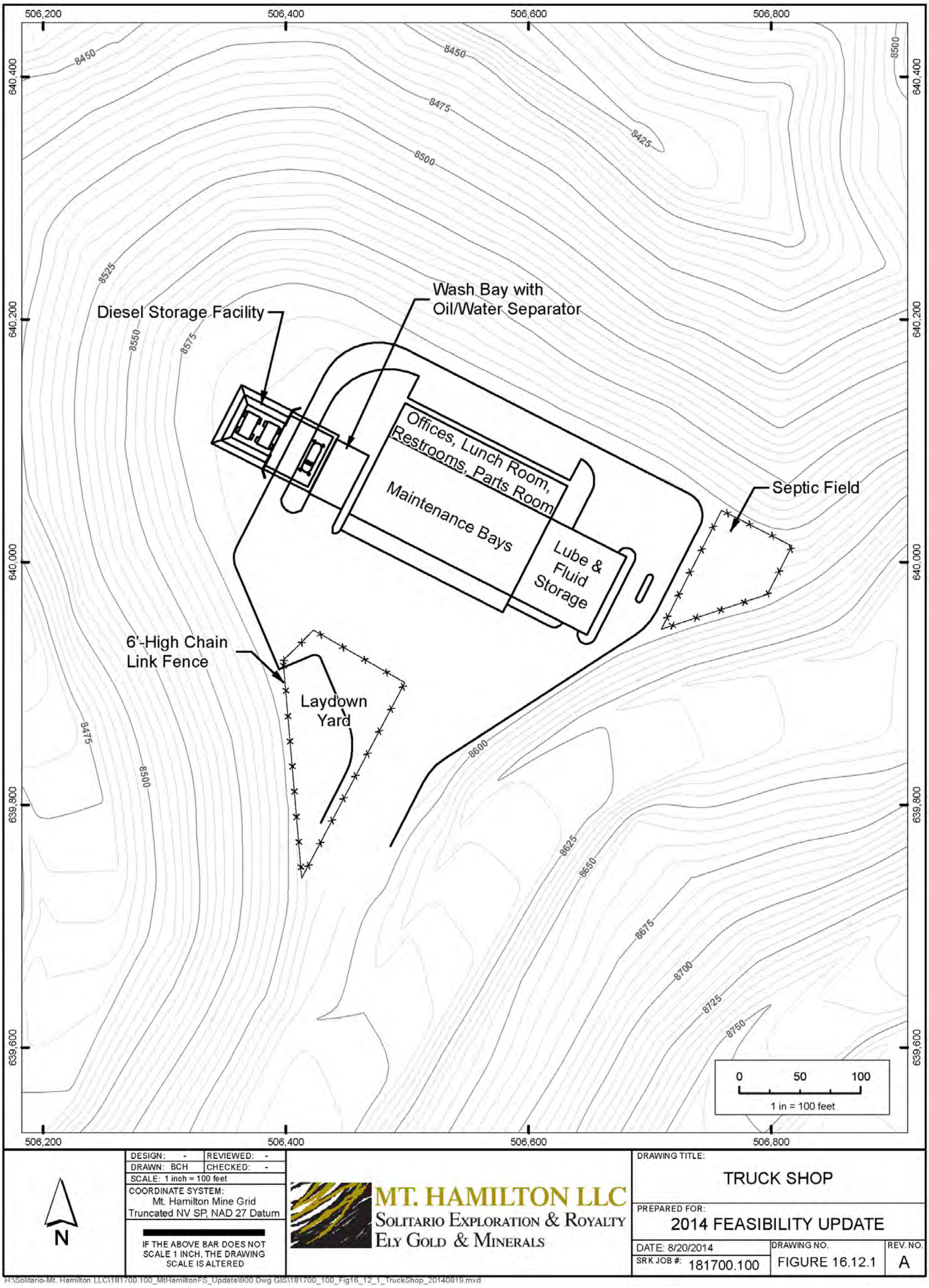
## 16.12 Truck Shop and Mine Operations Office

The truck maintenance shop will be located directly west of the Seligman deposit at elevation approximately 8,585 ft amsl. The truck shop site layout is shown in Figure 16.12.1. The proposed facility has three bays sized to handle 100 t trucks, and is 150 ft x 72 ft x 50 ft. The bays will be serviced by a 10 t overhead crane. The truck shop building will also include offices for the mine and maintenance staff, a lunch room, a tool crib, a storage room and changing rooms. The office area is 112 ft long x 30 ft wide and 12 ft high, along the back of the shop. Bulk lubrication and hydraulic oils, anti-freeze and grease will be stored in an attached 72 ft x 40 ft x 20 ft partition of the building. The proposed truck shop and mine operations office details are shown in Figure 16.12.2 and Figure 16.12.3.

## 16.13 Explosives Storage

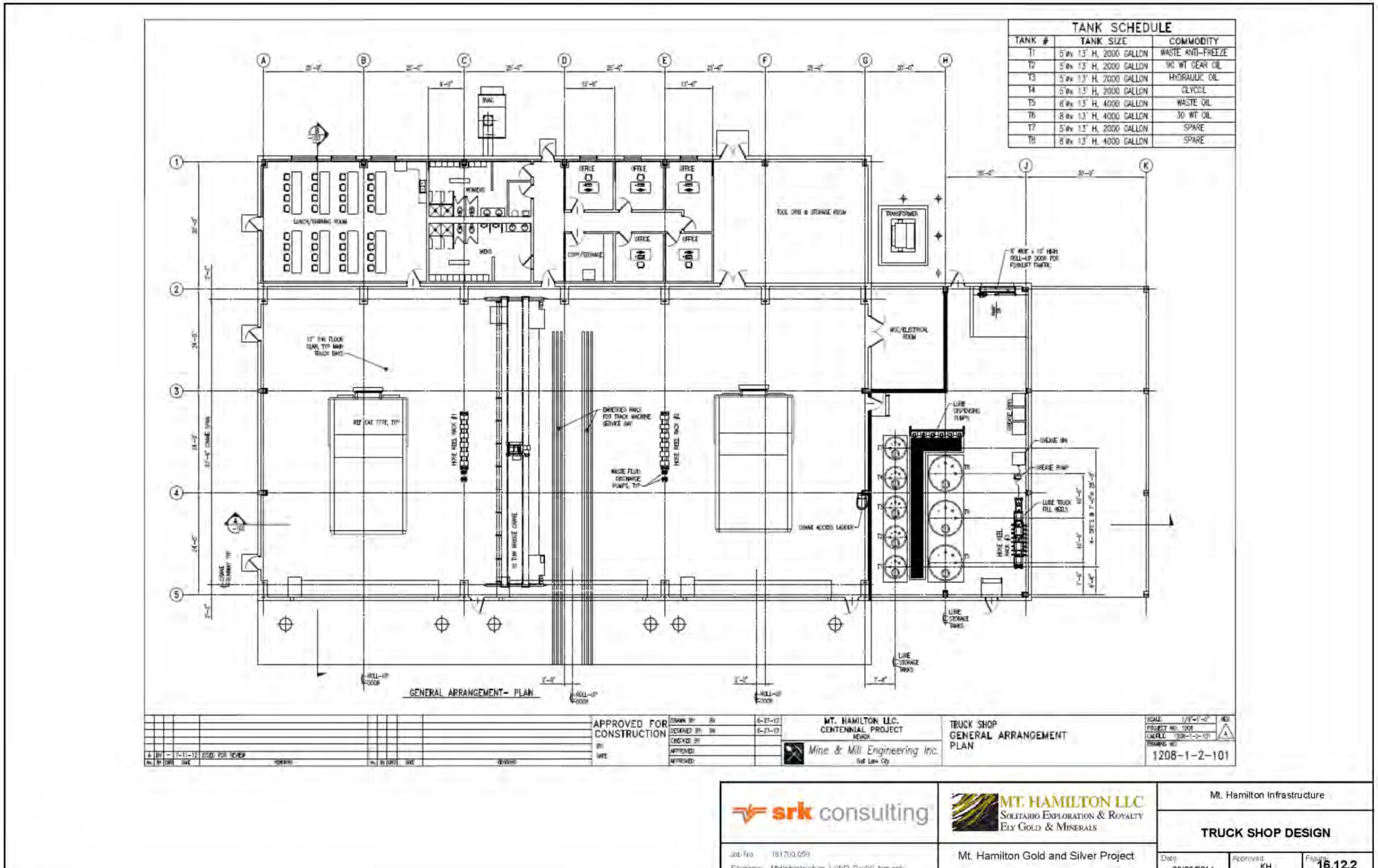
The explosives magazine area will be about 500 ft from the truck shop, in a drainage below the shop pad. Magazines 8 ft x 8 ft x 8 ft will house the boosters and blasting caps separately.

The ammonium nitrate prill storage silo will be located off the existing North Access road, at the old haul road into the NES 1 pit. The explosives magazine and prill silo locations are indicated in Figure 16.1.



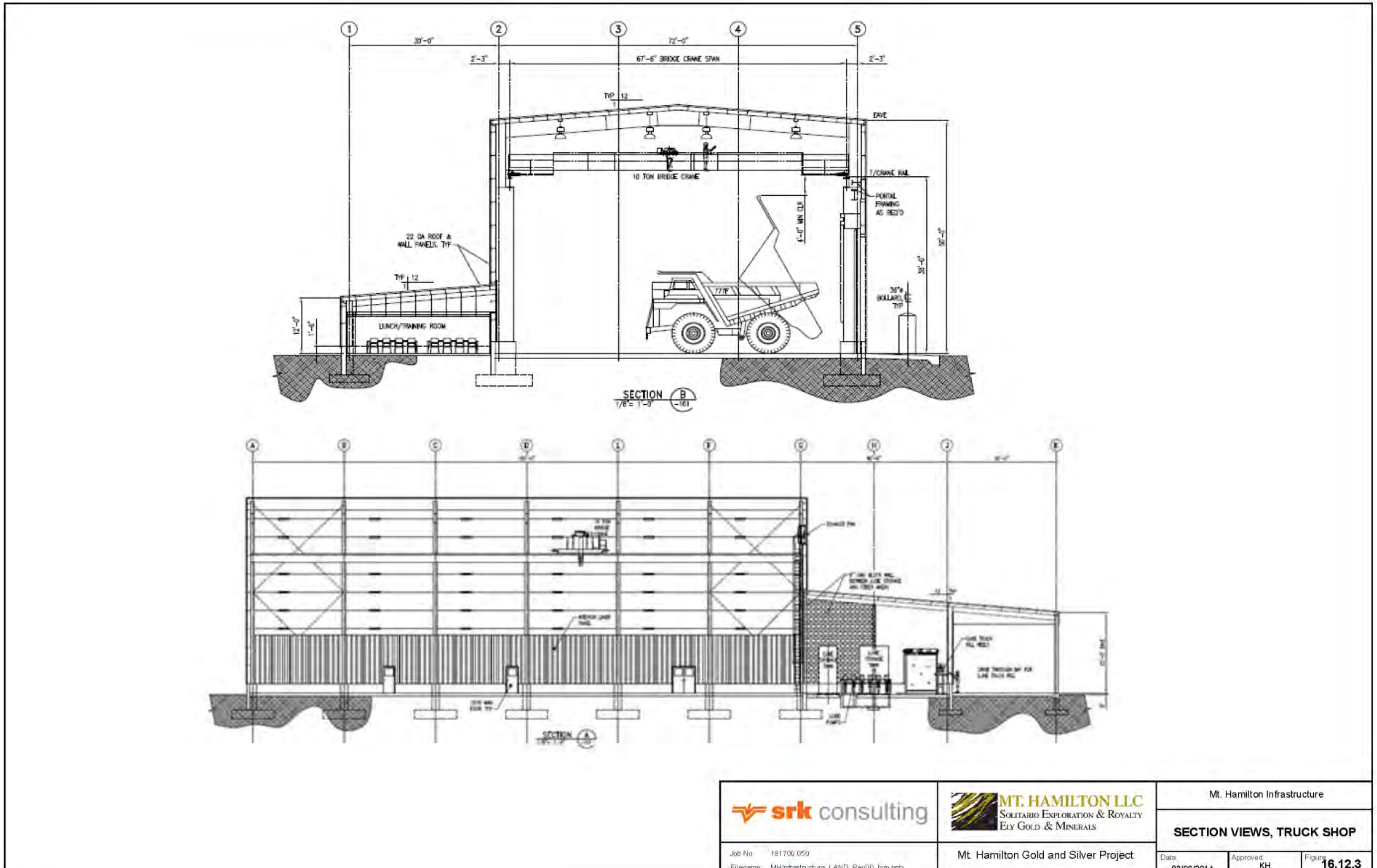

Source: SRK, 2014

**Figure 16.12.1: Truck Shop Site Layout**




Source: SRK, 2014

**Figure 16.12.2: Truck Shop and Mine Operations Office Layout**




Source: SRK, 2014

**Figure 16.12.3 Section Views of Truck Shop**

# 17 Market Studies and Contracts (Item 19)

The process facility proposed for this operation will produce gold doré bars between 80-99% purity. Gold bars will be weighed and assayed at the mine to establish value. The bars will be shipped regularly to a commercial refiner where their value will be verified. Sale prices are obtained based on world spot or London Metals Exchange market pricing and are easily transacted.

## 17.1 Relevant Market Studies

A market study for the gold product was not undertaken for this 2014 FS. Gold is sold through commercial banks and market dealers. The gold market is stable in terms of commodity price and investment interest.

## 17.2 Commodity Price Projections

This study assumes a static price curve for the gold market price. In the economic evaluations, the gold price was set at US$1,300/oz and the silver price was set at US$20/oz. Precious metal prices have stabilized at these levels for the last 12-18 months after hitting historic highs in 2011.

## 17.3 Contracts and Status

Terms for an off-take and smelting agreement are based on recent communications with Johnson Matthey, an international smelting and refining company with a facility at 4601 West 2100 South, Salt Lake City, Utah 84120.

Contract terms and doré treatment charges listed below are current as of Q4, 2011. These terms are suitable for use in this study:

- Treatment Charge: US$0.35/oz net weight received;
- Refining Charge: US$1.00/oz fine gold credited;
- Gold Return: 99.85% of assayed content;
- Silver Return: 99.00% of assayed content; and
- Settlement: 25 working days from receipt.

# 18 Environmental Studies, Permitting and Social or Community Impact (Item 20)

## 18.1 Environmental Liabilities

SRK is unaware of any outstanding environmental liabilities aside from minor reclamation obligations associated with existing drill roads that are still actively used.

A portion of the Mt. Hamilton Property which was mined during the 1990's by a previous operator has been extensively reclaimed by the U.S. Department of Agriculture, United States Forest Service (USFS, or Forest Service). The leach pad associated with previous mining has also been covered with soil, contoured, and revegetated. At the time of SRK's site visits, seeding was successful and the pad is now completely grass-covered. The site of the former mine-associated infrastructure has been completely reclaimed and virtually all remains of buildings have been removed. The only significant elements of the former mining operation are the haulage road from the old leach pad to the NE Seligman Mine site and the open pit mining areas from prior operations. This road remains in good repair and provides ready access to both of the deposit areas. MH-LLC currently has no environmental liabilities related to this previous mining activity.

## 18.2 Required Permits and Status

The Project is being permitted separately on National Forest System (NFS) lands and patented mining claims, where the mining and access will occur, and on private land owned by MH-LLC where the processing of the ore is planned and administrative infrastructure will be located.

A Mine Plan of Operations (MPO) was submitted to the Forest Service for mining activities on NFS lands. The MPO was determined to be complete by the USFS and scoping of the project was conducted in order to determine the issues to be evaluated to comply with the National Environmental Policy Act (NEPA). The USFS determined that an Environmental Assessment (EA) was required. Upon completion of the EA, a Finding of No Significant Impact and a draft Decision Notice were published on July 4, 2014. The Objection Period ended on August 18$^{th}$, 2014. No objections were filed. Phased bonding for reclamation of the mining areas will be required. The initial bonding of the first phase was submitted to the Forest Service, reviewed and accepted on September 24, 2014. The bill of collection, receipt and issue is pending.

Road access to the mine and to the administration/processing areas each requires crossing BLM land in order to enter the MPO area on Forest Service property. These two access routes are subject to a Right of Way grant by the BLM, which was issued to MH-LLC in 2013.

A Nevada Reclamation Permit (NRP) application has been submitted for the area covered by the MPO. The application for this permit is under review by the Nevada Division of Environmental Protection (NDEP), Bureau of Mining Regulation and Reclamation (BMRR).

The private land used for processing the ore and administrative functions is being permitted and bonded separately through the NDEP BMRR and will have a separate Nevada Reclamation Permit. An application for this permit has been filed and is under review. The USFS will not be involved in this permit approval although operations on private land are considered in the NEPA analysis as a connected action.

A Water Pollution Control Permit (WPCP) has been issued by the NDEP. The WPCP covers the entire project including both public and private land.

An Air Quality Permit (AQP) application has been submitted to NDEP for review and has been deemed to be complete. The technical review of the permit application is in process. A preliminary ADR plant design has been completed in order to provide the detail necessary for design of the mercury control systems to be incorporated in and reviewed under the Air Quality Permit.

**Other State of Nevada Permits**

Water appropriations are processed through the Nevada Division of Water Resources (NDWR) and the State Engineer's Office. Currently MH-LLC has appropriated 875 AFA of water, an amount sufficient for peak water requirements for the operation including construction.

**Local Permitting**

A Special Use Permit for septic and excavation/building permits will be required from White Pine County; usually a copy of the MPO provides sufficient information for the County to review and issue this permit.

To the best of SRK's knowledge, MH-LLC is in full compliance with all contractual and regulatory obligations. Because of previously permitted mining activity at the Project, SRK currently has no reason to believe that the few remaining permits to mine the mineral resources at Mt. Hamilton could not be reasonably obtained from the state and federal regulatory agencies.

Major permits for future mining operations are summarized in Table 18.2.1.

**Table 18.2.1: Summary of Major Permits Required for Mining Operations**

| Regulatory Agency | Permit Name |
|---|---|
| **Federal Permits** | |
| US Forest Service | • Approved Plan of Operations/Decision Memo |
| Bureau of Alcohol, Tobacco, Firearms, and Explosives | • Authorization to purchase, transport, or store explosives |
| Mine Safety and Health Administration | • Notification of Commencement of Operation<br>• Employee and Facility Health and Safety |
| Environmental Protection Agency | • Hazardous Waste ID No. (small quantity generator) |
| Bureau of Land Management | • Roads and Utility Rights-of-Way |
| **State Permits** | |
| ***Nevada Division of Environmental Protection*** | |
| Bureau of Mining Regulation and Reclamation | • Water Pollution Control Permit<br>• Reclamation Permit |
| Bureau of Air Pollution Control | • Class I Air Quality Operating Permit<br>• Mercury Operating Permit |
| Bureau of Water Pollution Control | • Septic Permit |
| Bureau of Waste Management | • Approval to Operate a Solid Waste System<br>• Hazardous Waste Management Permit |
| Bureau of Safe Drinking Water | • Potable Water Permit |
| ***Nevada Division of Water Resources*** | |
| | • Permit to Appropriate Water<br>• Permit to Construct a Dam<br>• Hole Plugging |
| ***Nevada Department of Wildlife*** | |
| | • Industrial Artificial Pond Permit |
| ***State Fire Marshall*** | |
| | • Hazardous Materials Permit |
| **Local Permits** | |
| ***White Pine County*** | |
| | • Special Use Permit<br>• Building Permit<br>• Business License |

Source: SRK, 2014

# 18.3 Environmental Study Results

## 18.3.1 Groundwater Monitoring

Two monitoring wells were installed by MH-LLC in late 2011. The wells were designed to characterize groundwater up gradient and down gradient from the proposed Heap Leach Pad (HLP) and were developed for sampling, which started in 2012. The well location coordinates and completion details are provided in Table 18.3.1.1 and Table 18.3.1.2, respectively. The locations of the monitoring wells were presented in Figure 14.1 (this report).

### Table 18.3.1.1: Groundwater Monitoring Well Locations, UTM NAD27 Coordinates

| Hole ID | Easting (Meters) | Northing (Meters) | Collar Elevations (ft amsl) | Total Depth (ft-bgs) | Static Water Depth (ft-bgs) | Bedrock Depth (ft-bgs) |
|---|---|---|---|---|---|---|
| MW-01 | 623079 | 4343939 | 7,570 | 862 | 782.5 | 200 |
| MW-02 | 622543 | 4343796 | 7,291 | 750 | 365 | Not Encountered |

Source: SRK, 2012

### Table 18.3.1.2: Well Completion Details

| Well ID | Well Completion Date | Well Casing Material | Total Depth (ft-bgs) | Screen Bottom (ft-bgs) | Screen Top (ft-bgs) | Bottom Filter Pack (ft-bgs) | Top Filter Pack (ft-bgs) | Bentonite (ft-bgs) | Cement Grout (ft-bgs) | Depth to Water (ft-bgs) | Geologic Unit Screened |
|---|---|---|---|---|---|---|---|---|---|---|---|
| MW-01 | 10/8/11 | 4 inch PVC | 862 | 862 | 662 | 862 | 660 | 660 - 647 | 647 - 0 | 782.5 | Kqp/Csc |
| MW-02 | 9/22/11 | 4 inch PVC | 750 | 750 | 590 | 750 | 586 | 586 - 577 | 577 - 0 | 365 | Qal |

Source: SRK, 2012

Well MW-01 (up gradient well) was advanced through a 200 ft alluvial sequence overlaying tight shale with interbedded chert to 600 ft-bgs and quartz monzonite porphyry to a depth of 862 ft-bgs. Static groundwater was initially recorded in MW-01 at 782.5 ft-bgs, well within the intrusive unit.

Groundwater monitoring well MW-02 (down gradient) was advanced through an alluvial sequence (Qal) to a depth of 750 ft-bgs. Bedrock was not encountered in this hole. The static groundwater level measured in MW-02 was 365 ft-bgs.

Results from the hydrology program indicate that the static water level in the bedrock aquifer is at a lower elevation than the lowest level of the planned open pit mine, so a pit lake will not develop.

During the hydrogeologic investigation programs at the mine and the WRDA, the static water level was not encountered in any of the drillholes reaching depths of greater than 500 ft. Therefore, groundwater is estimated to be greater than 500 ft-bgs in these areas.

Groundwater chemistry data were obtained from up-gradient monitoring well MW-01, which is screened in bedrock (shale with interbedded chert), and from downgradient monitoring well MW-02, which is screened in gravel (valley fill alluvium). Groundwater chemistry data for samples collected in 2012 were provided by Enviroscientists, Inc. (Reno, Nevada) and data for samples collected in 2013 were provided by MH-LLC. The average data for each well are presented in Table 18.3.1.3. The laboratory reports for the groundwater chemistry data are available in the 2014 FS.

The groundwater chemistry results demonstrate that baseline concentrations of arsenic and manganese are slightly elevated above the respective NDEP reference values in MW-01, the bedrock well. All dissolved constituents are less than the respective NDEP reference values in MW-02, the alluvium well. The first sample from MW-02 had a pH value of 11.06 s.u.; the average pH for the other four samples is 8.00 s.u., and is comparable to the pH range of MW-01 samples. For the purpose of calculating the averages presented in Table 18.3.1.3, the first sample was not included for MW-02. The measured static water levels and water quality from the well samples confirm the existence of separate bedrock and alluvium aquifers in the Project area, as predicted.

**Table 18.3.1.3: Average Groundwater Chemistry for MW-01 and MW-02**

| Parameter | NDEP Reference Value | MW-01 Average data for September 2012 to August 2013 | MW-02 Average data for March 2012 to August 2013 |
|---|---|---|---|
| Alkalinity | - | 190 | 120 |
| Aluminum | 0.2 | <0.05 | <0.05 |
| Antimony | 0.006 | <0.002 | 0.003 |
| Arsenic | 0.01 | 0.036 | 0.008 |
| Barium | 2 | 0.13 | 0.02 |
| Beryllium | 0.004 | <0.002 | <0.002 |
| Cadmium | 0.005 | <0.002 | 0.003 |
| Calcium | - | 49 | 34 |
| Chloride | 400 | 3.3 | 2.5 |
| Chromium | 0.1 | 0.003 | 0.003 |
| Copper | 1.3 | 0.003 | 0.003 |
| Fluoride | 4 | 0.2 | 0.2 |
| Iron | 0.6 | 0.09 | <0.05 |
| Lead | 0.015 | <0.002 | <0.002 |
| Magnesium | 150 | 9.7 | 8.8 |
| Manganese | 0.1 | 0.14 | <0.002 |
| Mercury | 0.002 | <0.0001 | <0.0001 |
| Nickel | 0.1 | 0.003 | 0.003 |
| $NO_3$+$NO_2$ | 10 | <1 | <1 |
| pH | 6.5-8.5 | 7.95 | 8.00 |
| Potassium | - | 1.7 | 1.7 |
| Selenium | 0.05 | <0.01 | <0.01 |
| Silver | 0.1 | <0.002 | <0.002 |
| Sodium | - | 29 | 7.125 |
| Sulfate | 500 | 41 | 16 |
| Thallium | 0.002 | <0.001 | <0.001 |
| TDS | 1000 | 307 | 157 |
| Zinc | 5 | 0.04 | 0.04 |

All values reported in mg/L except pH, which is in standard units (s.u.)
< indicates less than the specified method detection limit.
Shaded values exceed the respective comparative value from NDEP Form 0190 for Profile II constituents.
Source: SRK, 2014

## 18.3.2 Waste Rock and Ore Characterization and Management

**Introduction**

SRK prepared the Waste Rock and Ore Characterization Report (2014b) to present the methodology and results of the geochemical characterization program conducted for the Centennial and Seligman gold and silver deposits as part of the Mt. Hamilton Gold Project (the Project). The scope of the completed characterization program is the rock mass in the MPO design pits for the Centennial and Seligman deposits, based on 2014 geologic and resource models of each deposit. The purpose of the characterization program was to assess the acid rock drainage and metal leaching (ARDML) potential of the rock mass in the mine plan. The results of this geochemical characterization program and numerical predictions of water quality are summarized herein.

The Waste Rock Management Plan (SRK, 2014c) is a companion document to the Characterization Report, and was developed to support the WPCP application. The Waste Rock Disposal Area (WRDA) design considerations are based on the conclusions presented in the Characterization Report. Conclusions from the Management Plan are summarized herein.

Although the expanded 2013 resource and mine plan include additional material beyond the scope of the geochemical characterization program, the rock mass in the expanded pits is predicted to be analogous to the materials characterized previously.

**Summary of Characterization Results**

The results of the static and kinetic geochemical test work demonstrate that the majority of waste rock material from the Centennial and Seligman deposits is net neutralizing and presents a low risk for acid generation. This is based on a low sulfide sulfur content (typically <0.1 wt%) and an excess of neutralizing capacity. From the Acid-Base Accounting (ABA) tests, 75% of the 97 Centennial samples and 55% of the 66 Seligman drill samples are shown to be non-acid forming based on the BLM criteria. All but eight of the 163 samples meet the NDEP criteria for classification as non-acid generating rock with Neutralization Potential Ratio (NPR) values greater than 1.2. Only a small sub-set of drill samples show a higher risk for acid generation in the ABA test. These samples consist of ore-grade skarn which represent material that will not be placed in the WRDA. ABA results for the remaining samples are inconclusive and kinetic testing was conducted to address these uncertainties.

During Humidity Cell Tests (HCTs), the hornfels and skarn ore samples that showed an uncertain potential for acid generation from the ABA results produced alkaline leachates, as did the samples of hornfels waste and skarn waste that were predicted to be non-acid forming from the ABA tests. Two samples of igneous intrusive waste were selected for humidity cell testing. Based on the humidity cell test results, the sample of unoxidized igneous intrusive with visible pyrite generated acid during the HCT. The other sample of igneous intrusive was oxidized and did not contain visible sulfides. This sample maintained alkaline conditions throughout the duration of the HCT program and demonstrated a low potential for ARDML. Based on these results, a sub-set of the igneous sulfide material has the potential to produce acid, but the static test results show that this material type is not always predicted to be acid generating and in some cases, the unoxidized igneous intrusive is predicted to be net neutralizing from the ABA.

Based on the results of the geochemical characterization program, the only material types to demonstrate a potential for acid generation include a sub-set of the ore grade skarn and the unoxidized igneous intrusive ore and waste rock material. This material has undergone incomplete oxidation, which consumed available carbonate and left some pyrite intact. The potentially acid forming unoxidized igneous intrusive material will comprise approximately 10% by weight of waste rock generated by the combined Centennial and Seligman deposits.

The spent ore samples included in this study were also found to contain significant neutralizing capacity and are predicted to be non-acid generating from both the ABA and net acid generating (NAG) results.

Although the excess of neutralizing capacity means that net acid conditions are unlikely to develop at Mt. Hamilton, several metal(loid)s are likely to be mobile under the circum-neutral to moderately alkaline conditions. Meteoric Water Mobility Procedure (MWMP) tests indicated that aluminum, antimony, arsenic, mercury and thallium may be released during meteoric rinsing of the waste rock (i.e. during rainfall events) from one or more material types. These constituents have the potential to be leached from the hornfels and skarn material at concentrations above NDEP reference values. However, concentrations of these constituents are generally less than an order of magnitude higher than the respective reference value, indicating a low probability to impact groundwater or surface water resources. From the HCT results, there was also an initial flush of mercury from two of the six cells, but mercury release from the other cells was low. These results suggest mercury has limited potential to impact groundwater or surface water resources. This is further reduced by the significant depth to groundwater under the mine facilities (>500 ft under the Waste Rock Disposal Area) and the absence of any surface water resources down-gradient of the mine facilities.

Results of the characterization program indicate the sulfide-bearing unoxidized igneous material is characterized by limited neutralization potential and exhibits a higher risk for metal leaching. This material type showed release of aluminum, antimony, arsenic, beryllium, cadmium, chromium, fluoride, iron, lead, magnesium, manganese, nickel, sulfate and thallium above NDEP reference values under low pH conditions during the MWMP and HCT tests.

**Summary of Numerical Modeling Results**

Predictive geochemical modeling was carried out using mass balanced HCT results to develop source term for the waste rock associated with the Mt. Hamilton project. This modeling effort uses the USGS-developed software PHREEQC to predict the concentrations of constituents that could be released from the waste rock in response to meteoric rinsing by precipitation and evaluates the potential for waste rock to degrade groundwater resources down gradient of the WRDA.

During mining operations and post-closure, any recharge from the Mt. Hamilton WRDA will be derived from internally-stored moisture and also from infiltration of meteoric waters (i.e., precipitation). During closure, the WRDA surface will be regraded and the majority of precipitation is likely to either evaporate or infiltrate the facility. The water that infiltrates the WRDA will be held until the field capacity of the waste rock material is exceeded, after which recharge to groundwater may occur. The precipitation infiltrating the WRDA will have a period of contact that enables desorption and dissolution of solutes from each lithology that yields a unique water chemistry. The resultant chemistry of leachate waters reporting from the WRDA can be represented as the weighted sum of the leachate associated with each waste rock material type. Thus material types that comprise a larger proportion of the waste rock will exert a greater control on the net WRDA leachate chemistry. This solution will migrate through the WRDA, into the underlying bedrock and may eventually make its way to the groundwater below.

The geochemical model assumes waste rock from the Centennial and Seligman pits will consist of hornfels, skarn and igneous intrusive and will be co-disposed within the WRDA. Segregation of potential acid generating (PAG) waste rock is not currently proposed for the Project. For the base case (i.e., most-likely) model scenario it was assumed that PAG waste rock (i.e., unoxidized intrusive unit, which will comprise approximately 10% of the total waste rock) will be blended with material with a higher neutralizing capacity in order to preclude the development of ARD.

Representative leachate chemistries for the waste rock were obtained from site specific HCT data and used as input solutions to the geochemical model. The HCT samples were collected from exploration drill core within the proposed Centennial pit. No humidity cells were run on material from the Seligman pit, but the lithologies in both pits are the same and they have been assigned the same geochemical properties.

Results of the geochemical modeling indicate that blending of PAG waste rock will be sufficient to prevent acidification of groundwater, with groundwater under the WRDA predicted to be moderately alkaline (pH 8.45). Numerical predictions also confirm that solutes are predicted to be below NDEP reference values in groundwater underlying the facility. Exceptions include manganese and antimony, which are predicted to be marginally elevated above the respective reference values for these parameters. Baseline concentrations of manganese are already noted to be elevated in the baseline groundwater and any seepage from the WRDA is not expected to increase groundwater manganese concentrations.

Antimony is predicted to be slightly elevated above the NDEP reference value in groundwater underlying the WRDA, which likely reflects leaching of antimony from the hornfels waste rock material under circum-neutral to moderately alkaline conditions. However, the current geochemical predictions are conservative in that they assume that 7% of mean annual precipitation (MAP) will recharge to groundwater and that there will be no attenuation of solutes in the bedrock underlying the WRDA. In reality, the significant depth to groundwater (~570 ft) in the vicinity of the proposed WRDA may mean that only a very small proportion (if any) of MAP may report to groundwater. Furthermore, there is likely to be both storage of seepage waters and attenuation of constituents between ground surface and the water table, resulting in a reduced solute loading to groundwater. Antimony is known to attenuate under circum-neutral to alkaline conditions, and therefore antimony would be effectively attenuated between the base of the WRDA and the water table. However, without site-specific attenuation data it has not been possible to account for the attenuation of antimony in the current geochemical model. Although arsenic and mercury showed elevated release during the MWMP and HCT tests, these constituents are attenuated onto iron (oxy) hydroxide minerals within the WRDA and these constituents are not predicted to be elevated above NDEP reference values in groundwater underlying the facility.

**Summary of Waste Rock Management Plan**

Results of the geochemical testing indicate that the rock type and oxidation state can be used to define the acid generating potential of the waste rock material. This produces a classification system that is sufficiently sensitive to the indicators of metal leaching and acid generation as defined by the characterization program, but simple enough for operational waste management.

Waste rock from the Mt. Hamilton project can be classified into the following two waste rock management categories based on material type as defined by rock type and oxidation:

- Not Potentially Acid Generating (Non-PAG); and
- Potentially Acid Generating (PAG).

A summary of the waste rock classification for waste rock management is provided in Table 18.3.2.1.

**Table 18.3.2.1: Summary of Predicted Waste Rock Geochemistry**

| Material Type | Percentage of Waste Rock to be Mined (by volume percent) | | | Waste Rock Classification |
|---|---|---|---|---|
| | Centennial Deposit | Seligman Deposit | Total | |
| Hornfels | 31.4 | 40.9 | 33.6 | Non-PAG |
| Skarn | 54 | 20.6 | 46.1 | Non-PAG |
| Intrusive – Ox | 3.2 | 25.8 | 8.6 | Non-PAG |
| Intrusive – Unox | 10.3 | 8.3 | 9.9 | PAG |
| Sedimentary | <1 | 4 | <1 | Non-PAG |
| **Total** | **100** | **100** | **100** | **--** |

Source: SRK, 2014c

The results of the geochemical characterization program demonstrate that the bulk of the Mt. Hamilton waste rock material is Non-PAG (90%) and there is limited potential for metals to mobilize from the waste rock dumps and potentially impact waters of the State. Waste rock classified as PAG is limited to the igneous rock containing unoxidized sulfide, which will comprise approximately 10% by volume of the total waste from the Centennial and Seligman pits. Geochemical modeling completed for the Project demonstrates that PAG waste rock can be blended with Non-PAG waste rock with no resultant degradation of waters of the State of Nevada.

Based on these conclusions, segregation of PAG material is not required (Rob Kuczynski, NDEP, verbal communication). Management of waste rock may be achieved by blending the small amount of PAG waste rock with Non-PAG material. To ensure that the WRDA is constructed as a blended mass, MH-LLC proposes to track and quantify the PAG tonnage mined throughout the life of the mine.

Monitoring and testing of waste rock generated by the Project will be conducted on a quarterly basis, as discussed in the WRMP. If the ongoing monitoring program indicates that greater quantities of acid generating material are encountered than originally predicted (i.e., if more than 10% of the total waste rock is projected to be PAG), then MH-LLC will investigate best practices for management of the waste material at that time, and update the WRMP accordingly.

### 18.3.3 Meteorological Station

The Mt. Hamilton Mine Meteorological Station (Met Station) was installed in 2012, to establish baseline temperature, wind and precipitation parameters at the site. The instruments and data collection are managed by Inter-Mountain Labs Air Science (IML), of Sheridan, Wyoming. Instrumentation management includes data quality assurance audits in the first and third quarters of the year, and instrument calibration in the second and fourth quarters. The instrument was struck by lightning several days after the initial installation. After repairs were completed, a continuous data record began on August 23, 2012.

The following excerpt is from the IML report for the data collected during the first quarter of 2013:

“The Centennial Mine meteorological station is operated in compliance with the guidelines set forth by the Nevada Division of Environmental Protection - Bureau of Air Pollution Control (NBAPC).

The meteorological monitoring system consists of a ten-meter instrumented tower and utilizes a Campbell Scientific CR1000 data logger to continuously measure wind speed, direction, standard deviation of horizontal wind direction, temperature (with $\Delta T$) at 2 and 10 meters, relative humidity, barometric pressure, solar radiation and precipitation. Hourly aggregate parameters are logged by the system.”

The site is located in section 20 of Township 16N and Range 57E. The specific State Plane Nevada East (NAD27) site coordinates are 1,633,844 ft N and 502,651 ft E with an elevation of 7546 ft (MH-LLC).

IML provides quarterly results to MH-LLC in spreadsheet and text files, and in summary reports that include data analysis. Site precipitation results available as of November 2013 were considered to determine the Mean Annual Precipitation value used for the geochemical model.

### 18.3.4 Baseline Studies

Class III archeological surveys have been conducted over all lands proposed for development at the mine site and access areas as well as on private land to be used for processing and administrative infrastructure. No archeological sites identified as eligible for inclusion in the National Register of Historic Places will be impacted on public lands during development or operation.

Prehistoric and historic eligible sites have been identified on private land at the processing facility site and a data recovery plan was developed and approved by the State Historic Preservation Office (SHPO). A mitigation plan was implemented in 2012. A Memorandum of Agreement has been drafted by the USFS, SHPO and MH-LLC that defines a process for review under Section 106 of the National Historic Preservation Act in compliance with NEPA analysis of the site as a connected action. Tribal consultations and SHPO review are ongoing in this process.

All areas to be affected by development and operation have undergone Biological Surveys and those surveys have been updated, where appropriate. The Surveys have been reviewed by the USFS and Biological Evaluations have been finalized and were utilized in the Environmental Assessment for which a Finding of No Significant Impact has been issued.

## 18.4 Operating and Post Closure Requirements and Plans

As part of both the State Water Pollution Control Permit and the USFS Plan of Operations, MH-LLC has submitted a detailed plan for monitoring that is designed to demonstrate compliance with the approved MPO and other federal or state environmental laws and regulations, to provide early detection of potential problems, and to supply information that will assist in directing corrective actions, should they become necessary. The plan includes discussion on water quality in the area; monitoring locations, analytical profiles, and sampling/reporting frequency. Examples of monitoring programs which may be necessary include surface- and ground-water quality and quantity, air quality, revegetation, stability and wildlife mortality.

A process fluid management plan has been submitted and approved as part of the Water Pollution Control Permit. This plan describes the management of process fluids, including the methods to be used for the monitoring and controlling of all process fluids. The plan provides a description of the means to evaluate the conditions in the fluid management system, so as to be able to quantify the available storage capacity for meteoric waters and to define when and to what extent the designed containment capacity may have been exceeded. The management of non-process (non-contact) stormwater around and between process facilities is a necessary part of the Nevada General Permit for Stormwater Discharges Associated with Industrial Activity from Metals Mining Activities (NVR300000), and is detailed in the site-wide Stormwater Pollution Prevention Plan (SWPPP).

A Waste Rock Management Plan, supported by a Rock and Ore Geochemistry Characterization Report was submitted and approved in support of the Water Pollution Control Permit Application. This Plan concludes that no special waste handling procedures are required due to the high neutralizing capacity of the material.

### 18.4.1 Reclamation Bonds

The Project is located on National Forest System Lands administered by the USFS and on private land owned by MH-LLC. Bonding of the project is required by the USFS and by the State of Nevada. Three applications; the MPO, the Nevada Reclamation Permit for surface disturbance on National

Forest System lands, and the Nevada Reclamation Permit for surface disturbance on private land have been submitted.

The BMRR will hold the bond for the Nevada Reclamation Permit (NRP) that is required for activities conducted on private land. The BMRR has indicated that a phased bonding approach will be allowed and the Reclamation Cost Estimate for the first phase of bonding is in preparation.

Reclamation bonding for the mining operations on National Forest System lands will be held by the USFS. The Forest Service has agreed that the bonding will be phased. Application for the first phase of the bonding has been submitted and posting of the bond is expected in Q4 2014.

## 18.5 Social and Community

The Project workforce (including short-term construction contractors) will reside mainly in the towns of Ely, Eureka, Duckwater and the surrounding communities in White Pine, Eureka, or Elko County. White Pine County, where the project is located, is largely rural and has its main population center in Ely. According to the Nevada State Demographer (2014), the population of White Pine County was 10,030 in 2010, up 9.2% from 9,181 in 2000. This population growth has been slow, but steady, mainly because of increased mining activity in the area.

An important part of the income of predominantly rural counties in Nevada, like White Pine, is produced by sales tax and the net proceeds tax on mining activity. Sales tax revenues are collected by the county in which delivery of the goods are taken. For the Mt. Hamilton Project, this would be White Pine County. The median household income in the county rose from US$46,600 in 2000 to US$48,545 in 2010, indicating an increase in personal income for the residents of the county.

Other proposed or existing mining projects in White Pine County include the Pan Gold Mine currently under construction and located about 9 miles west of the Mt. Hamilton project and the Gold Rock Project located near the Pan Mine, both operated by Midway Gold. In neighboring Eureka County, nearby mining projects include General Moly Inc.'s Mt. Hope Project and Barrick's recently closed Ruby Hill Mine.

## 18.6 Mine Closure and Reclamation Cost

After operations cease, solution in the heap leach pad will be recirculated until the rate of flow from the facility can be passively managed through evaporation in the ponds or a combination of evaporation and infiltration. Given the physical characteristics of the ore, heap draindown is expected to conclude within a relatively short period of time compared to the average heap leach operation. This is due to the fact that the permeability of the ore in the pad is much higher than typical.

Waste rock dumps will be regraded. Soil for revegetation is not readily available to accommodate all of the disturbed areas, especially the WRDA. This shortfall in growth media is accounted for in the approved Plan of Operations and the NRP under review.

The Nevada Reclamation Permit (NRP) for private land proposes full revegetation for the heap leach pad and sufficient growth media will be available for this purpose.

All buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase. It is assumed that the majority of exploration disturbance will be mined out. Reclamation and closure activities will be conducted concurrently, to the extent practicable, to reduce the overall reclamation and closure costs, minimize environmental liabilities, and limit bond exposure. At this time, the Upper Cabin Gulch (UCG) waste rock dump is anticipated to be available for reclamation in year 3 of operations.

The revegetation release criteria for reclaimed areas are presented in the "Guidelines for Successful Revegetation for the Nevada Division of Environmental Protection, the Bureau of Land Management, and the U.S.D.A. Forest Service." The revegetation goal is to achieve the permitted plant cover as soon as possible.

Conceptual reclamation and closure methods were used to evaluate the various components of the project to estimate reclamation costs. Version 1.4.16 of the Nevada Standardized Reclamation Cost Estimator (SRCE) was used to prepare this cost estimate for the purpose of this Feasibility Study economic model. The Nevada SRCE is also utilized to calculate the actual phased bonding to be posted with the state and a similar federal version of the program is used for posting bonds with the Forest Service.

The SRCE uses first principles methods to estimate quantities, productivities and work hours required for various closure tasks based on inputs from the user. The physical layout, geometry and dimensions of the proposed project components used in the SRCE calculations for the Feasibility Study were based on the current mine plan and facilities layout. These included current designs for the main project components including the open pit, infrastructure, waste rock facilities, heap leach pad, and process ponds. Equipment and labor costs were conservatively estimated using State and BLM-approved costs.

The closure cost associated with the Mt. Hamilton Project is currently estimated by SRK to be US$8.9 million (including contractor profit without contingency). This total is the undiscounted MH-LLC cost to reclaim and close the facilities associated with the mining and processing operation. The major elements of the bond cost estimate were: 1) regrading and stabilization of the waste rock dumps; 2) regrading and seeding the heap leach pad; and, 3) leach pad fluid management and monitoring.

# 19 Capital and Operating Costs (Item 21)

## 19.1 Capital Cost Estimates

A summary of total estimated capital expenditures for Mt. Hamilton is presented in Table 19.1.1.

**Table 19.1.1: Capital Cost Summary**

| Initial Capital Cost Item | Cost US$000's |
|---|---|
| Mining | $17,837 |
| Processing | $25,380 |
| Leach Pad | $7,401 |
| Owner and Infrastructure | $32,116 |
| Contingency | $9,011 |
| **Initial Capital Total** | **$91,745** |
| Sustaining Capital | $17,197 |
| Closure Costs | $8,815 |
| Contingency | $3,760 |
| **LoM Total Capital** | **$121,518** |

Source: SRK, 2014

The support for this cost estimate is provided in the sections below.

### 19.1.1 Basis for Capital Cost Estimates

Capital costs used in the 2014 FS for Mt. Hamilton were based heavily on vendor and specialist quotations. A total of 98% of mining equipment, 94% of process, and 78% of owner and infrastructure capital costs are linked to vendor quotes. SRK has applied addition contingencies to these estimates for omissions. Similarly, operating costs, as driven by consumables or labor rates were supported by recent relevant vendor information or public domain mining services cost providers, typically InfoMine®.

The size of the mining equipment was based on matching the projected mine life to an equipment life cycle of 30,000 to 40,000 hours, which equates to about 7 to 8 years of continuous mining operation. A determination was made that a single equipment spread, consisting of one loading unit and a fleet of 100 t trucks would be used. A hydraulic shovel was selected as the primary loading unit due to its ability to selectively separate ore and waste on a bench. A large wheel loader was selected as the back-up loading unit. The wheel loader would also be used to feed the crusher when ore from the pit was not available to directly dump ore into the primary crusher hopper.

Once the mine layout, including pit design, haulage roads, dump and crusher locations were determined, the haulage cycles were determined and the number of trucks required to make the scheduled production was calculated. Initially five haul trucks are required, with two more trucks added to the fleet at the start of production.

Support equipment required for a 100 t truck fleet was based on experience of similar sized mines with similar loading and hauling fleets. One Caterpillar D9 and one D10 size dozers were selected to maintain the dumps and for cleanup in the pit. A Caterpillar 14 size motor grader will be used to maintain the roads and remove snow. An 8,000 gal water truck was sized to maintain dust control on the haul roads.

Metallurgical testing indicated that high recovery was possible by crushing the ore and using a heap leach process. A number of locations for the leach pad and methods to get the ore to these locations were explored. It was determined that best location to operate the leach pad was on a parcel of private land located in the valley approximately 1,600 ft in elevation below the pit. To get the ore to the leach pad, the ore will be dropped down a vertical ore pass to a conveyor in an underground drift. The crushing and conveying system was sized to handle a mining production rate of 3.5 Mt/y.

The ADR processing plant was sized to meet the expected gold and silver values recovered from the leach pad based on tons of ore placed, the leaching cycle time, and the anticipated metal recovery from column leach tests.

The flow rate capacity of the ADR plant will be 3,000 gpm. The flow rate will allow 750,000 $ft^2$ of heap area to be under leach. Provision has been made in the heap design to recirculate low-grade solutions for an additional 200,000 $ft^2$ of heap area. The ADR (carbon) plant acid wash, and desorption systems were designed to handle silver to gold ratios of up to 6:1. The ADR plant will contain a mercury retort and all mercury control systems as currently required by the State of Nevada regulations.

### 19.1.2 Mining Capital

**Mining Equipment**

The Owner intends to lease all of the major mining equipment. The lease costs were included in the mining cost. A pre-payment (deposit), due at the beginning of the lease period, was included the capital. All leased equipment was priced as new equipment. Table 19.1.2.1 shows a comparison of the purchase cost of the leased equipment, the monthly lease payments and residuals. All mining equipment was priced with options commonly specified for mining operations, including fire suppression systems. Purchase price and lease payments were supplied by the equipment suppliers. Purchase price included taxes, delivery and assembly.

**Table 19.1.2.1: Primary Equipment Capital Unit Costs**

| Equipment | Number of Units | Unit Capital Cost US$000's (each) | Monthly Lease Payment (US$) (60 Month Term) | Residual Payment US$ |
|---|---|---|---|---|
| Atlas Copco DM45 | 1 | $1,214 | $22,824 | $0 |
| Atlas Copco T45 | 1 | $798 | $13,984, | $0 |
| | | | **Average Quarterly Lease Payment (US$) (60 Month Term)** | **Initial Capital Payment** |
| CAT 6030FS | 1 | $4,858 | $217,720 | $971,680 |
| CAT 992K | 1 | $2,553 | $114,435 | $510,617 |
| CAT 777F | 7 | $1,777 | $557,797 | $2,488,448 |
| CAT D10T | 1 | $1,681 | $75,341 | $336,102 |
| Cat D9T | 1 | $1,205 | $54,057 | $241,089 |
| CAT 14M | 1 | $557 | $25,027 | $111,495 |

Source: SRK, 2014

Other Mining Equipment

Other support equipment that will be required include a fuel/lube truck, two mechanics trucks, an 8,000 gal water truck, and light plants. These items will also be leased preconstruction and are listed in Table 19.1.2.2.

**Table 19.1.2.2: Support Equipment Unit Costs**

| Equipment | Number of Units | Capital Cost US$000's (each) | Average Quarterly Lease Payment (US$) (60 Month Term) | Initial Capital Payment (US$) |
|---|---|---|---|---|
| Fuel/Lube Truck | 1 | $282 | $16,218 | $72,176 |
| Mechanics Truck | 2 | $361 | $28,390 | $112,896 |
| Light Plant | 6 | $11 | $3,341 | $13,573 |
| Cat 740 Water Truck | 1 | $845 | $37,889 | $168.913 |

Source: SRK, 2014

It is assumed that the blasting is to be performed by a contractor. The contractor will supply explosive magazines, prill silos, ANFO loading truck and a skid steer loader to stem holes. Capital costs for these items are not included.

In a similar fashion, it is assumed that mine tire supplier will also supply a tire truck on an "as needed" basis as part of the tire supply contract.

A 5% contingency was added to mine equipment.

**Mine Development and Pre-stripping**

Preproduction mining was broken into Development and Pre-Stripping material and Pre-Production Mining. Development and Pre-Stripping involved building a number of access roads needed to initiate mining. These roads were described in Section 14.2.1.

Pre-Stripping material included tons that were to be mined before full production rates were achieved and before the front shovel begins operating. During this period a total of 4.9 Mt of Waste and 97.6 kt of ore will be mined.

Pre-Production Mining includes 2.2 Mt of waste and 131.4 kt of ore. This material will be primarily moved with the front shovel before production begins (first gold pour). Work prior to gold production was identified as a capital expense. The cost for Company miners to remove all preproduction material is estimated at US$12.56 million. A 10% contingency was added to the pit development and pre-stripping capital.

### 19.1.3 Process Capital

Process capital will include the cost to purchase and install the process components of this project, including crushing and conveying equipment, leach pad construction, ADR plant, mobile equipment required for the process plant and maintenance equipment. Details of these items are supplied in Section 15 of this report. A summary of the process capital cost is included in Table 19.1.3.1 for initial and sustaining capital.

**Table 19.1.3.1: Process Capital Cost Summary**

| Capital Summary | Initial Capital (US$000's) | Sustaining Capital (US$000's) |
|---|---|---|
| Primary Crusher | 3,145 | 0 |
| Drift - Mechanical and Process | 3,565 | 0 |
| Secondary Crush and Convey (Drift to Leach pad) | 6,471 | 0 |
| Electrical | 1,094 | 0 |
| Leach Pad Conveyor and Piping | 2,116 | 799 |
| ADR | 8,805 | 931 |
| Process Mobile Equipment | 184 | 175 |
| **Process Total** | **$25,380** | **$1,904** |

Source: SRK, 2014

Process capital was developed using supplier quotes for process components and contractor quotes for installation to make a complete working system. Process plant engineering and costs were developed as a turnkey package in August, 2013 by Kappes Cassiday & Associates, a Nevada mining contractor with recent experience designing and constructing ADR plants. Sources of the quotes for the support elements of processing included: 1) a civil contractor for the earthworks quotes; and 2) a mechanical contractor to install the components and supply the buildings for the process components. SRK allowances and estimates totaled approximately 6% of the total process capital cost.

Process Mobile Equipment capital cost is based on leasing the equipment. The total purchase price of this equipment, including sales tax and delivery, would total US$1.8 million if directly purchased. Lease payments were included in operating costs.

A contingency was applied to each of these items depending on the detail of the underlying engineering and level of confidence of the completeness of the items and their construction or application. Equipment supported by vendor quotations received a 5% contingency. If the work element contained a mix of contractor quotations and SRK estimates, a contingency of 10% was assigned. The average contingency for process capital expenditure averaged 8.2%.

The leach pad will be constructed in four phases, with Phases III and IV being built simultaneously. Table 19.1.3.2 lists the capital costs for the individual phases.

**Table 19.1.3.2: Leach Pad Capital Cost**

| Capital Summary | Year Constructed | Phase Size($ft^2$) | Capital (US$000's) |
|---|---|---|---|
| Phase I (Initial Capital) | 0 | 1,196,000 | $7,401 |
| Phase II | 1 | 1,226,000 | $5,562 |
| Phase III & IV | 2 | 1,240,000 | $5,931 |
| **Total** | | **4,343,000** | **$18,894** |

Source: SRK, 2014

Leach pad costs were developed by a Nevada mining contractor with recent experience constructing leach pads.

A 15% contingency was added to the leach pad capital estimate due to the steep terrain and the requirement for an underliner soil amendment. The cost estimate for the underliner was based on an amended soil rather than a sourced native clay-rich soil. The amended soil represents the more expensive alternative; however, if a local low permeability soil can be used this will provide a cost savings.

### 19.1.4 Infrastructure and Owners Capital

The major components of the Owner and Infrastructure capital are shown in Table 19.1.4.1. Owner and Infrastructure capital costs are shown in Table 19.1.4.2. The costs include light vehicles for administration and production, administration and warehouse buildings and site development. Pre-production activities consist mainly of permitting, technical studies and Owner overheads during construction.

**Table 19.1.4.1: Major Components of Owner and Infrastructure Capital**

| Item | Size/Description | Max Required |
|---|---|---|
| **Drift Construction** | | |
| Ventilation Fan | 100 HP | 1 |
| Drift and Raise Construction | 15 ft H x 12 ft W | 4,396 ft |
| **Water Supply System** | | |
| Water well | Including downhole pump, booster pump and tank | 1 |
| Peerless Submersible Pumps | Well pump, 75 hp | 1 |
| Peerless Booster Pump | 125 hp | 1 |
| Tanks and installation | 750,000 gal main storage, misc. smaller tanks | 1 |
| **Power System** | | |
| 455 kW Generator Set | For primary crusher and truck shop power | 2 |
| Generators, 150 kW | Portable standby power for truck shop | 1 |
| 725 kW Generator Set | for operating secondary crusher, ADR and all lower elevation facilities | 4 |
| Power lines | Installed | 4.5 miles |
| **Other Infrastructure** | | |
| **Buildings** | | |
| Admin Building | 3,150 $ft^2$ | 1 |
| Mine Maintenance | 150 ft x 72 ft shop area with 112 x 13 office area | 1 |
| Ancillary Areas | Includes 65 ft x 45 ft lab building, 5,500 $ft^2$ warehouse/maintenance bldg., utilities and piping | 1 |
| **Owners Cost** | | |
| Access Road Development | | |
| County Road Upgrade | | |
| GPS Survey Equipment | | |
| Radio System | | |
| Laboratory Equipment and Supply | | |
| EPCM | | |
| Freight for Crusher and Conveyor System | | |
| Contractor Overhead and Profit | | |
| **Light Vehicles** | | |
| Pickup trucks-Extended Cab | | 8 |
| Pickup trucks-Standard Cab | | 8 |
| Pickup trucks-Crew Cab | | 5 |
| Staff Commuter Vans | | 5 |
| **Mobile Equipment** | | |
| Caterpillar 14M Motor Grader | 230 hp, 14 ft blade | 1 |
| Caterpillar P5000-LE Fork Lift | 63 hp, 5,000 lb lifting capacity | 1 |
| 4,000 gal water Truck | | 1 |
| Fuel truck small | | 1 |
| **Initial Fills** | | |
| **Pre-Production Activities** | | |
| G&A Pre-production | | |
| Process Pre-Production | | |
| Technical Studies | | |
| Admin+Permitting | | |

Source: SRK, 2014

**Table 19.1.4.2: Owner and Infrastructure Capital Cost Summary**

| Capital Cost Item | Initial Capital US$000's | Sustaining Capital US$000's |
|---|---|---|
| Drift Construction | 10,686 | 0 |
| Water Supply System | 1,947 | 0 |
| Power System | 1,603 | 2,866 |
| Other Infrastructure | 5,159 | 0 |
| Owners Costs | | |
| Other construction and purchases | 1,073 | 0 |
| EPCM and Contractor Costs | 5,631 | 355 |
| Light vehicles and mobile equipment | 279 | 364 |
| Initial Fills | 533 | 0 |
| Preproduction Operations and Permitting | 5,204 | 215 |
| **Owner and Infrastructure Total** | **$32,116** | **$3,800** |
| Mine Reclamation | - | $8,815 |

Source: SRK, 2014

Infrastructure and owners capital was developed using supplier quotes for components, a civil contractor for the earthworks, an underground contractor for the conveyor drift and a mechanical contractor to install the components and supply the buildings. SRK allowances and estimates totaled approximately 22% of the total Infrastructure and Owner Capital cost.

A contingency was applied to each of these items depending on the detail of the underlying engineering and level of confidence of the completeness of the items and their construction or application. Equipment supported by vendor quotations received a 5% contingency. If the work element contained a mix of contractor quotations and SRK estimates, a contingency of 15% was assigned. The average contingency for process capital expenditure averaged 9.8%.

Mine closure capital was developed using the Standardized Reclamation Cost Estimator (SRCE) the Nevada State-approved method of calculating reclamation bonds. The "in-ground" reclamation cost including contractor profit was US$8.8 million. A 15% contingency was added to the mine closure capital.

## 19.2 Operating Cost Estimates

Total operating cost estimates for the Project are presented in Table 19.2.1. The unit operating costs are based on total mined material of 88,468 kt of which 65,968 kt is waste material, which includes 1,075 kt of rehandled waste, and 22,500 kt is ore. Operating costs include only those activities that occurred after commencing metal production. The estimated mine life is 7 years.

**Table 19.2.1: Operating Cost Summary**

| Operating Costs | (US$000) | US$/t-ore |
|---|---|---|
| Mining | $134,740 | $5.99 |
| Processing | $92,427 | $4.11 |
| G&A | $18,863 | $0.84 |
| **Total Operating** | **$246,029** | **$10.93** |

Source: SRK, 2014

### 19.2.1 Basis for Operating Cost Estimates

Mining costs were dictated by the equipment selected and the conditions of the mine environment. Maintenance cost for most of the mining equipment was supplied by the equipment supplier. Infomine® CostMine™ data was used to determine other equipment hourly costs such as tires and wear components. When maintenance costs were not available from suppliers, CostMine™ was used to for estimating these costs. CostMine™ was used for determining hourly wage rates. Both CostMine™ and confidential sources from nearby operating mines were used for supervisory, technical and administrative salaries. The equipment productivities were determined from published manufacturer's data. These factors were treated in a conservative manner to reflect the difficulties of operating at over 9,000 ft elevation in rural Nevada.

Processing costs were developed from: 1) wage rates from similar projects in Nevada; 2) reagent consumption as determined by site-specific test programs or industry standards and current prices; and 3) wear and replacement parts by testing or manufactures recommendations.

The process staffing plan allows for the climatic conditions and the wide separation of the processing units.

The supervisory and administrative support staff was sized to efficiently handle the administrative, technical and management functions required for the proposed operation. Provisions for training, and regulatory mandated safety functions were also included.

Lease initial and residual payments are included in capital costs. Lease payments, including sales tax, are included in the operating cost.

### 19.2.2 Operating Costs - Mining

Mining equipment operating costs, on a US$/hour basis, were developed from equipment supplier information and Infomine® CostMine™ Surface Mining Equipment cost guide. Operating cost included fuel and lube, tires, overhaul and maintenance parts and wear items and Diesel fuel at US$3.20/gal. A breakdown of equipment costs are shown in Table 19.2.2.1.

**Table 19.2.2.1: Operating Costs for Primary Mining Equipment**

| Equipment | Fuel (gph) | Lube (gph) | Tires (US$/hr) | Repair and Overhaul (US$/hr) | Wear Items (US$/hr) |
|---|---|---|---|---|---|
| Atlas Copco DM45 | 22.0 | 0.9 | $0.00 | $48.22 | $18.32 |
| Atlas Copco T45 | 11.3 | 0.8 | $0.00 | $29.75 | $9.74 |
| CAT 6030FS | 53.7 | 3.0 | $0.00 | $135.30 | $11.39 |
| CAT 992K | 24.3 | 1.1 | $61.38 | $53.85 | $1.13 |
| CAT 777G | 18.1 | 0.8 | $26.59 | $39.93 | $0.00 |
| CAT D9T | 13.8 | 0.6 | $0.00 | $59.34 | $15.22 |
| CAT D10T | 19.3 | 0.7 | $0.00 | $48.94 | $16.33 |
| CAT 14M | 8.8 | 0.3 | $2.15 | $25.52 | $1.01 |
| Cat 740 Water Truck | 7.9 | 0.5 | $11.48 | $26.59 | $0.00 |

Source: SRK, 2014

Labor rates for mining are shown in Table 19.2.2.2.

**Table 19.2.2.2: Labor Rates Mining**

| Job Classification | Average Number Required | Base Rate (US$/hr) | Hours per Year | Base (US$/yr) | Burden (%) | Overtime Factor (%) |
|---|---|---|---|---|---|---|
| **Salary** | | | | | | |
| Mine Superintendent | 1 | - | - | $105,000 | 40.0% | 0.0% |
| Chief Engineer | 1 | - | - | $118,000 | 40.0% | 0.0% |
| Mining Engineer | 1 | - | - | $79,500 | 40.0% | 0.0% |
| Geologist | 2 | - | - | $72,500 | 40.0% | 0.0% |
| Surveyor | 2 | - | - | $46,000 | 40.0% | 0.0% |
| Mine Foreman | 4 | - | - | $72,500 | 40.0% | 0.0% |
| Maintenance Supervisor | 1 | - | - | $79,500 | 40.0% | 0.0% |
| **Hourly** | | | | | | |
| Driller | 4 | $26.60 | 2,080 | $55,328 | 40.0% | 9.0% |
| Loader Operator | 8 | $29.15 | 2,080 | $60,632 | 40.0% | 9.0% |
| Truck Driver | 19 | $23.05 | 2,080 | $47,944 | 40.0% | 9.0% |
| Equipment Operator | 14 | $27.05 | 2,080 | $56,264 | 40.0% | 9.0% |
| Laborer | 1 | $20.25 | 2,080 | $42,120 | 40.0% | 9.0% |
| Lead Mechanic | 1 | $33.30 | 2,080 | $69,264 | 40.0% | 9.0% |
| Mechanic | 8 | $27.80 | 2,080 | $57,824 | 40.0% | 9.0% |
| Electrician | 1 | $28.90 | 2,080 | $60,112 | 40.0% | 9.0% |
| Maintenance Worker | 4 | $22.90 | 2,080 | $47,632 | 40.0% | 9.0% |
| **Annual Mine Labor Cost** | | | | | | |

Source: SRK, 2014

Yearly mine cost statistics are shown in Table 19.2.2.3. Operating costs average US$1.66/t, when lease cost is included. Without lease cost, LoM costs average US$US1.38/t.

**Table 19.2.2.3: Mine Operating Costs Summary**

| Mining | Unit | Total | 1 | 2 | 3 | 4 | 5 | 6 | 7 |
|---|---|---|---|---|---|---|---|---|---|
| Drill & Blast | **US$000's** | 32,638 | 5,840 | 5,597 | 5,515 | 5,019 | 5,964 | 4,466 | 236 |
| Loading | **US$000's** | 15,844 | 2,763 | 2,758 | 2,817 | 2,783 | 2,598 | 1,885 | 241 |
| Hauling | **US$000's** | 29,560 | 5,469 | 4,714 | 4,648 | 4,745 | 5,394 | 4,406 | 184 |
| Roads & Dumps | **US$000's** | 19,193 | 3,264 | 3,380 | 3,159 | 3,312 | 3,289 | 2,248 | 541 |
| Stockpile Rehandle | **US$000's** | 2,077 | 136 | 297 | 0 | 0 | 651 | 315 | 678 |
| Mine Support | **US$000's** | 2,598 | 443 | 453 | 451 | 455 | 447 | 300 | 49 |
| Mine G&A Labor | **US$000's** | 9,695 | 1,591 | 1,619 | 1,619 | 1,619 | 1,619 | 1,242 | 386 |
| Mine Opex | **US$000's** | **111,606** | **19,505** | **18,818** | **18,210** | **17,933** | **19,962** | **14,862** | **2,315** |
| **Total US$/t** | | **1.38** | **1.36** | **1.32** | **1.21** | **1.20** | **1.51** | **1.61** | **18.11** |
| Lease Cost | **US$000's** | 23,135 | 4,937 | 4,949 | 4,963 | 4,977 | 3,308 | 0 | 0 |
| Mine Cost | **US$000's** | **134,740** | **24,442** | **23,768** | **23,173** | **22,910** | **23,271** | **14,862** | **2,315** |
| **Total US$/t** | | **1.66** | **1.71** | **1.67** | **1.54** | **1.53** | **1.76** | **1.61** | **18.11** |

Source: SRK, 2014

Detailed mine operating costs, by area, are shown in Table 19.2.2.4.

**Table 19.2.2.4: Detailed Mining Operating Costs**

| Item | US$000's | $/t-mined | $/t Ore |
|---|---|---|---|
| Drilling & Blasting | $32,638 | $0.40 | $1.29 |
| Loading | $15,844 | $0.20 | $0.63 |
| Hauling | $29,560 | $0.36 | $1.17 |
| Roads & Dumps | $19,193 | $0.24 | $0.76 |
| Stockpile Rehandle | $2,077 | $0.03 | $0.08 |
| Mine Support | $2,598 | $0.03 | $0.10 |
| Mine G&A Labor | $9,695 | $0.12 | $0.38 |
| Leasing Cost | $23,135 | $0.29 | $0.92 |
| **Total Mining Cost** | **$134,740** | **$1.66** | **$5.99** |

Source: SRK, 2014

### 19.2.3 Operating Costs – Processing

The major processing cost elements include: labor, power, consumables, maintenance materials and other for the processing areas of crushing, leaching, ADR, laboratory and administration. The life of mine operating cost to process 22.5 Mt of ore is US$92.9 million or 4.130/t ore processed. The life-of- mine process cost by major element is provided in Table 19.2.3.1.

**Table 19.2.3.1: LoM Major Element Process Cost**

| Item | LoM (US$000) | US$/t-ore |
|---|---|---|
| Labor | $32,803 | $1.46 |
| Power | $10,382 | $0.46 |
| Consumables | $30,324 | $1.35 |
| Maintenance Materials | $9,480 | $0.42 |
| Other | $9,438 | $0.42 |
| **Total Processing Cost** | **$92,427** | **$4.11** |

Source: SRK, 2014

Table 19.2.3.2 provides the annual operating cost by area for a full year of production, 3.5 Mt, with power supplied by the utility company. The full year full production cost is estimated at US$13.1 million or US$3.77/t ore processed.

**Table 19.2.3.2: Annual Cost by Process Area**

| Item | LoM (US$000) | US$/t-ore |
|---|---|---|
| Crushing | $2,740 | $0.783 |
| Leaching | $6,789 | $1.940 |
| ADR | $1,941 | $0.555 |
| Laboratory | $797 | $0.228 |
| Administration | $926 | $0.265 |
| **Total Processing Cost** | **$13,194** | **$3.770** |

Source: SRK, 2014

Process labor rates are built up from base rates to which a 40% burden factor has been applied. In addition SRK assumes an average 9% overtime for hourly job classifications. Base labor rates were based on the 2013 CostMine data for Nevada. The full year full production labor cost is estimated at US$4.74 million per year which equates to US$1.36/t-processed. Table 19.2.3.3 provides the annual labor cost for a full production year.

**Table 19.2.3.3: Labor**

| Job Classification | Average Number Required | Base Rate (US$/hr) | Base (US$/yr) | Burden (%) | Overtime (%) | Annual Cost ($) |
|---|---|---|---|---|---|---|
| **Salary** | | | | | | |
| Shift Foremen | 4 | - | $77,000 | 40.00% | 0.00% | $431,200 |
| Plant Superintendent | 1 | - | $112,000 | 40.00% | 0.00% | $156,800 |
| Metallurgist | 1 | - | $97,000 | 40.00% | 0.00% | $135,800 |
| Clerk | 1 | $19,00 | $39,520 | 40.00% | 0.00% | $ 55,328 |
| **Hourly** | | | | | | |
| **24X7 Schedule** | | | | | | |
| Primary Crusher Operator | 4 | $24.00 | $53,202 | 40.00% | 8.00% | $304,317 |
| Underground Conveyor Operator | 4 | $24.00 | $53,202 | 40.00% | 8.00% | $304,317 |
| Secondary Crusher Operator | 4 | $24.00 | $53,202 | 40.00% | 8.00% | $304,317 |
| Surface Conveyor Operator | 4 | $20.00 | $44,139 | 40.00% | 8.00% | $252,473 |
| Utility Operator | 8 | $27.00 | $59,077 | 40.00% | 8.00% | $668,754 |
| Plant Operator | 4 | $28.00 | $61,152 | 40.00% | 8.00% | $349,789 |
| Plant Helper | 4 | $20.00 | $44,139 | 40.00% | 8.00% | $252,473 |
| **5x10 Schedule** | | | | | | |
| Heap Piping | 4 | $24.00 | $53,202 | 40.00% | 8.00% | $304,317 |
| Labor | 1 | $21.00 | $55,146 | 40.00% | 8.00% | $ 83,822 |
| Refiner | 1 | $28.00 | $72,800 | 40.00% | 8.00% | $ 110,656 |
| **Assay Laboratory** | | | | | | |
| Assayers | 3 | $24.00 | $53,202 | 40.00% | 8.00% | $229,834 |
| Technicians | 1 | $20.00 | $52,546 | 40.00% | 8.00% | $79,870 |
| Sample Preparation | 3 | $20.00 | $44,139 | 40.00% | 8.00% | $190,679 |
| **Maintenance** | | | | | | |
| Mechanics/Electricians | 2 | $28.00 | $73,762 | 40.00% | 8.00% | $215,385 |
| Helpers | 3 | $23.00 | $59,540 | 40.00% | 8.00% | $257,213 |
| **Total Labor** | | | | | | **$4,745,308** |

Source: SRK, 2014

Power costs for the first two years and two months of production are generated on site until a utility power line can be constructed. Generated power cost is estimated at US$0.252/kWh with a delivered fuel price of US$3.20/gal. The utility power cost estimate, provided by the utility company, including peak demand is estimated US$0.064 kWh. The power consumption is based on a utilization and demand factor for each installed nameplate motor. Table 19.2.3.4 provides the annual power cost for generated and utility power and the power consumption.

**Table 19.2.3.4: Power**

| Item | Cons. (kWh) | Generated (US$000) | Generated (US$/t) | Utility (US$000) | Utility (US$/t) |
|---|---|---|---|---|---|
| Crushing | 577 | $1,275 | $0.349 | $322 | $0.088 |
| Leaching | 612 | $1,352 | $0.371 | $341 | $0.094 |
| ADR | 432 | $954 | $0.262 | $241 | $0.066 |
| Laboratory | 34 | $76 | $0.021 | $19 | $0.005 |
| **Total Power** | **$1,655** | **$3,657** | **$1.002** | **$923** | **$0.253** |

Source: SRK, 2014

The two major reagents consumed in the process are lime and sodium cyanide. Table 19.2.3.5 provides a full year cost of consumables for a full production year. The unit prices are delivered and include tax.

**Table 19.2.3.5: Consumables**

| Consumable | Consumption (lb/t) | Unit Cost (US$/lb) | (US$000) | (US$/t) |
|---|---|---|---|---|
| Lime (CaO) | 4.0 | 0.10 | $1,346 | 0.385 |
| Sodium Cyanide | 0.6 | 1.46 | $3,074 | 0.878 |
| Other consumables (1) | | | $255 | 0.073 |
| Lab Chemicals | | | $143 | 0.041 |
| **Total Consumables** | | | **4,819** | **1.377** |

Source: SRK, 2014

(1) Other reagents include, antiscalant, HCl, NaOH, carbon, propane and fluxes.

Annual maintenance material cost is estimated as a percentage of the equipment capital cost. Table 19.2.3.6 provides the full year full production cost by area and factors.

**Table 19.2.3.6: Maintenance Materials**

| Item | Equipment Capital ($M) | Factor | (US$000) | (US$/t) |
|---|---|---|---|---|
| Crushing | 8.0 | 12% | $965 | $0.276 |
| Leaching | 0.3 | 10% | $31 | $0.009 |
| ADR | 4.6 | 10% | $464 | $0.133 |
| Laboratory | 1.0 | 5% | $48 | $0.014 |
| **Total Materials** | | | **$1,509** | **$0.431** |

Source: SRK, 2014

Other costs include administration office costs, safety supplies, training and travel, consultants, maintenance contracts, light vehicle operating costs and a mercury disposal cost during the production period. The total full year full production cost is US$0.6 million or US$0.165/t ore processed.

An equipment lease allocation is also included in this category at an estimated full year full production cost of $0.99 million or $0.283/t ore processed.

A leased heap leach operations dozer is also included in the other category at an estimated full year full production cost of US$0.7 million or US$0.188/t ore processed.

## 19.2.4 General and Administrative Cost

General and Administrative costs average US$0.84/t ore crushed. General and administrative costs are summarized in Table 19.2.4.1.

**Table 19.2.4.1: G&A Operating Cost Summary**

| Item | US$000's | US$/t-total | US$/t-ore |
|---|---|---|---|
| G&A Costs | $8,578 | $0.11 | $0.38 |
| G&A Labor | $10,285 | $0.13 | $0.46 |
| **Total Operating Cost** | **$18,863** | **$0.21** | **$0.84** |

Source: SRK, 2014

Table 19.2.4.2 shows the breakdown of costs by cost area. G&A costs include lease payments on the generators, all highway vehicles and other G&A equipment. It also includes the finance costs associated with the leases.

**Table 19.2.4.2: G&A Costs**

| Item | LoM Total (US$000) | Production Years | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | 6 | 7 |
| Environmental, Compliance | 169 | 25 | 25 | 25 | 25 | 25 | 25 | 19 |
| Equipment Operation | 760 | 113 | 113 | 113 | 113 | 113 | 113 | 84 |
| Insurance | 2,025 | 300 | 300 | 300 | 300 | 300 | 300 | 225 |
| Licensing & Fees | 203 | 30 | 30 | 30 | 30 | 30 | 30 | 23 |
| Power Line Maintenance | 203 | 30 | 30 | 30 | 30 | 30 | 30 | 23 |
| Communications | 243 | 36 | 36 | 36 | 36 | 36 | 36 | 27 |
| Rentals/leases | 3,302 | 684 | 710 | 542 | 500 | 463 | 287 | 116 |
| Safety Supplies | 243 | 36 | 36 | 36 | 36 | 36 | 36 | 27 |
| Office/Training Supplies | 81 | 12 | 12 | 12 | 12 | 12 | 12 | 9 |
| Software/Computers | 608 | 90 | 90 | 90 | 90 | 90 | 90 | 68 |
| Small Vehicles | 68 | 10 | 10 | 10 | 10 | 10 | 10 | 8 |
| Finance Fees | 0 | 0 | 0 | 0 | 0 | 0 | - | - |
| Outside Services | 675 | 100 | 100 | 100 | 100 | 100 | 100 | 75 |
| **G&A Costs** | **8,578** | **1,465** | **1,492** | **1,324** | **1,282** | **1,244** | **1,068** | **702** |
| $/t crushed | 0.38 | 0.50 | 0.43 | 0.38 | 0.37 | 0.36 | 0.31 | 0.37 |

Source: SRK, 2014

Table 19.2.4.3 and Table 19.2.4.4 show the G&A labor rates and yearly labor costs, respectively.

**Table 19.2.4.3: G&A Labor Rates**

| Job Classification | Average Number Required | Base Rate (US$/hr) | Hours per Year | Base (US$/yr) | Burden (%) | Overtime (%) |
|---|---|---|---|---|---|---|
| **Salaried** | | | | | | |
| General Manager | 1 | - | - | $165,000 | 40.0% | 0.0% |
| Accountant | 1 | - | - | $72,500 | 40.0% | 0.0% |
| Purchasing Agent | 1 | - | - | $58,500 | 40.0% | 0.0% |
| Environmental Manager | 1 | - | - | $105,000 | 40.0% | 0.0% |
| Safety Engineer | 1 | - | - | $72,500 | 40.0% | 0.0% |
| Human Resources Manager | 1 | - | - | $79,500 | 40.0% | 0.0% |
| **Hourly** | | | | | | |
| Technician | 3 | $25.30 | 2,080 | $52,624 | 40.0% | 5.0% |
| Clerk | 1 | $19.00 | 2,080 | $39,520 | 40.0% | 2.0% |
| Secretary | 1 | $19.00 | 2,080 | $39,520 | 40.0% | 2.0% |
| Security Guard | 6 | $20.25 | 2,080 | $42,120 | 40.0% | 7.0% |
| Janitor | 1 | $17.75 | 2,080 | $36,920 | 40.0% | 2.0% |

Source: SRK, 2014

**Table 19.2.4.4: G&A Labor**

| G&A Labor | LoM Total US$000's | Production Years | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | 6 | 7 |
| General Manager | 1,559 | 231 | 231 | 231 | 231 | 231 | 231 | 173 |
| Accountant | 685 | 102 | 102 | 102 | 102 | 102 | 102 | 76 |
| Purchasing Agent | 553 | 82 | 82 | 82 | 82 | 82 | 82 | 61 |
| Environmental Manager | 992 | 147 | 147 | 147 | 147 | 147 | 147 | 110 |
| Human Resources Manager | 751 | 111 | 111 | 111 | 111 | 111 | 111 | 83 |
| Safety Engineer | 685 | 102 | 102 | 102 | 102 | 102 | 102 | 76 |
| Technician | 1,507 | 229 | 229 | 229 | 229 | 229 | 229 | 134 |
| Clerk | 351 | 56 | 56 | 56 | 56 | 56 | 56 | 14 |
| Secretary | 379 | 56 | 56 | 56 | 56 | 56 | 56 | 42 |
| Security Guard | 2,508 | 371 | 371 | 371 | 371 | 371 | 371 | 279 |
| Janitor | 315 | 52 | 52 | 52 | 52 | 52 | 52 | - |
| **G&A Labor Total** | **10,285** | **1,539** | **1,539** | **1,539** | **1,539** | **1,539** | **1,539** | **1,049** |
| $/t Crushed | 0.46 | 0.53 | 0.44 | 0.44 | 0.44 | 0.44 | 0.44 | 0.55 |

Source: SRK, 2014

# 20 Economic Analysis (Item 22)

The financial results of this report have been prepared on an annual basis. All costs are in Quarter 3 2014 US dollars.

## 20.1 Principal Assumptions

A financial model was prepared on an unleveraged, pre- and post-tax basis the results of which are presented in this section. Key criteria used in this analysis are discussed in detail throughout this report. Financial assumptions used in this analysis are shown summarized in Table 20.1.1.

**Table 20.1.1: Financial Assumptions for Economic Modeling**

| Model Parameter | Technical Input |
|---|---|
| Pre-Production Period | 0.75 years |
| Mine Life | 7 years |
| Life-of-Mine Gold Price (US$/oz) | $1,300 |
| Life-of-Mine Silver Price (US$/oz) | $20.00 |
| Operating Days per Year | 350 |
| Discount Rate | 8.0% |

Source: SRK, 2014

The pre-production period allows for pre-production stripping and facilities construction. The mine will have a seven year life given the Mineral Reserve described in this report.

The analysis assumes static conditions for the gold and silver market price over the seven year mine life. The gold price was set at US$1,300/oz. The silver price was set at US$20.00/oz. These prices are the rounded equivalent of spot metal prices as of August, 2014, which were lower than the 36 month trailing average.

## 20.2 Cash flow Forecasts and Annual Production Forecasts

The economic results, at a discount rate of 8%, indicate a NPV of US$60.8 million with an IRR of 26.0% (after estimated taxes). Payback will be in 2.9 years from the start of production. The following provide the basis of the SRK LoM plan and economics:

- A mine life of 7 years;
- An overall average gold recovery rate of 76%;
- An average operating cost of US$558 /AuEq oz-produced;
- Capital costs of US$121.5 million, comprised of initial capital costs of US$91.7 million, and sustaining capital over the LoM of US$29.8 million;
- Mine closure cost estimate including contingency is US$10.1 million; and
- The analysis does not include any allowance for end of mine salvage value.

Table 20.2.1 Mine Production Summary below shows the LoM production, ore grades and contained metal used in the economic analysis.

**Table 20.2.1: Mine Production Summary**

| Item | Value | Units |
|---|---|---|
| **Mine Production** | | |
| Waste | 63,005 | kt |
| Ore [(1)] | 25,463 | kt |
| **Total Material** | **88,468** | **kt** |
| Stripping Ratio | 2.47 | waste:ore |
| Avg. Daily Ore Capacity | 10,000 | t/d |
| **RoM Grade** | | |
| Gold | 0.024 | oz/t |
| Silver | 0.197 | oz/t |
| **Contained Metal** | | |
| Gold | 606.6 | koz |
| Silver | 5,012.5 | koz |

Source: SRK, 2014
Notes: 1. Ore tonnage includes 2.9Mt of material mined as ore but not processed due to the leach pad capacity limitation.

Table 20.2.2 Process Production Summary shows the LoM process tonnage, recoveries for gold and silver from the heap leach operation and recovered metal used in the economic analysis.

**Table 20.2.2: Process Production Summary**

| Item | Value | units |
|---|---|---|
| **Crushed Ore Leached** | 22,500 | kt |
| Avg. Daily Capacity | 10,000 | t/d |
| **Process Plant** | | |
| **Contained Metal** | | |
| Gold | 545.4 | koz |
| Silver | 4,459.6 | koz |
| **Recovery** | | |
| Gold | 76% | |
| Silver | 39% | |
| **Recovered Metal** | | |
| Gold | 415.8 | koz |
| Silver | 1,742.7 | koz |

Source: SRK, 2014

Table 20.2.3 Project Economic Results shown below contains the calculated project cash flow and NPV at a 5% and 8% discount rate along with the IRR for the project.

**Table 20.2.3: Project Economic Results**

| Description | Units | With Tax | Without Tax |
|---|---|---|---|
| **Market Prices** | | | Without Tax |
| Gold (LoM Avg) | /oz-Au | $1,300 | $1,300 |
| Silver (LoM Avg) | /oz-Ag | $20.00 | $20.00 |
| **Estimate of Cash Flow (all values in US$000's** | | | |
| **Payable Metal** | | | |
| Gold | koz | 415.0 | 415.0 |
| Silver | koz | 1,690.40 | 1,690.40 |
| **Gross Revenue** | - | | |
| Gold | | $539,494 | $539,494 |
| Silver | | $33,808 | $33,808 |
| **Revenue** | | **$573,302** | **$573,302** |
| Refinery & Transport | | ($3,273) | ($3,273) |
| **Gross Revenue** | | **$570,030** | **$570,030** |
| Royalty | | ($17,015) | ($17,015) |
| **Net Revenue** | | **$553,015** | **$553,015** |
| **Operating Costs** | $/t-ore | | |
| Mining | $5.99 | $134,740 | $134,740 |
| Processing | $4.11 | $92,427 | $92,427 |
| G&A | $0.84 | $18,863 | $18,863 |
| Property & Net Proceeds Tax | $0.58 | $12,943 | $12,943 |
| **Total Operating** | **$11.51** | **$258,972** | **$258,972** |
| **Operating Margin (EBITDA)** | | **$294,042** | **$294,042** |
| LoM Capital | | $121,518 | $121,518 |
| Income Tax | | $56,643 | $0 |
| **Cash Flow Available for Debt Service** | | **$115,882** | **$184,760** |
| **NPV 5%** | | **$78,466** | **$131,835** |
| **NPV 8%** | | **$60,817** | **$106,951** |
| **IRR** | | **26.0%** | **35.4%** |

Source: SRK, 2014

Table 20.2.4 Cash Cost contains the LoM cash cost for the project and cost per ton processed based on total revenue, total operating cost and total operating margin.

**Table 20.2.4: Cash Cost**

| Cash Costs | Value | Units |
|---|---|---|
| Gold | $1,300 | per oz |
| Silver | $20.00 | per oz |
| Leached Ore | 22,500 | kt |
| **Gross Revenue** | | |
| Gold | $539,494 | |
| Silver | $33,808 | |
| **Gross Revenue** | **$573,302** | |
| $/t-ore | $25.48 | |
| **Costs** | | |
| Refining and Transport | $3,273 | |
| Royalty | $17,015 | |
| Operating Costs | $258,972 | |
| **Cash Cost** | **$279,260** | |
| $/t-ore | $12.41 | |
| **Margin** | | |
| Operating Margin | $294,042 | |
| $/t-ore | $13.07 | |

Source: SRK, 2014

**Table 20.2.5: Annual Production and Cash flow Summary**

| year | Waste (kt) | Ore (kt)) | Ore Crushed (kt) | Gold Contained Metal (koz) | Silver Contained Metal (koz) | Gold Payable Metal (koz) | Silver Payable Metal (koz) | Silver: Gold ratio | Free Cashflow (US$000) | NPV @ 8% (US$000) |
|---|---|---|---|---|---|---|---|---|---|---|
| -2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -3,660 | -3,660 |
| -1 | 7,091 | 229 | 180 | 3 | 30 | 0 | 0 | | -88,886 | -82,302 |
| 1 | 11,250 | 3,086 | 2,920 | 64 | 614 | 45 | 205 | 4.6 | 194 | 166 |
| 2 | 10,248 | 3,957 | 3,480 | 94 | 614 | 74 | 234 | 3.2 | 41,000 | 32,547 |
| 3 | 9,812 | 5,188 | 3,500 | 107 | 833 | 83 | 320 | 3.8 | 57,707 | 42,416 |
| 4 | 8,292 | 6,708 | 3,500 | 134 | 1,194 | 102 | 444 | 4.3 | 66,034 | 44,942 |
| 5 | 10,963 | 2,294 | 3,500 | 54 | 432 | 43 | 182 | 4.3 | 13,625 | 8,586 |
| 6 | 5,339 | 3,884 | 3,500 | 69 | 536 | 54 | 219 | 4.1 | 28,311 | 16,519 |
| 7 | 11 | 117 | 1,920 | 20 | 206 | 15 | 87 | 5.9 | 8,878 | 4,796 |
| 8 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -200 | -100 |
| 9 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -2,197 | -1,017 |
| 10 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -4,288 | -1,839 |
| 11 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -303 | -120 |
| 12 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -143 | -52 |
| 13 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -190 | -65 |
| 14 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -190 | -65 |
| 15 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -190 | -65 |
| 16 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | -190 | -65 |
| 17 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | 0 | 0 |
| **Total** | **63,005** | **25,463** | **22,500** | **545** | **4,460** | **415** | **1,690** | **30** | **115,310** | **60,623** |
| Annualized Production | | | | | | 68.6 | 279.4 | 5.0 | | |

Source: SRK, 2014

# 20.3 Taxes, Royalties and Other Interests

MH-LLC will be subject to the following taxes as they relate to the Project:

- Federal Income Tax; and
- Net Proceeds Tax

MH-LLC is also subject to royalties as described in Section 20.3.3.

## 20.3.1 Federal Income Tax

Corporate Federal income tax is determined by computing and paying the higher of a regular tax or a Tentative Minimum Tax (TMT). If the TMT exceeds the regular tax, the difference is called the Alternative Minimum Tax (AMT). Regular tax is computed by subtracting all allowable operating expenses, overhead, depreciation, amortization and depletion from current year revenues to arrive at taxable income. The tax rate is then determined from the published progressive tax schedule. An operating loss may be used to offset taxable income, thereby reducing taxes owed, in the previous three and following 15 years. The highest effective corporate income tax is 35%.

The AMT is determined in three steps. First, regular taxable income is adjusted by recalculating certain regular tax deductions, based on AMT laws, to arrive at AMT Income (AMTI). Second, AMTI is multiplied by 20% to determine TMT. Third, if TMT exceeds regular tax, the excess is the AMT amount payable in addition to the regular tax liability.

Federal taxation has been applied to the SRK Technical Economic Model using the following guidelines provided by MH-LLC:

- A five year Modified Accelerated Cost Recovery System (MACRS) table. US Tax laws allow five year depreciation for assets with a life of 4 to 10 years. As the project has a life of 7 years, the five year table was selected.
- Development costs (examples include preproduction stripping, pit access development, construction EPCM, feasibility study and permitting) will be expensed at 70% the year they occur. The remaining 30% will be deductible over five years
- Depreciable costs (examples include buildings, equipment, and mobile equipment) are deductible on MACRS depreciation.
- Depletable costs (examples include preproduction property acquisition and holding costs) will be amortized on a unit of production basis over the life of the project.
- Reclamation costs were accrued over the life of the project on a per ton mined basis.

### 20.3.2 Net Proceeds Tax

In Nevada, if the net proceeds of a mine in the taxable year totals US$4 million or more the tax rate is 5%. The gross proceeds from the sale of the minerals minus the allowable deductions determine the taxable net proceeds. The allowable deductions include the actual cost of:

- Extraction;
- Transportation of the mineral from the mine or point of extraction to the point of processing and sale;
- Processing;
- Marketing and delivery;
- Repair and maintenance of equipment;
- Fire insurance on plant and equipment;
- Depreciation of the cost of machinery and equipment;
- Contributions or payments for unemployment insurance, social security, fringe benefits for Employees, etc.;
- Royalties paid to claim holders, which are taxable to the recipient; and
- Development in or about the mine or group of mines that are operated as a unit.

Included in these costs are the cost of labor, supplies, and materials required to perform these activities. Only costs incurred in the process of performing these tasks in the current tax year may be deducted. Costs cannot be carried forward to future tax years or carried back to previous tax years.

Costs that are unrelated to the direct production of minerals, such as property and income taxes, charitable contributions, liability insurance or lobbying expenses are not deductible (Nevada Tax Payers Association, 2008).

### 20.3.3 Royalties

MH-LLC is subject to a 3.4% Net Smelter Return royalty. Royalty is prepaid at the rate of US$300,000 per year. As of time of this report, a total of US$2.0 million has been paid in prepaid royalties.

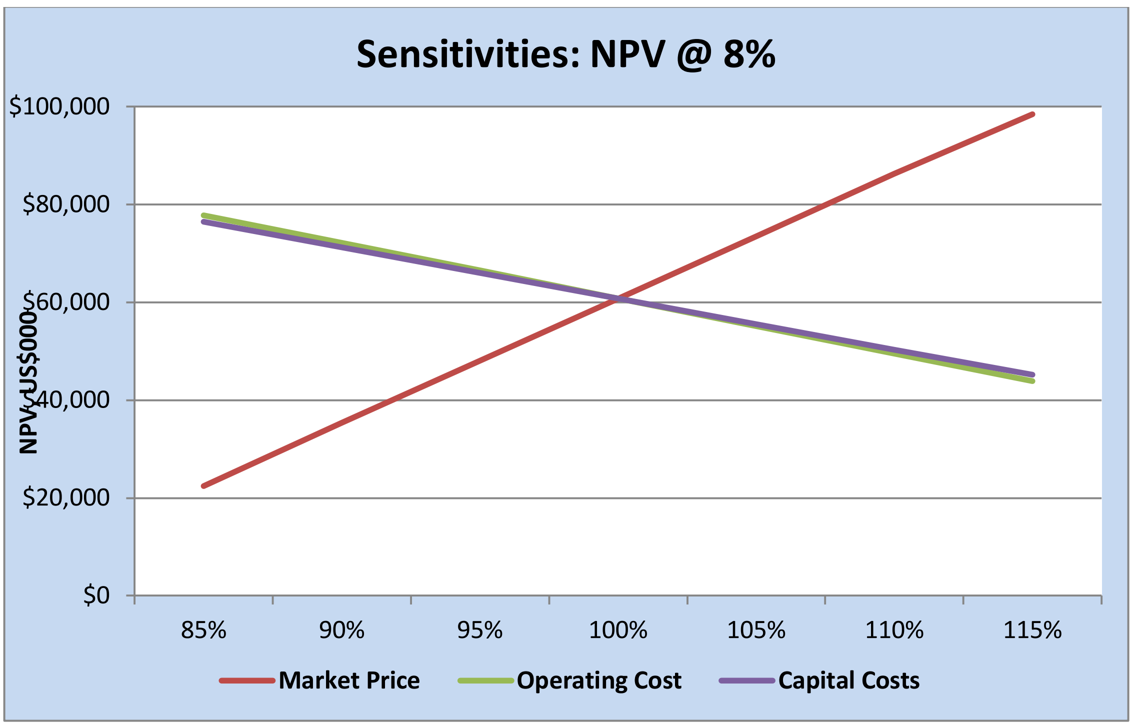
## 20.4 Sensitivity Analysis

Based on sensitivities of Market Price, Operating costs and Capital costs, the Project is most sensitive to changes in Market Price and least sensitive to both Operating and Capital Costs. The overall sensitivity at 8% discount rate is detailed in Table 20.4.1 and illustrated in Figure 20.4.1.

**Table 20.4.1: Project Sensitivities**

| NPV (8%) | 85% | 90% | 95% | 100% | 105% | 110% | 115% |
|---|---|---|---|---|---|---|---|
| **Market Price** | $22,368 | $35,383 | $48,102 | $60,817 | $73,532 | $86,274 | $98,504 |
| **Operating Cost** | $77,801 | $72,140 | $66,478 | $60,817 | $55,156 | $49,495 | $43,833 |
| **Capital Costs** | $76,498 | $71,271 | $66,044 | $60,817 | $55,590 | $50,363 | $45,136 |

Source: SRK, 2014




Source: SRK, 2014

**Figure 20.4.1: Project Sensitivity @8% Discount Rate**

Table 20.4.2 contains the metal price sensitivities at an 8% discount rate.

### Table 20.4.2: Project Sensitivity to Metal Prices

| **Gold US$/oz** | $1,000 | $1,200 | $1,250 | **$1,300** | $1,350 | $1,400 | $1,500 |
|---|---|---|---|---|---|---|---|
| **Silver US$/oz** | $15.40 | $18.50 | $19.25 | **$20.00** | $20.75 | $21.50 | $23.10 |
| **Pre-Tax** | | | | | | | |
| Cash Flow, US$M | $56,475 | $141,920 | $163,340 | **$184,760** | $206,180 | $227,600 | $269,507 |
| NPV 8% US$M | $16,110 | $76,627 | $91,789 | **$106,951** | $122,113 | $137,275 | $166,968 |
| IRR | 12.8% | 28.4% | 32.0% | **35.4%** | 38.8% | 42.1% | 48.4% |
| Payback (Years) | 3.6 | 3.0 | 2.8 | **2.7** | 2.6 | 2.5 | 2.3 |
| **With Tax (Federal=35%, State=5%)** | | | | | | | |
| Cash Flow, US$M | $36,206 | $89,425 | $102,653 | **$115,882** | $129,110 | $142,339 | $168,255 |
| NPV 8% US$M | $2,752 | $41,762 | $51,290 | **$60,817** | $70,345 | $79,872 | $98,534 |
| IRR | 8.9% | 20.6% | 23.3% | **26.0%** | 28.5% | 31.1% | 36.0% |
| Payback (Years) | 3.8 | 3.1 | 3.0 | **2.9** | 2.8 | 2.7 | 2.5 |

Source: SRK, 2014, Base case is bolded

# 21 Adjacent Properties (Item 23)

No mineral interests are immediately adjacent to the Mt. Hamilton Project area.

The historical Hamilton-Belmont-Treasure Hill silver district, which has been inactive since the 1950s, lies east of the Project area on the east flank of the Pogonip Range, and this area has not been explored with modern techniques.

## 21.1 Verification

SRK has not done any form of verification of the information concerning the nearby Hamilton District deposits or other prospects. None of the adjacent properties have a current economic impact on the development of the Project as described in this report.

# 22 Other Relevant Data and Information (Item 24)

There is no additional relevant technical or socio-economic information that SRK is aware of that would materially impact the conclusions of this report. The details of mining, processing and economics at a feasibility-level of detail are presented in earlier sections of this report.

# 23 Interpretation and Conclusions (Item 25)

Mt. Hamilton is an advanced pre-development project with a strong economic projection based on feasibility-level capital and operating costs from a thorough mining and processing development plan. A LoM Net after tax Present Value of US$60.8 million is forecast (US$1,300 gold/US$20 silver) with and internal rate of return of 26.0% and a payback period of 2.9 years on a 7-year mine life using a discount rate of 8%.

In 2014, MH-LLC retained SRK to complete a feasibility study for the Mt. Hamilton Project. During 2012-13, several data collection programs with the primary objective to bring parts of the Seligman resources into reserves and advance the combined Centennial-Seligman Project to feasibility-level including:

- Infill drilling to confirm historic metal grades and continuity of mineralization;
- Drilling and metallurgical testing to confirm gold recovery projections and process design;
- Drilling and geotechnical pit slope stability evaluation for mine planning;

The significant findings of the 2014 FS are summarized in this section.

The purpose of the 2014 FS was to collect and analyze sufficient data to reduce or eliminate risk in the technical components of the project and to refine economic projections based on current cost data. SRK offers the following conclusions and recommendations for key components of the proposed mining operation at Mt. Hamilton following the addition of the adjacent Seligman reserves.

**Geology, Drilling and Exploration Data Quality**

The geological and drilling database for the Mt. Hamilton property is robust, and recent drilling (2008-2012) carried out by MH-LLC, involving modern quality controls, has validated historic drilling in areas where new and old drilling overlap. Hole location and survey risk is considered very low as most drill sites can be confirmed using current and pre-mining aerial photography and topography. MH-LLC has a well-organized core storage and sample preparation facility in Ely, Nevada and follows industry standard protocols for material handling and documentation.

There is still a large dependence on historic data in parts of Seligman that were collected before current quality controls were in place. Infill drilling has improved the overall quality of the assay database by adding a higher proportion of validated samples to the total. Additional infill drilling will continue to improve data quality and also fill gaps in the cyanide-soluble data set. Installation of the proposed conveyor incline and ore pass will expose new geology, which will improve the geologic model and could define new drill targets accessible from the new conveyor incline.

**Recommendations:**

- Continue to compile and review assay results from future drilling converting resources to reserves as the results are received to improve batch quality when the analytical program is still active;
- Include a second split from a minimum of 5% of the coarse reject samples to verify the adequacy of crush size for assay repeatability;
- Randomly select roughly 5% of pulp samples from future drilling for check assay at a second independent laboratory, for all parameters used in resource estimation;
- Map and sample the conveyor incline during development.

**Mineral Resources**

From an exploration perspective, additional infill drilling could upgrade resource classifications to make more gold ounces eligible for reserves at price assumptions utilized in the current reserve statement. Inferred mineralization within the resource pit especially between the Seligman and Centennial deposits has strong prospect for upgrading. Successful conversion of resources to reserves in this area will likely reduce the overall stripping ratio as more tons are accessed in a combined pit. Additionally, a meaningful portion of the mineralized material within the current reserve pit is currently categorized Inferred resources, and consequently, treated as waste in the economic model.

The quality of the historic data used in the resource estimate has been verified by recent drilling and confirmed by an analysis of quality control data by SRK. Resources in the 2014 Mineral Resource Statement reflect a refinement of tonnage and grade estimates that used updated and more conservative density estimates and infill drilling results for Seligman and Centennial. Measured and Indicated mineral resources for the combined Mt. Hamilton gold-silver deposit are reported at 828,000 AuEq ounces with an additional 136,000 AuEq Inferred ounces. These resources are contained within an open pit mining configuration (resource pit) driven by US$1,300/oz gold and US$19.60/oz silver values.

There are more than 230,000 oz of in situ gold modeled outside the resource pit that are not categorized at this time, and not reportable as NI 43-101 compliant resources due to current economics or drill density. The majority of the uncategorized material is down-dip to the east and into the hill slope requiring an increasing proportion of stripping to access mineralization. Higher metal prices would convert some of this material into reportable resources where drill density is sufficient.

Exploration potential outside of the planned operational area has been demonstrated in surface soil gold anomalies located mostly east of Seligman and along strike south of Centennial. Principal targets include Chester/Wheeler Ridge, U4, Five Way and White Pine. Sparse or historic drilling in other exploration areas may have missed additional resources, which might be sterilized by the current mine design (e.g. Five Way, Cabin Gulch). Near-term condemnation drilling should address this possibility. Future exploration should also consider sulfide-hosted gold/silver as well as other commodities (Mo, W, Cu) that may be economic in a milling scenario.

**Recommendations:**

- Targeted infill drilling to characterize material in expanded pits and to upgrade resources from Inferred to Indicated classification and confirm continuity in narrow mineralized zones;
- Exploration drilling to test the large and strong Wheeler Ridge gold-in-soil anomaly south of the Centennial resource.
- Continue to build the multi-element database to get spatial distribution of base and transition metals;
- Improve geologic logging methods to capture material properties that affect rock mechanics and metallurgy for future feasibility analysis;
- Detailed stratigraphic/structural geology modeling (from historic mapping data) to identify step-out exploration targets that could add to the resource;

## Mineral Reserves and Mining

A conventional truck and shovel operation is proposed for operations at a mining rate of 10,000 t/d ore. Only Measured and Indicated resources were converted to reserves using US$840/oz gold and US$12.68/oz silver pricing along with conservative operating cost assumptions. Recovery and dilution were addressed in the definition of ore. The assumption of low metal prices in the reserve model mitigates down-side price risk while providing high-quality ore to the leach pad, which has a private-property limited capacity of 22.8 Mt of ore.

Dedicated oriented core drilling and geotechnical characterization of the rock mass has been applied to reserves. SRK's analysis of the geotechnical data supports an overall pit slope of 50°. Flatter slopes, which include ramps, were designed on the west side of the open pit.

The mining production schedule was built around detailed phase designs that include full mining equipment access. The designs contain detailed haulage profiles used to determine haulage costs.

Mining on 10 and 20 ft benches, (triple benched to 60 ft locally) using a hydraulic shovel allows for selectivity in tabular ore. The use of a wheeled loader will aid mining precision in thinner ore zones. Oxide ore is visibly distinguishable from un-oxidized waste, and in most cases this will improve grade control efficiency.

All previous drilling at Centennial and mining in the adjacent NE Seligman mine indicate that groundwater greatly exceeds the depth of proposed mining. Therefore, the proposed open pit will be dry and will require no provisions for dewatering.

SRK has proposed a design for ore delivery that accommodates winter operating conditions at high elevations. The predominantly underground ore-flow system will protect conveyors and should require less maintenance with less weather-related down-time. Although some geotechnical work has been completed regarding the adit and ore pass, there remain some uncertainties in the ore-flow system related to the geotechnical characterization of the proposed adit and ore-pass chamber. Ideally, both of these excavations would have received a complete geotechnical evaluation at a feasibility level based on pilot-hole drilling; however, permitting and seasonal limitations have precluded this assessment. To mitigate the uncertainty, SRK, based on outside underground subcontractor pricing, applied heavy contingencies for ground support, which added costs to the planned underground development. This was deemed necessary in the absence of geotechnical supporting data.

Other components of the ore flow system, including the conveyor and stacker array are well understood, vendor quoted, and considered to be of low risk for consistent ore delivery. Excavation and construction for the underground ore-flow system are scheduled to begin in Q4, 2014.

**Recommendations:**

- Additional oriented geotechnical diamond core drilling in the extreme southernmost Seligman; and
- Improve geologic and geotechnical engineering confidence for the ore pass and conveyor incline using oriented core drilling to better predict and cost ground support requirements.

## Metallurgy and Processing

Overall, the results of the 2013 Seligman-focused metallurgical testing of oxide ores were comparable, if not more favorable than previous results for Centennial. One of the key findings from the drilling and testing of the Seligman and North Centennial ores was the favorable leach profile of Seligman igneous oxide, which had been largely overlooked by previous operators.

Metallurgical characterization is at a feasibility level for all of the drilled or re-drilled parts of Centennial and North Seligman, leaving only extreme south Seligman needing further test work. Metallurgical risk for this area is considered low. Column test work on the oxide ores of both the Centennial and Seligman deposits demonstrates recovery of 79-80%. Sulfidic ores were also evaluated and found to be refractory in carbon-in-column processing. Testing showed that transitional ores were economically feasible to process in some cases. Cyanide soluble assay techniques have been shown to be effective to readily identify economic ore from waste in transitional ore. The projected average overall gold recovery of 76% is a result of the inclusion of some economic transitional ore in the mine plan. Modelled gold recovery based on paired cyanide soluble and fire assays provides a high degree of detail in characterization of expected operational recoveries in comparison to assigned recoveries based solely on observed oxidation of the ore.

There could be economic benefit to additional comminution and hydraulic conductivity testing on Seligman igneous material. Additional comminution testing on igneous material may show that less work is needed to crush igneous than skarn material. The current assumption is that all material will crush as skarn.

The recent 2014 detailed design work and contractual cost projections for the ADR plant by Kappes Cassiday and Associates (KCA) has improved confidence in cost estimates related to plant construction and operation. The strip rate of the plant was designed for 4.5 t of carbon to accommodate a throughput of 10,000 t/d. The crushing circuit planned can accommodate this tonnage, with variable belt speeds to match ore delivery rates.

Remodeling and rescheduling the reserves in 2014 largely removed concerns about overloading silver in the process circuit, but there are still phases in the production schedule when Ag:Au ratio should be monitored. In situations where the ratio is high, it can be remedied by blending stockpiled ore.

The current plan for leach pad underliner is to amend soils in place. There are less expensive options for underliner from known local clay borrow sources that should be investigated to reduce costs.

The selected processing methodology is considered low risk. The ADR carbon-in-column method for gold and silver recovery is proven technology and widely used in analogous operations in Nevada.

Power will be initially supplied at the mine and ADR by generators. The production water supply has been defined and water rights sufficient for project start-up have been secured by MH-LLC. This 2014 FS used the existing Seligman well as the primary source for production water, but further hydrogeologic exploration is planned to locate a source closer to the planned leach operation to reduce costs.

There is no tailings risk associated with this processing plan as no tailings will be generated. Spent ore will remain on containment (HDPE liner) after leaching and the facility will be reclaimed in place during closure.

**Recommendations:**

- Additional comminution and hydraulic conductivity testing on igneous material;

- Additional metallurgical characterization in conjunction with reserve drilling at South Seligman; and
- Further investigate local clay borrow source for leach pad underliner.

**Infrastructure**

Power and water are the key elements of the project infrastructure. Both systems are at a feasibility level for design and costing. There are opportunities to upgrade both systems and these have been built into the economic evaluation. In year three of operations, MH-LLC expects to convert from generated power to line power reducing unit costs from US$0.25/kWh to US$0.05/kWh. With such a change in costs, there is both a risk and an opportunity related to power costs depending on the timing of the installation compared to plan.

Water supply costs are currently based on the existing Seligman water well as the primary source. MH-LLC plans to install a new well, 1.5 miles closer to the process plant with lower pumping costs and piping risk. The new well(s) will likely become the primary water supply for operations.

**Recommendations:**

- Develop a second water supply well to supply up to 500 gpm during peak construction and operation. The current plan to install a new water supply at the Admin Parcel near the process plant is considered a top priority as this could become the primary water supply for the operation, securing availability during peak demand.

**Environmental Studies and Permitting**

Permits for activity on both private and Forest Service land have been submitted to the appropriate State agencies for review. A Water Pollution Control Permit has been issued by the State of Nevada including a recently updated Waste Rock Management Plan. The approved method of waste rock placement is blending which requires no special segregation of ore by geochemical character.

Air Quality Permit applications have been submitted to the state and approval of a permit to construct is expected in Q4 2014. Separate Reclamation Permit applications for the mine and processing facilities have been submitted to the State of Nevada. These are in the final stages of review and approval.

The Mine Plan of Operations (MPO) for activity on public land (USFS) has been submitted to the Forest Service. Baseline biological and archaeological surveys have been completed and approved for the area inside the MPO boundary. The EA, required under NEPA, is complete and the Objection Period passed. A Right of Way grant has been issued by the BLM for access to Forest Service land where mining will occur.

Water quality sampling from existing monitoring wells is ongoing and is reported to the state. Water supply capacity was confirmed by pump testing in 2013. Water rights for the operation have been secured.

The Project has several characteristics that are favorable from a permitting and compliance standpoint including: 1) No anticipated pit lake; 2) Acid neutralizing waste rock; 3) Deep groundwater beneath the proposed leach pad; and 4) Process components operated and closed on private land.

The mine closure cost without contingency as calculated using the Standard Reclamation Cost Estimator is US$8.8 million.

**Recommendations:**

- Permitting is advanced and no further recommendations apply

**Projected Economic Outcomes**

The additional metal brought into reserves by the 2011-2012 exploration drilling, geotechnical and metallurgical test work has helped to offset fixed capital requirements and lower assumed gold prices improving project economics in 2014 FS compared to the 2012 FS. Metal prices of US$1,300/oz gold and US$20.00/oz silver were applied to the 2014 economic evaluation. Anticipated mine production is 22.5 Mt. of ore with a 2.5:1 waste: ore stripping ratio (including residual ore stockpile), at a mining rate of 10,000 t/d ore, resulting in 545.4 koz contained gold and 4,459.6 koz of contained silver. Metal recoveries are projected at 76% and 39% for gold and silver, respectively.

The economic results, at a discount rate of 8%, indicate a NPV of US$60.8 million with an IRR of 26.0% (after estimated taxes). Payback will be in 2.9 years from the start of production. Initial capital costs are projected at US$91.8 million with a total capital cost for the Project of US$121.5 million. The cash costs per gold-equivalent ounce recovered is US$558.0

Economics of the Mt. Hamilton Project are fairly insensitive to commodity prices due to the low $840/oz gold price used to drive the pit in this study. Metal prices have fallen over the last three years from near all-time highs in 2011. The current metal prices have slowed down production at neighboring Nevada mines and made available additional skilled labor to support the Mt. Hamilton operation.

# 24 Recommendations (Item 26)

Work programs recommended to advance the Project include drilling, engineering designs and technical studies as follows:

Drilling:

- Resource conversion drilling (RC) (Inferred upgrade to Measured/Indicated outside of but adjacent to the ore within the current mine plan);
- Seligman south area resource/metallurgical confirmation RC and core drilling;
- Geotechnical drilling and analysis for underground development of the ore flow system; and
- Supplemental (closer to processing) water supply well drilling and piping design.

Engineering Designs:

- Staff engineer for detailed design project management;
- Detailed designs for underground reclaim chamber and infrastructure; and
- Construction-level designs on ancillary facilities.

Technical Studies:

- Seligman south metallurgical and geotechnical studies; and
- Finalize environmental permitting.

A total anticipated cost for advancement of the project during the Pre-Construction phase is US$2.9 million. The cost break-down for the work programs described above are presented in Table 24.1.

**Table 24.1: Recommended Pre-Construction Work Program Costs**

| Work Program | Estimated Cost US$ | Assumptions/Comments |
|---|---|---|
| Priority 1a and 1b resource/reserve conversion drilling (RC) | 400,000 | 31 holes for 9,000 ft @ US$45/ft |
| Priority 2 and 3 reseource/reserve conversion drilling (RC) | 390,000 | 24 holes for 8,700 ft @ US$45/ft |
| Geotechnical drilling for underground development (DD) | 500,000 | 2,500 ft @ 200/ft incl. supervision |
| Relocate water supply well closer to processing | 350,000 | pump tests and pumps, design |
| **Total Drilling** | **1,640,000** | |
| Detailed design project management | 200,000 | salaried new hire or contract PM |
| Detailed design for underground reclaim chamber and infrastructure | 50,000 | specialist contractor/engineer |
| Detailed design for crushing, process and infrastructure and preliminary EPCM | 500,000 | specialist contractor/engineer |
| **Total Detailed Design** | **750,000** | |
| Seligman geotechnical analysis | 25,000 | consultant engineer |
| Environmental permitting | 150,000 | environmental contractor |
| **Total Technical Studies** | **175,000** | |
| **Sub Total** | **2,565,000** | |
| **Contingency @15%** | **384,750** | |
| **Total** | **2,949,750** | |

Source: SRK, 2014

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# 26 Glossary

The mineral resources and mineral reserves have been classified according to the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (May 10, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

## 26.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

## 26.2 Mineral Reserves

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a pre-feasibility study or feasibility study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

## 26.3 Definition of Terms

The following general mining terms may be used in this report.

**Table 26.3.1: Definition of Terms**

| Term | Definition |
|---|---|
| Assay | The chemical analysis of mineral samples to determine the metal content. |
| Capital Expenditure | All other expenditures not classified as operating costs. |
| Composite | Combining more than one sample result to give an average result over a larger distance. |
| Concentrate | A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore. |
| Crushing | Initial process of reducing ore particle size to render it more amenable for further processing. |
| Cut-off Grade (CoG) | The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration. |
| Dilution | Waste, which is unavoidably mined with ore. |
| Dip | Angle of inclination of a geological feature/rock from the horizontal. |
| Fault | The surface of a fracture along which movement has occurred. |
| Footwall | The underlying side of an orebody or stope. |
| Gangue | Non-valuable components of the ore. |
| Grade | The measure of concentration of gold within mineralized rock. |
| Hangingwall | The overlying side of an orebody or slope. |
| Haulage | A horizontal underground excavation which is used to transport mined ore. |
| Hydrocyclone | A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials. |
| Igneous | Primary crystalline rock formed by the solidification of magma. |
| Kriging | An interpolation method of assigning values from samples to blocks that minimizes the estimation error. |
| Level | Horizontal tunnel the primary purpose is the transportation of personnel and materials. |
| Lithological | Geological description pertaining to different rock types. |
| LoM Plans | Life-of-Mine plans. |
| LRP | Long Range Plan. |
| Material Properties | Mine properties. |
| Milling | A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product. |
| Mineral/Mining Lease | A lease area for which mineral rights are held. |
| Mining Assets | The Material Properties and Significant Exploration Properties. |
| Ongoing Capital | Capital estimates of a routine nature, which is necessary for sustaining operations. |
| Ore Reserve | See Mineral Reserve. |
| Pillar | Rock left behind to help support the excavations in an underground mine. |

| Term | Definition |
|---|---|
| RoM | Run-of-Mine. |
| Sedimentary | Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks. |
| Shaft | An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste. |
| Sill | A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness. |
| Smelting | A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase. |
| Stope | Underground void created by mining. |
| Stratigraphy | The study of stratified rocks in terms of time and space. |
| Strike | Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction. |
| Sulfide | A sulfur bearing mineral. |
| Tailings | Finely ground waste rock from which valuable minerals or metals have been extracted. |
| Thickening | The process of concentrating solid particles in suspension. |
| Total Expenditure | All expenditures including those of an operating and capital nature. |
| Variogram | A statistical representation of the characteristics (usually grade). |

## 26.4 Abbreviations

The following abbreviations may be used in this report.

| Abbreviation | Unit or Term |
|---|---|
| % | percent |
| ° | degree (degrees) |
| °C | degrees Centigrade |
| AA | atomic absorption |
| AAS | atomic absorption spectroscopy |
| ADR | adsorption-desorption-recovery |
| AFA | acre-feet per annum |
| Ag | silver |
| AGP | Acid Generation Potential |
| amsl | above mean sea level |
| ANP | Acid Neutralization Potential |
| Au | gold |
| AuEq | gold equivalent |
| bgs | below ground surface |
| cfs | cubic feet per second |
| CIM | Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 |
| cm | centimeter |
| CoG | cut-off grade |
| EA | environmental assessment |
| EIS | environmental impact statement |
| FA | fire assay |
| FS | feasibility study |
| ft | foot (feet) |
| $ft^2$ | square foot (feet) |
| $ft^3$ | cubic foot (feet) |
| g | gram |
| g/t | grams per metric tonne |
| gal | gallon |
| gpm | gallons per minute |
| $H_2O_2$ | hydrogen peroxide solution |

| Abbreviation | Unit or Term |
|---|---|
| HCT | humidity cell test |
| HDPE | high density polyethylene |
| hp | horsepower |
| ICP | induced couple plasma |
| ID2 | inverse distance squared |
| IDW | inverse distance weighted |
| koz | thousand troy ounce |
| kt | thousand short tons |
| kW | kilowatt |
| kWh | kilowatt-hour |
| lb | pound (pounds) |
| LoM | life-of-mine |
| MH-LLC | Mt. Hamilton LLC |
| Mt | million short tons |
| Mt/y | million tons per year |
| $Myd^3$ | million cubic yards |
| NAG | Net Acid Generation |
| NI 43-101 | Canadian National Instrument 43-101 |
| NN | Nearest Neighbor |
| NNP | Net Neutralization Potential |
| NPR | Neutralization Potential Ratio |
| OK | ordinary kriging |
| oz | troy ounce |
| oz/t | ounces per short ton |
| $oz/yd^2$ | ounces per square yard |
| MPO | plan of operations |
| ppm | parts per million |
| QA/QC | quality assurance/quality control |
| QP | qualified person |
| RC | rotary circulation |
| RoM | run-of-mine |
| s.u. | standard units |
| sec | second |
| t | short ton (2,000 pounds) |
| t/d | short tons per day |
| t/h | short tons per hour |
| t/y | short tons per year |
| tonne | metric tonne (1,000 kg or 2,204.6 lb) |
| US$ | United States Dollar |
| V | volts |
| W | watt |
| y | year |

# Appendices

# Appendix A: Certificates of Authors

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## CERTIFICATE OF AUTHOR

I, J. B. Pennington, a Certified Professional Geologist do hereby certify that:

1. I am a Principal Mining Geologist of:

   SRK Consulting (U.S.), Inc.
   5250 Neil Road, Suite 300
   Reno, Nevada 89502

2. This certificate applies to the technical report titled "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012 and a Report Date of October 25, 2012 (the "Technical Report").

3. I graduated with a Bachelor of Science Degree in Geology from Tulane University, New Orleans, La., USA; May 1985; and a Master of Science Degree in Geology from Tulane University, New Orleans, La., USA; December 1987. I am a Certified Professional Geologist through membership in the American Institute of Professional Geologists, CPG - 11245. I have been employed as a geologist in the mining and mineral exploration business, continuously, for the past 27 years, since my undergraduate graduation from university. My relevant experience for the purpose of the Technical Report is:

   - Project Geologist, Archaen gold exploration with Freeport-McMoRan Australia Ltd. Perth Australia, 1987-1989;
   - Exploration Geologist, polymetallic regional exploration, Freeport-McMoRan Inc; Papua, Indonesia, 1990-1994;
   - Chief Mine Geologist, mine geology and resource estimation, Grasberg Cu-Au Deposit, Freeport-McMoRan Inc, Papua, Indonesia 1995-1998;
   - Corporate Strategic Planning: Geology and Resources, Freeport-McMoRan Inc., New Orleans, LA., 1999;
   - Senior Geologist, environmental geology and mine closure, MWH Consulting, Inc., Steamboat Springs, CO., 2000-2003;
   - Principal Mining Geologist, precious and base metal exploration, resource modeling, and mine development, SRK Consulting (U.S.), Inc., 2004 to present;
   - Experience in the above positions working with, reviewing and conducting resource estimation and technical studies in precious and base metal both domestically and overseas; and
   - As a consultant, I have managed and prepared several NI 43-101 Technical reports, 2006-2012.

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

5. I visited the Mt. Hamilton property most recently on June 29, 2011.

6. I am responsible for the preparation of introduction, geology, and the Mineral Resource Estimate; Sections 1, 2 except for 2.5, 3, 4,12, 21, 22, 23, 24, 25, 26 and portions of the Summary, Sections 23 and 24 summarized therefrom of the Technical Report.

7. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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North America
South America

8. I had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the coordination and preparation of the reports titled, "NI 43-101 Preliminary Economic Assessment, Ely Gold & Minerals Inc., Centennial Gold and Silver Deposit, Mt. Hamilton Property, White Pine County, Nevada" and dated May 8, 2009, "NI 43-101 Preliminary Economic Assessment, Ely Gold & Minerals Inc., Centennial Gold and Silver Deposit, Mt. Hamilton Property, White Pine County, Nevada" dated July 9, 2010, "NI 43-101 Technical Report on Resources and Reserves, Mt Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012 and the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold Project, Centennial Deposit, White Pine County, Nevada" and dated February 22, 2012.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014

*"Signed"* *"Sealed"*

J. B. Pennington, C.P.G., MSc

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**CERTIFICATE OF AUTHOR**

I, Brooke J. Miller, M.Sc., CPG, do hereby certify that:

1. I am a Senior Consultant of SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, Nevada, USA, 89502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with a Bachelor of Arts degree in Geology from Lawrence University in 2002 and a Master of Science degree in Geological Sciences from The University of Oregon in 2004. I am a Certified Professional Geologist of the American Association of Professional Geologists (AIPG). I have worked as a Geologist for a total of 9 years since my graduation from university. My relevant experience includes mining and exploration geology.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Mt. Hamilton property on October 24, 2011.
6. I am responsible for the preparation of geology and resources Sections 2, 3, 4, 5, 6, 7, 8, 9, 10 and portions of the Summary, Sections 23 and 24 summarized therefrom of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101-F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*
Brooke J. Miller, M.Sc., CPG

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## CERTIFICATE OF QUALIFIED PERSON

I, Richard DeLong, MS, PG, RG, CEM do hereby certify that:

1. I am I am President of Enviroscientists, Inc., 1650 Meadow Wood Lane, Reno, Nevada 890502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with two master degrees in Resource Management and Geology from University of Idaho in 1984 and 1986, respectively. I am a Professional Geology, Registered Geologist, and a Certified Environmental Manager in the States of Idaho, California, and Nevada, respectively. I have worked as a Environmental Professional in Permit Acquisition for a total of 26 years since my graduation from university. My relevant experience includes the acquisition of permits for over 500 mining and exploration projects in the western United States.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Mt. Hamilton Property.
6. I am responsible for the preparation of environmental, permitting and community impact Sections 2.5, 18 and portions of the Summary, Sections 23 and 24 summarized therefrom of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

Richard DeLong, MS, PG, RG, CEM

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**CERTIFICATE OF QUALIFIED PERSON**

I, Kent W. Hartley, P.E. Mining, BSc, do hereby certify that:

1. I am Principal Consultant of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Michigan Technological University in 1979. I have worked as an Engineer for over 32 years since my graduation from university. My relevant experience includes mine planning and project engineering at a number of open pit and underground mines as well as construction management and cost estimating experience. I am a registered Professional Engineer in Nevada, license number 021612.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Mt. Hamilton property on April 25, 2014 for 1 day.
6. I am responsible for the preparation of mineral reserves, mining methods, infrastructure, market studies, capital and operating costs and economic analysis Sections 13, 14 except 14.4, 16, 17, 19, 20 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012.
9. I have read NI 43-101 and Form 43-101-F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*

Kent W. Hartley, P.E. Mining, BSc

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**CERTIFICATE OF QUALIFIED PERSON**

I, Michael Levy, MSc, PE, PG do hereby certify that:

1. I am Senior Geotechnical Engineer of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated from the University of Iowa in 1998 with a B.Sc. in Geology and from the University of Colorado in 2004 with a M.Sc. in Civil-Geotechnical Engineering. I am a registered Professional Engineer in the states of Colorado (#40268) and California (#70578) and a registered Professional Geologist in the state of Wyoming (#3550). I have 16 years of experience in civil and mining geotechnical projects. I am skilled in both soil and rock mechanics engineering and specialize in the analysis and design of open pit slopes. My primary area of expertise includes statistical characterization of geotechnical data and probabilistic modeling techniques for pit slope design. I'm also experienced in numerical modeling and LiDAR scanning techniques for rock slope projects
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Mt. Hamilton property.
6. I am responsible for the preparation of the pit slope geotechnical Section: 14.4 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012 and the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold Project, Centennial Deposit, White Pine County, Nevada" and dated February 22, 2012.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*

Michael Levy, MSc, PE, PG

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**CERTIFICATE OF QUALIFIED PERSON**

I, Evan Nikirk, P.E., M.Sc., do hereby certify that:

1. I am a Principal Consultant of SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, Nevada 89502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with a Bachelor's of Science degree in Civil Engineering from the University of California at Berkeley in 1986. In addition, I have obtained a Master's of Science degree in Civil / Environmental Engineering from the University of California at Berkeley in 1994. I am a member of the American Society of Civil Engineers and the Association of State Dam Safety Officials. I have worked as a civil engineer for a total of 27 years since my graduation from university. My relevant experience includes conceptual and detailed site design, hydrologic and hydraulic design, technical specifications, cost estimating, construction supervision, permitting, planning, closure design, monitoring system design, environmental assessments, surveying, and ground water and landfill gas monitoring and reporting at numerous mining, waste management, and industrial properties throughout the western United States, Mexico, Thailand, and Australia.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Centennial property on June 29, 2011 for one day.
6. I am responsible for the preparation of heap leach pad design Section 15.3 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012 and the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold Project, Centennial Deposit, White Pine County, Nevada" and dated February 22, 2012.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*

Evan Nikirk, P.E., MSc.


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**CERTIFICATE OF QUALIFIED PERSON**

I Herbert Osborne, P.E., do hereby certify that:

1. I am an Associate of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with a Metallurgical Engineers Degree from Colorado School of Mines in 1961.I am a Registered Member of the Society of Mining Engineers and the Colorado Mining Association .I have worked as a Metallurgical Engineer for over 50 years since my graduation from university. My relevant experience includes design, construction, operation, troubleshooting and closure of 20+ heap leach operations.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Mt. Hamilton property since visiting the property with Martin Quick, a capital consultant, to assess the value of the plant in 2000.
6. I am responsible for the preparation of the crushing, conveying and stacking Section 15.2 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of July 31, 2012 and the report titled, "NI 43-101 Technical Report on Resources and Reserves, Mt. Hamilton Gold Project, Centennial Deposit, White Pine County, Nevada" and dated February 22, 2012.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*
Herbert Osborne, P.E. SME Registered Member 2430050

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**CERTIFICATE OF AUTHOR**

I, Chris Sheerin, RM-SME, MSc. do hereby certify that:

1. I am Principal Consultant, Metallurgy and Process Engineer of SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, Nevada, USA, 89502.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Mt. Hamilton Gold and Silver Project, Centennial Deposit and Seligman Deposit, White Pine County, Nevada" with an Effective Date of August 14, 2014 (the "Technical Report").
3. I graduated with a degree in Bachelor of Science in Chemical Engineering from University Nevada Reno in 1987. I am a Register Member of the Society for Mining, Metallurgy and Exploration. I have worked as a Metallurgical Engineer for 29 years since my graduation from university. My relevant experience includes; principal consulting on feasibility and metallurgical testing studies; engineering reviews; operational readiness; and commissioning planning. My expertise includes; ore characterization, feasibility studies and process design for the precious and base metal industries.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Mt. Hamilton property.
6. I am responsible for the preparation of process, metallurgical testing and recovery Sections: 11, 15 except for 15.2 and 15.3, and portions of the Summary, Sections 23 and 24 summarized therefrom of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report
9. I have read NI 43-101 and Form 43-101-F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th Day of October, 2014.

*"Signed"* *"Sealed"*
Chris Sheerin, RM-SME, MSc.

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