Preliminary Economic Assessment – Technical Report

Imperial Gold Project

California, USA

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Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for KORE Mining Ltd. ("KORE Mining") by Global Resource Engineering ("GRE") based in part on the Mineral Resource Estimate and Technical Report titled "Technical Report for the Imperial Gold Project, California, USA" dated December 30, 2019, by SRK Consulting (Canada) Inc. ("SRK"). The geology and Mineral Resource Estimation portions of the SRK report is included here in its entirety for completeness. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in GRE's and SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by KORE Mining subject to the terms and conditions of its contract with GRE and SRK and relevant securities legislation. The contract permits KORE to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

KORE Mining Ltd (“KORE Mining”, “KORE” or the “Company”) commissioned Global Resource Engineering (“GRE”) to prepare this NI43-101 compliant Preliminary Economic Assessment (PEA) of the Imperial Gold Project, located in Imperial County California, USA. GRE based its work on a Mineral Resource Estimate and Technical Report prepared by SRK Consulting (Canada) Inc. (“SRK”), titled “Technical Report for the Imperial Gold Project, California, USA” with an effective date of December 30, 2019 (“SRK (2019) Mineral Resource Estimate and Technical Report”).

The SRK mineral resource model was prepared following the guidelines of the Canadian Securities Administrators National Instrument (“NI”) 43-101 and Form 43-101F1. The 2019 SRK technical report was required by securities law to support the first-time disclosure of mineral resources by the KORE Mining.

This report is based on the SRK (2019) Mineral Resource Estimate and Technical Report.

The GRE Qualified Persons (“QP’s”) site visit occurred on January 9-10, 2020. SRK QP’s visited the site on November 26, 2019 and February 9-10, 2012. Both groups collected information while on site and both used additional information provided by KORE Mining.

The mineral resource estimate and preliminary economic analysis reported herein was prepared in conformity with generally accepted Canadian Institute of Mining’s (“CIM”) “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (as adopted by the CIM Council on November 29, 2019). The work in 2012 was undertaken on a database last updated in 1996 and no additional exploration data or property activity has occurred since that time.

All monetary units herein are in United States dollars (“US$” or “$”) unless otherwise specified.

1.2 Property Description, Location, Access, and Physiography

The Imperial Project is located in Imperial County in the desert region of southeastern California, USA Figure 1-1. It is located along the Indian Pass Road approximately 26 road-miles northwest of the city of Yuma, Arizona, and is approximately 45 miles east-northeast of El Centro, California. The project is located on public land administered by the Bureau of Land Management (“BLM”).

The operating Mesquite Mine and the closed Picacho Mine are located roughly 10 miles to the west and east, respectively, of the property. The closed American Girl Mine is about 8 miles south of the project. The Imperial project which is the subject of this assessment is owned by Imperial USA Corp. (“IUC”), formerly named, Glamis Imperial Corporation.

As per information supplied by KORE Mining and a Title Report supplied by Mitchell Chadwick LLP, the project property consists of 654 unpatented mining claims. The total area of all the claims is approximately 5,721 acres.

In March 2017, KORE Mining acquired Imperial USA Corp. from Newmont Goldcorp (formerly Goldcorp) (the "Vendor") for an initial payment of US$150,000, and future payments of US$1,000,000 payable upon the announcement of a revised Preliminary Economic Assessment (PEA) or similar report, and US$1,000,000 payable 30 days after the date that gold is poured from ore mined from the related properties. In addition, the Company has committed to incur US$5 million in exploration and evaluation expenditures (which includes permitting and development activities) on the Imperial Project on or before March 2022, the fifth anniversary of the date of the agreement. In the event the Company does not incur these expenditures within this timeframe, the Company must then pay US$1,000,000 to the Vendor.

Figure 1-1: Location of the Imperial Gold Project

The Imperial Property can be maintained in good standing by:

- Firstly, paying an annual claim maintenance fee to the Bureau of Land Management ("BLM") for each claim which is due prior to the end of the fiscal tenure year which starts and ends at noon on September 1st of the current year, and
- Recording an affidavit that the maintenance fees have been paid with the local County Recorder. Failure to comply will result in forfeiture of the claims.

Both of these requirements have been met for the 2019 assessment year, and all Claims are marked as active on BLM's Land & Mineral Legacy Rehost 2000 System.

After review of the required permits and authorizations as well as environmental considerations, the authors of this report conclude that the owner of the validated mineral claims (i.e., the claims within the area defined by the Imperial Gold Project) has the right to advance its exploration and mining interests subject to obtaining permits to carry out the activities as authorized by the appropriate government agencies.

1.3 History

Due to the extent of the alluvial cover on the Imperial Project, exploration has historically consisted primarily of drilling. Initial exploration strategies focused on wide-spaced definition drilling of buried gravity and structural anomalies. Mineralized zones were projected down dip and followed with additional drilling to depths exceeding 1,000 feet. Later exploration strategies focused on the development of the entire deposit and tested down-dip areas for economic mining limits. To date, 349 exploration boreholes totaling 195,047 ft have delineated the mineralized zones defined in the geology and mineral resource modeling.

Mineral exploration on the Imperial Gold project was undertaken between 1980 and 1996 by the following exploration entities:

- Gold Fields Mining Corporation (1980-1986)
- Exploration by Imperial County Joint Venture (1987-1993)
- Exploration by Glamis Gold (1994-1996)

1.4 Geology and Mineralization

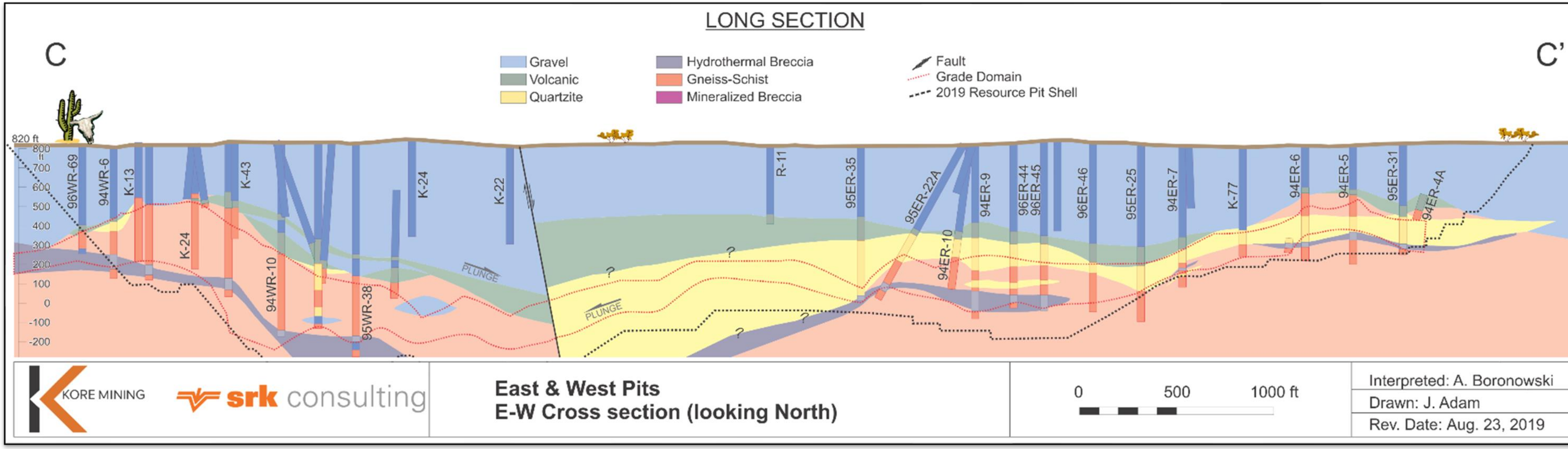
The Imperial Gold Project is located on the southern flank of the Chocolate Mountains, structurally aligned and equidistant between the Picacho and Mesquite gold deposits. The project area is underlain by a sequence of Jurassic age gneisses and schists. The overlaying stratigraphy is made up of fanglomerates and alluvium that vary in thickness from 10 to 700 feet (“ft”) and cover 95% of the project area.

Gold mineralization occurs primarily within haematitic and limonitic altered breccias, microfractures, and gouge zones developed in the host biotite gneiss and sericite gneiss units. Minor quartz veining, very-fine grained pyrite pseudomorphs and silicified zones are also common.

The Imperial gold deposit is believed to represent epithermal gold mineralization related to Tertiary-age low angle detachment faults and associated extensional faults. The epithermal gold mineralization is structurally controlled and transitional between low and high-sulphidation systems.

A cross section of the deposit is shown in Figure 1-2.

Figure 1-2: Conceptual East-West Long Section* Across the Imperial Gold Deposit (Looking North)



Section line C-C[I] is indicated on a plan in Appendix B of this report

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Exploration and Drilling

Exploration on the project occurred between 1982 and 1996 and was comprised mainly of reverse circulation and core drilling. A total of 349 reverse circulation ("RC") drill holes totaling 195,047 ft, and nine core holes totaling approximately 4,900 ft, were drilled.

Table 1-1 summarizes the drilling activities by year, drilling type and operator.

Table 1-1: Summary of Drilling on the Imperial Gold Project

Year	Operator	Type	No. Holes	Total (ft)
1982-1986	Gold Fields	RC	53	27,880
1987-1992	Imperial County Joint Venture	RC	169	71,539
1994	Glamis Gold	RC	45	34,565
1995	Glamis Gold	RC	32	29,890
1994-1995	Glamis Gold	Core[a]	9	4,913
1996[b]	Glamis Gold	RC	41	26,260
Total	**All**	**All**	**349**	**195,047**

a. Core drilling was dedicated to metallurgical testwork and was not used in the previous or current resource estimates.
b. Drilling in 1996 was not utilized in the previous mineral resource estimate found in the Western States Engineering 1996 FS but was included in this study.

1.5 Sample Preparation, Analyses, Security and Data Verifications

Sample preparation, analyses and security procedures for historical samples taken by the previous operators, Gold Fields and Glamis Gold, are not specifically documented and therefore difficult to review. The authors of this report understand that samples were assayed for gold at the Mesquite and Picacho mine laboratories. The preparation and assaying technique were not documented. Assay records are preserved on paper logs, level maps, and sections.

The majority of the recently completed gold analysis was conducted by American Assay Laboratory ("AAL") and Chemex Labs Inc. ("Chemex") at undisclosed locations. Chemex is accredited to ISO/IEC standards to provide complete assurance regarding quality performance in sample preparation and analysis. AAL is not accredited.

Verification sampling completed by Delta was conducted at ALS Canada Ltd. (ALS Minerals) in North Vancouver, British Columbia in order to verify selected historically sampled intervals. The management system of the ALS Group of laboratories is accredited ISO 9001:2000 by QMI Management Systems Registration.

In the opinion of the qualified person of this report, the sample preparation, security, and analytical procedures used by previous operators is poorly documented and therefore difficult to assess. The known analytical quality control measures implemented on the Imperial Project is limited to field duplicates and umpire check assays in 1991-1992 and umpire check assays in 1994-1996. Other checks on the data were likely performed by each operator but are not known to the qualified person.

1.6 Mineral Processing and Metallurgical Testing

A review of the all historical test work by GRE, indicated that the Imperial Gold Project material should be amenable to heap leaching. Run-of-mine ("ROM") heap leaching has been utilized with an estimated average gold recovery of 73% using a primary extraction cycle of 90 days and a total cycle of 270 days.

1.7 Environmental, Permitting and Social Impact

Environmental evaluations and reviews completed by previous project owner, Glamis, may still be relevant and may continue to apply, along with confirmatory baseline studies to establish their validity.

Permitting requirements remain similar to the previous attempts to produce gold at the Imperial Project. Land use and mineral resource are the same, and this report focuses on a changed mine plan that complies with all California and US regulations. This means the permitting will require an update to the Plan of Operations and the Reclamation plan, as well as updates to the baseline environmental studies performed in the late 1990's by the previous operators of the Project.

Although mining for metallic ore in California has not been legally prohibited, the restrictions placed on the development of new mines has created a more difficult regulatory environment in which to design and authorize a new operation. The key environmental, permitting, and social considerations for the future development of the Imperial Gold Project include, but are not necessarily limited to:

- The environmental baseline studies conducted during the previous permitting process in the late 1990's will need to be updated for all environmental resources.
- The use of spent heap leach pad material as pit backfill was previously proposed, but there has been no modelling conducted to evaluate potential impacts to groundwater. Initial analysis of spent heap material is generally good and demonstrates there is a low risk of groundwater contamination from contact with this material. However, additional modelling is required to more definitively assess this issue.
- While the California's Surface Mining and Reclamation Act (SMARA) backfilling regulations have a profound effect on mine planning for the Imperial Gold Project, these regulations from time to time come under review and may be revised in the future. Any project development will need to incorporate design elements that are consistent with and comply with the current regulatory environment in California.
- Proactive social and community engagement will be essential to any future mine development, especially with respect to local tribal engagement.

1.8 Mineral Resource and Mineral Reserve Estimates

The Mineral Resource Statement presented herein represents the second mineral resource evaluation prepared for the Imperial Project in accordance with the Canadian Securities Administrators NI 43-101. As no additional data has been generated for the project since 2012, the mineral resource model described in this report is unchanged from that generated by SRK (2012) but has been re-stated to consider current 2019 economics.

No mineral reserve has been estimated for the Imperial Gold Project.

The mineral resource model prepared by the qualified person considers 349 RC boreholes drilled by various operators during the period of 1987-1996. The resource estimate was completed under the supervision of Glen Cole, PGeo. (APGO #1416), who is an independent qualified person as this term is defined in NI 43-101. The effective date of this resource estimate is December 30, 2019.

Gold grades were estimated by ordinary kriging constrained within modeled grade zone domain solids. Gold grades were estimated within each domain separately using capped composites from within that domain and applying appropriate search parameters.

The qualified person considers that the blocks located within the conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a mineral resource. Mineral resources are reported at a Cut-Off-Grade of 0.003 oz/t Au and include all resource blocks above cut-off inside the conceptual pit shell. The COG was based on a gold price of $1,500/oz gold and a gold metallurgical recovery of 80%.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration

The Mineral Resource Statement for the Imperial Gold Project is presented in Table 1-2.

Table 1-2: Mineral Resource Statement*, Imperial Gold project, SRK 2019

Classification	Quantity (‘000 tons)	Grade Gold (oz/t)	Contained Gold (‘000 ounces)
Indicated			
Grade Zone (Domains 100, 120)	50,379	0.0174	877
Total Indicated	**50,379**	**0.0174**	**877**
Inferred			
Grade Zone (Domains 100, 110, 120)	79,869	0.0156	1,245
Gravel with grade (Domain 200)	10,557	0.0041	43
Bedrock with grade (Domain 300)	9,748	0.0050	48
Total Inferred	**100,174**	**0.0133**	**1,336**

*Reported at a cut-off grade of 0.003 oz/t Au using a price of $1,500 /oz Au inside a conceptual pit shell optimized using mining operating costs of $1.40 per ton, metallurgical and process recovery of 80%, combined processing and G&A costs of $2.30 per ton, $0.50 per ton of sustaining capital and overall pit slope of 45 degrees.
All figures rounded to reflect the relative accuracy of the estimates.

1.9 Mining Methods

The Imperial Mine deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the estimated grade of the resource model and Whittle pit shell analysis. The results are summarized in Table 1-3.

Table 1-3: Mine Plan Quantities

Pit	Indicated Material			Inferred Material			Waste Tons	Stripping Ratio
	Tons	Au (opt)	Au (tr oz)	Tons	Au (opt)	Au (tr oz)		
West P1	13,930,919	0.013	183,460	2,563,509	0.015	37,555	22,194,139	1.3
West P2	4,417,325	0.014	62,996	14,002,624	0.016	219,805	40,160,246	2.2
East P1	6,153,719	0.018	111,596	1,781,270	0.016	27,897	39,544,618	5.0
East P2	16,223,124	0.021	348,355	3,837,004	0.017	65,585	40,637,029	2.0
East P3	3,081,872	0.025	75,974	8,120,222	0.018	147,923	43,488,065	3.9
East P4	5,614,028	0.018	101,009	7,657,766	0.020	149,351	62,721,500	4.7
Singer P1	0	-	0	2,741,791	0.015	41,600	5,536,997	2.0
Singer P2	0	-	0	1,361,528	0.016	22,262	1,659,162	1.2
Totals	**49,420,987**	**0.018**	**883,390**	**42,065,714**	**0.017**	**711,978**	**255,941,756**	**2.8**

1.10 Recovery Methods

The Imperial project would employ open pit mining with a conventional heap leach system on a 365 day per year 24 hour per day basis. The heap leach will utilize run-of mine (ROM) material. The ROM is delivered directly from the open pit to the heap via the mine haul trucks. The trucks will pass under a silo that will deposit a measured amount of lime on the load for pH control.

The heap leach would consist of a suitable area lined with a containment system. The material lifts are targeted at 32 ft in height with a total heap height of 328 ft. Once a suitable area has been stacked (cell), the cell would be irrigated with dilute cyanide solution. The solution leaches gold from the heap materials and is transported to the gold recovery circuit as pregnant leach solution (PLS) and recovered in the Adsorption-Desorption-Recovery plant (ADR). The ADR plant consists of a series of columns containing activated carbon (CIC) that adsorb the gold. The gold is recovered by a desorption system and recovered as doré.

1.11 Project Infrastructure

A limited amount of infrastructure is currently available on site. Power, water, and all other systems necessary for a mining and processing operation will be required.

Sufficient water appears to be available on the Imperial property. One ground water well currently exists, and a second well is planned for this project. Groundwater supplies would be developed to meet the project water requirements.

Power is available near the mine site from the grid through a 161kV power line. There are no electrical substations at the site. Local labor for mining is available.

1.12 Market Studies and Contracts

The primary metal of economic interest for the Imperial project is gold. Gold has a readily available market for sale in the form of gold doré or gold concentrates. Figure 19-1 presents the gold market London PM fixed pricing through April 14, 2020. The selected Gold price for the PEA is $1,450/oz which represents the 3-year trailing average, $1,325/oz weighted by 60% and $1,620/oz projected gold price weighted by 40%, these were the values at the time of the preparation of this Technical Report.

1.13 Capital and Operating Costs

A breakdown of capital and operating costs is shown in Table 1-4.

Table 1-4: Imperial Capital Costs

Initial and Sustaining Capital Costs ($ millions)	
Mining & mine Infrastructure	$35.31
Heap leach pads and plant	$47.00
Infrastructure & G&A	$15.68
Working capital	$7.49
Contingency (25%)	$23.65
Total Pre-Production Capital	**$129.13**
Pre-production mining	$14.34
Total Pre-Production Cost	**$143.47**
Sustaining capital	$60.54
Closure, incl. Backfill and reclamation	$147.68

The average operating cash costs, once sustained positive cash flow has been achieved, are shown in Table 1-5.

Table 1-5: Imperial Operating Costs

Operating Costs	Unit	Cost
Mining costs (owner)	$/st mined	$1.45
Mining costs	$/st processed	$5.51
Processing costs	$/st processed	$1.85
G&A costs	$/st processed	$0.74
Total site operating costs	$/st processed	$8.11

1.14 Economic Analysis

The results of the economic analysis are summarized in Table 1-6.

Table 1-6: Summary of Imperial Economic Results

Economics	Unit	Pre-Tax	Post-Tax
Net present value (NPV 5%)	$ millions	$438	$343
Net present value (NPV 5%) see note	C$ millions	$584	$458
Internal rate of return (IRR)	%	52%	44%
Payback (undiscounted)	Years	2.3	2.7
LOM average annual cash flow	$ millions	$105	$90
LOM cumulative cash flow	$ millions	$697	$580
Cumulative cash flow (undiscounted)		$438	
Gold price assumption	per ounce	$1,450	

Note 1: Canadian dollar to US dollar exchange rate assumed to be 1.33:1

1.15 Interpretations and Conclusions

A total of 349 boreholes, of which 344 are located within resource estimation area (comprising a total of 190,047 ft of reverse circulation drilling) have been drilled by various operators (including Gold Fields, Glamis Gold, and other historical operators) on the Imperial Gold Project from 1982 to 1996.

No exploration activity has been undertaken on the project since 1996, with minimal documentation of the historical exploration activity available to review. Although a significant amount of drilling has occurred on the property to delineate significant gold mineralization, minimal evidence of exploration procedures or protocols are available to confirm that best practice exploration methodologies were adopted. Additionally, with most of the drilling having been reverse circulation, detailed geological reviews of drill core have not been possible to define a more detailed geological / structural model for the property or to generate a better understanding of the spatial controls of gold mineralization.

In the opinion of the qualified person, the sample preparation, security, and analytical procedures used to generate exploration data upon which the resource model is based is poorly documented and therefore difficult to assess. The known analytical quality control measures implemented on the Imperial Gold Project is limited to field duplicates and umpire check assays in 1991-1992 and umpire check assays in 1994-1996. Other checks on the data were likely performed by each operator but are not known to the qualified person.

Despite the uncertainty outlined above, limited data verification measures undertaken by Delta Gold in 2012 and the authors of this report suggest that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support the evaluation and classification of mineral resources in accordance with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices and CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2019).

The qualified person is satisfied that the geological modelling honors the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The mineral resource model is largely based on geological knowledge derived from boreholes drilled sections spaced at approximately 150 ft apart in the east and west portions of the deposit and over 250 ft in the rest of the deposit.

The geological information gathered from the RC drilling is sufficiently dense to allow modelling with reasonable confidence of the gold mineralization boundaries (domains 100, 110, and 120), as well as the base of gravel contact, which delimited the unconstrained domains (domains 200 and 300). However, uncertainty remains in the structural framework of the deposit. Normal faults are believed to displace the lithological units including gold mineralization but have not been modelled. The south dipping domain 110 is potentially the result of faulting. The geological continuity can only be inferred at the current drill spacing within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

The mineral resources classification was also reviewed using a combination of tools including: confidence in the geological interpretation, variography results, search ellipse volume, and kriging variance.

Generally, for mineralization exhibiting good geological continuity investigated at an adequate spacing and displaying low structural complexity, the qualified person considers that blocks estimated according to defined parameters could be classified into the Indicated and Inferred categories within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

The mineral resource model documented in this report indicates that the Imperial Gold project hosts significant gold mineralization, but additional exploration would need to be undertaken in areas of lower drilling density to upgrade the Inferred portions of the mineral resource model to be suitable for advanced mining study applications.

The mine plan is based on 33,000 tons per day of mineralized material production. The pits were divided into 6 phases, plus two satellite pits. Initial phases of both the east and west pits were designed as low strip-ratio volumes in order to lower the initial capital cost. The plan produces 91.5 million tons of mineralized material at an average grade of 0.017 oz/ton or 0.60 g/tonne in an 8-year mine life. Stripping requirements include a life of mine total of 255.9 million waste tons, 208.5 million tons of which are alluvium. Waste management for the mine includes 3 waste dumps and concurrent backfilling. At the end of production, the heap leach pad will be rinsed and neutralized. After which, it will be transported into the remaining open pit along with 2 dumps and a portion of the main dump. 94.7 million tons of aggregate material remain on the surface after the pits have been backfilled and could be used as an aggregate source either after closure or during operations.

Operating cost in production years for the Imperial project amount to $1.45 per short ton mining cost, $1.85 per short ton processed processing cost, and $0.74 per short ton processed G&A cost. Total capital cost for the project are $72.3 million mine, $47.0 million plant, $0.77 million G&A, $11.7 million infrastructure, $17.2 million sustaining, $27.8 million reclamation, and $37.3 million contingency for a total of $214.1 million.

The PEA used a base gold price of $1,450/oz with an estimated overall recovery of 73% which resulted in an After-Tax Net Present Value at 5% of $343 million and an Internal Rate of Return of 44%. This report includes inferred mineral resources. Inferred resources are based on limited information, and as such are not suitable to be categorized as mineral reserves.

1.16 Recommendations

The geological setting and character of the gold mineralization delineated to date on the Imperial Gold Project are of sufficient merit to justify additional exploration and development expenditures. The qualified person recommends that further work be conducted to increase the confidence in the resource model, metallurgy and geotechnical knowledge. The authors of this report recommend a data collection program that includes exploration drilling and technical data collection aimed at completing the characterization of the project in preparation for a more advanced technical study and to support project permitting.

The objective of this work will be to upgrade the category of the resources that are presently inferred to indicated resource classification. As such it will require more diamond drilling than RC drilling. The core drilling is needed to twin previously drilled RC holes and provide representative samples for metallurgical, geotechnical and other materials testing. The RC drilling will infill where present drill spacing in the targeted resources is inadequate.

The following recommendations are divided into resource and geology, engineering and metallurgy, and permitting and other, categories.

Resource and Geology

- Drill within the PEA pit to convert resources to higher levels of confidence.
- Prioritize permitting efforts. Project permitting is possibly the highest risk factor for the project.
- Continue drill hole exploration within the Imperial project area, as the deposit is open down dip and laterally in several areas.
- Drilling within the project should be done by core drilling to help improve the geological and structural models.
- Conduct additional geophysical exploration of the land package extending from the Mesquite to Picacho mines.
- Drill high priority geophysical targets.

Engineering and Metallurgy

- Conduct additional column leach tests focusing on ROM, crush size, deposit location, mineralogy and grade.
- Percolation and drain down testing with simulated heap loading to ensure that the heap will perform as predicted.
- Geotechnical investigations into the heap stability.
- Perform geotechnical testing of soils under the leach pad, ponds, and plant site.
- Conduct geotechnical testing of consolidated alluvium and the pit wall rock mass.
- Conduct metallurgical variability leach tests.
- Conduct column leaching of coarse samples obtained from drill core and bulk samples from outcropping mineralized material.
- Following metallurgical testing re-evaluate the crushing option and also the possible timing of when a crushing circuit could be installed.

Permitting and Other

- Closure testing on the spent heap materials should be conducted.
- Prioritize permitting efforts. Project permitting is possibly the highest risk factor for the project.
- Negotiate with the local native population and other stakeholders to obtain a mutually beneficial project.
- Following the completion of the above items, proceed to a pre-feasibility or feasibility study.

It is estimated that the proposed drilling and exploration work and the engineering and other studies would cost approximately US $8,340,000 (Table 1-7).

Table 1-7: Estimated Cost for the Exploration Program and Engineering Studies Proposed by SRK and GRE for the Imperial Gold Project

Description	Total (US$)
Drilling and Exploration	
Reverse Circulation Infill (48,000ft)	2,400,000
Core Drilling (16,000ft)	2,000,000
Geology / Structural Studies	125,000
Exploration QAQC	400,000
Subtotal	**4,925,000**
Engineering and Other Studies	
Environmental baseline studies	500,000
Advance all environmental Permits	1,000,000
Update mineral resource model with new drilling	75,000
Geotechnical / HL design studies	500,000
Metallurgical test work	500,000
Subtotal	**2,575,000**
Community Engagement Program	140,000
Stakeholder Mapping	60,000
Subtotal	**200,000**
Contingency (10%)	640,000
Total	**8,340,000**

1.17 Risks

The main risks associated with the project are related to permitting and California mining regulations. This risk could potentially cause long delays in acquiring permits and additional holding costs during these delays.

There is a risk that the project will encounter serious opposition during the permitting process if the permitting effort is not properly managed. To mitigate this risk the Company plans to initiate an industry best practice community engagement program to build local support with all stakeholders.

The change in California mining regulations in the early 2000's with the introduction of the backfill law severely impacted new projects. With the current higher gold price, the backfill requirement can be met without severely impacting the project economics. There is a risk other regulation could be implemented that further impact project economics.

Somewhat lesser risks include the historic sampling assay results and ROM heap leach recovery. The historic sampling assay results can be mitigated with additional infill drilling including some twin holes to validate the existing database. The ROM heap leach recovery can be verified by bulk sampling outcropping mineralization and leach testing in large columns. Variability tests of coarsely crushed large diameter core will provide additional confidence in ROM leach recovery.

1.18 Opportunities

The project also has some potential upside primarily from new resource discovery as extensions from the currently defined mineralization and new resource bodies along the Mesquite/Imperial/Picacho trend. A lessor opportunity is slightly higher ROM leach recovery that might be demonstrated by large column leach test work.

The study also shows that there will be approximately 95 million tons of alluvial sand and gravels left over after backfilling of the mine pits. No value has been ascribed in the economic analysis to this potential construction aggregate resource. There is an opportunity to deliver this material to the Los Angeles area via the nearby rail line that goes straight to Los Angeles, or possibly using this material as a remediation cover material for the Salton Sea contaminated beaches.

2.0 INTRODUCTION AND TERMS OF REFERENCE

During December 2019, KORE commissioned GRE to prepare a NI 43-101 compliant PEA technical report of the Imperial Gold Project located in Imperial County California, USA. GRE's mandate was to utilize the recently updated mineral resource estimate prepared by SRK and published in a technical report, with an effective date December 30, 2019, titled "Technical Report for the Imperial Gold Project, California, USA." The geology and mineral resource sections of the SRK report have been included in their entirety in this report for completeness. The PEA reported here was prepared in conformity with generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019)".

KORE Mining acquired 100% of the Imperial Gold Project in 2017 and did not consider the findings of the 2012 Preliminary Economic Assessment to be current and therefore requested SRK to update the mineral resource model for the Imperial gold project to current conditions and to document the findings in a report prepared following the guidelines of the Canadian Securities Administrators National Instrument (NI) 43-101 and Form 43-101F1. The mineral resource estimate reported in the December 2019 SRK report was prepared in conformity with generally accepted Canadian Institute of Mining's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019)". The SRK December 30, 2019 technical report is also required by securities law to support the first-time disclosure of mineral resources by KORE Mining.

GRE QP's Dr. Todd Harvey and Terre Lane visited the site on January 9, 2020, and the KORE storage locker in Yuma Arizona on January 10, 2020 where some core, geologic maps and sections, and file cabinets with project data and reports are stored and reviewed by the team. Additional data was provided by KORE via electronic files and access to the company data room.

QPs from SRK including Mr. Glen Cole visited the project site on November 26, 2019, accompanied by a KORE Mining representative. Much of the technical information documented in the December 30 SRK technical report was sourced from the SRK (2012) technical report, with that information being updated as appropriate.

In addition to inspecting the project site and access roads, the SRK QPs visited a storage locker in Yuma, Arizona, where drill core and chip samples and project documentation (maps, sections, reports, correspondence, and data) were inspected. the QPs believe that they were given full access to all available data.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the authors of this report do not consider them to be material.

Both GRE and SRK are not an insider, associate or an affiliate of KORE Mining, and neither GRE, SRK nor any affiliate has acted as an advisor to KORE Mining, its subsidiaries or its affiliates in connection with this project. The results of the PEA and mineral resource evaluation are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings. All of the QPs who authored this report are independent of KORE.

Table 2-1: QP Authorship by Report Section

Section		Author/QP
1 Executive Summary		
1.1	Introduction	Terre Lane
1.2	Property Description, Location, Access, and Physiography	Glen Cole
1.3	History	Glen Cole
1.4	Geological and Mineralization	Glen Cole
1.5	Sample Preparation, Analyses, Security and Data Verifications	Glen Cole
1.6	Mineral Processing and Metallurgical Testing	Todd Harvey
1.7	Environmental, Permitting and Social Impact	Lane, reliance on David Brown
1.8	Mineral Resource and Mineral Reserve Estimates	Glen Cole
1.9	Mining Methods	Terre Lane
1.10	Recovery Methods	Todd Harvey
1.11	Project Infrastructure	Terre Lane
1.12	Market Studies and Contracts	Terre Lane
1.13	Capital and Operating Costs	Lane and Harvey
1.14	Economic Analysis	Lane and Harvey
1.15	Interpretations and Conclusions	All QPs
1.16	Recommendations	All QPs
1.17	Risks	Terre Lane
1.18	Opportunities	Terre Lane
2 Introduction		Cole, Lane and Harvey
3 Reliance on Other Experts		Lane and Harvey
4 Property Description and Location		Glen Cole
5 Accessibility, Climate, Local Resources, Infrastructure and Physiography		Glen Cole
6 History		Glen Cole
7 Geology Setting and Mineralization		Glen Cole
8 Deposit Types		Glen Cole
9 Exploration		Glen Cole
10 Drilling		Glen Cole
11 Sample Preparation, Analyses and Security		Glen Cole
12 Data Verification		Glen Cole
13 Mineral Processing and Metallurgical Testing		Lane and Harvey
14 Mineral Resource Estimates		Glen Cole
15 Mineral Reserve Estimates		

Section		Author/QP
16 Mining Methods		Terre Lane
17 Recovery Methods		Lane and Harvey
18 Project Infrastructure		Lane and Harvey
19 Market Studies and Contracts		Lane and Harvey
20 Environmental Studies, Permitting and Social or Community Impact		Lane, reliance on David Brown
21 Capital and Operating Costs		Lane and Harvey, reliance on Mining Tax Plan LLC
22 Economic Analysis		Lane and Harvey
23 Adjacent Properties		Glen Cole
24 Other Relevant Data and Information		All QPs
25 Interpretation and Conclusions		All QPs
26 Risks and Recommendations		All QPs
27 References		All QPs

3.0 RELIANCE ON OTHER EXPERTS

The authors of this report have not performed an independent verification of the land titles and tenures as summarized in Section 4 of this report. They have relied upon the Title Report for the project claims as provided by Mitchell Chadwick LLP to KORE Mining in a memorandum dated May 3, 2019 and later in a confirmation email of the continued status for the title information in Section 4.2 and Appendices A for this report. The environmental and permitting section of this report were provided by Dave Brown and Kerry Shipiro who act as independent permitting consultant and legal counsel for permitting matters to Kore mining, respectively, and both have experience in permitting mining projects in California and the rest of the USA.

The authors relied on Mining Tax Plan LLC to estimate the federal and California state tax schedule. Mining Tax Plan LLC specializes in U.S. federal, state, local and foreign taxation of precious metal, non-metallic ores, coal and quarry mining based in Centennial, Colorado.

As of the date of this report, the authors are not aware of any litigation that could potentially affect the Imperial Gold Project.

4.0 PROPERTY DESCRIPTION AND LOCATION

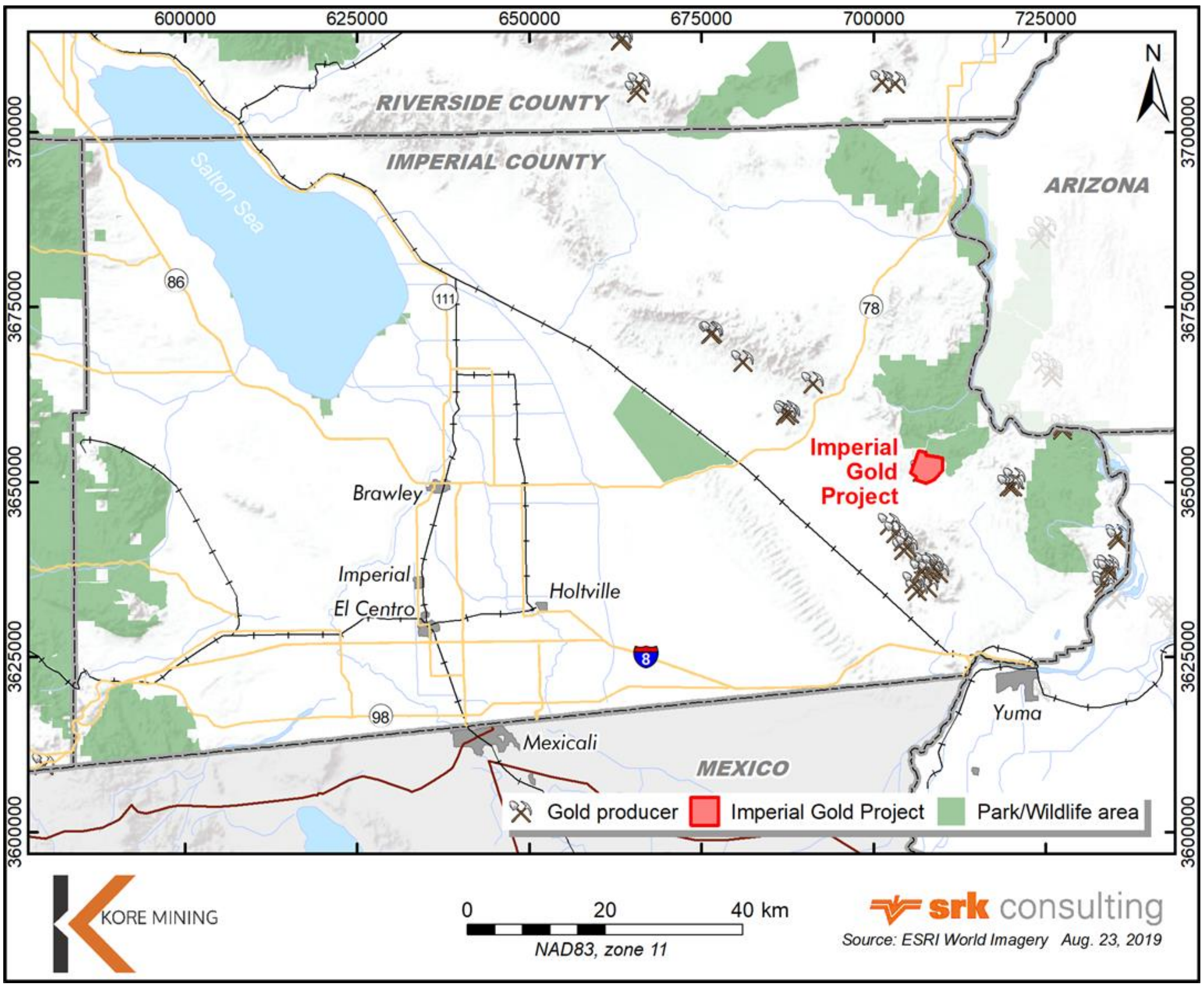
The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

The information contained in this section has not been verified by an independent legal entity. The authors of this report have relied upon land title, tenure and underlying agreement information provided by KORE Mining received from the firm of Mitchell Chadwick LLP.

4.1 Location

The Imperial Gold Project is located in Imperial County in the desert region of southeast California, USA. It is located along the Indian Pass Road approximately 26 road-miles northwest of Yuma, Arizona (see Figure 4-1).

Figure 4-1: Location Map for the Imperial Gold Project



The property is contained within the San Bernardino base meridian:

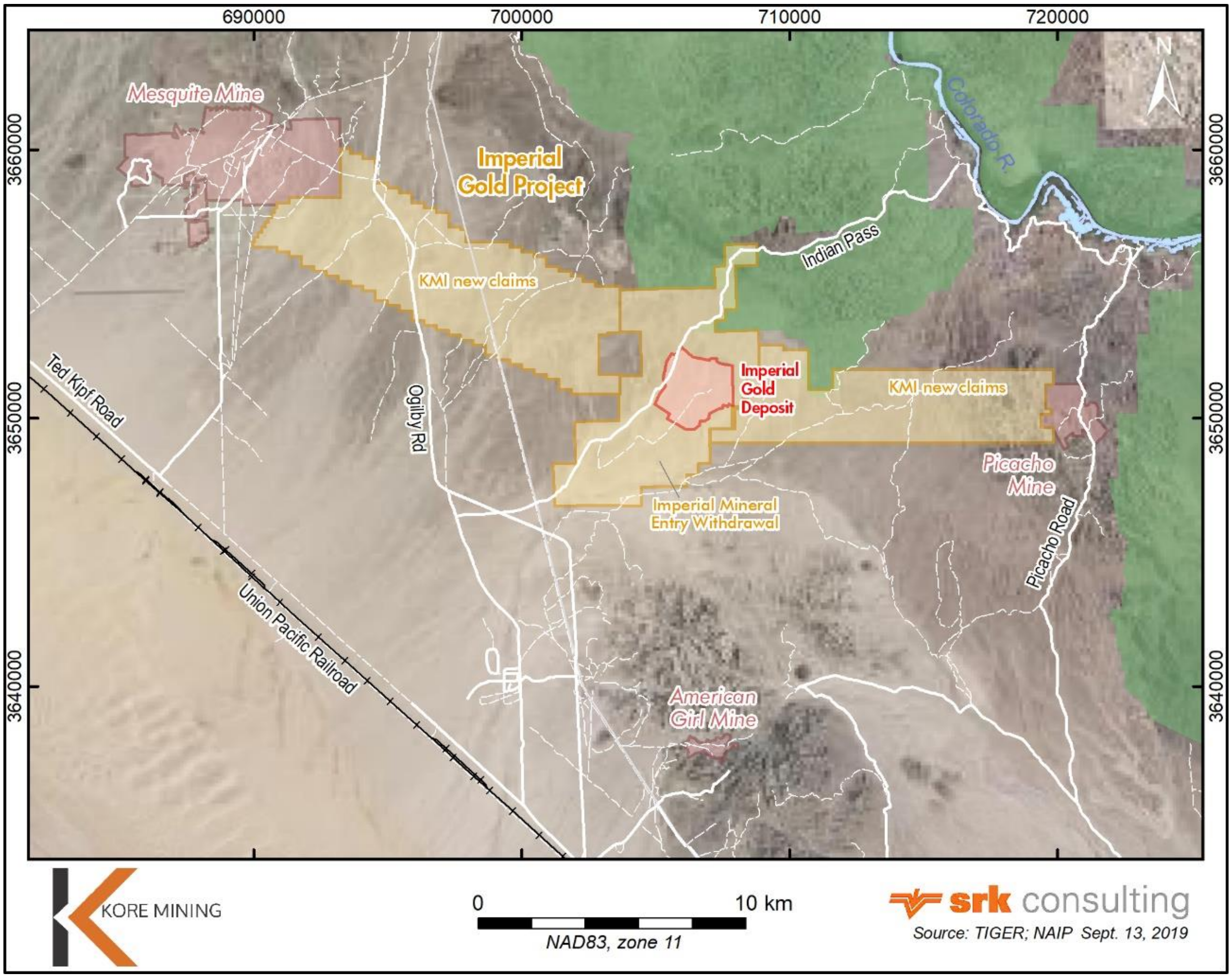
- Sections 31, 32, and 33, Township 13 South, Range 21 East and
- Section 5, Township 14 South Range 21 East, San Bernardino base meridian.

The centroid of the property is at approximately 32°59′ N and 114°47′ W.

The project is located on public land administered by the Bureau of Land Management ("BLM").

The operating Mesquite Mine and the closed Picacho Mine are located roughly ten miles to the northwest and east, respectively, of the property. The closed American Girl Mine is about eight miles south of the project (Figure 4-2).

Figure 4-2: Map Showing the Outline of the Imperial Gold Project Claim Boundaries



4.2 Mineral Tenure

As per mineral tenure information supplied by KORE Mining and a Title Report supplied by Mitchell Chadwick LLP, the project property consists of contains 654 unpatented mining claims. The total area of all the claims is approximately 5,721 acres held by Imperial USA Corp. Within the defined project boundary area there are 468 claims covering 2,020 acres made up of the UYA and BB claims that have been validated by the Mineral Examiner of the Bureau of Land Management. Appendix A contains a complete list of all the project claims.

The Imperial Gold Project that is the subject of this assessment is owned by Imperial USA Corp. (IUC), formerly named, Glamis Imperial Corporation.

Figure 4-3 shows the outline of the Imperial Gold Project claims, with those containing the mineral resource highlighted in red. The project claims tabulated in Appendix A are depicted in plan in Figure 4-3 (Kore Mining is depicted as KMI).

The following Sections 4.2.1 to 4.2.5 describe KORE Mining's option agreement and tenure information.

4.2.1 KORE Mining's Share Purchase Option Agreement

In March 2017, Kore Mining acquired Imperial USA Corp. from Newmont Goldcorp (formerly Goldcorp) (the "Vendor") for an initial payment of US$150,000, and future payments of US$1,000,000 payable upon the announcement of a revised Preliminary Economic Assessment (PEA) or similar report, and US$1,000,000 payable 30 days after the date that gold is poured from ore mined from the related properties. The Vendor has the option to receive these future payments in either cash or shares, up to a maximum 4.9% ownership interest in the Company, above which further share consideration is at the option of the Company. Upon receiving shares, the Vendor also retains the right to participate in future equity issuances on a pro-rata basis. The Vendor also retains a 1% NSR on the property.

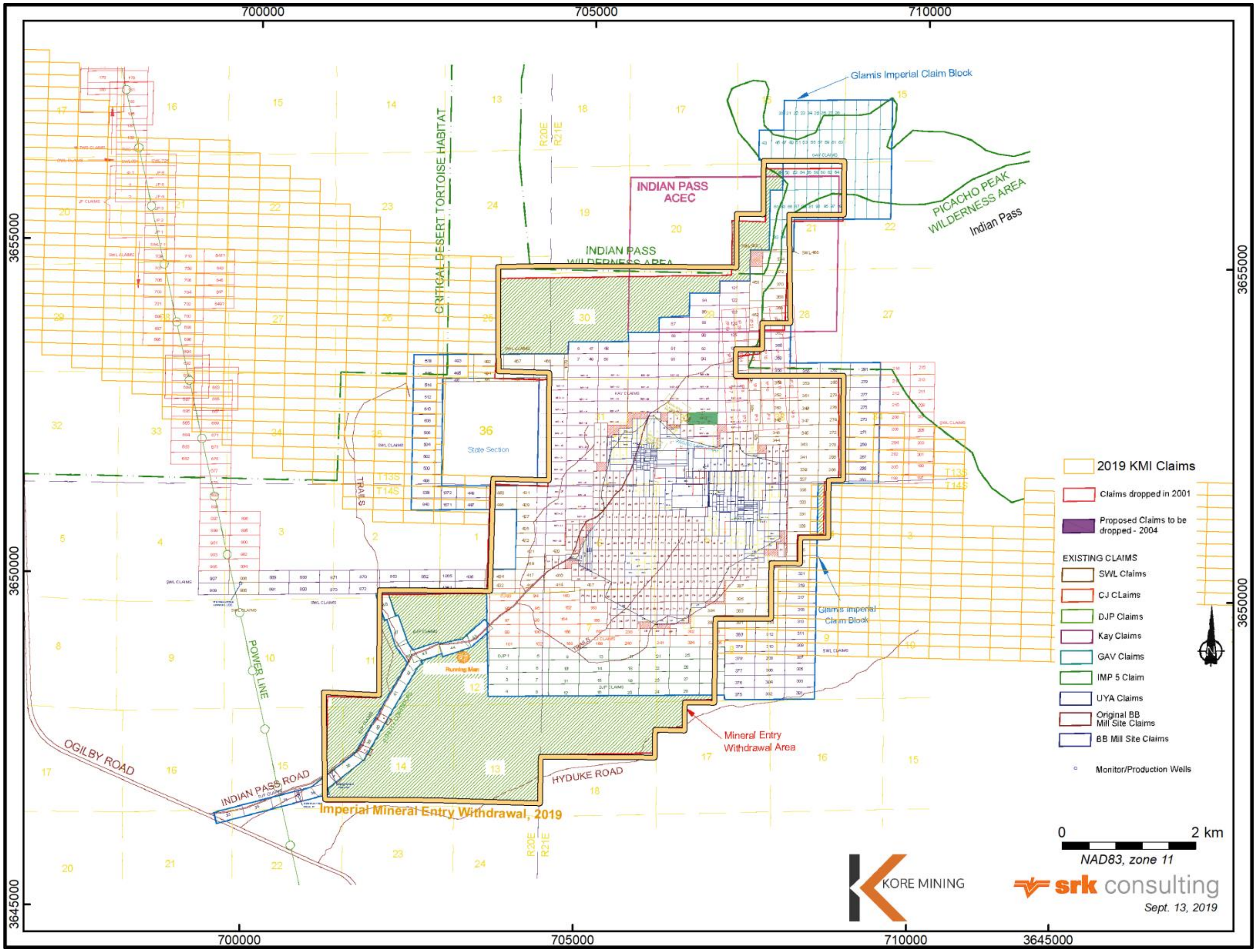
In addition, the Company has committed to incur US$5 million in exploration and evaluation expenditures (which includes permitting and development activities) on the Imperial Project on or before March 2022, the fifth anniversary of the date of the agreement. In the event the Company does not incur these expenditures within this timeframe, the Company must then pay US$1,000,000 to the Vendor.

4.2.2 Revised Title Review Summary

To undertake the title review update, Mitchell Chadwick LLP, examined the following material:

- Performed a fee payment review on the government website in May 2020
- Unpatented Mining Claim and Mill Site Title Opinion (May 17, 2012);
- Certificate of Amendment, recorded in the Imperial County Recorder's Office on April 26, 2019 (June 4, 2012);
- Title Review for Unpatented Mining Claims (January 23, 2017);
- Validity of Claims on Imperial Property (September 19, 2017);
- Title Review Update (October 5, 2018);
- Online search of BLM LR2000 database (April 17, 2019);
- Affidavit Notice of Intent to Hold Payment of Annual Maintenance Fee In lieu of Assessment Work for years 2017 and 2018;
- Maintenance Fee Payment form for years 2017 and 2018; and
- Results of search of Imperial County Official Records, maintained by Chicago Title in El Centro conducted on April 30, 2019.

Figure 4-3: Map Showing the Claim Details of the Imperial Gold Project



4.2.3 Details of Imperial Property Mining Claims and Licences – The BLM Mineral Claim Validity Report

In July 2002, the BLM completed the Validity Report which required a detailed examination and study of the Imperial Property by government representatives of various disciplines using the following guiding principles: "BLM conducts validity examinations to recognize valid claims, eliminate invalid ones and preserve the rights of the public. Any examination must be consistent with the law and must confirm that each mining claim contains a discovery of a valuable mineral deposit, and that each mill site is supported by a qualifying use."

The Validity Report concluded that, "Glamis (now Imperial USA Corp.) appears to have conducted the necessary work within the scope of the statutory requirements, and of a 'prudent operator in usual, customary, and proficient operations of similar character' (43 CFR 3809.0-5(k)) to support their claims, as valid existing rights, within the project area. Within the scope and limitations of this investigation we conclude that Glamis could mine the Imperial gold project as proposed and process gold from mineralized rock on the property at a profit as a surface mine, but not as an underground mine."

The Validity Report pertains to a specific area within the Imperial Property referred to as the Project Boundary which contains the entire underlying West, East and Central (Singer) deposits and known gold mineralization that comprise the geological resource model and covers all the area that encompassed the Plan of Operations that Glamis submitted initially into the federal/state Environmental Impact Statement/Environmental Review ("EIS/EIR") permitting process that started in 1995. The BLM identified the area-of-interest as covering 2,020 acres (817.5 hectares) made up of the 187 UYA Lode Claims and 281 BB Mill Site Claims.

4.2.4 Requirements to Maintain the Imperial Property

The Imperial Property can be maintained in good standing by:

- Firstly paying an annual claim maintenance fee to the Bureau of Land Management ("BLM") for each claim which is due prior to the end of the fiscal tenure year which starts and ends at noon on September 1st of the current year, and
- Secondly by recording an affidavit that the maintenance fees have been paid with the local County Recorder. Failure to comply will result in forfeiture of the claims.

Both of these requirements have been met for the 2020 assessment year, and all Claims are marked as active on BLM's Land & Mineral Legacy Rehost 2000 System as of May 2020.

The BLM maintains an online database named "LR2000" that contains updated information on all unpatented mining claims that have been filed with the BLM. To confirm that all 654 claims are shown as active, with fee payment, Mitchell Chadwick LLP searched the LR2000 database for all claims located in the same townships as the claims on the Mine Site and manually reviewed the status of each claim. After reviewing the LR2000 reports and cross-checking against the Claims list set forth on Appendix A, Mitchell Chadwick LLP confirmed that all of the Claims are marked as "active" in the current BLM database.

An annual inspection/survey of the location corner posts must be conducted to ensure that posts and information contained with the posts is legible and in good condition.

Annual taxes are assessed from July 1st to June 30th of the following year by Imperial County and due for payment on Nov 1st of the current year and February 1st of the following year. Notice of taxes is mailed to the recorded owner.

4.2.5 Royalties and Other Property Encumbrances

There is a 1% net smelter return royalty payable to Newmont Goldcorp (formerly Goldcorp) on any mineral production from the Imperial Project pursuant to the March 2017 Share Purchase Agreement.

In May 2019, the Company issued a 1% net smelter return royalty to Macquarie Americas Corp. on any mineral production from the Imperial Project. The Company has the right to buy back this royalty upon payment of:

- C$4,750,000 until November 2019 if, by this date, all of the outstanding shares of the Company are acquired, by take-over bid, amalgamation, arrangement or similar acquisition transaction, at and for any price per common share of C$0.75 or greater (adjusted for share consolidation/split) in a) cash or b) equity consideration; or
- C$6,750,000 until May 2020 if, by this date, all of the outstanding shares of the Company are acquired, by take-over bid, amalgamation, arrangement or similar acquisition transaction, at and for any price per common share of C$1.00 or greater (adjusted for share consolidation/split) in a) cash or b) equity consideration.

Pursuant to the May 2019 investment by Macquarie Bank Ltd and its affiliates (collectively "Macquarie") where Macquarie acquired the 1% net smelter return royalty, Macquarie also acquired the right of first offer and first refusal on a) project financing for the Imperial Project, b) new royalties on the Imperial project; and c) purchase of the 1% net smelter return royalty issued to Newmont Goldcorp.

4.2.6 Present Environmental Liabilities on the Property

No environmental liabilities have been identified or believed to exist on the Imperial Property. However, it should be noted that the area was utilized during World War II for tank, infantry and weaponry training by General Patton and his troops.

4.3 Permits and Authorization

4.3.1 Lead Agencies and Major Guiding Regulations

The U.S. Department of the Interior – Bureau of Land Management ("BLM") is responsible for administering mineral access on federal public lands on which the project is located, as authorized by the General Mining Law of 1872. The project area comprises approximately 1,648 acres of federal public lands in the form of unpatented mining claims, which were staked in accordance with the General Mining Law. Under this law, qualified "prospectors" are entitled to reasonable access to mineral deposits on these lands. Management of these public lands, including administration of the unpatented mineral claims, falls under the *Federal Land Policy and Management Act* ("FLPMA"), and the governing regulations for FLPMA are found under Title 43 of the Code of Federal Regulations ("CFR"), with specific mineral regulations in 43 CFR §3800 *et seq*. The BLM would function as Lead Agency with respect to compliance with the National Environmental Policy Act ("NEPA") under which the potential environmental impacts from the project would be analyzed and disclosed.

On the local level, the Imperial County Planning/Building Department ("ICPBD") would be the Lead Agency with respect to compliance with California's Surface Mining and Reclamation Act ("SMARA") and applicable sections of Title 14 of the California Code of Regulations, as well as the *California Environmental Quality Act* ("CEQA"). These comprise the major guiding regulations for permitting a mine operation on public land in California.

Currently there are no active federal, state, or local permits authorizing exploration, development, or any other mining activities on the Imperial Property.

4.4 Environmental Considerations

The project is located within the *California Desert Conservation Area* ("CDCA"), which was identified by Congress in FLPMA as a unique area in need of special management by the BLM. Use of the lands and natural resources within the CDCA are guided by the 1980 CDCA Plan (as amended). The project is also located within the *Indian Pass Area of Critical Environmental Concern* ("ACEC") and within the *Indian Pass-Running Man Area of Traditional Cultural Concern*.

Essentially, all of the public lands in the CDCA under BLM management have been designated under a multiple-use classification system. Four multiple-use classes have been established: C – Controlled (the most restrictive), L – Limited, M – Moderate, and I – Intensive (the least restrictive). The Imperial Project area is located entirely within Class L, which is intended to protect sensitive, natural, scenic, ecological, and cultural resource values. Public lands designated as Class L are managed to provide for generally lower-intensity, carefully controlled multiple use of resources, while ensuring that sensitive values are not significantly diminished.

The QP reviewed an environmental assessment proposing the *Indian Pass-Running Man Area of Traditional Cultural Concern* but could find no evidence that it was authorized by the BLM.

The CDCA Plan recognizes that "judgement is called for in allowing consumptive uses only up to the point that sensitive natural and cultural values might be degraded." The multiple use guidelines adopted for implementing the CDCA Plan in Class L lands recognize that locatable mineral operations are non-discretionary, but state that the development of minerals on Class L lands would be limited to activities necessary to achieve extraction with minimum environmental impact, using best available mitigation technology, and most effective feasible reclamation practices.

The project is located on a property not previously developed for commercial use. The project area contains some existing public roads, one set well point (pump not installed) and two monitoring wells previously installed. There is no evidence of previous commercial use or any other use that may have created an environmental liability.

4.4.1 Cultural

Various cultural resource surveys and studies were completed for the project area during the previous NEPA Environmental Impact Statement ("EIS") process. These studies documented the existence of numerous historic trails through the project area, as well as rock features, ground figures, and lithic and ceramic scatters. The EIS public review process progressively revealed that the local Quechan Tribe ascribes very high religious and cultural significance to this area.

Although the BLM and Imperial County issued draft EIS and CEQA Environmental Impact Report (EIR) reviews in 1995 and 1996 that would have allowed the project to proceed, these were rescinded, and in their January 2001 Record of Decision ("ROD") the BLM chose the no-action alternative, effectively denying the project. In part, this decision was based on the determination that the proposed project would cause unavoidable adverse impacts to the cultural resources identified in the area.

However, the subsequent federal administration vacated this ROD in early 2002. As a result of this federal ROD rescission, the California State Mining and Geology Board (SMGB) revised 14 CCR Section 3704.1 (hereinafter Metallic Mine Backfill Regulations). The SMGB adopted these regulations under the guise that the large open-pit quarries resulting from the extraction of metallic minerals were not necessarily left in a useful and beneficial condition, contrary to the intent of the SMARA.

4.4.2 Botanical

A biological survey report conducted for the EIS indicated that no state or federal listed, proposed, or special status plant species were reported in the Project area. A single sensitive plant species, the fairy duster (*Calliandra eriophylla*), was observed within the project area. This assessment has not been updated as part of this report but would need to be completed as part of project permitting.

4.4.3 Wildlife

A biological survey report conducted for the EIS indicated the presence of two federal and/or state listed species, the desert tortoise (*Gopherus agassizii*) and the Gila woodpecker (*Melanerpes uropygial*), are potentially within the Project area. Several special status species were also recorded during the survey. These include the chuckwalla, logger head shrike (*Lanuis ludovicianus*), sharp-shinned hawk (*Falco striatus*), northern harrier (*Circus cyaneus*) and American badger (*Taxidea taxus*). This assessment has not been updated as part of this report but would need to be completed as part of project permitting.

4.4.4 Visual Resources

The Project area landscape consists of a series of gently rolling ridge lines and upland areas interspersed with a series of slightly incised sub-parallel ephemeral drainage channels which all gently slope from north-northeast to south-southwest at approximately 1%. The Project area is relatively undisturbed, with only a few roads, trails, and minor disturbances from historic and ongoing mineral exploration activities.

The landscape color consists principally of browns, tans, and grays, while vegetation colors are generally browns, greens, yellows, and tans. Because of the sparse vegetation cover, the existing landscape colors meld with vegetation colors from distant points.

4.4.5 Land Use

The entire Project area is located within a remote area of eastern Imperial County on undeveloped public lands administered by the BLM. Current land uses in the area consist of mineral exploration and development, aerial military training, utility corridors, and dispersed recreational activities by the general public. Similar public lands with similar uses generally surround the Project area.

However, access to these similar lands off Indian Pass Road for recreational use by motorized vehicles is limited to designated trails. The nearest residence to the project site and process area is at Gold Rock

Ranch, which is located approximately seven miles southwest of the project site and process area. No other permanent residences are known to exist within ten miles of the project area.

There are two wilderness areas located near the project. The Picacho Peak wilderness is located half a mile north of the project and the Indian Pass wilderness is located 1.5 miles north of the project. Both areas are accessed via the Indian Pass Road. Land use status will need to be updated as part of the project permitting process.

4.5 Mining Rights in Imperial County, California

[1]Federal law and policy recognize the importance of a viable domestic mining industry and also recognize the importance of protecting natural resources from the potential damaging effects of mining. For example, the Mining Law of 1872 allows miners to secure exclusive rights to mine public lands through the location of valid mining claims, and the Mining and Mineral Policy Act sets forth a federal policy to "foster and encourage" mining, (30 U.S.C. §§ 21a, 22). On the other hand, Section 302(b) of the FLPMA directs that the Secretary "shall by regulation or otherwise, take any action necessary to prevent unnecessary or undue degradation of the lands" (43 U.S.C. §1732(b)). Section 601 of FLPMA also provides, in part:

> *Subject to valid existing rights, nothing in this Act shall affect the applicability of the United States mining laws on the public lands within the California Desert Conservation Area, except that all mining claims located on public lands within the California Desert Conservation Area shall be subject to reasonable regulations as the Secretary may prescribe to effectuate the purposes of this section. Any patent issued on any such mining claim shall recite this limitation and continue to be subject to such regulations. Such regulations shall provide for such measures as may be reasonable to protect the scenic, scientific, and environmental values of the public lands of the California Desert Conservation Area against undue impairment, and to assure against pollution of the streams and waters within the California Desert Conservation Area (43U.S.C. §1781(f)).*

BLM regulations concerning the surface use of mining claims on public land reflect the dual purposes behind this policy. The regulations provide that it is the policy of the Department of the Interior to "encourage the development of Federal mineral resources," but to do so consistently with the obligation to prevent "unnecessary or undue degradation of the lands." (43 CFR 3809.0-6). The term "unnecessary or undue degradation" is defined in BLM's regulations as follows:

> *Unnecessary or undue degradation means surface disturbance greater than what would normally result when an activity is being accomplished by a prudent operator in usual, customary, and proficient operations of similar character and taking into consideration the effects of operations on other resources and land uses, including those resources used outside the area of operations. Failure to initiate and complete reasonable mitigation measures, including reclamation of disturbed areas or creation of a nuisance, may constitute unnecessary or undue degradation.*

[1] The first four paragraphs of this section are taken from the Final Environmental Impact Statement, September 2000.

Failure to comply with applicable environmental protection statutes and regulations thereunder will constitute unnecessary or undue degradation. Where specific statutory authority requires the attainment of a stated level of protection or reclamation, such as in the California Desert Conservation Area, Wild and Scenic Rivers, areas designated as part of the National Wilderness System administered by the Bureau of Land Management and other such areas, that level of protection shall be met. (43 CFR 3809.05(k)).

The Solicitor for the Department of the Interior under the Clinton administration issued a legal opinion signed on January 3, 2000 by the Secretary of the Interior that reviewed the regulation of hardrock mining as it applied to the Proposed Action. This opinion found that the unnecessary or undue degradation standard, as defined above, allowed BLM to require reasonable mitigation measures to protect resources, but did not by itself give BLM the authority to prohibit mining altogether on public lands. Because the Proposed Action would be located within the CDCA, the opinion went on to analyze the “undue impairment” standard (43 U.S.C. §1781(f), quoted above).

The opinion noted that use of the lands and natural resources within the CDCA are guided by the 1980 CDCA Plan (as amended), and that all of the Project facilities would be located within multiple use Class L - Limited Use, which is the second-most restrictive of the four classifications. The opinion found that the “undue impairment” standard would permit BLM to impose reasonable mitigation measures to prevent undue impairment, and that the standard might also permit denial of a plan of operations if the impairment of other resources is particularly “undue,” and no reasonable measures are available to mitigate that harm.

The Solicitor for the Department of Interior under the Bush administration issued a legal opinion signed on October 23, 2001, by the Secretary of the Interior that again reviewed the regulation of hardrock mining and the former Solicitors opinion. This opinion found that the former Secretary improperly applied the concept of “undue impairment” and that this standard could not be used to deny a Plan of Operation (“PoO’) until the agency formally defined the term through a rulemaking process.

Since the agency has not to-date conducted any rulemaking process to define “undue impairment” future proposed mining operations would likely be subject to the “Unnecessary or undue degradation” standard as defined above.

On October 27, 2000, the Secretary of the Department of Interior (“DOI”) issued a final withdrawal for the Indian Pass area, which includes the area of the proposed Project. This withdrawal precludes entry under the public land laws, including mining laws, for a period of twenty years, subject to valid existing rights. Because these lands were not withdrawn from mineral entry before Glamis located its mining claims, the withdrawal is subject to Glamis’s (and KORE’s) mining claims to the extent the claims were valid on the date of the withdrawal and continue to be valid today. The DOI conducted a Validity Examination of the Glamis claims and issued a final report on September 27, 2002 in which they concluded the claims were valid.

Mining operations in the State of California are conducted under the mining regulations provided in the Surface Mining and Reclamation Act of 1975 (as amended). This act states, The Legislature hereby finds and declares that the extraction of minerals is essential to the continued economic well-being of the state and to the needs of the society, and that the reclamation of mined lands is necessary to prevent or minimize adverse effects on the environment and to protect the public health and safety.

The Legislature further finds that the reclamation of mined lands as provided in this chapter will permit the continued mining of minerals and will provide for the protection and subsequent beneficial use of the mined and reclaimed land.

The Legislature further finds that surface mining takes place in diverse areas where the geologic, topographic, climatic, biological, and social conditions are significantly different and that reclamation operations and the specifications therefore may vary accordingly.

Therefore, QP concludes that the owner of the validated mineral claims (i.e., the claims within the area defined by the Project Boundary) has the right to advance its exploration and mining interests subject to obtaining permits to carry out the activities per the permits and authorisations referred to in Section 3.3.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

5.1 Accessibility

Road access to the site from Yuma is eight miles west on Interstate Highway 8 to State Highway S34 (Ogilby Road), 13 miles north on S34 to Indian Pass Road, and five miles northwest along Indian Pass Road. Highways 8 and S34 are paved roads, while Indian Pass Road is a good gravel road maintained by the county. Approximately one mile of the Indian Pass Road would have to be temporarily re-located around the West Pit.

It is assumed that workers at the project would travel from Yuma and surrounding communities to the site each day.

5.2 Climate

The project site is located in the Colorado Desert and has a typical desert climate with very hot summers, warm winters, and very low annual precipitation of 3 to 5 inches. The region enjoys over 4,000 hours of sunshine per year. The maximum temperatures generally occur in July when the maximum temperature averages about 100°F and the average minimum temperature is 80°F. In December, the coldest month, the average high is about 70°F and the average low about 45°F.

The majority of the precipitation in the region occurs in winter with very little rain falling in April, May and June. Evaporation rates are estimated to be 100 inches per annum and the probable maximum precipitation event is 4.65 inches caused by localized thunderstorms with the potential to cause flash flooding (WSE, 1996). In 1997, 3.6 inches of rain was recorded at the near-by Marine Corps Air Station Yuma as a result of the landfall of Hurricane Nora.

The project operation is not anticipated to be materially impacted by weather.

5.3 Local Resources and Infrastructure

The project is located near Yuma, Arizona a city of over 100,000 people. There are abundant mining support services and skilled labour available in Yuma.

Water for the site would be provided from wells located approximately five miles away, near the junction of Indian Pass and Ogilby Roads.

Electrical power is available within five miles of the project site.

5.4 Vegetation

Vegetation in the project area is typical of a hot desert climate in the region (Figure 5-1). The lack of precipitation and high temperatures limits vegetation growth to specialized species. Ocotillo and Jumping

Cholla are common in the area and occur as single, widely spaced individuals. Mesquite and palo verde trees occur in and around the stream beds.

Figure 5-1: Typical Landscape in the Project Area

KORE MINING

srk consulting

Source SRK, 2012

5.5 Physiography

The project is located at between 700 ft and 900 ft above sea level on a plain southwest of the Chocolate Mountains and north of the Cargo Muchacho Mountains. The project area is generally flat with rolling pediments of up to about 100 ft in height (Figure 5-1).

6.0 HISTORY

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

Due to the extent of the alluvial cover on the Imperial Gold Project, exploration has historically consisted primarily of drilling. Initial exploration strategies focused on wide-spaced definition drilling of buried gravity and structural anomalies. Mineralized zones were projected down dip and followed with additional drilling to depths exceeding 1,000 ft. Later exploration strategies focused on the development of the entire deposit and tested down-dip areas for economic mining limits. To date, 349 exploration boreholes totaling 195,047 ft have delineated the mineralized zones defined in the geology and mineral resource modeling completed.

6.1 Exploration by Gold Fields Mining Corporation (1980-1986)

Gold Fields Mining Corporation (Gold Fields), between 1980 and 1986, acquired a 16,000-acre land holding and conducted a regional exploration program searching for low-grade, heap leachable gold deposits similar to their discovery at the Mesquite mine. Gold Fields was attracted to the Imperial Gold Project area by encouraging geochemical dry stream wash gold results, favourable widely spaced gravity, resistivity and aeromagnetic results, and the presence of placer gold and lode gold underlying Anna M. and Richard L. Singer's claims within the Imperial Gold Project area.

Drilling on the Imperial Gold Project by Gold Fields is summarized in Section 9.

6.2 Exploration by Imperial County Joint Venture (1987-1993)

In 1987, Gold Fields entered into an option agreement with the Imperial County Joint Venture comprising of Glamis Gold (65%) and Amir Mines Inc. (35%).

In 1987, the Imperial County Joint Venture conducted an exploration program consisting of 1,066 samples of experimental gas vapour phase geochemical survey over the strike of the gravity-resistivity trend, as well as reverse circulation drilling in the West, East, and Golden Queen areas (located east of the East area), and on a few of the gas vapour anomalies.

In 1989, Amir Mines Inc. changed its name to Imperial Gold Corporation and again in 1990 to Arizona Star Resources Limited.

Exploration by the joint venture between 1989 and 1992 consisted solely of drilling. A summary of the drilling activities by the Imperial County Joint Venture can be found in Section 9.

6.3 Exploration by Glamis Gold (1994-1996)

In 1994, Glamis Gold, under the name of wholly-owned subsidiary Chemgold Inc., became the sole owner and operator of the property and initiated an accelerated development drilling and pre-feasibility program.

The 1994, 1995, and 1996 exploration programs focused on definition drilling within the East, West, and Central areas, as well as metallurgical testing, engineering studies, environmental studies, density studies and culminated with a feasibility study completed in April 1996.

A summary of the drilling activities by the Glamis Gold can be found in Section 9.0, Exploration.

6.4 Previous Mineral Resource Estimates

Following the completion of exploration drilling by the Imperial County Joint Venture, the overall geological reserve in 1990 was estimated by Mine Development Associates (MDA) from Reno, Nevada as 13.3 Mt at 0.022 oz/t gold (Garagan, 1990). The reader is cautioned that this historical mineral resource and mineral reserve estimate was prepared prior to the implementation of the NI 43-101 guidelines and, therefore, the values reported should not be relied upon. A qualified person has not done sufficient work to classify this historical estimate as current mineral resources and they have not verified to determine their relevance or reliability. This historical mineral resource and mineral reserve estimate is superseded by the mineral resource statement reported herein. The Company is not treating this historical estimate as a current mineral resource. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In 1996, MDA from Wheat Ridge, Colorado prepared an updated mineral resource estimate that was applied in an historical feasibility mining study commissioned by Glamis Gold (MDA, 1996). Open pit mineral resources were constrained by the East and West conceptual pits. The conceptual pit envelopes were designed at a gold price of $400/oz. The mineral resources were reported at a cut-off grade of 0.007 oz/t gold. A qualified person has not done sufficient work to classify this historical estimate as current mineral resources. The issuer is not treating this historical estimate as a current mineral resource and they have not verified to determine their relevance or reliability. This historical mineral resource and mineral reserve estimate is superseded by the mineral resource statement reported herein. The Company is not treating this historical estimate as a current mineral resource. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In 2012, Delta commissioned SRK to prepare an updated mineral resource model upon which a preliminary economic assessment was based (SRK, 2012). This mineral resource model was the first mineral resource evaluation prepared for the Imperial Project in accordance with the Canadian Securities Administrators NI 43-101 guidelines. and was based on a database comprising 349 RC boreholes, 344 of which were located within the resource estimation area.

Analytical data used for the SRK (2012) mineral resource model was primarily sourced from drilling completed between 1987 and 1996 by Gold Fields, Glamis Gold, and other historical operators. The mineral resource statement which was informed by a total of 190,134 ft of RC drilling, is tabulated in Table 6-1.

These mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines” (November 23, 2003). The qualified person for this mineral resource statement were Dominic Chartier, PGeo. (OGQ #874), Dorota El Rassi, P.Eng. (APEO #100012348) and Glen Cole, PGeo. (APGO #1416), who were independent qualified persons as this term is defined in NI 43-101. The effective date of this resource estimate was March 31, 2012. The mineral resource statement documented in this report replaces this version.

Table 6-1: Mineral Resource Statement, Imperial Gold Project, SRK 2012

Classification	Quantity ('000 tons)	Grade Gold (oz/t)	Contained Gold ('000 ounces)
Indicated			
Grade Zone (Domains 100, 120)	50,445	0.0174	879
Total Indicated	**50,445**	**0.0174**	**879**
Inferred			
Grade Zone (Domains 100, 110, 120)	78,298	0.0160	1,251
Gravel with grade (Domain 200)	1,403	0.0067	9
Bedrock with grade (Domain 300)	4,443	0.0085	38
Total Inferred	**84,144**	**0.0154**	**1,298**

Reported at a cut-off grade of 0.005 oz/t Au using a price of $1,400 /oz Au inside a conceptual pit shell optimized using metallurgical and process recovery of 80%, overall mining and processing costs of $3.60 per ton and overall pit slope of 45 degrees.
All figures rounded to reflect the relative accuracy of the estimates.
Mineral resources are not mineral reserves and do not have demonstrated economic viability

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

7.1 Regional Geology

The Imperial Gold Project is located on the southern flank of the Chocolate Mountains, structurally aligned and equidistant between the Picacho and Mesquite gold deposits. The project area is underlain by a sequence of Jurassic age gneisses and schists. This package of rocks is part of the amphibolite grade metamorphic suite of the Chocolate Mountain thrust sequence. The thrust system has displaced metamorphic and igneous rocks north-eastward over metamorphic greenschist facies Pelona and Orocopis schists during the Mesozoic time period. The metamorphic rocks are unconformably overlain by Cenozoic andesite, basalt flows, and tuffs. Overlying the volcanic rocks are Paleocene age fanglomerate gravels with variable thicknesses reaching up to 700 ft. A thin veneer of Miocene flood basalts and Quaternary age alluvium locally caps the gravels. A plan showing the regional geology setting is provided in Figure 7-1.

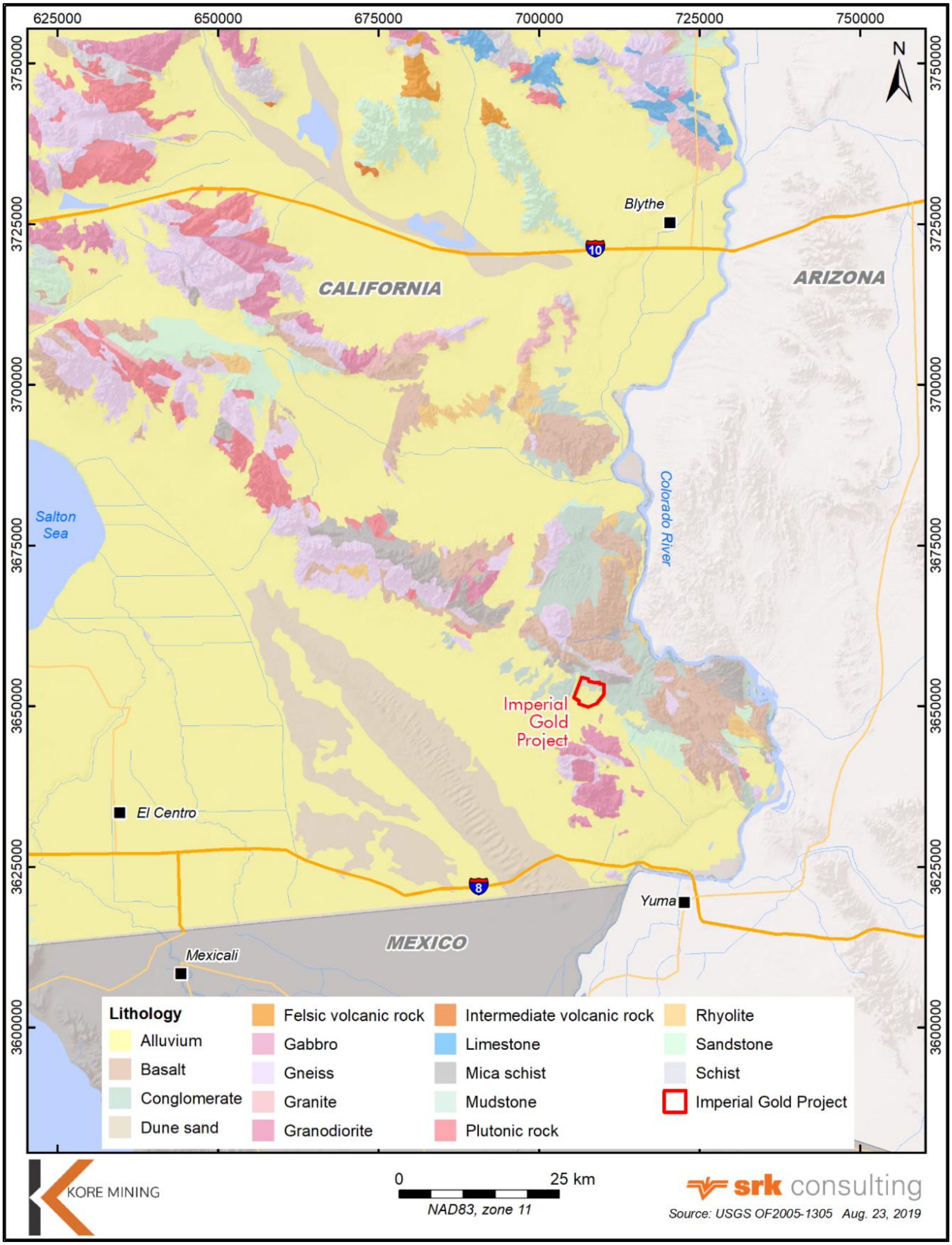
7.2 Property Geology

The Jurassic age metamorphic gneisses and schists underlying the Imperial Project have similarities to rocks found at the Mesquite and Picacho gold mines. There are very few outcrops which necessitated that the geological model be developed by interpreting drilling results. The dominant application of reverse circulation drilling and the local variations of texture and composition within the stratigraphic sequence currently make it difficult to correlate between boreholes. Core and rock chip logging placed more emphasis on recognizing changes in alteration, mineralization, and apparent structural discontinuities in order to correlate stratigraphy between boreholes and sections. Surface geological information was limited to examining a few outcrops in the Singer deposit area, which is located between the West and East portions of the deposit.

The predominant rock type intersected in the boreholes below the Paleocene gravels is the Jurassic- age biotite gneiss. The biotite gneiss contains numerous gradational divisions of biotite-chlorite gneiss and quartz feldspathic gneiss with gradational sequences into their schistose equivalents. The biotite gneiss package occurs across the entire project, while a muscovite-sericite rich unit is prevalent in the East portion of the deposit. Gold mineralization is hosted within the biotite gneiss and the sericite gneiss units.

The biotite gneiss units are capped by an upper felsic gneiss, logged commonly as a quartzite, which is predominant in the Central area of the project hosting the Singer mineralization. The quartzite is possibly a silicified version of the quartz feldspathic gneiss and may have acted as a cap to upwelling mineralized fluids (Scott 1992). If correct, then the Singer area, which is part of the Central area, may represent the top or peripheral top of the mineralizing hydrothermal system.

Figure 7-1: Regional Geology Setting of the Imperial Gold Project



The metamorphic units are unconformably overlain by thin andesite basalt flows that are generally less than 100 ft in thickness. Paleocene age fanglomerates and alluvium with variable thicknesses of 10 ft to 700 ft cover 95% of the project area. A thin veneer of Miocene flood basalts and Quaternary age alluvium locally caps the gravels.

The footwall of the metamorphic units usually consists of a siliceous breccia unit, which varies from 10 ft to 170 ft in thickness. The unit appears to parallel the fault planes of the low angle thrust sheet. The breccia is interpreted to have been injected along fault contacts as the result of the pressure release of hydrothermal fluids. A 1990 petrographic report describes the rock type as having a highly variable grain size and consisting of brecciated gneiss and dacite fragments in a rock flour matrix (Garagan, 1990). There is no indication of strain or rotation in drill cuttings and surface rock specimens have uncrushed zoned feldspars, suggesting the unit is not of tectonic origin. The siliceous breccia is flat lying to gently inclined with dips of 5° to 15° southward steepening in dip to 60° to 70° south along thrust planes.

Below the siliceous breccia unit, a footwall gneiss unit consisting of hornblende biotite gneisses occurs. This footwall unit tends to be very hard and shows rare and thin mineralized intercepts. Below this, the footwall conglomerate unit is a well indurated, clay-carbonate cemented material with coarse sub-angular gneissic fragments varying from 10 ft to 200 ft in thickness.

An interpretative East-West longitudinal section across the deposit is shown in Figure 7-2, whereas two other interpretative cross sections are provided in Appendix B.

7.2.1 Lithology

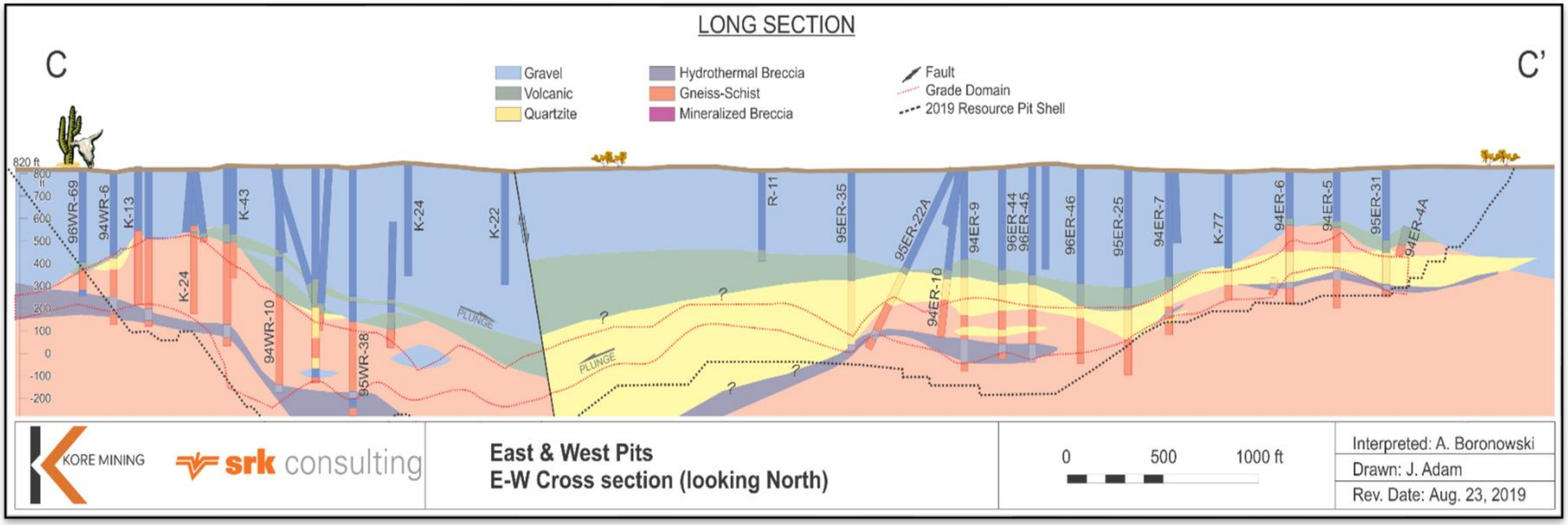
The following rock type codes are described in WSE (1996):

Gravel – Contains material eroded from the metamorphic units. Narrow mineralized horizons within the gravels are believed to represent placer material eroded from exposed mineralized horizons. Gravels occur above and below the West deposit. Gravels below the West deposit may be explained by a positive-type flower structure, which has thrust older stratigraphy over the younger gravels.

Gneiss/Schist – Predominantly consisting of biotite gneiss and sericite gneiss but locally contains quartz, feldspar, chlorite, hornblende, and grades in schistose members. The West portion of the deposit contains mostly biotite gneiss and the East portion contains predominantly sericite schist. A petrographic report shows that a mineralized haematitic gneiss sample consists of quartz feldspathic schist that was recrystallized. Limonite occurs in fractures and as interstitial films and pores. Gold mineralization is primarily hosted within the biotite gneiss and the sericite gneiss units on the Imperial gold project.

Hydrothermal Breccia – Occurs along fault contacts. It can be tabular and shallowly dipping southward to narrow and dipping steeply to the south or north. Contemporaneous or post gold mineralization. The tabular hydrothermal siliceous breccia is locally stacked. The rock consists of a siliceous, fine grained, blue-grey to brownish-yellow unit consisting of brecciated gneiss and dacite fragments in a rock flour matrix. Uncrushed, zoned feldspar crystals suggest that the breccia was formed by a hydrothermal event rather than a tectonic event. Locally, the hydrothermal breccia is mineralized with gold.

Figure 7-2: Conceptual East-West Long Section Across the Imperial Gold Deposit (looking North)



Section line C-C' is indicated on a plan in Appendix B of this report

A petrographic examination describes five samples of the siliceous breccias as protomylonite (granulated rhyolite to granitic gneiss) or a microbreccia. The rock contains a pseudoporphyritic texture with coarse grained fragments in a finer grained cataclastic matrix. The rock contains no directional fabric, suggesting crushing rather than shearing was the method of fracturing.

The breccias probably represent a gas-charged phreatic breccias formed as the result of the pressure release of hydrothermal fluids. These breccias are common in epithermal environments.

Volcanics – Grey-brown to maroon coloured fine-grained andesite to basalt flows and tuffs overlay unconformably the metamorphic package. Gold mineralization is rare in the volcanic rocks.

Quartzite – An upper felsic gneiss is commonly logged as a quartzite and is predominant in the Central area. The quartzite is probably a silicified version of the quartz feldspathic gneiss and may have acted as a cap to upwelling mineralized fluids.

Mineralized Gravel – Low grade (0.010 oz/t gold to 0.015 oz/t gold) mineralization also occurs within the overlying cemented gravel units as narrow layers eroded from exposed mineralized gneissic units.

Mineralized Breccia – Hosts sporadic gold mineralization associated commonly with limonitic fracture fillings, variable silicification, pyrite pseudomorphs and quartz veining.

High Grade Vein – Elevated gold values are directly related to the pervasiveness of the haematitic and limonitic alteration, the fracture density of the host, and most significantly, the presence of quartz veining and haematitic gouge zones.

7.2.2 Structural Geology

The dominant structural feature in the project area is a west-northwest trending thrust sheet that places Jurassic age gneisses and schists northeast over Paleocene gravels.

The thrust sheet appears as a network of curved faults (flower faults) that dip approximately 30 degrees to the south and steepen southward along the curve. Flower structures are typical of structures formed in a transpressional strike-slip environment and are common on parts of the San Andreas Fault System where shortening has thrust pre-Cretaceous granodiorite over Paleocene sediments (Boulter, 1989, and, Willis and Tosdal, 1992).

Riedel shear structures related to the dextral shear regime are formed during this phase of deformation. The shear regime structures likely prepared the rock for hydrothermal fluid migration.

Post-mineralization, high angle, east-west striking normal faults (step faults) have down faulted the mineralized zones to the south. Depth of mining would be determined by economics relating to amounts of displacement in these down dip mineralized zones.

The low angle footwall thrust contact forms the north side of the mineralized zone and defines mineable limits.

High angle, north to northeast trending faults bound the mineralized zones, forming the east and west economic limits of the proposed East and West pits. The full extent of these faults is not yet well understood.

7.2.3 Mineralization and Alteration

Gold mineralization occurs primarily within haematitic and limonitic altered breccias, microfractures and gouge zones developed in the host biotite gneiss and sericite gneiss units. Minor quartz veining, very-fine grained pyrite pseudomorphs and silicified zones are also common.

The density of fractures, extent of the red-brown to yellow haematitic/limonitic coatings and pyrite pseudomorphs within the host units are notable mineralized features. Logging of core and cuttings samples from the project site indicated no fresh pyrite or sulphide mineralization is present due to the oxidized state exhibited throughout the deposit.

The deposits were oxidized to a depth in excess of 750 ft indicating that the deposits were oxidized near surface and down dropped by faulting to their current locations.

The majority of gold mineralization occurs stratigraphically above a siliceous breccia horizon. This distinct relationship between the siliceous breccia and the overlying host rock units is traceable across the deposit. Sporadic mineralization is also noted along the cemented gravel and volcanic contacts and in fault structures within the brecciated volcanic and conglomeritic units. Low grade mineralization also occurs within the overlying cemented gravel units as narrow layers eroded from exposed mineralized gneissic units.

The mineralization and alteration character of the deposit varies across the deposit as described below.

East Area

Gold mineralization in the East area occurs within a west-northwest trending fault zone with a strike length of 3,200 ft, a variable width of up to 800 ft, and an average thickness of approximately 85 ft. The mineralized zone is a tabular body, predominantly flat lying to gently dipping 5° to 15° south. The mineralized body is cut by a series of east-west striking normal faults. The fault bound mineralized lenses of the tabular body are offset progressively deeper southward across the series of faults.

The east-west normal faulting may represent extension or possibly a change from a positive flower structure to a negative flower structure. It was noted that the dip of the mineralized lenses to the north steepen to 45° to 70° to the south. It was explained that the change in dip may be coincidental with the inflection of the flower structure thrust sheet where it steepens to a 60° to 70° dip to the south (Scott, 1992).

Another explanation may be that the shallow mineralized lenses were thrust over the adjacent, relatively stable stratigraphy, and then during the extensional period, a section of the shallow mineralized lenses located along the edge of the relatively stable stratigraphy was dragged down and southward along the south dipping normal fault. The mineralized lenses are cut by north-northeast trending normal faults that drop stratigraphy to the east and west. Paleocene to recent gravels covers the East portion of the deposit, averaging approximately 200 ft in thickness.

Gold values in the East area are elevated where the pervasiveness of limonitic alteration increases and is accompanied by silicification, quartz veining, pyritization and gouge zones. The distribution of the hematitic and limonitic alteration zones within the East area exhibit a definite spatial association to the siliceous breccias. A vertical zonation is noted in several mineralized intersections associated with the breccias from limonitic to hematitic alteration moving up in the stratigraphy. The thickness of the limonitic zone is variable, ranging from 10 ft to 75 ft. The hematitic zones are typically thicker, up to 150 ft. Hematitic and limonitic alteration show crude correlation with an increase in gold grade/thickness along linear trends oriented to the east-northeast. The linear trends are believed to reflect the presence of high angle mineralized structures. Similar structures also occur in the nearby Picacho and Mesquite mine sites.

West Area

The West area is similar to the East area and was modelled by the QP as an extension of the same mineralized body. Mineralization occurs as a tabular body made up of several zones with planar dimensions of 1,200 ft in length, 1,000 ft in width and an average thickness between 90 ft and 120 ft. Mineralization intercepts occur as shallow as 20 ft from surface and average 80 ft to 120 ft below surface.

The gold mineralization is down faulted to the south by a series of east-west trending vertical to steeply south dipping normal faults. Vertical displacement on these structures is variable from 80 ft to 260 ft. Drill data suggests that the mineralized zone is cut off to the west by a north-northeast trending structure that displaces stratigraphy down to the west. The amount of strike slip displacement is unknown on this structure. The West area gold mineralization is limited to the east by a northeast trending fault and to the east of this fault is situated the Central area. Mineralization to the north tapers into a series of discontinuous lenses or is cut off by a north dipping antithetic fault to the flower structure.

Central Area

The Central area is a down faulted block of the same stratigraphy encountered in the West and East pits. Structurally the area differs slightly from the West and East pits. Bedrock intersections occur predominantly in the shallow portion of the "flower structure". Mineralization is not as prevalent in the shallowest portion of the thrust structure in the West and East pits. This may be the result of the structural preparation of the host and explain the narrow (10 ft to 40 ft) sporadic intersections in the Singer Pit area.

Mineralization is hosted by biotite to biotite-chlorite quartz-feldspar gneisses and to a lesser degree sericite schists. Mineralization is also spatially related to a fault gouge zone that represents the fault contact between the gneissic package and underlying gravels. Gold values are associated with hematite fractured gneisses with localized zones of quartz veining, gouge zones, and to a lesser degree limonite alteration, silicification and brecciation of the host rock. Mineralization commonly occurs stratigraphically below a fine-grained, quartz-rich unit that has a variable thickness (5 ft to 180 ft). This unit, descriptively-logged as "quartzite", may represent a facies change within the gneissic package or more likely a silicified quartz feldspathic unit that acted as a cap to mineralizing fluids. The "quartzite" is fractured and altered by hematite along fractures but seldom hosts any mineralization.

A siliceous breccia unit in the Central area has mineralization occurring stratigraphically above although not directly adjacent to the breccia unit. However, in areas where the breccia appears to have a steep dip to the south mineralization may occur both above and below the breccia horizon. An example is drill hole I-11, which intersected 0.045 oz/t gold over 20 ft below the breccia.

8.0 DEPOSIT TYPES

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

The Imperial gold deposit is believed to represent epithermal gold mineralization related to Tertiary-age low angle detachment faults and associated extensional faults. The epithermal gold mineralization is structurally controlled and transitional between low and high-sulphidation systems.

Structural data from the Mesquite mining district suggests that the gold mineralization accompanied dextral strike-slip faulting during Oligocene (Willis & Tosdal, 1992). Dextral strike-slip faults in the mining district have northwesterly strikes and extension fault and veins strike northerly, consistent with a north south-oriented shortening and east-west-oriented extensional strains during mineralization (Willis & Tosdal, 1992).

9.0 EXPLORATION

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

Exploration work conducted on the Imperial gold project was completed prior to KORE Mining involvement.

Historical exploration is summarized in Section 6.0. Exploration drilling completed by historical operators is described in Section 10.0.

10.0 DRILLING

The following was prepared by SRK for the December 30, 2019 technical report. Nothing has changed in this section since the SRK report was published and the text for Section 10 of the SRK report is included here verbatim.

Exploration drilling conducted on the Imperial Gold Project was completed prior to KORE Mining involvement. The following section summarizes the drilling efforts completed by previous operators. Table 10-1 summarizes the drilling activities by year, drilling type and operator. A plan map of drilling, by operator, in relation to the 2019 mineral resource open pit shell and grade domains on the Imperial Gold Project shown in Figure 10-1.

Table 10-1: Summary of Drilling on the Imperial Gold Project

Year	Operator	Type	No. Holes	Total (ft)
1982-1986	Gold Fields	RC	53	27,880
1987-1992	Imperial County Joint Venture	RC	169	71,539
1994	Glamis Gold	RC	45	34,565
1995	Glamis Gold	RC	32	29,890
1994-1995	Glamis Gold	Core[a]	9	4,913
1996	Glamis Gold	RC	41	26,260
Total	**All**	**All**	**349**	**195,047**

a. Core drilling was dedicated to metallurgical testwork and was not used in the previous or current resource estimates.

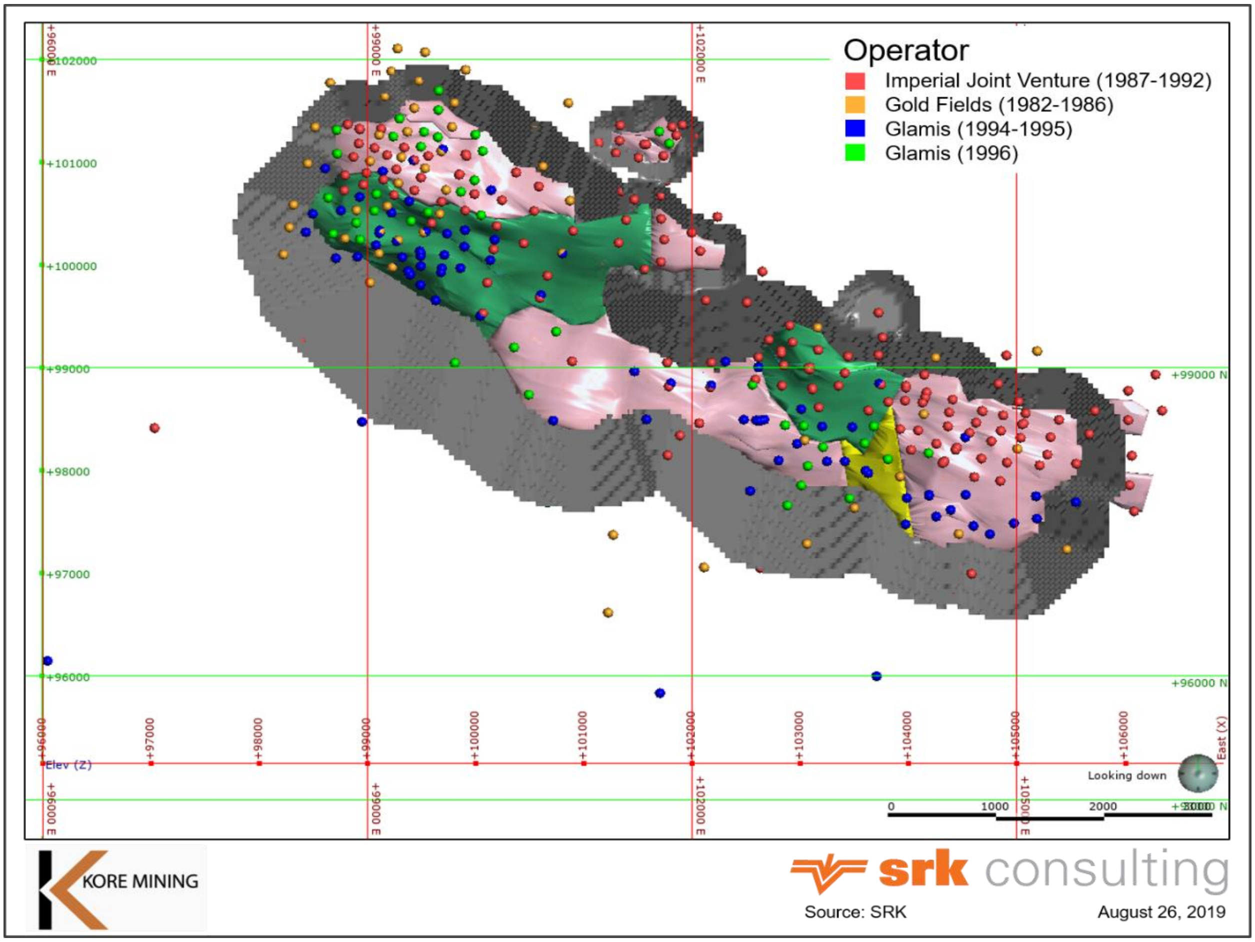
Aside from nine core boreholes, all drilling on the property utilized reverse circulation (RC) methods. Initial RC drilling methods varied with the preference of the operator, the borehole depth and individual borehole conditions. Generally, areas with thick overlying gravel units (greater than 500 ft) required wet drilling methods to prevent borehole wall collapse.

Dry RC drilling methods were utilized when possible during the later drilling programs. Groundwater was encountered at the southern end of the East and West areas, generally at the 100 ft elevation (approximately at 700 ft borehole depth). Groundwater necessitated wet drilling and sampling methods. Later exploration programs utilized dual walled reverse circulation, drilling dry with a tri-cone bit and low air pressure. This combination produced better chip recoveries of 75% to 95%. Samples were collected at five-foot intervals, irrespective of geological contacts.

In 1994 and 1995, a core drilling program was completed by Glamis Gold which included seven HQ (2.5-inch diameter) and two PQ (3.3-inch diameter) holes drilled in the East and West deposits. All core drilling was performed utilizing wireline, triple-tube technology.

Drilling was completed on a local mine grid coordinate system.

Figure 10-1: Plan Map of Drilling on the Imperial Gold Project by Operator in Relation to the Resource Pit Shell and Grade Domains



10.1 Drilling by Gold Fields (1982-1986)

Between 1982 and 1986, reconnaissance drilling by Gold Fields testing gravity high anomalies along a regional gravity trend resulted in the initial mineralized intersections in the Indian Rose (West area), located 2,000 ft west of the original Singer showings, and the Ocotillo (East area), approximately 4,500 ft east-southeast of the West area in a southwesterly trend. The Singer area (or Central area) is located between the East and West areas. These three mineralized zones appeared at the time to potentially be part of the same deposit.

Gold Fields drilled a total of 53 boreholes for 27,880 ft. Boreholes K-77, K-78, K-149 to K 154, and K-156 tested a gravity anomaly trend and intersected gold mineralization in the East area. Individual significant intersection and composite weighted averages were 0.135 oz/t gold from 450 ft to 455 ft in K-77; 0.21 oz/t gold over 140 ft and averaging 0.016 oz/t over 180 ft in K-149; 0.019 oz/t gold over 130 ft in K-153; and 0.035 oz/t gold over 90 ft in K-77. However, the initial investigations suggested the deposit did not meet Gold Fields' corporate criteria for size and grade.

10.2 Drilling by Imperial County Joint Venture (1987-1992)

In 1987, the Imperial County Joint Venture conducted approximately 20,000 ft of RC drilling in the West area, East area, and Golden Queen area (located east of the East area), and on a few gas vapour anomalies. The 17-borehole drilling program tested the southeast continuity of mineralization from the West area to the East area. Five of the boreholes intersected gold mineralization (Nordin, 1988).

In 1989, 32 RC boreholes, totaling 11,265 ft, were drilled in the project area. Eighteen of the boreholes tested the East area, three of the holes tested the Golden Queen area and eleven holes tested three gas vapour anomalies. The pre-existing gravity data were reinterpreted. Gold mineralization was further intersected in the East area and a large alteration zone was intersected in the Golden Queen area (Garagan, 1989).

Exploration in 1990 consisted of the drilling of 44 RC boreholes totaling 22,120 ft. A total of 15,480 ft in 29 boreholes were drilled in the East and West areas. The remaining holes were drilled on gravity anomalies. A resistivity survey was carried out on the horst block between the eastern boundary of the East area and the Golden Queen area. A compilation of the West and East areas was completed. The drilling program intersected significant gold mineralization and resulted in the substantial increase in the size of the resource (Garagan, 1990).

Exploration from July 1991 to February 1992 consisted of 94 RC boreholes totaling 40,705 ft. In addition, geological mapping and sampling were completed, as well as an airborne photographic survey. The objective of the program was to further delineate known mineralized zones in the West and East areas and determine mineralogical and structural characteristics of the zones.

10.3 Drilling by Glamis Gold (1994-1996)

Drilling by Glamis Gold between 1994 and 1996 focused on definition drilling within the East, West, and Central areas. Between 1994 and 1995, definition drilling totaled 86 RC boreholes for 69,368 ft. In 1996, a total of 41 RC boreholes were drilled for 26,260 ft including infill between the East and West areas which were not included in the WSE 1996 FS reserve and resource estimate.

A total of nine HQ (2.5-inch diameter) and four PQ (3.3-inch diameter) core boreholes were drilled in the East and West areas between 1994 and 1995. The core drilling program was dedicated to obtaining bulk mineralized samples and independent metallurgical testwork. The core was also logged for alteration, structural, and geotechnical information and utilized for metallurgical and analytical testing.

10.4 Sampling Method and Approach

To ensure proper collection and assaying of RC borehole cuttings, carefully designed sampling procedures were maintained throughout the drilling programs. To minimize sample contamination, dry drilling, and sampling was utilized wherever possible. Approximately 75% of the total footage drilled was completed with dry drilling and sample collection.

The typical sample collection system used at the Imperial Gold Project consists of an in-line cyclone discharging through a three-tier Jones Splitter. Individual samples weighing approximately 15 pounds were collected at 5 ft intervals.

The sampling system and splitter assembly were thoroughly blown out with compressed air between each sample. Double samples were taken in gneissic units. One sample split was shipped directly to an unnamed independent assay laboratory for fire assay. The remaining split was retained for in-house assay and metallurgical testing.

Wet drilling utilized similar sampling procedures with a rotating wet splitter. Samples were caught on five-foot intervals in buckets lined with sample bags. The sample weight averaged 15 pounds.

The 1994 core sampling procedures consisted of logging, photographing, and sawing the core in half. The split core was separated and bagged into five-foot intervals for independent assay. The remaining core was utilized for metallurgical testing, comparison of adjacent RC borehole assays, overall geotechnical characteristics and rock type apparent bulk density.

1995 core was photographed, logged, analyzed for geotechnical properties and sent to McClelland Laboratory for metallurgical test work. Sampling procedures are described in the metallurgical test work section.

The sampling method and approach utilized during the various drill campaigns appears to be conducted well and supervised by professional geologists.

10.5 SRK Comments

Historical sampling methods and approach are difficult to assess retrospectively. The chip sampling data were meticulously recorded on paper records and later transposed to digital format. Although much of the RC drill chips have not been preserved, representative drill chips from the Glamis Gold drill campaign during 1994 to 1996 were preserved in chip trays (Figure 10-2). The QP was able to check a limited selection of the original paper logs and found these to fairly represent the material in the chip trays and similar to that reflected in the digital logs used for geological and mineral resource modeling.

Based on historical reports, SRK considers that the sampling approach used by the historical operators did not introduce a sampling bias.

In the opinion of the QP, the personnel from Gold Fields, Imperial County Joint Venture and Glamis Gold used industry best practices in the collection of assay samples from drilling. There is no evidence that the sampling approach and methodology used by the historical operators introduced any sampling bias.

Figure 10-2: Preserved Chip Trays From the 1994 Glamis Gold Drill Campaign Reviewed by the Qualified Person

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

11.1 Sample Preparation and Analyses

Sample preparation, analyses and security procedures for historical samples taken by the previous operators, Gold Fields and Glamis Gold, are not specifically documented and therefore difficult to review. The QP of this report understands that some samples were assayed for gold at the Mesquite and Picacho mine laboratories. The preparation and assaying technique were not documented. Assay records are preserved on paper logs, level maps, and sections.

The majority of the gold analysis was conducted by American Assay Laboratory ("AAL") and Chemex Labs Inc. ("Chemex") at undisclosed locations. Chemex is accredited to ISO/IEC standards to provide complete assurance regarding quality performance in sample preparation and analysis. AAL is not accredited. It is believed that Monitor Geochemical Laboratory Inc., Nevada Geochemical Services Inc., and the private laboratories of Gold Fields, and Glamis Gold were also utilized but it is unclear in what capacity.

According to previous reports on the Imperial Gold Project, sampling preparation documentation suggests that the laboratories followed similar sample preparation techniques used most commonly for chip and core samples. Industry standards require that the sample be weighted, dried, and fine crushed to produce a crush product with 70% of the material to be less than 2 millimetres in diameter. A split sample of between 250 grams ("g") to 400 g was pulverized to better than 85% passing 75 microns.

The quantitative analysis of gold followed the industry standard fire assay of a 1-assay-ton sample and analysis by atomic absorption spectrometry or gravimetric finish.

It is unclear whether all laboratories followed the same sample preparation and analytical procedures on samples collected between 1987 and 1996 by various operators.

Verification sampling completed by previous operator Delta was conducted at ALS Canada Ltd. ("ALS Minerals") in North Vancouver, British Columbia in order to verify selected historically sampled intervals. The management system of the ALS Group of laboratories is accredited ISO 9001:2000 by QMI Management Systems Registration. Selected historical sample pulps were delivered to North Vancouver for assaying. The North Vancouver laboratory is accredited ISO/IEC 17025:2005 by the Standards Council of Canada for certain testing procedures, including those used to assay samples submitted by Delta. ALS Minerals also participated in international proficiency tests such as those managed by CANMET and Geostats Pty Ltd.

Verification RC chip samples were prepared for assaying at the ALS Minerals preparation facility using a conventional preparation procedure (dry at 60°C, crushed and sieved to 70% passing 10 mesh ASTM, pulverised to 85% passing 75 micron or better). Prepared samples were then assayed for gold using a conventional fire assay procedure (ICP-AES) on 30-gram sub-samples.

11.2 Density Data

A review of the apparent bulk density data collected from 1994 to 1996 was conducted by the authors of this report. The review was conducted to determine the cause for the apparent bulk density differences for gravel between the 1994 and 1996 data. The following summarizes the results of that review.

The sample preparation and procedure for determining the apparent bulk density for the tested core samples consisted of drying samples at 100°C for 24 hours, cooled at room temperature and weighed on a top loading balance. Samples were weighed with an accuracy of approximately 1.0 grams. After weighing, each sample was coated with a thin film of paraffin wax in order to eliminate any excess moisture. Each individual sample was then immersed in a receptacle that allowed for the containment of the overflow of distilled & degassed water. The overflow volume was measured and recorded.

Appendix A of the Western States Engineering (1996) report contains an “ore” reserves estimate conducted by Mine Reserves Associates Inc. (“MRA”) It reported tonnage factors (ft^3/ton) results for “ore” at 13.00 ft^3/ton, waste at 13.10 ft^3/ton and gravel at 14.90 ft^3/ton (Table 11-1).

The 1994 apparent bulk density reported for gravel of 16.50 ft^3/ton was based on averaging two samples, whereas the 1996 gravel density of 14.90 ft^3/ton was based on the average of 17 samples. Therefore, the 1996 gravel density average of 14.90 ft^3/ton is more representative of the apparent bulk density. Delta’s check of the 17 gravel, conglomerate/gravel samples yielded an average of 14.93 ft^3/ton.

The density checks by Delta appear to match reasonably well with the results reported by WSE (1996). The QP applied the same tonnage factors in the current resource estimate to that used by WSE (1996). The QP recommends however, that more mineralized material and waste density measurements be collected during future drill campaigns.

Table 11-1: Density Results Reported by WSE (1996)

Rock Type	Range (ft^3/ton)	Tonnage Factor (ft^3/ton)	Density (ton/ft^3)
Mineralized Rock*		13.00	0.077
Biotite Gneiss (6 samples assaying > 0.007oz/t Au.)	11.82-14.80	12.83	0.078
Sericite Gneiss (3 samples assaying > 0.007oz/t Au.)	12.81-14.36	13.41	0.075
Waste**		13.10	0.076
Biotite Gneiss (7 samples assaying < 0.007oz/t Au.)	11.57-14. 76	12.90	0.078
Sericite Gneiss (4 samples assaying < 0.007oz/t Au.)	13.20-15.60	14.26	0.070
Combined Biotite Gneiss and Sericite Gneiss***		13.23	0.076
Biotite Gneiss (combined 13 samples)	11.57-14.80	12.87	0.078
Sericite Gneiss (combined 7 samples)	12.81-15.60	13.90	0.072
Volcanics (4 samples)	12.5-15.95	14.16	0.071
Gravel		14.90	0.067
Gravel (combined 17 samples)****		14.93	0.067

*Note, the combined density for the biotite gneiss and the sericite gneiss samples assaying greater than 0.007 oz/t gold averages 13.03 ft^3/ton.

**Note, the combined density for the biotite gneiss and the sericite gneiss samples assaying less than 0.007 oz/t gold averages 13.40 ft^3/ton.

***Note, the combined density for all of the biotite gneiss and sericite gneiss samples averages 13.23 ft^3/ton.

****Note, the combined density for all of the gravel and conglomerate/gravel samples averages 14.93 ft^3/ton.

11.3 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of the exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for the project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying processes. They are also important to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and insertion of quality control samples. Check assaying is typically performed as an additional reliability test of assaying results. This typically involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

There are too few records available to the QP to indicate if specific analytical quality control measures were implemented by previous operators. It does not appear that any of the previous operators inserted external quality control samples to their sample streams.

There are no records of assay checks being conducted by a second laboratory during drilling campaigns between 1984 and 1990. However, internal pulp duplicate samples assays were conducted approximately every 15 to 20 samples by AAL. AAL also inserted two standards and one blank per batch of 50 samples. It is believed that most reputable laboratories used similar quality control standards between 1984 and 1990.

A selection of field duplicates (92 pairs) and umpire check assays from the 1991 to 1992 drilling program by the Imperial County Joint Venture were recovered by Delta.

WSE (1996) reported that check assay analysis was conducted using information from the pre-feasibility and feasibility drilling programs. AAL was the primary laboratory used by Glamis Gold with checks conducted by Chemex. Neither the QP nor KORE has been able to review this data.

11.4 SRK Comments

In the opinion of the authors of this report, although some of the sample preparation, security, and analytical procedures used by previous operators is poorly documented and therefore difficult to assess, the QP has undertaken sufficient independent checks on data quality to consider that the drilling data is adequate for geological and mineral resource modeling. Analytical quality control measures implemented on the Imperial Gold Project by previous operators included field duplicates and umpire check assays in 1991-1992 and umpire check assays in 1994-1996.

12.0 DATA VERIFICATION

The content of this section has been directly re-produced from the SRK (2019) technical report, which is still considered current.

12.1 Verifications by Previous Operators

There are too few records available to indicate if specific analytical quality control measures were implemented by previous operators. Imperial County Joint Venture sampled field duplicates and umpire check assays in 1991-1992 as well as umpire check assays in 1994-1996.

WSE (1996) report that check assay analysis were conducted using information from the pre-feasibility and feasibility drilling programs. AAL was the primary laboratory used by Glamis Gold with checks conducted by Chemex. Check assay comparisons were limited to samples greater than or equal to 0.005 oz/t Au and any obvious outliers were eliminated prior to analysis. The Wilcoxon Matched Pairs non-parametric test was used. Pre-feasibility results showed no bias between the AAL and Chemex laboratories (WSE, 1996). Feasibility results did show a statistical bias with AAL, showing an average higher grade on the order of 0.001 oz/t Au. Neither the QP nor Kore has been able to review this data.

Assay certificates from the pre-feasibility drilling campaign were spot checked by MDA. It was MDA's opinion that the transfer of assay information from the certificates to the computer database appeared to have been done with care and that the database can be assumed to be an accurate representation of the original assay certificates.

12.2 Verifications by SRK

12.2.1 Introduction

The SRK QP, in collaboration with previous operator Delta, reviewed the available reports, files and limited RC chip boxes and drill pulps in a Goldcorp storage facility in Yuma, Arizona in 2012 to determine the following:

- What quality assurance and quality control programs were implemented during the exploration campaigns between 1984 and 1996;
- To validate transcribing of approximately 100 assay certificate results to the digital borehole database;
- To collect 24 drill pulps from the mineralized horizon in the East and West areas in order to check the precision and accuracy of the results by submitting the pulps to an umpire laboratory.

Approximately 50 pages of AAL assay certificates were examined by the QP and the internal pulp duplicates within these pages were consistently within 20% of the original assay. Approximately 100 assay certificate results were compared to assays within the digital borehole database and no errors were found in transcribing the information. However, in a single case, the slightly higher duplicate value rather than the original value was entered into the database. Subsequently, one of the 2012 pulps showed a transcribing error from the assay certificate to the digital borehole database. The 2012 follow-up assay check for this pulp showed that the original AAL assay result was acceptable.

12.2.2 Site Visit

In accordance with NI 43-101 guidelines, QPs visited the Imperial Gold Project site between February 9 and 10, 2012 and more recently on November 26, 2019. In addition to inspecting the project site and access roads, two consultants from SRK (Mr. Anoush Ebrahimi, PEng and Mr. Glen Cole, PGeo) and a KORE Mining representative (Mr. Dan Purvance) visited a storage locker in Yuma, Arizona on November 26, 2019 where drill core samples and project documentation (maps, sections, reports, correspondence, and data) were inspected. The authors of this report believe they were given full access to all relevant data. All aspects that could materially impact the integrity of the resource data were reviewed.

The chip boxes from various historical RC holes were examined by the QP. The degree of alteration, oxidation, quartz, and sulphide content was checked against the logs showing the auriferous intervals. No discrepancies were found by the QP between the observations on the chip samples and the entries in the paper log sheets and digital database. The QP also examined split core from several boreholes and found the logging information to accurately reflect actual drill core (Figure 12-1).

Figure 12-1: Preserved Split Core Boxes Located in the Yuma Storage Facility

All the project data within the Yuma storage Facility was examined. This data and information included paper log sheets, geology maps, land holdings plans, historical project reports from all disciplines and historical RC sample pulps.

On November 26, 2019, QPs of this technical report also interviewed Mr. Dan Purvance, a former project geologist and former employee of Glamis Gold who was personally responsible for the generation of much of the project data used in the mineral resource estimate. Mr. Purvance described the drilling and sampling procedures undertaken on the project. The QPs are satisfied that these procedures reflect that described in this technical report.

12.2.3 Verifications of Analytical Quality Control Data

The QP reviewed exploration spreadsheet data. This database aggregated the assay results for the quality control samples received from the historical borehole database. the QP aggregated the assay results for the external quality control samples for further analysis. No sample blanks or certified reference materials are known to have been inserted with borehole samples on the Imperial Gold Project.

External analytical quality control data analyzed by the QP included:

- blind field duplicates from 1991 to 1992 drilling (92 pairs),
- umpire check assays also from 1991 to 1992 sampling (77 pairs), and
- verification sampling conducted on RC samples from 1994 to 1996 (24 pairs).

This paired data was analyzed by the QP using bias charts, quantile-quantile and relative precision plots. Analytical quality control data are summarized in graphical format in Appendix C.

Historical paired assay data from 1991 to 1992 produced by Chemex and examined by the QP suggest that gold grades can be reasonably reproduced despite the small population of data pairs. Rank half absolute difference ("HARD") plots suggest that 62% of the blind RC field duplicate sample pairs and 59.7% of the umpire check assay sample pairs sent to Monitor Geochemical have HARD below 10%. Quantile-quantile plots show acceptable reproducibility for both types of duplicate pairs. However, a bias towards higher values in the original assays is apparent at values above 0.5 oz/t gold in two blind field duplicate pairs which is likely attributed to a nugget effect. The QP does not consider this bias material. In general, however, the reproducibility is worse nearing the detection limits, as expected.

The 24 samples submitted to ALS Minerals in 2012 show good reproducibility. These samples, originally collected in 1994-1996 by Glamis Gold, show that only four samples have a HARD above 10% and only one sample above 20%. The QP considers this encouraging in the process of validating the original dataset. However, the dataset of 24 samples is currently insufficient and the QP recommends that further assay verification checks be undertaken.

In the opinion of the QP, that although limited in number, the analytical data available for the Imperial Gold Project does not present evidence of bias and the QP, therefore, concludes that despite the lack of extensive analytical quality control data for a portion of the exploration database, the analytical data are sufficiently reliable to support geology and resource modelling.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

A series of metallurgical tests and analysis were conducted on the Imperial Project material between 1988 and 1996. No additional test work has been undertaken on the deposit since that time. Given the age of the test work, the fragmented nature of reports, the lack of definition of the samples employed, and the lack of significant column leach testing, additional metallurgical testing is recommended for pre-feasibility or feasibility studies to confirm the performance of the deposit and the metallurgical assumptions employed.

13.1 Mineralogy

Mineralogical studies were conducted on Imperial project samples site by PMET Laboratories (Pittsburgh Mineral & Environmental Technology, Inc. (PMET), 1995). Tests included microscopy analyses (both optical and SEM-EDX), X-Ray Diffraction tests, particle size analysis, fire assays, and gravimetric tests on two drill-core composites, as well as thin-section petrographic analyses on three individual (drill-core) rock samples. The five analyzed samples are described in Table 13-1.

Table 13-1: Summary of Samples Used for Mineralogical Analyses (PMET Laboratories, 1995)

Name	Description	Assay (g/t)	Weight (g)
BGN-1	West Pit Composite, Biotite Gneiss, crushed split of assayed core; major ore type used in column tests	0.686	14,061
SGN-1	East Pit Composite, Sericite Gneiss, crushed split of assayed core, 2nd major ore type used in column tests	0.514	11,793
IP-1	Rock sample, Siliceous Breccia Unit		396
IP-2	Rock sample, Fractured Biotite Gneiss		407
IP-3	Rock sample, Altered Gneiss		177

The composite samples were subjected to SEM-EDX and "microscopic modal" analyses, which revealed their compositions. The results are shown in Table 13-2.

Table 13-2: Summary of Chemical Analyses Performed on Composite Samples

Composition	BGN-1	SGN-1
Fe_2O_3	8.26% (Mass)	3.96%
MnO	0.59%	0.24%
TiO_2	0.87%	0.66%
CaO	5.50%	0.77%
K_2O	6.75%	7.57%
SiO_2	53.93%	67.34%
Al_2O_3	14.99%	14.38%
MgO	1.97%	1.10%
Na_2O	3.60%	2.20%
Sulphides	< 0.1% (Vol)	0.6%
Iron Oxides	14%	4%
Gangue	86%	95.3%

(PMET Laboratories, 1995)

Petrographic analyses were conducted on the drill-core samples IP-1, IP-2, and IP-3 (Chemgold, Inc., 1995). Mineralogical classifications were assigned to each of these, and the composition of the samples were determined. The results are given in Table 13-3.

Table 13-3: Results of Petrographic Analyses on Rock Samples

IP-1		IP-2		IP-3	
Initial Classification	Siliceous Breccia Unit	Initial Classification	Fractured Biotite Gneiss	Initial Classification	Altered Gneiss
K-Feldspar	28.3%	Plagioclase [3]	54.7%	Quartz	44.2%
Plagioclase	26.8%	Hematite / Goethite [4]	17.6%	Carbonate	20.6%
Quartz	18.9%	Quartz	9.8%	Plagioclase	15.4%
Goethite	11.0%	Sericite	6.1%	Muscovite	8.2%
Carbonate [1]	8.0%	Pyrite [5]	4.7%	Goethite / Hematite	7.8%
Sericite [2]	5.5%	Fe-Carbonates	4.0%	Sericite	2.4%
Hematite	1.1%	K-Feldspar	2.1%	K-Feldspar	1.4%
Misc.	0.4%	Misc.	1.0%		
Final Classification	Granite Breccia	Final Classification	Plagioclase Gneiss	Final Classification	Quartz-Feldspar Gneiss

1: Goethite rich calcite and siderite
2: Plagioclase alteration
3: Heavily altered, differentiating between plagioclase and feldspar was difficult
4: Occurred in veinlets and fractured rock matrix
5. Possibly elevated levels of Pyrite not representative of bulk material
(PMET Laboratories 1995, Chemgold, Inc., 1995)

Gravimetric tests were performed on the composite samples (BGN-1 and SGN-1) to determine gold deportment by particle size. Each sample was ground to a 48-mesh size and all "-400 mesh" material (fine slimes) were removed. The -48 to +400 material was subjected to "super-panning" to separate it into size fractions, which were then assayed. The results of the tests are given below in Table 13-4.

Table 13-4: Results of Gravimetric Tests on Composite Samples

	BGN-1			SGN-1		
Product	Weight (%)	Gold Assay (g/t)	Gold Distribution (%)	Weight (%)	Gold Assay (g/t)	Gold Distribution (%)
Calculated Head	100	0.446	100	100	0.411	100
Gravity Concentration	2.28	0.754	3.59	1.06	3.086	8.22
Gravity Tails	69.59	0.206	29.91	77.50	0.206	40.06
-400 Fine Slimes	28.13	1.131	66.50	21.44	0.960	51.72

(PMET Laboratories, 1995)

As shown above, only minor amounts of gold reported to the gravity concentrates. Over 65% of the gold from the BGN material reported to the fine slimes that were removed prior to gravity separation; similarly, over 50% of the gold from the SGN material reported to fine slimes. Approximately 30 – 40% of gold reported to gravity separation tails, indicating that traditional gravity separation may not be effective.

Overall, the gravity tests showed that the gold present in the BGN and SGN material was relatively fine grained.

Optical analysis (microscopy) of the BGN-1 material indicated that the Biotitic material contained a high level of iron oxides and hydroxides in the veinlets and fractured rock matrix of the sample. It was concluded that it is highly likely that gold was associated with these iron oxides/hydroxides and that these compounds were the result of prior pyrite oxidation. The hematite occurred with biotite alteration and feldspar, and within fractured quartz-feldspar matrix as coarse, specular hematite. Specular hematite was found to be coarse, approximately 50 – 60 µm in diameter, but could be as fine as 1 – 8 µm, or as coarse as 500 µm.

Analysis of the Biotite material found that the brittle/fractured rock matrix could result in increased permeability of lixiviants for gold leaching. No significant presence of sulphides were found. The Biotite material was found to be extremely oxidized, with hematite, limonitic material, magnetite, goethite, jarosite, manganese oxides, quartz-feldspar-biotite matrix, calcite, and "micaceous alteration" such as muscovite and chlorite.

Optical analysis of the SGN-1 material indicated that it consisted of strongly oxidized Sericite Gneiss, some hydrous iron oxides, and a quartz-sericite-Na-feldspar-K-feldspar matrix, with chlorite and trace amounts of calcite. Gold was found to be extremely finely disseminated through the fractured rock matrix, encapsulated by quartz and sericite. The average size of gold particles was found to be 1 – 10 µm, with some gold grains as coarse as 45 µm. The SGN-1 material contained a significantly higher proportion of silica, and slightly higher amounts of sulphides compared to the BGN-1 material.

Small amounts of both carbonaceous material and mercury were found in several samples as shown in Table 13-5 (Chemex Labs, Inc., 1994) and Table 13-6 (Chemgold, Inc., 1995).

Table 13-5: Results of Carbon Assays Performed on Imperial Project Material

Sample	Total Carbon (%)	Inorganic Carbon (%)	Total Organic Carbon (%)	Total Sulphur (%)
BGN Feed	0.77	0.65	0.1	< 0.01
SGN High Grade	0.10	< 0.65	0.1	0.03
SGN Low Grade	0.17	0.05	0.1	0.01

(Chemex Labs, Inc., 1994)

Table 13-6: Results of Mercury Assays Performed on Imperial Project Core Samples

Sample	Date Tested / Sampled	Mercury (PPB)	Comments	Grade (g/t)
WP-1	10/11/1994	600	BR Tails	0.926
WP-2	10/11/1994	700	BR Tails	0.754
WP-3	10/11/1994	300		0.034
WP-4	10/11/1994	< 200		
WP-5	10/11/1994	< 200		
EP-1	10/11/1994	200	BR Tails	0.617
EP-2	10/11/1994	< 200	BR Tails	0.514
EP-3	10/11/1994	800	BR Tails	0.754
EP-4	10/11/1994	400	BR Tails	0.789
EP-5	10/11/1994	200		0.069

Sample	Date Tested / Sampled	Mercury (PPB)	Comments	Grade (g/t)
EP-6	10/11/1994	300		0.034
EP-7	10/11/1994	200		0.000
EP-8	10/11/1994	200		
WP-6	2/21/1995	0.6	BR Tails	0.651
WP-7	2/21/1995	< 200		0.103
WP-8	2/21/1995	< 200	BR Tails	1.063
WP-9	2/21/1995	< 200		
EP-9	2/21/1995	< 200	BR Tails	1.371
EP-10	2/21/1995	< 200		0.034
EP-11	2/21/1995	< 200	BR Tails	0.754
Average		< 285		0.529

(Chemgold, Inc., 1995)

Further analysis of organic carbon and mercury should be considered for subsequent samples given the occasional occurrence in the samples analyzed.

13.2 Bulk Density Measurements

Multiple bulk density measurements have been conducted on the Imperial deposit (Chemgold, Inc., 1995; Chemgold, Inc., 1994; Chemgold, Inc., 1995; McClelland Laboratories, Inc., 1996; McClelland Laboratories, Inc., 1995; McClelland Laboratories, Inc., 1995). The results ranged from approximately 1,600 kg/m^3 for sericite gneiss material to approximately 3,200 kg/m^3 for quartz breccia material with an abundance of calcite. The results were segregated by deposit area: the average bulk density of the Western area of the deposit was 2,417 kg/m^3, and 2,183 kg/m^3 for the Eastern area. The average for all results is 2,283 kg/m^3. Table 13-7 and Table 13-8 show the bulk density measurements.

Table 13-7: Bulk Density Measurements East and West Areas of the Imperial Deposit

Hole Number	Rock Type	Interval (m)	Specific Gravity		Avg.	Grade (g/t)	Bulk Density	
			Volume Displ. Method	Weight Diff. Method			kg/m^3	m^3/t
95WC-4	Conglomeration	18.9	2.59	2.58	2.59	0.069	2589	0.386
95WC-4	BGN	72.5	2.44	2.44	2.44	0.171	2459	0.407
95WC-5	Basalt	100.3	2.38	2.38	2.38	0.034	2428	0.412
95WC-5	BGN	114.9	2.68	2.68	2.68	0.411	2688	0.372
95WC-5	Quartz Breccia	207.3	2.44	2.44	2.44		2428	0.412
94WC-3	Conglomeration	27.4	2.15	2.15	2.15		2159	0.463
95WC-4	Conglomeration	22.6	2.42	2.42	2.42		2439	0.410
95WC-4		46.6	2.16	2.14	2.15		2159	0.463
95WC-4		101.8	2.35	2.30	2.33		2348	0.426
WC-1	BGN					0.446	1927	0.519
WC-2	BGN					0.583	2680	0.373
WC-3	BGN					0.651	1866	0.536
Average (West)			2.40	2.39	2.40		2348	0.432
95EC-3	Conglomeration	68.9	2.42	2.42	2.42	0.034	2419	0.413

Hole Number	Rock Type	Interval (m)	Specific Gravity Volume Displ. Method	Specific Gravity Weight Diff. Method	Avg.	Grade (g/t)	Bulk Density kg/m³	Bulk Density m³/t
95EC-3	SGN	115.8	2.45	2.45	2.45	0.034	2449	0.408
95EC-3	Basalt	150.3	2.49	2.49	2.49	0.000	2558	0.391
95EC-3	SGN	187.1	2.06	2.06	2.06	0.171	2058	0.486
94EC-1	Conglomeration	11.3	2.10	2.10	2.10		2089	0.479
94EC-1	Conglomeration	18.6	2.14	2.14	2.14		2139	0.467
94EC-1	Conglomeration	20.1	2.09	2.09	2.09		2109	0.474
94EC-1	Conglomeration	25.9	2.02	2.02	2.02		2029	0.493
94EC-1	Conglomeration	77.7	2.29	2.29	2.29		2279	0.439
95EC-3			2.53	2.52	2.53		2529	0.395
EC-1&2	SGN					0.377	1592	0.628
EC-1&2	BGN					0.000	1629	0.614
Average (East)			2.26	2.26	2.26		2156.69	0.474

Table 13-8: Bulk Density Measurements Imperial Project Deposit

Sample #	Rock Type	Area	Hole	Depth (m)	Grade (g/t)	Bulk Density kg/m³	Bulk Density m³/t	Comments
1	Gravel	West	WC-3	22.25	0.000	1900	0.526	Full core, cemented
2	Gravel	East	EC-1	26.82	0.000	1972	0.507	SAA
3	Ftwall Conglomerate	West	WC-1	149.05	0.000	2267	0.441	SAA
4	Volcanic	West	WC-1	25.60	0.000	2135	0.468	Full core, Breccia texture
5	Volcanic	West	WC-1	15.85	0.000	2009	0.498	SAA
6	Unmin SGN	East	EC-1	48.77	0.171	2417	0.414	Full core, unbrecciated
7	Unmin SGN	East	EC-1	63.40	0.103	2137	0.468	Full core, brecciated
8	Min SGN	East	EC-1	57.91	0.411	2451	0.408	Partial core
9	Min SGN	East	EC-2A	96.01	2.229	2500	0.400	SAA
10	Min SGN	East	EC-2A	99.97	0.960	2231	0.448	SAA
11	Unmin BGN	East	EC-1	91.44	0.206	2171	0.461	+Quartz Biotite/Sericite
12	Unmin BGN	West	WC-1	35.05	0.651	2478	0.404	Strong Hematite
13	Unmin BGN	West	WC-1	41.76	0.069	2770	0.361	Unaltered, blocky
14	Min BGN	West	WC-2	19.20	1.029	2592	0.386	Oxide/Hematitic Breccia
15	Min BGN	West	WC-2	105.16	0.069	2651	0.377	+Blocky, hematitic
16	Min BGN	West	WC-1	82.60	0.583	2432	0.411	Strong hematitic veinlets
17	Min BGN	West	WC-3	93.27	0.137	2542	0.393	Sheared, hard
18	Granite Pegmatite	West	WC-3	60.96	0.000	2840	0.352	Full core, pegmatite

Sample #	Rock Type	Area	Hole	Depth (m)	Grade (g/t)	Bulk Density kg/m³	Bulk Density m³/t	Comments
19	Quartz Breccia Dike	West	WC-3	120.09	1.063	3217	0.311	+Calcite cement
20	Min BGN	East	EC-2A ore comp.		0.000	2710	0.369	Composite, +1" material
A	Min BGN?	East	EC-2A	13.41		1230	0.813	Rock
B	Min BGN?	East	EC-2A	13.41		2540	0.394	Fines
C	Min BGN?	East	EC-2A	45.42		2250	0.444	
D	Min BGN?	East	EC-2A	47.55		1970	0.508	
E	Min BGN?	East	EC-2A	48.46		2170	0.461	
F	Min BGN?	East	EC-2A	49.07		2150	0.465	
G	Min BGN?	East	EC-2A	53.04		2100	0.476	
H	Min BGN?	East	EC-2A	56.08		2060	0.485	
I	Min BGN?	East	EC-2A	58.83		2080	0.481	
				Average		2309.34	0.433	

(Chemgold, Inc.; McClelland Laboratories, Inc.)

13.3 Indian Rose Zone Testing (1988 – 1994)

A range of tests were performed on samples taken from the Indian Rose area. According to a previous PEA report by SRK Consulting, the Indian Rose area was part of the West mineralized zone (SRK Consulting (Canada) Inc., 2012). Current project maps of the area indicate that the boundary of the Imperial Project now at least partially intersects what was then known as the Indian Rose deposit. A summary of these results is presented below.

13.3.1 1988 Bottle Roll Tests

Fourteen samples from the Indian Rose zone were coarse cyanide leached (-10 mesh) in 1988 (Chemgold, Inc., 1991). According to the procedure given in the report, bulk samples were blended using a traditional tarp rolling technique to mix material. Approximately 1,200 g of material was used with 1,500 mL of process waster, 2.5 g of caustic soda (NaOH), and 2.0 g of sodium cyanide (NaCN). The material was rolled and leached for 72 hours without solution removal or chemical addition. The tests results were calculated based on fire assay (FA) and hot cyanide assays (HCL) as shown by the calculations below:

Fire Assay Recovery = 100 x (FA Head Grade – FA Tails Grade) / FA Head Grade

Solution Recovery = 100 x (Solution Head Grade – FA Tails Grade) / Solution Head Grade

HCL Recovery = 100 x (HCL Head Grade – FA Tails Grade) / HCL Head Grade

The HCL recovery technique is designed to provide an indicative recovery of what a heap leach should provide when utilized with a comparable column leach result. This method needs to be utilized in conjunction with column leach tests to provide a suitable basis for extrapolating the HCL recovery to field performance. The method is often used by producing mines as a method to provide predictive recovery without the long duration of a column test. The results of these tests are shown in Table 13-9.

Table 13-9: 1988 Bottle Roll Test on Material from the Indian Rose Area (Chemgold, Inc.)

Sample #	Interval (m)		Hot Cyanide Leach			Fire Assay				Rec by Sol			
	From	To	Pulp Head (g/t)	Pulp Tails (g/t)	Calc Rec	Head Grade (g/t)	Pulp Head (g/t)	Pulp Tails (g/t)	Calc Rec	Calc Rec	CN Cons (kg/t)	NaOH Cons (kg/t)	Rock Type
R-16	56.39	57.91	1.097	0.103	90.6%	1.029	0.720	0.000	100.0%	89.7%	0.435	0.85	Tertiary Volcanics
R-16	57.91	59.44	0.583	0.000	100.0%	0.480	0.583	0.000	100.0%	100.0%	0.155	0.66	Tertiary Volcanics
K-12	36.58	38.10	0.549	0.103	81.3%	0.411	0.446	0.000	100.0%	72.7%	0.05	1.19	Felsic Gneiss
K-12	59.44	60.96	0.446	0.034	92.3%	0.446	0.309	0.000	100.0%	90.9%	0.05	1.245	Felsic Gneiss
K-12	76.20	77.72	0.446	0.034	92.3%	0.274	0.309	0.000	100.0%	90.9%	0.20	1.15	Felsic Gneiss
K-21	51.82	53.34	4.149	0.137	96.7%	4.183	4.457	0.171	96.2%	95.6%	0.00	0.7	Hematitic Gneiss
K-21	54.86	56.39	6.754	0.343	94.9%	8.366	8.057	0.583	92.8%	93.8%	0.04	0.8	Hematitic Gneiss
K-21	67.06	68.58	1.029	0.034	96.7%	1.097	0.651	0.000	100.0%	95.8%	0.04	0.81	Felsic Gneiss
K-21	83.82	85.34	0.549	0.034	93.8%	0.754	0.411	0.000	100.0%	93.8%	0.04	0.98	Felsic Gneiss
K-22	41.15	42.67	1.097	0.240	78.1%	1.269	0.754	0.240	68.2%	80.6%	0.04	1.12	Felsic Gneiss
K-34	47.24	48.77	0.514	0.069	86.7%	0.651	0.446	0.000	100.0%	83.3%	0.04	1.03	Quartz. Biotitic Gneiss
K-34	65.53	67.06	1.440	0.137	90.5%	1.474	0.891	0.000	100.0%	87.5%	0.04	0.725	Quartz. Biotitic Gneiss
K-40	86.87	88.39	0.651	0.000	100.0%	0.720	0.343	0.000	100.0%	100.0%	0.08	0.725	Quartz. Biotitic Schist
K-40	117.35	118.87	1.234	0.069	94.4%	1.440	0.823	0.000	100.0%	93.3%	0.08	1.01	Chloritic Biotitic Schist
Average			1.467	0.096	92.0%	1.614	1.371	0.071	96.9%	90.6%	0.092	0.928	

The results indicate that direct coarse cyanidation results in high recovery with the fire assay (FA) recovery averaging 96.9%, 92% for the HCL method and 90.6% for the solution method. The location of the samples is not known but the original report indicates that a map of the sample locations was originally supplied.

13.3.2 1991 Bottle Rolls and Fractional Assays

Four "oxidized" samples from the Indian Rose area were separated by size fraction after crushing to -10 mesh and then subjected to Hot Cyanide Leach tests (Chemgold, Inc., 1991). The results of the bottle roll tests are shown in Table 13-10. The resultant mass distributions were analyzed to determine the deportment of gold. Both size-weighted fire assay and HCL pulp/solution techniques were used to determine initial head grade for each sample. The results of the tests are given in Table 13-11.

Table 13-10: 1991 Bottle Roll Tests Oxidized Samples from the Indian Rose Area (Chemgold, Inc.)

	Hot Cyanide Leach			Fire Assay			Rec by Sol	
Sample #	Pulp Head (g/t)	Pulp Tails (g/t)	Calc Rec	Pulp Head (g/t)	Pulp Tails (g/t)	Calc Rec	Calc Rec	Rock Type
84517	1.337	0.240	82.1	1.337	0.343	74.4	77	Chloritized Muscovite Schist
84519	0.994	0.137	86.2	1.371	0.103	92.5	90	Chloritized Muscovite Schist
169059	1.646	0.137	91.7	1.817	0.274	84.9	94	Biotite Gneiss
169060	1.509	0.103	93.2	1.509	0.206	86.4	87	Biotite Gneiss
Average Schist	1.166	0.189	84.1	1.354	0.223	83.4	83.5	
Average Biotite	1.337	0.146	88.7	1.431	0.214	84.9	85.3	

Table 13-11: 1991 Sieved Oxidized Gneiss Samples Gold Deportment from the Indian Rose Area (Chemgold, Inc.)

						Average Head Grade (g/t)	
Sample	Particle Size Distribution	Mass Distribution	Gold Assay (HCL) (g/t)	Particle Distribution %	Gold Within Size (%)	Size-Weighted Assay (HCL)	Calculated (HCL)
84517	-4+10	8.5%	20.469	9%	51%	3.562	1.337
	-10+100	52.8%	1.783	55%	28%		
	-100+200	37.4%	0.514	20%	4%		
	-200	54.8%	3.977	16%	17%		
84519	-4+10	4.0%	0.651	4%	4%	0.774	0.994
	-10+100	51.6%	0.274	54%	19%		
	-100+200	21.0%	0.171	22%	5%		
	-200	19.1%	2.811	20%	72%		
169059	-4+10	21.4%	2.331	24%	34%	1.612	1.646
	-10+100	37.4%	1.543	40%	39%		
	-100+200	9.5%	0.343	10%	2%		
	-200	9.3%	1.543	26%	25%		
169060	-4+10	16.4%	2.263	17%	24%	1.564	1.509
	-10+100	54.8%	1.097	56%	39%		
	-100+200	9.3%	0.857	9%	5%		
	-200	17.9%	2.709	18%	32%		

The results indicate that the gold deportment within the samples varied considerably. Samples 84517 and 84519 had virtually the same particle size distributions, but highly variable gold deportment. Samples 169059 and 169060 shared similar particle sizes, and similar gold deportments. The samples employed in these tests were also assayed by hot cyanide leaching (HCL) to give an extractable gold grade indication.

13.3.3 1992 Bottle Roll Leach Tests on Ocotillo and Indian Rose Samples

Ten samples from the Indian Rose (West) zone, and five samples from the Ocotillo (East) zone were taken from what was then the Imperial County J. V. property (Chemgold, Inc., 1992). The samples originated from mixed oxidized/non-oxidized mineral zones according to the test reports. The material was crushed to 10 mesh and leached for 72 hours. Both Fire Assay and HCL pulp/solution techniques were used to determine head grade, tail grade, and final recoveries. The results of these tests are given in Table 13-12.

Table 13-12: 1991 Bottle Roll Tests on Mixed Samples from the Indian Rose and Ocotillo Zones

		Interval (m)			HCL Assay (g/t)			Fire Assay (g/t)		Solution		
Area	Hole	To	From	Sample #	Head	Tail	Recovery (%)	Head	Tail	Recovery (%)	Recovery (%)	Rock Type
Ocotillo (East)	O-47			40631	1.029	0.137	87	1.029	0.240	77		
Ocotillo (East)	O-47			40628	0.514	0.103	80	0.583	0.103	82		
Ocotillo (East)	O-47			40632	2.263	0.411	82	2.880	0.549	81		
Ocotillo (East)	O-47			40633	2.640	0.240	91	3.017	0.343	89		
Ocotillo (East)	O-47			40642	0.720	0.171	76	0.857	0.274	68		
Indian Rose (West)	O-4	260	265	84517	1.337	0.240	82	1.337	0.343	74	77	Chloritized Muscovite Schist
Indian Rose (West)	O-4	270	275	84519	0.994	0.137	86	1.371	0.103	93	90	Chloritized Muscovite Schist
Indian Rose (West)	R-16	465	470	169059	1.646	0.137	92	1.817	0.274	85	94	Biotite Gneiss
Indian Rose (West)	R-16	470	475	169060	1.509	0.103	93	1.509	0.206	86	87	Biotite Gneiss
Indian Rose (West)	O-31	295	297.5	83730	1.131	0.069	94	1.097	0.000	100	100	Micaceous Schist
Indian Rose (West)	O-31	295	297.5	83733	0.446	0.103	77	0.411	0.000	100	100	Micaceous Schist
Indian Rose (West)	O-30	215	220	83194	1.131	0.069	94	1.029	0.137	87	81	Chloritic Schist
Indian Rose (West)	O-15	270	275	104206	0.754	0.034	95	0.583	0.137	76	71	Quartz Sericite Schist
Indian Rose (West)	R-16	475	480	169061	1.714	0.069	96	1.646	0.137	92	89	Chloritic Schist
Indian Rose (West)	R-16	490	495	169064	0.549	0.069	88	0.617	0.103	83	78	Biotite Gneiss
Average				Overall	1.225	0.139	87.5	1.319	0.197	84.9	86.7	
				Ocotillo	1.433	0.213	83.2	1.673	0.302	79.4	-	
				Indian Rose	1.121	0.103	89.7	1.142	0.144	87.6	86.7	

(Chemgold, Inc.)

13.3.4 1994 Bottle Roll Test on Indian Rose Samples

Six bottle roll tests were completed from the Indian Rose/Imperial Project area: two samples with the label "WP", and four labelled "EP" (Chemgold, Inc., 1994). These are presumed to be from the West Pit and East Pit of the mine at the time, respectively.

The tests were conducted using the hot cyanide assay technique. The tests employed 1,200 g of feed material (the size was not reported), 1,800 mL of process water, 1.0 g of lime to control pH at 11, and 1.8 g of sodium cyanide added to the leach. The leach was run for 72 hours before filtering, and solution and tails were assayed.

No silver was recorded in the pregnant leach solutions of any of the samples. The pregnant leach solution (PLS) from samples EP2 – EP4 showed an average copper concentration of 0.45 ppm, indicating little or no copper present in the material. Table 13-13 provides a summary of the results.

Table 13-13: 1994 Bottle Roll Tests on Indian Rose/Imperial Project Material (Chemgold, Inc.)

Sample #	Head Gold Grade (HCL) (g/t)	Recovery (%)	CN Consumed (kg/t)
WP-1	0.823	86	0.14
WP-2	0.754	94	0.18
EP-1	0.377	81	0.22
EP-2	0.309	81	0.22
EP-3	0.549	77	0.20
EP-4	0.480	91	0.20
East Average	**0.429**	**82.5**	**0.210**
West Average	**0.789**	**90.0**	**0.160**
Overall Average	**0.564**	**85.3**	**0.191**

The average recoveries of the tests shown above were relatively high, between 77% and 94%. The material from the East Pit showed a moderately lower recovery compared to the West Pit. Cyanide consumption was low averaging 0.19 kg/t.

13.3.5 1994 Bottle Roll Tests

Standard bottle roll tests using the HCL assay technique were conducted on seventy-five (75) biotite and sericite samples taken from the Imperial Project deposit. Metallurgical testing was conducted by ChemGold. For these tests, hole and interval data were recorded for each sample, and a description of each sample's mineralogy was given (Chemgold, Inc., 1994). The average fire assay head grade of the samples was 1.43 g/t. Gold head grade and tail grade was also calculated for each test using the HCL method. The HCL average gold head grade was 1.23 g/t.

The results of the campaign showed an average gold extraction of 86% with a minimum result of 60% and a maximum result of 100%. A complete summary of the results of these tests is given in Table 13-14.

Table 13-14: Results of 1994 Hot Cyanide Leach Assays (Chemgold Inc.)

#	Sample #	Hole No.	Interval (m) From	Interval (m) To	Head Fire Assay (g/t)	Description	HCL Head (g/t)	HCL Tail (g/t)	Tail Fire Assay (g/t)	HCL Recovery (%)
1	15611	O-57	64.0	65.5	0.686	Schist	0.686	0.069	0.069	91%
2	15612	O-57	65.5	67.1	0.343	Schist	0.411	0.034	0.034	93%
3	15622	O-57	80.8	82.3	0.309	Schist Ch-bio	0.549	0.103	0.034	91%
4	39551	I-91-5	131.1	132.6	0.377	Schist bio-chl	0.343	0.034	0.034	94%
5	39839	I-91-5b	115.8	117.3	0.720	Schist bio-chl, epl altered	0.686	0.103	0.171	78%
6	39841	I-91-5b	121.9	123.4	0.411	Gneiss biotitic	0.549	0.103	0.103	84%
7	38591	I-91-5b	45.7	47.2	0.583	Quartz 70%, leucogneiss	0.549	0.137	0.206	60%
8	39601	I-91-6	32.0	33.5	1.646	Schist bio-chl, weakly oxidized	0.617	0.103	0.171	77%
9	39607	I-91-6	41.1	42.7	0.411	Schist bio-chl, fault contact	0.206	0.034	0.000	100%
10	39606	I-91-6	39.6	41.1	0.446	Schist bio-chl	0.240	0.034	0.000	100%
11	39975	I-91-6b	103.6	105.2	0.480	Gneiss bio silicifed, with minor hem	0.377	0.137	0.206	67%
12	40110	I-91-6b	157.0	158.5	0.617	Gneiss	0.686	0.034	0.103	87%
13	40111	I-91-6b	158.5	160.0	0.309	Gneiss	0.514	0.103	0.034	94%
14	20637	I-91-9	102.1	103.6	1.509	Schist chl-musc schist, trace red clay	0.583	0.069	0.034	98%
15	20638	I-91-9	105.2	106.7	0.823	Schist with 20-30% gneiss	0.583	0.206	0.240	83%
16	20667	ES-19	62.5	64.0	1.783	Gneiss, chloritic gneiss well brecciated	1.646	0.103	0.206	88%
17	20690	ES-19	97.5	99.1	1.920	Gneiss schist, mix bio-chl	2.126	0.103	0.171	91%
18	20699	ES-19	111.3	112.8	0.720	Gneiss	0.514	0.069	0.377	75%
19	20701	ES-19	114.3	115.8	0.857	Gneiss, 5% hem qtz	0.926	0.069	0.034	98%
20	20703	ES-19	117.3	118.9	1.303	Schist bio-chl, 5% qtz	0.651	0.137	0.137	86%
21	20704	ES-19	120.4	121.9	1.817	Schist bio-chl, 3% hem qtz	1.817	0.274	0.309	84%
22	10740	ES-20	94.5	96.0	2.674	Schist bio-chl, 5-10% red clay	2.503	0.343	0.171	92%
23	10737	ES-20	89.9	91.4	1.440	Schist bio-chl, 5-10% red clay	1.474	0.171	0.171	88%
24	10743	ES-20	99.1	100.6	2.194	Gneiss, <10% lim schist	1.577	0.103	0.034	98%
25	10747	ES-20	105.2	106.7	0.617	Gneiss	0.377	0.069	0.000	100%
26	21642	O-1A	111.3	112.8	2.674	Quartzite, strongly hematitic	3.017	0.343	0.377	87%
27	21644	O-1A	83.8	85.3	1.680	Gneiss, hematitic	1.234	0.171	0.137	88%
28	21717	O-1A	115.8	117.3	0.446	Breccia	0.446	0.034	0.137	90%
29	21635	O-1A	70.1	71.6	0.754	Schist bio-chl	1.097	0.309	0.171	81%
30	21726	O-1A	132.6	134.1	0.549	Breccia	0.446	0.103	0.069	82%
31	21728	O-1A	135.6	137.2	0.686	Breccia	1.200	0.103	0.103	93%
32	21636	O-1A	71.6	73.2	0.274	Breccia	0.754	0.034	0.103	75%
33	21758	ES-20	121.9	123.4	1.269	Breccia	0.926	0.103	0.137	86%
34	82302	O-42D	51.8	53.3	0.343	Gneiss gravel contact	0.343	0.034	0.034	86%
35	82304	O-42D	54.9	56.4	0.411	Gneiss	0.206	0.069	0.103	83%
36	82344	O-42D	115.8	117.3	1.029	Schist bio-chl lim stained	1.166	0.274	0.343	72%
37	82413	O-42D	146.3	147.8	1.269	Schist bio-chl/gneiss	0.754	0.137	0.137	85%
38	82343	O-42D	114.3	115.8	1.097	Quartz Schist	1.509	0.343	0.377	75%
39	82414	O-42D	147.8	149.4	0.686	Schist bio-chl gneiss	0.377	0.069	0.000	100%
40	82327	O-42D	89.9	91.4	0.274	Quartzite, hematitic traces	0.171	0.034	0.069	72%
41	40577	O-47	32.0	33.5	0.617	Schist mixed with breccia	0.651	0.137	0.034	94%
42	40580	O-47	36.6	38.1	1.200	Quartz Schist, 10% clay gangue	1.543	0.103	0.171	84%
43	40585	O-47	44.2	45.7	0.789	Breccia with 5% free quartz	0.514	0.103	0.137	69%
44	15588	O-56	97.5	99.1	0.823	Schist, qtz, <3% oxidized pyrite	0.960	0.103	0.103	94%

#	Sample #	Hole No.	Interval (m) From	To	Head Fire Assay (g/t)	Description	HCL Head (g/t)	HCL Tail (g/t)	Tail Fire Assay (g/t)	HCL Recovery (%)
45	15590	O-56	99.1	100.6	0.686	Schist, bio-chl	0.994	0.103	0.103	94%
46	15597	O-56	115.8	117.3	2.126	Schist, Impact hem gneiss, str hem	2.263	0.103	0.103	94%
47	15535	O-55	76.2	77.7	0.994	Gneiss	0.994	0.069	0.137	85%
48	40644	O-47	134.1	135.6	0.617	Pyritic 1-3%, Quartz	0.480	0.069	0.000	100%
49	41370	O-60	93.0	94.5	0.651	Quartz mixed with bio-chl schist	0.514	0.069	0.069	85%
50	41375	O-60	100.6	102.1	1.166	Schist bio-chl	1.543	0.240	0.240	82%
51	41380	O-60	108.2	109.7	1.714	Schist bio-chl	1.371	0.171	0.069	84%
52	41386	O-60	117.3	118.9	0.309	Breccia	0.274	0.034	0.000	100%
53	10590	O-49	125.0	126.5	0.411	Schist	0.411	0.103	0.069	85%
54	10597	O-49	135.6	137.2	1.646	Gneiss mixed with schist	2.023	0.137	0.137	93%
55	10599	O-49	138.7	140.2	0.891	Schist	1.097	0.103	0.034	96%
56	10604	O-48	146.3	147.8	1.509	Schist	1.680	0.103	0.171	89%
57	15694	O-58	89.9	91.4	0.926	Gneiss	1.303	0.103	0.137	87%
58	15874	O-58	59.4	61.0	0.651	Gneiss mixed with schist	0.480	0.103	0.069	84%
59	40712	O-73B	76.2	77.7	1.269	Breccia	0.686	0.103	0.103	84%
60	40221	O-64	79.2	80.8	0.377	bio qtz fep gneiss	0.549	0.034	0.034	92%
61	41222	O-64	80.8	82.3	0.446	Breccia 50%, bio-ch schist 30%	0.291	0.069	0.137	60%
62	41226	O-64	86.9	88.4	0.343	Quartz	0.343	0.069	0.103	69%
63	41233	O-64	97.5	99.1	2.503	Quartz Schist	3.051	0.137	0.103	84%
64	41235	O-64	100.6	102.1	0.583	Quartz mixed with bio-chl gneiss	0.549	0.034	0.034	92%
65	41237	O-64	103.6	105.2	0.960	Gneiss	1.200	0.069	0.034	96%
66	41238	O-64	105.2	106.7	1.783	Gneiss	2.091	0.069	0.137	89%
67	41472	O-83	77.7	79.2	0.480	Quartz, moderate oxidized	0.446	0.069	0.034	82%
68	21592	I-91-14	86.9	88.4	1.234	Schist	2.366	0.206	0.309	85%
69	21591	I-91-14	85.3	86.9	0.789	Schist	1.337	0.103	0.103	89%
70	21649	O-1A	91.4	93.0	2.469	Gneiss	3.051	0.206	0.274	90%
A	21650	O-1A	93.0	94.5	1.303	Gneiss	1.440	0.137	0.171	87%
B	21376	I-18	74.7	76.2	1.646	Gneiss	1.474	0.583	0.034	84%
C	41363	O-60	82.3	83.8	5.486	Schist bio-chl, mod limonitc	4.011	0.309	0.240	82%
D	41381	O-60	109.7	111.3	0.857	Schist bio-chl, limonitic	0.754	0.034	0.069	85%
E	41362	O-60	80.8	82.3	28.800	Schist bio-chl, mod limonitc, 20% clay	14.914	1.166	1.166	92%
				AVG	1.433		1.233	0.137	0.136	86%
				STD DEV	3.303		1.785	0.153	0.153	9%
				MIN	0.274		0.171	0.034	0.000	60%
				MAX	28.80		14.91	1.166	1.166	100%

Discussion of the results in the original report concludes that material with the “Schist” designation reacted the most favorably to testing, with an average gold extraction of 89% from all tests. Quartz and Quartzite material, in contrast, showed the lowest average extractions at 81% and 80%, respectively. Tests with Gneiss and Breccia material resulted in 87% and 83% gold extraction, respectively. The same report concludes that gold extraction decreases with increasing “elevation”: 92% average recovery at 300 – 400 feet (91 – 122m), 88% at 400 – 500 feet (122 – 152m), 85% at 500 – 600 feet (152 – 183m), and 83% at 600 – 800 feet (183 – 244m). However, this correlation does not factor in mineralogy of the samples. Figure 13-1 shows the relationship between recovery and sample elevation. Although there appears to be a correlation the R-squared value is very low (0.05).

Figure 13-1: Gold Recovery with Elevation 1994 Bottle Roll Tests

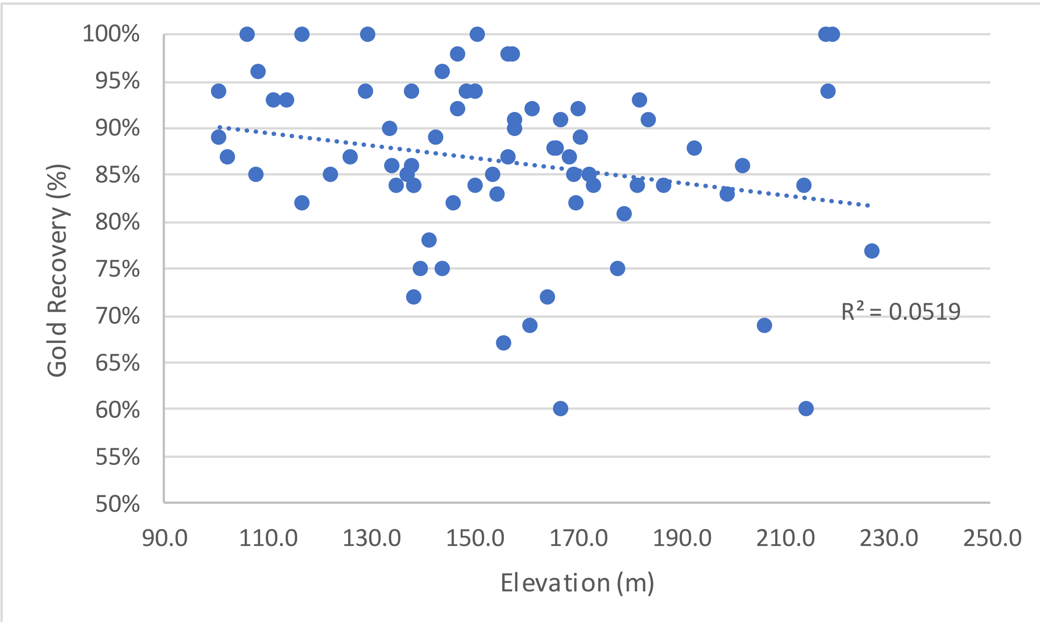


Figure 13-2 shows the correlation between HCL head grade and fire assay head grade. A reasonable correlation exists that could likely be improved if rock type were employed as an additional segregation method.

Figure 13-2: HCL Head Grade vs. Fire Assay Head Grade for 1994 Bottle Roll Tests

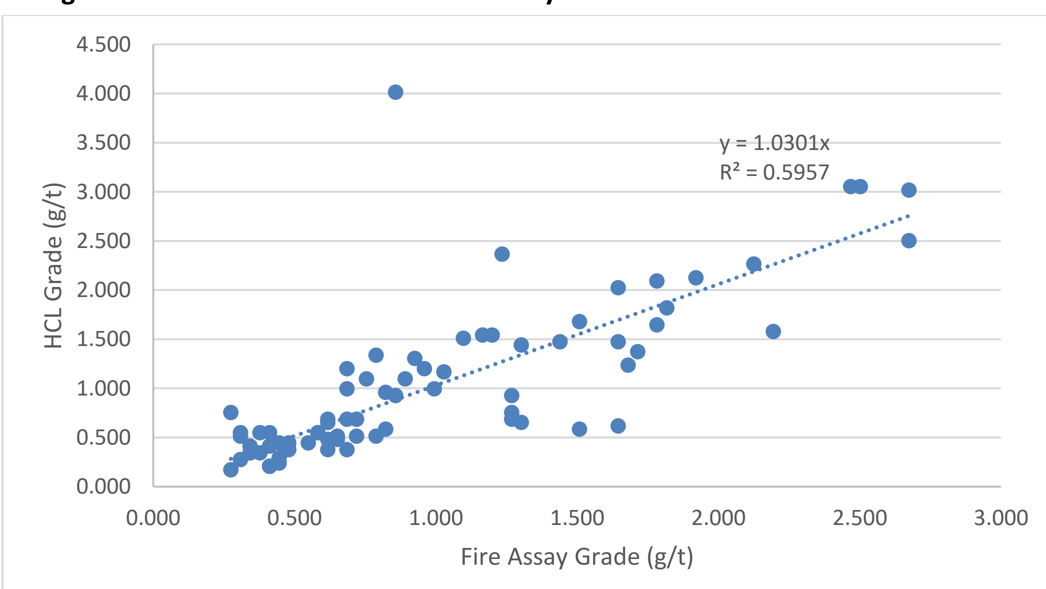
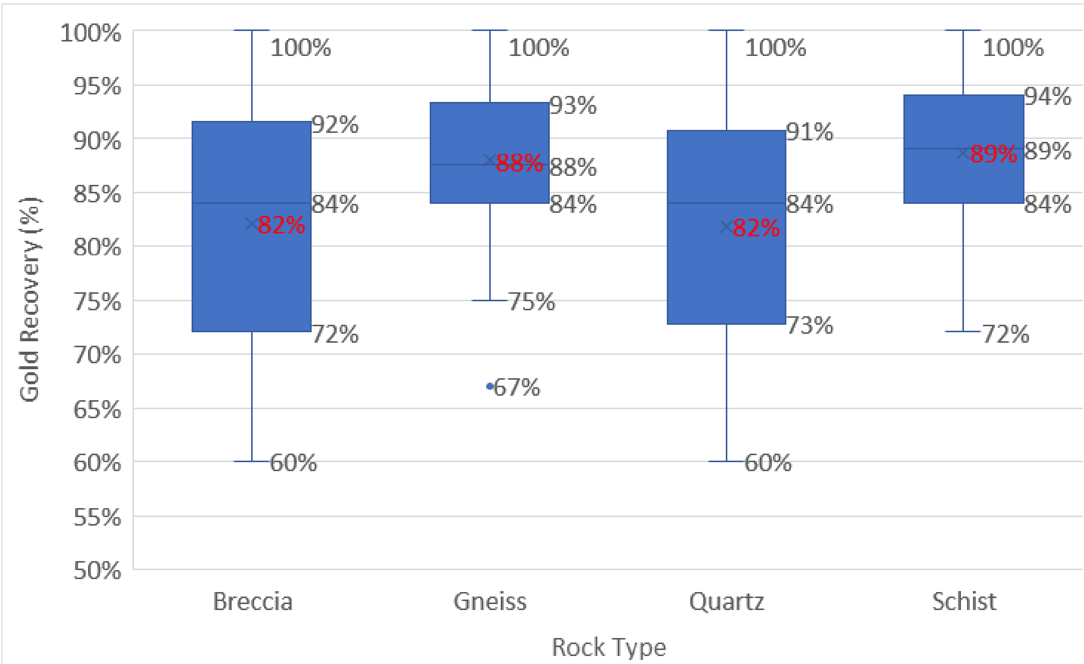


Figure 13-3 shows a Whisker Plot of the recovery and rock type. As shown, there is a significant variability in the gold recovery within each rock type and between rock types. The gneiss and schist had average recoveries ranging from 88 to 89% and the breccia and quartz's recoveries both averages 82%. The problem with this type of analysis is that it relies heavily on the proper categorization of the mineralization. The plot still shows the variability and rock type differences well.

Figure 13-3: Gold Recovery and Rock Type Whisker Plot 1994 Bottle Roll Tests



Bottle roll tests were also completed on samples employed for column leach testing (Chemgold, Inc., 1994) as shown in Table 13-15.

Table 13-15: Bottle Roll Tests on 1994 Column Leach Feed (Chemgold, Inc.)

Sample #	Geology*	Column Head Fire Assay (g/t)	Head Gold Grade		Tail Gold Grade HCL (g/t)	Recovery (HCL) (%)	CN Consumed (kg/t)
			Fire Assay (g/t)	HCL (g/t)			
B-7	SGN	0.411	0.411	0.411	0.069	79	0.10
B-8	SGN	0.343	0.309	0.377	0.069	84	0.14
B-9	BGN	0.309	0.309	0.309	0.034	90	0.10
B-10	BGN	0.411	0.411	0.377	0.514	75	0.13
B-11	BGN	0.480	0.480	0.446	0.034	90	0.19
B-12	BGN	1.371	0.480	0.583	0.034	88	0.12
B-13	BGN	0.583	0.583	0.686	0.103	75	0.17
B-14	BGN	0.583	0.549	0.754	0.034	89	0.16
Avg	SGN	0.377	0.360	0.394	0.069	81.5	0.12
	BGN	0.623	0.469	0.526	0.126	84.5	0.15
	Overall	0.561	0.441	0.493	0.111	83.75	0.14

*SGN = sericite gneiss, BGN = biotite gneiss

The results of the bottle roll tests showed slightly lower gold recovery and cyanide consumption values when compared with previous tests conducted on Imperial Project material. The average extraction for sericite gneiss (SGN) and biotite gneiss (BGN) samples was 81.5% and 84.5%, respectively. The report indicates that these samples were produced from core and were coarser than the previous tests that were based on exploration samples. We assume that this implies that the exploration samples were RC drill chips and the core materials were crushed to -10 mesh. The column leach test results on these samples follows in Sections 13.4 and 13.6.

Numerous "gravel" samples were also examined (Chemgold, Inc., 1994). The conditions used were considered standard: 0.1% CN content in the leaching solution, 12.0 pH, and a 5-day leach duration. Table 13-16 provides a summary of the results from these tests.

Table 13-16: 1994 Bottle Roll Tests on Gravel Material from the Imperial Project (Chemgold, Inc.)

Sample	Geology	Total Weight (g)	Total Volume (mL)	Gold Head Grade (g/t)		Recoverable Gold (g/t)
				Solution	Original HCL	
BR-ER9-310	Gravel?	6577	9866	0.069	0.171	0.103
BR-ER9-290	Gravel?	6513	9770	0.069	0.171	0.103
BR-ER14-130	Gravel?	3440	5160	0.069	0.274	0.103
BR-ER14-140	Gravel?	5300	7950	0.034	0.206	0.034
BR-ER11-290	Gravel	1002	1503	0.069	0.103	0.103
BR-ER11-310	Gravel	3960	7920	0.103	0.206	0.206
BR-ER11-130	Gravel	3440	6880	0.034	0.069	0.069
BR-ER11-170	Gravel	5703	8555	0.069	0.103	0.103
BR-ER14-120	Gravel	4353	6336	0.034	0.171	0.034
BR-ER14-190	Gravel	4335	6503	0.034	0.069	0.034
BR-ER15-290	Gravel	6293	9440	0.103	0.103	0.137
BR-ER15-280	Gravel	5075	7613	0.103	0.171	0.137
BR-ER14-160	Gravel	5860	8790	0.103	0.103	0.137
BR-ER14-100	Gravel	4330	6495	0.034	0.137	0.034
BR-ER15-260	Gravel	4386	6579	0.137	0.034	0.206
BR-ER8-385	Gravel	8995	13493	0.103	0.103	0.171
BR-ER9-305	Gravel	6527	9791	0.069	0.171	0.103
BR-ER9-225	Gravel	4881	7322	0.069	0.103	0.103
BR-ER9-285	Gravel	3912	5668	0.103	0.069	0.137
BR-ER14-150	Gravel	5660	8490	0.069	0.343	0.103
BR-ER12-250	Gravel	6520	9780	0.103	0.206	0.137
BR-ER12-180	Gravel	5048	7572	0.034	0.069	0.034
BR-ER15-250	Gravel	5050	7575	0.103	0.103	0.137
BR-ER8-280	Gravel	5811	8717	0.034	0.103	0.034
BR-ER9-335	Gravel	5234	7851	0.034	0.069	0.034
BR-ER9-265	Gravel	6964	10446	0.034	0.137	0.034
BR-ER12-295	Gravel	6448	9672	0.034	0.103	0.034
BR-ER12-235	Gravel	7237	10856	0.069	0.171	0.069
BR-ER12-280	Gravel	5731	8597	0.069	0.171	0.103
BR-ER8-220	Gravel	5430	8145	0.069	0.103	0.069
BR-ER12-220	Gravel	493	740	0.034	0.103	0.069

Sample	Geology	Total Weight (g)	Total Volume (mL)	Gold Head Grade (g/t)		Recoverable Gold (g/t)
				Solution	Original HCL	
BR-ER8-410	Gravel	5535	8302	0.137	0.240	0.206
BR-ER9-190	Gravel	6099	9149	0.069	0.274	0.103
BR-ER8-285	Gravel	5845	8768	0.069	0.309	0.103
	Average	5235	7950	0.070	0.148	0.098

In the case of the gravel materials there was not significant gold present and the estimated recoverable gold is less than 0.21 g/t.

13.4 1994/1995 Column Leach Tests

Column leach tests were performed on the same samples previously described above in Section 13.3.5 (Chemgold, Inc., 1994). The results of bottle roll leach tests on the same samples can be found in Table 13-15. The column leach test material (core samples) was crushed to under two inches (5.1 cm) and leached for 35 days. Approximately 15% of the ore was between -2” to +1”, 25% between -1” and +¼” with balance being – ¼”. The estimated P80 for these columns was ~2”. The ore was combined with 1.25 kg/tonne of lime prior to loading in the column. The results of these tests are given in Table 13-17.

Table 13-17: 1994 Column Leach Results on Samples Previously Bottle Leached (Chemgold, Inc.)

Column ID	Core Sample ID	Ore Type*	Dia. (cm)	Column Weight (kg)	Solution Rate (LPH/m^2)	Head Grade (g/t)	Recovery (Solution) (%)	Cyanide Cons. (kg/t)
B-7	EC-1 & EC-2[1]	SGN	15.2	49.0	8.5	0.617	88	0.30
B-8[1]	EC-1 & EC-2[1]	SGN	15.2	48.5	9.3	0.446	79	0.27
B-9	EC-1 & EC-2[2]	BGN	15.2	54.9	10.7	0.446	89	0.29
B-10[2]	EC-1 & EC-2[2]	BGN	15.2	54.9	10.5	0.583	87	0.28
B-11	WC-1 & WC-3	BGN	30.5	272.2[3]	6.6	0.823	94	0.17
B-12[3]	WC-1 & WC-3	BGN	15.2	68.0[3]	8.1	1.131	95	0.26
B-13	WC-2	BGN	30.5	272.2[4]	6.3	1.029	91	0.19
B-14[4]	WC-2	BGN	15.2	68.0[4]	8.1	0.891	93	0.21
				Average	**8.5**	**0.746**	**89.5**	**0.25**

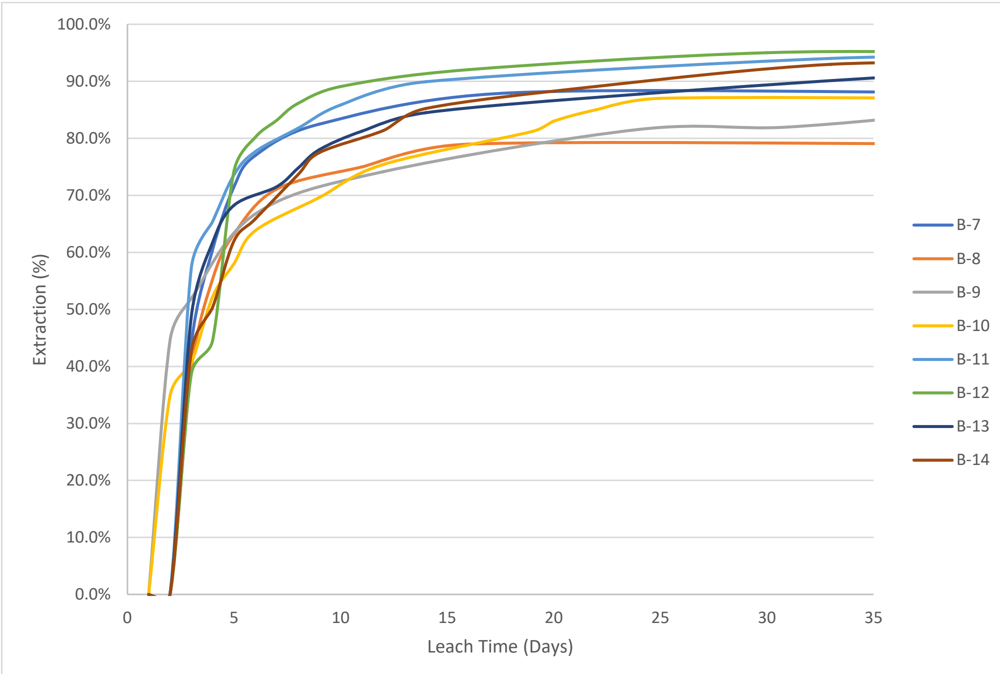
SGN = sericite gneiss, BGN = biotite gneiss
1: Upper portions of both cores used
2: Lower portions of both cores used
3: Material from both cores combined with 22.7 kg of -1/4 rejects
4: Addition of 22.7 kg of material broken down to -1+1/4 size and addition of 84.1 kg of -1/4 size material

Gold recovery ranged between 79% and 95% with an average recovery of 90%. The report also indicates that similar Picacho tests on oxide ore averaged 82%. Size fraction recoveries were uniform across all size fractions analyzed. Cyanide consumption averaged 0.25 kg/t.

There were no reported percolation problems in columns with the Biotite Gneiss material, however, the Sericite Gneiss column was reported to have had ponding issues. Percolation issues in the form of ponding were noted in column B-8 and column B-7 late in the leach cycle (Chemgold, Inc., 1995). However, despite these issues, the results of these tests show uniformly good response to traditional column leaching.

The leach rate for each test varied, with significant leaching starting early in the tests, and completion of gold extraction generally occurred between 16 and 33 days. Figure 13-4 shows the gold extraction curves for the tests.

Figure 13-4: Gold Extraction vs. Leach Time for 1994/1995 Column Leach Tests Conducted on BGN and SGN Samples (Chemgold, Inc.)



The report on these columns references the neighboring Picacho mine on several occasions as a method to help validate and predict the potential heap leach results from the Imperial Project. The report states:

The major ore type mined at Picacho Mine has been oxidized Jurassic biotite gneiss. This gneiss is characterized by strong hematitic staining and is dark reddish brown in color. Other oxidized ore material is ore breccia that is a red-brown colored mix of volcanic and oxidized gneiss. Leaching run-of-mine oxide ore on four completed heap sites at Picacho Mine has yielded a cumulative 72% gold recovery.

Table 13-18: Picacho Mine Gold Recovery – Oxide ROM (Unknown Report)

Location	Au Stacked (oz)	Au Recovered (oz)	Recovery (%)
Site 1	112,066	83,449	74.8
Site 2	26,985	17,212	63.8
Site 3	90,413	66,074	73.1
Site 4	55,512	38,939	70.1
Cumulative	**284,976**	**205,674**	**72.2**

SGN = sericite gneiss, BGN = biotite gneiss

The report further indicates that column leach tests performed on Picacho biotite ore typically yield recoveries averaging 82%. Cyanide consumption averages 0.26 pounds per ore ton. No indication was provided as to the particle size employed in the Picacho columns.

13.5 1995/1996 Bottle Roll Tests

Additional bottle roll tests were conducted on biotite gneiss and sericite gneiss samples, presumably by McClelland Laboratories Inc (McClelland Laboratories Inc., 1996). The results of these tests closely matched the previous bottle roll tests (McClelland Laboratories Inc., 1996). Column ID (sample #) information was recorded for these tests, corresponding to HCL assays and drill core data previously described above. The average fire assay head grade for the biotite gneiss samples was 0.953 g/t. The average fire assay head grade for sericite gneiss was 2.59 g/t (with the presence of two outlying samples; excluding these high-grade samples, the average was 1.135 g/t).

For biotite gneiss, the average gold recovery was 86.5%. The minimum observed recovery was 60%, and the maximum recovery was 100% (HCL). For the sericite gneiss samples, the average gold recovery was 86.1%. The minimum observed recovery was 71.6% and the maximum recovery was again 100%. The two different types of samples showed similar average cyanide consumptions – for biotite gneiss, 0.16 kg/t, and 0.17 kg/t for sericite gneiss. Table 13-19 and Table 13-20 provide the details of the tests.

Table 13-19: 1995 Bottle Roll Tests Conducted on Biotite Gneiss Samples (McClelland Laboratories, Inc.)

Column ID	Fire Head Grade (g/t)	Calculated Head Grade (g/t)	Tail Grade (g/t)	Recovery (%)	CN Consumed (kg/t)
21717	0.446	0.343	0.034	90.1%	0.16
21726	0.549	0.377	0.069	81.8%	0.08
21728	0.686	1.269	0.103	93.3%	0.08
21636	0.274	0.343	0.103	75.0%	0.10
21758	1.269	0.994	0.137	86.1%	0.08
40585	0.789	0.446	0.137	69.0%	0.08
41386	0.309	0.309	0.000	100.0%	0.08
40712	1.269	0.651	0.103	84.3%	0.16
41221	0.377	0.411	0.034	91.7%	0.20
41222	0.446	0.343	0.137	60.0%	0.16
39841	0.411	0.651	0.103	83.9%	0.28
39975	0.480	0.583	0.206	67.0%	0.12
40110	0.617	0.789	0.103	87.1%	0.12
40111	0.309	0.583	0.034	94.0%	0.20
20667	1.783	1.714	0.206	87.9%	0.12
20690	1.920	1.817	0.171	91.5%	0.12
20699	0.720	0.549	0.137	75.0%	0.20
20701	0.857	0.720	0.034	97.6%	0.16
10743	2.194	1.440	0.034	97.6%	0.12
10747	0.617	0.617	0.000	100.0%	0.16
83202	0.343	0.309	0.034	84.9%	0.15
82304	0.411	0.274	0.103	63.4%	0.15
15535	0.994	0.926	0.137	85.2%	0.16
15694	0.926	1.063	0.137	87.2%	0.32
41237	0.960	0.891	0.034	96.2%	0.16
41238	1.783	1.303	0.137	89.4%	0.16

Column ID	Fire Head Grade (g/t)	Calculated Head Grade (g/t)	Tail Grade (g/t)	Recovery (%)	CN Consumed (kg/t)
21649	2.469	2.640	0.274	90.3%	0.20
21650	1.303	1.371	0.171	87.4%	0.20
21375	1.646	4.011	0.651	83.8%	0.28
41226	0.343	0.343	0.103	69.5%	0.05
41233	2.503	1.886	0.309	83.6%	0.12
41235	0.583	0.411	0.034	91.9%	0.16
15611	0.686	0.583	0.069	91.3%	0.12
15612	0.343	0.514	0.034	93.4%	0.08
15622	0.309	0.377	0.034	91.3%	0.20
39551	0.377	0.617	0.034	93.9%	0.16
39839	0.720	0.754	0.171	77.8%	0.12
39591	0.583	0.514	0.206	60.0%	0.16
39601	1.646	0.720	0.171	76.7%	0.32
39607	0.411	0.411	0.000	100.0%	0.16
39606	0.446	0.446	0.000	100.0%	0.16
20637	1.509	0.789	0.034	95.7%	0.20
20638	0.823	1.920	0.377	83.2%	0.12
20703	1.303	1.029	0.137	85.8%	0.05
20704	1.817	1.886	0.309	83.7%	0.28
10740	2.674	2.194	0.171	92.1%	0.12
10737	1.440	1.406	0.171	87.8%	0.08
21635	0.754	0.891	0.171	80.7%	0.12
82413	1.269	0.789	0.137	84.8%	0.28
82414	0.686	0.686	0.000	100.0%	0.08
40577	0.617	0.617	0.034	94.4%	0.08
15589	0.823	1.680	0.103	93.9%	0.28
15590	0.686	1.680	0.103	93.9%	0.28
Average	0.953	0.960	0.122	90.0%	0.16
Std Dev	0.635	0.711	0.113	75.0%	0.07

Table 13-20: 1995 Bottle Roll Tests Conducted on Sericite Gneiss Samples

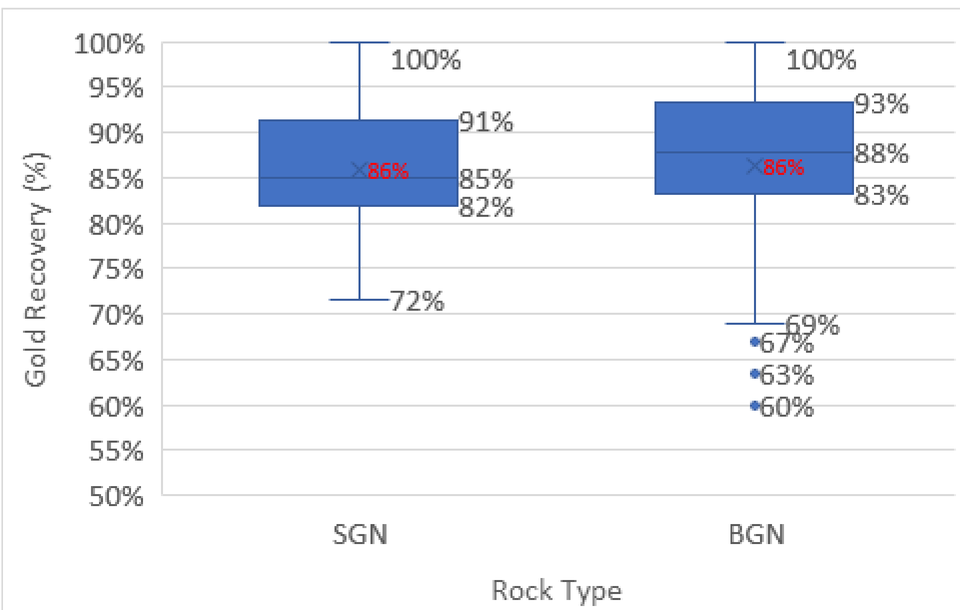
Column ID	Fire Head Grade (g/t)	Calculated Head Grade (g/t)	Tail Grade (g/t)	Recovery (%)	CN Consumed (kg/t)
21644	1.680	1.166	0.137	88.1%	0.20
10597	1.646	1.886	0.137	92.7%	0.16
15674	0.651	0.411	0.069	83.6%	0.20
15597	2.126	1.680	0.103	93.9%	0.28
41375	1.166	1.303	0.240	81.8%	0.16
41380	1.714	1.200	0.069	94.3%	0.12
10590	0.411	0.446	0.069	85.0%	0.16
10599	0.891	0.857	0.034	96.1%	0.08
10604	1.509	1.577	0.171	89.1%	0.16
21592	1.200	1.989	0.309	84.6%	0.16

Column ID	Fire Head Grade (g/t)	Calculated Head Grade (g/t)	Tail Grade (g/t)	Recovery (%)	CN Consumed (kg/t)
21591	0.789	0.960	0.103	89.2%	0.16
41363	5.486	1.303	0.240	81.8%	0.40
41381	0.857	0.446	0.069	84.9%	0.20
82343	1.097	1.406	0.377	74.6%	0.12
40580	1.200	1.097	0.171	84.4%	0.08
40644	0.617	0.617	0.000	100.0%	0.08
41370	0.651	0.480	0.069	85.4%	0.04
41472	0.480	0.206	0.034	82.4%	0.16
41362	28.800	14.366	1.166	92.0%	0.26
21642	2.674	2.983	0.377	87.4%	0.12
82344	1.029	1.200	0.343	71.6%	0.20
82327	0.309	0.171	0.069	72.2%	0.16
Average	2.590	1.716	0.198	86.1%	0.17
Std Dev	5.955	2.904	0.245	7.3%	0.08

(McClelland Laboratories, Inc.)

Figure 13-5 shows the Whisker plot of the gold recovery for the two rock types sericite gneiss (SGN) and biotite gneiss (BGN).

Figure 13-5: Gold Recovery and Rock Type Whisker Plot of 1995 Bottle Roll Tests



The plot shows that the average gold recovery for both rock types was 86% with recovery ranging widely for both materials, from 69% to 100%.

13.6 1995 Column Leach Tests

A series of additional column tests were conducted at the Picacho mine by Glamis Gold (Chemgold, Inc.). These follow-up tests used the same procedure as the previous tests. However, the samples were derived from reverse circulating drill hole material, not diamond drill core. Four of these follow-up tests used – ¼" (6.35 mm) material. Only test B-15 could be used to replicate conditions of the B-7 and B-8 tests described above (minus 2" material). The results of the column tests are given in Table 13-21:

Table 13-21: 1995 Column Leach Tests on BGN and SGN Samples (Chemgold, Inc.)

Column ID	Sample Type / Source	Ore Type	Dia. (cm)	Column Weight (kg)	Solution Rate (LPH/m²)	Head Assay (g/t)		Recovery (Solution) (%)	Cyanide Cons. (kg/t)
						Fire	Calc.		
B-15	Bulk Sample	SGN	30.5	275.3	11.7	0.240	0.309	89.0	0.15
B-17	94WR-24A	BGN	30.5	228.6	11.7*	1.029	1.029	77.5	NR
B-18	94WR-8	BGN	30.5	127.5	11.7*	0.377	0.549	71.4	NR
B-19	94WR-14A	BGN	30.5	108.4	11.7*	0.720	1.131	66.8	NR
G1	94ER-18A	BGN	30.5	145.1	11.7*	0.171	0.686	61.5	NR
				Average	**11.7**	**0.507**	**0.741**	**73.2**	

*: Not recorded, calculated SGN = sericite gneiss, BGN = biotite gneiss

The overall cyanide consumption average was noted as 0.33 lb./ton (1.65 kg/t) with 2 lb./ton of lime (1 kg/t). There was some indication of percolation issues near the end of the column operation.

13.7 1996 Column Leach Tests and Bottle Roll Tests

13.7.1 Column Leach Tests on Biotite/Sericite Gneiss Composites

Additional column leach tests were undertaken to further evaluate the heap leach amenability of the material by (McClelland Laboratories, Inc., 1996), (MLI Job No. 2230). A total of 161 ten-foot drill core intervals were received for crushing and interval assay, and subsequent compositing and heap leach test work. A single biotite gneiss (BGN) composite was prepared from drill holes 9SWC-4 (PQ core) and 9SWC-S (HQ core) after interval assays were reviewed by Chemgold personnel Two composites were prepared from sericite gneiss (SGN) core intervals from drill holes 95EC-3 (PQ core) and 9SEC-S (HQ core), and were designated SGN low-grade and SGN high-grade. All core interval and core composite preparation was done according to instructions provided by Glamis Gold.

Head screen analyses, column leach tests, and tail screen analyses were conducted in duplicate on the BGN composite and on the SGN-LG. and SGN-RG. composites at a 90% minus l" (25,4 mm) feed size. Average head grades for the BGN, SGN-LG., and SGN-RG. composites were 0.0159, 0.0032, and 0.0324 opt, respectively (0.545, 0.110 and 1.111 g/t). Silver content in each composite was below fire assay detection limits (<0.05 opt or 1.71 g/t). Consequently, silver recovery data are not discussed in this report.

All three Imperial project ore composites were readily amenable to heap leach cyanidation treatment at the P90 l" (25,4mm) feed size. Gold recoveries of 91.8% (initial) and 90.1% (duplicate) were achieved from the BGN core composite in about 86 days of leaching and washing (including rest cycles). Gold recovery rate was rapid and about 79% gold recovery was achieved in 20 days of continuous leaching. Gold recovery rate slowed markedly after 20 days, and an additional 54 days of cyanide solution contact (including rest cycles) was required to achieve ultimate recovery (~91 %). No additional gold was recovered during the water wash cycles (6 to 8 days). Cyanide consumption was moderate at 1.24 (average of 2 tests) lbs./ton of ore. Cyanide consumption from column tests is usually (absence of cyanicides) substantially higher than that experienced in commercial production. It is expected that commercial consumption from BGN ore at a 1" (25.4 mm) crush size would not exceed 0.3 lb./ton of ore (0.15 kg/t). The 2.0 Ibs lime/ton of ore (1 kg/t) added before leaching was sufficient to maintain protective alkalinity at above pH 10.3 throughout the cyanide leach cycles.

A gold recovery of 77.8% was achieved from the SGN-L.G. composite in 40 days of cyanide solution contact. Gold recovery rate was very rapid for the extremely low-grade feed, and extraction was complete in 10 days of continuous leaching. Rest cycles were not effective in improving gold recovery. Cyanide consumption was low at 0.75 lbs./ton of ore (0.375 kg/t) and should be even lower in commercial production. The 2.0 lbs. lime/ton of ore (1 kg/t) added before leaching was sufficient to maintain leaching pH at above 10.2.

A gold recovery of 93.9% was achieved from the SGN-H.G. core composite in 89 days of leaching and washing. Gold recovery rate data was nearly the same as for the BGN composite. Cyanide consumption was moderate at 1.5 lbs./ton of ore (0.75 kg/t). Commercial consumption should not exceed 0.4 Ibs NaCN/ton of ore (0.2 kg/t). The 2.0 lbs lime/ton of ore (1 kg/t) added before leaching was sufficient to maintain leaching pH at above pH 10.0 through 78 days of cyanide solution contact. Pregnant solution pH dropped to as low as pH 9.8 the last 6 days of leaching.

Samples from each composite were sieved to produce size distributions that were then assayed to reveal the deportment of gold for each size fraction. Composites were then leached in 12-inch (30.5 cm) diameter columns (10 feet or 3.05 m high) using 0.35 kg/tonne NaCN and 1.0 kg/tonne lime, applied in a solution at a rate of approximately 9.8 LPH/m^2. Solution samples were taken every 24-hours, and leaching was continued until no appreciable recovery increases were observed. Table 13-22 shows the results of the tests.

Table 13-22: Results of 1995/1996 Column Leach Tests on Sized Composites

Material	Size Fraction	Max size (µm)	Head Weight (%)	Cum. Weight (%)	Tails Weight (%)	Head Assay (g/t)	Tail Assay (g/t)	Au Distribution		Au Recovery (Fire Assay) (%)
								%	Cum. %	
BGN Feed (Test 1)	+2	-	0.4	0.4	0	0.069	-	0.1%	0%	-
Actual Column Recovery (Solution)	-2+1	50.8	10.8	11.2	11.1	0.686	0.069	13.5%	14%	90.0%
91.8%	-1+3/4	25.4	6.5	17.7	6.4	0.823	0.034	9.8%	23%	95.8%
Leach time (days)	-3/4+1/2	19.05	9.6	27.3	8.8	1.029	0.034	18.0%	41%	96.7%
79	-1/2+1/4	12.7	12.8	40.1	12.6	0.411	0.069	9.6%	51%	83.3%
Cyanide Consumption (g/t)	-1/4+10	6.35	17	57.1	17.3	0.446	0.034	13.8%	65%	92.3%
0.64	-10+20	1.68	8.3	65.4	8.8	0.411	0.069	6.2%	71%	83.3%

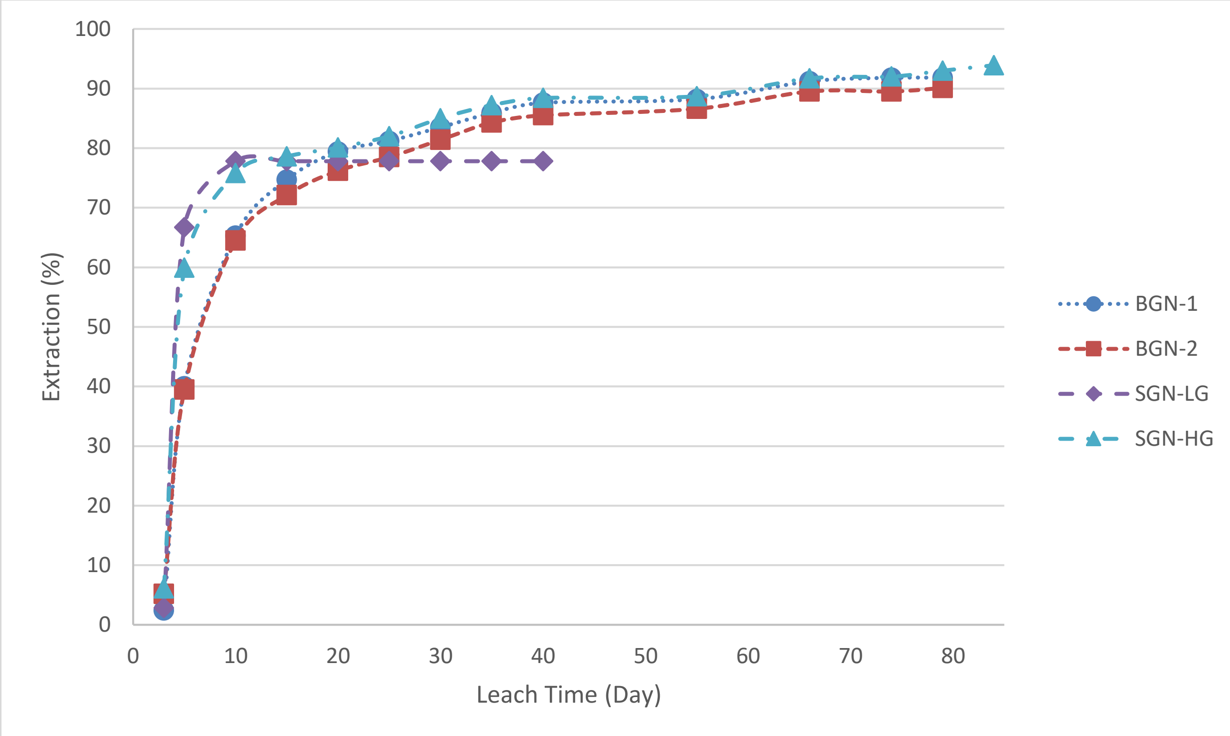
Material	Size Fraction	Max size (µm)	Head Weight (%)	Cum. Weight (%)	Tails Weight (%)	Head Assay (g/t)	Tail Assay (g/t)	Au Distribution %	Au Distribution Cum. %	Au Recovery (Fire Assay) (%)
Drainage in 120 hours (L)	-20+35	0.841	6.6	72	7.7	0.274	0.069	3.3%	74%	75.0%
22.7	-35+65	0.42	6.1	78.1	5.1	0.206	0.034	2.3%	77%	83.3%
	-65	0.21	21.9	100	22.2	0.583	0.027	23.3%	100%	95.3%
	Comp.		**100%**		**100%**	**0.547**	**0.047**	**100%**	**100%**	**91.3%**
BGN Feed (Test 2)	+2	-	0.4	0.4	0	0.069	-	0.1%	0%	-
Actual Column Recovery (Solution)	-2+1	50.8	10.8	11.2	17.8	0.686	0.069	13.5%	14%	90.0%
90.1%	-1+3/4	25.4	6.5	17.7	7.9	0.823	0.069	9.8%	23%	91.7%
Leach time (days)	-3/4+1/2	19.05	9.6	27.3	9.8	1.029	0.069	18.0%	41%	93.3%
79	-1/2+1/4	12.7	12.8	40.1	12.5	0.411	0.069	9.6%	51%	83.3%
Cyanide Consumption (g/t)	-1/4+10	6.35	17	57.1	14.8	0.446	0.069	13.8%	65%	84.6%
0.60	-10+20	1.68	8.3	65.4	7.1	0.411	0.069	6.2%	71%	83.3%
Drainage in 120 hours (L)	-20+35	0.841	6.6	72	5.6	0.274	0.034	3.3%	74%	87.5%
22.2	-35+65	0.42	6.1	78.1	4.5	0.206	0.034	2.3%	77%	83.3%
	-65	0.21	21.9	100	20	0.583	0.027	23.3%	100%	95.3%
	Comp.		**100%**		**100%**	**0.547**	**0.057**	**100%**	**100%**	**89.4%**
SGN-12 (Low Grade) Feed	+2	-	0.2	0.2	0	0.034	-	0.1%	0%	-
Actual Column Recovery (Solution)	-2+1	50.8	10.2	10.4	6.9	0.034	0.034	3.0%	3%	0.0%
77.8%	-1+3/4	25.4	5.9	16.3	3.5	0.103	0.034	5.3%	8%	66.7%
Leach time (days)	-3/4+1/2	19.05	7.3	23.6	6.4	0.069	0.034	4.4%	13%	50.0%
40	-1/2+1/4	12.7	10.9	34.5	11.3	0.069	0.034	6.5%	19%	50.0%
Cyanide Consumption (g/t)	-1/4+10	6.35	18.2	52.7	18.9	0.103	0.031	16.3%	36%	70.0%
0.38	-10+20	1.68	7	59.7	9.1	0.137	0.027	8.4%	44%	80.0%
Drainage in 120 hours (L)	-20+35	0.841	6.8	66.5	7.9	0.069	0.034	4.1%	48%	50.0%
41.9	-35+65	0.42	6.7	73.2	7	0.069	0.021	4.0%	52%	70.0%
	-65	0.21	26.8	100	29	0.206	0.014	48.0%	100%	93.3%
	Comp.		**100%**		**100%**	**0.115**	**0.026**	**100%**	**100%**	**75.0%**
SGN-22 (High Grade) Feed	+2	-	0.2	0.2	0.6	1.371	0.069	0.3%	0%	50.0%

Material	Size Fraction	Max size (µm)	Head Weight (%)	Cum. Weight (%)	Tails Weight (%)	Head Assay (g/t)	Tail Assay (g/t)	Au Distribution		Au Recovery (Fire Assay) (%)
								%	Cum. %	
Actual Column Recovery (Solution)	-2+1	50.8	10.6	10.8	13.1	0.926	0.069	9.1%	9%	92.6%
93.9%	-1+3/4	25.4	6.2	17	5.7	1.714	0.103	9.9%	19%	94.0%
Leach time (days)	-3/4+1/2	19.05	9.2	26.2	8.4	1.269	0.069	10.8%	30%	94.6%
84	-1/2+1/4	12.7	13.6	39.8	15.4	0.720	0.069	9.1%	39%	90.5%
Cyanide Consumption (g/t)	-1/4+10	6.35	18.7	58.5	18.9	0.754	0.069	13.1%	52%	90.9%
0.75	-10+20	1.68	6.9	65.4	6.7	0.549	0.137	3.5%	56%	75.0%
Drainage in 120 hours (L)	-20+35	0.841	5.9	71.3	5.5	0.480	0.137	2.6%	58%	71.4%
20.6	-35+65	0.42	6.4	77.7	4.9	0.411	0.069	2.4%	61%	83.3%
	-65	0.21	22.3	100	20.8	1.886	0.024	39.1%	100%	98.7%
	Comp.		**100%**		**100%**	**1.076**	**0.070**	**100%**	**100%**	**93.6%**

(McClelland Laboratories, Inc.)

The results show good gold recovery from most samples except from the low-grade Sericite Gneiss material showed the poorest response to column leaching at 75% gold extraction. The high-grade Sericite Gneiss material produced 93.6% recovery. Overall, the rate of leaching for all tests was high shown in Figure 13-6.

Figure 13-6: Gold Extraction BGN and SGN Column Leach Tests 1996 (McClelland Laboratories, Inc.)



After the tests were stopped, drainage was recorded, frequently for the first 24-hours, and then every 24-hours after that until drainage of the columns was completed. The results indicate that between 13.3 to 14.8% moisture was required to achieve breakthrough and that the final drain down moisture ranged between 7.78 to 10.5%.

Three main areas of concern were raised were confirmed by this test work. The three concerns were:

- Ore degradation during column leaching
- Column test cyanide consumptions being higher than commercial and/or bottle roll test consumptions
- Fire assay head grades are lower than calculated heads from metallurgical tests

A further comparison was added by Glamis Gold (Ron Wyrick) between the Picacho Mine and the potential Imperial Project:

Sites 1 and 3 at the Picacho Mine were stacked with primarily oxide material. Since the Imperial Project ore is all oxide, the overall recovery of the Imperial Project heap leach pad is expected to resemble that achieved on Sites I and 3. Historic results from Sites I and 3 show 5,604,680 tons of ore were stacked containing 202,479 ounces. Of those contained ounces, 149,523 ounces of fine gold were produced for an average recovery of 73.8%.

Based on these relationships Mr. Wyrick recommended a recovery for the Imperial Project of 73%.

13.7.2 Bottle Roll Tests and Column Leach Tests on BGN and SGN Composites

McClelland Laboratories also conducted column leach tests and bottle roll tests on additional samples of BGN and SGN samples from the Imperial deposit. Some material was rejected from columns due to size limitations and was then subjected to bottle roll testing to determine metallurgical recovery. Bottle roll conditions employed a 0.10% cyanide solution, 72-hour duration of tests, and pH 11.0 (McClelland Laboratories Inc., 1996).

The remaining material was loaded into columns. According to the previous Feasibility Study conducted by Western States Engineering, the material used for these tests originated from four drill core holes – 2 cores (PQ and HQ) coming from the East pit, and 2 cores (PQ and HQ) coming from the West pit (Western States Engineering, 1996). However, this information could not be verified in the metallurgical data provided. All four cores were crushed and composited according to ore type: biotite gneiss and sericite gneiss. Multiple sericite gneiss composites were created, to delineate "high-grade" and "low-grade" material, although no explanation of where the distinction lies between the two designations was provided.

Two columns were prepared for the biotite gneiss material, measuring 38.1 cm (15") in diameter, each containing approximately 500 kg (1,100 lbs.) of material. Two columns were prepared for the sericite gneiss material (one for high grade and one for low grade material). The columns measured 30.5 cm (12") in diameter and contained approximately 317.5 kg (700 lbs.) of material each. All columns were operated with 350 ppm cyanide, at a rate of 11 LPH/m^2 (0.0045 gal/min/ft^2). No caustic was added to the columns, and the pH of the solution was maintained at least 10.3 for the duration of the leach cycle. The durations of the biotite gneiss column tests were 86 days; the low-grade sericite gneiss column was operated for 40 days, the high-grade sericite gneiss column was run for 89 days (Western States Engineering, 1996).

The results of the column tests as well as the bottle roll tests on column size rejects are given in Table 13-23.

Table 13-23: 1996 Bottle Roll Tests and Column Leach Tests for Phase 2 Metallurgical Testing (McClelland Laboratories, Inc.)

	Head Grade		Recovery		
Phase 2 Bottle Roll Tests (column rejects)	**Fire Assay (g/t)**	**Calculated Head Grade (g/t)**	**Fire Assay (%)**	**Calculated Recovery (%)**	**Cyanide Consumption (kg/t)**
Biotite Gneiss	0.549	0.514	81.1	79.2	0.21
Sericite Gneiss (Low Grade)	0.103	0.103	74.8	71.0	0.12
Sericite Gneiss (High Grade)	1.097	0.926	91.7	88.9	0.12
Sericite Gneiss (Averaged)	0.600	0.514	83.3	80.0	0.12
Phase 2 Column Tests					
Biotite Gneiss	0.549	0.583	90.4	90.9	0.62
Sericite Gneiss (Low Grade)	0.103	0.137	75.0	77.8	0.38
Sericite Gneiss (High Grade)	1.063	1.131	93.6	93.9	0.75
Sericite Gneiss (Averaged)	0.583	0.634	84.3	85.9	0.56

As shown above, the average recoveries for biotite gneiss samples were 79.2% for the oversized column rejects compared to 90.9% for column test material. Low grade sericite samples showed similar recoveries for both bottle roll tested column rejects (71.0%) and column leached material (77.8%). High grade sericite material also showed similar results for bottle roll tests (88.9%) and column leach tests (93.9%). The results of the tests on sericite material show that there is at least a moderate correlation between gold head grade and gold recovery.

Additionally, various size fractions of biotite gneiss and sericite gneiss material were subjected to bottle roll tests (McClelland Laboratories Inc., 1996). Two runs of tests were performed on each type of material (biotite gneiss, low-grade sericite gneiss, and high-grade sericite gneiss), and the average of these two runs was calculated. No procedural details could be determined from the metallurgical report, but details of these tests can be determined from previous Feasibility and PEA reports (Western States Engineering, 1996). The average results are given in Table 13-24.

Table 13-24: 1996 Bottle Roll Tests on Size Fractions from Biotite- and Sericite Gneiss Samples (McClelland Laboratories, Inc.)

	Size Fraction	Max size (µm)	HCL Head Grade (g/t)	Fire Head Grade (g/t)	Head Wt. (%)	Sol'n Grade (g/t)	HCL, Tail (g/t)	Fire Tail (g/t)	Tail Wt. (%)	Rec. by HCL (%)	Fire Rec. (%)	Rec. (calc) (%)	CN Cons. (kg/t)
BGN-2	-2+1	50.8	0.686	0.771	11.2%	0.291	0.086	0.117	17.8%	88.5%	84.9%	85.0%	0.22
3	-1+3/4	25.4	0.411	0.514	6.5%	0.189	0.163	0.195	7.9%	68.0%	62.0%	84.0%	0.16
4	-3/4+1/2	19.05	0.754	0.720	9.6%	0.257	0.094	0.137	9.8%	85.9%	91.0%	80.7%	0.22
5	-1/2+1/4	12.7	0.617	0.583	12.8%	0.240	0.163	0.120	12.5%	72.6%	78.8%	78.3%	0.20
6	-1/4+10	6.35	0.617	0.583	17.0%	0.291	0.146	0.105	14.8%	72.2%	81.5%	78.4%	0.20
7	-10+20	1.68	0.514	0.377	8.3%	0.154	0.086	0.084	7.1%	75.2%	77.7%	75.6%	0.29
8	-20+35	0.841	0.480	0.377	6.6%	0.111	0.077	0.094	5.6%	72.1%	75.0%	67.9%	0.20
9	-35+65	0.42	0.343	0.240	6.1%	0.086	0.034	0.052	4.5%	76.3%	78.2%	67.8%	0.16
10	-65	0.21	0.617	0.583	21.9%	0.223	0.034	0.053	20.0%	93.1%	90.6%	89.7%	0.22
	Average		0.560	0.528	100.0%	0.205	0.098	0.106	100.0%	78.2%	80.0%	78.6%	0.21
SGN-12	-2+1	50.8	0.034	0.048	10.4%	0.000	0.000	0.024	6.9%				0.16
13	-1+3/4	25.4	0.000	0.065	5.9%	0.000	0.000	0.031	3.5%				0.16
14	-3/4+1/2	19.05	0.274	0.274	7.3%	0.000	0.000	0.041	6.4%				0.16
15	-1/2+1/4	12.7	0.069	0.082	10.9%	0.034	0.017	0.045	11.3%				0.12
16	-1/4+10	6.35	0.103	0.062	18.2%	0.034	0.103	0.062	18.9%				0.08
17	-10+20	1.68	0.069	0.055	7.0%	0.034	0.034	0.003	9.1%				0.12
18	-20+35	0.841	0.137	0.055	6.8%	0.000	0.000	0.003	7.9%				0.08
19	-35+65	0.42	0.103	0.072	6.7%	0.000	0.000	0.007	7.0%				0.08
20	-65	0.21	0.137	0.171	26.8%	0.069	0.000	0.014	29.0%	100.0%	92.0%	91.0%	0.12
	Average		0.103	0.098	100.0%	0.019	0.017	0.026	100.0%	100.0%	92.0%	91.0%	0.12
SGN-22	-2+1	50.8	0.617	0.497	10.8%	0.206	0.069	0.070	13.7%	87.2%	85.9%	86.9%	0.12
23	-1+3/4	25.4	1.954	2.674	6.2%	0.343	0.154	0.139	5.7%	79.5%	94.8%	81.2%	0.08
24	-3/4+1/2	19.05	0.823	0.737	9.2%	0.223	0.094	0.094	8.4%	81.2%	87.2%	81.2%	0.16
25	-1/2+1/4	12.7	1.303	1.354	13.6%	0.291	0.120	0.067	15.4%	81.3%	95.1%	88.7%	0.11
26	-1/4+10	6.35	0.686	0.686	18.7%	0.257	0.137	0.120	18.9%	77.6%	82.5%	79.8%	0.11
27	-10+20	1.68	0.549	0.651	6.9%	0.206	0.129	0.117	6.7%	78.8%	82.3%	80.4%	0.14
28	-20+35	0.841	0.686	0.549	5.9%	0.343	0.146	0.137	5.5%	84.3%	74.2%	85.1%	0.12

	Size Fraction	Max size (µm)	HCL Head Grade (g/t)	Fire Head Grade (g/t)	Head Wt. (%)	Sol'n Grade (g/t)	HCL, Tail (g/t)	Fire Tail (g/t)	Tail Wt. (%)	Rec. by HCL (%)	Fire Rec. (%)	Rec. (calc) (%)	CN Cons. (kg/t)
29	-35+65	0.42	0.514	0.446	6.4%	0.189	0.060	0.093	4.9%	87.4%	78.4%	81.8%	0.12
30	-65	0.21	1.749	1.851	22.3%	0.874	0.129	0.068	20.8%	93.7%	96.3%	96.6%	0.10
	Average		0.987	1.050	100.0%	0.326	0.115	0.100	100.0%	83.4%	86.3%	84.6%	0.12
	Overall Avg									87.2%	86.1%	84.7%	0.15

13.8 Test Work Summary

Several bottle roll cyanidation and column leach cyanidation tests have been completed from 1988 – 1996 on samples from the Imperial project deposit.

Coarse material bottle roll recoveries ranged from 60% to 100%, with an average of approximately 86.3% when employing the hot cyanide assay technique. Column test recoveries ranged from 61.5% to 95%, with an average of 84.2%.

The above averages use both biotite- and sericite-type ores. Testing of biotite gneiss material result in approximately 86.5% recovery from bottle roll tests, and 83.9% recovery from column leach tests. Sericite Gneiss material shows approximately 86.1% recovery from bottle roll tests, and 84.9% recovery from column leach tests for crushed ore ranging from -2” in 1994-1995 tests to a P90 of 1” for tests completed in 1996. The average for the column tests by material type and size is given in Table 13-25.

Table 13-25: Summary of Column Leach Tests by Material Size and Type

Year	Size of Material	Material Type		Overall Average
		BGN	SGN	
1994-1995	-2” (50.8 mm)	82.6%	85.3%	83.2%
1996	P90: 1” (25.4 mm)	90.4%	84.3%	87.3%
Overall		83.9%	84.9%	84.2%

The bottle roll tests tended to have low cyanide consumption, with the average consumption from all material tested at a level of 0.16 kg/tonne of ore, with biotite gneiss tests using slightly less cyanide (0.16 kg/t) compared to sericite gneiss (0.17 kg/t). Column Leach tests recorded approximately 0.35 kg/tonne ore cyanide consumption, with tests involving biotite gneiss recording 0.35 kg/t, and 0.37 kg/t for sericite gneiss.

Overall, the Imperial Project material test was amenable to coarse sized cyanidation. Two major types of mineralogy have been identified: biotite- and sericite gneiss; both types of material exhibited good recovery with fast leach kinetics. There was some indication that lower grade materials may have lower gold recovery due to the constant tail effect.

13.9 Recommended Process Variables

The original feasibility study Western States Engineering in 1996 used the average Picacho gold extraction of 73% for Imperial material, assuming a conventional dedicated leach pad and effective leach period of

210 days. Each lift of 25 ft or 50 ft would be leached for 90 days before new material was dumped directly from trucks. An ultimate pad height of 300 ft was indicated based on the production rate of 20,000 t/d to 30,000 t/d.

The PEA produced by SRK in 2012 concluded that crushed material would have a higher recovery than ROM ore as column leach test work was conducted on minus 2-inch feed and achieved over 80% recovery for both BGN and SGN samples. Based on this their gold recovery recommendation for a 2-inch crushed product was 83%.

GRE has developed a hybrid heap leach system consisting of both a crushed feed and a ROM feed to the heap leach facility (HLF). Approximately 20,000 tpd of crushed product is proposed to be truck dumped on the HLF along with approximately 13,000 tpd of ROM material.

For a ROM only option GRE agrees with the previous recommendations and believes that an ultimate gold recovery of 73% should be achievable. This fits well with the data provided by Picacho and GRE's experience with other neighboring mines that utilize a ROM HLF.

ROM Only Option

- ROM Particle Size: Nominal minus 6"
- ROM Gold Recovery: 73% recovery
- Primary Leach Duration: 90 days with two secondary cycles of similar duration

Given that the new design is a hybrid of crush and ROM a modified recovery calculation is required. A cutover grade will be employed to determine what material is directed to crushing and a cutoff grade (COG) will determine what is sent to ROM or waste. The current cut-over grade for crushing has a minimum of 0.014 opt (0.47 g/t). Given that the ROM material will be lower grade a more conservative gold recovery estimate has been applied of 65%. The crushed material gold recovery is predicted at 80% slightly lower than the SRK prediction of 83%. GRE lowered this recovery because of the variability in the metallurgical test data. Although most of the column and bottle roll tests performed exceptionally, there are a few outliers that still lack explanation.

Combined Crush/ROM Option

- Crush Particle Size: P80 1"
- Crush Gold Recovery: 80% recovery
- Primary Leach Duration: 90 days with two secondary cycles of similar duration
- ROM Particle Size: Nominal minus 6"
- ROM Gold Recovery: 65% recovery
- Primary Leach Duration: 90 days with two secondary cycles of similar duration

The reagent consumptions were estimated from both the test work and from data provided by Picacho and neighboring mines. These are conservative estimates.

<u>Reagent Consumptions</u>

- CN consumption: 0.42 lb./t (0.21 kg/t)
- Lime Consumption: 2.4 lb./t (1.2 kg/t)

13.10 Mineral Processing Recommendations

The following recommendations have been put forward to allow the Imperial Project to advance to the next phase of development. The main area of interest is to ensure that test work is conducted on samples that are representative of the deposit in spatial, mineralogical and grade terms.

- Additional column leach tests focusing on ROM, crush size, deposit location, mineralogy and grade.
- Comparable bottle roll leach tests on column samples to confirm the relationship between the two testing methods.
- Percolation and drain down testing with simulated heap loading to ensure that the heap will perform as predicted.
- Geotechnical investigations into the heap stability.
- Organic carbon and mercury assays should be conducted on some existing exploration materials to confirm the potential of these elements across the deposit.
- Carbon loading kinetic test work should be conducted to confirm no issues with solution metal content as well as estimate the ADR circuit capacity.
- Closure testing on the spent heap materials should be conducted.

It is estimated that this additional work will cost approximately $500,000 including drilling for new metallurgical samples. For a complete list of recommendations see Section 26.0.

14.0 MINERAL RESOURCE ESTIMATION

The following was prepared by SRK for the December 30, 2019 technical report. Nothing has changed in this section since the SRK report was published and the text for Section 14 of the SRK report is included here verbatim.

The Mineral Resource Statement presented herein represents the second mineral resource evaluation prepared for the Imperial Gold Project in accordance with the Canadian Securities Administrators National Instrument 43-101. As no additional data has been generated for the project since 2012, the mineral resource model described in this section is unchanged from that generated by SRK (2012) but has been re-stated to consider late 2019 economics. There has been no restating of resources as part of this updating of the Technical Report to include an Preliminary Economic Analysis, as the assumptions used to estimate resources remain valid 5 months after the publishing of the previous Technical Report.

The mineral resource model prepared by the SRK QP considers 349 reverse circulation (RC) boreholes drilled by various operators during the period of 1987-1996. The resource estimate was completed under the supervision of Glen Cole, PGeo. (APGO #1416), who is an independent qualified persons as this term is defined in NI 43-101. The effective date of this mineral resource estimate is December 30, 2019.

This section describes the resource estimation methodology and summarizes the key assumptions considered by the QP. In the opinion of the QP, the resource evaluation reported herein is a reasonable representation of the global gold mineral resources found in the Imperial Gold Project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (November 29, 2019) and are reported in accordance with the Canadian Securities Administrators NI 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

The database used to estimate the Imperial Gold Project mineral resources was audited by the SRK QP. The QP is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for the gold mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

Gemcom GEMS™ ("GEMS TM") software was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate mineral resources. GoCad and Leapfrog software were used to create the 3D geological model. The Geostatistical Software Library- family of software was used for geostatistical analysis and variography.

14.1 Resource Estimation Procedures

The mineral resources reported herein were estimated using a geostatistical block modelling approach informed from borehole data.

The evaluation of mineral resources for the Imperial Gold Project involved the following procedures:

- Database compiling and verifying

- Resource modelling
- Modelling of 3D wireframe models for the topography, gold mineralized zone, gravel zone and below gravel/bedrock zone
- Validating of database and wireframe models
- Data processing (compositing and capping), statistical analysis and variography
- Selecting of estimation strategy and estimation parameters
- Block modelling and grade estimating
- Validating, classifying and tabulating
- Assessing of "reasonable prospects for economic extraction" and selecting reporting COG
- Preparing of mineral resource statement

14.2 Resource Database

14.2.1 General

Data used to evaluate the mineral resource was provided by Delta as comma delimited tables containing borehole data. The Imperial Gold Project database contains 349 boreholes, 344 of which are located within the resource estimation area. Analytical data for the Imperial Gold Project is primarily sourced from drilling completed between 1987 and 1996 by Gold Fields, Glamis Gold, and other historical operators. The borehole data includes collar location, down-hole survey data, lithology codes and 36,361 sample intervals assayed for gold. The mineral resource statement is informed by a total of 190,047 ft of RC drilling.

Geological (gravel and bedrock) and gold mineralization wireframes were generated by the SRK QP was based on borehole lithological contacts and assay results.

14.2.2 Data Validation

The authors performed the following validation steps on the borehole data:

- Check minimum and maximum values for each quality value field and confirming and editing those outside of expected ranges
- Check for gaps, overlaps, and out of sequence intervals for both assays and lithology tables

The original assay database contained a few minor errors (including out of sequence or negative intervals). The errors were corrected by the QP. Additionally, four boreholes were removed from the estimation database due to overlapping collar and survey information (K15, O10, R23, and R16).

On completion of the validation procedure, the QP considers the database and modelled mineralization wireframes suitable for resource estimation.

14.3 Solid Body Modelling

The gold mineralization on the Imperial gold project occurs primarily within structurally controlled hematite and limonite altered breccias and fault filled gouge zones hosted in biotite or sericite altered gneiss.

The SRK QP's geological interpretation includes wireframes of the gold mineralization and the surfaces defining the contact between the Quaternary gravel sediments and the Mesozoic bedrock (Figure 14-1 and Figure 14-2). The gold mineralized zone was estimated using a traditional wireframe interpretation constructed from a sectional interpretation of drilling data. Sections were spaced 200 ft apart and angled at a 15° to 195° orientation. The modelled gold mineralized zone was then subdivided into three domains displaying different strike or dip directions. All modeled domains and surfaces created by the SRK QP are shown in Figure 14-1 and Figure 14-2. Each wireframe was assigned a numerical rock code by the QP to facilitate identification during resource estimation and tabulation (Table 14-1).

Figure 14-1: Oblique Section and Long Section Showing Gold Mineralization Domains, Topography, Gravel/Bedrock Contact and Block Model Extent at the Imperial Gold Project

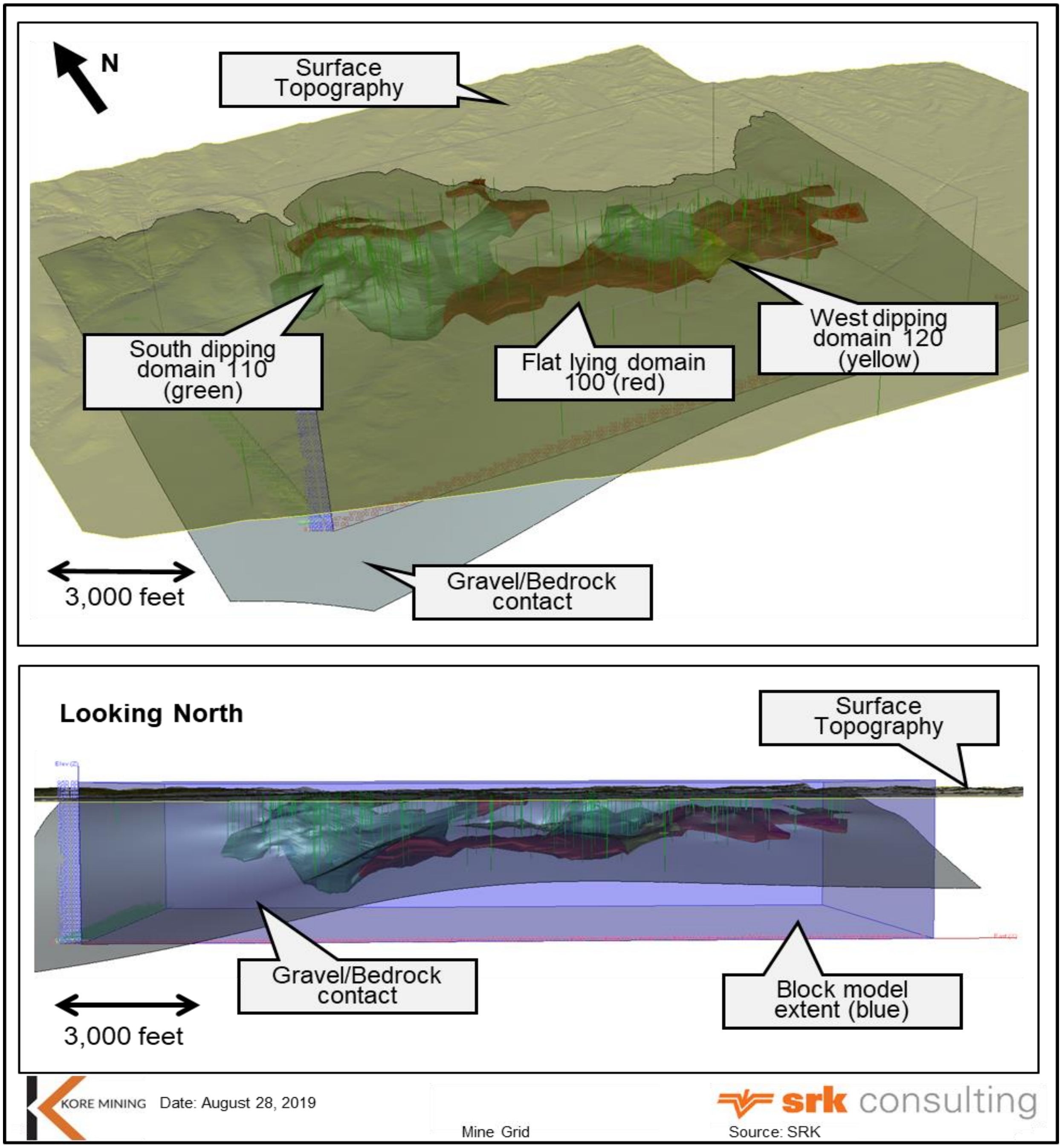


Figure 14-2: Cross-section Showing the Lithology-Gold Mineralization Relationship at the Imperial Gold Project

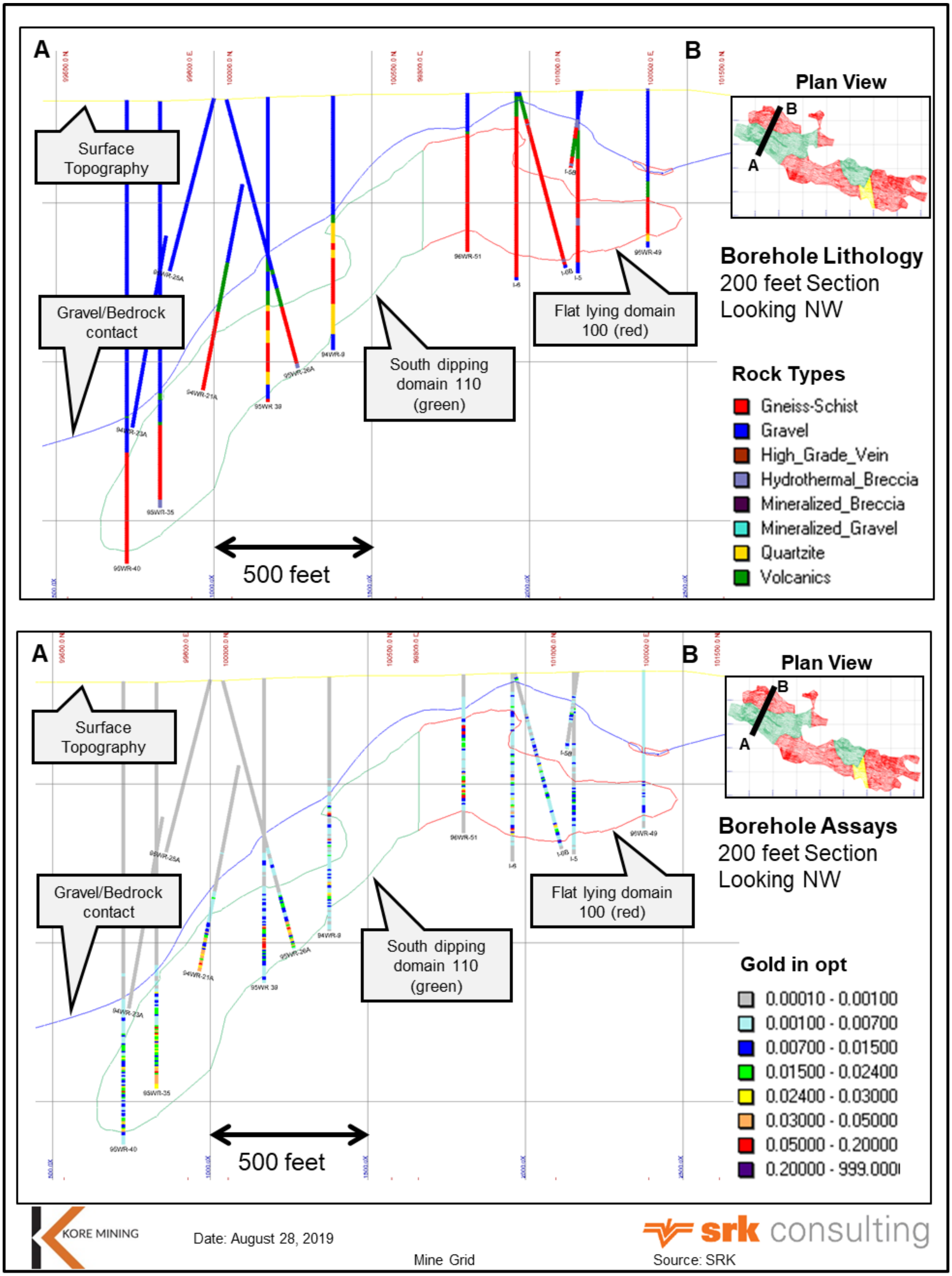


Table 14-1: Rock Codes in the Imperial Gold Project Block Model

Zone	Domain	Rock Code	Density (t/ft^3)
Grade Zone	Flat lying wireframe	100	0.077
	South dipping wireframe	110	0.077
	West dipping wireframe	120	0.077
Outside Wireframe Model	Gravel with grade	200	0.067
	Bedrock with grade	300	0.076

14.4 Compositing, Outlier Analyses and Statistics

The wireframes representing the interpreted gold zones were used to code a zone field into a block model (Table 14-1). Table 14-2 illustrates the basic sample gold grade and sample length statistics of the original borehole data. For unsampled borehole intervals intersecting geological wireframes, SRK assigned a detection limit grade of 0.0005 oz/t gold.

Table 14-2: Basic Statistics of Raw Borehole Samples for the Imperial Gold Project

Domain	Unit	Count	Min	Max	Mean	Std. Dev.	Variance	COV
100	Au oz/t	4,521	0.0005	1.522	0.0161	0.0345	0.0012	2.1416
110		3,942	0.0005	0.262	0.0160	0.0197	0.0004	1.2345
120		187	0.0005	0.227	0.0216	0.0281	0.0008	1.3020
All 100's		8,650	0.0005	1.522	0.0162	0.0286	0.0008	1.7678
200		15,917	0.0005	0.144	0.0010	0.0021	0.0000	2.2337
300		10,601	0.0005	0.226	0.0015	0.0038	0.0000	2.4954
100	ft	4,521	2	23	5.0495	0.5895	0.3475	0.1167
110		3,942	2	11	5.0342	0.3784	0.1432	0.0752
120		187	5	5	5.0000	0.0000	0.0000	0.0000
All 100's		8,650	2	23	5.0415	0.4970	0.2470	0.0986
200		15917	2	500	5.6268	6.8690	47.1828	1.2208
300		10,601	1	185	5.2032	2.6725	7.1425	0.5136

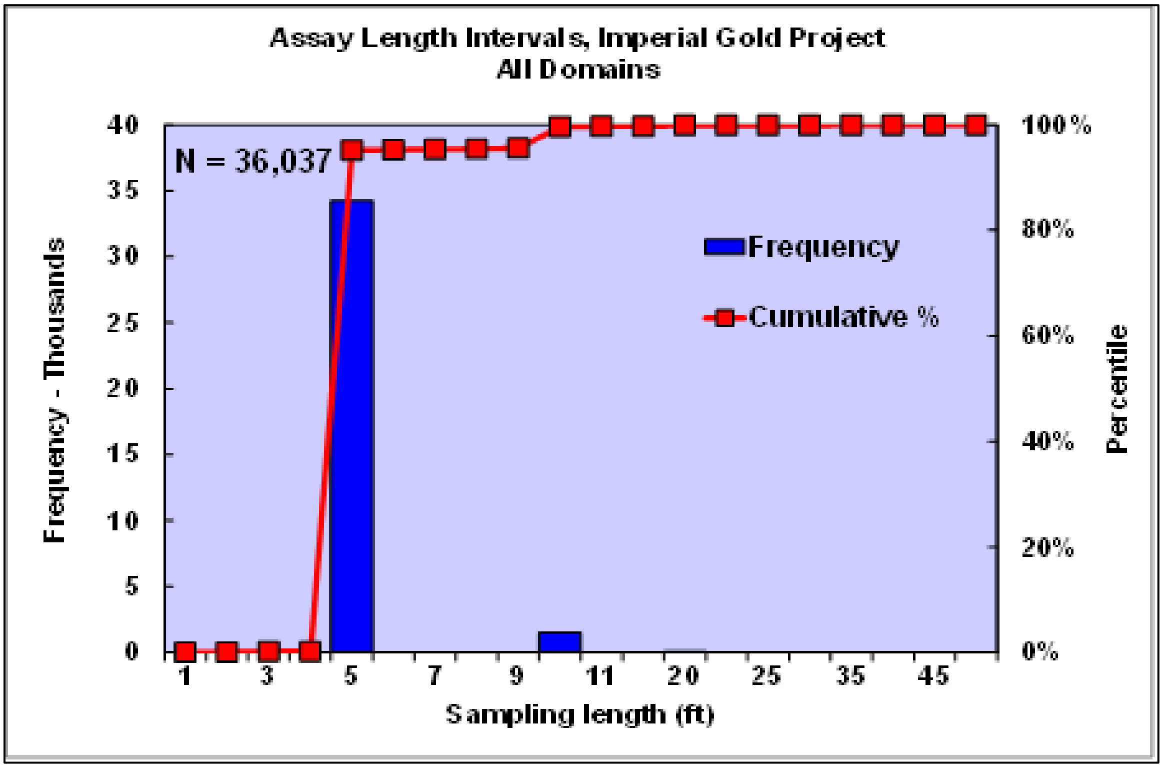
The majority of RC assay samples were collected at five ft intervals (Figure 14-3), irrespective of geological contacts. After a review of sample length histograms for each zone, gold assays were composited to 10 and 20 ft intervals for comparative geostatistical analysis and variography. The SRK QP examined the impact of composite length on grade continuity and estimation and observed that 20 ft composite intervals yielded reasonable resource estimates for the anticipated block size. All subsequent analysis was performed using 20 ft composites.

For each zone, a capping value was determined by analyzing histograms and cumulative frequency plots of gold composites (Figure 14-3).

Capping values were adjusted iteratively by reference to summary statistics to ensure robustness of statistics to chosen capping values (Table 14-3).

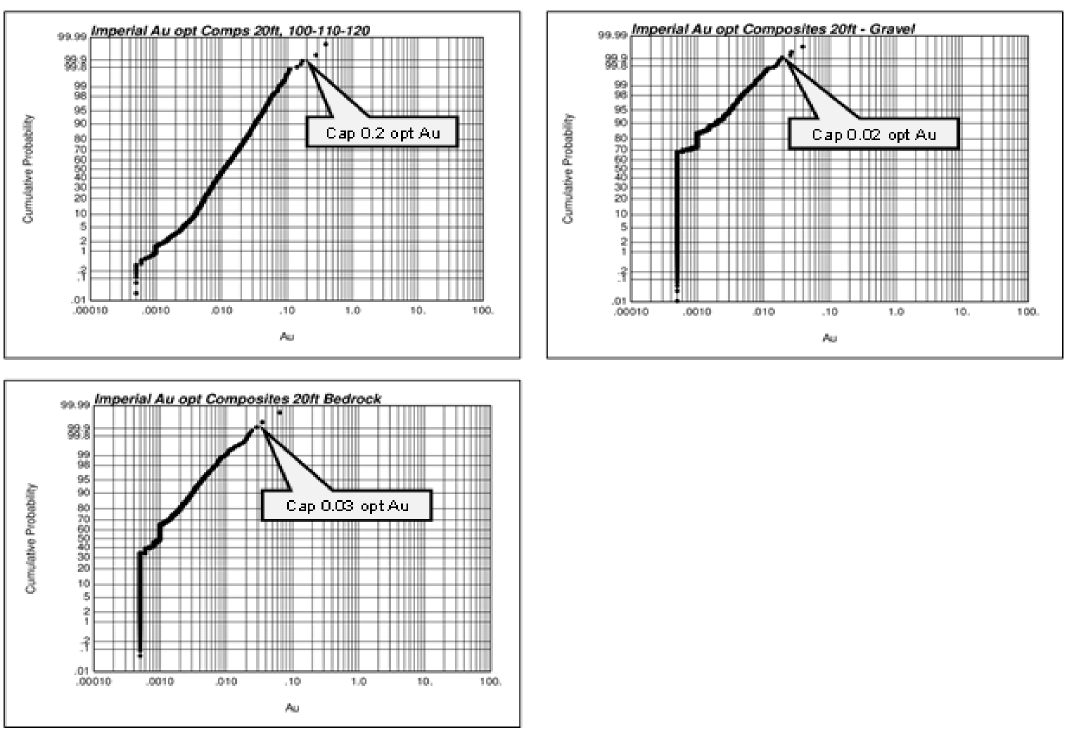
Basic statistics for uncapped and capped gold composites are shown in Table 14-4.

Figure 14-3: Sample Length Histograms for all Domains (100, 110, 120, 200 and 300)



Source: SRK 2012

Figure 14-4: Cumulative Frequency Plots for Gold Composites Within Gold Mineralization Wireframes, Within the Gravel Domain 200, and Within Bedrock Domain 300). Selected Capping Value as Illustrated



Source: SRK 2012

Table 14-3: Imperial Project Capping Values on 20-foot Composites

Domain	Cap Grade (Au oz/t)	# Capped	Percentile Cap
All 100's	0.2	2	99.9
200	0.02	4	99.9
300	0.03	2	99.9

Table 14-4: Statistics for Uncapped and Capped Gold Composites

Domain	Variable	Count	Min	Max	Mean	Std. Dev.	Variance	COV
100	Uncapped Grade (Au oz/t)	1,194	0.0005	0.395	0.0159	0.0205	0.0004	1.2877
110		1,027	0.0005	0.111	0.0159	0.0143	0.0002	0.9012
120		49	0.0021	0.093	0.0210	0.0191	0.0004	0.9108
All 100's		2,270	0.0005	0.395	0.0160	0.0180	0.0003	1.1206
200		4,579	0.0005	0.041	0.0010	0.0016	0.0000	1.5935
300		2,908	0.0005	0.064	0.0015	0.0023	0.0000	1.5372
100	Capped Grade (Au oz/t)	1,194	0.0005	0.200	0.0157	0.0173	0.0003	1.1027
110		1,027	0.0005	0.111	0.0159	0.0143	0.0002	0.9012
120		49	0.0021	0.093	0.0210	0.0191	0.0004	0.9108
All 100's		2,270	0.0005	0.200	0.0159	0.0161	0.0003	1.0115
200		4,579	0.0005	0.020	0.0010	0.0014	0.0000	1.3928
300		2,908	0.0005	0.030	0.0015	0.0020	0.0000	1.3658

14.5 Density

The density data was sourced from the WSE (1996). In 1994 and 1995, a core drilling program consisting of nine boreholes was conducted to obtain bulk mineralized samples. Samples were analysed for metallurgical testing, independent assay verification, geotechnical characteristics and rock type bulk density.

A total of 32 core samples were collected for bulk density determination. Average tonnage factors were assigned to “ore”, waste rock and gravel based on weighted average bulk density results. For all other domains, a weighted average density value was assigned (Table 14-1):

- Grade zones (Domains 100, 110 and 120): 0.077 t/ft^3;
- Gravel (Domain 200): 0.067 t/ft^3; and
- Bedrock outside grade zone (Domain 300): 0.076 t/ft^3.

14.6 Variography

The SRK QP evaluated the spatial distribution of gold by calculating a variogram and correlogram for capped composites of gold and also for its normal score transform. A total of four spatial metrics was considered to infer the correlation structure that was used in grade estimation. Continuity directions were assessed based on the orientation of each domain, composites and the spatial distribution of gold grades. Further, variogram calculation considered sensitivities on orientation angles prior to finalizing the correlation orientation. All variogram analysis and modelling was performed using the Geostatistical

Software Library (GSLib; Deutsch and Journal, 1998), Isatis was used to confirm principal orientations and in some cases, the lack thereof.

Variogram modelling is based on the combination of the four metrics; however, the correlogram tends to give reasonably clear continuity structures that are often amenable to variogram fitting. The fitted models are based on the inverted correlogram of capped gold composites (Table 14-5 and Figure 14-5).

The variograms were fitted in GEMS TM using the principal azimuth, dip, intermediate azimuth method. The methodology to set up this rotation is outlined as follows:

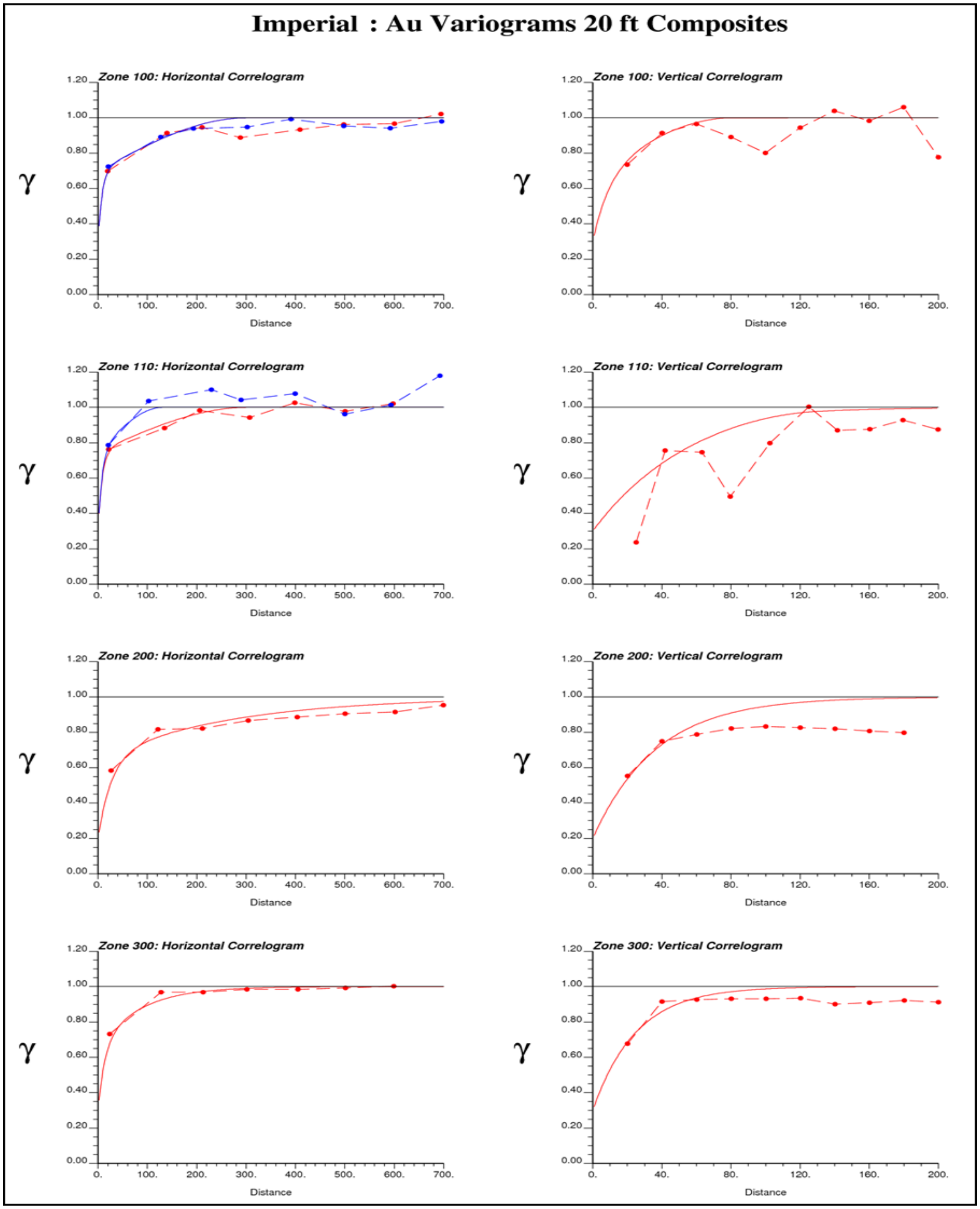
- Principal azimuth is the true azimuth of the anisotropy X axis in degrees;
- Principal dip is the dip angle of the anisotropy X axis in degrees (negative downward); and
- Intermediate azimuth is the azimuth of the anisotropy Y axis in degrees.

Table 14-5: Variogram Models and GEMS TM Angles for the Imperial Gold Project

	GEMS TM Angles			Variogram Model						
Domain	Princ. Azimuth	Princ. Dip	Interm. Azimuth	Nugget	Structure No.	Type	Var. Cont.	Rx	Ry	Rz
100	110	-5	20	0.3	1	Exponential	0.40	25	25	30
					2	Spherical	0.30	300	300	80
110	130	-25	65	0.3	1	Exponential	0.45	25	25	130
					2	Spherical	0.25	300	130	130
120*	110	35	5	0.3	1	Exponential	0.40	25	25	30
					2	Spherical	0.30	300	300	80
200	0	0	0	0.2	1	Exponential	0.45	80	80	110
					2	Exponential	0.35	800	800	110
300	0	0	0	0.3	1	Exponential	0.35	40	40	75
					2	Exponential	0.35	250	250	75

* Ranges were borrowed from Domain 100 as Domain 120 had insufficient data for variogram modeling.

Figure 14-5: Modelled Gold Variograms for the Imperial Gold Project Domains 100, 110, 200, and 300



Source: (SRK 2012)

Note: The correlogram is inverted for the purposes of variogram modeling. The solid lines correspond to the fitted model, while the dashed lines correspond to the experimental variogram in those same directions.

14.7 Block Model and Grade Estimation

14.7.1 Block Model

A block model was created in GEMS TM to cover the entire area of gold mineralization at the Imperial Gold Project. The block model was based on the WSE (1996) block model set on a grid of 50 feet by 50 feet by 40 feet. The model parameters are summarized in Table 14-6.

Table 14-6: Imperial Gold Project Block Model Parameters

Direction	Size (ft)	Minimum*	Maximum*	Number of Blocks
East-West	50	97,000	106,900	198
North-South	50	96,250	102,450	124
Vertical	40	(-)1020	980	50

* Mine Grid.

14.7.2 Grade Interpolation

Gold grades were estimated by ordinary kriging. The variogram models used for estimation are summarized in Table 14-5. Gold grades were estimated in each domain separately using capped composites from within that domain and search parameters summarized in Table 14-7.

The SRK QP evaluated the impact of varying estimation parameters in order to select optimal estimation parameters for block grade interpolation. The results of this comparative study indicate that the grade estimation for these domains is not very sensitive to slight variations of estimation parameters.

Three estimation runs were used to populate the block model with gold grades for zones 100, 110, and 120, whereas only two passes were used for the 200 and 300 domains not constrained by hard mineralization wireframes. The first and second estimation passes considered full variogram ranges with the third pass doubled the variogram range. For comparison, gold grades were also estimated using an inverse distance algorithm (power of two) using the same estimation parameters.

Table 14-7: Grade Estimation Search and Rotation Parameters

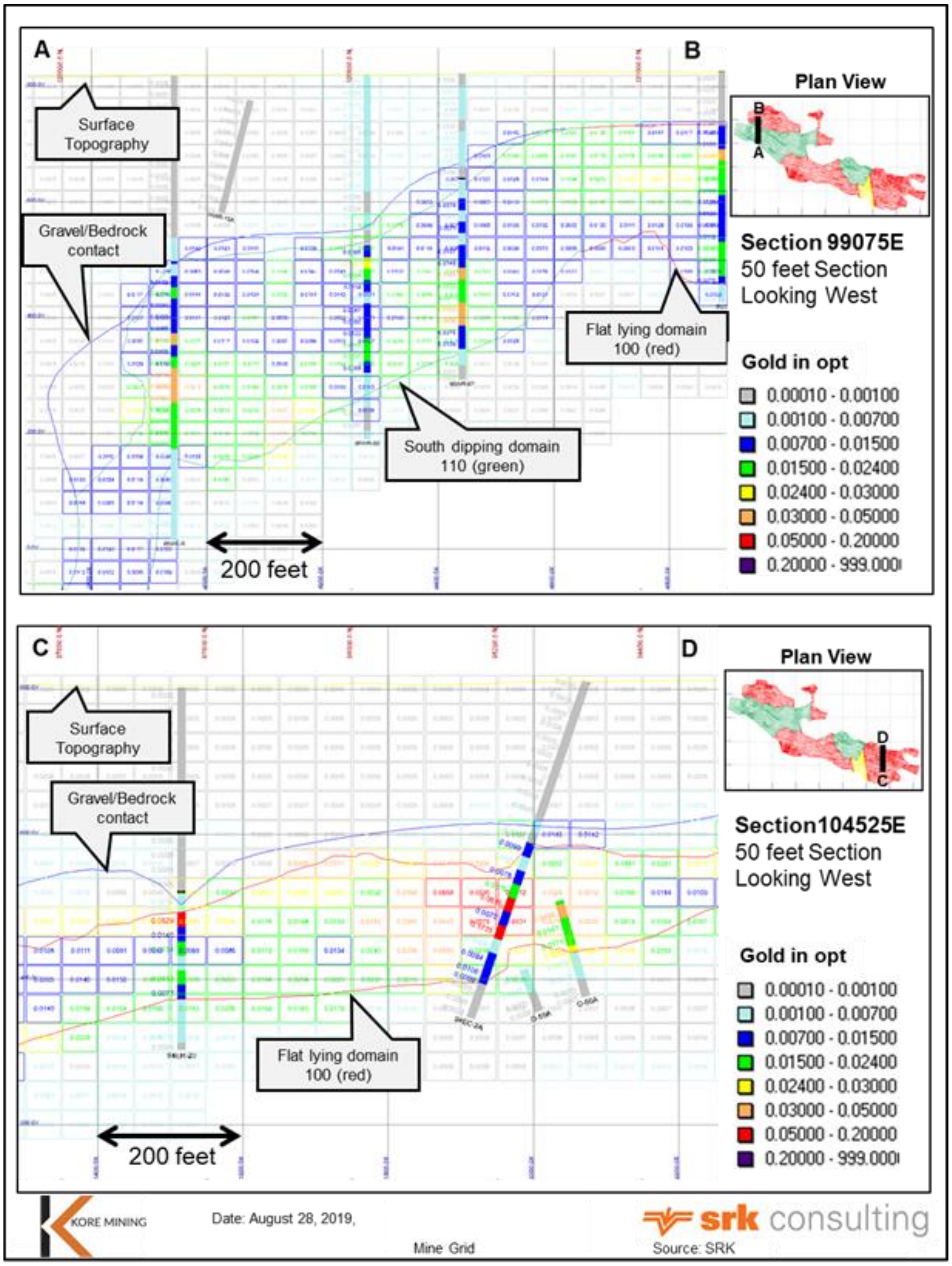
Interpolation Parameters	1st Pass	2nd Pass	3rd Pass
Domains 100, 110 and 120			
Interpolation Method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Search Type	Octant	Ellipsoidal	Ellipsoidal
Minimum Number of Octants	2	-	-
Maximum Composite per Octant	5	-	-
Maximum Composite per Borehole	2	-	-
Minimum Number of Composites	3	2	1
Maximum Number of Composites	8	10	12
Search Distance	1 x variogram	1 x variogram	2 x variogram
Domains 200 and 300			
Interpolation Method	Ordinary Kriging	Ordinary Kriging	-
Search Type	Octant	Ellipsoidal	-
Minimum Number of Octants	2	-	-
Maximum Composite per Octant	5	-	-
Maximum Composite per Borehole	3	-	-

Interpolation Parameters	1st Pass	2nd Pass	3rd Pass
Minimum Number of Composites	3	2	-
Maximum Number of Composites	8	10	-
Search Distance	1 x variogram	1 x variogram	-

14.8 Resource Model Validation

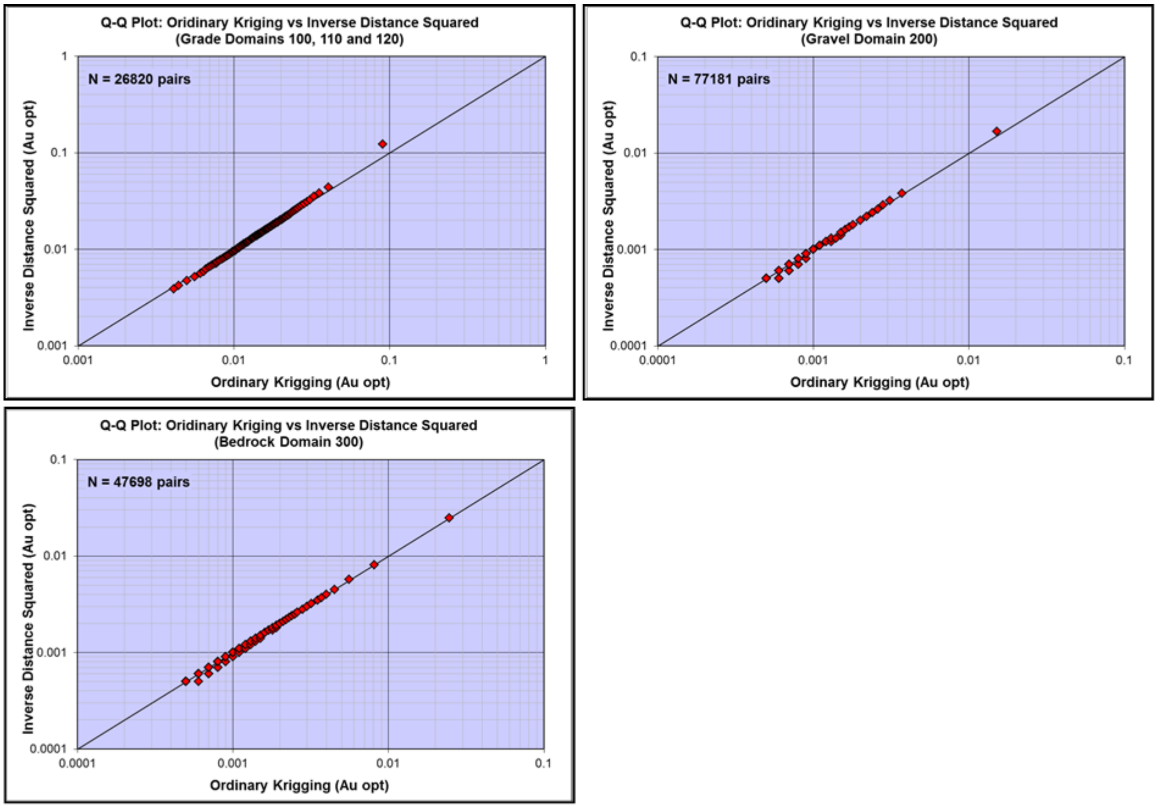
The mineral resource model prepared by SRK was validated by visually comparing block estimates with informing borehole data on section by section and elevation by elevation basis. Two representative cross sections showing block model gold grades in relation to geology zones and composited drilling data are presented in Figure 14-6.

Figure 14-6: Vertical Cross-sections Comparing Estimated Blocks with Informing Composited Drilling Data



Quantile-quantile plots comparing block model grades interpolated by ordinary kriging and an inverse distance algorithm (power of two) data were constructed for the blocks within the gold mineralization wireframe (domains 100, 110, and 120 combined) and outside the wireframe (domains 200 and 300). These plots confirm that block estimates using different interpolation methods with the same estimation parameters do not create an important bias at low grades. At gold grades above 0.03 oz/t within the gold mineralization wireframe; a slight bias towards higher grades occurs with inverse distance squared data (Figure 14-7).

Figure 14-7: Quantile-Quantile Plots Comparing Block Model Grades Interpolated from Ordinary Kriging Compared to an Inverse Distance Algorithm (Power of Two)



Source: SRK 2012

14.9 Mineral Resource Classification

Mineral resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider both the confidence in the geological interpretation and geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the quality of the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas of similar resource classification.

Mineral resources for the Imperial gold project was classified according to CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by SRK under the supervision of Glen Cole, PGeo. (APGO#1416), an independent QP for the purpose of a NI 43-101.

The SRK QP is satisfied that the geological modelling honors the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The mineral resource model is largely based on geological knowledge derived from boreholes drilled sections spaced at approximately 150 ft apart in the east and west portions of the deposit and over 250 ft in the rest of the deposit. The geological information gathered from the RC drilling is sufficiently dense to allow modelling with reasonable confidence of the gold mineralization boundaries (domains 100, 110, and 120), as well as the base of gravel contact, which delimited the unconstrained domains (domains 200 and 300). However, uncertainty remains in the structural framework of the deposit. Normal faults are believed to displace the lithological units including gold mineralization but have not been modelled. The south dipping domain 110 is potentially the result of faulting. The geological continuity can only be inferred at the current drill spacing within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

The mineral resources classification was also reviewed using a combination of tools including: confidence in the geological interpretation, variography results, search ellipse volume, and kriging variance.

Generally, for mineralization exhibiting good geological continuity investigated at an adequate spacing and displaying low structural complexity, the SRK QP considers that blocks estimated according to parameters in Table 14-8 could be classified in the Indicated category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (Figure 14-8). For those blocks, the QP considers that the level of confidence is sufficient to allow appropriate application of technical and economic parameters to support mine planning and to allow evaluation of the economic viability of the deposit. The majority of these blocks are found within the flat lying domain 100 showing little structural complexity.

Table 14-8: Search Parameters Used to Code the Indicated Blocks

Interpolation Parameters	Indicated
Domains 100 and 120	
Interpolation Method	Ordinary Kriging
Search Type	Octant
Estimation Run	1st Pass
Minimum number of Boreholes	2
Kriging Efficiency	Greater than 10%
Maximum anisotropic search distance	150 ft. (90% of Variogram Sill)

The SRK QP considers that with the current confidence in historical data and geological interpretation, all other blocks estimated during the three estimation runs allowing for full and double variogram ranges can be classified in the Inferred category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

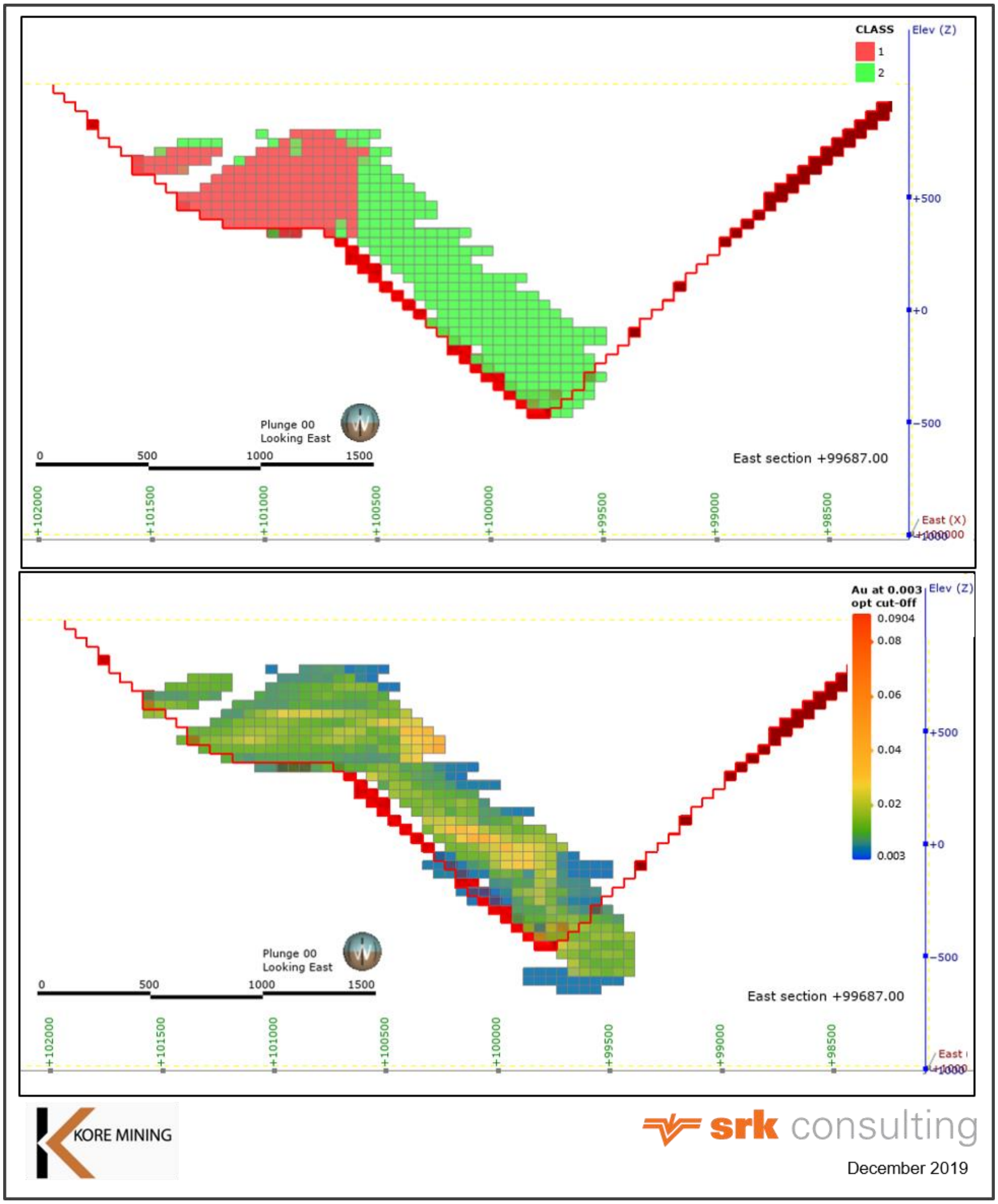
14.10 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

"a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate COG taking into account extraction scenarios and processing recoveries. the QPs consider that the gold mineralization of the Imperial Gold Project is amenable for open pit extraction.

Figure 14-8: Cross Section Through the Mineral Resource Model Showing: Classification (Top) and Grade Distribution (Below) in Relation to the Resource Pit Shell Outline



Note: Class 1 = Indicated and Class 2 = Inferred

To determine the quantities of material offering "reasonable prospects for economic extraction" by an Open Pit, the Lerchs-Grossman optimizing algorithm was applied by Anoush Ebrahimi, PEng, a Principal Consultant (Mining) with SRK to evaluate the profitability of each resource block based on its value. Optimization parameters summarized in Table 14-9 were selected in discussions between KORE Mining and SRK staff. The input parameters for the project have been set up using the recent experience for similar projects and consensus market forecast reports available to SRK. To recover gold from mineral resources a heap leach processing method is expected to be used. Mineralized rocks are mined, crushed and sent to the pad for leaching.

Model blocks located within a conceptual shell are considered to have reasonable prospects for economic extraction by the Open Pit and therefore can be reported as a mineral resource (Figure 14-8). The reader is cautioned that the pit optimization results are used solely for the purpose of testing the "reasonable prospects" for economic extraction and do not represent an attempt to estimate mineral reserves. Mineral reserves can only be estimated with an economic study. There are no mineral reserves for the Imperial Gold Project. The results are used to assist with the preparation of a Mineral Resource Statement for the Imperial Gold Project.

Table 14-9: Assumptions for the Mineral Resource Constraining Shell Optimization

Input for Pit Optimization	Au	Units
Mining cost (ore and waste)	$1.40	US$/t
General and administration costs	$0.50	US$/t milled
Off-site costs	$5.00	US$/oz
Processing operating costs	$1.80	US$/t milled
Sustaining capital cost	$0.50	US$/t milled
Assumed Mill Throughput	25,000	tpd
Gold Price	$1,500	US$/oz
Gold processing recovery	80%	%
Specific Gravity - Ore	0.0680	ton/ft^3
Specific Gravity - Waste	0.0708	ton/ft^3
Specific Gravity - dumps	0.0453	ton/ft^3
Dilution	2%	%
Mining recovery	98%	%
Overall pit slope angles	45	Degrees

The SRK QP considers that the blocks located within the conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a mineral resource. Mineral resources are reported at a COG of 0.003 oz/t Au and include all resource blocks above resource cut-off inside the conceptual pit shell.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The mineral resource statement for the Imperial Gold Project is presented in Table 14-10 (stated in imperial units) and
Table 14-11 (stated in metric units).

Table 14-10: Mineral Resource Statement, SRK Consulting (Canada) Inc., December 13, 2019 Imperial Gold Project (Imperial Units)

Classification	Quantity ('000 tons)	Grade Gold (oz/t)	Contained Gold ('000 ounces)
Indicated			
Grade Zone (Domains 100, 120)	50,379	0.0174	877
Total Indicated	**50,379**	**0.0174**	**877**
Inferred			
Grade Zone (Domains 100, 110, 120)	79,869	0.0156	1,245
Gravel with grade (Domain 200)	10,557	0.0041	43
Bedrock with grade (Domain 300)	9,748	0.0050	48
Total Inferred	**100,174**	**0.0133**	**1,336**

Reported at a cut-off grade of 0.003 oz/t Au using a price of $1,500 /oz Au inside a conceptual pit shell optimized using mining operating costs of $1.40 per ton, metallurgical and process recovery of 80%, combined processing and G&A costs of $2.30 per ton, $0.50 per ton of sustaining capital and overall pit slope of 45 degrees.
All figures rounded to reflect the relative accuracy of the estimates.
Mineral resources are not mineral reserves and do not have demonstrated economic viability

Table 14-11: Mineral Resource Statement, SRK Consulting (Canada) Inc., December 13, 2019 Imperial Gold Project (Metric Units)

Classification	Quantity ('000 tonnes)	Grade Gold (g/t)	Contained Gold ('000 ounces)
Indicated			
Grade Zone (Domains 100, 120)	45,703	0.59	877
Total Indicated	**45,703**	**0.59**	**877**
Inferred			
Grade Zone (Domains 100, 110, 120)	72,456	0.54	1,245
Gravel with grade (Domain 200)	9,577	0.14	43
Bedrock with grade (Domain 300)	8,843	0.17	48
Total Inferred	**90,876**	**0.46**	**1,336**

Reported at a cut-off grade of 0.1g/ton Au using a price of US$1,500 /oz Au inside a conceptual pit shell optimized using mining operating costs of US$1.54 per tonne, metallurgical and process recovery of 80%, combined processing and G&A of US$2.53 per tonne, $0.55 per tonne of sustaining capital and overall pit slope of 45 degrees.
All figures rounded to reflect the relative accuracy of the estimates.
Mineral resources are not mineral reserves and do not have demonstrated economic viability

Reported at a cut-off grade of 0.003 oz/t Au using a price of $1,500 /oz Au inside a conceptual pit shell optimized using metallurgical and process recovery of 80%, overall mining costs of $1.40 per ton and processing costs and general and administration costs of $2.30 per ton and overall pit slope of 45 degrees.

The qualified person is not aware of any known legal, political, environmental, or other risks that could materially affect the potential development of the mineral resources. All figures rounded to reflect the relative accuracy of the estimates.

14.11 Grade Sensitivity Analysis

The mineral resources of the Imperial Gold Project are sensitive to the selection of reporting cut-off grade. To illustrate this sensitivity, within 2019 resource pit tonnage and grade estimates for Indicated and Inferred material are tabulated in Table 14-12 and Table 14-13 at various cut-off grades. The corresponding grade tonnage curves for within pit Indicated and Inferred material are presented in Figure 14-9 and Figure 14-10, respectively.

The reader is cautioned that these figures should not be misconstrued as a Mineral Resource Statement. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade.

Table 14-12: Grade Tonnage Sensitivity Chart for Within Pit Indicated Material at Various COGs

Cut-off Grade (oz/t)	Quantity ('000 tons)	Grade Gold (oz/t)	Contained Gold ('000 ounces)
0.0001	50,426	0.0171	878
0.001	50,426	0.0171	878
0.002	50,426	0.0171	878
0.0025	50,426	0.0171	878
0.003	**50,379**	**0.0174**	**877**
0.004	50,356	0.0148	877
0.0045	50,319	0.0153	877
0.005	50,208	0.0156	876
0.006	49,611	0.0158	873
0.007	48,117	0.0163	866
0.008	45,820	0.0168	846
0.009	43,465	0.0173	826
0.01	41,000	0.0180	803
0.015	28,229	0.0221	643
0.02	16,659	0.0265	444
0.025	7,927	0.0318	249
0.03	3,688	0.0381	134
0.04	791	0.0498	37
0.05	222	0.0615	12
0.06	23	0.0693	1
0.07	-	-	-
0.08	-	-	-

The reader is cautioned that the figures presented in this table should not be misconstrued as a mineral resource statement. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of COG.

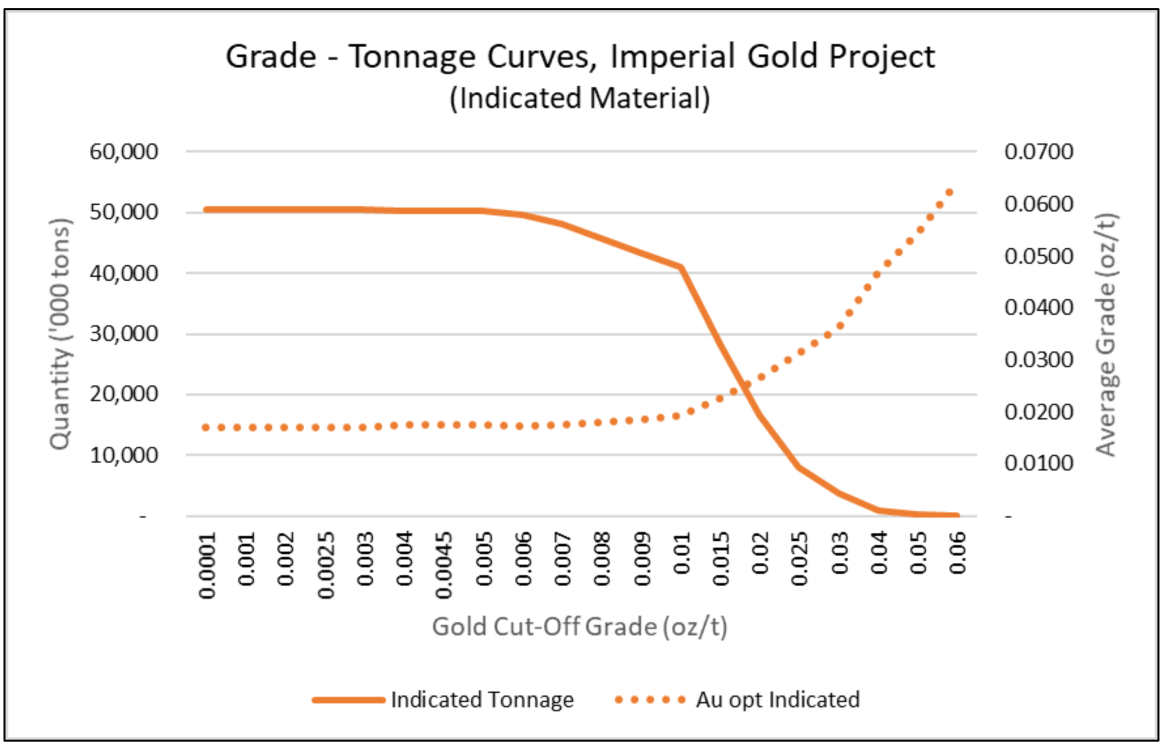
Table 14-13: Grade Tonnage Sensitivity Chart for Within Pit Inferred Material at Various COGs

Cut-off Grade (oz/t)	Quantity ('000 tons)	Grade Gold (oz/t)	Contained Gold ('000 ounces)
0.0001	517,887	0.0037	1,724
0.001	251,356	0.0068	1,566
0.002	131,022	0.0112	1,410
0.0025	112,577	0.0125	1,370
0.003	**100,174**	**0.0133**	**1,336**
0.004	87,000	0.0148	1,290
0.0045	83,593	0.0153	1,276

Cut-off Grade (oz/t)	Quantity ('000 tons)	Grade Gold (oz/t)	Contained Gold ('000 ounces)
0.005	81,261	0.0156	1,265
0.006	78,059	0.0158	1,248
0.007	74,269,	0.0163	1,224
0.008	70,340	0.0168	1,194
0.009	66,326	0.0173	1,160
0.01	61,498	0.0180	1,115
0.015	36,269	0.0221	802
0.02	19,568	0.0265	516
0.025	8,899	0.0318	280
0.03	3,721	0.0381	140
0.04	923	0.0498	45
0.05	303	0.0615	19
0.06	149	0.0693	10
0.07	49	0.0786	4
0.08	28	0.0858	2

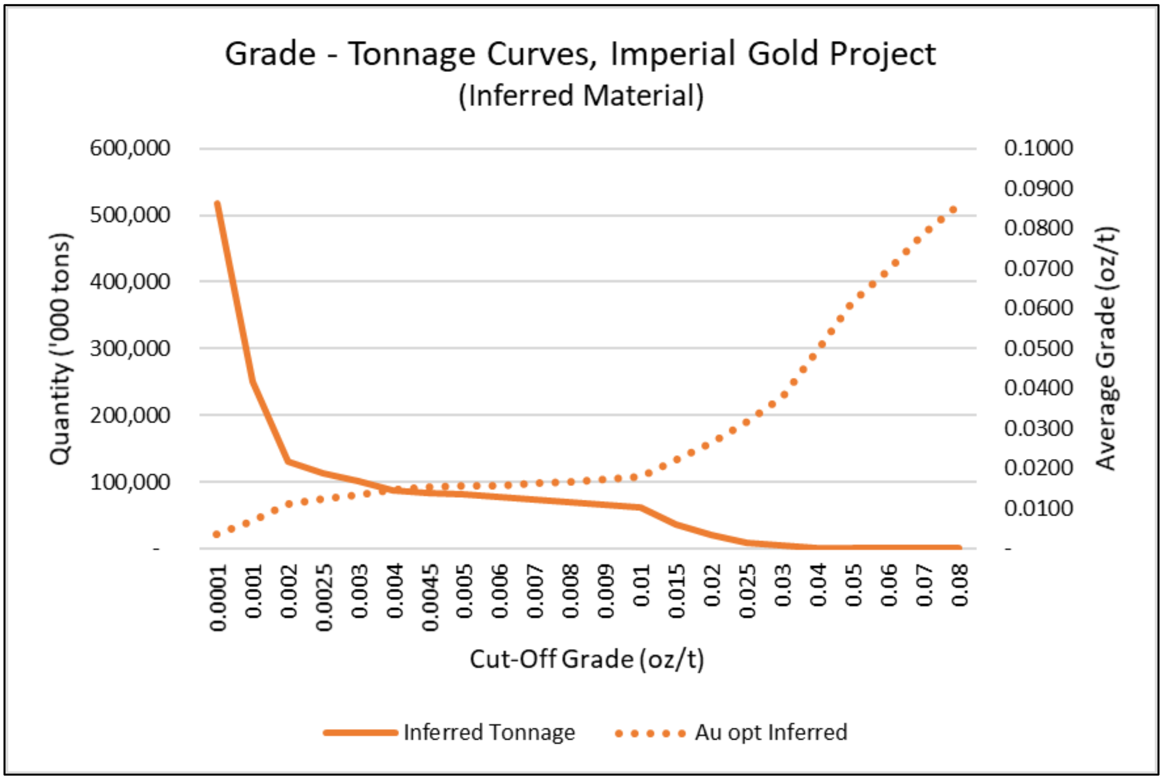
The reader is cautioned that the figures presented in this table should not be misconstrued as a mineral resource statement. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of COG

Figure 14-9: Grade Tonnage Curves for Within Pit Indicated Material for the Imperial Gold Project



Source: SRK 2012

Figure 14-10: Grade Tonnage Curves for Within Pit Inferred Material for the Imperial Gold Project



Source: SRK 2012

15.0 MINERAL RESERVE ESTIMATES

There are no Mineral Reserve Estimates in this Technical Report.

16.0 MINING METHODS

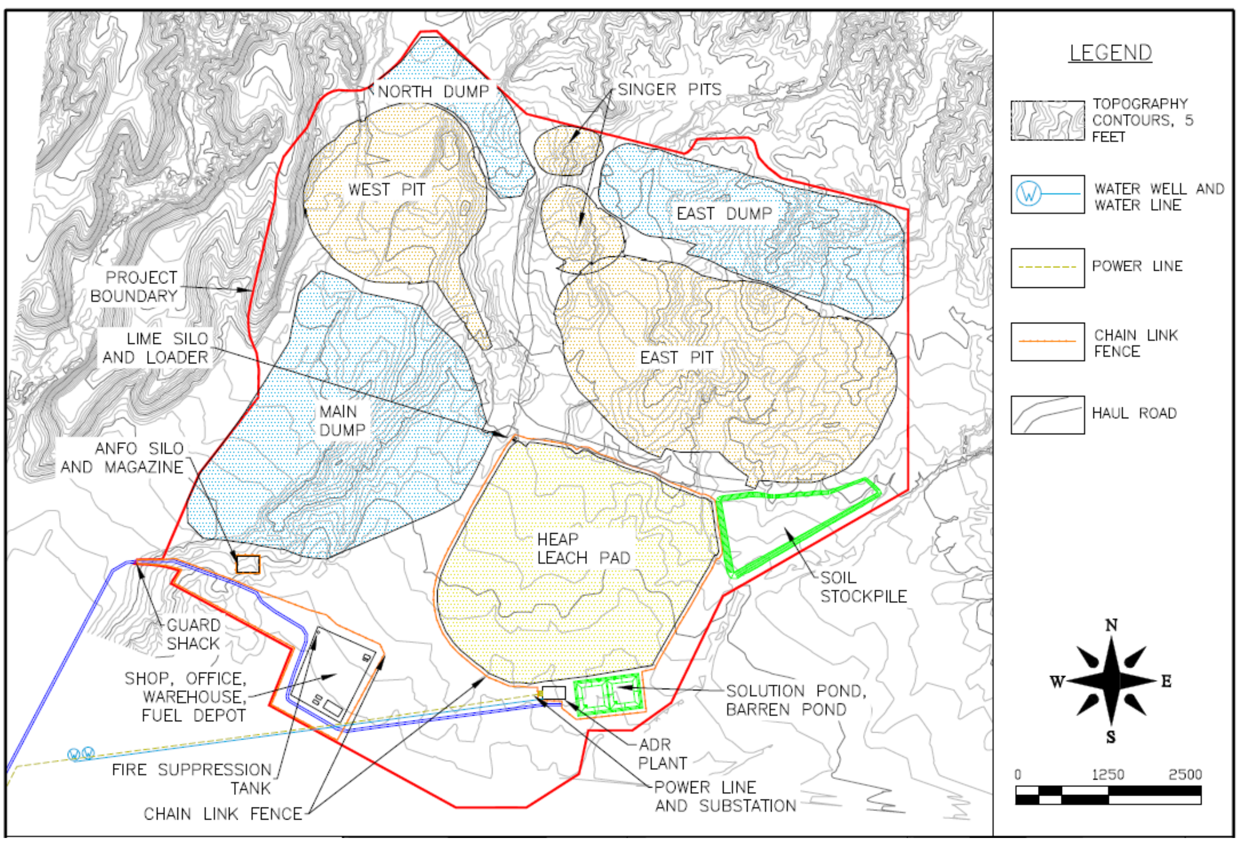
The Imperial Mine deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the estimated grade of the resource model and Whittle pit shell analysis.

16.1 Summary

The mine plan calls for the extraction of run of mine (ROM) potentially economic material from the pits to the heap leach pad at a rate of 12 million short tons per year. Normally, the word “ore” is used to describe proven and probable mineral reserves. In this report, ore is occasionally used as a mining industry term for potentially-economic mineralized material to differentiate it from unmineralized material. The reader is cautioned that no mineral reserves have been presented in this report and the use of “ore” is simply used to identify mineralized material and no future economic viability should be assumed for this material.

The mine plan includes ultimate pit design including ramps and benches, internal phases, production schedule, waste storage, yearly drawings, and capital and operating costs. Figure 16-1 shows the General Facilities Arrangement.

Figure 16-1: General Facilities Arrangement



16.2 Pit Parameters

16.2.1 Bench size

Benches are 40 feet in height, matching the block model level sizes. The catch berm on each bench is 23 feet.

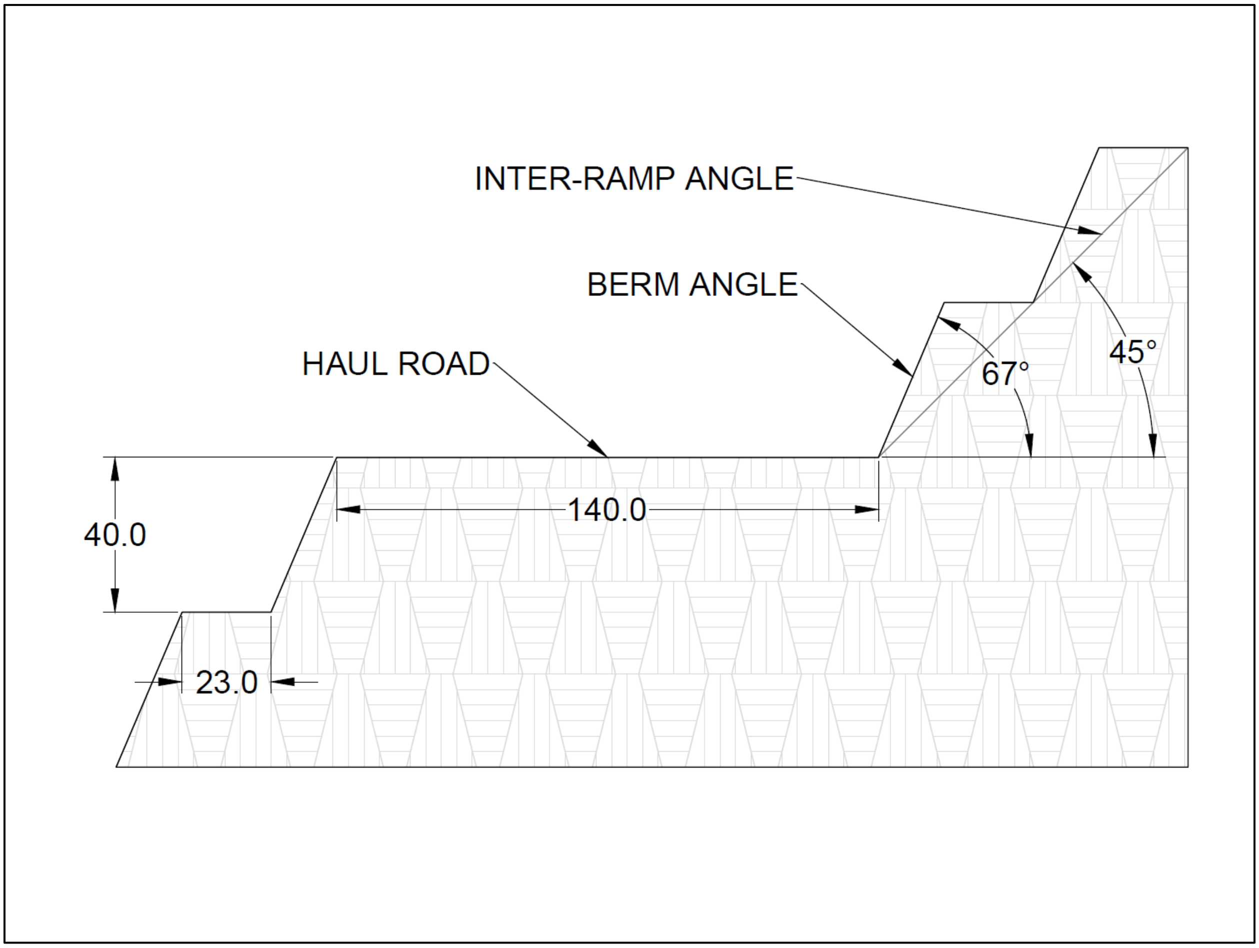
16.2.2 Ramp Size

The ramp was designed to a 140 feet width. This includes a required 100 feet width based on haul truck operating width multiplied by 4 and an additional 40 feet for wall crumbling in the alluvium. The target grade for ramps is 10%.

16.2.3 Pit Slope

WESTEC Inc. (Westec) analyzed the geotechnical properties of the project in a 1997 report. Recommendations from the report assert that the overall pit slopes range from 40° to 50°. When applied to the project, the east pit has an overall slope of 45° in the azimuth range of 115° to 285°, and the west pit has an overall slope of 50° in the azimuth range of 115° to 285°. All other slopes are 40°. Figure 16-2 shows the pit slope profile.

Figure 16-2: Pit Slope Profile



16.3 Ultimate Pit Design

For the ultimate pit size, GRE used Whittle ™ software to generate a series of pit shells of incrementally increasing total value. The shells differed from each other by a revenue factor applied to gold price. Compliance with California Code of Regulations (CCR) §3704.1 Metallic Mine Backfill Regulations requires that the original topo must be reclaimed to ±25 feet. Therefore, the largest possible pit shell with waste rock and heap leach tailings generated during mining plus the additional volume of swell does not exceed the total excavation volume plus the volume between topo and a +25 feet offset above topo.

Within Whittle, the Lerchs-Grossman algorithm was used to create a set of pit shells. Each pit shell was based on a revenue factor applied to the base gold price of 1400 $/troy oz. All parameters controlling the evaluation are shown in Table 16-1.

Table 16-1: LG Input Parameters

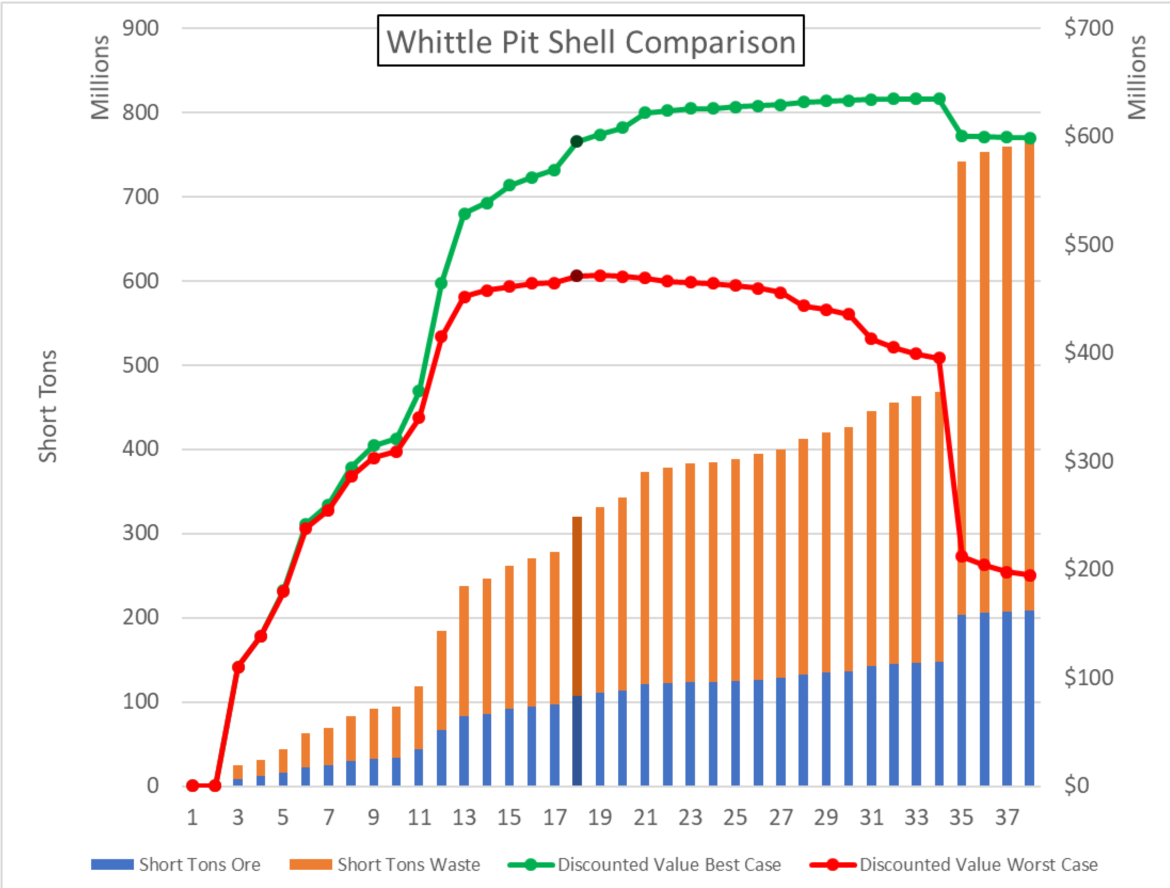
Item	Unit	Value
Gold Price	$/oz	1400
Gold Price of Selected Shell	$/oz	1400
Selling Cost	$/oz	7
Mining Cost	$/ton	2.10
Processing Cost	$/ton	2.36
Process Recovery	%	73
Pit Slope	**degrees**	
West Pit	115 to 285 Azi	50
West Pit	285 to 115	40
East Pit	115 to 285 Azi	45
East Pit	285 to 115	40

Figure 16-3 shows mineralized tons, waste tons, discounted (5%) best case value, and discounted (5%) worst case value. The best case is defined as a schedule in which each pushback is fully mined before the next pushback in a sequence. The worst case is a schedule in which no pushbacks are considered, and the entire pit is scheduled on a top to bottom basis. For this analysis, a processing rate of 9,125,000 tons/year (25,000 tons/day) was used as a basis for the discounted value of the pit shells.

The pit shell selected as the basis for a designed pit needed to fulfill the backfill volume requirements and offer a better economic outlook than the pit shells with a higher revenue factor. For backfill volume requirements, a maximum volume of the pits was determined by estimating the total permissible cover of the project extents. This represents the maximum additional volume created by the swell factor of the mined rock, 30% of the total volume. Approximately 5.1 x 10^9 ft^3 or 191.7 x 10^6 yd^3 of rock can be mined with room in the project for reclamation.

In the Whittle LG analysis, the pit shell best suited for the basis for a designed pit is number 18, shaded dark in Figure 16-3. Pit 18 fulfills the volume requirements of the project, and a significant jump in discounted value over pit 17.

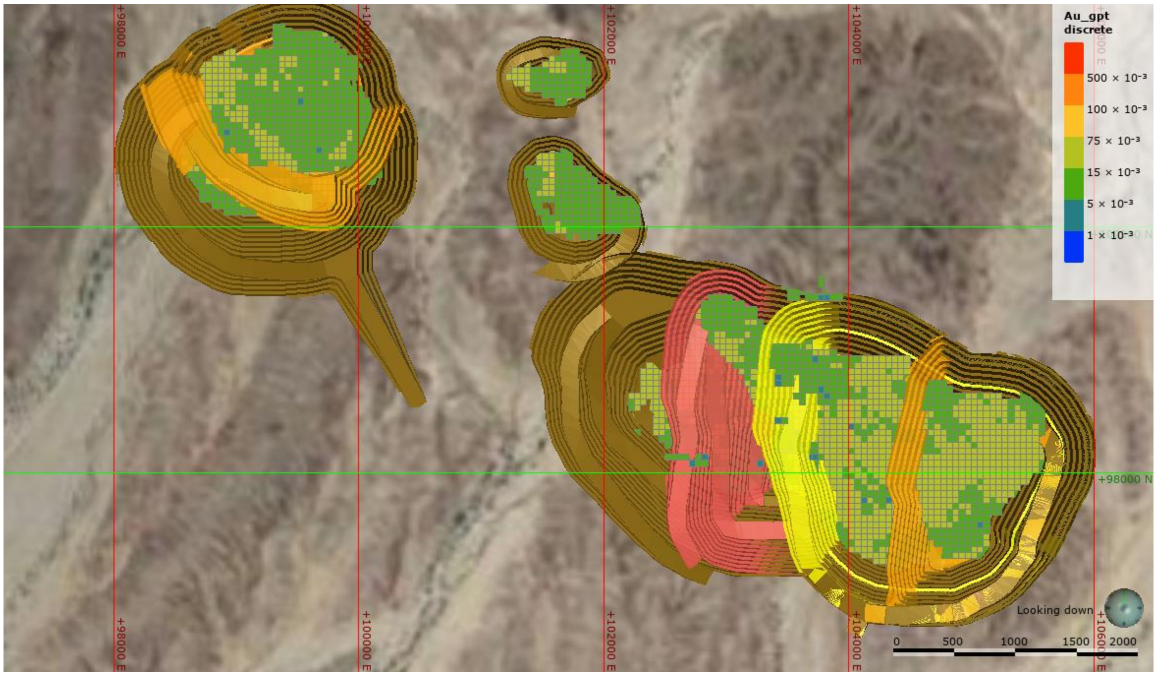
Figure 16-3: Whittle Pit Shell Comparison Graph



16.4 Pit Phases

Initial Whittle analysis showed that high grades could be achieved early on by taking the west pit as one phase, and the east pit in 3 phases (see Figure 16-4). Revised schedule analysis revealed that splitting the west pit and first east pit phase into smaller initial phases would decrease the pre-stripping requirements substantially.

Figure 16-4: Pit Phases



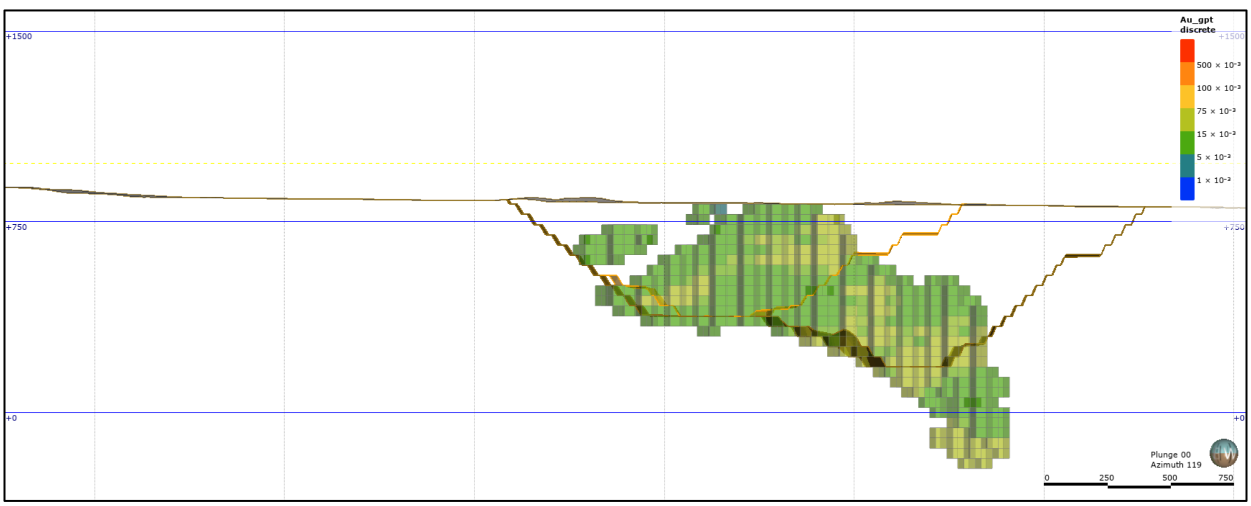
16.4.1 West Pit 2 Phases

Mineralized tonnage, mineralized gold, and waste quantities are summarized in Table 16-2.

The first phase in the west pit is designed to keep waste stripping low and produce gold from the top benches. Ramps are placed on the southwest wall, where they will not cut off access for the second phase. West phase 1 extends down 12 benches to 380 feet (see Figure 16-5).

The second phase in the west pit mines to the ultimate limits of the west pit. It mines 17 benches down to 180 feet in elevation.

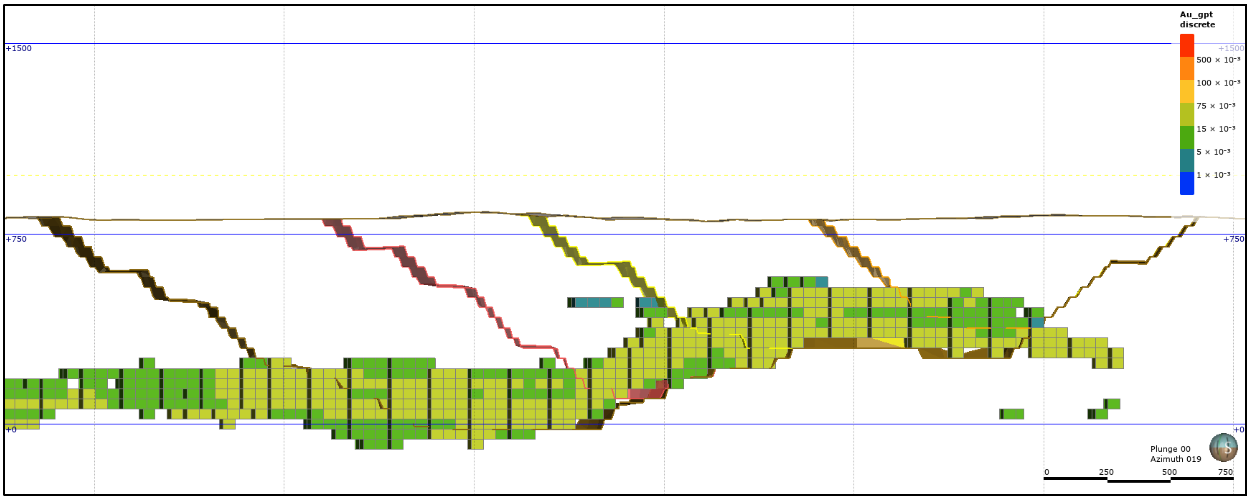
Figure 16-5: West Pit Cross Section, Looking Northeast



16.4.2 East Pit 4 Phases

Phases in the east pit start in the east end and proceed to the west. This is due in part to the ultimate pit shape. The east pit bottom slopes down from east to west (see Figure 16-6). Pit phases pushing back in one direction are an excellent fit for mineral deposits in this geometry.

Figure 16-6: East Pit Phases Cross Section, Looking North



The first phase of the east pit uses one ramp descending counterclockwise into the pit. It will continue to be used to access the bottom of each phase throughout all the phases of the east pit. This phase is 11 benches deep down to an elevation of 380 feet. Each subsequent phase has a ramp in the west wall to access the upper benches before the continuous east ramp can be used.

The East Pit's second phase goes from 820 to 260 elevation in 15 benches. The west ramp allows access to benches down to level 500 where the previous phase's ramp will be used.

East Pit third phase from 820 to 100 in 19 benches. 420 connects to ramp from previous phase.

East Pit fourth phase from 820 to -20 in 22 benches. Level 380 connects to ramp from previous phases and continues to the bottom of the pit.

Table 16-2: Phase Quantities

Pit	Indicated Material			Inferred Material			Waste Tons	Stripping Ratio
	Tons	Au (opt)	Au (tr oz)	Tons	Au (opt)	Au (tr oz)		
West P1	13,930,919	0.013	183,460	2,563,509	0.015	37,555	22,194,139	1.3
West P2	4,417,325	0.014	62,996	14,002,624	0.016	219,805	40,160,246	2.2
East P1	6,153,719	0.018	111,596	1,781,270	0.016	27,897	39,544,618	5.0
East P2	16,223,124	0.021	348,355	3,837,004	0.017	65,585	40,637,029	2.0
East P3	3,081,872	0.025	75,974	8,120,222	0.018	147,923	43,488,065	3.9
East P4	5,614,028	0.018	101,009	7,657,766	0.020	149,351	62,721,500	4.7
Singer P1	0	-	0	2,741,791	0.015	41,600	5,536,997	2.0
Singer P2	0	-	0	1,361,528	0.016	22,262	1,659,162	1.2
Totals	**49,420,987**		**883,390**	**42,065,714**		**711,978**	**255,941,756**	

16.4.3 Singer Satellite Pits

The Singer pits combined contain approximately 4 million tons of mineralized material. This is about 1/3 of nominal production for a year. They are mined early in the mine plan to keep pre-stripping requirements low.

16.5 Production Rates and Mine Life

The mine plan produces 33,000 tons per day or 12 million tons per year for 7.75 years. Table 16-3 shows the quantities produced in the mine plan by production years. Figure 16-7 through Figure 16-14 show plan maps of through the mine life.

Table 16-3: Mine Production Values per Year

Mine Production	Year -1	Year 1	Year 2	Year 3	Year 4
Mineralized Tons	0	12,013,714	12,242,104	11,925,634	12,085,395
Mineralized Tons per Day	0	32,914	33,540	32,673	33,111
Gold Troy Oz	0	173,140	181,781	179,011	188,860
Gold (oz/ton)	0.000	0.014	0.015	0.015	0.016
Gold (gram/tonne)	0.000	0.494	0.509	0.515	0.536
Waste Tons	297,484	33,770,657	36,351,570	35,831,564	31,557,720
Stripping Ratio	0.0	2.8	3.0	3.0	2.6
Alluvium Waste Tons	297,484	23,308,335	34,289,831	32,037,306	25,436,635
Hard Rock Waste Tons	0	10,462,323	2,061,740	3,794,258	6,121,084
Mine Production	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Total**
Mineralized Tons	12,800,760	11,276,396	12,572,820	6,569,878	**91,486,702**
Mineralized Tons per Day	35,071	30,894	34,446	30,990	**33,063**
Gold Troy Oz	265,995	210,688	266,319	129,574	**1,595,368**
Gold (oz/ton)	0.021	0.019	0.021	0.020	**0.017**
Gold (gram/tonne)	0.712	0.641	0.726	0.676	**0.598**
Waste Tons	39,003,447	37,429,937	40,541,239	1,158,135	**255,941,755**
Stripping Ratio	3.0	3.3	3.2	0.2	**2.8**
Alluvium Waste Tons	34,779,864	33,323,544	24,978,176	0	**208,451,176**
Hard Rock Waste Tons	4,223,584	4,106,393	15,563,063	1,158,135	**47,490,579**

Figure 16-7: Mine Plan, Year 1

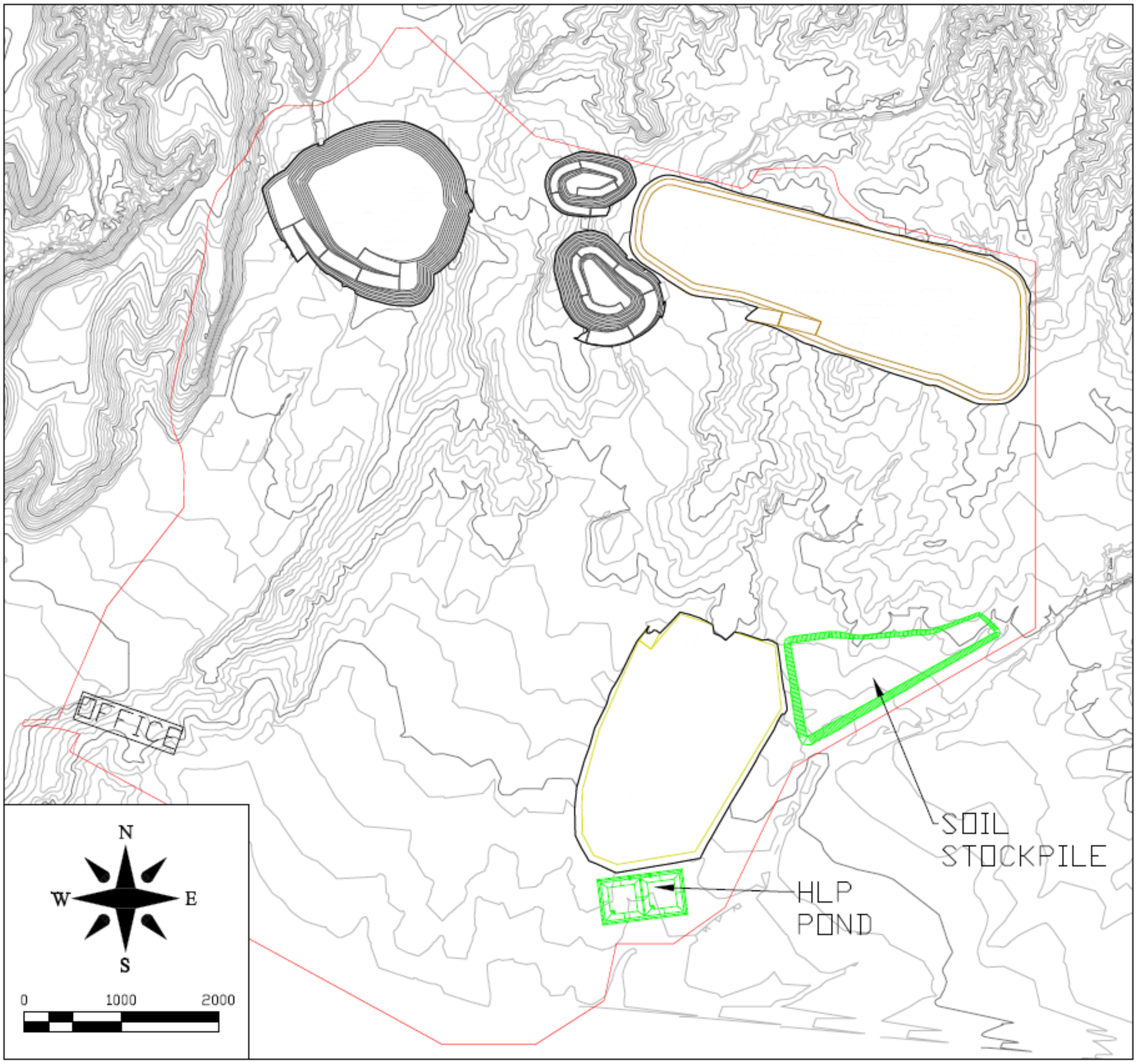


Figure 16-8: Mine Plan, Year 2

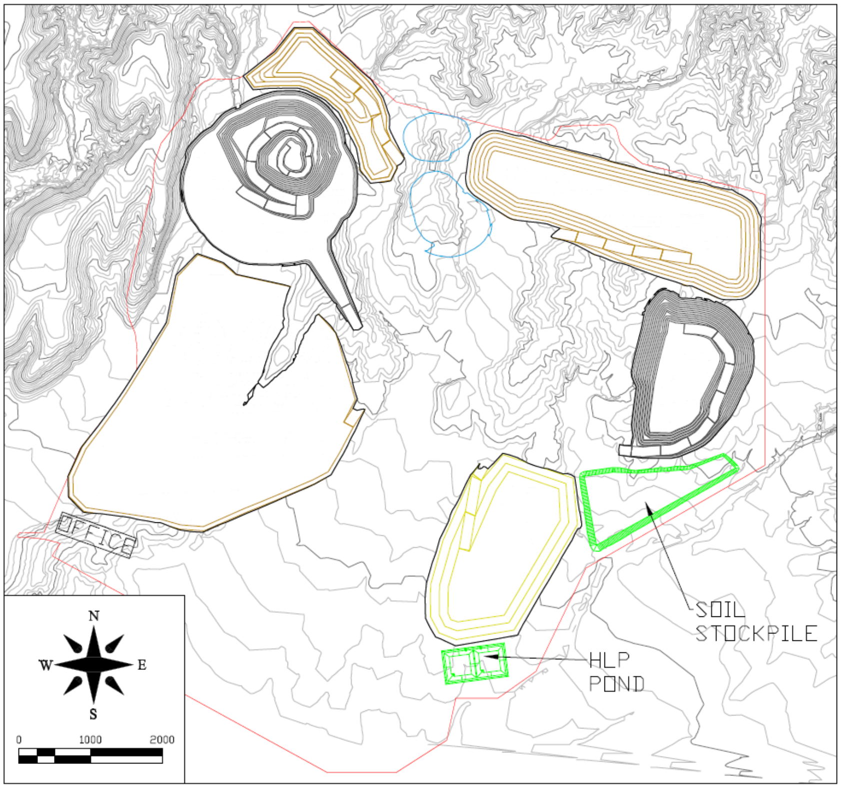


Figure 16-9: Mine Plan, Year 3

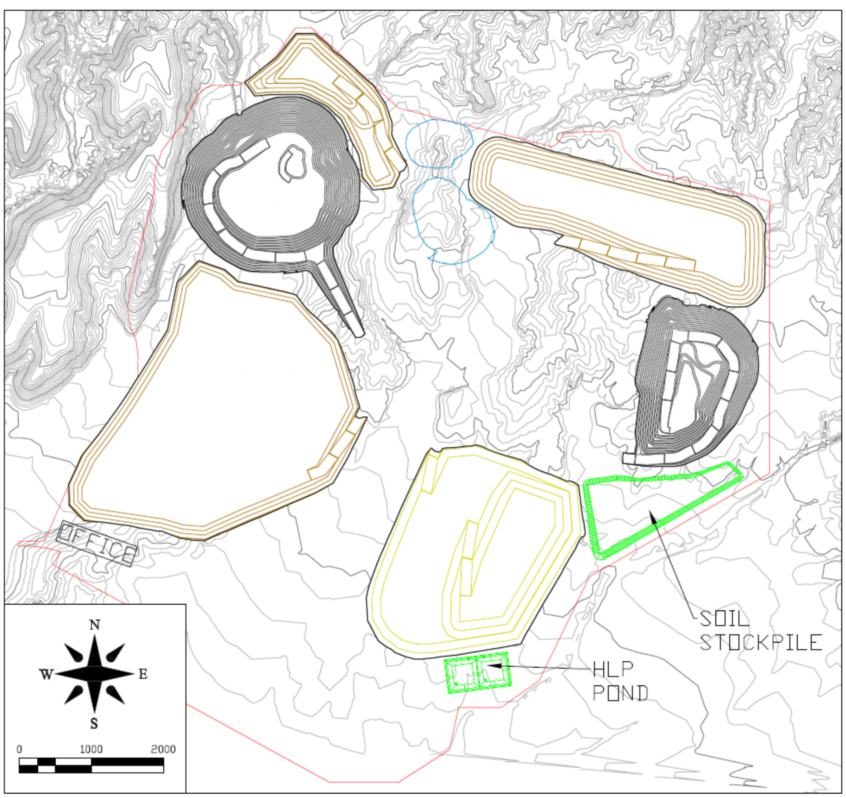


Figure 16-10: Mine Plan, Year 4

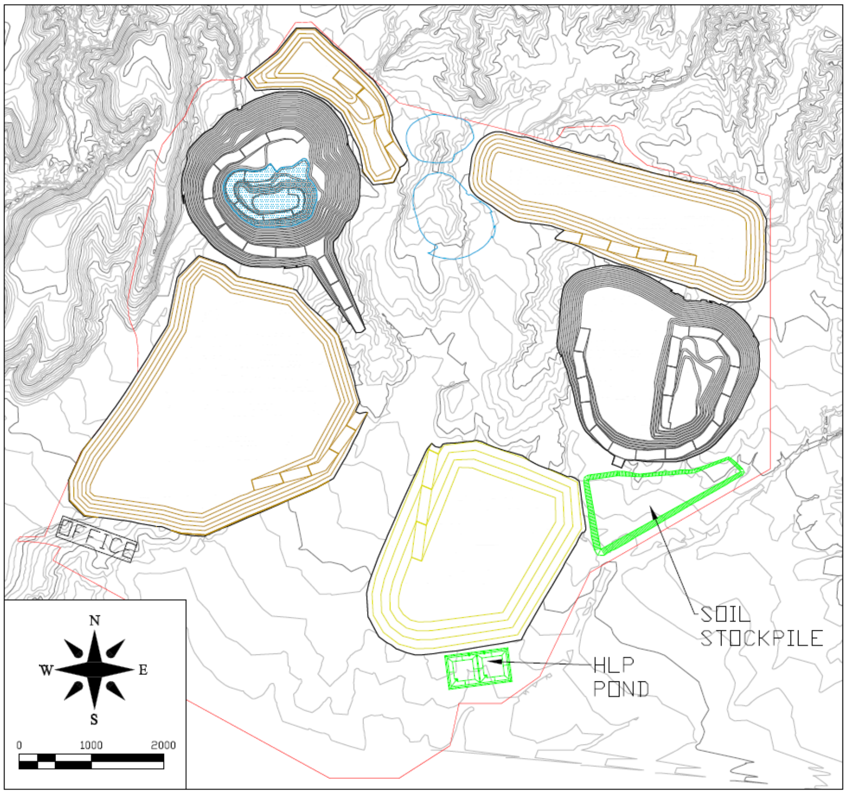


Figure 16-11: Mine Plan, Year 5

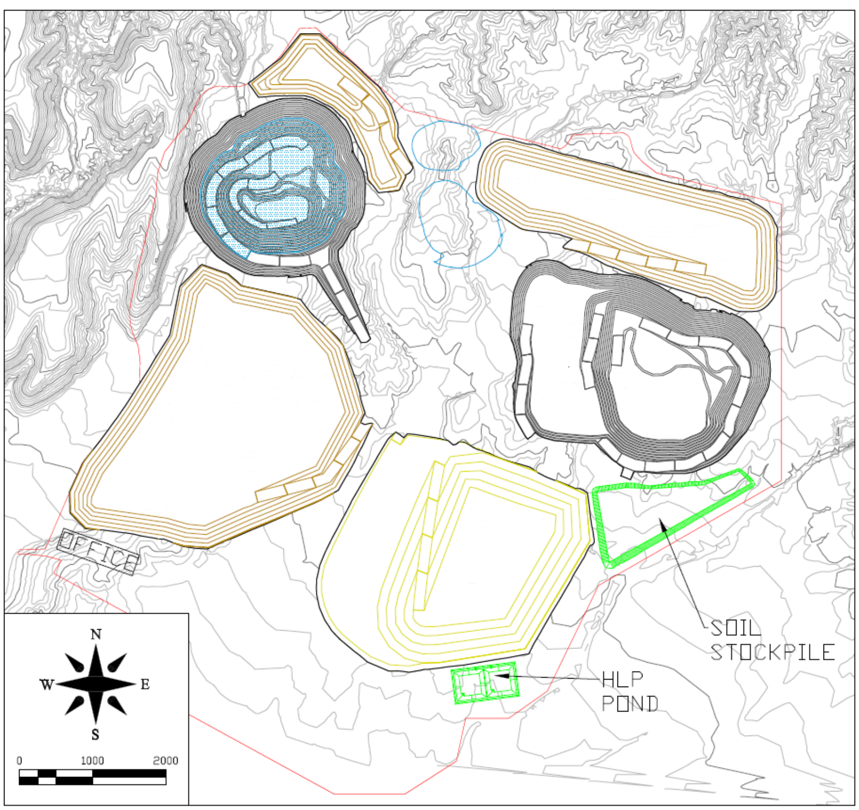


Figure 16-12: Mine Plan, Year 6

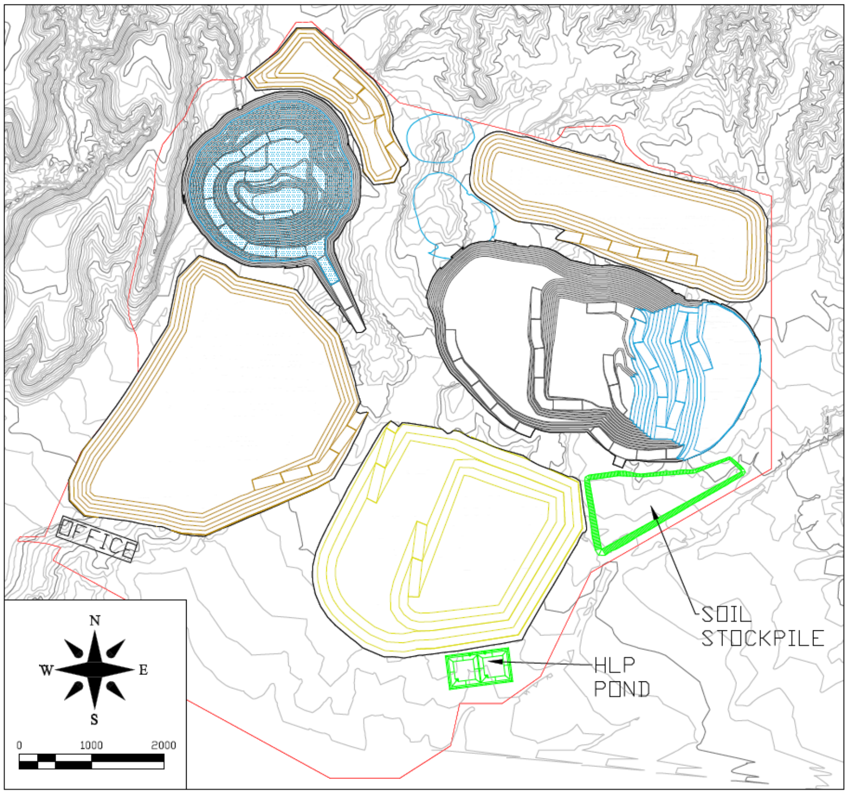


Figure 16-13: Mine Plan, Year 7

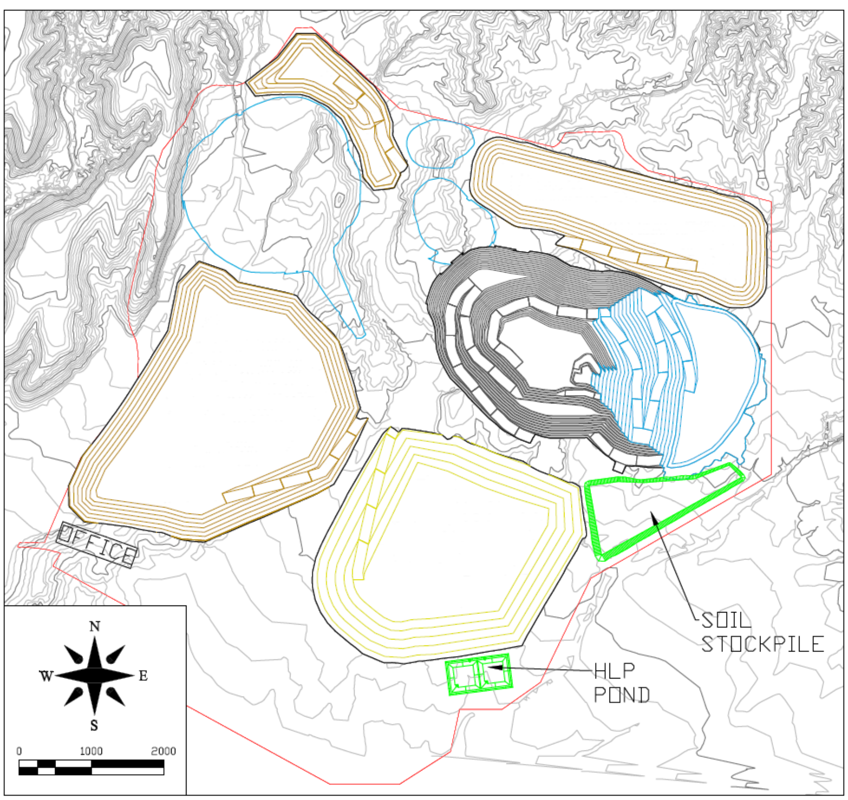
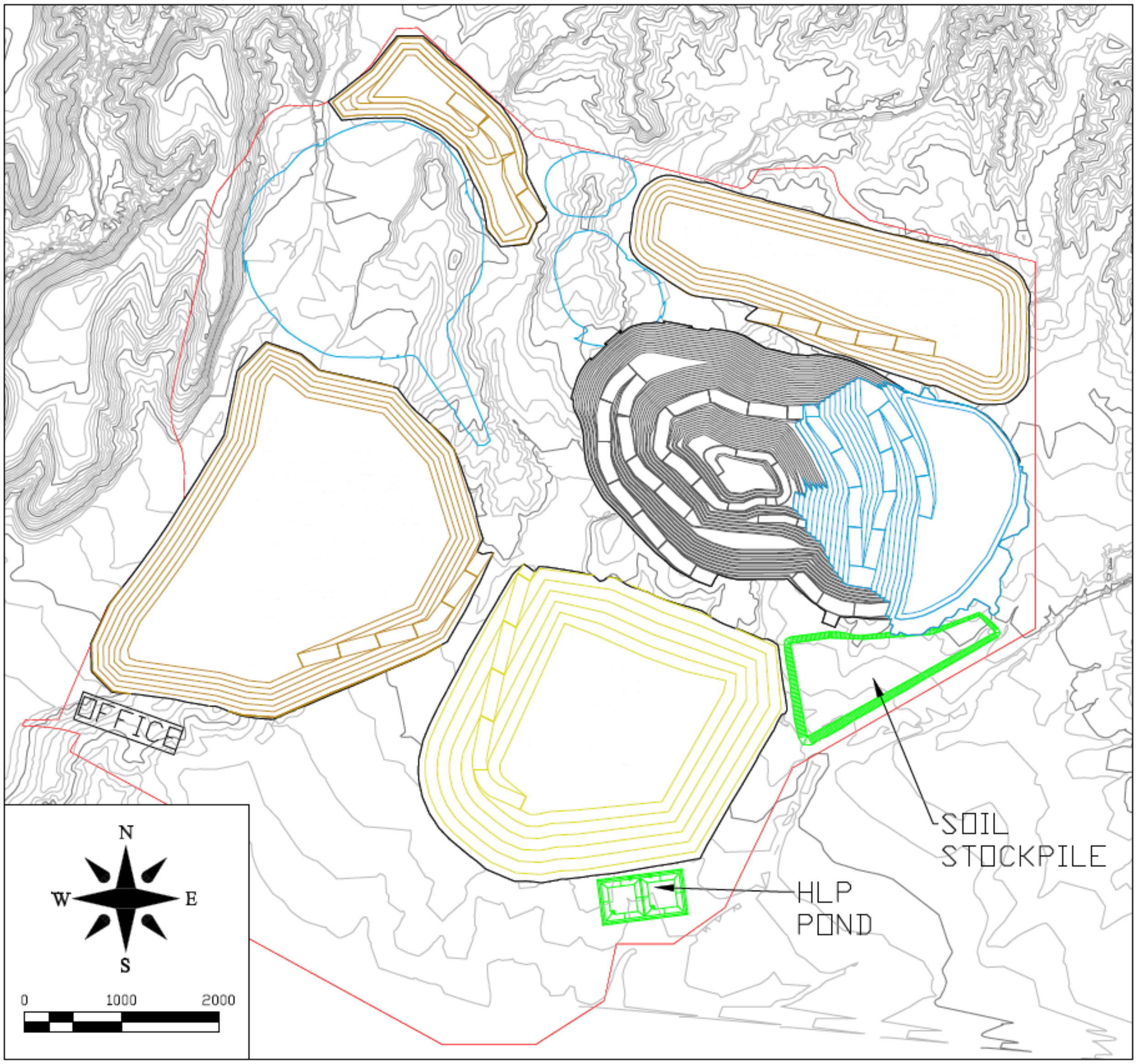


Figure 16-14: Mine Plan, Year 8



16.6 Stripping and Backfilling

The mine plan was designed with an eye on keeping the strip ratio low early on. The desired effect would be a low capital cost of pre-production. The mine plan requires only 297 thousand tons of pre-stripping before production mining starts. This quantity may be mined in as short of a time period as a month.

16.7 Machinery

Kore owned equipment and contract mining scenarios are both considered for the project. Primary mining is done with two CAT 6040 28.7 cubic yard bucket excavators or equivalent and a maximum of 13 CAT 789 trucks or 200 short ton capacity equivalent. Drilling is based on a 9- to 10.66-inch diameter capable drill, of which two are needed. Bulldozer needs are met by three CAT D10 or equivalent.

Table 16-4: Quantities of Major, Support, and Minor Equipment Needed for Life of Mine

Major Equipment	Quantity
Excavator CAT 6040	2
Haul Truck CAT 789D	13
Bulldozer D10	3
Drill	2
Support Equipment	**Quantity**
Wheel Dozer	1
Wheel Loader	1
Water Truck	2
ANFO Truck	1
Lube Truck	2
Mechanics Truck	2
Grader	1
Minor Equipment	**Quantity**
Small Excavator	1
Backhoe	1
Small Crane	1
Light Plant	6
Dewatering Pump	1
4x4 Pickup	10

17.0 RECOVERY METHODS

17.1 Process Description

The Imperial project would employ open pit mining with a conventional heap leach system on a 365 day per year 24 hour per day basis. The heap leach will utilize run-of mine (ROM) material. The ROM is delivered directly from the open pit to the heap via the mine haul trucks. The trucks will pass under a silo that will deposit a measured amount of lime on the load for pH control.

The heap leach would consist of a suitable area lined with a containment system, typically a linear low-density polyethylene (LLDPE) liner with an over liner of sized material to facilitate drainage. Within this over liner would be placed drainage pipes to conduct the leach solution to the centralized collection ponds. The ROM material is stacked in lifts on the lined pad by means of truck dumping. The lifts are targeted at 32 ft (10 meters) in height with a total heap height of 328 ft (100 m). Once a suitable area has been stacked (cell), the cell would be irrigated with dilute cyanide solution. Stacking would continue to advance, and each area irrigated with cyanide solution for a set period of time (primary leach cycle). The solution leaches gold from the heap materials and is transported to the gold recovery circuit as pregnant leach solution (PLS).

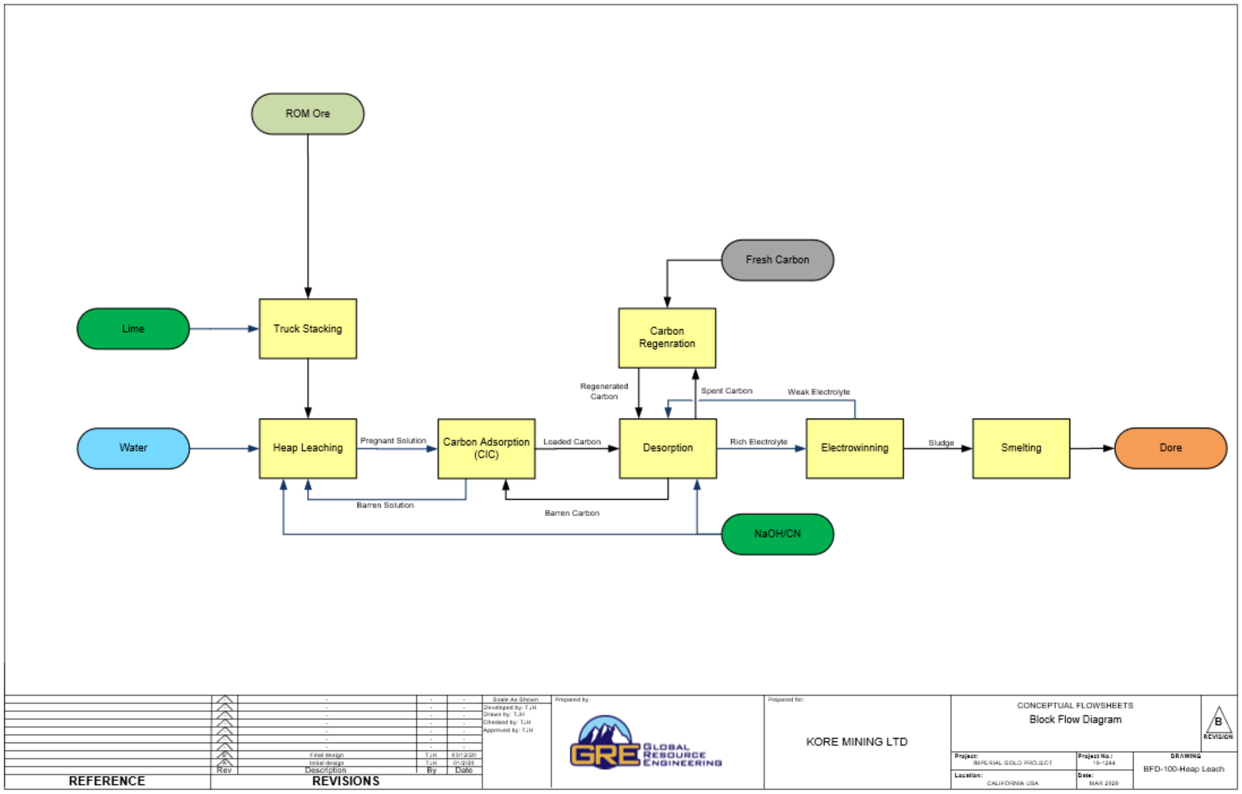
This PLS would be processed in the Adsorption-Desorption-Recovery plant (ADR), diverted to a dedicated pond or recirculated to the heap. The gold in the solution is collected on activated carbon in a series of carbon-in-column (CIC) vessels (from 4 to 8 columns is typical). The depleted “barren” solution would report to the heap leach barren pond/tank and be recirculated back to the heap after having the reagent levels adjusted (pH and cyanide).

Once the gold level on the carbon in the CIC circuit reaches a specific setpoint (3,000 g/mt) in the lead column, the carbon is advanced, and a set amount removed for gold recovery. Gold recovery takes place through stripping the activated carbon using a specifically designed process (ZADRA or Anglo American Research Laboratory [AARL] are typical). The gold is stripped from the carbon into an enriched solution that reports to an electrowinning circuit where the gold is recovered as a sludge that is ultimately smelted into doré bars (gold and silver).

The heap leach is typically designed to have multiple lifts installed. Each new lift goes on top of the last lift until the heap reaches its ultimate height. Heap leaches often utilize 10 or more lifts to reach an ultimate height of 328 ft to 492 ft (100 to 150 meters). The configuration of the heap leach is heavily dependent on the permeability characteristics of the material, the terrain available, and the geotechnical aspects of the site. Figure 17-1 shows the complete conceptual flowsheet.

There is an option to utilize a crushing circuit to treat the higher-grade mineralized material and develop a combination ROM and crushed material heap leach facility. The crushed material showed significantly higher gold extraction during testing and this could improve the overall project economics. However, this study only presents the lowest capital cost option of the represented by the ROM HLF.

Figure 17-1: Conceptual Heap Leach Flowsheet



17.2 Heap Leach Circuit

Ore would be stacked for a sufficient period to allow enough surface area to be created for irrigation, this also allows operations personnel to be a safe distance from active irrigation areas. Irrigation is provided by an emitter-type irrigation system designed to deliver 0.005 gallons per minute per square foot (gpm/ft^2) (12 liters per hour per square meter [lph/m^2]). Emitter layout is designed to provide suitable ore wetting. The heap would be placed under primary irrigation for a period of approximately 90 days. After the primary leach, irrigation would be discontinued and advanced to the next cell. No rinse phase is included because of the multiple lift system employed. The subsequent lift will be placed on top up to a total of 10 lifts. Rinsing will only occur before closure or once the heap reaches its ultimate height.

High concentration gold leach solutions or pregnant leach solutions (PLS) flow from the pad to the PLS sump by gravity. The solution is pumped from the sump to the ADR circuit. Excess solution is diverted to the PLS pond. Solution is collected from each heap cell by a series of drain pipes under the heap that transport the solution to perimeter piping. The solution can be placed in either the PLS or Event Pond piping. Storm water collected from the pad during heavy precipitation events can be diverted to a storm water pond. The storm water can be used as fresh make up water to the circuit.

17.3 Adsorption, Desorption, Recovery (ADR)

During normal operations, PLS solution is pumped to the CIC tanks. The CIC circuit consist of two trains of five CIC vessels, each containing six tons of carbon. Carbon is advanced counter current to the PLS flow as the first tank in the series reaches its loading limit. The target carbon loading is 3,000 g/t of gold. Carbon is advanced by recessed impeller pumps.

The loaded carbon from the first tank is pumped across the loaded carbon screen to the acid wash column. The screen under flow is returned to the PLS flow.

The barren solution exiting the last of the CIC is returned to the heap leach barren solution tank/pond after passing through a carbon safety screen. Fine carbon from the screen underflow is stockpiled and sent for separate off-site recovery. Loaded carbon is acid washed with dilute nitric acid to remove calcium and adsorbed metals. Spent acid is neutralized and disposed. After acid washing, the carbon is passed to an elution column. Elution is conducted by the modified ZADRA system. A solution of caustic and cyanide is passed through the elution column to remove the adsorbed gold. The rich electrolyte is pumped to electrowinning cells, where the gold and silver are recovered on the cathodes. The cathodes are washed, and the recovered sludge is refined in a conventional induction furnace after drying. The circuit is designed to conduct two strip cycles per day. The doré produced is assayed and stored in a vault before being shipped off-site for refining and payment. All thermal devices are to be equipped with mercury abatement systems.

Barren carbon from the elution column is returned to the CIC circuit after passing across a carbon sizing screen. Fine carbon from the screen underflow is stockpiled and sent for separate off-site recovery. Approximately 50% of the barren carbon reports to an indirect fired kiln for thermal regeneration. The regenerated carbon reports to a quench tank before being pumped to the carbon sizing screen. Fresh makeup carbon is first sent to an attrition tank for fines removal before being pumped to the carbon sizing screen. The fine carbon from the screen underflow is captured in a plate and frame filter.

17.4 Conceptual Heap Leach Pad and Pond Design

The HLF consists of the following system components:

- Heap leach pad
- Liner system
- Leachate (solution) collection system
- Storm pond
- Stormwater management system
- Freshwater supply

To minimize capital expenditure, the heap leach pad has been designed in phases, with each phase requiring advanced expansion of the engineered pad. The HLF would be constructed in three phases, with the pad foundation preparation, liner installation, and collection piping advanced as the leach pad expands. The capacity of each stacking stage includes an initial three-year period two additional two-year period.

The initial HLF development (Phase 1) would also include the full development of the solution handling system, storm pond, and perimeter diversion ditches prior to commencing ore stacking and leaching.

Table 17-1 shows the development phases and the lift capacity in ore volume and duration. Design details for each of the HLF components are discussed further in the following sections.

Table 17-1: Heap Capacity

Development Phase	Elevation (abs m)	Lift Capacity (days)	Mine Life (years)	Mineralized Material Volume (m^3)	Mineralized Material Volume (cum m^3)
1	10	257	0.7	4,806,633	4,806,633
	20	475	1.3	4,066,316	8,872,950
	30	656	1.8	3,380,205	12,253,155
	40	804	2.2	2,748,295	15,001,451
	50	920	2.5	2,170,579	17,172,030
	60	1008	2.8	1,647,042	18,819,072
	70	1071	2.9	1,177,652	19,996,724
	80	1112	3.0	762,318	20,759,042
	90	1134	3.1	400,679	21,159,720
	100	1138	3.1	87,083	21,246,803
2	10	1270	3.5	2,467,542	23,714,346
	20	1396	3.8	2,337,405	26,051,750
	30	1514	4.1	2,207,275	28,259,025
	40	1625	4.5	2,077,157	30,336,182
	50	1729	4.7	1,947,058	32,283,239
	60	1827	5.0	1,816,992	34,100,232
	70	1917	5.3	1,686,992	35,787,224
	80	2000	5.5	1,557,146	37,344,370
	90	2077	5.7	1,427,813	38,772,182
	100	2147	5.9	1,304,640	40,076,822
3	10	2279	6.2	2,467,444	42,544,266
	20	2404	6.6	2,337,286	44,881,552
	30	2522	6.9	2,207,129	47,088,680
	40	2634	7.2	2,076,972	49,165,653
	50	2738	7.5	1,946,817	51,112,470
	60	2835	7.8	1,816,664	52,929,134
	70	2926	8.0	1,686,514	54,615,648
	80	3009	8.2	1,556,367	56,172,014
	90	3085	8.5	1,426,226	57,598,241
	100	3155	8.6	1,296,095	58,894,336

17.5 Heap Leach Pad

The heap leach pad consists of a perimeter berm, pad liner system, and leachate collection system to collect and convey the leachate solution to the ADR plant, which should be located adjacent to the heap leach facility. The leach pad has an approximate final footprint area of 10,763,910 square feet (1,000,000 square meters). The heap leach pad is designed to be operated as a fully drained system with no leachate storage within the HLF. Prior to the start of each of the development stages, the pad foundation must be prepared. Foundation preparation involves stripping the topsoil and vegetation and the removal of any rocks. The topsoil would be stockpiled at a convenient location and used for reclamation of the HLF area at closure. The underlying soils would be excavated down to a competent, stable foundation to provide a uniform and graded surface for the pad liner. Grading and backfill would be used to level the surface and to ensure that the pad grading will promote leachate flow towards the collection piping system and sump. A minimum pad grade of 1-2% is required.

17.6 Liner System

A liner system is planned to maximize solution recovery and minimize environmental impacts by minimizing leachate losses through the bottom of the leach heap pad. The liner system consists of both barrier and drainage layers using a combination of synthetic and natural materials to provide leachate solution containment that meets the accepted standards for leach pad design. The pad is designed to operate with minimal solution storage within the pad structure during normal operating conditions. The liner system is designed to meet the required performance standards assuming fully saturated solution storage conditions. A double liner system has been employed with two layers of synthetic material.

17.7 Liner Design

A liner system has been developed for the pad using an engineered composite double liner design. The double liner system is designed to be installed as the primary liner system under the entirety of the HLF. The double liner system consists of the following components:

- 1.6-foot-thick (0.5-meter-thick) over liner (1.5-inch [38-mm] minus with less than 10% fines content) using ore as the material
- 80-mil (2-mm) LLDPE geomembrane
- 1-foot-thick (0.3-meter-thick) compacted low permeability soil liner
- Leak Detection and Recovery System (LDRS)
- 60-mil (1.5-mm) LLDPE geomembrane.
- LLDPE was proposed for the geomembrane liner systems for the heap leach pad because it has the following benefits (Lupo, 2005):
    - Generally higher interface friction values, compared to other geomembrane materials
    - Ease of installation in cold climates due to added flexibility,
    - Good performance under high confining stresses (large heap height)
    - Higher allowable strain for projects where moderate settlement may become an issue.

17.8 Construction

Development of the heap leach liner would be constructed in three phases, with pad expansions proposed after three years of initial production to meet ore stacking requirements. The liner system would be constructed with both the synthetic and natural layers extending to the top of the perimeter berms to provide full containment. The synthetic liners and geotextiles would be anchored and backfilled in a trench along the heap leach pad perimeter and perimeter berms to ensure that ore loading does not compromise the liner coverage of the heap leach pad footprint by pulling the liner into the pad. Along the pad toe, all liners would be tied into their corresponding liner layer along the foundation of the pad to provide a continuous seal and drainage connection.

The perimeter berm would be constructed as part of the liner tie-in around the perimeter of the pad footprint to ensure that heap solution is contained within the pad and to prevent surface runoff entering the pad collection system. A 1-foot-thick (0.3-meter-thick) bedding sand layer would be placed on the face of the confining embankment directly underneath the second (bottom) geomembrane liner to provide additional integrity protection to the liner.

17.9 Over Liner

A protective layer of approximately 1.5 feet (½ meter) of coarse crushed ore/waste would be placed over the entire liner system footprint to protect the liner's integrity from damage during ore placement. The over liner acts as the drainage layer, allowing solution drainage into the pipe collection system. The over liner material must be competent and be free from fines.

17.10 Solution Collection System

Collection and recovery of the leach solution is facilitated by the solution collection system in conjunction with the heap leach liner, over liner, and LDRS. The collection system consists of the following pipe and sump components:

- Lateral collection pipes
- Collection header pipes
- Main header collection pipes
- Leachate collection sumps

The solution collection system would be designed to facilitate quick and efficient solution conveyance off the pad to reduce the potential risk of solution losses through liner system. The entire piping system would be constructed from perforated corrugated plastic tubing (CPT), which is embedded within the over liner layer.

The lateral collection pipes, which would be spaced approximately 16 feet (five meters) apart under the entire pad footprint, feed directly into the collection header pipes, which then flow into the main header. The main header pipes would be positioned along the centerline of each heap leach pad cell and terminate at the upstream toe of the perimeter berm at the leachate collection ditch. Two leachate collection ditches allow solution to flow by gravity to the required storage pond. The collection pipes would be fitted with gate valves to allow solution to be directed to one of the three perimeter collection ditches – PLS, Barren, or Storm.

17.11 Leak Detection and Recovery System

The LDRS would be designed to capture and convey any solution that may leak through the overlying primary geomembrane layer. The LDRS consists of a 1-foot-thick (0.3-meter-thick) sand layer embedded with 4-inch (100-mm) diameter perforated CPT collection pipes. Any leakage recovered by the LDRS would be conveyed into the LDRS sump at the downstream toe of the HLF. A level-switch controlled submersible sump pump would transfer the recovered solution via a pipe installed within the LDRS sand layer and connect into the main solution recovery line for processing. Monitoring of the leakage recovery would be undertaken by recording pump operating hours.

17.12 Leakage Detection Cells

To facilitate more accurate leak identification, the entire pad solution collection system is typically subdivided into multiple independently monitored areas (cells) separated by small berms. Each of these cells has a dedicated leakage detection collection system comprising a drain gravel layer beneath the inner composite liner system which conveys the leakage to a 4-inch (100-mm) diameter perforated collection pipe within the LDRS collection trench. The LDRS ditches flow by gravity at a minimum 0.5 % slope towards the LDRS collection sump, located along the sides of the leach pad. The flow rates from the dedicated collection pipes are continuously monitored and measured prior to discharging into a sump.

17.13 Solution Storage

17.13.1 Storm Pond

The Storm Pond is designed to provide storage for excess leachate and runoff generated as a result of rainfall events. The pond is situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping inside lined ditches. The Storm Pond is designed to meet the following design criteria:

- Storage capacity to contain the excess HLF leachate and surface runoff from the 1 in 100-year 24-hour storm event without discharge
- Overflow designed to discharge the 1 in 200-year 24-hour storm event

The storage requirements for the Storm Pond were established based on containment of the entire estimated surface runoff generated from the HLF (at the Phase 3 footprint) during the 1 in 100-year 24-hour storm event. Based on the surface runoff estimates, the following storage requirements for the events pond were identified:

Total runoff estimates for 1 in 100-year 24-hour storm event 3,032,600 cubic feet (85,885 cubic meters)

- 10% additional factor of safety 303,360 cubic feet (8,588 cubic meters)
- Total pond storage capacity 3,335,860 cubic feet (94,500 cubic meters)

Solution stored in the Storm Pond would be pumped back to the heap leach pad using the Storm Pond pump station. The pump station is designed to be able to drain the storm volume over a period of approximately ten days.

17.13.2 PLS Pond and Barren Tank

The PLS and Barren tank/ponds are designed to provide storage for leachate and CIC return solutions. The ponds are situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The PLS and Barren ponds are designed to meet the following design criteria:

- Storage capacity to contain sufficient solution volumes to maintain irrigation and feed to the CIC circuits
- The PLS Pond is designed to contain up to 24 hours of solution assuming a maximum irrigation rate of 15 lph/m^2

- The PLS Pond is designed with a capacity of approximately 2,349,880 cubic feet (66,550 cubic meters).
- The Barren tank is designed to hold 15 minutes of solution at a capacity of 24,717 cubic feet (700 cubic meters).

Excess solution flows to any of these ponds/tanks would be diverted to the PLS or Storm Pond for recycle back to the heap.

17.13.3 Pond Liner System

The engineered double liner system designed for the ponds uses the same design principles as the HLF pad liner system. The liner design consists of the following layer configuration:

- 60-mil (1.5 mm) high-density polyethylene (HDPE) geomembrane
- 1-foot-thick (0.3-meter-thick) low permeability soil liner
- Geosynthetic “geonet” drainage layer
- 60-mil HDPE geomembrane.

The liner system installed on the upslope of the pond embankment would have an additional 1-foot-thick (0.3-meter-thick) bedding sand layer that would interface with the lower geomembrane layer to protect the integrity of the liner.

Installation of a LDRS is not required for the Storm Pond as the pond is operated as a dry facility and would only receive and store runoff water during significant storm events. In the event that leakage does occur through the double liner system, this water would be conveyed via the geonet layer to a 3-foot-thick (1-meter-thick) drainage blanket that underlies the Storm Pond embankment. This drainage blanket discharges to a sump for solution return to the pond.

It is recommended that HDPE geomembrane be used for the pond liner system rather than LLDPE. Unlike the heap leach pad, the pond liner system would not be subjected to high confining stresses from ore stacking, and HDPE has a higher ultraviolet resistance, which is critical for exposed surfaces like that of the ponds.

17.14 Runoff Collection and Diversion

The surface water management system proposed for the site consists of a series of ditches constructed around the perimeter of the HLF to intercept overland surface runoff around the HLF pad and to convey surface water away from the active site. The ditches are designed to meet the following design criteria:

- Conveys the 1 in 100-year 24-hour duration storm event
- Minimum freeboard = 1-foot (0.3 meters)
- Minimum ditch grade = 0.01 foot/foot (meter/meter)
- Side slopes = 2H:1V
- Channel shape = trapezoidal

Lining and protection of the ditch channels from erosion and scouring may be required for all permanent ditches. Temporary ditches would be constructed between heap phases.

18.0 PROJECT INFRASTRUCTURE

A limited amount of infrastructure is currently available on site. Power, water, and all other systems necessary for a mining and processing operation will be required.

Sufficient water appears to be available on the Imperial property. One ground water well currently exists, and a second well is planned for this project. Groundwater supplies would be developed to meet the project water requirements.

Power is available near the mine site from the grid through a 161kV power line. There are no electrical substations at the site.

Local labor for mining is available.

18.1 Water Supply

Modeling of the heap operation on a monthly basis over the projected mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 1,100 gpm (250 m^3/hr). An additional 150 gpm (34 m^3/h) is required for mine, shop, and office water consumption.

18.2 Electrical Power

Electrical power is proposed to be supplied by the 161 kV power transmission line running parallel to the Ogilby Road. The existing line is located about five miles west of the project site. An above-ground power transmission line would be erected to connect the existing Imperial Irrigation District line to the site. A second lower voltage line would be run along the new high voltage poles to the proposed well site which is on the way to the project along the Indian Pass Road.

Site power requirements were estimated to be 5,000 KW based on the design of the equipment for this technical report. Power requirements are mainly for the well pumps, leaching pumps, the ADR plant and the site office and shop facilities. No mining equipment, other than sump pumps in the bottom of the pit, is proposed to be electric powered.

18.3 Access Roads

The mine is accessed by Indian Pass Road from highway S-31 (Ogilby Road) north off of California Interstate 8. One mile of existing Indian Pass Road would need to be re-aligned around the west side of the west pit and returned to its original position during reclamation. The trafficability of the Indian Pass Road should be improved by grading and additional road bed materials as required.

18.4 Water Balance and Water Supply

The following summarizes key components of the hydrologic analysis completed for the project by GRE.

Using a combination of HLF design data, project data, climate information obtained from publicly available sources and previous reports (SRK 2012), GRE completed a preliminary hydrological assessment of the Imperial Project site.

Meteorological information was acquired from the Western Regional Climate Center (WRCC), and gauging station information for the area compiled from US Geological Survey databases. Annual pan evaporation records were obtained from a technical report prepared by Farnsworth and Thompson (1982). Monthly distribution of pan evaporation was obtained from WRCC.

18.4.1 Water Balance

Modeling of the heap operation on a monthly basis over the projected mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 1,100 gpm (250 m^3/hr). An additional 150 gpm (34 m^3/hr) is required for mine, shop, and office water consumption.

A water balance around the heap leach was produced using average rainfall, evaporation and temperatures. Key parameters included in the hydrologic assessment were average precipitation, average runoff, and pan evaporation. No simulation was conducted to incorporate major events at this stage of the study. Table 18.1 presents the distribution of average precipitation at the project site.

Table 18-1: Imperial Site Average Climate Conditions

Month	Precipitation	High Average	RH	Low Average	RH	Pan Evaporation
	(mm)	(deg C)	(%)	(deg C)	(%)	(mm)
Jan	9.9	25.56	29	1.11	49	90.9
Feb	8.9	28.89	24	2.78	47	110.7
Mar	7.4	33.33	21	5.00	46	173.0
Apr	2.8	37.22	17	7.78	38	232.9
May	1.0	41.11	15	11.67	38	298.5
June	0.3	45.00	15	16.11	36	335.0
July	5.3	45.56	21	21.11	46	351.8
Aug	11.7	45.00	26	21.11	53	311.9
Sept	10.2	43.33	22	16.67	51	241.6
Oct	6.6	38.33	22	10.00	47	175.5
Nov	5.8	31.11	24	4.44	45	112.5
Dec	11.2	25.56	32	1.67	51	85.6
	81.0					2519.9

SRK had estimated the mean annual runoff for the mine site to be approximately 0.04 inches/year.

18.4.2 Ground Water

WSE 1996 FS describes the ground water potential from three aquifers underlying the project area; a confined alluvial aquifer, an unconfined aquifer and a bedrock aquifer. The alluvial aquifers are found in consolidated and unconsolidated sands and gravel while the bedrock aquifer occurs in metamorphic rock.

The general ground water flow is northeast to southwest from the Chocolate mountains to the alluvial basin of the valley floor. A combination of piezometer and monitoring wells were installed to determine static groundwater elevations, to evaluate water chemistry and to estimate the in-situ aquifer hydraulic properties associated with each aquifer system.

10 monitoring wells were tested in January 2020 to establish the depth to groundwater in and around the site and the average depth to groundwater was 615 feet below the surface. The wells close to the proposed pits showed ground water at approximately 730 feet below the surface and 550 feet below surface at the existing water well. The monitoring showed that most of the water levels had remained very consistent from when the wells were completed in the mid 1990s.

18.4.3 Water Balance

A preliminary operational average monthly water balance model was developed for the HLF. The intent of the modeling was to estimate the magnitude and extent of any water surplus or deficit conditions in the HLF based on annual average climatic conditions. The modeling timeline was for 9 years of HLF operations.

The model incorporates the following major project components:

- Heap Leach Pad
- Mine Usage
- Shop Usage
- General Usage
- Fresh Water Supply
- Pond and Tank Storage – PLS, Barren and Event

The findings of the water balance were that the HLF would operate in a water deficit. The deficit is most pronounced in the early years and is reduced as water stored within the ore is released from the earlier leaching stages. The total make-up required by the HLF is estimated at 4.8 billion gallons 18 million m^3 over the life of the facility. The HLF water requirement ranges from 470 million gallons to 500 million gallons annually (1.8 million m^3 to 1.9 million m^3 annually). The project requires a significant amount of water at start up due to the initial ore wetting requirements and the solution retention in the heap. GRE estimates that approximately 180 million gallons (675,000 m^3) of fresh water would be necessary at the start of heap operations.

The water balance was based on assumed moisture content values for the stacked ore and climatic conditions for the site. The model is sensitive to these values and they should be reviewed and confirmed for future design studies. The following criteria were employed in the water balance:

- Natural Moisture Content – Ore 3%
- Field Moisture Content – Ore 12%
- Drain-Down Final Moisture Content 8%
- Evaporation Losses – 11% total
- Pan Evaporation for pond based on Yuma Arizona.
- Average Irrigation Rate 0.005 gpm/ft^2 (12.2 lph/m^2)

- Pad Area – Phase 1,2, and 3: 5,381,955 ft^2, 8,072,932 ft^2 and 10,763,910 (500,000, 750,000, 1,000,000 m^2)
- Climate Conditions monthly temperature, precipitation and evaporation

18.5 Mine Facilities

GRE has provided conceptual design of facilities required for mine operations. These include access roads, offices, warehouses, shops, leach pad, and waste dumps (see Figure 16-1).

18.5.1 Waste Dump Facilities

The waste dump facilities (WDFs) are planned to be temporarily located adjacent to the final pit limits for the East, West, and Singer deposits. The South WRF is the largest of the three facilities planned, which also include the North WRF (north of the West Pit) and the East WRF (directly north of East Pit). Backfilling into previously mined out areas is also planned for the West and Singer pits, as well as the eastern portion of the East pit.

The three WRFs would be built in a series of lifts in a “bottom-up” approach in order to maximize stability. The WRFs would be constructed by placing material at an angle of 1.5:1.

The backfilling of the previously mined out pits during the active mine life is planned to minimize the amount of waste material that needs to be reclaimed at the end of the mining operation. The pits would be backfilled to no more than 25 ft above original ground elevations. Table 18-1 summarizes the waste emplacement volumes during the mining operations in the various WRFs including backfills.

Table 18-1: Design Capacities of the Various WRFs (including Backfills)

WRF	Capacity (million cu yd)	Capacity (million Tons)
East WSF	25.8	35.0
North WSF	5.0	6.5
South WSF	68.6	91.6
East Backfill Part 1	21.2	28.0
East Backfill Part 2	12.8	18.0
West Backfill	51.3	68.5
Singer Backfill Part 1	4.4	6.0
Singer Backfill Part 2	1.7	2.2
Total	**190.8**	**255.8**

18.5.2 Mine WRF Development Schedule

Table 18-2 provides the annual sequential development of the various WRFs during the mining operation. Waste material will be left in the various storage facilities such that the reclamation surface will be approximately 25 feet above the original topography.

Table 18-2: Waste Storage Total per Year (Ktons)

WSF	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
East WSF	297	25,517	9,165	0	0	0	0	0	0
North WSF	0	0	6,497	0	0	0	0	0	0
South WSF	0	0	20,690	35,832	22,163	0	0	11,910	1,056
East Backfill Part 1	0	0	0	0	0	0	27,964	0	0
East Backfill Part 2	0	0	0	0	0	0	0	17,988	0
West Backfill	0	0	0	0	9,395	39,003	9,466	10,643	0
Singer Backfill Part 1	0	6,041	0	0	0	0	0	0	0
Singer Backfill Part 2	0	2,213	0	0	0	0	0	0	0
Total	**297**	**33,771**	**36,352**	**35,832**	**31,558**	**39,003**	**37,430**	**40,541**	**1,056**

The mine development sequence and the geometric shape of the pits provide limited concurrent backfilling opportunities of the pits. In Year 1, the Singer pits are exhausted which allows backfilling to commence. The Singer Pits are backfilled to the original ground surface. Once the Singer Pits are filled, waste storage returns to the East, North, and South WSFs. In the fourth quarter of year 4, waste backfilling commences in West pit. The West pit backfill is used exclusively for waste storage until year 6, when Stage 1 of East Pit is mined out, which provides the opportunity to commence backfilling the pit with the advantage of a shorter haulage distance than hauling to West Pit or an external WRF. The West pit backfill and the East pit Stage 2 store waste through Year 7 when all available backfill storage is filled.

To meet the regulatory guidelines of restoring the site to be within +/- 25 ft of the original ground surface, a total of 40.2 Mt of waste material stored in the WRFs and 91.5 Mt of heap leach material stored on the HLF pad will be re-handled and placed into the East pit. The North and East WRFs will be drawn down to within 25 ft of original topo and placed in the pit; the entirety of the HLF material will be moved to the pit, and 8.2 Mt of waste material from the South WRF will be placed on the top of the HLF material in the West Pit to return the pit areas to original topography.

The backfill material will be utilized to re-create the washes with sufficiently high berms, as well as curtain the runoff to the stream channel. The design would mimic the existing wash topography and physiological characteristics. The following are some conceptual design criteria that would be incorporated into the next phases of engineering.

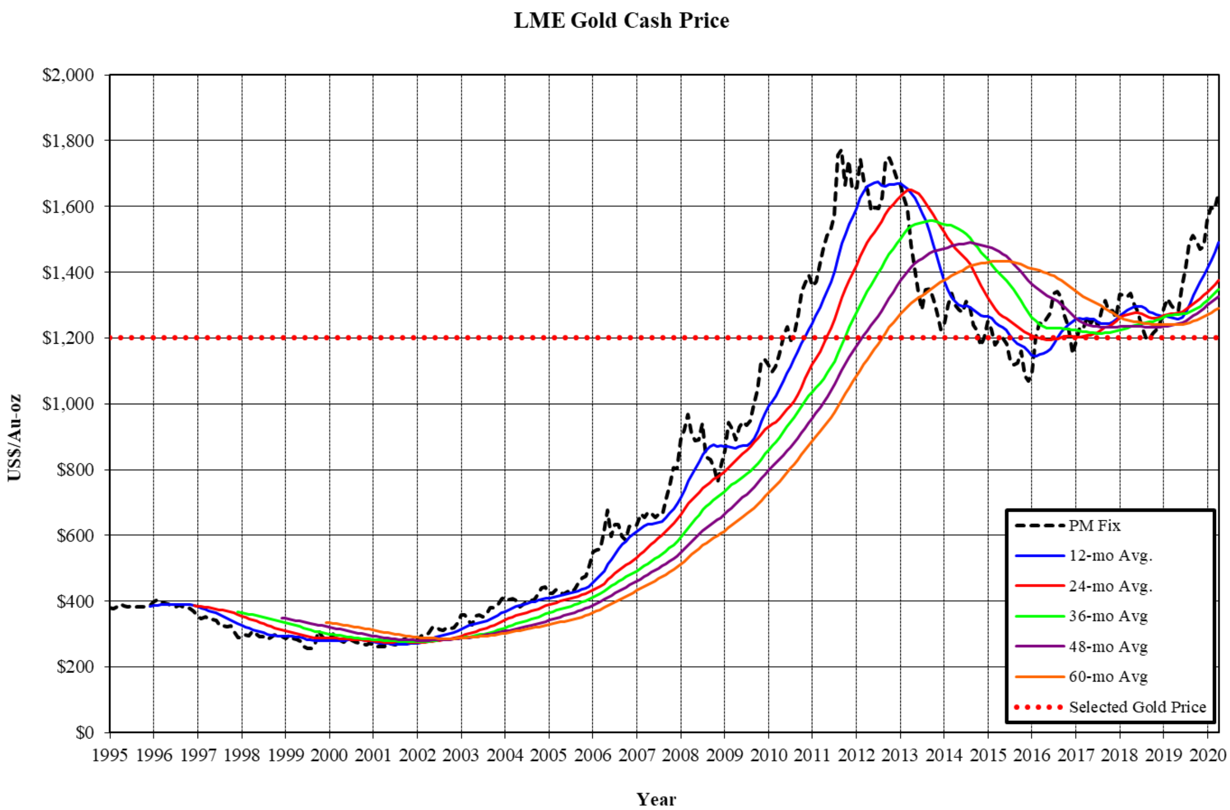
- The backfill area would not impound water.
- Any washes would be rebuilt to pre-mining elevations.
- The centerline of the wash through the pit backfill area would maintain the pre-mining slope (fall) of the original wash. The entrance and exit of the wash through the pit area should not include any drops or rises, but should smoothly match to the existing slope.
- The wash bottom would be reconstructed with stockpiled wash materials (sands and gravels).
- The pit backfill areas outside the washes can be below the pre-mining topography, but should mimic the morphology of the pre-mining slopes in that vicinity unless they are steeper than 3H:1V.
- The final reclamation surface will be less than or equal 25 ft above the current surface topography over almost all of the project area if the waste dumps and HLF material are required to be removed to within 25 ft of original topography.

- The maximum slope would be 3H:1V.
- The portion of the South WRF remaining after reclamation will be used as a construction aggregates source for nearby construction projects. There has been no value escribed to the future value of these aggregates in this report.

19.0 MARKET STUDIES AND CONTRACTS

The primary metal of economic interest for the Imperial project is gold. Gold has a readily available market for sale in the form of gold doré or gold concentrates. Figure 19-1 presents the gold market London PM fixed pricing through April 14, 2020. The selected Gold price for the PEA is $1,450/oz which represents the 3-year trailing average, $1,325/oz weighted by 60% and $1,620/oz projected gold price weighted by 40%. The Company nor any of the authors of this report have conducted a market study in relation to the gold doré or gold concentrates that will be produced at the Imperial Gold Project. The refining treatment charge in this study is assumed to be $5 per ounce.

Figure 19-1: London Metals Exchange PM Gold Price



20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Permits and Authorization

20.1.1 Site Permitting Background

In 1994, an application was submitted to the U.S. Department of the Interior (DOI) Bureau of Land Management (BLM) for approval of a plan of operations for the Glamis Imperial Project (Glamis Project) under the Federal Land Policy and Management Act (FLPMA). An application was also submitted to the County of Imperial (County) for approval of a reclamation plan pursuant to the California Surface Mining and Reclamation Act (SMARA). The County and BLM coordinated the preparation of an environmental impact statement/environmental impact report (EIS/EIR) under the National Environmental Policy Act (NEPA) and the California Environmental Quality Act (CEQA).

Opposition to the project elevated its consideration to the DOI Secretary and based on a then-recent legal opinion of the DOI Solicitor, a Record of Decision (ROD) was initially issued in early 2001 denying the Glamis plan of operations, primarily because of unavoidable adverse impacts to Native American cultural resources. However, following a change of Administration, later that year the new DOI Solicitor reconsidered and rescinded the prior Solicitor's legal opinion and recommended that DOI reconsider the ROD on Glamis Imperial's plan of operations. On November 23, 2001, the DOI Secretary concurred and formally rescinded the prior ROD denying the plan of operations. The BLM subsequently issued its final mineral report on September 27, 2002, confirming that Glamis Imperial held valid existing rights to the mining claims and the vast majority of the mill sites, and that Glamis Imperial could profitably produce from an open pit mine substantial gold reserves from the Glamis Project as proposed.

Meanwhile, in September 2002, the California Legislature added Section 2773.3 to the California Public Resources Code, requiring the backfilling of metallic mines and mines "located on, or within one mile of, any Native American sacred site and located in an area of special concern." In December 2002, the California State Mining and Geology Board approved a new regulation implementing the requirements of Section 2773.3. At the time, these statutes and regulations made open-pit gold mining cost prohibitive in California because of the cost of backfilling relative to the price of gold, and Glamis therefore suspended its effort to develop the Glamis Project. However, the mineral claims have been maintained in good standing for more favorable economics.

KORE's revised project (the Imperial Project) would include mining at least the same mineral resources as the Glamis Project but would include a re-evaluated engineering design for the mineral resource and updated environmental data. From a permitting perspective, the site conditions and land use entitlement requirements have not substantially changed since the proposal of the Glamis Project. Certain updates (analysis of greenhouse gas emissions, for example) will be necessary to update and amend the existing plan of operations in conformance with current requirements. However, because technology has been significantly improved since the original Glamis Project was considered, air emissions from mining equipment, for example, are much reduced compared to the prior environmental estimates.

The following provides an overview of the permits and other land use entitlements required for a modern precious metal mine in California, and the approach to amending and updating the existing plan of operations and environmental documentation.

20.1.2 Primary Entitlements

The plan of operations and reclamation plan are the primary plans required for a mining project on federal lands.

20.1.2.1 Plan of Operations

As the Project applicant, KORE must file a plan of operations with BLM (43 CFR § 3809.11). The BLM 3809 regulations apply to mining activities on BLM-managed lands in the western United States. The plan of operations must demonstrate that the proposed operations would not result in "unnecessary or undue degradation" of public lands. The plan of operations must also include operator information, a description of operations, a reclamation plan, a monitoring plan, an interim management plan, and a reclamation cost estimate (43 CFR § 3809.401). The existing plan of operations is substantially complete and would need to be updated to apply to current regulations and the details of KORE's Imperial Project.

20.1.2.2 Reclamation Plan

Under federal law, KORE must file a reclamation plan for the Project (43 CFR § 3601.42) that specifies the proposed manner in which the areas disturbed by operations will be reclaimed and the associated schedule for reclamation. In addition, SMARA, applies to surface mining operations on federal land in California, and requires the submittal of a reclamation plan. The existing reclamation plan is substantially complete and would be updated to address current regulations and the details of KORE's Imperial Project.

20.1.3 Environmental Review and Key Environmental Permits

20.1.3.1 NEPA/CEQA Environmental Review

Discretionary actions that qualify as "projects" in California require environmental review under CEQA. In addition, projects that either occur on federal land or require federal approval require environmental review under NEPA. The joint NEPA/CEQA environmental review was previously completed for the Glamis Project, including detailed technical evaluations. These evaluations remain substantially applicable to KORE's Imperial Project, requiring only certain revisions necessary to account for changed regulatory requirements, changes to the existing environmental setting, if any, and design changes in comparison to the Glamis Project. Thus, the previously prepared joint NEPA/CEQA environmental document and associated technical studies can be incorporated by reference, allowing the updated NEPA/CEQA documents prepared for KORE's Imperial Project to be focused on any regulatory, environmental, and design changes.

The following environmental permits are required subsequent to NEPA/CEQA review and project approval**:**

<u>Section 7 of the Federal Endangered Species Act (ESA)</u>

In conjunction with the environmental review for any federal approvals needed for the Imperial Project (e.g., BLM approval of a plan of operations), under Section 7 of the (ESA), the approving federal agency will need to consult with the U.S. Fish and Wildlife Service (USFWS) regarding the potential for "take" of federally listed species. The Imperial Project site is located in an area known to contain desert tortoise and Yuma clapper rail habitat. The desert tortoise is listed as "threatened" and the Yuma clapper rail is listed as "endangered" under the ESA. However, no critical habitat has been identified for either species within the existing mining claims. A biological opinion for the Glamis Project was previously issued by the USFWS.

Section 2081 of the California Endangered Species Act (CESA)

If implementation of KORE's Imperial Project has the potential to adversely affect state-listed endangered or threatened fish and wildlife, the California Department of Fish and Wildlife (CDFW) must be contacted and advised of the Project and its potential impacts. If a federal incidental take permit is required under the ESA for impacts to a federally listed species, and the same species is also protected under CESA, the Project proponent may submit the federal incidental take statement to CDFW to determine whether the federal document is "consistent" with CESA. If the federal permit is found to be "consistent" with CESA, a state incidental take permit would also be issued.

Section 404 Permit of the Federal Clean Water Act (CWA)

CWA Section 404 requires a permit from the U.S. Army Corps of Engineers (USACE) for the discharge of dredge or fill material into the Waters of the United States, including streams and wetlands (33 USC § 1344). Because the Imperial Project site was previously determined to include desert washes that were determined to be jurisdictional Waters of the United States, potential impacts to those desert washes, if still in existence, could trigger the need for a CWA Section 404 permit. USACE would review the permit application and consult with the U.S. Environmental Protection Agency (EPA) before issuing the Section 404 permit.

The Final EIS released for the prior Glamis Project determined that 114.5 acres of Waters of the U.S. were present on the mine site. Since that determination, there have been several court decisions regarding the scope of federal jurisdiction under the CWA. For example, in *Solid Waste Agency of Northern Cook County v. U.S. Army Corps of Engineers*, 531 U.S. 159 [2001]) *(SWANCC)*, a plurality of U.S. Supreme Court Justices held that the CWA did not give the USACE authority to assert federal jurisdiction over "isolated waters" (i.e., the ponds that were not connected with or adjacent to a traditional navigable water of the United States). Additionally, the Court held that where the use of waters for migratory birds was the only basis for asserted CWA jurisdiction, and no "significant nexus" to navigable waters existed, the CWA did not apply. Later, in *Rapanos v. United States*, 547 U.S. 715 (2006), the U.S. Supreme Court determined that the scope of federal agency regulatory authority should extend only to "relatively permanent, standing or continuously flowing bodies of water" connected to traditional navigable waters, and to "wetlands with a continuous surface connection to" such relatively permanent waters."

Note also that the Trump Administration has directed the EPA to reconsider the definition of Waters of the United States and the EPA is in the process of publishing a revised rule to define the scope of CWA Section 404 authority.

Streambed Alteration Agreement of California Fish and Game Code Section 1602

The California Fish and Game Code (Section 1602) requires anyone proposing an activity that may substantially modify a stream to notify CDFW. The notification requirement applies to activities proposed in or near a stream, even if water only flows intermittently through a bed or channel. After receiving notification of the proposed activity, if CDFW determines that the activity may substantially adversely affect fish and wildlife resources, a streambed alteration agreement would be prepared. The agreement would contain conditions to mitigate the Imperial Project's expected impacts on the waterbody.

The technical studies prepared for the Glamis Imperial Project identified several desert washes that appeared as "blue-line streams" on standard U.S. Geological Survey maps and therefore, were presumed to be "waters of the state" subject to the jurisdiction of CDFW. Accordingly, a streambed alteration agreement was required to permit disturbance of these desert washes. If those desert washes still exist and the proposed project plan will disturb the desert washes, KORE's Imperial Project may require a streambed alteration agreement.

, "Timeline for Key Permit and Approvals," summarizes the key approvals, typical time frames, and approach for the KORE Imperial Project.

Table 20-1: Timeline for Key Permits and Approvals

Permit/Authorization	Timeline	Work Needed
Environmental Impact Statement (EIS) *(NEPA, 42 USC § 4321 et seq.)*	18–24 months The BLM would become involved in the process at the time of pre-application meeting and application submittal.	A revised or amended EIS is needed to address changed conditions or circumstances, if any, and design revisions to the Glamis Project.
Environmental Impact Report (EIR) *(CEQA, PRC § 21000 et seq.; 14 CCR § 15000 et seq.)*	Prepared concurrently with the NEPA document. The County would become involved in the process at the time of pre-application meeting and application submittal.	The analysis for KORE's Imperial Project would be revised for new requirements under CEQA (e.g., GHG).
Plan of Operations *(FLPMA, 43 USC § 1701)*	Processed concurrently with the NEPA document. The BLM would become involved in the process at the time of pre-application meeting and application submittal.	Update for potential changed conditions and revised design and operation. Review of regulations and guidance to confirm whether additional revisions are necessary.
Mining/Reclamation Plan and Financial Assurance *(SMARA) (PRC § 2710 et seq.)*	Processed concurrently with the CEQA and NEPA review. The County would become involved in the process at the time of	Update for potential changed conditions and revised design and operation. This would be done as part of the reclamation plan process with Imperial County. KORE's Imperial will be phased to comply with the current backfilling regulations.

Permit/Authorization	Timeline	Work Needed
	pre-application meeting and application submittal.	
Biological Assessment, Section 7 Consultation, Biological Opinion (BO) *(ESA, 16 USC § 1531-1544)*	Section 7 consultation, incidental take statement: 6 to 12 months	Update the BO for potential changed conditions and revised design and operation.
California Endangered Species Act Section 2081 Permit *(CESA) ((Fish and Game Code § 1603)*	9-18 months; can be sought concurrent with other approvals	Obtain for project as approved.
Water Discharge Permit (Water Code 13000 et seq.)	6–9 months to obtain, after CEQA document is complete	Obtain for project as approved.
Individual/Nationwide Section 404 Discharge Permit *(Clean Water Act, 33 USC § 1341)*	12-18 months	Obtain for project as approved.
Lake/Streambed Alteration Agreement *(Fish and Game Code § 1603)*	6–9 months to obtain, after CEQA document is complete	Obtain for project as approved.
Section 401 (Water Quality) Certification *(CWA, 33 USC § 1251: If the Project Requires USACE 404 permit)*	2–6 months, after CEQA document is complete	Obtain for project as approved.
Authority to Construct *(Local district rules, per Health and Safety Code § 42300 et seq.)*	6 months, after CEQA document is complete	Obtain for project as approved.

Notes: BLM = U.S. Bureau of Land Management; CEQA = California Environmental Quality Act; CCR = California Code of Regulations; CWA= Clean Water Act; NEPA = National Environmental Policy Act; PRC = Public Resources Code; USC = U.S. Code; USACE = U.S. Army Corps of Engineers.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The capital cost estimate has been prepared for the PEA under the assumption of processing 33,000 short tons per annum of gold ore on a run of mine heap leach. Project costs were estimated using cost data from Infomine (Infomine, 2019) and experience of senior staff. The initial capital costs are incurred in the year prior to production. GRE expects there will be 3-5 years of continued exploration, engineering, and permitting prior to a production decision.

Initial capital costs are defined as all costs until a sustained positive cash flow is reached. This includes labor and development costs in the pre-production year. Sustaining capital is defined as the capital costs incurred in the periods after a sustained positive cash flow is achieved through the end of mine life.

All capital cost estimates cited in this Report are referenced in US dollars with an effective date of April 2020.

Table 21-1: Imperial Capital Costs

Initial and Sustaining Capital Costs ($ millions)	
Mining & mine Infrastructure	$35.31
Heap leach pads and plant	$47.00
Infrastructure & G&A	$15.68
Working capital	$7.49
Contingency (25%)	$23.65
Total Pre-Production Capital	**$129.13**
Pre-production mining	$14.34
Total Pre-Production Cost	**$143.47**
Sustaining capital	$60.54
Closure, incl. Backfill	$147.68

21.1.1 Facilities

All buildings and associated infrastructure installed on the property on a permanent or semi-permanent basis are considered facilities. They include material and installation cost.

Each item’s capital cost was estimated based on knowledge of nearby mine operations or senior engineers’ experience. Table 21-2 shows total cost for each facility item.

Table 21-2: Capital Cost, Facilities

Item	Cost
Haul Roads	$460,000
Office	$787,500
Warehouse	$1,000,000
Mine Shop	$3,500,000
Fuel Bay	$100,000
Wash Bay	$200,000
Security and Fencing	$250,000

Item	Cost
Surface Water Management	$500,000
Water Well with Pump	$1,250,000
New Well Pump	$67,200
Back Up Gen Set	$346,400
Sub-Station	$1,500,000
Power Line 33KV	$1,767,000

21.1.2 Process Plant

The $47,003,000 cost of the process plant, including the first phase of the heap leach pad, is incurred in the preproduction year. Heap leach expansion occurs in years four and seven of production with a cost of $8,610,000 in each of those years for a total of $64,223,000 (rounded to the nearest thousand). The breakdown of the unit costs of the process plant is shown in Table 21-3.

Table 21-3: Capital Costs – Process Plant

Capital Costs	Cost USD
Fixed Equipment	
Lime Handling	$461,400
Leach Pad, Ponds, Sol'n Dist and Collection	$26,100,800
ADR	$4,217,800
Utilities	$1,462,700
Total Equipment	$32,242,700
Installation Labor	$8,883,600
Concrete	$671,800
Piping	$2,782,500
Structural Steel	$806,600
Instrumentation	$797,400
Insulation	$321,300
Electrical	$835,600
Coatings and Sealants	$333,600
Spares and First Fill	$2,991,400
Engineering/Management	$7,991,400
Total - Fixed Equipment	**$58,657,900**
Mobile Equipment	
For Pad	$5,200,000
Maintenance	$125,000
Light Vehicles	$240,000
Total - Mobile Equipment	**$5,565,000**
Total - Mobile and Fixed Equipment	**$64,222,900**

21.1.3 Mine Equipment

Initial major mobile equipment is purchased in the pre-production year and the first operating year (see Table 21-4).

Table 21-4: Initial Equipment Purchase

Description	Quantity	Each	Total
Excavator CAT 6040	2	$8,420,200	$16,840,400
Haul Truck CAT 789D	9	$3,081,700	$27,735,300
Bulldozer D10	3	$1,090,600	$3,271,800
Drill	2	$2,000,000	$4,000,000
Wheel Dozer	1	$1,044,700	$1,044,700
Wheel Loader	1	$2,208,100	$2,208,100
Water Truck	2	$1,140,000	$2,280,000
ANFO Truck	1	$219,800	$219,800
Lube Truck	2	$84,200	$168,400
Mechanics Truck	2	$70,600	$141,200
Grader	1	$443,300	$443,300
Small Excavator	1	$305,109	$305,109
Backhoe	1	$128,840	$128,840
Small Crane	1	$395,216	$395,216
Light Plant	6	$25,300	$151,800
Dewatering Pump	1	$164,887	$164,887
4x4 Pickup	10	$46,100	$461,000
Total			**$59,959,852**

21.1.4 Working Capital

Working capital is the necessary cash on hand for the next period's operating cost. The estimated total is $7,487,500. Note that this cost is recovered at the end of production.

21.1.5 Closure

Closure costs are estimated over six years at the end of production due to the need to rinse and neutralize the leached ore. Total cost for site closure is $25.4 million for rinsing and neutralizing the heap leach pad, backfill is $107.5 million, and general and administrative costs during this time add up to $15.0 million. The combined cost for the three parts of closure is $147.9 million.

21.2 Operating Cost Estimate

Operating costs are presented in Table 21-5.

Table 21-5: Imperial Operating Costs

Operating Costs	Unit	Cost
Mining costs (owner)	$/st mined	$1.45
Mining costs	$/st processed	$5.51
Processing costs	$/st processed	$1.85
G&A costs	$/st processed	$0.74
Total site operating costs	$/st processed	$8.11

21.2.1 Labor

Hourly labor in the project is based on the number of people needed to operate and support equipment for each shift in a day plus additional crew to fill in for absences. Salaried labor in the project is based on job positions filled regardless of production changes or equipment units needed. Table 21-6 through Table 21-9 show the required labor.

Table 21-6: Hourly Laborers by Year

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Heavy Equipment Operator	7	40	40	40	40	40	40	40	24	0	0	0	40	40	40
Blasters	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Laborer	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Drill Operator	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Production Truck Driver	4	36	36	36	32	44	40	52	16	0	0	0	16	16	16
Oilers/Mechanic	6	38	38	38	36	42	40	46	20	0	0	0	28	28	28
Total	**35**	**132**	**132**	**132**	**126**	**144**	**138**	**156**	**78**	**18**	**18**	**18**	**102**	**102**	**102**

Table 21-7: Salaried Workers, Mine Management

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Engineer	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0
Geologist	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0
Surveyor/Tech	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
General Foreman	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Shift Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Total Salaried	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**7**	**7**	**7**	**7**	**7**	**7**

Table 21-8: General and Administrative Positions by Year

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
General Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Purchasing Manager	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Purchaser	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0
Chief Accountant	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Accounting Clerk	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Human Resources/Relations Manager	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Human Resources/Payroll Clerk	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Security/Safety/Training Manager	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Safety Officer	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Environmental Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Environmental Technicians	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0
Logistics Administrator	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
IT Manager	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Warehouseman ON SITE	4	4	4	4	4	4	4	4	4	1	1	1	1	1	1
Accounts Payable Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Receptionist/Secretary	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Guards	4	4	4	4	4	4	4	4	4	0	0	0	0	0	0
Drivers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Laborers / Janitorial ON SITE	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0
Total G&A	**31**	**31**	**31**	**31**	**31**	**31**	**31**	**31**	**31**	**8**	**8**	**8**	**8**	**8**	**8**

Table 21-9: Processing Positions by Year

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
General Foreman	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Maintenance Foreman	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Shift Foreman	4	4	4	4	4	4	4	4	4	0	0	0	0	0	0
Chief Assay Chemist	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Sr Metallurgist	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Metallurgist	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Instrument Technician	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0
Irrigation Operator	4	4	4	4	4	4	4	4	4	0	0	0	0	0	0
Reagent Operator	4	4	4	4	4	4	4	4	4	0	0	0	0	0	0
Dozer/FEL Operator	4	4	4	4	4	4	4	4	4	0	0	0	0	0	0
Assayers	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Mechanic	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Electrician	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Carbon Handling	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
EW Operators	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Cathode Striping	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Refiners	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Samplers	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Reagent Operator	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Mechanic	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Electrician	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Total Processing	**54**	**54**	**54**	**54**	**54**	**54**	**54**	**54**	**54**	**35**	**35**	**2**	**0**	**0**	**0**

21.2.2 Mining

The average $1.45 per ton mining cost was determined by summing the costs of equipment, consumable materials, maintenance, and labor costs and dividing by the number of tons mined during the production years in the life of mine. The average mining cost per ton of ore is $5.51.

21.2.3 Process Plant

The average $1.85 per ore ton processing cost was determined by summing the costs of equipment, materials, electricity, labor, and maintenance associated with operating the heap leach pad and ADR plant and dividing by the total ore tons produced through the life of the project.

21.2.4 Taxes and Royalties

GRE relied on Mining Tax Plan LLC to estimate the federal and California state tax schedule. Mining Tax Plan LLC has prepared the U.S federal and state income tax computation based on the Internal Revenue

Code of 1986, as amended and the regulations thereunder and the CA Revenue and Taxation Code as in effect as of March 20, 2020. We have not audited or verified any of the economic or operating assumptions of the Preliminary Economic Assessment Model but have made inquiries to properly classified revenue, expenses and capital expenditures consistent with federal and state income tax statutes, regulations and case law.

The following is a summary of tax elections incorporated into this tax computation:

- The Imperial Project consists of a single mine and property under Section 614.
- The Imperial Project will elect to expense exploration expenditures as incurred.
- The Imperial Project will elect to treat mine development costs as incurred as deferred expenses under Section 616(b).
- The Imperial Project will elect out of Section 168(k) bonus depreciation.
- The Imperial Project will elect depreciate long-lived assets under the unit of production basis under Section 168(f)(1) and all other assets will be depreciated under MACRS in accordance with Rev. Proc. 87-56.
- The Imperial Project will elect to deduct reclamation costs under Section 468.

21.2.5 General and Administrative

General and administrative costs were estimated for two phases of the mine plan: Production Operating and Rinse and Closure. The G&A costs include both salaried and hourly labor, supplies, office equipment, and anticipated regular expenses. Production years have a G&A cost of $7.8 million per year. Reclamation and closure years have a G&A cost of $2.5 million per year. The average for production years is $0.74 per ton.

22.0 ECONOMIC ANALYSIS

22.1 Project Forecast

Analysis of the Imperial project includes statements addressing events in the future. Conditions regarding these events have potential to change, and as such, present an inherent risk. Actual results could differ from the projections estimated in this report. The economic analysis is modeled at the time of a production decision. It allows for 1 year of preproduction and construction. Costs incurred for exploration, engineering, and permitting over 3 to 5 years leading up to a production decision are not include.

Table 22-1: Summary of Imperial Economic Results

Economics	Unit	Pre-Tax	Post-Tax
Net present value (NPV 5%)	$ millions	$438	$343
Net present value (NPV 5%)	C$ millions	$584	$458
Internal rate of return (IRR)	%	52%	44%
Payback (undiscounted)	Years	2.3	2.7
LOM average annual cash flow *	$ millions	$105	$90
LOM cumulative cash flow *	$ millions	$697	$580
Cumulative cash flow (undiscounted)	$ millions	$438	
Gold price assumption	per ounce	$1,450	

22.2 Taxes and Royalties

The economic analysis includes the 1% NSR royalty payable to Macquarie Bank and the second 1% NSR royalty that is payable to Newmont for a total of a 2% NSR royalty. The undiscounted value of the 2% total NSR royalty for the base case is $33.8 million.

The U.S. federal income tax is based on the Internal Revenue Code of 1986, as amended and the relevant state and local statutes, the regulations thereunder, and judicial and administrative interpretations thereof, on the following assumptions and tax return elections by the taxpayer, based on the PEA cashflows and capital expenditures. As of April 6, 2020, the U.S. federal corporate income tax rate is twenty-one (21) percent, the State of California rate is (8.86) percent and the federal and state income tax is based on the following assumptions and tax elections:

The Imperial Project is owned by a California Corporation (“taxpayer”) which is a wholly owned direct or indirect subsidiary of KML.

The Imperial Project has acquired an economic interest in the minerals in place and is operated and treated as a single mine under Section 614.

The Imperial Project will elect to expense exploration expenditures under Section 617(a) as incurred.

The Imperial will deduct mine development costs as incurred under Section 616(a).

The Imperial Project will elect out of Section 168(k) bonus depreciation.

The Imperial Project will elect to accrue and deduct reclamation costs under Section 468.

California Property Tax is imposed under Revenue and Taxation Code 20584 and the regulations on real and personal property based upon the municipality and county where the mine is located.

22.3 Mine Life

The project has a short pre-production period of less than 1 year, a production life of 8 years, and a reclamation and closure time of 6 years.

22.4 Economic Model

The mine plan is based on an ore production rate of 33,000 tons per day at a gold cut-off of 0.005 troy oz per ton. All ore material is sent to the heap leach pad as run of mine. Recovery is assumed to be 73%.

This technical report is a preliminary economic assessment and is preliminary in nature and partially utilizes inferred mineral resources. Inferred mineral resources are considered too speculative, geologically, to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The following table summarizes the results of the PEA.

22.4.1 Results

The economic model results are summarized in Table 22-2.

Table 22-2: Economic Model Results Years 1 - 14

Economic Value	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Recovered Gold ('000s troy oz)	1,167	0	109.0	127.0	137.0	137.0	187.0	161.0	193.0	115.0	1.0	0.0	0.0	0.0	0.0	0.0
Gold Production Revenue (million$)	$1,692	$0	$158.3	$184.4	$198.4	$198.2	$270.6	$233.7	$279.1	$167.1	$1.6	$0.0	$0.0	$0.0	$0.0	$0.0
Refining/Selling Cost (million$)	$5.8	$0.0	$0.5	$0.6	$0.7	$0.7	$0.9	$0.8	$1.0	$0.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Royalty (million$)	$33.8	$0.0	$3.2	$3.7	$4.0	$4.0	$5.4	$4.7	$5.6	$3.3	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
OpEx Mine Equipment (million$)	$408.1	$3.1	$39.1	$39.9	$56.3	$37.4	$43.4	$49.6	$59.2	$17.0	$0.0	$0.0	$0.0	$21.0	$21.0	$21.0
OpEx Mine Labor (million$)	$212.9	$3.6	$21.0	$21.0	$21.0	$20.0	$23.1	$22.1	$25.2	$9.2	$0.8	$0.8	$0.8	$14.8	$14.8	$14.8
OpEx Process Incl Labor (million$)	$169.7	$0.0	$22.0	$22.3	$21.9	$22.1	$23.1	$21.0	$22.8	$14.4	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0
OpEx G&A (million$)	$85.4	$7.6	$7.7	$7.8	$7.8	$7.9	$7.9	$7.9	$7.9	$7.9	$2.6	$2.6	$2.6	$2.6	$2.4	$2.3
EBITA - Earnings Before Interest, Taxes, Amortization (million$)	$775.8	($14.3)	$64.7	$89.1	$86.7	$106.1	$166.8	$127.7	$157.6	$114.8	($1.9)	($3.3)	($3.3)	($38.4)	($38.3)	($38.2)
Depreciation (million$)	$86.5	$0.0	$5.5	$6.4	$6.9	$8.4	$11.4	$9.9	$17.1	$13.7	$1.8	$1.5	$1.5	$1.1	$0.8	$0.4
Depletion (million$)	$238.3	$0.0	$22.8	$26.6	$28.6	$28.6	$39.0	$33.7	$35.1	$24.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Other Deductions (million$)	$395.7	$0.1	$27.0	$23.4	$41.2	$47.9	$53.4	$61.1	$93.7	$43.0	($1.8)	($1.7)	($1.5)	($1.5)	($0.9)	$12.3
Loss Carry Forward (million$)	($7.7)	$0.0	($7.7)	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
CA Tax (million$)	$40.0	$0.0	$2.5	$6.1	$4.5	$4.8	$8.7	$5.0	$4.8	$3.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Fed Tax (million$)	$77.4	$0.0	$4.2	$14.1	$8.1	$9.8	$19.8	$8.0	$7.4	$6.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
After Tax Operating Cash Flow (million$)	$658.5	($14.3)	$58.0	$68.9	$74.2	$91.5	$138.2	$114.7	$145.4	$105.1	($1.9)	($3.3)	($3.3)	($38.4)	($38.3)	($38.2)
CAPEX Mine Equipment (million$)	$72.3	$35.3	$24.7	$0.0	$0.0	$0.0	$6.2	$0.0	$6.2	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
CAPEX Infrastructure/Facilities (million$)	$11.7	$11.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
CAPEX Process Plant (million$)	$47.0	$47.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0

Economic Value	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
CAPEX G&A (million$)	$0.8	$0.6	$0.0	$0.0	$0.1	$0.0	$0.0	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
CAPEX Closure (million$)	$27.8	$2.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$9.8	$11.8	$1.5	$0.0	$0.0	$2.4
CAPEX Working Capital (million$)	$0.0	$7.5	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	($7.5)	$0.0	$0.0	$0.0	$0.0	$0.0
CAPEX Contingency (million$)	$37.3	$23.6	$6.2	$0.0	$0.0	$2.2	$1.5	$0.0	$3.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Cash Flow (million$)	$438.5	($143.5)	$26.1	$68.0	$73.3	$80.1	$130.0	$114.3	$126.7	$105.0	($4.2)	($15.2)	($4.8)	($38.4)	($38.3)	($40.5)
Cumulative Cash Flow (million$)	$438.5	($143.5)	($117.4)	($49.4)	$23.9	$104.0	$233.9	$348.2	$474.9	$579.9	$575.7	$560.5	$555.7	$517.3	$479.0	$438.5

Note: Numbers are rounded to the nearest 1000 and may not sum correctly due to rounding.

22.4.2 Sensitivity

Table 22-3 and Figure 22-1 summarize the sensitivity to gold price. Table 22-4 and Table 22-5 summarize the sensitivity to operating costs and capital costs.

Table 22-3: Project Economics Sensitivity to Gold Price

Item	Units	Base Case $1450/oz	Gold Price $1300/oz	Gold Price $1600/oz
Pre-Tax NPV (5%)	US$ million	$438.2	$301.2	$575.2
Pre-Tax NPV (10%)	US$ million	$340.0	$228.2	$451.8
Post-Tax NPV (5%)	US$ million	$343.4	$234.5	$450.0
Post-Tax NPV (10%)	US$ million	$262.1	$173.3	$348.9
Post-Tax IRR	%	44%	34%	52%
Cash Flow	US$ million	$438.5	$301.8	$572.1
Average Annual Cash Flow Production Years	US$ million	$90.4	$73.4	$107.1
Average Gross Revenue Production Years	US$ million	$188.0	$168.5	$207.4

Figure 22-1: Economic Model Sensitivity to Gold Price

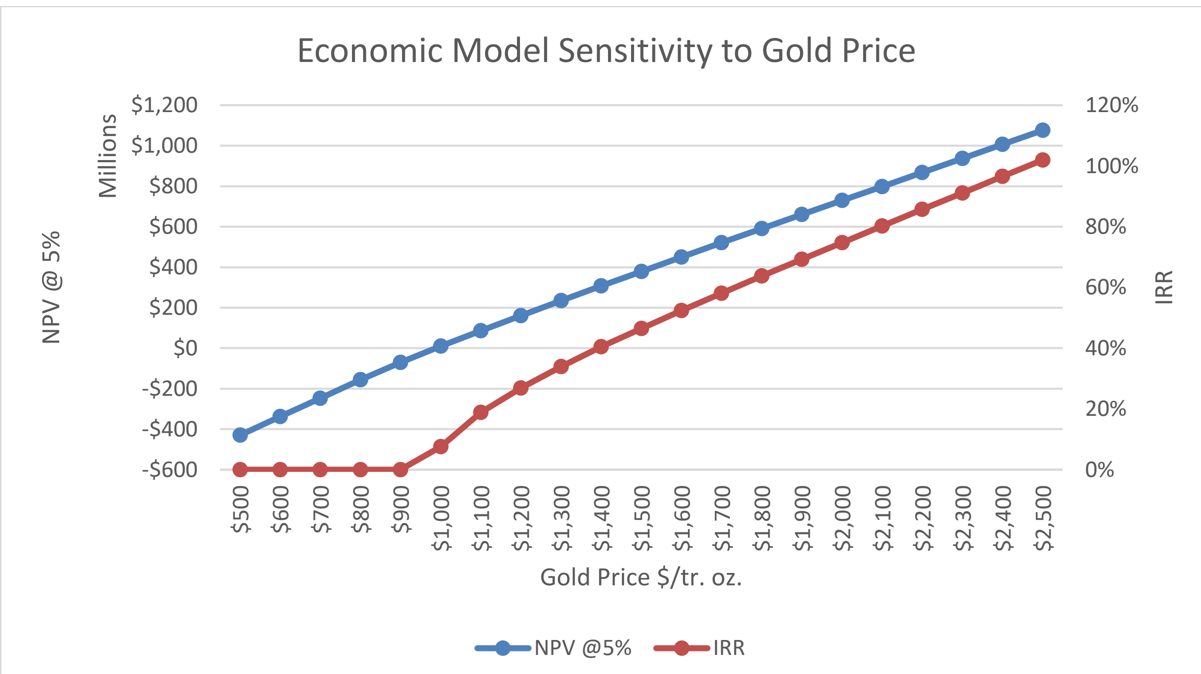


Table 22-4: Project Economics Sensitivity to Operating Costs

Sensitivity To OpEx	OpEx $/ton	OpEx $/rec oz	NPV 0%	NPV 5%	NPV 10%	IRR
80%	$6.57	$515	$571.0	$445.7	$344.3	53%
85%	$6.98	$547	$538.4	$420.5	$324.0	51%
90%	$7.39	$579	$505.5	$395.1	$303.6	48%
95%	$7.80	$612	$472.3	$369.5	$283.0	46%
100%	$8.21	$644	$438.5	$343.4	$262.1	44%

Sensitivity To OpEx	OpEx $/ton	OpEx $/rec oz	NPV 0%	NPV 5%	NPV 10%	IRR
105%	$8.62	$676	$404.6	$317.3	$241.1	41%
110%	$9.03	$708	$369.7	$290.4	$219.4	39%
115%	$9.44	$740	$334.3	$263.0	$197.4	36%
120%	$9.85	$773	$298.8	$235.6	$175.3	34%

Table 22-5: Project Economics Sensitivity to Capital Costs

Sensitivity To CapEx	CapEx $ millions	CapEx $/rec oz	NPV 0%	NPV 5%	NPV 10%	IRR
80%	$115.35	$99	$470.4	$374.0	$291.9	53%
85%	$122.20	$105	$462.6	$366.6	$284.6	50%
90%	$129.17	$111	$454.7	$359.0	$277.2	48%
95%	$136.26	$117	$446.6	$351.3	$269.7	46%
100%	$143.47	$123	$438.5	$343.4	$262.1	44%
105%	$150.79	$129	$430.2	$335.5	$254.3	42%
110%	$158.24	$136	$421.8	$327.4	$246.4	40%
115%	$165.80	$142	$413.3	$319.2	$238.4	38%
120%	$173.48	$149	$404.7	$310.9	$230.3	36%

22.5 Alternate Economic Cases

GRE estimated cost and revenue for two other options: implementing a crusher to increase gold recovery from the heap leach and hiring a contract mining company to run mining operations to decrease the initial capital cost. Neither of these cases are incorporated into the final economic analysis, conclusions, or recommendations. They are only presented here for discussion.

22.5.1 Crushing Feed to Heap Leach Pad

GRE evaluated the project using a crushed material component. A crushing circuit sized for an average throughput of 8.2 million short tons per year (7.5 tonnes per year) was added to the processing evaluation. The expected change in recovery was ROM decreasing to 65% and crushed material increasing to 80% from the ROM only case of an overall recovery of 73%. The minimum gold grade required for crushing was estimated at 0.014 oz/ton (0.48 grams per tonne). However, by using the highest grade from the mine to fill the crusher feed, the minimum grade of material crushed never falls below the minimum required. Average recovery based on this method of utilizing the crusher results in an increase of overall gold recovery to 78%. Processing cost increases to $2.48/ton ($2.73/tonne); NPV at a discount rate of 5% increases to $355 million, and IRR drops to 40%.

22.5.2 Contract Mining

GRE also evaluated the Imperial Project with mining operations performed by a contract mining company. Contract mining would enable the project to lower capital costs overall, but especially the initial capital costs which can have a great impact on NPV. The tradeoff would be an increased operating cost. Using estimates from an owner-operator cost with a profit factor and industry quotes from contract mining companies, GRE established that an average cost for contract mining is $2.04/ton ($2.25/tonne). Capital costs related to mine operation drop to $461,000; NPV at a discount rate of 5% drops to $272 million and IRR increases to 49%.

23.0 ADJACENT PROPERTIES

The operating Mesquite Mine and the closed Picacho Mine are located roughly ten miles to the northwest and east, respectively, of the property. The closed American Girl Mine is about eight miles south of the project.

24.0 OTHER RELEVANT DATA AND INFORMATION

This section is intentionally left blank. Relevant data is included in other sections.

25.0 INTERPRETATION AND CONCLUSIONS

A total of 349 boreholes, of which 344 are located within resource estimation area (comprising a total of 190,047 ft of reverse circulation drilling) have been drilled by various operators (including Gold Fields, Glamis Gold, and other historical operators) on the Imperial Gold Project from 1982 to 1996.

No exploration activity has been undertaken on the project since 1996, with minimal documentation of the historical exploration activity available to review. Although a significant amount of drilling has occurred on the property to delineate significant gold mineralization, minimal evidence of exploration procedures or protocols are available to confirm that best practice exploration methodologies were adopted. Additionally, with most of the drilling having been reverse circulation, detailed geological reviews of drill core have not been possible to define a more detailed geological / structural model for the property or to generate a better understanding of the spatial controls of gold mineralization.

In the opinion of the QP's, the sample preparation, security, and analytical procedures used to generate exploration data upon which the resource model is based is poorly documented and therefore difficult to assess. The known analytical quality control measures implemented on the Imperial Gold Project is limited to field duplicates and umpire check assays in 1991-1992 and umpire check assays in 1994-1996. Other checks on the data were likely performed by each operator but are not known to SRK.

Despite the uncertainty outlined above, limited data verification measures undertaken by KORE and SRK suggest that the exploration data are sufficiently reliable to interpret with confidence the boundaries of the gold mineralization and support the evaluation and classification of mineral resources in accordance with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

The geological information gathered from the RC drilling is sufficiently dense to allow modelling with reasonable confidence of the gold mineralization boundaries (domains 100, 110, and 120), as well as the base of gravel contact, which delimited the unconstrained domains (domains 200 and 300). However, uncertainty remains in the structural framework of the deposit. Normal faults are believed to displace the lithological units including gold mineralization but have not been modelled. The south dipping domain 110 is potentially the result of faulting. The geological continuity can only be inferred at the current drill spacing within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

Generally, for mineralization exhibiting good geological continuity investigated at an adequate spacing and displaying low structural complexity, the QP considers that blocks estimated according to parameters in Table 13.8 could be classified in the Indicated category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). For those blocks, SRK considers that the level of confidence is sufficient to allow appropriate application of technical and economic parameters to support mine planning and to allow evaluation of the economic viability of the deposit. The majority of these blocks are found within the flat lying domain 100 showing little structural complexity.

The QP considers that the mineral resource model documented in this report indicates that the Imperial Gold project hosts significant mineralization but notes that additional exploration would need to be undertaken in areas of lower drilling density to upgrade the Inferred portions of the mineral resource model to be suitable for advanced mining study applications.

The mine plan is based on 33,000 tons per day of ore production. The pits were divided into 6 phases, plus two satellite pits. Initial phases of both the east and west pits were designed as low strip-ratio volumes in order to lower the initial capital cost. The plan produces 91.5 million ore tons at an average grade of 0.017 oz/ton or 0.60 g/tonne in an 8-year mine life. Stripping requirements include a life of mine total of 255.9 million waste tons, 208.5 million tons of which are alluvium. Waste management for the mine includes 3 waste dumps and concurrent backfilling. At the end of production, the heap leach pad will be rinsed and neutralized. After which, it will be transported into the remaining open pit along with 2 dumps and a portion of the main dump. 94.7 million tons of aggregate material remain on the surface.

Operating cost in production years for the Imperial project amount to $1.45 per short ton mining cost, $1.85 per short ton processed processing cost, and $0.74 per short ton processed G&A cost. Total capital cost for the project are $72.3 million mine, $47.0 million plant, $0.77 million G&A, $11.7 million infrastructure, $17.2 million sustaining, $27.8 million reclamation, and $37.3 million contingency for a total of $214.1 million.

The PEA used a base gold price of $1,450/oz with an estimated overall recovery of 73% which resulted in an After-Tax Net Present Value at 5% of $343 million and an Internal Rate of Return of 44%. This technical report includes inferred mineral resources. Inferred resources are based on limited information, and as such are not suitable to be categorized as mineral reserves.

26.0 RECOMMENDATIONS

This section describes the principal project risks, and subsequently, the recommendations to mitigate the principal risks.

26.1 Discussion of Principal Project Risks

As with most mining projects, there are risks that could affect the economic viability of the Imperial Gold Project. Many of these risks are based on a lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted at the next study stages. Below is a discussion of some of the principal risks the Imperial Gold Project faces moving forward. External risks are, to a certain extent, beyond the control of the Imperial Gold Project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the Imperial Gold Project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates. There are significant opportunities that could improve the economics, timing, and/or permitting potential of the Imperial Gold Project. Further information and assessments are needed before these opportunities should be included in the Imperial Gold Project economics.

Historic Sampling and Assay Risks – The QP fully discusses this issue in Section 25.0 of this report where the QP points out that the drilling and assay testing that the resource is based upon was performed prior to 1997 and performing check assays is difficult as only assay pulps remain from that period. Also, there is very little core remaining from the 9 drill holes that were drilled at the site as most of the core samples were used in metallurgical test work. **Recommendation** - The Company needs to prioritize in their next phase of drilling a program that will help to confirm the validity of the current assay database through hole twinning and conformational core holes in pit bottoms. In addition, the project requires new geological/structural interpretation and an industry best practice QA/QC sampling program.

Permitting Risk – In Section 20.0 there is a full discussion of the permitting history and path forward for the project permits. There is a risk that the project will encounter serious opposition during the permitting process if it is not properly managed. **Recommendation** – The Company needs to employ experts in the area of mine permitting in California. It is true there have been very few metallic mines permitted in California since the introduction of the backfill law, but California has continued to permit construction aggregate mines and the process is basically the same for the Imperial project with the exception of the pit backfilling requirement. The Company should also initiate an industry best practice community engagement program to build local support with stakeholders.

Recovery – Section 13.0 details the historic metallurgical studies and presents the recommendation for appropriate recovery assumptions (73% for ROM). With the exception of the work performed by McClelland Labs, most of the test work was performed by the project operators back in the 1990’s. This means that some of the work is difficult to confirm (in particular the material employed in the test work). There is a strong correlation in the tests that show that the material responds well to heap leaching methods which can also been seen in other nearby operations that process material that is similar to the material found at Imperial. **Recommendation** – A full metallurgical test program should be performed to establish the gold recovery at different particle sizes, and to provide better spatial and grade representation in the testing database. The program recommendations are detailed below.

Changes to Regulations – The Project was detrimentally impacted in the early 2000’s with the introduction of the backfill law. There is a risk that this same thing could happen again if project opponents are successful in convincing regulators that the project should not move forward. **Recommendations** – All the plans for project development should completely comply with all current regulation/requirements, including the backfill law, as the plan put forward in this Technical Report does. The Company should also execute a community engagement plan at all levels of government (County, State and Federal) to educate the different levels of government on the benefits of the project to the local economy and to demonstrate that the project is complying with all US and California regulations.

26.2 Exploration, Geology and Mineral Resource Modeling Recommendations

The geological setting and character of the gold mineralization delineated to date on the Imperial Gold Project are of sufficient merit to justify additional exploration and development expenditures. The authors of this report recommend that further work be conducted to increase the confidence in the resource model. SRK recommends a data collection program that includes exploration drilling and technical data collection aimed at completing the characterization of the project in preparation for additional engineering/economic evaluation.

The objective of this work will be to upgrade the category of the resources that are presently inferred to indicated resource classification. As such, it will require more diamond drilling than RC drilling. The core drilling is needed to twin previously drilled RC holes and provide representative samples for metallurgical, geotechnical, and other materials testing. The RC drilling will infill where present drill spacing in the target resources is inadequate. On completion of Phase 1 the information gained will be assessed and if positive a Phase 2 Program will be initiated.

Specific recommendations related to geology and mineral resources, and their anticipated costs, are listed below.

26.2.1 Resource Drilling

The SRK QP considers that additional drilling is required to:

- Infill gaps in the drilling data with the potential to increase the classification of the mineral resources;
- Test the lateral and depth extensions of the gold mineralization; and
- Diamond drilling is recommended to twin and confirm selected historical reverse circulation drilling and also to better understand the stratigraphy/lithologies for 3D modeling.

The SRK QP proposes a reverse circulation infill drilling program of 47,000 ft targeting areas within the mineral resource pit shell. This reverse circulation program will cost an estimated US$2.4 million ($50/ft). An additional 16,000 ft of core drilling is recommended to twin and confirm selected historical reverse circulation drilling and also to better understand the stratigraphy/lithologies for 3D modeling. This core drilling program will cost an estimated US$2.0 million ($125/ft).

26.2.2 Geological Studies

The SRK QP recommends that geological/structural studies be initiated to build on existing knowledge of and improve the confidence in the interpretation of the boundaries of the gold mineralization; to understand its distribution; and to update the 3D geological model. Geotechnical and hydrogeological logging should be incorporated into standard field practices for all future drilling.

A budget of $125,000 should be allocated to increasing the geological understanding of the gold grade distribution, which would incorporate structural studies and 3D modeling of the deposit.

26.2.3 Exploration QA/QC

The SRK QP recommends that KORE Mining consider:

- Acquiring additional density data from each geological domain;
- Re-surveying the collar positions to validate the current collar database;
- Establishing the relationship between the currently used mine/local grid and UTM and consider migrating, as well as considering migration of the project to a new validated coordinate system;
- Inserting control samples into the sample stream of future sampling; and
- Further checking sampling of historical pulps (5% to 10% of total sample database), which is required to further validate historical assays.

The above work should provide the necessary support to migrate certain resources characterized by dense drilling from Inferred to Indicated classification. SRK recommends that a budget of $400,000 be allocated to the check sampling of historical pulps, acquiring more specific gravity data and to establish a new validated coordinate system.

26.3 Recommendations in Other Project Areas

The authors of this report recommend that KORE Mining initiates further engineering, metallurgical, geotechnical, environmental, permitting, and other studies aimed at evaluating at a conceptual level the viability of an open pit mine, with heap leach processing at the Imperial Gold project.

The proposed work program should include:

- Collection of geotechnical, hydrology, and hydrogeology data;
- Additional metallurgical test work to characterize the metallurgical variability of the gold mineralization;
- Additional metallurgical test work to confirm expected gold extraction using ROM heap leaching and other particle sizes:
- Following metallurgical testing, re-evaluate the crushing option and also the possible timing of when a crushing circuit could be installed.

- Conduct percolation and drain down testing with simulated heap loading to ensure that the heap will perform as predicted.
- Execute conceptual mine design work to evaluate which mine design options offer the best potential for economic return.
- Work to prioritize permitting efforts; currently, project permitting is one of the highest risk factors for the project.
- Expand environmental baseline studies to document baseline site conditions. This should include the monitoring of water quality, wildlife habitats, and other aspects for which long-term and seasonal data are required.
- Perform closure testing on the spent heap materials to determine if the material can cause water quality impacts.
- Execute geotechnical investigations into the heap stability.
- Perform geotechnical testing of soils under the leach pad, ponds, and plant site.
- Conduct geotechnical testing of consolidated alluvium and the pit wall rock mass.
- Negotiate with the local native population and other stakeholders to obtain a mutually beneficial project.
- Following the completion of the above items, proceed to a pre-feasibility or feasibility study.

26.4 Community Engagement and Stakeholder Mapping

The authors of this report recommend that KORE Mining initiates industry best practices community engagement program to help the project with local stakeholders and advance local project acceptance. Additionally, the authors recommend that KORE Mining go through a stakeholder mapping exercise to develop a plan to engage with local and national groups, and with different levels of governmental authorities. The total cost for this is estimated to be $200,000.

26.5 Recommendation Budget

It is estimated that the proposed drilling and exploration work and the engineering and other studies would cost approximately US$8,340,000 (Table 26-1) which includes a 10% contingency.

Table 26-1: Estimated Cost for the Exploration Program and Engineering Studies Proposed by SRK and GRE for the Imperial Gold Project

Description	Total (US$)
Drilling and Exploration	
Reverse Circulation Infill (48,000ft)	2,400,000
Core Drilling (16,000ft)	2,000,000
Geology / Structural Studies	125,000
Exploration QAQC	400,000
Subtotal	**4,925,000**
Engineering and Other Studies	
Environmental baseline studies	500,000
Advance all environmental Permits	1,000,000
Update mineral resource model with new drilling	75,000

Description	Total (US$)
Geotechnical / HL design studies	500,000
Metallurgical test work	500,000
Subtotal	**2,575,000**
Community Engagement Program	140,000
Stakeholder Mapping	60,000
Subtotal	**200,000**
Contingency (10%)	640,000
Total	**8,340,000**

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Appendices

Appendix A

Tabulated Claims List

The following claims are located in Imperial County, California within Townships 13S 21E, 14S 20E, and 14S 21E of the San Bernardino Meridian and Baseline.

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
Indian Rose Placer #11	Bk 1372 Pg 1600	13S	21E	31 & 32	CAMC24623
Indian Rose Placer #12	Bk 1372 Pg 1601	13S	21E	31 & 32	CAMC24624
H.R.J. Research #51 (amended)	Bk 1486 Pg 1792	13S	21E	15 & 22	CAMC112875
KAY 16	Bk 1479 Pg 687	13S	21E	31	CAMC105539
KAY 18	Bk 1479 Pg 689	13S	21E	31	CAMC105541
KAY 20	Bk 1479 Pg 691	13S	21E	31	CAMC105543
KAY 22	Bk 1479 Pg 693	13S	21E	31	CAMC105545
KAY 24	Bk 1479 Pg 695	13S	21E	31	CAMC105547
KAY 26	Bk 1479 Pg 697	13S	21E	31	CAMC105549
KAY 27	Bk 1479 Pg 698	14S	20E & 21E	1 & 6	CAMC105550
KAY 28	Bk 1479 Pg 699	14S	21E	6	CAMC105551
KAY 29	Bk 1479 Pg 700	14S	20E & 21E	1 & 6	CAMC105552
KAY 30	Bk 1479 Pg 701	14S	21E	6	CAMC105553
KAY 31	Bk 1479 Pg 702	14S	20E & 21E	1 & 6	CAMC105554
KAY 32	Bk 1479 Pg 703	14S	21E	6	CAMC105555
KAY 33	Bk 1479 Pg 704	14S	20E & 21E	1 & 6	CAMC105556
KAY 35	Bk 1479 Pg 703	14S	20E & 21E	1 & 6	CAMC105558
KAY 56	Bk 1479 Pg 727	13S	21E	32	CAMC105579
KAY 57	Bk 1479 Pg 728	13S	21E	31	CAMC105580
KAY 58	Bk 1479 Pg 729	13S	21E	32	CAMC105581
KAY 59	Bk 1479 Pg 730	13S	21E	31	CAMC105582
KAY 89	Bk 1479 Pg 760	13S	21E	29	CAMC105612
KAY 97	Bk 1479 Pg 768	13S	21E	29 & 32	CAMC105620
KAY 98	Bk 1479 Pg 769	13S	21E	29 & 32	CAMC105621
KAY 99	Bk 1479 Pg 770	13S	21E	32	CAMC105622
KAY 100	Bk 1479 Pg 771	13S	21E	32	CAMC105623
KAY 101	Bk 1479 Pg 772	13S	21E	32	CAMC105624
KAY 102	Bk 1479 Pg 773 Amended Bk 1855 Pg 1259	13S	21E	32	CAMC105625
KAY 106	Bk 1479 Pg 777 Amended Bk 1855 Pg 1262	13S	21E	32	CAMC105629
KAY 129	Bk 1479 Pg 800	13S	21E	28, 29, 32, & 33	CAMC105652
KAY 130	Bk 1479 Pg 801	13S	21E	32 & 33	CAMC105653
KAY 131	Bk 1479 Pg 802	13S	21E	32 & 33	CAMC105654
KAY 132	Bk 1479 Pg 803	13S	21E	32 & 33	CAMC105655
KAY 133	Bk 1479 Pg 804 Amended Bk 1855	13s	21E	32 & 33	CAMC105656

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
	Pg 1270				
GAV 20	Bk 1509 Pg 49	13S	21E	16	CAMC133403
GAV 21	Bk 1509 Pg 50	13S	21E	16	CAMC133404
GAV 22	Bk 1509 Pg 51	13S	21E	16	CAMC133405
GAV 23	Bk 1509 Pg 52	13S	21E	16	CAMC133406
GAV 24	Bk 1509 Pg 53	13S	21E	15 & 16	CAMC133407
GAV 25	Bk 1509 Pg 54	13S	21E	15	CAMC133408
GAV 26	Bk 1509 Pg 55	13S	21E	15	CAMC133409
GAV 27	Bk 1509 Pg 56	13S	21E	15	CAMC133410
GAV 28	Bk 1509 Pg 57	13S	21E	15	CAMC133411
GAV 43	Bk 1509 Pg 72	13S	21E	16	CAMC133426
GAV 45	Bk 1509 Pg 74	13S	21E	16	CAMC133428
GAV 47	Bk 1509 Pg 76	13S	21E	16	CAMC133430
GAV 48	Bk 1509 Pg 77	13S	21E	16 & 21	CAMC133431
GAV 49	Bk 1509 Pg 78	13S	21E	16	CAMC133432
GAV 50	Bk 1509 Pg 79	13S	21E	16 & 21	CAMC133433
GAV 51	Bk 1509 Pg 80	13S	21E	16	CAMC133434
GAV 52	Bk 1509 Pg 81	13S	21E	16 & 21	CAMC133435
GAV 53	Bk 1509 Pg 82	13S	21E	16	CAMC133436
GAV 54	Bk 1509 Pg 83	13S	21E	16 & 21	CAMC133437
GAV 55	Bk 1509 Pg 84	13S	21E	15 & 16	CAMC133438
GAV 56	Bk 1509 Pg 85	13S	21E	15, 16, & 21	CAMC133439
GAV 57	Bk 1509 Pg 86	13S	21E	15	CAMC133440
GAV 58	Bk 1509 Pg 87	13S	21E	15 & 22	CAMC133441
GAV 59	Bk 1509 Pg 88	13S	21E	15	CAMC133442
GAV 60	Bk 1509 Pg 89	13S	21E	15 & 22	CAMC133443
GAV 61	Bk 1509 Pg 90	13S	21E	15	CAMC133444
GAV 62	Bk 1509 Pg 91	13S	21E	15 & 22	CAMC133445
GAV 63	Bk 1509 Pg 92	13S	21E	15	CAMC133446
GAV 64	Bk 1509 Pg 93	13S	21E	15 & 22	CAMC133447
GAV 81	Bk 1509 Pg 110	13S	21E	21	CAMC133464
GAV 82	Bk 1509 Pg 111	13S	21E	21	CAMC133465
GAV 83	Bk 1509 Pg 112	13S	21E	21	CAMC133466
GAV 84	Bk 1509 Pg 113	13S	21E	21	CAMC133467
GAV 85	Bk 1509 Pg 114	13S	21E	21	CAMC133468
GAV 87	Bk 1509 Pg 116	13S	21E	21	CAMC133470
GAV 89	Bk 1509 Pg 118	13S	21E	21	CAMC133472
GAV 91	Bk 1509 Pg 120	13S	21E	21 & 22	CAMC133474
GAV 93	Bk 1509 Pg 122	13S	21E	22	CAMC133476
GAV 95	Bk 1509 Pg 124	13S	21E	22	CAMC133478
GAV 97	Bk 1509 Pg 126	13S	21E	22	CAMC133480
GAV 99	Bk 1509 Pg 128	13S	21E	22	CAMC133482
SWL 316	Bk 1510 Pg 1337	14S	21E	8 & 9	CAMC135612
SWL 318	Bk 1510 Pg 1339	14S	21E	8 & 9	CAMC135614

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
SWL 320	Bk 1510 Pg 1341	14S	21E	4, 5, 8, & 9	CAMC135616
SWL 322	Bk 1510 Pg 1343	14S	21E	4 & 5	CAMC135618
SWL 323	Bk 1510 Pg 1344	14S	21E	4	CAMC135619
SWL 324	Bk 1510 Pg 1345	14S	21E	4 & 5	CAMC135620
SWL 325	Bk 1510 Pg 1346	14S	21E	4	CAMC135621
SWL 327	Bk 1510 Pg 1348	14S	21E	4	CAMC135623
SWL 329	Bk 1510 Pg 1350	14S	21E	4	CAMC135625
SWL 331	Bk 1510 Pg 1352	14S	21E	4	CAMC135627
SWL 333	Bk 1510 Pg 1354	14S	21E	4	CAMC135629
SWL 335	Bk 1510 Pg 1356	14S	21E	4	CAMC135631
SWL 337	Bk 1510 Pg 1358	13S & 14S	21E	33 & 4	CAMC135633
SWL 339	Bk 1510 Pg 1360	13S	21E	33	CAMC135635
SWL 341	Bk 1510 Pg 1362	13S	21E	33	CAMC135637
SWL 343	Bk 1510 Pg 1364	13S	21E	33	CAMC135639
SWL 344	Bk 1510 Pg 1365	13S	21E	33	CAMC135640
SWL 345	Bk 1510 Pg 1366	13S	21E	33	CAMC135641
SWL 346	Bk 1510 Pg 1367	13S	21E	33	CAMC135642
SWL 347	Bk 1510 Pg 1368	13S	21E	33	CAMC135643
SWL 348	Bk 1510 Pg 1369	13S	21E	33	CAMC135644
SWL 349	Bk 1510 Pg 1370	13S	21E	33	CAMC135645
SWL 350	Bk 1510 Pg 1371	13S	21E	33	CAMC135646
SWL 351	Bk 1510 Pg 1372	13S	21E	33	CAMC135647
SWL 352	Bk 1510 Pg 1373	13S	21E	33	CAMC135648
SWL 353	Bk 1510 Pg 1374	13S	21E	28 & 33	CAMC135649
SWL 354	Bk 1510 Pg 1375	13S	21E	28 & 33	CAMC135650
SWL 370	Bk 1510 Pg 1391	13S	21E	28	CAMC135666
SWL 372	Bk 1510 Pg 1393	13S	21E	21 & 28	CAMC135668
SWL 374	Bk 1510 Pg 1395	13S	32E	21	CAMC135670
SWL 382	Bk 1510 Pg 1403	14S	21E	8	CAMC135678
SWL 383	Bk 1510 Pg 1404	14S	21E	8	CAMC135679
SWL 384	Bk 1510 Pg 1405	14S	21E	8	CAMC135680
SWL 385	Bk 1510 Pg 1405	14S	21E	8	CAMC135681
SWL 387	Bk 1510 Pg 1408	14S	21E	5 & 8	CAMC135683
SWL 407	Bk 1512 Pg 564	14S	21E	6 & 7	CAMC137648
SWL 414	Bk 1512 Pg 571	14S	21E	6 & 7	CAMC137655
SWL 415	Bk 1512 Pg 572	14S	20E	1 & 12	CAMC137656
SWL 416	Bk 1512 Pg 573	14S	21E	6	CAMC137657
SWL 417	Bk 1512 Pg 574	14S	20E	1	CAMC137658
SWL 419	Bk 1512 Pg 576	14S	20E	1	CAMC137660
SWL 420	Bk 1512 Pg 577	14S	21E	6	CAMC137661
SWL 421	Bk 1512 Pg 578	14S	20E	1	CAMC137662
SWL 423	Bk 1512 Pg 580	14S	20E	1	CAMC137664
SWL 425	Bk 1512 Pg 582	14S	20E	1	CAMC137666
SWL 427	Bk 1512 Pg 584	14S	20E	1	CAMC137668
SWL 428	Bk 1512 Pg 585	13S	21E	33	CAMC137669

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
SWL 430	Bk 1512 Pg 587	13S	21E	33	CAMC137671
SWL 450	Bk 1512 Pg 607	13S	21E	28 & 33	CAMC137691
SWL 453	Bk 1512 Pg 610	13S	21E	28	CAMC137694
SWL 455	Bk 1512 Pg 612	13S	21E	21	CAMC137696
SWL 456	Bk 1512 Pg 613	13S	21E	21	CAMC137697
SWL 906	Bk 1513 Pg 126	14S	20E	3	CAMC138444
SWL 908	Bk 1513 Pg 128	14S	20E	3 & 10	CAMC138446
CJ 93	Bk 1520 Pg 1171	14S	20E	1 & 12	CAMC148160
CJ 94	Bk 1520 Pg 1172	14S	20E & 21E	1, 12, 6, & 7	CAMC148161
CJ 95	Bk 1520 Pg 1173	14S	20E	12	CAMC148162
CJ 96	Bk 1520 Pg 1174	14S	20E & 21E	12 & 7	CAMC148163
CJ 97	Bk 1520 Pg 1175	14S	20E	12	CAMC148164
CJ 98	Bk 1520 Pg 1176	14S	20E & 21E	12 & 7	CAMC148165
CJ 99	Bk 1520 Pg 1177	14S	20E	12	CAMC148166
CJ 100	Bk 1520 Pg 1178	14S	20E & 21E	12 & 7	CAMC148167
CJ 101	Bk 1520 Pg 1179	14S	20E	12	CAMC148168
CJ 102	Bk 1520 Pg 1180	14S	20E & 21E	12 & 7	CAMC148169
CJ 160	Bk 1520 Pg 1238	14S	21E	6 & 7	CAMC148227
CJ 162	Bk 1520 Pg 1240	14S	21E	7	CAMC148229
CJ 163	Bk 1520 Pg 1241	14S	21E	7	CAMC148230
CJ 164	Bk 1520 Pg 1242	14S	21E	7	CAMC148231
CJ 165	Bk 1520 Pg 1243	14S	21E	7	CAMC148232
CJ 166	Bk 1520 Pg 1244	14S	21E	7	CAMC148233
CJ 167	Bk 1520 Pg 1245	14S	21E	7	CAMC148234
CJ 168	Bk 1520 Pg 1246	14S	21E	7	CAMC148235
CJ 169	Bk 1520 Pg 1247	14S	21E	7	CAMC148236
CJ 238	Bk 1520 Pg 1308	14S	21E	7	CAMC148297
CJ 240	Bk 1520 Pg 1310	14S	21E	7	CAMC148299
CJ 241	Bk 1520 Pg 1311	14S	21E	7 & 8	CAMC148300
CJ 302	Bk 1520 Pg 1372	14S	21E	8	CAMC148361
CJ 303	Bk 1520 Pg 1373	14S	21E	8	CAMC148362
CJ 304	Bk 1520 Pg 1374	14S	21E	8	CAMC148363
CJ 305	Bk 1520 Pg 1375	14S	21E	8	CAMC148364
DJP 1	Bk 1829 Pg 229	14S	20E	12	CAMC266932
DJP 2	Bk 1829 Pg 230	14S	20E	12	CAMC266933
DJP 5	Bk 1829 Pg 233	14S	20E & 21E	12 & 7	CAMC266936
DJP 6	Bk 1829 Pg 234	14S	20E & 21E	12 & 7	CAMC266937
DJP 9	Bk 1829 Pg 237	14S	21E	7	CAMC266940
DJP 10	Bk 1829 Pg 238	14S	21E	7	CAMC266941
DJP 13	Bk 1829 Pg 241	14S	21E	7	CAMC266944
DJP 14	Bk 1829 Pg 242	14S	21E	7	CAMC266945
DJP 17	Bk 1829 Pg 245	14S	21E	7	CAMC266948
DJP 18	Bk 1829 Pg 246	14S	21E	7	CAMC266949
DJP 21	Bk 1829 Pg 249	14S	21E	7 & 8	CAMC266952

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
DJP 22	Bk 1829 Pg 250	14S	21E	7 & 8	CAMC266953
DJP 25	Bk 1829 Pg 253	14S	21E	8	CAMC266956
DJP 26	Bk 1829 Pg 254	14S	21E	8	CAMC266957
DJP 29	Bk 1829 Pg 257	14S	21E	8	CAMC266960
DJP 30	Bk 1829 Pg 258	14S	21E	8	CAMC266961
DJP 33	Bk 1829 Pg 261	14S	20E	21 & 22	CAMC266964
DJP 34	Bk 1829 Pg 262	14S	20E	15 & 22	CAMC266965
DJP 35	Bk 1829 Pg 263	14S	20E	15 & 22	CAMC266966
DJP 36	Bk 1829 Pg 264	14S	20E	15	CAMC266967
DJP 37	Bk 1829 Pg 265	14S	20E	14 & 15	CAMC266968
DJP 38	Bk 1829 Pg 266	14S	20E	14	CAMC266969
DJP 39	Bk 1829 Pg 267	14S	20E	14	CAMC266970
DJP 40	Bk 1829 Pg 268	14S	20E	14	CAMC266971
DJP 41	Bk 1829 Pg 269	14S	20E	11 & 14	CAMC266972
DJP 42	Bk 1829 Pg 270	14S	20E	11	CAMC266973
DJP 43	Bk 1829 Pg 271	14S	20E	11 & 12	CAMC266974
DJP 44	Bk 1829 Pg 272	14S	20E	12	CAMC266975
DJP 45	Bk 1829 Pg 273	14S	20E	12	CAMC266976
DJP 46	Bk 1829 Pg 274	14S	20E	11	CAMC266977
IMP 5	Bk 1855 Pg 1254	13S	21E	32	CAMC269532
BB 1	Bk 1927 Pg 1534	13S	21E	31	CAMC273771
BB 2	Bk 1927 Pg 1535	13S	21E	31	CAMC273772
BB 9	Bk 1927 Pg 1542 Amended Bk 1946 Pg 871	13S	21E	32	CAMC273779
BB 12	Bk 1927 Pg 1545	13S	21E	32	CAMC273782
BB 13	Bk 1927 Pg 1546	13S	21E	32	CAMC273783
BB 14	Bk 1927 Pg 1547	13S	21E	32	CAMC273784
BB 15	Bk 1927 Pg 1548	13S	21E	32	CAMC273785
BB 16	Bk 1927 Pg 1549	13S	21E	31	CAMC273786
BB 17	Bk 1927 Pg 1550	13S	21E	31	CAMC273787
BB 26	Bk 1927 Pg 1559 Amended Bk 1946 Pg 872	13S	21E	32	CAMC273796
BB 29	Bk 1927 Pg 1562	13S	21E	32	CAMC273799
BB 30	Bk 1927 Pg 1563	13S	21E	32	CAMC273800
BB 31	Bk 1927 Pg 1564	13S	21E	31	CAMC273801
BB 32	Bk 1927 Pg 1565	13S	21E	31	CAMC273802
BB 36	Bk 1927 Pg 1569	13S	21E	31	CAMC273806
BB 37	Bk 1927 Pg 1570	13S	21E	31	CAMC273807
BB 38	Bk 1927 Pg 1571	13S	21E	31	CAMC273808
BB 39	Bk 1927 Pg 1572	13S	21E	31	CAMC273809
BB 40	Bk 1927 Pg 1573	13S	21E	31	CAMC273810
BB 41	Bk 1927 Pg 1574	14S	21E	6	CAMC273811
BB 43	Bk 1927 Pg 1576	13S	21E	32	CAMC273813
BB 44	Bk 1927 Pg 1577	13S	21E	32	CAMC273814

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
BB 45	Bk 1927 Pg 1578	13S	21E	33	CAMC273815
BB 46	Bk 1927 Pg 1579	13S	21E	33	CAMC273816
BB 47	Bk 1927 Pg 1580 Amended Bk 1932 Pg 680	13S	21E	33	CAMC273817
BB 50	Bk 1927 Pg 1583	13S	21E	32	CAMC273820
BB 51	Bk 1927 Pg 1584	13S	21E	33	CAMC273821
BB 52	Bk 1927 Pg 1585	13S	21E	33	CAMC273822
BB 56	Bk 1927 Pg 1589	13S	21E	33	CAMC273826
BB 57	Bk 1927 Pg 1590	13S	21E	33	CAMC273827
BB 58	Bk 1927 Pg 1591	13S	21E	33	CAMC273828
BB 59	Bk 1927 Pg 1592	13S	21E	33	CAMC273829
BB 60	Bk 1927 Pg 1593 Amended Bk 1946 Pg 873	13S	21E	32	CAMC273830
BB 61	Bk 1927 Pg 1594 Amended Pg 1946 Pg 874	13S	21E	32	CAMC273831
BB 62	Bk 1927 Pg 1595	13S	21E	32	CAMC273832
BB 63	Bk 1927 Pg 1596	13S	21E	33	CAMC273833
BB 64	Bk 1927 Pg 1597	13S	21E	33	CAMC273834
BB 65	Bk 1927 Pg 1598 Amended Bk 1946 Pg 875	13S	21E	33	CAMC273835
BB 66	Bk 1927 Pg 1599 Amended Bk 1946 Pg 876	13S	21E	33	CAMC273836
BB 67	Bk 1927 Pg 1600 Amended Bk 1946 Pg 877	13S	21E	33	CAMC273837
BB 68	Bk 1927 Pg 1601	13S	21E	33	CAMC273838
BB 69	Bk 1927 Pg 1602	13S	21E	33	CAMC273839
BB 71	Bk 1927 Pg 1604	13S	21E	33	CAMC273841
BB 84	Bk 1927 Pg 1617	13S & 14S	21E	33, 4, & 5	CAMC273854
BB 85	Bk 1927 Pg 1618	13S & 14S	21E	33 & 4	CAMC273855
BB 87	Bk 1927 Pg 1620	13S & 14S	21E	33 & 4	CAMC273857
BB 90	Bk 1927 Pg 1623 Amended Bk 1946 Pg 878	14S	21E	4 & 5	CAMC273860
BB 93	Bk 1927 Pg 1626	14S	21E	4	CAMC273863
BB 94	Bk 1927 Pg 1627	14S	21E	6	CAMC273864
BB 95	Bk 1927 Pg 1628	14S	21E	6	CAMC273865
BB 96	Bk 1927 Pg 1629	14S	21E	6	CAMC273866
BB 97	Bk 1927 Pg 1630	14S	21E	6	CAMC273867
BB 98	Bk 1927 Pg 1631	14S	21E	6	CAMC273868
BB 103	Bk 1927 Pg 1636	14S	21E	6	CAMC273873

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
BB 104	Bk 1927 Pg 1637	14S	21E	6	CAMC273874
BB 105	Bk 1927 Pg 1638	14S	21E	6	CAMC273875
BB 106	Bk 1927 Pg 1639	14S	21E	6	CAMC273876
BB 107	Bk 1927 Pg 1640	14S	21E	6	CAMC273877
BB 108	Bk 1927 Pg 1641	14S	21E	6	CAMC273878
BB 109	Bk 1927 Pg 1642	14S	21E	6	CAMC273879
BB 110	Bk 1927 Pg 1643	14S	21E	6	CAMC273880
BB 111	Bk 1927 Pg 1644	14S	21E	6	CAMC273881
BB 112	Bk 1927 Pg 1645	14S	21E	5 & 6	CAMC273882
BB 114	Bk 1927 Pg 1647	14S	21E	6	CAMC273884
BB 115	Bk 1927 Pg 1648	14S	21E	6	CAMC273885
BB 116	Bk 1927 Pg 1649	14S	21E	6	CAMC273886
BB 117	Bk 1927 Pg 1650	14S	21E	6	CAMC273887
BB 118	Bk 1927 Pg 1651	14S	21E	6	CAMC273888
BB 119	Bk 1927 Pg 1652	14S	21E	6	CAMC273889
BB 120	Bk 1927 Pg 1653	14S	21E	6	CAMC273890
BB 121	Bk 1927 Pg 1654	14S	21E	6	CAMC273891
BB 122	Bk 1927 Pg 1655	14S	21E	6	CAMC273892
BB 123	Bk 1927 Pg 1656	14S	21E	6	CAMC273893
BB 124	Bk 1927 Pg 1657	14S	21E	6	CAMC273894
BB 125	Bk 1927 Pg 1658	14S	21E	5 & 6	CAMC273895
BB 126	Bk 1927 Pg 1659	14S	21E	5	CAMC273896
BB 127	Bk 1927 Pg 1660	14S	21E	5	CAMC273897
BB 128	Bk 1927 Pg 1661	14S	21E	5	CAMC273898
BB 129	Bk 1927 Pg 1662	14S	21E	5	CAMC273899
BB 130	Bk 1927 Pg 1663	14S	21E	5	CAMC273900
BB 131	Bk 1927 Pg 1664	14S	21E	5	CAMC273901
BB 133	Bk 1927 Pg 1666	14S	21E	6	CAMC273903
BB 134	Bk 1927 Pg 1667	14S	21E	6	CAMC273904
BB 135	Bk 1927 Pg 1668	14S	21E	6	CAMC273905
BB 137	Bk 1927 Pg 1670	14S	21E	6	CAMC273907
BB 138	Bk 1927 Pg 1671	14S	21E	6	CAMC273908
BB 139	Bk 1927 Pg 1672	14S	21E	6	CAMC273909
BB 140	Bk 1927 Pg 1673	14S	21E	6	CAMC273910
BB 141	Bk 1927 Pg 1674	14S	21E	6	CAMC273911
BB 142	Bk 1927 Pg 1675	14S	21E	6	CAMC273912
BB 143	Bk 1927 Pg 1676	14S	21E	6	CAMC273913
BB 144	Bk 1927 Pg 1677	14S	21E	6	CAMC273914
BB 145	Bk 1927 Pg 1678	14S	21E	6	CAMC273915
BB 146	Bk 1927 Pg 1679	14S	21E	5 & 6	CAMC273916
BB 147	Bk 1927 Pg 1680	14S	21E	5	CAMC273917
BB 148	Bk 1927 Pg 1681	14S	21E	5	CAMC273918
BB 149	Bk 1927 Pg 1682	14S	21E	5	CAMC273919
BB 150	Bk 1927 Pg 1683	14S	21E	5	CAMC273920
BB 151	Bk 1927 Pg 1684	14S	21E	5	CAMC273921

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BB 153	Bk 1927 Pg 1686	14S	21E	5	CAMC273923
BB 158	Bk 1927 Pg 1691	14S	21E	6	CAMC273928
BB 159	Bk 1927 Pg 1692	14S	21E	6	CAMC273929
BB 162	Bk 1927 Pg 1695	14S	21E	6	CAMC273932
BB 163	Bk 1927 Pg 1696	14S	21E	6	CAMC273933
BB 164	Bk 1927 Pg 1697	14S	21E	6	CAMC273934
BB 165	Bk 1927 Pg 1698	14S	21E	6	CAMC273935
BB 166	Bk 1927 Pg 1699	14S	21E	6	CAMC273936
BB 167	Bk 1927 Pg 1700	14S	21E	6	CAMC273937
BB 168	Bk 1927 Pg 1701	14S	21E	6	CAMC273938
BB 169	Bk 1927 Pg 1702	14S	21E	6	CAMC273939
BB 170	Bk 1927 Pg 1703	14S	21E	6	CAMC273940
BB 171	Bk 1927 Pg 1704	14S	21E	5 & 6	CAMC273941
BB 172	Bk 1927 Pg 1705	14S	21E	5	CAMC273942
BB 173	Bk 1927 Pg 1706	14S	21E	5	CAMC273943
BB 174	Bk 1927 Pg 1707	14S	21E	5	CAMC273944
BB 175	Bk 1927 Pg 1708	14S	21E	5	CAMC273945
BB 176	Bk 1927 Pg 1709	14S	21E	5	CAMC273946
BB 177	Bk 1927 Pg 1710	14S	21E	5	CAMC273947
BB 178	Bk 1927 Pg 1711	14S	21E	5	CAMC273948
BB 179	Bk 1927 Pg 1712	14S	21E	5	CAMC273949
BB 180	Bk 1927 Pg 1713	14S	21E	5	CAMC273950
BB 181	Bk 1927 Pg 1714	14S	21E	5	CAMC273951
BB 191	Bk 1927 Pg 1724	14S	21E	4	CAMC273961
BB 195	Bk 1927 Pg 1728	14S	21E	6	CAMC273965
BB 196	Bk 1927 Pg 1729	14S	21E	6	CAMC273966
BB 197	Bk 1927 Pg 1730	14S	21E	6	CAMC273967
BB 198	Bk 1927 Pg 1731	14S	21E	6	CAMC273968
BB 199	Bk 1927 Pg 1732	14S	21E	6	CAMC273969
BB 200	Bk 1927 Pg 1733 Amended Bk 1946 Pg 879	14S	21E	6	CAMC273970
BB 202	Bk 1927 Pg 1735 Amended Bk 1946 Pg 880	14S	21E	6	CAMC273972
BB 204	Bk 1927 Pg 1737	14S	21E	5 & 6	CAMC273974
BB 205	Bk 1927 Pg 1738	14S	21E	5	CAMC273975
BB 206	Bk 1927 Pg 1739	14S	21E	5	CAMC273976
BB 207	Bk 1927 Pg 1740	14S	21E	5	CAMC273977
BB 208	Bk 1927 Pg 1741	14S	21E	5	CAMC273978
BB 209	Bk 1927 Pg 1742	14S	21E	5	CAMC273979
BB 210	Bk 1927 Pg 1743	14S	21E	5	CAMC273980
BB 211	Bk 1927 Pg 1744	14S	21E	5	CAMC273981
BB 212	Bk 1927 Pg 1745	14S	21E	5	CAMC273982
BB 213	Bk 1927 Pg 1746	14S	21E	5	CAMC273983

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BB 216	Bk 1927 Pg 1749	14S	21E	5	CAMC273986
BB 217	Bk 1927 Pg 1750	14S	21E	5	CAMC273987
BB 218	Bk 1927 Pg 1751 Amended Bk 1946 Pg 881	14S	21E	5	CAMC273988
BB 221	Bk 1927 Pg 1754 Amended Bk 1946 Pg 882	14S	21E	4 & 5	CAMC273991
BB 223	Bk 1927 Pg 1756	14S	21E	4	CAMC273993
BB 224	Bk 1927 Pg 1757	14S	21E	6	CAMC273994
BB 225	Bk 1927 Pg 1758	14S	21E	6	CAMC273995
BB 226	Bk 1927 Pg 1759	14S	21E	6	CAMC273996
BB 227	Bk 1927 Pg 1760 Amended Bk 1946 Pg 883	14S	21E	6	CAMC273997
BB 228	Bk 1927 Pg 1761	14S	21E	6	CAMC273998
BB 231	Bk 1927 Pg 1764	14S	21E	6	CAMC274001
BB 232	Bk 1927 Pg 1765	14S	21E	6	CAMC274002
BB 233	Bk 1927 Pg 1766	14S	21E	6	CAMC274003
BB 234	Bk 1927 Pg 1767	14S	21E	6	CAMC274004
BB 235	Bk 1927 Pg 1768	14S	21E	6	CAMC274005
BB 236	Bk 1927 Pg 1769 Amended Pg 1946 Pg 884	14S	21E	6	CAMC274006
BB 237	Bk 1927 Pg 1770 Amended Bk 1954 Pg 848	14S	21E	6	CAMC274007
BB 240	Bk 1927 Pg 1773	14S	21E	5 & 6	CAMC274010
BB 241	Bk 1927 Pg 1774	14S	21E	5	CAMC274011
BB 242	Bk 1927 Pg 1775	14S	21E	5	CAMC274012
BB 243	Bk 1927 Pg 1776	14S	21E	5	CAMC274013
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BB 245	Bk 1927 Pg 1778	14S	21E	5	CAMC274015
BB 246	Bk 1927 Pg 1779	14S	21E	5	CAMC274016
BB 247	Bk 1927 Pg 1780	14S	21E	5	CAMC274017
BB 248	Bk 1927 Pg 1781	14S	21E	5	CAMC274018
BB 249	Bk 1927 Pg 1782	14S	21E	5	CAMC274019
BB 250	Bk 1927 Pg 1783	14S	21E	5	CAMC274020
BB 251	Bk 1927 Pg 1784	14S	21E	5	CAMC274021
BB 252	Bk 1927 Pg 1785	14S	21E	5	CAMC274022
BB 253	Bk 1927 Pg 1786	14S	21E	5	CAMC274023
BB 254	Bk 1927 Pg 1787	14S	21E	5	CAMC274024
BB 255	Bk 1927 Pg 1788	14S	21E	5	CAMC274025
BB 256	Bk 1927 Pg 1789	14S	21E	6	CAMC274026

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BB 259	Bk 1927 Pg 1792	14S	21E	6	CAMC274029
BB 260	Bk 1927 Pg 1793	14S	21E	6	CAMC274030
BB 261	Bk 1927 Pg 1794	14S	21E	6	CAMC274031
BB 262	Bk 1927 Pg 1795	14S	21E	6	CAMC274032
BB 263	Bk 1927 Pg 1796	14S	21E	6	CAMC274033
BB 264	Bk 1927 Pg 1797	14S	21E	6	CAMC274034
BB 265	Bk 1927 Pg 1798	14S	21E	5 & 6	CAMC274035
BB 266	Bk 1927 Pg 1799	14S	21E	5	CAMC274036
BB 267	Bk 1927 Pg 1800	14S	21E	5	CAMC274037
BB 268	Bk 1928 Pg 1	14S	21E	5	CAMC274038
BB 269	Bk 1928 Pg 2	14S	21E	5	CAMC274039
BB 270	Bk 1928 Pg 3	14S	21E	5	CAMC274040
BB 271	Bk 1928 Pg 4	14S	21E	5	CAMC274041
BB 272	Bk 1928 Pg 5	14S	21E	5	CAMC274042
BB 273	Bk 1928 Pg 6	14S	21E	5	CAMC274043
BB 274	Bk 1928 Pg 7	14S	21E	5	CAMC274044
BB 275	Bk 1928 Pg 8	14S	21E	5	CAMC274045
BB 276	Bk 1928 Pg 9	14S	21E	5	CAMC274046
BB 277	Bk 1928 Pg 10	14S	21E	5	CAMC274047
BB 278	Bk 1928 Pg 11	14S	21E	6 & 7	CAMC274048
BB 279	Bk 1928 Pg 12	14S	21E	6 & 7	CAMC274049
BB 280	Bk 1928 Pg 13	14S	21E	6 & 7	CAMC274050
BB 281	Bk 1928 Pg 14	14S	21E	6 & 7	CAMC274051
BB 283	Bk 1928 Pg 16	14S	21E	6 & 7	CAMC274053
BB 284	Bk 1928 Pg 17	14S	21E	6 & 7	CAMC274054
BB 285	Bk 1928 Pg 18	14S	21E	6 & 7	CAMC274055
BB 286	Bk 1928 Pg 19	14S	21E	6 & 7	CAMC274056
BB 287	Bk 1928 Pg 20	14S	21E	6 & 7	CAMC274057
BB 288	Bk 1928 Pg 21	14S	21E	6 & 7	CAMC274058
BB 289	Bk 1928 Pg 22	14S	21E	6 & 7	CAMC274059
BB 290	Bk 1928 Pg 23	14S	21E	5, 6, 7, & 8	CAMC274060
BB 291	Bk 1928 Pg 24	14S	21E	5 & 8	CAMC274061
BB 292	Bk 1928 Pg 25	14S	21E	5 & 8	CAMC274062
BB 293	Bk 1928 Pg 26	14S	21E	5 & 8	CAMC274063
BB 294	Bk 1928 Pg 27	14S	21E	5 & 8	CAMC274064
BB 295	Bk 1928 Pg 28	14S	21E	5 & 8	CAMC274065
BB 296	Bk 1928 Pg 29	14S	21E	5 & 8	CAMC274066
BB 297	Bk 1928 Pg 30	14S	21E	8	CAMC274067
BB 298	Bk 1928 Pg 31	14S	21E	8	CAMC274068
BB 299	Bk 1928 Pg 32	14S	21E	8	CAMC274069
BB 300	Bk 1928 Pg 33	14S	21E	8	CAMC274070
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BB 306	Bk 1928 Pg 39	14S	21E	7	CAMC274076
BB 307	Bk 1928 Pg 40	14S	21E	7	CAMC274077
BB 308	Bk 1928 Pg 41	14S	21E	7 & 8	CAMC274078
BB 309	Bk 1928 Pg 42	14S	21E	8	CAMC274079
BB 310	Bk 1928 Pg 43	14S	21E	8	CAMC274080
BB 311	Bk 1928 Pg 44	14S	21E	8	CAMC274081
BB 312	Bk 1928 Pg 45	14S	21E	8	CAMC274082
BB 313	Bk 1928 Pg 46	14S	21E	8	CAMC274083
BB 314	Bk 1928 Pg 47 Amended Bk 1947 Pg 215	14S	21E	8	CAMC274084
BB 315	Bk 1928 Pg 48	14S	21E	7	CAMC274085
BB 316	Bk 1928 Pg 49	14S	21E	7	CAMC274086
BB 317	Bk 1928 Pg 50	14S	21E	7	CAMC274087
BB 318	Bk 1928 Pg 51	14S	21E	7	CAMC274088
BB 319	Bk 1928 Pg 52 Amended Bk 1946 Pg 887	14S	21E	7	CAMC274089
BB 321	Bk 1928 Pg 54	14S	21E	7 & 8	CAMC274091
BB 322	Bk 1928 Pg 55	14S	21E	8	CAMC274092
BB 324	Bk 1928 Pg 57 Amended Bk 1946 Pg 888	14S	21E	8	CAMC274094
BB 325	Bk 1928 Pg 58 Amended Bk 1946 Pg 889	14S	21E	8	CAMC274095
BB 328	Bk 1928 Pg 61	14S	21E	7	CAMC274098
BB 329	Bk 1928 Pg 62	14S	21E	7 & 8	CAMC274099
BB 330	Bk 1928 Pg 63	14S	21E	8	CAMC274100
BB 335	Bk 1928 Pg 68	14S	20E	15 & 22	CAMC274105
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BB 337	Bk 1928 Pg 70	14S	20E	14	CAMC274107
BB 338	Bk 1928 Pg 71	14S	20E	14	CAMC274108
BB 340	Bk 1946 Pg 891	13S	21E	32	CAMC274465
BB 341	Bk 1946 Pg 892	13S	21E	32	CAMC274466
BB 342	Bk 1946 Pg 893	13S	21E	32	CAMC274467
BB 343	Bk 1946 Pg 894	13S & 14S	21E	32 & 5	CAMC274468
BB 344	Bk 1946 Pg 895	14S	21E	5	CAMC274469
BB 345	Bk 1946 Pg 896	14S	21E	5	CAMC274470
BB 346	Bk 1946 Pg 897	14S	21E	5	CAMC274471
BB 347	Bk 1946 Pg 898	14S	21E	5	CAMC274472
BB 348	Bk 1946 Pg 899	13S	21E	31 & 32	CAMC274473

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BB 351	Bk 1946 Pg 902	13S	21E	31	CAMC274476
BB 352	Bk 1946 Pg 903	13S	21E	31	CAMC274477
BB 353	Bk 1946 Pg 904	13S	21E	31 & 32	CAMC274478
BB 354	Bk 1946 Pg 905	13S	21E	32	CAMC274479
BB 355	Bk 1946 Pg 906	13S	21E	32	CAMC274480
BB 356	Bk 1947 Pg 219	14S	21E	7	CAMC274481
BB 357	Bk 1946 Pg 907	13S	21E	33	CAMC274482
BB 358	Bk 1946 Pg 908	13S & 14S	21E	33 & 4	CAMC274483
BB 359	Bk 1946 Pg 909	14S	21E	4 & 5	CAMC274484
BB 360	Bk 1946 Pg 910	14S	21E	5	CAMC274485
BB 361	Bk 1946 Pg 911	14S	21E	5	CAMC274486
BB 362	Bk 1946 Pg 912	14S	21E	5	CAMC274487
BB 363	Bk 1946 Pg 913	14S	21E	5	CAMC274488
BB 364	Bk 1946 Pg 914	14S	21E	6	CAMC274489
BB 365	Bk 1946 Pg 915	14S	21E	6	CAMC274490
BB 366	Bk 1946 Pg 916	14S	21E	6	CAMC274491
BB 367	Bk 1946 Pg 917	14S	21E	6	CAMC274492
BB 368	Bk 1946 Pg 918	14S	21E	6	CAMC274493
BB 369	Bk 1946 Pg 919 Amended Bk 1974 Pg 1106	14S	21E	6	CAMC274494
BB 370	Bk 1946 Pg 920	14S	21E	6	CAMC274495
BB 371	Bk 1946 Pg 921	14S	21E	4 & 5	CAMC274496
UYA 1	Bk 1946 Pg 922	14S	21E	4 & 5	CAMC274497
UYA 2	Bk 1946 Pg 923	14S	21E	4 & 5	CAMC274498
UYA 3	Bk 1946 Pg 924	14S	21E	4 & 5	CAMC274499
UYA 4	Bk 1946 Pg 925	14S	21E	4 & 5	CAMC274500
UYA 5	Bk 1947 Pg 216	14S	21E	5	CAMC274501
UYA 6	Bk 1946 Pg 926	14S	21E	5	CAMC274502
UYA 7	Bk 1946 Pg 927	14S	21E	5	CAMC274503
UYA 8	Bk 1946 Pg 928	14S	21E	5	CAMC274504
UYA 9	Bk 1946 Pg 929	14S	21E	5	CAMC274505
UYA 10	Bk 1946 Pg 930	14S	21E	5	CAMC274506
UYA 11	Bk 1946 Pg 931	14S	21E	5	CAMC274507
UYA 12	Bk 1946 Pg 932	14S	21E	5	CAMC274508
UYA 13	Bk 1946 Pg 933	14S	21E	5	CAMC274509
UYA 14	Bk 1946 Pg 934	14S	21E	5	CAMC274510
UYA 15	Bk 1946 Pg 935	13S & 14S	21E	33 & 5	CAMC274511
UYA 16	Bk 1946 Pg 936	13S & 14S	21E	33 & 5	CAMC274512
UYA 17	Bk 1946 Pg 937	14S	21E	5	CAMC274513
UYA 18	Bk 1946 Pg 938	14S	21E	5	CAMC274514
UYA 19	Bk 1946 pg 939	14S	21E	5	CAMC274515
UYA 20	Bk 1946 Pg 940	14S	21E	5	CAMC274516
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UYA 25	Bk 1946 Pg 945	14S	21E	5	CAMC274522
UYA 26	Bk 1946 Pg 946	14S	21E	5	CAMC274523
UYA 27	Bk 1946 Pg 947	14S	21E	5	CAMC274523
UYA 28	Bk 1946 Pg 948	13S & 14S	21E	33 & 5	CAMC274524
UYA 29	Bk 1946 Pg 949	14S	21E	5	CAMC274525
UYA 30	Bk 1946 Pg 950	14S	21E	5	CAMC274526
UYA 31	Bk 1946 Pg 951	14S	21E	5	CAMC274527
UYA 32	Bk 1946 Pg 952	14S	21E	5	CAMC274528
UYA 33	Bk 1946 Pg 953	14S	21E	5	CAMC274529
UYA 34	Bk 1946 Pg 954	14S	21E	5	CAMC274530
UYA 35	Bk 1946 Pg 955	14S	21E	5	CAMC274531
UYA 36	Bk 1946 Pg 956	14S	21E	5	CAMC274532
UYA 37	Bk 1946 Pg 957	14S	21E	5	CAMC274533
UYA 38	Bk 1946 Pg 958	14S	21E	5	CAMC274534
UYA 39	Bk 1946 Pg 959	14S	21E	5	CAMC274535
UYA 40	Bk 1946 Pg 960	14S	21E	5	CAMC274536
UYA 41	Bk 1946 Pg 961	14S	21E	5	CAMC274537
UYA 42	Bk 1946 Pg 962	14S	21E	5	CAMC274538
UYA 43	Bk 1946 Pg 963	13S & 14S	21E	33 & 5	CAMC274539
UYA 44	Bk 1946 Pg 964	14S	21E	5	CAMC274540
UYA 45	Bk 1946 Pg 965	14S	21E	5	CAMC274541
UYA 46	Bk 1946 Pg 966	14S	21E	5	CAMC274542
UYA 47	Bk 1946 Pg 967	13S & 14S	21E	32, 33, & 5	CAMC274543
UYA 48	Bk 1946 Pg 968	14S	21E	5	CAMC274544
UYA 49	Bk 1946 Pg 969	14S	21E	5	CAMC274545
UYA 50	Bk 1946 Pg 970	14S	21E	5	CAMC274546
UYA 51	Bk 1946 Pg 971	14S	21E	5	CAMC274547
UYA 52	Bk 1946 Pg 972	14S	21E	5	CAMC274548
UYA 53	Bk 1946 Pg 973	13S & 14S	21E	32 & 5	CAMC274549
UYA 54	Bk 1946 Pg 974	13S & 14S	21E	32 & 5	CAMC274550
UYA 55	Bk 1946 Pg 975	14S	21E	5	CAMC274551
UYA 56	Bk 1946 Pg 976	14S	21E	5	CAMC274552
UYA 57	Bk 1946 Pg 977	14S	21E	5	CAMC274553
UYA 58	Bk 1946 Pg 978	14S	21E	5	CAMC274554
UYA 59	Bk 1946 Pg 979	14S	21E	5	CAMC274555
UYA 60	Bk 1946 Pg 980	14S	21E	5	CAMC274556
UYA 61	Bk 1946 Pg 981	14S	21E	5	CAMC274557
UYA 62	Bk 1946 Pg 982	14S	21E	5	CAMC274558
UYA 63	Bk 1946 Pg 983	13S & 14S	21E	32 & 5	CAMC274559
UYA 64	Bk 1946 Pg 984	14S	21E	5	CAMC274560
UYA 65	Bk 1946 Pg 985	14S	21E	5	CAMC274561
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UYA 69	Bk 1946 Pg 989	14S	21E	5	CAMC274565
UYA 70	Bk 1946 Pg 990	14S	21E	5	CAMC274566
UYA 71	Bk 1946 Pg 991 Amended Bk 1949 Pg 797	14S	21E	5	CAMC274567
UYA 72	Bk 1946 Pg 992	14S	21E	5	CAMC274568
UYA 74	Bk 1946 Pg 994	14S	21E	5	CAMC274570
UYA 75	Bk 1946 Pg 995	14S	21E	5	CAMC274571
UYA 76	Bk 1946 Pg 996	14S	21E	5	CAMC274572
UYA 77	Bk 1946 Pg 997	13S & 14S	21E	32 & 5	CAMC274573
UYA 78	Bk 1946 Pg 998	14S	21E	5	CAMC274574
UYA 79	Bk 1946 Pg 999	14S	21E	5	CAMC274575
UYA 80	Bk 1946 Pg 1000	14S	21E	5	CAMC274576
UYA 81	Bk 1946 Pg 1001	14S	21E	5	CAMC274577
UYA 82	Bk 1946 Pg 1002	13S & 14S	21E	32 & 5	CAMC274578
UYA 83	Bk 1946 Pg 1003	14S	21E	5	CAMC274579
UYA 84	Bk 1946 Pg 1004	13S & 14S	21E	32 & 5	CAMC274580
UYA 85	Bk 1946 Pg 1005	13S & 14S	21E	32 & 5	CAMC274581
UYA 86	Bk 1946 Pg 1006	13S & 14S	21E	32 & 5	CAMC274582
UYA 87	Bk 1946 Pg 1007	14S	21E	5	CAMC274583
UYA 88	Bk 1946 Pg 1008	14S	21E	5	CAMC274584
UYA 89	Bk 1946 Pg 1009	14S	21E	5	CAMC274585
UYA 90	Bk 1946 Pg 1010	14S	21E	5	CAMC274586
UYA 91	Bk 1946 Pg 1011	14S	21E	5	CAMC274587
UYA 92	Bk 1946 Pg 1012	13S & 14S	21E	32 & 5	CAMC274588
UYA 93	Bk 1946 Pg 1013	13S & 14S	21E	32 & 5	CAMC274589
UYA 94	Bk 1946 Pg 1014	13S & 14S	21E	32 & 5	CAMC274590
UYA 95	Bk 1946 Pg 1015	14S	21E	5	CAMC274591
UYA 96	Bk 1946 Pg 1016	14S	21E	5	CAMC274592
UYA 97	Bk 1946 Pg 1017	14S	21E	5	CAMC274593
UYA 98	Bk 1946 Pg 1018	13S & 14S	21E	33 & 5	CAMC274594
UYA 99	Bk 1946 Pg 1019	14S	21E	5	CAMC274595
UYA 100	Bk 1946 Pg 1020	13S	21E	32	CAMC274596
UYA 101	Bk 1946 Pg 1021	13S	21E	32	CAMC274597
UYA 102	Bk 1946 Pg 1022	13S	21E	32	CAMC274598
UYA 103	Bk 1946 Pg 1023	13S	21E	32	CAMC274599
UYA 104	Bk 1946 Pg 1024	13S	21E	32	CAMC274600
UYA 105	Bk 1946 Pg 1025	13S	21E	32	CAMC274601
UYA 106	Bk 1946 Pg 1026	13S & 14S	21E	32, 5, & 6	CAMC274602
UYA 107	Bk 1946 Pg 1027	13S	21E	32	CAMC274603
UYA 108	Bk 1946 Pg 1028	13S & 14S	21E	32 & 6	CAMC274604
UYA 109	Bk 1946 Pg 1029	13S	21E	32	CAMC274605
UYA 110	Bk 1946 Pg 1030	13S & 14S	21E	32 & 6	CAMC274606
UYA 111	Bk 1946 Pg 1031	13S & 14S	21E	32 & 6	CAMC274607

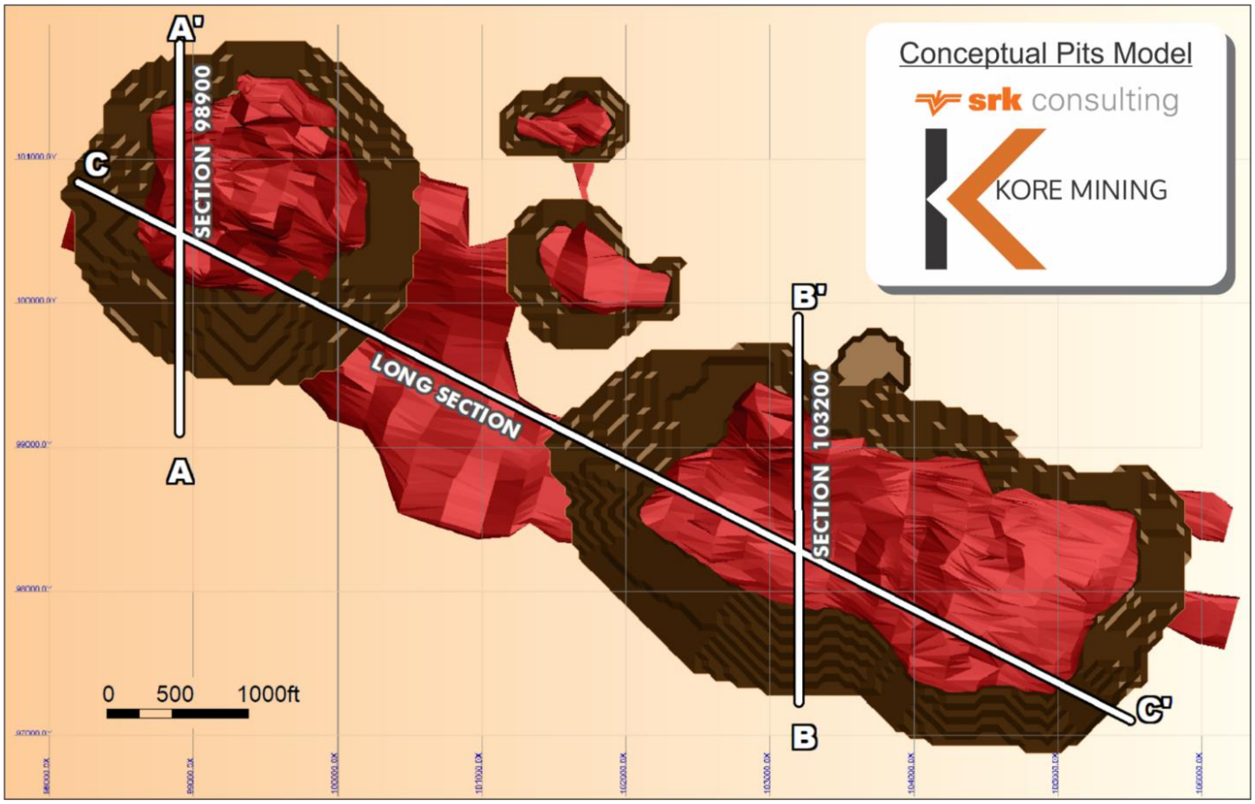
Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
UYA 112	Bk 1946 Pg 1032	13S	21E	32	CAMC274608
UYA 113	Bk 1946 Pg 1033	13S	21E	32	CAMC274609
UYA 114	Bk 1946 Pg 1034	13S	21E	31 & 32	CAMC274610
UYA 115	Bk 1946 Pg 1035	13S	21E	31 & 32	CAMC274611
UYA 116	Bk 1946 Pg 1036	13S	21E	31 & 32	CAMC274612
UYA 117	Bk 1946 Pg 1037	13S	21E	32	CAMC274613
UYA 118	Bk 1946 Pg 1038	13S	21E	32	CAMC274614
UYA 119	Bk 1946 Pg 1039	13S	21E	32	CAMC274615
UYA 120	Bk 1946 Pg 1040	13S	21E	32	CAMC274616
UYA 121	Bk 1946 Pg 1041	13S	21E	32	CAMC274617
UYA 122	Bk 1946 Pg 1042	13S	21E	32	CAMC274618
UYA 123	Bk 1946 Pg 1043	13S	21E	32	CAMC274619
UYA 124	Bk 1946 Pg 1044	13S	21E	32	CAMC274620
UYA 125	Bk 1946 Pg 1045	13S	21E	32	CAMC274621
UYA 126	Bk 1946 Pg 1046	13S	21E	32	CAMC274622
UYA 127	Bk 1946 Pg 1047	13S	21E	32	CAMC274623
UYA 128	Bk 1946 Pg 1048	13S	21E	32	CAMC274624
UYA 129	Bk 1946 Pg 1049	13S	21E	32	CAMC274625
UYA 130	Bk 1946 Pg 1050	13S	21E	32	CAMC274626
UYA 131	Bk 1946 Pg 1051	13S	21E	32	CAMC274627
UYA 132	Bk 1946 Pg 1052	13S	21E	32	CAMC274628
UYA 133	Bk 1946 Pg 1053	13S	21E	32	CAMC274629
UYA 134	Bk 1946 Pg 1054	13S	21E	32	CAMC274630
UYA 135	Bk 1946 Pg 1055	13S	21E	32	CAMC274631
UYA 136	Bk 1946 Pg 1056	13S	21E	32	CAMC274632
UYA 137	Bk 1946 Pg 1057	13S	21E	32	CAMC274633
UYA 138	Bk 1946 Pg 1058	13S	21E	32	CAMC274634
UYA 139	Bk 1946 Pg 1059	13S	21E	32	CAMC274635
UYA 140	Bk 1946 Pg 1060	13S	21E	32	CAMC274636
UYA 141	Bk 1946 Pg 1061	13S	21E	32	CAMC274637
UYA 142	Bk 1946 Pg 1062	13S	21E	32	CAMC274638
UYA 143	Bk 1946 Pg 1063	13S	21E	32	CAMC274639
UYA 144	Bk 1946 Pg 1064	13S	21E	32	CAMC274640
UYA 145	Bk 1946 Pg 1065	13S	21E	32	CAMC274641
UYA 146	Bk 1946 Pg 1066	13S	21E	32	CAMC274642
UYA 147	Bk 1946 Pg 1067	13S	21E	32	CAMC274643
UYA 148	Bk 1946 Pg 1068	13S	21E	32	CAMC274644
UYA 149	Bk 1946 Pg 1069	13S	21E	32	CAMC274645
UYA 150	Bk 1946 Pg 1070	13S	21E	32	CAMC274646
UYA 151	Bk 1946 Pg 1071	13S	21E	32	CAMC274647
UYA 152	Bk 1946 Pg 1072	13S	21E	32	CAMC274648
UYA 153	Bk 1946 Pg 1073	13S	21E	32	CAMC274649
UYA 154	Bk 1946 Pg 1074	13S	21E	32	CAMC274650
UYA 155	Bk 1946 Pg 1075	13S	21E	32	CAMC274651
UYA 156	Bk 1946 Pg 1076	13S	21E	32	CAMC274652

Claim Name	County Book and Page No.	Township	Range	Section	BLM Serial No.
UYA 157	Bk 1946 Pg 1077	13S	21E	32	CAMC274653
UYA 158	Bk 1946 Pg 1078	13S	21E	32	CAMC274654
UYA 159	Bk 1946 Pg 1079	13S	21E	32	CAMC274655
UYA 160	Bk 1946 Pg 1080	13S	21E	32	CAMC274656
UYA 161	Bk 1946 Pg 1081	13S	21E	32	CAMC274657
UYA 162	Bk 1946 Pg 1082	13S	21E	32	CAMC274658
UYA 163	Bk 1946 Pg 1083	13S	21E	32	CAMC274659
UYA 164	Bk 1946 Pg 1084	13S	21E	32	CAMC274660
UYA 165	Bk 1946 Pg 1085	13S	21E	32	CAMC274661
UYA 166	Bk 1946 Pg 1086	13S	21E	32	CAMC274662
UYA 167	Bk 1946 Pg 1087	13S	21E	32	CAMC274663
UYA 168	Bk 1946 Pg 1088	13S	21E	32	CAMC274664
UYA 169	Bk 1946 Pg 1089	13S	21E	32	CAMC274665
UYA 170	Bk 1946 Pg 1090	13S	21E	32	CAMC274666
UYA 171	Bk 1946 Pg 1091	13S	21E	32	CAMC274667
UYA 172	Bk 1946 Pg 1092	13S	21E	32	CAMC274668
UYA 173	Bk 1946 Pg 1093	13S	21E	32	CAMC274669
UYA 174	Bk 1946 Pg 1094	13S	21E	32	CAMC274670
UYA 175	Bk 1946 Pg 1095	13S	21E	32	CAMC274671
UYA 176	Bk 1946 Pg 1096	13S	21E	32	CAMC274672
UYA 177	Bk 1946 Pg 1097	13S	21E	32	CAMC274673
UYA 178	Bk 1946 Pg 1098	13S	21E	32	CAMC274674
UYA 179	Bk 1946 Pg 1099	13S	21E	32	CAMC274675
UYA 180	Bk 1946 Pg 1100	13S	21E	32	CAMC274676
UYA 181	Bk 1946 Pg 1101	13S	21E	32	CAMC274677
UYA 184	Bk 1946 Pg 1102	13S	21E	32	CAMC274678
UYA 185	Bk 1946 Pg 1103	13S	21E	32	CAMC274679
UYA 186	Bk 1946 Pg 1104	13S	21E	32	CAMC274680
UYA 187	Bk 1946 Pg 1105	13S	21E	32	CAMC274681
UYA 188	Bk 1946 Pg 1106	13S	21E	32	CAMC274682
UYA 189	Bk 1946 Pg 1107	13S	21E	32	CAMC274683
UYA 190	Bk 1946 Pg 1108	14S	21E	5	CAMC274684

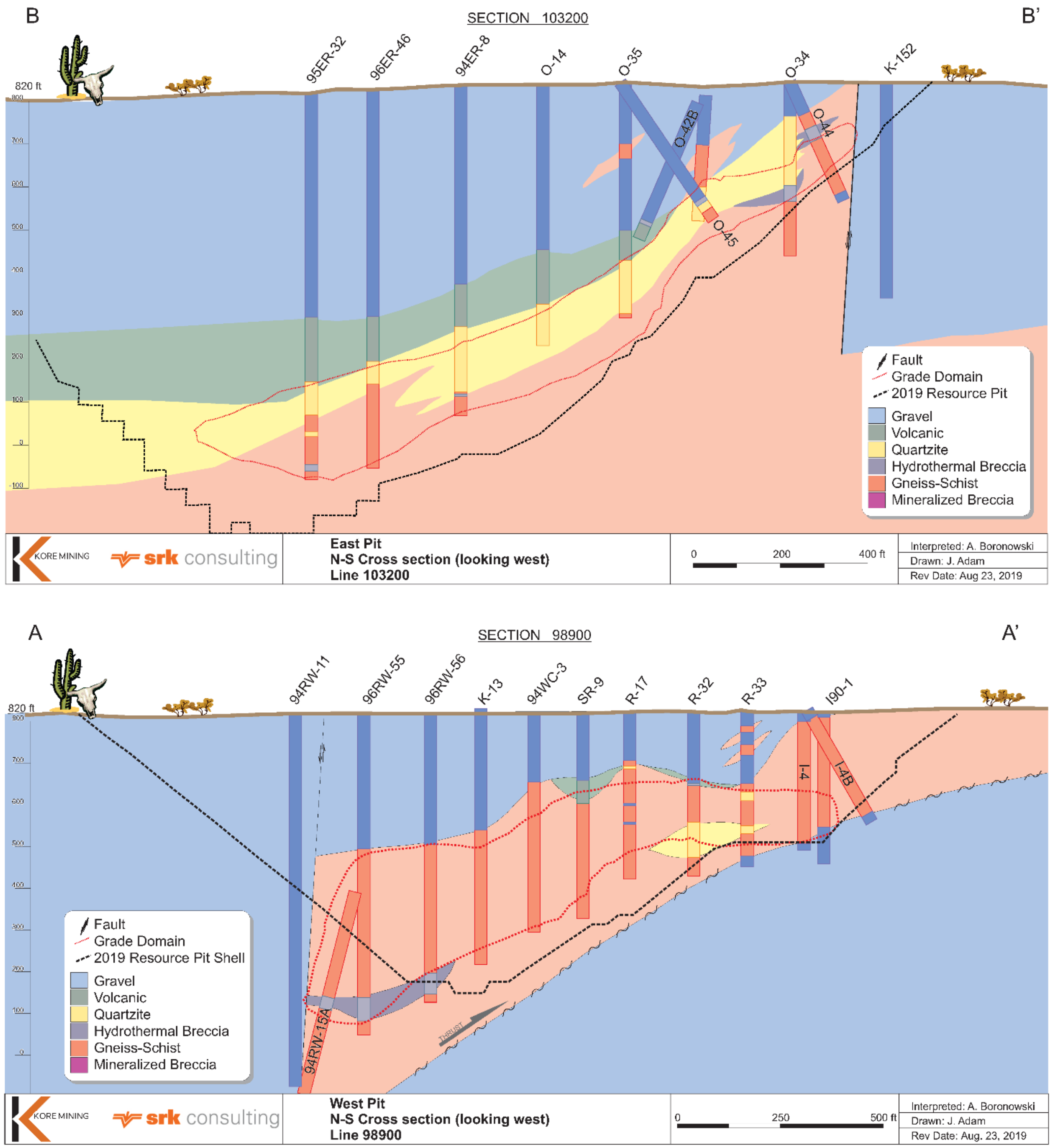
Appendix B

Conceptual Geological Cross Sections

Conceptual Plan Across the Imperial Gold Deposit, Showing Geological Cross Section Locations (Looking North)



Geological Cross-Sections Along Sections 103200 (Top) and 98900 (Below)



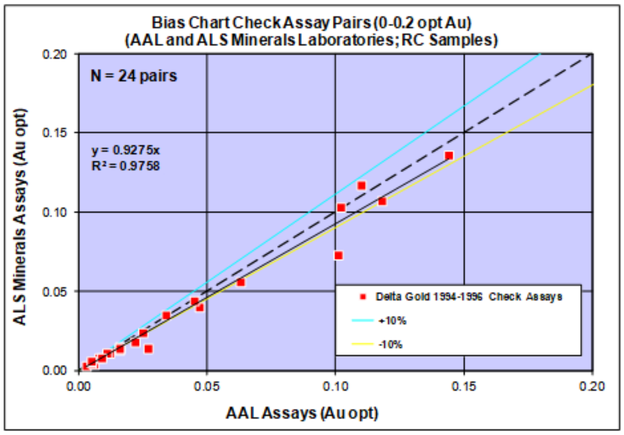
APPENDIX C

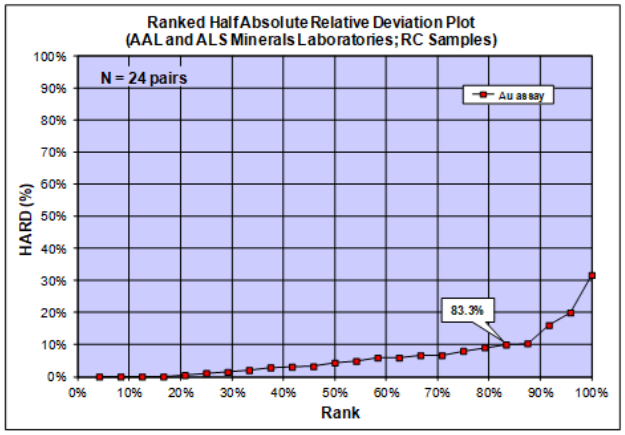
Analytical Quality Control Data and Relative Precision Charts

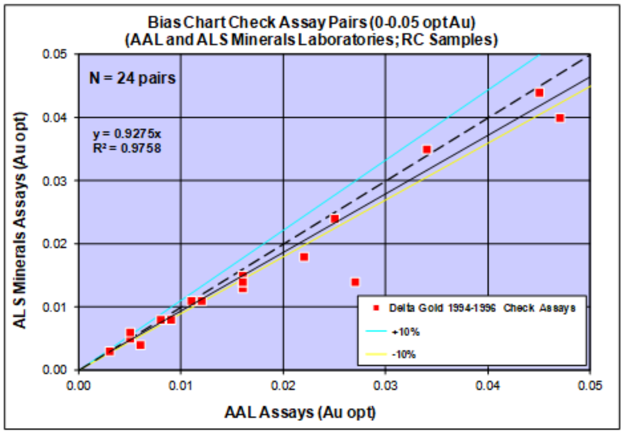
Charting of verification sampling conducted by Delta in 2012 on RC samples from 1994 to 1996 (24 pairs)

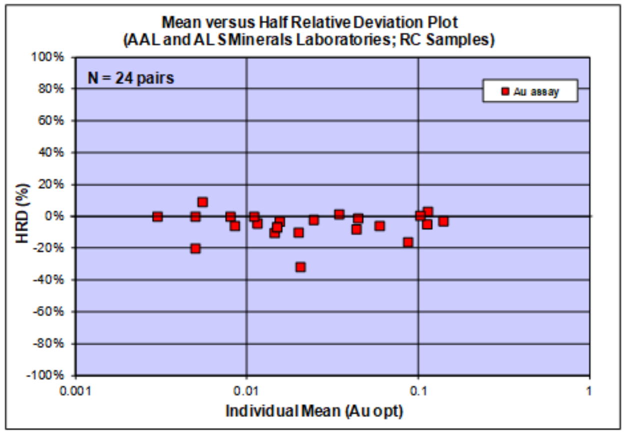
Project	Delta Gold - Imperial
Data Series	Delta Gold 1994-1996 Check Assays
Data Type	RC Samples
Commodity	Au in opt
Analytical Method	Fire Assay
Detection Limit	0.001 opt Au
Original Dataset	AAL Assays
Paired Dataset	ALS Minerals Assays

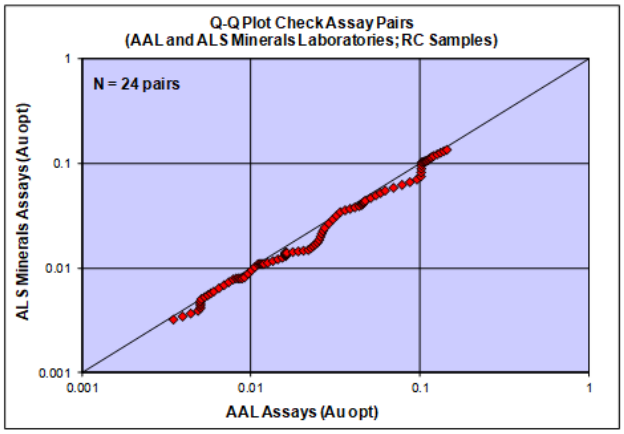
Statistics	**AAL**	**ALS Minerals**
Sample Count	24	24
Minimum Value	0.003	0.003
Maximum Value	0.144	0.136
Mean	0.040	0.037
Median	0.019	0.015
Standard Error	0.009	0.008
Standard Deviation	0.043	0.040
Correlation Coefficient	0.9880	
Pairs ≤ 10% HARD	83.3%	

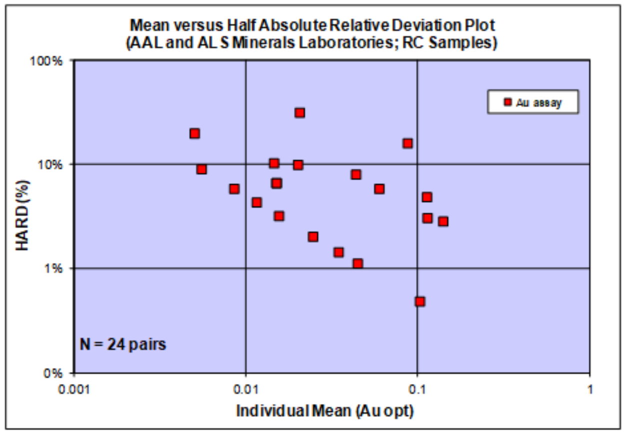










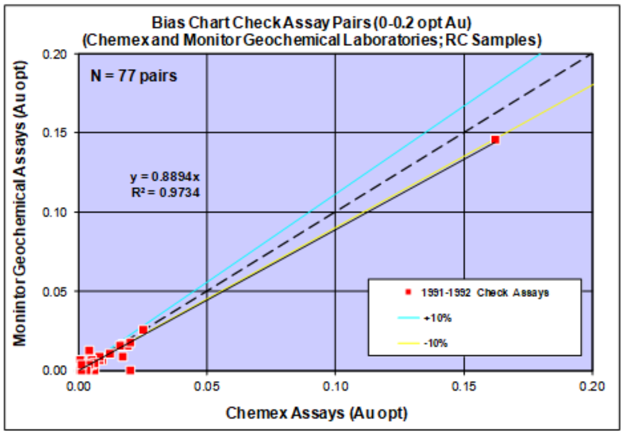


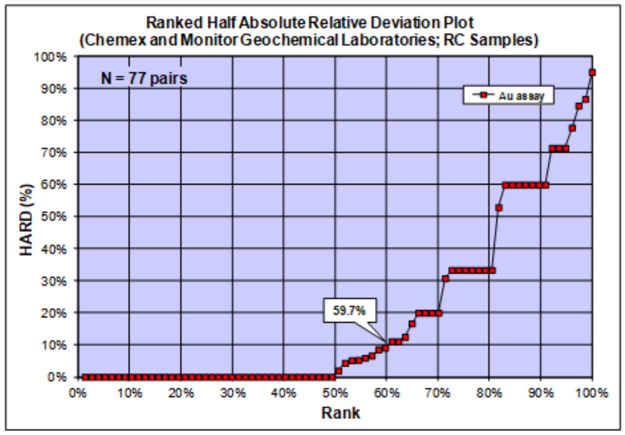
Charting of umpire check assays from 1991 to 1992 sampling (77 pairs)

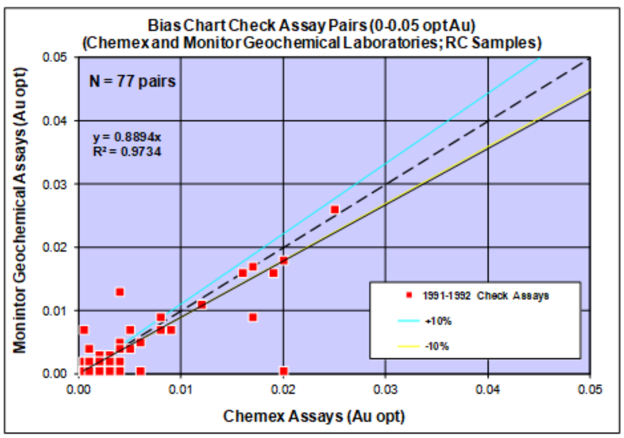
srk consulting

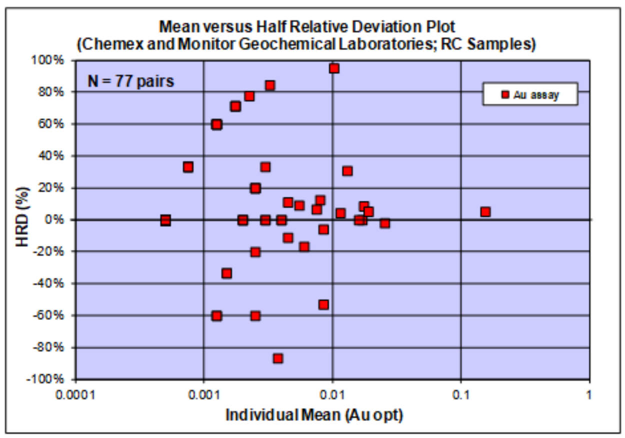
Project	Delta Gold - Imperial
Data Series	1991-1992 Check Assays
Data Type	RC Samples
Commodity	Au in opt
Analytical Method	Fire Assay
Detection Limit	0.001 opt Au
Original Dataset	Chemex Assays
Paired Dataset	Monintor Geochemical Assays

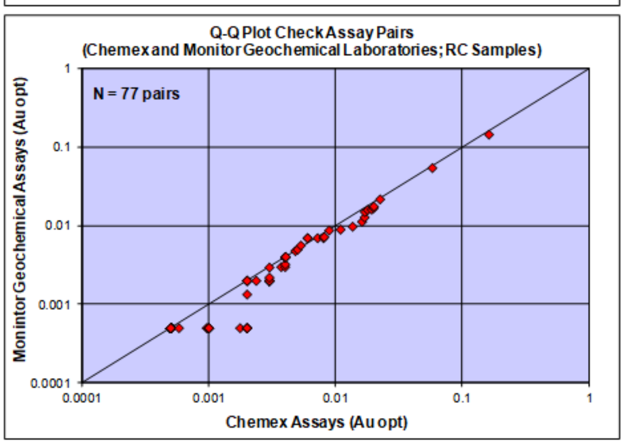
Statistics	**Cehemex**	**Monitor**
Sample Count	77	77
Minimum Value	0.0005	0.0005
Maximum Value	0.162	0.146
Mean	0.006	0.005
Median	0.001	0.001
Standard Error	0.002	0.002
Standard Deviation	0.019	0.017
Correlation Coefficient	0.9866	
Pairs ≤ 10% HARD	59.7%	

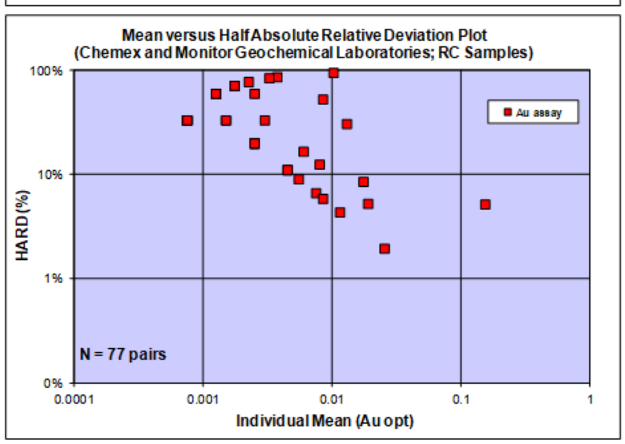








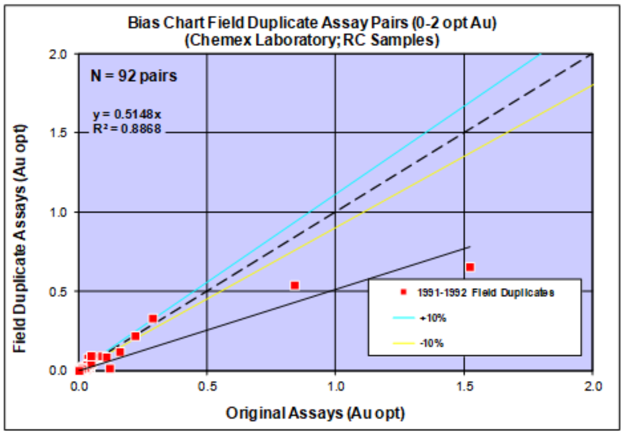


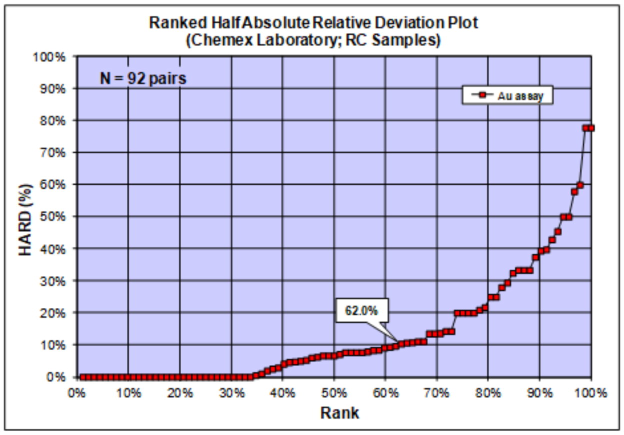


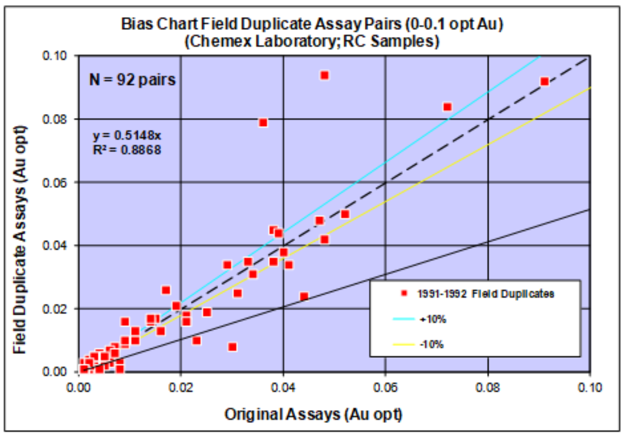
Charting of blind field duplicates from 1991 to 1992 drilling (92 pairs)

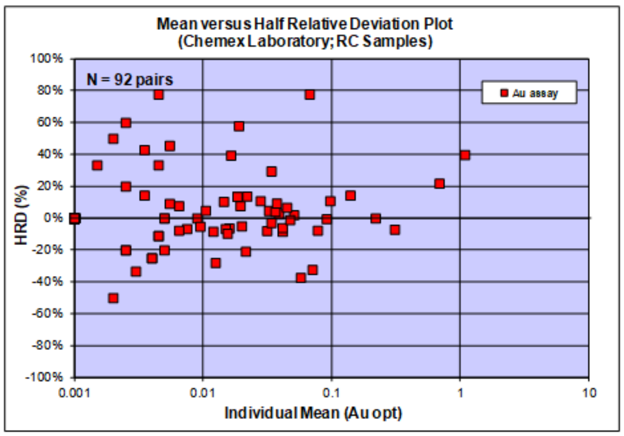
Project	Delta Gold - Imperial
Data Series	1991-1992 Field Duplicates
Data Type	RC Samples
Commodity	Au in opt
Analytical Method	Fire Assay
Detection Limit	0.001 opt Au
Original Dataset	Original Assays
Paired Dataset	Field Duplicate Assays

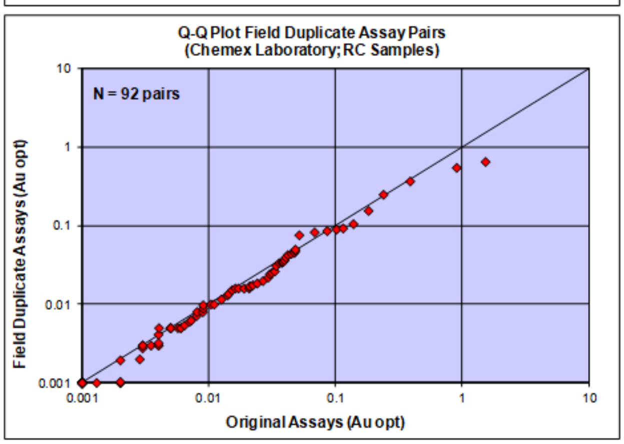
Statistics	Original	Field Duplicate
Sample Count	92	92
Minimum Value	0.001	0.001
Maximum Value	1.522	0.656
Mean	0.048	0.035
Median	0.006	0.005
Standard Error	0.019	0.010
Standard Deviation	0.183	0.096
Correlation Coefficient	0.9475	
Pairs ≤ 10% HARD	62.0%	

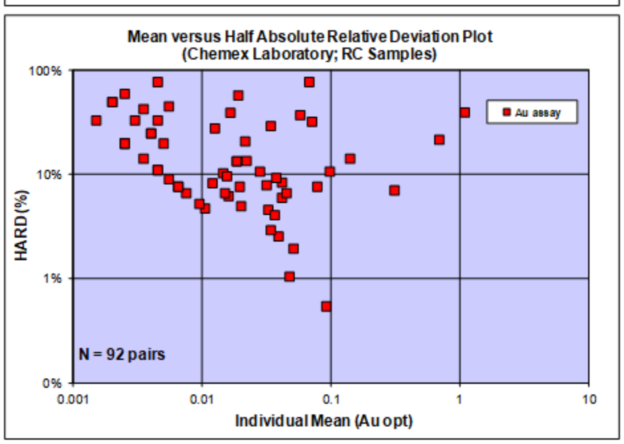












CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled, Preliminary Economic Assessment - Technical Report for the Imperial Gold Project, California, USA, May 19, 2020

I, Glen Cole, do hereby certify that:

1) I am a Principal Resource Geologist with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1500, 155 University Avenue, Toronto, Ontario, Canada;
2) I am a graduate of the University of Cape Town in South Africa with a B.Sc. (Hons) in Geology in 1983; I obtained a M.Sc. (Geology) from the University of Johannesburg in South Africa in 1995 and an MEng in Mineral Economics from the University of the Witwatersrand in South Africa in 1999. I have practiced my profession continuously since 1986, having worked on multi-commodity international exploration and mining projects. I worked on gold exploration projects, underground and open pit mining gold operations in Africa and held positions of Mineral Resource Manager, Chief Mine Geologist and Chief Evaluation Geologist, with the responsibility for estimation of mineral resources and mineral reserves for development gold projects and operating gold mines;
3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of the Province of Ontario (APGO#1416) and am also registered as a Professional Natural Scientist with the South African Council for Scientific Professions (Reg#400070/02);
4) I have personally visited the project area during February 9 to 10, 2012 and on November 26, 2019;
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am co-author of this report and responsible for Sections 4 to 12, 14 and 23 of the report and am a supporting author for Sections 1, 2 and 24-27. I accept professional responsibility for those sections;
8) I have had prior involvement with the subject property, having previously contributed to a technical report on the property in 2012 and in 2019:
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by KORE Mining Ltd. to prepare a technical report of the Imperial gold project. The technical report is based on a site visit, a review of project files and discussions with KORE Mining Ltd. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Imperial gold project or securities of KORE Mining Ltd; and
12) That, as of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]

Toronto, Ontario, Canada
May 19, 2020

Glen Cole, PGeo (APGO#1416), PrSciNat. (Reg#400070/02)
Principal Consultant (Resource Geology)

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled, Preliminary Economic Assessment - Technical Report for the Imperial Gold Project, California, USA, May 19, 2020

I, Terre A. Lane, do hereby certify that:

1) I am the Director of Mining Engineering with the firm of Global Resource Engineering Ltd ("GRE") with an office at 600 Grant Street, Suite 975, Denver Colorado, 80203, U.S.A;
2) I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this MRE is project management, mineral resource estimation, mine capital and operating costs estimation, and economic analysis with 25 or more years of experience in each area. I have created or overseen the resource estimation, mine design, capital and operating cost estimation, and economic analysis of well over a hundred open pit projects. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
3) I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
4) I have personally visited the project area on January 9-10, 2020;
5) 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;
6) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for section 16, and am a co-author on sections 1, 2, 3, 17-22, and 24-27, this technical report.
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) Global Resource Engineering Ltd. ("GRE") was retained by KORE Mining Ltd. to prepare a technical report of the Imperial gold project. The technical report is based on a site visit, a review of project files and discussions with KORE Mining Ltd. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Imperial gold project or securities of KORE Mining Ltd; and
12) That, as of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

["signed and sealed"]

Denver Colorado, U.S.A.
May 19, 2020

Terre A. Lane
Principal Consultant (Mining)

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled, Preliminary Economic Assessment - Technical Report for the Imperial Gold Project, California, USA, May 19, 2020

I, Jeffrey Todd Harvey, do hereby certify that:

1) I am the Director of Process Engineering with the firm of Global Resource Engineering Ltd ("GRE") with an office at 600 Grant Street, Suite 975, Denver Colorado, 80203, U.S.A;
2) I graduated with Ph.D. in Mining Engineering from the Queen's University at Kingston in 1994, a Master's degree in Mining Engineering from the Queen's University at Kingston in 1990 and a Bachelors degree in Mining Engineering in 1988 all with a specialization in mineral processing. I also hold a degree in Metallurgical Engineering and Computer Science from Ryerson University in Toronto Canada graduating in 1986 as well as an MBA from the University of New Brunswick in Saint John Canada graduating in 2001. I have worked as a Process Engineer for over 35 years since my graduation from university. My relevant experience includes process due diligence/competent persons evaluations of developmental phase and operational phase mines throughout the world, including mines in the USA, Canada, Kazakhstan, Brazil, Mexico, and Africa to name a few. I have a wide range of experience in multiple mineral fields including precious metal processing and base metals such as copper, lead, and zinc;
3) I am a Registered Member (No. 04144120) of the Society for Mining, Metallurgy & Exploration Inc. (SME). I am also a member of the Association for Mineral Exploration (AME), Minerals Engineering Journal Review Board, and the Journal of Hydrometallurgy Review Board;
4) I have personally visited the project area on January 9-10, 2020;
5) 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;
6) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for sections 13 and 17 of this technical report, and a supporting author on sections 1, 18, 22 and 26 and accept responsibility for those sections;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) Global Resource Engineering Ltd. ("GRE") was retained by KORE Mining Ltd. to prepare a technical report of the Imperial gold project. The technical report is based on a site visit, a review of project files and discussions with KORE Mining Ltd. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Imperial gold project or securities of KORE Mining Ltd; and
12) That, as of the effective date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Denver Colorado, U.S.A.
May 19 2020

["signed and sealed"]
Jeffrey Todd Harvey
Principal Consultant (Processing)