MINE DEVELOPMENT ASSOCIATES
MINE ENGINEERING SERVICES

TECHNICAL REPORT AND FEASIBILITY STUDY FOR THE RELIEF CANYON PROJECT, PERSHING COUNTY, NEVADA, U.S.A.

Prepared for

PERSHING GOLD CORPORATION

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1.0 EXECUTIVE SUMMARY

Mine Development Associates ("MDA") has prepared this Technical Report and Feasibility Study on the Relief Canyon project, located in Pershing County, Nevada, at the request of Pershing Gold Corporation ("Pershing Gold"). The Relief Canyon project is owned by Gold Acquisition Corp. ("GAC"), a wholly owned subsidiary of Pershing Gold.

The purpose of this Technical Report is to present a Feasibility Study ("FS") and updated estimate of mineral reserves for the Relief Canyon project. Pershing Gold is listed in Canada on the Toronto Stock Exchange and on the NASDAQ Global Market with stock symbol PGLC. MDA has prepared this report and the estimates provided herein in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

The effective date of the mineral resource estimate in this report is November 1, 2016. The effective date of this report and the new, updated reserve estimate is May 24, 2018. There are minor differences between the Reserves tables and the Feasibility production schedule, due to rounding in the mine scheduling program. Note that some of the tables in this report may not appear to add properly, however, this is due to rounding, and the totals in the tables are correct.

This report discusses a number of phases in the development of the Relief Canyon mine. When discussing the two permitting phases, Roman numerals will be used. When discussing a mine design or scheduling phase, Arabic numerals will be used.

1.1 Property Description and Ownership

The Relief Canyon property is located at the southwestern flank of the Humboldt Range about 16 miles east-northeast of Lovelock in Pershing County, Nevada, and about 100 miles northeast of Reno. As a result of an Asset Purchase Agreement ("APA") dated January 13, 2015, by and between Pershing Gold and its wholly owned subsidiary GAC as buyer, and Newmont USA Limited ("Newmont"), and the actions taken to effectuate the terms of the APA, the property currently consists of approximately 12,100 acres and includes 391 unpatented lode mining claims, 120 unpatented millsite claims, and approximately 4,373 acres of leased or subleased private mineral rights (fee land). The parcels that comprise the property are owned by Pershing Gold or GAC, or are leased by GAC from New Nevada Resources, LLC ("NNR") and New Nevada Lands, LLC ("NNL"), or are leased or subleased by Pershing Gold from Newmont.

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Pursuant to the APA, the June 15, 2006 Minerals Lease and Sublease ("2006 Lease Agreement") with Newmont was further amended by the Third Amendment dated January 15, 2015 whereby, among other amended terms: (i) 1,594 acres of fee land previously subleased from Newmont, were released from the 2006 Lease Agreement, the prior underlying leases terminated as to those lands, and converted to a new Mining Lease between GAC and NNR and NNL; (ii) 74 unpatented lode mining claims owned by Newmont were released from the 2006 Lease Agreement and conveyed to GAC; and (iii) the area of interest modified to exclude the GAC interests referenced in (i) and (ii) above, as well as other proximal lands owned or controlled by GAC. By the Third Amendment, none of the GAC owned mining claims, mill sites, or the fee lands directly leased by GAC are subject to the 2006 Lease Agreement. The mineral resources and reserves discussed in this report are all on GAC owned mining claims or GAC leased fee lands. The 2006 Lease Agreement, as amended, is further discussed in Section 4.3.1. Throughout this report, the reference to Pershing Gold may be used to apply to either Pershing Gold the parent, or its subsidiary GAC.

GAC-owned Lode and Millsite Claims, Leased Fee Lands:

The 120 unpatented millsite claims and 254 of the unpatented lode mining claims are owned by GAC. There is a 2 percent net smelter return ("NSR") royalty payable to either Newmont or Royal Crescent Valley Inc. on 141 of the lode mining claims owned by GAC. GAC leases 1,594 acres of the fee lands directly from NNR and NNL pursuant to Mining Lease NNR # 500135, dated January 6, 2015; these lands are subject to a 2.5 percent NSR payable to NNR and a 2 percent NSR payable to Newmont. None of the GAC owned mining claims, mill sites, or leased fee lands are subject to the 2006 Lease Agreement and therefore the mineral resources discussed by this report are not subject to the 2006 Lease Agreement.

Pershing Gold-owned and Leased Lode Claims, Subleased Fee Lands:

Pershing Gold owns or leases 137 unpatented lode mining claims and subleases ~ 2,779 acres of fee lands that remain subject to the 2006 Lease Agreement with Newmont. The 2006 Lease Agreement provides Newmont the option ("Newmont Option") under certain circumstances to enter into a joint venture with Pershing Gold or to convey the property to Pershing Gold and receive a sliding scale three to five percent NSR royalty, and a right to a $1.5 million-dollar production bonus payment upon conveyance. With regard to the subleased fee land, there is an offset provision in the event of underlying royalties such that Newmont's three to five percent NSR will be reduced by the underlying royalty, provided that Newmont's royalty shall not be less than two percent. Pershing Gold leases from Newmont 81 of the 137 unpatented lode mining claims that are subject to the 2006 Lease Agreement and owns the other 56. Pershing Gold's 56 owned claims are made subject to the 2006 Lease Agreement by the Agreement area of interest provision. These claims are subject to the Newmont Option. Pershing Gold subleases Newmont's leasehold rights on approximately 2,779 acres of fee land. Newmont holds these rights under two leases with NNR and NNL: (i) Minerals Lease NNR # 182092, dated August 17, 1987 covering 320 acres, with no underlying royalty; and (ii) Mining Lease NNR # 500136, dated December 31, 2014 covering approximately 2,459 acres, with a 2.5 percent underlying NSR payable to NNR. These subleased fee lands are subject to the Newmont Option.

1.2 Geology and Mineralization

The Relief Canyon property is located on the western flank of the southern Humboldt Range, one of the generally north-trending, fault-bounded ranges of the Basin and Range physiographic province. The oldest rocks exposed in the range are mafic and silicic volcanic rocks of the arc-related Lower Triassic Koipato Group, which are overlain by marine carbonate rocks of the Middle to Late Triassic Star Peak Group. The Cane Spring Formation lies at the top of the Star Peak Group and hosts the gold mineralization at Relief Canyon. Overlying the Star Peak Group is a fluvial-deltaic sequence called the Auld Lang Syne Group, of which the basal Grass Valley Formation overlies the gold mineralization at Relief Canyon.

During Middle Jurassic to Middle Cretaceous time, east-directed folding and thrusting, as well as metamorphism to at least greenschist facies, affected the Mesozoic carbonate and deltaic sedimentary rocks of the Relief Canyon area. Isolated remnants of Miocene basaltic and rhyolitic volcanic rocks in the southern Humboldt Range attest to Tertiary volcanism. Cenozoic northeast and north-northwest-trending normal faults are present on the property.

Mesozoic tectonostratigraphy in the vicinity of the Relief Canyon mine consists chiefly of a metamorphosed footwall mafic volcanic package; a metamorphosed, foliated, and highly deformed carbonate-dominant package with intercalations of conglomerate and mafic volcanic rocks; a tectonically thickened, thick-bedded to massive limestone unit; and a tectonically thickened package of siliciclastic rocks of the Late Triassic Grass Valley Formation. Altered feldspar porphyry dikes and sills, and dikes of gabbro are found in the Relief Canyon mine area. Gold mineralization occurs in both the foliated carbonate package and the thick-bedded limestone unit and is spatially correlated with the contacts of the gabbro intrusions.

Gold mineralization at the Relief Canyon mine is primarily found in three mineral zones that are structurally controlled and characterized by distinctive host rocks. From structurally lowest to highest, the zones are the Jasperoid Zone, the Lower Zone, and the Main Zone. The Main Zone hosts the bulk of the current and historical gold resources at Relief Canyon, while the Lower and Jasperoid zones are newly discovered mineral zones encountered below the Main Zone in the North Target area. Quartz-illite+fluorite+kaolinite alteration is associated with gold mineralization in all three of these mineral zones. Recognition of these three zones has provided the context for evaluating data from metallurgical testing, and for the selection of metallurgical test samples.

The modeled Relief Canyon Main Zone gold mineralization lies primarily within a collapse breccia at the top of the Cane Spring Formation (formerly the Natchez Pass Formation) massive limestone, immediately below the Grass Valley Formation. Within the deposit area, the contact between the Grass Valley Formation and Cane Spring Formation, as well as the mineralized breccia horizon lying between the two units, forms a broad, northeast-trending antiform that plunges about 10° to the southwest. The thickest portions of the breccia, as well as the associated mineralization, lie primarily along the broad crest of the antiform, and the breccia and accompanying mineralization thins and pinches out down dip on the northwest limb, and is very thin to nonexistent on the southeast limb. Locally, the breccia-hosted mineralization extends a short distance (usually less than 10 feet) into the overlying Grass Valley Formation.

The Lower Zone and Jasperoid Zone gold mineralization is hosted within the foliated deformed limestone package below the massive limestone unit. Lower Zone gold mineralization displays a strong spatial association with gabbro sills and/or transposed dikes, and mineralization is hosted in, or is proximal to, complex tectonic breccias and local carbonate-dissolution collapse breccias. The Jasperoid Zone occurs within a sequence of limey ductile tectonites with local stretched and boudinaged quartz veins, stretched-quartz-pebble conglomerate and sandstone, folded and foliated limestone, and altered gabbro, all of which have been replaced by dark-colored quartz.

1.3 Exploration and Mining History

Relief Canyon is located in the Relief-Antelope Springs mining district, which had antimony, silver, and mercury production, and fluorite prospecting, dating back to the late 1800s. The property was staked in 1978 for high-purity limestone by Falconi Cement Inc. ("Falconi"), which drilled one core hole to test the quality of the limestone. Gold was not identified in the area until 1979, when a regional precious metals prospecting program by the Duval Corporation ("Duval") generated anomalies in the area. Drilling by Duval in 1981 and 1982 confirmed the presence of a low-grade zone of gold.

Lacana Mining Inc. ("Lacana") purchased the property from Duval in 1982. After drilling 204 reverse circulation holes and undertaking pilot-scale heap-leach test work, Lacana opened the open pit Relief Canyon mine in September 1984, only to close it in October 1985 due to poor gold recoveries. Various sources report that from 1984 to 1985 Lacana produced about 14,000 ounces of gold from heap-leach processing of run-of-mine ore. Southern Pacific Land Company (later Santa Fe Pacific Corp.; "Santa Fe") owned the private property adjacent to the deposit, participated with Lacana in the pilot-scale metallurgical program, and drilled 147 reverse circulation holes on their property to test for continuation of the mineralization.

In 1986, Pegasus Gold Corp. ("Pegasus") purchased the property from Lacana and re-opened the mine in October 1987. Mining ceased in 1989 after having extracted material from three open pits. Production by Pegasus from the Relief Canyon mine is considered to be a little over 100,000 ounces.

J. D. Welsh and Associates of Reno, Nevada ("Welsh") purchased the property from Pegasus in September of 1993 and reportedly produced several thousand ounces of gold by continuing to rinse the existing heaps.

Newgold, Inc., which later changed its name to Firstgold Corporation (collectively "Firstgold"), purchased the Relief Canyon property from Welsh in January of 1995. In the first year, Firstgold processed pregnant pond solution until July of 1995.

Through April of 1997, Firstgold drilled 73 reverse circulation holes to examine the areas north, west, and southwest of the old pits for continuation of mineralization. The property was apparently then idle until 2003. In 2006, a ground magnetic survey was conducted. Subsequent exploration by Firstgold focused on the potential for mineralization between the existing pits and to the north and northwest. A total of 105 reverse circulation holes and four core holes were completed at Relief Canyon by Firstgold in 2007 and 2008.

Firstgold redeveloped and reconstructed the Relief Canyon heap-leach processing facilities in 2007 and 2008. They attempted to reprocess some of the previously leached material in late 2008 and early 2009, but shut the project down within a few months. In January of 2010, Firstgold filed for bankruptcy protection.

Platinum Long Term Growth LLC acquired the Relief Canyon assets, from whom Pershing Gold acquired the Relief Canyon mine on August 30, 2011. Since acquiring the project, Pershing Gold has conducted geologic mapping, rock and soil sampling for geochemical analysis, and geophysical surveying. As of

September, 2016, Pershing Gold has drilled 415 core and 89 reverse circulation holes to expand the resource and to develop and test targets away from the historical pits that are included in the database used to prepare the grade model of the deposit. Since September, 2016, Pershing Gold has drilled about 50 additional holes that are not included in the grade model.

1.4 Drilling and Sampling

Falconi, Duval, Lacana, Pegasus, Santa Fe, Firstgold, and Pershing Gold all drilled at the Relief Canyon property. No information on the single core hole drilled by Falconi in 1978 is included in the database. The database used for the resource estimate described in this report includes 419 core holes and 676 reverse circulation holes, for a total of 482,755 feet of drilling, of which 415 core holes and 89 reverse circulation holes were drilled by Pershing Gold from 2011 to September 2016. The updated mineral resource estimate described herein is based on the drill database through October of 2016.

Lacana and Santa Fe reported different experiences with drill-hole sampling of the mineralized breccia. Nearly all of the Lacana samples in the breccia were collected by dry drilling methods, as the breccia was intersected above the water table, and they made an effort to mitigate and quantify any effects of contamination. To the west, Santa Fe drilled deeper and encountered heavy formational water flows in the breccia, and their early sampling procedures allowed fine, clay-sized material (grain size is classified as "clay" if the particle diameter is <0.002 mm) to overflow the sampling bucket. Santa Fe revised its sampling procedures to improve collection of the fines, and comparison of the two types of sampling procedures by Santa Fe showed average increases of 8 percent to 19 percent in the gold values for intervals for which the fines were caught. Drill logs suggest that most holes by other operators drilled the breccia dry when it was intersected above the water table.

The drill data suggest that down-hole contamination of gold values occurred in some portion of the pre-Pershing Gold reverse circulation drill-hole sample database for drill-hole intervals below the water table. In response to this issue, Pershing Gold converted to entirely core drilling, resulting in a much higher confidence for geologic and assay data, as well as a much-improved structural interpretation; the latter being the primary control on mineralization within the Relief Canyon deposit. The issue of down-hole contamination in historical reverse circulation samples has been mitigated to a significant extent in the North area resource modeling by the exclusion of suspect intervals and the reliance on the recent Pershing Gold core drilling program. The authors' opinion is that the procedures used in the resource modeling have minimized the effects of potentially contaminated intervals and the risk to the resource estimate is considered low.

The authors believe the sample preparation, security, and analytical procedures used by Pershing Gold and prior operators were acceptable procedures and the resulting analytical data are of sufficient quality for use in the resource estimation.

1.5 Metallurgical Testing

The Relief Canyon ore deposit contains an oxidized and partially oxidized gold mineral resource and reserve that metallurgical testing and historical mining experience indicate are amendable to cyanide heap-leach processing. In 2015, 2016, and 2018, Pershing Gold conducted metallurgical test work on drill core and bulk samples to confirm heap-leach processing on additional resources that have been identified under

the existing pit. The metallurgical test work was based on identifying three distinct metallurgical zones on cross-section called the Main, Lower, and Jasperoid zones.
The variability of recovery in the deposit was determined by taking samples from the three zones, Main, Lower and Jasperoid and using 10M bottle roll tests to confirm the projected recoveries. The samples were collected and collated according to the design Phase 1, Phase 2 and Phase 3 mining pits. Each of these phases refers to a different mining sequence at an increasing depth. In general, the variability testing supported the column leach test results shown in Table 1.1. There were two areas, one in the Main Zone of the north section of the design Phase 1 mine pit and one in the Main Zone of the north section of the design Phase 3 mine pit, which had lower recoveries than the bulk of the Main Zone, and these lower recoveries were incorporated into the reserve calculations.

Bottle roll tests conducted on Main, Lower and Jasperoid zone material support the conclusion that gold recovery in a heap leach is not dependent on a crush size between 3/8 inch and -2 inch that would be typical for a heap leach. For the Jasperoid and Lower zones it is likely that the ultimate recovery is not dependent on feed size but rather on leach time.

The column-leach and permeability tests indicate that agglomeration is required in order to achieve hydraulic conductivity and a corresponding gold recovery on a consistent basis. Tests conducted in 2018 on blends of high fines/low fines material from the bulk samples taken from the Main Zone indicated that permeability of these ore samples was dependent on the amount of contained minus 200M material. The results from two different laboratories indicated that for these samples of blended Main Zone ore, the permeability of the heap could be maintained at the planned application rate up to a maximum stacking height of 200 feet provided the amount of fine material could be controlled.

The planned processing method is to blend ores to a fines content (to be optimized during operations), primary crush the ore (80 percent -3 inch crush size), agglomerate the material with an average of 8 lb/ton of cement as a binder, heap-leach the material using dilute cyanide solutions for the recovery of gold and silver, recover the precious metals using carbon adsorption columns. The gold loaded carbon is eluted via strip vessel using sodium hydroxide, high temperature and pressure. The eluted pregnant solution then reports to the electrowinning cells. The final step is smelting the electowinning sludge to produce a doré bar. This method of processing and recovering the gold and silver values is supported by the testwork conducted to date and is technology that is known and practiced in the gold mining industry

Projected operational parameters based on multiple years of testwork summarized in Table 1.1 demonstrate that the major Relief Canyon rock types contained in the Main, Jasperoid and Lower Zones, generally would be amenable to heap-leach cyanidation treatment.

Table 1.1 Project Operational Metallurgical Parameters

Zone	Sample	Crush Size	Au Recovery (%)	Ag Recovery (%)	Solution to Ore Ratio	NaCN (lb/ton)	Cement (lb/ton)	Heap Leach Time (days)
Main	BS-1 & BS-2-NC	80% -3"	87	36	2:1	0.30	8.0	135
Lower	Composites 2 and 4 (2015)	80% -3"	77	18	5.1:1	0.60	8.0	270
Jasperoid	Composites 1 and 3 (2015)	80% -3"	77	28	7.8:1	1.20	8.0	540

Table 1.2 shows the tonnages and grades of material with a grade of 0.005 oz Au/ton, (0.025 oz Au/ton for mix or sulfide) that is within the final pit summarizing the metallurgical zones as defined by drilling and modeling through 2016.

Table 1.2 Metallurgical (Mineral) Zones

Material	Tons Placed	Grade Placed	Ounces Placed	Ounces Produced	Cumulative Rec.	% of Total
	000's	oz au/t	000's	000's	%	Ounces
Main	21,337	0.017	369	314	85%	58%
Lower	4,133	0.028	116	89	77%	18%
Jasperoid	4,766	0.031	146	105	72%	23%
Totals	30,237	0.021	631	509	81%	100%

Cumulative recoveries summarize the variability of the deposit and reflect certain areas that have different geological characteristics and lower recoveries. The cumulative recoveries in the Jasperoid zone include a minor tonnage of mixed and sulfide materials that have lower estimated gold recoveries than oxide materials.

1.6 Mineral Resource Estimation

The mineral resources at Relief Canyon were modeled and estimated by evaluating the drill data statistically, utilizing the geologic interpretations provided by Pershing Gold to interpret mineral domains on cross sections spaced at 50-foot intervals, rectifying the mineral-domain interpretations on long sections spaced at 10-foot intervals, analyzing the modeled mineralization geostatistically to establish estimation parameters, and estimating grades into a three-dimensional block model. All modeling of the Relief Canyon resources was performed using GEOVIA Surpac^{TM} mining software.

The Relief Canyon mineral resources are listed in Table 1.3 using a cutoff grade of 0.005 oz Au/ton for oxide material, 0.01 oz Au/ton for mixed material, and 0.02 oz Au/ton for sulfide material. The oxide and mixed cutoffs were chosen to capture mineralization potentially available to open-pit extraction and heap-leach processing, with the higher cutoff for mixed material reflecting the expected reduction in recovered gold. The sulfide cutoff was chosen to reflect the potentially higher costs associated with sulfide processing. Silver grade data are only available for a portion of the deposit. The effective date of the mineral resources estimate is November 1, 2016, and these resources are inclusive of the calculated reserves.

Table 1.3 Relief Canyon Reported Mineral Resources

Class	Cutoff (oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
Measured-Oxide	0.005	14,232,000	0.022	312,000	10,550,000	0.119	1,260,000
Measured-Mixed	0.010	259,000	0.058	15,000	259,000	0.251	65,000
Measured-Total	variable	14,491,000	0.023	327,000	10,809,000	0.123	1,325,000
Indicated-Oxide	0.005	26,854,000	0.016	439,000	6,236,000	0.094	584,000
Indicated-Mixed	0.010	162,000	0.033	5,000	162,000	0.206	33,000
Indicated-Sulfide	0.020	369,000	0.050	18,000	369,000	0.313	115,000
Indicated-Total	variable	27,385,000	0.017	462,000	6,767,000	0.108	732,000
Meas. + Ind Total	variable	41,876,000	0.019	789,000	17,576,000	0.117	2,057,000
Inferred-Oxide	0.005	5,238,000	0.009	45,000	781,000	0.066	52,000
Inferred-Mixed	0.010	4,000	0.018	100	4,000	0.125	1,000
Inferred-Sulfide	0.020	4,000	0.028	100	4,000	0.164	1,000
Inferred-Total	variable	5,246,000	0.009	45,200	789,000	0.068	54,000
Note: rounding may cause apparent inconsistencies							

The Inferred oxide resource total in Table 1.3 includes a historical waste dump/stockpile resource of 23,000 ounces gold at a 0.007 oz Au/ton average gold grade.

Measured resources are restricted to those model blocks defined by Pershing Gold's core holes due to the general lack of QA/QC data that could be used for verification purposes for the historical reverse circulation drilling database, and to some uncertainties in the reverse circulation-based geologic interpretations.

1.7 Estimated Mineral Reserves

This Feasibility Study has calculated Proven and Probable reserves for the deposit with an effective date of May 24, 2018, that are based on the estimated Measured and Indicated resources, which have an effective date of November 1, 2016. Section 16 describes the pit optimization procedure and economic and pit design parameters used in this study. A three-phase pit was designed (see Section 16 for design parameters) that contains the Proven and Probable material shown in Table 1.4 which constitutes the reserves for the property using a gold price of $1,290 per ounce, a cutoff grade of 0.005 oz Au/ton, and the detailed mine economics shown in Sections 21 and 22. The Proven and Probable reserves are contained in the designed Feasibility final pit. The Measured and Indicated resources are inclusive of the estimated reserves. The Proven and Probable reserves include approximately 80 percent of the gold ounces, and approximately 73 percent of the tons within the Measured and Indicated resources. Note the reserves are based on a pre-Feasibility final pit configuration of April, 2017.

Table 1.4 Relief Canyon Mineral Reserves (0.005 oz Au/ton cutoff)

Classification	Tons 000's	Grade oz Au/ton	Oz Au 000's
Proven	13,013.1	0.024	308.4
Probable	17,225.1	0.019	322.6
Proven & Probable	**30,238.1**	**0.021**	**631.0**

In addition, a silver reserve grade can be reported for a portion of the deposit shown in Table 1.4a.

Table 1.4a Reserve Gold Material with a Silver Grade

Classification	Tons 000's	Grade oz Ag/ton	Oz Ag 000's
Proven	10,185.6	0.121	1,232
Probable	3,914.4	0.093	364
Proven & Probable	**14,100.0**	**0.113**	**1,597**

1.8 Mining Methods

The Relief Canyon deposit has been mined in the past by open pit methods, followed by heap leaching. This Feasibility Study considers mining by open pit methods. To determine potentially minable material, a number of pit optimization runs were completed utilizing pit slope parameters developed by Golder and Associates. The property is currently permitted to mine inside a permit boundary down to an elevation of 5,080 feet. These limitations were used to constrain the pit design for design Phase 1. Additionally, a number of pit optimization runs were completed using the base case parameters shown in Table 1.5 and varying the gold price. In the earlier pre-Feasibility study, there was a consideration to not crush lower grade materials. All ore grade materials are now planned to be crushed, so no run-of-mine (“ROM”) material will be shown in the final production schedule.

Table 1.5 Pit Optimization Parameters
(ROM = run-of-mine material)

Mining Cost	**$/ton**		**Comments**
Mining Cost	$1.95		Contract mine
Processing Cost	**Crusher $/ton**	**ROM $/ton**	**Comments**
Crush, Convey, Process	$4.00		
ROM Process		$1.31	
Gold Recovery %	83.0%	65.0%	
Minimum Grade oz Au/ton	0.008	0.005	
Other			
Base Case Metal Price $/oz Au	$1,300	$1,300	
Transport, Refining $/oz Au	$15	$15	

The basis for the design Phase 1 pit utilized a $1,300 gold price and limited the material by the current permit boundary, which includes the 5,080-foot pit bottom elevation. To optimize access to the design Phase 1 pit, minor areas were added along the south and west permit boundary. The design Phase 2 pit design utilized a $600/oz gold price, but the design was altered to allow mining to the final pit walls in the northeast and expanded where necessary to the design Phase 1 outline. The $1,300/oz gold price pit was used to design a final pit (i.e., the design Phase 3 pit shown in Table 1.7).

Table 1.6 summarizes the material contained in the pit phases.

Table 1.6 Material Contained in the Design Pit Phases

Design Phase	Tons Ore 000's	Grade oz Au/ton	Ounces Au 000's	Tons with Silver Grade 000's	Grade oz Ag/ton	Ounces Ag 000's	Tons Rock Waste 000's	Tons Alluvium 000's	Tons Historic Mine Dump 000's	Total Waste Tons 000's	Total Tons 000's
1	10,527.9	0.017	176.5	2,495.5	0.064	159.2	14,994.6	907.9	725.8	16,628.3	27,156.2
2	1,324.6	0.013	16.9	1,213.4	0.048	58.1	16,077.3	4,880.5	0.6	20,958.4	22,283.0
3	18,385.6	0.024	437.6	10,391.2	0.133	1,379.6	72,305.4	5,176.3	1,791.3	79,273.1	97,658.7
Totals	**30,238.1**	**0.021**	**631.0**	**14,100.0**	**0.113**	**1,597.0**	**103,377.4**	**10,964.7**	**2,517.7**	**116,859.7**	**147,097.9**

A production schedule was developed based on crushing a maximum of 6.0 million tons per year and a maximum annual production, including waste, of a little over 31 million tons of material. Material to be processed is defined by a cutoff grade of 0.005 oz Au/ton or greater, and with a Measured or Indicated classification. The base case of this study assumes a contract miner will complete the mining, transport the ore to a crusher, and load the crusher using a front-end loader. The owner will crush, reclaim and stack material utilizing conveyor systems. The production schedule was based on assumptions that a permit to mine below 5,080 feet will be applied for during June, 2018, and the (Permitting) Phase II permit will be received during the fourth quarter of 2019. This allows the production schedule about a year of cushion to account for any permitting delays. If construction for the project does not start until after the third quarter of 2018, the cushion will increase. The production schedule assumes an eight-month construction period and a six-month pre-production mining period.

Three main design pit phases were utilized from the pit optimization results as templates for the pit design. All of the design phases were split into at least two scheduling pit phases. A total of seven scheduling pit phases were used to develop the production schedule for this study. Table 1.7 shows the relationship of the design pit phases to the scheduling pit phases.

Table 1.7 Design and Scheduling Pit Phases

Design Pit Phase	Top Bench (Ft)	Bottom Bench (Ft)	Schedule Pit Phase	Description	Pit Limits
1	5,720	5,080	1	Design Phase 1 to 5080 Elevation	Phase I permit - low stripping & optimization
2	5,600	5,080	2	Design Phase 2 to 5080 Elevation	Phase I permit & optimization
1	5,060	5,000	3	Design Phase 1 from 5060 to 5000 Elevation	permit boundary & optimization
2	5,060	4,740	4	Design Phase 2 from 5060 to 4740 Elevation	permit boundary & optimization
3	5,720	4,520	5	North portion of design Phase 3	optimization
3	5,440	4,940	6	Establish south temporary ramp system to 4940	establish temporary ramp (preliminary design)
3	5,440	4,840	7	Complete south design Phase 3 - mine out south ramp	optimization

Table 1.8 shows the material moved each year in the production schedule from the scheduling pit phases.

Table 1.8 Mine Production Schedule

Scheduling Phase	Material	Units	Pre-production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Phase 1	Ore to Crusher	000's Tons	24	5,377	5,126	-	-	-	-	10,526
		Oz Au/ton	0.014	0.016	0.017	-	-	-	-	0.016
		000's oz Au	0	86	88	-	-	-	-	174
Phase 2	Ore to Crusher	000's Tons	-	510	807	8	-	-	-	1,324
		Oz Au/ton	-	0.009	0.015	0.008	-	-	-	0.013
		000's oz Au	-	4	12	0	-	-	-	17
Phase3	Ore to Crusher	000's Tons	-	-	-	1,386	-	-	-	1,386
		Oz Au/ton	-	-	-	0.016	-	-	-	0.016
		000's oz Au	-	-	-	23	-	-	-	23
Phase4	Ore to Crusher	000's Tons	-	-	-	3,835	1,432	-	-	5,267
		Oz Au/ton	-	-	-	0.020	0.029	-	-	0.022
		000's oz Au	-	-	-	78	41	-	-	118
Phase5	Ore to Crusher	000's Tons	-	-	-	688	3,084	3,571	1,174	8,518
		Oz Au/ton	-	-	-	0.020	0.022	0.027	0.027	0.025
		000's oz Au	-	-	-	14	67	97	32	210
Phase6	Ore to Crusher	000's Tons	-	-	-	-	-	972	-	972
		Oz Au/ton	-	-	-	-	-	0.029	-	0.029
		000's oz Au	-	-	-	-	-	29	-	29
Phase7	Ore to Crusher	000's Tons	-	-	-	-	-	25	2,217	2,242
		Oz Au/ton	-	-	-	-	-	0.025	0.027	0.027
		000's oz Au	-	-	-	-	-	1	61	62
Total	Ore to Crusher	000's Tons	24	5,886	5,932	5,918	4,516	4,568	3,392	30,237
		Oz Au/ton	0.014	0.015	0.017	0.019	0.024	0.028	0.027	0.021
		000's oz Au	0	90	100	114	108	126	93	631
Phase 1	Waste to Dump	000's Tons	1,661	8,799	6,170	-	-	-	-	16,630
Phase 2	Waste to Dump	000's Tons	-	6,951	13,770	238	-	-	-	20,958
Phase 3	Waste to Dump	000's Tons	-	-	-	865	-	-	-	865
Phase 4	Waste to Dump	000's Tons	-	-	-	7,267	2,473	-	-	9,740
Phase 5	Waste to Dump	000's Tons	-	-	-	10,660	21,682	8,222	2,501	43,066
Phase 6	Waste to Dump	000's Tons	-	-	-	-	2,618	9,824	-	12,443
Phase 7	Waste to Dump	000's Tons	-	-	-	-	-	8,086	5,124	13,210
Total	Mine Waste Dump	000's Tons	25	701	-	-	578	1,214	-	2,518
Total	Alluvium	000's Tons	382	2,255	3,147	788	2,905	1,476	7	10,961
Total	Rock Waste	000's Tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618	103,433
Total	Total Waste	000's Tons	1,945	15,466	19,939	19,029	26,774	26,132	7,625	116,911
Total	Total Mined	000's Tons	1,969	21,353	25,872	24,947	31,290	30,701	11,017	147,148
Total	Strip Ratio	W:O	80.28	2.63	3.36	3.22	5.93	5.72	2.25	3.87

A six month mine pre-production period (Year -1 or Pre-production) is required to establish initial mine access roads and to mine and stockpile ore for the production startup in Year 1 when gold production begins. Access to the upper elevation material of the design Phase 1 and 2 pits will be difficult and some material is planned to be dozed to lower elevations so the fleet of loaders and trucks have the necessary room to place mined material into the trucks.

A minor amount of ore grade material will be stockpiled due to monthly crusher capacity. The material to be mined and stockpiled at the crusher is shown in Table 1.8. Because of these stockpiles there are slight differences between the tons hauled from the pit and the material crushed and transported to the leach pad shown in Table 1.9.

Table 1.9 Process Production Schedule

Ore Crushed	Item	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Main	Tons Placed	K tons	-	5,524	5,422	4,081	1,806	2,291	2,213	21,337
Material	Grade Placed	Oz Au/ton	-	0.015	0.017	0.016	0.013	0.019	0.027	0.017
	Ounces Placed	K Ozs Au	-	86	91	64	24	44	61	369
	Produced	K Ozs Au	-	67	76	57	18	38	58	314
	Cumulative Rec.	%		79%	82%	84%	83%	83%	85%	85%
Lower	Tons Placed	K tons	-	166	283	903	1,393	1,198	191	4,133
Material	Grade Placed	Oz Au/ton	-	0.012	0.014	0.029	0.031	0.029	0.031	0.028
	Ounces Placed	K Ozs Au	-	2	4	26	43	34	6	116
	Produced	K Ozs Au	-	1	3	15	36	25	10	89
	Cumulative Rec.	%		79%	79%	78%	76%	75%	76%	77%
Jasperoid	Tons Placed	K tons	-	43	285	1,024	1,348	1,052	1,015	4,766
Material	Grade Placed	Oz Au/ton	-	0.009	0.022	0.025	0.030	0.044	0.026	0.031
	Ounces Placed	K Ozs Au	-	0	6	26	40	47	27	146
	Produced	K Ozs Au	-	0	4	15	30	30	26	105
	Cumulative Rec.	%		51%	62%	60%	68%	67%	72%	72%
Total	Tons Placed	K tons	-	5,732	5,990	6,008	4,547	4,541	3,419	30,237
All	Grade Placed	Oz Au/ton	-	0.015	0.017	0.019	0.024	0.028	0.027	0.021
Material	Ounces Placed	K Ozs Au	-	88	101	115	108	125	94	631
	Produced	K Ozs Au	-	69	83	86	84	94	93	509
	Cumulative Rec.	%		78%	80%	78%	78%	77%	81%	81%
Ore with Silver	**Item**	**Units**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Total**
Main	Tons Placed	K tons	-	737	2,085	1,346	579	457	0	5,205
Material	Grade Placed	Oz Ag/ton	-	0.023	0.072	0.117	0.081	0.062	0.040	0.017
	Ounces Placed	K Ozs Ag	-	17	151	158	47	28	0	401
	Produced	K Ozs Ag	-	3	45	56	15	11	0	131
	Cumulative Rec.	%		20%	29%	32%	32%	33%	33%	33%
Lower	Tons Placed	K tons	-	166	283	903	1,393	1,198	187	4,129
Material	Grade Placed	Oz Ag/ton	-	0.030	0.068	0.149	0.109	0.120	0.132	0.017
	Ounces Placed	K Ozs Ag	-	5	19	134	152	144	25	479
	Produced	K Ozs Ag	-	0	2	15	31	19	12	80
	Cumulative Rec.	%		8%	10%	11%	16%	15%	17%	17%
Jasperoid	Tons Placed	K tons	-	43	285	1,024	1,348	1,052	1,015	4,766
Material	Grade Placed	Oz Ag/ton	-	0.044	0.052	0.124	0.135	0.202	0.176	0.017
	Ounces Placed	K Ozs Ag	-	2	15	127	182	213	178	717
	Produced	K Ozs Ag	-	0	2	15	47	47	65	176
	Cumulative Rec.	%		11%	14%	12%	20%	21%	24%	24%
Total	Tons Placed	K tons	-	945	2,653	3,272	3,321	2,707	1,202	14,100
with silver	Grade Placed	Oz Ag/ton	-	0.025	0.070	0.128	0.115	0.142	0.169	0.017
assay	Ounces Placed	K Ozs Ag	-	24	185	419	381	385	203	1,597
	Produced	K Ozs Ag	-	4	50	86	93	77	77	386
	Cumulative Rec.	%		17%	26%	22%	23%	22%	24%	24%

1.9 Recovery Methods

Processing will be conducted using a conventional heap leach with ADR (Adsorption, Desorption, Recovery) circuit. Ore will be crushed to 80 percent passing 3 inches, belt agglomerated using cement, and conveyed and stacked on the heap leach pad. Stacked ore will be leached with a dilute sodium cyanide solution and the resulting pregnant solution will be processed in an ADR plant for the recovery of precious metals from solution. The resulting gold and silver sludge from the ADR plant will be treated in a mercury retort and smelted to produce doré bars.

A single-stage crushing plant will process up to 16,700 tons daily, or a maximum of 6.0 million tons annually. The mine has an estimated life of 5.6 years.

Approximately 30.2 million tons of ore are planned for stacking and leaching over the life of mine. The Pad 5, 6, 7 heap leach will include both new and existing lined pad area for a combined total of approximately 3.3 million square feet of pad area. The existing Operating Pond East ("OPE") and Operating Pond West ("OPW") process ponds will be used for solution containment and management; no additional ponds are required during the first two to three years of operation. In Year 3, the leach pad will be expanded by an additional 2.2 million square feet and an additional process solution pond will be constructed for storm water management to provide additional process solution containment capacity and operational flexibility.

The gold recovery plant will process a nominal 3,000 gallons per minute of pregnant heap leach solution and consists of existing carbon-in-column adsorption, pressure strip, and refining circuits, a new electrowinning circuit and new melt furnace. The existing plant also includes carbon acid washing and regeneration. The plant will be retrofitted with the mercury control equipment authorized in the Mercury Operating permit, which includes a mercury retort, sulfur-impregnated carbon circuits and baghouse for removing mercury from process equipment exhaust gases.

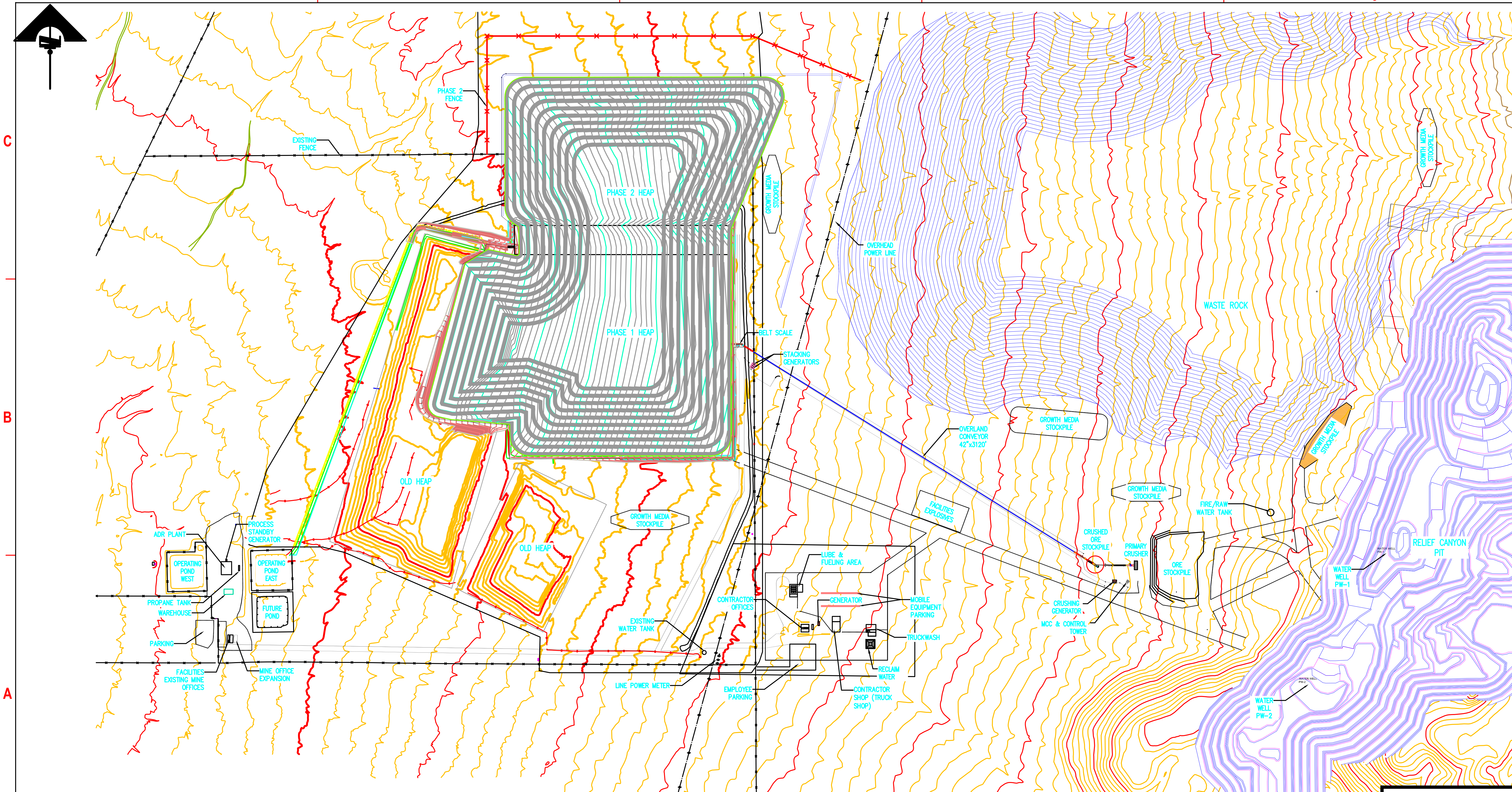
1.10 Infrastructure

Significant infrastructure currently exists at the Relief Canyon site from previous operations. Existing installations include site access and haul roads, ADR facility, process solution ponds, heap-leach pad, waste rock facilities, site buildings, electrical power supply, water wells, and fencing around the process facilities. Pershing Gold intends to use as much of the existing infrastructure as possible.

Electrical power will be by a combination of line power and diesel-fired generator units. Approximately 41% of the electrical demand will be supplied by line power to most of the existing infrastructure, including the existing administration building, mine offices, warehouse, ADR plant, and process solution management. The remaining demand for the crushing plant, overland conveyor, and heap stacking conveyor system will be met by installing local diesel generators. Mine facilities such as a truck shop, truck wash, mine office complex, and fuel station will be constructed, operated, and maintained by the mining contractor. Mining facilities will be constructed during Year 1 of operation.

The water demands for the project include make up water for the process facilities, fire water, crushing area dust suppression, road dust suppression, and potable water supply for the offices. Water for mining, the heap-leach facilities, fire suppression, and other uses will initially be supplied by existing production wells located west of the pit area. It is estimated that the total average site demand will be approximately 630 gpm in the summer months and 430 gpm in the winter months. Two new deeper wells will need to be constructed to replace the existing wells for future pit dewatering when the existing wells are ultimately mined out.

Water from the production wells will be pumped to a new raw/fire water storage head tank on the western side of the pit, just south of the ROM stockpile. This tank will be sized to contain the necessary fire water and process / raw water reserves and will provide raw process water for the crushing and stacking plant, and mine facilities such as the truck shop, truck wash, and mine offices. Figure 1.1 shows the general arrangement of the overall site. A new water tank will be connected to the existing process plant water tank to add additional capacity. The pit, waste area, and heap are shown at the end of the mine life in Figure 1.1.



PLAN VIEW

SCALE : 1"=30'

0 120 240 480
FEET

PRELIMINARY
NOT FOR CONSTRUCTION

REFERENCE DRAWINGS

DWG. NO.	TITLE

REVISIONS

NO.	BY	CK'D	APPROVED	DATE	DESCRIPTION

REVISIONS

NO.	BY	CK'D	APPROVED	DATE	DESCRIPTION
A	EDZ	GML		09JAN18	ISSUED FOR REVIEW
B	WCC	GML		01MAR18	ISSUED FOR REVIEW
C	PR	GML		29MAY18	ISSUED FOR REVIEW

DESIGNED BY	MPS
DRAWN BY	WCC
CHECKED BY	
APPROVED 1	
APPROVED 2	
CLIENT APP'L	

PREPARED FOR
pershing gold

PREPARED BY
Kappes, Cassiday & Associates
Reno, Nevada
KCA

Project: **RELIEF CANYON**

Title: FEASIBILITY STUDY
GENERAL ARRANGEMENT
OVERALL SITE LAYOUT
(ULTIMATE PIT & HEAP)

Job No.	Scale	ACAD Filename:
643 I	AS SHOWN	643-GA510.A.dwg

643	GA510	REV C

NAME: \\mda.com\Users\Neil\Relief_Canyon\2018_Feas\2018_EOY_Drawings\643-GA510_C 29May18\643-GA510_MDA.dwg LAST UPDATE: 13-Jun-18, 8:58 AM PLOT DATE:13-Jun-18 BY: CWALKER PLOT SCALE: 1:2.1053

1.11 Environmental Studies and Permitting

Currently, the Relief Canyon project has the following major environmental permits: 1) a Plan of Operations ("Plan") from the BLM; 2) a Nevada Reclamation Permit ("NRP") with the Nevada Division of Environmental Protection's ("NDEP") Bureau of Mining Regulation and Reclamation ("BMRR"); 3) a Water Pollution Control Permit ("WPCP") with the BMRR; 4) Water rights from the Nevada Division of Water Resources ("NDWR"); 5) Class I and Class II Air Quality Operating Permits ("AQOP") with the NDEP's Bureau of Air Pollution Control ("BAPC"); 6) a Mercury Operating Permit ("MOP") to Construct with the BAPC; and 7) a Special Use Permit from Pershing County. All of Pershing Gold's permitting efforts are conducted through GAC as the permittee. As discussed below, GAC has all of the state and federal permits necessary to start the Phase I mining and heap-leach processing operations.

Pershing Gold is planning a two-phase permitting and development scenario for the project. Permittng Phase I, which has been approved, is the re-purposing of previously approved disturbance for expanded mining to a pit bottom elevation of 5080 feet, partial backfilling of the design Phase I pit to approximately 20 feet above the historical groundwater elevation to eliminate a pit lake, expanded exploration operations, full build-out of the heap-leach pad space and stacking to a heap height of 200 feet, and construction of a new waste rock storage facility ("WRDF 5"). Permitting Phase II will include additional mine expansion activities and allow mining further below the water table. The Phase II permit will be applied for shortly after the completion of this Feasibility Study, which will be the basis for the mine plan to be submitted in the permit application.

During Permitting Phase I, Pershing Gold will expand the existing open pits creating one larger pit, build the new WRDF 5 on private land, conduct exploration activities outside of the existing pit area, and construct ancillary facilities. Permitting Phase I includes using 211.8 acres of previously authorized, but currently unused surface disturbance. The proposed disturbance would be needed for mine expansion and mineral exploration activities. The mined ore would be processed on the previously permitted heap-leach pad Cells 5, 6A, 6B, 7A and 7B, of which only one of the four permitted cells, Cell 6A, has been constructed.

Pershing Gold is in the final stage of planning and conducting baseline studies for the initiation of the permitting for Phase II of development at the mine. The design for Phase II will include a larger and deeper open pit that will be closed with a pit lake, an expanded waste rock storage area and two additional heap-leach cells. All of these facilities will be within the existing project boundary. In order to construct, operate, reclaim, and close mining operations during Phase II at the mine, Pershing Gold will be required to modify and obtain a number of environmental and other permits from the BLM, the NDEP, NDWR, and Pershing County. The principal permits necessary for the Phase II mine development are modifications to: 1) the Plan of Operations with the BLM; 2) the NRP with the BMRR; 3) the WPCP with the BMRR; 4) the dewatering water rights from the NDWR; and 5) the Special Use Permit with Pershing County. In order to obtain these permits, applications need to be submitted to each agency. In the case of the Plan and the NRP, there is a single application (Plan Application) that meets the requirements of both the BLM and BMRR.

Permitting Phase I is fully bonded. The current reclamation bond for the project is $12,398,386. For Phase II of the project the total bond is expected to decrease to approximately $9,000,000 due to the elimination of the need for the Phase I pit backfilling. The current annual surety fee for the bond is approximately $300,000. The bond annual fees for the surety under Phase II of the project will be approximately $220,000. Reclamation costs are expected to total about $8 million spread over three years, after mining has been completed.

1.12 Capital and Operating Costs

The capital and operating cost estimate is based on mining contractor detailed quotes for mining the deposit, and cost estimates derived from first principles based on quotes for major items. In addition, the crushing plant, stacking system, mobile equipment and the diesel generators are initially leased (to own) for a period of three years. The initial down payment on leased equipment is shown as a capital cost, while the leasing cost is shown as an operating cost. The base case assumes equipment leasing, and a second case was also considered to purchase all of the required process equipment. Most costs estimates were collected in the first or second quarter of 2018; all prices were quoted in US dollars and have an estimated accuracy of +/- 15 percent. Working capital has been included in Year 1 to total 2.5 months of operating costs, recovered in the following year. Table 1.10 shows the base case estimated capital cost for the mine.

Table 1.10 Estimated Base Case Capital Cost ($000's)

Activity	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
Process Equipment Leased										(Year 1 thru 8)
Process - General Facilities	303.7									
Process - Mobile Equipment	327.2									
Process Plant	2,220.8									
Crushing Plant & Reclaim	1,360.3									
Heap Leach & Solution Handling	8,986.4		208.0	6,220.4						6,428.4
Water Facilities	612.0									
Power Distribution	933.0									
Process Commisioning and Supervision	65.0									
Process Spare Parts	521.0									
Process First Fills	262.4									
Process Preproduction Labor	170.2									
Process Preproduction	0.0									
Owners Cost (Includes Preproduction G & A)	1,024.8									
Belt Stacking System	1,521.8									
Mine Contractor Facilities			660.0							660.0
Water Wells / Pit Dewatering			3,127.5	848.7						3,976.2
Fire Water Network		325.9								325.9
Mine - Radios, WiFi, Survey Equipment, Computers	200.0									0.0
Mine - Contractor Facilities (Shop, Fuel Storage)										
Mine - Preproduction	5,553.7									
Mine - Ramp System outside Pit										0.0
Mine - Light Vechicles	307.4			219.6						219.6
Reclamation							1,000.0	4,000.0	3,000.0	8,000.0
Return of Bond Collateral									(3690.0)	(3690.0)
Salvage of mine and process equipment							(4000.0)			(4000.0)
Salvage of existing crushing not used	(459.0)									
Subtotal	**23,910.7**	**325.9**	**3,995.5**	**7,288.6**	**0.0**	**0.0**	**(3,000.0)**	**4,000.0**	(690.0)	**11,920.1**
Mine Contingency	555.4	0.0	0.0	0.0						0.0
Process Contingency	2,848.4	48.9	500.3	1,060.4	0.0	0.0				1,609.6
EPCM	580.0	19.6	200.1	424.1	0.0	0.0				643.8
Indirects	344.3	19.6	200.1	424.1	0.0	0.0				643.8
Subtotals	28,238.8	413.9	4,896.1	9,197.3	0.0	0.0	(3000.0)	4,000.0	(690.0)	14,817.4
Working Capital		10,157.1	(10157.1)					0.0		
Totals	**28,238.8**	**10,571.0**	**(5,260.9)**	**9,197.3**	**0.0**	**0.0**	**(3000.0)**	**4,000.0**	**(690.0)**	**14,817.4**

The costs presented have been estimated using information provided by Pershing Gold, MDA, Kappes Cassiday and Associates ("KCA"), and their contractors. All equipment and material requirements are based on the design information described in this study. Capital cost estimates have been made using budgetary quotes from contractors and suppliers for most major items. Other items were estimated from consultants via their own databases.

A number of facilities and equipment already exist on site at Relief Canyon. Costs are based on a combination of the purchase of new equipment items, repair or refurbishment of existing items and facilities, and purchase of used equipment where reasonable.

A significant amount of new equipment will be financed, including the crushing plant, stacking system, mobile equipment and the diesel generators. The financing includes the equipment supply only without any installation costs. Financing will be with a single lender. Financed equipment is assumed to include a 20 percent down payment in pre-production, with five-year lease terms paid for the first three years and a balloon payment in Month 37 of the contract (Year 4). The down payment is included as a pre-production capital cost and all monthly and balloon payments are included as operating costs.

Operating costs for all areas have been estimated from first principles and include equipment data, vendor information and typical industry values. Mining costs are estimated based on a contract mining proposal. Labor costs are estimated using project-specific staffing, salary, wage, and benefit requirements. Unit consumptions of materials, supplies, power, water, and delivered supply costs are also estimated and are based on testwork, vendor quotes, and similar recent project data.

All operating costs are presented in first or second quarter 2018 dollars. The costs are believed to have an accuracy of +/-15 percent. No contingency has been added to the process operating costs. Year 6 process operating costs represent seven months of production plus continued operation of the heap-leach irrigation system and recovery plant, as additional gold is expected to be recovered. Operating costs for supporting infrastructure are included during the last five months of Year 6, at a reduced capacity.

Table 1.11 shows the estimated operating cost. The base case operating cost estimate indicates an average cost of $768.60 to produce an ounce of gold. The gold and silver prices used for this study are $1,290 per ounce of gold and $16.75 per ounce of silver. Note that year 7 operating cost is included in reclamation, (i.e., pad rinsing).

Table 1.11 Operating Cost Estimate

Item	Units	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Production Totals
Dozed Material	000's tons	131	114		141	147	120			522
Ore	000's tons	24	5,886	5,932	5,918	4,516	4,568	3,392		30,212
Waste Dump Material	000's tons	25	701	-	-	578	1,214	-		2,492
Alluvium	000's tons	383	2,255	3,147	788	2,905	1,476	7		10,579
Rock Waste	000's tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618		101,896
Total Waste	000's tons	1,945	15,467	19,939	19,029	26,774	26,132	7,625		114,967
Total Material		1,969	21,353	25,872	24,947	31,290	30,701	11,017		145,179
Crushed Material Summary										
Tons	000's tons		5,732	5,990	6,008	4,547	4,541	3,419		30,237
Grade	oz Au/ton		0.015	0.017	0.019	0.024	0.028	0.027		0.021
Ounces	000's ounces		88	101	115	108	125	94		631
Total Silver Produced	000's ounces		4.01	51.84	93.58	101.22	84.18	85.25	11.30	431.38
Total Gold Produced	000's ounces		68.66	83.27	86.39	83.85	93.52	93.17	0.65	509.51
Revenue	$000's		$88,576.8	$107,418.7	$111,441.9	$108,164.1	$120,639.7	$120,194.6	$837.8	$657,273.5
Refining and Transportation	000's		$686.6	$832.7	$863.9	$838.5	$935.2	$931.7	$6.5	$5,095.1
Royalties (2.15%)	$000's		$1,888.3	$2,273.8	$2,345.2	$2,272.7	$2,544.7	$2,534.8	$14.0	$13,873.4
Net Sales	$000's		$86,001.9	$104,312.2	$108,232.7	$105,052.9	$117,159.8	$116,728.0	$817.3	$638,304.9
Operating Cost										
Silver Credit	$000's		($64.2)	($829.4)	($1,497.2)	($1,619.6)	($1,346.8)	($1,364.1)	($180.8)	($6,902.1)
Mining	$000's		$41,670.4	$46,331.3	$49,889.0	$61,942.2	$64,225.1	$24,035.4		$288,093.5
Load Crusher	$000's		$2,121.0	$2,216.3	$2,222.8	$1,682.3	$1,680.2	$1,265.2		$11,187.7
Processing (Lease)	$000's		$15,760.0	$16,328.0	$16,728.0	$15,292.0	$12,364.0	$9,723.0		$86,195.0
G & A	$000's		$2,450.2	$2,450.2	$2,148.2	$2,148.2	$2,148.2	$1,675.6		$13,020.8
Total Operating Cost	$000's		$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$391,594.8
Cost $/ton Ore	$/ton ore		10.80	11.10	$11.57	17.47	17.41	10.33		$12.95
Cost $/ounce Au recovered	$/ounce Au		$902.03	$798.56	$804.39	$947.49	$845.50	$379.24		$768.56
Net after Operating Costs	$000's		$24,064.5	$37,815.7	$38,741.9	$25,607.8	$38,089.2	$81,392.8	$998.1	$246,710.1

1.13 Economic Analysis

The economic analysis of the Relief Canyon mine was completed both on a pre-tax and after-tax basis. Table 1.12 presents the pre-tax, base case cash-flow model of the mine using a $1,290 per ounce gold price and $16.75 per ounce silver price. The base case has a pre-tax net present value ("NPV") at a 5 percent discount rate of $153.7 million with an internal rate of return ("IRR") of 91.0 percent. The payback period is about 1.3 years. The case using purchase of the processing equipment resulted in lower NPV and IRR, and higher initial capital than the base case.

Table 1.12 Pre-Tax Base Case Cash Flow

Item	Units	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Production Totals
Dozed Material	000's tons	131	114		141						255
Ore	000's tons	24	5,886	5,932	5,918	4,516	4,568	3,392			30,212
Waste Dump Material	000's tons	25	701	0	0	578	1,214	0			2,492
Alluvium	000's tons	383	2,255	3,147	788	2,905	1,476	7			10,579
Rock Waste	000's tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618			101,896
Total Waste	000's tons	1,945	15,466	19,939	19,029	26,774	26,132	7,625			114,967
Total Material		1,969	21,353	25,872	24,947	31,290	30,701	11,017			145,179
Crushed Material Summary											
Tons	000's tons		5,732.4	5,990.0	6,007.6	4,546.6	4,541.0	3,419.4			30,237
Grade	oz Au/ton		0.015	0.017	0.019	0.024	0.028	0.027			0.021
Ounces	000's ounces		87.9	100.9	115.2	108.2	125.4	93.7			631.3
Total Silver Produced	000's ounces		4.0	51.8	93.6	101.2	84.2	85.3	11.3		431.4
Total Gold Produced	000's ounces		68.7	83.3	86.4	83.8	93.5	93.2	0.6		509.5
Revenue	$000's		$88,576.8	$107,418.7	$111,441.9	$108,164.1	$120,639.7	$120,194.6	$837.8		$657,273.5
Refining and Transportation	000's		$686.6	$832.7	$863.9	$838.5	$935.2	$931.7	$6.5		$5,095.1
Royalties (2.15%)	$000's		$1,888.3	$2,273.8	$2,345.2	$2,272.7	$2,544.7	$2,534.8	$14.0		$13,873.4
Net Profit	$000's		$86,001.9	$104,312.2	$108,232.7	$105,052.9	$117,159.8	$116,728.0	$817.3		$638,304.9
Operating Cost											
Silver Credit	$000's		($64.2)	($829.4)	($1,497.2)	($1,619.6)	($1,346.8)	($1,364.1)	($180.8)		($6,902.1)
Mining	$000's		$41,670.4	$46,331.3	$49,889.0	$61,942.2	$64,225.1	$24,035.4			$288,093.5
Load Crusher	$000's		$2,121.0	$2,216.3	$2,222.8	$1,682.3	$1,680.2	$1,265.2			$11,187.7
Processing	$000's		$15,760.0	$16,328.0	$16,728.0	$15,292.0	$12,364.0	$9,723.0			$86,195.0
G & A	$000's		$2,450.2	$2,450.2	$2,148.2	$2,148.2	$2,148.2	$1,675.6			$13,020.8
Total Operating Cost	$000's		$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$0.0	$391,594.8
Cost $/ton Ore			11.0	11.5	12.0	17.2	13.7	6.6			$12.95
Cost $/ounce Au recovered			922.1	824.3	837.0	931.8	636.8	155.8			$768.56
			$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$0.0	$391,594.8
Net after Operating Costs	$000's		$24,064.5	$37,815.7	$38,741.9	$25,607.8	$38,089.2	$81,392.8	$998.1	$0.0	$246,710.1
Cumulative Cashflow	$000's		$24,064.5	$61,880.2	$100,622.2	$126,230.0	$164,319.1	$245,712.0	$246,710.1		
Capital Cost	$000's	$28,238.8	$10,571.0	($5,260.9)	$9,197.3	$0.0	$0.0	($3,000.0)	$4,000.0	($690.0)	
Cash Flow with Capital	$000's	($28,238.8)	$13,493.5	$43,076.7	$29,544.7	$25,607.8	$38,089.2	$84,392.8	($3,001.9)	$690.0	$203,654.0
Cumulative Including Capital	$000's	($28,238.8)	($14,745.3)	$28,331.4	$57,876.1	$83,483.9	$121,573.0	$205,965.8	$202,964.0	$203,654.0	

1.13.1 Pre-Tax Sensitivity

The pre-tax sensitivity was assessed by varying the gold price, operating cost, and capital cost estimates in increments of ± 5 percent. The impact to the project NPV (at a 5 percent discount rate) and IRR are shown in Figure 1.2 and 1.3 respectively.

Figure 1.2 Pre-Tax NPV (5%) Sensitivity to Gold Price, Operating Cost and Capital Cost ($000's)

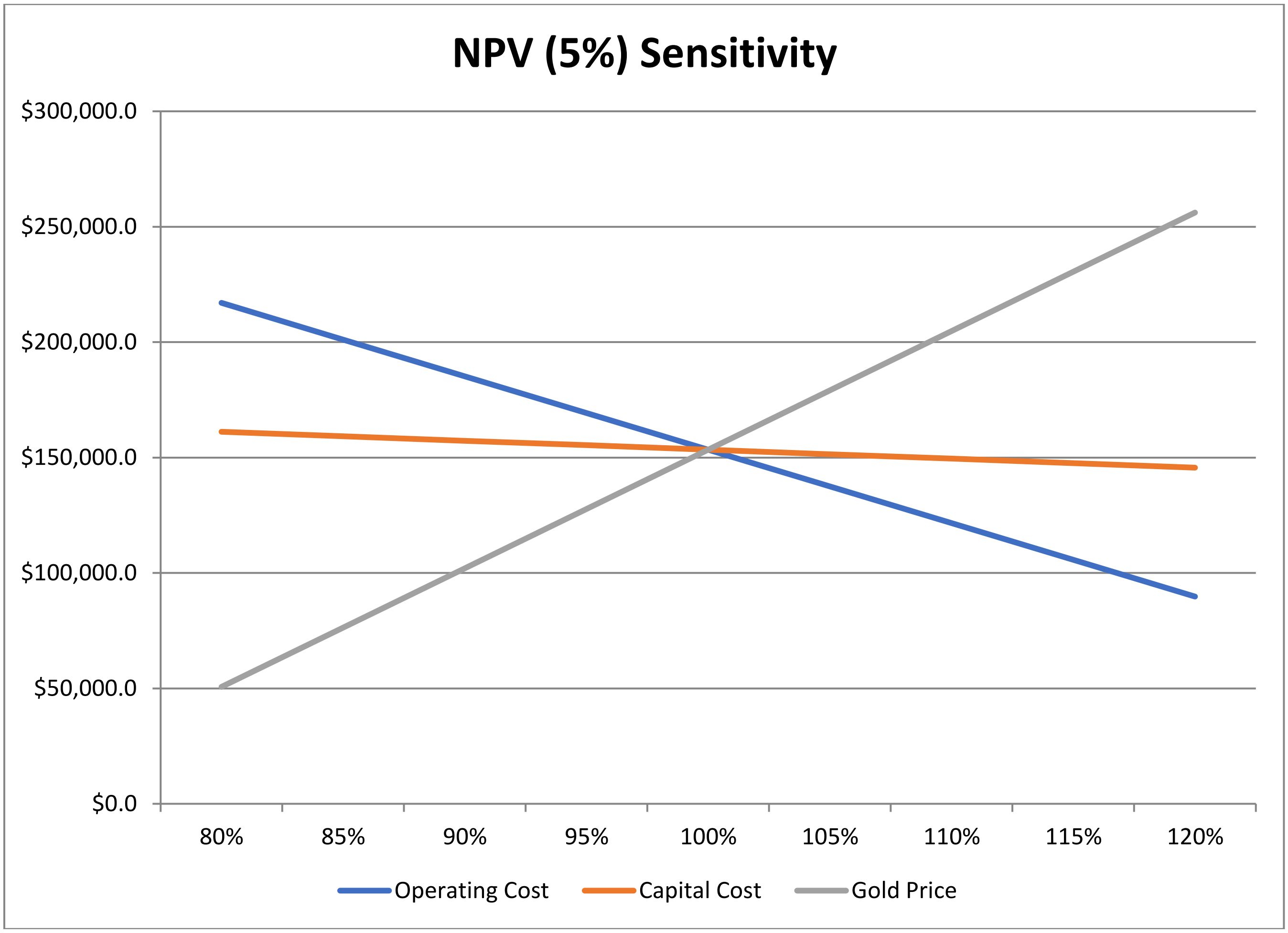
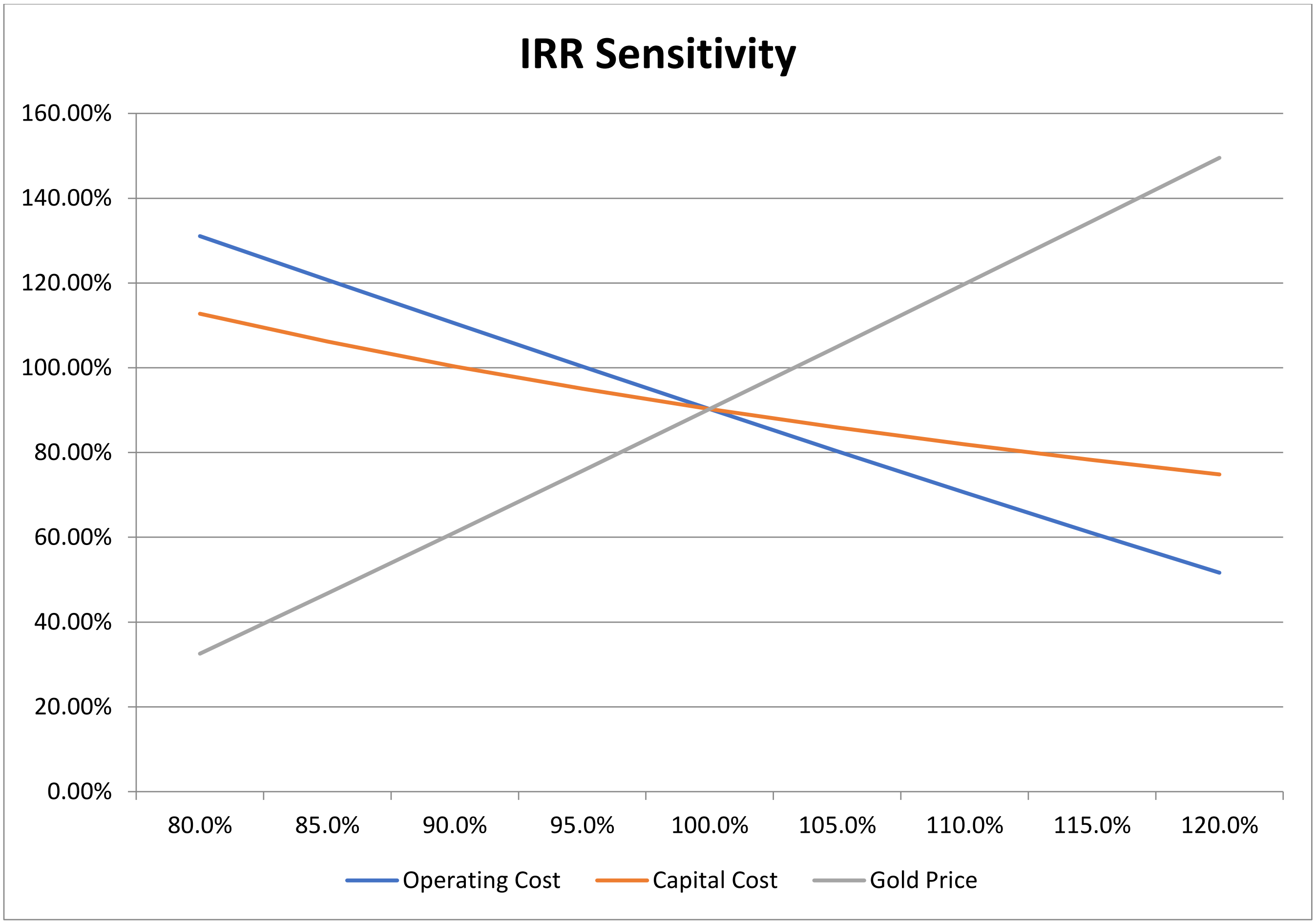


Figure 1.3 Pre-Tax IRR Sensitivity to Gold Price, Operating Cost and Capital Cost



The after-tax, base case cash-flow model is shown in Table 1.13. It should be noted that Pershing Gold has accrued approximately $73.2 million in net operating losses for the property that can be utilized to reduce the mine's taxable income, which will lower the amount of income taxes paid over the life of the project. The use of the accrued net operating losses was considered in the economic analysis.

The after tax, base case cash flow is an estimated $175.7 million. The after tax NPV (at 5 percent discount rate) is $133.2 million and the IRR is 86.5 percent. The economic evaluation reported here indicates that the project should proceed to production.

Table 1.13 Base Case After Tax Cash Flow

Item	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Totals
After Tax Evaluation-Lease Equipment											
Net Profit before Tax	**$000's**		$24,064.5	$37,815.7	$38,741.9	$25,607.8	$38,089.2	$81,392.8	$998.1	$0.0	$246,710.1
Nevada Net Proceeds	**$000's**		$476.6	$1,899.9	$1,295.1	$998.5	$1,652.1	$4,071.1	$0.0	$0.0	$10,393.3
Net after Net Proceeds	**$000's**		$23,587.9	$35,915.8	$37,446.9	$24,609.3	$36,437.1	$77,321.7	$998.1	$0.0	$236,316.8
Depreciation	**$000's**		$3,744.5	$5,295.5	$3,643.4	$5,637.9	$5,046.6	$2,970.1	$0.0	$0.0	$26,338.0
Net before Depletion	**$000's**		$19,843.4	$30,620.3	$33,803.5	$18,971.4	$31,390.4	$74,351.6	$998.1	$0.0	$209,978.7
Depletion (15%)	**$000's**		$12,900.3	$15,646.8	$16,234.9	$15,757.9	$17,574.0	$17,509.2	$0.0	$0.0	$95,623.1
Depletion (50% max)	**$000's**		$9,921.7	$15,310.2	$16,901.7	$9,485.7	$15,695.2	$37,175.8	$0.0	$0.0	$104,490.3
Depletion Taken	**$000's**		$9,921.7	$15,310.2	$16,234.9	$9,485.7	$15,695.2	$17,509.2	$0.0	$0.0	$84,156.9
Taxible Income	**$000's**		$9,921.7	$15,310.2	$17,568.6	$9,485.7	$15,695.2	$56,842.4	$998.1	$0.0	$125,821.8
Loss Carry Forward	**$000's**		$9,921.7	$15,310.2	$17,568.6	$9,485.7	$15,695.2	$5,218.7			$73,200.0
Taxible Income	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$51,623.7	$998.1	$0.0	$52,621.8
Income Tax (21%)	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$17,552.1	$0.0	$0.0	$17,552.1
Income After Tax	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$34,071.6	$998.1	$0.0	$35,069.8
Loss Carry Forward	**$000's**		$9,921.7	$15,310.2	$17,568.6	$9,485.7	$15,695.2	$5,218.7	$0.0	$0.0	$73,200.0
Depletion	**$000's**		$9,921.7	$15,310.2	$16,234.9	$9,485.7	$15,695.2	$17,509.2	$0.0	$0.0	$84,156.9
Depreciation	**$000's**		$3,744.5	$5,295.5	$3,643.4	$5,637.9	$5,046.6	$2,970.1	$0.0	$0.0	$26,338.0
Net After Tax	**$000's**		$23,587.9	$35,915.8	$37,446.9	$24,609.3	$36,437.1	$59,769.6	$998.1	$0.0	$218,764.7
Capital Cost	**$000's**	$28,238.8	$10,788.0	($5,478.0)	$9,197.3	$0.0	$0.0	($3,000.0)	$4,000.0	($690.0)	$43,056.1
After Tax Cashflow	**$000's**	($28,238.8)	$12,799.8	$41,393.8	$28,249.6	$24,609.3	$36,437.0	$62,769.6	($3,001.9)	$690.0	$175,708.6
Cumulative After Tax Cashflow	**$000's**	($28,238.8)	($15,438.9)	$25,954.9	$54,204.5	$78,813.8	$115,250.8	$178,020.4	$175,018.6	$175,708.6	
NPV (5%)	**$000's**										$133,208.4
NPV 7.5%	**$000's**										$116,544.3
NPV 10%	**$000's**										$102,252.6
IRR	**%**										86.5%

1.14 Risks and Opportunities

1.14.1 Risks

The estimated feed to the crusher is 80 percent passing six inches, and the grizzly and jaw appears adequate to process the nominal tonnage at the planned size distribution. There is a moderate risk that the actual feed size may be larger than the anticipated size, which could result in reduced throughputs at the crusher for the target product size.

Based on an initial fire code review and considering the present information on Pershing's existing infrastructure, it is believed that no fire main network will need to be installed in pre-production, and instead is deferred to Year 1 when new permanent structures (the truck shop and permanent mine offices) are added. At the time of construction and/or inspection there is a low to moderate risk that either for insurance or code compliance the pipe network may be required to be installed for some existing infrastructure, increasing pre-production capital costs.

There is a significant amount of used equipment planned for use in the ADR plant areas, and some used conveyors that have not been operated for several years. Although some allowances have been made for repair and refurbishment, there is a risk that during construction and commissioning the need for additional repairs and replacements may be discovered, increasing pre-production costs, and may also present some risk of additional unplanned maintenance during production over what is currently assumed (increasing operating costs). This plan incorporates lease-purchase of new crusher, reclaim, convey and stacking equipment.

There is a risk that a varying amount of fine clayey material that may be present in the pit and that an increase in fine material above what is currently predicted to occur could potentially cause permeability problems that would affect gold recovery and heap stability. The risk is high due to the potential economic impact to gold recovery. This risk can be mitigated by:

- Monitoring the amount of fine clayey material being mined and blending this material with other coarse sandy material
- Adding additional binder cement
- Constructing additional heap leach pad area and lowering the overall height of the heap leach
- Installing an inter-lift pad liner to reduce phreatic conditions in underlying lifts
- A combination of any of these identified scenarios.

Blending of clayey and sandy material is currently the considered method for resolving this risk. The mine plan currently identifies fine material that may be blended with coarser material to ensure permeability. Additionally, the amount of cement addition to the ore may be varied at will in order to improve permeability. In anticipation of this issue, under the current plan the pad liner build-out and stacking plan reaches a maximum height of 140 feet. Pershing Gold has a land position and ample space in the Permitting Phase II Modification that can provide an opportunity to build additional pad if required, which has the potential to further lower the overall stack height should that become necessary.

1.14.2 Opportunities

1.14.2.1 Exploration

There are several opportunities to improve the project. First, the project has a number of targets for resource expansion that should be followed up with more detailed mapping, sampling and drilling. Past production of silver from the deposit indicates that there will be a silver credit from the property, though about 1/3 of the current model resource blocks contain estimated silver grades. Additional silver assaying of the available pulps in continuous interval runs of the mineralized areas should be completed so that silver could be modeled and included for more of the resource.

Since the resource estimate was completed about fifty core holes have been drilled in three areas:

- Infill drilling in the northwest area of the final pit (eight holes to date). Drill hole results so far have indicated higher grades (about 30 percent) than predicted by the resource model. It is likely that this drilling to date will have a positive impact when the resource estimate is updated for the project;

- Extension drilling to the southwest of the north portion of the final pit (seven holes to date). This drilling has mostly been downdip of the past drilling and has intersected similar mineralization to the up-dip drilling;

- Twin hole drilling within the Main zone mineralization southwest of the existing South Pit (eight holes to date plus five infill holes). This drilling has confirmed the results from older drill holes and has also confirmed that the older holes may not have been deep enough. Drilling is continuing to confirm the extent of deeper mineralization.

About 20 percent of the Pershing Gold's 40 square mile land package has been explored. Recent exploration work has generated several targets.

The current resource is open to the west and additional drilling is recommend in this area. However, this material dips to the west and may become too deep to be contained in a future resource pit.

During late 2016, new zones of gold mineralization were identified by drilling southeast of the Lightbulb Pit. Additional drilling is warranted in this area to potentially elevate these zones of mineralization to a mineral resource status.

In addition, infill core drilling is recommended to:

- Identify the primary structural controls on the Main Zone mineralization that may result in the identification of higher-grade targets within and beneath the Main Zone;
- Expand and/or demonstrate continuity of the high-grade gold grade shells;
- The Main Zone is defined primarily by historical reverse circulation drilling. An enhanced geologic understanding of the Main Zone can be obtained by drilling core holes that will allow for improved delineation of the mineralized breccias, including those high-fines breccias that could negatively impact heap percolation. In addition, core drilling will result in an improved

understanding of the structural controls on the Main Zone mineralization that may allow the identification of higher-grade targets within, and at depth beneath, the Main Zone. This work has started with positive results;

- Expand on Phase 2 drilling during 2016 that identified new zones of mineralization southeast of the Lightbulb pit, with potential to elevate these zones of mineralization to a mineral resource status; and
- Continue exploration on several targets on Pershing Gold's 40 square mine land package.

Much of this work is on-going or planned in the future.

1.14.2.2 Silver Credit

Past production from the deposit indicates that there will be a silver credit from the property, but the current database contains limited silver assays within the Main Zone and silver therefore is not included within the resource model and estimate in the Main Zone. At present, approximately 1/3 of the resource blocks have silver grades estimated, and a little less than half of the material planned to be crushed has silver grades estimated. Additional silver data can be obtained by assaying the available Pershing Gold sample pulps, though there is just scattered Pershing Gold drilling within the Main Zone and these core holes do not provide the sample coverage needed for a classification of Measured and Indicated. Additional infill drilling and sampling is required.

1.14.2.3 Historical Mine Dump Inferred Mineralization

There is an Inferred resource contained in a historical waste dump. A total of 42 shallow reverse circulation drill holes have been drilled into this material, along with 20 trenches. About half of this dump is currently planned to be mined as waste in the production schedule. This material should be drilled and sampled as it is mined. It should also be possible to drill the easily accessible material prior to the scheduled production and determine if some of the material should be crushed and agglomerated to supplement production as required.

1.14.2.4 Other Opportunities

Preliminary test results indicate that agglomeration of blended ores with 8 lbs/ton of cement will allow heap stacking to the target and authorized heap height of 200 feet. Optimizing the quantities of cement addition needed for agglomeration pretreatment of the various material types, through continued testing could lead to improved permeability characteristics and decreased cement additions.

A trade-off study was conducted and indicated favorable economics for a case of purchasing and operating an on-site assay laboratory (vs. the current case of contract services). Purchasing an assay laboratory presents an opportunity to lower life of mine operating costs. This study should be updated to the present costs of building and operating a new lab to determine the present-day savings.

1.15 Conclusions and Recommendations

The authors have reviewed the project data and have visited the project site. It is the authors' opinion that the data presented by Pershing Gold are an accurate and reasonable representation of the Relief Canyon project and adequately support the mineral resources, reserves and Feasibility Study of the mine as reported herein. The mineral resource estimate is based on drilling data through September 2016.

Based on the positive results of this Feasibility Study, the project should be advanced to a production decision.

Since acquiring the mine, Pershing Gold has performed extensive metallurgical tests on the deposit. Column leach tests and variability bottle roll tests show an overall expected gold recovery of approximately 81 percent for the project. Additional study and development of the following concepts will help reduce operating costs and ensure gold recovery:

- Development of a blending strategy for all zones in order to minimize fines;
- Optimization of cement additions;
- Development of a heap leach loading strategy to ensure low permeability and/or slow leaching ores are not placed in areas where they would be covered and compacted.
- Develop a sample program and an analysis procedure for determining fines content of agglomerate prior to placement on the heap leach.

2.0 INTRODUCTION

Mine Development Associates ("MDA") has prepared this Technical Report and Feasibility Study on the Relief Canyon project, located in Pershing County, Nevada, at the request of Pershing Gold Corporation ("Pershing Gold"). Pershing Gold is a Nevada corporation listed on the Toronto Stock Exchange (PGLC) and the NASDAQ Global Market (PGLC). The Relief Canyon project is owned by Gold Acquisition Corp. ("GAC"), a wholly owned subsidiary of Pershing Gold. Throughout this report, with the exception of Section 4.0, "Pershing Gold" will refer to both Gold Acquisition Corp. and Pershing Gold Corporation.

The purpose of this report is to provide a technical summary of the Relief Canyon project in support of a Feasibility Study and estimate of mineral reserves. MDA prepared resource estimates for the Relief Canyon gold project in 2014 (Tietz and McPartland, 2014) and 2015 (Tietz and McPartland, 2015). In 2016 a Preliminary Economic Assessment ("PEA") was completed by MDA (Tietz et al., 2016), and in 2017 MDA completed a Pre-Feasibility Study for Relief Canyon as reported by Tietz et al. (2017). The resource estimate reported herein is the same as reported in the 2016 PEA and in the 2017 Pre-Feasibility for the project. Much of this report is modified from Tietz et al. (2017).

MDA has prepared this report and the estimates provided herein in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

The effective date of this report is May 24, 2018. The effective date of the mineral resource estimate is November 1, 2016.

2.1 Project Scope and Terms of Reference

The Mineral Resources presented in this report were estimated and classified under the supervision of Paul Tietz, C.P.G. and Senior Geologist for MDA. Mineral Reserves were estimated and classified by Neil B. Prenn, P.E. and Principal Engineer for MDA. Mining Methods (Section 16.0), Capital and Operating Costs (Section 21), and the Economic Analysis (Section 22.0) for the Feasibility Study were prepared by Mr. Prenn.

Section 13.0 on Mineral Processing and Metallurgical Testing was prepared by Mr. Mark Jorgensen of Jorgensen Engineering and Technical Services in Denver, Colorado, who is a Qualified Professional Member of the Mining and Metallurgical Society of America with special expertise in Metallurgy and Processing.

Section 17.0 on Recovery Methods and Section 18 (excluding sections 18.3 and 18.11 through 18.13 which were prepared by MDA) on Project Infrastructure were prepared under the supervision of Mr. Carl E. Defilippi, RM SME, Senior Engineer for Kappes, Cassiday and Associates ("KCA") in Reno, Nevada. The processing and general and administrative portions of Section 21.0 (21.1.2 through 21.1.23, 21.2.2 and 21.2.3) were also prepared under the supervision of Mr. Defilippi.

There is no affiliation between Mr. Tietz, Mr. Prenn, Mr. Jorgensen or Mr. Defilippi and Pershing Gold except that of an independent consultant/client relationship.

Section 20.0 on Environmental Studies, Permitting, and Social and Community Impact was prepared by Mr. Richard DeLong, President of EM Strategies, Inc. of Reno, Nevada, with special expertise in environmental compliance and permitting of mining projects in Nevada. There is no affiliation between Mr. DeLong and Pershing Gold, except that of an independent contractor/client relationship. Section 20 has benefited from information and the expertise of Debra W. Struhsacker, former Senior Vice President of Pershing Gold including special expertise in External Affairs, Permitting and environmental compliance, and now an independent consultant to the company.

Section 7.0 and Section 8.0 have benefitted from the addition of considerable updated information and new interpretations provided by Douglas Prihar, Manager of Exploration for Pershing Gold, based on the exploration efforts of the geologic staff of Pershing Gold.

The scope of this study included a review of pertinent technical reports and data provided to MDA by Pershing Gold relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy. The authors have made use of the data and information provided by Pershing Gold for the completion of this report, including the supporting data for the estimation of the mineral resources. In compiling the background information for this report, the authors used a 1996 review of the project by Watts, Griffis and McOuat Ltd. (Fernette *et al.*, 1996), a 2007 technical report prepared by John Mears, a 2013 technical report prepared by RPA Inc. (Evans and Altman, 2013), and other references as cited in Section 27.0. MDA prepared a previous technical report on the project for the prior operator in 2010 (Gustin, 2010). Based on the extensive work on the property by previous operators, including mining by two well-known companies, the authors presume that there is a considerable body of information that has been developed over the years, although only those references cited in Section 27.0 were available for review.

Mr. Tietz conducted site visits on October 17 and 18, 2013, January 15, 2015, September 30, 2015, and October 13, 2016. Mr. Prenn visited the site on September 25, 2015 and on October 5, 2016. The mine geology was reviewed, which included: a) a field tour of the deposit area; b) visual inspection of core holes; and c) a thorough review of the geologic cross-sections prepared by Pershing Gold. Drill-site and mineralization verification procedures were conducted, and core drilling and sampling procedures were appraised.

Mr. Defilippi visited the mine site on October 5, 2016. During that visit, meetings were held with site personnel to discuss project status. Mr. Defilippi then toured of the mine, crushing and stacking systems, heap leach facilities and recovery plant. During the tour, initial inspections of existing equipment were made to evaluate their condition and suitability for future operations. Mr. Jorgensen visited the site on March 30, 2017.

The authors have made the independent investigations deemed necessary in the professional judgment of the authors to be able to reasonably present the conclusions discussed herein. The authors believe that the data provided by Pershing Gold are generally an accurate and reasonable representation of the Relief Canyon project.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in Imperial units. Where information was originally reported in metric units, MDA has made conversions according to the formulas shown below; discrepancies may result in slight variations from the original data in some cases.

The term "mine" generally refers to the immediate area of the existing and planned pits, whereas the term "project" includes the associated facilities and other infrastructure within the Relief Canyon property.

Linear Measure

1 centimeter	= 0.3937 inch	
1 meter	= 3.2808 feet	= 1.0936 yard
1 kilometer	= 0.6214 mile	

Area Measure

1 hectare	= 2.471 acres	= 0.0039 square mile

Capacity Measure (liquid)

1 liter	= 0.2642 US gallons

Weight

1 tonne	= 1.1023 short tons	= 2,205 pounds
1 kilogram	= 2.205 pounds	

Currency Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States ("USD").

Acronyms and abbreviations that appear in report:

AA	atomic absorption spectrometry
ADR	adsorption-desorption recovery
Ag	silver
AOI	area of interest
APA	asset purchase agreement
Au	gold
BLM	United States Department of the Interior, Bureau of Land Management
Clay	A material in this report is classified as “clay” if the particle diameter is <0.002 mm.
core	diamond core-drilling method
cuft/ton	cubic feet per ton
GAC	Gold Acquisition Corp.
g Au/t	grams of gold per metric tonne
gpm	gallons per minute
ICP	inductively coupled plasma analytical method
kVA	kilovolt-ampere
lb(s)	pound/pounds
load/perm	load/permeability samples – metallurgical testing
LOM	life of mine
Lph/m^2	liters per hour per square meter
M	mesh
MDM	Mount Diablo meridian
NDEP/BMRR	Nevada Division of Environmental Protection/Bureau of Mining Regulation and Reclamation
MSHA	United States Mine Safety and Health Administration
NNR	New Nevada Resources, LLC
NNL	New Nevada Lands, LLC
NSR	net smelter return royalty
oz Au/ton	ounces of gold per short ton
P_{80}	particle-size distribution of 80% ≤ the nominal dimension
PEA	preliminary economic assessment
QA/QC	quality assurance and quality control
ROM	run of mine
RQD	rock-quality designation
SPLC	Southern Pacific Land Company
t or ton	Imperial short ton (2,000 pounds)

3.0 RELIANCE ON OTHER EXPERTS

The authors are not experts in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements. The authors did not conduct any investigations of the environmental or social-economic issues associated with the Relief Canyon project, and the authors are not experts with respect to these issues.

The authors have fully relied on Pershing Gold to provide all information concerning the legal status of Pershing Gold Corporation, as well as current legal title, material terms of all agreements, existence of applicable royalty obligations, and material environmental and permitting information that pertain to the Relief Canyon property. Sections 4.2 and 4.3 in their entirety are based on information provided by Pershing Gold, including 2013 title opinions (Thompson, 2013a, 2013b, 2013c, and 2013d), as updated in 2014 and 2015, and 2017 by Faillers (2014a, 2014b, 2015a, 2015b, 2017a, 2017b), and the 2013 technical report by RPA Inc. (Evans and Altman, 2013). Sections 4.4 and 20 on Environmental Studies, Permitting and Social or Community Impact was prepared by Mr. Richard DeLong of EM Strategies, Inc.

4.0 PROPERTY DESCRIPTION AND LOCATION

The authors are not experts in land, legal, environmental, and permitting matters. The information presented in this Section 4 is based entirely on information provided to MDA by Pershing Gold, including 2013 title opinions (Thompson, 2013a, 2013b, 2013c, and 2013d), as updated in 2014, 2015, and 2017 by Faillers (2014a, 2014b, 2015a, 2015b, 2017a, 2017b), and the 2013 technical report by RPA Inc. (Evans and Altman, 2013). The authors present the information in this Section in the interest of full disclosure but express no opinion regarding the legal or environmental status of the Relief Canyon property or any of the agreements and encumbrances related to the property.

4.1 Location

The Relief Canyon property is located at the southwestern flank of the Humboldt Range in northwestern Nevada, about 16 miles in a direct line east-northeast of Lovelock, in Pershing County, and about 100 miles northeast of Reno, Nevada (Figure 4.1). The center of the Relief Canyon property is located at approximately 40° 12' 15" North latitude and 118° 10' 13" West longitude. The property area is within the Lovelock 1:250,000 and Buffalo Mountain 1:50,000 scale USGS topographic maps.

4.2 Land Area

As a result of an Asset Purchase Agreement ("APA") dated January 13, 2015, by and between Pershing Gold and its wholly owned subsidiary Gold Acquisition Corp. ("GAC") as buyer, and Newmont USA Limited ("Newmont"), and the actions taken to effectuate the terms of the APA, the Relief Canyon property currently consists of approximately 12,100 acres and includes a total of 391 unpatented lode mining claims, 120 unpatented millsite claims, and approximately 4,373 acres of fee land (Figure 4.2). The parcels that comprise the property are owned by Pershing Gold or their wholly owned subsidiary GAC, or are leased by GAC from New Nevada Resources, LLC ("NNR") and New Nevada Lands, LLC ("NNL"), or are leased or subleased by Pershing Gold from Newmont. The Relief Canyon property is comprised of:

1. GAC-owned Lode and Millsite Claims and Leased Fee Land that are *not* subject to the 2006 Minerals Lease and Sublease Agreement between Victoria Resources (US) Inc. ("Victoria Resources") and Newmont ("2006 Lease Agreement"), which is described in Section 4.3.1:
   - 254 unpatented lode mining claims are owned by GAC and cover approximately 4,693 acres in Sections 3, 4, 9, 10, 14, 15, 16, 18, 20, and 30, Township 27 North, Range 34 East MDM and Section 12, Township 27 North, Range 33 East MDM;
   - 120 unpatented millsite claims owned by GAC covering approximately 592 acres in Section 18, Township 27 North, Range 34 East, MDM; and
   - 1,594 acres of the fee land in Sections 17, 19, and 21, T27N R34E, MDM, that GAC leases directly from NNR and NNL pursuant to a Mining Lease (NNR # 500135) dated January 6, 2015.

Figure 4.1 Location of the Relief Canyon Mine

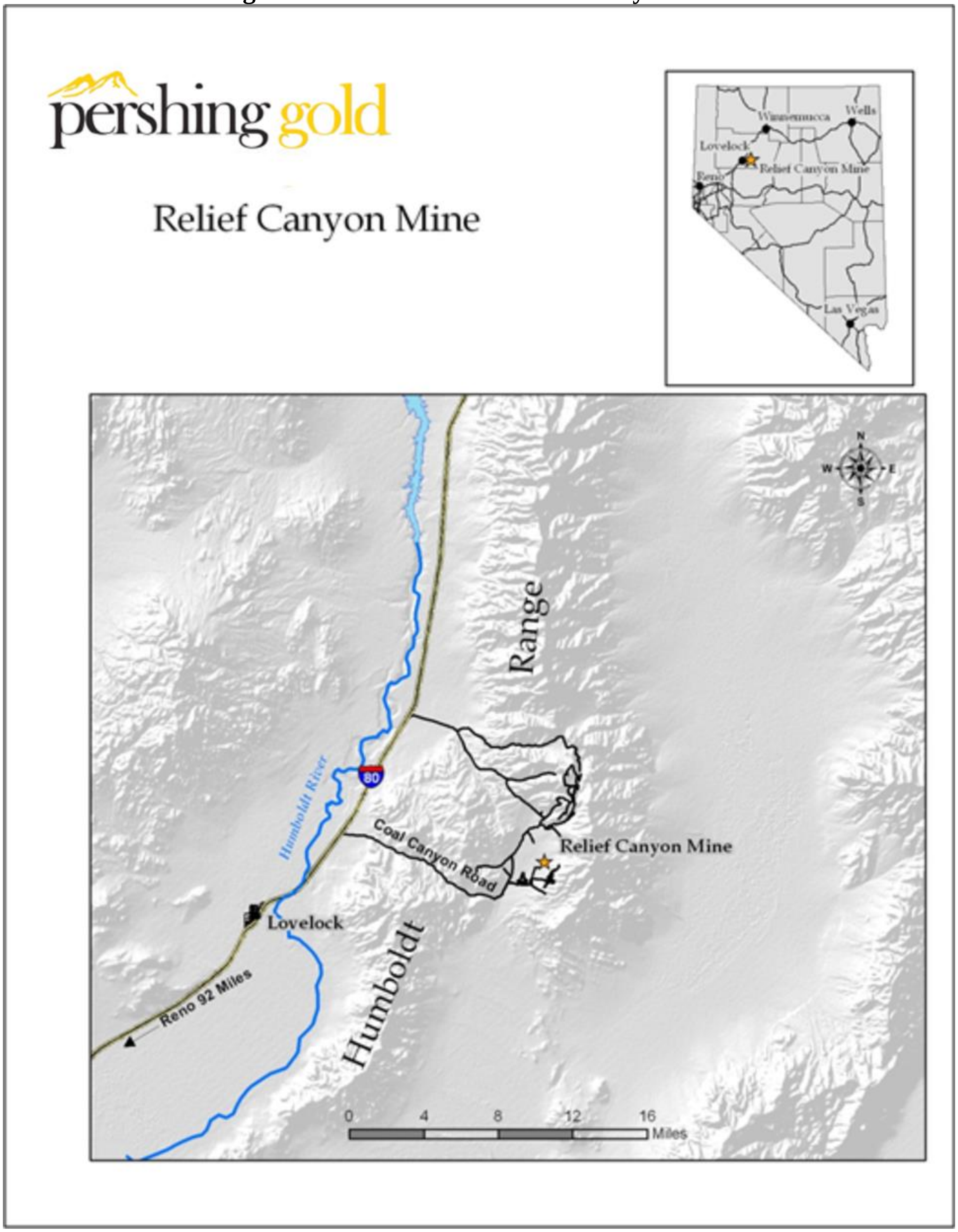
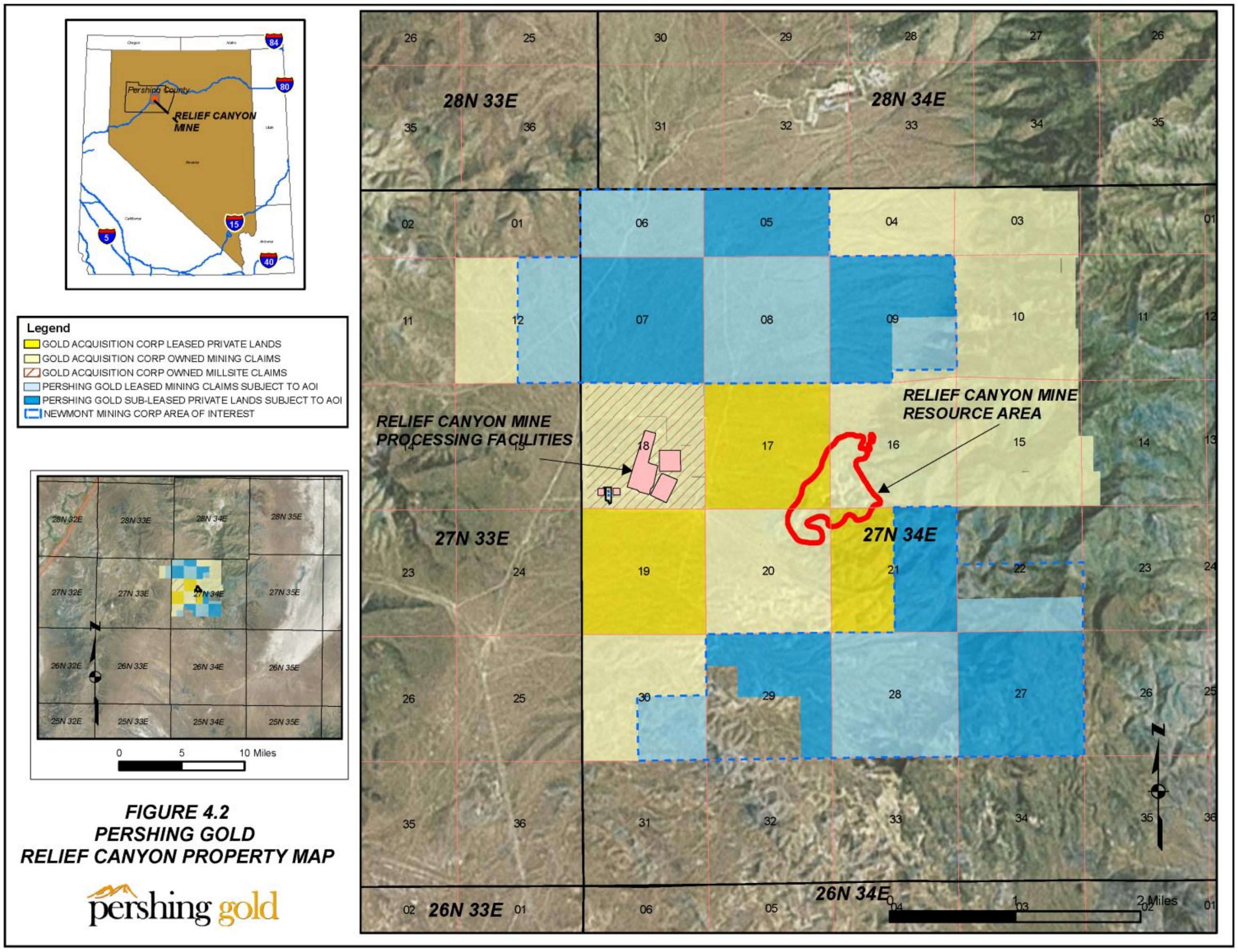


Figure 4.2 Relief Canyon Property Map
(from Pershing Gold, 2016)



2. Pershing Gold-owned or controlled Lode Claims and Subleased Fee Lands that *are* subject to the 2006 Lease Agreement between Victoria Resources and Newmont:
   - 137 unpatented lode mining claims (56 claims owned by Pershing Gold and 81 claims leased from Newmont) covering approximately 2,442 acres in Sections 6, 8, 9, 22, 28, and 30 Township 27 North, Range 34 East, MDM; and Section 12, Township 27 North, Range 33 East, MDM; and
   - One Minerals Lease and one Mining Lease covering approximately 2,779 acres of fee land in Sections 5, 7, 9, 21, 27, and 29, Township 27 North, Range 34 East, MDM.

The mineral resources and reserves discussed in this report are all within GAC owned mining claims or GAC leased fee lands. Table 4.1 lists the 254 lode mining claims and 120 millsites owned by GAC, and describes the 1,594 acres of fee land leased directly by GAC from NNR (not subject to the 2006 Lease Agreement). Table 4.2 provides additional information about the fee land subleased from Newmont. Figure 4.3 shows the portion of the property discussed in detail in this report.

Table 4.1 Unpatented Mining Claims and Leased Land of the Relief Canyon Property
(***not*** Subject to the 2006 Lease Agreement)

Claims	BLM Serial Nos.	County Recording	Count	Royalty
PF 63-109 lode claims PF 125-147 lode claims PF 152-155 lode claims	802860 - 802906 802922 - 802944 802949 - 802952	222734 - 222780 222796 - 222818 222823 - 222826	74	2.0% NSR – Newmont[1]
R 1 - 4 lode claims	902710 - 902713	243962 - 243965	4	2.0% NSR – RCVI[2]
RCL 60-63 lode claims	902722 - 902725	243974 - 243977	4	2.0% NSR – RCVI[2]
RC 1-57 millsite claims	902731 - 902787	243985 - 244041	57	RCVI[2] n/a - millsites
NGR 1-5 lode claims	929649 - 929653	249316 - 249320	5	2.0% NSR – RCVI[2]
RM 1-63 millsite claims	929654 - 929716	249321 - 249383	63	RCVI[2] n/a - millsites
NRC 1-5, 7-17 lode claims	947420 - 947435	354339 - 354354	16	2.0% NSR – Newmont[1]
Bobcat 1-30 lode claims	969360 - 969389 amended	357884 – 357913 492051 – 492080	30	2.0% NSR – RCVI[2]
RCD 5 lode claim	1036656 amended	470991 482546	1	2.0% NSR – RCVI[2]
DROR 181-206 lode claims	1073954 - 1073979	480004 - 480029	26	none
NGAC 1-87 lode claims	1078563 - 1078649	481355 - 481442	87	none
RCLR 46 – 50 lode claims	1082910 - 1082914	482538 - 482542	5	2.0% NSR – RCVI[2]
RR 6 lode claim	1082915	482543	1	2.0% NSR – RCVI[2]
RCDR 1 lode claim	1082916	482544	1	2.0% NSR – RCVI[2]
Total Unpatented Lode/Millsite Claims Owned By GAC			374	

Mining Lease NNR 500135 dated January 6, 2015, by and between NNR and NNL, collectively as Owners, and GAC, as Lessee, concerning the following property:

T. 27 N., R. 34 E., MDM, Pershing County, Nevada (1,593.60 acres):

Section 17: All
Section 19: Lots 1-4, E1/2W1/2, E1/2 (All)
Section 21: W1/2

Primary term of 20 years from the effective date of lease and continuing as long thereafter as any mining, development (which includes exploration or development drilling), or processing operations are being conducted on the leased premises. Annual Advance Royalty ("AMR") payments of one dollar per acre (escalates on 5th anniversary) are required to maintain the lease. AMR payments are current to the 2nd anniversary date January 6, 2018.

Royalty: 2.5% NSR payable to NNR pursuant to Mining Lease 500135, and

2.0% NSR – Newmont[1]

1 – 2.0% NSR payable to Newmont USA Limited pursuant to Royalty Deed dated January 15, 2015.

2 – 2.0% NSR payable to Royal Crescent Valley Inc. pursuant to Amended & Restated Net Smelter Return Royalty Agreement dated August 24, 2011, as amended February 13, 2013.

Table 4.2 Unpatented Mining Claims and Fee Land Subleased by Pershing Gold from Newmont

Claims	BLM Serial Nos.	County Recording	Count	Royalty
PF 1- 62 lode claims, leased PF 110 - 124 lode claims, leased PF 148 - 151 lode claims, leased Note PF 1 – 18 amended	802798 - 802859 802907 - 802921 802945 - 802948 -	222672 - 222733 222781 - 222795 222819 – 222822 492032 - 492049	81	Newmont Option[3]
GVA 1-36 lode claims, owned	939692 - 939727	352199 - 352234	36	Newmont Option[3]
TASK 63 - 72 lode claims, owned	940156 - 940165	351937 - 351946	10	Newmont Option[3]
NPGC 1-10 lode claims, owned	1078553 - 1078562	481344 - 481353	10	Newmont Option[3]
Total Claims Owned or Controlled by Pershing Gold Subject to 2006 Lease Agreement			137	

2,778 acres of Fee Land Subleased by Pershing Gold from Newmont Subject to the 2006 Lease Agreement

Lease A (320 acres): Minerals Lease NNR # 182092 dated August 17, 1987, by and between NNR and NNL, as successors-in-interest to Southern Pacific Land Company, and Newmont, successor-in-interest to Santa Fe Pacific Minerals, and subleased by Pershing Gold pursuant to the 2006 Lease Agreement, concerning the following property:

T. 27 N., R. 34 E., MDM, Pershing County, Nevada (320 acres):

Section 21: E2

Primary term of 25 years from effective date of lease and as long thereafter as the lessee exercises its rights.

Royalty: Subject to Newmont Option[3] - no underlying royalty – no annual advance royalty payment.

Lease B (2,458.88 acres): Mining Lease NNR 500136 dated December 31, 2014, by and between NNR and NNL, collectively as Owners, and Newmont, and subleased by Pershing Gold from Newmont pursuant to the 2006 Lease Agreement, concerning the following property:

T. 27 N., R. 34 E., MDM, Pershing County, Nevada (2,458.88 acres):

Section 5: Lots 1-8 (All)
Section 7: Lots 1-4, E1/2W1/2, E1/2 (All)
Section 9: N1/2, SW1/4
Section 27: All
Section 29: NE1/4, N1/2NW1/4, SE1/4NW1/4, E1/2SE1/4

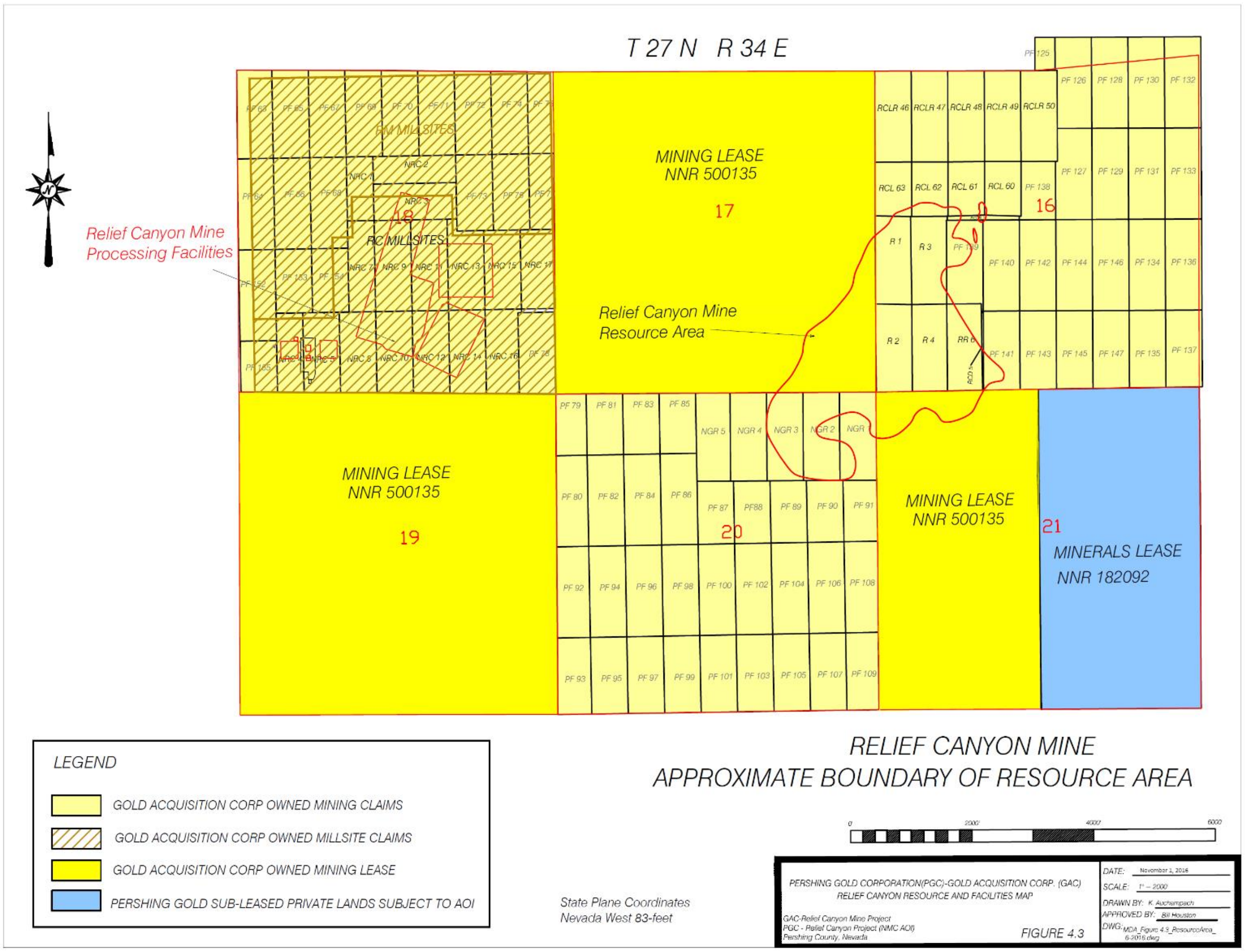
Primary term of 20 years from the effective date of lease and continuing as long thereafter as any mining, development (which includes exploration or development drilling), or processing operations are being conducted on the leased premises. Annual Advance Royalty ("AMR") payments of one dollar per acre (escalates on 5th anniversary) are required to maintain the lease. Newmont makes AMR payments which are reimbursed by Pershing Gold on invoice. AMR payments are current to the 1st anniversary date December 31, 2015.

Royalty: 2.5% NSR payable to NNR pursuant to Mining Lease 500136, and subject to Newmont Option[3]

[3] *The 2006 Lease Agreement provides Newmont the option ("Newmont Option") under certain circumstances to enter into a joint venture with Pershing Gold or to convey the property that is subject to the 2006 Lease Agreement to Pershing Gold and receive a sliding scale three to five percent NSR royalty, and a right to a $1.5 million dollar production bonus payment upon conveyance. With regard to the subleased fee land, there is an offset provision in the event of underlying royalties such that Newmont's three to five percent NSR will be reduced by the underlying royalty, provided that Newmont's royalty shall not be less than two percent.*

A $155 annual maintenance fee must be paid to the U.S. Bureau of Land Management ("BLM") for each claim on or before September 1st, to keep the claims in good standing for the following year. Additionally, an Affidavit and Notice of Intent to Hold Claims accompanied by a $12.00 per claim filing fee must be paid to Pershing County by November 1st of each year. Faillers (2017a, 2017b) reported that federal claim maintenance fee payments and proper filings with Pershing County have been timely made to maintain all of the claims and millsites (511 total) as listed in Table 4.1 and Table 4.2 current through the assessment year ending on August 31, 2017. The holding costs that were paid in 2016 for the 2016-2017 assessment year for the 511 claims listed in Table 4.1 and Table 4.2 were $79,205 to the BLM and $6,132.00 to Pershing County.

Figure 4.3 Detail of Land Holdings in the Vicinity of the Relief Canyon Resource Area



4.3 Agreements and Encumbrances

4.3.1 2006 Minerals Lease and Sublease ("2006 Lease Agreement") and Area of Interest

In April 2012, Pershing Gold executed an Asset Purchase Agreement and Assignment and Assumption of Mineral Lease and Sublease with Victoria Resources by which Pershing Gold acquired Victoria Resources' interests in a 2006 Minerals Lease and Sublease agreement with Newmont (the "2006 Lease Agreement"). On April 5, 2012, Victoria assigned its interests in the 2006 Lease Agreement to Pershing Gold. This acquisition gave Pershing Gold control of the lands surrounding the Relief Canyon mine that had previously been acquired by GAC and eliminated the land ownership constraints associated with the private land sections adjacent to the mining claims on which a portion of the mine is located.

Pursuant to the Asset Purchase Agreement ("APA") dated January 13, 2015, by and between Pershing Gold and its wholly owned subsidiary GAC as buyer, and Newmont, and the actions taken to effectuate the terms of the APA, the 2006 Lease Agreement was amended by the Third Amendment dated January 15, 2015 whereby:

- the 2006 Lease Agreement area of interest ("AOI") was significantly reduced and is now described as shown in Figure 4.2 to encompass

  <u>Township 27 North, Range 33 East, MDM,</u> Section 12 (E1/2),

  <u>Township 27 North, Range 34 East, MDM,</u> Sections 5 – 8; Section 9 (All - excepting area encompassed by PF 125, 126, 128, 130, and 132 lode claims); Section 21 (E1/2); Section 22 (S1/2); Sections 27 – 29; and Section 30 (area of Task 63 – 72, only - SE 1/4);

- 1,593.60 acres of fee land previously subleased from Newmont, were released from the 2006 Lease Agreement, the prior underlying leases terminated as to those lands, and converted to a new Mining Lease (NNR # 500135) dated effective January 6, 2015, by and between GAC and NNR and NNL, as described in Table 4.1. There is 2.5 percent NSR royalty payable to NNR pursuant to the Lease. These 1,594 acres are no longer subject to the 2006 Lease Agreement. Newmont was granted a 2 percent Net Smelter Return ("NSR") royalty on these lands pursuant to Royalty Deed dated January 15, 2015;

- 74 of the Newmont PF unpatented lode mining claims (PF 63 - 109, 125 - 147, and PF 152 - 155), as listed in Table 4.1, were released from the 2006 Lease Agreement and conveyed to GAC, and are no longer subject to the 2006 Lease Agreement. Newmont was granted a 2 percent NSR royalty on these claims pursuant to Royalty Deed dated January 15, 2015;

- 16 Pershing Gold-owned NRC claims (NRC 1 - 5 and 7 - 17), as listed in Table 4.1 that were within the prior AOI were conveyed from Pershing Gold to GAC, and are no longer subject to the 2006 Lease Agreement. Newmont was granted a 2 percent NSR royalty on these claims pursuant to Royalty Deed dated January 15, 2015;

- Mining Lease NNR # 189056 covering 494.66 acres of fee land in Section 33, T28N R34E, MDM was completely released from the 2006 Lease Agreement without any further force or effect;

- 320 acres of fee land (E1/2 Section 21) leased by Newmont under Minerals Lease NNR 182092, dated August 17, 1987, by and between NNR and NNL, and Newmont (as successors in interest), as described in Table 4.2 Lease A, remain subleased to Pershing Gold under the 2006 Lease

Agreement, and subject to the Newmont Option (JV or sliding scale NSR royalty offset by underlying royalty). There is no underlying royalty on this Minerals Lease;

- The prior Minerals Lease NNR # 182092 and Mining Lease NNR # 188095, covering 2,458.88 acres of fee land subleased from Newmont, were terminated as to those lands, and were converted to a new Mining Lease (NNR # 500136) dated effective December 31, 2014, by and between Newmont and NNR and NNL, as described in Table 4.2 Lease B. These 2,458.88 acres, now leased under NNR # 500136, remain subleased to Pershing Gold under the 2006 Lease Agreement, and subject to the Newmont Option (JV or sliding scale NSR royalty offset by underlying royalty), and an underlying 2.5 percent NSR royalty payable to NNR pursuant to the Lease;
- The 137 unpatented lode claims described in Table 4.2 (81 PF leased from Newmont, and 56 Pershing Gold-owned GVA, TASK, and NPGC claims included in the AOI) remain subject to the 2006 Lease Agreement and the Newmont Option.

Pershing Gold's 137 owned and leased claims, and its sub leasehold interest in the 2,788.88 acres of fee land, all as described in Table 4.2 are inside the AOI boundary defined in the 2006 Lease Agreement, as amended by the Third Amendment, and are AOI Lands on which Newmont has the option (Newmont Option) at any time until Pershing Gold delivers a positive feasibility study, and for a period of 90 days thereafter, to become 51-percent manager of a joint venture with Pershing Gold, upon payment to Pershing Gold of 250 percent of the expenditures made on those lands since March 2006. If Newmont elects not to enter into a joint venture, Newmont would convey the leased and subleased properties to Pershing Gold, reserve a sliding scale, 3 percent to 5 percent NSR royalty on the claims and the subleased properties (offset by underlying royalties) inside the AOI boundary, and have the right to a $1.5 million production bonus payment. The Newmont Option is limited to the AOI Lands. GAC's 374 mining claims and millsites, and the 1,594 acres held by GAC under NNR Lease # 500135 at the Relief Canyon property are not AOI Lands and are not subject to the 2006 Lease Agreement or to the Newmont Option.

The 2006 Lease Agreement provided for a $3.6 million work commitment scheduled to be spent over seven years and to be completed by June 15, 2013. Thereafter, starting with the eighth anniversary, $500,000 must be spent each year, or payment must be made to Newmont of an annual rental fee of $10/acre if the work commitment for the preceding year was not met. One-half of the work commitment has to be for direct exploration drilling and associated expenses, including but not limited to drill-road construction, sample assays, down-hole surveys, and reclamation. The Third Amendment provides for an additional work commitment starting on January 15, 2015 of $2.6 million, and scheduled to be spent over seven years, to be completed by January 15, 2022. Pershing Gold has met all work commitments under the original work commitment schedule through December 15, 2016. As of December 2016, Pershing Gold has spent in excess of $2.6 million, which satisfies the work commitment requirement under the new additional work commitment schedule in the Third Amendment. These expenditures keep the 2006 Lease Agreement, as amended, in good standing through the work requirement period due by January 15, 2023, with about $387,123 available towards satisfying the $500,000 expenditure requirement due on January 15, 2024.

Mining Lease NNR #500136 shown in Table 4.2, Lease B, requires Newmont to make annual advance royalty ("AMR") payments of one dollar per acre (escalates on fifth anniversary) to NNR. Newmont makes AMR payments, which are reimbursed to Newmont by Pershing Gold on invoice. Pershing Gold will have to reimburse Newmont approximately $2,500 to cover the 2016 advance royalty payment due by December 31, 2016. AMR payments are current to the next payment due by the second anniversary date, December 31, 2018.

Mining Lease NNR #500135 shown in Table 4.1 requires GAC to make annual AMR payments of one dollar per acre (escalates on fifth anniversary) to NNR. GAC has made the $1,594 AMR payment that was due January 6, 2017. AMR payments are current to the next payment due by the second anniversary date, January 6, 2019.

4.3.2 Royalties

There is a 2.0 percent NSR royalty payable to Royal Crescent Valley, Inc. on the GAC owned R, RCL, Bobcat, RR, RCD, RCDR, RCLR, and NGR lode claims described in Table 4.1 pursuant to the Amended & Restated Net Smelter Return Royalty Agreement dated August 24, 2011, as amended February 13, 2013.

There is a 2.0 percent NSR royalty payable to Newmont on the 74 PF and 16 NRC lode claims owned by GAC described in Table 4.1 pursuant to the Royalty Deed dated January 15, 2015.

There is a 2.5 percent NSR royalty payable to NNR on the 1,593.60 acres of fee land covered by Mining Lease NNR # 500135, described in Table 4.1, dated January 6, 2015, by and between NNR and NNL, collectively as Owners, and GAC, as Lessee.

There is a 2.0 percent NSR royalty payable to Newmont on the 1,593.60 acres of fee land covered by Mining Lease NNR # 500135, described in Table 4.1, pursuant to Royalty Deed dated January 15, 2015.

There is a 2.5 percent NSR royalty payable to NNR on the 2,458.88 acres of fee land covered by Mining Lease NNR # 500136, described in Table 4.2, dated December 31, 2014, by and between NNR and NNL and Newmont, and subleased by Pershing Gold from Newmont pursuant to the 2006 Lease Agreement. This royalty would be an offset to the Newmont royalty under the Newmont Option of the 2006 Lease Agreement.

Section 4.3.1 describes the sliding scale, 3 percent to 5 percent NSR royalty and joint venture option (Newmont Option) that apply to the unpatented lode claims and private fee lands within the AOI covered by the 2006 Lease Agreement with Newmont, as amended by the Third Amendment. Under the 2006 Lease Agreement with Newmont, there is an offset provision in the event of underlying royalties such that Newmont’s 3 to 5 percent NSR royalty will be reduced by the underlying royalty, provided that Newmont’s royalty shall not be less than 2 percent. The royalty that would be payable on Mining Lease 500136, Table 4.2, Lease B, would be 2.5 percent to NNR and 2.5 percent to Newmont. There is no underlying royalty on Minerals Lease 182092, Table 4.2, Lease A, or on the 137 lode claims described in Table 4.2, so there is no royalty offset applicable to these lands.

4.4 Environmental Permits and Potential Liabilities

The following section was prepared by EM Strategies, an expert on environmental and permitting issues. The information presented in this section is based entirely on information provided to EM Strategies by Pershing Gold. The authors cannot verify that the information provided below constitutes all of the permits required for future work on the property, including the possible development of the mineral reserves and resources discussed in this report.

All of Pershing Gold's project permits and licenses for Phase I of the Relief Canyon project are in good standing with no outstanding notices of deficiency or unresolved compliance issues. Permits issued by the U.S. Bureau of Land Management ("BLM") and the Nevada Division of Environmental Protection/Bureau of Mining Regulation and Reclamation ("NDEP/BMRR") authorize the Phase I pit expansion and deepening to a pit bottom elevation of 5,080 feet AMSL, partially backfilling this pit to eliminate a post-mining pit lake, construction of a new waste rock storage facility on private land in Section 17, heap leaching on the approved leach pads and processing solutions in the existing ADR plant. BLM and NDEP/BMRR have authorized Pershing to relocate the crushing facilities from its millsite claims in Section 18, Township 27 North, Range 34 East, to the adjacent private land on Section 17. Pershing Gold is also authorized to reconfigure the access and haul roads to the new crusher location. Those permit modifications that are necessary to construct and operate Phase II of the Relief Canyon project have yet to be obtained (refer to Section 20 for a detailed discussion).

The Water Pollution Control Permit issued by NDEP/BMRR, required Pershing Gold to provide a hydrologic study to document the elevation of the water table underneath the pit areas prior to resuming mining. Pershing Gold retained Schlumberger Water Services ("SWS") to perform this study, which was provided to NDEP/BMRR and BLM in July 2014. The hydrology study also presented information from a 28-day aquifer test and water elevation measurements from groundwater monitoring wells and piezometers located in the vicinity of the mine and elsewhere in the project area. In December 2015, Pershing Gold provided an updated hydrology study prepared by SWS that presents a groundwater model based on the drawdown and recovery data collected during the 28-day pump test. This model evaluates the rate of groundwater drawdown as a result of pumping the two on-site water wells for processing water for the heap-leach facilities and the projected groundwater elevation recovery rates following mining. Pershing Gold is currently completing with WSP a groundwater characterization report to evaluate further mining below the water table. Table 4.3 provides a list of all of Pershing Gold's permits for the Relief Canyon project.

There are no known environmental issues that could materially impact Pershing Gold's ability to extract the Mineral Resources and Reserves. With the exception of the reclamation obligations associated with the features created by previous site operators and owners, there are no known material environmental liabilities due to the previous mining activities. Pershing Gold has a $12.4 million reclamation bond covering all aspects of the Phase I development for the Relief Canyon project. In compliance with the NDEP/BMRR permits and the BLM Plan of Operations listed in Table 4.3, Pershing Gold personnel collect groundwater monitoring data that are reported on a quarterly basis and other environmental data that are submitted to the agencies as required. Both NDEP and BLM personnel conduct quarterly inspections of the mine site.

During August, 2016, the BLM approved Pershing Gold's modification to the Plan of Operations for Phase I mining to deepen and expand the boundary of the pit and to construct a new waste rock storage area on private land in Section 17, Township 27 North, Range 34 East. The authorized heap-leach pads have adequate capacity (21 million tons of ore) to accommodate ore mined during Phase I. In February 2017, the BLM approved a minor modification to the Plan. Pershing Gold has also obtained a Mercury Operating Permit to Construct Thermal Mercury Emissions Units from the NDEP/Bureau of Air Pollution Control pursuant to the Nevada Mercury Control Program ("NMCP"). The Mercury Operating Permit authorizes adding a gold recovery system (e.g., carbon stripping, electrowinning cells, a carbon regeneration kiln, a carbon soak tank, a doré furnace, and a mercury abatement system that includes a scrubber and a retort) to the existing ADR plant. In February 2017, NDEP issued a modification to Pershing Gold's Class II Air Quality Operating permit for the Relief Canyon project and the Class I Operating Permit to Construct in conjunction with issuance of the Mercury Operating Permit. Also, in February 2017, NDEP/BMRR approved Pershing Gold's Major Modification and Renewal for the Water Pollution Control Permit.

Table 4.3 Federal, State, and Local Permits and Authorizations for the Relief Canyon Project

Permit or Approval	Agency	Comments
Federal Permits and Authorizations		
Plan of Operations NVN-064634	BLM - Winnemucca District Office/Humboldt River Field Office	BLM approved the Phase I plan during August, 2016 and approved a Minor Modification on 2/20/18. Covers Phase I mining and heap leaching, and exploration. Reclamation bond amount is $12,398,386. Plan in force for the life of the project. Can be amended for Phase II mining operations. The June 2018 Plan Modification covers mining farther below the water table, expanded mining and mineral processing activities.
BLM Right-of-Way Grant NVN-083323	BLM - Winnemucca District Office/Humboldt River Field Office	Communications Site ROW for mine site radio repeater site.
EPA ID #NVR 000 083 709	US Environmental Protection Agency & NDEP Bureau of Waste Management	Site currently is a Conditionally Exempt Small Quantity Generator (CESQG).
State Permits and Authorizations		
Reclamation Permit No. 0264	NDEP/ Bureau of Mining Regulation & Reclamation	NDEP issued the Phase I permit on 12/22/16 and approved a Minor Modification on 2/13/18. Covers Phase I mining and heap leaching, and exploration drilling. Reclamation bond amount is $12,398,386. Permit is good for the life of the project. Can be modified to address Phase II mining operations. The SRCE associated with the June 2018 Major Modification to be submitted once Preferred Alternative defined in NEPA process.

Permit or Approval	Agency	Comments
Water Pollution Control Permit NEV2007105	NDEP/ Bureau of Mining Regulation & Reclamation	Effective date 3/1/2018; expires on 9/24/2021. Minor Modification to the Permit approved March 1, 2018. Major Modification to the Permit to be submitted in 4Q 2018 to add the expanded heap leach pads, Operating Pond #3, and expand WRSF 5 and other WRSFs.
Class II Air Quality Class Operating Permit No. AP1041-2441	NDEP/Bureau of Air Pollution Control	NDEP approved the modified Class II permit, which covers relocating and operating the crusher and other emission sources, on 2/23/17. BAPC is currently reviewing a modified air quality operating permit to reflect updated crushing and conveying operations that include all of the Phase II facilities, crushing rates, and configurations. Permit renewal application also currently under review by BAPC.
Class I Air Quality Operating Permit to Construct No. AP1041-3652	NDEP/Bureau of Air Pollution Control	NDEP issued the Class I permit on 2/23/17, which covers the thermal mercury emission units.
Mercury Operating Permit to Construct Thermal Mercury Emission Units Permit # AP1041-3585	NDEP/Bureau of Air Pollution Control Nevada Mercury Air Emissions Control Program	Issued on 6/21/16, with construction start-up date extended to 6/21/19. Construction of thermal mercury emission units must start within 18 months;
Class III Landfill Waiver No. F444	NDEP/Bureau of Waste Management	Valid from 12/22/16 - 1/11/22. This landfill will be closed and a new Class III-waivered landfill will be permitted and built in the new waste rock storage area.
Onsite Sewage Disposal System (OSDS) Permit #GNEVOSDS09-S-0392 (Capacity <5,000 gpd)	NDEP/Bureau of Water Pollution Control	Permit No. GNEVOSDS09-S-0392 Permit administratively continued pending Bureau issuance of new General Permit. Permit to be amended to include, or new permit application submitted Q4 2018 for new septic system for truck shop/warehouse to be constructed near mine pits.
Industrial Artificial Pond Permit S39298	NV Department of Wildlife	Recently renewed. Valid from 11/1/16 – 10/31/21.
COUNTY		
Pershing County Special Use Permit	Pershing County Planning Department and Board of County Commissioners	Good for life of project. Courtesy update provided to the Pershing County Commissioners for Phase I mining and heap leaching.

Permit or Approval	Agency	Comments
Building Permits	Pershing County Planning Department	Will be submitted once building designs are finalized.

Notes: BLM – Bureau of Land Management
EPA – U.S. Environmental Protection Agency
NDEP – Nevada Division of Environmental Protection

The Minor Modification approved on February, 2018 authorized the following: construction of heap leach pad extension Cell 5; realignment, repurposing and relocation of permitted but not yet disturbed surface disturbance; reconfiguration of the open pit boundary to improve stability and safety of the open pit highwall; increase the annual ore production rate from 6 million tons to 7.5 million tons; increase the permitted heap leach solution application rate; and removing low-grade ore material from WRSF 4 and placing that material on the heap leach facility.

5.0 ACCESS; CLIMATE; LOCAL RESOURCES; INFRASTRUCTURE; AND PHYSIOGRAPHY

5.1 Access

Access to the Relief Canyon property is via Interstate 80 northeast of Lovelock. From exit 112 located 7 miles northeast of Lovelock, access is by way of Coal Canyon Road about 10 miles southeast, turning north at Packard Flat onto a gravel road for about two miles to the property. Coal Canyon Road is a paved road maintained by Pershing County.

5.2 Climate

The climate at Relief Canyon is typical of the high desert. Summers are warm with cool nights; winters are cool to cold with occasional moderate snowfall. Precipitation is low and comes primarily in winter, although there are infrequent rains in the summer. The yearly precipitation is about 6 inches in the valleys and up to 20 inches in the mountains in Pershing County (Johnson, 1977). Average annual precipitation at the Relief Canyon mine was 4.85 inches from 2009 through 2013 (Schlumberger Water Services, 2014, citing Gold Acquisition Corp., 2014, WPCP Permit #NEV2007105, 2013 Annual Report). Mining can be conducted year-round on the property.

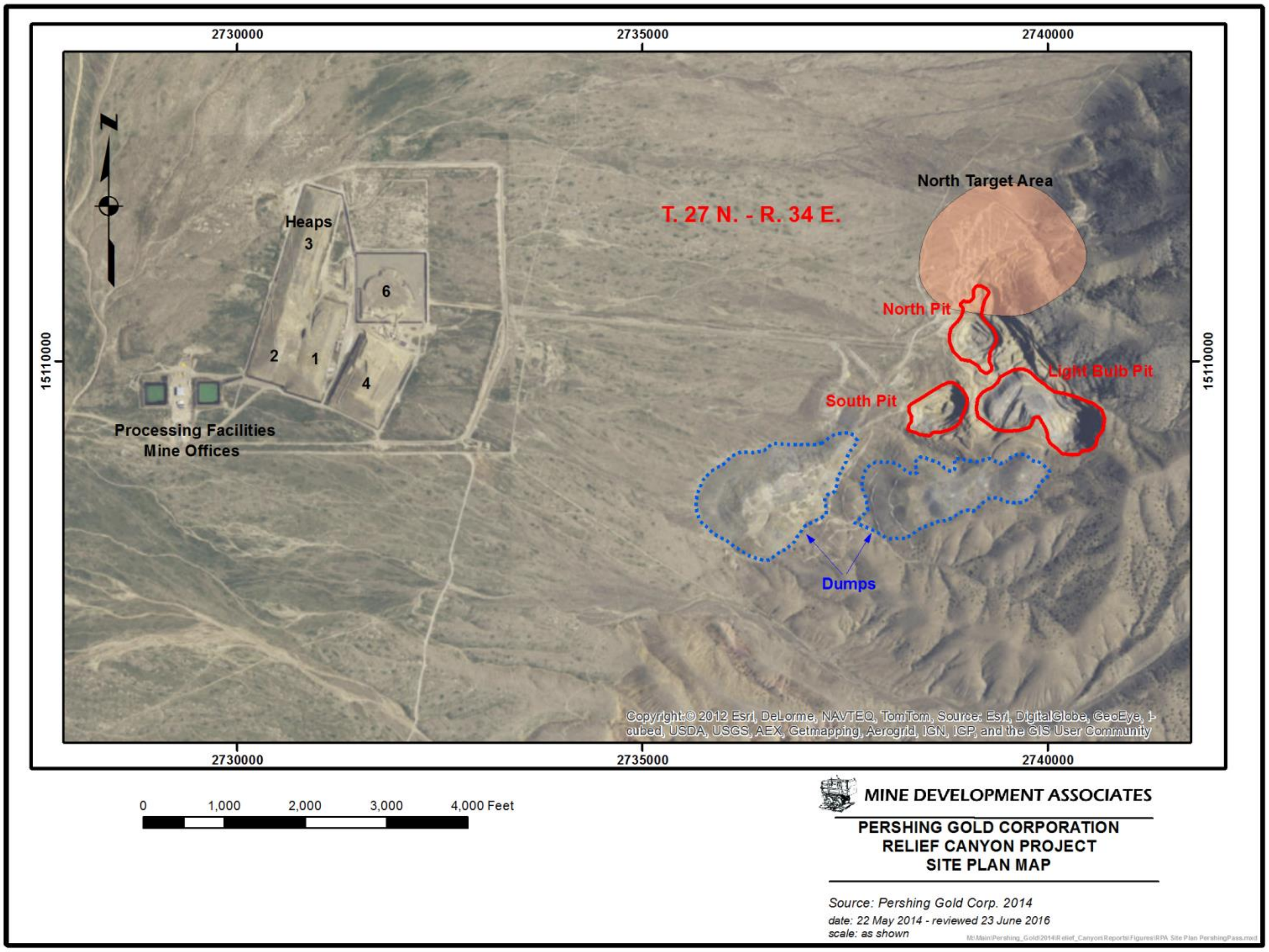
5.3 Local Resources and Infrastructure

The city of Lovelock lies about 19 miles by road west-southwest of the property and had an estimated population of 1,987 in 2013, according to the website of the Nevada State Demographer (http://nvdemography.org/data-and-publications/estimates/). The city of Reno, Nevada, is about 90 miles southwest of Lovelock on Interstate 80, and is part of a metropolitan area with a population of approximately 425,000 as of the 2010 census. Necessary supplies, equipment, and services for exploration and mine development would be available in the Reno area. A trained mining and industrial workforce is available in Lovelock and other communities nearby.

There are currently three open pits (North, South, and Light Bulb pits), several waste-rock dumps, growth media stockpiles, access roads, heap-leach pads, and a carbon-adsorption recovery ("ADR") plant on the property (Figure 5.1). An existing two-stage crushing plant that is also on the property will be sold.

Electricity is available on the property, and water is available from two wells located east of the process plant.

Figure 5.1 Infrastructure at the Relief Canyon Property
(Information from Evans and Altman, 2013)



5.4 Physiography

The Relief Canyon property is located on the southwestern flank of the Humboldt Range, one of the generally north-trending, fault-bounded ranges of the Basin and Range physiographic province. Within the project area, the topography varies from flat to hilly (Figure 5.2). Elevations in the project area vary between 4,600 feet in the valley on the west, to 5,500 feet in the range to the east of the open pits (Mears, 2007).

Vegetation is sparse, consisting of grasses and shrubs of the high desert, with a few trees in the higher elevations of the range.

Schlumberger Water Services ("Schlumberger") conducted a hydrogeological study of the Relief Canyon mine over a 2.5-year period in order to prepare a baseline hydrogeologic characterization report (Schlumberger, 2014). In February 2014, Schlumberger conducted a 28-day pump test of the two groundwater production wells, PW-1 and PW-2, (Schlumberger, 2015). Groundwater flow at the Relief

Canyon mine is controlled by an alluvial aquifer in the Quaternary alluvium and fractured bedrock aquifers in the Grass Valley and Cane Spring formations. As of January 2015, the groundwater elevation was 43 ft below the bottom of the North Pit, 47 ft below the bottom of the South Pit, and 303 ft below the bottom of the Light Bulb Pit. The two water production wells northwest and southwest of the South Pit, PW-1 and PW-2, were completed in the 1980s. The 2014 Schlumberger pump test described in the Schlumberger (2015) report, found that pumping PW-1 and PW-2 for process water would further lower the groundwater elevation in a localized area under the pits. Schlumberger concluded that "*in the event that future pit dewatering requirements exceed the future potential mine consumptive use, excess water would likely be managed through operation of Rapid Infiltration Basins…in the valley-fill alluvium*." Gold Acquisition Corp. owns water rights certificates 13402 and 13403 and water rights permit 76626 with a total annual duty of 618 acre-feet per year. (Schlumberger, 2014). In 2017, the Nevada Division of Water Rights granted GAC Permit No. 83438 for an additional 300 acre-feet per year.

Figure 5.2 Photograph Showing Physiography of Relief Canyon Area
(Lacana/Pegasus Heaps in middle ground)

6.0 HISTORY

The information summarized in this section of the report is derived from multiple sources, as cited. Paul Tietz has reviewed this information and believes this summary accurately represents the history of the Relief Canyon property.

6.1 History of Exploration and Mining

The project history has been compiled from the following references: Johnson (1977), Fiannaca (1982), Fiannaca and McKee (1983), Easdon (1983b), Fiannaca and Easdon (1984), Wittkopp *et al.* (1984), Atiyeh (1986), Parratt *et al.* (1987), Pegasus Gold Inc. (1987, 1988, 1990), Wallace (1989), Cuffney *et al.* (1991), Abbott *et al.* (1991), Wojcik (1996), Fernette *et al.* (1996), Firstgold (2006, 2007d, 2007e, 2008a, 2008c), Mears (2007), and Evans and Altman (2013). In some cases, references differ as to details of the history, but MDA has assembled what it believes to be an accurate description of events.

The Relief-Antelope Springs mining district, in which the property is situated, had historical production of silver, antimony, and mercury, but there is no evidence that it had produced gold prior to development of the Relief Canyon deposit in the 1980s.

Exploration began in the district in the early 1860s with discovery of antimony and silver in the same decade and mercury discovered in 1907. Historical production of silver, antimony, and mercury totaled about $3,000,000. There were a number of fluorite prospects in the immediate vicinity of the Relief Canyon deposit, where mining took place in the 1940s, although none have had reported production (Papke, 1979). The site of the current Relief Canyon deposit was originally known as the Bohannon or Emerald Spar fluorite prospect.

In 1978, the property was staked for high-purity limestone by Falconi Cement Inc. (“Falconi”), which drilled one core hole measuring 745 feet to test the quality of the limestone. That hole passed through mineralized breccia into Natchez Pass Formation limestone. That hole is not included in the current database.

As part of a regional, precious metals prospecting program with an emphasis on the Humboldt Range, Duval Corporation (“Duval”) explored the area in 1979 with mapping and stream-sediment sampling and detected 0.45ppm gold in a single stream sediment sample from the site. Duval then contacted Falconi, logged and assayed their single core hole, and ran a series of soil and rock-chip sample lines over Falconi’s property. Assays of the core showed the presence of gold. Duval negotiated a joint-venture agreement with Falconi in 1979 and staked an additional 2,300 acres. Duval initiated a detailed mapping and sampling program, which identified a gold anomaly that was 2,000 feet by 1,500 feet in area, ranging between 0.01 and 0.06 oz Au/ton in grade. Duval proceeded to drill reverse circulation percussion holes during 1981-1982 that confirmed the presence of a low-grade but potentially mineable zone that was 2,400 feet by 1,800 feet in size. Fiannaca and McKee (1983) and Fiannaca and Easdon (1984) reported that Duval drilled 40 reverse circulation holes, although the database shows 41; Mears (2007) reported that Duval drilled 44 holes. MDA cannot account for these discrepancies.

Lacana Mining Inc. (“Lacana”; Lacana Gold Inc. was the U. S. subsidiary, and this name is also referenced in the literature) first optioned and then purchased the property from Duval in 1982, including the remaining 10 percent interest held by Falconi. At the start of its investigation, Lacana undertook various sampling programs to verify Duval's assays and to understand the metallurgy of the mineralization. These are described in Section 11.10 and Section 13.2 of this report, respectively. In addition, Lacana commissioned aerial photography in order to prepare a topographic map of the main drilling area. Lacana took 48 samples from several trenches to evaluate variations among assays of the drill intervals along a vertical plane adjacent to the holes. The samples were taken on a 5-foot continuous chip basis on vertical lines spaced 10 feet apart horizontally. The end points of each sample interval were horizontally parallel to the end points of each 5-foot drill interval. Lacana's initial efforts also included mapping, sampling, drilling, and bench-scale metallurgical testing.

Lacana conducted detailed geological mapping of the property and then drilled 48 reverse circulation holes in order to provide details on the "inferred geological reserves."

In September 1983, Lacana undertook pilot-scale heap-leach test work, mining and cyanide heap leaching of two 5,000-ton blocks of mineralization. Bo-Ter Construction Company was contracted to mine the deposit and construct the leaching facility, crushing plant, and recovery circuit according to final design engineering by Mine and Mill Engineering. Additional reverse circulation drilling on 25-foot centers was conducted at each of four potential mining sites, and 140 blast holes were drilled on the two selected sites. The leach tests indicated that a net gold recovery approximating 70 percent could be achieved by standard cyanide heap leaching of run-of-mine material with 80 percent for agglomerated material.

Based on encouraging preliminary results midway through the pilot test, Lacana undertook a second drilling phase to sample the main zone of mineralization on 100-foot centers. A total of 99 reverse circulation holes were drilled in this phase. A third phase of drilling was undertaken to better define the pit perimeter and to condemn potential sites for waste dumps; this phase consisted of 57 reverse circulation holes. The 3 phases of exploration drilling reported here total 204 holes, but the database shows 205 holes. MDA cannot account for the discrepancy.

Southern Pacific Land Company ("SPLC"), who was an adjacent property owner, had participated with Lacana in the pilot-scale metallurgical program and according to Fiannaca and Easdon (1984) drilled 147 reverse circulation holes proximal to Lacana's deposit on their own property to test for continuation of mineralization; this drilling apparently took place in 1983 and possibly into 1984. The database includes 146 holes. Southern Pacific Land Company later merged with Santa Fe Industries, subsequently Santa Fe Pacific Corp., whose natural resource interests were spun off as Santa Fe Pacific Gold Corp. Santa Fe Pacific Gold Corp. later merged with Newmont Mining Corp., and Santa Fe Pacific Corp.'s property interests were sold to Nevada Land and Resource Company. For simplicity, SPLC and its later iterations are referred to as "Santa Fe" in this report.

At some point during their tenure on the property, Lacana commissioned an IP survey by Phoenix Geophysics, Inc. Results from a single IP line were provided to MDA.

Cooksley Geophysics Inc. conducted a reflection seismic exploration program in the alluvium of Packard Flat near Relief Canyon for Lacana in 1984 (Cooksley and McMahon, 1984). The program was designed to define stratigraphy and structure covered by the alluvium. Three lines were run – 2 trending northwest and 1 trending east. The alluvium ranged from less than 50 feet to over 300 feet in thickness. All three lines encountered well-stratified sections thought to represent Mesozoic units.

Lacana began mining the open-pit Relief Canyon mine and leaching run-of-mine material in August 1984 but closed it in October 1985 due to poor leach recoveries. Based on metallurgical testing, Lacana had expected gold recovery from crushing and agglomeration to be 75 percent and from run-of-mine material to be about 65 percent. No data is available from Lacana operations, but one source reported run-of-mine material to be about 65 percent, but actual gold recoveries were only 48 percent (Wojcik, 1996). According to Fernette *et al.* (1996, citing the 1986-1988 Canadian Mines Handbook), Lacana produced 13,826 ounces of gold in 1984-1985. However, the Nevada Bureau of Mines and Geology (1987; 2008) reported that 1984 production from Relief Canyon was 24,500 ounces of gold from 1,000,000 tons per year (this may be mined ounces, as opposed to recovered ounces, which is consistent with the material estimated to have been mined in Table 6.4, but would suggest a recovery of 56 percent); they did not report any production for 1985.

In March 1986, Pegasus Gold Corporation ("Pegasus") entered into an option agreement with Lacana to evaluate the property. Work began immediately on an evaluation of the mineable material (estimates described in Section 6.2). The project database includes 17 Pegasus holes totaling 5,100 feet that are stated to have been drilled in 1987 and 1988. Pegasus purchased the property in July 1986 and re-opened the mine in November 1986, using crushing and agglomeration to process the fine-grained ores. They also installed an adsorption-desorption recovery ("ADR") type cement block process plant. Mining ceased in 1989 with four heap-leach pads completed; Pegasus continued leaching operations until August 1990. Figure 6.1 shows one of the Relief Canyon pits as of fall 2008.

Figure 6.1 Photograph of One of the Relief Canyon Pits as of 2008

Table 6.1 shows production by Pegasus from Relief Canyon as reported by Pegasus to the Nevada Division of Minerals (NV Div. Minerals column), Pegasus' annual reports, and Fernette *et al.* (1996, citing Pegasus' 1990 annual report and the Canadian Mines Handbook for 1989-1991). Wojcik (1996)

reported that Pegasus recovered some 117,600 ounces of gold from 5.18 million tons of ore. According to Mears (2007), Pegasus processed about 2 million tons per year averaging 0.03 oz Au/ton. Pegasus reported that its 1989 gold production came from 1,600,000 tons of ore with a cutoff grade of 0.010 and that the gold and silver grades for that year were 0.028 oz Au/ton and 0.100 oz Ag/ton, respectively (Pegasus Gold Inc., 1989).

Various reports suggest that Pegasus achieved recoveries of 65 percent to 70 percent at Relief Canyon. The Pegasus 1989 annual report states that Pegasus' recovery from their heap leaching of crushed and agglomerated ore in 1987 exceeded 65 percent. Fernette *et al.* (1996) state that, "*no data is available from the Pegasus operation so actual gold recoveries are not known. According to the 1990 Pegasus Annual Report, during 1987 and 1988, the mine produced 83,600 ounces of gold from 4 million tons of ore with an average grade of 0.03 ounces of gold per ton. This would indicate that 70% of the gold in the ore was recovered.*"

Table 6.1 Production at Relief Canyon by Pegasus from 1987 through 1990

(Data from Pegasus Gold Inc. annual reports (1987, 1989, 1990) and data provided by the Nevada Commission on Mineral Resources, Division of Minerals (personal communication, 2010) based on figures provided by Pegasus)

Year	Gold (ounces)			Silver (ounces)	Tons Mined[2]
	NV Division of Minerals	Pegasus Annual Reports	Fernette et al. (1,996)	NV Division of Minerals	NV Division of Minerals
1986		1,800			
1987	41,177	41,600	41,600	31,868	4.9 million
1988	40,827	40,000	42,000	42,570	9.5 million
1989	29,906	29,900	29,900	29,828	9.6 million
1990	4,064	4,100	4,100	6,414	Not Reported
Total	**115,974**	**117,400**	**117,600**	**110,680**	**24.0 million**

[1] Table does not include 1984 through 1985 production by Lacana; see text.
[2] Tonnages reported here by the NV Division of Minerals appear to be total of ore and waste.

Based on the information presented herein, a total of about 131,000 ounces of gold and about 111,000 ounces of silver were produced by Lacana and Pegasus at Relief Canyon from 1984 through 1990; Wojcik (1996) and Fernette *et al.* (1996) both reported that production from the Relief Canyon mine in this period totaled about 131,000 ounces of gold.

J. D. Welsh and Associates ("Welsh") of Reno, Nevada, purchased the property from Pegasus in September 1993 and reportedly produced several thousand ounces of gold by continuing to rinse the heaps.

Newgold, Inc., which later changed its name to Firstgold Corp. in 2006 (collectively called "Firstgold" in this report), purchased the property from Welsh in January 1995 for $500,000. This acquisition originally included the unpatented claims, which were purchased from Welsh, and fee land that Welsh was subleasing from Santa Fe, for which Firstgold acquired an Assignment of Minerals Sublease. Nevada Land and Resource Company was the owner of the fee land, which was leased by Santa Fe and subleased by Firstgold (Newgold Inc., 1997).

Firstgold processed pregnant pond solution until July 1995. In addition, they acquired a drill-hole and assay data set with 400 drill holes and over 22,000 assays, and also acquired much of Lacana's metallurgical test data. Through April 1997, Firstgold drilled 73 reverse circulation holes totaling 43,220 feet, including 23 holes drilled on Santa Fe's ground that Firstgold was leasing at the time. The focus of Firstgold's initial drilling was just north of the North pit, west of the pits, and southwest of the pits. A ground magnetic study was conducted in 1999, but no reports or data from the magnetic study are available.

Firstgold placed the property in closure in 1997, and from 1998 until late 2004, the property was maintained on a care and maintenance basis. The Firstgold fee land sublease was dropped at some time during the period of 1997 to 2004 (likely in 1997). Firstgold began preparations to resume exploration and development of the project in 2003 and reactivated the project in 2005. Through early 2007, Firstgold carried out the following activities:

1. Available data were assembled and, where possible, digitized;
2. The land status was researched, and Firstgold staked additional claims;
3. Surface and trench samples were taken on the existing heap-leach piles and sent for analysis, including heap-leach column analysis, by an outside testing lab;
4. Aerial photography was flown to produce a topographic map of the property and surrounding area; and
5. The firm of Dyer and Associates was hired to bring the property into complete government compliance, to design new leach pads for re-processing of the existing heaps, and to expand capacity for processing.

In 2006, Zonge Geosciences, Inc. conducted a ground magnetic survey on the project. The following is Mears' (2007) description of the survey:

> *"From October 20th to October 25th 2006 Zonge Geosciences, Inc. performed a GPS-based ground magnetic survey on the Relief Canyon Gold Project. Ground Magnetic/GPS data were acquired on 29 lines oriented east-west and spaced approximately 90 meters apart, for a total of 40 line kilometers of data acquisition. Total magnetic field data were acquired with GEM Systems GSM-19 Overhausereffect magnetometers. The GSM-19 magnetometer has a resolution of 0.01 nT and an accuracy of 0.2 nT over the operating range. Positioning was made with a Trimble PRO-XRS GPS receiver. The GPS data were differentially corrected in real-time using OMNISTAR corrections. This system provides sub-meter accuracy under standard operating conditions."*

White (2008) reported on the interpretation of the ground magnetic data, noting that the data can be used to map near-surface mafic dikes or sills and possibly deeper, more felsic intrusions.

Firstgold resumed exploration drilling at Relief Canyon in 2007, initially focusing on the potential of the area between the existing pits and then expanding to the north and northwest. A total of 92 reverse circulation holes were completed by May 2008 within and adjacent to the resource area, with an additional 13 reverse circulation holes drilled in the North Target area. Firstgold also completed four core holes in 2008, one within the resource area and three in the North Target area; one additional hole was abandoned before completion.

Following acquisition of the necessary permits, Firstgold invested approximately $30 million in redeveloping and reconstructing the Relief Canyon heap-leach processing facilities in 2007 and 2008. This redevelopment effort consisted of completely refurbishing the plant and building a new leach pad cell designed to meet current regulatory requirements for engineering design and containment. The planned project at that time was to re-crush the leached ore on the old pads to minus one-half inch, agglomerate the crushed ore, and use a conveyor to stack the crushed and agglomerated ore on the newly constructed leach pad.

Firstgold attempted to reprocess the existing heaps in late 2008 and early 2009. It operated the reprocessing project for a few months but experienced very low recoveries which were insufficient to cover operating costs. The project was soon shut down.

In January 2010, Firstgold filed for bankruptcy protection. Platinum Long Term Growth LLC acquired the Relief Canyon assets. Gold Acquisition Corp., a newly formed, wholly owned subsidiary of Sagebrush Gold, Ltd., acquired the Relief Canyon assets on August 30, 2011. On March 5, 2012, Sagebrush Gold, Ltd. changed its name to Pershing Gold Corporation. Exploration by Pershing Gold is described in Section 9.0.

6.2 Historical Mineral Inventory Estimates

All estimates described in this section were prepared prior to 2000 and are presented herein merely as an item of historical interest with respect to the exploration targets at Relief Canyon. The classification terminology are presented as described in the original references, but it is not known if they conform to the meanings ascribed to the measured, indicated, and inferred mineral resource classifications or proven and probable reserve classifications of the CIM Standards. Accordingly, these estimates should not be relied upon. The authors have not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Pershing Gold is not treating these historical estimates as current estimates. These historical mineral resource estimates are superseded by the current mineral resource estimate discussed in Section 14.0 of this report.

Duval's initial drilling of the Relief Canyon deposit in 1981-1982 identified "inferred geological reserves" ranging from six million tons at a grade of 0.06 oz Au/ton and a waste-to-ore stripping ratio of 3.0:1.0, to 10 million tons at a grade of 0.04 oz Au/ton and a 1.5:1.0 stripping ratio (Fiannaca and Easdon, 1984).

Following their bench-scale metallurgical test program in 1983, Lacana recalculated "inferred geological reserves" based on Duval's drilling, categorizing the reserves into three cases. The total "inferred contained gold" was about 460,000 ounces (Fiannaca and Easdon, 1984). Table 6.2 compares Lacana's results to those of Duval.

Table 6.2 Preliminary Calculations of "Inferred Geological Reserves" by Duval and Lacana
(From Fiannaca and Easdon, 1984)

Company	Case	Tonnage (million tons)	Grade (oz Au/ton)	Stripping Ratio t waste/t ore
Duval	1	6		3.0
Duval	2	10		1.5
Lacana	1	2.1		2.1
Lacana	2	5		2.0
		10.4		1.2
Lacana	3	9.8*	0.042*	

* Recalculated following later drilling and heap-leach testing.

Subsequently, after two phases of reverse circulation drilling and pilot heap-leach testing, Lacana recalculated their "geological reserves" (Fiannaca and Easdon, 1984). They found that their third case of "inferred geological reserves" had decreased from 10.4 million tons at a grade of 0.044 oz Au/ton to 9.8 million tons at a grade of 0.042 oz Au/ton (Table 6.2).

As part of their final feasibility study in 1984, Lacana once again calculated the "mining reserves" and also constructed the pit plan by two methods. The first calculation was done entirely by hand. For confirmation, Pincock, Allen, and Holt ("PAH") of Tucson was commissioned to recalculate the "mining reserves" by kriging and to derive the pit and bench plans by the floating cone method. These reserves are shown in Table 6.3 and are based on a recovery factor of 70 percent, bench height of 15 feet, and pit slope of 50° (Fiannaca and Easdon, 1984). Santa Fe had drilled out a smaller volume of deeper mineralization on their own property that could perhaps be mined in the future, but these potential Santa Fe reserves are not included in Table 6.3.

Table 6.3 Historical Comparison of "Mining Reserves" Calculations by Lacana and Pincock, Allen, and Holt

(From Fiannaca and Easdon, 1984)

Company	Au Price	Cut-off	Mineable "Ore"	Diluted Grade	Strip Ratio
	$ US/oz	oz Au/ton	(million tons)	oz Au/ton	t waste/t ore
Lacana	400	0.02	7.25	0.036	2.80
Lacana	400	0.015 *	9.185	0.032	2.00
Lacana	400	0.010 *	11.403	0.028	1.40
PAH	400	0.02	7.793	0.035	1.49
PAH	400	0.010 *	10.972	0.029	0.77
PAH	500	0.02	9.175	0.034	1.87
PAH	500	0.010 *	14.391	0.027	0.94

* Internal to the pit based on the 0.02 cut-off

In 1986 while evaluating the Relief Canyon property under an option agreement with Lacana, Pegasus estimated the "mineable ore reserves" (Atiyeh, 1986). Pegasus used a statistical software package developed by Geostat Systems of Golden, CO that used the kriging method of grade interpolation. Drill-hole spacing was about 100 feet. "Total mineable reserves" were estimated to be 5,042,000 tons at an average grade of 0.030 oz Au/ton with a 0.015 oz Au/ton cutoff and a stripping ratio of 1.18:1. Compared to prior estimates of "reserves" by Lacana, Pegasus' estimate was lower due to elimination of deeper carbonaceous material, some limestone "ore," and a few outlying areas where Pegasus felt there were insufficient data. Pegasus' estimate used costs of $1.11/ton for mining and $3.60/ton for processing, a gold recovery of 70 percent, a density of 14.1 cubic feet/ton, and a gold price of $400/oz (Atiyeh, 1986). Table 6.4 shows "ore reserve" estimates from Lacana and Pegasus for 1985 mine-to-date, 1986 mine plan, and the ultimate pit. Table 6.5 shows the final "mineable ore reserve" estimates made by Pegasus in 1986.

Table 6.4 Comparison of 1985 and 1986 "Ore Reserve" Estimates of Lacana and Pegasus
(Modified from Atiyeh, 1986)

Project Stage	Company	Ore (tons)	Grade (oz Au/ton)	Total Ounces Au	Total Tons	Strip Ratio
1985 mine-to-date	Lacana	739,000	0.036	26,604	3,074,000	3.16
	Pegasus	652,000	0.036	23,472	2,310,000	2.54
1986 mine plan	Lacana	857,000	0.045	38,565	2,040,000	1.38
	Pegasus	1,372,000	0.033	45,276	2,789,000	1.03
Ultimate pit	Lacana	7,570,000	0.032	242,240	26,344,000	2.48
	Pegasus	8,775,000	0.028	245,700	27,804,000	2.17

Table 6.5 Relief Canyon "Total Mineable Reserves" Calculated by Pegasus in 1986
(Atiyeh, 1986)

Cutoff (oz Au/ton)	Ore (tons)	Grade (oz Au/ton)	Waste (tons)	Total Tons	Strip Ratio
Floating Cone Runs at $400/oz Au					
0.015	6,738,000	0.030	3,734,000	10,472,000	0.55
0.02	5,805,000	0.032	4,667,000	10,472,000	0.80
Hand Design Pit					
0.015	5,042,000	0.030	5,923,000	10,965,000	1.18

As of December 1, 1986, "reserves" at Relief Canyon were said to be 5.3 million tons grading 0.03 oz Au/ton (Engineering and Mining Journal, June 1987, cited in Abbott *et al.*, 1991).

Following acquisition of drill-hole data from the prior owner, Firstgold created a block model and estimated the tonnage and grade of the Relief Canyon deposit with Watts, Griffis and McOuat Limited ("WGM") reviewing Firstgold's methodology (Fernette *et al.*, 1996; Wojcik, 1996). A "resource and reserve" summary, believed to reflect this block model, was prepared by Kim Drossulis, Firstgold's mining engineer, and was based on 400 vertical reverse circulation drill holes with 22,188 assay intervals containing gold, silver, and rock codes (Drossulis, undated but presumed to be 1996). WGM reported that Firstgold's grade modeling approach was, in effect, a geologically constrained polygonal model. Using the inverse distance squared interpolation method and assigning a tonnage factor of 15 cubic feet/ton to all blocks in the model, Drossulis estimated a "geological resource" of 23,984,400 tons containing 0.017 oz Au/ton using a 0.004 oz Au/ton cutoff and digitizing the geopolygons in plan with 15-foot benches and section with 85-foot centers. Fernette *et al.* (1996) commented that "*The use of a single density value for all rock types and the lack of coding for alluvium and waste dumps could lead to overestimation of waste tonnage. Alternatively, the density of the Cane Spring limestone is probably greater than 15 cubic ft. per ton which would understate the tonnage of mineralized limestone and limestone waste blocks*." WGM concluded, "*Although the sample density on which the model is based is quite high, the criteria used to develop the grade and rock models, as noted above, are such that the estimated tonnage and grade for the Relief Canyon deposit meets the criteria for an 'Indicated Mineral Resource.*"

WGM used computerized floating cone methodology combined with the grade and rock model developed by Firstgold as described above to define the potential economic pit boundaries and make preliminary estimates of the potentially minable portion of the Relief Canyon "indicated mineral resource" (Fernette *et al.*, 1996). Two floating-cone resource estimates at different cutoff grades were developed from the Firstgold deposit model by WGM. "*The estimates range from 5.6 million tons at 0.022 oz Au/ton to 8.2* *million tons at 0.020 oz Au/ton, with the contained ounces of gold ranging from 121,000 to 163,000*" (Fernette *et al.*, 1996).

In 1997, Independent Mining Consultants, Inc. of Tucson, Arizona ("IMC") were commissioned to review the resource model developed by Firstgold's personnel (IMC, 1997). The IMC estimate was based on a total of 474 drill holes, including those from Falconi, Duval, Lacana, Santa Fe, Pegasus, and the first 73 holes drilled by Firstgold. All of these holes except the one drilled by Falconi were reverse circulation.

Table 6.6 shows the estimates of "model contained resources" (mineral inventory or geologic resource) from IMC, and Table 6.7 compares the "model-contained resource" at cutoffs of 0.003 and 0.019 oz Au/ton from IMC's work with that at the same cutoffs calculated earlier by Firstgold. According to Firstgold's website (http://www.firstgoldcorp.com/our_story.asp, January 2, 2009), the IMC estimates were based on areas considered proximal to the North, South, and Light Bulb pits, and Firstgold controlled about 75 percent of the surface area considered in these estimates. IMC identified the following differences between their revised block model and the one developed by Firstgold:

- IMC used a 22° dip for the search ellipse operator instead of a flat search;
- IMC used ordinary kriging instead of the inverse-distance-squared method;
- The location of Hole 88-3 was corrected in the IMC study;
- The atomic absorption assays of the Pegasus holes from 1988 increased using a factor of 1.25 to approximate fire assays, as opposed to the 1.53 factor used by Firstgold; and
- Firstgold's reverse circulation holes through the end of 1996 were included in IMC's study.

Table 6.6 Historical 1997 Estimate of Relief Canyon "Model-Contained Resources" by IMC
(IMC, 1997)

Cut-off (oz Au/ton)	Tons (000's)	Grade (oz Au/ton)	Contained Au ounces
0.003	76,782	0.012	921,384
0.010	30,483	0.022	670,626
0.015	18,642	0.028	521,976
0.019	12,939	0.033	426,987
0.030	5,629	0.046	258,934
0.040	2,702	0.060	162,120
0.050	1,592	0.072	114,624

Table 6.7 Historical Comparison of Firstgold 1996 and IMC 1997 Historical Estimates
(Modified from IMC, 1997)

	Cutoff Grade = 0.003 oz Au/ton			Cutoff Grade = 0.019 oz Au/ton		
Item	Tons (000's)	Grade (oz Au/ton)	Contained Au ounces	Tons (000's)	Grade (oz Au/ton)	Contained Au ounces
Firstgold 1996 Model	69,345	0.013	901,485	12,866	0.036	463,176
IMC 1997 Model	76,782	0.012	921,384	12,939	0.033	426,987
Variance	7,437	-0.001	19,899	73	-3	-36,189
% Variance	10.70%	-7.70%	2.20%	0.60%	-8.30%	-7.80%

IMC (1997) estimated the "approximate potential minable resources" for the Relief Canyon deposit using a floating cone on the IMC model, a gold price of \$390 per ounce, and a 50° slope angle (Table 6.8).

Table 6.8 Historical Potential "Minable Resources" for the Relief Canyon Deposit
(Modified from IMC, 1997)

Cut-Off (oz Au/ton)	Tons (000s)	Grade (oz Au/ton)	Contained Au ounces
0.050	486	0.065	31,590
0.040	958	0.055	52,690
0.030	1,792	0.045	80,640
0.019	3,697	0.034	125,698
0.015	5,061	0.029	146,769
0.010	7,708	0.023	177,284
0.003	10,870	0.018	195,660
The total material inside the floating cone geometry is 18,925,000 tons			

6.3 2010 Firstgold and 2013-2014 Pershing Gold Mineral Resource Estimates

MDA prepared a mineral resource estimate of the Relief Canyon deposit for the previous operator in 2010 (Gustin, 2010) that was the first estimate reported in accordance with NI 43-101 standards for disclosure at that time. After subsequent drilling and additions to the property that increased the reportable portion of the resource, updated mineral resource estimates were prepared by RPA Inc. in 2013 (Evans and Altman, 2013) and by MDA in 2014 (Tietz and McPartland, 2014), 2015 (Tietz and McPartland, 2015), and 2016 (Tietz et al, 2016).

All of these previous mineral resource estimates are superseded by the current estimate described in Section 14.0.

7.0 GEOLOGIC SETTING AND MINERALIZATION

The information presented in this section of the report is derived from multiple sources, as cited. Mr. Tietz has reviewed this information and believes this summary accurately represents the geology and mineralization of the Relief Canyon property as presently understood.

7.1 Geologic Setting

7.1.1 Regional Geology

The Relief Canyon property is located in northwestern Nevada near the southern end of the Humboldt Range within the Basin and Range physiographic province. The geologic map of the Buffalo Mountain quadrangle (Wallace *et al.*, 1969a) covers the project area. Johnson (1977) summarized the geology of Pershing County, from which much of the following discussion is taken, with additional information from Evans and Altman (2013) and Pershing Gold.

During Paleozoic time, western Nevada was the site of deep-water sedimentation and local marine volcanism, while to the east of Pershing County, predominantly carbonate rocks were deposited on the continental shelf. East-directed compression transported rocks of the western, deep-water assemblages to the east on the Roberts Mountains thrust during the Late Devonian to Early Mississippian Antler Orogeny; evidence for the Antler Orogeny is seen east of the Humboldt Range but not within the Relief Canyon area. A second compressional episode known as the Early Triassic Sonoma Orogeny produced more east-directed transport on the Golconda thrust. The Golconda thrust lies to the west of the Roberts Mountains thrust.

The geologic environment changed in the Early Triassic with deposition of a thick sequence of rhyolitic and andesitic volcanic rocks over much of what is now Pershing County. While part of this deposition may have been in a non-marine environment, drilling at Relief Canyon has intersected interbedded limestone and rhyolitic pyroclastic rocks, indicating that at least part of the Early Triassic volcanic rocks were deposited in a marine environment. A long period of sedimentation began in the late Early Triassic and deposited lithologically-variable marine and non-marine sedimentary sequences at first, followed by more uniform carbonate deposits, including the Smelser Pass Member of the Augusta Mountain Formation and the Cane Spring Formation (formerly the Natchez Pass Formation) in the Relief Canyon area. From Middle Triassic to Early Jurassic time, there were uplift and deposition of near-shore deltaic deposits of mudstone, shale, and sandstone that include the Grass Valley Formation in the Relief Canyon area.

A third episode of regional deformation occurred during Jurassic orogenesis when there was low-grade regional metamorphism, variably directed folding and thrust faulting, and extensive intrusion of granodiorite in the region, particularly within the Luning-Fencemaker fold and thrust belt.

Several episodes of plutonism are recorded in this region, ranging in age from Early Triassic to Tertiary. Early Triassic leucogranites and rhyolite porphyries were associated with contemporaneous volcanism. During the Middle Jurassic, gabbro was widely intruded as sills in the south-central part of the county, south and southwest of the Relief Canyon area. Late Cretaceous granitic stocks are present in the Humboldt Range along with some diabase dikes associated with these stocks. Other Cretaceous and Tertiary granodiorite plutons are exposed in the region.

During the Cenozoic, basaltic, andesitic, and rhyolitic flows, breccias, and tuffs with intercalated lacustrine deposits, fanglomerate, and fluvial sand and gravel were deposited across the county.

The structural regime changed dramatically to extensional events during the Neogene. High-angle normal faulting and tilting of Cenozoic units exemplify this period of regional extension that resulted in the present physiography of the Basin and Range Province.

During the Quaternary period, alluvial deposits accumulated in the structural basins between ranges in Pershing County and are also exposed on the flanks of the ranges.

The regional geology of the Humboldt Range and vicinity is shown in Figure 7.1. Also shown on Figure 7.1 are six gold and silver mines and deposits in the Humboldt and West Humboldt ranges, in addition to Relief Canyon. Those shown as red or blue dots are currently or have recently been active; numerous other historical prospects and mines are shown as small green dots. At the Coeur Rochester mine, about six miles north of Relief Canyon, silver and gold mineralization occurs in stacked, sub-parallel zones of quartz veins, veinlets, and stockworks hosted by folded and faulted volcanic rocks of the Lower Triassic Rochester and overlying Weaver formations (Evans and Altman, 2013). Mineralization occupies high- and low-angle, north- to northeast-trending fault systems (Robinson *et al.*, 2014). The Nevada Packard deposit is three miles south of the Rochester mine and contains mineralization similar to that at Rochester, with silver and gold occurring in vein arrays in northeast-trending, west-dipping faults (Robinson *et al.*, 2014). However, the Packard mineralization tends to have higher silver grades and lower gold grades than those at Rochester, and the mineralized zones at Packard are smaller than those at Rochester (Robinson *et al.*, 2014). Both Rochester and Nevada Packard are owned by Coeur Rochester, Inc., and details can be found at http://www.coeur.com/mines-projects/mines/rochester-nevada.

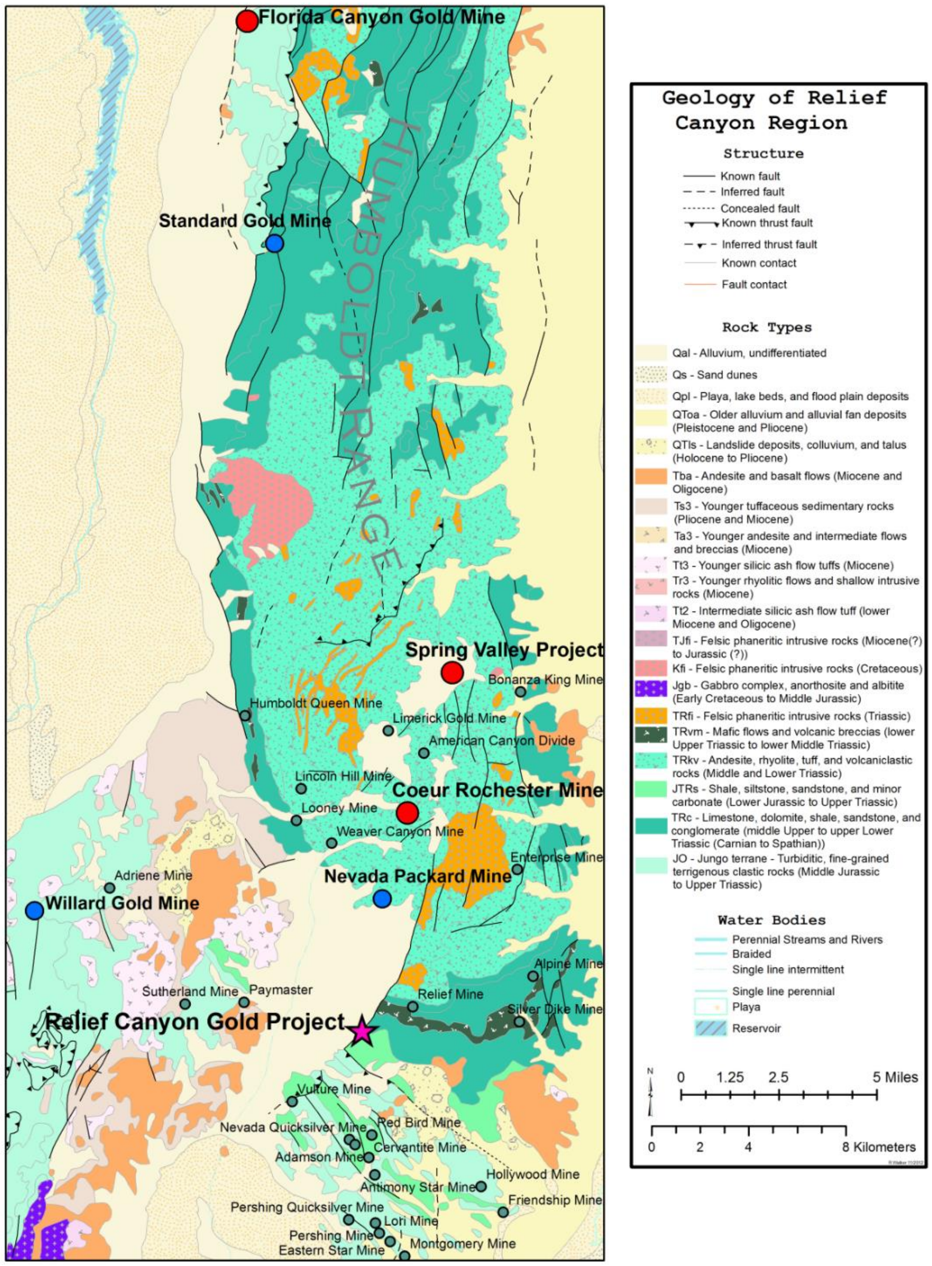
North-northeast of the Coeur Rochester mine is the Spring Valley project of Waterton Global Resource Management. Gold and base metal mineralization occurs in veins that are hosted by the Rochester and Limerick formations, which are part of the Permo-Triassic Koipato Group. The gold mineralization exhibits characteristics of orogenic (metamorphic) or reduced-type, intrusion-related origin (Crosby, 2012; Crosby and Thompson, 2015).

On the northwest flank of the Humboldt Range about 25 miles north-northwest of Relief Canyon, is the past-producing Florida Canyon mine (Figure 7.1). Disseminated gold-silver mineralization is hosted in Late Triassic siliciclastic metasedimentary rocks of the Grass Valley Formation and is localized along and in the footwall of a north-trending range-front fault. From 1986 to 2008, the mine produced 3.18 million ounces of gold from 180.2 million tonnes at an average grade of 0.62g Au/t (Fifarek *et al.*, 2011, cited in Evans and Altman, 2013). The purchase of the Florida Canyon property by Rye Patch Gold Corp. was announced in May, 2016. (Figure 7.1) Additional information can be found: (http://www.jipangu.co.jp/en/about_jipangu/jipangu_goldmine_01). At the Standard mine, low-grade gold mineralization is associated with jasperoid at the thrust contact of the Upper Triassic Grass Valley sandstone and siltstone overlying Middle to Upper Triassic Natchez Pass limestone (Evans and Altman, 2013). Mineralization at the Standard mine also occurs in collapse breccias that are proximal to steep faults and close to altered gabbro in a setting similar to that found at Relief Canyon (D. Prihar, 2014, personal communication).

The Willard gold mine, located 8.75 miles west-northwest of Relief Canyon, near the western margin of the West Humboldt Range (Figure 7.1), is part of the Wilco project of Alio Gold (formerly of Rye Patch

Gold Corp.) (http://www.ryepatchgold.com/projects/wilco/). Epithermal, low-sulfidation (quartz-calcite-adularia) gold mineralization occurs in two deposits at the Wilco project (Evans and Altman, 2013).

Figure 7.1 Regional Geologic Setting of the Relief Canyon Mine
(from Pershing Gold, 2014)



7.1.2 Local Geology

The following information is taken from Fiannaca and McKee (1983), Fiannaca and Easdon (1984), Wallace (1989), Mears (2007), Evans and Altman (2013), and Fifarek et al (2015) with additional details provided by Pershing Gold or from other references as cited. It should be noted that various historical references differ in nomenclature for one of the major units at Relief Canyon. What was formerly called the Natchez Pass Formation was subsequently separated into the Augusta Mountain Formation and the lower part of the overlying Cane Spring Formation (Nichols and Silberling, 1977), with the Cane Spring Formation said by some authors to host or form the footwall for much of the gold mineralization at Relief Canyon. In this report, the stratigraphic terminology currently used by Pershing Gold are Cane Spring Formation, which overlies the Deformed Limestone unit (described in Fifarek et al (2015), which in turn overlies the Footwall Volcanic Unit (Smelser Pass Member of the Augusta Mountain Formation).

The Relief Canyon property lies on the western flank of the southern Humboldt Range on the eastern side of Packard Wash. The Humboldt Range itself is the product of Cenozoic high-angle normal faulting and is one of the typical, generally north-trending fault-bounded ranges of the Basin and Range Province. According to Mears (2007), rocks within the range form a broad anticline with Cretaceous intrusions locally exposed in the central core. The oldest rocks exposed in the Humboldt Range are mafic and silicic volcanic rocks of the arc-related Lower Triassic Koipato Group, with silicic volcanic rocks predominating. The Limerick Canyon Greenstone, Rochester Rhyolite, and Weaver Rhyolite make up the Koipato Group in the southern Humboldt Range. Boron and fluorine-enriched leucogranite and rhyolite porphyry intrusions cut the upper part of the Koipato Group and are thought to be genetically related to the volcanic units. At the Rochester silver district, located about 5 miles north of Relief Canyon, low-grade disseminated and vein-controlled precious-metals mineralization has been mined from the Koipato Group. The Koipato Group is overlain by the Star Peak Group, with marine carbonate rocks that are interbedded with and overlie rhyolitic pyroclastic volcanic rocks. Included in the Star Peak Group are the Smelser Pass Member of the Augusta Mountain Formation and the Cane Spring Formation (formerly the Natchez Pass Formation) in the Relief Canyon area. In Late Triassic and Early Jurassic time, a fluvial-deltaic system deposited predominantly fine-grained sediments of the Auld Lang Syne Group, which is composed of the basal Grass Valley Formation and the overlying Osobb, Dun Glenn, Winnemucca, Raspberry, O'Neill, Singas, Adorno, and Mullinix formations.

There are varying descriptions in the literature of the nature of the contact between the Cane Spring Formation (formerly Natchez Pass) and the overlying Grass Valley Formation. At the Relief Canyon mine, the contact is not well defined because of collapse along it, in response to solution-related brecciation, where it is exposed in the pits and intersected in numerous core holes. Where exposed in the North pit, the contact is characterized by gently inclined to recumbent drag folds that formed in the Grass Valley units immediately above a thrust fault contact with the underlying Deformed Limestone Unit.

Within the Humboldt Range between Middle Jurassic and Middle Cretaceous time, coeval basinal sedimentary rocks were folded and thrust southeastward over the older platform and deltaic rocks, and all the units were deformed and metamorphosed to at least greenschist facies. This was followed by Late Cretaceous emplacement of granitic intrusions.

There are two intersecting structural zones in the region that are Late Mesozoic to Late Cenozoic in age. Possibly the older is a major northwest-trending, right-lateral strike-slip system that forms a topographically pronounced linear fault belt at least five miles wide cutting the southernmost portion of the Humboldt Range and the northernmost portion of the West Humboldt Range. A northeast-trending fault system cuts most of northern Nevada and forms the western margin of the West Humboldt Range. This fault system is Cretaceous or older in age, and the Black Ridge fault is a strand of this system.

Late Cenozoic volcanic rocks were deposited, but largely eroded, during Miocene and younger uplift of the range. There are isolated remnants of Miocene basaltic and rhyolitic volcanic rocks in the southern part of the Humboldt Range. In addition, in the southern part of the range, there are gabbro or weathered mafic dikes of unknown, but older age that were deformed in the Late Mesozoic Luning-Fencemaker fold and thrust belt.

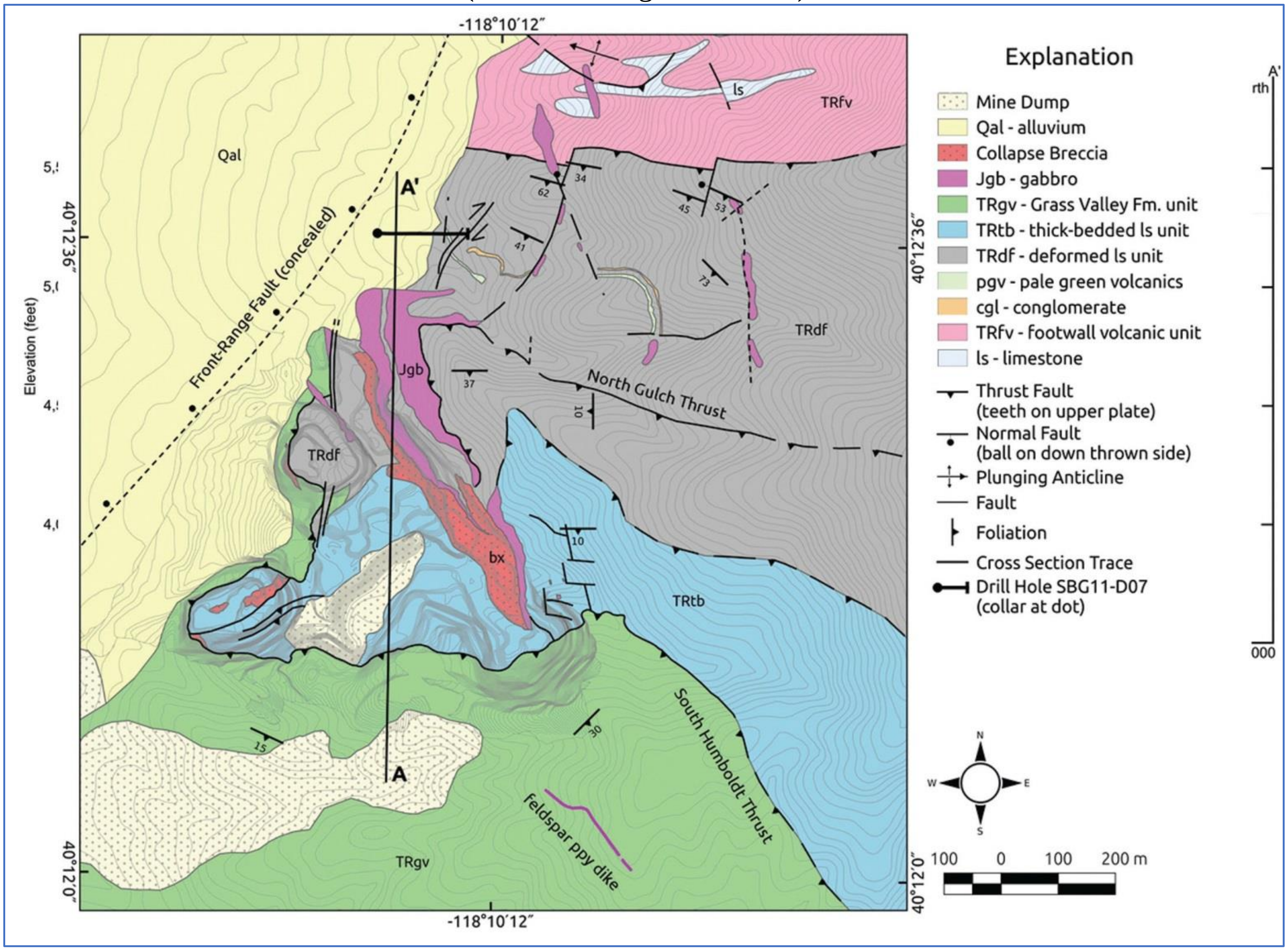
7.1.3 Property Geology

The information in this subsection has been provided by Pershing Gold.

Mesozoic tectonostratigraphy in the vicinity of the Relief Canyon mine consists chiefly of a metamorphosed footwall mafic volcanic package (Smelser Pass Member of the Augusta Mountain Formation); a metamorphosed, foliated, and highly deformed carbonate-dominant package with intercalations of conglomerate and mafic volcanic rocks (Deformed Limestone Unit); a tectonically thickened, thick-bedded to massive limestone unit (Cane Spring Formation); and a tectonically thickened package of siliciclastic rocks of the Late Triassic Grass Valley Formation. These units were all deeply buried sometime during Jurassic orogenesis and display moderate- to well-developed penetrative fabric that was generated by significant shortening during emplacement of the Luning Fencemaker fold and thrust belt (Oldow, 1984; Elison and Speed, 1989). Prior studies emphasized that Mesozoic deformation in northwestern Nevada occurred primarily in lower Mesozoic sedimentary strata deposited in a deep marine back-arc basin between a volcanic arc to the west and the continental shelf to the east (Speed, 1978 and Wyld, 2000). However, these authors did not recognize that metamorphism and development of penetrative fabric also affected continental shelf deposits, as demonstrated in the Relief Canyon mine area, that includes the lower to middle Triassic Shelf Sequence of Silberling and Wallace (1969). Development of penetrative fabric well east of the limit normally described for the Luning Fencemaker fold and thrust belt by Ellison and Speed (1989) and Wyld et al. (2003) was identified by Vikre (2014), but Vikre suggested rocks in the Relief Canyon mine area exhibit little deformation and no penetrative fabric. Field mapping and petrographic studies by Pershing Gold geologists document pervasive metamorphism and penetrative fabric in the Relief Canyon mine area. The eastern limit of the currently accepted Luning Fencemaker fold and thrust belt requires modification. Figure 7.2 shows the generalized geology in the vicinity of the Relief Canyon mine.

A southerly dipping, fault-bounded package of schistose intermediate to mafic metavolcanic rocks, including lenses or boudins of mafic meta-tuff breccia, forms the footwall to gold mineralization at the Relief Canyon mine. This metavolcanic unit (“TRfv – footwall volcanic unit” in Figure 7.2), crops out northeast of the Relief Canyon mine and is the basal unit encountered in drilling. These metavolcanic rocks possibly correlate with the Smelser Pass Member of the Augusta Mountain Formation (Nichols and Silberling, 1977) and were formerly termed the Lower Member volcanic and limestone units of the Natchez Pass Formation of Silberling and Wallace (1969) and Wallace et al. (1969b). These footwall units are host to fracture- and fault-filled quartz-tourmaline-Fe-carbonate-leucoxene veins with local formation of tourmaline and carbonatization of wall rocks adjacent to the veins. Pyrite, pyrrhotite, and arsenopyrite are usually found in these veins along with anomalous values of Ag, As (±W, ±Mo). Local values of silver greater than 1.0 oz Ag/ton, or gold greater than 0.200 oz Au/ton can occur within the veins. These quartz-tourmaline-sulfide veins formed during a period of hydrothermal activity preceding the epithermal gold mineralization that generated the gold resource described in this report.

Figure 7.2 Generalized Geology of the Relief Canyon Mine Area
(From Pershing Gold, 2015)



All of the area covered by this figure lies on land controlled by Pershing Gold.

Overlying the basal metavolcanic unit is a deformed and metamorphosed carbonate package consisting of limy ductile tectonites and calcareous mudstones to silty limestones intercalated with stretched-pebble conglomerate and mafic metavolcanic and intrusive rocks. On Figure 7.2, this unit is shown as "TRdf – deformed ls unit", for the Deformed Limestone Unit. The deformed and metamorphosed carbonate package was assigned by previous workers to the Upper Member of the Natchez Pass Formation (Silberling and Wallace, 1969; Wallace et al, 1969b). Fifarek et al (2015) indicates that the Deformed Limestone Unit may correlate with the Cane Spring Formation and/or Augusta Mountain Formation. Determining the appropriate stratigraphic assignment for this unit is problematical because lithologic and tectonic characteristics of both the underlying Smelser Pass Member of the Augusta Mountain Formation and the overlying Cane Spring Formation are present within the Deformed Limestone unit. These rocks host much of the gold mineralization in the Jasperoid and Lower zones and a minor amount of gold mineralization in the Main Zone; these mineral zones are described in Section 7.2. This deformed and metamorphosed carbonate package is distinguished by foliation that is subparallel to bedding as well as younger, steep axial plane foliation. Folding at various scales is common. Carbonate, quartz, and carbonate-quartz veins occur in this package but typically do not carry detectable gold or silver. The green metavolcanic unit hosts quartz-tourmaline-sulfide veins with trace amounts of gold. These veins are similar in style and interpreted age of emplacement as the veins within the footwall volcanic unit.

A tectonically thickened package of thick- to very thick-bedded limestone overlies the Deformed Limestone Unit. The thick-bedded limestone package, assigned to the Cane Spring Formation (Nichols and Silberling, 1977; Fianacca, 1985; Wallace, 1989), was previous described as constituting the Upper Member of the Natchez Pass Formation (Silberling and Wallace, 1969; Wallace et al, 1969a). On Figure 7.2, this unit is identified as “TRtb thick-bedded ls”, and is shown in Figure 7.3 as the “Massive Limestone” unit. Thick-bedded limestone is best observed in the pit walls of the Light Bulb and South pits at the Relief Canyon mine. Descriptions that follow are based on observations made at the outcrop and in drill core or have been modified from Wallace (1989). Thick-bedded limestone is predominantly micrite to fossiliferous micrite with interbedded silty micrite and biosparite (Wallace, 1989). Bedding-parallel stylolitic layers are common. The thick-bedded limestone package has a thickness of greater than 200 feet that is the product of imbricate stacking of thick to massive beds along low-angle thrust faults. The thick-bedded limestone package was subjected to significant carbonate dissolution through reaction with weakly acidic meteoric waters (Wallace, 1989) and/or weakly acidic hydrothermal fluids. Carbonate dissolution cavities up to 10 feet in length are visible in benches in the Light Bulb Pit. Cavities are locally occupied by stratified cave-fill material, coarse white calcite that probably precipitated from lower-temperature hydrothermal fluids, or drusy calcite that is probably forming today from alkaline meteoric water percolating through the area. The thick-bedded limestone package, especially where there are extensive karst features, is host to the bulk of the Main Zone current and historical gold resources at Relief Canyon.

The overlying Grass Valley Formation is comprised of a tectonically thickened package of siltstone and argillite with intercalated sandstone. The Grass Valley Formation was emplaced by the South Humboldt thrust over the underlying thick-bedded limestone package of the Cane Spring Formation or the Deformed Limestone Unit. The South Humboldt thrust cuts down the tectonostratigraphic section along the northwest portion of the Relief Canyon mine North pit, where it juxtaposes Grass Valley Formation over the deformed and metamorphosed carbonate package without any of the intervening thick-bedded limestone package. The overall package of siliclastic rocks of the Grass Valley Formation is highly deformed and exhibits foliation, intersection lineation, mullions, and multiple sets of folds similar to those in the underlying metamorphosed and deformed carbonate package.

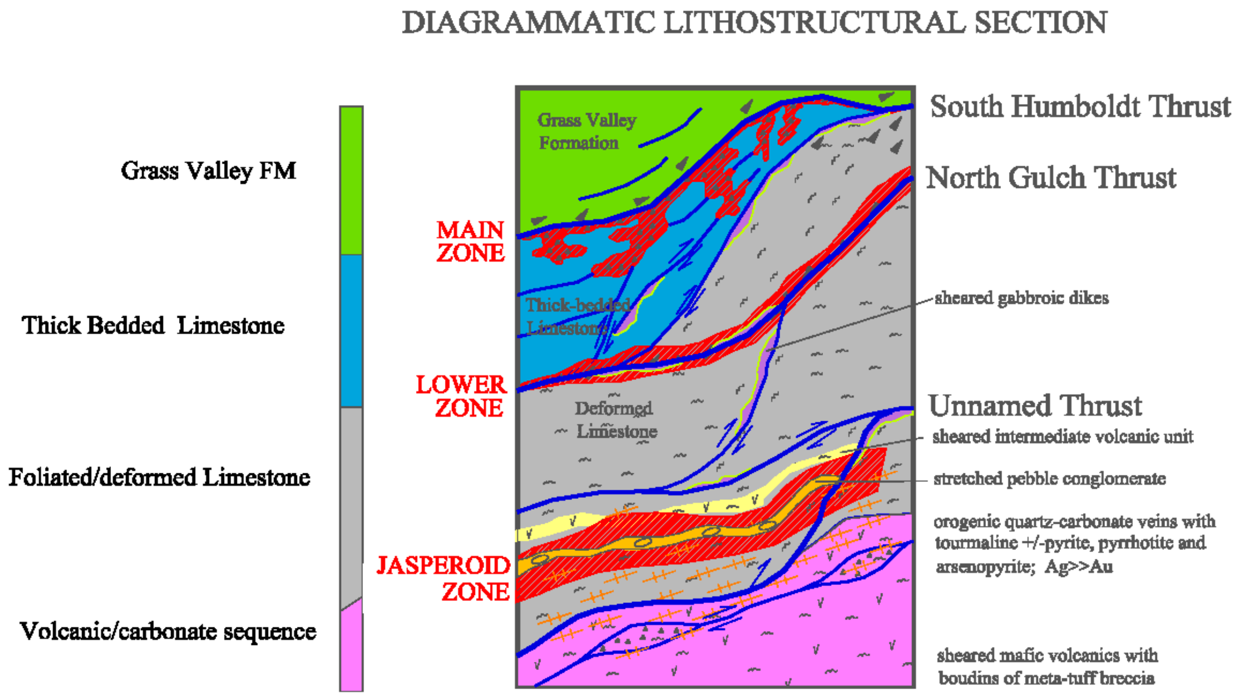
Various types of intrusive rocks have been identified in the Relief Canyon mine area. Dikes of altered feldspar porphyry have only been found cutting the Grass Valley Formation and are therefore late Triassic or younger in age. A swarm of mafic intrusions that vary from sill-like to dike-like are found throughout the southern Humboldt Range. A similar swarm of mafic dikes was described in the northern Humboldt Range by Wallace *et al.* (1969b) and was also mentioned by Wallace (1989). Gabbro is especially prominent in the north highwall of the Relief Canyon mine North and Light Bulb pits, where bodies of gabbro vary in width from a few inches to greater than 100 feet. Gabbro locally shows chilled border phases against limestone and against earlier bodies of gabbro. Gabbro in the Relief Canyon mine area is ubiquitously metamorphosed and/or hydrothermally altered. Penetrative fabric is observed in the majority of gabbro bodies in outcrop and in drill core. A few of the gabbro bodies do not possess penetrative fabric but look similar in hand specimen to those with penetrative fabric. It is unclear if there was more than one pulse of mafic magmatism that emplaced dikes and/or sills in the southern Humboldt Range. Gabbro in the north highwall area locally is attenuated and forms meso- to mega-scale boudinage. The assumption is that the gabbro intrusions were emplaced after deposition of the mid- to late-Triassic sedimentary rocks and were then deformed sometime during the Jurassic orogenesis.

Gold mineralization shows a strong spatial correlation with the gabbro intrusion. Post-emplacement movement along gabbro contacts served to channel hydrothermal solutions. In addition, gabbro in the Relief Canyon area is iron-rich and may have acted as a reductant that reacted with gold-bearing fluids. Locally, gabbro hosts economic grades of gold, chiefly along narrow 1- to 10-foot wide zones adjacent to the contact with mineralized limestone or in rare localities where quartz-calcite-adularia veins are found.

7.2 Mineralization

Gold mineralization at the Relief Canyon mine is primarily found in three zones that are structurally controlled and characterized by distinctive host rocks. From structurally lowest to highest, the zones are the Jasperoid Zone, the Lower Zone, and the Main Zone. The Main Zone hosts the bulk of the current and historical gold resources at Relief Canyon, while the Lower and Jasperoid zones are newly discovered mineral zones encountered below the Main Zone in the North Target area (see Figure 5.1 for location of the North Target area) during the recent Pershing Gold drill programs. Figure 7.3 shows a diagrammatic lithostructural section of the Relief Canyon mine area along with the mineral zone locations. Recognition of these three zones has provided the context for evaluating data from metallurgical testing, and for the selection of metallurgical test samples (see Section 13).

Figure 7.3 Diagrammatic Lithostructural Section of the Relief Canyon Mine Area
(From Pershing Gold, 2015)



The Jasperoid Zone gold mineralization is hosted within the Deformed Limestone package in a sequence of limey ductile tectonites with local stretched and boudinaged quartz veins, stretched-quartz-pebble conglomerate/sandstone, folded and foliated limestone, and altered gabbro, all of which have been replaced by dark-colored quartz. Silicification is also found in a set of sheeted, N30-35°E-striking, steep fractures that may be extensional in origin. In addition, auriferous fluids were localized by brecciation along contacts between lithologies of contrasting competency, such as between the stretched-pebble conglomerate and carbonate units, or less commonly, between pale green schistose volcanic and carbonate horizons. Locally, collapse breccia is also a site of gold deposition in the Jasperoid Zone.

Gold in the Jasperoid Zone shows a strong spatial correlation with silicification, fluorite, and white illite with trace kaolinite. Fine-grained disseminations of pyrite and coarse euhedral pyrite of metamorphic origin are also present. Silver is generally more abundant (> 10 g/t) in this zone than in the Lower or Main zones.

Lower Zone gold mineralization displays a strong spatial association with gabbro sills and/or transposed dikes (i.e., dikes in which progressive ductile deformation has transposed originally discordant contacts into contacts subparallel to foliation, giving the dikes a sill-like appearance). Mineralization is hosted in, or is proximal to, complex tectonic breccias that show multiple generations of structural reactivation and/or shearing, decalcification, locally superimposed carbonate-dissolution collapse breccia, and the presence of illite and/or kaolinite, sulfides iron-oxides and fluorite. Host rocks for mineralization in the Lower Zone include:

- a sheared, protomylonitic to mylonitic, jasperoidal limestone cut by quartz veinlets that are parallel to foliation;
- a carbonate-dissolution collapse breccia comprised of rotated limestone fragments supported by a quartz-chlorite-sulfide matrix that lies beneath the North Pit; and
- a complex mixture of tectonic and collapse breccia that lies under much of the North Target area.

The tectonic breccia was generated by multiple (minimum of two) episodes of movement in the North Target area. The latest movement was likely normal-fault offset in response to Tertiary extension. The principal clasts in the tectonic breccia comprise dark grey foliated limestone, jasperoid, and quartz. Fine-grained gouge material is chiefly composed of quartz flour and kaolinite.

Sheared, strongly foliated and crenulated, dark gray calcareous mudstones also host gold mineralization in the footwall of tectonic breccia in the North Target area. Locally, gabbro hosts gold mineralization along its contacts, especially where post-emplacement movement occurred. Two distinct hydrothermal pulses cut the gabbro. Proximal to a mineralized fault zone along the footwall contact seen in drill hole SBG11-D07, the gabbro is brecciated and completely altered to a very fine-grained assemblage of quartz-illite-kaolinite that can contain 5-50 volume percent sulfides (primarily pyrite). This breccia is cut by fluorite-illite-kaolinite-sulfide veins that represent one of the hydrothermal pulses. Slightly farther from the faulted contact similar breccia clasts comprised of fine-grained granular quartz-illite are cemented by quartz-illite-sulfides-kaolinite and are crosscut by late calcite veinlets. A second distinct gold-bearing alteration assemblage more distal from the mineralized fault contact is composed of chlorite and clays that are cut by quartz-calcite-adularia veins. No cross-cutting relationships have been seen to date that suggest the relative ages of the two hydrothermal phases.

The Main Zone was the source of the historical gold production from the Relief Canyon mine. Gold in the Main Zone is found in a quartz-illite±fluorite-cemented, polyphase, dissolution collapse breccia with cave-fill sedimentary rocks. This collapse breccia formed along and just beneath the South Humboldt thrust which emplaced the siltstones, argillite, and sandstones of the Grass Valley Formation over the Cane Spring Formation “Massive Limestone” unit within southern and central portions of the Relief Canyon mine area and over the “Deformed Limestone” unit at depth in the North Target area. The siliciclastics of the Grass Valley Formation are believed to have acted as a barrier to fluid movement.

Quartz-illite±fluorite±kaolinite alteration at the Relief Canyon mine is associated with gold mineralization in all three of the mineral zones described above. This assemblage is interpreted to have formed from a weakly acidic hydrothermal fluid. A relatively small volume of the gold resource is hosted by gabbro that is cut by quartz-calcite-adularia veins. This is a low-sulfidation assemblage that typically forms from slightly alkaline solutions that boiled at shallow crustal levels.

Anomalous amounts of arsenic, antimony, molybdenum, tungsten, chromium, mercury, and thallium are associated with gold at Relief Canyon, but some gold-mineralized intervals show distinctly low amounts of arsenic with only anomalous molybdenum and tungsten, which is similar to the geochemical signature of orogenic veins. Gold mineralization associated with quartz-calcite-adularia veins is characterized by relatively low amounts of arsenic (< 100 ppm), molybdenum, and tungsten.

At Relief Canyon, mineralization is controlled by low-angle thrust faults, the most important of which, the South Humboldt Thrust, hosts the Main Zone breccia that occurs at the contact between the Grass Valley and the Cane Spring Formations. This thrust contact has undergone significant post-thrust alteration as there is a variety of breccia types (jasperoid, clay matrix, and limestone) along the contact. Mineralization is often associated with a rubbly mixed breccia, which may represent slumping into elongate cavities or chimneys formed by karsting of the limestone along joints and fractures.

Detailed mapping also shows the presence of additional mineralized thrust faults below the lower Grass Valley contact within the thick-bedded limestone package and the underlying deformed and metamorphosed carbonate package. Locally, these faults are believed to be part of a series of stacked low-angle fault zones, and at Relief Canyon, these faults are believed to be the principal control for Lower Zone and Jasperoid Zone gold mineralization. These stacked zones are cut by faults and fractures that trend north-northeast to northeast and also west-northwest. Some geologists at the mine believe a series of northeast- or north-northwest-trending sub-vertical faults could exist below the mineralized stacked thrust faults and may have acted as hydrothermal fluid conduits for the Main Zone and Lower Zone mineralization. Relatively deep drilling of almost 40 holes, from >1,000 feet in depth to >1,800 feet beneath the deposit, has not identified the hypothesized sub-vertical feeder structures. Rather, the shallow southerly plunging, tabular form of the Main, Lower and Jasperoid zones strongly suggests lateral flow of hydrothermal fluids.

As defined by drilling through September 2016, Relief Canyon mineralization occurs as a continuous body over 4,000 feet long north-south and 3,000 feet wide east-west. Mineralization crops out at the surface within the historical open-pits and extends to depths of over 900 feet. See Figure 10.1 for a plan view of the deposit footprint and Section 14.0, for cross-sectional depictions of the structurally-controlled stacked mineral zones at depth.

8.0 DEPOSIT TYPE

The following discussion is taken from Evans and Altman (2013) and Fifarek et al (2015), with additional information from MDA and Pershing Gold. Paul Tietz has reviewed this information and believes this summary accurately represents the Relief Canyon property.

Gold and silver mineralization at Relief Canyon is believed to be largely epithermal in origin, but the deposit also exhibits similarities to Carlin-type deposits (Fifarek et al, 2015) and to orogenic vein deposits as mentioned in Section 7.2.

Gold-bearing jasperoid breccias as well as a suite of trace elements associated with the precious metal mineralization, such as mercury, antimony, and arsenic, are two of the more important similarities between the mineralization at Relief Canyon and Carlin-type gold deposits. Another characteristic of a Carlin-type system is the preferential weathering of the jasperoid material. The jasperoid breccias at Relief Canyon are brittle and highly fractured, which enhances permeability. In many places, they are weathered and oxidized to great depths, whereas surrounding rocks are generally oxidized to shallower depths. A single crystal of euhedral pyrite with a later overgrowth of pyrite has been described in a petrographic study of material from the Jasperoid Zone. Similar arsenic-rich pyrite overgrowths are typical of Carlin-type deposits. It has yet to be determined if the overgrowth in the Jasperoid Zone is arsenic-rich.

As discussed in Section 7.0, there is evidence at Relief Canyon for gold-bearing mineralization of three different styles representing possibly three different ages. The quartz-illite±fluorite±kaolinite alteration associated with gold occurs in all three mineralized zones – Jasperoid Zone, Lower Zone, and Main Zone – and is the most important type of alteration. This assemblage is believed to have formed by weakly acidic hydrothermal fluids. Illite from the quartz-illite±fluorite±kaolinite alteration assemblage has been dated with $^{40}Ar/^{39}Ar$ methods at 23.51±0.11 Ma (Fifarek et al, 2015). Low-sulfidation quartz-calcite-adularia veins with associated gold are hosted by gabbro and constitute a relatively small volume of the mineral resource. This assemblage typically forms from slightly alkaline solutions. Adularia from this assemblage has been dated at 14.92±0.05 Ma with 40Ar/39Ar methods (Fifarek et al, 2015). The presence of orogenic vein mineralization is indicated by the alteration mineral assemblage of quartz-carbonate-tourmaline-leucoxene, in which some gold-mineralized intervals are associated with low amounts of arsenic, but with anomalous molybdenum and tungsten.

9.0 EXPLORATION

Since acquiring the Relief Canyon mine in August 2011, Pershing Gold has conducted additional core and reverse circulation drilling in order to expand the resource and has also conducted drilling for some target development and testing away from the pits. Pershing Gold's drilling is discussed in Section 10.0.

Pershing Gold has also conducted geophysical surveying, geologic mapping, and rock and soil sampling around the mine and within the district. A program of 10 east-trending Controlled Source Audio Magneto-Telluric ("CSAMT") lines totaling 23.54 miles was completed across the Packard Flat alluvial sediments. Two additional CSAMT lines totaling 2.92 miles and oriented N30°E were completed south of Packard Flat, in the northwest corner of the Pershing Pass area. Two IP-resistivity lines totaling 3.48 miles were completed in an east-west orientation across the north edge of Packard Flat, just south of Coeur Mining Inc.'s Nevada Packard open-pit mine.

Three geophysical anomalies beyond the resource area were tested with 4 exploration drill holes totaling approximately 2,800 feet.

Pershing Gold undertook detailed mapping in the pits and adjacent areas in 2012 to collect structural data that would help refine understanding of the complex geology at Relief Canyon and to identify additional drill targets. They have also collected surface rock-chip samples for geochemical analyses.

MDA and Mr. Tietz have not analyzed the sampling methods, quality, and representativity of surface sampling on the Relief Canyon property because drilling results, and not the surface samples, form the basis for the mineral resource estimate described in Section 14.0.

10.0 DRILLING

The information presented in Section 10.0 is derived from multiple sources, as cited. Mr. Tietz has reviewed this information and believes this summary accurately represents the drilling conducted at the Relief Canyon property.

10.1 Summary

The mineral resources discussed in this report were estimated using the data provided by reverse circulation and core drilling completed by Duval, Lacana, Santa Fe, Pegasus, Firstgold, and Pershing Gold.

As described by Tietz et al. (2016) and updated herein, the Relief Canyon drilling has defined a zone of gold mineralization within the jasperoidal-clay breccia lying immediately below the Grass Valley Formation. This Main Zone mineralization was exploited by past mining activity. Recent drilling by Pershing Gold to the north of the area of past mining activity ("North area") has encountered structurally controlled mineralization in the Lower Zone and Jasperoid Zone at depth beneath the Main Zone.

The project database used to estimate the current mineral resources now has a total of 1,095 drill holes for 482,755 feet of drilling: 419 core holes for 244,353 feet and 676 reverse circulation holes for 238,402 feet. This total includes drilling by Pershing Gold through September, 2016. Table 10.1 is a summary of the drilling included in the project database. The database does not include the hole drilled by Falconi in 1978, and it is likely missing some holes drilled by some of the companies listed in Table 10.1, as well. As noted in Section 6.0 and discussed further below, there are a number of discrepancies in the total numbers of holes drilled by the various companies in reports reviewed by MDA. Figure 10.1 shows the location of the drill holes.

The Main Zone breccia that hosts most of the Relief Canyon mineralization modeled by MDA forms a broad antiform. Much of the crest of the antiform is subhorizontal to shallowly plunging, while the limbs generally dip at angles less than 30°. Drilling targeting the Main Zone breccia are primarily short vertical reverse circulation holes drilled by operators before Pershing Gold. The vertical holes are generally well oriented with respect to the Main Zone breccia-hosted mineralization. The database contains 341 holes drilled at an angle. The majority of the angle holes were core holes drilled by Pershing Gold targeting North area structurally controlled Lower Zone and Jasperoid Zone mineralization beneath the Main Zone breccia.

The project database also includes 20 excavator trenches within the dump/stockpile area located west-southwest from the historical mined South pit. The trenches were dug by Pershing Gold in 2014 and 2015 to provide additional confidence in the mineralization previously defined by shallow core and RC drilling. Each trench is about 50 feet long and 15 feet deep and samples were collected from the excavated material along the length of the trenches (Casaceli, 2015). These trenches are not included in the Table 10.1 drill summary.

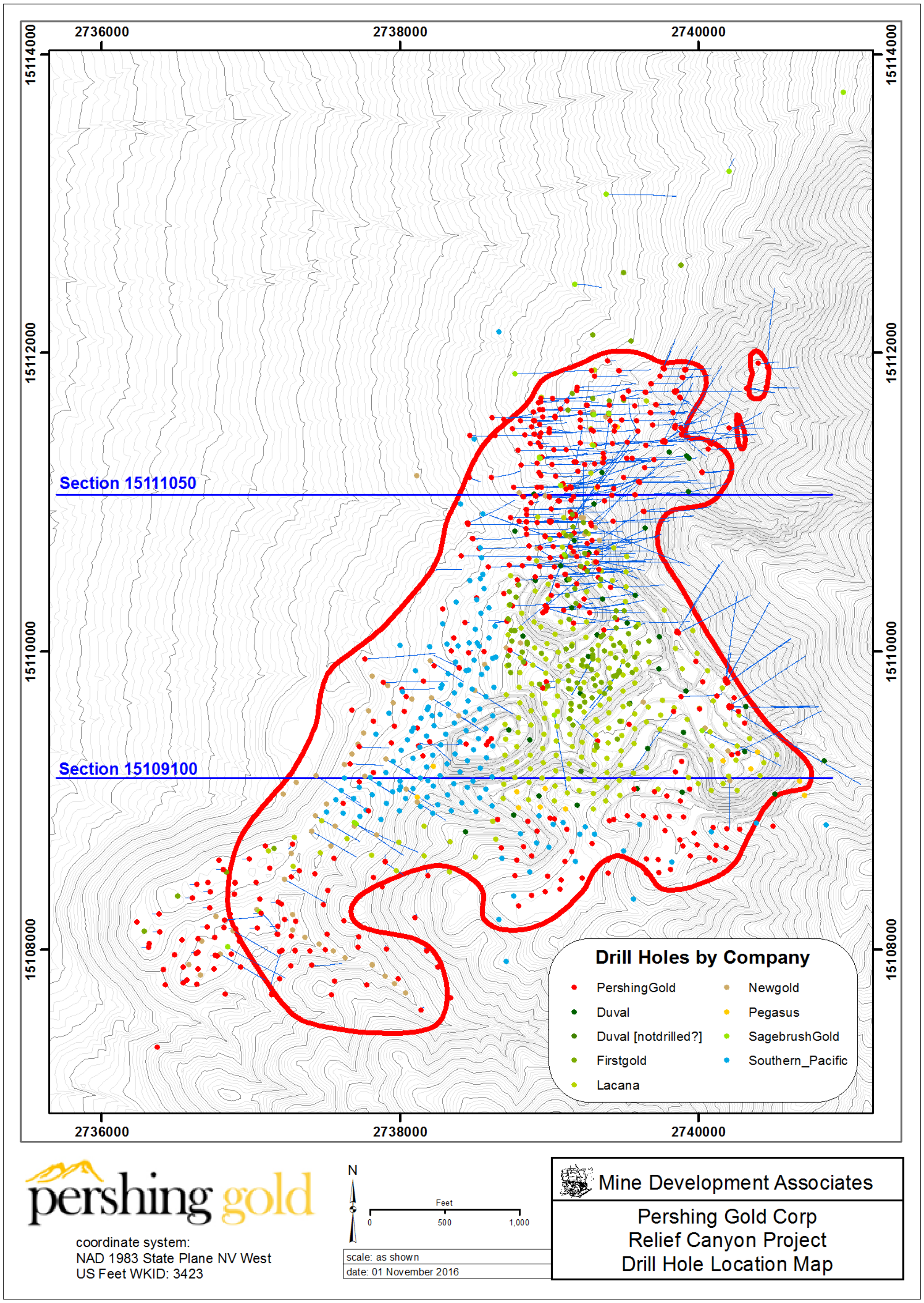
Table 10.1 Relief Canyon Mineral Resource Drill Database Summary

Company	Period	Hole Numbers	Core		Reverse Circulation		Total	
			No.	Feet	No.	Feet	No.	Feet
Duval	1981-1982	DVR1 - 45[1] (excludes 27, 39-41)			41	13,663	41	13,663
Lacana	1982-1983	LRC1 - LRC203 (includes two re-drilled holes[2])			205	50,453	205	50,453
Santa Fe	1983-1984?	SPRC1 – 148[3] (excludes 80,114-119,129) (includes six “A” holes)			146	47,688	146	47,688
Pegasus	1987-1988	PRC87-02, 03, 06 - 15[4] (excludes 04, 05) PRC88-1 through 5			17	5,100	17	5,100
Firstgold (Identified as Newgold holes on Figure 10.1)	1996-1997	9601-9640 (excludes 07, 14, 16, 22-24, 39) 9702-9743 (excludes 18, 41)			73	50,420	73	50,420
Firstgold	2007-2008	RCM07-01 - 75[5] RCM08-01 – 19 RC - D1 NT07-01, NT08-01 – 10 NT08-D01, 03, 04	4	4,578	105	39,113	109	43,691
Pershing Gold	2011-2013	SGB11-D01 – D03, D05 – D08 SBG11-RC01 – RC05 SGB12-D01 - D07 RC12-008 – 030 RC12-031R – 114R RC13-115 – 121, 122M – 126M, 127-129, 129A, 130 – 148, 148A, 149-152	77	53,827	89	31,965	166[6]	85,792
Pershing Gold	2014-2016	RC14-158 – 262 RC15-263 – 458 RC15-286M – 288M[7] RC16-459 – 475	338[8]	185,948			338	185,948
TOTAL			*419*	244,353	*676*	*238,402*	*1,095*	482,755

[1]DVR42 - 45 may have been drilled by conventional rotary
[2]In cases of original and re-drilled hole sets, assay data available for re-drilled holes only
[3]Assay data unavailable for 13 holes
[4]Assay data unavailable for PRC87-03
[5]Assay data unavailable for RCM07-24
[6]Total does not include four piezometer holes (RC13-153 – 156) and one monitor well (MW13-01)
[7]Metallurgical holes; assay data not available
[8]Includes eight holes re-drilled due to poor recovery in original holes

Figure 10.1 Relief Canyon Drill-Hole Location Map

(Red outlines show current resource footprint; cross-sections are shown in Figure 14.1 and Figure 14.2)



10.2 Historical Drilling

10.2.1 Falconi

While exploring the property for high-purity limestone, Falconi drilled a single core hole to a depth of 745 feet (Fiannaca and Easdon, 1984). MDA has no further information on this drilling, and no information from this hole is included in the database.

10.2.2 Duval

According to Fiannaca and Easdon (1984), Duval Corporation drilled 40 reverse circulation holes on the property in 1981-1982 for a total of 13,148 feet. Fiannaca and McKee (1983) had previously reported that Duval drilled 38 reverse circulation holes in 1981-1982. Mears (2007) reported that Duval drilled 44 holes totaling 15,080 feet. However, the database used by MDA contains 41 holes totaling 13,663 feet. The author cannot account for these discrepancies. MDA has no information on the drill contractor or type of equipment used for Duval's drilling.

Drill logs indicate that the Duval reverse circulation holes drilled in 1981 were drilled dry until encountering the water table.

10.2.3 Lacana

Upon exercising its option and acquiring Relief Canyon, Lacana drilled 48 reverse circulation percussion holes totaling 12,610 feet in a first phase of drilling to provide details on the deposit as defined by Duval. The following information on Lacana's drilling is taken from Fiannaca and Easdon (1984) and Fiannaca and McKee (1983).

Lacana's holes were drilled on 200-foot centers such that the holes fell at the apices of a series of contiguous equilateral triangles. This pattern was chosen to eliminate directional bias in the grid and to permit construction of cross sections from three directions. Eklund Drilling Company of Elko, Nevada, was the drill contractor for these 48 holes. Data in the drill-hole database indicate that Eklund used a TH-60 rig for these holes, although for the last few holes the database indicates that LL Enterprises Eklund was the contractor using a TD-105 or TD-100 rig.

Nearly all of the samples collected in the breccia unit were dry. Apparently some wet conditions were encountered in other units.

Results of this drilling led to Lacana's constructing a pilot heap-leaching facility, during which each of four potential mining sites were drilled by reverse circulation methods on 25-foot centers. After 2 sites were selected for the pilot test, 140 blastholes were drilled on 4- to 6-foot centers. MDA has no details on this drilling except that all blasthole cuttings were fire assayed for head-grade control. The blastholes are not included in the database used by MDA.

While the pilot test was underway, Lacana began a second phase of reverse circulation drilling to sample the main zone of mineralization on 100-foot centers. A total of 99 reverse circulation holes were drilled during this phase for a total of 24,038 feet. A third phase of drilling was then undertaken to better define the pit perimeter and to condemn waste dump sites; this phase consisted of 57 reverse circulation holes totaling 13,715 feet. The total of the three phases of drilling is 204 holes, although the drill-hole database used by MDA shows 205 Lacana holes with a total of 50,453 feet of drilling; MDA cannot account for the discrepancy, although it may involve counting of holes that were started and then re-drilled.

MDA has few details concerning the drilling contractors, rigs, or drilling conditions for the second and third phase of Lacana's drilling. Where a drill contractor is shown in the database, it is listed as Eklund using a TH-60 or TH-100 rig.

10.2.4 Santa Fe

In the 1980s, Santa Fe owned property adjacent to Lacana's property and drilled 147 reverse circulation holes to test for continuation of mineralization onto their property, according to Fiannaca and Easdon (1984). A total of 146 of these holes are included in the drill-hole database with a total of 47,688 feet drilled; MDA cannot account for the difference. MDA has no information on the drilling contractor or type of rig used for Santa Fe's drilling.

Santa Fe drilled the portion of the Relief Canyon deposit under their control in Sections 17 and 21, which consists primarily of the northwest-dipping limb of the antiformal mineralized breccia horizon. Most Santa Fe holes were therefore drilled deeper than the Lacana holes and encountered considerably more groundwater, which caused sampling problems. Drill logs indicate that Santa Fe drilled their holes dry until the water table was intersected.

10.2.5 Pegasus

According to Mears (2007), Pegasus drilled 11 reverse circulation holes totaling 3,545 feet on the property, but the drill-hole database used by MDA contains 17 holes totaling 5,100 feet; MDA cannot account for the difference. According to the database, these holes were drilled in 1987 and 1988. The 5 holes shown as drilled in 1988 appear from the database either to have been drilled by Eklund Drilling or with equipment leased from Eklund. Drill logs indicate that these 5 holes were drilled dry until either drilling conditions required the injection of water or ground water was intersected.

MDA has no further information on drilling equipment or procedures.

10.2.6 Firstgold

The only drilling by Firstgold described in this section was for exploration and identification of remaining and potential new resources. Firstgold also investigated the potential for reprocessing heaps remaining from previous mining, including drilling of the heaps, but this drilling program is not discussed in this section.

From its acquisition of the property in 1995 through April 1997, Firstgold drilled 73 reverse circulation drill holes that were 6.5 inches in diameter (Mears, 2007). These holes are shown as having been drilled by Newgold on Figure 10.1; Newgold changed its name to Firstgold in 2006. Based on the database used by MDA in 2010, this drilling totaled 50,420 feet; however, according to Mears (2007), the 73 holes totaled 43,220 feet. MDA cannot account for the differences in total length drilled between the database and that reported by Mears (2007). The focus of this drilling was just north of the North pit, west of the pits, and southwest of the pits (Mears, 2007). MDA has no details on the drilling contractor or type of rig used except for an entry in the database for the first hole drilled in 1996 that indicates Five-O was the drilling contractor.

In 2007, Firstgold again began exploration drilling at Relief Canyon, initially focused on the area of the existing pits from prior mining operations (Firstgold, 2007e). Their drilling included shallow twin holes and infill drilling to confirm grade and continuity of the gold mineralization (Firstgold, 2007d). Later drilling tested deeper targets within the pit and outside it to the northwest in the pediment area, between the North and South pits, and in the North Target (Firstgold, 2008a, 2008c, 2008d). In 2007 and 2008, Firstgold drilled four core holes totaling 4,578 feet and 105 reverse circulation holes totaling 39,113 feet, based on the database used for the resource estimate.

The information that follows was provided by Firstgold to MDA in 2010.

Firstgold drilled these holes using their own crew and equipment. Reverse circulation holes RCM07-1 through RCM07-72 were drilled with an MPD 1000 rig using a 5 1/8-inch diameter hammer bit, with the exception of holes RCM07-24, RCM07-29, RCM07-31, and RCM07-38, which used a 4 ¾-inch rock bit. Holes RCM07-73 through RCM07-75 and holes RCM08-1 through RCM08-16 were drilled with an IR TH-75E rig; most used a 5 ¾-inch hammer bit, except for holes RCM08-15 and RCM08-16, which used 5 3/8-inch rock bits and hole RCM07-73, which used a 5 ¾-inch hammer and 5 ½-inch and 5 ¼-inch rock bits. Holes RCM08-18 and RCM08-19 were drilled with a Schram 685 rig using a 6-inch hammer bit. Additional holes were numbered NT07-1 and NT08-1 through NT08-10. All except NT07-01, NT08-01, and NT08-03 were drilled with a Schram 685, while the remaining holes were drilled with an IR TH-75E rig. The bits were either rock or hammer and ranged from 5 3/8-inch to 6-inch in diameter.

Water was encountered at depths of between 120 and 460 feet in 33 of the reverse circulation holes.

For its core drilling, Firstgold drilled one core hole between the pits (RC-D1) and four core holes in the North Target (NT08-D1 through NT08-D4), of which one was abandoned. The core holes were drilled with a UDR 200DLS rig using HQ bits.

10.3 Pershing Gold

From 2011 through September 2016, Pershing Gold drilled 415 core holes and 89 reverse circulation holes. Information on the rig types was taken from the current database and information provided by Pershing Gold.

The initial drilling in 2011 and early 2012 consisted of 14 core holes and five reverse circulation holes testing targets north of, and at depth below, the previous resource estimate. Carpenter Drilling of Benton City, Washington, was the drill contractor for the 2011 and early 2012 core holes (SBG11-D01 – D03, D05-D08; SBG12-D01-D07), using an LH 90D rig. O'Keefe Drilling Company drilled the 2011 reverse circulation holes (SBG11-RC01 – RC05).

The 2012 drilling consisted of 30 core holes and 84 reverse circulation holes. Boart Longyear drilled 13 of the 2012 core holes using a BL28 rig (RC12-08, 10, 12, 14, 16, 18, 20, 22, 24-26, 29-30). Ruen Drilling Inc. drilled 10 of the 2012 core holes (RC12-09, 11, 13, 15, 17, 19, 21, 23, 27, and 28), using a track-mounted LF90 rig. The core drilling primarily focused on extending mineralization on the west side of the historical pits. The 2012 reverse circulation holes were drilled by National Exploration, Wells and

Pumps using a 178 rig (RC12-031-114R). The reverse circulation drilling targeted the southern extension of the Main Breccia Zone and evaluated the gold mineralization within the waste-rock dump.

In 2013, West-Core Drilling, LLC ("West-Core") of Elko, Nevada, drilled 40 core holes, using a rubber-track-mounted Atlas Copco CS-14-1 rig (RC13-115-121, 122M-126M, 127-129, 129A, 130-148, 148A, 149-152). The 2013 core drilling primarily targeted structurally controlled mineralization at depth north of the previous resource estimate. Five metallurgical holes (RC13-122M-126M) were drilled within the Main Breccia Zone.

In 2014 and through December, 2015, West-Core completed 99 core holes using an Atlas Copco CS-14 rig. Timberline Drilling, Inc. of Hayden Lake, Idaho, completed 39 of the holes in 2014 using a modified Sandvik DE130 (formerly Hagby) drill. Primary goals of this drill program were to extend and confirm high-grade zones in the Lower and Jasperoid zones as well as extend mineralization southwest of the historical mining areas. The core drilling also served to systematically replace much of the North area reverse circulation drill data, due to possible down-hole contamination in the reverse circulation holes, and thereby increase confidence in the Lower Zone and Jasperoid Zone gold resource.

The 2016 drilling (through September 2016) included seven core holes drilled for geotechnical purposes and 11 core holes targeting expansion of the resource downdip and to the west. The geotechnical holes were drilled by AK Drilling of Butte, Montana using a track-mounted LF-90D drill rig, and by Titan Drilling of Elko, Nevada using a track-mounted, modified Atelier Val'Dor 5000 drill rig. The resource expansion core holes were completed by Timberline Drilling using a track-mounted, modified Sandvik DE130 diamond core rig and by Titan Drilling using the Atelier Val'Dor 5000 rig.

Drilling by Pershing is ongoing as of the date of this report. Approximately fifty-five core holes have been completed since the effective date of the current resource estimate. See Section 14.11 for a discussion of these drill results and their potential impact on the current resource estimate.

10.4 Drill-Hole Collar and Down-Hole Surveys

Uncertainties with respect to collar elevations and x-y positions in several historical holes remain, although these are not likely to be material to the resource modeling discussed in Section 14.0.

For Pershing Gold's 2011-2012 drill program, collars were staked with hand-held GPS and marked with a temporary hole identification. Final collar locations were then surveyed by an in-house surveyor with a Trimble GPS and uploaded directly to the database (Evans and Altman, 2013). Similar collar location and survey procedures were used for Pershing Gold's 2013 through 2016 drilling. A re-survey in late 2013 of approximately 20 Pershing Gold drill holes by a third-party surveyor using a sub-meter-accuracy Trimble indicated that the 84 reverse circulation drill holes completed in 2012 (RC12-031R thru RC12-114R) had a consistent 32-foot shift in the Northing coordinate. The collar locations for these reverse circulation holes were corrected based on the third-party surveys. A material shift in drill-hole coordinates was not recognized in the other drill campaigns.

No down-hole survey data are available for any of the historical drill holes in the database, so all of these holes are assumed to have constant dip angles. This assumption is likely to introduce increasing error with increasing depth of the drill holes, although the shallow nature of most of the modeled mineralization and the prevalence of vertical holes (only 31 of the 591 historical holes in the database were drilled at an angle) likely minimizes any impacts in the resource modeling.

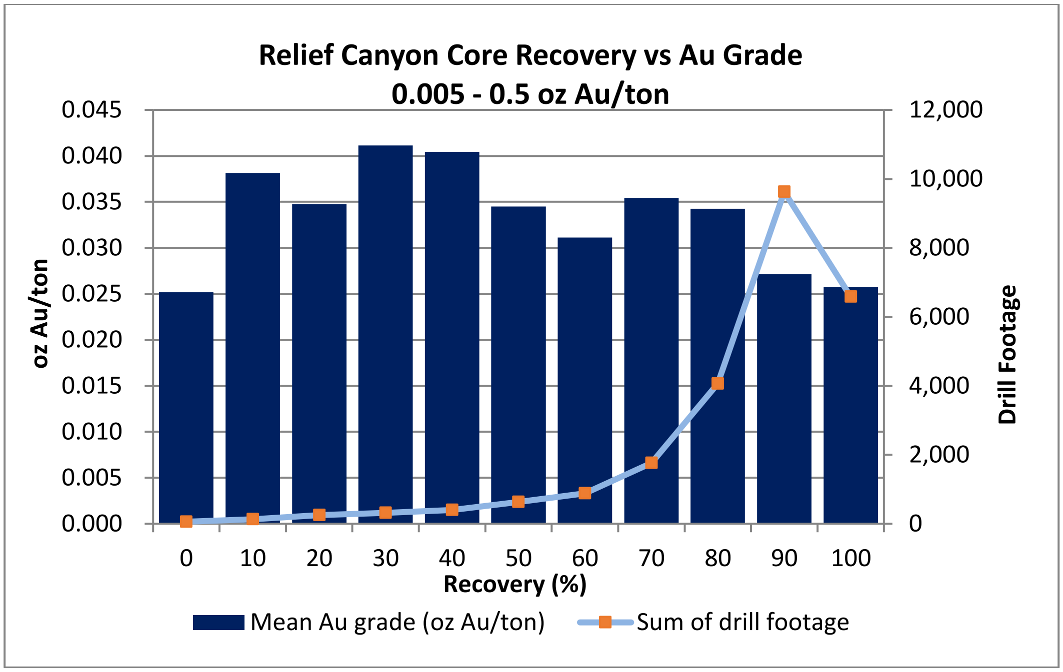
For Pershing Gold's drilling, the drillers recorded down-hole survey measurements using a Reflex tool on 395 of the 415 Pershing Gold core holes, and 5 of the 89 reverse circulation holes. The 84 unsurveyed reverse circulation holes were shallow (<400 feet) vertical holes, which likely minimizes any impacts of potential hole deviation on the resource.

10.5 Core Recovery/RQD Analyses

Average core recovery for all Pershing Gold sample intervals is 90 percent, while average core recovery for those intervals assaying over 0.004 oz Au/ton is 89 percent. The core is generally highly fractured within the mineralized horizons, and rock quality designation "RQD" measurements are typically low, averaging about 10-20 percent.

Poor core recovery may have an impact on grade assessment. For the mineral resource and PEA completed in June 2016 (Tietz et al, 2016), MDA analyzed the drill data to determine if there was a deposit-wide relationship between poor recovery intervals and gold grades. Figure 10.2 shows the mean gold grade (blue vertical bars) and the drill footage (light blue line with orange data points) plotted in the vertical axes, while core recovery is plotted along the horizontal axis. The figure includes those mineralized intervals assaying 0.005 oz Au/ton or greater, with the very high-grade (>0.5 oz Au/ton) intercepts excluded due to their tendency to skew the statistics. The core-recovery data have been separated into distinct bins for each 10 percent increase in recovery. So the "70" value in the horizontal axis contains all data points which have core recovery values between 70 and 79 percent. The "100 percent" core recovery bin includes all drill intercepts with 100 percent or greater core recovery. Approximately 2 percent of these drill intercepts have calculated core recovery values greater than 100 percent. These infrequent intercepts are the result of core re-drill or minor footage measurement errors.

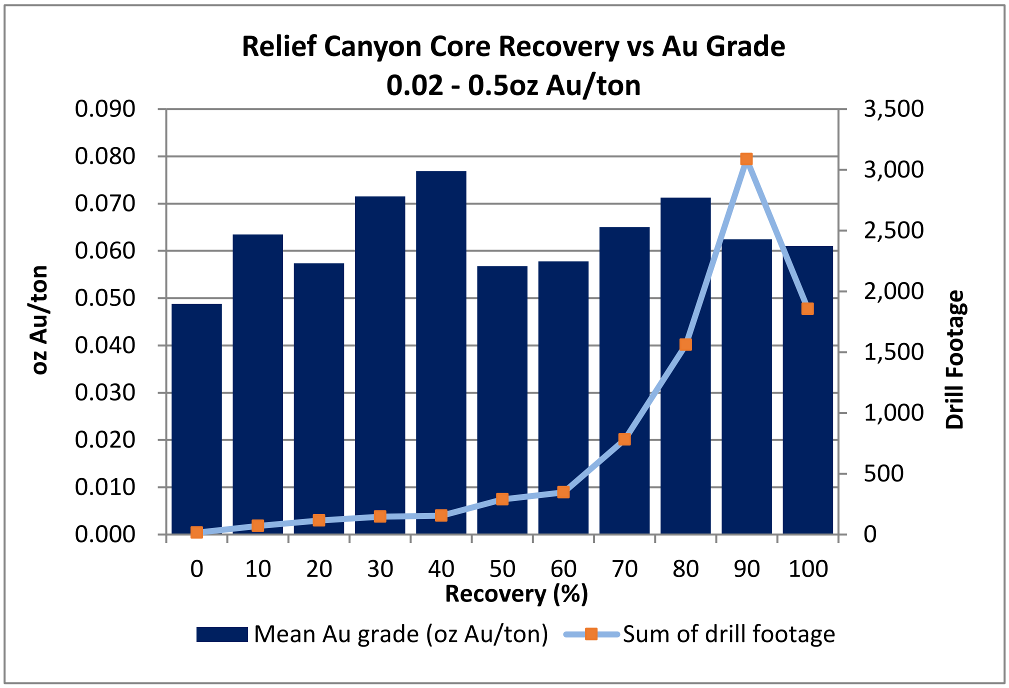
Figure 10.2 Core Recovery versus Gold Grade - >0.004oz Au/ton



There is an increase in gold grade of about 30 percent associated with decreasing core recoveries down to about 70 percent recovery. Below 70 percent, gold grades become more erratic though remain 20 to 35 percent above the gold grades for those intervals above 90 percent recovery. The drill intervals having less than 70 percent core recovery represent approximately 10 percent of the total mineralized core intervals.

MDA reviewed the drill data using 0.02oz Au/ton lower cut-off grade for the data set to see whether the increase in gold grades in the 70 to 90 percent recovery ranges is observed within the higher-grade drill intercepts (Figure 10.3). The grade increase is still observed, though the increase is now approximately 5 to 10 percent. There are also the same erratic gold grades below 70 percent, though the gold grades are now both higher and lower than those intervals with minimal core loss.

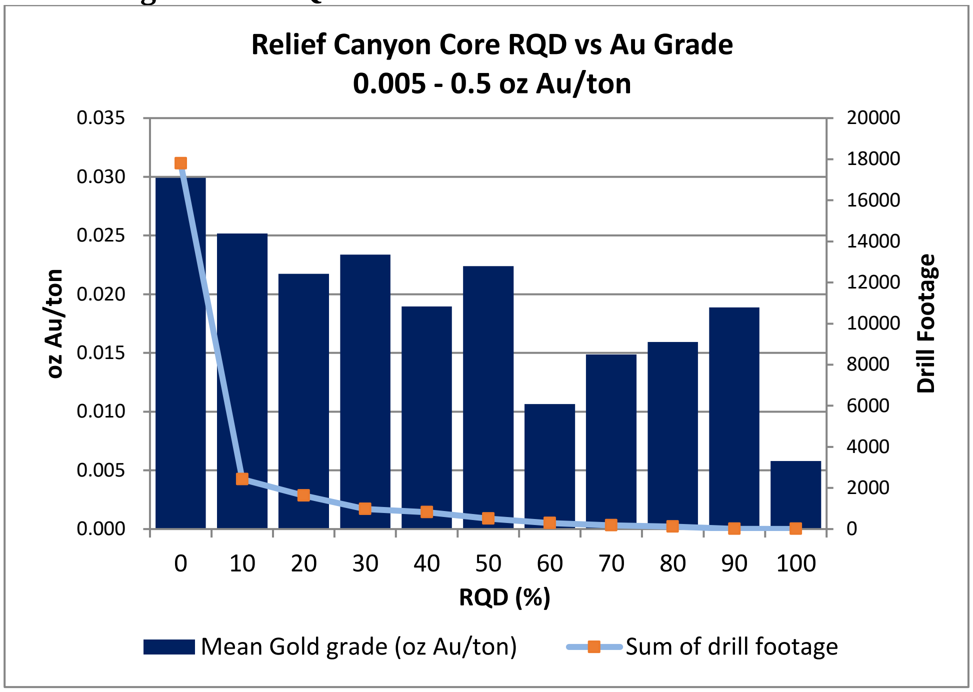
Figure 10.3 Core Recovery versus Gold Grade – >0.019oz Au/ton



The observed increase in gold grades with decreasing recovery occurs in all grade ranges but is most prevalent, on a percentage increase basis, within the lower grade ranges.

Figure 10.4 has the same format as the previous figures, but RQD replaces the core recovery data thereby showing the relationship between rock quality and gold grade. The drilled footage ("Drill Length") line clearly indicates the very low RQD values prevalent within the deposit, with the greatest number of intervals having RQD values within the 0 to 10 percent RQD bin. The gold grades clearly show the relationship between increasing RQD and decreasing gold grades.

Figure 10.4 RQD versus Gold Grade - >0.004oz Au/ton



It is expected that core intervals with very low RQD values would have a tendency to have lower core recovery as compared to those weak to moderately fractured core intervals with increased RQD values. Due to the observed correlation throughout the deposit between higher gold grades and more highly fractured/brecciated core intervals, it is not surprising that increased gold grades are associated with intervals of lower core recovery. It is unclear though to MDA whether the observed increase in grade is directly related to any preferential core loss of the weakly mineralized portion of the core interval, or just reflects the inherent relationship between increasing grade and the highly fractured rock. MDA believes the uncertainties associated with increasing grade and low core recovery represents a low risk to the resource estimate.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

The Relief Canyon database includes assay data from reverse circulation and core drill holes. While it is likely that down-hole contamination presents a sample integrity issue in some reverse circulation holes below the water table, the author of this section believes techniques employed during the resource modeling have limited the problem, as discussed below, and any potential remaining issues are appropriately reflected in the resource classification.

The preponderance of samples for all drill programs of all operators were taken at 5-foot intervals, as is customary for reverse circulation drilling, and which is significantly less than the thickness of the bulk-tonnage style of mineralization at Relief Canyon. Each drill sample interval is, therefore, a fraction of the true thickness of the mineralized zones.

11.1 Falconi

Fiannaca and Easdon (1984) state that although the single Falconi core hole was drilled through mineralized breccia into the Natchez Pass Formation limestone, but at the time, the core was not assayed for gold.

11.2 Duval

MDA has no information on the sampling methods used by Duval during their drilling at Relief Canyon, other than some data regarding wet or dry sampling (see Section 10.2.2), including Duval's sampling and assaying of the Falconi core. The only information MDA has on Duval's analyses is that pulps were assayed by ½-assay-ton fire assays (Fiannaca, 1982; Fiannaca and Easdon, 1984).

11.3 Lacana

The following information is taken from Fiannaca and Easdon (1984).

Lacana took the following measures to mitigate and quantify any effects of down-hole contamination in their reverse circulation drilling program:

- Lacana personnel were used for sample collection;
- Holes were air-cleaned at the bottom of each 5-foot interval;
- Water and/or stabilizers were injected to minimize caving as needed;
- The sample-collection cyclone and splitter were continuously cleaned; and
- When drilling below the water table, contamination samples were collected while the drill continued to circulate for three-minute intervals.

Nearly all of the samples that Lacana collected in the mineralized breccia unit were dry. Each 5-foot interval in the breccia was collected and assayed. When drilling in dry conditions, each sample was collected in a cylindrical cyclone device equipped with a 10 foot-high baffled stack to minimize the loss of fines. The dry samples were dumped into a 42-inch Jones-type splitter, and both splits were remixed and re-split twice to insure homogeneity of the final split. During drilling of the 48 holes of the phase-one program, three identical, equal-volume samples of each drill interval were collected by designing inserts for the splitter pans which contained exactly 1/3 the volume of each pan. All samples of the first split were sent to the primary assay laboratory; every fifth sample of the second split was sent to a secondary assay laboratory for check; and all the samples of the third split were stored for future metallurgical testing.

All samples were collected in pre-labeled polyethylene sample bags and marked with identification tags placed into each bag. Sample weights commonly varied between 10 pounds and 15 pounds, and the drill-sample recovery was stated to be "generally excellent".

When drilling in wet conditions, each sample was discharged through the opened cyclone onto either a single-deck or a triple-deck Jones splitter, and the water-rock mixture was collected in 40-gallon plastic cans. The mixture was allowed to settle, and CaCl was added to rapidly enhance flocculation of the slimes. Clarified water was removed by siphon, with care taken to prevent remixing of the sample slurry. The sample was removed from the plastic can and was shipped to the laboratory either in pre-labeled, double-lined polyethylene bags or in plastic metallurgical sample buckets. At the laboratory, the wet samples of a given interval were mixed and thoroughly dried together in drying sheds. The laboratory handled the final mixing and splitting of the dried sample.

During the drilling of Lacana's second phase, in which 99 reverse circulation holes were drilled, the same sampling procedures were followed, except that only two sample splits were collected instead of three; the metallurgical sample split was omitted. MDA has no specific details about the sampling in Lacana's third phase of 57 reverse circulation holes.

Samples from the 48 reverse circulation holes drilled by Lacana in their first phase of drilling and the 99 reverse circulation holes drilled in their second phase underwent sample preparation at the assay laboratory and then were analyzed by one-assay-ton fire assay. Sample preparation consisted of crushing the entire sample to 85 percent -10 M. A 7-ounce to 12-ounce split was taken and pulverized to 80 percent -200 M. The pulverized samples were rolled 30 times before selecting a 1.03 ounce split for fusion. The pulverizer was routinely cleaned with compressed air, and every fourth sample pulverized was barren silica sand. MDA has no information on sample preparation during Lacana's third phase of drilling.

The only information available to MDA on the laboratories used by Lacana for analysis is in the drill-hole database. For many holes, no laboratory is listed. Those holes from the first phase of drilling that do show a laboratory indicate that Monitor Geochemical Laboratory, Inc. ("Monitor") of Elko, Nevada was the laboratory used. For the second phase of drilling, some holes list Shasta Analytical Geochemistry Laboratory ("Shasta"); others list both Shasta and Monitor; and later holes list Shasta and Hunter Mining Laboratory, Inc. ("Hunter"). For the third phase of drilling, Shasta, Legend Metallurgical Laboratory Inc. ("Legend"), or both are listed.

11.4 Santa Fe

According to Wittkopp *et al.* (1984) and Parratt *et al.* (1987), early sampling procedures used by Santa Fe allowed fine material to overflow the sample bucket when heavy water flows were encountered at the upper contact of the mineralized breccia unit, which resulted in sufficient loss of sample to raise concerns. Assays of grab samples of the overflow indicated gold values up to five times those returned from assaying of the primary samples from the bucket. Two previously drilled holes were offset about 16 feet and re-drilled. During re-drilling, material from the cyclone was passed through a riffle splitter, taking a 50-50 split. On one split, the bucket was allowed to overflow, losing the fines. On the second split, the entire five-foot sample, including the water, was recovered in several buckets. Flocculent was added to each bucket and the water decanted over a 24-hour period. Assay comparisons of the 2 types of sampling procedures showed average increases of 8 percent to 19 percent in the gold values for intervals for which the fines were caught.

While not stated explicitly, it appears that Santa Fe did not allow overflow of the sample buckets following this exercise.

Assuming the results from the re-drilling exercise are generally applicable to the early Santa Fe holes, and perhaps some holes by other operators as well, some portion of the drill-hole database may have grades that understate the true grades drilled. This potential understatement of grade would result in the resource estimate being locally conservative adjacent to some of the historical drilling.

MDA has no information on sample preparation, analysis, or security used by Santa Fe during their drilling programs at Relief Canyon. Section 10.2.4 discusses groundwater issues with respect to Santa Fe's drilling.

11.5 Pegasus

As part of a review of Firstgold's resource model, Independent Mining Consultants, Inc. ("IMC") found that the drill samples for the 5 holes drilled by Pegasus in 1988 were analyzed only by AA methods, which could understate the gold contents yielded from fire assaying (IMC, 1997). The five reverse circulation holes are shallow holes drilled in the North area and are included within the mineral domains modeled by MDA. The effect of these holes on the resource estimate is considered minimal due to the close proximity of Pershing Gold drill holes.

MDA has no further information regarding the sample preparation, analysis, and security procedures implemented during the Pegasus drilling programs.

11.6 Firstgold

With very few exceptions, all Firstgold reverse circulation samples were collected at 5-foot intervals. Review of drill-hole logs indicates that all 1996 and 1997 reverse circulation holes were drilled dry until the water table was encountered, while at least some, and perhaps all, of the later reverse circulation Firstgold holes were drilled similarly.

The following description of Firstgold's sampling methods during the 1996 and 1997 reverse circulation drill program is taken from Ball (1997). Each 5-foot sample to be sent for laboratory analysis was collected by the drill crew, while chips for logging and petrologic samples (samples of the cuttings that are sufficiently representative of the rock to be adequate for petrologic studies) were usually collected by Firstgold geologists. The primary samples were collected in 10 x 17-inch olefin sample bags, labeled by the geologists with the hole number and footage interval, after passing through a rotary wet splitter. The split sample was either directly captured into the sample bag or into a five-gallon bucket from one of the splitter discharge outlets. The splitter was regulated to emit just enough sample to fill the sample bags.

Chip samples for geological reference were collected from each 5-foot interval from a different sample discharge of the splitter than the primary samples for analysis and stored in plastic chip trays for later examination and lithologic logging.

An undated Firstgold protocol reviewed by MDA described the proposed sampling procedures for reverse circulation and core sampling of Firstgold projects for 2007 and beyond. MDA cannot verify to what extent this protocol was followed for the drilling at Relief Canyon. According to the Firstgold protocol, reverse circulation samples were to be collected with a rotating wet pie splitter attached to the drill rig. Samples were to be collected in 5-gallon buckets, with the pie splitter to be rinsed with water after each five-gallon sample was collected. A flocculent was to be added to the finished sample, and the entire sample was to be agitated. The water was to be decanted and the sample poured into an 11-inch by 17-inch cloth sample bag; the entire sample was to be placed in the bag. Samples were either to be air dried or dried in a drying oven at Firstgold's preparation laboratory facility. Once dried, the samples were to be crushed using a jaw crusher, and a 1,000-gram (2.2 pounds) split, made with a riffle splitter, was to be placed in a paper envelope for transport to ALS Chemex (now ALS; "ALS"). Rejects were to be stored at Firstgold's warehouse.

According to Firstgold's protocol, core samples were to be a maximum length of 5 feet unless conditions dictated an extension to no greater than 10 feet. Unless dictated otherwise by the situation, the smallest core sample length was to be 2.5 feet. Changes in the minimum or maximum lengths might be warranted because of alteration, poor recovery, or changes in lithology. Core samples were to be cut in half using a diamond-blade rock saw. The Firstgold preparation laboratory would then crush the samples using a jaw crusher and take a 1,000-gram (2.2 pounds) sample split using a riffle splitter. The sample splits would be put into paper sample envelopes. Samples would either be delivered to the laboratory by ALS staff or by Firstgold employees.

For the 1995-1997 drilling, the Firstgold reverse circulation samples were ordered sequentially on the ground at the drill site while each hole was being drilled. Personnel from the laboratory picked up the samples 2 to 3 days after the holes were drilled. Pick-up days, times, and laboratory personnel were recorded by Firstgold geologists (Ball, 1997). According to Mears (2007), Firstgold ran 32-element ICP (inductively coupled plasma) analyses on several of the reverse circulation drill holes from their 1995-1997 drilling program; 4 standards identified as "standard C2" were apparently included and reported in the results.

For their drilling program from 1995 through 1997, Firstgold used ALS and American Assay Laboratories ("American Assay") for analysis. Both are currently ISO-rated laboratories. According to Mears (2007), for samples submitted to ALS, "*Chemex picked up the lab bag samples at each drill site pad and transported them directly to its sample preparation facility in Sparks, Nevada, using chain-of-custody identification and tracking procedures. Chemex prepared the samples for assay and geochemical analysis. If the samples were wet, they were dried in low temperature ovens. Then, depending on the type of analysis requested, the samples were split, sieved, crushed, and pulverized. Finally, Chemex shipped the pulps to its laboratory in Vancouver, British Columbia for final chemical analysis, maintaining custody of the samples the entire time.*" For samples submitted to American Assay, the same procedures were used except that the final analysis was performed in Sparks, Nevada (Mears, 2007).

Mears (2007) reported that for Firstgold's 1995 to 1997 programs, samples were protected from contamination or disturbance from third parties by storage away from other activity at the drill site. Only drillers or laboratory pick-up personnel handled the sample bags. Exploration personnel were present seven days a week, and at night the access gate was locked.

According to Mears (2007), for its subsequent drilling Firstgold used protocols for handling, bagging, transportation, security, preparation and analysis as defined in an internal memorandum by B. Ball in 1997. MDA has reviewed that report as well as a copy of an undated Firstgold protocol for sampling and QA/QC that was developed for their 2007 and subsequent drilling. MDA cannot verify that the procedures in the undated protocol were followed in practice.

According to the Firstgold protocols, the following QA/QC program was developed:

1. Duplicate samples of reverse circulation chips were collected after initial assay results were received. Duplicates for core were taken from coarse rejects. Duplicates were to be submitted at a rate of one for every 40 samples submitted, with a second duplicate to be sent to a second laboratory at a rate of one for every 80 samples.
2. Commercial standards of medium and low-grade oxide gold from Rocklabs of New Zealand were to be inserted into the sample stream prior to assaying at the rate of one standard for every 20 samples. Any standard that differed from the expected results by +/- 10 percent was to be reported to the assay laboratory, which was to run their own checks on the particular batch of samples containing the anomalous results. If the laboratory reported problems with the batch, they would perform re-runs at their cost on all of the samples contained within that batch.
3. One blank composed of cuttings with known assays of less than 0.005ppm will be submitted for every 100 samples.
4. Internal QA/QC procedures from ALS and American Assay will also be used.

For its 2007 and 2008 drilling, Firstgold continued to use ALS and American Assay for sample preparation and assaying (Mears, 2007), with ALS as the primary laboratory (Firstgold, 2007d, 2007e). All assays for their 2007 drilling were fire assays with an atomic absorption ("AA") finish on either a 30 gram or 50-gram charge (Firstgold, 2007d). Pulps and coarse rejects were returned to Firstgold from American Assay on a quarterly basis and were stored at Relief Canyon (Mears, 2007). Firstgold reports that in the pit areas, samples from reverse circulation hole RCM08-19 and core hole RC-D1 were assayed by Firstgold's in-house laboratory, but most of the intervals were then assayed by ALS to check the values. All of the assaying of the North Target holes was done in Firstgold's in-house laboratory, but these holes lie outside of the mineral resource modeling. No assay data from the Firstgold in-house laboratory were used in grade interpolation related to the resource estimation.

The sampling and QA/QC protocols for Firstgold stipulated that no samples would be collected or handled by officers or directors of the company or any associate of the issuer prior to the reporting of final analytical results. Samples were to be picked up on site by ALS personnel and delivered to the ALS preparation laboratory in Winnemucca or Elko, Nevada. Following preparation, the pulps were trucked to Reno by ALS staff for assaying or shipment to their Vancouver assay laboratory.

Mears (2007) reported that most of the stored reverse circulation cuttings from the various drilling programs at Relief Canyon were destroyed or lost.

11.7 Pershing Gold

The following information is taken from Evans and Altman (2013) with additional information provided by Pershing Gold. MDA has verified much of the core sampling and assaying procedures during four site visits.

Drill core was boxed and sealed at the drill rig, then moved to the Relief Canyon logging and sample preparation facilities in Lovelock, Nevada by trained personnel. Pershing Gold sampled core in variable lengths up to a maximum of about 5 feet. Sample lengths were based on lithology or the presence of mineralization. Generally, the entire length of the hole was sampled, although in some instances, sampling was limited to known gold-bearing lithologies. Core was split down the center using a table-fed circular rock saw. One-half of the core was sent for assay, while the remaining half was returned to the core box and stored at Relief Canyon in a secure, fenced-off area. Reverse circulation chips were split at the drill rig with approximately 3 to 6 kg of cuttings saved for assay from each 5-foot sample (Pershing Gold news release, December 4, 2012). Core density measurements were completed in-house using the wax-coated water-immersion method.

Core and reverse circulation samples were stored in a fenced area until picked up by the laboratory (Pershing Gold news release, December 4, 2012).

For the five reverse circulation holes completed in 2011, and the 35 core holes completed in 2011 and 2012, ALS was the principal laboratory. For the remaining 84 reverse circulation holes completed in 2012, and 26 of the core holes completed in 2013, Inspectorate America Corp. ("Inspectorate", now part of Bureau Veritas) in Sparks, Nevada, was used for assaying. The five metallurgical core holes in 2013 were assayed at McClelland Laboratories ("McClelland") in Sparks, Nevada while the remaining 2013 core holes were sent for analyses to Skyline Assayers and Laboratories ("Skyline") in Sparks, Nevada. Both Skyline and Inspectorate were used for the 2014 and 2015 core drill program while McClelland was used for the 2016 drill program. Sample preparation was performed by the assay laboratories. ALS analyzed for gold by fire assay on a 30g sample with an atomic absorption finish (their code Au-AA23). Inspectorate performed one-assay-ton fire assays with an atomic absorption finish (their code AU-1AT-AA). Skyline analyzed for gold by fire assay on a 30g sample with an atomic absorption finish (their code FA-1,) while over-limits (>3ppm) were analyzed by gravimetric assay (code FA-2). McClelland analyzed for gold by fire assay with an atomic absorption finish (their code FA-AAS-30-Au).

In addition to the gold analyses, silver analyses on the 2016 drill samples were completed by McClelland by fire assay with an atomic absorption finish (their code FA-AAS-30-Ag). Pershing also sent pulp samples from the 2014 and 2015 drill programs to Inspectorate for silver analyses by aqua regia digestion and atomic absorption analyses.

For the 2011 and 2012 Quality Assurance/Quality Control ("QA/QC") programs, 12 different certified standards were inserted into the sample stream at a rate of one standard for every 25 samples. The certified standards were purchased from CDN Laboratories of British Columbia and an independent sample preparation laboratory in Reno, Shea Clark Smith MEG ("MEG"). A total of 163 blanks were inserted into the drill samples at a rate of one blank in 75 samples (slightly more than one percent of the total number of samples). Blank samples consisted of landscape rock (scoria) purchased from a local hardware store. In addition, the primary sample stream also included replicate analyses of 455 pulp within the original sample stream at a rate of one in 25 samples, representing 3.6 percent of the total number of samples analyzed. Field duplicates were also used at a rate of one in 200 samples. For core samples, the field duplicate consisted of a quarter cut of the remaining core; for reverse circulation samples, the field duplicate consisted of all the cuttings from the reject pipe of the revolving splitter. A total of 84 second-laboratory check assays were performed by Inspectorate on pulps used for the original ALS assays.

The 2013 drill program followed the same QA/QC protocol for blanks, standards, and replicate pulps as in 2011-2012. A total of 77 blanks and 266 standards were submitted to the primary laboratory (Inspectorate or Skyline). Blank samples consisted of landscape rock (scoria), while the certified standards were purchased from CDN Laboratories. The primary sample stream also included replicate analyses of 68 pulps in the original laboratory batches. The 2013 QA/QC program did not include any coarse reject pulp duplicate or field duplicate samples nor were any check samples sent to a second umpire laboratory.

The 2014-2015 drill program followed the same QA/QC protocol for blanks and standards as in 2011, 2012 and 2013. A total of 568 blanks and 1,599 standards were inserted into the primary sample stream submitted to the primary laboratories (Inspectorate, Skyline and to a more limited extent ALS). The 2014-2015 QA/QC program also included 134 quarter-core field duplicates, 593 same-laboratory replicate pulps, 153 second laboratory duplicate pulp analyses, and 68 second laboratory check assays on original pulps ("replicate pulp analyses").

The QA/QC program for the 2016 drilling (drill holes RC16-458 thru RC16-475) used 42 blanks and 101 standards inserted into the sample stream sent to McClelland. Another 13 blanks and 29 standards were inserted into the sample stream of pulps sent to Inspectorate for silver analyses. See Section 11.12 through 11.16 for a discussion of the Pershing Gold QA/QC procedures and results.

11.8 Reverse-Circulation Sample Contamination

Due to the nature of reverse circulation drilling, the possibility of contamination of drill cuttings from intervals higher in the hole is a concern, especially when groundwater is encountered or fluids are added during drilling. A number of holes at Relief Canyon intersected groundwater. Wittkopp *et al.* (1984) report that *"most [Santa Fe] drill holes hit a heavy flow of water at the point where they hit the [mineralized] breccia units."* In addition, based on comments recorded in drill logs, MDA was able to document that a significant number of holes in the database encountered water while drilling.

Evidence for down-hole contamination can be documented in some of the historical reverse circulation holes drilled below the water table in the North area. When compared with nearby core holes, the reverse circulation drill holes have long continuous runs of mineralization, while mineralization within the adjacent core holes occurs within discrete horizons. Within the core holes, the contact between the mineralized interval and the weakly anomalous wallrock is typically very sharp, while the reverse circulation holes show continuous mineralization below the first significant gold intercept.

In recognition of the strong evidence of down-hole contamination in at least some reverse circulation holes, the mineral-domain modeling and the gold assays used in the resource estimation described in Section 14.0, have excluded the suspect mineralized reverse circulation samples. A total of 2,914 sample intervals from all or part of 31 reverse circulation holes have been removed from use in the geologic model and resource estimate. Most of these removed intervals have been "replaced" in the drill database by nearby core holes recently drilled by Pershing Gold.

11.9 Historical Density Data

There are very limited historical rock density data available to MDA. Atiyeh (1986) reports that Pegasus used tonnage factors of 15.25 ft^3/ton for mineralized and unmineralized breccia and 12.39 ft^3/ton for unmineralized Grass Valley Formation in a 1986 estimation. Firstgold used tonnage factors of 15 (Fernette *et al.*, 1996) and 18 (Drossulis, undated) for internal estimations.

The only documented density measurements that pre-date work by Pershing Gold, and that are known to MDA, are reported by Hopkins (1985), who summarized tests completed by Lacana's Engineering Department of the Relief Canyon mine. A blast-hole air-track rig was used to drill 50 four-inch-diameter holes to depths of 5.5 to 14 feet. Cuttings from each hole were "meticulously" collected to prevent loss; the wet and dry weights of the cuttings were determined; the hole depth was measured; and the hole was immediately filled by weighed amounts of screened flux sand of known density (to determine hole volume). Using these data, the density of the material was determined for each of 48 holes (two holes were discarded from the study; Table 11.1).

Table 11.1 Lacana Density Study

Rock Type	Tonnage Factor (ft^3/ton)	Specific Gravity	No. of Holes
Breccia	15.19	2.11	40
Limestone	12.99	2.47	5
Shale	14.29	2.24	3

Mears (2007) summarized a density study by Pegasus in 1987; MDA does not have further documentation of this study, and Mears states that little is known of the methods used in the density determinations. The weighted average (using "percent tonnage") of the Pegasus breccia tonnage factors is 14.3 (Table 11.2).

Table 11.2 Pegasus Density Study
(Walker *et al*., 1987 as reported in Mears, 2007)

Item	Grass Valley Formation	Dike	Limestone	High-Clay Limestone Breccia	Limestone Breccia	Siliceous Breccia
Tonnage Factor (ft^3/ton)	13.25	11.30	12.27	14.60	13.70	14.93
Specific Gravity	2.42	2.83	2.61	2.19	2.34	2.15
Percent Tonnage	15	9	8	31	25	12
Number of Samples	23	14	12	59	42	24
oz Au/ton	0.011	0.013	0.011	0.017	0.011	0.012

Using the above historical data, the tonnage factors MDA chose to use in the 2010 resource model are shown in Table 11.3. These historical data were not used for the current resource model as discussed in Section 14.5.

Table 11.3 Tonnage Factors Applied to MDA's 2010 Resource Model

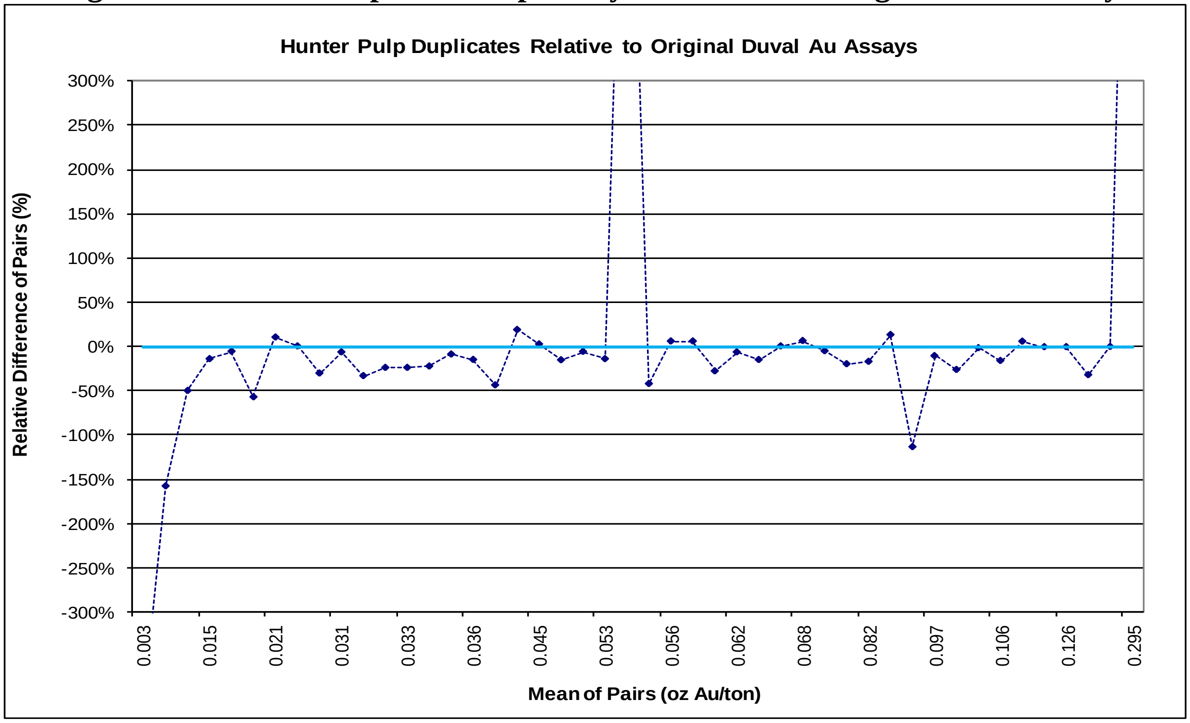
Unit	Tonnage Factor (ft^3/ton)	Specific Gravity
Mine Dump	20	1.60
Alluvium	18	1.78
Grass Valley Formation	14	2.29
Mineralization (Breccia)	15	2.14
Natchez Pass Formation	13	2.46

11.10 Lacana QA/QC Data

The following information is derived from Fiannaca (1982) and Fiannaca and Easdon (1984). Upon optioning the Relief Canyon property from Duval, Lacana undertook a program in 1982 to verify the Duval assay database. Lacana submitted 46 coarse-reject splits from original Duval drill samples to Hunter Laboratories, who prepared duplicate pulps from the rejects and analyzed the new pulps by one-assay-ton fire assays; the Duval drill samples were originally analyzed by fire assaying of ½-assay-ton charges.

MDA compiled the Lacana duplicate-assay data from tables and a copy of an original Monitor assay certificate provided by Fiannaca (1982). The Hunter duplicate-pulp analyses are compared to the original Duval assays in Figure 11.1 which is a graph showing the relative difference, plotted on the y-axis, between each original assay and the pulp-duplicate assay. The x-axis plots the means of the paired data, with each pair consisting of an original assay and the pulp-duplicate assay.

Figure 11.1 Hunter Duplicate-Pulp Analyses Relative to Original Duval Assays

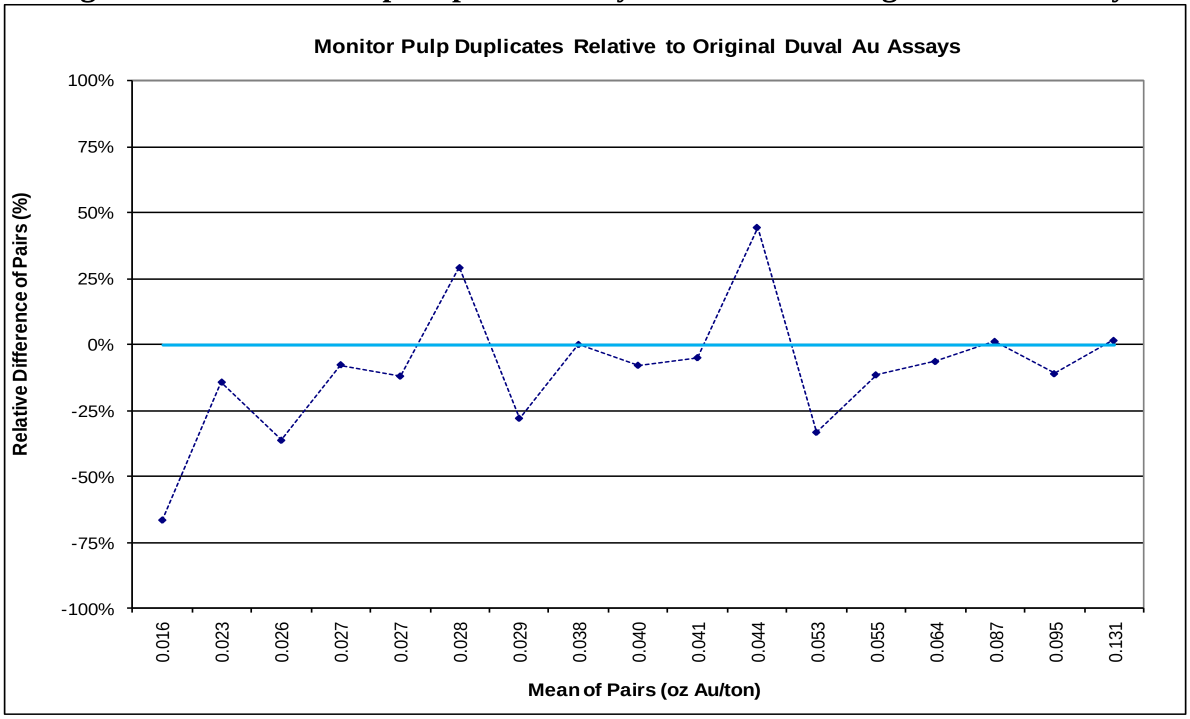


Although the mean of the Lacana duplicate pulps assayed at Hunter (0.068 oz Au/ton) is 9 percent higher than the mean of the original Duval analyses (0.062 oz Au/ton), the graph shows that the Hunter duplicate assays are actually systematically lower than the Duval assays (the difference in the mean lowers to -8 percent if the highest-grade pair is removed).

As an additional check, Lacana submitted, "*selected sample intervals of the pulps which Hunter had earlier analyzed*" to Monitor. While this explanation does not clarify the type of sample Monitor analyzed, handwritten notes on the original Monitor assay certificate suggest that Monitor prepared a new pulp from the coarse rejects. These samples were selected as a subset of 76 Duval sample intervals analyzed by cyanide leach by Hunter. Monitor analyzed these samples by both one-assay-ton fire assay and hot cyanide extraction-AA finish. MDA compiled the one-assay-ton fire-assay check data and compared it to the original ½-assay-ton fire assays of Duval (Figure 11.2).

The mean of the Monitor analyses of the Duval pulp duplicates (0.047 oz Au/ton) is 5 percent lower than the original Duval results, which is supported by the low bias evident in the relative difference graph.

Figure 11.2 Monitor Pulp-Duplicate Analyses Relative to Original Duval Assays



Lacana collected field duplicate samples at the rig during the drilling of their 3-phase, 203-hole drilling program. These field duplicates consisted of secondary splits of the drill cuttings when the drilling was dry; no field duplicates were collected when the drilling was wet. While MDA does not have the field-duplicate data, Fiannaca and Easdon (1984) report that comparisons of the field duplicates to the primary assay samples for each of the 3 drill phases yield differences in the means of -1.4 percent to +7.3 percent for all 1,463 field-duplicate/primary sample pairs and -3.8 percent to +10.4 percent for sample pairs with primary assays >0.010 oz Au/ton (608 pairs); it is not clear if the stated percentages are based on the field duplicates relative to the primary samples or *vice versa*.

Lacana also inserted one standard for every 9 primary samples, at least for the first phase of drilling (Fiannaca and Easdon, 1984); MDA does not have the results of the standard analyses.

11.11 Firstgold QA/QC Data

Other than check assaying of petrological samples from hole DH-9723 by a third-party, MDA has no QA/QC data from the Firstgold 1996-1997 drilling programs. The petrological samples were washed of fines, represent only a small, hand-selected portion of the sample interval, and therefore are not useful for the purposes of a QA/QC analysis.

Firstgold instituted a formal QA/QC program in 2007, described in an undated and anonymous QA/QC protocol document, that included the insertion of analytical standards and blanks into the primary sample stream, as well as the analysis of field-duplicate samples that were submitted after the primary sample assays were received. MDA compiled and analyzed the resultant QA/QC data provided by Firstgold.

11.11.1 Firstgold Analytical Standards

Standards are used to evaluate the analytical accuracy and precision of the assay laboratory during the time the primary drill samples were analyzed.

MDA was provided with 287 standard results from the 2007 and 2008 drilling programs. Samples from 18 of the 109 holes drilled do not have accompanying standard analyses; 13 of the 18 holes without standards contribute samples to the resource estimation. The insertion ratio implied by the number of standards and the total gold analyses for these holes in the database is one standard for every 17 drill samples. The drill samples and associated standards were analyzed by ALS.

MDA does not have any documentation of the 13 standards used by Firstgold, nor was the name (the standards are simply numbered 1 through 13) or source of the standards provided, although the QA/QC protocol document states that standards were to be acquired from Rocklabs of New Zealand. The expected values were provided by Firstgold without standard deviation data.

The standard results are summarized in Table 11.4 (samples with insufficient material for analysis and two results that were likely to actually be from blank samples were excluded). The means of the standard analyses are quite close to the expected values for most of the standards, although they tend to be lower than the expected values of the standards in 2007. A low bias in the ALS analyses might be present in both 2007 and 2008, however, as evidenced by the higher number of analyses below the expected value than above. A more comprehensive review of the data would require full documentation of the standards.

Table 11.4 Summary of Results of Firstgold Analytical Standards

Drill Hole Series	Standard	Expected Value	No. of Analyses	Mean	Percent Diff.	Min.	Max.	No. Below Exp. Value	No. Above Exp. Value
2007 holes	1	0.012	26	0.012	0.0%	0.011	0.013	7	2
2007 holes	2	0.030	28	0.029	-3.3%	0.025	0.033	18	6
2007 holes	3	0.054	27	0.052	-3.7%	0.046	0.058	19	5
2007 holes	4	0.012	14	0.011	-8.3%	0.009	0.013	5	3
2007 holes	5	0.030	17	0.029	-3.3%	0.024	0.032	9	5
2007 holes	6	0.054	12	0.051	-5.6%	0.045	0.056	10	1
2008 holes + NT07-01	7	0.012	24	0.011	-8.3%	0.009	0.013	11	4
2008 holes + NT07-01	8	0.030	26	0.030	0.0%	0.027	0.032	7	8
2008 holes	9	0.053	10	0.054	1.9%	0.049	0.062	7	3
2008 holes + NT07-01	10	0.012	29	0.012	0.0%	0.009	0.013	13	3
2008 holes + NT07-01	11	0.030	21	0.030	0.0%	0.027	0.032	8	5
2008 holes + NT07-01	12	0.053	13	0.052	-1.9%	0.049	0.055	8	3
2008 holes	13	0.012	6	0.012	0.0%	0.011	0.013	1	1

11.11.2 Firstgold Check Assays

MDA has 9 American Assay check assays of original ALS pulps, with one check from each of 9 holes within the sequence RCM07-01 to 19. The American Assay and ALS means are identical (0.032 oz Au/ton).

11.11.3 Firstgold Preparation Blanks

The Firstgold QA/QC protocol document states that blank material was to be prepared from approximately 100 kilograms (220 pounds) of drill cuttings with assay results of less than 0.005 g Au/t (0.0001 oz Au/ton). The samples were to be thoroughly mixed and split into one kilogram (2.2 pounds) packets for insertion into the sample stream.

Preparation blanks are used to test for cross contamination between drill samples in the analytical laboratory, which is most common during sample-preparation stages. In order for the blanks to accomplish this, they must be sufficiently coarse to require the same crushing stages as the drill samples. It is also important for blanks to be placed in the sample stream immediately after mineralized samples (which would be the source of most cross-contamination issues). Blank results that are greater than 5 times the detection limit are typically considered failures that require further investigation and possible re-assay of associated drill samples.

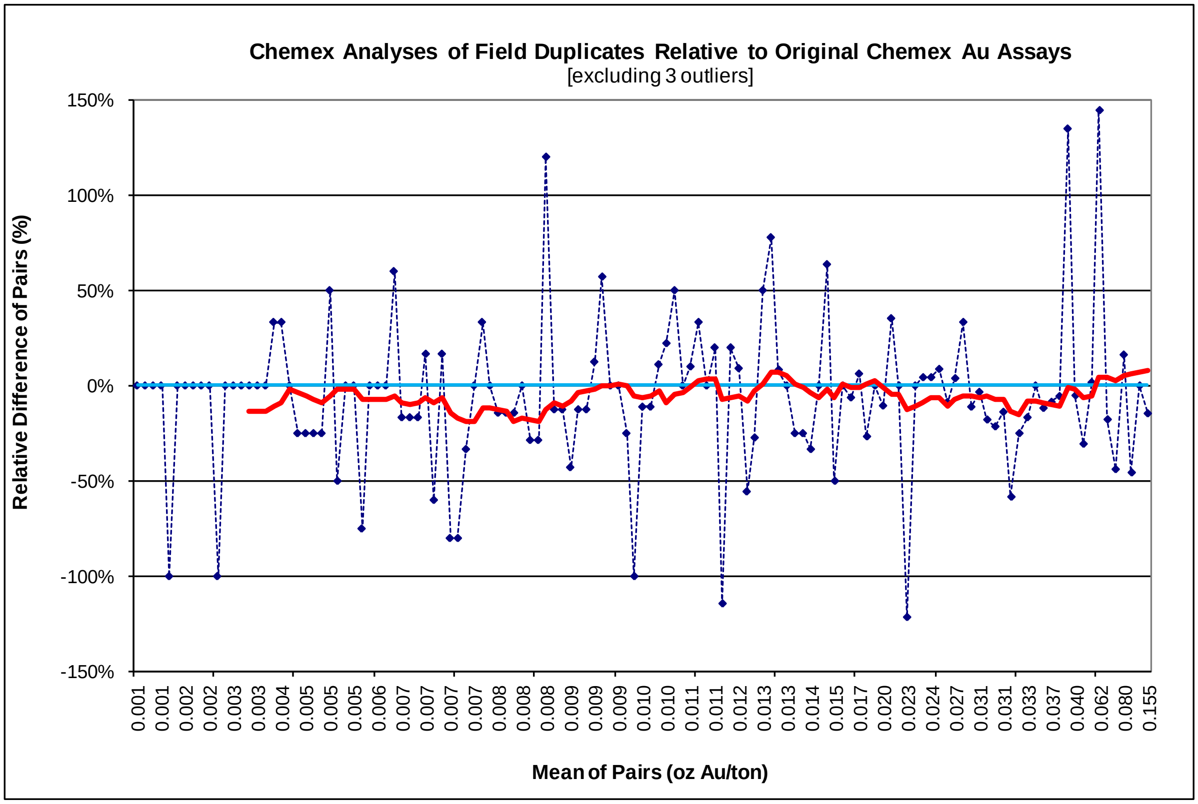
MDA has the results of 24 blanks inserted into the sample stream of 18 holes drilled in 2008; the gold values from the previous sample are not known. Twelve of the blanks have values less than the detection limit (<0.0001 oz Au/ton), 10 lie between 0.0001 and 0.0004, and 2 have anomalously high results (0.0062 and 0.013 oz Au/ton).

11.11.4 Firstgold Field Duplicates

The Firstgold reverse circulation field duplicates are secondary splits of the drill cuttings collected at the rig at the same time as the primary samples. Field duplicates are mainly used to assess inherent geologic variability and sampling variance.

MDA was provided with 131 field-duplicate analyses by ALS that can be compared to the original ALS analyses. The field duplicates are from 66 reverse circulation holes, including RCM07-01 through 75, RCM08-01 through 19, and NT07-01. The number of duplicates relative to the associated primary assays implies an analytical rate of one field duplicate for every 32 primary samples. The field-duplicate analyses are compared to the primary assays in Figure 11.3, with three outlier pairs excluded.

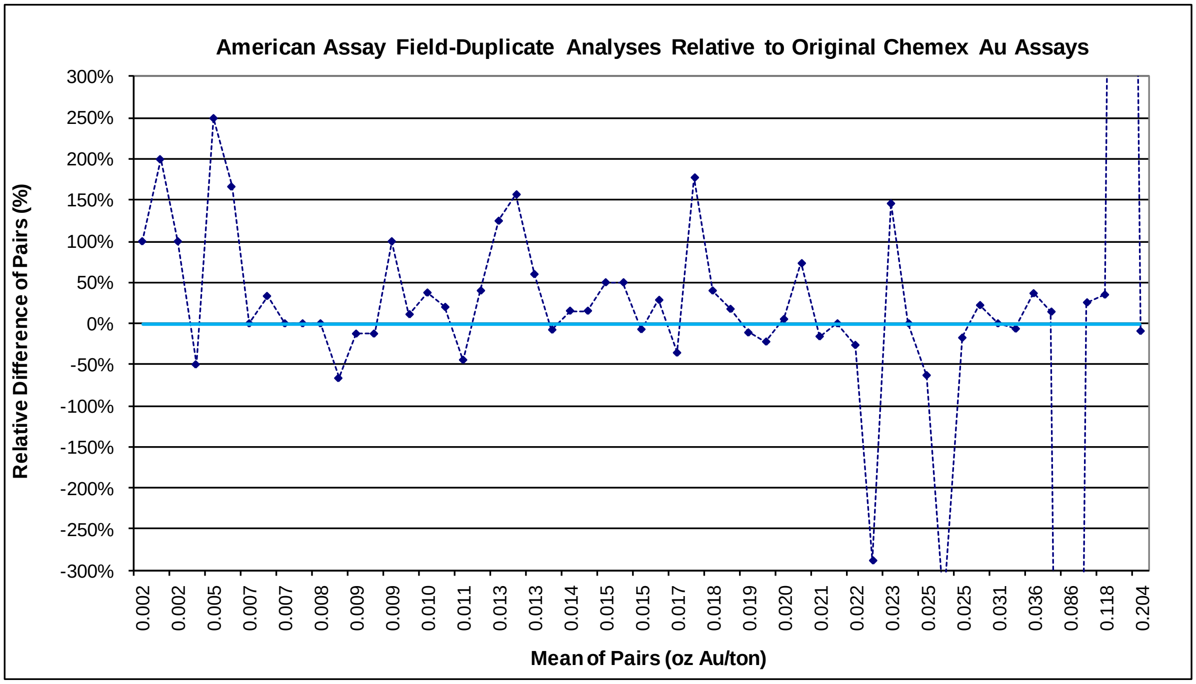
Figure 11.3 Firstgold ALS Field-Duplicate Analyses Relative to Original ALS Assays



A very slight low bias in the field-duplicate analyses relative to the original assays may be evident in the graph, but the means of the two sets of data are identical (0.017 oz Au/ton). Additional data are needed to prove the existence of a low bias. The absolute value of the relative differences between the sample pairs averages 25 percent.

There are an additional 57 field-duplicate samples analyzed by American Assay that can be compared to the original samples, which were assayed by ALS, although the differing laboratories lead to additional factors when analyzing the data statistically. These reverse circulation field duplicates are from 27 reverse circulation holes, including NT08-03 and 26 holes in the sequence of RCM07-20 through 75, with an implied analytical rate of one duplicate for every 33 primary samples. The field-duplicate results are compared to the original ALS assays in Figure 11.4.

Figure 11.4 Firstgold American Assay Field-Duplicate Analyses Relative to Original ALS Assays



The relative difference plot shows that the field duplicates tend to be higher grade than the original samples, and the mean of the duplicates (0.027 oz Au/ton) is higher than the original samples (0.024 oz Au/ton). The variability indicated by the relative differences in the paired data exceeds that seen in Figure 11.3 More data are required before definitive conclusions can be reached, however.

11.12 Pershing Gold 2011-2012 QA/QC Data

A detailed evaluation of Pershing Gold's 2011-2012 QA/QC program was provided in Evans and Altman (2013). MDA has reviewed these data and has summarized the results in the following sections. Refer to Evans and Altman (2013) for additional details.

11.12.1 Pershing Gold Preparation Blanks 2011-2012

A total of 163 blanks were inserted into the primary sample stream at a rate of one blank in 75 samples (slightly more than one percent of the total number of samples). Blank samples consisted of landscape rock (scoria) purchased from a local hardware store. No evidence of gold contamination was observed in any of the blank samples.

11.12.2 Pershing Gold Standards 2011-2012

Twelve different certified standards were inserted into the sample stream at a rate of one standard for every 25 samples. The certified standards were purchased from CDN Laboratories of British Columbia and an independent sample preparation laboratory in Reno, Shea Clark Smith MEG ("MEG"). Samples were considered failures if they assayed more than three standard deviations above or below the expected mean value. The average failure rate for these standards was about three percent. One of the standards had a high failure rate of 17 percent. Evans and Altman (2013) state that the sample batch was re-run whenever a standard was considered to have failed. This procedure has not been verified by MDA.

11.12.3 Pershing Gold Pulp Duplicates 2011-2012

A total of 455 pulp duplicate analyses were completed at a rate of one in 25 samples, representing 3.6 percent of the total number of samples analyzed. In general, the pulps produced good reproducibility with only a 2.2 percent difference between the sample pair means. The data set included 7 sample pairs having very high variability, which resulted in a high 36 percent average variability between sample pairs. MDA believes these outliers do not appear to be representative of the sample population as a whole and likely should have been excluded from the analyses.

11.12.4 Pershing Gold Field Duplicates 2011-2012

Thirty-two field duplicates were analyzed at a rate of one in 200 samples. For core samples, the field duplicate consisted of a quarter cut of the remaining core; for reverse circulation samples, the field duplicate consisted of all the cuttings from the reject pipe of the revolving splitter. The results indicated a less than one percent difference in the mean of the sample pairs and a six percent relative standard deviation between pairs.

11.12.5 Pershing Gold Check Analyses 2011-2012

A total of 84 check assays were analyzed by Inspectorate of ALS original assays. The check assay consisted of a duplicate analysis of the same pulp shipped from ALS to Inspectorate. The second laboratory mean sample value for the Inspectorate analyses is about three percent lower than the original ALS mean value. The correlation coefficient between data sets is 0.993 indicating very good reproducibility.

11.13 Pershing Gold 2013 QA/QC Data

The 2013 drill program followed the same QA/QC protocol for blanks and standards as in 2011-2012. A total of 77 blanks and 266 standards were inserted into the primary sample stream submitted to the primary laboratory (Inspectorate or Skyline). Also included in the primary sample stream were 68 pulp replicate analyses that consisted of a second analysis of the original pulp done in the same laboratory run as the original analyses. The 2013 QA/QC program did not include any pulp duplicate or field duplicate samples nor were any check samples sent to a second umpire laboratory.

11.13.1 Pershing Gold Preparation Blanks 2013

A total of 77 blanks were inserted into the primary sample stream at a rate of one blank in 75 samples. Blank samples consisted of landscape rock (scoria) purchased from a local hardware store. Elevated gold values were found in three samples though the anomalous values were still very low (<0.002 oz Au/ton). Any contamination at this grade is not considered material to the resource estimate.

11.13.2 Pershing Gold Standards 2013

Four different certified standards were inserted into the sample stream at a rate of one standard for every 25 samples. The certified standards were purchased from CDN Laboratories of British Columbia. Samples were considered failures if they assayed more than three standard deviations above or below the expected mean value. The standard results are summarized in Table 11.5. Three outlier samples were removed from the standard data set due to extreme differences with the expected value. MDA has no explanation for these extreme differences though it is likely due to clerical error.

Table 11.5 Summary of Results of Pershing Gold 2013 Analytical Standards

Standard	Lab	Expected Value	Std Dev	No. of Analyses	Mean	Percent Diff.	Min	Max.	Low Failures	High Failures	% Failures
CDN-GS-1K	Inspectorate	0.867	0.098	38	0.818	-5.7%	0.619	0.966	1	0	3%
CDN-GS-1K	Skyline	0.867	0.098	9	0.780	-10.1%	0.663	0.920	2	0	22%
CDN-GS-P3C	Inspectorate	0.263	0.02	46	0.265	0.6%	0.231	0.328	1	6	15%
CDN-GS-P3C	Skyline	0.263	0.02	17	0.249	-5.3%	0.233	0.275	1	0	6%
CDN-GS-P6	Inspectorate	0.626	0.074	63	0.627	0.1%	0.397	0.747	2	2	6%
CDN-GS-P6	Skyline	0.626	0.074	38	0.602	-3.8%	0.403	0.705	1	0	3%
CDN-GS-P7H	Inspectorate	0.799	0.05	37	0.796	-0.3%	0.709	0.972	1	4	14%
CDN-GS-P7H	Skyline	0.799	0.05	4	0.768	-3.9%	0.735	0.825	0	0	0%
All	Inspectorate			184					5	12	9%
All	Skyline			68					4	0	6%

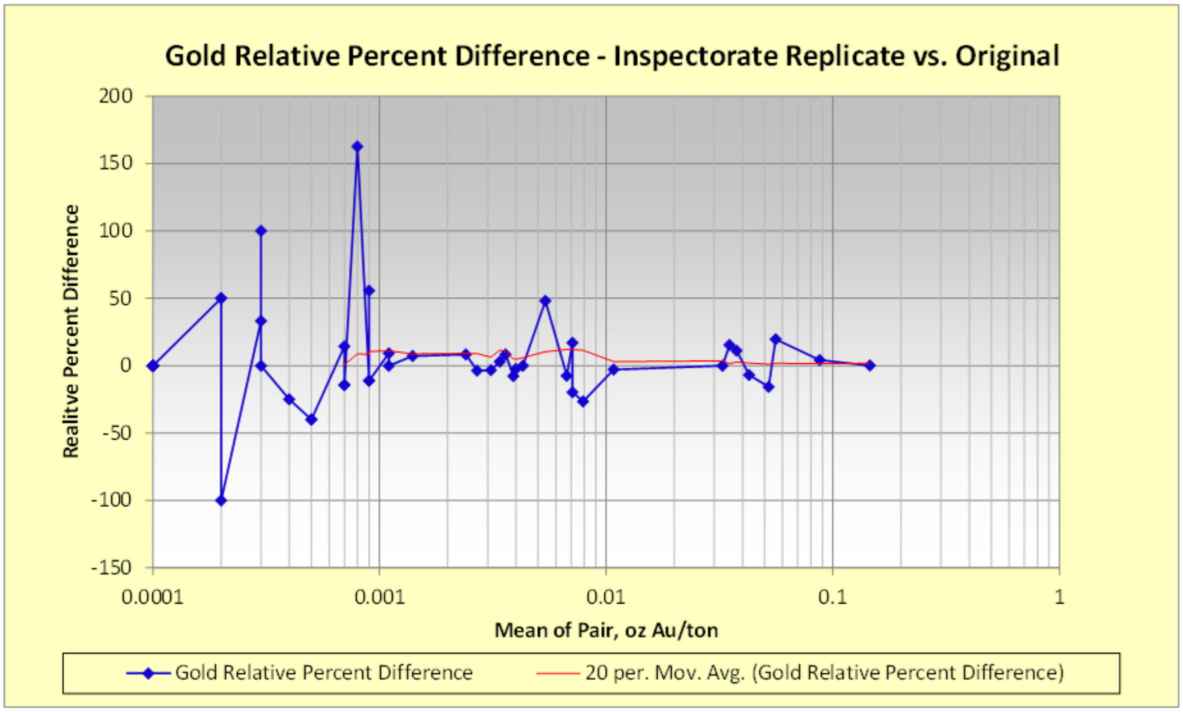
The average failure rate for all Inspectorate standards was nine percent, while the Skyline standards had an average failure rate of six percent. All of the high failures were within the Inspectorate analyses. Except for the first standard (CDN-GS-1K), where both laboratories are biased low versus the standard's expected value, Inspectorate's mean value is very similar to the expected values, while Skyline has a low bias of three percent to six percent below the expected values.

11.13.3 Pershing Gold Pulp Replicates 2013

Replicate analyses on a total of 68 pulp samples were completed by Inspectorate and Skyline. The replicate data are two analyses of the same pulp created in the original laboratory and reported on the same original laboratory certificate. Since the same original pulp is used in the second analysis, the replicate data provides information on sample variability only at the final laboratory analytical stage.

The Inspectorate replicate analyses are compared to the original assays in Figure 11.5. There is just a minor high bias (three percent) in the replicate mean grade as compared to the original analyses. Above a 0.001 oz Au/ton grade, the average relative difference between sample pairs also shows a minor three percent high bias in the replicate data. The average variability (absolute value of the relative difference between sample pairs) at gold grades above 0.001 oz Au/ton is 10 percent, which is an acceptable value for replicate pairs.

Figure 11.5 Inspectorate Replicate Analyses Relative to Original Inspectorate Assays



There are 17 Skyline replicate analyses, though only four sample pairs have mean gold grades greater than 0.001 oz Au/ton. One sample pair has high (90 percent) variability, but the data are too few to allow a meaningful evaluation.

11.14 Pershing Gold 2014 - 2015 QA/QC Data

The 2014-2015 drill program followed the same QA/QC protocol for blanks and standards as in 2011, 2012 and 2013. A total of 568 blanks and 1,599 standards were inserted into the primary sample stream submitted to the primary laboratories (Inspectorate, Skyline and to a more limited extent ALS). The 2014-2015 QA/QC program also included 134 quarter-core field duplicates, 593 same-laboratory replicate pulps, 153 second laboratory duplicate pulp analyses, and 68 second laboratory check assays on original pulps ("replicate pulp analyses").

MDA complete an initial review of the 2014 and early 2015 QA/QC program in the spring of 2015 in preparation for the July 2015 resource model update. This current analyses includes the 2014 through early 2015 data along with all remaining holes completed by Pershing in 2015 and which are included within the current resource model and estimate.

11.14.1 Pershing Gold Preparation Blanks 2014 - 2015

A total of 568 blanks were inserted into the primary sample stream at a rate of about one blank in 75 samples. Blank samples consisted of landscape rock (scoria) purchased from a local hardware store. Elevated gold values were found in just two samples assayed at Inspectorate though the anomalous values were still very low (<0.002 oz Au/ton). Any contamination at this low grade is not considered material to the resource estimate.

11.14.2 Pershing Gold Standards 2014 - 2015

Nine different certified standards were inserted into the sample stream at a rate of one standard for every 25 samples. The certified standards were purchased from CDN Laboratories of British Columbia. Samples were considered failures if they assayed more than three standard deviations above or below the expected mean value. The standard results for the gold analyses are summarized. Note that the gold values shown in Table 11.6 are in g Au/t. Four outlier samples were removed from the standard data set due to extreme differences with the expected value. MDA has no explanation for these extreme differences though it is likely due to clerical error.

Table 11.6 Summary of Results of Pershing Gold Analytical Standards
(data shown in g Au/t values)

Standard	Lab	Expected Mean Value	Expected Std Dev	No. of Analyses	Analyses Mean	Percent Diff.	Min	Max.	Low Failures	High Failures	% Failures
CDN-GS-P4B	Inspectorate	0.417	0.023	155	0.413	-1.0%	0.335	0.533	4	3	5%
CDN-GS-P4B	Skyline	0.417	0.023	212	0.431	3.4%	0.218	0.633	3	13	8%
CDN-GS-1K	Inspectorate	0.867	0.049	116	0.836	-3.6%	0.572	0.860	1	0	1%
CDN-GS-1K	Skyline	0.867	0.049	66	0.854	-1.5%	0.719	1.090	0	0	0%
CDN-GS-P3C	Skyline	0.263	0.01	3	0.254	-3.4%	0.238	0.269	0	0	0%
CDN-GS-P6	Inspectorate	0.626	0.037	119	0.627	0.2%	0.725	0.860	0	0	0%
CDN-GS-P6	Skyline	0.626	0.037	43	0.660	5.4%	0.605	0.687	0	0	0%
CDN-GS-P7H	Inspectorate	0.799	0.025	84	0.808	1.1%	0.725	0.907	1	1	2%
CDN-GS-P7H	Skyline	0.799	0.025	212	0.817	2.3%	0.664	0.943	3	7	5%
CDN-GS-1L	Inspectorate	1.16	0.05	9	1.170	0.9%	1.082	1.226	0	0	0%
CDN-GS-1L	Skyline	1.16	0.05	91	1.180	1.7%	1.035	1.328	1	1	2%
CDN-GS-P8C	Inspectorate	0.784	0.028	13	0.792	1.0%	0.74	0.822	0	0	0%
CDN-GS-P8C	Skyline	0.784	0.028	92	0.823	5.0%	0.607	0.896	2	0	2%
CDN-GS-P5C	Inspectorate	0.571	0.024	143	0.577	1.1%	0.384	0.826	2	1	2%
CDN-GS-P5C	Skyline	0.571	0.024	88	0.597	4.6%	0.417	0.818	1	5	7%
CDN-GS-P4C	Inspectorate	0.362	0.018	100	0.374	3.3%	0.297	0.600	0	5	5%
CDN-GS-P4C	Skyline	0.362	0.018	48	0.373	3.0%	0.33	0.539	0	1	2%
All	Inspectorate	0.884		504	0.881	0.1%			8	10	4%
All	Skyline	0.688		855	0.706	2.9%			10	27	4%

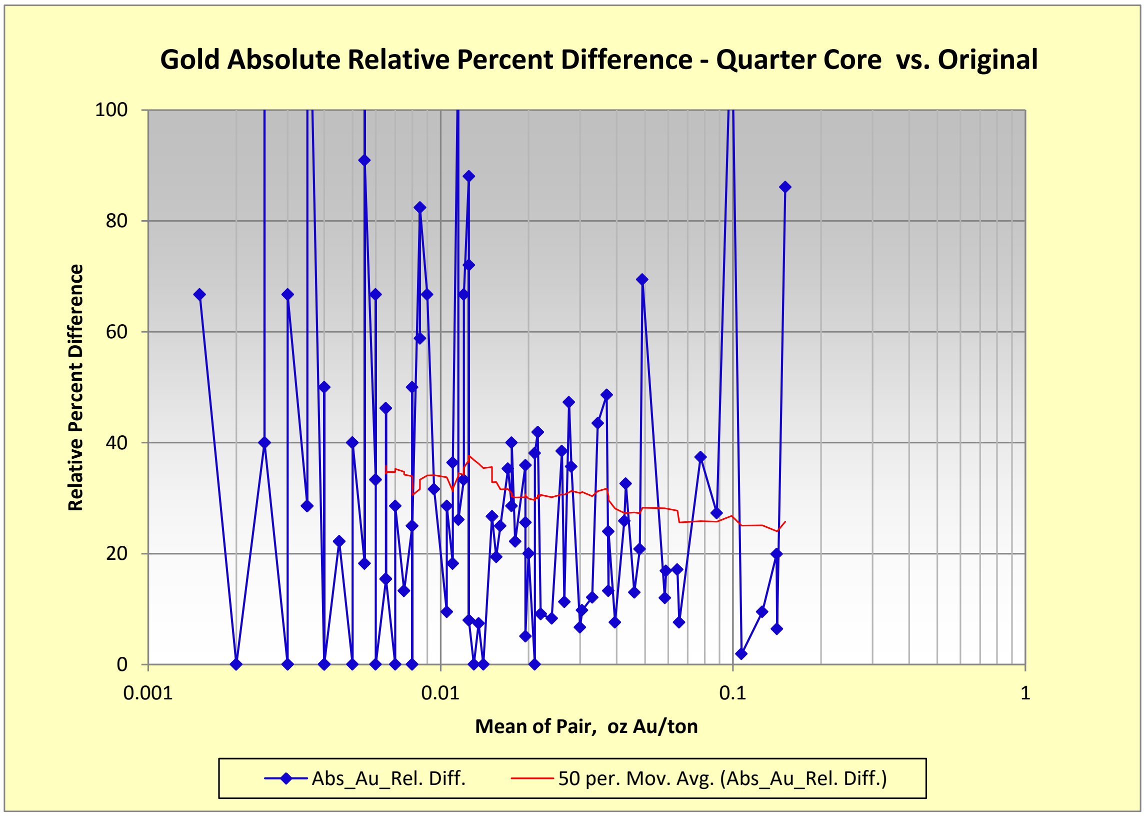
The average failure rate for both Inspectorate and Skyline standards was four percent. The Inspectorate failures were split relatively evenly between low and high failures while about 70 percent of the Skyline failures were high failures. The Inspectorate analyses showed no significant bias in any of the individual standards and the average difference between the total population of standards and Inspectorate analyses is less than one percent. There is no significant bias in the Skyline analyses for any of the individual standards though the Skyline analyses for the total standards population has a 2.9 percent positive bias. This bias is not considered significant but does introduce a low risk to the current resource estimate.

11.14.3 Pershing Gold Quarter-Core Field Duplicates 2014 - 2015

Duplicate analyses on a total of 134 quarter-core samples were completed by Skyline, the same lab that assayed the initial half-core sample.

The absolute value of the relative difference between the quarter-core sample pairs versus the sample pair mean grade is shown in Figure 11.6. The average variability at gold grades above 0.005 oz Au/ton is about 30 percent with variability decreasing on average with increasing grade. There is just a minor 4 percent high bias in the duplicate mean grade as compared to the original analyses. The quarter-core results provide the best indicators of total expected sample variability since the results include both the mineral variability inherent within the deposit and also the sample preparation and analytical variability.

Figure 11.6 Quarter-Core Duplicate Analyses– Absolute Relative Difference



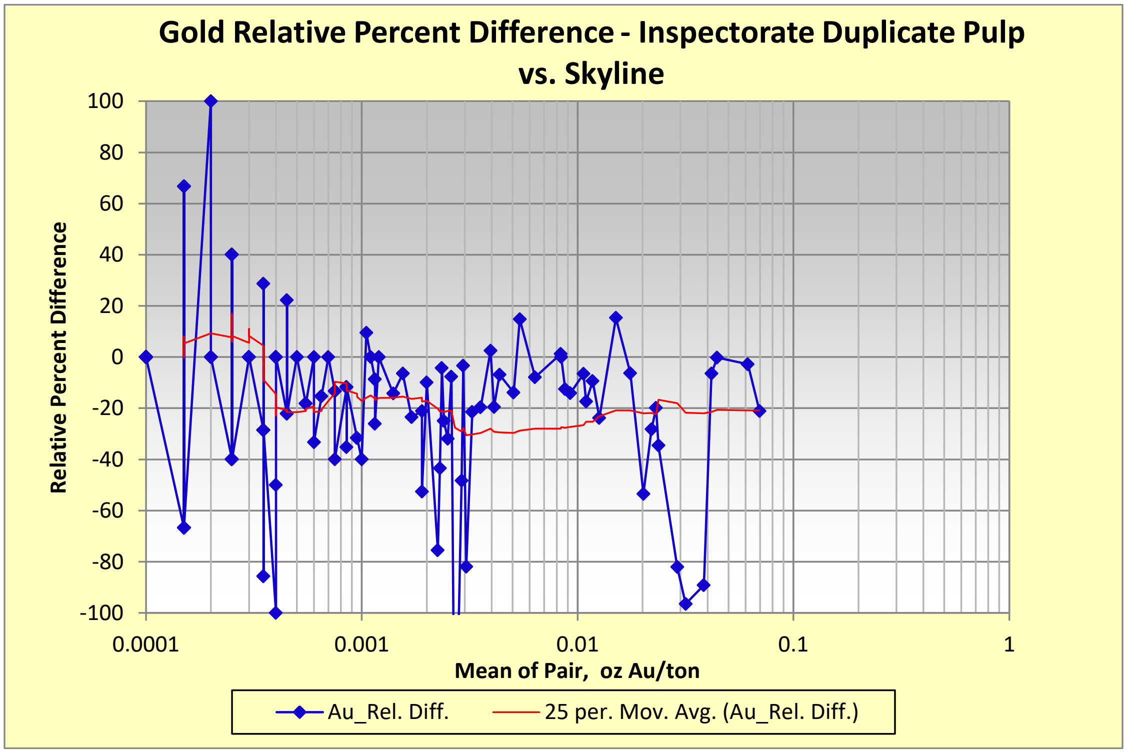
11.14.4 Pershing Gold Second Laboratory Duplicate Pulps 2015

The coarse rejects of 152 samples, all from drill-hole RC15-294, were shipped from Skyline to Inspectorate where duplicate pulps (preparation duplicates) were created and assayed for gold. Duplicate pulp analyses are usually run at the same original lab and provides an indication of expected assay variability due to sample preparation and analytical imprecision. There should be little to no bias between the duplicate pairs. For these 152 analyses, using a second lab likely adds to the observed variability and potentially inserts a bias into the results, due to the differences between labs, both in sample preparation and analytical techniques. For a discussion of lab variability, see Section 11.14.6.

The relative difference between the duplicate pulps versus the sample pair mean grade is shown in Figure 11.7. The Inspectorate analyses are consistently 20 percent lower in value than the Skyline analyses. Average total variability is 25 percent so most of the variability between the sample pairs is due to the significant sample bias between the laboratories. It is not known which laboratory is more accurate, only that there can be a material difference between analytical results from the two labs.

These duplicate pair results are based on data from one drill hole with all analytical results coming from the same original Skyline job and a single Inspectorate job. Therefore, these results are limited in time and it is uncertain whether these laboratory differences occur throughout the project life. Additional second lab testing is warranted.

Figure 11.7 Inspectorate Duplicate Pulp Analyses– Absolute Relative Difference

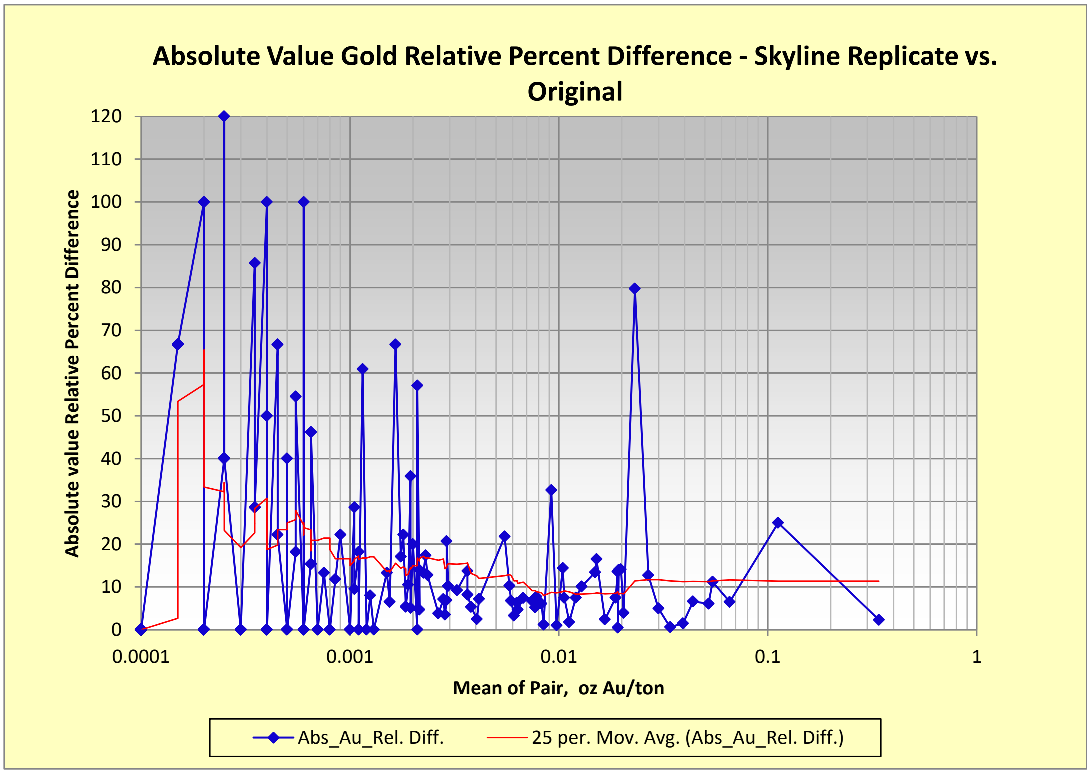


11.14.5 Pershing Gold Same-Laboratory Replicate Pulps 2014 – 2015

Replicate pulp analyses on a total of 593 samples were completed by Skyline and Inspectorate. The replicate data are two analyses of the same pulp created in the original laboratory and reported on the same original laboratory certificate. Since the same original pulp is used in the second analysis, the replicate data provides information on sample variability only at the final laboratory analytical stage.

The absolute value of the relative difference between the 304 Skyline replicate pulp analyses versus the sample pair mean grade is shown in Figure 11.8. The majority of replicate pairs are just weakly mineralized and no meaningful results can be taken from the data at such low gold grades. For the replicate pairs with mean gold grades above 0.005 oz Au/ton (a total of 38 replicate pairs), the average variability of the sample pairs is 10 percent, which is an acceptable value for replicate pairs. There is just a minor high bias (four percent) in the replicate mean grade as compared to the original analyses for samples at gold grades above 0.005oz Au/ton.

Figure 11.8 Skyline Replicate Pulp Analyses– Absolute Relative Difference



There are a total of 258 Inspectorate replicate pulps though only 21 have sample pair mean gold grades above 0.005oz Au/ton. Average variability for this small data set is 8 percent with just a one percent low bias in the replicate pulp versus the original sample.

11.14.6 Pershing Gold Check Assays 2014

A total of 68 original pulps were assayed at a second and third laboratory as a check on the original laboratory analyses. Inspectorate completed the check analyses if the original laboratory was Skyline while Skyline was the check laboratory for the Inspectorate original pulps. ALS Minerals (Reno, Nevada) was the third umpire laboratory for all 68 samples.

In this round-robin testing, the Inspectorate analyses were nine percent lower on average that the Skyline data for those sample pairs with mean grades greater than 0.005oz Au/ton. The ALS analyses were on average three percent higher than Inspectorate and six percent lower than Skyline. If these relationships occur within the larger project data set, the Skyline assay data, and the resource model estimate associated with the Skyline data, would show a potential nine percent increase in gold grade as compared to the Inspectorate assay data and associated block model estimate, solely from the change in laboratory alone. The risk to the over-all deposit resource is considered low because the drill holes with samples sent to the two laboratories are spatially mixed primarily within the North target area. Additional check analyses are warranted to confirm these initial results.

11.15 Pershing Gold 2016 QA/QC Data

The 2016 drill program completed up through September 2016 (drill hole RC16-475) followed the same QA/QC protocol for blanks and standards as in 2011 through 2015. A total of 42 blanks and 101 standards were inserted into the primary sample stream for the 18 drill holes submitted to the primary laboratory (McClelland). The 2016 QA/QC program also included 14 quarter-core field duplicates.

Pershing Gold also included QA/QC samples (13 blanks and 29 standards) in the batch of pulps sent to Inspectorate for silver analyses.

MDA completed an initial review of the 2016 QA/QC data in preparation for estimating the current, November 2016 resources, and no significant concerns were noted. Due to the limited data sets, no further analyses were conducted by MDA. It is expected that these data will be reviewed in greater detail after completion of additional project drilling in 2018.

11.16 Discussion of QA/QC Results

Based on their duplicate-pulp check assaying of original Duval results, Lacana concluded that their one-assay-ton fire assays were uniformly slightly lower than the original ½-assay-ton fire assays of Duval. While the two sets of Lacana duplicate-pulp data are indeed systematically lower-grade than the original Duval assays, the discrepancy could also be due to heterogeneities in the coarsely crushed samples, which could lead to sub-samples of the coarse material having varying gold contents. Another possibility is there were problems in the sub-sampling itself that caused the bias.

The ALS analyses of the Firstgold reverse circulation field duplicates show no significant issues with respect to sample bias caused by splitting at the drill rig or in the variability of the data. The remaining Firstgold QA/QC data are either insufficiently documented or too few to allow for meaningful conclusions.

The Pershing Gold QA/QC data show no significant issues with sample contamination and no evidence of analytical bias in the ALS, Inspectorate, or McClelland standard analyses. The Skyline standard analyses show more variable results with a three percent to six percent low bias in the 2013 data set, but a three percent high bias in the 2014-2015 data set. The 2011-2012 field and pulp duplicates, and the second-laboratory check analyses, show no material sampling or analytical bias, and there is a good correlation between sample pairs. The 2013 pulp replicate data are limited, though no material issues were noted. The 2014-2015 quarter core and pulp replicate data show no material biases or higher than expected variability. The limited data from the 2014 second laboratory check analyses and 2015 second laboratory duplicate pulp analyses did indicate a high bias in the Skyline data as compared to the Inspectorate and ALS data. This bias is not considered significant but does introduce a low risk to the current resource estimate.

While the QA/QC data are limited for much of the historical drilling, no significant issues were noted that would prohibit the use of these data in the current resource estimate. The uncertainty with these data though has contributed to a reduction in classification for those portions of the resource which rely solely on these data. No material issues were noted with the Pershing Gold data, and these data are suitable for use in the resource estimation.

11.17 Comparison of Drill Programs

As Firstgold's 2007 and 2008 drilling programs were the best documented prior to Pershing Gold's drilling, used up-to-date reverse circulation drilling equipment, and implemented the most QA/QC protocols to that time, statistical comparisons were completed by MDA that compared the results from Firstgold's and Pershing Gold's programs to those of the earlier companies.

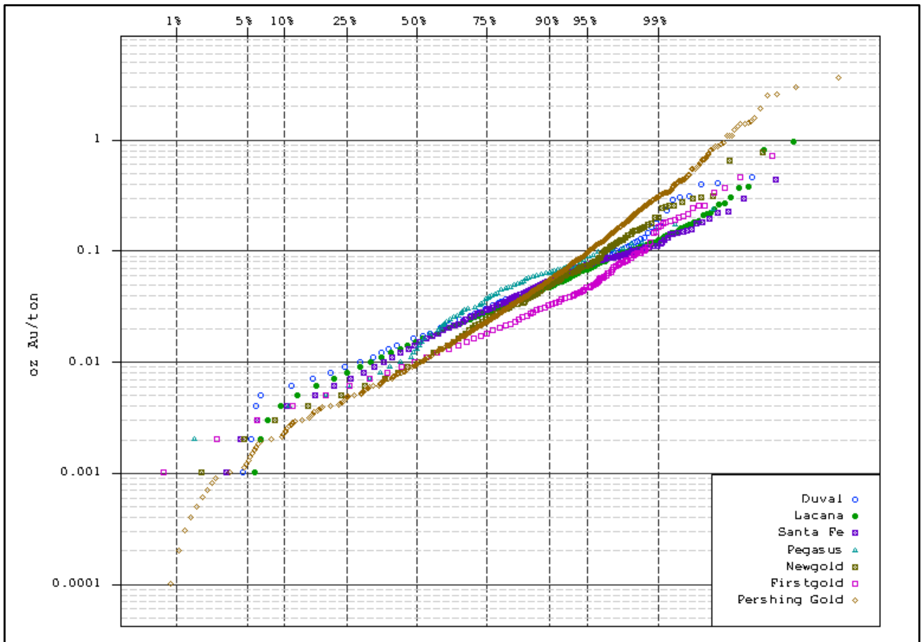
Figure 11.9 is a quantile-quantile plot that compares the assay populations of the various operators at Relief Canyon. The sample data is standardized to include only those samples used in the grade estimate.

The graph shows the early historical drilling (thru Pegasus) has higher grades within the population of samples within the lower grade ranges (up to about 0.05 oz Au/ton), while the newer drilling post-Pegasus has generally lower grades. This difference reflects the pre- versus post-mining drilling in the Main zone. The Firstgold holes were drilled on post-mining topography, so that many holes are collared in the footwall of significant mineralization and therefore drilled proportionally less mineralized material.

In comparing individual drill campaigns, historical land constraints prohibit meaningful comparisons involving Santa Fe holes (Figure 10.1), and Pegasus had too few holes for meaningful comparisons.

The most useful comparisons can be made using Lacana and Duval holes. These two populations are spatially similar, and their mean and median gold values compare well (0.023 and 0.015 oz Au/ton for Lacana *vs.* 0.026 and 0.016 oz Au/ton for Duval, respectively).

Figure 11.9 Q-Q Plot of Gold Assays by Company



	Count	Mean	Median	StdDev	CV	Min	Max
Duval	795	0.026	0.016	0.038	1.44	0.000	0.452
Lacana	3572	0.023	0.015	0.033	1.43	0.000	0.961
Santa Fe	1823	0.023	0.015	0.027	1.18	0.000	0.436
Pegasus	231	0.027	0.013	0.031	1.18	0.002	0.238
Newgold	1133	0.023	0.010	0.045	1.96	0.000	0.759
Firstgold	1630	0.017	0.010	0.033	1.87	0.000	0.718
Pershing Gold	6860	0.0282	0.0099	0.0922	3.27	0.0000	3.6180

The Pershing Gold drilling has a significantly more variable assay population than the historical drilling, as indicated both by the much higher coefficient of variation ("CV") and the steepness of the assay plot. The Pershing Gold assay data has a much higher maximum value and a higher percentage of samples assaying greater than 0.1 oz au/ton as compared to the historical drilling. These differences are attributed to both a spatial bias, in that the Pershing Gold drilling is located primarily within the North area Lower and Jasperoid zones, which on average is higher grade than the Main zone, and the fact that the Pershing drilling is all core so there is not the relative smoothing of gold grades that is common within RC drilling.

11.18 Additional Comments

While documentation of sample preparation, analysis, and security for the various companies that operated at Relief Canyon prior to Firstgold is incomplete, all of the companies were reputable, well-known mining or exploration companies that likely followed accepted industry practices.

All of the laboratories discussed above are, or were, well-known commercial analytical laboratories and were independent of Pershing Gold. Only the drill samples of Firstgold and Pershing Gold were assayed in a laboratory with present-day certification (ALS, ISO 9001:2000; American Assay, ISO/IEC 17025; Inspectorate ISO 9001:2000); all other assaying was completed prior to the institution of formal certifications. It is Mr. Tietz's opinion that the sample preparation, security, and analytical procedures used by Pershing Gold and prior operators were acceptable procedures and the resulting analytical data are of sufficient quality for use in the resource estimation.

12.0 DATA VERIFICATION

12.1 Site Visit

Mr. Paul Tietz visited the Relief Canyon project office and field site on October 17 and 18, 2013, January 15, 2015, September 30, 2015, and October 13, 2016. During all site visits, the project geology was reviewed, which included: a) a field tour of the deposit area; b) visual inspection of core holes; and c) discussion with Pershing personnel of the current geologic interpretations. Drill site and mineralization verification procedures were conducted, and core drilling and sampling procedures were appraised. MDA has also maintained a relatively continual line of communication through telephone calls and emails with Pershing Gold project personnel in which the project status, procedures, and geologic ideas and concepts have been discussed. The result of the site visits and communications is that MDA has no significant concerns with the project procedures.

Mr. Tietz has also verified the Relief Canyon project database and compiled and analyzed available quality control/quality assurance ("QA/QC") data collected by Lacana, Firstgold, and Pershing Gold; no QA/QC data collected by Duval, Pegasus, or Santa Fe are available. In addition to a review of the database verification and available QA/QC data, a comparison of the drill data by company is also discussed, as is a sample-pair analysis of closely spaced drill intervals from adjacent holes.

Data verification, as defined in NI 43-101, is the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. There were no limitations on, or failure to conduct, the data verification for this report. Additional confirmation on the drill data's suitability for use are the analyses of the Relief Canyon QA/QC procedures and results as described in Section 11.10 through Section 11.17.

12.2 Database Verification

The project database includes information derived from 1,095 drill holes. For this resource estimate, Mr. Tietz completed a full audit of the Pershing Gold 2016 drill data (18 drill holes). The earlier Pershing Gold and historical data had previously been audited by MDA in 2014, 2015, and January 2016 as summarized below.

12.2.1 MDA 2010 Audit of Historical Data

The following text is taken from MDA's 2010 report on the Relief Canyon project (Gustin, 2010).

Using various digital survey files provided by Firstgold, MDA validated the collar locations of 540 holes, found discrepancies of 2.0 feet or less in 23 holes, and found significant discrepancies in the collar coordinates of 16 holes, only some of which could be resolved (11 of the 16 holes provide data to the resource estimation). After fixing the collar locations to the extent possible, MDA still noted several holes lying significantly above or below the ground surface, which indicate errors in the x, y, and/or z coordinates, or whose mineralized intervals seemed out of place in context with surrounding holes. These database uncertainties, in conjunction with other factors, led to the lack of resources classified as Measured resources in MDA's 2010 estimate. No records of down-hole survey data were found that could be used to validate the database.

The pre-2007 assay database was audited using original or photocopied assay certificates and typed or handwritten assays on drill logs. A total of 3,997 sample intervals out of 31,579 (~13 percent) were audited. A total of 62 audited sample intervals had substantive errors (≥0.007 oz Au/ton), 40 had marginally significant errors (≥0.003 to 0.006 oz Au/ton), and 106 had insignificant errors. All errors found were corrected.

The audit revealed problems in conversions of gold values from ppb to oz/ton and *vice versa*, as well as assigned values for less-than-detection limit and trace assays, all of which MDA corrected to the extent possible. A minor amount of original and check assay data were added to the project database by MDA.

There are 6,914 sample intervals from the Firstgold 2007 and 2008 drill holes with assays from independent laboratories in the database. Using original digital copies of the assay certificates received directly from the analytical laboratories, MDA was able to check 6,890 of the intervals. Eight errors were found and corrected, two of which were substantive.

12.2.2 MDA 2014 Database Verification

MDA validated the collar locations of the 2011 through 2013 holes drilled by Pershing Gold against the original in-house and third-party survey data. MDA revised the collar locations for the 2012 reverse circulation drill holes to incorporate the 32-foot shift in Northing coordinate as discussed in Section 10.4. No other changes were made to the collar survey data.

The Pershing Gold down-hole survey data were checked against the drill-log data and the available original survey card data. In reviewing the data, it was observed that the Pershing Gold database contained down-hole azimuth readings for the 2012 core holes (RC12-008 thru RC12-030) that matched the original Reflex survey data. However, these data had not been adjusted for declination, so the MDA database was revised to account for a -13.5 degree declination similar to that used in the 2013 drill campaign.

MDA made some other minor changes to the survey database, including removing the unsurveyed "0" depth collar set-up orientation for those holes that have down-hole survey data.

Following up on MDA's 2010 assay audit, approximately 8,800 pre-2007 assay intervals (one fourth of the total data set) were checked against copies of the original handwritten drill logs and assay sheets. The audit primarily focused on less-than-detection limit and trace assay conversion values. Besides the occasional random error, there were over 200 Lacana drill intervals with 0.005 oz Au/ton values in the database that in the original assay sheets were noted as "<0.005". These data were corrected to the <0.005 value.

All of the 2007-2008 assay data were checked against the original digital laboratory certificates. Only three significant errors were noted, and these were corrected. There were many insignificant "rounding" errors, and these were also corrected.

MDA compared all Pershing Gold assay data against original digital data downloaded from the assay laboratories. A total of 177 errors were noted, though 161 were less than detection or trace value errors and are considered insignificant. Material errors were just 0.1 percent of the database. All errors were corrected.

12.2.3 MDA February 2015 Database Verification

MDA validated the collar locations of the 2014 through February 2015 holes drilled by Pershing Gold against the original in-house and third-party survey data. One small error of less than one foot discrepancy was noted and the database was changed to match the original collar survey data.

The Pershing Gold down-hole survey data were checked against the drill-log data, along with the available original survey card and spreadsheet data. Working with Pershing Gold, MDA made some minor changes to the survey database including removing bad readings due to an abnormal local magnetic field, adding survey data at the hole TD, and removing the unsurveyed "0" depth collar set-up orientation for those holes that have nearby down-hole survey data.

MDA compared all Pershing Gold assay data against original digital data downloaded from the assay laboratories. A total of 443 errors were noted, though 387 were less than detection or trace value errors and are considered insignificant. Only 30 errors were considered material (samples with greater than 0.005oz Au/ton difference) which totaled just 0.2 percent of the database. All errors were corrected.

12.2.4 MDA January 2016 Database Verification

MDA validated the collar locations of the March 2015 through December 2015 holes (RC15-289 through RC15-458) drilled by Pershing Gold against the original in-house survey data. No errors were noted, though MDA made a minor change to the total depth of one drill hole.

The Pershing Gold down-hole survey data were checked against the drill-log data, along with the original third party (Devjco) survey spreadsheet data. As in previous survey audits, MDA worked with Pershing Gold and made minor changes to the survey database including removing erroneous readings due to an abnormal local magnetic field, adding survey data at the hole TD, and removing the unsurveyed "0" depth collar set-up orientation for those holes that have nearby down-hole survey data.

MDA compared all Pershing Gold assay data for the March 2015 through December 2015 drill holes against original digital data downloaded from the assay laboratories. A total of 217 errors were noted, though only 16 errors were considered material (samples with greater than 0.005oz Au/ton difference). These material errors totaled just 0.1 percent of the database. All errors were corrected.

12.2.5 MDA October 2016 Database Verification

Using the same comparison procedures as in previous years, MDA validated all of the collar, downhole survey, and assay data for the 18 core holes (RC16-459 through RC16-475) drilled June 2015 through October 2016. No errors were noted in the collar locations, while, as in previous audits, a few changes were made to the survey data, including removing erroneous readings due to an abnormal local magnetic field, and removing the unsurveyed "0" depth collar set-up orientation for those holes that have nearby down-hole survey data.

MDA compared all of the 2016 Pershing Gold assay data against original digital data downloaded from the assay laboratories. No errors were noted.

The author believes the corrected MDA database is of sufficient quality to be used for the resource estimation at Relief Canyon.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section was prepared by Mark Jorgensen of Jorgensen Engineering and Technical Services, LLC ("JE&TS"). Mr. Jorgensen has reviewed the information cited below and it is his opinion that the information accurately represents the Relief Canyon processing and metallurgical testing. The term "ore" is used frequently in this section and refers to mineralized material. In some cases, the mineralized material constitutes portions of the estimated reserves. In other cases, the term has no economic significance, but merely refers to metallurgical test material.

13.1 Introduction

The Relief Canyon project is comprised of a predominantly oxidized to partially oxidized gold mineral resource that metallurgical testing and historical mining experience indicate is amenable to heap-leach cyanidation processing.

13.2 Historical Operating Data

Heap leaching of Relief Canyon ore was conducted by Lacana in 1984 and 1985 and by Pegasus between 1987 and 1990. Subsequently, Newgold (later Firstgold) redeveloped the project in 2007 and 2008, and reprocessed existing heap leach tailings by crushing and re-leaching (heap leaching). Observations made in this chapter concerning operational experience are based on previous reviews (Gustin, 2010; Altman, 2013; and Janke, 2013). None of the original operational data or reports, or other third-party reviews were available.

13.2.1 Lacana Operations

Lacana operated the Relief Canyon mine between 1984 and 1985 by heap leaching ROM ore. The average recoveries realized during Lacana's operations were reportedly 45 percent to 48 percent. This was significantly lower than expected, based on Lacana's pilot heap-leach testing. No reasons for the lower than expected gold recoveries were found in the reviews referenced above.

13.2.2 Pegasus Operations

Pegasus operated the Relief Canyon mine between 1987 and 1990. The ore was primary crushed, agglomerated, stacked on the leach pad with trucks and heap leached. The first annual report for the project listed a gold recovery in "excess of 65 percent". A final recovery number that would represent what the heap leach process could have produced over a lengthier leach time was not available. (Janke 2013)

13.2.3 Newgold Operations

During a few months in 2008 and 2009, Newgold reprocessed approximately 250,000 tons of previously leached material at the Relief Canyon mine (Janke, 2013). Reprocessing consisted of re-crushing historical heap residue material to -1/2 inch and heap leaching (re-leaching). The project was reportedly shut down after a few months because of low gold recoveries.

Testing to identify and process new resources that had not been mined was conducted by Firstgold (formerly Newgold) in 2009.

In 2009, Firstgold had two column-leach tests conducted at KCA (KCA, 2010) on two bulk samples of mineralized material taken from the bottom of the Relief Canyon North Pit. Both samples were from the Main Zone. Sampling procedures were described in an internal Firstgold memorandum (Beck, 2010). One sample was designated as clay-matrix limestone breccia. The other was designated as jasperoid hosted. The samples had assayed head grades of 0.016 and 0.017 oz Au/ton, respectively. The samples were crushed to minus ¾ inch before shipment to KCA. Agglomeration testing on the samples indicated that no cement agglomeration would be required, but that a cement addition of 3.0 lb/ ton of ore would be sufficient for the purposes of maintaining alkalinity during leaching.

Gold recoveries from the clay-matrix limestone breccia and jasperoid-hosted samples were 83 percent and 85 percent, respectively. Respective silver recoveries were 29 percent and 4 percent. Cyanide consumptions were 0.58 and 0.47 lb/ton.

13.2.4 Summary of Historical Operations

A summary list of metallurgical testing programs and historical commercial production periods for the project is shown in Table 13.1. Correlating historical test work with the resource that is currently available is difficult due to the lack of reports which would provide important information such as lithology and location. The summary list in Table 13.1 is provided for historical context only.

Table 13.1 Summary of Metallurgical Testing and Commercial Production

Dates					
Metallurgical Testing	**Commercial Operations**	**Commisioning Company**	**Laboratory**	**Description**	**References**
1982 - 1983		Lacana	DML	Bottle Roll, Agitated Leach, Column Leach, Barrel Leach and Gravity Concentration Tests	Dawson, Salisbury, 1982
1983		Lacana	KCA	Bottle Roll and Column Leach Testing	Dawson, Salisbury, 1982
1983		Lacana	N/A	Lacana Pilot Heap	Gustin, 2010
	1984 - 1985	Lacana	N/A	Lacana Operations	
1987		Pegasus	Unknown	Size Fraction Leaching - Jasperoid Material	Gustin, 2010
	1987 - 1990	Pegasus	N/A	Pegasus Operations	
2006		Newgold	MLI	Column Leach Testing - Heap Residue Samples	McPartland, 2007
	2007 - 2008	Newgold	N/A	Newgold Project Redevelopment Operations	Janke, 2013
2008		Firstgold	KCA	Agglomeration and Column Leach Testing - Heap Residue Samples	KCA, 2008
2009		Firstgold	KCA	Agglomeration and Column Leach Testing - North Pit Samples	KCA, 2010
2011 - 2012		Pershing Gold	Unknown	Pershing Gold Cyanide Shake Study	Janke, 2013
2012		Pershing Gold	KCA	Bottle Roll Testing	KCA, 2012 & Janke, 2013
2013 - 2014		Pershing Gold	MLI	Bottle Roll and Column Leach Testing	Olson, 2014a
2014		Pershing Gold	MLI	Bottle Roll Tests on 3 Eploration Samples	Olson, 2014b
2015		Pershing Gold	MLI	Metallurgical Testing - Bottle Roll and Column Leach Testing	Olson, 2015a & Olson 2015b
2014 - 2015		Pershing Gold	MLI	Agglomeration and Load/Permeability Testing	Olson, 2016a
2015 - 2016		Pershing Gold	MLI	Bottle Roll and Column Leach Testing	Olson, 2016b
2018		Pershing Gold	MLI	Bottle Roll Deposit Variability Testing	McPartland, 2018
2018		Pershing Gold	Geo-logic/ Newfields	Load Permeability Testing	Hillman, 2018/ Magner, 2018

DML=Dawson Metallurgical Laboratories Inc.; KCA = Kappes Cassiday and Accociates; MLI = McClelland Laboratories Inc.

13.3 Ore Characterization

The Relief Canyon ore that is currently considered for mining and heap leaching is made of three types of material that are classified as zones: Main, Lower and Jasperoid. Characterization of these ore zones has evolved since Pershing Gold began exploration on the property.

Beginning in 2012 potential ore grade (>0.006 oz/ton Au) samples from all drill holes were analyzed for cyanide soluble gold. The results from this data, as well as detailed core logging lead to the development of metallurgical classifications based on rock type and cyanide solubility for the Main Zone.

In 2013, five composites were prepared, which were designated “1-LSBXH” (limestone breccia with high cyanide solubility), “2-LSBXL” (limestone breccia with low cyanide solubility), “3-CMBXH” (clay-matrix breccia with high cyanide solubility), “4-CMBXL” (clay-matrix breccia with low cyanide solubility), and “5-JSPBX” (jasperoid breccia). These initial classifications considered rock type but did not consider the geologic placement of gold in the ore deposit.

In 2014, a composite was made that modeled an overall recovery for the Main Zone portion of the project based on the percentage of rock types contained within the five drill hole composites from 2013 described above. Additional study and modeling would show that these percentages would change, especially for rock types that showed reduced heap leaching characteristics, such as the jasperoid breccia. This early Pershing Gold metallurgical work showed reasonable metallurgical recoveries, but may have underestimated recoveries because the geologic model was still being refined with the results from additional drilling in 2014 - 2016.

In 2015, additional drilling verified the extent of all zones, especially the Lower and Jasperoid. Geologic and mine modeling showed that the ore near surface was comprised of the Main Zone type. The Lower Zone and Jasperoid Zones were below the Main Zone, but trended upward to meet the Main Zone on the north end of the proposed pit. The zones are generally sub-parallel and hosted by carbonate bearing units.

Gold recoveries for the ore planned for crush/agglomerate/heap-leach processing are defined by these three zones. An approximation of the reserve tonnage and contained ounces associated with each zone as defined by drilling and modeling through 2016 is shown in Table 13.2.

Table 13.2 Metallurgical Zones, Tons Grade and Ounces

Material	Tons Placed	Grade Placed	Ounces Placed	Ounces Produced	Cumulative Rec.	% of Total
	000's	oz au/t	000's	000's	%	Ounces
Main	21,337	0.017	369	314	85%	58%
Lower	4,133	0.028	116	89	77%	18%
Jasperoid	4,766	0.031	146	105	72%	23%
Totals	30,237	0.021	631	509	81%	100%

13.4 Recent Metallurgical Testing

In 2011, Pershing Gold conducted fire assays on exploration drill-core samples with comparative cyanide soluble gold analysis on "ore grade" intervals (Janke, 2013). The data analysis indicated average gold recoveries (cyanide soluble versus fire assay) for the oxide, mixed oxide, and sulfide sample types of 82 percent, 76 percent, and 34 percent, respectively. There was a significant range of recoveries for the oxide and mixed oxide sample types, but no strong trends with respect to sample depth. No information was referenced concerning the drill holes tested. Therefore, the results of these tests are not considered in this report.

During 2012, Pershing Gold submitted eight samples to KCA for testing (KCA, 2012) to estimate mercury liberation during leaching for sizing the planned mercury emission control equipment (Janke, 2013). The samples were described as either generated from pit exposures or made up from drill-hole samples and were intended to represent three host-rock types and three areas from the current resource. As part of this testing, bottle-roll tests were conducted on the samples at a 100 percent -10 mesh feed size. Gold and silver extraction was measured during this testing and varied from 0 percent to 81 percent. Due to the limited representation of the samples, the results of these tests are not considered in this report.

In 2014, Pershing Gold commissioned column-leach tests on five PQ drill-core composites at McClelland Laboratories, Inc. ("MLI") (MLI - Olson, 2014a). Also in 2014, Pershing Gold submitted three exploration samples, representing jasperoid material, for bottle-roll testing at MLI (Olson, 2014b).

In 2015, Pershing Gold undertook a metallurgical testing program on four drill-core composites from the Lower and Jasperoid zones, and on a single bulk sample, representing mineralization rich in clay-size material from the Main Zone. The testing program was conducted at MLI (Olson, 2015a; Olson, 2015b).

In late 2015, Pershing Gold submitted a single bulk sample to MLI for agglomeration optimization and load/permeability testing (Olson, 2016a). The sample was described as Dump 4 material.

Also in late 2015, Pershing Gold submitted another bulk sample to MLI for bottle-roll and column-leach testing. The sample represented low fines material from the North Pit, Main zone. That sample, along with the high fines bulk sample also received in late 2015, were used in column testing (Olson, 2016b).

In 2018 Pershing Gold submitted 38 core samples for variability testing for gold recovery to MLI. The samples were taken from drill holes along a north to south strike at different elevations to represent Main, Lower and Jasperoid zones.

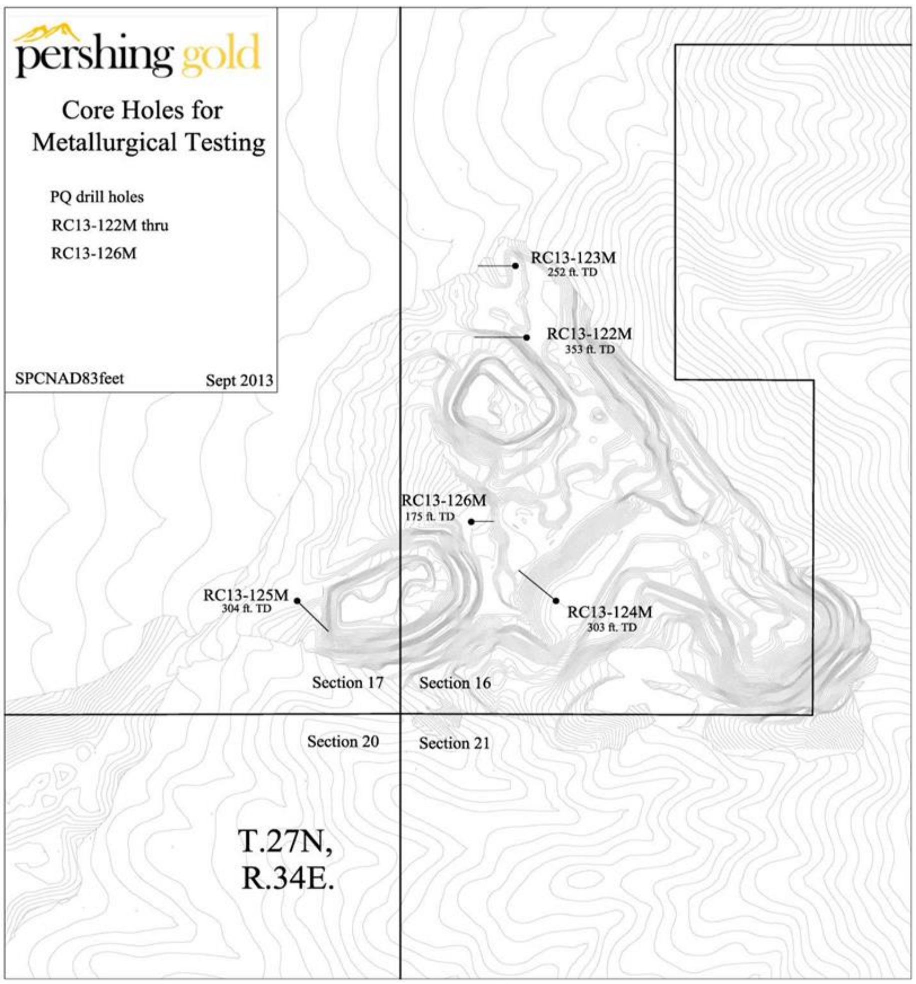
In 2018 Pershing Gold submitted a mixture of two bulk samples that had been collected in 2015 and had been mixed and agglomerated for permeability testing. The samples contained different quantities of fine material. The samples were sent to two different laboratories, Geo-Logic in Grass Valley, California and Newfields in Elko, Nevada.

13.4.1 Pershing Gold - 2014

In September 2013, a total of 276 drill-core samples from five drill holes were submitted by Pershing Gold to MLI for metallurgical testing. Results from that testing were described in an MLI report (Olson 2014a).

A detailed description of the sampling program used to generate the drill core for the MLI testing was prepared by Pershing Gold exploration staff (GAC memo, Sept. 10, 2013). Five PQ holes (RC13-122M to 126M) were completed for a total of 1,388 feet. The samples "*were designed to intercept significant intervals of gold mineralization in the block model throughout the pit area as well as above and below the water table.*" It was also noted that "*this drilling only tested the Main zone, other mineral zones including the Lower zone were not the target of this drilling*." Samples were described by lithology, alteration, and mineralization. A plan view of the drill-hole locations is shown in Figure 13.1. A summary of the lithologies encountered in the drill core is shown in Table 13.3.

Figure 13.1 Plan View of the 2013 Metallurgical Drill-Hole Locations



Based on fire assay and cyanide solubility results, along with core logging information, five composites were prepared. Those composites were designated "1-LSBXH" (limestone breccia with high cyanide solubility), "2-LSBXL" (limestone breccia with low cyanide solubility), "3-CMBXH" (clay-matrix breccia with high cyanide solubility), "4-CMBXL" (clay-matrix breccia with low cyanide solubility), and "5-JSPBX" (jasperoid breccia). The limestone breccia and clay-matrix breccia composites were comprised only of drill core. The jasperoid breccia composite was comprised of drill core (58 percent by weight), supplemented with a bulk sample (42 percent by weight) because of the limited amount of drill core available.

Table 13.3 Lithologies Encountered in the 2013 Metallurgical Drilling

Rock Code	Lengths (Ft)	Percent of total	Description	Type
QAL	5.0	0.36%	Alluvium	Waste
GVCMBX	250.4	18.04%	Clay Breccia Grass Valley	Waste
PMG	61.0	4.39%	Gabbro Intrusion	Waste?
LSTB	156.9	11.30%	Thin Bedded limestone	Waste?
LSCR	24.6	1.77%	carbonaceous limestone	Waste?
LSCMBX	47.7	3.44%	Clay-matrix breccia, multi lithic	Ore
FLTBX	178.8	12.88%	Fault breccia	Ore
BXCF	99.2	7.15%	Cave Fill Breccia	Ore
LSBX	552.2	39.78%	Limestone Breccia	Ore
JSBX	12.2	0.88%	Jasperiod Breccia	Ore
	1,388.0	100.00%		

Composites were prepared by combining selected intervals in their entirety, using the descriptions provided above. Intervals logged as unoxidized material generally were left out of the composites. In some cases, individual interval widths included in the composites were relatively small (as small as 1.3 feet), in order to generate sufficient sample for testing, while keeping the composites separated by ore type. Some of the smaller interval widths included in the composites would be considered impractically small for open pit mining techniques. Of the 438 lineal feet of drill core that comprised the five metallurgical composites, 185 feet came from intervals that were less than five-feet thick. Pershing Gold included these thinner intervals in the composite columns because they are representative of larger, potentially minable intervals of the same material within the Relief Canyon deposit.

Head analysis results for the metallurgical composites are shown in Table 13.4.

Table 13.4 Head Analyses, Relief Canyon Drill-Core Composites, MLI 2014 Testing

Analysis	Unit	1-LSBXH	2-LSBXL	3-CMBXH	4-CMBXL	5-JSPBX
Interval Analyses[1)]						
Fire Assay, Au	oz/ton	0.016	0.011	0.023	0.045	N/A
Fire Assay, Ag	oz/ton	0.06	0.04	0.06	0.11	N/A
CN Sol. : Fire Assay, Au	%	92.2	37.9	94.7	77.4	N/A
CN Sol. : Fire Assay, Ag	%	79.9	65.5	71.8	74.2	N/A
Composite Analyses						
Fire Assay, Au	oz/ton	0.017	0.015	0.024	0.040	0.033
Fire Assay, Ag	oz/ton	0.06	0.06	0.06	0.12	0.22
CN Sol. : Fire Assay, Au	%	82.4	57.1	96.0	85.3	93.5
CN Sol. : Fire Assay, Ag	%	72.2	31.6	91.7	83.3	84.4
Preg-Rob Analysis, Au	%	1.2	2.4	1.2	3.6	3.6
C (Organic)	%	0.04	0.22	0.05	0.11	0.03
S (Sulfide)	%	0.01	0.61	0.01	0.05	<0.01
1) Composite results, calculated based on properly weighted interval analysis results.						

Metallurgical testing conducted on each of the five individual composites included bottle-roll tests at feed sizes of 100 percent -2 inch, 80 percent -3/4 inch, 80 percent -3/8 inch, 80 percent -10M, and 80 percent -200M, and a column leach test at an 80 percent -3/4 inch feed size.

The same scope of bottle-roll testing and two column-leach tests (80 percent -3/4 inch) were conducted on a master composite, which was made up of proportional contributions from each of the individual composites to represent the Main Zone resource as a whole.

Bottle-roll test results showed that four of the five composites were amenable to agitated cyanidation treatment. Gold recoveries obtained from composites 1-LSBXH, 3-CMBXH, 4-CMBXL, and 5-JSPBX at an 80 percent -3/4 inch feed size ranged from 65.5 percent to 87.0 percent and averaged 75.3 percent in four days of leaching.

The jasperoid breccia composite (5-JSPBX) was quite sensitive to feed size, and gold recovery increased from 48.4 percent at the 100 percent -2 inch feed size, to 87.1 percent at the 80 percent -200M feed size. The three other composites discussed above were not particularly sensitive to feed size, though recoveries for the milled (200M) feeds tended to be moderately higher than obtained from the crushed feeds. It should be noted that the JSPBX was a Main Zone ore type and made up less than 1 percent of the lithologies encountered in the drilling as shown in Table 13.3 and was not considered as a major ore deposit constituent in later testwork.

The limestone breccia with low cyanide solubility composite (2-LSBXL) gave significantly lower gold recoveries than the four other composites. Gold recoveries for the crushed feeds were somewhat erratic and ranged from 16.7 percent at the 80 percent -3/4 inch feed size, to 30.8 percent at both the 100 percent -2 inch and 80 percent -3/8 inch feed sizes. Gold recovery from the 80 percent -200M feed was moderately higher (40.0 percent). There were some indications of a mild preg-robbing character for this composite. Locking of contained gold in sulfide mineral grains or in silica are among other possible explanations for the low gold recoveries obtained from this composite.

Bottle-roll test reagent consumptions were low. Cyanide consumptions at the ¾ inch feed size ranged from <0.14 to 0.39 lb NaCN/ton. Lime requirements ranged from 2.1 to 3.3 lbs/ton ore.

Summary results from the bottle-roll tests are presented in Table 13.5.

Table 13.5 Summary Results, Bottle-roll Tests, MLI 2014

Composite	Feed Size	Au Recovery %	Calculated Head ozAu/ton	Reagent Requirements lb/ton ore	
				NaCN Cons.	Lime Added
1-LSBXH	100%-2"	82.4	0.017	0.16	1.9
1-LSBXH	80%-3/4"	81.8	0.011	<0.14	3.0
1-LSBXH	80%-3/8"	83.3	0.012	0.20	3.8
1-LSBXH	80%-10M	83.3	0.012	<0.14	4.0
1-LSBXH	80%-200M	92.3	0.013	<0.14	2.7
2-LSBXL	100%-2"	30.8	0.013	<0.14	2.7
2-LSBXL	80%-3/4"	16.7	0.012	0.39	3.3
2-LSBXL	80%-3/8"	30.8	0.013	0.36	3.6
2-LSBXL	80%-10M	28.6	0.014	0.55	4.0
2-LSBXL	80%-200M	40.0	0.015	0.59	2.7
3-CMBXH	100%-2"	90.9	0.022	0.16	1.8
3-CMBXH	80%-3/4"	87.0	0.023	<0.14	2.7
3-CMBXH	80%-3/8"	81.8	0.022	<0.14	2.6
3-CMBXH	80%-10M	87.0	0.023	0.21	2.4
3-CMBXH	80%-200M	91.3	0.023	<0.14	2.0
4-CMBXL	100%-2"	70.3	0.037	0.30	2.0
4-CMBXL	80%-3/4"	66.7	0.036	0.32	3.3
4-CMBXL	80%-3/8"	74.3	0.035	<0.14	3.6
4-CMBXL	80%-10M	73.7	0.038	0.23	4.4
4-CMBXL	80%-200M	81.1	0.037	<0.14	3.1
5-JSPBX	100%-2"	48.4	0.031	<0.14	1.6
5-JSPBX	80%-3/4"	65.5	0.029	<0.14	2.1
5-JSPBX	80%-3/8"	63.3	0.030	<0.14	2.0
5-JSPBX	80%-10M	74.2	0.031	<0.14	2.0
5-JSPBX	80%-200M	87.1	0.031	<0.14	2.1

Column tests were performed on each of the five composites. Summary results from the column-leach tests are presented in Table 13.6. Gold leach-rate profiles for the master composite are shown in Figure 13.2.

Column test results showed that three of the five individual composites were readily amenable to simulated heap leach cyanidation treatment at an 80 percent -3/4 inch feed size. Gold recoveries obtained from composites 1-LSBXH, 3-CMBXH, and 4-CMBXL were 85.7 percent, 91.3 percent, and 78.0 percent, respectively, in 76 days of leaching and rinsing. Gold recovery rates for those composites were rapid, and gold extraction was substantially complete in 30 days of leaching. Final solution to ore ratios varied from 3.2:1 to 3.8:1.

Table 13.6 Summary Results, Column Leach Tests, MLI 2014 Testing, 80% -3/4"

Composite	Au Recovery, (%)	Calculated Head, (oz Au/ton)	Solution to Ore Ratio	Leach Time (days)	Reagent Requirements	
					NaCN Usage (lbs/ton)	Cement Added (lbs/ton)
1-LSBXH	85.7	0.014	3.2:1	76	1.1	8.0
1-LSBXL	38.5	0.013	3.2:1	76	1.2	8.0
3-CMBXH	91.3	0.023	3.2:1	76	1.1	8.0
4-CMBXL	78.0	0.041	3.8:1	76	1.7	8.0
5-JSPBX	65.5	0.029	9.4:1	194	3.0	8.0
MASTER (TEST P6)	79.2	0.024	2.8:1	76	1.3	8.0
MASTER (TEST P7)	81.8	0.022	4.3:1	91	1.2	8.0
MASTER (TEST P8)	72.7	0.022	3.3:1	76	1.3	8.0

Gold recovery from the jasperoid breccia composite (5-JSPBX) was somewhat lower (65.5 percent), in large part because of a very slow leach rate. Gold extraction was progressing from that test at a slow but significant rate when the test was ended after 194 days.

The limestone breccia with low cyanide solubility, composite (2-LSBXL), was not as amenable to simulated heap leach cyanidation treatment at an 80 percent -3/4 inch feed size. A gold recovery of 38.5 percent was obtained in 76 days of leaching and rinsing.

Column test cyanide consumptions were moderate (1.07-1.74 lbs/ton of ore) for the individual composite tests run for 76 days. Cyanide consumption for the jasperoid breccia composite test was significantly higher (3.02 lbs/ton of ore), in large part because of the column test leach cycle that was extended to 194 days. Column test cyanide consumptions are usually substantially higher than experienced in commercial production. Considering the column test and bottle-roll test data, it is expected that commercial consumptions for the ore types represented by the composites tested probably would not exceed 0.5 lbs/ton of ore.

A single master composite was prepared from the five individual composites based on the expected weighting of the five ore types in the Relief Canyon resource. Composite make-up information for the master composite is shown in Table 13.7. This master composite included a proportional contribution of the low recovery LSBXL sample to reflect the more refractory portion of the resource that is planned for heap-leach processing.

Figure 13.2 Leach Rate Profiles, Master Composite P7, Column Leach - 80% -3 inch Feed

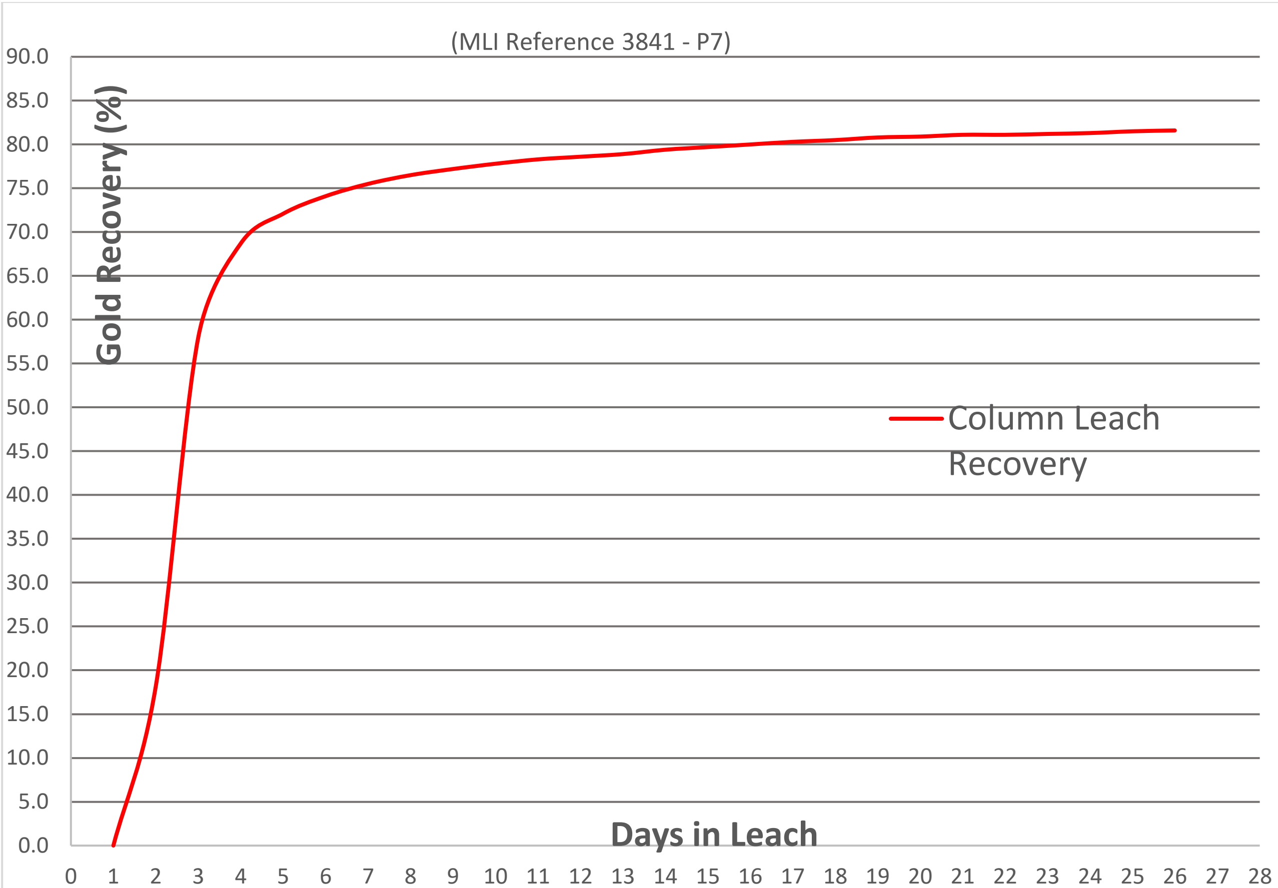


Table 13.7 Composite Make-up, Relief Canyon Master Composite, McClelland - 2014

Composite	Weight to Comp. %	Grade (Fire Assay) ozAu/ton	Au Contribution to Composite, %
1-LSBXH	30.8	0.017	22.0
2-LSBXL	13.2	0.015	8.3
3-CMBXH	31.5	0.024	31.8
4-CMBXL	13.5	0.040	22.7
5-JSPBX	11.0	0.033	15.2
Composite	100.0	0.024	100.0

Three column tests were conducted on the master composite. Two duplicate columns utilized only master-composite sample while a third column test was conducted on the master composite (80 percent -3/4 inch) after it was placed on top of a layer of the heap residue sample. The purpose for this test was to evaluate if there was any negative impact of using the historic heap residue as a drain and liner protection layer on the leaching of freshly mined, crushed, and agglomerated ore.

A bulk sample of leached residue from the historical Relief Canyon heaps was also received for testing. That sample was identified as "Composite 6 - 2009 Tailings" and was not included in the composite summarized in Table 13.7.

Results of the master composite tests are shown in Table 13.6. The two column tests P6 and P7, which were essentially duplicate tests, were conducted on the master composite, at an 80 percent -3/4 inch feed size. Gold recoveries from the two tests were very similar and within the margin of error for this type of test (79.2 percent and 81.8 percent). These gold recoveries agreed closely with the gold recovery that would be expected (78.0 percent), based on the composition of the master composite and the gold recoveries obtained during column testing on the five individual composites.

The LSBXL ore type, which was added to the master composite in a proportional amount to its discovery in the drill holes, had low recoveries but did not have a deleterious effect on the recovery of the other ore types. The inference would be that the low recoveries associated with this ore type were due to the gold being encapsulated in sulfide minerals or in fine-grained silica, and not due to "preg-robbing" material, such as active carbon if carbon were found in this ore. Had there been carbon found in the ore it would have absorbed gold out of solution and the recovery would likely have been less than the calculated recovery. Since gold recovery did not increase significantly with size reduction as noted in the bottle-roll tests, it is likely that the gold for the LSBXL ore type is encapsulated in sulfide minerals.

A column-leach test was conducted by placing the master composite (90 percent by weight) on top of the historical heap residue material from the project (10 percent by weight). Results from this test showed that gold extraction from the master composite material was similar to the gold extraction obtained from the two master composite column tests described above. No significant adverse effects resulted from leaching the master composite on top of the historical heap residue material. Gold recovery from the historical heap residue material was negligible. Gold recovery rate and reagent consumptions were similar to those obtained from the two other master composite tests.

All of the column charges were agglomerated before leaching using a cement (Portland type I/II) binder addition of 8 lbs/ton. This binder addition was based on results from scoping agglomeration ("agglomerate strength and stability") tests conducted on each of the five individual composites and the master composite. The binder addition was sufficient for maintaining adequate solution percolation during column leach testing. After column leaching, selected column-leached residues were sampled and submitted to a geotechnical testing laboratory (GeoLogic Associates) for fixed-wall hydraulic conductivity testing (ASTM method D-2434--load permeability testing). This Load/Permeability testing program is summarized in Section 13.6.

13.4.2 Pershing Gold - 2015

In February 2015, a heap-leach testing program was undertaken by MLI on drill-core composites and a bulk surface sample (designated "BS-1") submitted by Pershing Gold. The drill composites represented material from the Lower and Jasperoid zones of the Relief Canyon mine. The bulk sample, BS-1, represented rich in clay size material from the Main Zone. Results from that testing program were discussed in an MLI report (Olson 2015a).

In November 2015, a single bulk sample, identified as "Dump 4" material, was received for agglomeration and load/permeability testing at MLI. Results from that testing were included in an MLI report (Olson, 2016a), and described here in Section 13.6.

In October 2015, a single bulk sample, identified as BS-2-NC, and described as a low-fines content bulk sample, was received for column-leach testing at MLI. Column tests were conducted on this sample alone (100 percent minus 6 inch and 1.5 inch feed sizes), and on a blend of this sample (70 percent by weight) with the high-fines bulk sample received in February 2015 (30 percent by weight), at a 100 percent minus 3.0 inch feed size.

13.4.2.1 Main Zone Bulk Sample (BS-1) and Lower Zone and Jasperoid Zone Drill-Core Composite Testing

A total of 338 drill-core samples and a single bulk sample were submitted by Pershing Gold to MLI in February 2015, for metallurgical testing. Eighty-two selected drill-core intervals were crushed, split, and fire assayed to determine gold and silver content, and were subjected to cyanide solubility analysis procedures to determine soluble gold and silver contents. Based on results from those interval sample analyses, four drill-core composites were prepared. The composites were described as representing material from the Lower and Jasperoid zones of the Relief Canyon mine. Assayed head grades for the composites ranged from 0.020 to 0.103 oz Au/ton. Figure 13.3 shows the location of the drill holes that supplied the drill core for the 2015 Jasperoid and Lower Zone composite samples.

The high-fines bulk sample (BS-1) was described as a "Composite of Bulk Metallurgical Sample from North Pit, North Target and the Southwest Lacana Dump". A description of the sample and the procedures used for generating the sample was detailed in a memo by Peter Dilles (2015). A map showing where the samples were taken is shown in Figure 13.4.

The sample was prepared on-site by blending more than four tons of "North Target dump" (clay rich or high fines material) stockpile with equal amounts of East North Pit limestone breccia stockpile and Southwest dump trenched stockpile. The shipped composite was projected to have an average head grade of 0.020 oz Au/ton, and weighed approximately 12 tons. Assayed head grade for the sample was 0.018 oz Au/ton.

Testing conducted on each of the four drill-core composites and one bulk sample included detailed head analyses, bottle-roll tests at five feed sizes, ranging from 100 percent -2 inch to 80 percent -200M, and a column-leach test at an 80 percent -3/4 inch feed size. Column-leach tests were also conducted on the bulk ore sample at feed sizes ranging from 100 percent -6 inch to 80 percent -1.5 inch. Testing on the bulk sample also included the evaluation and optimization of agglomeration pretreatment.

Summary results from the bottle-roll tests are presented in Table 13.8.

Figure 13.3 Locations of Jasperoid and Lower Zone Drill Holes, March 2015

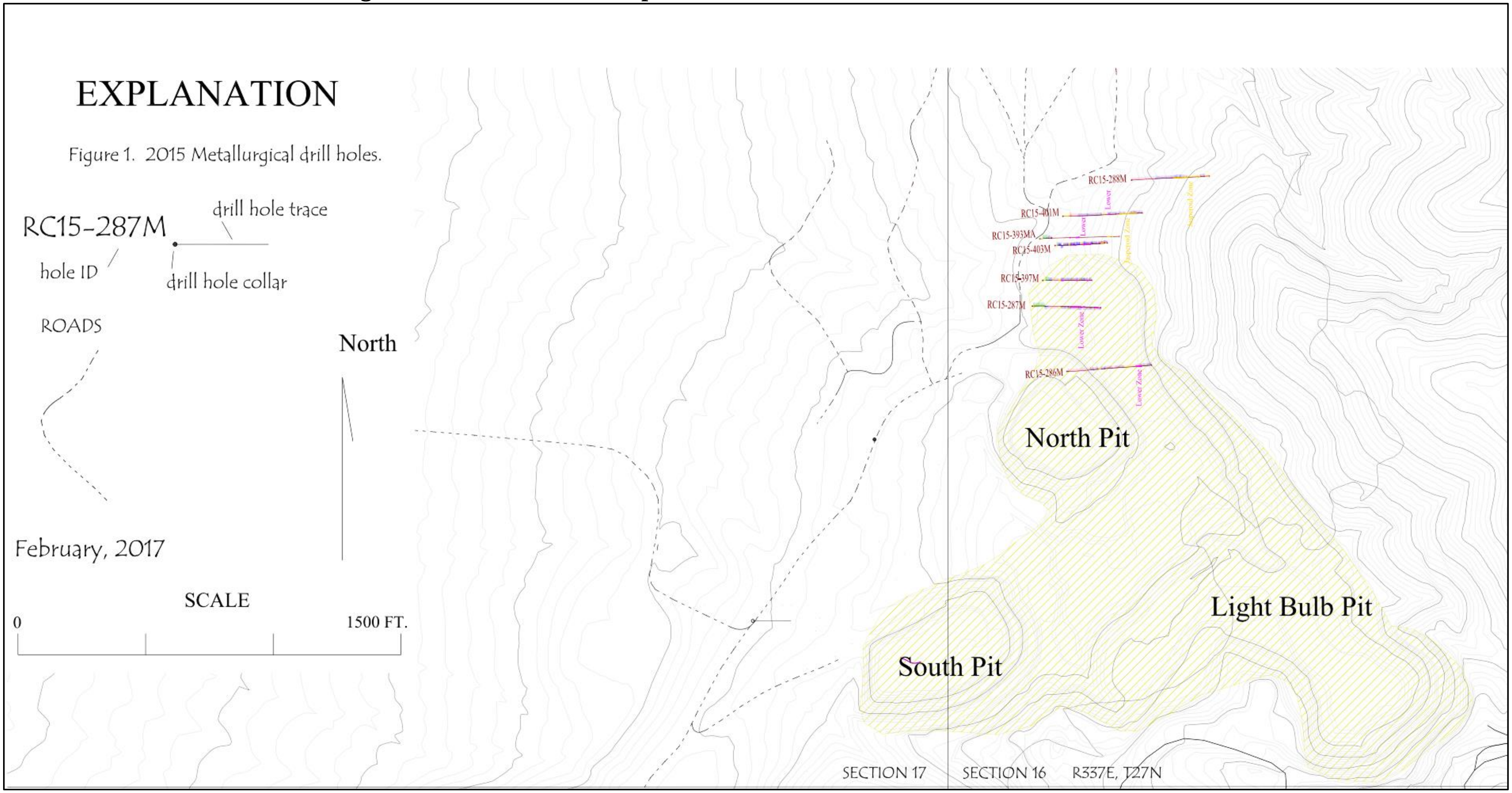


Figure 13.4 Locations of Source Material for Bulk Sample BS-1, March 2015

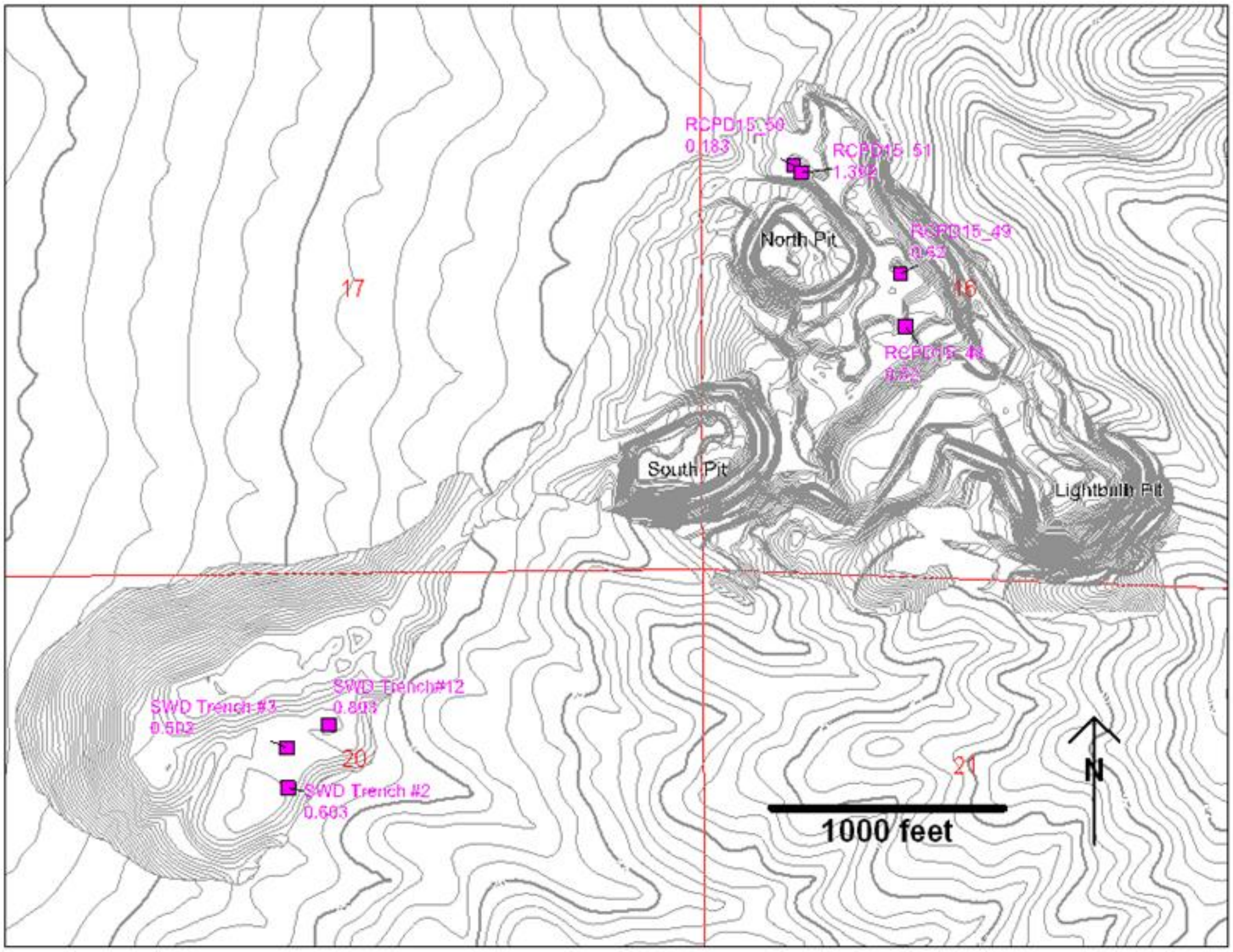


Table 13.8 Summary of 2015 MLI Bottle-Roll Tests, BS-1 and Drill-Core Composites

Sample	Description	Feed Size	Au Recovery, %	Calculated Head, ozAu/ton	Reagent Requirements lb/ton NaCN Cons.	Lime Added
BS-1	Bulk Surface Sample	100%-2"	86.7	0.015	<0.14	1.9
		80%-3/4"	77.8	0.018	<0.14	3.0
		80%-3/8"	82.4	0.017	<0.14	3.6
		80%-10M	81.3	0.016	<0.14	3.4
		80%-200M	86.7	0.015	0.15	3.7
Comp. 1	Jasperoid Zone -Fault Breccia and Jasperoid Breccia, High CN Solubility	100%-2"	67.9	0.109	<0.14	1.6
		80%-3/4"	71.6	0.116	0.25	2.6
		80%-3/8"	67.6	0.145	<0.14	2.9
		80%-10M	79.0	0.124	<0.14	2.7
		80%-200M	97.8	0.134	<0.14	2.5
Comp. 2	Lower Zone - Limestone Breccia and Thin-Bedded Limestone, High CN Solubility	100%-2"	65.2	0.023	<0.14	0.7
		80%-3/4"	78.3	0.023	<0.14	1.6
		80%-3/8"	80.0	0.020	0.15	1.7
		80%-10M	83.3	0.024	<0.14	2.4
		80%-200M	92.0	0.025	0.19	2.1
Comp. 3	Jasperoid Zone - Fault Breccia, Jasperoid Breccia, and White Clay (Illite)-Fluorite, Low CN Solubility	100%-2"	60.5	0.086	0.15	1.9
		80%-3/4"	65.9	0.085	0.23	2.1
		80%-3/8"	68.6	0.086	<0.14	2.8
		80%-10M	69.9	0.093	<0.14	2.9
		80%-200M	95.5	0.089	<0.14	2.9
Comp. 4	Lower Zone - Thin-Bedded Limestone and Gabbro, Low CN Solubility	100%-2"	65.0	0.020	<0.14	0.9
		80%-3/4"	64.7	0.017	<0.14	1.3
		80%-3/8"	68.2	0.022	0.31	2.5
		80%-10M	70.4	0.027	0.27	2.4
		80%-200M	92.3	0.026	<0.14	2.1

Bottle-roll test gold recoveries from the high-fines bulk sample, BS-1, ranged from 77.8 percent to 86.7 percent in 96 hours of leaching, but the sample was described as being not particularly sensitive to feed size. Variations in recovery resulted mainly from the low-grade nature of the sample, and normal analytical and experimental variability.

Gold recoveries from the four Lower and Jasperoid drill core composites increased with decreasing feed size. At 2-inch crush size recoveries ranged from 60 to 70 percent. The high cyanide samples gradually increased to 80 percent recovery at a 10-mesh feed size. The low cyanide samples gradually increased to 70 percent recovery at a 10-mesh feed size. At a 200-mesh feed size all samples showed a significant increase in recovery in excess of 92 percent.

Bottle-roll test cyanide consumption was low for all feeds tested and ranged from less than 0.14 to 0.31 lb NaCN/ton. Cyanide consumption was not strongly correlated with feed size for any of the samples tested. Lime requirements were low and tended to increase with decreasing feed size.

Summary results from the column-leach tests are presented in Table 13.9.

Table 13.9 MLI 2015 Column-Leach Tests, Bulk Sample and Drill-Core Composites

Sample	Description	Feed Size	Au Recov. (%)	Ag Recov. (%)	Calc Head, (oz Au/ton)	Sol. to Ore Ratio	Leach Time (days)	Reagent Requirements NaCN Usage (lbs/ton)	Reagent Requirements Cement Added (lbs/ton)
BS-1	Main Zone - Bulk Sample (High Fines Mineralization)	100% -3 inch	86.7	44.1	0.015	1.9:1	86	1.01	8.0
BS-1	Main Zone - Bulk Sample (High Fines Mineralization)	100% -1.5 inch	85.7	50.0	0.014	2.2:1	64	0.99	8.0
BS-1	Main Zone - Bulk Sample (High Fines Mineralization)	100% -3/4 inch	81.3	57.1	0.016	0.8:1	12	0.51	8.0
Comp. 1	Jasperoid Zone - Fault Breccia and Jasperoid Breccia, High CN Solubility	80% -3/4 inch	82.4	40.9	0.015	7.9:1	168	4.6	8.0
Comp. 2	Lower Zone - Limestone Breccia and Thin Bedded Limestone, High CN Solubility	80% -3/4 inch	84.2	22.2	0.017	5.2:1	109	1.7	8.0
Comp. 3	Jasperoid Zone - Fault Breccia, Jasperoid Breccia, and White Clay (Illite) - Fluorite, Low CN Solubility	80% -3/4 inch	82.4	33.3	0.016	7.8:1	168	4.5	8.0
Comp. 4	Lower Zone - Thin Bedded Limestone and Gabbro, Low CN Solubility	80% -3/4 inch	76.2	-	0.016	5.1:1	96	1.7	8.0

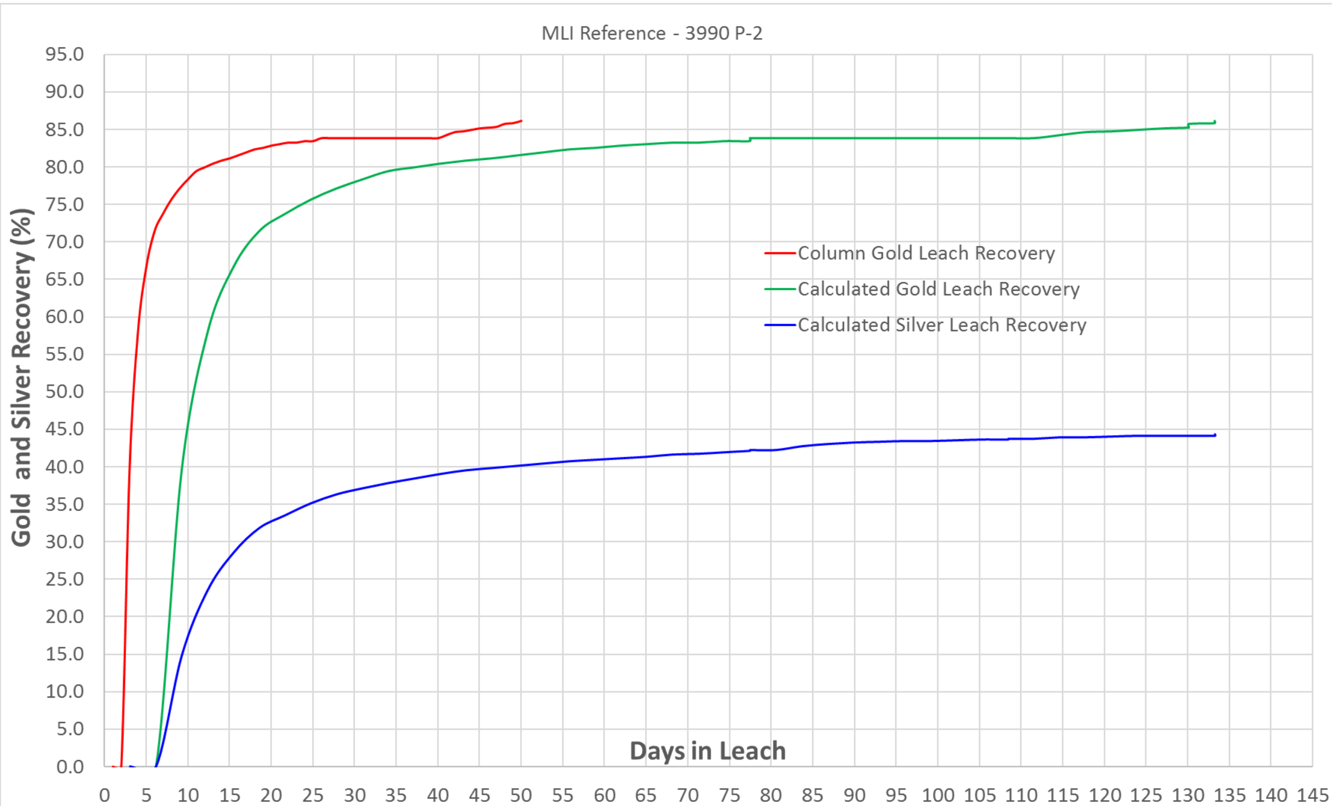
Column-leach test results showed that the bulk sample, BS-1, was amenable to simulated heap-leach cyanidation treatment at feed sizes ranging from 100 percent -3.0 inch to 80 percent -3/4 inch. All of these feeds were agglomerated before leaching, using a cement binder addition of 8.0 lbs/ton. Gold recoveries at these sizes ranged from 85.7 percent to 86.7 percent in 64 to 96 days of leaching and rinsing.

Figure 13.5 shows the column-leach recovery for BS-1 (100 percent - 3 inch) as a function of time. After approximately 50 days of leaching, the gold recovery was 86.7 percent.

The figure also shows the calculated leach recovery rate for a 20-foot lift with an irrigation application rate of 0.003 gal/min/ft^2. Figure 13.5 illustrates the relationship between laboratory column leach rates and the calculated leach cycle for comparable recovery on an operating heap, which for this sample may be 140 days and represents the time required for the corresponding solution to ore ratio to be achieved.

Silver recovery was determined for composite BS-1 in a column-leach test and the same calculation that was used to estimate a gold recovery rate was used to estimate a silver recovery rate. Figure 13.5 illustrates an estimated silver recovery of 44.1 percent for a 20-foot lift after approximately 140 days of leaching.

Figure 13.5 Column-Leach Rate Profiles, Bulk Sample BS-1, - 100% - 3 inch Feed



An additional column-leach test was attempted at a 100 percent -6 inch feed size. In this case, the ore charge was not agglomerated. Ponding was observed early in the leach cycle of this column and percolation rate was very slow. As a result, this test was discontinued. These results indicate that the high-fines Main Zone material represented by the bulk sample will require agglomeration pretreatment and blending with lower fines material.

All four drill-core composites from the Lower and Jasperoid zones also were amenable to simulated heap-leach cyanidation, at an 80 percent -3/4 inch feed size. Gold recovery from these composites ranged from 76.2 percent in 96 days of leaching to 84.2 percent in 168 days of leaching and rinsing, and represents the time required for the corresponding solution to ore ratio to be achieved.

Figure 13.6 shows the column-leach recovery for Jasperoid Zone sample, Composite 1, as a function of time. After approximately 168 days of leaching, the calculated recovery was 82 percent.

The figure also shows the calculated leach recovery rate for a 20-foot lift with an irrigation application rate of 0.003 gal/min/ft^2. Figure 13.6 illustrates the relationship between laboratory column-leach rates and the calculated leach cycle for comparable recovery on an operating heap, which for this sample may be 540 days and represents the time required for the corresponding solution to ore ratio to be achieved.

Silver recovery was determined for the Jasperoid Zone sample in a column-leach test and the same calculation that was used to estimate a gold recovery rate was used to estimate a silver recovery. Figure 13.6 illustrates an estimated silver recovery of 33.3 percent for a 20-foot lift after approximately 540 days of leaching.

Figure 13.6 Column-Leach Rate Profiles, Jasperoid Zone, - 80% - 3/4 inch Feed

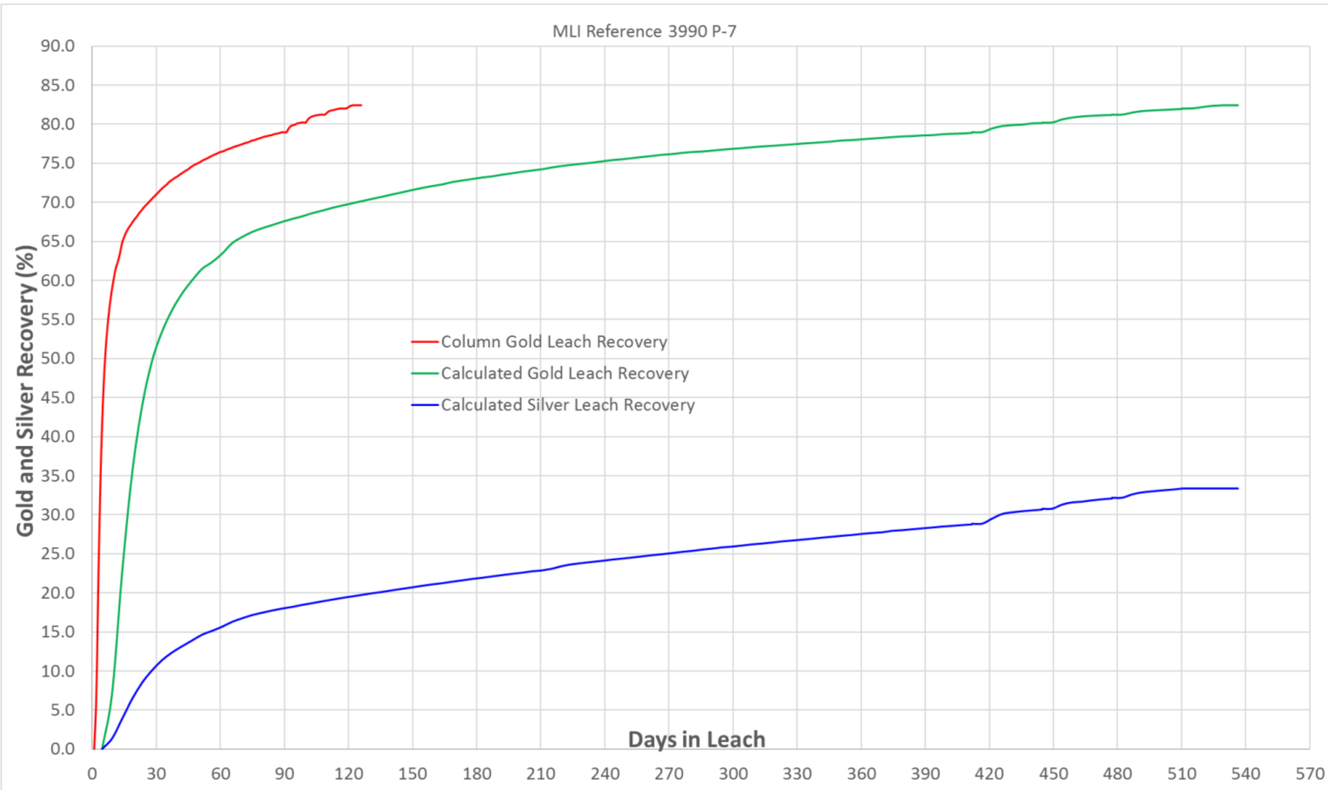


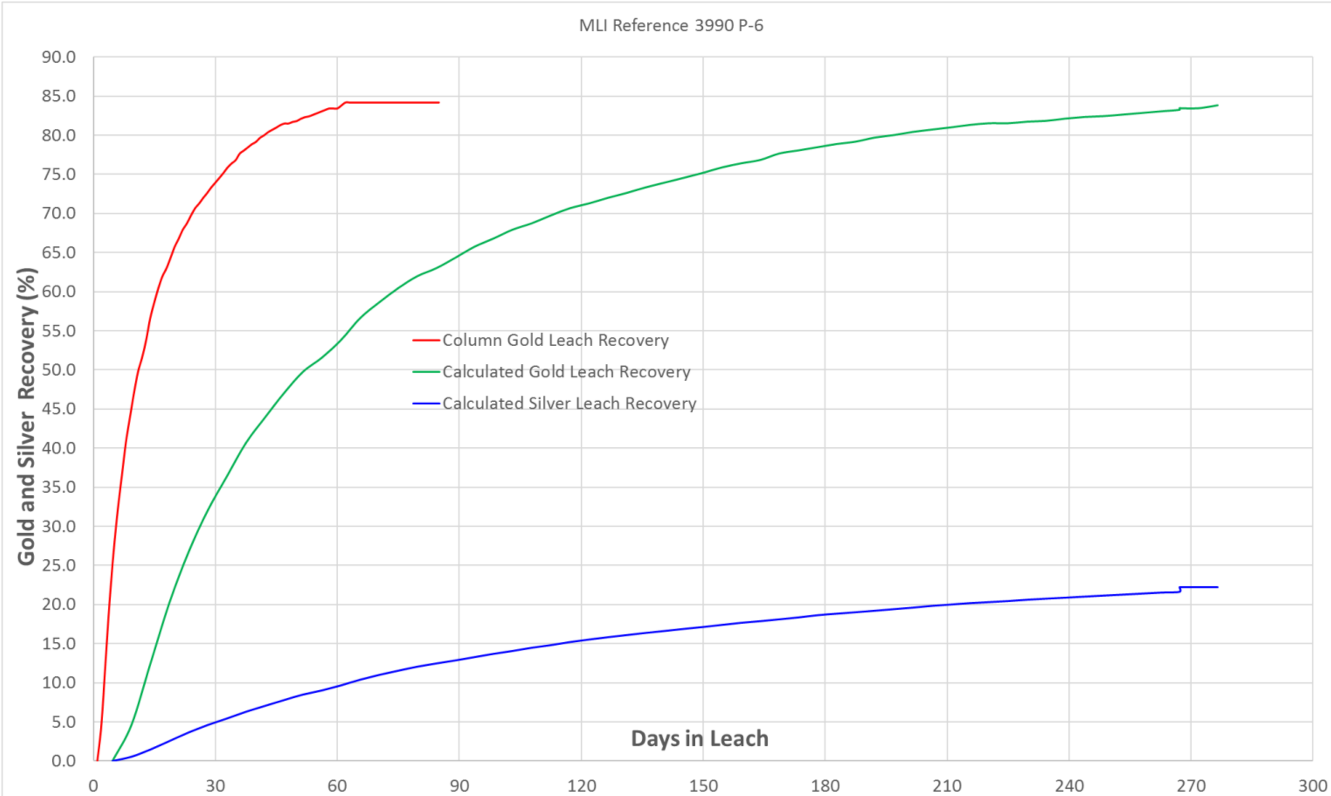
Figure 13.7 shows the column-leach recovery for Lower Zone sample, Composite 2, as a function of time. After approximately 90 days of leaching, the calculated recovery was 84 percent.

The figure also shows the calculated leach recovery rate for a 20-foot lift with an irrigation application rate of 0.003 gal/min/ft^2.

Figure 13.7 illustrates the relationship between laboratory column-leach rates and the calculated leach cycle for comparable recovery on an operating heap, which for this sample may be 270 days, and represents the time required for the corresponding solution to ore ratio to be achieved.

Silver recovery was determined for the Lower Zone sample in a column-leach test and the same calculation that was used to estimate a gold recovery rate was used to estimate the silver recovery. Figure 13.7 illustrates an estimated silver recovery of 21 percent for a 20-foot lift after approximately 270 days of leaching.

Figure 13.7 Column -Leach Rate Profiles, Lower Zone, - 80% - 3/4 inch Feed

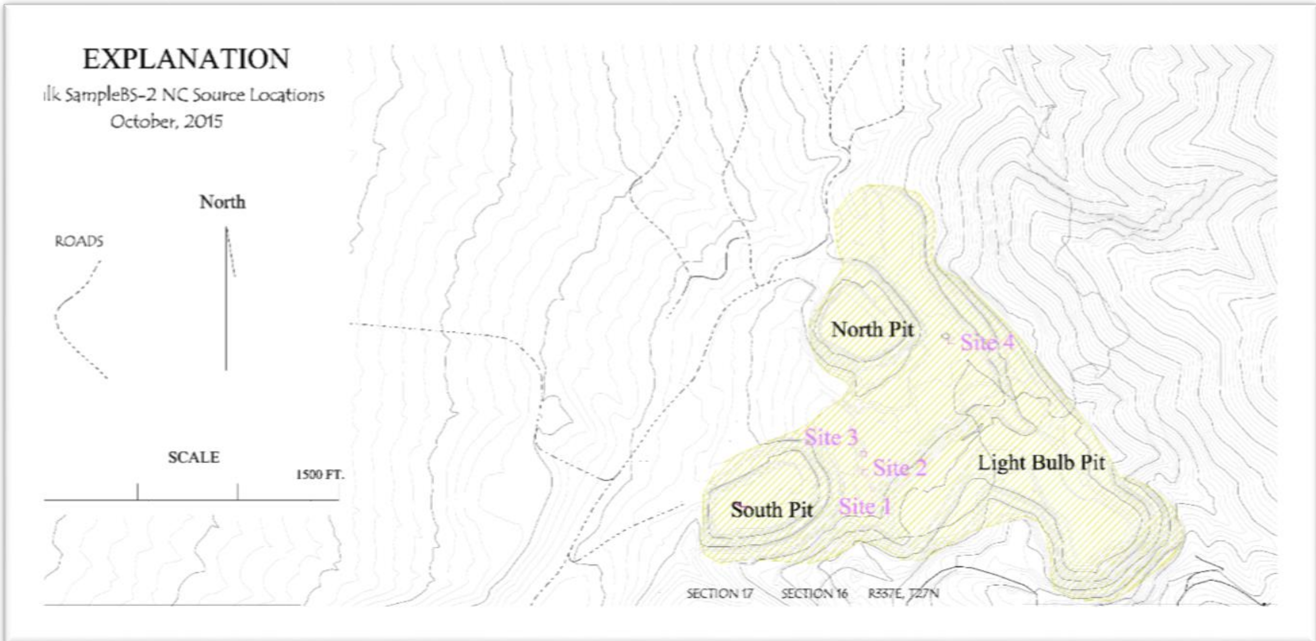


The results from the column leach tests for the Jasperoid and Lower zone samples for the ¾ inch feed size are higher than those achieved in the bottle-roll tests recoveries ranging from 76 to 84 percent. The bottle-roll tests indicated a recovery of 65 to 78 percent for the same feed size. The plus 90 percent recoveries for these zones for the minus 200 mesh bottle-roll tests indicate that the gold is not locked in sulfides or hindered by preg robbing carbon. Additionally, the recovery continues to increase over time for the column leach tests for the Jasperoid and Lower zone samples. Based on this information it is likely that the ultimate recovery is not dependent on feed size but rather on leach time. A finer feed size would likely decrease the leach time

13.4.2.2 Low Fines Bulk Sample

A metallurgical testing program was undertaken in October 2015 at MLI, using a bulk sample ("BS-2-NC") described as being a low fines content Main Zone sample, and on a Main Zone bulk sample rich in clay-size material (high fines), BS-1, stored from the earlier MLI testing program. A May 2016 memorandum (Prihar, 2016) described in detail the sampling program used to generate the BS-2-NC (low fines content) bulk sample. The sample was received at MLI for testing in November 2015. A map showing where the samples were taken is shown in Figure 13.8.

Figure 13.8 Locations of Source Material for Bulk Sample BS-2-NC, October 2015



Bottle-roll tests on the bulk sample BS-2-NC were completed at feed sizes ranging from 100 percent minus 2 inch to 80 percent minus 200M. Results from those tests were described in the 2016 MLI (Olson, 2016b). Bottle-roll testing showed that the low fines content bulk sample was readily amenable to agitated cyanidation treatment at the feed sizes tested and was not particularly sensitive to crush size with respect to gold recovery. Gold recoveries obtained from the short term (4 day) bottle-roll tests, at feed sizes ranging from 80 percent minus 2 inch to 80 percent minus 10M, ranged from 74.9 percent to 81.7 percent. Gold recovery obtained at the 80 percent minus 200M feed size was somewhat higher (88.5 percent). Bottle test reagent requirements were very low. Cyanide consumptions were 0.20 lb NaCN/ton or lower. Lime additions ranged from 1.4 to 3.1 lbs/ton.

A column leach test was conducted on the bulk sample BS-2-NC on a feed size of 100 percent minus 6 inch, which was not agglomerated. The purpose of this column test was to estimate the recovery for a run-of-mine ("ROM") heap leach.

Column-leach tests were also performed on the bulk sample BS-2-NC at 100 percent passing 1.5 inches, which was agglomerated.

Column-leach tests were performed on what was considered a low fines and high fines mix. This sample was generated by mixing 70 percent of the low fines material, which was from the bulk sample BS-2-NC and 30 percent of the high fines material, which was from the bulk sample BS-1. The mixed material was agglomerated.

Table 13.10 shows the results of the bulk sample BS-2-NC column-leach tests and the blended BS-2-NC and BS-1 samples.

Table 13.10 MLI 2015 Column-leach Tests BS-2-NC AND WITH BS-1 BLENDED

Sample	Description	Feed Size	Au Recov. (%)	Ag Recov. (%)	Calc. Head, (oz Au/ton)	Solut. to Ore Ratio	Leach Time (days)	Reagent Requirements	
								NaCN Usage (lbs/ton)	Cement Added (lbs/ton)
BS-2-NC	Low Fines Bulk Sample	100% -6 inch	74.2	12.5	0.015	5.4:1	213	0.84	No Cement 1.9 lbs/ton Lime
BS-2-NC	Low Fines Bulk Sample	100% -1.5 inch	87.1	-	0.015	1.3:1	76	0.46	8.0
Blend	70% Low Fines Bulk Sample BS-2-NC and 30% High Fines Bulk Sample BS-1 (All Agglomerated)	100% -3 inch	87.3	28.6	0.017	1.9:1	71	0.25	8.0
Blend	70% Low Fines Bulk Sample BS-2-NC and 30% High Fines Bulk Sample BS-1. Fine and coarse screened. Fine material agglomerated.	100% -3 inch	86.6	-	0.016	1.9:1	71	0.26	8.0

The low fines bulk sample BS-2-NC non-agglomerated -6 inch feed had a gold recovery of 74.2 percent in 213 days of leaching. The previous BS-1 sample contained greater than 25 percent minus 200 M material was not amenable to column leaching. The lower fines content of less than 25 percent for BS-2-NC provided permeability for the test. Cyanide consumption was 0.84 pounds per ton and 1.9 pounds per ton of lime was used to maintain alkalinity. The calculated leach time for this ore would be 368 days, which would be the time required to achieve a solution to ore ratio of 5.4:1 based on a 20-foot lift with an irrigation application rate of 0.003 gal/min/ft^2.

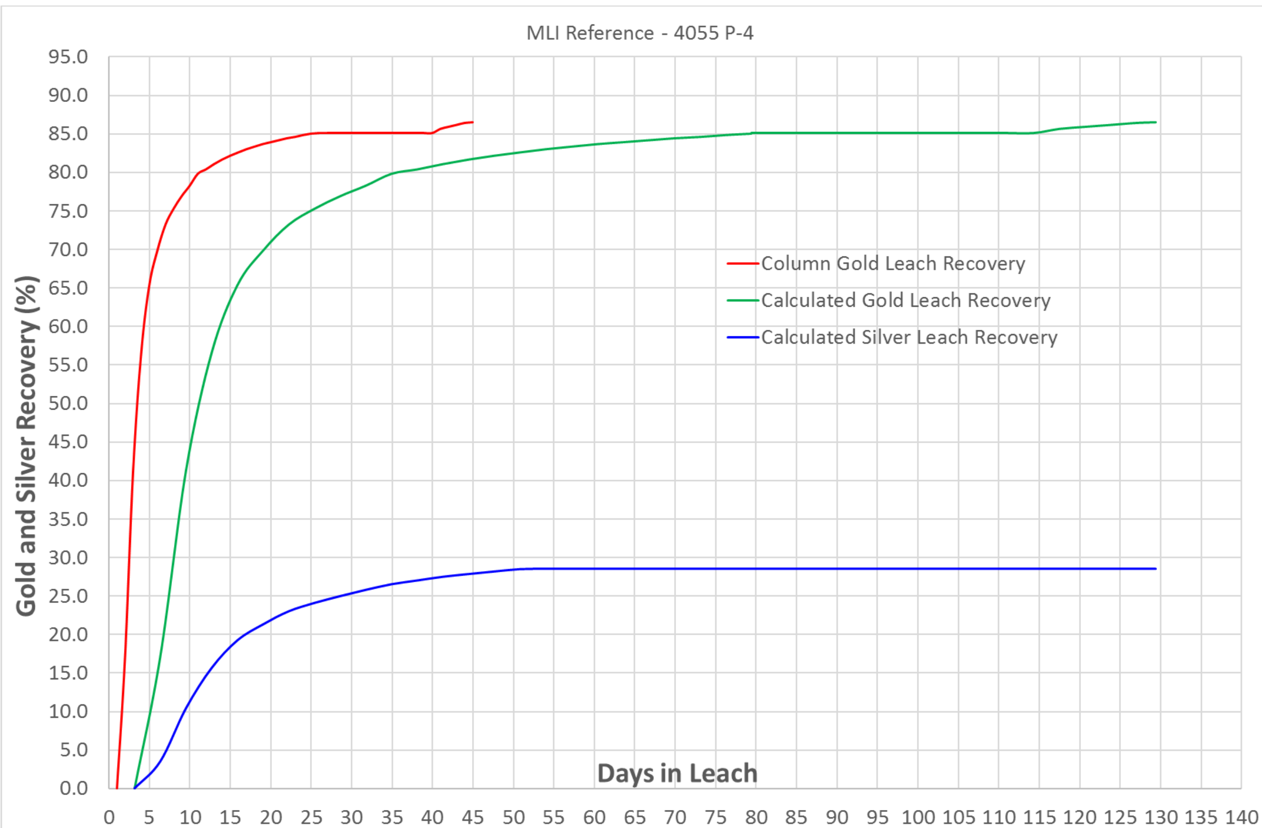
Column test gold recovery obtained from the low fines content sample, at a 100 percent minus 1.5-inch feed size, was 87.1 percent, in 76 days of leaching and rinsing. Gold recovery rate was very rapid. Cyanide consumption was 0.46 lbs/ton of ore. This sample was agglomerated before leaching, using a cement addition of 8 lbs/ton.

Column test gold recoveries obtained from the 70/30 weighted blend of low fines content and high fines content samples, at a 100 percent minus 3 inch feed size were 87.3 percent and 86.6 percent. Both of these feeds were agglomerated using a cement addition equivalent to 8.0 lbs/ton. In one case, the entire column test feed was agglomerated. In the other case, only the minus 2.0-inch material (removed by dry screening) from the minus 3.0-inch feed was agglomerated, and then recombined with the coarser material before leaching. The testing showed no benefit to recovery from the screening-agglomeration-recombination option.

Figure 13.9 shows the column leach recovery for a mix of 70 percent low fines BS-2-NC and 30 percent high fines BS-1 as a function of time. After approximately 71 days of leaching, the calculated recovery was 87 percent.

Figure 13.9 also shows the calculated leach recovery rate for gold and silver for a 20-foot lift with an irrigation application rate of 0.003 gal/min/ft^2. Figure 13.9 illustrates and estimated gold recovery of 87 percent for a 20-foot lift after approximately 130 days of leaching.

Figure 13.9 Leach Rate Profiles, Bulk Sample BS-2-NC 70/30 Low Fines High Fines Mix Column Leach 100% - 3 inch Feed



Silver recovery was determined for a mix of 70 percent low fines BS-2-NC and 30 percent high fines BS-1 sample in a column-leach test and the same calculation that was used to estimate a gold recovery rate was used to estimate the silver recovery. Figure 13.9 illustrates an estimated silver recovery of 28.6 percent for a 20-foot lift after approximately 130 days of leaching.

13.5 Variability Tests

In January of 2018, Pershing Gold identified 38 samples for variability testing (McClelland, April 2018). The purpose of the variability tests was to confirm the recoveries obtained from the composites used in the column leach testing. The samples were identified in cross-sections from the pit design completed for the PFS. An attempt was made to represent different ore zones, including Main, Lower and Jasperoid, at different elevations in the pit as well as along a north to south strike. The samples were collected and collated according to the design Phase 1, Phase 2 and Phase 3 mining pits. Each of these phases refers to a different mining sequence at an increasing depth.

After a sample interval was identified in the cross-section, the availability of the sample in the core shed was determined. Although a number of intervals were identified, only 38 samples were actually available.

The core samples were collected from a number of different drilling campaigns starting in 2012 through 2015. The samples were assembled from the core shed at site and shipped to MLI for sample preparation and bottle-roll testing.

Bottle-roll testing was conducted at a 10M grind. Tests were charged with 1.0 gram per liter of cyanide. Lime was added to the sample as required to maintain alkalinity.

The maximum depth of the variability samples that were taken for the design Phase 1 pit was at an elevation of 5100 feet. The Phase 1 pit is primarily in Main Zone ore, with minor amounts of Lower and Jasperoid ores.

The design Phase 2 pit is a combination of ores from the Main, Lower and Jasperoid zones. The maximum depth of the variability samples that were taken for this phase was at an elevation of 4750 feet.

The design Phase 3 pit is also a combination of ores from the Main, Lower and Jasperoid zones. The maximum depth of the variability samples that were taken for this phase is the same as for the Phase 2 pit, but the pit has expanded to the west and is several hundred feet deeper.

13.5.1 Mine Design Phase 1 – Variability Bottle-Roll Tests

Table 13.15 lists the results for the 2018 variability tests for the design Phase 1 pit. All of the samples tested were from the Main Zone ore. The recoveries for the calculated head varied between 61.5 percent and 93.8 percent. Average elevations for the samples varied between 5100 and 5370 feet.

A distinct anomaly was noted in Sections 10800 through 10950 of the pit, where recovery dropped from an average of plus 80 percent to the mid 60's. Consultation with exploration geologists and pit designers identified approximately 600,000 tons of Main Zone ore represented by these samples that would have lower recovery. Consequently, a recovery of 65 percent was assigned to this area.

The bottle-roll tests for the remainder of the Main Zone ore in the design Phase 1 pit had a weighted head average recovery of approximately 85 percent. These samples would have a direct comparison to the material represented in the two bulk samples, BS-2 and BS-NC-2, that were taken in 2015. The bottle-roll tests for these bulk samples had recoveries of 81.3 and 81.7 percent, respectively. As shown in Tables 13.9 and 13.10, the column-leach test recoveries for these bulk samples was approximately 87 percent. The results of the variability bottle rolls for the Main Zone in Sections 9550 through 10750 confirmed the recoveries achieved in the column tests for BS-1 and BS-NC-2 and the applicability of the estimated recovery to this ore in this section of the pit.

Table 13.11 MLI 2018 Variability Tests – Phase 1 Design Pit

Sample Information					Gold Recovery Information								
Composite	Drill Hole	Drill Interval from	Drill Interval to	Calc'd Head Recovery	Measured Head Recovery	Extracted	Tail	Calc'd Head	Measured Head	Drill Assay Head	Drill Hole AA/Fire	Section (South to North)	Mineralogical Zone
		(ft)	(ft)	%	%	(opt)	(opt)	(opt)	(opt)	(opt)			
4293-011	RC16-477	153	181	75.0	76.2	0.015	0.005	0.020	0.021	0.023		9550	Main
4293-015	RC15-442	5	68	89.5	87.5	0.017	0.002	0.019	0.016	0.045	98%	9700	Main
4293-002	RC14-213	75	130	82.4	82.4	0.014	0.003	0.017	0.017	0.018	90%	9800	Main
4293-008	RC14-239	40	120	90.9	91.3	0.020	0.002	0.022	0.023	0.023	93%	10500	Main
4293-004	RC14-222	121	166	94.7	93.8	0.018	0.001	0.019	0.016	0.016	86%	10600	Main
4293-009	RC14-242	45	82	84.2	87.0	0.016	0.003	0.019	0.023	0.029	81%	10650	Main
4293-007	RC14-236	65	165	80.8	83.3	0.021	0.005	0.026	0.030	0.038	63%	10700	Main
4293-001	RC-14-211A	85	144	81.3	77.8	0.18	0.060	0.024	0.27	0.020	80%	10750	Main
4293-005	RC14-226	50	75	68.8	72.2	0.011	0.005	0.016	0.018	0.019	85%	10800	Main
4293-012	RC14-160	75	116	68.0	68.0	0.017	0.008	0.025	0.025	0.024	75%	10850	Main
4293-006	RC14-234	63	115	61.5	66.7	0.008	0.005	0.013	0.015	0.017	65%	10950	Main

13.5.2 Mine Design Phase 2 – Variability Bottle-Roll Tests

Table 13.12 lists the results for the variability tests for the Phase 2 pit. The Phase 2 Pit contained a mixture of Main, Lower and Jasperoid zone ores. Recoveries were evaluated taking into account elevation and sectional distribution. Average elevations for the samples varied between 4750 and 5120 feet.

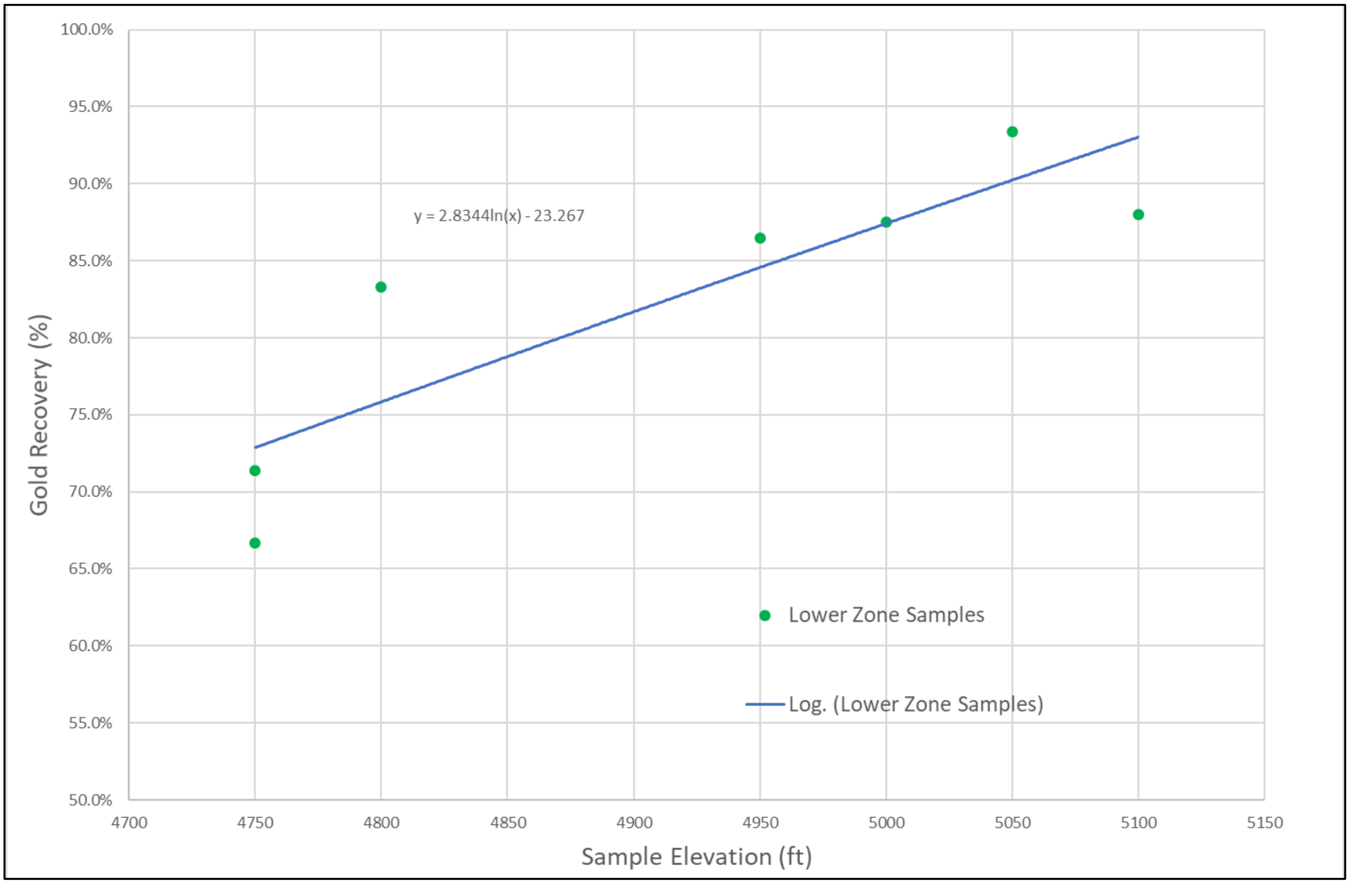
Table 13.12 MLI 2018 Variability Tests – Phase 2 Design Pit

Sample Information					Gold Recovery Information								
Composite	Drill Hole	Drill Interval from	Drill Interval to	Calc'd Head Recovery	Measured Head Recovery	Extracted	Tail	Calc'd Head	Measured Head	Drill Assay Head	Drill Hole AA/Fire	Section (South to North)	Mineralogical Zone
		(ft)	(ft)	%	%	(opt)	(opt)	(opt)	(opt)	(opt)			
4293-014	RC14-221A	218	265	83.3	83.3	0.010	0.002	0.012	0.012	0.016	93%	10200	Main
4293-019	RC14-255	150	197	83.3	84.4	0.025	0.005	0.030	0.032	0.038	73%	10450	Main
4293-018	RC14-242	266	362	86.5	85.7	0.032	0.005	0.037	0.035	0.023	85%	10650	Lower
4293-033	RC14-240	409	463	66.7	65.1	0.044	0.022	0.066	0.063	0.095	84%	10700	Lower
4293-017	RC14-211A	262	326	93.4	92.5	0.085	0.006	0.091	0.080	0.147	92%	10750	Lower
4293-023	RC15-369	355	390	83.3	85.7	0.025	0.005	0.030	0.035	0.036	75%	10800	Lower
4293-024	RC15-264	278	315	87.5	88.2	0.014	0.002	0.016	0.017	0.043		10850	Lower
4293-027	RC15-272	238	259	87.3	87.5	0.048	0.007	0.055	0.056	0.067		10950	Main
4293-022	RC15-300	213	252	88.0	89.1	0.044	0.006	0.050	0.055	0.044	102%	11050	Lower
4293-016	RC13-135	311	360	71.4	74.2	0.020	0.008	0.028	0.031	0.025	74%	11100	Lower
4293-035	RC15-284	398	451	71.4	72.4	0.020	0.008	0.028	0.029	0.029	80%	11150	Jasperoid
4293-026	RC15-268	246	326	86.2	88.9	0.025	0.004	0.029	0.036	0.031		11250	Jasperoid
4293-025	RC15-266	267	308	92.9	93.1	0.026	0.002	0.028	0.029	0.046	43%	11350	Jasperoid
4293-020	RC15-273	290	333	76.7	76.2	0.033	0.010	0.043	0.042	0.037	91%	11400	Jasperoid
4293-028	RC13-121	334	456	71.1	74.5	0.032	0.013	0.045	0.051	0.049		11450	Jasperoid
4293-021	RC15-278	268	323	78.0	81.4	0.046	0.013	0.059	0.070	0.039	79%	11550	Jasperoid

There were three Main Zone samples in the design Phase 2 pit. The recoveries for these three samples averaged approximately 85 percent. This recovery, which is similar to the behavior in the Main Zone section in the Phase 1 pit, confirms applicability of the column-leach recoveries achieved in the column tests for BS-1 and BS-NC-2 for this ore in this section of the pit.

The bottle-roll recoveries for the Lower Zone samples in the design Phase 2 pit had a weighted average of 83 percent, which is greater than the 2015 composite bottle-roll recovery of 70 and 79 percent shown in Table 13.8 but supports the column-leach test recovery for the 2015 composite of 84 percent. Lower Zone recovery was also analyzed as a function of elevation and sectional location. There was no relationship between sectional location and recovery. There was a relationship between elevation and recovery, which is illustrated in Figure 13.10.

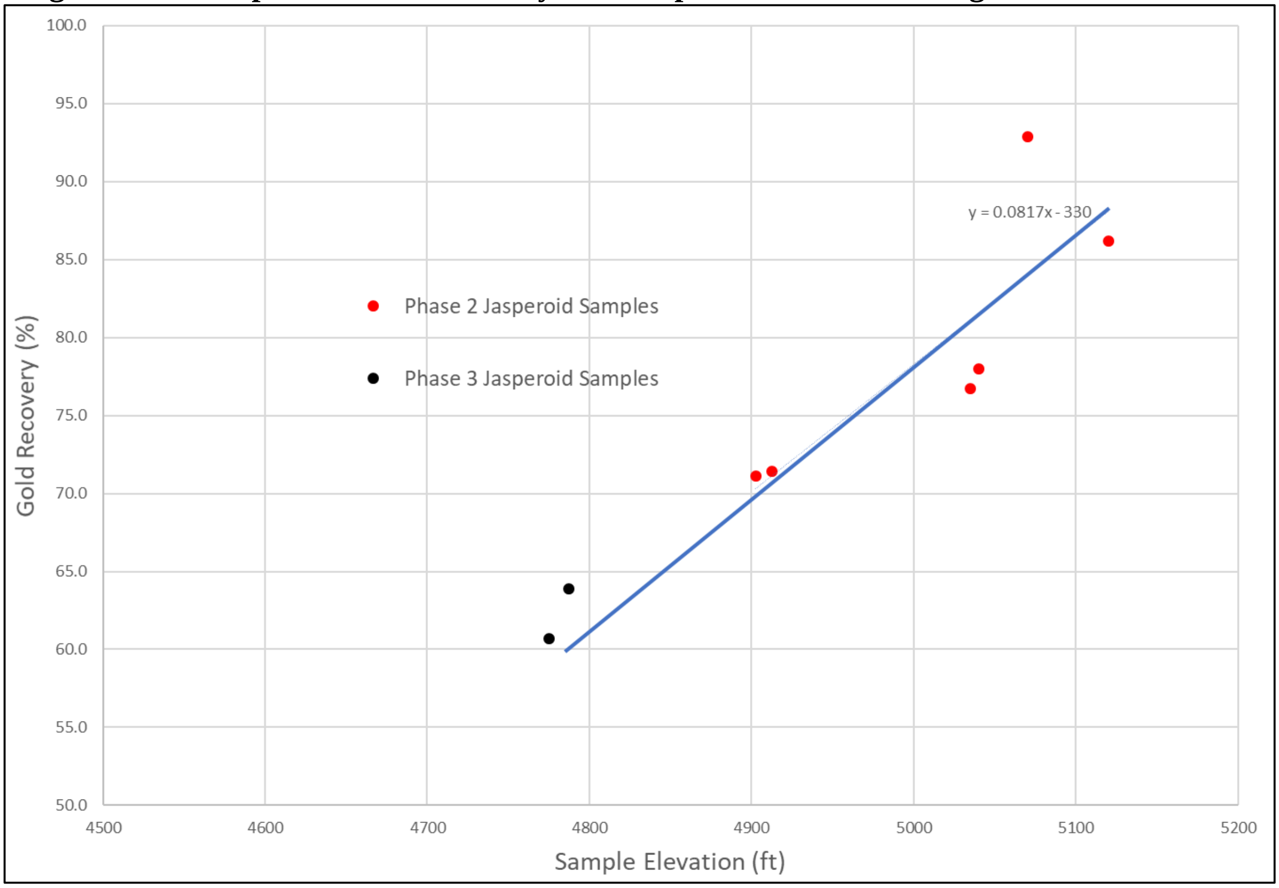
Figure 13.10 Lower Zone Recovery vs. Sample Elevation for design Phase 2 and 3 Pits



A weighted recovery for the Lower Zone for design Phases 2 and 3 was calculated using the linear relationship for recovery shown in Figure 13.10 and the tons at given elevations, which resulted in an overall recovery of 80.5 percent. This result also compares favorably with the bottle-roll leach tests that were performed in 2015, as shown in Table 13.8 that achieved recoveries of 70 and 83 percent and with the corresponding column-leach tests that achieved 76 and 84 percent gold recovery.

The bottle-roll recoveries for the Jasperoid Zone samples in the design Phase 2 and Phase 3 pit were analyzed for recovery as a function of sectional location and elevation. There was no relationship between recovery and sectional location. There was a relationship between recovery and elevation, which is illustrated in Figure 13.11.

Figure 13.11 Jasperoid Zone Recovery vs. Sample Elevation for Design Phase 2 and 3 Pits



A grade-weighted recovery for the Jasperoid Zone, for mining pit design Phases 2 and 3, was calculated using the linear relationship for recovery shown in Figure 13.11, and the tons and grade at given elevations, which resulted in an overall recovery of 70.1 percent. This result compares favorably with the bottle-roll leach tests performed in 2015, shown in Table 13.8, that achieved recoveries of 70 and 79 percent with the corresponding column-leach tests achieving an 82 percent gold recovery.

13.5.3 Mine Design Phase 3 – Variability Bottle-Roll Tests

Table 13.13 lists the results for the variability tests for the Phase 3 pit. The number of drill core samples that were available in the proposed design Phase 3 pit were limited.

Table 13.13 MLI 2018 Variability Tests – Design Phase 3 Pit

Sample Information					Gold Recovery Information					Sample Location and Information				
Composite	Drill Hole	Drill Interval from	Drill Interval to	Calc'd Head Recovery	Measured Head Recovery	Extracted	Tail	Calc'd Head	Measured Head	Drill Assay Head	Drill Hole Aa/Fire	Section South to North	Avg Elev	Mineralogical Zone
		(ft)	(ft)	%	%	(opt)	(opt)	(opt)	(opt)	(opt)		(ft above amsl)		
4293-032	RC14-222	537	573	71.4	83.3	0.020	0.008	0.028	0.048	0.084	82%	10600	4750	Lower
4293-037	RC14-234	519	563	60.7	52.7	0.068	0.044	0.112	0.093	0.056	75%	10950	4775	Jasperoid
4293-029	RC14-017	226	286	78.3	81.5	0.018	0.005	0.023	0.027	0.037		11000	4990	Main
4293-030	RC14-028	229	295	78.6	81.3	0.011	0.003	0.014	0.016	0.015		11350	4995	Main
4293-036	RC14-310	466	504	63.9	67.5	0.023	0.013	0.036	0.040	0.033	82%	11500	4788	Jasperoid

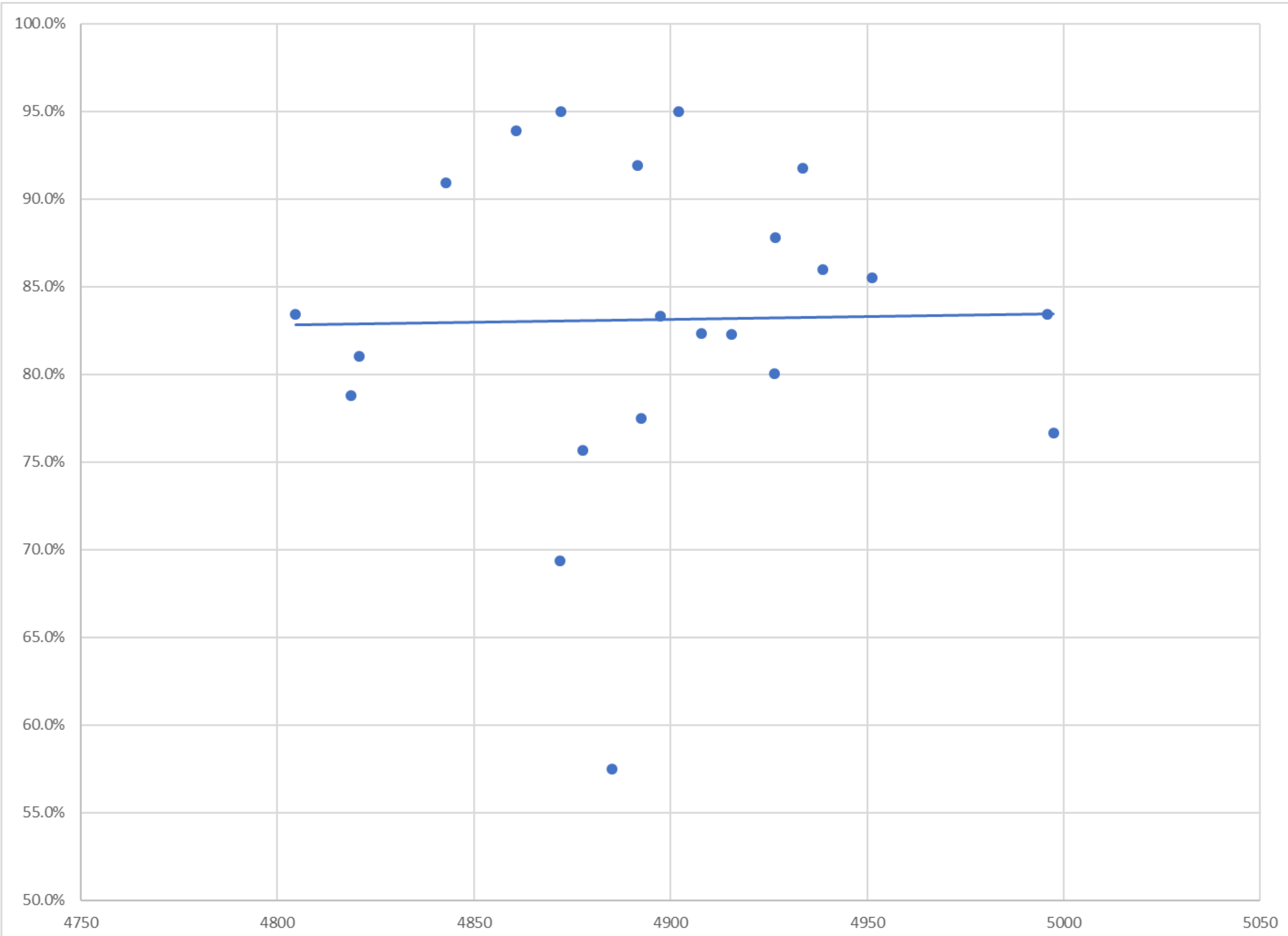
The Lower Zone bottle-roll recoveries for the design Phase 3 pit are illustrated in Figure 13.10. These results were combined with the Phase 2 pit bottle-roll tests and as explained in Section 13.5.2 support the column leach test recovery results of 70 and 84 percent gold recovery.

The Jasperoid Zone bottle-roll recoveries for the design Phase 3 pit are illustrated in Figure 13.11. These results were combined with the Phase 2 pit bottle-roll tests and, as explained in Section 13.5.2, support the column-leach test results of 82 percent gold recovery.

The Main Zone bottle-roll average recoveries of 78.5 percent in the design Phase 3 pit represent approximately 3.1 million tons of ore at an average elevation of 4990 feet located north of Section 11000. These recoveries were adjusted by 3.5 percent to reflect the difference between the bottle-roll tests and column-leach test recoveries achieved for the Main Zone ore in the composites that were completed in 2014. A recovery of 82 percent was used for these tons.

The design Phase 3 pit contains approximately 3.2 million tons of Main Zone ore that was not represented by variability samples. The ore is located between Sections 7990 to 8620 at an elevation of 4800 to 5000. The AA/Fire assay ratios are a means of determining the cyanide soluble portion of mineralized material. The AA/Fire ratios were obtained for mineralized zones for the drill holes that were in this area and are plotted as a function of elevation as illustrated in Figure 13.12. The figure shows that while the AA/Fire ratios for this area have a wide range of scatter, the trend for gold recovery would be in the mid to low 80s percent range, which would indicate that the recovery from this ore would be similar to the Main Zone ore represented by the bulk samples BS-1 and BS-NC-2.

Figure 13.12 Design Phase 3 Pit Main Zone Ore - South Area Drill Hole AA/Fire Ratios (Scheduling Phase 6 and 7)



13.5.4 Silver Recoveries – Variability Bottle-Roll Tests

The variability bottle-roll tests were used to determine if the recoveries selected for silver from the column-leach test results could be projected through the three different ore zones. Analysis of the variability tests indicated that silver recovery could not be correlated with sectional location in the pit or with elevation in the pit. Silver recovery appeared to be random.

The Main Zone weighted-average silver recovery was 41 percent for the 10M bottle-roll variability tests, based on the calculated head grades. The variability test silver recoveries ranged from 7.1 to 75 percent, indicating a high degree of variability. Table 13.9 shows that the silver recovery for the BS-1 column-leach test was 44.1 percent. The 10M bottle-roll test for the BS-1 bulk sample was 45.4 percent. The variability tests appear to support the average estimated silver recovery for the column test.

The Lower Zone weighted-average silver recovery for the 10M bottle-roll variability tests, based on the calculated head, was 38 percent. The variability test silver recoveries ranged from 20 to 77 percent, indicating a high degree of variability. Table 13.9 shows that the silver recovery for the Composite 2 column-leach test was 22 percent. The 10M bottle-roll test for this column test was 40 percent. The variability tests appear to support the average estimated silver recovery for the column test.

The Jasperoid Zone weighted-average silver recovery for the 10M bottle-roll variability tests, based on the calculated head, was 41 percent. The variability test silver recoveries ranged from 22 to 52 percent. Table 13.9 shows that the silver recovery for the Composite 3 column leach tests was 33 percent. The 10M bottle-roll test for Composite 3 was 36 percent. The variability tests appear to support the average estimated silver recovery for the column test.

13.5.5 Mixed Ore and Sulfide Ore Recovery

The current mine resource model identifies several areas in the deep part of the north pit were gold grades are sufficient to warrant inclusion despite the mixed amounts of sulfide and expected poor recovery from the ore. There are only 160,000 tons of mixed material grading 0.085 oz Au/ton, and 124,000 tons of sulfide material grading 0.077 oz Au/ton contained in the Feasibility final pit. A 0.025 oz Au/ton cutoff grade was used for mixed and sulfide materials. Nearly all of this material is contained in the Jasperoid Zone.

The AA/Fire ratios were obtained for mineralized zones for the drill holes that were in this area. The AA/Fire ratios for this area have a wide range of scatter. By inspection, the gold recovery for these ores would be in the mid to low 50’s.

13.6 Hydraulic Conductivity Testing Summary

A total of 47 fixed-wall hydraulic conductivity (“load/permeability”) test series were conducted on samples of Relief Canyon mineralized material. A total of 32 of these samples were completed during the 2014 to 2016 metallurgical testing programs. The purpose for these tests was to evaluate permeability characteristics of the material with different cement additions and a range of compressive loadings expected during commercial, multi-lift heap leaching.

An additional 15 samples were tested at two laboratories in early 2018 for permeability characteristics at a fixed reagent addition rate of 8 lbs/ton of cement, but with varying concentrations of fines material from the Main Zone bulk samples, BS-1 and BS-NC-2. The purpose of these tests was to confirm the effects of blending different amounts of high fines material and to compare different analytical procedures employed by two laboratories.

13.6.1 Load/Permeability Tests 2014 – 2016

Summary results from the load/permeability tests from 2014 to 2016 are presented in Table 13.14. The table includes the equivalent heap stack height (proportional to the applied compressive loading) where the measured hydraulic conductivity was equal to an assumed heap solution application rate of 0.005 gpm/ft^2 (“1x”), and ten times this rate (“10x”). The planned field solution application rate is 0.003 gpm/ft^2.

Table 13.14 Fixed-Wall Hydraulic Conductivity (Load/Permeability) Tests 2014 -2106

Sample	Zone	Leach Residue	Feed Size		Cement, lb/ton	Maximum Stack Depth for		Job/Test #	Geo lab
			Nominal	% - 200M		1 x App. Rate	10 x App. Rate		
3841 Load/Perm Bulk Comp.	Main	No	80%-3/4"	20	4.0	>200'	>200'	3841 LP-1	GLA
3841 Load/Perm Bulk Comp.	Main	No	80%-3/4"	20	6.0	>200'	176'	3841 LP-2	GLA
Reconstituted Load/Perm Comp.	Main	No	80%-3/4"	35[1)]	4.0	157'	80'	3842 LP-3	GLA
Reconstituted Load/Perm Comp.	Main	No	80%-3/4"	35[1)]	6.0	>200'	90'	3842 LP-4	GLA
Reconstituted Load/Perm Comp.	Main	No	80%-3/4"	35[1)]	10.0	>73'	44'	3843 LP-5	GLA
Drill Core Comp. 3841-3 (CMBXH)	Main	Yes	80%-3/4"	37	8.0	87'	55'	3841 P-3	GLA
Drill Core Master Comp.	Main	Yes	80%-3/4"	23	8.0	>200'	>200'	3841 P-8	GLA
Dump 4 Bulk	N/A	No	100%-2"	27	0.0	29'	18'	4068 LP-5	GLA
Dump 4 Bulk	N/A	No	100%-2"	27	2.0	23'	16'	4068 LP-4	GLA
Dump 4 Bulk	N/A	No	100%-2"	27	4.0	>120'	99'	4068 LP-3	GLA
Dump 4 Bulk	N/A	No	100%-2"	27	6.0	>82'	64'	4068 LP-2	GLA
Dump 4 Bulk	N/A	No	100%-2"	27	8.0	>128'	101'	4068 LP-1	GLA
BS-1 Bulk Sample	Main	No	100%-3"[2)]	26	8.4	29'	19'	3990 LP-3	ASWT
BS-1 Bulk Sample	Main	No	100%-3"[2)]	26	9.4	62'	40'	3990 LP-2	ASWT
BS-1 Bulk Sample	Main	No	100%-3"[2)]	26	10.5	171'	105'	3990 LP-1	ASWT
Drill Core Comp. 3990-1	Jasperoid	Yes	80%-3/4"	17	8.0	Failed[3)]		3990 P-5	GLA
Drill Core Comp. 3990-3	Jasperoid	Yes	80%-3/4"	16	8.0	96'	54'	3990 P-7	GLA
BS-2 Bulk Sample	Main	Yes	100%-1.5"	22	8.0	>200'	>200'	4055 P-2	GLA
High Fines/Low Fines Blend (30%/70%)	All	No	100%-2"	14	5.0	>200'	>200'	4055 LP-1	GLA
High Fines/Low Fines Blend (10%/90%)	All	No	100%-2"	10	0[4)]	>200'	>200'	4055 LP-2	GLA
High Fines/Low Fines Blend (10%/90%)	All	No	100%-2"	10	3.0	>200'	>200'	4055 LP-3	GLA
High Fines/Low Fines Blend (40%/60%)	All	No	100%-2"	16	5.0	>200'	>200'	4055 LP-4	GLA
High Fines/Low Fines Blend (50%/50%)	All	No	100%-2"	17	5.0	>200'	>200'	4055 LP-5	GLA
High Fines/Low Fines Blend (60%/40%)	All	No	100%-2"	19	5.0	>200'	>200'	4055 LP-6	GLA
High Fines/Low Fines Blend (20%/80%)	All	No	100%-2"	22	5.0	>200'	>200'	4055 LP-7	GLA
BS-1 Bulk Sample	Main	No	100%-2"	26	5.0	>200'	>200'	4055 LP-8	GLA
BS-1 Bulk Sample	Main	No	100%-2"	26	8.0	>200'	>200'	4055 LP-9	GLA
BS-1 Bulk Sample	Main	No	100%-2"	26	3.0	>200'	>200'	4055 LP-10	GLA
BS-1 Bulk Sample	Main	No	100%-2"	26	4.0	>200'	>200'	4055 LP-11	GLA
BS-1 Bulk Sample	Main	No	100%-2"	26	4.0	64'	44'	4055 LP-12	NF
BS-1 Bulk Sample	Main	No	100%-2"	26	5.0	64'	44'	4055 LP-13	NF
BS-1 Bulk Sample	Main	No	100%-2"	26	8.0	129'	87'	4055 LP-14	NF

1) The 3841 Load/Perm composite was reconstituted in order to produce a feed with a higher fines content.
2) The 100%-3" feed was produced by crushing to 100%-3", screening to separate -2" material, agglomerating -2" material and recombining with screened -3"+2" material.
3) Failed to meet the criteria at any of the compressive loads evaluated.
4) Sample combined with 1.5 lb/ton lime, no agglomeration.

Two column leached residues, 3841 Load/Perm Bulk Comp. – Main Zone, were tested at the end of the 2014 metallurgical testing program to evaluate the expected overall average blend of mineralization types from the Main Zone. Test results indicated that the weighted blend of the Main Zone mineralization types performed satisfactorily using 4.0 and 6.0 lbs/ton cement addition rates. Hydraulic conductivity was higher than the planned solution application rate (0.003 gpm/ft^2) at simulated heap stack heights of greater than 200 feet. The design stack height is 140 feet, while the permitted maximum stack height is 200 feet.

The Main Zone Drill-Core Comp. 3841-3, CMBXH, was agglomerated with 8.0 lbs/ton of cement, but performed poorly having a maximum stack depth of 87 feet at the planned application rate. This composite had the highest fines content of any of the samples tested (37 percent -200M).

Column leached residues from samples representing material from the Jasperoid Zone (Comps. 3990-1 and 3990-3) were selected for evaluation during the 2015 metallurgical testing program because they both had a higher fines content (16 percent to 17 percent passing 200M) than the other samples being evaluated during that program. Both samples were evaluated at an 80 percent passing ¾ inch feed size, with a cement binder addition rate of 8.0 lbs/ton. Composite 3990-1 failed to achieve a hydraulic conductivity equivalent to at least the planned solution application rate, at the lowest compressive load (simulated heap stack height) evaluated. Results for composite 3990-3 indicated a hydraulic conductivity equivalent to the planned solution application rate was maintained to a simulated heap stack height of only 97 feet.

Sample BS-2-NC was described as a low fines content sample and was tested at a 100 percent minus 1.5 inch feed size, using a cement binder addition of 8.0 lbs/ton. That leached residue performed well, and the hydraulic conductivity was equivalent to greater than 10 times the planned solution application rate at simulated heap stack heights of greater than 200 feet (Olson, 2016a).

All other load/permeability testing was conducted on freshly prepared samples or samples that had not been previously leached.

Testing was conducted in 2014 on a 3841 load/perm bulk composite, which was sampled to represent material with a high fines content. These tests were conducted at an 80 percent passing ¾ inch feed size. Initial testing on that composite showed hydraulic conductivities in excess of 10 times the planned solution application rate, to simulated heap stack heights of greater than the planned 140 feet, after being agglomerated using a cement binder addition of 6.0 lbs/ton. Screen analysis of that sample showed that it contained a somewhat lower quantity of fines (20 percent passing 200M) than was expected. Consequently, the composite was reconstituted, by screening, splitting size fractions and recombining material, in a manner to give the sample a significantly higher fines content (35 percent passing 200M). Results from testing on the reconstituted sample, using 4.0 and 6.0 lbs/ton cement addition rates, indicated the expected improvement in permeability with increasing cement addition. Results from the test conducted using a 10 lbs/ton cement addition rate were anomalous and suspect.

A bulk sample, identified as "Dump 4" material was received by MLI for testing in November 2015. The sample was crushed to 2.0 inches and subjected to hydraulic conductivity testing after agglomeration pretreatment using varied cement additions. A head screen analysis was also conducted on the material to determine gold content and distribution as well as particle size distribution. The average assayed head grade of the Dump 4 bulk sample was 0.011 oz Au/ton. Hot cyanide shake test results indicated that the contained gold was readily cyanide soluble. The Dump 4 material was not tested for recovery with either bottle-roll tests or column tests.

The Dump 4 location would coincide with the southern trenches where the BS-1 bulk sample was taken. These trenches are illustrated in Figure 13.4 as Trenches SWD #2, #3 and #12. Testing was conducted on a Dump 4 bulk sample, at a 100 percent minus 2.0 inch feed size, using cement binder additions ranging from zero (not agglomerated) to 8.0 lbs/ton. That sample had a fines content of 27 percent passing 200M. Results indicated hydraulic conductivity at standard application rates increased from a maximum stack depth of 29 feet with no cement binder, to 128 feet with the addition of 8.0 lbs/ton of cement binder**.**

Testing was conducted on the bulk sample (BS-1) rich in clay size material (high fines content), at a feed size of 100 percent passing 3.0 inches. The sample was stage crushed to 100 percent passing 3.0 inches. Material coarser than 2.0 inches was removed from the feed by dry screening the crushed feed on a 2.0 inch screen. The minus 2.0 inch feed was agglomerated and then recombined with the minus 3.0 inch plus 2.0 inch size fraction for testing. Cement binder additions were equivalent to 8.4, 9.4 and 10.5 lbs/ton whole feed (including the plus 2.0 inch material, which was not agglomerated). Results indicated hydraulic conductivity at standard application rates increased from a maximum stack depth of 29 feet with 8.4 pounds of cement binder to 171 feet with the addition of 10.4 lbs/ton of cement binder.

A test was conducted on a blend of the clay-size rich (high fines content) bulk sample (BS-1) and low fines content drill-core composite. The drill-core composite was prepared to represent low fines content mineralization, on a properly weighted basis, from the three major zones (Main, Jasperoid and Lower). Both samples were crushed to 100 percent passing 2.0 inch, and were combined in a range of weighted blends of high fines to low fines sample. The blends ranged from 10 percent low fines to 70 percent low fines. Results showed that, after agglomerating with a cement addition of 5 lbs/ton, the hydraulic conductivity for all samples was equivalent to greater than 10 times the planned solution application rate at simulated heap stack heights of greater than 200 feet (Olson, 2016b).

In order to test consistency in the permeability testing results, material from the BS-1 bulk sample was sent to two other laboratories. The comparable tests are listed as 3990 LP-1 through LP-3 and 4055 LP-8 through LP-14. The results were inconsistent. For the same cement addition rates for agglomeration, hydraulic conductivities at standard application rates yielded a maximum stack depth that ranged from 29 feet to over 200 feet.

13.6.2 Load/Permeability Tests 2018

Inconsistent results from different laboratories were identified in the 2017 Pre-Feasibility (Tietz et al., 2017) as an issue that required additional study. A program to test the procedures of the different laboratories at different mixes of high fines was developed.

Quantities of the bulk samples BS-1 and BS-NC-2 were identified at MLI. BS-1 was classified by Pershing as a high fines sample, or a sample containing greater than 25 percent minus 200 mesh high fines material. BS-NC-2 was classified by Pershing as a low fines sample, containing less than 25 percent high fines material. The samples were mixed at ratios of 30/70, 40/60, 50/50 and 60/40 of BS-1 to BS-NC-2. MLI next agglomerated the samples with 8 lbs/ton of cement, sealed the agglomerate in plastic buckets and shipped them to Newfields in Elko, Nevada and Geo-Logic Associates in Grass Valley, California. The samples were provided in triplicate to Newfields and Geo-Logic Associates for testing the repeatability of the results (Criely, 2018)/ (Magner, 2018).

Testing procedures for each laboratory were requested and received. Both laboratories used the USBT 5600/5605 procedure. There was difference identified in the consolidation criteria used in the procedure. Geo-Logic was using a 0.1 inch change in sample height under load for a specific time period, while Newfields was using a 0.002 inch change in sample height under load over a 16 hour period. The consolidation criteria employed by Newfields would result in greater compression, thereby decreasing the permeability. This difference in consolidation criteria also allowed for the Geo-Logic testing to progress at a much faster rate.

Due to the excessive amount of time required by NewFields to complete the testwork, the laboratory chose to test only the high fines/low fines (30/70 ratio of BS-1 to BS-NC-2) samples. Summary results from the 2018 repeated load/permeability tests at Newfields and Geo-Logic are presented in Table 13.15. The table includes the equivalent heap stack height (proportional to the applied compressive loading), where the measured hydraulic conductivity was equal to the planned heap solution application rate of 0.003 gpm/ft^2 (“1x”), and ten times this rate (“10x”).

Table 13.15 Fixed Wall Hydraulic Conductivity (Load/Permeability) Tests - 2018

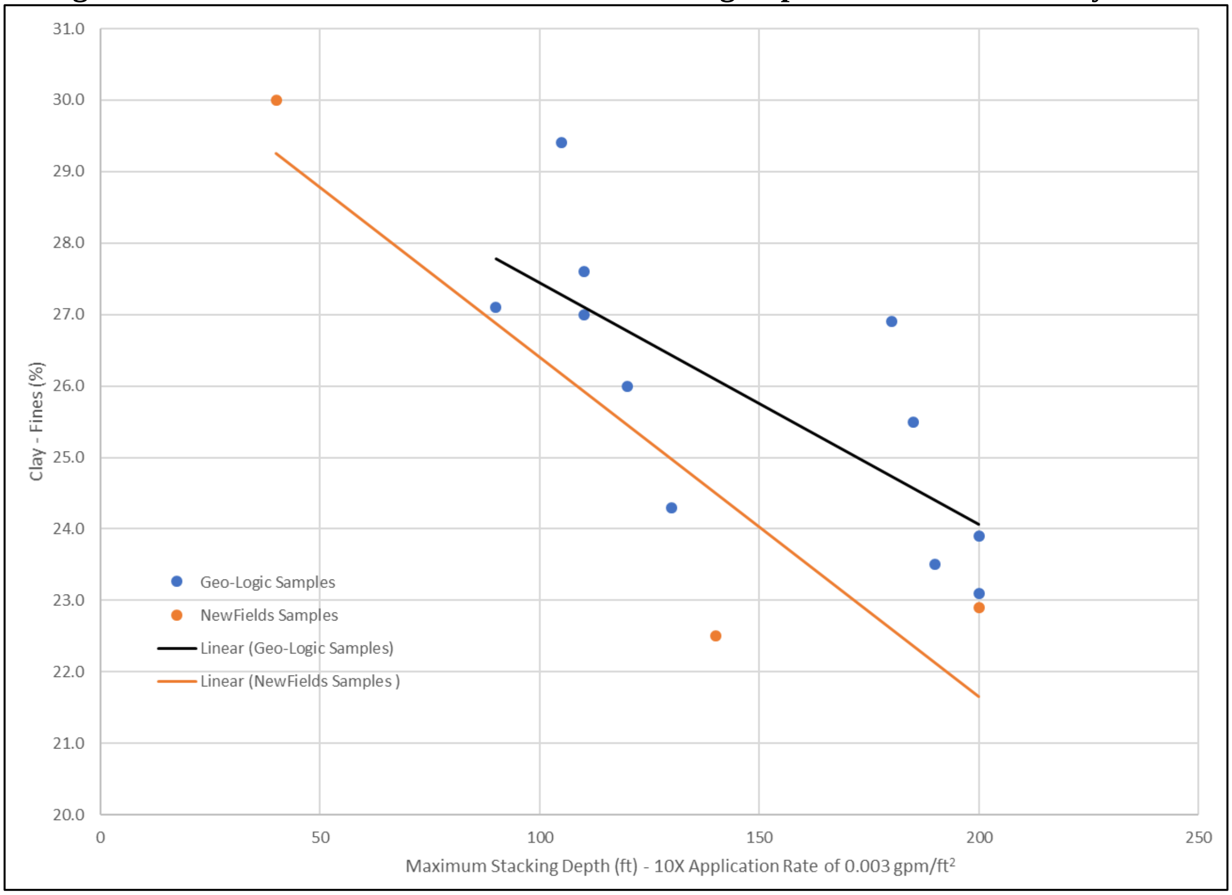
Sample	Zone	Leach Residue	Feed Size		Cement	Maximum Stack Depth in feet for		Geo-Lab
				(% -200 M)	(lb/ton)	1 x App. Rate	10 x App. Rate	
High Fines/Low Fines (60%/40%)	Main	No	100% -2 inch	27.1	8.0	155	90	Geo-Logic
High Fines/Low Fines (60%/40%)	Main	No	100% -2 inch	26.0	8.0	200	120	Geo-Logic
High Fines/Low Fines (60%/40%)	Main	No	100% -2 inch	23.9	8.0	>200	>200	Geo-Logic
High Fines/Low Fines (50%/50%)	Main	No	100% -2 inch	23.1	8.0	>200	>200	Geo-Logic
High Fines/Low Fines (50%/50%)	Main	No	100% -2 inch	29.4	8.0	140	105	Geo-Logic
High Fines/Low Fines (50%/50%)	Main	No	100% -2 inch	27.0	8.0	>200	110	Geo-Logic
High Fines/Low Fines (40%/60%)	Main	No	100% -2 inch	27.6	8.0	155	110	Geo-Logic
High Fines/Low Fines (40%/60%)	Main	No	100% -2 inch	24.3	8.0	200	130	Geo-Logic
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	30.0	8.0	120	40	NewFields
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	22.5	8.0	175	140	NewFields
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	22.9	8.0	>200	>200	NewFields
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	25.5	8.0	>200	185	Geo-Logic
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	26.9	8.0	>200	180	Geo-Logic
High Fines/Low Fines (30%/70%)	Main	No	100% -2 inch	23.5	8.0	>200	190	Geo-Logic

Table 13.15 shows that there does not appear to be a good relationship between the planned blend and the resultant 200M content and maximum stack depth. As an example, for the High Fines/Low Fines (30/70) sample the maximum stack depth varied from 40 to over 200 feet at NewFields. Another example is for the High Fines/Low Fines (40/60) sample the maximum stack depth varied from 90 to over 200 feet at Geo-Logic.

The difference in the permeabilities for these samples appears to be the amount of minus 200M material in the sample. Figure 13.13 shows that as a general trend the maximum stack depth increases as the amount of minus 200M material decreases. The testwork that was performed by Geo-Logic showed that, generally, if the amount of fines is less than 24 percent the maximum stack depth could be 200 feet. The testwork that was performed by NewFields showed that, generally, if the amount of fines is less than 22 percent the maximum stack depth could be 200 feet. The difference in the two laboratory results can only be verified by operational experience.

It must be noted that permeability tests have a number of variables that affect the outcome and that general trends are indicative of what might be achieved. Additionally, actual operations with agglomerated material may be very different from what is observed in the laboratory.

Figure 13.13 Fines Content vs Maximum Stacking Depth – 2018 Permeability Tests



13.6.3 Load/Permeability Summary

Despite the inconsistencies from the different laboratories, the results indicate that material with a higher fines content would require higher cement binder additions and would benefit from blending with lower fines content material. Using a cutoff of 25 percent passing 200M (high fines/low fines material types), review of the resource model indicates that approximately 10 percent of the resource would be categorized as high fines content, and 90 percent would be classified as low fines content. Segregation of mined material into low fines and high fines stockpiles at the crusher would allow blending of this material. This blended feed could then be agglomerated with a cement dosage appropriate for the fines content of the blended feed, and the ultimate burial depth in the leach pad where it would be stacked.

13.7 Summary

Results from column-leach testing demonstrate that the major Relief Canyon ore types (limestone breccia, clay matrix breccia, and jasperoid) contained in the Main Zone, as well as the Jasperoid and Lower Zones, generally would be amenable to heap-leach cyanidation treatment. A summary of column-leach test results from samples representing the current resource is shown in Table 13.15.

Table 13.15 Summary of Column Test Results for Recovery Estimate

Zone	Sample	Description	Feed Size	Au Recovery	Ag Recovery	Leach Time	Solution to Ore Ratio	NaCN Consumed based on Bottle Roll	Cement Added	Reference
				(%)	(%)	(days)		(pounds per ton)	(pounds per ton)	
Main	BS-1 (Bulk)	High Fines Content Bulk Dump Sample	100% -3"	86.7	44.1	86	1.9:1	0.15	8.0	Olson, 2015
			100% -1.5"	85.7	50.0	64	2.2:1	0.15	8.0	
			100% - 3/4"	81.3	57.1	12	0.8:1	0.15	8.0	
Main	BS-2-NC (Bulk)	Low Fines Content Bulk Dump Sample	100% -1.5"	87.1	-	76	1.3:1	0.15	8.0	Olson, 2016a
Main	BS-1 & BS-2-NC	70%/30% Blend of Low Fines BS-2-NC/High Fines BS-1	100% -3"	87.3	28.6	71	1.9:1	0.15	8.0	Olson, 2016a
Main	BS-1 & BS-2-NC	70%/30% Blend of Low Fines BS-2-NC/High Fines BS-1 (Agglomeration of fines)	100% -3"	86.6	-	71	1.9:1	0.15	8.0	Olson, 2016a
Main	BS-2-NC (Bulk)	Low Fines Content Bulk Dump Sample	100% -6"	74.2	12.5	213	5.4:1	0.15	N/A	Olson, 2016a
Main	Master Comp. (Drill Core)	Weighted Comp. of Main Zone Rock Types	80%-3/4"	79.2	-	76	2.8:1	0.18*	8.0	Olson, 2014a
Main	Master Comp. (Drill Core)	Weighted Comp. of Main Zone Rock Types	80%-3/4"	81.8	-	91	4.3:1	0.18*	8.0	Olson, 2014a
Main	1-LSBXH (Drill Core)	Limestone Breccia, High CN Sol.	80%-3/4"	85.7	-	76	3.2:1	0.16	8.0	Olson, 2014a
Main	2-LSBXL (Drill Core)	Limestone Breccia, Low CN Sol.	80%-3/4"	38.5	-	76	3.2:1	0.15	8.0	Olson, 2014a
Main	3-CMBXH (Drill Core)	Clay Matrix Breccia, High CN Sol.	80%-3/4"	91.3	-	76	3.2:1	0.16	8.0	Olson, 2014a
Main	4-CMBXL (Drill Core)	Clay Matrix Breccia, Low CN Sol.	80%-3/4"	78	-	76	3.2:1	0.30	8.0	Olson, 2014a
Main	5-JSPBX (Drill Core)	Jasperoid Breccia, High CN Sol.	80%-3/4"	65.5	-	194	9.4:1	0.14	8.0	Olson, 2014a
Jasperoid	Comp. 1 (Drill Core)	Fault Breccia and Jasperoid Breccia, High CN Sol.	80%-3/4"	82.4	40.9	168	7.9:1	0.15	8.0	Olson, 2015
Jasperoid	Comp. 3 (Drill Core)	Fault Breccia, Jasperoid Breccia and White Clay (Illite) - Fluorite, Low CN Sol.	80%-3/4"	82.4	33.3	168	7.8:1	0.15	8.0	Olson, 2015
Lower	Comp. 2 (Drill Core)	Limestone Breccia and Thin Bedded Limestone, High CN Sol.	80%-3/4"	84.2	21.0	109	5.2:1	0.15	8.0	Olson, 2015
Lower	Comp. 4 (Drill Core)	Thin-Bedded Limestone and Gabbro, Low CN Sol.	80%-3/4"	76.2	-	96	5.1:1	0.15	8.0	Olson, 2015

* - weighted average for composite

Table 13.15 shows that the gold recoveries for the two Bulk Samples BS-1 and BS-2-NC for the Main Zone average 87 percent for the coarser crush size of 100% -3 inch. Other column tests for BS-1 at 100 percent -1.5 inches and 100 percent -¾ inches also showed gold recoveries greater than 80 percent. The BS-2-NC sample that was crushed to 100 percent -1.5 inches had the same gold recovery as the blended material at a 100 percent -3.0 inch crush. These tests indicate that the gold recovery is not crush size dependent for the range tested, except for the fines content, and that the coarser -3-inch crush would be adequate.

Bottle-roll tests conducted on Main Zone material in 2015, reported in Table 13.8, also support the conclusion that gold recovery is not dependent on a fine crush size for feed to the heap leach. Gold recoveries were approximately 80 percent for sizes from 100 percent -2.0 inch, to 80 percent -10 M.

The results from the column-leach tests averaged 82 percent gold extraction for the Jasperoid Zone for the ¾ inch crush size. Bottle-roll tests conducted on Jasperoid Zone material in 2015 and reported in Table 13.8 support the conclusion that gold recovery is not dependent on a crush size. Gold recoveries were similar at approximately 67 percent for sizes from 100 percent -2.0 inch to 80 percent -3/8 inch crush size.

The results from the column-leach tests averaged 80 percent gold extraction for the Lower Zone for the ¾ inch crush size. Bottle-roll tests conducted on Lower Zone material in 2015 and reported in Table 13.8 support the conclusion that gold recovery may not be dependent on a crush size. Gold recoveries were similar at approximately 65 percent for one column test for sizes from 100 percent -2.0 inch to 80 percent -3/8 inch crush size. The second column test had a gold recovery of 65 percent at the 100 percent -2.0 inch crusher size, and increased to 80 percent for the -3/8 inch crush size.

The plus 90 percent gold recoveries for the Jasperoid and Lower Zone ores for the minus 200 mesh bottle-roll tests shown in Table 13.8 indicate that the gold is not locked in sulfides or hindered by preg robbing carbon. Additionally, the recovery continues to increase over time for the column-leach tests for the Jasperoid and Lower zone samples.

In summary, it is likely that the ultimate recovery is not dependent on feed size but rather on leach time. A finer feed size would likely decrease the leach time.

13.7.1 Main Zone Gold Recovery and Reagent Consumption

Table 13.11 shows that gold recovery averaged 87 percent for the samples from the Main Zone bulk samples, BS-1 and BS-2-NC, with crush size greater than 1.5 inches. As discussed in Section 13.3 the composite that was assembled in 2014 using the percentage of rocks types encountered in five holes drilled in 2013 and shown as the “Master Comp. (Drill Core)” most likely did not accurately represent the distribution of rock types. Additional drilling, study, and modeling subsequently revealed that the percentages for rock types with reduced heap leaching characteristics, such as the jasperoid breccia, changed. Although this metallurgical work showed good metallurgical recoveries, it may have underestimated recovery because the geologic model at that time did not accurately reflect the distribution of rock types.

In 2015 additional drilling verified the extent of all zones, especially the Lower and Jasperoid zones. Geologic and mine modeling showed that the ore near surface was characteristic of the Main Zone. The

Lower Zone and Jasperoid Zone were below the Main Zone but trended upward to meet the Main Zone on the north end of the proposed pit. Hence, the bulk samples collected on the surface contact with the Main Zone are a better representation of the ore and are used for the basis for gold recovery, reagent consumption and heap leach cycle time.

In view of the above, gold recovery for the Main Zone in the design Phase 1, Phase 2 and Phase 3 pits is estimated at 87 percent, based on the recoveries from the two bulk samples BS-1 and BS-2-NC. Gold recovery would be dependent upon achieving a solution to ore ratio of approximately 2:1 over a period of 135 days, which would include rinsing residual values from the heap.

Variability testing identified approximately 0.6 million tons in the north part of the Phase 1 pit that would have an estimated 65 percent gold recovery. Variability testing also identified approximately 3.1 million tons in the north section of the design Phase 3 pit that would have an estimated recovery of 82 percent.

Silver recovery for the Main Zone is estimated at 36 percent, based on an average of the recoveries of 44.1 and 28.6 percent from the two bulk sample column tests for BS-1, and a 30/70 mix of BS-2 to BS-2-NC, as shown in Table 13.9 and Table 13.10.

Sodium cyanide consumptions were recorded as less than 0.15 lb/ton for the bottle-roll leach tests, which for the industry is very low. An estimated consumption during operation would be 0.30 lb/ton.

Projected gold and silver recoveries, leach times and reagent consumptions for the Main Zone are shown in Table 13.16.

Table 13.16 Main Zone Gold Recovery

Zone	Location	Sample	Crush Size	Au Recovery	Ag Recovery	Solution to Ore Ratio	NaCN	Cement	Heap Leach Cycle Time
				(%)	(%)		(lb/ton)	(lb/ton)	(days)
Main	Phase 1, 2 and 3 Pit	BS-1 & BS-2-NC & Variability Samples	80% -3"	87	36	2:1	0.30	8.0	135
Main	Phase 1 Pit - North of Section 10800	Variability Samples	80% -3"	65	36	2:1	0.30	8.0	135
Main	Phase 3 Pit - North of Section 11000	Variability Samples	80% -3"	82	36	2:1	0.30	8.0	135

13.7.2 Jasperoid Zone Gold Recovery

The 2015 column-leach tests conducted on Jasperoid Zone drill composites showed identical recoveries of 82.4 percent after approximately 120 days of leaching. The calculated heap leach cycle time for the Jasperoid Zone is 540 days. This long leach cycle could result in multiple lifts being leached before the expected recovery can be realized. Multiple lifts carry an increased risk of channeling. Due to the long heap leach cycle time, a five percent recovery penalty is assessed, resulting in a predicted recovery of 77 percent. Gold recovery would be dependent upon achieving a solution to ore ratio of approximately 7.8:1 over a period of 540 days, which would include rinsing residual values from the heap.

Variability samples supported the overall column-leach recovery of 82 percent. A weighted ton average recovery for mining pit Phases 2 and 3 was calculated using a relationship that was derived for recovery as a function of elevation, which resulted in an overall recovery of 70 percent. This result also compares favorably with the bottle-roll tests that were performed in 2015, shown in Table 13.8 that achieved recoveries of 70 percent and with the corresponding column-leach tests that achieved an 82 percent gold recovery.

Silver recovery for Composite 3 in the column-leach test was 33.3 percent (Table 13.9). As with the estimate for gold recovery, the long heap leach cycle of 540 days and multiple lifts carry an assessed penalty of five percent, resulting in a predicted recovery of 28 percent.

As with the Main Zone samples, sodium cyanide consumptions were recorded as less than 0.15 lb/ton for the Jasperoid Zone bottle-roll tests, which for the industry is very low. However, due to the long heap leach cycle, cyanide will continue to be consumed by natural oxidation. The heap leach cycle Jasperoid Zone is four times the cycle for the Main Zone. Hence, cyanide consumption is estimated at four times the Main Zone consumption at 1.2 lb/ton.

Projected gold and silver recoveries for a 3-inch crush, leach times and reagent consumptions for the Jasperoid Zone are shown in Table 13.17. The Jasperoid Zone bottle-roll tests indicate that the recovery is not size dependent for the crush sizes tested.

Table 13.17 Jasperoid Zone Gold Recovery

Zone	Sample	Crush Size	Au Recovery	Ag Recovery	Solution to Ore Ratio	NaCN	Cement	Heap Leach Cycle Time
			(%)	(%)		(lb/ton)	(lb/ton)	(days)
Jasperoid	Composites 1 and 3 (2015)	80% -3"	77	28	7.8:1	1.20	8.0	540

13.7.3 Lower Zone Gold Recovery

The 2015 column leach tests conducted on Lower Zone drill composites showed recoveries of 84.2 percent and 76.2 after approximately 90 days of leaching. The average recovery for these two columns is 80.2 percent. The calculated heap leach cycle time for the Lower Zone is 270 days. Because of the heap leach cycle time, a three percent recovery penalty is assessed, resulting in a predicted recovery of 77 percent. Gold recovery would be dependent upon achieving a solution to ore ratio of approximately 5.1:1 over a period of 270 days, which would include rinsing residual values from the heap.

Variability samples supported the overall column-leach recovery of 80 percent. A weighted ton average recovery for mining pit Phases 2 and 3 was calculated using a relationship that was derived for recovery as a function of elevation, which resulted in an overall recovery of 80.5 percent. This result also compares favorably with the bottle-roll tests that were performed in 2015, shown in Table 13.8 that achieved recoveries of 70 and 83 percent and with the corresponding column-leach tests that achieved 76 and 84 percent gold recovery.

Silver recovery for the Composite 2 column-leach test was 21 percent as shown in Table 13.9. As with the estimate for gold recovery, the long heap leach cycle of 270 days and multiple lifts carry an assessed penalty of three percent, resulting in a predicted recovery of 18 percent.

Lower Zone sodium cyanide consumptions were recorded as less than 0.15 lb/ton for the bottle-roll leach tests, which for the industry is very low. However, due to the heap leach cycle time, cyanide will continue to be consumed by natural oxidation. The heap leach cycle is twice the cycle for the Main Zone. Hence, cyanide consumption is estimated at 0.6 lb/ton, or two times the Main Zone consumption.

Projected gold and silver recoveries for a 3-inch crush, leach times and reagent consumptions for the Lower Zone are shown in Table 13.18. The Lower Zone bottle-roll tests indicate that the recovery is not size dependent for the crush sizes tested.

Table 13.18 Lower Zone Gold Recovery

Zone	Sample	Crush Size	Au Recovery	Ag Recovery	Solution to Ore Ratio	NaCN	Cement	Heap Leach Cycle Time
Units			**(%)**	**(%)**		**(lb/ton)**	**(lb/ton)**	**(days)**
Lower	Composites 2 and 4 (2015)	80% -3"	77	18	5.1:1	0.60	8.0	270

13.7.4 Mixed Ores and Sulfides

The resource block model includes high grade mixed and sulfide ores found in the design Phase 3 pit at depth. Gold recovery for these resources is estimated at 50 percent. Leach times and reagent consumptions are assumed to be similar to the Jasperoid Zone material.

13.7.5 Recoveries by Mining Phase

The mine plan uses two terms “Design Pit Phase” and “Scheduling Pit Phase”. The “Design Pit Phase” is composed of three distinct boundaries. These boundaries were derived in the early phases of the mine design and were used to describe major milestones in the progression of the pit limits for metallurgical and environmental planning.

The “Scheduling Pit Phase” was developed for mine equipment planning purposes. As the size of the resource acquired better definition, the movement of mine equipment and the capital and operating costs of mine equipment to extract the resource were optimized to the extent possible for the relevant study phase.

A summary of recoveries for the different pit phases is shown in Table 13.19. The different recoveries for the different pit phases are derived from information obtained from the column-leach tests and from the variability tests, which are found in Table 13.16 through Table 13.18.

Table 13.19 Gold and Silver Recoveries by Mine Phase

Design Pit Phase	Scheduling Pit Phase	Ore Zone	Gold Recovery	Silver Recovery
			(%)	(%)
1	1	Main	87	36
1 (north)	1	Main	65	36
2	2	Main	87	36
2	2	Lower	77	18
2	2	Jasperoid	77	28
1	3	Main	87	36
2	4	Main	87	36
2	4	Lower	77	18
2	4	Jasperoid	77	28
3 (north)	5	Main	82	36
3	5	Lower	77	18
3	5	Jasperoid	77	28
3	6,7	Main	87	36
3	6,7	Lower	77	18
3	6,7	Jasperoid	77	28
2,3	all	Mixed	50	28
3	all	Sulfide	50	28

13.7.6 Agglomeration and Permeability

Permeability testwork completed in 2014 and 2015 indicated that the different ore zones could be mixed with different amounts of cement and agglomerated and stacked to a maximum height of 200 feet. Tests were conducted on a blend of a bulk sample from the Main zone and drill-core composites that were prepared to represent low fines content mineralization from the three major zones (Main, Jasperoid and Lower). The blends contained from 10 to 70 percent low fines drill composites and had a fines content that ranged from 10 to 22 percent minus 200M. Results showed that, after agglomerating with a cement addition of 5 lb/ton, the hydraulic conductivity for all samples was equivalent to greater than 10 times the planned solution application rate at simulated heap stack heights of greater than 200 feet (Olson, 2016b).

Permeability testwork was completed in 2018 at two different laboratories to ensure consistency in the testing results. Blended and agglomerated samples of Main zone ore containing different amounts of the bulk sample BS-1 and BS-NC-2 were prepared and tested. While the findings were different, both laboratory results indicated that by blending to maintain the feed below approximately 22 percent of contained minus 200M material, and agglomerating the ore with 8 lb/ton of cement, permeability could be maintained at a maximum stacking depth of 200 feet.

The planned processing method is heap-leach cyanidation of primary crushed ore (80 percent passing 3 inch feed size), after agglomeration pretreatment using 8.0 lbs/ton cement as binder. Cement additions for agglomeration pretreatment will require further optimization. Areas for improvement/resolution include:

- Development of a blending strategy for all zones to minimize fines;
- Optimization of cement additions;
- Development of a heap leach loading strategy to ensure low permeability and/or slow leaching ores are not placed in areas where they would be covered and compacted; and
- Develop a sample program and an analysis procedure for determining fines content of agglomerate prior to placement on the heap leach.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Mineral resource estimation described in this section follows CIM standards and the disclosure and reporting requirements set forth in NI 43-101. The modeling and estimation of the Mineral Resources were done under the supervision of Paul G. Tietz. There is no affiliation between Mr. Tietz and Pershing Gold except that of an independent consultant/client relationship. Although MDA and Mr. Tietz are not experts with respect to any of the following aspects, Mr. Tietz is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Relief Canyon mineral resources as of the date of this report. The effective date of the mineral resource estimate is November 1, 2016.

Pershing has conducted additional drilling within and adjacent to the current resource since the resource estimate's effective date. The impacts of this drilling on the current resource are discussed in Section 14.11.

MDA classifies resources in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories to be in accordance with the CIM Standards. CIM mineral resource definitions are given below, with CIM's explanatory material shown in italics:

Mineral Resource

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

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

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

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic *extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.*

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

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

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of 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.

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

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are 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.

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

Modifying Factors

> Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

MDA reports resources at cutoffs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the requirements that a resource exists "*in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction*."

14.2 Resource Data

A geologic model for estimating the gold resources at Relief Canyon was created from drilling data generated by historical operators, over a period from 1981 through 2008, and also from Pershing Gold's drilling in 2011 through September 2016. The Relief Canyon drill-hole database contains 1,095 holes for total drill footage of 482,755 feet. Of these holes, 419 are core holes with total core footage of 244,353 feet, and the remainder are reverse circulation holes for a total of 238,402 feet. Almost all the core drilling was completed by Pershing Gold (415 holes), with much of this drilling targeting the deep, structurally controlled mineralization occurring within the northern portion of the resource area.

The resource database also includes 20 excavator trenches within the dump/stockpile area located west-southwest from the historically mined South pit. The trenches were dug by Pershing Gold in 2014 and 2015 to provide additional confidence in the mineralization previously defined by shallow core and RC drilling. Each trench is about 50 feet long and 15 feet deep. Samples were collected from the excavated material along the length of the trenches.

The Relief Canyon assay database contains 92,602 gold assays and 43,512 silver assays. Due to the limited silver data within the Main zone, only gold was estimated in the current resource for this portion of the deposit. With the additional silver analyses completed for the North area in 2016, the silver data set is considered robust enough to allow for including silver within the North area resource estimate. All less-than-detection values were converted to "0" for use in the resource estimate.

The database contains down-hole survey information only for the Pershing Gold drilling. The historical drill holes were not surveyed, though the risk to the resource estimate is mitigated by the fact that most historical drill holes were short vertical holes in which only minor down-hole deviation is expected.

The reverse circulation drilling was conducted using both dry and wet drilling techniques. Down-hole contamination was noted in the reverse circulation drilling below the water table, especially in the deeper North area holes, Pershing's 2013 through 2015 core drilling targeted this deep North area mineralization allowing MDA to remove most, if not all, suspect reverse circulation sample intervals from use in the geologic model and resource estimate. A total of 2,914 samples from all or parts of 31 reverse circulation holes were removed from use in the estimate.

Project digital topography was provided by Pershing Gold. These data were incorporated into a digital database using State Plane coordinates, Nevada West zone, NAD83 datum, expressed in US Survey feet, and all subsequent modeling of the Relief Canyon resource was performed using GEOVIA Surpac^{TM} mining software.

14.3 Deposit Geology Pertinent to Resource Modeling

The majority of the modeled Relief Canyon gold mineralization, and all the mineralization within the historical mining area, lies within an envelope of jasperoid-clay breccia (the Main Zone) that lies immediately below the Grass Valley Formation. Within the project area, the thrust fault contact between the Grass Valley Formation and underlying Cane Spring Formation, as well as the mineralized breccia horizon lying between the two units, forms a broad, northeast-trending antiform that plunges about 10° to the southwest. The thickest portions of the breccia, as well as the associated mineralization, lie primarily along the broad crest of the antiform, and the breccia and accompanying mineralization thins and pinches out down dip on the northwest limb and is very thin to nonexistent on the southeast limb. Locally, the breccia-hosted mineralization extends a short distance (usually less than 10 feet) into the overlying Grass Valley Formation.

The exception to the limited mineralization within the Grass Valley occurs within the southwest extension of the Main zone. Drilling in 2015 encountered east-dipping mineralization within the Grass Valley that has been interpreted to be a ramp structure extending off the west-dipping main thrust horizon. This mineralization was encountered at depths of less than 200 feet, and there is no visual or geochemical evidence at the surface for this mineralized zone.

While the reverse circulation data often indicate significant jasperoid development within the upper portions of the Main Zone breccia, the few core holes drilled through this feature indicate breccias with a clay-size particle rich matrix are also prevalent. It is likely that the clay-size component is poorly recovered using reverse circulation drilling techniques, at least in the geologic samples used for logging purposes.

Deeper drilling within the North area has encountered structurally controlled mineralization within the Cane Spring Formation (within both the Massive and Deformed Limestone packages as discussed in Section 7.0) at depth beneath the Main Zone breccia. Labeled the Lower Zone and Jasperoid Zone by Pershing Gold, both are sub-horizontal structural zones characterized by fault brecciation, moderate to pervasive silicification, and sporadic illite, kaolinite development. Higher-angle splay structures, along with thrust fault-related duplex structures, are common extending up from and between the main sub-horizontal structures, and mineralization is localized at these structural intersections.

Mafic dikes within the Lower Zone show a strong spatial correlation with gold and associated silver mineralization though the dikes themselves are generally only weakly mineralized. Post-emplacement fault movement along the intrusive/limestone contacts served to channel hydrothermal solutions, and higher-grade gold mineralization often occurs within these fault contact zones. The dikes themselves are generally poor hosts and often displace and/or create gaps within the mineralized horizons.

Above the water table, the Main Zone breccia mineralization is primarily oxidized, though some remnant unoxidized pockets are in evidence. Below the water table, oxidation is highly variable within the Main Zone and Lower Zones, and portions of the Jasperoid Zone, with the presence of oxide, mixed oxide/unoxidized, and some remnant unoxidized mineralization.

14.4 Geology Modeling

MDA used the same modeling procedures in the current resource estimate as were used in the previous MDA resource block models and estimates. The geologic interpretations and assay analyses have been updated with the new 2016 drill data.

Pershing Gold provided MDA with a set of digitized 50-foot-spaced, east-west-oriented cross sections that define the limits of alluvium, the Grass Valley Formation, mafic intrusive sills and dikes, the post-mineral(?) diorite intrusion within the northeast corner of the deposit, as well as low- and high-angle fault zones within the Cane Spring and Smelser Pass Formations beneath the Main Zone breccia. Included within the cross-sectional interpretation were areas of moderate to strong illite-altered breccias within the Main Zone mineralization. The clay-size matrix breccia model, which more accurately could be described as a fines-dominant model, is used primarily for metallurgical purposes. The Pershing Gold sections were created using geologic logs of Pershing Gold's drill holes in combination with interpretations of logging codes and historical cross sections from older holes.

MDA made some minor revisions to the interpreted geology to best fit the validated drill data and also added the historical mine dumps and the updated topography to the cross-sections. The cross-sectional data were rectified in 3-dimensions to the drill data and modeled using solids (the mine dumps and all geology) and planar surfaces (the prominent faults). Individual solids were created for the alluvium, Grass Valley Formation, fines-dominant breccias, mafic sills and dikes, and the diorite intrusive.

14.4.1 Water Table

Pershing Gold provided MDA with a groundwater surface created by Schlumberger Water Services (Reno, Nevada) using the data from two pumping wells, four piezometers, and approximately 15 exploration drill holes located throughout the resource area. This groundwater surface is coded into the block model and will be used by Pershing Gold for mine planning.

14.4.2 Oxidation

The project database includes oxidation codes (1, 2, and 3), with 1 = completely oxidized, 2 = mixed oxidized/unoxidized, and 3 = completely unoxidized) for 860 drill holes, representing approximately 80 percent of the drill holes in the Relief Canyon database. The codes were interpreted by a number of different geologists from a variety of companies based on logging of reverse circulation drill chips and diamond core. These data are therefore subjective by nature and may or may not correlate with ultimate recoveries achieved in a cyanide heap-leach scenario.

In addition to the standard fire assay completed for Pershing Gold, most of the mineralized intervals within their 2012 through 2016 core drilling programs were assayed for gold using hot cyanide-leach technique. The database includes a total of 5,364 cyanide-leach gold assays. In conjunction with both the subjective drill-log oxidation codes and the observed correlation between the hot cyanide data and bottle-roll and column-leach test, MDA used these data to determine general areas within the deposit that are predominantly "oxide," "mixed oxide/sulfide," and "sulfide."

A review of the oxidation data in the context of the geology reveals that the Main Zone breccia horizon is predominantly oxide material, though the deeper extensions, especially below the water table, can be partially oxidized with some minor remnant unoxidized material. The Lower Zone structurally controlled mineralization at depth beneath the Main Zone is also predominantly oxide material, as indicated by the metallurgical test work, though isolated areas of partially oxidized, and some remnant unoxidized material are present.

The oxide zone transitions to mixed oxide/sulfide material near the base of the Lower Zone and much of the Jasperoid Zone mineralization at depth within the North area. The transition between oxidized, mixed, and sulfidic material within individual drill holes can be very sharp, though it also can be highly variable both within and between drill holes.

Due to the reduced gold recoveries associated with the mixed and sulfide material, as indicated by the cyanide leach assay data and the metallurgical test work, mixed and sulfide 3D solids were created and used to code the block model. An increased resource cut-off grade was applied to blocks within these solids.

14.5 Density

Historical density data are discussed in Section 11.9. Due to questions of sample provenance and procedures, these data have not been used to develop the current specific gravity model.

Pershing Gold collected over 2,900 samples for density determinations from their 2012, 2013 and 2014 core drilling programs. The samples were from all significant rock types and gold grade ranges, and the procedures used the water immersion method. No additional density data was collected in 2015 or 2016 and the density values used in the current model are the same as those used in the 2015 and June 2016 resource models and estimates.

MDA analyzed the data and the general statistics by modeled rock type and gold mineral domain, along with the tonnage factor used in the block model, as shown in Table 14.1. Due to the often highly fractured nature of the deposit, and the fact that voids resulting from many of the open fractures cannot be accurately reflected in density determinations, the measured density values were reduced by 1.5 percent to account for the unavoidable sample-selection bias. The alluvium density was reduced by a larger percentage due to the few samples collected and the sample bias in collecting representative overburden material. There is no density data on dump material, so a tonnage factor of 20 cuft/ton was assigned by MDA. The factored data, shown in the "Model TF" column in Table 14.1, reflect the actual density values assigned to the Relief Canyon block model.

Table 14.1 Descriptive Statistics of Relief Canyon Density Values by Rock Type

Rock Type	Specific Gravity Statistics (g/cm3)						Model TF (cuft/ton)
	Count	Mean	Median	Min.	Max.	Std.Dev.	
Alluvium	14	2.19	2.24	1.64	2.69	0.33	**18**
Dump	NA	-	-	-	-	-	**20**
Grass Valley Fm	44	2.36	2.31	2.14	2.60	0.14	**14**
Main Zone Breccia	70	2.50	2.57	1.98	2.76	0.21	**12.8**
Cane Springs Ls	558	2.59	2.63	1.85	2.84	0.15	**12.5**
Mafic dike	91	2.34	2.41	1.68	2.75	0.29	**13.7**
Volcanic tuff/flow	99	2.66	2.74	2.18	2.85	0.16	**12**
Debris Flow Breccia	6	2.56	2.56	2.42	2.70	0.12	**12.7**
Diorite Intrusive	NA	-	-	-	-	-	**12.5***
Mineralized Material**							
Low-grade Au (MZ-Ox)	668	2.47	2.57	1.70	2.75	0.24	**12.9**
Low-grade Au (MZ-Sulf)	11	2.39	2.42	2.27	2.43	0.05	**13.5**
Low-grade Au (LZ-Ox)	658	2.51	2.58	1.82	2.78	0.19	**12.8**
Low-grade Au (LZ-Sulf)	113	2.55	2.61	1.99	2.77	0.17	**12.6**
Mid-grade Au (MZ-Ox)	157	2.41	2.51	1.86	2.74	0.23	**13.2**
Mid-grade Au (MZ-Sulf)	6	2.36	2.39	2.14	2.48	0.12	**13.5**
Mid-grade Au (LZ-Ox)	284	2.49	2.54	1.85	2.86	0.20	**12.9**
Mid-grade Au (LZ-Sulf)	50	2.53	2.58	2.05	2.71	0.17	**12.7**
High-grade Au (LZ-Ox)	73	2.39	2.42	1.99	2.66	0.18	**13.5**
High-grade Au (LZ-Sulf)	18	2.53	2.57	2.30	2.69	0.11	**12.7**

* SG data received after resource completion indicates an actual tonnage factor of 11.7

** MZ= Main zone; LZ=Lower zone; Ox=Oxide/Mixed area; Sulf=unoxidized area

As noted in Table 14.1, specific gravity measurements on the diorite intrusive were conducted by Pershing after the current resource model and estimate were completed. Analyses of the 17 core samples indicate an average specific gravity of 2.8g/cm^3.

14.6 Gold and Silver Modeling

The mineral resources at Relief Canyon were modeled and estimated by evaluating the drill data statistically, utilizing the geologic interpretations provided by Pershing Gold to interpret gold mineral domains on cross sections spaced at 50-foot intervals, rectifying the gold mineral-domain interpretations on long sections spaced at 10-foot intervals, analyzing the modeled mineralization geostatistically to establish estimation parameters. This was followed by estimating gold and silver grades into a three-dimensional block model. All modeling of the Relief Canyon resources was performed using Geovia® mining software.

A separate silver model was not constructed due to 1) the generally close association with gold, and 2) the relatively low silver values and metallurgical recoveries which results in silver having a minor impact on projected economics. The silver assay coding and grade estimate were controlled using the same mineral domains and estimation parameters as the gold model.

14.6.1 Gold Mineral Domains

A total of 87 vertical, north-looking cross sections spaced at 50-foot intervals across the deposit were used for the initial modeling of the Relief Canyon gold mineral domains. A mineral domain is a natural grade population of a metal that occurs within a specific geologic setting. In order to define the gold mineral domains at Relief Canyon, the natural populations were first identified on quantile graphs that plot the gold-grade distributions of the drill-hole assays. Quantile graphs of the combined core and reverse circulation assay data, along with graphs of core-only and reverse-circulation-only data, were created and evaluated. This analysis led to the identification of low- (~0.004 to ~0.025 oz Au/ton), medium- (~0.025 to 0.10 oz Au/ton), and high-grade (>~0.10 oz Au/ton) gold populations, assigned to domains 100, 200, and 300, respectively. Ideally, each of these populations can be correlated with specific geologic characteristics that are captured in the project database to aid in the definition of the mineral domains. These population breaks were most clearly observed in the core data, while the reverse circulation data have only subtle breaks indicating a smoothing of the assay data. This smoothing versus the core results is expected from the reverse circulation data due to the often greater sample lengths and the nature of the drilling and sampling procedures.

The drill-hole traces, topographic profile, and Pershing Gold's geologic interpretations were plotted on the sections with gold assays (colored by the grade-domain population ranges) plotted along the drill-hole traces, and these data were used as the base for MDA's interpretations of the mineral domains. Mineral-domain envelopes were interpreted on the sections to more-or-less capture assays corresponding approximately to each of the defined grade populations. The domains were modeled through all available drill data, including volumes that had been mined.

Due to inconsistencies in the geologic logs of the historical reverse circulation holes, as well as the fact that essentially all subsurface geologic information in the Main Zone is derived from reverse circulation chips, it was difficult to correlate the three mineral domains to specific Main Zone geologic characteristics. In a general sense, medium-grade zones of mineralization (domain 200) typically lie in the upper portions of the Main Zone breccia, often associated with jasperoid-clay breccias. The crests of small, sympathetic folds that lie within the crest of the larger antiformal structure appear to exert some control on the mineralization. While high-grade assays occur within the often jasperoid-rich breccia, they are primarily isolated and show limited continuity. As such, a high-grade domain 300 was not modeled within the Main Zone. The low-grade (domain 100) zones envelope the domain 200 mineralization, but they extend progressively further laterally within the breccia, especially down the dip of the northwest limb of the antiformal structure. Despite the lack of geologic definition of the mineral domains, the modeled 100 and 200 domains exhibit excellent continuity throughout the Main Zone.

In the North area, medium- and high-grade mineralization (domains 200 and 300) occurs within sub-horizontal to steeply-dipping breccia zones within the Cane Spring Formation carbonates (the Massive and Deformed Limestone packages). The breccias are interpreted as thrust fault-related features that are often sub-parallel to bedding orientation. Low-grade mineralization is associated with limited brecciation/fracturing within the Cane Spring wallrock. The mineral domain contacts, as exhibited in core, are often sharp and clearly define the mineralized structures. Adjacent reverse circulation drilling often shows a much thicker mineral intercept with low- to medium-grade values occurring down hole beneath the projected mineralized zone. This is presumed to be due to down-hole contamination (See Section 11.8 for a discussion of core versus reverse circulation drilling in the North area). The significant amount of core drilling completed by Pershing from 2013 through 2016 has allowed for the removal of the suspect reverse circulation intervals from use in the geologic model and resource estimate.

Representative cross sections showing gold mineral-domain interpretations are shown for the South area Main Zone in Figure 14.1 and the stratigraphically deeper Lower and Jasperoid zones mineralization within the North area in Figure 14.2.

The cross-sectional mineral-domain polygons were digitized and then three-dimensionally rectified to the drill data. Vertical slices of the polygons were created at 10-foot intervals orthogonal to the cross sections, and the mineral domains were then modeled on 10-foot-spaced long sections. The final product of the long-section work is a set of 10-foot-spaced mineral-domain envelopes that three-dimensionally honor the drill data at the resolution of the block model.

Dump/Stockpile: Mineralized material was encountered in drill holes within portions of the mine dumps that overlie the existing *in-situ* resource and also within Waste Rock Dump 4 situated to the west of the South pit. A total of 42 shallow reverse circulation holes and 17 core holes had been drilled through the latter dump/stockpile and results indicated the potential to develop a small resource. Pershing Gold excavated 20 trenches within this dump area in 2014 and 2015 to better define the gold mineralization. The dump mineralization was modeled within a unique cross-sectional mineral domain (domain 10) polygon. The polygons were then used to create 3D solids, and gold grades were estimated within these solids.

Figure 14.1 Cross Section 15109100 Showing Gold Mineral Domains

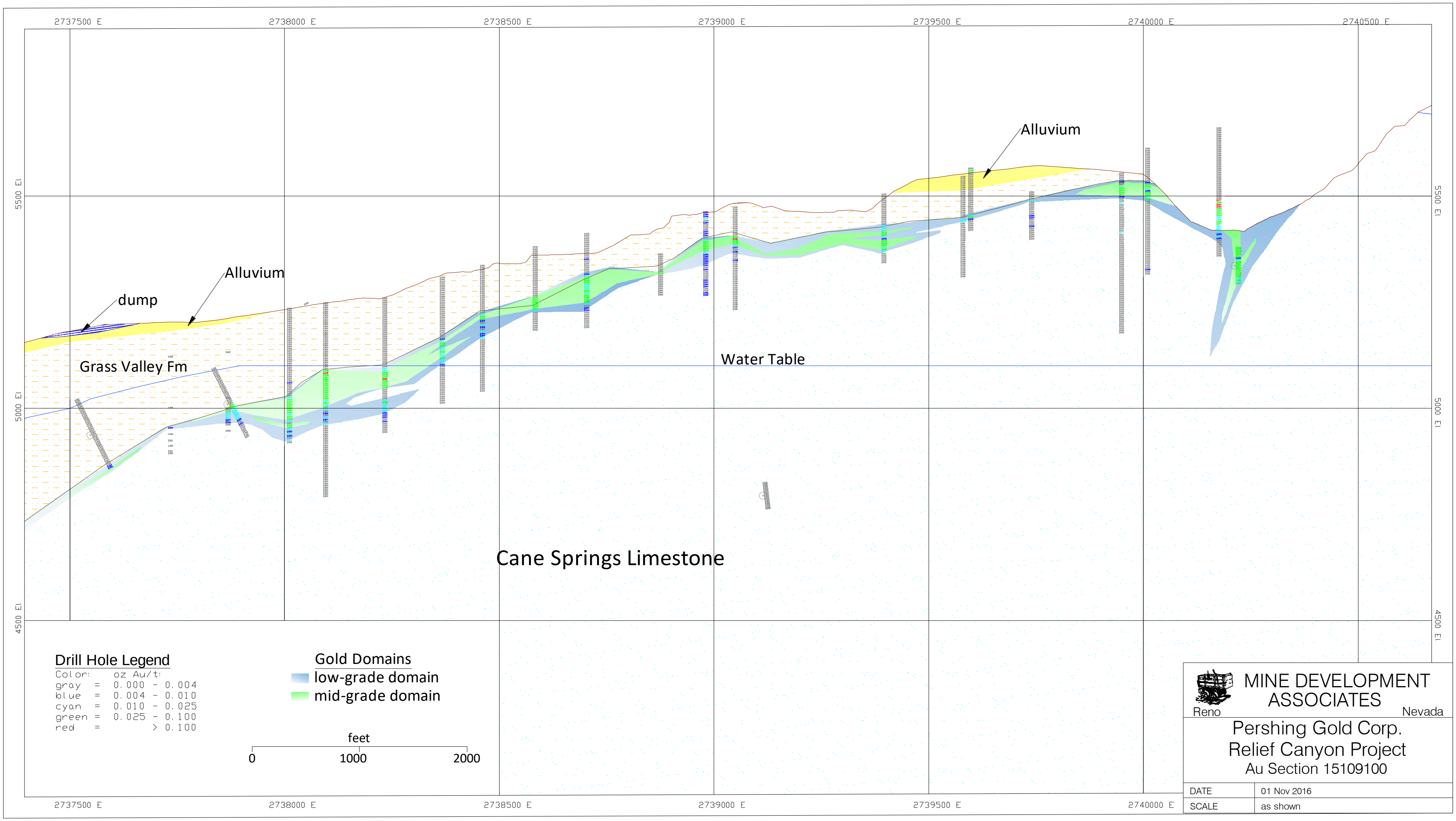
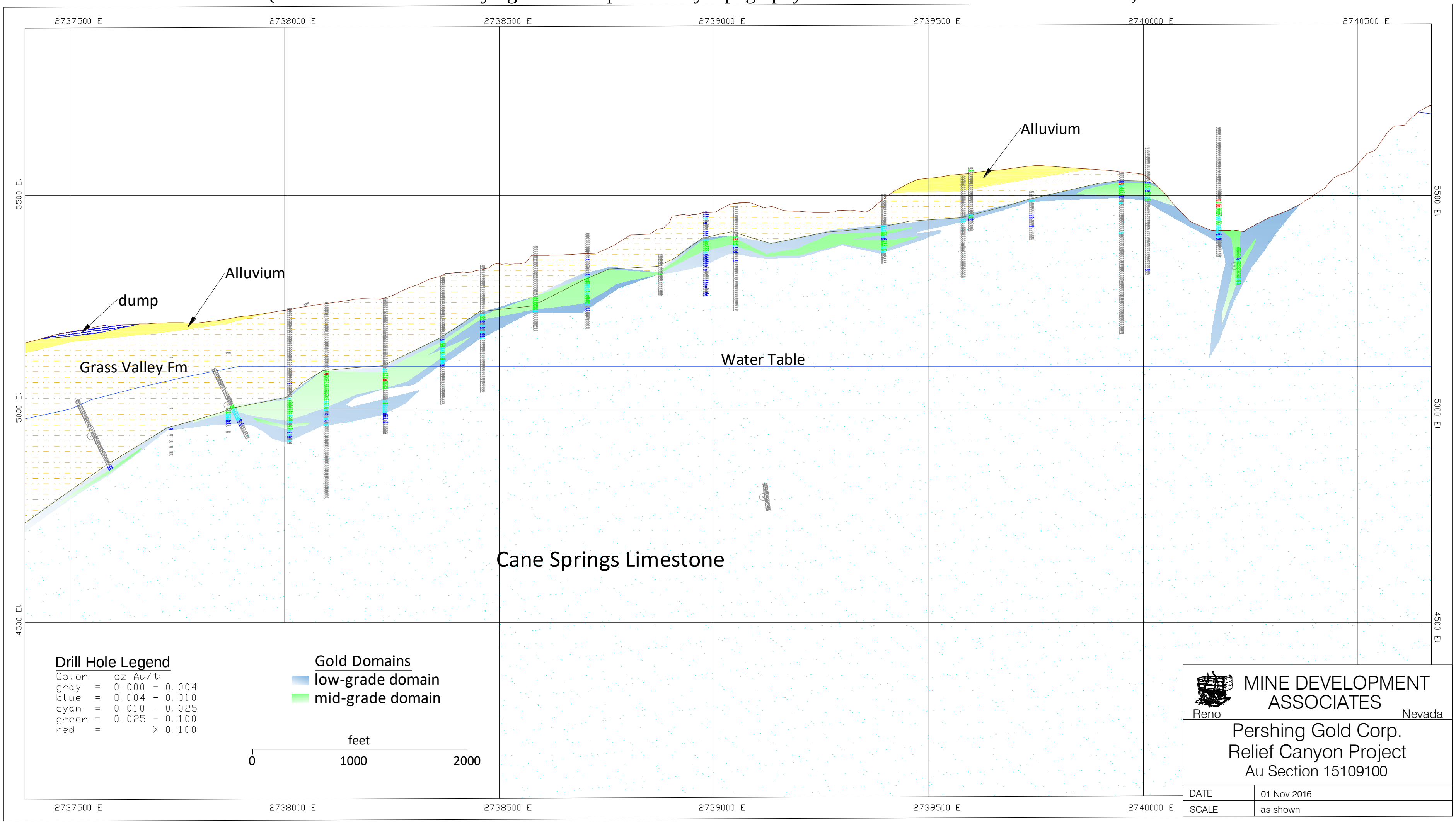


Figure 14.2 Cross Section 15111050 Showing Gold Mineral Domains

(modeled mineralization lying above the present-day topography is not included in the mineral resources)



14.6.2 Assay Coding, Capping, and Compositing

Drill-hole gold and silver assays were coded to the mineral domains using the cross-section mineral-domain envelopes. Only in the North area, where there are enough silver assays to provide confidence in the resulting grade estimate, were the silver assays coded and used in the subsequent resource estimate. Descriptive statistics of the coded gold and North area silver assays are provided in Table 14.2 and Table 14.3.

Table 14.2 Descriptive Statistics of Coded Gold Assays

Domain	Assays	Count	Mean (oz Au/ton)	Median (oz Au/ton)	Std. Dev.	CV	Min. (oz Au/ton)	Max. (oz Au/ton)	# Capped
100	Au	11445	0.010	0.008	0.009	0.860	0.000	0.170	0
	Au Cap	11445	0.010	0.008	0.009	0.860	0.000	0.170	
200	Au	4652	0.047	0.036	0.043	0.910	0.000	0.961	4
	Au Cap	4652	0.047	0.036	0.039	0.830	0.000	0.500	
300	Au	229	0.363	0.253	0.390	1.070	0.004	3.618	0
	Au Cap	229	0.363	0.253	0.390	1.070	0.004	3.618	
All	*Au*	*16326*	*0.025*	*0.012*	*0.063*	*2.560*	*0.000*	*3.618*	*4*
	Au Cap	*16326*	*0.025*	*0.012*	*0.063*	*2.530*	*0.000*	*3.618*	

Table 14.3 Descriptive Statistics of Coded Silver Assays – North Area

Domain	Assays	Count	Mean (oz Ag/ton)	Median (oz Ag/ton)	Std. Dev.	CV	Min. (oz Ag/ton)	Max. (oz Ag/ton)	# Capped
100	Ag	3950	0.105	0.047	1.304	12.440	0.000	98.331	11
	Ag Cap	3950	0.086	0.047	0.142	1.650	0.000	1.500	
200	Ag	1383	0.337	0.158	0.845	2.510	0.000	16.867	17
	Ag Cap	1383	0.301	0.158	0.466	1.550	0.000	3.000	
300	Ag	203	1.104	0.460	3.365	3.050	0.000	42.851	6
	Ag Cap	203	0.791	0.460	0.878	1.110	0.000	4.000	
All	*Ag*	*5536*	*0.193*	*0.067*	*1.347*	*6.970*	*0.000*	*98.331*	*34*
	Ag Cap	*5536*	*0.161*	*0.067*	*0.337*	*2.100*	*0.000*	*4.000*	

The process of determining assay caps began with inspection of quantile plots of the coded assays by domain to assess the mineral-domain populations and identify possible high-grade outliers that might be appropriate for capping. Descriptive statistics of the coded assays by domain and visual reviews of the spatial relationships of the possible outliers and their potential impacts during grade interpolation were also considered in the process of determining appropriate assay caps. After this review, just four gold samples, all in the 200 domain, were capped at 0.5 oz Au/ton. A total of 34 silver assays were capped, with samples requiring capping occurring in all three mineral domains. The effects of the final assay caps can be qualitatively evaluated by examination of the descriptive statistics of the capped and uncapped mineral-domain assays (Table 14.2 and Table 14.3).

The capped assays were composited at 10-foot down-hole intervals respecting the gold mineral domains, and length-weighted composites were used in the block-model grade estimation. The volume inside each mineral domain was estimated using only composites from inside that domain. Descriptive statistics of the gold and silver composites are shown in Table 14.4 and Table 14.5.

Table 14.4 Descriptive Statistics of Relief Canyon Gold Composites

Domain	Count	Mean (oz Au/ton)	Median (oz Au/ton)	Std. Dev.	CV	Min. (oz Au/ton)	Max. (oz Au/ton)
100	6265	0.010	0.009	0.007	0.69	0.000	0.170
200	2642	0.047	0.037	0.033	0.71	0.001	0.500
300	180	0.360	0.259	0.329	0.91	0.054	3.618
All	*9087*	*0.025*	*0.012*	*0.057*	*2.31*	*0.000*	*3.618*

Table 14.5 Descriptive Statistics of Relief Canyon Silver Composites – North Area

Domain	Count	Mean (oz Ag/ton)	Median (oz Ag/ton)	Std. Dev.	CV	Min. (oz Ag/ton)	Max. (oz Ag/ton)
100	2339	0.086	0.053	0.122	1.430	0.000	1.500
200	884	0.301	0.166	0.416	1.380	0.000	3.000
300	165	0.791	0.513	0.785	0.990	0.000	4.000
All	*3388*	*0.161*	*0.074*	*0.307*	*1.910*	*0.000*	*4.000*

14.6.3 Block Model Coding

The 10-foot-spaced long-sectional mineral-domain polygons were used to code a north-south three-dimensional block model that is comprised of 10-foot (width) x 10-foot (length) x 10-foot (height) blocks. In order for the block model to better reflect the irregularly shaped limits of the various gold domains, as well as to explicitly model dilution, the percentage volume of each mineral domain within each block is stored (the "partial percentages").

Each block is assigned a tonnage factor listed on Table 14.1 based on its coded lithology and mineral domain. The blocks are coded as lying above or below the Schlumberger and MDA-modeled water-table surface, and the percentage of each block that lies below the topographic surface is also stored.

14.7 Resource Estimation

The resource estimate reflects the general northerly trend and variably west-dipping nature of the Relief Canyon gold mineralization. To replicate the change in orientation observed along the strike of the deposit, four search-ellipse orientations were used to control the resource estimate (see Table 14.6). The first three orientations are controlled by the Northing coordinate (designated as Areas 10, 20, and 30) with a general steepening of dip within the Main Zone breccia as one progresses north through the deposit. The search-ellipse orientation in the northern area (Area 30) also represents the general orientation of the mineralized fault structures at depth within the Cane Spring Formation. The fourth orientation area (Area 40) is coded into the block model using a solid and represents an area of east-dipping mineralization located within the southwestern extension of the deposit below and to the east of the low-grade stockpile dump. This structurally-controlled mineralization, which lies along the Cane Spring-Grass Valley thrust contact and also extends above the thrust into the overlying Grass Valley formation, was the focus of drilling in 2015.

A variographic study was performed using the gold composites from each mineral domain, collectively and separately, at various azimuths, dips, and lags. Acceptable variogram models were obtained from composites from the combined 100 and 200 gold domains in the Main zone and the 200 and 300 gold domains in the North area. A maximum range of about 250 feet was obtained in the Main Zone horizontal strike (azimuth 0°, plunge 0°) and dip (azimuth 270, plunge 20°) directions, while a maximum range of about 150 feet was obtained in the North area horizontal strike (azimuth 90°, plunge 25°) direction; these are geologically reasonable orientations for the global strike and dip of the mineralization, respectively. Parameters obtained from the variography study were used in an ordinary-krige interpolation and also provided information relevant to both the estimation parameters used in an inverse-distance interpolation and in resource classification.

The estimation parameters applied at Relief Canyon are summarized in Table 14.7 The estimation used two search passes in the Main zone (estimation areas 10 and 40) and three passes in the North area (estimation areas 20 and 30) with successive passes not overwriting previous estimation passes. The first-pass search distances take into consideration the results of both the variography and drill-hole spacing. The second and third passes were designed to estimate grade into all blocks coded to the mineral domains that were not estimated in the first pass.

The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded by that domain. The estimated grades were coupled with the partial percentages of the mineral domains to enable the calculation of a single weight-averaged block-diluted grade for each block.

Gold and silver grades were interpolated using inverse distance to the third power, ordinary-krige, and nearest-neighbor methods. The mineral resources reported herein were estimated by inverse-distance interpolation, as this technique was judged to provide results superior to those obtained by ordinary kriging. The nearest-neighbor estimation was also completed as a check on the other interpolations.

Gold grades were estimated into all blocks coded by the gold mineral domains, including those blocks coded as "mined out" (the block centroid occurs above the current post-mining surface but below the material coded as dump). Silver grades were estimated into only the North area blocks coded by the gold mineral domains. Due to the more limited amount of silver data, silver was not estimated into the Main zone blocks.

Table 14.6 Relief Canyon Search Ellipse Orientations

Estimation Area	Major Bearing	Plunge	Tilt
Area 10; <15110400 North	0°	0°	15°
Area 20; 15110400 North to 15111000 North	0°	10°	25°
Area 30; >15111000 North	0°	10°	35°
Area 40; Southwest area east-dipping structural zones	0°	0°	-25°

Table 14.7 Summary of Relief Canyon Estimation Parameters

Estimation Pass	Search Ranges (ft)			Comp Constraints		
	Major	S-Major	Minor	Min	Max	Max/hole
1 (area 10,40)	200	200	100	2	12	3
2 (area 10, 40)	400	400	400	1	15	3
1	150	150	75	2	9	3
2	300	300	150	1	12	3
3	400	400	400	1	12	3

For the dump/stockpile mineralization, gold grades were estimated into the one domain using inverse distance to the second power, an isotropic search, and no restrictions on the number of dump composites used in the estimate. These relaxed parameters were used to attain a smoothed estimate that approximated the average grade of the dump assay data. Silver was not estimated into the dump/stockpile.

14.8 Relief Canyon Mineral Resources

The Relief Canyon resources are classified on the basis of the distance of the model blocks to the nearest composite, and the minimum number of composites and drill holes used in the grade interpolation of each block (Table 14.8). Measured resources are restricted to blocks defined by Pershing Gold's core holes due to the general lack of QA/QC data for the reverse circulation drilling that could be used for verification purposes and to some uncertainties in the reverse-circulation-based geologic interpretations. The stockpile/dump mineralization located southwest of the historical mined open-pit is restricted to an Inferred only classification.

Table 14.8 Relief Canyon Classification Parameters

Class	Estimation Pass	Min. Number of Drill holes	Min. Number of Composites	Avg. Dist. to Nearest 2 Composites
Measured	1	2*	3	75
Indicated	1	2	2	150
"	1	1	2	100
Inferred	all other modeled in-place mineralization			
"	all dump/stockpile mineralization			

* minimum one core hole

The Relief Canyon mineral resources are listed in Table 14.9 using a cutoff grade of 0.005 oz Au/ton for oxide material, 0.01 oz Au/ton for mixed material, and 0.02 oz Au/ton for sulfide material. The oxide and mixed cutoffs were chosen to capture mineralization potentially available to open-pit extraction and heap-leach processing, with the higher cutoff for mixed material reflecting the expected reduction in recovered gold. The sulfide cutoff was chosen to reflect the potentially higher costs associated with sulfide processing. Reported cutoff grades were applied, based on Mr. Tietz's judgement as a Qualified Person, in order to determine that the "*material has reasonable prospects for eventual economic extraction*".

Table 14.9 Relief Canyon Reported Mineral Resources

Class	Cutoff (oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton
Measured-Oxide	0.005	14,232,000	0.022	312,000	10,550,000	0.119
Measured-Mixed	0.010	259,000	0.058	15,000	259,000	0.251
Measured-Total	variable	14,491,000	0.023	327,000	10,809,000	0.123
Indicated-Oxide	0.005	26,854,000	0.016	439,000	6,236,000	0.094
Indicated-Mixed	0.010	162,000	0.033	5,000	162,000	0.206
Indicated-Sulfide	0.020	369,000	0.050	18,000	369,000	0.313
Indicated-Total	variable	27,385,000	0.017	462,000	6,767,000	0.108
Meas. + Ind Total	variable	41,876,000	0.019	789,000	17,576,000	0.117
Inferred-Oxide	0.005	5,238,000	0.009	45,000	781,000	0.066
Inferred-Mixed	0.010	4,000	0.018	100	4,000	0.125
Inferred-Sulfide	0.020	4,000	0.028	100	4,000	0.164
Inferred-Total	variable	5,246,000	0.009	45,200	789,000	0.068
Note: rounding may cause apparent inconsistencies						

The Inferred oxide resource total in Table 14.9 includes a dump/stockpile resource of 23,000 ounces gold at a 0.007oz Au/ton average gold grade. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to the Indicated classification with continued exploration.

Mineralized materials at various cutoff grades are shown in Table 14.10, Table 14.11, and Table 14.12, for oxide, mixed, and sulfide material, respectively. These block-diluted resources are tabulated at additional cutoffs in order to provide grade-distribution information, as well as to provide for economic conditions other than those envisioned by the reported resource cutoffs. It should be noted that the estimated resources include the calculated Mineral Reserves presented in Section 15 and discussed in Section 16 and 22 of this report.

Table 14.10 Relief Canyon Oxide Mineralized Material

Cutoff	Measured Resource - Oxide					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.004	15,768,000	0.020	319,000	11,788,000	0.111	1,310,000
0.005	***14,232,000***	***0.022***	***312,000***	***10,550,000***	***0.119***	***1,260,000***
0.006	12,822,000	0.024	305,000	9,480,000	0.128	1,209,000
0.008	10,492,000	0.028	290,000	7,816,000	0.143	1,117,000
0.010	8,681,000	0.032	275,000	6,578,000	0.157	1,036,000
0.012	7,282,000	0.036	260,000	5,637,000	0.172	968,000
0.015	5,810,000	0.042	241,000	4,650,000	0.190	884,000
0.020	4,313,000	0.050	216,000	3,544,000	0.217	770,000

Cutoff	Indicated Resource - Oxide					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.004	29,786,000	0.015	451,000	7,137,000	0.087	620,000
0.005	***26,854,000***	***0.016***	***439,000***	***6,236,000***	***0.094***	***584,000***
0.006	24,212,000	0.018	426,000	5,493,000	0.100	549,000
0.008	19,666,000	0.020	396,000	4,315,000	0.113	487,000
0.010	15,745,000	0.023	363,000	3,404,000	0.127	431,000
0.012	12,789,000	0.026	332,000	2,783,000	0.139	387,000
0.015	9,719,000	0.030	292,000	2,107,000	0.156	328,000
0.020	6,834,000	0.036	244,000	1,370,000	0.182	249,000

Cutoff	Measured + Indicated - Oxide					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.004	45,554,000	0.017	770,000	18,925,000	0.102	1,930,000
0.005	***41,086,000***	***0.018***	***751,000***	***16,786,000***	***0.110***	***1,844,000***
0.006	37,034,000	0.020	731,000	14,973,000	0.118	1,758,000
0.008	30,158,000	0.023	686,000	12,131,000	0.132	1,604,000
0.010	24,426,000	0.026	638,000	9,982,000	0.147	1,467,000
0.012	20,071,000	0.030	592,000	8,420,000	0.161	1,355,000
0.015	15,529,000	0.034	533,000	6,757,000	0.179	1,212,000
0.020	11,147,000	0.041	460,000	4,914,000	0.207	1,019,000

Cutoff	Inferred Resource - Oxide*					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.004	5,742,000	0.008	46,000	975,000	0.061	60,000
0.005	**5,238,000**	**0.009**	**45,000**	**781,000**	**0.066**	**52,000**
0.006	4,613,000	0.009	42,000	620,000	0.072	44,000
0.008	2,203,000	0.012	26,000	386,000	0.082	32,000
0.010	968,000	0.016	15,000	252,000	0.086	22,000
0.012	594,000	0.019	12,000	173,000	0.088	15,000
0.015	373,000	0.024	9,000	110,000	0.093	10,000
0.020	203,000	0.029	6,000	61,000	0.102	6,000

*includes dump/stockpile

Table 14.11 Relief Canyon Mixed Mineralized Material

Cutoff	Measured Resource -Mixed					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.008	279,000	0.054	15,000	279,000	0.240	67,000
0.010	**259,000**	**0.058**	**15,000**	**259,000**	**0.251**	**65,000**
0.011	250,000	0.060	15,000	250,000	0.254	64,000
0.012	242,000	0.061	15,000	242,000	0.259	63,000
0.013	233,000	0.063	15,000	233,000	0.264	61,000
0.015	216,000	0.067	15,000	216,000	0.274	59,000
0.020	186,000	0.075	14,000	186,000	0.291	54,000
0.030	141,000	0.092	13,000	141,000	0.319	45,000

Cutoff	Indicated Resource - Mixed					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.008	193,000	0.029	6,000	193,000	0.187	36,000
0.010	**162,000**	**0.033**	**5,000**	**162,000**	**0.206**	**33,000**
0.011	151,000	0.035	5,000	151,000	0.214	32,000
0.012	142,000	0.036	5,000	142,000	0.220	31,000
0.013	133,000	0.038	5,000	133,000	0.227	30,000
0.015	117,000	0.041	5,000	117,000	0.242	28,000
0.020	84,000	0.051	4,000	84,000	0.275	23,000
0.030	50,000	0.070	3,000	50,000	0.320	16,000

Cutoff	Measured + Indicated - Mixed					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.008	472,000	0.044	21,000	472,000	0.218	103,000
0.010	**421,000**	**0.048**	**20,000**	**421,000**	**0.234**	**98,000**
0.011	401,000	0.051	20,000	401,000	0.239	96,000
0.012	384,000	0.052	20,000	384,000	0.245	94,000
0.013	366,000	0.054	20,000	366,000	0.251	91,000
0.015	350,000	0.058	20,000	350,000	0.263	89,000
0.020	270,000	0.068	18,000	270,000	0.286	77,000
0.030	191,000	0.086	16,000	191,000	0.319	61,000

Cutoff	Inferred Resource - Mixed					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.008	5,000	0.016	100	5,000	0.116	1,000
0.010	**4,000**	**0.018**	**100**	**4,000**	**0.125**	**1,000**
0.011	3,000	0.019	100	3,000	0.133	0
0.012	3,000	0.021	100	3,000	0.138	0
0.013	3,000	0.021	100	3,000	0.140	0
0.015	2,000	0.024	0	2,000	0.153	0
0.020	2,000	0.027	0	2,000	0.167	0
0.030	0	0.030	0	0	0.193	0

Table 14.12 Relief Canyon Sulfide Mineralized Material

Cutoff	Indicated Resource - Sulfide					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.010	724,000	0.032	23,000	724,000	0.237	172,000
0.015	517,000	0.041	21,000	517,000	0.281	145,000
0.020	**369,000**	**0.050**	**18,000**	**369,000**	**0.313**	**115,000**
0.030	215,000	0.069	15,000	215,000	0.342	74,000
0.050	118,000	0.095	11,000	118,000	0.366	43,000

Cutoff	Inferred Resource - Sulfide					
(oz Au/ton)	Tons	oz Au/ton	oz Au	Tons	oz Ag/ton	oz Ag
0.010	9,000	0.020	200	9,000	0.124	1,000
0.015	5,000	0.027	100	5,000	0.156	1,000
0.020	**4,000**	**0.028**	**100**	**4,000**	**0.164**	**1,000**
0.030	2,000	0.032	100	2,000	0.190	0
0.050	0	0.000	0	0	0.000	0

Figure 14.3 and Figure 14.4 show cross sections of the block model that correspond to the mineral-domain cross sections in Figure 14.1 and Figure 14.2, respectively.

Figure 14.3 Cross Section 15109100 Showing Block Model Gold Grades

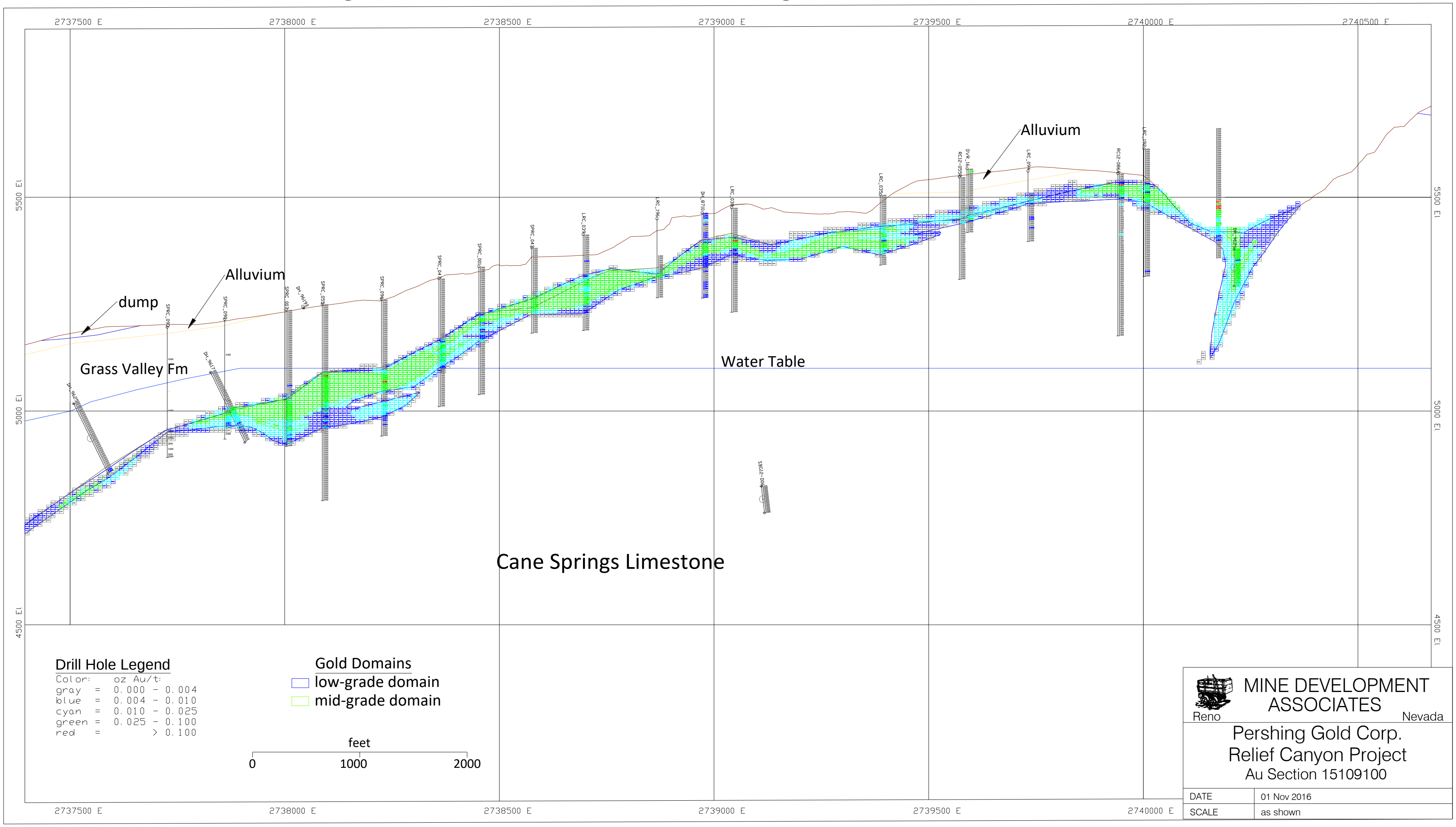
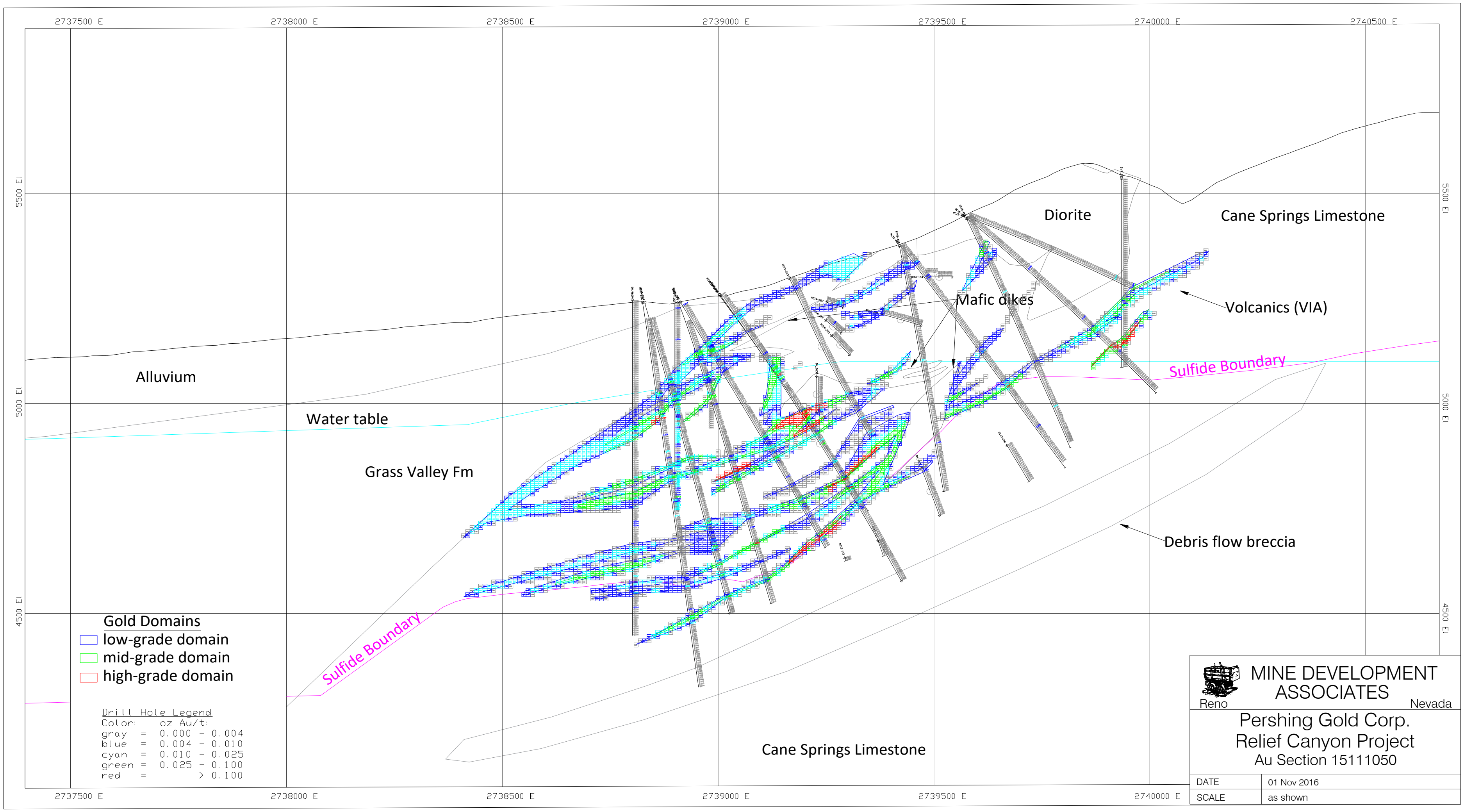


Figure 14.4 Cross Section 15111050 Showing Block Model Gold Grades



14.9 Model Checks

Volumes indicated by the sectional mineral-domain modeling were compared to the long-section volumes and those coded to the block model to assure close agreement, and all block-model coding was checked visually on the computer. Nearest-neighbor and ordinary-krige estimates of the Relief Canyon resources were undertaken as a check on the inverse-distance-cubed resource model. Grade-distribution plots of assays and composites versus the nearest-neighbor, krige, and inverse-distance block grades were also evaluated as a check on the estimation. Finally, the inverse-distance-cubed grades were visually compared to the drill-hole assay data to assure that reasonable results were obtained.

The estimate of the mined-out grade and tons can be compared to the reported production from the pits as a very rough check on the resource estimation. At a cutoff of 0.005 oz Au/ton, the MDA model estimates that a total of about 10.4 million tons of material grading 0.020 oz Au/ton (208,000 ounces) were mined by Lacana and Pegasus. Using the available production data and estimates of recoveries, approximately 202,000 ounces of gold were mined by Lacana and Pegasus, which is 3 percent lower than the MDA model. Given uncertainties regarding mining cutoff grades, reported production, and actual recoveries realized by Lacana and Pegasus, the MDA model and estimated production numbers are very close.

14.10 Comments on the Resource Modeling

The size and tenor of the Relief Canyon deposit reflects the strong structural and lithologic controls observed with detailed surface mapping and the recent Pershing Gold core drilling. The Main Zone breccia, which was the focus of past mining activity, extends at a shallow dip to the west and south beneath surface cover, while the mineralized zones at depth within the North area (represented by the Lower Zone and Jasperoid Zone) are controlled by sub-horizontal structures sub-parallel to the overlying Main Zone. Higher-angle splays and fault duplexes within the deeper structures can extend up into the Main Zone mineralization, resulting in increased brecciation and associated mineralization. MDA believes that the resource model adequately reflects the current project knowledge as it pertains to geologic controls on mineralization.

The primary risks with the resource model are the reliance on reverse circulation drilling within the Main Zone, which have been somewhat ameliorated by recent core drilling as discussed in Section 14.11, and the highly variable oxidation and its effect on metallurgical recoveries, both within the deeper portions of the Main Zone and at depth within the North area mineralized structures.

Additional core drilling in the Main Zone would result in the incorporation of detailed geologic controls that correlate with, and therefore assist in the definition of, the various mineral domains. The ability to better define the jasperoid-dominant versus fines-dominant breccias would aid in the determination of mineral domains and also would be useful in the metallurgical characterization studies.

Down-hole contamination, which is of limited extent within the Main Zone, is more readily apparent in the North area where the pre-Pershing Gold reverse circulation drilling extends beneath the water table. The current resource estimate excludes those anomalous drill-sample assay values believed to be the result of down-hole contamination and the current geologic model and estimate is based primarily on the significant amount of core drilling completed by Pershing Gold in 2011 through 2016. While the resource estimate likely still includes some intervals of possible down-hole contamination in the older reverse circulation drilling, the risk to the resource estimate is considered low. MDA recommends that future drilling below the water table be limited to core drilling.

The North area resource at depth beneath the Main Zone is open to the south, down-dip to the west below the Humboldt thrust, and to a limited extent to the east. Further core drilling is recommended to expand the resource and better define the limits of mineralization. The core drilling should be predominantly east-directed angle holes to better test the generally west-dipping structural targets. Deep high-grade targets beneath the current resource, which may require underground mining methods, are available and should be tested with exploration drilling.

The recent drilling in 2015 and 2016 has shown that the Main zone mineralization continues to the south and southwest, and can occur at depths amenable to open-pit mining. This mineralization is geochemically blind from the surface, occurring under unmineralized Grass Valley Formation, and Pershing should consider both drilling and geophysical exploration techniques to aid in expanding the resource.

14.11 Post-Resource Drilling (2016 through 2018)

Pershing has continued to conduct exploration and development drilling within and adjacent to the current 2016 resource model. Fifty-five core holes have been drilled as of the report date and the drilling is ongoing. The post-resource drilling is primarily in three areas: 1) infill drilling within the North area along the proposed west edge of the Feasibility reserve pit, 2) expansion drilling southwest of the North area resource, and 3) twin hole drilling within the Main zone mineralization southwest of the historical South pit. Drilling in the first two target areas is meant to upgrade the resource model, resulting in a potential enlargement of the reserve pit, and also to extend the current mineral resource farther to the west and southwest. The Main zone drilling is twinning pre-Pershing reverse circulation drill holes to confirm and provide confidence in the resource model. Upon completion of the current drill program, Pershing is planning on updating the resource model in the second half of 2018.

Drill results to-date from eight infill holes in the first target area are on average about 30% higher in gold grade than predicted within the resource model. It is likely that these results would have a positive impact on the local reserve economics and should result in an enlargement of the current Feasibility pit.

The resource expansion drilling southwest of the North area (seven holes completed to-date) has returned favorable results within the Lower and Jasperoid zones with mineral intercepts of the same tenor as seen up-dip within the current resource model. These intercepts are over an area of approximately 700 feet northeast/southwest and 500 feet northwest/southeast and mineralization is still open to the west and southwest. Cyanide solubility testing indicates the material is amenable to standard heap-leach processing as proposed in the current Feasibility Study and there are plans to follow up with metallurgical testing of composites from these holes. There is the potential that these positive drill results, especially if further drilling returns similar results, could drive further pit expansion.

The third area of drilling is in the Main Zone, within and immediately to the west of the current reserve pit. Eight core holes have been drilled twinning pre-Pershing Main Zone reverse circulation drilling and an additional five core holes were drilled to infill gaps in the Main Zone drill spacing and test the western extension of Main Zone mineralization. The infill drill results indicate only a minor variation with the predicted resource estimate grades and there is close match in geology and sample results between the

Pershing and pre-Pershing drill holes. The positive Main zone twin drill results serve to verify and provide confidence in the pre-Pershing drilling, which forms the basis for much of the Main Zone model and resource estimate. The twin and expansion holes also confirm a previous observation that the historical drilling may not have been deep enough to fully test both the base of Main zone mineralization and also the western extension of mineralization downdip from the current reserve pit. Ore grade intercepts in drill holes 516 and 512, holes located 600ft apart along a north-trending strike, and 400ft to 700ft, respectively, west of the reserve pit, indicate that Main zone mineralization is present to the west past the current drill spacing. Mineralization is open between these two holes and also farther along strike to the north. Drilling is ongoing to determine the extent of this opportunity for resource and reserve growth.

15.0 MINERAL RESERVE ESTIMATE

CIM mineral reserve definitions are given below, with CIM's explanatory guidance shown in italics:

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

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.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

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.

The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

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.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

Proven and Probable Mineral Reserves for the Relief Canyon deposit are shown in Table 15.1 based on applying modifying factors to the Measured and Indicated resources estimated with an effective date of November 1, 2016. Section 16 describes the pit optimization procedure and economic and pit design parameters applied as modifying factors used by Mr. Prenn, the Qualified Person, responsible for these sections in this Feasibility Study. The resulting pit optimizations were used to design an ultimate pit based on three-phase design (see Section 16.0 for design parameters) containing the Proven and Probable material shown in Table 15.1 that constitutes the reserves for the property. The final pit used in this Feasibility Study is based on the Pre-Feasibility design, but the cutoff grade for mixed and oxide materials was increased to 0.025 oz Au/ton. Pit optimizations and designs were based on a gold price of $1,300 per ounce as used for previous studies. A final gold price for economics and reserves of $1,250 per ounce was used for estimating the reserves. The difference between pit designs completed for a $1,300 and a $1,250 gold price is minimal and not considered to be material. Mineral Reserves are stated using a cutoff grade of 0.005 oz Au/ton, and the detailed mine economics shown in Sections 0 and 22.0 show that the Proven and Probable Reserves are economic. The Mineral Reserves have an effective date of May 24, 2018. The Proven and Probable Reserves are contained in the designed Feasibility final pit. As stated in Section 14, the mineral resources include the Mineral Reserves; specifically, the Measured and Indicated oxide resources are inclusive of the estimated Reserves.

Table 15.1 Relief Canyon Mineral Reserves at 0.005 oz Au/ton Cutoff

Classification	**Tons**	**Grade**	**Oz Au**
	000's	**oz Au/ton**	**000's**
Proven	13,013.1	0.024	307.3
Probable	17,225.1	0.019	324.0
Proven & Probable	**30,238.1**	**0.021**	**631.3**

About 42 percent of the resource model blocks contain an estimated silver grade. The Proven and Probable reserves that contain a silver grade estimate are summarized in Table 15.2

Table 15.2 Relief Canyon Mineral Reserves with Silver Grade at 0.005 oz Au/ton Cutoff

Classification	**Tons**	**Grade**	**Oz Ag**
	000's	**oz Ag/ton**	**000's**
Proven	10,185.6	0.121	1,231.2
Probable	3,914.4	0.093	365.9
Proven & Probable	**14,100.0**	**0.113**	**1,597.1**

All of Pershing Gold's mine permits and licenses for the Relief Canyon project are in good standing with no outstanding notices of deficiency or unresolved compliance issues. Permits issued by the BLM and the NDEP/BMRR authorize the Phase 1 pit expansion and deepening to a pit bottom elevation of 5,080 feet AMSL, heap leach stacking to a height of 200 feet on the approved leach pads and processing of solutions in the existing Adsorption-Desorption Recovery ("ADR") plant. These permits also include the construction of a new waste rock storage area on private land in Section 17, Township 27 North, Range 34 East. The permitting status is described in more detail in Sections 4.4 and 20.0, and their limitations on mining of the reserves are discussed in Section 16.0. Mr. Prenn is not aware of any legal, political, or other risks that could materially affect the development of the Mineral Reserves.

16.0 MINING METHODS

The Relief Canyon deposit has been mined in the past by open-pit methods. This Feasibility Study assumes mining by open-pit methods for the proposed renewal of mining. Conveyor hauage from the crusher and contact mining was considered to be the base case for the study.

16.1 Pit Optimization

To design final and intermediate phase pits, Whittle Pit Optimization was completed for the in-situ resources at the Relief Canyon mine. Table 16.1 summarizes the pit optimization assumptions used. Note that at the start of the study there was a consideration to not crush and agglomerate lower grade materials and utilize ROM methods, but all ore grade materials are now planned to be crushed and agglomerated. A number of pit designs and production schedules were completed, ranging from three to six million tons of ore grade material per year. The final production schedule planned to mine six million tons of ore grade material annually. After the pit optimization was completed, along with a number of pit design and scheduling iterations, it was decided to crush and agglomerate all material above a 0.005 oz Au/ton cutoff grade, so no ROM material will be shown in the final production schedule.

Table 16.1 Pit Optimization Parameters

Mining Cost	$/ton		Comments
Mining Cost	$1.95		Contract mine
Processing Cost	**Crusher $/ton**	**ROM $/ton**	**Comments**
Crush, Convey, Process	$4.00		
ROM Process		$1.31	
Gold Recovery %	83.0%	65.0%	
Minimum Grade oz Au/ton	0.008	0.005	
Other			
Base Case Metal Price $/oz Au	$1,300	$1,300	
Transport, Refining $/oz Au	$15	$15	

The economic parameters used for pit optimization provide a cutoff grade below 0.004 oz Au/ton, so minimum cutoff grades were used to keep the pit optimization from considering material below 0.005 oz Au/ton from being included as ore grade material for the pit optimization. The cutoff grade used for this study for all ore grade material is 0.005 oz Au/ton.

The initial pit optimization that limited the depth of the pit to the 5,080 elevation was used to design the Phase 1 pit for the Feasibility mine plan. The Phase 1 pit was designed to be inside the permit boundary, but a small portion is outside the eastern boundary; this is in an area with prior disturbance (waste dump).

The second pit optimization used the existing permit boundary but was allowed to mine below the 5,080 elevation. An optimized pit utilizing an approximate $600 gold price was used as a template to design a Phase 2 pit.

The final pit optimization used a base price of $1,300 per ounce of gold to be consistent with previous studies. The final economics and reserve statements are based on $1,290 per ounce of gold. The difference between pit designs completed for a $1,300 and a $1,290 gold price pit shell is minimal and not considered to be material.

Golder Associates ("Golder") completed a pit slope study for the PFS, which also applies to the Feasibility Study as the pit designs are the same. Figure 16.1 shows Golder recommended slopes, which were used in the pit optimization study. Golder's pit slopes ranged from 40 to 49 degrees and were based on triple benching 20-foot benches, a 63 to 67-degree face angle, and a catch bench to make the desired inter-ramp slope. The recommended Golder parameters are shown in Table 16.2 and Table 16.3 The width of the catch bench varies as tabulated in Table 16.3.

Table 16.2 Golder Associates Recommended Slope Parameters

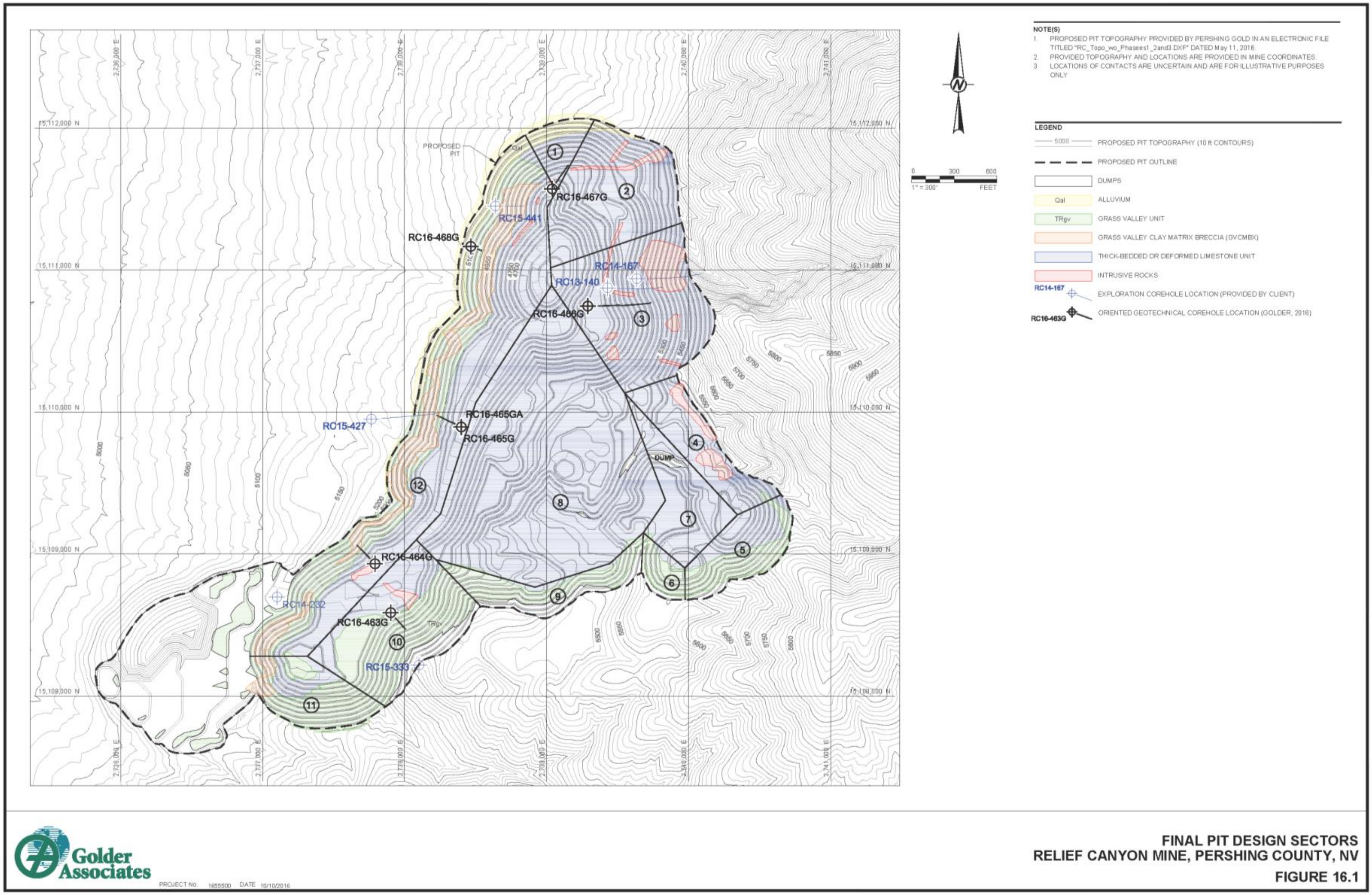
Geotechnical Unit	Slope Dip Direction[1]	Sectors[2]	Operating Practice	Bench Configuration and Height	Design Catch Bench Width (ft)	Design Bench Face Angle (degrees)	Design Inter-Ramp Slope (Degrees)
Alluvium	All	1,2, and 12	Dozer Trim and Track Hoe Scaling	Triple Bench (3 x 20 ft) 60ft Between catch benches	30	63° (0.5H:1V)	45°
Limestone (Thick Bedded and Deformed)	190° to 250°	2	Trim Blasting[3] and Track Hoe Scaling (scale bench face to goliation where required)	Triple Bench (3 x 20 ft) 60ft Between catch benches[5]	30	63° (0.5H:1V)	45°
Grass Valley Formation (Foliated and Breccia)	160° to 250°	none	N/A	N/A	N/A	N/A	N/A
Limestone (Thick Bedded and Deformed)	250° to 160°	1,3,4,5, 7,8,10, 11, and 12	Trim Blasting[3] and Track Hoe Scaling	Triple Bench (3 x 20 ft) 60ft Between catch benches	27	63° (0.5H:1V)	45°
Grass Valley Formation (Foliated and Breccia)	250° to 160°	5,6,9,10,11 and 12	Trim or Cushion Blasting with Track Hoe Scaling	Triple Bench (3 x 20 ft) 60ft Between catch benches	Varies w/ Slope Height	67°	45° to 49° Depending on Slope Height

Notes: 1) Measured clockwise
2) Location of sectors shown in Figure 16.1
3) Pre-splitting under benches to define crest may be appropriate where limestone is massive

Table 16.3 Golder Associates Variable Recommended Slope Parameters

Slope Height (Feet)	Pit Sector	Operating Practice	Bench Configuration and Height (Feet)	Bench Face Angle (Degrees)	Catch Bench Width (Feet)	Design Inter-Ramp Angle (Degrees)
240	All	Trim or Cushion Blasting with Track Hoe Scaling	Triple Bench (3 x 20 ft) 60ft Between catch benches	67°	27	49°
300					27	49°
340					27	49°
400					31	47°
440					35	45°

Figure 16.1 Final Pit Design Sectors



The initial base case pit optimization results are shown in Table 16.4.

Table 16.4 Base Case Pit Optimization Results

Pit	Gold Price $/oz Au	Ore Tons 000's	Waste Tons 000's	Total Tons 000's	Strip Ratio t w/t o	Gold Grade oz Au/ton	Gold Ounces 000's
1	390	1,281.6	1,626.7	2,908.3	1.27	0.036	46.4
3	442	2,059.6	2,722.6	4,782.1	1.32	0.032	66.5
5	494	3,276.9	5,742.2	9,019.1	1.75	0.031	102.7
7	546	4,885.9	7,690.7	12,576.7	1.57	0.027	133.5
9	598	10,811.3	30,229.2	41,040.5	2.80	0.028	300.5
11	650	13,148.2	36,267.2	49,415.3	2.76	0.027	351.7
13	702	15,594.9	41,890.1	57,485.0	2.69	0.026	399.0
15	754	16,659.9	43,254.8	59,914.7	2.60	0.025	413.8
16	780	17,710.5	46,996.2	64,706.7	2.65	0.025	435.4
17	806	18,433.5	48,491.0	66,924.4	2.63	0.024	446.2
18	832	19,353.6	51,997.6	71,351.2	2.69	0.024	464.6
19	858	20,362.2	56,527.4	76,889.6	2.78	0.024	485.9
20	884	21,046.9	58,460.2	79,507.1	2.78	0.024	496.6
21	910	21,928.5	61,319.6	83,248.2	2.80	0.023	511.2
22	936	22,263.8	62,364.5	84,628.4	2.80	0.023	516.5
23	962	22,876.0	63,300.6	86,176.5	2.77	0.023	523.2
24	988	24,506.6	69,097.6	93,604.1	2.82	0.022	549.2
25	1014	25,029.5	71,229.0	96,258.5	2.85	0.022	558.1
26	1040	25,435.0	72,124.6	97,559.6	2.84	0.022	562.9
27	1066	26,044.6	74,327.2	100,371.7	2.85	0.022	571.9
28	1092	26,626.4	77,886.4	104,512.8	2.93	0.022	583.9
29	1118	26,961.7	79,044.8	106,006.5	2.93	0.022	588.4
30	1144	28,160.8	83,860.6	112,021.4	2.98	0.022	606.2
31	1170	28,587.6	85,707.1	114,294.7	3.00	0.021	612.6
32	1196	28,808.0	86,912.6	115,720.6	3.02	0.021	616.4
33	1222	29,342.3	88,868.4	118,210.7	3.03	0.021	623.4
34	1248	29,951.0	92,761.0	122,712.1	3.10	0.021	634.5
35	1274	30,289.8	94,541.5	124,831.3	3.12	0.021	639.8
36	**1300**	**30,769.7**	**97,782.9**	**128,552.6**	**3.18**	**0.021**	**648.5**
37	1326	31,101.2	100,012.9	131,114.1	3.22	0.021	654.4
38	1352	31,324.3	101,486.3	132,810.6	3.24	0.021	658.2
39	1378	31,402.3	101,718.6	133,120.9	3.24	0.021	659.0
40	1404	31,575.1	102,442.1	134,017.2	3.24	0.021	661.1
41	1430	31,833.7	103,968.1	135,801.8	3.27	0.021	665.0
42	1456	31,931.3	104,496.0	136,427.3	3.27	0.021	666.3
43	1482	32,277.7	107,152.1	139,429.8	3.32	0.021	672.4
44	1508	32,423.9	108,047.4	140,471.3	3.33	0.021	674.5
46	1560	32,917.4	111,886.2	144,803.6	3.40	0.021	682.8
48	1612	33,537.3	115,558.4	149,095.7	3.45	0.021	691.1
50	1664	33,813.5	117,518.6	151,332.1	3.48	0.021	695.2
52	1716	34,019.1	119,226.2	153,245.3	3.50	0.021	698.5
54	1768	34,162.7	120,056.9	154,219.6	3.51	0.02	700.2
56	1820	34,363.4	121,372.7	155,736.2	3.53	0.02	702.8
58	1872	34,639.6	123,629.4	158,269.0	3.57	0.02	706.9
60	1924	34,895.6	125,779.0	160,674.6	3.60	0.02	710.6
62	1976	35,111.3	127,409.8	162,521.1	3.63	0.02	713.4
70	2184	35,660.5	131,893.9	167,554.5	3.70	0.02	720.5
80	2444	36,401.1	138,189.9	174,591.0	3.80	0.02	729.5
86	2600	36,752.9	141,852.9	178,605.8	3.86	0.02	734.1

16.2 Mine Operating Plan

The Relief Canyon mine is currently permitted to mine to a pit bottom elevation of 5,080 feet AMSL. Prior mining was concentrated in three open pits, with several waste facilities, a heap-leach pad, a heap-leach fluid process plant, and associated infrastructure. The mine site has been in temporary closure for a number of years. A modified Plan of Operations was filed in August 2015 with the BLM and the NDEP. The agencies approved this Plan of Operations and associated permit modifications in 2016 and 2017. The Plan was further modified in February 2018.

A six month mine pre-production period is planned to start when financing for the operation is in place, and during the process and heap construction, which is expected to require an eight month pre-production period. The mine production schedule was developed by designing a Phase 1 mine plan that was planned only to mine to the 5,080 elevation, and a Phase 2 plan that was developed to stay inside the current permit boundary, but mine below the 5,080 elevation. Using this plan would enable ore supply from the design Phase 1 pit and mostly waste stripping from the design Phase 2 pit. Annual maximum ore production is planned to be about six million tons (about 16,700 tons per day), while total production is limited to a total of about 31 million tons of material. Both of these design pit phases utilized out-of-pit haul roads to move material that was above the 5,220 to 5,240 elevation and move the remainder of material out of the pit by nominal 85 foot wide, in-pit, 10 percent ramp systems. Phase 1 was later redesigned to include some material that was originally in the design Phase 2 pit, as well as some material that was outside of the permit boundary in the Light Bulb area.

A total of three design pit phases were used, and these pits were split into seven scheduling pit phases. Figures 16.3, 16.4, and 16.5 show the design Phase 1, Phase 2, and Phase 3 (Final) pits, respectively. Scheduling pit Phases 3 and 4 used the material that could be mined from design Phases 1 and 2, respectively, below the 5,080 elevation.

The final design Phase 3 was split into three scheduling pit Phases of 5, 6, and 7. For scheduling production Phase 3 was split into north and south areas. Scheduling Phase 5 corresponds to the north area of Phase 3. These scheduling phases are apparent in the end of year drawings in Figures 16.6 through 16.12. Scheduling Phases 6 and 7 are the south portion of design Phase 3. Scheduling Phase 6 establishes a ramp system to the 4,940 bench. As scheduling Phase 7 is mined, material is transported down to the 4,940 elevation and then out of the pit with the design Phase 3 ramp system.

The ore-grade material mined will be processed using a new heap-leach pad close to the existing pad and associated tanks, ponds, and the existing ADR facility that will be updated and modified. Water for the mining and heap-leach operation will be obtained from the existing water supply wells, PW-1 and PW-2, located west of the pit area. However, new wells will be necessary for both water supply and dewatering the pit over the life of the project. Power will be obtained from the existing power supply system consisting of an overhead power line and on-site generators. A new crushing plant will be installed capable of producing about 16,700 tons per day of crushed material to a size of 80 percent passing a 3 inch screen, with a conveyor system that will transport and stack material on the leach pad.

A permit modification application for Phase II of the project will be submitted shortly after the completion of this study, which will be used as the foundation of the modification application. This permit is expected to be granted prior to 2 years of production from material from within the current permit and above the

5,080 elevation (about third quarter 2020). Detailed plans need to be developed to allow mine production to continue should the permit take longer than expected, or if production proceeds faster than planned.

16.3 Pit Design

Three main pit phases were designed utilizing the pit optimization results as templates for the design. All of the pit phases were split into at least two scheduling pit phases. A total of seven scheduling pit phases were used to develop the production schedule. Table 16.5 shows the relationship of the design pit phases to the scheduling pit phases.

Table 16.5 Design and Scheduling Pit Phases

Design Pit Phase	Top Bench Elev Ft	Bottom Bench Elev Ft	Scheduling Pit Phase	Description	Pit Limits
1	5720	5080	1	Design Phase 1 to 5080 Elevation	above water table - low stripping & optimization
2	5600	5080	2	Design Phase 2 to 5080 Elevation	above water table & optimization
1	5060	5000	3	Design Phase 1 from 5060 to 5000 Elevation	permit boundary & optimization
2	5060	4740	4	Design Phase 2 from 5060 to 4740 Elevation	permit boundary & optimization
3	5720	4520	5	North portion of design Phase 3	optimization
3	5440	4940	6	Establish south temporary ramp system to 4940	establish temporary ramp (preliminary design)
3	5440	4840	7	Complete south design Phase 3 - mine out south ramp	optimization

The Phase 1 pit extends slightly beyond the southern permit boundary on previously disturbed ground and finalizes the southeast portion of the pit. The final pit is designed based on the $1,300 per ounce gold, base-case optimized pit, and includes an internal ramp system. Figure 16.3 shows the Phase 1 designed pit with the southeast area expanded to the final pit, while Figure 16.4 shows the Phase 2 designed pit, with the northeast area expanded to the final pit. Figure 16.5 shows the designed final pit. It should be noted that the final pit contains about 20 percent more waste than the optimized pit. The resulting ultimate pit design is approximately 14 percent larger than the optimized pit due to inclusion of the ramp system and requirements for mining room between pit phase designs. It may be possible to improve on this design, but it is important to consider all pit phases when redesigning the final pit. Some of the additional waste included in the final design is due to the permit boundary size, which causes access issues between the pit phases requiring additional waste to be mined to maintain working space between the pit phases. If the boundary on the east and south could be expanded by 200 feet, design issues and stripping may be reduced.

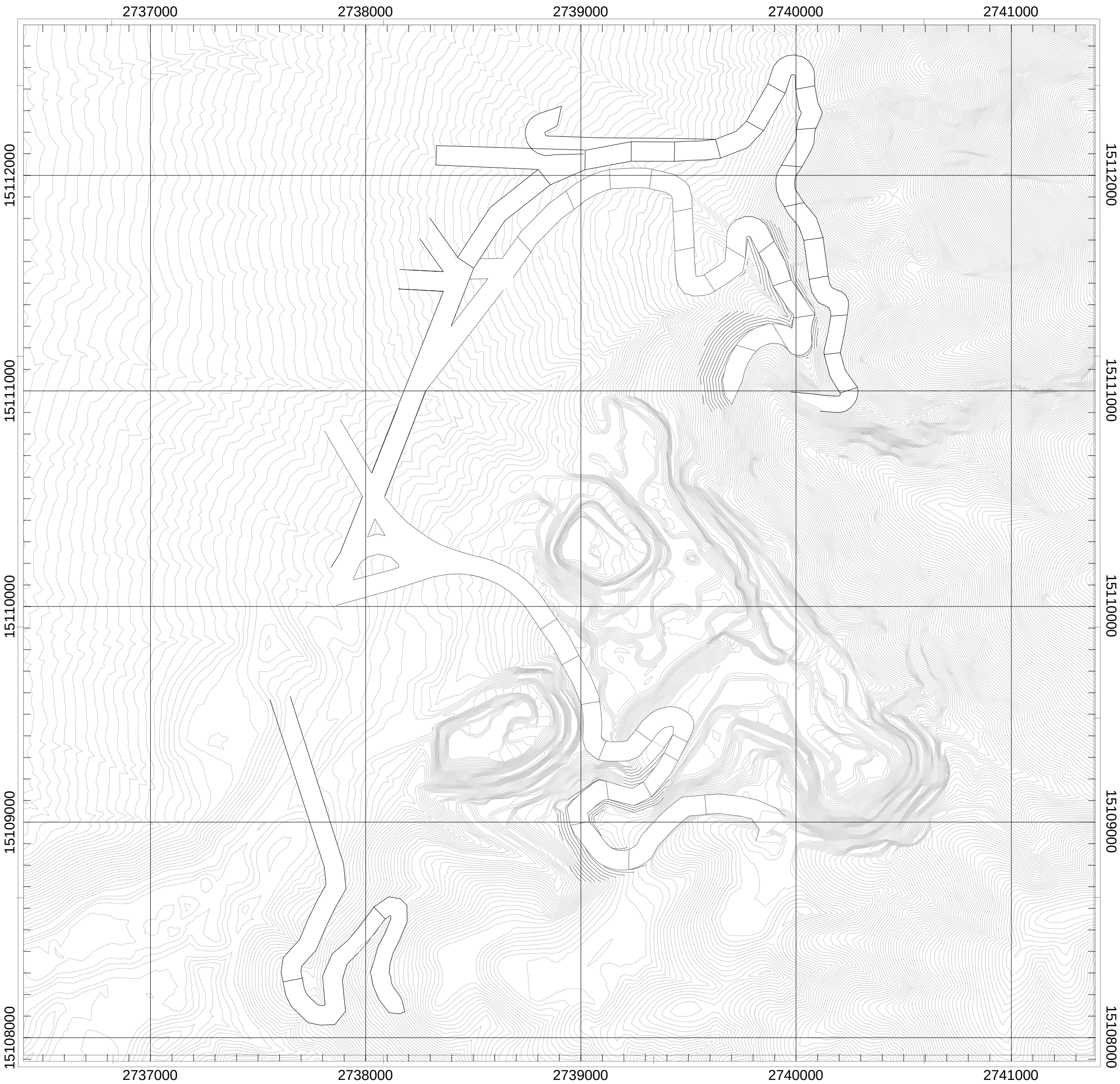
The total material in each pit phase is shown in Table 16.6.

Table 16.6 Summary of Material by Pit Phase

Design Phase	Tons Ore 000's	oz Au/ton	Ounces Au 000's	Rock Waste Tons 000's	Alluvium Tons 000's	Old Dump Tons 000's	Total Waste Tons 000's
1	11,914.3	0.016	196.1	15,859.5	907.9	725.8	17,493.2
2	6,591.9	0.021	135.3	25,817.3	4,880.5	0.6	30,698.4
3	11,731.9	0.026	299.8	61,750.6	5,176.3	1,791.3	68,718.2
Totals	**30,238.1**	**0.020878**	**631.3**	**103,427.4**	**10,964.7**	**2,517.7**	**116,909.8**

Most of the access to material in each pit phase above the 5,220 or 5,240 elevation will be from access roads prepared prior to mining the pit phase. In the case of the Phase 1 access road, most of this road will be mined out as mining in the phase proceeds. In the case of Phase 2 and Phase 3, access to the upper pit benches will be by roads that will be constructed mostly outside the pit phase as mining proceeds. Figure 16.2 shows the out-of-pit access roads for all the pit phases. Design Phase 1, Phase 2, and the ultimate pit designs are shown in Figure 16.3, Figure 16.4, and Figure 16.5 respectively.

Figure 16.2 Pit Access Roads



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PROJECT

RELIEF CANYON GOLD PROJECT
PERSHING COUNTY, NEVADA

CONSULTANT

MINE DEVELOPMENT
ASSOCIATES

TITLE

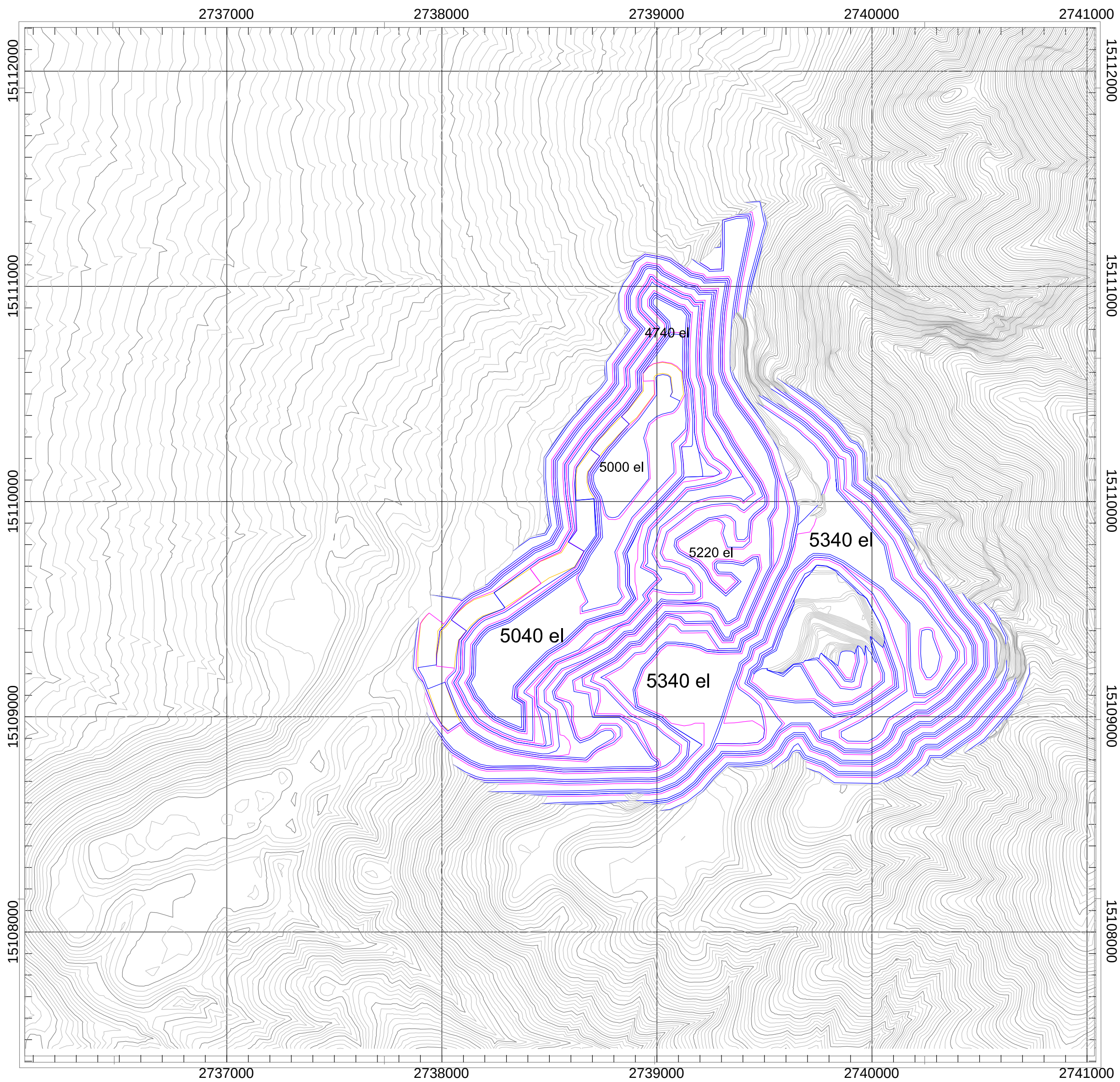
PIT ACCESS ROADS

N
W E
S

0 500 1000
Feet

COORDINATE SYSTEM: State Plane Nevada West Zone NAD 83
MM-DD-YYYY: 16-May-17

Figure 16.3 Phase 1 Pit Design



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PROJECT
RELIEF CANYON GOLD PROJECT
PERSHING COUNTY, NEVADA

CONSULTANT
MINE DEVELOPMENT
ASSOCIATES

TITLE
PHASE 1 PIT MAP

N W E S
0 500 1000 Feet

COORDINATE SYSTEM: State Plane Nevada West Zone NAD 83
MM-DD-YYYY: 16-May-17

Figure 16.4 Phase 2 Pit Design

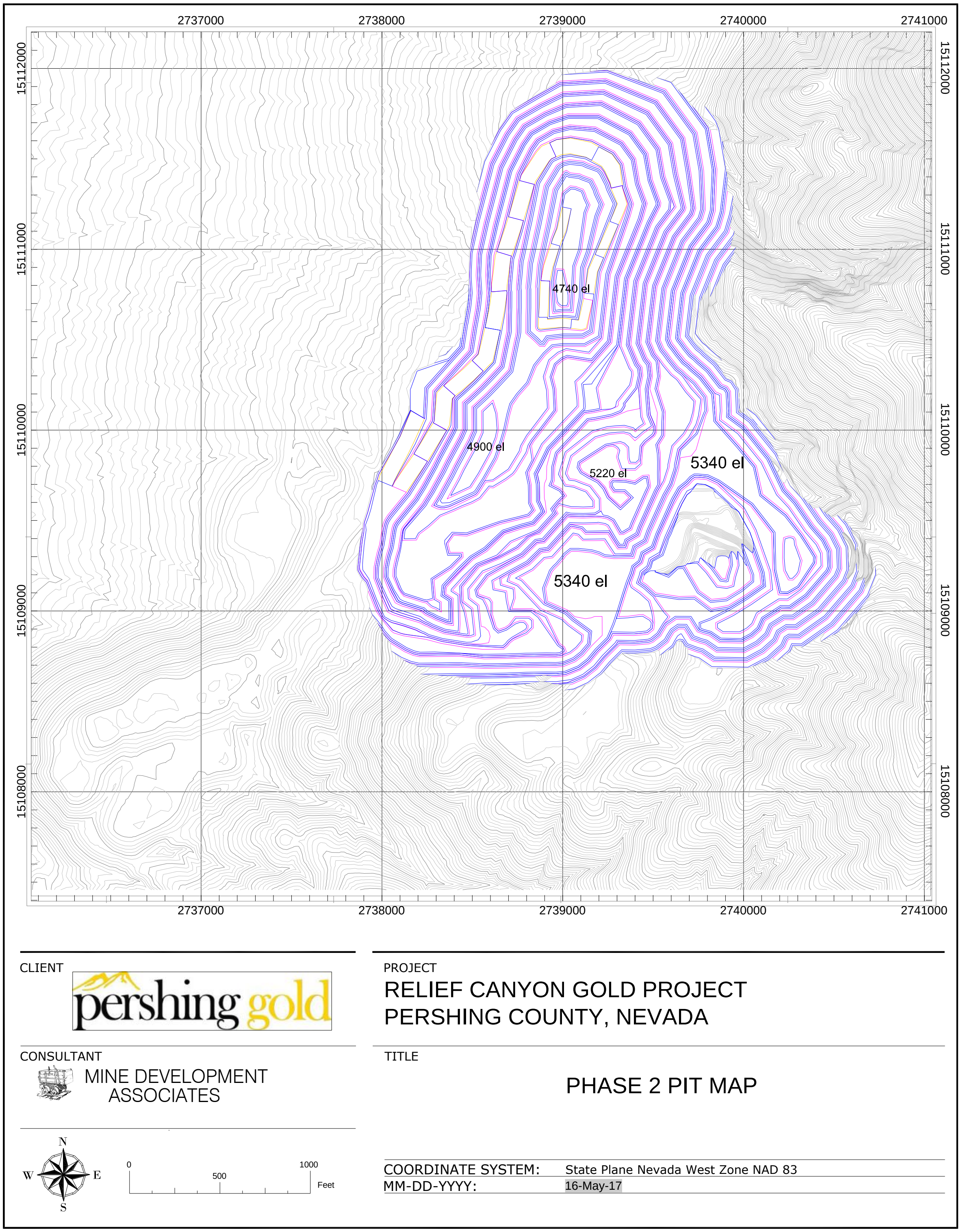
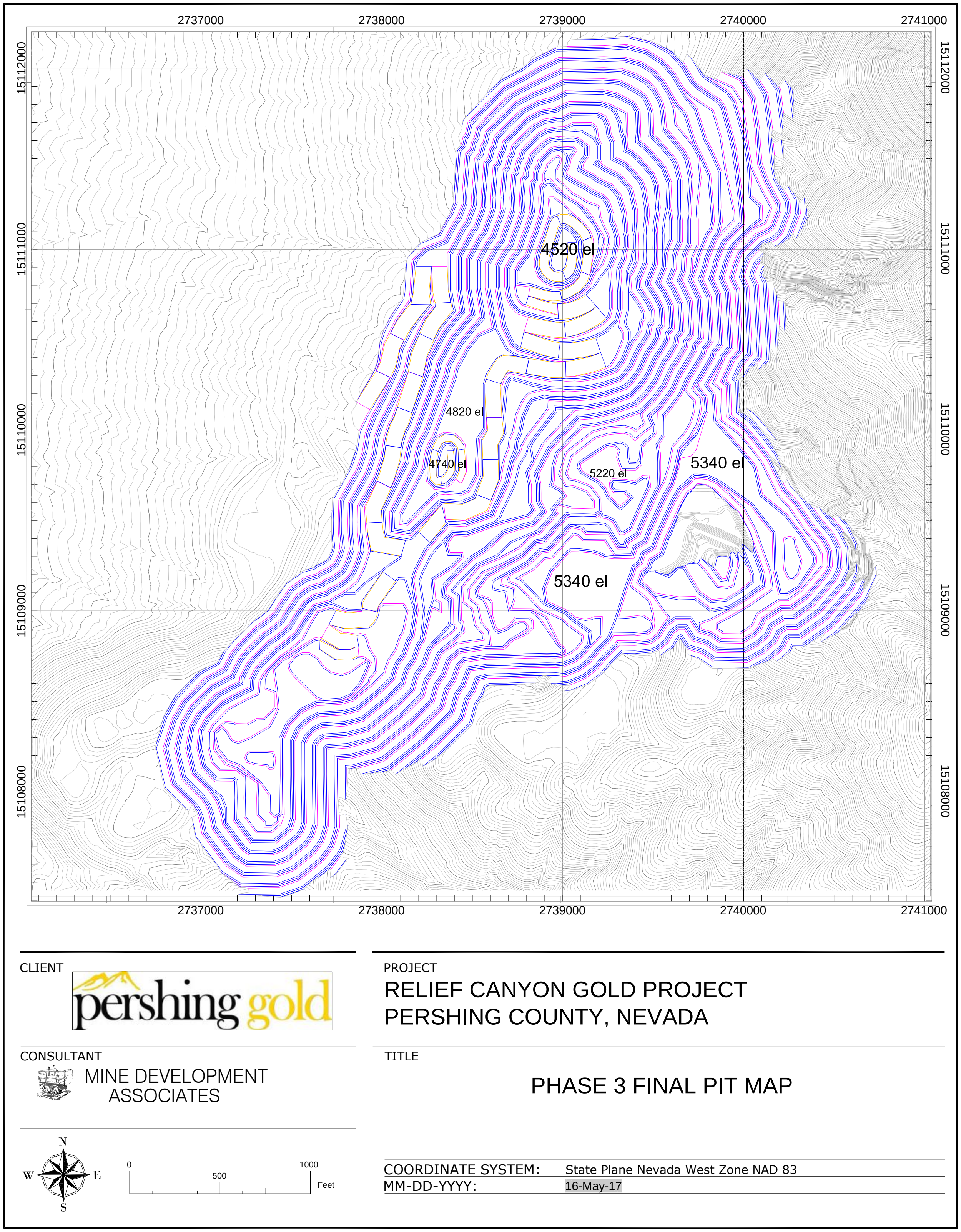


Figure 16.5 Final Pit Design



A production schedule was developed based on crushing a maximum of six million tons of material per year. All ore grade material that is planned to be crushed is defined by a cutoff grade of greater than or equal to 0.005 oz Au/ton of material classed as Proven or Probable. The production schedule assumes a four month mine pre-production schedule starting in the second quarter of 2018 and assumes pre-production (Year -1) mining will commence during the third quarter of 2018 Table 16.7 shows the material moved each year in the schedule.

Table 16.7 Mine Production Schedule

Scheduling Phase	Design Phase	Material	Units	Pre-production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Phase 1	1	Ore to Crusher	000's Tons	24	5,377	5,126	-	-	-	-	10,526
			Oz Au/ton	0.014	0.016	0.017	-	-	-	-	0.016
			000's oz Au	0	86	88	-	-	-	-	174
Phase 2	2	Ore to Crusher	000's Tons	-	510	807	8	-	-	-	1,324
			Oz Au/ton	-	0.009	0.015	0.008	-	-	-	0.013
			000's oz Au	-	4	12	0	-	-	-	17
Phase3	1	Ore to Crusher	000's Tons	-	-	-	1,386	-	-	-	1,386
			Oz Au/ton	-	-	-	0.016	-	-	-	0.016
			000's oz Au	-	-	-	23	-	-	-	23
Phase4	2	Ore to Crusher	000's Tons	-	-	-	3,835	1,432	-	-	5,267
			Oz Au/ton	-	-	-	0.020	0.029	-	-	0.022
			000's oz Au	-	-	-	78	41	-	-	118
Phase5	3	Ore to Crusher	000's Tons	-	-	-	688	3,084	3,571	1,174	8,518
			Oz Au/ton	-	-	-	0.020	0.022	0.027	0.027	0.025
			000's oz Au	-	-	-	14	67	97	32	210
Phase6	3	Ore to Crusher	000's Tons	-	-	-	-	-	972	-	972
			Oz Au/ton	-	-	-	-	-	0.029	-	0.029
			000's oz Au	-	-	-	-	-	29	-	29
Phase7	3	Ore to Crusher	000's Tons	-	-	-	-	-	25	2,217	2,242
			Oz Au/ton	-	-	-	-	-	0.025	0.027	0.027
			000's oz Au	-	-	-	-	-	1	61	62
Total		Ore to Crusher	000's Tons	24	5,886	5,932	5,918	4,516	4,568	3,392	30,237
			Oz Au/ton	0.014	0.015	0.017	0.019	0.024	0.028	0.027	0.021
			000's oz Au	0	90	100	114	108	126	93	631
Phase 1	1	Waste to Dump	000's Tons	1,661	8,799	6,170	-	-	-	-	16,630
Phase 2	2	Waste to Dump	000's Tons	-	6,951	13,770	238	-	-	-	20,958
Phase 3	1	Waste to Dump	000's Tons	-	-	-	865	-	-	-	865
Phase 4	2	Waste to Dump	000's Tons	-	-	-	7,267	2,473	-	-	9,740
Phase 5	3	Waste to Dump	000's Tons	-	-	-	10,660	21,682	8,222	2,501	43,066
Phase 6	3	Waste to Dump	000's Tons	-	-	-	-	2,618	9,824	-	12,443
Phase 7	3	Waste to Dump	000's Tons	-	-	-	-	-	8,086	5,124	13,210
Total		Mine Waste Du	000's Tons	25	701	-	-	578	1,214	-	2,518
Total		Alluvium	000's Tons	382	2,255	3,147	788	2,905	1,476	7	10,961
Total		Rock Waste	000's Tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618	103,433
Total		Total Waste	000's Tons	1,945	15,466	19,939	19,029	26,774	26,132	7,625	116,911
Total		Total Mined	000's Tons	1,969	21,353	25,872	24,947	31,290	30,701	11,017	147,148
Total		Strip Ratio	W:O	80.28	2.63	3.36	3.22	5.93	5.72	2.25	3.87

Figure 16.6 through Figure 16.12 show pit maps for mining progress by year.

Figure 16.6 Pre-production Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan

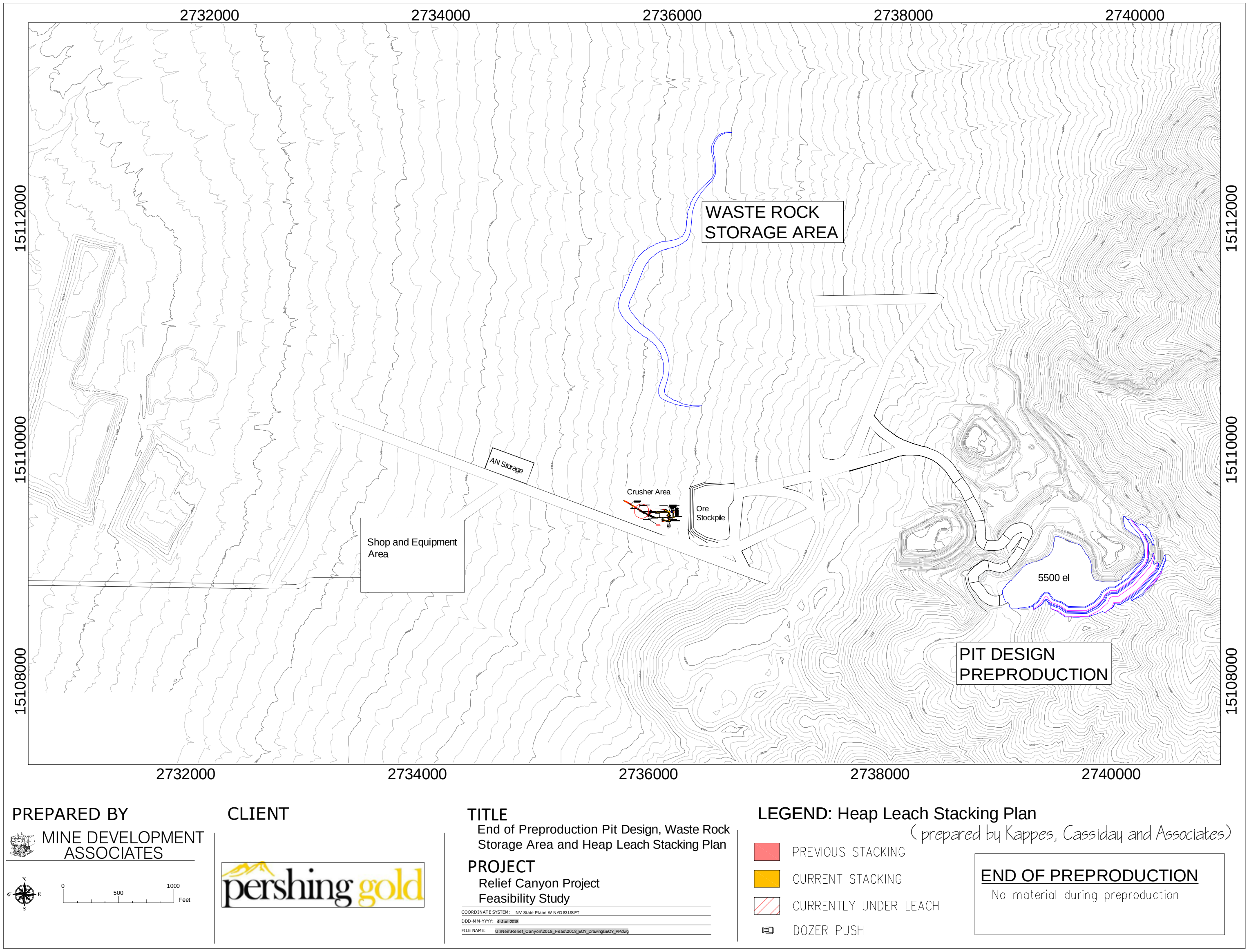


Figure 16.7 Year 1 Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan

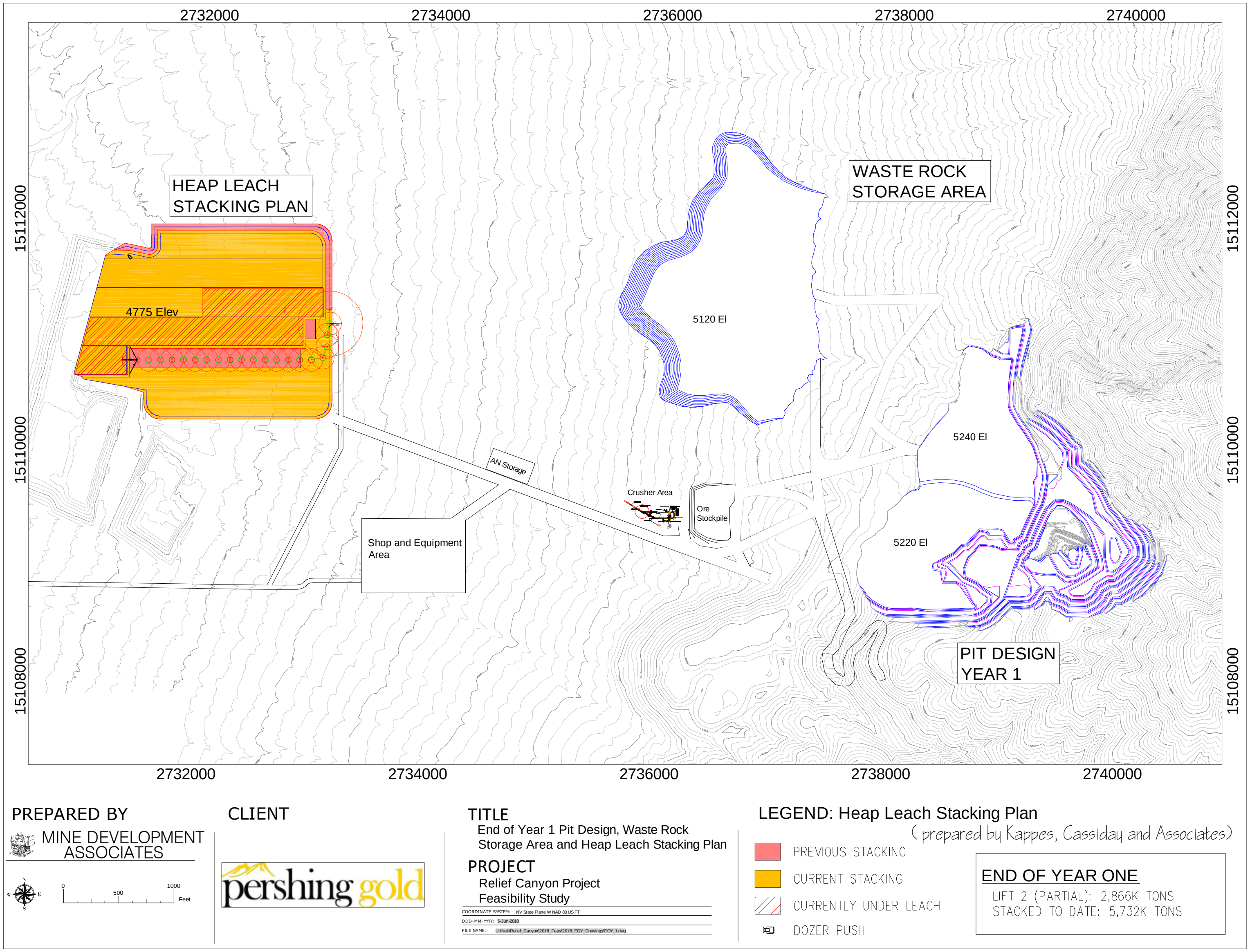


Figure 16.8 Year 2 Pit Design, Waste Rock Storage Area an d Heap Leach Stacking Plan

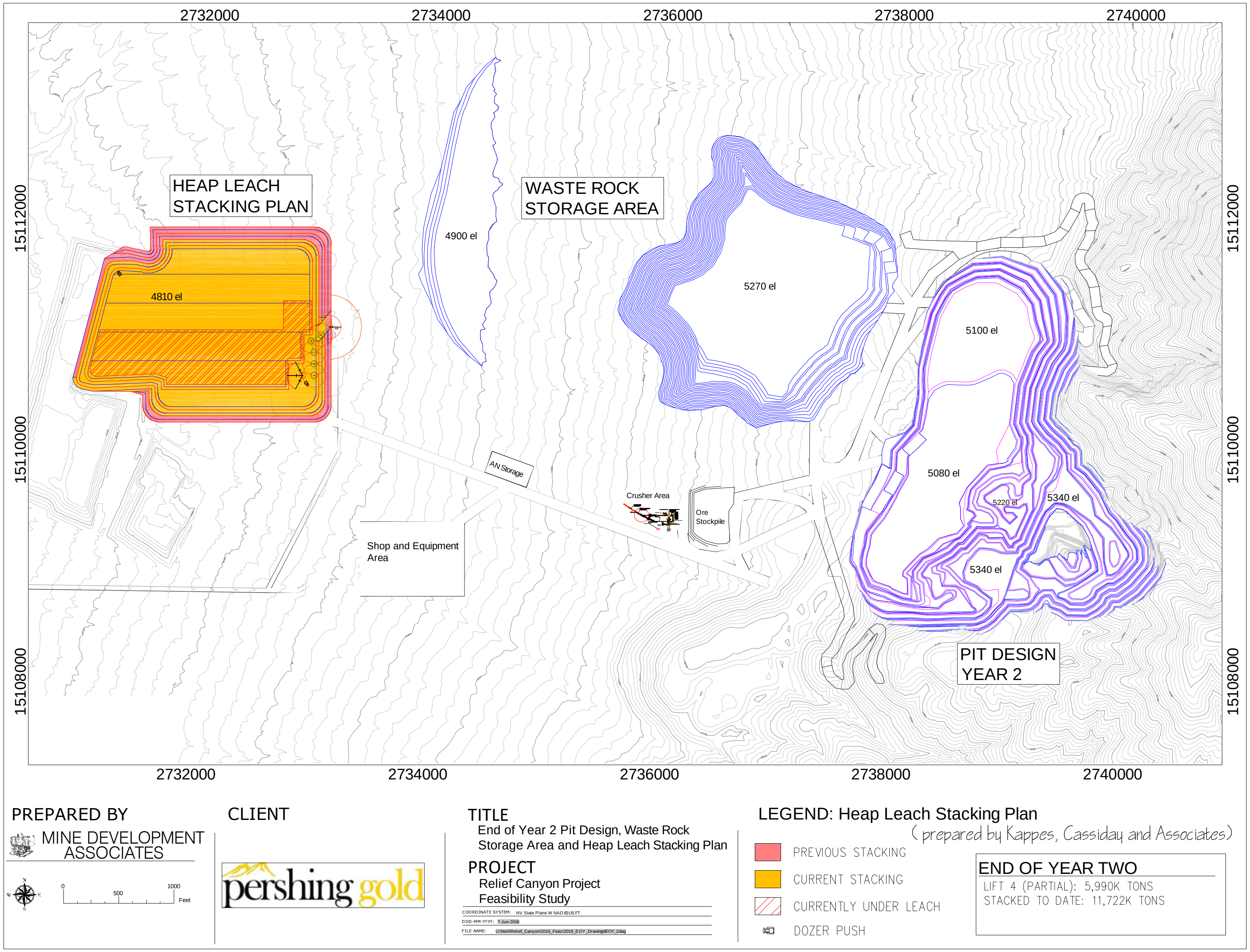


Figure 16.9 Year 3 Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan

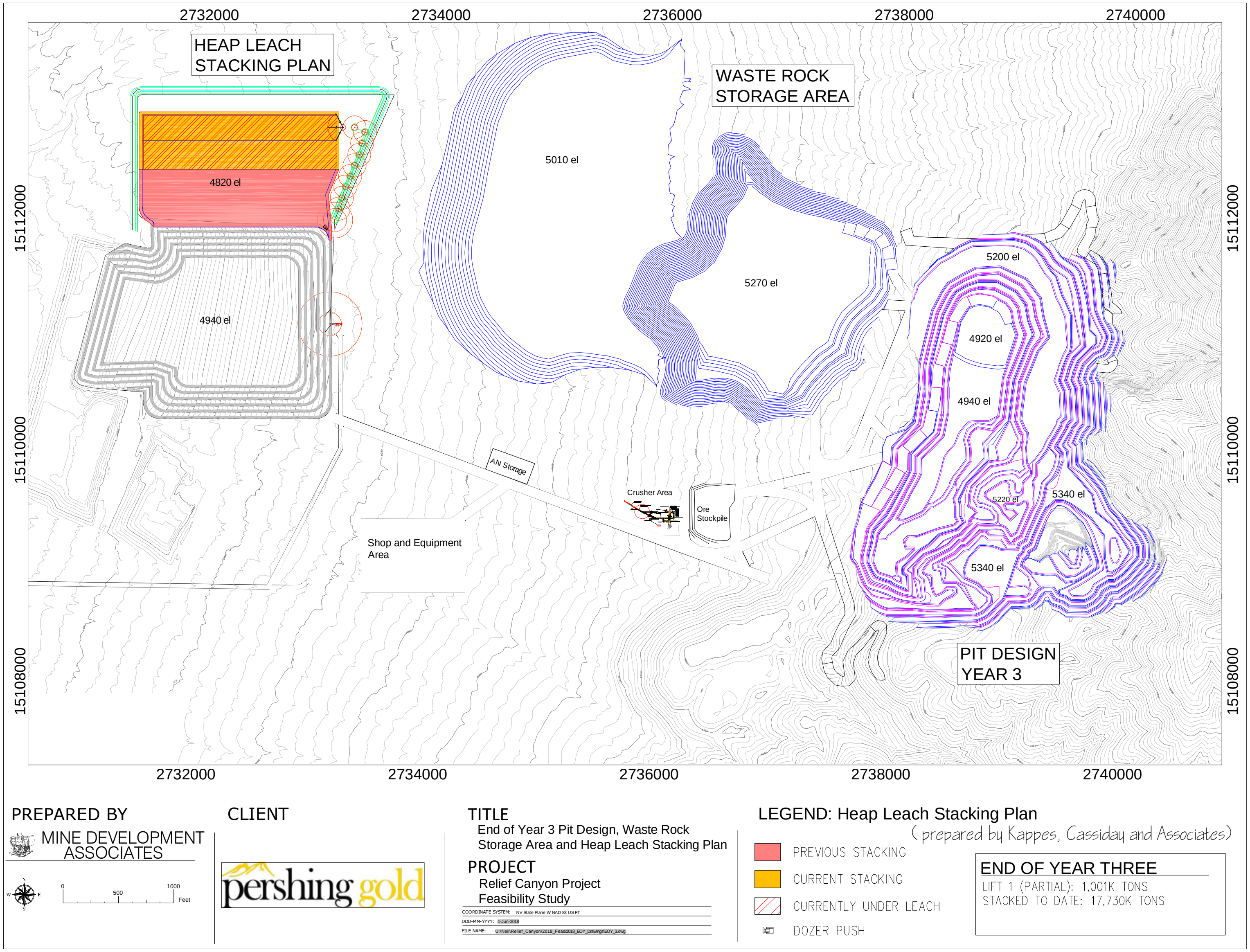


Figure 16.10 Year 4 Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan

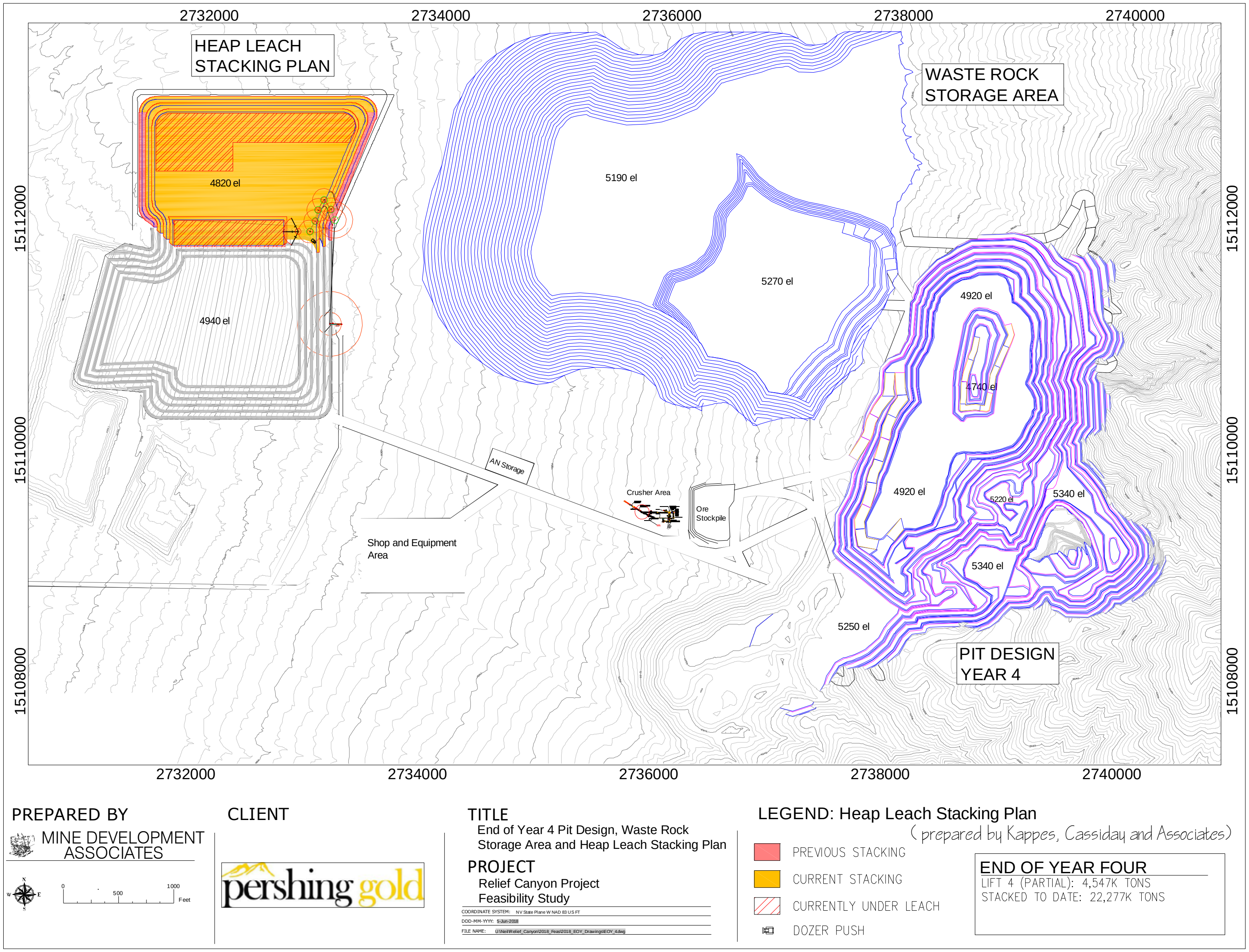


Figure 16.11 Year 5 Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan

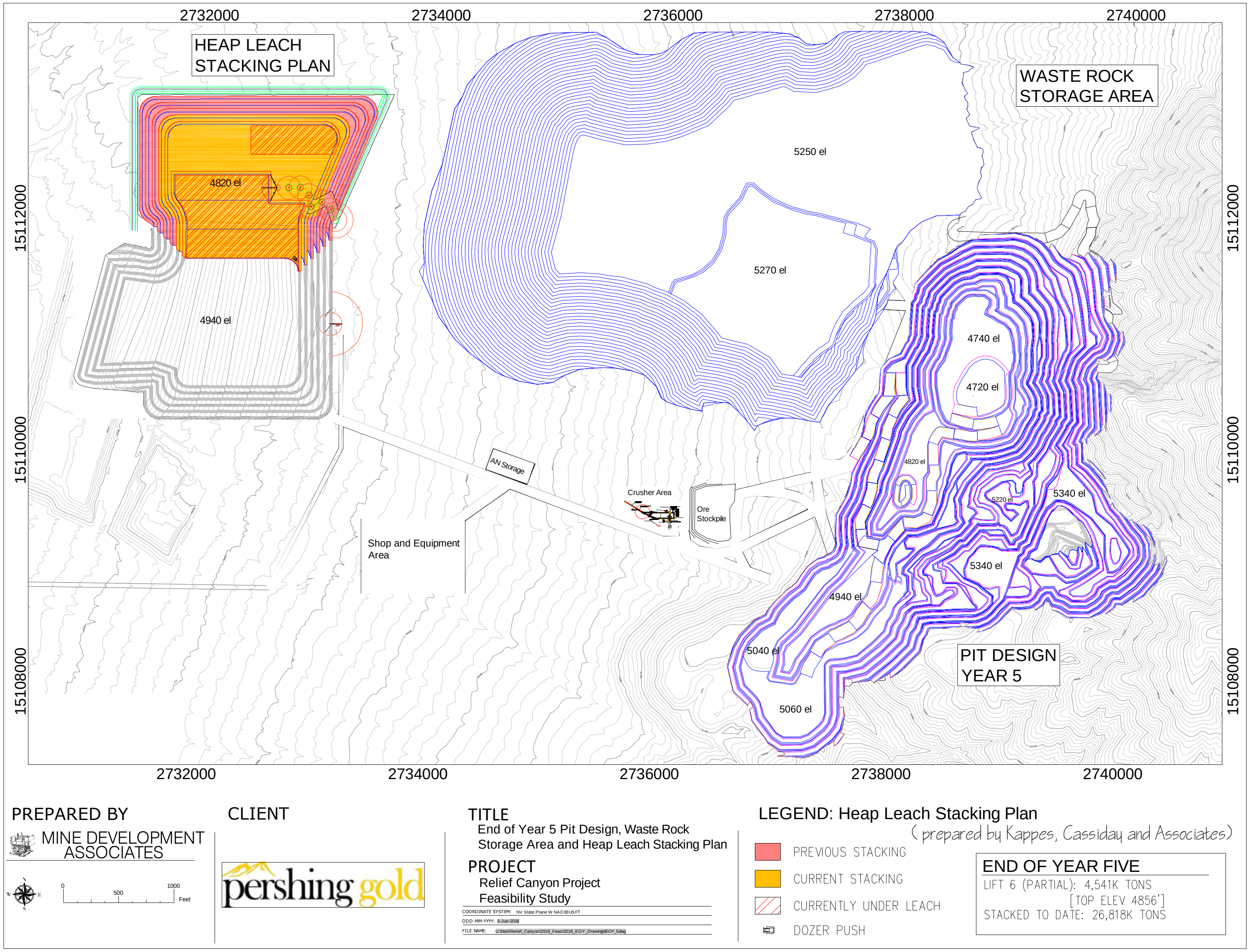
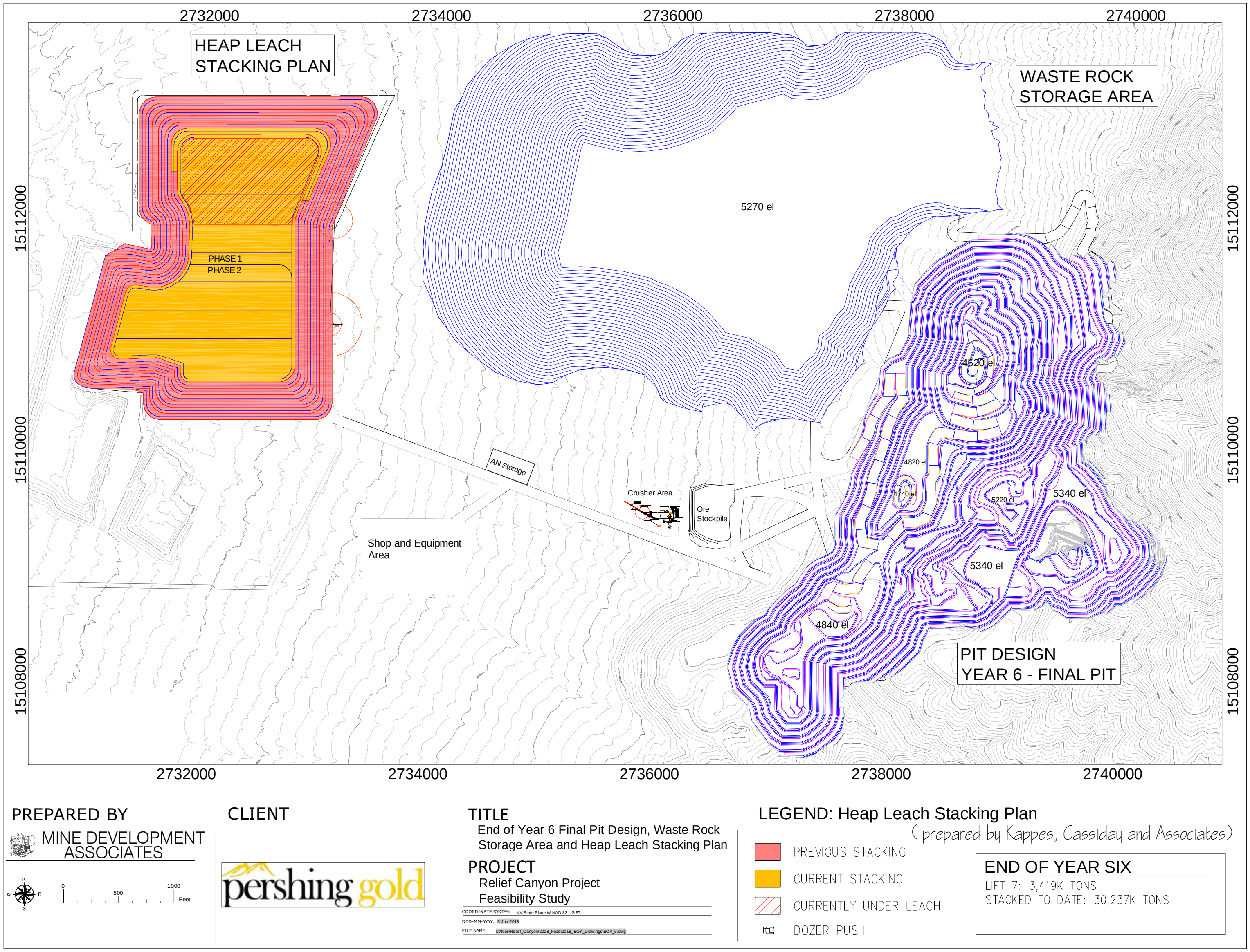


Figure 16.12 Year 6 Final Pit Design, Waste Rock Storage Area and Heap Leach Stacking Plan



The base case mine plan assumes contract mining based on proposals from mining contractors. The base case also assumes that the contractor will load the crusher using a front end loader, and the crushed material will be transported by conveyor to the leach pad.

Initial mine haulage roads will be developed to transport material from the upper pit benches to the waste dumps and the crusher utilizing existing roads as much as possible. However, most of the existing roads will be mined out by the Phase 1 design pit. These roads are the critical path to achieving the planned production, and they require the dozed material as fill for completion. Both roads should be constructed to a width of 85 feet and gradient not exceeding 10 percent, as all materials above the 5,220 elevation, or so, will be hauled out of the pits on these roads. Access roads to the south of the pit will be required for design Phase 1, and it will be necessary to soon develop access roads to the north of the pit. The design Phase 1 pit will require access roads on the south to elevations between 5,200 and 5,500, and on the north between 5,360 and 5,200.

Design Phase 2 will require out-of-pit south access to the upper benches starting at the 5580 bench, and north access starting at the 5600 bench. Material above the 5600 bench in design Phase 2 will be dozed to the 5600 bench. All of the dozed material is considered as waste and has not been included in Proven or Probable reserves. A considerable amount of organization and detailed pit and access road planning needs to occur prior to the start of mining.

Production from both design Phase 1 and 2 will be limited to benches at or above the 5080 bench until the permit to deepen the pit is received. This permit is expected to be applied for during June, 2018 and is expected to be granted by the third quarter of 2020.

16.4 Contractor Mine Equipment

A detailed quote received from Ledcor CMI Inc., a mining contractor, was the basis used for estimating contract mining costs in this Feasibility Study. In addition, MDA completed an independent estimate of owner mining, staffing and equipment. This was to compare the owner mining cost and contract mining cost, as well as to provide estimates of fuel and explosives, to compare with the contractor estimate of fuel and explosives use. The owner mining case considers mining with Cat 993 loaders and 150 ton trucks, while the contractor case assumes a fleet of Caterpillar 992 front end loaders, and 100 ton trucks. Table 16.8 shows the mine equipment that the contractor is planning to use at Relief Canyon.

Table 16.8 Planned Mining Contractor Equipment

Mine Equipment	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Atlas Copco DM45	2	3	3	3	4	4	3
Cat 992 Front End Loader	3	5	5	5	6	6	5
100 Ton Trucks	6	16	16	16	23	23	19
Cat 16M Grader	2	2	2	2	2	2	2
Cat D10T Dozer	2	2	2	2	2	2	2
Cat D9T Dozer	2	2	2	2	2	2	2
Cat 777 Water Truck	1	1	1	1	1	1	1
Cat 773 Water Truck	1	1	1	1	1	1	1
Cat 330 Excavator	1	1	1	1	1	1	1
Maintenance Equipment*	1	1	1	1	1	1	1
*I lot consiting of Lube Truck, Mechanics Truck, IT-28, and Tire Truck							

16.5 Planned Contractor Mine Personnel

Table 16.9 shows the contractor personnel estimated to be required for the Relief Canyon operation for years 1-3 and 6, while Table 16.11 shows the contractor personnel required for years 4 and 5.

Table 16.9 Mine Contractor Personnel (Years 1-3 and 6)

Description	**5x2**	**4X3**	**5x4**	**5x4**	**5X4**	**5X4**	**Total**
	d/s	**d/s only**	**A Crew**	**B Crew**	**C Crew**	**D Crew**	
Project Manager	1	-	-	-	-	-	1
Operations Superintendent	2	-	-	-	-	-	2
Maintenance Superintendent	1	-	-	-	-	-	1
Project Engineer	1	-	-	-	-	-	1
Administration Clerk-Maint.	1	-	-	-	-	-	1
Trainers	1	-	-	-	-	-	1
Safety Advisors/EMT	1	-	-	-	-	-	1
Ops. Foremen	-	-	1	1	1	1	4
Total Salary	**8**	**-**	**1**	**1**	**1**	**1**	**12**
Ops. Truck Drivers	-	-	11	11	11	11	44
Ops. Dozer	-	-	2	2	2	2	8
Ops. 992K	-	-	3	3	3	3	12
Ops. Grader	-	-	1	1	1	1	4
Ops. Water Truck	-	-	1	1	1	1	4
Ops. Laborer	-	-	1	1	1	1	4
Ops. Drillers	-	-	2	2	2	2	8
Ops. Blasting Crew	-	5	-	-	-	-	5
Total Ops Hourly	**-**	**5**	**21**	**21**	**21**	**21**	**89**
Maint. Mechanics	-	1	3	3	3	3	13
Maint. Welders	-	1	-	-	-	-	1
Maint. Serviceman	-	-	2	2	2	2	8
Maint. Tire Tech (Contract)	-	1	-	-	-	-	1
Maint. Contract Mechanics	-	2	-	-	-	-	2
Total Maintenance Hourly	**-**	**5**	**5**	**5**	**5**	**5**	**25**
Totals Salary and Hourly	**8**	**10**	**27**	**27**	**27**	**27**	**126**

Table 16.10 Mine Contractor Personnel (Years 4 and 5)

Description	5x2	4X3	5x4	5x4	5X4	5X4	Total
	d/s	d/s only	A Crew	B Crew	C Crew	D Crew	
Project Manager	1	-	-	-	-	-	1
Operations Superintendent	2	-	-	-	-	-	2
Maintenance Superintendent	1	-	-	-	-	-	1
Project Engineer	1	-	-	-	-	-	1
Administration Clerk-Maint.	1	-	-	-	-	-	1
Trainers	1	-	-	-	-	-	1
Safety Advisors/EMT	1	-	-	-	-	-	1
Ops. Foremen	-	-	1	1	1	1	4
Total Salary	**8**	**-**	**1**	**1**	**1**	**1**	**12**
Ops. Truck Drivers	-	-	17	17	17	17	68
Ops. Dozer	-	-	3	3	3	3	12
Ops. 992K	-	-	4	4	4	4	16
Ops. Grader	-	-	1	1	1	1	4
Ops. Water Truck	-	-	1	1	1	1	4
Ops. Laborer	-	-	1	1	1	1	4
Ops. Drillers	-	-	2	2	2	2	8
Ops. Blasting Crew	-	5	-	-	-	-	5
Total Ops Hourly	**-**	**5**	**29**	**29**	**29**	**29**	**121**
Maint. Mechanics	-	1	3	3	3	3	13
Maint. Welders	-	1	-	-	-	-	1
Maint. Serviceman	-	-	2	2	2	2	8
Maint. Tire Tech (Contract)	-	1	-	-	-	-	1
Maint. Contract Mechanics	-	2	-	-	-	-	2
Total Maintenance Hourly	**-**	**5**	**5**	**5**	**5**	**5**	**25**
Totals Salary and Hourly	**8**	**10**	**35**	**35**	**35**	**35**	**158**

16.6 Owner's Mine Personnel

Table 16.11 shows the owner's mine personnel estimated to be required for the Relief Canyon operation.

Table 16.11 Owner Mine Personnel

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Mine Manager	1	1	1	1	1	1	1
Mine Clerk		1	1	1	1	1	1
Light Vehicle Mechanic	1	1	1	1	1	1	1
Senior Mine Engineer	1	1	1	1	1	1	1
Surveyor	1	1	1	1	2	2	2
Surveyor Assistant	1	2	2	2	2	2	2
Ore Control Geologist	1	1	1	1	1	1	1
Sampler	1	2	2	2	2	2	2
Dewatering Crew				2	2	2	2
Totals	7	10	10	12	13	13	13

17.0 RECOVERY METHODS

Kappes, Cassiday and Associates prepared this section. Metallurgical test work indicates that ore from the Main Zone, Jasperoid Zone and Lower Zone are amenable to cyanide leaching for the recovery of gold and silver. Ore will be mined by standard open-pit mining methods and processed at an average rate of 16,700 tons per day. Mined ore will be single-stage crushed to approximately 80% passing 3 inches in size, belt agglomerated using cement, conveyor stacked onto the heap-leach pad in 20-foot lifts and processed in a conventional heap-leach recovery circuit. Stacked ore will be leached with a dilute cyanide solution, and the resulting pregnant solution will be processed in an adsorption, desorption and recovery plant ("ADR") for the recovery of precious metals from solution. The gold will be stripped from the loaded carbon using a pressurized desoption process, followed by electrowinning to produce a precipitate sludge. Gold and silver sludge from the ADR plant will be treated in a mercury retort and smelted to produce doré bars.

The project has an estimated reserve of 30.2 million tons, and an estimated mine life of 5.6 years. The crushing and stacking circuit is selected to process up to 6.0 million tons of ore annually, or 16,700 tons per day. A summary of the process design criteria is presented in Table 17.1.

Table 17.1 Process Design Criteria Summary

Item	Rate
Annual Tons Processing Capacity	6,010,000
Daily Production Rate	16,700
Hours per Shift	12
Shift per Day	2
Days per Week	7
Days per Year	360
Crushing Availablity	75%
Crushed Product Size	80% passing 3"
Heap Leach Cycle	60 days

17.1 Reagent Consumptions

The reagent consumptions for heap leaching of the Main Zone, Jasperoid Zone and Lower Zone mineralized materials have been taken from Section 13 of this report. The estimated cyanide consumption for heap leaching of the crushed, agglomerated ore is 0.5 lb NaCN/ton ore. The estimated nominal cement addition rate for agglomeration and pH control of the crushed ore is 8.0 lb/ton.

For other process reagents used primarily at the recovery plant, such as hydrochloric acid, caustic, make-up activated carbon, antiscalant, and fluxes, consumptions are based on the anticipated production of gold and silver.

A summary of the estimated reagent consumptions at the Relief Canyon mine is presented in Table 17.2 below. These estimates may vary depending on the metallurgical conditions encountered during operations. Pershing Gold may elect to substitute reagents with similar chemical compositions for those listed if higher efficiencies can be realized.

Table 17.2 Reagent Consumptions

Item	Form	Annual Consumption
Sodium Cyanide	30% Solution, Liquid Tankers	1,509 tons (dry basis)
Cement	Bulk Pneumatic Trucks	24,048 tons
Sodium Hydroxide	Dry Solid Sacks / Bags	112 tons (dry basis)
Hydrochloric Acid	Liquid Tote 1 m^3 Bins	212 tons
Activated Carbon	500 kg Supersacks	32 tons
Antiscalant	Liquid Tote 1 m^3 Bins	25 tons
Fluxes	Dry Solid Sacks	14 tons

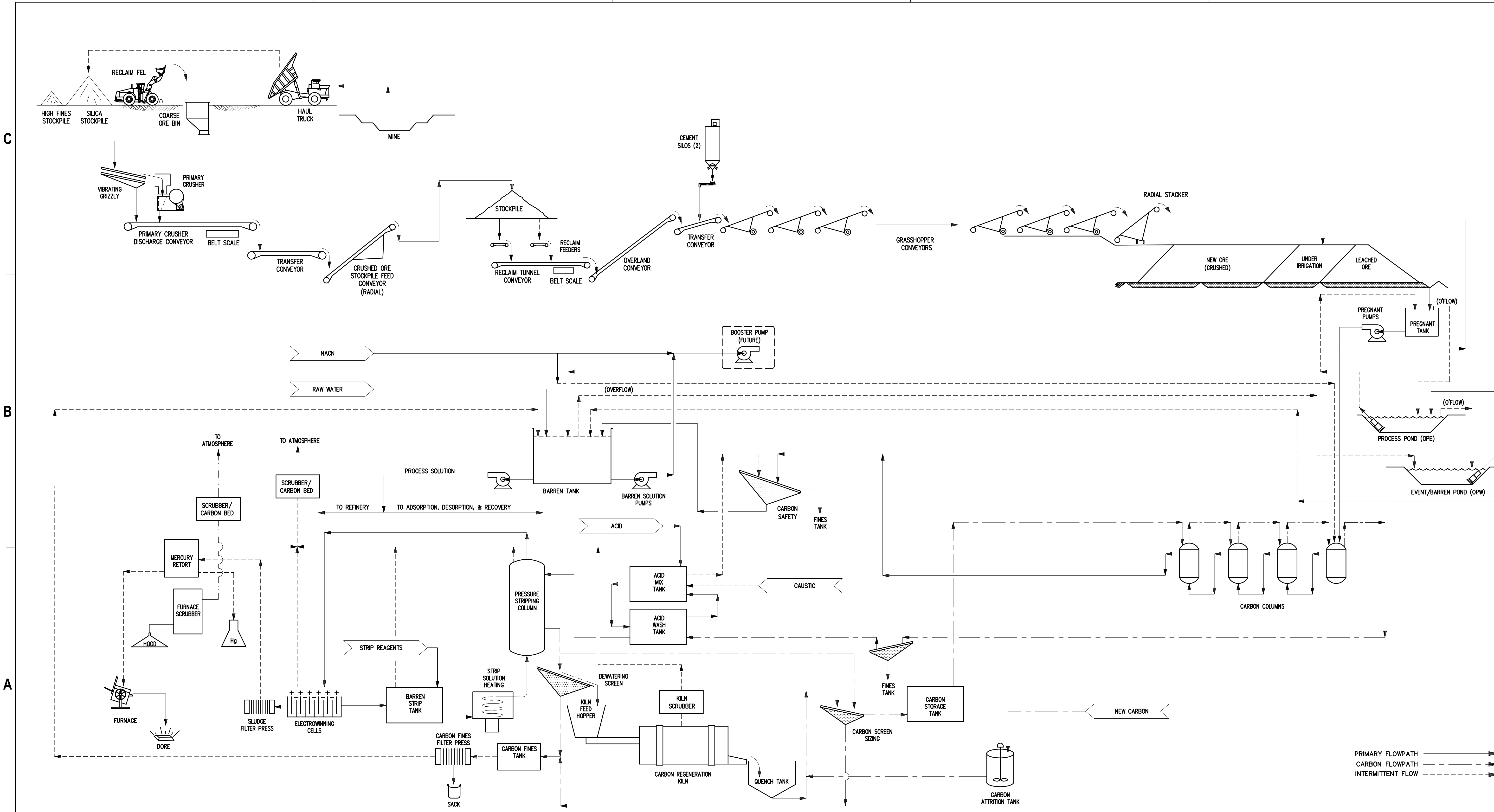
17.2 Process Description

Ore for the Relief Canyon project will be delivered to the ROM stockpile using 100 ton haul trucks. ROM ore will be reclaimed and crushed to a P80 of 3.0 inches in a single-stage crushing circuit at an average rate of 16,700 tons per day. Crushed ore will be stockpiled and reclaimed using vibrating pan feeders before being transferred to the conveyor stacking system by an overland conveyor. Cement is to be added ahead of the conveyor stacking system at an average rate of 8 lbs per ton of ore for pH control and permeability, and belt agglomerated by the conveyor stacking system. The process solution for agglomeration will be barren solution and directly dosed on the conveyor transfer points after the cement addition.

The conveyor stacking system includes mobile grasshopper conveyors which feed a radial stacking system. The leach pad will be stacked in 20-foot lifts. Drip tubes will be used to irrigate the ore with a low concentration sodium cyanide solution to leach gold and silver values with a 60-day leach cycle. Pregnant solution from the heap will flow by gravity to a pregnant solution tank where it will be pumped to a carbon in column (“CIC”) adsorption circuit. Gold and silver will be loaded onto activated carbon and will periodically be stripped from the carbon using a modified Zadra pressure-strip circuit, electrowon and smelted to produce the final doré product. A mercury retort will be utilized to remove mercury prior to smelting.

The existing solution ponds, Operating Pond East (OPE) and Operating Pond West (OPW), will be used to contain process solution in the event of a large storm event or other upset conditions that cannot be managed during normal operations with Pad 5, 6, 7 (pre-production leach pad). An additional contingency pond will be constructed with Pad 8 (Years 3+ leach pad) to account for the added heap-leach pad expansion. Solution collected in the process solution ponds will be returned to the system as makeup solution.

Electric power will be supplied by a combination of line power and on-site diesel generators. A process flowsheet for the Relief Canyon project is presented in Figure 17.1.



REFERENCE DRAWINGS	
DWG. NO.	TITLE

REVISIONS

NO.	BY	CK'D	APPROVED	DATE	DESCRIPTION
A	EDZ	GML		09JAN18	ISSUED FOR REVIEW
B	WCC	MPS		12JUN18	ISSUED FOR REVIEW

DESIGNED BY MPS
DRAWN BY WCC
CHECKED BY
APPROVED 1
APPROVED 2
CLIENT APP'L

PREPARED FOR
pershing gold

PREPARED BY
KCA Kappes, Cassiday & Associates
Reno, Nevada

Project: RELIEF CANYON
Title: FEASIBILITY STUDY
SIMPLIFIED OVERALL FLOWSHEET

Job No.	Scale	ACAD Filename:
643 I	N.T.S.	643-FS500.B.dwg

643 | FS500 | REV B

NAME: J:\DRAWINGS\Active Projects\Relief Canyon Feasibility Study\DWG_Working\11-Flowsheets\643-FS500.B.dwg LAST UPDATE: 27-Apr-18, 2:11 PM PLOT DATE:12-Jun-18 BY: ANDY PLOT SCALE: 1:2.58

17.2.1 Crushing and Agglomeration

The following major components are included in the crushing, reclaim and agglomeration facility:

- 70-ton rock box (new);
- 62” x 24’ Vibrating grizzly feeder (new);
- 33” x 65” Jaw crusher (new);
- 36” x 150’ Fixed crushed product stacker (existing);
- Two ea. 48” x 72” Vibrating pan feeders (new);
- 42” x 3,120’ Overland conveyor (new);
- Two ea. 100-ton cement storage and feeding system (one existing, one new); and,
- Associated transfer conveyors (new).

Ore will be transported from the mine in 100-ton haul trucks and dumped at the crusher area stockpiles. Ore will be fed to the crusher from one of two ROM stockpiles: a low fines material stockpile, or a high fines material stockpile. All feed to the crusher will be reclaimed by a Cat 992 loader, or similar, from the stockpiles in a manner to produce a desired blend according to the fines content. Based on a preliminary blast simulation evaluation, the average ROM feed to the crusher will be approximately 80% passing 6 inches.

A new primary crusher will be purchased for crushing the ore. Ore will be fed to a 70-ton rock box over a grizzly feeder, which separates ore at approximately 3.5 inches. Grizzly oversize material will be fed to a 33” x 65” jaw crusher, with a closed-side setting of 3.5 inches, and grizzly undersize will bypass the jaw and combine with the jaw product on the primary discharge conveyor. The combined product will be approximately 80% passing 3 inches. No permanent rock breaker or fixed grizzly is included; Pershing anticipates a minimum of oversize lumps in the rock box by visual screening by the loader operators and will address any oversize material with a mobile rock breaker included in the contract mining fleet.

Dust suppression at the primary crusher is to be accomplished using foggers at the rock box and water sprays at conveyor transfer points. The available working pressure in the dust suppression raw water line makes it so no water pump or air compressor are required.

Ore discharged from the primary crusher will be transferred to a transfer conveyor, and then stacked on an approximated 11,500-ton stockpile using an existing 150-foot stacker. The stockpile has been sized to accommodate 16.5 hours of production with an estimated live capacity of 3.4 hours.

Stockpiled crushed ore will be reclaimed using a conveyor reclaim system, which will consist of a 10-foot diameter x 100-foot reclaim tunnel constructed underneath the stockpile, with two reclaim feeders and a reclaim tunnel conveyor.

From the tunnel conveyor, ore will be transported to the heap by an overland conveyor of approximately 3,120 feet in length with a 42” belt. Ore will then be fed to a 55-foot, 42” belt transfer conveyor where cement will be added from two 100-ton cement silos. Barren solution will be slowly added to bring up the ore moisture content to approximately 8% by weight over the course of various transfer points. A pipeline will be installed for transport of barren solution to the cement addition point. The ore will then be agglomerated over the subsequent grasshopper conveyors.

The crushing system will operate 24 hours a day at an assumed availability of 75%, which at the maximum planned throughput of 16,700 tons a day is approximately 928 dry tons per hour.

17.2.2 Heap Stacking

The heap stacking system includes the following major components:

- 7 ea. 42” x 100’ Grasshopper conveyors (six existing, one new);
- 12 ea. 42” x 110’ Grasshopper conveyors (new);
- 4 ea. 42” x 110’ Grasshopper ramp conveyors (new);
- 42” x 136’ Radial stacker (new); and,
- Index tugger unit (new).

The heap stacking system will consist of a series of 42” belt conveyors, starting with 100-foot and 110-foot portable and ramp grasshopper conveyors, followed by a 76-foot radial stacker with 60-foot retractable stinger (136 feet fully extended). A “tugger” unit is included to move the stacker.

Conveyor stacking will start at the downward slope of the heap and proceed in a retreating fashion up slope, parallel to the graded surface.

Ore will be stacked on the heap-leach pad using conveyors exclusively, seven lifts in total. Ore will be stacked in 20-foot high lifts to a maximum total height of 140 feet over the pad liner, with typical 20-foot benches between lifts to give an overall average slope of approximately 2.88 to 1. Dozer ripping to facilitate improved percolation will be performed after each lift has been stacked to the determined height.

17.2.3 Leaching and Solution Handling

After each irrigation cell has been stacked, the irrigation system will be installed. Prior to stacking the next lift, the panel will be dozer ripped. Dripline emitters will be used to apply a sodium cyanide solution with a concentration of 0.3 to 0.5 pounds of sodium cyanide per ton of solution, at an application rate of 0.003 gpm per square foot. A leach cycle of 60 days is planned for a nominal flow rate of 3,060 gpm.

Barren leach solution pH will be at least 10 and controlled by the cement that has been added for agglomeration of crushed ore.

Barren solution will be delivered from the existing 24,000-gallon barren tank located at the ADR plant, by a pair of existing vertical multi-stage pumps at the nominal flow rate. This solution will be carried by a 14” steel pipeline to the base of the heap and then to a network of subheaders and risers to the top of the heap where it is delivered by the emitters. The barren solution line also includes a 100M fine carbon filter. High strength sodium cyanide and antiscalant agent will be added to the barren solution by metering pumps. A barren solution booster pump will be required prior to the Pad 8 expansion in Year 3 to meet the head requirements of the heap.

Solution passing through the heap will dissolve gold and silver contained therein and then collect in a network of perforated solution collection pipes, which feed to a common discharge point at the base of the heap. The solution will then be carried by gravity in a single solid HDPE 16-inch collection pipe down to the existing 22,500-gallon pregnant solution tank. Excess solution from the heap will overflow from the pregnant tank to the existing process solution pond, also called the Operating Pond East ("OPE"). Solution is to be pumped from the pregnant tank to the adsorption carbon column circuit at the ADR plant.

17.2.4 Recovery Plant

The process considers utilizing and upgrading the existing recovery plant at site for the recovery of gold and silver by an ADR process. Precious metals in the heap leach pregnant solution will be processed in the existing CIC adsorption circuit where gold and silver values are adsorbed onto activated carbon (adsorption). Loaded carbon from the CIC circuit will then be stripped in a high-temperature and pressurized elution process coupled to an electrowinning circuit, which includes new and refurbished existing equipment (desorption), followed by drying the electrowinning sludge in a mercury retort for removal of mercury values, and smelting to produce doré bullion (recovery). Prior to elution, each batch of carbon will be acid washed in the existing acid wash circuit to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon. An existing rotary kiln is in place and will be refurbished to reactivate the carbon and remove any organic contaminants.

17.2.4.1 Adsorption

Adsorption of gold and silver onto carbon will occur in the CIC circuit. Heap leach pregnant solution will be pumped to the CIC circuit from the pregnant solution tank; antiscalant agent will be added at the pregnant solution tank to prevent scaling of the piping systems and carbon.

The carbon adsorption circuit consists of four existing closed-top, pressurized carbon columns in series, each column approximately 12.6 feet in diameter and 10 feet tall, and each holding 8 tons of carbon. Pregnant solution flows through the columns and discharges from the last column as barren solution. Barren solution leaving the last column passes through a 4 x 10-foot carbon safety screen to trap any carbon leaving the system and is directed to the barren tank. Cyanide make up is added to the barren solution and returned to the heap for further leaching. Overflow from the barren tank is directed to a process pond, also called the Operating Pond West ("OPW").

Adsorption of gold and silver from pregnant solution is a continuous process with loaded carbon from the CIC circuit periodically being advanced for further processing. Loaded carbon in the first column will be passed over an existing carbon dewatering screen, where process solution and any carbon fines in the undersize will be transferred to a carbon fines tank, and the oversize loaded carbon directed to the acid wash circuit. Carbon in the remaining columns is advanced to the next column in series with new or freshly stripped and regenerated carbon being added to the fourth column. Generally, stripping of carbon will occur approximately two to three times each week.

17.2.4.2 Carbon Acid Wash

Acid washing consists of circulating a dilute hydrochloric acid solution through the bed of carbon to dissolve and remove scale from the carbon and is performed on a batch basis. The existing acid wash circuit includes an acid wash tank, acid mix tank, acid circulation pump, and acid metering pump, and can process 8.0 tons of carbon per batch. Acid washing will be performed before every strip, which is expected for optimum performance of the carbon.

After the loaded carbon is dewatered and carbon fines removed by the loaded carbon dewatering screen, it will be transferred to the acid wash tank where it is rinsed to remove any entrained cyanide solution. Rinse solution will be discarded to the barren solution tank. A dilute acid solution is then prepared in the acid mix tank and circulation of acid solution is established between the acid mix tank and acid wash tank. Concentrated acid is injected into the recycle stream to achieve and maintain a pH ranging between 1.0 and 2.0. Completion of the cycle is indicated when the pH stabilizes between 1.0 and 2.0 without acid addition for a minimum of one hour of circulation.

After acid washing is complete, spent acid solution from the acid mix tank and wash vessel are sent to the dilute acid storage tank. The carbon is then rinsed with water and neutralized using a dilute caustic solution to remove any residual acid. Acid washed carbon is then transferred to the elution circuit using an acid wash carbon advance pump. Total time for acid washing is approximately six hours for one batch.

17.2.4.3 Desorption and Carbon Regeneration

Loaded, acid-washed carbon will be batch stripped in a single pressure stripping vessel, using a modified Zadra circuit. During the desorption process, gold and silver are continuously extracted by electrowinning from the pregnant eluate concurrently with desorption. The desorption process requires 18-24 hours to completely strip one batch of carbon.

The desorption circuit includes the following major components:

- 1,350 ft^3 barren strip solution tank, 8-ton capacity (existing);
- Two each eluate solution pumps (one operating, one standby) (existing);
- 5 MM BTU/hr hot water heater (new);
- Heat exchanger system including heat up, heat recovery, and cool down heat exchangers (existing); and
- 5.8 ft dia. x 22 ft tall elution column (existing).

The loaded carbon will be eluted using a sodium hydroxide solution at a rate of about two bed volumes per hour, at a temperature of 250-275°F. Eluate solution is pumped through the heat recovery and primary heat exchangers and introduced to the elution vessel. Under normal operating conditions, barren eluent solution from the barren strip solution tank will pass through the heat recovery exchanger to be preheated by hot pregnant eluate leaving the elution column.

The elution column contains screens at the inlet to hold carbon in the column and distribute the incoming strip solution evenly in the column. Pregnant eluate leaving the column passes through additional stainless-steel screens before passing through the heat recovery and cool down heat exchanger, which reduces the temperature to 185°F (to prevent boiling). The cooled pregnant solution is then sent to the electrowinning cells.

The eluted pregnant solution will report to the electrowinning cells.

After the desorption cycle is complete, stripped carbon will be passed over a sizing screen and is then transferred to the kiln feed hopper. Stripped carbon will be fed to the kiln by a screw feeder. Hot regenerated carbon leaving the kiln discharges to a water filled quench tank for cooling and storage. Regenerated carbon will be screened to remove fine carbon from the circuit, as required; carbon fines from the dewatering screens will be stored in a carbon storage tank and periodically sent to a filter press for collection.

The existing kiln has a capacity to regenerate 125 lb/hr of carbon, or approximately 50% of the total carbon stripped.

17.2.4.4 Electrowinning

The electrowinning circuit will operate in series with the elution circuit. Solution will be pumped continuously from the barren strip solution tank, through the elution vessel, then through the electrowinning cells, and back to the barren strip solution tank in a continuous closed loop.

The electrowinning circuit includes a new 100 ft^3 electrowinning cell with 2,000-amp rectifier, e-cell barren return tank with pump, and an existing sludge filtering system. Gold and silver are won from the eluent in the electrowinning cells using stainless steel cathodes. Caustic (sodium hydroxide) in the eluate solution acts as an electrolyte to encourage free flow of electrons and promote the precious metal winning from solution. Barren eluate leaving the electrolytic cells will be discharged to an existing 11,000-gallon e-cell barren return tank where it is to be pumped back to the barren solution strip tank.

Periodically, all or part of the barren eluent will be dumped to the barren solution tank and new solution will be added to the tank. Typically, about one-third of the barren eluent will be discarded after each strip cycle. Sodium hydroxide will be added as required to the barren eluent tank during fresh solution make-up.

The precious metal-laden cathodes in the electrolytic cells are to be removed periodically and processed to remove mercury and produce the final doré product. Loaded cathodes will be pressure washed to remove gold and silver sludge. The sludge will then be pumped to a plate-and-frame filter press to remove filter cake, which will be loaded into pans to be dried in the mercury retort system and smelted in the smelting furnace.

17.2.4.5 Refining and Smelting

The refining and smelting circuit includes the following major components:

- 5.0 ft^3 mercury retort (new);
- Propane-fired smelting furnace (new); and
- Smelting furnace hood and off-gas extraction blower (new).

Cathode sludge from the filter press will be loaded into pans and manually transported to a 5.0 ft^3 retort oven for drying and removal of mercury. After retorting, the dried sludge will be mixed with fluxes at a 1.5:1 ratio and fed to a tilting, propane-fired smelting furnace (300 lb red brass capacity). After melting, the slag will be poured off into cast iron molds until the remaining furnace charge is mostly molten metal (doré). The doré will be poured into bar molds, cooled, cleaned and stored in a vault pending shipment off site for final processing and sale. The doré poured from the furnace represents the final product of the processing circuit. A hood will collect fumes from the furnace, which will pass through a sulfonated carbon bed and bag house to remove any remaining mercury and particulates present in the exhaust fumes.

Periodically, slag produced from the smelting operation is to be re-smelted on a batch basis to recover residual metal values, or will be broken and reprocessed on the heap-leach pad.

17.2.4.6 Mercury Control

A mercury abatement system will be installed in the existing ADR plant. This system will consist of two mercury scrubbing systems. One system will consist of a hood over the smelting furnace, which will direct furnace exhaust air to a baghouse and then to a tray-style carbon scrubber filled with 1,500 pounds of sulfur-impregnated carbon for removal of mercury vapor. The second system consists of a wet scrubber fitted to the regeneration kiln exhaust, which combines with air exhaust from the mercury retort, electrowinning cells, and eluate solution tank, and is to be directed to a demister, air heater, and finally to a 3,000-pound deep bed scrubber filled with sulfur impregnated carbon, for removal of mercury vapor.

At the design process rate, potentially up to 4,000 pounds of mercury are expected to be recovered as metallic mercury per year. The captured mercury will be stored on-site and properly disposed in accordance to local and federal regulations.

17.2.5 Reagents and Utilities

Most of the reagent handling systems required for the process, including the caustic mix and storage system, and dosing systems, are existing at site and require little or no modifications or refurbishment. A new cyanide storage tank and metering pumps are required, as well as a second storage silo for cement.

The major reagent handling systems include:

- Two 100-ton cement storage silos each equipped with variable speed screw feeders, bin vents, bin activators and dust collectors (one new, one existing);
- 12,000 gallon sodium cyanide storage skid-mounted tank with flowmeter and metering pump (new);
- Caustic mix and storage system with 4.5 ft dia. X 4.5 ft tall caustic mix and storage tank, caustic mix tank agitator, and caustic transfer pump (existing); and
- 1,000 gallon and 550-gallon antiscalant storage tanks with metering pumps (existing).

Approximate annual consumptions are provided in Table 17.2.

17.2.5.1 Sodium Cyanide

Sodium cyanide will be delivered as a 30% solution from tanker trucks to a new 12,000-gallon, skid-mounted storage tank. Strong cyanide solution will be added to the barren pump discharge line near the preg tank using a metering pump, to maintain a target cyanide concentration in the leach circuit. Cyanide is expected to be consumed at a rate of 0.5 lb per ton of ore processed, or 8,350 lb per day.

The cyanide storage tank is sized for approximately four days of operation.

17.2.5.2 Cement

Cement is to be added at an average rate of 8 lb per ton of ore processed for pH control and heap permeability. Cement will be delivered to site by bulk trucks and transferred pneumatically to each of the two 100-ton storage silos (one new and one existing).

The cement silos are sized for approximately three days of operation.

17.2.5.3 Sodium Hydroxide (Caustic)

Sodium hydroxide (caustic) will be delivered in beads or prills in 50-lb sacks and dissolved in an existing mix tank for distribution to the acid wash circuit for neutralization, and to the elution circuit.

For elution, concentrated caustic will be pumped from the caustic mix and storage tank to the barren strip solution tank where it is to be mixed with raw water or process solution to produce a 1.5% (by weight) sodium hydroxide eluent solution.

For carbon acid wash neutralization, caustic solution will be pumped to the acid wash mix tank where it will be mixed with water and circulated through the acid wash column.

17.2.5.4 Hydrochloric Acid

Hydrochloric acid will be used for the acid wash section of the elution circuit prior to desorption. Hydrochloric acid will be delivered as a 32-35% solution in tote bins. Approximately 41 gallons of HCl are required per ton of carbon washed.

17.2.5.5 Carbon

Activated carbon will be used to absorb precious metals in the CIC tanks. Make-up carbon will be 6 x 12 mesh and delivered in supersacks. Carbon losses are estimated at 3% of the weight of carbon stripped in the elution circuit.

17.2.5.6 Antiscalant Agent

Antiscalant agents will be used to prevent the build-up of scale in the process solution and heap irrigation lines. Antiscalant is normally added at the process pump intakes or pipelines and consumptions will vary depending on the concentration of scale-forming species in the process stream.

Antiscalant is expected to be delivered in totes and will be metered to the barren tank, pregnant solution tank, and barren strip solution tank.

17.2.5.7 Propane

Liquid propane will be delivered in tankers to a new 30,000-gallon, skid-mounted tank, and will be used for the strip solution heater, regeneration kiln, hot water boiler, mercury retort, smelting furnace, and for building heating. The existing, on-site propane vaporizer will be utilized in conjunction with new rotary gas meters for each of the delivery points.

17.3 Heap-Leach System

The heap-leach system includes the heap-leach pad and associated solution storage tanks and ponds. The leach pad is to be located north east of the ADR facility and west of the waste rock dump and mine pit. The heap-leach pad will be constructed in two phases, beginning with Pad 5/6/7 and expanding with Pad 8, with a maximum heap height of 140 ft and a total capacity of 30.2 million tons.

The Pad 5/6/7 construction will consist of unloading an existing lined pad area (called “Cell 6B”) which currently contains approximately 185,000 cubic yards of previously leached ore, along with construction of a new pad lined area (Cells 5, 6A, 7A, and 7B), which will be tied into the existing Cell 6B pad.

Pad 8 will be built to the north of Pads 5/6/7 in Year 3 and will consist of the final heap construction to contain the ultimate heap tonnage.

17.3.1 Pad 5/6/7 (Pre-Production Construction)

The heap-leach Pad 5/6/7 design was developed by Welsh Hagen of Reno, NV. KCA has not reviewed the design in detail but the design appears to KCA to be consistent with standard practice. The concepts are summarized below.

The pad lining system will consist of a geosynthetic clay liner (“GCL”) overlain by a single-sided texture 80-mil HDPE primary liner. An 80-mil HDPE secondary liner will be placed and welded across divider berms that will connect the new pad with the existing pad (between the 6W-6E and 6W-7 cell tie-ins).

Solution collection pipes will consist of corrugated and perforated HDPE pipe at regularly spaced laterals, which drain to main-cell collection pipes of 4” to 24” in diameter. These collection pipes are placed over the liner system.

Overliner (consisting of crushed and screened material) will be placed over the lining and collection pipe system to an average depth of two feet to act as a blanket drain for solution percolating through the heap and to protect the lining system. Ore will then be stacked on the overliner.

A leak detection system will be installed, which will consist of a piping network between the primary and secondary liners underneath the main solution collection pipes, for detecting any leaks underneath the geomembranes below the primary solution collection system.

The lined area for Pad 5/6/7 will be approximately 3.3 million ft^2 (including the existing ~0.8 million ft^2 Cell 6W lined area). It is estimated that approximately 15.6 million tons can be stacked over Pad 5/6/7 before additional pad area must be made available for leaching.

17.3.2 Pad 8

In Year 3, the Pad 8 will need to be constructed, adding approximately 2.2 million square feet of lined area to the pad, for a total of 5.5 million square feet. The Pad 8 will be an extension of the Pad 5/6/7 and will be built to the north. Pad 8 will provide sufficient area to stack to the ultimate pad tonnage.

During the Pad 8 construction, other supporting equipment and infrastructure such as an additional 1,000-foot overland conveyor, additional barren and pregnant solution piping, additional contingency pond and perimeter fencing will be constructed.

17.3.3 Solution Storage and Management

Solution management for the Relief Canyon project is based on precipitation data and estimated process and facility water demands. The existing solution storage management system includes the 6.6 million-gallon OPE pregnant / process solution pond, the 6.6 million-gallon OPW barren / event pond, a 24,000-gallon, barren solution tank, and a pregnant solution tank.

The Relief Canyon project is designed as a zero-discharge facility. The solution ponds and storage tanks have been evaluated and sized to ensure that leach solutions can be managed and controlled under all foreseeable operating conditions.

Pregnant solution from the leach pad will be collected in the pregnant solution tank and pumped to the adsorption circuit, which will discharge to the barren solution tank where reagents are added and the solution is pumped back to the heap. In the event of a surge of solution from the heap or other upset condition, the pregnant solution tank will overflow to the OPE pond and the barren solution tank will overflow to the OPW pond. In the event of a significant storm event, the OPE pond will overflow to the OPW pond. Make-up solution for the process is primarily added to the barren solution tank as needed.

Solution management for the system is generally simple. Both the OPE and OPW ponds should normally be maintained at empty or low levels whenever possible. Any solution collected in the OPE or OPW ponds should be pumped back to the leach system as soon as practical.

An additional contingency pond will be constructed as part of the Pad 8 expansion. The contingency pond will be similar in size to the OPE and OPW ponds and will be managed in the same way. Solution will overflow to the contingency pond after the OPE and OPW ponds are full. Any solutions collected in the contingency pond will be pumped back to the leach system as soon as practical. No long-term storage of solutions is considered for the ponds.

17.3.3.1 Ponds (OPE and OPW)

The OPE and OPW process solution ponds already exist at the Relief Canyon site and have a capacity of approximately 6.6 million gallons each. Welsh Hagen conducted a technical drainage study in September, 2015 and determined that these existing ponds combined have sufficient volume to contain the 100-year 24-hour storm flow, plus a 24-hour heap draindown for the Pad 5/6/7 area, given that the pond operating volumes are monitored and controlled to a sufficiently low level.

Based on data available, the existing ponds do not have sufficient capacity to handle a major rain event after the leach pad expansion in Pad 8. A contingency pond has been included in the Pad 8 heap expansion. The contingency pond will be constructed south of the OPE pond and will be sized to hold 6.6 million gallons of solution.

17.4 Laboratory Facilities

Analytical support, including fire assays and metallurgical testing, required to support the project operations, will be conducted off-site by contract laboratories. On-site facilities may be considered at a future date. It is anticipated that approximately 200 samples will be delivered from the mine for cyanide shake tests, with a portion requiring fire assay. A small number of fire assays, solutions, and carbon assays will also be required for metallurgical control for processing. Solution assays will be conducted on-site in the ADR plant using a Flame Atomic Absorption Spectrometer ("AA") instrument that will be purchased. Sodium cyanide titrations and pH measurements will also be conducted in this area.

17.5 Process Power Requirements

Power for the process and infrastructure have been estimated based on connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation are assigned and coupled with estimated on-stream times to determine the average energy usage.

The average power demands for the project are estimated at 12.6 MW during Year 1, 13.4 MW for Year 2, and 13.7 MW for the remaining project life. Electrical power will be provided using both generated and line power.

17.6 Water Balance

KCA had previously developed a preliminary water balance to estimate heap and crusher water consumption and took estimates from Pershing Gold for water consumption for road dust suppression. The water demands for the project include make-up water for the process facilities, fire water, crushing area dust suppression, road dust suppression, and potable water supply for the offices. It is estimated that the total average site demand will be approximately 630 gpm in the summer months and approximately 430 gpm in the winter months, for an overall average of 530 gpm.

17.7 Process Manpower Requirements

Process labor required for the process facilities is summarized in Table 17.3.

Table 17.3 Process Manpower Requirements

Job Title	# at Position	# of Crews	Total Qty.
PROCESS			
Supervision			
Process Manager	1	1	1
Process Foreman	1	1	1
Maintenance Supervisor	1	1	1
Clerk	1	1	1
Crushing			
Loader Operator (under mining)			
Crusher Operator	1	4	4
Utility/Multifunction Operator	1	1	1
Heap Leach			
Stacking*	1	4	6
Piping	3	2	6
Recovery Plant			
ADR	1	4	4
Process Maintenance			
Mechanic	2	2	4
Electrical	1	2	2
Subtotal Process			**31**

*Assumes two 12-hour shifts, two workers on day and on nights

18.0 PROJECT INFRASTRUCTURE

KCA prepared this section with the exception of Sections 18.3; 18.11 through 18.13, which were prepared by MDA. Figure 18.1 shows the planned locations of the crusher, leach pad, waste dump, plant, and other facilities for the project.

18.1 Existing Infrastructure

Significant infrastructure currently exists at the Relief Canyon site from previous operations. Existing installations include site access and haul roads, ADR facility, Operating Pond East and Operating Pond West process solution ponds, heap-leach pad, waste rock facilities, site buildings, electrical power supply, water wells and fencing around the process facilities. Pershing Gold intends to use as much of the existing infrastructure as possible, with only minor modifications where required, as discussed in this section.

18.2 Heap-Leach Facility

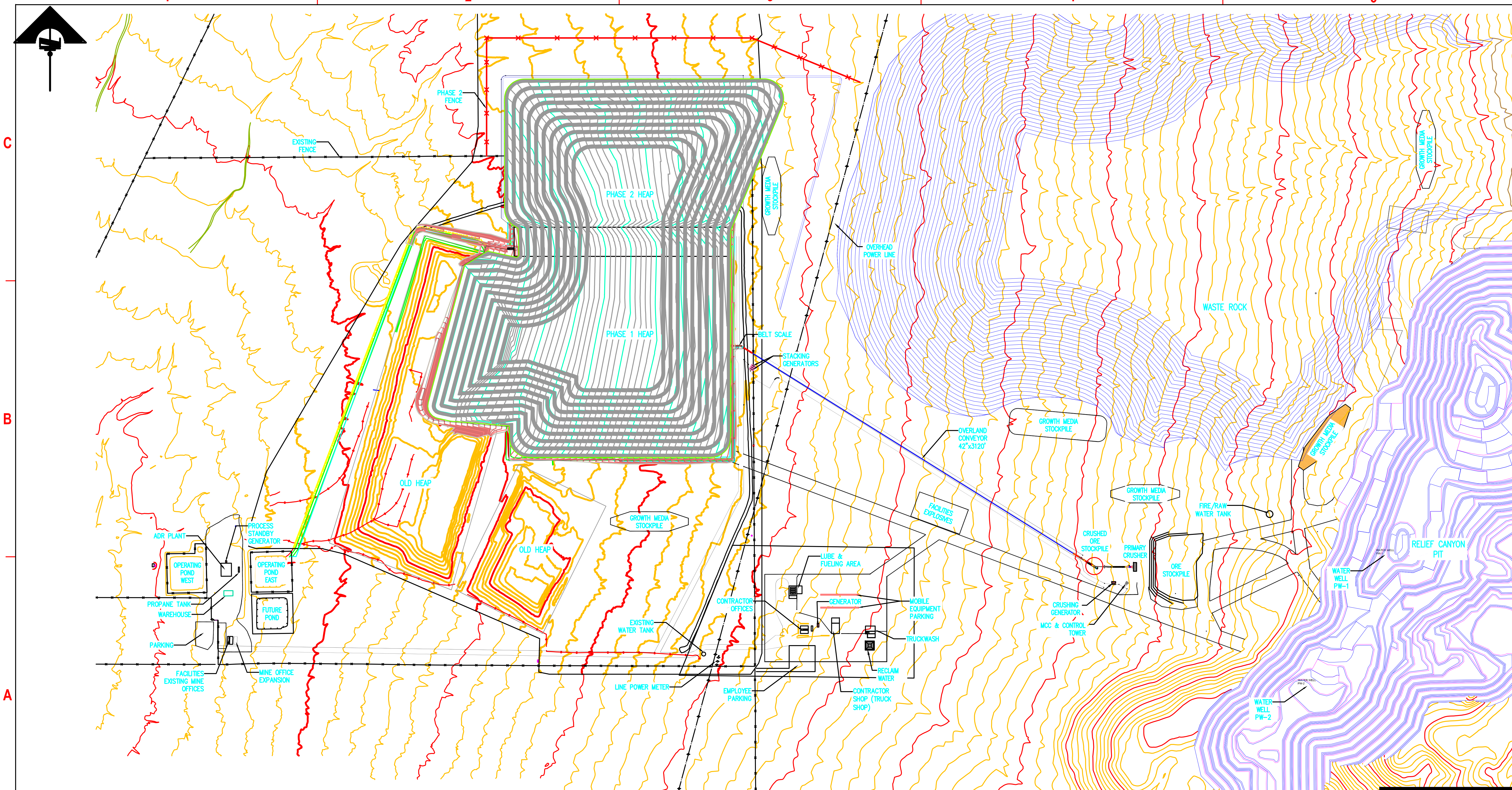
The existing heap-leach facility includes a heap-leach pad and the OPE and OPW process solution ponds which will be used for the planned production. The heap-leach pad (designated as Cell 6W) has previously been constructed with all necessary lining and solution collection gravel and piping, and is approximately 900 by 900 feet in size. The pad also contains approximately 185,000 cubic yards of previously stacked and leached ore, which will be moved off the pad before stacking new ore. During initial construction, the existing leach pad Cell 6W will be expanded to include Cells 5, 6E, 7W and 7E. The lining and collection systems for the new cells will be tied into the existing pad. Based on a water balance evaluation, the existing OPE and OPW ponds will be suitable for process solution containment for the heap-leach cells initially constructed.

18.3 Waste Dumps

Most of the mined waste rock from the first two years of mining will be stored in the Phase I permitted Waste Rock Storage Facility. An enlarged waste area will be part of the Phase II modification. This permit application is based on the waste areas shown in the end of year drawings in Section 16.2.

To the maximum extent possible, waste rock will be placed within mined-out portions of the pit. Some alluvium waste material from the Grass Valley Formation will be placed on former leach pads as cover material and some waste alluvium will be placed as growth media on other reclaimed facilities.

Prior to construction, vegetation will be cleared from the footprint of permitted Waste Rock Area and the available growth media will be salvaged and placed into growth media stockpiles near the dumps. Trucks will place the waste rock in 50-foot lifts.



PLAN VIEW

SCALE : 1"=30'

0 120 240 480
FEET

PRELIMINARY
NOT FOR CONSTRUCTION

1 2 3 4 5

REFERENCE DRAWINGS

DWG. NO.	TITLE

REVISIONS

NO.	BY	CK'D	APPROVED	DATE	DESCRIPTION

REVISIONS

NO.	BY	CK'D	APPROVED	DATE	DESCRIPTION
A	EDZ	GML		09JAN18	ISSUED FOR REVIEW
B	WCC	GML		01MAR18	ISSUED FOR REVIEW
C	PR	GML		29MAY18	ISSUED FOR REVIEW

DESIGNED BY	MPS
DRAWN BY	WCC
CHECKED BY	
APPROVED 1	
APPROVED 2	
CLIENT APP'L	

PREPARED FOR
pershing gold

PREPARED BY
Kappes, Cassiday & Associates
Reno, Nevada
KCA

Project: **RELIEF CANYON**

Title: FEASIBILITY STUDY
GENERAL ARRANGEMENT
OVERALL SITE LAYOUT
(ULTIMATE PIT & HEAP)

Job No.	Scale	ACAD Filename:
643 I	AS SHOWN	643-GA510.A.dwg

		REV
643	GA510	C

NAME: \\ymda.com\Users\Neil\Relief_Canyon\2018_Feas\2018_EOY_Drawings\643-GA510_C 29May18\643-GA510_MDA.dwg LAST UPDATE: 13-Jun-18, 8:58 AM PLOT DATE:13-Jun-18 BY: CWALKER PLOT SCALE: 1:2.1053

The final configurations for the Waste Rock Storage Areas have been designed to provide long-term stability and to promote surface run-off to minimize ponding of water and infiltration, and to limit erosion and channel scouring.

18.4 Roads

Access to the Relief Canyon site is by an existing, two-mile gravel road off of the paved Coal Canyon Road. Many of the access and haul roads required for mining and processing will require only the re-opening of pre-existing haul roads and access roads that were used by previous operators. Haul road running surfaces outside of the pit will be approximately 100 feet wide to accommodate haul trucks, while secondary roads will be approximately 30 feet in width. The actual road disturbance width may be wider, depending on topography. Roads will be bermed in accordance with MSHA regulations and best management practices will be used where necessary to control erosion. Pershing Gold will control fugitive dust emission from roads using water or chemical dust suppressant, such as magnesium chloride or lignin sulfonate, where appropriate.

18.5 Power Transmission, Generation and Distribution

Electrical power will be provided by a combination of line power and diesel-fired generator units. The site currently receives electrical service from an NV Energy medium-voltage (14.4 kV) primary service line that feeds from the Limerick substation to the north of the site and runs along the eastern side of the planned heap-leach pad. However, the existing power line does not have enough capacity to meet the power needs of the entire site. Approximately 41% of the electrical demand will be supplied by line power to most of the existing infrastructure, including the existing mine offices, warehouse, process solution management, and ADR plant. The remaining demand for the crushing plant, overland conveyor, and heap stacking conveyor system will be met by installing local diesel generators. Mine facilities such as the mine truck shop, wash bay, mine offices, and others are counted independently and on separate generated power.

18.5.1 Site Power Demand

The project on-site power demand was estimated based on a developed electrical load list for motorized equipment and other loads such as lighting and electric heating. Load factors and simultaneity factors were applied to estimate average power consumption for operating costs. For the crushing and stacking areas powered by generators, a peak demand was developed, which was used as the basis for generator sizing. A breakdown of estimated power consumption by area is presented in Table 18.1 below.

Table 18.1 Estimated Mine Power Consumption

Project Area	Power Source	YEAR 1		YEAR 2		YEAR 3-5	
		Peak kW	kWh/year	Peak kW	kWh/year	Peak kW	kWh/year
Area 00 - General Facilities	Line	39	259,200	39	259,200	39	259,200
Area 02 - Crushing & Reclaim	Generator	319	2,134,500	319	2,134,500	319	2,134,500
Area 06 - Stacking	Generator	696	4,947,000	696	4,947,000	740	5,237,200
Area 10 - Heap Leach & Solution Handling	Line	383	3,245,300	458	3,750,800	458	3,750,800
Area 10 - ADR Recovery Plant	Line	317	1,126,800	312	1,023,200	312	1,023,200
Area 20 - Water Facilities	Line	54	619,000	119	1,031,600	119	1,031,600
Area 20 - Power Supply & Distribution	Generator	35	220,800	35	220,800	35	220,800
TOTAL ESTIMATED DEMAND		**1843**	**12,552,700**	**1978**	**13,367,200**	**2022**	**13,657,300**
TOTAL DEMAND FOR GENSETS		1049	7,302,300	1049	7,302,300	1094	7,592,400
TOTAL DEMAND FOR LINE POWER		793	5,250,400	929	6,064,900	929	6,064,900

** Mine facilities (truck shop, wash bay, mine offices, etc.) of ~120kW are on separate generated power; Peak KW from Prefeasibility.*

18.5.2 Line Power

Line power will be supplied by NV Energy at a medium-voltage (14.4 kV) to a new metering point and further branched off to supply the process facilities to the west (existing mine offices, warehouse, process solution management, and ADR plant), and water wells to the east. The power line is further stepped-down in voltage using new transformers to be located near the source. For example, the ADR facilities building will utilize a 1,000 kVA transformer to step-down from 14.4 kV to 480 V. The mine offices and warehouse will utilize smaller transformers.

Line power modifications are required due to the increase in electrical power and includes the removal of the existing power meter, main step-down transformer and power lines. The new power equipment will include power poles, power lines, larger step-down transformer, and a primary meter. These power line upgrades are expected to be completed in the pre-production stage.

18.5.3 Power Generation (Process)

Diesel generators will be installed on-site in three separate areas to power the crushing plant, belt stacking system, and mine facilities. At the crushing plant, one 725 kW prime rated diesel generator, switchgear, and a 2,000-gallon diesel fuel tank are included. For the Pad 5/6/7 stacking system, two additional generators are included, one 725 kW prime rated generator and one 455 kW prime rated generator, both located near the overland conveyor head pulley. The Pad 8 stacking system will add one additional 455 kW generator to work in parallel with the existing two generators.

The peak demand is expected at approximately 334 kW for the crushing plant, and at 785 kW for the overland and stacking system with the conveyor line fully extended.

Fuel for the generators will be in the form of diesel and stored locally near the generators in tanks. The 725 kW diesel generators will include a 2,000-gallon tank, while the 455 kW generators include a 660-gallon tank.

The LOM average annual diesel fuel consumption for all generators is estimated to be 552,360 gallons per year.

18.5.4 Generator at Mine Facilities

A 150kW diesel generator will be installed at the mine facilities (truck shop, offices, fuel station) to power this infrastructure.

18.5.5 Emergency Generator

A 400 kW diesel generator currently exists on site and was previously used to power half of the MCC at the ADR plant. After the line power modification is complete in pre-production, the ADR plant will run 100% on line power and this generator will be used as an emergency backup for powering the barren solution pumps and other essential systems in the event of line power failure.

18.6 Water

As discussed in Section 17.6, the average water demand estimate includes approximately 360 gpm for crusher dust suppression and heap leach operations, 150 gpm for road dust control, and 10 gpm for miscellaneous uses.

18.6.1 Supply

Water for mining, the heap-leach facilities, fire suppression, and other uses will initially be supplied by existing production wells PW-1 and PW-2, located west of the pit area. These wells have the capacity to deliver approximately 700 gpm with a power upgrade, as proposed in a water management report by Schlumberger.

In Year 1 and 2 of operations, both PW-1 and PW-2 will be mined out, requiring new wells be drilled to replace them (these new wells also will be designed for dewatering of the pit, which will be required beginning in Year 3). PW-3 will replace PW-2 in Year 2, and PW-4 will replace PW-1 in Year 3. The maximum anticipated dewatering rate will be sufficient for meeting the site water needs, and higher than the consumptive use in later years of mining. Excess water from pit dewatering will be directed to Rapid Infiltration Basins (“RIB”s).

Water from the production wells will be pumped to a new raw/fire water storage head tank on the western side of the pit, just south of the ROM stockpile. This tank will be sized to contain the necessary fire water and process / raw water reserves, and will provide raw process water for the crushing and stacking plant, and mine facilities such as the truck shop, truck wash, and mine offices. A total storage capacity of 220,000 gallons was calculated using the fire water code requirements, investigated by KCA's subcontractor, along with the process water make-up requirements. The required fire water storage is on the order of 180,000 gallons, based on the fire hydrant flow and time requirements of 1,500 gpm for two hours.

The raw water storage capacity was determined to be around 120,000 gallons based on the process make-up solution requirements with a 3.0 hour working time, and this total required capacity is divided between the existing raw water tank and the new combination raw/fire water tank. An existing raw water tank with a capacity of 80,000 gallons is on-site, located near the power line metering point, and currently provides raw water needs for the ADR plant and other facilities.

18.6.2 Fire Water

Fire suppression for the site facilities will be provided as a gravity feed system from a head tank that will be constructed during pre-production and located on the north of the ROM stockpile. The fire system will have a minimum 2.0 hour dedicated fire reserve capacity with 12" HDPE mains and 6" HDPE submains to hydrants around the property. Due to the classifications of existing structures, a retrofit with sprinkler or hydrant systems is not required for existing facilities, which have been previously permitted. New facilities, mainly the truck shop, will be constructed in Year 1 of production and will require an addition to the fire water system at that time.

All buildings will have hand-held fire extinguishers available in accordance with MSHA regulations and industry standards.

18.7 Sewage

Septic systems and leach fields currently exist near the administrative building, process plant, and warehouse. Biosolids will be pumped as necessary by a licensed septic waste hauler and transported to a licensed repository.

A new, second septic system will be installed for servicing the truck shop and mine offices.

18.8 Existing Buildings

18.8.1 Warehouse

A warehouse is located near the process plant building and will continue to be used to store supplies and small equipment.

18.8.2 Administration Offices

The existing administration building includes a reception area, offices for administrative staff, and a meeting room. A safety/security/training area is also located in the administration building. A new parking area for personal vehicles will be located outside of the mine fence. The safety/security/training area includes first-aid supplies and a meeting/training room. An emergency response vehicle will be stationed at the administration building to respond to accidents and incidents.

18.8.3 ADR Plant

The existing ADR plant building has some small areas for storage and maintenance equipment, and also has a control room and a lunch/break room.

18.9 Mine Facilities

Several mine facilities will be constructed, operated and maintained by the mining contractor, which include a truck shop, truck wash, mine offices, and fuel storage tanks. Pershing Gold will provide septic, raw and fire water to these facilities.

18.9.1 Truck Shop

The truck shop will include maintenance bays to support mobile equipment maintenance. The mining contractor will provide the mine shop facilities. Lubricants and antifreeze will be managed and stored in the area as required by MSHA and other state and federal regulations.

18.9.2 Truck Wash

A truck wash facility will be located adjacent to the truck shop. Wash water will be directed to a settling basin where water and solids will be separated. Water will be treated with an oil-water separator and re-circulated. Solids collected from the settling basin will be tested and handled as petroleum contaminated soil if necessary. This system will be installed with the truck shop in Year 1 of operation.

18.9.3 Mine Office Complex

The mine office complex includes an office building for mine contractor staff, an office building for Pershing Gold mining staff, and a common break/lunch room building. A parking area for personal vehicles will be located adjacent to the offices area. Temporary office trailers will be used during the first year of production until the permanent mine office is constructed during Year 1.

18.9.4 Fuel Storage

A fuel station will be installed in the mine facilities area. The fuel station will consist of two 20,000-gallon off-road diesel tanks, a split 20,000-gallon tank for gasoline and clear diesel, and five 500-gallon oil/lube storage tanks, plus associated dispensing equipment and controls.

18.10 Communications

Communication facilities currently exist at the site, including telephone, internet and radio. Modifications are planned to add additional radio repeater capabilities and wireless links to the water wells and the crushing and process systems. The potential for installation of a tower to add cellular phone coverage to the site will be investigated.

18.11 Reagent, Fuel and Explosives Storage

Reagents, fuel and explosives will be transported to the Relief Canyon project on trucks from suppliers located throughout Nevada, and possibly from other western states, via US Interstate 80 and the main Coal Canyon access road.

Most reagent tanks will be located outside of the process facilities in secondary containment. The secondary containment will hold 110 percent of the largest volume tank or tanks in series, and if located outside, will have additional capacity to hold the 100 year, 24 hour storm event. The fuel storage areas will be located in lined areas with secondary containment with 110 percent containment capacity of the largest tank or tank in series.

Table 18.2 presents the fuel, explosives and consumables that will be used and their approximate daily consumption. Process reagent consumptions are presented in Section 17.2.5 and Table 17.2.

Table 18.2 Fuels and Reagents Consumption

Fuel / Consumable	Approximate Consumption/day	Unit
Off-road Diesel (Mobile Equipment)	10,000	gallons
Off-road Diesel (Process Area Gensets)	1,200	gallons
Highway Diesel	100	gallons
Gasoline	250	gallons
Automatic Transmission Fluid	20	gallons
Engine Oil	60	gallons
Hydraulic Fluid	45	gallons
Gear Oil	40	gallons
Antifreeze	15	gallons
Used Oil	165	gallons
Used Antifreeze	15	gallons
Ammonium Nitrate	50,000	lbs
Ammonium Nitrate Emulsion	10,000	lbs
Propane	1,200	gallons

Explosive agents will be purchased, transported, stored, and used in accordance with the Bureau of Alcohol, Tobacco, and Firearms, Department of Homeland Security provisions, MSHA regulations and other applicable federal, state, or local legal requirements. The primary explosive used will be ANFO. Ammonium nitrate prill will be stored in a silo in a secure area and other explosive agents, boosters, and blasting caps will be stored within a separate secured area near the pit. The explosives storage facilities will be provided by the mining contractor.

18.12 Mobile Equipment Ready Line

Haul trucks and other mobile mine equipment will be temporarily staged at the ready line located at the mine contractor's yard when not in use. The equipment will be parked there during shift changes and when required for light maintenance. The area will be illuminated for safety and security.

18.13 Petroleum Contaminated Soil Storage Area

The hazardous waste storage area will be located next to the truck shop. Petroleum contaminated soils resulting from spills or leaks of hydrocarbons will be removed from the spill site and placed in a lined petroleum contaminated soils storage area prior to shipment offsite to an appropriately permitted facility.

18.14 Existing Crushing Plant

A two-stage crushing plant currently exists on site at Relief Canyon, which includes a jaw crushing plant (Cr 3042 with 5620 AMI VGF, complete structure), cone crusher (MVP450, with structure), cone feed conveyor (36x60, with structure), 3-deck screen (CR6x20, with structure), screen feed conveyor (48x60, with structure), transfer conveyor (36x50 and supports), tunnel feed stacker (36x100, with structure), reclaim tunnel with feeder and discharge conveyor, crushing motor control center ("MCC"), and six 100 foot grasshoppers. Most of this equipment does not meet the design criteria required for the selected crushing process rates at Relief Canyon, and therefore will be traded in for credit with a supplier or sold on the open market. However, the existing radial stacker will be refurbished and used to create the primary crushed ore stockpile, the existing MCC will be retained and upgraded for use with the planned operation, and the six 100-foot grasshopper conveyors will also be refurbished and used in the belt line for stacking.

19.0 MARKET STUDIES AND CONTRACTS

Gold is sold in an open market with supply and demand determining the daily price. Table 19.1 shows the average gold price over the last three years was $1,238 per ounce, based on Kitco average prices. Two future prices were used with the three year average to arrive at a gold price of $1,290 per ounce to be used for this study. Mr. Prenn believes that a gold price of $1,290 per ounce is justified and is used as the base case for this study.

Table 19.1 Kitco Average Gold Prices – Last Three Years

Month	2015	2016	2017	2018
January		$1,097.37	$1,192.62	$1,331.67
February		$1,199.91	$1,234.36	$1,331.52
March		$1,246.34	$1,231.09	$1,324.66
April		$1,242.26	$1,265.63	$1,313.20
May		$1,259.40	$1,245.00	$1,324.35
June	$1,181.50	$1,278.40	$1,260.26	
July	$1,130.04	$1,337.33	$1,236.22	
August	$1,117.47	$1,341.09	$1,282.32	
September	$1,124.53	$1,326.03	$1,314.98	
October	$1,159.25	$1,266.59	$1,279.51	
November	$1,085.70	$1,235.98	$1,282.28	
December	$1,068.25	$1,151.40	$1,261.05	
3 year back average				$1,237.8
CME future prices 2019				$1,351.3
CME future prices 2020				$1,384.2
average 3 year back and 2 year forward				$1,289.8

20.0 ENVIRONMENTAL STUDIES, PERMITTING, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

EM Strategies, Inc., a permit acquisition strategy and government relations consulting firm, provided the following information on environmental considerations, permitting, and social and community impacts. Pershing Gold's wholly owned subsidiary GAC, has in the first quarter of 2018, all the local, state, and federal permits necessary to commence permitting Phase 1 operations of mining and heap-leach processing. As discussed below, Pershing Gold plans to submit permit applications to expand and modify the permits for permitting Phase II operations, which can generally be described as mining farther below the water table. All of Pershing Gold's permitting efforts are conducted through GAC as the permittee.

20.1 Introduction

The permitting activities for the Relief Canyon Mine began in 1981 with the original Mine Plan of Operations approved in 1984 by the BLM, which included 485 acres of authorized surface disturbance. Subsequent Plan modifications in 1986, 2007, 2008, 2014, and 2015 approved additional surface disturbance by 137.6 acres for a total of 622.6 acres approved under the current Plan. At the present time, 225.6 acres of the authorized disturbance is not in use. Subsequently, The Mine Plan of Operations was amended to accommodate Phase I operations. Table 20.1 presents the authorized Phase I total acres by land ownership. The currently authorized disturbance is 636.7 acres.

Table 20.1 Authorized Surface Disturbance Acres by Land Ownership and Component

Facility	Total Acres		
	Public	**Private**	**Total**
Exploration Roads & Pads	16.3	10.2	26.5
Access Roads	11.0	9.0	20.0
Haul Roads	1.0	24.4	25.4
Wells/Pipelines	2.2	2.2	4.4
Pits	116.3	49.8	166.1
Ponds	4.5		4.5
Process Solution Ditch	3.0		3.0
Heap Leach Cells [1] – 4	62.0		62.0
heap Leach Cells 5	13.5		13.5
Heap Cells 6 and 7	72.2		72.2
Waste Rock Storage Areas [2]	55.8	108.1	163.9
Process Area, Buildings, Lab, Warehouse	0.6		0.6
Crusher Yard	3.6	17.9	21.5
Overland Conveyor	0.1	1.7	1.8
Storage/In-Fill	12.0		12.0
Contractor's Yard		14.5	14.5
Growth Media Stockpiles	6.4	12.0	18.4
Class III Landfill [3]	0.2		0.2
Materials Storage		1.8	1.8
Drainage Control	4.4		4.4
Total Acres	**385.1**	**251.6**	**636.7**

[1] *These acres are authorized in the 1984 Plan and subsequent Plan Modifications and Plan updates in 1986, 2007, 2008, 2014, 2015, and 2018. A total of 622.6 acres of surface disturbance was authorized. Current project facilities utilize 396.9 acres.*

[2] *Includes 42 acres of reclaimed waste rock storage areas that cannot be re-disturbed without additional surety.*

[3] *New Class III waivered landfill to be built in Waste Rock Storage Area; therefore, no additional acres of disturbance are needed for this facility.*

It should be noted that all acreages in Table 20.1 have been rounded to the nearest tenth of an acre, so acreages in this table may not reflect the exact permitted acres of disturbance in the approvals.

Currently, the mine has the following major environmental permits: 1) a Plan from the BLM; 2) a Nevada Reclamation Permit ("NRP") with the NDEP/BMRR); 3) a Water Pollution Control Permit ("WPCP") with the NDEP/BMRR; 4) Water rights from the Nevada Division of Water Resources; 5) an Air Quality Operating Permit ("AQOP") with the NDEP's Bureau of Air Pollution Control ("BAPC"); 6) a Mercury Operating Permit ("MOP") to Construct with the BAPC; and 7) a Special Use Permit from Pershing County.

Table 20.2 outlines the mine permits and their status.

Table 20.2 Mine Permits

Permit or Approval	Agency	Comments
Federal Permits and Authorizations		
Plan of Operations NVN-064634	BLM - Winnemucca District Office/Humboldt River Field Office	BLM approved during August, 2016 and approved a Minor Modification on 2/20/18. Covers Phase I mining and heap leaching, and exploration. Reclamation bond amount is $12,398,386. The Plan is in force for the life of the project. The June 2018 Plan Modification covers mining farther below the water table, expanded mining and mineral processing activities.
BLM Right-of-Way Grant NVN-083323	BLM - Winnemucca District Office/Humboldt River Field Office	Communications Site ROW for mine site radio repeater site.
EPA ID #NVR 000 083 709	US Environmental Protection Agency & NDEP Bureau of Waste Management	Site currently is a Conditionally Exempt Small Quantity Generator ("CESQG").
State Permits and Authorizations		
Reclamation Permit No. 0264	NDEP/ Bureau of Mining Regulation & Reclamation	NDEP issued the permit on 12/22/16 and approved a Minor Modification on 2/13/18. Covers Phase I mining and heap leaching, and exploration drilling. Reclamation bond amount is $12,398,386. Permit is good for the life of the project. The SRCE associated with the June 2018 Major Modification to be submitted once Preferred Alternative defined in NEPA process.
Water Pollution Control Permit NEV2007105	NDEP/ Bureau of Mining Regulation & Reclamation	Effective date 3/1/2018; expires on 9/24/2021. Minor Modification to the Permit approved March 1, 2018. Major Modification to the Permit to be submitted in 4Q 2018 to add the expanded heap leach pads, Operating Pond #3, and expand WRSF 5 and other WRSFs.
Class II Air Quality Class Operating Permit No. AP1041-2441	NDEP/Bureau of Air Pollution Control	Effective date 2/23/17. BAPC is currently reviewing a modified air quality operating permit to reflect updated crushing and conveying operations that include all of the Phase II facilities, crushing rates, and configurations. Permit renewal application also currently under review by BAPC.
Class I Air Quality Operating Permit to Construct No. AP1041-3652	NDEP/Bureau of Air Pollution Control	NDEP issued the Class I permit on 2/23/17, which covers the thermal mercury emission units.
Mercury Operating Permit to Construct Thermal Mercury Emission Units	NDEP/Bureau of Air Pollution Control Nevada Mercury Air Emissions Control Program	Issued on 6/21/16. Construction start-up date extended to 6/21/19.

Permit or Approval	Agency	Comments
Permit # AP1041-3585		Construction of thermal mercury emission units must start within 18 months (extension to 6/21/19 granted)
Class III Landfill Waiver No. F444	NDEP/Bureau of Waste Management	Valid from 12/22/16 - 1/11/22. This landfill will be closed and a new Class III-waivered landfill will be permitted and built in the new waste rock storage area.
Onsite Sewage Disposal System ("OSDS") Permit #GNEVOSDS09-S-0392 (Capacity <5,000 gpd)	NDEP/Bureau of Water Pollution Control	Permit No. GNEVOSDS09-S-0392. Permit administratively continued pending Bureau issuance of new General Permit. Permit to be amended to include, or new permit application submitted Q4 2018 for new septic system for truck shop/warehouse to be constructed near mine pits.
Industrial Artificial Pond Permit S39298	NV Department of Wildlife	Recently renewed. Valid from 11/1/16 – 10/31/21.
County Authorization		
Pershing County Special Use Permit	Pershing County Planning Department and Board of County Commissioners	Good for life of project. Courtesy update provided to the Pershing County Commissioners for Phase I mining and heap leaching.
Building Permits	Pershing County Planning Department	Will be submitted once building designs are finalized.

DNA is a BLM Determination of NEPA Adequacy and Land Use Conformance

Pershing Gold is planning a two-phase development scenario for the mine. Phase I is the re-purposing of previously approved disturbance for expanded mining and exploration operations. Phase II will be for additional mine expansion activities, including dewatering and mining further below the water table.

Under Phase I, Pershing Gold will a) expand the existing open pits, creating one larger pit that will be partially backfilled, b) build a new waste rock storage facility on private land, c) conduct exploration activities outside of the existing pit area, and d) construct ancillary facilities. This includes using 211.8 acres of previously authorized but currently unused surface disturbance. The planned disturbance is needed for mine expansion and mineral exploration activities. The mined ore will be processed on the previously permitted heap-leach pad Cells 6 and 7, of which only cell 6A is currently constructed.

The previously authorized disturbance acres will be re-purposed (i.e., used for different mining purposes in different locations within the project area) as compared to the surface disturbance authorized in the 1984 Plan and subsequent amendments. Generally, Phase I will create a larger pit than originally authorized, the heap-leach pads will be smaller, and there will be a new waste rock storage facility on private land. Phase I will also involve constructing several new roads, a new pipeline, closing and reclaiming the old heap-leach pads (Cells 1-4), adding an analytical laboratory, a contractor's yard, stockpiling growth media, and expanding the exploration drilling areas. Table 20.1 provides an acreage breakdown of the authorized facilities on public land and private land. The remainder of the project components and activities will use, and in some cases expand, the existing infrastructure within the Project Area. Process solution from ore mined under Phase I will be processed in the existing ADR facility. Water for the mining and heap-leach operations will be obtained from the existing water supply wells, PW-1 and PW-2, located west of the pit area, until it becomes necessary to relocate one or both of the wells as mining proceeds, given their proximity to the pit, in which case, replacement wells will be drilled. Power will be obtained from the existing power supply system consisting of an overhead powerline, and on-site generators.

The Minor Modification approved on February, 2018 authorized the following: construction of heap leach pad extension Cell 5; realignment, repurposing and relocation of permitted but not yet disturbed surface disturbance; reconfiguration of the open pit boundary to improve stability and safety of the open pit highwall; increase the annual ore production rate from 6 million tons to 7.5 million tons; increase the permitted heap leach solution application rate; and removing low-grade ore material from WRSF 4 and placing that material on the heap leach facility.

Pershing Gold plans to modify the project under Phase II to expand and deepen the Relief Canyon open pit mine and process the mined ore at the authorized crushing facility and on expanded heap-leach facilities. The specific expansion facilities and activities under Phase II include the following:

- Expand the footprint of the existing approved pit area by approximately 83.45 acres with resultant elimination of a portion of Waste Rock Storage Facility (WRSF) 4;
- Mine to a final pit bottom elevation of 4,420 feet above mean sea level (ft amsl), which will involve mining further below the water table, and result in a post-mining pit lake that is predicted to reach equilibrium at elevation 4,870 ft amsl 30 years after completion of mining. The lake is predicted to be a hydrologic sink with no appreciable outflow and water quality that will be safe for wildlife;
- Backfill a portion of the waste rock into selected areas of the pit;
- Expand the footprint of WRSF 5 in Section 17 and eastward into Section 16 and expand the footprint of WRSFs 1 through 3 in Sections 20 and 21. Reduce the footprint of WRSF4 from 43.3 acres to 6.0 acres, to allow for southern expansion of pit. The total surface disturbance associated with WRSFs 1-3, 4 and 5 will cover approximately 382 acres.
- Expand the heap-leach pad to the north in Section 18, and to the east into Section 17 (proposed Cell 8); and to the south in Section 18 (proposed Cell 9). The expanded heap-leach pad will additionally cover approximately 106 acres.
- Upgrade the electric line power to the site;
- Construct a third process fluid pond in Section 18 for additional storage of process fluid needed for the expanded heap-leach pads;
- Construct a dewatering well conveyance pipeline and Rapid Infiltration Basins (RIBs) in Sections 17 and 18 to re-infiltrate up to 900 gpm of mine dewatering water in excess of that needed for heap-leach process make-up water and other permitted consumptive uses during the last three months of Phase II mining; and construct associated up- and down-gradient groundwater monitoring wells, and water level measurement piezometers around the immediate RIBs perimeters;

- Install up to 50 vertical and horizontal drains in the pit wall to ensure pit slope stability;
- Convert up to 50 exploration drill holes located in and adjacent to the pit as vertical or near vertical drains and/or piezometers to monitor water levels to ensure pit slope stability;
- Construct new growth media stockpiles, diversion ditches for stormwater control, and reconfigure certain roads and fence lines necessary for the expanded facilities;
- Expand yard areas and crusher-conveyor areas to support the mining and heap-leach pad areas proposed in the 2018 Modification; and
- Close and reclaim all project facilities at the completion of the project operations.

The ore to be mined in Phase II will be processed on expanded heap-leach facilities, using the existing associated tanks, ponds, plus a new process fluid pond, and the existing Adsorption-Desorption Recovery (ADR) facility. Water for the mining and heap-leach operation will be obtained from water supply wells located west of the pit area until such time that expansion of the pit will result in the need to construct the replacement wells. The existing power supply system consisting of an overhead and buried power lines and on-site generators will be augmented with the proposed upgrade of the line power from NV Energy's Limerick substation that enters the mine site from the north. Phase II operations will continue using the existing approved ancillary facilities including the mine administration building and associated parking area, the warehouse, contractor's yard, fuel storage and dispensing areas, reagent storage, septic systems, communication facilities, yards, and groundwater monitoring wells. Additional groundwater monitoring wells will be constructed as directed by NDEP and BLM in conjunction with the expanded heap-leach pad and the RIBs.

Table 20.3 outlines the proposed surface disturbance, by land ownership, for the Phase II operations. The total surface disturbance in Table 20.3 is 1,222.4 acres, which is a 588.7-acre increase for Phase II of the operations.

Table 20.3 Proposed Surface Disturbance Acres by Land Ownership and Component

Facility	Total Acres		
	Public	Private	Total
Exploration Roads & Pads	16.3	10.2	26.5
Access Roads	10.1	13.3	23.4
Haul Roads	0.3	17.4	17.7
Wells/Pipelines	2.2	2.2	4.4
Pits	184.3	65.5	249.8
Ponds	6.8		6.8
Process Solution Ditch	3.0		3.0
Heap Leach Cells 1 – 4	59.4		59.4
Heap Leach Cells 5	13.5		13.5
Heap Cells 6 and 7	72.2		72.2
Heap Leach Cells 8 and 9	87.0	19.0	106.0
Waste Rock Storage Areas	62.6	319.0	381.6
Process Area, Buildings, Lab, Warehouse	0.6		0.6
Crusher Yard	0.7	90.8	91.5
Overland Conveyor Corridor	0.1	1.7	1.8
Storage/In-Fill	12.0		12.0
Contractor’s Yard		14.5	14.5
Growth Media Stockpiles	17.3	19.4	36.7
Class III Landfill	0.2		0.2
Materials Storage		1.8	1.8
Other Yards	41.3	30.1	71.4
RIBs	4.9	9.6	14.5
Drainage Control	8.7	4.4	13.1
Total Acres	**603.5**	**618.9**	**1222.4**

In order to construct, operate, reclaim, and close mining operations under Phase II at the project, Pershing Gold will be required to modify and obtain a number of environmental and other permits from the BLM, the NDEP, NDWR, and Pershing County. The principal permits necessary for the mine development are: 1) the Plan with the BLM; 2) the NRP with the BMRR; 3) the WPCP with the BMRR; 5) the dewatering water rights from the NDWR; and 6) the Special Use Permit with Pershing County. In order to obtain these permits, applications need to be submitted to each agency. In the case of the Plan and the NRP, there is a single application (Plan Application) that meets the requirements of both the BLM and BMRR. Pershing Gold submitted the 2018 Plan Modification for Phase II to BLM and the modified NRP for Phase II to NDEP in mid-June 2018.

Pershing Gold will comply with applicable federal and state environmental statutes, standards, regulations, and guidelines in the permitting of the project. Environmental baseline studies have been conducted as part of the permit acquisition activities for Phase II of the project to meet federal and state requirements.

The review and approval process for the Phase II Plan by the BLM constitutes a federal action under the NEPA and BLM regulations. Thus, for the BLM to process the Plan Application the BLM is required to comply with NEPA and prepare either an EA, or an Environmental Impact Statement ("EIS"). The following sections provide additional detailed information on the principal permits necessary to develop each phase of the project and the NEPA process, as well as the status relative to each permit process.

20.2 BLM Plan of Operations / Nevada Bureau of Mining Regulation and Reclamation, Nevada Reclamation Permit

The BLM and the BMRR have implemented a process for Plan Application processing that commences prior to the submittal of the Plan Application and continues through the review and approval process for the Plan Application. Pershing Gold submitted a Plan Modification for Phase II of the project in mid-June 2018 and BLM approval of this application will likely occur in the fourth quarter of 2019.

20.2.1 Bureau of Land Management Pre-Application Planning

As part of the pre-application planning process with the BLM, an initial, pre-application meeting is scheduled between the proponent and the BLM to discuss the anticipated scope of the mining operation and review the likely environmental resource baseline data that will be required for the processing of the Plan Application by the BLM. This initial meeting generally occurs some time prior to the submittal of the Plan, depending on the anticipated complexity of the mining operations and baseline data needs, which varies for each project. Several meetings between Pershing Gold and the BLM Humboldt River Field Office have occurred over the last 18 months.

The process for collecting baseline data generally includes the development of baseline data collection work plans, which are submitted to the BLM for review and approval prior to initiating the baseline data collection. Following approval, field surveys are carried out to collect relevant baseline data. Depending on the environmental resource to be evaluated, desktop studies may be utilized in lieu of field surveys. Findings of the field surveys are then summarized in a report that documents the data collected. This report is then submitted to the BLM for review and approval. In some cases, the baseline data collection process will also involve the State of Nevada, depending on the resource being assessed, particularly for geochemical and hydrological surveys. Baseline data for Phase II of the project has been collected, but the reports have yet to be submitted and reviewed/accepted by the BLM. These reports are included in the Plan Application submittal. For Phase II of the project, the required environmental baseline data include the following: ore and waste rock geochemical characterization; hydrogeological characterization; a pit lake evaluation; and air quality modeling. Other necessary baseline data for Phase II was collected as part of the evaluation of Phase I of the project and does not need to be repeated.

20.2.2 Plan of Operations Processing

The Plan Application is submitted to the BLM and the BMRR for any surface disturbance in excess of five acres. The single application utilizes the format of the Plan document accepted by the BLM and the BMRR. The Plan Application describes the operational procedures for the construction, operation, and closure of the project. As required by the BLM and BMRR, the Plan Application includes a waste rock management plan, quality assurance plan, a storm water plan, a spill prevention plan, reclamation plan, a monitoring plan, and an interim management plan. In addition, a reclamation report with a Reclamation Cost Estimate ("RCE") for the closure of the project is required. The content of the Plan Application is based on the mine plan design and the data gathered as part of the environmental baseline studies. The Plan Application includes all mine and processing design information and mining methods. The BLM determines the completeness of the Plan Application and, when the completeness letter is submitted to the proponent, the NEPA process begins. The RCE is reviewed by both agencies and the bond is determined prior to the BLM issuing a decision record on the Plan Application and BMRR issuing the NRP.

The BLM will need to complete their review of the baseline reports in the Plan Application and approve the final version of the reports prior to moving on to the NEPA Process.

20.3 National Environmental Policy Act ("NEPA")

The NEPA process is triggered by a federal action. In this case, the issuance of a completeness letter for the Plan triggers the federal action. The NEPA review process is completed with either an EA or an EIS.

20.3.1 Environmental Assessment ("EA")

The EA process is conducted in accordance with NEPA regulations (40 CFR 1500 et. seq.), BLM guidelines for implementing the NEPA in BLM Handbook H-1790-1 (updated January 2008), and BLM Washington Office Bulletin 94-310. The intent of the EA is to assess the direct, indirect, residual, and cumulative effects of the proposed project, and to determine the significance of those effects. Scoping is conducted by the BLM and includes a determination of the environmental resources to be analyzed in the EA, as well as the degree of analysis for each environmental resource. The scope of the cumulative analysis is also addressed during the scoping process. Following scoping and baseline information collection, the EA is either prepared by the BLM, or prepared by a third-party contractor for the BLM. When the BLM determines that the EA is complete, a Preliminary EA is made available to the public for a 30-day review period. Comments received from the public would be incorporated into a Final EA, or included in the decision record and Finding of No Significant Impacts.

20.3.2 Environmental Impact Statement ("EIS")

The EIS process is conducted in accordance with NEPA regulations (40 CFR 1500 et. seq.), BLM guidelines for implementing the NEPA in BLM Handbook H-1790-1 (updated January 2008), and BLM Washington Office Bulletin 94-310. The intent of the EIS is to assess the direct, indirect, residual, and cumulative effects of the project and to determine the significance of those effects. Scoping is conducted by the BLM and includes a determination of the environmental resources to be analyzed in the EIS, as well as the degree of analysis for each environmental resource. The scope of the cumulative analysis is also addressed during the scoping process. Following scoping and baseline information collection, the Draft EIS is prepared for the BLM by a third-party contractor. When the BLM determines the Draft EIS is complete, it will be submitted to the public for review. Comments received from the public will be incorporated into a Final EIS, which will in turn be reviewed by the BLM and the public prior to a record of decision ("ROD"). Under an EIS there can be significant impacts. The preparation of an EIS is a lengthier and more expensive process than an EA. The project proponent pays for the third-party contractor to prepare the EIS, and also pays recovery costs to the BLM for any work on the project by BLM specialists.

For Phase II of the project it is expected that the BLM will require the preparation of an EIS to comply with the NEPA for this project, which under the new Secretarial Order 3355 has to be completed in 365 days (for the Notice of Intent publication in the Federal Register to the signing of the Record of Decision) and must be less than 150 pages (unless a Department of Interior wavier is obtained, which then allows for 300 pages).

20.4 State of Nevada Permits

As listed above, there are a number of environmental permits issued by the NDEP that are necessary to develop Phase II at Relief Canyon. Pershing Gold currently has all the state and federal permits for Phase I activities. However, a number of these permits will need to be modified for Phase II activities. The NDEP issues permits that address water and air pollution, as well as land reclamation. The NDWR issues water rights for the use and management of water.

20.4.1 Water Pollution Control Permit

A WPCP from the BMRR is needed to construct, operate, and close a mining facility in the State of Nevada. The contents of the application are prescribed in the Nevada Administrative Code ("NAC") Section 445A.394 through 445A.399.

A WPCP application for Phase II of the mine will be prepared and will be based on the following:

- Open pit mining, with an anticipated post-mining pit lake formation;
- Storage of non-acid generating waste rock;
- Exploration;
- Dewatering water management;
- Heap-leach pad expansion; and
- Ancillary facilities that include storm water diversions, and sediment control basin.

WPCP applications will include an engineering design for waste rock storage areas and heap-leach facilities, waste rock characterization reports, hydrogeological summary reports, engineering design for process components including methods for the control of storm water runoff, and containment reports detailing specifications for containment of process fluids. Applications will also contain the appropriate WPCP plans, including a process fluid management plan, a monitoring plan, an emergency response plan, a temporary closure plan, and a tentative plan for permanent closure of the mine.

20.4.2 Air Quality Operating Permit

Pershing Gold already has an air quality operating permit which expires July 1, 2018, for which a modification to streamline the Phase I operations was submitted to BAPC in October 2017. A renewal application was timely submitted in April 2018. No modification to the AQOP is anticipated for Phase II of the project.

20.4.3 Mercury Operating Permit

Pershing Gold has a MOP to Construct that authorizes adding the following gold recovery system components to the ADR plant: carbon stripping, electrowinning cells, a carbon regeneration kiln, a carbon soak tank, a doré furnace, and a mercury abatement system that includes a scrubber and a retort. Changes to the MOP are not anticipated for Phase II of the mine.

20.5 Pershing County

Pershing Gold already has a Special Use Permit issued by Pershing County. Updated information describing Phase II of the mine has been submitted to Pershing County for incorporation in the Special Use Permit.

20.6 Other Permits

In addition to the principal environmental permits outlined above, Table 20.4 lists other notifications or ministerial permits that may likely be necessary to operate the Phase I and II of the mine.

Table 20.4 Ministerial Permits, Plans, and Notifications

Notification/Permit	Agency	Timeframe	Comments
Mine Registry	Nevada Division of Minerals	30 days after mine operations begin	
Mine Opening Notification	State Inspector of Mines	Before mine operations begin	
Solid Waste Landfill	Nevada Bureau of Waste Management	180 days prior to landfill operations	
Hazardous Waste Management Permit	Nevada Bureau of Waste Management	Prior to the management or recycling of hazardous waste	
General Storm Water Permit	Nevada Bureau of Water Pollution Control	Prior to construction activities	
Hazardous Materials Permit	State Fire Marshall	30 days after the start of operations	
Fire and Life Safety	State Fire Marshall	Prior to construction	
Explosives Permit	Bureau of Alcohol, Tobacco, and Firearms	Prior to purchasing explosives	Mining contractor may be responsible for permit
Mine Identification Number	Mine Safety and Health Administration	Prior to start-up	
Notification of Commencement of Operation	Mine Safety and Health Administration	Prior to start-up	
Radio License	Federal Communications Commission	Prior to radio use	

20.7 Environmental Study Results and Known Issues

As previously outlined, the mine has been in place for over 30 years; however, there have been very long periods of non-operation. There are no known ongoing environmental issues with any of the regulatory agencies. Pershing Gold has been conducting baseline data collection for a couple of years for environmental studies required to support the Plan Application and permitting process. Results indicate limited biological and cultural issues, air quality impacts appear to be within State of Nevada standards, traffic and noise issues are present but at low levels, and socioeconomic impacts are positive.

20.8 Waste Disposal, Monitoring, Water Management

Waste rock characterization has been conducted and results indicate that the waste rock and ore are generally non-reactive, not acid generating, and do not leach metals. As a result, waste rock management is expected to be by random placement with only quarterly sampling of the placed materials.

20.9 Social and Community Issues

Social and community impacts have been and are being considered and evaluated for the various plan amendments performed for the project in accordance with the NEPA and other federal laws. Potentially affected Native American tribes, tribal organizations and/or individuals are consulted during the preparation of all plan amendments to advise on the proposed projects that may have an effect on cultural sites, resources, and traditional activities.

The most recent Master Plan of Pershing County, Nevada, is consulted during the preparation of plan amendments. Potential community impacts to existing population and demographics, income, employment, economy, public finance, housing, community facilities and community services are evaluated for potential impacts as part of the NEPA process.

There are no known social or community issues that would have a material impact on the project's ability to extract mineral resources. Identified socioeconomic issues (employment, payroll, services and supply purchases, and state and local tax payments) are anticipated to be positive.

20.10 Mine Closure

A Tentative Plan for Permanent Closure ("TPPC") for the project was submitted to the BMRR with the WPCP application for Phase I of the project in January 2016. As described in the TPPC, the proposed heap leach closure approach consisted of fluid management through evaporation, covering the heaps with waste rock and growth media, and then revegetating. The process of managing the solutions from the heap leach draindown would require approximately two years at a cost of approximately $2,200,000, with an additional $1,400,000 after that to finish the solution management. Residual heap drainage will be managed with evaporation cells. Costs associated with this phase of the heap leach closure are approximately $500,000. Waste Rock dumps and other facilities would be regraded, covered with growth media and revegetated. Under Phase 1 the open pit will be partially backfilled to approximately 20 feet above the historical ground water elevation. Under Phase 2 the open pit will be closed with a pit lake. The closure scenario for the heap-leach pads will result in conditions that require long-term management of the evaporation cells and associated ancillary facilities at the site, which will require a financial instrument to cover those cost into the future.

The current bond at the project is approximately $12,400,000 to commence construction of the expanded operations under Phase 1 of the project. The annual fees for the surety is approximately $290,000. Under Phase II of the project the total bond should decrease to approximately $8,800,000. This decrease is principally because Phase II does not have the same amount of backfilling of the open pit as under Phase

I. The annual fees for the surety under Phase II of the project will be approximately $220,000. It should be noted that the actual costs to complete the reclamation, as outlined in the reclamation costs estimate for the bonds is approximately 35 percent less than the bond amount, due to the regulatory agencies' markups and overhead. Therefore, based on the reclamation cost estimate for the purpose of the bond, the anticipated reclamation costs are approximately $8,000,000 for Phase II. In addition, the actual costs for Pershing Gold to implement reclamation of Phase II would likely be less than those outlined in the reclamation cost estimate, due to the use of on-site personnel and equipment. The anticipated expenditures for the three years of reclamation are approximately $1,000,000 in year one (year 6 of the operation), $4,000,000 in year two (year 7 of the operation, and $3,000,000 in year three (year 8 of the operation).

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

Capital costs were developed based on a contract mining proposal, owner operated crushing, and new heap-leach pads. The base case was developed on a scenario in which the mining contractor will feed the crusher, with some of the equipment planned to be acquired on a lease-purchase option. The crushed material will be conveyed and stacked on the leach pad. Table 21.1 shows the capital cost estimate for the Relief Canyon heap-leach mine. The capital cost estimate was developed from quotations and detailed estimates based on first principles by KCA, MDA, and Pershing Gold, and has an accuracy of about +/- 15 percent. The capital and operating cost estimate is in terms of first or second quarter 2018 costs. MDA was responsible for providing the mine portion of the estimate, while KCA provided the process and infrastructure estimate (with inputs from Pershing Gold). Working capital is based on the first 2.5 months of operating cost, recovered during the second year of operation.

Table 21.1 Capital Cost Estimate ($000's)

Activity	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
Process Equipment Leased										(Year 1 thru 5)
Process - General Facilities	303.7									
Process - Mobile Equipment	327.2									0.0
Process Plant	2,220.8									
Crushing Plant & Reclaim	1,360.3									
Heap Leach & Solution Handling	8,986.4		208.0	6,220.4						6,428.4
Water Facilities	612.0									
Power Distribution	933.0									
Process Commisioning and Supervision	65.0									
Process Spare Parts	521.0									
Process First Fills	262.4									
Process Preproduction Labor	170.2									
Process Preproduction	0.0									
Owners Cost (Includes Preproduction G & A)	1,024.8									
Belt Stacking System	1,521.8									
Mine Contractor Facilities			660.0							660.0
Water Wells / Pit Dewatering			3,127.5	848.7						3,976.2
Fire Water Network		325.9								325.9
Mine - Radios, WiFi, Survey Equipment, Computers	200.0									0.0
Mine - Contractor Facilities (Shop, Fuel Storage)										
Mine - Preproduction	5,553.7									
Mine - Ramp System outside Pit										0.0
Mine - Light Vechicles	307.4			219.6						219.6
Reclamation							1,000.0	4,000.0	3,000.0	8,000.0
Return of Bond Collateral									(3,690.0)	(3,690.0)
Salvage of mine and process equipment							(4000.0)			(4,000.0)
Salvage of existing crushing not used	(459.0)									
Subtotal	**23,910.7**	**325.9**	**3,995.5**	**7,288.6**	**0.0**	**0.0**	**(3,000.0)**	**4,000.0**	**(690.0)**	**11,920.1**
Mine Contingency	555.4	0.0	0.0	0.0						0.0
Process Contingency	2,848.4	48.9	500.3	1,060.4	0.0	0.0				1,609.6
EPCM	580.0	19.6	200.1	424.1	0.0	0.0				643.8
Indirects	344.3	19.6	200.1	424.1	0.0	0.0				643.8
Subtotals	28,238.8	413.9	4,896.1	9,197.3	0.0	0.0	(3,000.0)	4,000.0	(690.0)	14,817.4
Working Capital		**10,157.1**	**(10,157.1)**					**0**		
Totals	**28,238.8**	**10,571.0**	**(5,260.9)**	**9,197.3**	**0.0**	**0.0**	**(3,000.0)**	**4,000.0**	**(690.0)**	**14,817.4**

21.1.1 Contract Mining

The capital cost is based on a contract mining proposal that would use 100 ton trucks and Cat 992 loaders to mine the Relief Canyon deposit. The contractor included a mobilization fee and the cost to install his facilities, as well as a monthly fee for support labor and facilities. In addition, the owners mine staff, the cost of generating power for the mine shop, the contract cost of loading the crusher, and the cost of placing crushed material into trucks and transporting to the pad by stacker conveyor are included. The mining contractor plans to subcontract blasting services, the cost of which is included in the contract proposal. The blasting subcontractor included downhole and shot consumables other than ANFO or emulsion in his estimate. The contractor was supplied with estimated haulage cycles for each bench of each pit phase based on detailed haulage cycles calculated for the 2017 PFS. The waste dump location was modified in this study and cycles to these areas were modified for waste materials. The contract quote estimates fuel and explosives use, but these consumables were not included in the contract mining cost. MDA obtained quotes for these consumables and accepted the contractor consumption estimate.

21.1.1.1 Mine Pre-production Costs

The mine pre-production period is estimated to last for at least six months and includes establishing haul roads to access higher elevation material on both the north and south sides of the planned pits. All costs during this period are treated as capital costs. Mining during this period starts by dozing material from small higher elevation benches. Some of the waste that is dozed off smaller benches is not hauled to the waste dump during the pre-production period. A total of 1.97 million tons of material is mined during the pre-production period, including 24,000 tons of ore grade material. The contractor included the cost of a smaller drill to be used when the large production drills could not be used on some small higher elevation benches. The cost associated with preparing access roads, dozing material off high elevation benches, and mining and transporting most of the pre-production material to the waste dump or stockpile near the crusher is included as the mine pre-production cost, as are the owners cost for the mine personnel over this six-month period. Table 22.2 shows the mine pre-production cost details.

Table 21.2 Mine Pre-production Cost

MINE OPERATIONS - PRODUCTION - SUMMARY		Units	Month -6	Month -5	Month -4	Month -3	Month -2	Month -1	Total Preproduction
	Dozed Material	(t x 1000)	24	67	40	0	0	0	131
	Ore	(t x 1000)			0	0	0	24	24
	Waste Dump Material	(t x 1000)	5	17			0	3	25
	Alluvium	(t x 1000)	19	50	40	47	127	100	383
	Rock Waste	(t x 1000)			104	383	362	688	1,537
	Total Waste	(t x 1000)	24	67	144	430	489	791	1,945
	TOTAL Mine Production	(t x 1000)	24	67	144	430	489	815	1,969
	Pit Ore	(t/day)			0	0	4	777	197
	Pit Waste	(t/day)			5,945	14,338	15,779	25,509	16,881
	TOTAL DAILY PRODUCTION	(t/day)			5,945	14,338	15,784	26,286	17,078
	Stockpile Ore	(t/day)			0	0	0	0	5,681
	STRIP RATIO	w/o			0	0	0.01	32.83	80.3
GENERAL MINE EXPENSE & ENGINEERING									
	Contract Staff - Monthly Fee	($ x 1000)	$15.0	$15.0	$135.2	$185.2	$185.2	$185.2	$720.8
	Contract Facilities and Equipment per Month	($ x 1000)	$15.0	$15.0	$24.0	$64.0	$64.0	$64.0	$245.9
	Owner Staff (preproduction included KCA)	($ x 1000)							$0.0
	Owner Supplies	($ x 1000)	$3.0	$3.0	$3.0	$5.0	$5.0	$5.0	$24.0
	Generator operating cost less fuel	($ x 1000)			$4.4	$4.3	$4.4	$4.4	$17.5
	Generator Fuel	($ x 1000)			$6.0	$10.2	$16.6	$16.6	$49.4
	Subtotal General Mine	($ x 1000)	**$33.0**	**$33.0**	**$172.6**	**$268.6**	**$275.1**	**$275.1**	**$1,057.5**
DRILLING	Small Diameter	($ x 1000)			67.5				67.5
	Large Diameter Rock Waste	($ x 1000)				85.3	80.6	153.1	319
	Large Diameter Rock Ore	($ x 1000)				0	0	5.2	5.2
	Large Diameter Alluvium	($ x 1000)				10.7	28.7	22.6	62
	Fuel	($ x 1000)			2.1	8.9	10.1	16.7	37.8
	Owner Assaying Fixed Fee	($ x 1000)							
	Owner Assaying	($ x 1000)				$2.00	$3.00	2.9	7.9
	Subtotal Drilling	($ x 1000)			**$69.7**	**$106.8**	**$122.5**	**$200.5**	**$499.4**
BLASTING	Small Diameter	($ x 1000)			$51.9				$51.9
	Contract Mine Blasting-alluvium	($ x 1000)				$6.0	$16.0	$12.6	$34.6
	Contract Mine Blasting-ore	($ x 1000)						$2.4	$2.4
	Contract Mine Blasting-rock waste	($ x 1000)				$39.8	$37.6	$71.4	$148.8
	ANFO	($ x 1000)			$11.0	$44.7	$50.8	$81.8	$188.2
	Elmulsion	($ x 1000)							
	Fuel	($ x 1000)			$1.3	$3.9	$4.4	$7.4	$17.0
	Subtotal Blasting	($ x 1000)			**$64.2**	**$94.3**	**$108.9**	**$175.7**	**$443.1**
LOADING	Salaries & Wages	($ x 1000)							
	Contract Mine Loading	($ x 1000)			$18.7	$77.4	$88.1	$146.7	$330.9
	Fuel	($ x 1000)			$4.5	$13.3	$15.2	$25.3	$58.2
	Subtotal Loading	($ x 1000)			**$23.2**	**$90.8**	**$103.2**	**$171.9**	**$389.1**
HAULING	Ore	($ x 1000)			$0.0	$0.0	$0.1	$14.7	$14.8
	Ore Fuel	($ x 1000)			$0.0	$0.0	$0.0	$2.3	$2.3
	Waste	($ x 1000)			$77.9	$322.6	$366.9	$593.1	$1,360.5
	Waste Fuel	($ x 1000)			$11.0	$45.6	$51.9	$83.8	$192.3
	Subtotal Hauling	($ x 1000)			**$88.9**	**$368.2**	**$418.8**	**$693.9**	**$1,569.9**
SUPPORT	Contract Mine Support	($ x 1000)	$11.3	$31.5	$67.6	$202.2	$230.0	$383.0	$925.5
	Support and Maintenance Fuel	($ x 1000)	$1.6	$4.4	$9.4	$28.0	$31.8	$53.0	$128.0
	Mine Dewatering	($ x 1000)							
	Subtotal Support	($ x 1000)	**$12.8**	**$35.8**	**$77.0**	**$230.1**	**$261.8**	**$435.9**	**$1,053.5**
SUMMARY	Owner Staff and Supplies	($ x 1000)	$3.0	$3.0	$13.4	$19.5	$26.0	$26.0	$90.8
	Fuel & Power	($ x 1000)	$1.6	$4.4	$28.3	$99.7	$113.4	$188.5	$435.7
	ANFO and Assesories	($ x 1000)			$11.0	$44.7	$50.8	$81.8	$188.2
	Assaying	($ x 1000)			$0.0	$2.0	$3.0	$2.9	$7.9
	Pioneering & Dozed Material	($ x 1000)			$119.4	$75.0	$75.0	$75.0	$344.4
	Contract Mine	($ x 1000)	$11.3	$31.5	$164.2	$743.9	$848.1	$1,404.8	$3,203.7
	Contract Mine Staff	($ x 1000)	$15.0	$15.0	$135.2	$185.2	$185.2	$185.2	$720.8
	Contract Mine Facilities	($ x 1000)	$15.0	$15.0	$24.0	$64.0	$64.0	$64.0	$245.9
	Mine Dewatering	($ x 1000)			$0.0	$0.0	$0.0	$0.0	$0.0
	Mob + Facilities Recovery	($ x 1000)				$105.4	$105.4	$105.4	$316.2
	Subtotal Total Preproduction	($ x 1000)	**$45.8**	**$68.8**	**$495.5**	**$1,339.2**	**$1,470.7**	**$2,133.5**	**$5,553.7**
	Total Material Mined	(t x 1000)	24	67	143.9	430.2	489.3	814.9	1,969.20
UNIT COST	Contract Mine Labor & Facilities	$/t Mined	$1.25	$0.45	$1.11	$0.82	$0.73	$0.44	$0.65
	General Mine Expense	$/t Mined	$0.13	$0.05	$0.09	$0.05	$0.05	$0.03	$0.05
	Drilling	$/t Mined			$0.48	$0.25	$0.25	$0.25	$0.25
	Blasting	$/t Mined			$0.45	$0.22	$0.22	$0.22	$0.23
	Loading	$/t Mined			$0.16	$0.21	$0.21	$0.21	$0.20
	Hauling	$/t Mined			$0.62	$0.86	$0.86	$0.85	$0.80
	Support	$/t Mined	$0.54	$0.54	$0.54	$0.54	$0.54	$0.54	$0.54
	Total Preproduction	$/t Mined	**$1.91**	**$1.03**	**$3.44**	**$2.94**	**$2.85**	**$2.53**	**$2.71**

21.1.2 Process and Infrastructure Capital Costs

The required pre-production capital expenditures for the Relief Canyon project are summarized in Table 21.3. These costs are stated in US dollars (USD), are based on the design outlined in this study and are considered to have an accuracy of +/-15 percent. The scope of these costs includes all process facilities and project infrastructure.

The costs in Table 21.3 have been estimated using information provided by Pershing Gold (directly and via sub-contractors), KCA and MDA. A significant amount of the process and infrastructure costs in the study were provided to KCA by Pershing's consultants, who obtained budgetary quotations from several vendors and contractors, and provided these to KCA for review. KCA has determined the costs to be reasonable.

All equipment and material requirements are based on the design information described in this study. Capital cost estimates have been made using budgetary quotes from contractors and suppliers for most major items, with almost all quotes having at least two sources. Other items were estimated from consultants via their own database and KCA's project files for recent projects in Nevada, where contractor and supplier quotes were available for similar works and equipment.

A number of facilities and process equipment already exist on site at Relief Canyon. Costs are based on a combination of the purchase of new equipment items, repair or refurbishment of existing items and facilities, and purchase of used equipment where reasonable. Additionally, some of the equipment will be financed, including the crushing plant, stacking system, mobile equipment and the diesel generators. The financing includes equipment supply only, without any installation costs.

Financing will be with a single lender. Financed equipment is assumed to include a 20 percent down payment in pre-production, with five-year lease terms at an annual percentage rate of 5.89 percent paid for the first three years and a balloon payment in Month 37 of the contract (Year 4). The down payment is included as a pre-production capital cost and all monthly and balloon payments are included as operating costs.

Most costs have been collected or updated in the first quarter of 2018; all prices were quoted in US dollars. Where quotes were obtained prior to first quarter of 2017, appropriate inflation allowances were made.

Table 21.3 Summary of Pre-Production Process Capital Costs by Area

Plant Totals Direct Costs	Supply US$ 000's	Install US$ 000's	Grand Total US$ 000's
Area 00 - General Facilities	$149.0	$154.6	$303.7
Area 01 - Mobile Equipment	$327.2	$0.0	$327.2
Area 02 - Crushing & Reclaim	$709.8	$650.5	$1,360.3
Area 06 - Stacking	$1,070.5	$451.2	$1,521.8
Area 10 - Heap Leach & Solution Handling	$3,842.6	$5,143.8	$8,986.4
Area 10 - ADR Recovery Plant	$1,860.6	$360.2	$2,220.8
Area 20 - Water Facilities	$546.8	$65.2	$612.0
Area 20 - Power Supply & Distribution	$568.4	$364.6	$933.0
Area 25 - Laboratory	$0.0	$0.0	$0.0
Other	$0.0	$0.0	$0.0
Process Plant Total Direct Costs	**$9,075.0**	**$7,190.2**	**$16,265.2**
Comissioning & Supervision			$65.0
Spare Parts			$521.0
Contingency			$2,848.4
Total Direct Costs			**$19,699.6**
Indirect Costs			$344.3
Initial Fills			$262.4
EPCM			$580.0
Owner's Costs & Indirects			$1,024.8
SUBTOTAL Before Working Capital			**$21,911.1**
Pre-Production Labor			$170.2
Pre-Production Operating Costs			$0.0
SUBTOTAL Pre-Production Capital			**$22,081.3**
Salvage / Resale of Existing Crushing Plant (Credit)			($459.0)
TOTAL Pre-Production Capital			**$21,622.3**

21.1.3 Process Direct Costs – Basis

The capital costs in the capital cost table for each process area including facilities, mobile equipment, crushing and reclaim, stacking, heap leach and solution handling, recovery plant, water supply and distribution, power supply and distribution, and laboratory were built up from the following disciplines, where applicable:

- Major earthworks (includes pad/pond liner);
- Concrete;
- Structural steel;
- Platework;
- Mechanical equipment;
- Piping;
- Electrical and instrumentation; and
- Infrastructure.

Supply, freight, tax and installation costs are included in the capital cost buildup for each discipline, where applicable. Engineering, procurement, and construction management ("EPCM"), commissioning and supervision, indirect costs, and initial fills inventory, owner's costs and indirects are included in the total direct costs.

21.1.4 Freight

Freight costs, if not included as part of quotes, were estimated based on recent project information or were factored as percentage of mechanical equipment supply costs. When factored, 7.0 percent was used based on similar project experience.

21.1.5 Tax

State of Nevada and Pershing County sales taxes are applicable to most purchased goods. Tax was included in many of the vendor quotes. If tax was not included it was applied at a rate of 7.1 percent to equipment and parts supply, and to equipment rentals. Tax was not applied to installation costs or labor.

21.1.6 General Facilities Area

The general facilities area includes general supporting infrastructure items. The truck shop, truck wash, fuel station, and haul roads are included under the mining contract and thus are excluded from the process capital costs. Items included under the process area are site fencing, landfill, septic system for the truck shop and mine offices, and electrical distribution equipment for the truck shop area.

21.1.7 Mobile Equipment Area (Process)

The mobile equipment includes a combination of existing, used and new equipment. Purchase of mobile equipment includes a skid steer loader, loader, dozer, backhoe, mechanic/electrician service truck, forklift, flatbed service truck, water truck, telehandler, mobile crane, and four light plants. Most of the selected equipment is used and assumed to be purchased with low hours, excluding the skid steer loader, flatbed truck, and portable lights which are assumed to be purchased new.

Pershing Gold currently owns a small telehandler and lube truck. Some cost allowances were estimated by Pershing for minor repairs and maintenance for this equipment. Pershing Gold also currently owns several light trucks. Additional trucks for purchase new are included under the mining capital costs.

A single Cat 992 loader will be used for feeding the primary crusher. This loader will be provided by the mining contractor and is included under the mining costs.

21.1.8 Crushing and Reclaim Area

Costs for the equipment in the crushing plant were based on supplier quotes and included the primary jaw crusher, 70-ton rock box and supporting steel structures, conveyors, and dust suppression hardware. The reclaim system was also based on supplier quotes and included a corrugated plate tunnel, with two vibrating reclaim feeders and tunnel conveyor.

Install costs were estimated by KCA using tiered percentages based on equipment value and an hourly rate of $65; install costs closely match a single-source contractor quote obtained for the crushing, reclaim, conveying and stacking areas. The stockpile stacker exists on site and costs were included for refurbishment and repair. Costs of the retaining wall at the crusher were based on a supplier quote, designed as a welded wire retaining wall.

The costs for related earthworks were based on preliminary general arrangements of the crushing and conveying areas, while unit costs were quoted by a contractor. Costs include grading around the crusher platforms and stockpile areas.

Concrete costs were calculated based on general arrangement layouts and unit rates from contractor quotes for crusher, conveyors, and MCC foundations.

Electrical costs were quoted by equipment vendors. The existing MCC and control room will be reused with the new primary crushing system as described in Section 18, with some upgrades required and associated costs included.

21.1.9 Stacking Area

The heap-leach stacking system will be purchased from a different vendor than the crushing equipment. The overland conveyor will be furnished by the crushing vendor, but is included in the stacking cost area. Costs for the equipment for the belt stacking system were based on supplier quotes and included a 3,120-foot overland conveyor, 55-foot transfer conveyor, two 100-ton cement silos (one existing with costs to relocate and refurbish; and one new from a supplier quote), one 100-foot mobile grasshopper, twelve 110-foot mobile grasshoppers, four 110-foot ramp conveyors, 136-foot radial stacker, and a tugger unit. The stacking system will not include a horizontal index conveyor or index feed conveyor. It was assumed six existing 100-foot conveyors would also be used in the belt line for stacking, and refurbishing costs of these existing conveyors were also included.

Future costs include a 1,000-foot overland conveyor to be purchased in Year 3, to feed the belt stacking system to the north for Pad 8.

Grading of the conveyor corridor and concrete sleepers were also included in the costs.

Electrical costs were included based on quotes by various contractors. The electrical design for the stacking system was based on stepping up to medium voltage from the generator plant at the edge of the heap, distributing medium voltage to three portable step-down transformers on skids on top of the heap that would be moved with the stacking belt line, and running low voltage from the transformers to individual grasshoppers. Costs were included for weather-resistant mine duty mobile transformers along with all low and medium voltage distribution cabling and related electrical accessories.

21.1.10 Heap Leach and Solution Handling Area

This area includes heap-leach pad construction and items related to the pumping, drainage and leach solution handling systems.

Costs for Pad 5/6/7 heap-leach earthworks, geomembrane liner, gravel overliner, and piping were based on material take-offs generated by Welsh Hagen and multiple contractor and supplier quotes for supply and installation, provided by one of Pershing's consultants. The Pad 8 heap-leach costs were determined from material quantities estimated by Knight-Piesold, scaled-down by KCA based on the current developed stacking plan, and the final costs calculated using the unit rates from Pad 5/6/7. Surveying and QA/QC activities for the heap were also included in these costs. An additional allowance for building a new ditch, and for the run of barren and pregnant solution piping was estimated by Pershing and was based on preliminary quantities for earthworks and piping, and on supplier and contractor quotes for supply and install. Some existing barren and pregnant solution piping from the previous operation was assumed to be dismantled and reinstalled for the new operation.

Additional costs were included by KCA for purchase of warehouse spare pumps for pregnant and barren solutions, and installation of barren solution strainers for the heap-leach irrigation system.

Future costs include a barren solution booster pump to be purchased in Year 2.

The existing heap Cell 6W contains approximately 185,000 cubic yards of leached material from the previous operation that will require off-loading during the pre-production period. Costs for this activity were quoted by a contractor on a unit rate basis. Pershing estimated a small allowance for the inspection and potential repair of the existing liner.

21.1.11 ADR Recovery Plant Area

Costs in this area are for refurbishing and upgrading the existing ADR plant to proper working condition.

Several quotations were provided for upgrading, replacement and/or refurbishing of the carbon regeneration kiln, electrowinning cells, smelting furnace, recovery area boiler, and addition of a mercury abatement system. The carbon regeneration kiln will be the only refurbished item, while the remaining equipment will be purchased new.

KCA inspected the plant and added costs for a cyanide storage tank system, insulation for strip tanks, modification of the acid wash area, modifications to the carbon handling system, modifications to the refinery room and area partitioning, and building infrastructure.

In an effort to secure lower unit pricing for propane, the propane storage system was upgraded to utilize a single 30,000-gallon propane tank.

For electrical and instrumentation, contractor quoted costs were included to upgrade the plant PLC, add an AA instrument, and reconfigure the MCC to run completely on-line power (and for the existing generator to serve as a backup generator).

21.1.12 Water Facilities Area

Costs in this area include all equipment and works related to water wells, tanks, piping and distribution of raw, fire, and potable water.

Costs were also included for production wells PW-1 and PW-2 repairs and installation of a new larger capacity motors. These costs were provided in a water management report by Schlumberger. KCA reviewed the costs and determined them to be reasonable.

KCA used a subcontractor to conduct a site-specific fire code review and identified basic requirements for fire system protection including water holding capacity and sizing requirements. The fire water design includes a main fire water tank, which will include added capacity for raw water, piping distribution, a combination of hydrants and sprinkler system, and fire annunciation panels. Costs were estimated by KCA's contractor based on recent equipment supply and install quotes on similar projects and quotes obtained directly by Pershing.

Based on the code review, Pershing elected to remove site-specific infrastructure that would require a fire main distribution network in pre-production. Pershing elected to build the fire storage tank in pre-production but defer the installation of the fire main pipe network (including hydrants and alarms) until Year 1, when the truck shop and mine contractor offices become permanent structures (and require fire protection).

Raw water costs were added for piping between the existing 80,000-gallon raw water tank and the new combination raw/fire storage tank, and piping to supply raw water to the dust suppression distribution manifolds on the crushing system. These costs were based on budgetary quotes provided by vendors and contractors for supply and install.

Costs for three separate potable water treatment systems are included at the mine contractor offices, main office and ADR plant, along with an allowance for interconnecting piping from the well and new fire/raw water storage tank.

Future costs included in Years 2 and 3 comprise replacing the current water wells with newly constructed wells (due to the current wells being mined out), and various pit dewatering equipment.

21.1.13 Power Supply and Distribution Area

The project electrical power includes both generator and line power. Generators and associated switchgear will be purchased new with pricing from vendor quotes, and include: one 725 kW prime rated generator for the crushing area, one 725 kW and one 455 kW prime rated generators for the stacking area, and one 150 kW prime rated generator for the truck shop, mine offices, and other miscellaneous facilities. Each generator includes a fuel storage tank. Costs were also included by KCA for supporting concrete, fuel piping, and electrics to the crushing plant MCC.

Line power will be provided by the power company, NV Energy, and will be distributed on-site via overhead and underground power lines. A distribution line upgrade/reconfiguration is planned, based on using the existing 14.4 kV power line to provide the necessary line power to the ADR plant and facilities. Costs for this line and associated electrical equipment such as transformers and metering upgrades were included by Pershing Gold and were based on communications with NV Energy.

21.1.14 Crushing, Stacking, Generator and Mobile Equipment

Financed equipment includes the crushing plant, stacking system, mobile equipment and the diesel generators. The financing includes equipment only, without any installation costs. Financing will be with a single lender.

The crushing plant and stacking system includes the jaw crusher plant, transfer conveyors, reclaim tunnel, reclaim feeders, overland conveyor, belt stacking grasshoppers and stacker, and all electrical equipment. Freight and taxes are included in the financeable amount, but earthworks and concrete were excluded. Approximately $1.67 million will be financed for the crushing plant and $3.56 million financed for the stacking system.

The diesel generators will be financed through the same lender. Generators included are one 150kW, one 455kW, and two 755kW for the crushing, stacking, and mine facilities areas. Approximately $0.91 million will be financed for the diesel generators.

Mobile equipment includes a mix of new and used equipment that will be financed. Included in the process mobile equipment is a skid steer loader, loader, dozer, backhoe, mechanic/electrician service truck, forklift, flatbed service truck, water truck, telehandler, mobile crane, and four light plants. Approximately $1.26 million will be financed for the mobile equipment.

21.1.15 Spare Parts

The cost of spare parts for the mechanical equipment was estimated at a rate of 5 percent of the equipment cost. An allowance for spare parts for the ADR plant was also included.

21.1.16 Contingency

Contingency is applied to the pre-production direct costs in the process areas and disciplines to cover various levels of uncertainty in the estimated costs for the study. KCA has applied contingencies varying from 10 percent to 20 percent to different areas of the project based on the full value of all equipment and contractor costs (i.e. any equipment financing is disregarded in the contingency calculation). The overall average contingency is $2.85 million, approximately 13.1 percent of the direct costs.

21.1.17 Indirect Capital Costs

Indirect costs include costs for items during the construction period such as equipment rentals, temporary construction facilities, quality control, survey support, mobilization and demobilization fees, operation of the warehouse and fenced yard, consumables such as fuel and power, security, and commissioning of certain equipment items. Miscellaneous consultants, and an allowance for updating the mine design are also included in the indirect costs. These costs have been estimated based on quotes, information supplied by Pershing Gold, estimated equipment requirements and KCA’s experience with similar projects.

21.1.18 Owner’s Costs

Owner’s costs are included which cover pre-production G&A costs and construction indirect costs, and were estimated by Pershing Gold. Costs include office operating expenses, legal fees, phones/internet service, office supplies, insurance, IT services and computers, travel, community assistance and environmental expenses. Labor G&A costs include management, office, safety and security staff.

21.1.19 EPCM Costs

EPCM will be managed and executed primarily by Pershing Gold staff, with support from contractors as needed. The EPCM phase is estimated at approximately nine months in total, with six months of that period as on-site construction.

Pershing will staff a project manager, engineers, purchasing agent, survey support, and administrative personnel for EPCM initially, and will then transition most of these personnel into operations. Detailed process, civil, mechanical, piping, and electrical design and drafting, along with electrical and controls commissioning, will be supported by contractors as necessary.

21.1.20 Initial Fills

A separate initial fills component of the pre-production capital costs is included, which consists of critical consumable items purchased and stored on site at the start of operations. Initial fills items in the process capital cost areas include sodium cyanide, cement for agglomeration, diesel fuel for generators, propane, activated carbon, antiscalant, caustic soda, hydrochloric acid and fluxes (silica, borax, niter, and soda ash). This inventory of initial fills ensures adequate consumables are available for plant commissioning and operation.

21.1.21 Pre-Production Labor

The Pre-production labor is the cost of labor during initial hiring and training in the pre-production period. Some senior staff will be hired 3-6 months in advance of Year 1 start, and most operators will be hired 0.5-1.0 months before the start of production (or will not be hired until right before production starts).

21.1.22 Crushing Plant Credit

Pershing Gold currently owns an existing two-stage crushing plant, which includes a 30”x42” jaw plant, cone crusher, screen, reclaim tunnel, associated transfer conveyors, and MCC. This equipment is not planned for use in the Feasibility Study, with the exception of the MCC, and is believed to be in reasonably good condition and hold some resale value. Pershing approached the crushing equipment vendors to assess the equipment’s value in the resale market. Pershing has determined it can sell the equipment for approximately $459,000 and intends to sell in the pre-production period. This amount is therefore credited against the pre-production total capital cost.

21.1.23 Sustaining Process Capital Costs

Sustaining capital expenditures included those which are required to maintain the existing operation, but do not increase annual gold production at the mine site and exclude all expenditures which are deemed expansionary in nature. These capital items are presented in Table 21.4 and include the costs of the leach pad expansion.

Table 21.4 Process Sustaining Capital

Activity	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
	$000's	$000's	$000's	$000's	$000's	$000's	$000's
Heap Leach Expansion		$208.	$6,220.	$0.			$6,428.
Water Wells / Pit Dewatering	$0.	$3,127.	$849.				$3,976.
Fire Water Network	$326.						$326.
Totals	**$326.**	**$3,336.**	**$7,069.**	**$0.**	**$0.**	**$0.**	**$10,731.**
Contingency	$49.	$500.	$1,060.	$0.	$0.	$0.	$1,610.
EPCM	$20.	$200.	$424.	$0.	$0.	$0.	$644.
Indirects	$20.	$200.	$424.	$0.	$0.	$0.	$644.
Grand Total	**$414.**	**$4,236.**	**$8,978.**	**$0.**	**$0.**	**$0.**	**$13,628.**

Values in table may not sum due to rounding or truncation of digits displayed.

21.1.23.1 Pad 8 Heap-Leach Pad Expansion

The Pad 8 heap-leach pad expansion is expected to occur during Year 3 of process operations. Costs in this area include earthworks, liner, piping, surveying and QA/QC activities for construction for additional areas of the heap leach for Pad 8. The expansion also includes additional barren and pregnant header piping, construction of an additional event pond to handle storm flows over the new additional areas of the heap, and a 1,000-foot overland conveyor.

It is also determined that the existing barren solution pumps will not provide the necessary head for irrigating the heap once the heap has been stacked above a certain height; in Year 2 costs are included for installing a barren solution booster pump which will provide the necessary head for the remainder of the mine life.

21.1.23.2 Water Wells / Pit Dewatering

Costs for managing pit dewatering and continued water supply occur in Year 2 and Year 3 of operation. Year 2 costs include construction of a new deep well PW-3 to replace PW-2, installation of piezometers, drilling of vertical and horizontal drains for pit dewatering, and construction of RIBs and associated pipelines for excess water. Year 3 costs include construction of a new deep well PW-4 to replace PW-1 and installation of additional piezometers. Costs in this area were estimated in a pit dewatering and hydrology study prepared by Schlumberger.

21.1.23.3 Closure and Reclamation

Reclamation costs are estimated to total $8 million and occur over a three year period, after mining has been completed. The reclamation bond cash collateral total of $3.69 million will be refunded after the bonds are released.

21.1.23.4 Exclusions

The following process capital costs are excluded from the capital cost estimate:

- Finance charges and interest during construction;
- Escalation costs; and
- Currency exchange fluctuations.

21.2 Operating Cost Summary

The estimated operating costs are based on mining contractor quotes and detailed cost estimates based on first principles for the Feasibility Study base case of initially leasing some of the process equipment. These are summarized in Table 21.5. The base case operating costs are expected to average about $768.56 per ounce of gold, or $12.95 per ton of ore. All costs are presented in first or second quarter 2018 dollars. The costs are believed to have an accuracy of +/-15 percent. No contingency has been added to the operating costs.

Table 21.5 Estimated Base Case Operating Costs

Item	Units	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Production Totals
Dozed Material	000's tons	131	114		141	147	120			522
Ore	000's tons	24	5,886	5,932	5,918	4,516	4,568	3,392		30,212
Waste Dump Material	000's tons	25	701	-	-	578	1,214	-		2,492
Alluvium	000's tons	383	2,255	3,147	788	2,905	1,476	7		10,579
Rock Waste	000's tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618		101,896
Total Waste	000's tons	1,945	15,467	19,939	19,029	26,774	26,132	7,625		114,967
Total Material		1,969	21,353	25,872	24,947	31,290	30,701	11,017		145,180
Crushed Material Summary										
Tons	000's tons		5,732	5,990	6,008	4,547	4,541	3,419		30,237
Grade	oz Au/ton		0.015	0.017	0.019	0.024	0.028	0.027		0.021
Ounces	000's ounces		88	101	115	108	125	94		631
Total Silver Produced	000's ounces		4.01	51.84	93.58	101.22	84.18	85.25	11.30	431.38
Total Gold Produced	000's ounces		68.66	83.27	86.39	83.85	93.52	93.17	0.65	509.51
Revenue	$000's		$88,576.8	$107,418.7	$111,441.9	$108,164.1	$120,639.7	$120,194.6	$837.8	$657,273.5
Refining and Transportation	000's		$686.6	$832.7	$863.9	$838.5	$935.2	$931.7	$6.5	$5,095.1
Royalties (2.15%)	$000's		$1,888.3	$2,273.8	$2,345.2	$2,272.7	$2,544.7	$2,534.8	$14.0	$13,873.4
Net Sales	$000's		$86,001.9	$104,312.2	$108,232.7	$105,052.9	$117,159.8	$116,728.0	$817.3	$638,304.9
Operating Cost										
Silver Credit	$000's		($64.2)	($829.4)	($1,497.2)	($1,619.6)	($1,346.8)	($1,364.1)	($180.8)	($6,902.1)
Mining	$000's		$41,670.4	$46,331.3	$49,889.0	$61,942.2	$64,225.1	$24,035.4		$288,093.5
Load Crusher	$000's		$2,121.0	$2,216.3	$2,222.8	$1,682.3	$1,680.2	$1,265.2		$11,187.7
Processing (Lease)	$000's		$15,760.0	$16,328.0	$16,728.0	$15,292.0	$12,364.0	$9,723.0		$86,195.0
G & A	$000's		$2,450.2	$2,450.2	$2,148.2	$2,148.2	$2,148.2	$1,675.6		$13,020.8
Total Operating Cost	$000's		$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$391,594.8
Cost $/ton Ore (1)	$/ton ore		10.80	11.10	$11.57	17.47	17.41	10.33		$12.95
Cost $/ounce Au recovered(1)	$/ounce Au		$902.03	$798.56	$804.39	$947.49	$845.50	$379.24		$768.56
Net after Operating Costs	$000's		$24,064.5	$37,815.7	$38,741.9	$25,607.8	$38,089.2	$81,392.8	$998.1	$246,710.1

21.2.1 Mining Operating Costs

A monthly production schedule was prepared for the life of the mine. The details of the monthly production schedule can be found in Appendix A. The contractor estimated unit rates for each year, which were applied to the monthly material mined totals for drilling, blasting, loading, hauling, and support. In addition, the owners mine staff, the cost of generating power for the mine shop, the contract cost of loading the crusher, monthly contractor fees for operations support, maintenance support, maintenance support equipment, and facilities operation were included. The average mining cost over the life of the mine averaged $1.98 per ton mined.

Mine operating costs are based on contractor quotes for the base case, and use a fuel cost quote of $2 per gallon delivered to the mine site, obtained in February, 2018, when the crude oil price was about $50 per barrel), and a February, 2018, ANFO cost quote from Southwest Energy of $460 per ton delivered to the site ($492.66 with sales tax). The fuel cost estimate was increased to $2.10 per gallon as oil prices rose during the study. The estimated fuel consumption is about 14 million gallons of fuel over the life of the mine. Table 21.6 summarizes the mine operating cost estimate.

Table 21.6 Mine Operating Cost Estimate

Item	Year	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	TOTALS
MINE OPERATIONS - PRODUCTION - SUMMARY	Days	365	365	366	365	365	244	2,070
Dozed Material	(t x 1000)	114	0	141			0	255
Ore	(t x 1000)	5,886	5,932	5,918	4,516	4,568	3,392	30,212
Waste Dump Material	(t x 1000)	701	0	0	578	1,214	0	2,493
Alluvium	(t x 1000)	2,255	3,147	788	2,905	1,476	7	10,578
Rock Waste	(t x 1000)	12,510	16,792	18,241	23,292	23,443	7,618	101,896
Total Waste	(t x 1000)	15,466	19,939	19,029	26,775	26,133	7,625	114,967
Load Crusher	(t x 1000)	5,732	5,990	6,008	4,547	4,541	3,419	30,237
Total Waste	(t x 1000)	15,466	19,939	19,029	26,775	26,133	7,625	114,967
TOTAL Mine Production	**(t x 1000)**	**21,352**	**25,871**	**24,947**	**31,291**	**30,701**	**11,017**	**145,179**
Pit Ore	(t/day)	16,126	16,252	16,169	12,373	12,515	13,902	14,595
Pit Waste	(t/day)	42,373	54,627	51,992	73,356	71,597	31,250	55,540
TOTAL DAILY PRODUCTION	(t/day)	58,499	70,879	68,161	85,729	84,112	45,152	70,135
Stockpile Ore	(t/day)	15,704	16,411	16,415	12,458	12,441	14,012	14,607
STRIP RATIO	w/o	2.63	3.36	3.22	5.93	5.72	2.25	3.81
MINE OPERATING COST								
SUMMARY (Not Including Stockpile)								
Owner Staff and Supplies	($ x 1000)	$1,475.2	$1,513.5	$1,590.1	$1,590.1	$1,590.1	$754.8	$8,513.7
Fuel & Power	($ x 1000)	$3,988.2	$4,531.3	$4,941.5	$6,337.6	$6,864.9	$2,588.1	$29,251.6
ANFO and Emulsion	($ x 1000)	$1,551.5	$2,070.3	$2,050.2	$2,801.6	$2,665.0	$815.5	$11,954.2
Assaying	($ x 1000)	$1,036.6	$1,039.3	$1,038.6	$958.0	$956.3	$531.7	$5,560.4
Pioneering & Dozed Material	($ x 1000)	$131.1	$0.0	$81.2	$0.0	$0.0	$0.0	$212.4
Contract Mine	($ x 1000)	$27,835.1	$32,463.8	$35,374.3	$44,598.8	$47,335.9	$17,540.8	$205,148.7
Contract Mine Staff	($ x 1000)	$3,936.6	$3,936.6	$3,936.6	$3,936.6	$3,936.6	$1,434.2	$21,117.2
Contract Mine Facilities	($ x 1000)	$767.4	$767.4	$767.4	$767.4	$767.4	$306.9	$4,143.9
Mine Dewatering	($ x 1000)	$0.0	$9.1	$109.0	$109.0	$109.0	$63.6	$399.7
Mob + Facilities Recovery	($ x 1000)	$948.6			$843.2			$1,791.8
Subtotal	**($ x 1000)**	**$41,670.4**	**$46,331.3**	**$49,889.0**	**$61,942.2**	**$64,225.1**	**$24,035.4**	**$288,093.5**
MINE - UNIT OPERATING COST								
Total Material Mined	(t x 1000)	21,352	25,871	24,947	31,291	30,701	11,017	145,179
Contract Mine Labor & Facilities	$/t Mined	$0.26	$0.18	$0.19	$0.18	$0.15	$0.16	$0.19
General Mine Expense	$/t Mined	$0.07	$0.06	$0.06	$0.05	$0.05	$0.07	$0.06
Drilling	$/t Mined	$0.29	$0.28	$0.29	$0.27	$0.26	$0.29	$0.28
Blasting	$/t Mined	$0.19	$0.20	$0.20	$0.20	$0.20	$0.19	$0.20
Loading	$/t Mined	$0.21	$0.21	$0.21	$0.21	$0.21	$0.21	$0.21
Hauling	$/t Mined	$0.64	$0.58	$0.77	$0.83	$0.97	$0.97	$0.79
Support	$/t Mined	$0.29	$0.28	$0.29	$0.24	$0.25	$0.30	$0.27
Total Mine Operations	**$/t Mined**	**$1.95**	**$1.79**	**$2.00**	**$1.98**	**$2.09**	**$2.18**	**$1.98**
Total Mine Operations	**$/t ore Mi**	**$7.08**	**$7.81**	**$8.43**	**$13.72**	**$14.06**	**$7.09**	**$9.54**

The mining contractor also estimated the cost of feeding the crusher. The estimated contractor cost for loading the crusher is $0.43 per ton, including fuel.

21.2.2 Process Operating Costs

Process operating costs for the Relief Canyon project are stated in US dollars and have been based on the information presented in earlier sections of this report. Life-of-mine average process operating costs are estimated to be $2.85 per ton total ore processed (excluding costs for loader feed of the crusher which are under mining costs). The costs are summarized in US$ and in US$ per ton of ore processed, in Table 21.7 and Table 21.8, respectively.

Table 21.7 Relief Canyon Operating Cost Summary, ($000's)

Category	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM Total
Total Tons							
Crushed Ore	5,732	5,990	6,008	4,547	4,541	3,419	30,237
ROM Ore	0	0	0	0	0	0	0
TOTAL Ore	**5,732**	**5,990**	**6,008**	**4,547**	**4,541**	**3,419**	**30,237**
Crushed Ore (US$ 000's)							
Labor (All Process Areas)	$2,474	$2,474	$2,474	$2,474	$2,474	$1,710	$14,082
Crushing & Reclaim	$907	$948	$950	$719	$718	$541	$4,783
Stacking	$5,255	$5,492	$5,555	$4,204	$4,199	$3,162	$27,868
Heap Leach & Solution Handling	$4,052	$4,263	$4,275	$3,297	$3,294	$3,057	$22,239
Recovery Plant	$1,038	$1,048	$1,049	$961	$961	$720	$5,777
Water Facilities	$159	$229	$229	$214	$214	$163	$1,209
Laboratory	$16	$16	$16	$16	$16	$12	$93
General Facilities	$487	$487	$808	$487	$487	$357	$3,113
Equipment Financing	$1,371	$1,371	$1,371	$2,918	$0	$0	$7,032
TOTAL Crushed Ore	**$15,760**	**$16,328**	**$16,728**	**$15,292**	**$12,364**	**$9,723**	**$86,195**

Table 21.8 Relief Canyon Process Operating Cost Summary, US$ Per Ton of Ore Processed

Category	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM Average
Total Tons							
Crushed Ore	5,732	5,990	6,008	4,547	4,541	3,419	30,237
ROM Ore	0	0	0	0	0	0	0
TOTAL Ore	**5,732**	**5,990**	**6,008**	**4,547**	**4,541**	**3,419**	**30,237**
Crushed Ore							
Labor (All Process Areas)	$0.43	$0.41	$0.41	$0.54	$0.54	$0.50	$0.47
Crushing & Reclaim	$0.16	$0.16	$0.16	$0.16	$0.16	$0.16	$0.16
Stacking	$0.92	$0.92	$0.92	$0.92	$0.92	$0.92	$0.92
Heap Leach & Solution Handling	$0.71	$0.71	$0.71	$0.73	$0.73	$0.74	$0.74
Recovery Plant	$0.18	$0.17	$0.17	$0.21	$0.21	$0.26	$0.19
Water Facilities	$0.03	$0.04	$0.04	$0.05	$0.05	$0.06	$0.04
Laboratory	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
General Facilities	$0.08	$0.08	$0.13	$0.11	$0.11	$0.24	$0.10
Equipment Financing	$0.24	$0.23	$0.23	$0.64	$0.00	$0.00	$0.23
TOTAL Crushed Ore	**$2.75**	**$2.50**	**$2.56**	**$2.72**	**$2.72**	**$2.89**	**$2.85**

Operating costs for all areas of the process have been estimated from equipment data, vendor information and typical industry values. Labor costs are estimated using project-specific staffing, salary, wage, and benefit requirements. Unit consumptions of materials, supplies, power, water, and delivered supply costs are also estimated, and are based on test work, vendor quotes, and similar recent project data.

All costs are presented in first quarter 2018 dollars. The costs are believed to have an accuracy of +/-15 percent. No contingency has been added to the process operating costs.

Year 6 operating costs in Table 21.7 and Table 21.8 represent 7 months of production plus continued operation of the heap-leach irrigation system and recovery plant, as additional gold is expected to be recovered. Operating costs for supporting infrastructure are included during the last five months of Year 6, at a reduced capacity.

21.2.2.1 Labor

Process labor was based on a proposed staffing list by year, with wages and benefits for staffing provided by Pershing Gold. A preliminary crew schedule was developed for estimating standard and overtime hours for hourly personnel, and for calculating total labor costs. An average of 31 personnel (hourly and salary) are estimated for the process during full-scale operation Years 1 through 5, excluding personnel for loading ore to the crusher. In Year 6, full-scale operation is expected for only the initial seven months and the remaining five months will include a leach-only scenario. The leach-only stage of the project, end of Year 6, will have an average of eight personnel (hourly and salary).

21.2.2.2 Power

Operating costs for power were derived from the preliminary load list and total power consumptions discussed previously in Section 18. Costs for power include both generated power, which serves the crushing plant, conveyor stacking system, and mine facilities, and existing line power, which serves all other areas of the site. Power costs are included in each process area based on the connected loads assigned to that area.

Generated power costs were based on vendor specifications for diesel fuel consumption at anticipated loads and were estimated at $0.164/kWh based on a red-dye diesel price of $2.10 per gallon. This rate was applied to the applicable areas. Note generator maintenance costs are included under general facilities operating costs. Line power costs were based on estimated rates from discussions with NV Energy and were applied at $0.068/kWh.

21.2.2.3 Crushing and Reclaim

Crushing and reclaim costs include generated power, wear parts, overhaul and maintenance. Costs for loader feeding to the crusher are included under mining operating costs.

21.2.2.4 Stacking

Heap stacking costs include generated power, maintenance and operating costs for the overland conveyor, silos, mobile conveyor stacking equipment, cement consumption, and for a dozer on the heap.

21.2.2.5 Heap Leach and Solution Handling

Costs in this area include line power for the irrigation pumping system, cyanide consumption, replacement of irrigation piping and drip tubing, and general maintenance supplies for the area.

21.2.2.6 Recovery Plant

Operating costs for the recovery plant include line power, propane for thermal equipment and building heating, miscellaneous operating and maintenance supplies, and reagents and consumables such as activated carbon, acid, caustic, antiscalant, fluxes, and cyanide.

21.2.2.7 Water Facilities

Costs include line power for operating well pumps, costs for operating and maintaining the potable water treatment system, and other general maintenance supplies.

21.2.2.8 Laboratory / Assays

Pershing Gold will be using a contract laboratory for all analytical services for the project. Costs in this area are based on the estimated number of samples and associated assays, and preliminary contract pricing from regional laboratories to perform the work. The laboratory operating costs, including allowances for samples generated by the process and operation of the on-site AA, are included under mining operating costs.

21.2.2.9 General Facilities

Costs in this area include the diesel generator plant maintenance, line power and heating for buildings and other infrastructure, propane tank rental, and mobile equipment operating costs.

Generator maintenance was based on a preliminary maintenance schedule and costs from vendor information.

Power and heating of buildings includes the main office, warehouse, and ADR plant. Generated power and propane costs for the truck shop and mine offices area are included under the mine contractor's costs.

Mobile equipment includes a skid-steer loader, backhoe, telehandler, light plants, front loader, service truck, forklifts, flatbed truck, water truck, and large telehandler. Operating and maintenance costs were based on estimated monthly operating hours and published information for hourly costs.

21.2.2.10 Equipment Financing

Financed equipment includes the crushing plant, stacking system, mobile equipment and the diesel generators. The financing includes equipment only, without any installation costs. A total of approximately $7.4 million will be financed with a single lender. The lender has proposed initial terms of a 20 percent down payment which is capitalized, with financing for 36 months, which will report to operating costs, and the balance due in month 37. The balance paid in month 37 will be included in the operating costs. The financing rate is fixed at 5.89 percent.

21.2.3 G&A Costs

General and Administrative personnel is presented per year in Table 21.9. The costs, which include G&A labor and expenses, are presented in Table 21.10 by year. Year 6 operating costs represent seven months of production plus continued operation of the heap-leach irrigation system and recovery plant, as additional gold is expected to be recovered. Operating costs for supporting infrastructure are included during the last five months of Year 6, at a reduced capacity.

Note G&A costs do not include off-site corporate overheads.

Table 21.9 G & A Labor Personnel Totals

Job Title	Total Qty. (per year)
Mine General Manager	1
Administrative Assistant/ HR	1
Safety, Security, Environmental Superintendent	1
Controller	1
Accounts Receivable / Payable	1
Purchasing Agent	1
Warehousemen	2
Janitor	1
Subtotal G&A	**9**

Table 21.10 G&A Labor and Expenses ($US 000's)

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Labor						
Mine General Manager	$238	$238	$238	$238	$238	$139
Administrative Assistant/ HR	$95	$95	$95	$95	$95	$95
Safety, Security, Environmental Superintenden	$109	$109	$109	$109	$109	$109
Controller	$122	$122	$122	$122	$122	$71
Accounts Receivable / Payable	$88	$88	$88	$88	$88	$52
Purchasing Agent	$95	$95	$95	$95	$95	$56
Warehousemen	$115	$115	$115	$115	$115	$67
Janitor	$57	$57	$57	$57	$57	$33
Subtotal Labor	**$920**	**$920**	**$920**	**$920**	**$920**	**$622**
Expenses						
Supplies & General Maintenance	$144	$144	$144	$144	$144	$99
Land Holdings	$93	$93	$93	$93	$93	$93
Off Site Overhead	$0	$0	$0	$0	$0	$0
Legal, Audits, Consulting	$48	$48	$48	$48	$48	$48
Computers, IT, Internet, Software, Hardware	$25	$25	$25	$25	$25	$17
Environmental, Montoring Wells, Reporting	$145	$145	$145	$145	$145	$145
Bond Surety Payments	$290	$290	$195	$195	$195	$195
Donations, Dues, Public Relations	$30	$30	$30	$30	$30	$30
Fees, Licenses, Misc Taxes, Insurance	$240	$240	$240	$240	$240	$165
Permitting - Phase II Pit / WRD	$207	$207	$0	$0	$0	$0
Travel, Lodging, Meals, Entertainment	$25	$25	$25	$25	$25	$17
Telephones, Computers, Cell Phones	$25	$25	$25	$25	$25	$17
Light Vehicle Maintenance, Fuel	$50	$50	$50	$50	$50	$34
Small Tools, Janitorial, Safety Supplies	$50	$50	$50	$50	$50	$50
Equipment Rentals	$60	$60	$60	$60	$60	$60
Access Road Maintenance	$48	$48	$48	$48	$48	$33
Office Power	$50	$50	$50	$50	$50	$50
Subtotal Expenses	**$1,530**	**$1,530**	**$1,228**	**$1,228**	**$1,228**	**$1,054**
TOTAL G&A Costs	**$2,450**	**$2,450**	**$2,148**	**$2,148**	**$2,148**	**$1,676**

22.0 ECONOMIC ANALYSIS

22.1 Pre-Tax Analysis

The economic analysis of the mine was completed both on a pre-tax and after-tax basis for the base case considering contract mining and conveyor stacking of material on the leach pad. Table 22.1 presents the pre-tax economics of the mine using a $1,290 per ounce gold price and $16.75 per ounce silver price. The base case has a pre-tax Net Present Value ("NPV") at a 5.0 percent discount rate of $153.7 million (Table 22.1), with an Internal Rate of Return ("IRR") of 91.0 percent. Payback of the initial capital investment occurs in about 1.3 years of the operation.

Table 22.1 Pre-Tax Base Case Cash Flow

Item	Units	Preproduction	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Production Totals
Dozed Material	000's tons	131	114		141						255
Ore	000's tons	24	5,886	5,932	5,918	4,516	4,568	3,392			30,212
Waste Dump Material	000's tons	25	701	0	0	578	1,214	0			2,492
Alluvium	000's tons	383	2,255	3,147	788	2,905	1,476	7			10,579
Rock Waste	000's tons	1,537	12,510	16,792	18,241	23,292	23,443	7,618			101,896
Total Waste	000's tons	1,945	15,466	19,939	19,029	26,774	26,132	7,625			114,967
Total Material		1,969	21,353	25,872	24,947	31,290	30,701	11,017			145,179
Crushed Material Summary											
Tons	000's tons		5,732.4	5,990.0	6,007.6	4,546.6	4,541.0	3,419.4			30,237
Grade	oz Au/ton		0.015	0.017	0.019	0.024	0.028	0.027			0.021
Ounces	000's ounces		87.9	100.9	115.2	108.2	125.4	93.7			631.3
Total Silver Produced	000's ounces		4.0	51.8	93.6	101.2	84.2	85.3	11.3		431.4
Total Gold Produced	000's ounces		68.7	83.3	86.4	83.8	93.5	93.2	0.6		509.5
Revenue	$000's		$88,576.8	$107,418.7	$111,441.9	$108,164.1	$120,639.7	$120,194.6	$837.8		$657,273.5
Refining and Transportation	000's		$686.6	$832.7	$863.9	$838.5	$935.2	$931.7	$6.5		$5,095.1
Royalties (2.15%)	$000's		$1,888.3	$2,273.8	$2,345.2	$2,272.7	$2,544.7	$2,534.8	$14.0		$13,873.4
Net Profit	$000's		$86,001.9	$104,312.2	$108,232.7	$105,052.9	$117,159.8	$116,728.0	$817.3		$638,304.9
Operating Cost											
Silver Credit	$000's		($64.2)	($829.4)	($1,497.2)	($1,619.6)	($1,346.8)	($1,364.1)	($180.8)		($6,902.1)
Mining	$000's		$41,670.4	$46,331.3	$49,889.0	$61,942.2	$64,225.1	$24,035.4			$288,093.5
Load Crusher	$000's		$2,121.0	$2,216.3	$2,222.8	$1,682.3	$1,680.2	$1,265.2			$11,187.7
Processing	$000's		$15,760.0	$16,328.0	$16,728.0	$15,292.0	$12,364.0	$9,723.0			$86,195.0
G & A	$000's		$2,450.2	$2,450.2	$2,148.2	$2,148.2	$2,148.2	$1,675.6			$13,020.8
Total Operating Cost	$000's		$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$0.0	$391,594.8
Cost $/ton Ore			10.8	11.1	11.6	17.5	17.4	10.3			$0.01
Cost $/ounce Au recovered			902.0	798.6	804.4	947.5	845.5	379.2			$768.56
			$61,937.4	$66,496.5	$69,490.8	$79,445.1	$79,070.7	$35,335.2	($180.8)	$0.0	$391,594.8
Net after Operating Costs	$000's		$24,064.5	$37,815.7	$38,741.9	$25,607.8	$38,089.2	$81,392.8	$998.1	$0.0	$246,710.1
Cumulative Cashflow	$000's		$24,064.5	$61,880.2	$100,622.2	$126,230.0	$164,319.1	$245,712.0	$246,710.1		
Capital Cost	$000's	$28,238.8	$10,571.0	($5,260.9)	$9,197.3	$0.0	$0.0	($3,000.0)	$4,000.0	($690.0)	
Cash Flow with Capital	$000's	($28,238.8)	$13,493.5	$43,076.7	$29,544.7	$25,607.8	$38,089.2	$84,392.8	($3,001.9)	$690.0	$203,654.0
Cumulative Including Capital	$000's	($28,238.8)	($14,745.3)	$28,331.4	$57,876.1	$83,483.9	$121,573.0	$205,965.8	$202,964.0	$203,654.0	

22.1.1 Pre-Tax Sensitivity

The pre-tax sensitivity was assessed by varying the gold price, operating cost, and capital cost estimates in increments of ± 5 percent. The impact to the project NPV (at a 5.0 percent discount rate) and IRR are shown in Figure 22.1 and Figure 22.2 respectively.

Figure 22.1 Pre-Tax NPV (5%) Sensitivity to Gold Price, Operating Cost and Capital Cost ($000's)

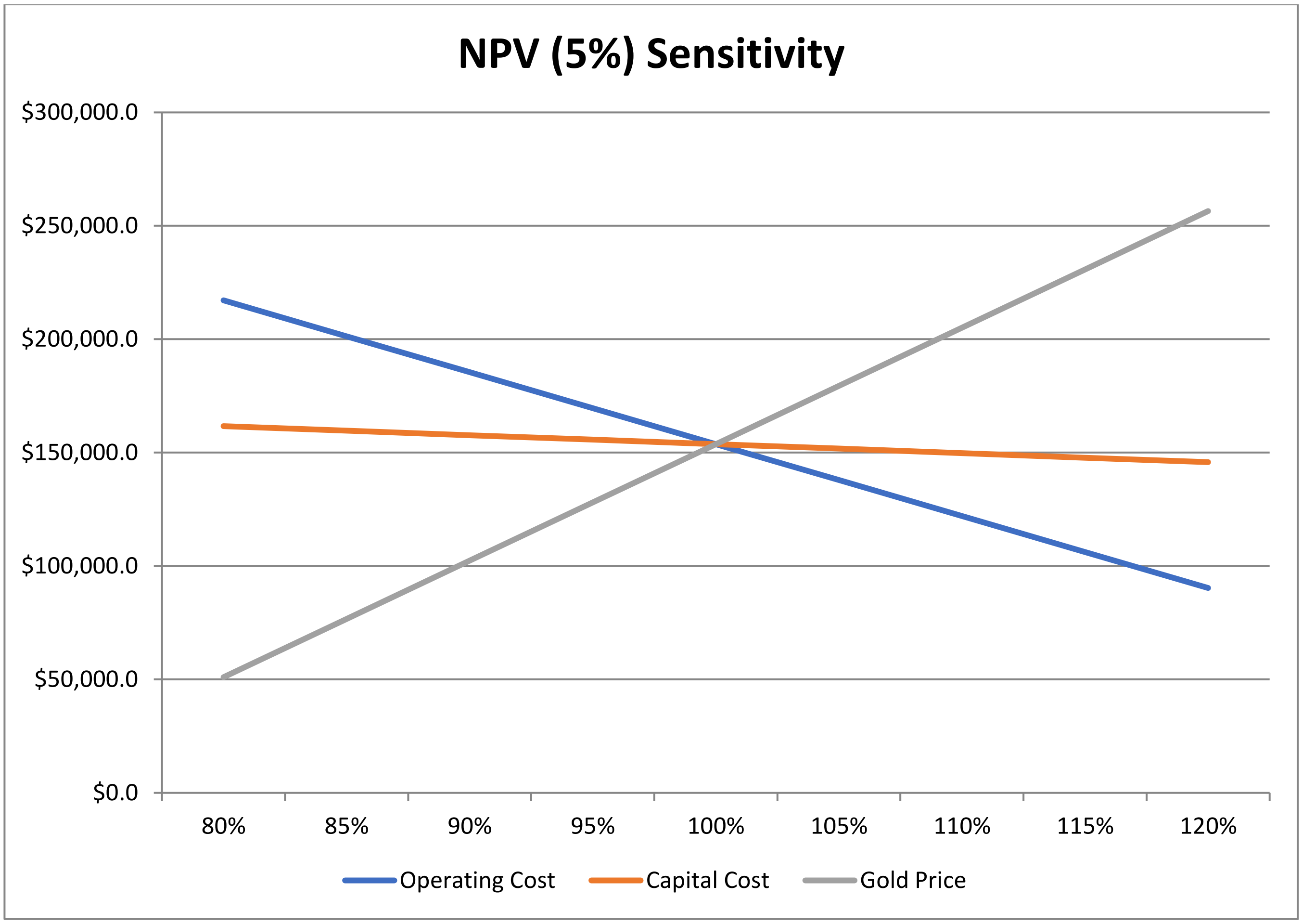
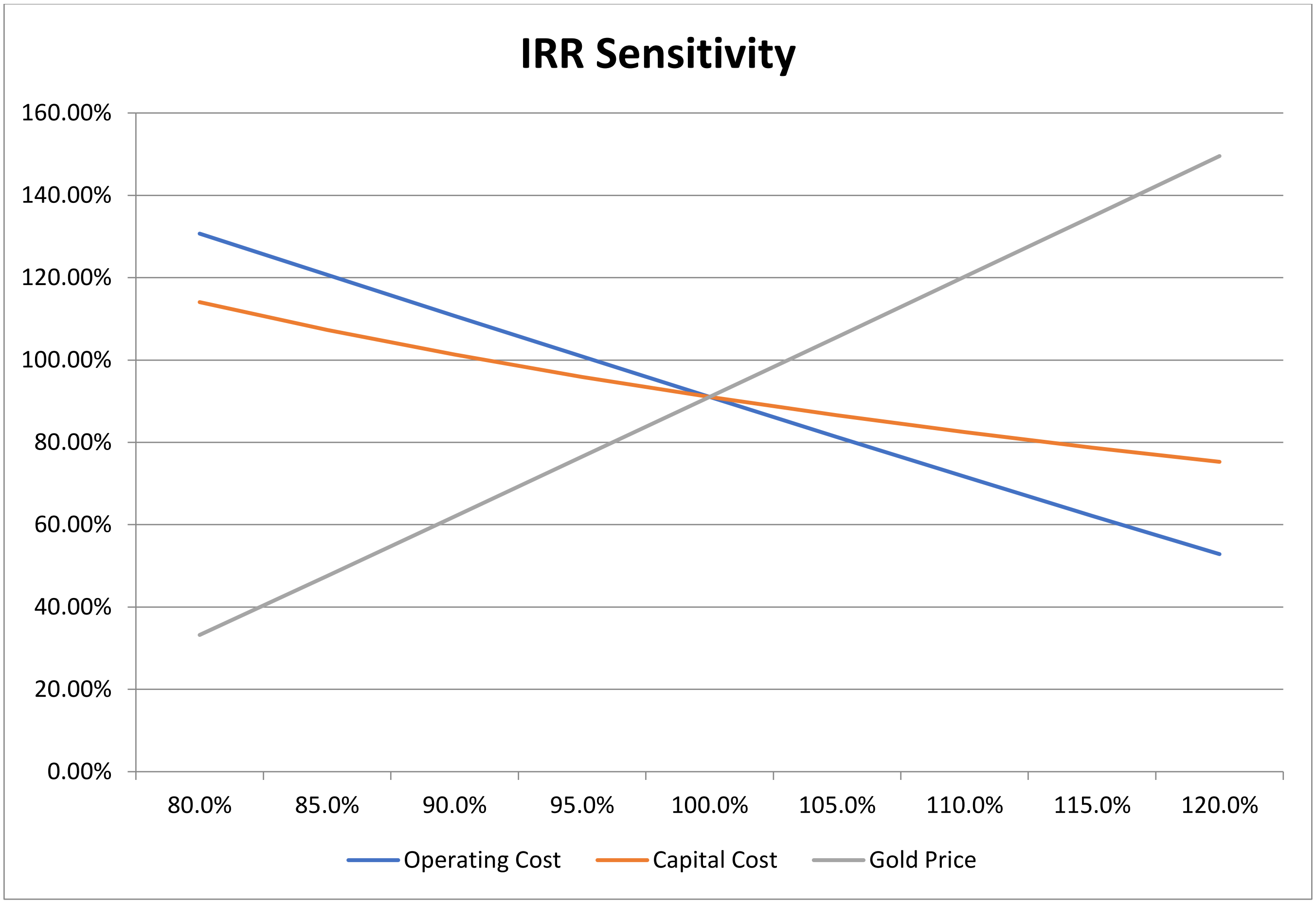


Figure 22.2 Pre-Tax IRR Sensitivity to Gold Price, Operating Cost and Capital Cost



The graphs show the mine is most sensitive to changes in gold price and operating cost.

22.2 After-Tax Cash Flow

The after-tax cash flow is shown in Table 22.2. It should be noted that Pershing Gold has approximately $73.2 million in net operating losses that can be utilized to reduce the mine's taxable income, which will lower the amount of income taxes paid over the life of the mine. This was considered in the after-tax analysis.

The after-tax cash flow is estimated at $175.7 million. The after-tax NPV at a 5.0 percent discount rate is $133.2 million and the IRR is 86.5 percent. The economic evaluation reported here indicates that the project should proceed.

Table 22.2 After Tax Cash Flow

Item	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Totals
After Tax Evaluation-Lease Equipment											
Net Profit before Tax	**$000's**		$24.1	$37.8	$38.7	$25.6	$38.1	$81.4	$1.0	$0.0	$246.7
Nevada Net Proceeds	**$000's**		$0.5	$1.9	$1.3	$1.0	$1.7	$4.1	$0.0	$0.0	$10.4
Net after Net Proceeds	**$000's**		$23.6	$35.9	$37.4	$24.6	$36.4	$77.3	$1.0	$0.0	$236.3
Depreciation	**$000's**		$3.7	$5.3	$3.6	$5.6	$5.0	$3.0	$0.0	$0.0	$26.3
Net before Depletion	**$000's**		$19.8	$30.6	$33.8	$19.0	$31.4	$74.4	$1.0	$0.0	$210.0
Depletion (15%)	**$000's**		$12.9	$15.6	$16.2	$15.8	$17.6	$17.5	$0.0	$0.0	$95.6
Depletion (50% max)	**$000's**		$9.9	$15.3	$16.9	$9.5	$15.7	$37.2	$0.0	$0.0	$104.5
Depletion Taken	**$000's**		$9.9	$15.3	$16.2	$9.5	$15.7	$17.5	$0.0	$0.0	$84.2
Taxible Income	**$000's**		$9.9	$15.3	$17.6	$9.5	$15.7	$56.8	$1.0	$0.0	$125.8
Loss Carry Forward	**$000's**		$9.9	$15.3	$17.6	$9.5	$15.7	$5.2			$73.2
Taxable Income	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$51.6	$1.0	$0.0	$52.6
Income Tax (21%)	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$17.6	$0.0	$0.0	$17.6
Income After Tax	**$000's**		$0.0	$0.0	$0.0	$0.0	$0.0	$34.1	$1.0	$0.0	$35.1
Loss Carry Forward	**$000's**		$9.9	$15.3	$17.6	$9.5	$15.7	$5.2	$0.0	$0.0	$73.2
Depletion	**$000's**		$9.9	$15.3	$16.2	$9.5	$15.7	$17.5	$0.0	$0.0	$84.2
Depreciation	**$000's**		$3.7	$5.3	$3.6	$5.6	$5.0	$3.0	$0.0	$0.0	$26.3
Net After Tax	**$000's**		$23.6	$35.9	$37.4	$24.6	$36.4	$59.8	$1.0	$0.0	$218.8
Capital Cost	**$000's**	$28.2	$10.8	($5.5)	$9.2	$0.0	$0.0	($3.0)	$4.0	-$0.7	$43.1
After Tax Cashflow	**$000's**	($28.2)	$12.8	$41.4	$28.2	$24.6	$36.4	$62.8	($3.0)	$0.7	$175.7
Cumulative After Tax Cashflow	**$000's**	($28.2)	($15.4)	$26.0	$54.2	$78.8	$115.3	$178.0	$175.0	$175.7	
NPV (5%)	**$000's**										$133.2
NPV 7.5%	**$000's**										$116.5
NPV 10%	**$000's**										$102.3
IRR	**%**										86.5%

23.0 ADJACENT PROPERTIES

This technical report does not include information from adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other information relevant to this technical report and Feasibility Study for the Relief Canyon project.

25.0 INTERPRETATIONS AND CONCLUSIONS

Based on the positive results of this Feasibility Study, the project should continue on a path to production. There are several opportunities to improve the project.

The authors have visited the project site and believe that the data provided by Pershing Gold are generally an accurate and reasonable representation of the Relief Canyon project. Mr. Tietz has reviewed the project data and the Relief Canyon drill-hole database and. concludes that it adequately supports the mineral resource and mineral reserve estimation. It is Mr. Tietz's opinion that the sample preparation, security, and analytical procedures used by Pershing Gold and prior operators were acceptable procedures and the resulting analytical data are of sufficient quality for use in the resource estimation. After updating the resource estimate with drilling results through September 2016, Measured and Indicated resources of all oxidation categories total 41.9 million tons at an average grade of 0.019 oz Au/ton, for 789,000 contained ounces of gold.

Proven and Probable reserves were calculated based on the Measured and Indicated resources by applying Modifying Factors through a pit optimization study and designed final pit with estimated costs, recoveries and gold prices applied as Modifying Factors, as described in Sections 15.0 and 16.0. Proven and Probable reserves at Relief Canyon include mostly oxide material of the Measured and Indicated resource classifications within the final design pit, as summarized in Table 25.1.

Table 25.1 Relief Canyon Mineral Reserves

Classification	Tons	Grade	Oz Au
	000's	oz Au/ton	000's
Proven	13,013.1	0.024	307.3
Probable	17,225.1	0.019	324.0
Proven & Probabl	**30,238.1**	**0.021**	**631.3**

The mine plan considers a conventional truck and shovel, open pit operation for 5.6 years with a base case of contractor mining. Pre-production mining would commence could start as early as the fouth quarter of 2018 and production mining could commence in the second quarter of 2019, depending on the timing of financing the operation. The operation would involve crushing about of six million tons per year and a maximum annual production of about 31 million tons of material. The nominal processing rate through the crusher will be 16,700 tons per day. Crushed material will be conveyed to the leach pad and stacked for cyanide heap leaching. The leached gold will be recovered from solution using a carbon adsorption circuit. Gold recoveries are estimated to be 81 percent from crushed and agglomerated material.

Total capital costs are estimated to be $28.2 million during the pre-production period, and $14.8 million in sustaining capital. A number of facilities and much of the process equipment already exist on site at Relief Canyon. The above costs are based on a combination of the purchase of new equipment items, repair or refurbishment of existing items and facilities, and purchase of used equipment where reasonable. The main sustaining capital items are expansion of the leach pad, purchase of mine equipment, and costs related to mine dewatering.
The base case operating costs are expected to average about $768.6 per ounce of gold, or $12.95 per ton of ore. This includes contract mining costs estimated at $1.98 per ton mined, and average LOM processing

costs of $2.85 per ton including the equipment lease. All costs are presented in second quarter 2018 dollars. The costs are believed to have an accuracy of +/-15 percent.

The economic analysis of the mine was completed both on a pre-tax and after-tax basis using a $1,290 per ounce gold price. The base case has a pre-tax Net Present Value ("NPV") at a 5.0 percent discount rate of $153.7 million, with an Internal Rate of Return ("IRR") of 91.0 percent. The after-tax cash flow is estimated at $175.7 million. The after tax NPV at a 5.0 percent discount rate is $133.2 million and the IRR is 86.5 percent.

25.1 Project Risks

The selected mine feed to the crusher will be 80 percent passing six inches, and the existing grizzly/jaw appears adequate to process the feed material. There is a moderate risk that the actual feed size may be larger than the anticipated size, which could result in reduced throughputs at the target product size. This may require supplemental equipment and costs to achieve the desired throughput, and/or increasing the closed side setting on the jaw crusher, the latter of which may present risks to the selected gold recovery. Alternatively, a higher powder factor and/or more tight blast hole pattern could be implemented if the feed size is larger than desired (i.e. increased mining operating costs).

Based on an initial fire code review and considering the present information on Pershing's existing infrastructure, it is believed that no fire main network will need to be installed in pre-production, and instead is deferred to Year 1 when new permanent structures (the truck shop and permanent mine offices) are added. At the time of construction and/or inspection there is a low to moderate risk that either for insurance or code compliance the pipe network may be required to be installed for some existing infrastructure, increasing pre-production capital costs.

There is a four-month period in the current production schedule during Year 4 (months 41 through 44) where mine production averages about 200,000 tons of ore per month, due mainly to one phase ending and another beginning. These months produce a negative cashflow and should be noted by operating personnel that as this time approaches that the production schedule needs to be revised during this period. Once the Phase II permit is received, it is believed that this will allow mining outside the current permit boundary earlier than planned in the Feasibility Study. This should allow more stripping in the later high stripping phase to occur, which should remove this issue. Also, possible additional ore grade material being developed by 2017 -2018 drilling should also help relieve this potential issue. Additionally, the project generates positive cash flow prior to this four-month period which is more than sufficient to absorb this negative cash flow.

There is a significant amount of used equipment planned for use in the ADR plant areas, and some used conveyors that have not been operated for several years. Although some allowances have been made for repair and refurbishment, there is a risk that during construction and commissioning the need for additional repairs and replacements may be discovered, increasing pre-production costs, and may also present some risk of additional unplanned maintenance during production over what is currently assumed (increasing operating costs). This plan incorporates purchase of new crusher, reclaim, convey and stacking equipment.

There is a risk that a varying amount of fine clayey material that may be present in the pit and that an increase in fine material above what is currently predicted to occur could potentially cause permeability problems that would affect gold recovery and heap stability. The risk is high due to the potential economic impact to gold recovery. This risk can be mitigated by:

- Monitoring the amount of fine clayey material being mined and blending this material with other coarse sandy material
- Adding additional binder cement
- Constructing additional heap leach pad area and lowering the overall height of the heap leach
- Installing an inter-lift pad liner to reduce phreatic conditions in underlying lifts
- A combination of any of these identified scenarios.

Blending of clayey and sandy material is currently the considered method for resolving this risk. The mine plan currently identifies fine material that may be blended with coarser material to ensure permeability. Additionally, the amount of cement addition to the ore may be varied at will in order to improve permeability. In anticipation of this issue, under the current plan the pad liner build-out and stacking plan reaches a maximum height of 140 feet. Pershing Gold has a land position and ample space in the Permitting Phase II Modification that can provide an opportunity to build additional pad if required, which has the potential to further lower the overall stack height should that become necessary.

The project site has previously been mined, which may lead to confined bench access at times, in addition to areas high in the designed pits that require dozing of materials. Careful planning is necessary in some areas to maintain proper mining widths between pit phases.

25.2 Project Opportunities

25.2.1 Exploration and Resource Expansion

There are several opportunities to improve the project. First, the project has a number of targets for resource expansion that should be followed up with more detailed mapping, sampling and drilling. Past production of silver from the deposit indicates that there will be a silver credit from the property, though about 1/3 of the current model resource blocks contain estimated silver grades. Additional silver assaying of the available pulps in continuous interval runs of the mineralized areas should be completed so that silver could be modeled and included for more of the resource.

Since the resource estimate was completed about fifty core holes have been drilled in three areas:

- Infill drilling in the northwest area of the final pit (eight holes to date). Drill hole results so far have indicated higher grades (about 30 percent) than predicted by the resource model. It is likely that this drilling to date will have a positive impact when the resource estimate is updated for the project;

- Extension drilling to the southwest of the north portion of the final pit (seven holes to date). This drilling has mostly been downdip of the past drilling and has intersected similar mineralization to the up-dip drilling;

- Twin hole drilling within the Main zone mineralization southwest of the existing South Pit (eight holes to date plus five infill holes). This drilling has confirmed the results from older drill holes and has also confirmed that the older holes may not have been deep enough. Drilling is continuing to confirm the extent of deeper mineralization.

About 20 percent of the Pershing Gold's 40 square mile land package has been explored. Recent exploration work has generated several targets.

The current resource is open to the west and additional drilling is recommend in this area. However, this material dips to the west and may become too deep to be contained in a future resource pit.

During late 2016, new zones of gold mineralization were identified by drilling southeast of the Lightbulb Pit. Additional drilling is warranted in this area to potentially elevate these zones of mineralization to a mineral resource status.

In addition, infill core drilling is recommended to:

- Identify the primary structural controls on the Main Zone mineralization that may result in the identification of higher-grade targets within and beneath the Main Zone;
- Expand and/or demonstrate continuity of the high-grade gold grade shells;
- The Main Zone is defined primarily by historical reverse circulation drilling. An enhanced geologic understanding of the Main Zone can be obtained by drilling core holes that will allow for improved delineation of the mineralized breccias, including those high-fines breccias that could negatively impact heap percolation. In addition, core drilling will result in an improved understanding of the structural controls on the Main Zone mineralization that may allow the identification of higher-grade targets within, and at depth beneath, the Main Zone. This work has started with positive results;
- Expand on Phase 2 drilling during 2016 that identified new zones of mineralization southeast of the Lightbulb pit, with potential to elevate these zones of mineralization to a mineral resource status; and
- Continue exploration on several targets on Pershing Gold's 40 square mine land package.

Much of this work is on-going or planned in the future.

25.2.2 Silver Credit

Past production from the deposit indicates that there will be a silver credit from the property, but the current database contains limited silver assays within the Main Zone and silver therefore is not included within the resource model and estimate in the Main Zone. At present, approximately 1/3 of the ore grade blocks have silver grades estimated for the blocks. About ½ the blocks that are planned to be crushed have silver grades estimated. Additional silver data can be obtained by assaying the available Pershing Gold sample pulps, though there is just scattered Pershing Gold drilling within the Main Zone and these core holes do not provide the sample coverage needed for a classification of Measured and Indicated. Additional infill drilling and sampling is required.

25.2.3 Historical Mineralized Waste Dump

There is an historic waste dump that has been identified as a possible source of mineralized material. This dump has been sampled by 42 shallow reverse circulation holes, and has been trenched. Currently an inferred resource is carried for this area. About half of the dump is scheduled to be mined as waste in the current production schedule of this report. Additional drilling is recommended for this area to determine easily accessible mineralized areas that could supplement the crusher stockpile as needed.

25.2.4 Other Opportunities

Preliminary test results indicate that agglomeration of blended ores with 8 lbs/ton of cement will allow heap stacking to the target and authorized heap height of 200 feet. Optimizing the quantities of cement addition needed for agglomeration pretreatment of the various material types, through continued testing could lead to improved permeability characteristics and decreased cement additions.

A trade-off study was conducted and indicated favorable economics for a case of purchasing and operating an on-site assay laboratory (vs. the current case of contract services). Purchasing an assay laboratory presents an opportunity to lower life of mine operating costs. This study should be updated to the present costs of building and operating a new lab to determine the present-day savings.

26.0 RECOMMENDATIONS

This Feasibility Study indicates the Relief Canyon project is a viable mining and heap-leach processing operation and work should continue on advancing the project to production.

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28.0 DATE AND SIGNATURE PAGE

Effective Date of report:	May 24, 2018
Completion Date of report:	July 6, 2018

"Neil B. Prenn"
Neil B. Prenn

July 6, 2018
Date Signed

"Paul Tietz"
Paul Tietz, C.P.G.

July 6, 2018
Date Signed

"Carl E. Defilippi"
Carl E. Defilippi

July 6, 2018
Date Signed

"Mark Jorgensen"
Mark Jorgensen

July 6, 2018
Date Signed

29.0 CERTIFICATE OF AUTHOR

Neil Prenn, P.Eng.

I, Neil Prenn, P.E., do hereby certify that I am currently employed as Principal Mining Engineer with Mine Development Associates, Inc. 210 South Rock Blvd, Reno, Nevada 89502, USA, and;

1. I am a graduate of the Colorado School of Mines with an Engineer of Mines degree, 1967. I have practiced my profession continuously since 1967. I have been an independent consultant for over 28 years;

2. I am a Registered Professional Mining Engineer in the states of Nevada, USA (#7844). I am a registered 'QP' member with the Mining and Metallurgical Society of America (MMSA-01283QP). I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME). I have worked in technical, operations and management positions at mines in the United States;

3. 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;

4. I visited the Relief Canyon project site on October 5, 2016;

5. I am responsible for Section 1.7 through 1.13 pertaining to mining, Section 15, 16, 18, 19, 21, 22 23, 24 and those parts of Section 25 and 26 pertaining to mining of this report titled, *Technical Report and Preliminary Feasibility Study for the Relief Canyon Gold Project, Pershing County, Nevada, U.S.A.* for Pershing Gold Corporation ("Technical Report"), with an effective date of May 24, 2018.

6. Other than my work with Pershing Gold Corporation, I have had no prior involvement with the Relief Canyon property or project that is the subject of this Technical Report, and I am independent of Pershing Gold Corporation, and all of its affiliates and subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

7. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading;

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

Dated July 6, 2018

"Neil Prenn"
Neil Prenn, P.E.

CERTIFICATE OF AUTHOR

PAUL TIETZ, C.P.G.

I, Paul Tietz, C.P.G., do hereby certify here that I am currently employed as Senior Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.

1. I graduated with a Bachelor of Science degree in Biology/Geology from the University of Rochester in 1977 and a Master of Science degree in Geology from the University of North Carolina, Chapel Hill in 1981. I also received a Master of Science degree in Geological Engineering from the University of Nevada, Reno in 2004. I have worked as a geologist for a total of 35 years since receiving my Master of Science degree in Geology.

2. I am a Certified Professional Geologist (#11004) with the American Institute of Professional Geologists. I have drilling, exploration, and resource modeling experience in similar sediment-hosted epithermal deposits throughout Nevada and the western U.S.

3. 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. I am independent of Pershing Gold Corporation and its subsidiaries, applying all of the tests in section 1.5 of National Instrument 43-101.

4. I am responsible or jointly responsible for Sections 1 (excluding 1.5) through 12, 14, and those parts of 25 and 26 pertaining to the resource model, of this technical report titled *Technical Report and Preliminary Feasibility Study for the Relief Canyon Gold Project, Pershing County, Nevada, U.S.A.* for Pershing Gold Corporation ("Technical Report"), and with an effective date of May 24, 2018.

5. I have had involvement with this project having worked on three previous resource estimates for Pershing Gold Corporation on Relief Canyon as described in this report. I visited the Relief Canyon project on October 17 and 18, 2013, January 15, 2015, September 30, 2015, and October 13, 2016.

6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Dated July 6, 2018

"Paul Tietz"
Paul Tietz

CERTIFICATE OF THE AUTHOR

Carl Defilippi, C.E.M.

I, Carl E. Defilippi, M.Sc., C.E.M., do hereby certify that I am currently employed as a Project Manager for Kappes, Cassiday & Associates, 7950 Security Circle, Reno, Nevada 89506.

1. I graduated with a Bachelor of Science degree in Chemical Engineering from the University of Nevada in 1978 and a Master of Science degree in Metallurgical Engineering from the University of Nevada in 1981. I have practiced my profession continuously since 1981.
2. I am a Registered Member in good standing of the Society for Mining, Metallurgy and Exploration (775870RM).
3. I have read 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.
4. I am independent of Pershing Gold Corp. and its related companies, as independence is described in Section 1.5 of NI 43-101.
5. I am one of the authors of this Technical Report titled *Technical Report and Preliminary Feasibility Study for the Relief Canyon Gold Mine, Pershing County, Nevada, U.S.A*, prepared for Pershing Gold Corp., with an effective date of 24 May 2018. I am responsible for Sections 17, 18, 21.1.2 through 21.1.21, 21.1.23, 21.1.24, 21.2.2, 21.2.3, and all parts of the Summary, Interpretations and Conclusions, and Recommendations which pertain to those Sections. This technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
6. I visited the Relief Canyon project site on October 5, 2016.
7. At the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I hereby consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 6 July, 2018,

"Carl Defilippi"
Carl E. Defilippi, SME Registered Member

CERTIFICATE OF AUTHOR

Mark K. Jorgensen, Q.P. Metallurgy

I, Mark K. Jorgensen am a Principal Consultant for Jorgensen Engineering and Technical Services located at 1230 E Jamison Ave, Centennial, CO 80122.

1. I graduated with a Bachelor of Science degree in Chemical Engineering from the University of Nevada (Reno) in 1977. I have worked as a metallurgist for a total of 35 years since receiving my degree.

2. I am a Qualified Professional (Q.P.) in Metallurgy (Member Number 01202QP) with the Mining and Metallurgical Society of America. I have heap leach operating experience. I have designed heap leach test programs, heap leach operating plants and have helped construct several heap leach facilities.

3. 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. I am independent of Pershing Gold Corporation and its subsidiaries, applying all of the tests in section 1.5 of National Instrument 43-101.

4. I am responsible or jointly responsible for Sections 13, Mineral Processing and Metallurgical Testing, of this technical report titled *Technical Report and Preliminary Feasibility Study for the Relief Canyon Gold Project, Pershing County, Nevada, U.S.A.* for Pershing Gold Corporation ("Technical Report"), and with an effective date of May 24, 2018.

5. My involvement with this project started in December 2016. I visited the Relief Canyon project on March 1, 2017. I visited the laboratory that conducted the test work on February 28, 2017. where I reviewed their procedures and results.

6. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, those parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Dated this 6th day of July, 2018

"Mark K. Jorgensen"

Mark K. Jorgensen, Q.P. Metallurgy

APPENDIX A - MONTHLY PRODUCTION SCHEDULE AND OPERATING COSTS

Table A_1 Preproduction

Item	Units	Month -8	Month -7	Month -6	Month -5	Month -4	Month -3	Month -2	Month-1	Totals
Dozed Material	000's tons			24	67	40				131
Ore	000's tons							0	24	24
Waste Dump Material	000's tons			5	17			-	3	25
Alluvium	000's tons			19	50	40	47	127	100	383
Rock Waste	000's tons					104	383	362	688	1,537
Total Waste	000's tons			24	67	144	430	489	791	1,945
Total Material	000's tons			24	67	144	430	489	815	1,969
Crushed Material Summary										
Tons	000's tons									
Grade	oz Au/t									
Ounces	000's ounces									
Total Silver Produced	000's ounces									
Total Gold Produced	000's ounces									
Revenue Gold	$000's									
Refining and Transportation (Au)	$000's									
Royalties (2.15%)	$000's									
Net Sales	$000's									
Operating Cost										
Silver Credit	$000's									
Mining	$000's									
Load Crusher	$000's									
Processing (Lease)	$000's									
G & A	$000's									
Total Operating Cost	$000's									
Cost $/ton Ore										
Cost $/ounce Au recovered										
Net after Operating Costs	$000's									
Cumulative Cashflow	$000's									
Capital Cost	$000's	$790.2	$3,886.6	$4,731.9	$5,054.3	$4,250.5	$2,986.9	$3,232.3	$3,306.0	$28,238.8
Cash Flow with Capital	$000's	($790.2)	($3,886.6)	($4,731.9)	($5,054.3)	($4,250.5)	($2,986.9)	($3,232.3)	($3,306.0)	($28,238.8)
Cumulative Including Capital	$000's	($790.2)	($4,676.8)	($9,408.6)	($14,462.9)	($18,713.4)	($21,700.4)	($24,932.7)	($28,238.8)	

Table A_2 Year 1

Item	Units	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Year 1
Dozed Material	000's tons			35	29	50								114
Ore	000's tons	311	357	532	515	532	421	422	532	600	620	515	532	5,886
Waste Dump Material	000's tons	32	44	63	11	254	259	24	12	2	-	-	-	701
Alluvium	000's tons	52	15	0	0	3	4	146	323	162	488	431	631	2,255
Rock Waste	000's tons	760	1,035	672	783	669	1,879	1,851	897	560	1,219	1,232	1,018	12,573
Total Waste	000's tons	843	1,093	735	793	926	2,143	2,021	1,232	725	1,707	1,663	1,648	15,530
Total Material	000's tons	1,154	1,450	1,267	1,308	1,457	2,564	2,443	1,764	1,325	2,327	2,177	2,180	21,416
Crushed Material Summary														
Tons	000's tons	335	319	532	515	532	448	432	529	514	532	515	532	5,732
Grade	oz Au/t	0.018	0.013	0.014	0.020	0.017	0.012	0.023	0.015	0.012	0.014	0.014	0.013	
Ounces	000's ounces	6	4	7	10	9	5	10	8	6	7	7	7	88
Total Silver Produced	000's ounces		0.01	0.00	0.51	0.20	0.15	0.40	0.39	0.64	0.56	0.41	0.74	4.01
Total Gold Produced	000's ounces		4.64	3.64	6.08	8.41	7.82	4.85	8.14	6.89	5.67	6.29	6.22	68.66
Revenue Gold	$000's		$5,991.2	$4,698.7	$7,847.6	$10,850.4	$10,084.1	$6,256.9	$10,501.7	$8,890.7	$7,320.7	$8,112.8	$8,022.0	$88,576.8
Refining and Transportation (Au)	$000's		$46.4	$36.4	$60.8	$84.1	$78.2	$48.5	$81.4	$68.9	$56.7	$62.9	$62.2	$686.6
Royalties (2.15%)	$000's		$127.8	$100.2	$167.2	$231.4	$215.1	$133.3	$223.9	$189.5	$156.0	$172.9	$170.9	$1,888.3
Net Sales	$000's		$5,816.9	$4,562.0	$7,619.6	$10,534.9	$9,790.8	$6,075.1	$10,196.4	$8,632.4	$7,108.0	$7,877.0	$7,788.9	$86,001.9
Operating Cost														
Silver Credit	$000's		($0.2)	($0.0)	($8.2)	($3.2)	($2.4)	($6.3)	($6.2)	($10.2)	($8.9)	($6.6)	($11.9)	($64.2)
Mining	$000's	$2,489.6	$2,959.4	$2,671.7	$2,833.4	$2,911.5	$4,657.1	$4,572.5	$3,466.2	$2,745.8	$4,271.2	$4,042.2	$4,049.8	$41,670.4
Load Crusher	$000's	$123.8	$118.0	$196.7	$190.4	$196.7	$165.7	$160.0	$195.6	$190.4	$196.7	$190.4	$196.7	$2,121.0
Processing (Lease)	$000's	$920.0	$877.0	$1,461.6	$1,414.5	$1,461.6	$1,231.2	$1,188.7	$1,453.0	$1,414.5	$1,461.6	$1,414.5	$1,461.6	$15,760.0
G & A	$000's	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$2,450.2
Total Operating Cost	$000's	$3,737.6	$4,158.4	$4,534.2	$4,634.2	$4,770.8	$6,255.8	$6,119.0	$5,312.8	$4,544.7	$6,124.9	$5,844.6	$5,900.5	$61,937.4
Cost $/ton Ore		11.16	$13.03	$8.52	$9.00	$8.97	$13.96	$14.15	$10.05	$8.83	$11.51	$11.35	$11.09	$11.04
Cost $/ounce Au recovered			$894.92	$1,243.89	$761.22	$566.78	$799.90	$1,260.99	$652.18	$658.92	$1,078.67	$928.81	$948.29	$922.13
Net after Operating Costs	$000's	($3,737.6)	$1,658.6	$27.8	$2,985.4	$5,764.1	$3,535.0	($44.0)	$4,883.6	$4,087.7	$983.1	$2,032.4	$1,888.4	$24,064.5
Cumulative Cashflow	$000's	($3,737.6)	($2,079.0)	($2,051.2)	$934.2	$6,698.3	$10,233.3	$10,189.4	$15,073.0	$19,160.7	$20,143.7	$22,176.1	$24,064.5	
Capital Cost	$000's	$10,370.0	$207.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$10,577.0
Cash Flow with Capital	$000's	($14,107.6)	$1,451.6	$27.8	$2,985.4	$5,764.1	$3,535.0	($44.0)	$4,883.6	$4,087.7	$983.1	$2,032.4	$1,888.4	$13,487.5
Cumulative Including Capital	$000's	($42,346.4)	($40,894.8)	($40,867.0)	($37,881.6)	($32,117.5)	($28,582.4)	($28,626.4)	($23,742.8)	($19,655.1)	($18,672.0)	($16,639.7)	($14,751.3)	

Table A_3 Year 2

Item	Units	Month 13	Month 14	Month 15	Month 16	Month 17	Month 18	Month 19	Month 20	Month 21	Month 22	Month 23	Month 24	Year 2
Dozed Material	000's tons													
Ore	000's tons	373	364	473	498	498	532	497	532	515	532	665	455	5,932
Waste Dump Material	000's tons	-	-	-	-	-	-	-	-	-	-	-	-	
Alluvium	000's tons	717	493	373	721	486	-	-	4	72	92	124	64	3,147
Rock Waste	000's tons	1,615	1,761	1,858	1,399	1,720	1,746	1,161	886	1,006	648	806	2,186	16,792
Total Waste	000's tons	2,332	2,254	2,231	2,120	2,206	1,746	1,161	891	1,079	740	930	2,250	19,939
Total Material	000's tons	2,705	2,617	2,705	2,617	2,705	2,278	1,658	1,422	1,593	1,272	1,594	2,705	25,872
Crushed Material Summary														
Tons	000's tons	532	378	460	473	518	508	497	532	514	532	515	532	5,990
Grade	oz Au/t	0.013	0.018	0.014	0.013	0.017	0.017	0.019	0.020	0.019	0.013	0.022	0.016	
Ounces	000's ounces	7	7	6	6	9	9	10	11	10	7	11	9	101
Total Silver Produced	000's ounces	2.22	4.45	1.14	1.13	7.96	1.83	4.80	8.02	2.21	2.78	10.07	5.22	51.84
Total Gold Produced	000's ounces	5.80	5.51	5.88	5.21	5.50	7.14	7.62	7.81	8.71	8.67	6.31	9.11	83.27
Revenue Gold	$000's	$7,480.2	$7,105.7	$7,590.4	$6,716.5	$7,091.2	$9,209.3	$9,833.8	$10,078.5	$11,235.4	$11,187.2	$8,142.3	$11,748.3	$107,418.7
Refining and Transportation (Au)	$000's	$58.0	$55.1	$58.8	$52.1	$55.0	$71.4	$76.2	$78.1	$87.1	$86.7	$63.1	$91.1	$832.7
Royalties (2.15%)	$000's	$158.8	$150.1	$161.5	$142.9	$148.5	$195.8	$208.1	$212.2	$238.9	$237.7	$170.2	$248.8	$2,273.8
Net Sales	$000's	$7,263.4	$6,900.5	$7,370.0	$6,521.5	$6,887.7	$8,942.1	$9,549.4	$9,788.1	$10,909.4	$10,862.8	$7,909.0	$11,408.4	$104,312.2
Operating Cost														
Silver Credit	$000's	($35.5)	($71.2)	($18.2)	($18.1)	($127.3)	($29.3)	($76.8)	($128.4)	($35.4)	($44.5)	($161.1)	($83.5)	($829.4)
Mining	$000's	$4,719.5	$4,582.1	$4,704.6	$4,579.0	$4,705.9	$4,035.3	$3,085.2	$2,719.5	$2,995.6	$2,499.8	$2,988.1	$4,716.7	$46,331.3
Load Crusher	$000's	$196.7	$139.8	$170.3	$175.2	$191.5	$188.0	$184.0	$196.7	$190.4	$196.7	$190.4	$196.7	$2,216.3
Processing (Lease)	$000's	$1,449.2	$1,030.0	$1,254.3	$1,290.4	$1,410.9	$1,384.9	$1,355.7	$1,449.2	$1,402.5	$1,449.2	$1,402.5	$1,449.2	$16,328.0
G & A	$000's	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$204.2	$2,450.2
Total Operating Cost	$000's	$6,534.1	$5,884.8	$6,315.2	$6,230.7	$6,385.2	$5,783.1	$4,752.4	$4,441.2	$4,757.2	$4,305.4	$4,624.0	$6,483.4	$66,496.5
Cost $/ton Ore		$12.29	$15.57	$13.72	$13.16	$12.34	$11.38	$9.56	$8.35	$9.25	$8.10	$8.99	$12.19	$11.46
Cost $/ounce Au recovered		$1,126.83	$1,068.36	$1,073.28	$1,196.70	$1,161.57	$810.07	$623.42	$568.45	$546.20	$496.45	$732.58	$711.89	$824.32
Net after Operating Costs	$000's	$729.3	$1,015.7	$1,054.8	$290.8	$502.5	$3,159.0	$4,797.0	$5,346.9	$6,152.2	$6,557.4	$3,285.0	$4,925.1	$37,815.7
Cumulative Cashflow	$000's	$24,793.8	$25,809.6	$26,864.3	$27,155.2	$27,657.7	$30,816.6	$35,613.7	$40,960.6	$47,112.7	$53,670.2	$56,955.2	$61,880.2	
Capital Cost	$000's	($9,943.1)	$220.0	$220.0	$0.0	$0.0	$0.0	$0.0	$0.0	$993.0	$993.0	$1,125.1	$1,125.1	($5,267.0)
Cash Flow with Capital	$000's	$10,672.4	$795.7	$834.8	$290.8	$502.5	$3,159.0	$4,797.0	$5,346.9	$5,159.2	$5,564.4	$2,159.9	$3,800.0	$43,082.7
Cumulative Including Capital	$000's	($4,078.9)	($3,283.1)	($2,448.4)	($2,157.6)	($1,655.0)	$1,503.9	$6,301.0	$11,647.9	$16,807.1	$22,371.5	$24,531.4	$28,331.4	

Table A_4 Year 3

Item	Units	Month 25	Month 26	Month 27	Month 28	Month 29	Month 30	Month 31	Month 32	Month 33	Month 34	Month 35	Month 36	Year 3
Dozed Material	000's tons	141												141
Ore	000's tons	532	515	532	600	386	281	560	532	515	532	403	532	5,918
Waste Dump Material	000's tons	-	-	-	-	-	-	-	-	-	-	-	-	-
Alluvium	000's tons	-	-	-	-	-	0	8	14	49	196	294	227	788
Rock Waste	000's tons	1,236	868	1,042	1,640	2,319	2,423	1,855	559	1,559	1,392	1,920	1,429	18,241
Total Waste	000's tons	1,236	868	1,042	1,640	2,319	2,423	1,862	573	1,608	1,588	2,215	1,656	19,029
Total Material	000's tons	1,768	1,383	1,574	2,240	2,705	2,705	2,422	1,105	2,122	2,120	2,617	2,187	24,947
Crushed Material Summary														
Tons	000's tons	532	515	532	514	532	334	425	532	514	532	514	532	6,008
Grade	oz Au/t	0.018	0.015	0.017	0.020	0.015	0.021	0.022	0.018	0.016	0.023	0.028	0.017	0.019
Ounces	000's ounces	10	8	9	10	8	7	9	10	8	12	14	9	115
Total Silver Produced	000's ounces	7.63	3.78	2.49	4.46	9.43	11.98	6.90	8.69	4.77	6.30	15.40	11.75	93.58
Total Gold Produced	000's ounces	7.33	7.88	6.82	6.95	7.55	6.58	4.81	6.85	8.68	6.26	7.36	9.32	86.39
Revenue Gold	$000's	$9,455.3	$10,165.8	$8,792.4	$8,966.5	$9,742.2	$8,482.2	$6,202.2	$8,836.6	$11,199.8	$8,078.4	$9,498.0	$12,022.5	$111,441.9
Refining and Transportation (Au)	$000's	$73.3	$78.8	$68.2	$69.5	$75.5	$65.8	$48.1	$68.5	$86.8	$62.6	$73.6	$93.2	$863.9
Royalties (2.15%)	$000's	$199.1	$215.6	$186.7	$189.7	$204.6	$176.8	$129.9	$185.5	$237.3	$170.2	$197.3	$252.4	$2,345.2
Net Sales	$000's	$9,182.9	$9,871.4	$8,537.5	$8,707.2	$9,462.0	$8,239.6	$6,024.2	$8,582.6	$10,875.7	$7,845.6	$9,227.0	$11,676.9	$108,232.7
Operating Cost														
Silver Credit	$000's	($122.1)	($60.4)	($39.8)	($71.4)	($150.9)	($191.7)	($110.4)	($139.1)	($76.3)	($100.8)	($246.5)	($187.9)	($1,497.2)
Mining	$000's	$3,694.0	$2,930.8	$3,261.3	$4,413.4	$5,253.4	$5,268.4	$4,738.2	$2,443.3	$4,222.1	$4,218.4	$5,107.6	$4,338.1	$49,889.0
Load Crusher	$000's	$196.7	$190.4	$196.7	$190.4	$196.7	$123.7	$157.4	$196.7	$190.4	$196.7	$190.4	$196.7	$2,222.8
Processing (Lease)	$000's	$1,480.4	$1,432.6	$1,480.4	$1,432.6	$1,480.4	$931.0	$1,184.4	$1,480.4	$1,432.6	$1,480.4	$1,432.6	$1,480.4	$16,728.0
G & A	$000's	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$2,148.2
Total Operating Cost	$000's	$5,428.0	$4,672.4	$5,077.6	$6,144.0	$6,958.6	$6,310.4	$6,148.6	$4,160.3	$5,947.8	$5,973.8	$6,663.1	$6,006.2	$69,490.8
Cost $/ton Ore		$10.21	$9.08	$9.55	$11.94	$13.09	$18.87	$14.46	$7.83	$11.56	$11.24	$12.95	$11.30	$12.03
Cost $/ounce Au recovered		$740.55	$592.91	$744.97	$883.94	$921.42	$959.71	$1,278.84	$607.33	$685.07	$953.92	$904.98	$644.46	$836.96
Net after Operating Costs	$000's	$3,754.9	$5,199.0	$3,460.0	$2,563.2	$2,503.5	$1,929.2	-$124.4	$4,422.3	$4,927.9	$1,871.8	$2,563.9	$5,670.7	$38,741.9
Cumulative Cashflow	$000's	$65,635.1	$70,834.1	$74,294.1	$76,857.2	$79,360.7	$81,289.9	$81,165.5	$85,587.9	$90,515.8	$92,387.6	$94,951.5	$100,622.2	
Capital Cost	$000's	$1,675.9	$1,675.9	$1,675.9	$1,316.6	$1,426.4	$1,426.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$9,197.3
Cash Flow with Capital	$000's	$2,079.0	$3,523.1	$1,784.0	$1,246.5	$1,077.0	$502.8	($124.4)	$4,422.3	$4,927.9	$1,871.8	$2,563.9	$5,670.7	$29,544.7
Cumulative Including Capital	$000's	$30,410.4	$33,933.4	$35,717.5	$36,964.0	$38,041.0	$38,543.8	$38,419.5	$42,841.8	$47,769.7	$49,641.5	$52,205.4	$57,876.1	

Table A_5 Year 4

Item	Units	Month 37	Month 38	Month 39	Month 40	Month 41	Month 42	Month 43	Month 44	Month 45	Month 46	Month 47	Month 48	Year 4
Dozed Material	000's tons										147			147
Ore	000's tons	524	515	532	170	94	212	176	342	485	532	515	421	4,516
Waste Dump Material	000's tons	-	-	-	-	-	-	-	-	-	-	456	121	578
Alluvium	000's tons	333	296	470	865	559	237	122	2	-	-	0	20	2,905
Rock Waste	000's tons	1,848	1,642	1,368	1,583	2,052	2,256	2,145	2,361	2,132	2,116	1,647	2,142	23,292
Total Waste	000's tons	2,181	1,938	1,838	2,448	2,611	2,493	2,267	2,363	2,132	2,116	2,103	2,283	26,774
Total Material	000's tons	2,705	2,453	2,370	2,617	2,705	2,705	2,443	2,705	2,617	2,648	2,617	2,705	31,290
Crushed Material Summary														
Tons	000's tons	532	503	532	205	94	212	172	315	485	498	514	485	4,547
Grade	oz Au/t	0.027	0.027	0.026	0.018	0.012	0.021	0.037	0.025	0.030	0.019	0.018	0.020	0.024
Ounces	000's ounces	14	14	14	4	1	5	6	8	14	9	9	10	108
Total Silver Produced	000's ounces	8.42	8.58	8.45	9.72	8.82	7.48	7.81	7.40	9.10	9.80	8.44	7.21	101.22
Total Gold Produced	000's ounces	8.56	8.20	9.19	9.88	6.63	3.78	3.89	4.12	5.29	8.63	7.83	7.83	83.85
Revenue Gold	$000's	$11,044.5	$10,577.9	$11,857.6	$12,749.4	$8,549.7	$4,876.9	$5,023.8	$5,318.8	$6,825.5	$11,135.8	$10,097.3	$10,106.9	$108,164.1
Refining and Transportation (Au)	$000's	$85.6	$82.0	$91.9	$98.8	$66.3	$37.8	$38.9	$41.2	$52.9	$86.3	$78.3	$78.3	$838.5
Royalties (2.15%)	$000's	$232.7	$222.7	$250.1	$268.6	$179.4	$101.5	$104.5	$110.9	$142.5	$234.2	$212.5	$213.1	$2,272.7
Net Sales	$000's	$10,726.2	$10,273.2	$11,515.6	$12,381.9	$8,304.0	$4,737.6	$4,880.4	$5,166.6	$6,630.1	$10,815.2	$9,806.5	$9,815.4	$105,052.9
Operating Cost														
Silver Credit	$000's	($134.7)	($137.3)	($135.3)	($155.5)	($141.2)	($119.6)	($124.9)	($118.4)	($145.7)	($156.7)	($135.0)	($115.3)	($1,619.6)
Mining	$000's	$5,354.9	$4,914.8	$4,769.2	$5,304.9	$5,468.1	$5,431.1	$4,976.3	$5,392.1	$5,098.6	$5,139.9	$4,880.2	$5,212.1	$61,942.2
Load Crusher	$000's	$196.7	$186.1	$196.7	$75.9	$34.7	$78.5	$63.6	$116.6	$179.3	$184.2	$190.4	$179.5	$1,682.3
Processing (Lease)	$000's	$1,788.1	$1,691.7	$1,788.1	$690.1	$315.1	$714.0	$577.8	$1,060.3	$1,630.1	$1,674.2	$1,730.5	$1,632.0	$15,292.0
G & A	$000's	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$2,148.2
Total Operating Cost	$000's	$7,384.0	$6,834.4	$6,797.9	$6,094.4	$5,855.7	$6,283.0	$5,671.7	$6,629.8	$6,941.4	$7,020.5	$6,845.0	$7,087.4	$79,445.1
Cost $/ton Ore		$13.89	$13.59	$12.79	$29.70	$62.51	$29.60	$33.02	$21.03	$14.32	$14.10	$13.30	$14.61	$272.45
Cost $/ounce Au recovered		$862.45	$833.46	$739.55	$616.63	$883.52	$1,661.94	$1,456.36	$1,607.96	$1,311.92	$813.28	$874.50	$904.60	$12,566.17
Net after Operating Costs	$000's	$3,342.1	$3,438.9	$4,717.7	$6,287.6	$2,448.4	-$1,545.4	-$791.3	-$1,463.1	-$311.4	$3,794.7	$2,961.5	$2,728.0	$25,607.8
Cumulative Cashflow	$000's	$103,964.3	$107,403.2	$112,120.9	$118,408.5	$120,856.9	$119,311.4	$118,520.1	$117,057.0	$116,745.7	$120,540.4	$123,501.9	$126,230.0	
Capital Cost	$000's	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	
Cash Flow with Capital	$000's	$3,342.1	$3,438.9	$4,717.7	$6,287.6	$2,448.4	($1,545.4)	($791.3)	($1,463.1)	($311.4)	$3,794.7	$2,961.5	$2,728.0	$25,607.8
Cumulative Including Capital	$000's	$61,218.2	$64,657.1	$69,374.8	$75,662.4	$78,110.8	$76,565.3	$75,774.0	$74,310.9	$73,999.6	$77,794.3	$80,755.8	$83,483.9	

Table A_6 Year 5

Item	Units	Month 49	Month 50	Month 51	Month 52	Month 53	Month 54	Month 55	Month 56	Month 57	Month 58	Month 59	Month 60	Year 5
Dozed Material	000's tons		120											120
Ore	000's tons	436	416	343	346	349	216	261	262	362	532	515	532	4,568
Waste Dump Material	000's tons	284	108	218	138	35	65	82	220	53	11	0	-	1,214
Alluvium	000's tons	120	305	211	57	34	17	10	140	188	108	252	34	1,476
Rock Waste	000's tons	1,865	1,789	1,932	2,077	2,287	2,406	2,091	2,083	2,014	1,454	1,379	2,066	23,443
Total Waste	000's tons	2,269	2,201	2,362	2,271	2,356	2,488	2,182	2,443	2,256	1,573	1,631	2,099	26,132
Total Material	000's tons	2,705	2,617	2,705	2,617	2,705	2,705	2,443	2,705	2,617	2,105	2,146	2,631	30,701
Crushed Material Summary														
Tons	000's tons	418	433	339	346	350	219	262	259	352	516	514	532	4,541
Grade	oz Au/t	0.025	0.033	0.031	0.024	0.025	0.025	0.020	0.029	0.036	0.027	0.028	0.025	0.028
Ounces	000's ounces	10	14	11	8	9	5	5	8	13	14	15	13	125
Total Silver Produced	000's ounces	7.62	7.92	8.39	8.56	6.82	6.54	6.72	5.68	5.40	6.50	6.75	7.28	84.18
Total Gold Produced	000's ounces	7.37	7.16	8.89	8.56	6.67	6.68	5.28	4.84	5.78	8.93	11.88	11.47	93.52
Revenue Gold	$000's	$9,510.3	$9,231.8	$11,470.6	$11,048.8	$8,598.0	$8,621.4	$6,813.9	$6,248.3	$7,460.0	$11,515.8	$15,326.1	$14,794.7	$120,639.7
Refining and Transportation (Au)	$000's	$73.7	$71.6	$88.9	$85.6	$66.7	$66.8	$52.8	$48.4	$57.8	$89.3	$118.8	$114.7	$935.2
Royalties (2.15%)	$000's	$200.3	$194.2	$241.8	$232.8	$181.1	$181.7	$143.1	$131.3	$157.3	$243.4	$324.6	$313.1	$2,544.7
Net Sales	$000's	$9,236.3	$8,966.0	$11,139.8	$10,730.4	$8,350.3	$8,372.9	$6,618.0	$6,068.5	$7,244.9	$11,183.1	$14,882.7	$14,366.9	$117,159.8
Operating Cost														
Silver Credit	$000's	($121.9)	($126.7)	($134.2)	($136.9)	($109.1)	($104.7)	($107.5)	($90.9)	($86.4)	($104.0)	($108.1)	($116.5)	($1,346.8)
Mining	$000's	$5,527.2	$5,455.2	$5,595.2	$5,459.8	$5,673.0	$5,706.9	$5,179.7	$5,621.4	$5,497.0	$4,466.2	$4,561.2	$5,482.2	$64,225.1
Load Crusher	$000's	$154.8	$160.2	$125.6	$128.2	$129.4	$80.9	$97.0	$95.9	$130.1	$191.0	$190.4	$196.7	$1,680.2
Processing (Lease)	$000's	$1,139.1	$1,178.9	$924.0	$943.2	$952.2	$595.4	$714.1	$705.7	$957.4	$1,405.7	$1,400.9	$1,447.6	$12,364.0
G & A	$000's	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$2,148.2
Total Operating Cost	$000's	$6,878.2	$6,846.6	$6,689.5	$6,573.3	$6,824.5	$6,457.5	$6,062.3	$6,511.1	$6,677.2	$6,138.0	$6,223.4	$7,189.0	$79,070.7
Cost $/ton Ore		$16.44	$15.81	$19.71	$18.98	$19.52	$29.53	$23.12	$25.12	$18.99	$11.89	$12.10	$13.52	$13.74
Cost $/ounce Au recovered		$932.97	$956.70	$752.31	$767.47	$1,023.91	$966.23	$1,147.71	$1,344.26	$1,154.64	$687.58	$523.82	$626.83	$636.82
Net after Operating Costs	$000's	$2,358.1	$2,119.4	$4,450.3	$4,157.1	$1,525.8	$1,915.3	$555.7	($442.6)	$567.7	$5,045.0	$8,659.3	$7,178.0	$38,089.2
Cumulative Cashflow	$000's	$128,588.1	$130,707.5	$135,157.9	$139,314.9	$140,840.7	$142,756.0	$143,311.8	$142,869.2	$143,436.8	$148,481.9	$157,141.2	$164,319.1	
Capital Cost	$000's	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	
Cash Flow with Capital	$000's	$2,358.1	$2,119.4	$4,450.3	$4,157.1	$1,525.8	$1,915.3	$555.7	($442.6)	$567.7	$5,045.0	$8,659.3	$7,178.0	$38,089.2
Cumulative Including Capital	$000's	$85,842.0	$87,961.4	$92,411.8	$96,568.8	$98,094.6	$100,009.9	$100,565.7	$100,123.1	$100,690.7	$105,735.8	$114,395.1	$121,573.0	

Table A_7 Year 6

Item	Units	Month 61	Month 62	Month 63	Month 64	Month 65	Month 66	Month 67	Month 68	Month 69	Month 70	Month 71	Month 72	Year 6
Dozed Material	000's tons													
Ore	000's tons	499	515	532	515	532	532	269						3,392
Waste Dump Material	000's tons	-	-	-	-	-	-	-						-
Alluvium	000's tons	7	-	-	-	-	-	-						7
Rock Waste	000's tons	2,199	2,069	1,985	610	414	240	101						7,618
Total Waste	000's tons	2,206	2,069	1,985	610	414	240	101						7,625
Total Material	000's tons	2,705	2,584	2,516	1,125	946	772	370						11,017
Crushed Material Summary														
Tons	000's tons	526	484	532	514	532	532	300						3,419
Grade	oz Au/t	0.028	0.029	0.027	0.031	0.025	0.026	0.025						
Ounces	000's ounces	15	14	14	16	13	14	8						94
Total Silver Produced	000's ounces	7.94	11.14	11.74	11.76	8.69	7.05	6.60	5.34	4.53	3.55	3.72	3.18	85.25
Total Gold Produced	000's ounces	9.63	10.01	11.05	11.55	15.28	12.54	12.69	7.73	1.41	0.72	0.39	0.18	93.17
Revenue Gold	$000's	$12,419.4	$12,912.1	$14,254.4	$14,904.7	$19,710.6	$16,174.0	$16,368.7	$9,971.1	$1,814.2	$923.6	$508.5	$233.2	$120,194.6
Refining and Transportation (Au)	$000's	$96.3	$100.1	$110.5	$115.5	$152.8	$125.4	$126.9	$77.3	$14.1	$7.2	$3.9	$1.8	$931.7
Royalties (2.15%)	$000's	$262.2	$271.6	$300.1	$313.9	$417.5	$342.6	$346.9	$210.9	$37.1	$18.5	$9.6	$3.9	$2,534.8
Net Sales	$000's	$12,060.9	$12,540.4	$13,843.9	$14,475.2	$19,140.3	$15,706.0	$15,894.8	$9,683.0	$1,762.9	$898.0	$495.0	$227.6	$116,728.0
Operating Cost														
Silver Credit	$000's	($127.0)	($178.2)	($187.9)	($188.2)	($139.0)	($112.9)	($105.6)	($85.5)	($72.5)	($56.8)	($59.6)	($50.9)	($1,364.1)
Mining	$000's	$5,862.8	$5,614.2	$5,470.8	$2,471.5	$2,058.3	$1,707.5	$850.4						$24,035.4
Load Crusher	$000's	$194.8	$179.1	$196.7	$190.4	$196.7	$196.7	$110.9						$1,265.2
Processing (Lease)	$000's	$1,324.9	$1,218.0	$1,338.1	$1,295.0	$1,338.1	$1,338.1	$754.1	$223.3	$223.3	$223.3	$223.3	$223.3	$9,723.0
G & A	$000's	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$179.0	$84.5	$84.5	$84.5	$84.5	$84.5	$1,675.6
Total Operating Cost	$000's	$7,434.5	$7,012.1	$6,996.8	$3,947.7	$3,633.2	$3,308.5	$1,788.8	$222.3	$235.3	$251.0	$248.2	$256.9	$35,335.1
Cost $/ton Ore		$14.12	$14.49	$13.16	$7.67	$6.83	$6.22	$5.97						$10.33
Cost $/ounce Au recovered		$772.05	$700.39	$633.04	$341.53	$237.67	$263.74	$140.89	$28.76	$167.31	$350.53	$629.78	$1,420.70	$379.24
Net after Operating Costs	$000's	$4,626.4	$5,528.3	$6,847.1	$10,527.5	$15,507.2	$12,397.6	$14,106.1	$9,460.7	$1,527.7	$647.0	$246.7	($29.3)	$81,392.9
Cumulative Cashflow	$000's	$168,945.5	$174,473.8	$181,320.9	$191,848.4	$207,355.6	$219,753.2	$233,859.3	$243,319.9	$244,847.6	$245,494.6	$245,741.3	$245,712.0	
Capital Cost	$000's	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	$83.3	($3,916.7)	($3,000.0)
Cash Flow with Capital	$000's	$4,543.0	$5,445.0	$6,763.8	$10,444.2	$15,423.8	$12,314.2	$14,022.7	$9,377.3	$1,444.3	$563.7	$163.4	$3,887.3	$84,392.9
Cumulative Including Capital	$000's	$126,116.1	$131,561.1	$138,324.9	$148,769.0	$164,192.8	$176,507.1	$190,529.8	$199,907.2	$201,351.5	$201,915.2	$202,078.6	$205,965.9	

Table A_8 Year 7

Item	Units	Month 73	Month 74	Month 75	Month 76	Month 77	Month 78	Month 79	Month 80	Month 81	Year 7
Dozed Material	000's tons										
Ore	000's tons										
Waste Dump Material	000's tons										
Alluvium	000's tons										
Rock Waste	000's tons										
Total Waste	000's tons										
Total Material	000's tons										
Crushed Material Summary											
Tons	000's tons										
Grade	oz Au/t										
Ounces	000's ounces										
Total Silver Produced	000's ounces	3.04	2.35	1.81	1.42	1.01	0.89	0.51	0.26	0.00	11.30
Total Gold Produced	000's ounces	0.14	0.12	0.10	0.09	0.08	0.07	0.04	0.02	0.00	0.65
Revenue Gold	$000's	$186.5	$150.3	$123.1	$114.4	$100.6	$84.3	$50.1	$28.0	$0.3	$837.8
Refining and Transportation (Au)	$000's	$1.4	$1.2	$1.0	$0.9	$0.8	$0.7	$0.4	$0.2	$0.0	$6.5
Royalties (2.15%)	$000's	$2.9	$2.4	$2.0	$2.0	$1.8	$1.5	$0.9	$0.5	$0.0	$14.0
Net Sales	$000's	$182.2	$146.8	$120.2	$111.6	$98.0	$82.2	$48.8	$27.3	$0.3	$817.3
Operating Cost											
Silver Credit	$000's	($48.6)	($37.7)	($28.9)	($22.8)	($16.1)	($14.3)	($8.2)	($4.2)	($0.0)	($180.8)
Mining	$000's										
Load Crusher	$000's										
Processing (Lease)	$000's										
G & A	$000's										
Total Operating Cost	$000's	(48.6)	(37.7)	(28.9)	(22.8)	(16.1)	(14.3)	(8.2)	(4.2)	(0.0)	(180.8)
Cost $/ton Ore											
Cost $/ounce Au recovered											
Net after Operating Costs	$000's	$230.8	$184.4	$149.1	$134.3	$114.1	$96.5	$57.1	$31.5	$0.4	$998.1
Cumulative Cashflow	$000's	$245,942.8	$246,127.2	$246,276.3	$246,410.6	$246,524.7	$246,621.2	$246,678.3	$246,709.8	$246,710.1	
Capital Cost	$000's	$333.3	$333.3	$333.3	$333.3	$333.3	$333.3	$333.3	$333.3	$333.3	$3,000.0
Cash Flow with Capital	$000's	($102.6)	($148.9)	($184.3)	($199.0)	($219.2)	($236.9)	($276.3)	($301.8)	($333.0)	($2,001.9)
Cumulative Including Capital	$000's	$205,863.3	$205,714.4	$205,530.2	$205,331.2	$205,112.0	$204,875.1	$204,598.8	$204,297.0	$203,964.0	