RPMGLOBAL

NI 43-101 Technical Report, Preliminary Economic Assessment of the Gold Road Mine, Arizona, USA

Para Resources Inc.

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1. Executive Summary

RPM Global (RPM) has been retained by Para Resources Inc. (Client) to recommend actions to bring the project into compliance with the CIM standards and produce an NI 43-101 Preliminary Economic Assessment (PEA) Technical Report. In addition the Client requested the following activities which will make up a part of the Technical report and provide information for mine planning and resource estimation:

- Review the processing facility and infrastructure and comment on the conditions of the plant and facilities.
- Complete trade-off studies for transporting the ore from underground to the surface.
- Evaluate various mining method to determine the most effective and efficient mining method
- Develop a sampling program to verify the historical sampling results in order to bring the historical results into compliance with current industry reporting standards.
- Complete a resource estimation consistent with current industry reporting standards.
- The PEA is based on resources that have been classified as Inferred. This PEA is preliminary in nature in that it includes Inferred Mineral Resources that 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.

1.1 Property Description and Encumbrances

The Gold Road property (Property) is 15 miles (24 km) northeast of the Arizona-California-Nevada border and 25 miles (40 km) southwest of Kingman, Arizona.

The Gold Road property fully owned or controlled by Para Resources Inc. (Company) consists of 21 patented claims (299 acres), 4 patented millsite claims (20 acres or 8.08 ha) and 82 unpatented claims (1,525 acres). The Company also has under lease an additional 31 patented claims (466 acres) from Cruski Mines. Total acreage owned or controlled by the Company for the Gold Road property is 2,290. In addition to the Gold Road property, Para Resources, through Gold Road Corporation controls an additional 2846 acres of land that includes 76 patented claims and 74 unpatented claims within the Oatman Mining District.

The annual holding cost for the 82 unpatented mining claims is $12,792 payable to the US Federal and Mohave County. The annual holding costs for the Cruski claims are $8,000. Holding costs for the 48 claims to be recorded and filed will be approximately $7,930.

The Gold Road Property was purchased from Mohave Desert Minerals LLC in 2017 for a total of $7,000,000 payable annually in $1 million dollar payments. The initial payment of $1 million has been made and the remaining $6 million is included in the cash flow model. If there is a default in the annual payment the entirety of the property reverts to the seller. Currently with the mine shut down the property taxes is approximately $13,000 per year. Once the mine goes back into operation the property taxes will increase.

Current environmental liability is limited to a $37,319 cash Arizona State Mine Inspector Bond and a cash Arizona State APP bond totaling $81,603. The current surface disturbance is on patented mining claims which are not subject to federal reclamation regulation. The State of Arizona does not have a mined land reclamation act and the counties in Arizona have no jurisdiction to regulate mining activities. A recent third party review of recent water quality compliance sampling shows no recent water quality violations.

1.2 Geology

The Oatman mining district is dominated by Tertiary volcanic rocks representing at least four major cycles of late Oligocene to early Miocene volcanism. These cycles of volcanism are represented by the Lower Volcanics series, the Middle Volcanics series, the Upper Volcanics series and younger basalt-dominated volcanism. Rock units range from basaltic to rhyolitic in composition, but the bulk of the volcanic sequence consists of alkalic to subalkalic, intermediate rocks with latitic to andesitic compositions. An eruptive center for at least some of the volcanic rocks is inferred to be near the town of Oatman.

The major structures which host important gold mineralization in the Oatman district form a roughly radial pattern outward and southeast from an area centered near the Oatman Amalgamated prospect. This area may be near the center of a three-mile diameter (5 km) circular feature defined by a concentric fracture and joint set, inwardly-dipping faults and dikes and lineaments detected using Landsat satellite imagery and high altitude aerial photographs. These may reflect concentric fractures developed during the ascent or descent of magma within a near-surface magma chamber.

The gold-bearing mineralized bodies in the Oatman mining district are tabular to lens-shaped quartz+calcite+adularia veins localized along northwest- to north-northwest-trending faults and fractures. The structures typically dip steeply north with a few exceptions, notably the Gold Ore and Moss veins and several structures in the far southern portion of the district. Almost without exception, the most important gold mines in the district are from veins hosted by either the Oatman latite or Gold Road latite.

At the Gold Road Mine, the vein system is exposed on the surface for about 1.5 miles (2.4 km) and the ore-grade segment is nearly continuous for about 1 mile (1.6 km). The Gold Road Mine has been mined in the vertical dimension down to the elevation of 2,200 ft (671 m) above sea level. Mining has extracted ore from the Gold Road vein system for a horizontal distance of 7,000 ft (2,133 m) and for a vertical distance of 1450 ft (442 m). Individual lodes on the Gold Road vein structure are up to 2,100 ft (640 m) in length, 620 ft (190 m) in height and vary in width from 3-7 ft (1-2 m) within the Gold Road latite and up to 23 ft (7 m) within the Oatman latite.

1.3 Exploration

The exploration conducted on the Gold Road mine area has been over the history of the mine has been drifting on the vein and sampling along the face and back. Production sampling in stopes was also completed. Beyond the sampling maps, there is little information about exploration programs prior to 2005. From Q4 2005 to Q1 2007 Addwest Mineral Inc. (AMI) carried out a systematic, multi-faceted exploration program on the Gold Road property.

1.4 Drilling

RPM has been provided no documentation of any drilling prior to Addwest Minerals Inc's (AMI) work in the early 1990s.

Underground core drilling took place at many sites within the mine at various times in the recent history of development and production. All of the documented holes are within an area that is below the 900 level and into an area of excellent potential. The reported historical drill intercepts serve as a strong confirmation that both the vein and associated gold mineralization continue to depth.

1.5 Sample Preparation, Analysis, and Security

The work done by Addwest Minerals in the late 1990s and early 2000s used some check assays but relied on the ISO certified labs to provide any QA/QC testing.

RPM has been tasked with the mission of developing a work program to verify the historical sampling along the drifts and certify that it of sufficient quality to be used in a manner consistent with industry standards for reporting resource estimation. This work program consisted of resampling certain areas of the min to verify the historical results.

1.6 Data Validation

The data available to the Author was historical sampling from the pre-1942 era and the post 1990 era. One of the major issues in preparing a resource report and the PEA was devising a protocol to validate the historical assays so they could be used to estimate, at a minimum, an Inferred resource. The program developed included validation sampling on two levels of the mine, the 700 Level and the 840 Level. The 700 Level was sampled prior to 1942 and the assays of the original samples were by fire assay (FA) from the mine lab. The 840 Level was sampled after 1990 and the samples were assayed using cyanide soluble methodology in the mine lab.

Thirty nine samples were collected on the 700 Level along a drift length of 400 ft. Forty three samples were collected along about 440 ft of drift on the 840 Level. All samples were assayed by both FA and CN soluble methods. The results of the original sampling were compared to the validation sampling using techniques similar to comparing twin drill holes.

The average of the fire assays for the 39 verification samples on the 700 Level was 0.227 oz. Au per ton. The average of the corresponding original sampling was 0.180 oz. Au per ton. Further there is a reasonable correspondence between those intervals of the verification sampling with grades greater than 0.1 oz. Au per ton and intervals of the original samples with grades greater than 0.1 oz. Au per ton.

The average of the cyanide soluble assays for the 43 verification samples on the 840 Level was 0.181 oz. Au per ton. The average of the corresponding original sampling was 0.186 oz. Au per ton. The average for the fire assays was 0.243 oz. Au per ton. The average of all samples ≥ 0.10 oz. Au per ton was 0.229 oz. Au per ton for the verification sampling and 0.231 oz. Au per ton for the original sampling, and 0.30 oz. Au per ton for the fire assays.

RPM opines the original sampling is of a quality that can be used to estimate Inferred resources.

1.7 Metallurgical Testwork and Mineral Processing

The processing parameters for Gold Road ore have long been established by both actual processing of the ore over many years and by metallurgical testwork. The gold is present as very fine particles in extremely hard, chalcedonic quartz. Processing requires grinding to 80% passing 325 mesh and 24-hour leaching which results in about 95% gold extraction.

Excluding the ore processing in the early-1900s, about one million tons of Gold Road ore grading about 0.2 ounces gold/ton have been processed by the following two plants:

- A 400-ton/day counter-current-decantation/Merrill-Crowe plant, operated from 1937 to 1941
- A 500-ton/day carbon-in-pulp (CIP) plant, operated from 1996 to 2016

The latter plant is still in existence and is in good condition. Tailings from the current plant are filtered and dry-stacked in a tailings storage facility (TSF) close to the plant.

1.8 Resource Estimation

Resources were supported by 14,768 channel samples. The database contains 19,400 assay data excluding the 16 drill holes which were not used in the resource estimate. The channel samples were taken within the mined stopes and the development workings. Channels are spaced every five ft along the drifts.

All the Inferred mineral resources are reported using a 0.1 opt Au cutoff, which is roughly the current economic cutoff. In order to meet the international requirement of reasonable prospect for eventual economic extraction, the mineral resources quoted in this report are constrained within a maximum vertical distance of 200 ft from a drift.

Table 1-1 Gold Road Mineral Resources at 0.1 opt Au cutoff

Distance (ft)	Au (opt)	Tons	Ounces
<50	0.23	160,000	36,200
50-100	0.21	268,000	56,600
100-200	0.22	550,000	121,000
Total	**0.22**	**978,000**	**214,000**

1.9 Reserve Estimates

This PEA is preliminary in nature and is based on 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.

1.10 Mining

The underground mine historically used Shrinkage Stoping Mining (SSM). During the most recent mining phase haulage of the ore and waste to the surface was with underground trucks. RPMGlobal reviewed the continued use of SSM for the Gold Road Mine and has also reviewed an alternative mining method known as Raise Access Mining. Raise Access Mining (RAM) incorporates an Alimak style raise climber to develop the access raise in ore from the sill level to the top level. Once the raise is established the climber is used to drill and blast horizontal production holes from the bottom of the ore block up in retreat. The ore is loaded from the bottom sill level using a mucker loading into nearby muck bay, a truck for the haul up the decline or to the proposed new shaft loadout. Gold Road used a form of RAM in a test stope in the past with mixed results. RPM advises that the use of RAM is the preferred method and has been used in the PEA as part of the restart and full mining method for the mine. RAM provides a safer mining process, with limited, to no exposure of the miners to unsupported ground, as well as lower operating cost. Both mining methods are discussed in the following sections of the PEA.

RPM also reviewed alternative haulage methods using the current truck haulage decline (11,000 ft. one way) as well as a truck/shaft ore haulage scenario. As expected the truck/shaft method was the preferred alternative due to costs and efficiency.

1.11 Infrastructure

The mine and plant are close to established towns with a paved public access road passing alongside the operation. Water use is minimal since the tailings are filtered; however, water is plentifully available from the mine and from adjoining mine shafts. Grid power is provided to the mine and plant. All the buildings required for the operation are in place and functional.

1.12 Environmental Studies, Permitting and Social or Community Impact

The Company stated all necessary permits for mining and production are in place. The environmental issues including permitting, bonding, and closure plans and costs will be discussed in more detail in Section 20 of this report. The most recent permit amendment allows for toll milling at the Gold Road Mill of mineralized material similar to those of the Gold Road Mine.

Current Permits:

- Aquifer Protection Permit (APP) 2015 – Permit No. 102805

- Air Quality Control Permit – No. 65238 as amended LTF No. 67979
- Permit to Appropriate Public Water of The State of Arizona – Permit No. 33-96287-000
- Nationwide Permit 404 – File No. 930128500 (Clean Water Act)
- EPA NPDES Storm Water Discharge Permit – Permit No. AZCN68776
- NPDES Construction Storm Water Permit – Permit No. AZCN68776
- Mining Safety and Health Administration Mine Identification # 02-02620

1.13 Capital and Operating Costs

The total capital costs for the restart of the Gold Road Mine include US$5.4M for the restart, preliminary development and shaft / hoist installation and purchase of the raise climbers completed in Year 1. All other development and other capital cost are considered to be sustaining capital for the remaining LOM.

Estimated capital cost to bring the ore-processing plant into operational condition is US$ 0.5 million. No capital costs are anticipated for the infrastructure.

The underground mine operating costs were developed on a per foot of advance basis for the waste development. Other operating costs, includes raise mining (in ore), production mining, contractor labor and management and Gold Road site management for the underground mine. Total mining costs are US$64.90 per ton of mineralized material.

Estimated ore-processing operating costs are US$27.06/ton. Estimated G&A costs are US$5.89/ton of mineralized material processed or about US$2.5M/year.

1.14 Economic Analysis

This PEA is preliminary in nature and is based on 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.

The total capital costs for the restart of the Gold Road Mine include US$9.5M for the restart, preliminary development and shaft/hoist installation and purchase of the raise climbers completed in Year 1 and 2, development drilling and Year 1 property payments. All other development and other capital, is considered to be sustaining capital for the remaining LOM. Table 1-2 shows the economic assumptions and results.

Table 1-2 Economic Assumptions and Results

Gold Road	Units	LOM Value
Mineralized Material	Tons	1,110,274
Gold - Mined Grade	Gold oz. per ton	0.19
Gold Recovery	%	95%
Payable Gold	oz.	203,569
Gold Price	US$/oz.	$1,200
Net Revenue	US$ 000's	$238,175
Capital Cost	US$ 000's	$5,744
Sustaining Capital	US$ 000's	$6,454
Total Capital	US$ 000's	$12,198
Total Operating Cost	US$ 000's	$110,362
Total All-in Sustaining Cost (AISC)	US$/oz. gold	$632.79
Total All in Cost	US$/oz. gold	$659.29
Payback Period	Year	1.5
Cumulative Net Cash flow	US$ 000's	$103,964
Pre Tax NPV @ 5 %	US$ 000's	$81,309
Pre Tax IRR	%	238%
Post Tax NPV @ 5 %	**US$ 000's**	**$56,739**
Post Tax IRR	**%**	**175%**

The NPV is still robust with changes in gold prices and capital and operating costs. Table 1-3 shows the sensitivities to these changes.

Table 1-3 Economic Sensitivities

	Percent Change from Base Case	Gold Price US$	NPV at 5% Discount X $1,000,000
Gold Price	Base Case	1,200	58.2
	-20.00%	960	28.7
	-10.00%	1,080	42.7
	10.00%	1,320	70.7
	20.00%	1,440	84.7
OPEX	Base Case	1,200	58.2
	-20.00%	1,200	70.0
	-10.00%	1,200	63.3
	10.00%	1,200	50.1
	20.00%	1,200	43.5
CAPEX	Base Case	1,200	58.2
	-20.00%	1,200	61.3
	-10.00%	1,200	59.7
	10.00%	1,200	56.6
	20.00%	1,200	55.0

2. Introduction

RPMGlobal (RPM) is pleased to provide this NI 43-101 Preliminary Economic Assessment (PEA) Technical Report for Para Resources Inc. (“Para” or the “Company”) to support Canadian stock exchange listings and financing activities of the Company. This report reviews the resources as of April 15, 2018.

2.1 Terms of Reference

RPM’s assignment for this report was to review the available information and complete the following tasks that form the basis of this report. The tasks are:

- Review the processing facility and infrastructure and comment on the conditions of the plant and facilities.
- Complete trade-off studies for transporting the ore from underground to the surface.
- Evaluate various mining method to determine the most effective and efficient mining method.
- Develop a sampling program to verify the historical sampling results in order to make the historical results meet current industry reporting standards.
- Complete a resource estimation consistent with current industry reporting standards.

This PEA is preliminary in nature and is based on 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.

The PEA is based on Inferred Resources defined by the results of historical sampling that has been verified by a sampling program specifically designed for this PEA. The QP has assumed that the current sampling program has verified the historical sampling to a level that the historical results can be used to define Inferred Resources. While some of the inferred resources are bounded on at least one side by mine workings and geostatistical studies along with the sampling support the continuity of the mineralization, the Author feels that there is enough uncertainty in the historical sampling assay values that only Inferred Resources can be defined. The author also assumed the resources defined would be accessible to exploitation by the mining methods proposed. No detailed mine plan has been developed.

This PEA provides the basis for further studies of a more detailed nature. Any future studies should address the detailed mine plan to better define those Inferred Resources that can economically be accessed for mining. The mining method proposed (Alimak) is different than the mining method used historically (Shrink Stope). Mining costs are based on estimates from a mining contractor experienced in Alimak mining. Processing costs are based on historical processing costs updated to today. Capital costs are estimated based on the results of the site visit and benchmarking with similar operations. The cost to raise bore a shaft are a quote from a contractor. The author would anticipate these costs would change little with further studies with the possible exception of development costs to access some of the Inferred Resources.

2.2 Sources of Information

The following documents were utilized in support of the PEA.

- Guilinger, James R., 2017, 2017 Technical Report on the Gold Road Mine, San Francisco District , Oatman Arizona *for* Para Resources, Inc.
- World Industrial Minerals, 2009, 2009 Technical Report on the Gold Road Mine, Addwest Minerals Inc. San Francisco District, Oatman Arizona *for* Addwest Minerals, Inc.
- Wojcik, Joseph R., 2003, Technical Report on Plan for Exploration to Resume Production at the Gold Road Mine, Mohave County, Arizona

- Addwest Minerals Inc., 2004, Gold Road Exploration, Development & Production Plan
- Behre Dolbear & Company, 1997, Technical Due Diligence of the Gold Road Mine, Oatman Arizona, Addwest Minerals, Inc., *for* Standard New York, Inc.
- Various documents from the mine including maps, sample results, and assay certificates
- Addwest Minerals, Inc., 1993, Gold Road Project Metallurgy and Historic Mill Summary
- Bauer, Charlie, 2017, Mill Costs Estimate, *for* Gold Road Mining

2.3 Qualified Persons and Responsibilities

Table 2-1 lists the authors of this report along with their contribution.

Table 2-1 Gold Road PEA Authors Contributing to Report

Authors	Contribution Section of Report	Comments
Richard Kehmeier	Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 15, 19, 20, 23, 24, 25, 26, 27	March 27 & 28 2018 site visit
David Young	Sections 16, parts of 21, 22	October, 4, 2017 site visit
Dick Addison	Sections 13, 17, 18, and parts of 21	October, 4, 2017 site visit
Esteban Acuna	Section 14	

2.4 Units, Terms, Abbreviations, and Acronyms

Tonnages are reported as dry tons of 2,000 pounds. Metal values are given as weight percent. Precious metals are given as troy ounces per short ton. Other units, abbreviations, and acronyms are shown below.

Abbreviation	Unit or Term
AAS	Atomic absorption spectroscopy
ACC	Arizona Corporation Commission
Acre	43,560 square feet
ADEQ	Arizona Depart of the Environmental Quality
Ag	Silver
Au	Gold
Ai	Abrasion index
ANFO	Ammonium Nitrate and Fuel Oil
APP	Aquifer Protection Permit
AWQS	Aquifer Water Quality Standards
AZPDES	Arizona Pollutant Discharge Elimination System
BTU	British Thermal Unit
BWI	Bond ball mill work index
CAA	US - Clean Air Act
Capex	Capital expenditure
CFM	Cubic feet per minute
CO	Carbon monoxide
CO_2	Carbon dioxide
COG	Cut-off grade
Con	Concentrate
Corps	U.S. Army Corp of Engineers
Cu	Copper
$CuSO_4$	Copper Sulphate

CWA	Clean Water Act
DDH	Diamond drill hole
EPCM	Engineering, Procurement and Construction Management
ft	Feet
FOS	Factor of Safety
FW	Foot wall
G	Grams
gpm	US Gallons per minute
gpt	Grams per tonne (grams per metric tonne)
ha	hectare
HDPE	High density polyethylene
hr	hour
hp	horsepower
HW	Hanging wall
ID2	Inverse distance squared
K	1000's or "kilo"
Kg	Kilogram
Km	Kilometer
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt hour
L	Liter
lb.	Pound
LHD	Load haul dump
LOM	Life of mine
M	Million – mega
masl	meters above sea level
Mile	5,280 ft.
MSHA	US – Mine Safety & Health Administration
Mt	Million tonnes or million metric tonnes
MW	Mega Watt
NaCN	Sodium Cyanide
NAG	Net acid generation
NEPA	National Environmental Policy Act
NI 43-101	National Instrument 43-101 (Canadian standards)
NPV	Net Present Value
NSR	Net smelter royalty
Oz	Troy ounces
opT	Troy ounces per short ton (2,000 lbs)
Opex	Operating expenditure
ppb	Parts per billion
ppm	Parts per million
P&ID	Process and instrument diagram
PEA	Preliminary Economic Assessment
POO	Plan of Operations

Q	Q-system (rock mass quality)
QA/QC	Quality Assurance/Quality Control
QP	Qualified person as defined by NI 43-101
RAR	Return air raise
RC	Reverse circulation drilling
RMR	Rock mass rating
ROM	Run of Mine
RPMGlobal	RPMGlobal USA, Inc.
RQD	Rock quality designation
SAG	Semi-autogenous grinding
T	Short ton (1 short ton = 0.90718 tonne)
t	Tonne or metric tonne
US$	U.S. dollar

3. Reliance on Other Experts

Employees of Para and Gold Road Mining have been working on the Gold Road Mine since the mid-1990s and have provided much information on the past history of the mine, mining and processing costs and mining methodology that was successful. These employees of Para and formerly Gold Road Mining are not Qualified Persons as defined in Part 1 Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects

RPM Global has not investigated the land status and has accepted the data supplied by Para for the status and the ownership status of the project.

The overall QP and primary author of this report accepts the input from the persons above and considers it consistent with the standards of reporting for an NI 43-101.

4. Property Description and Location

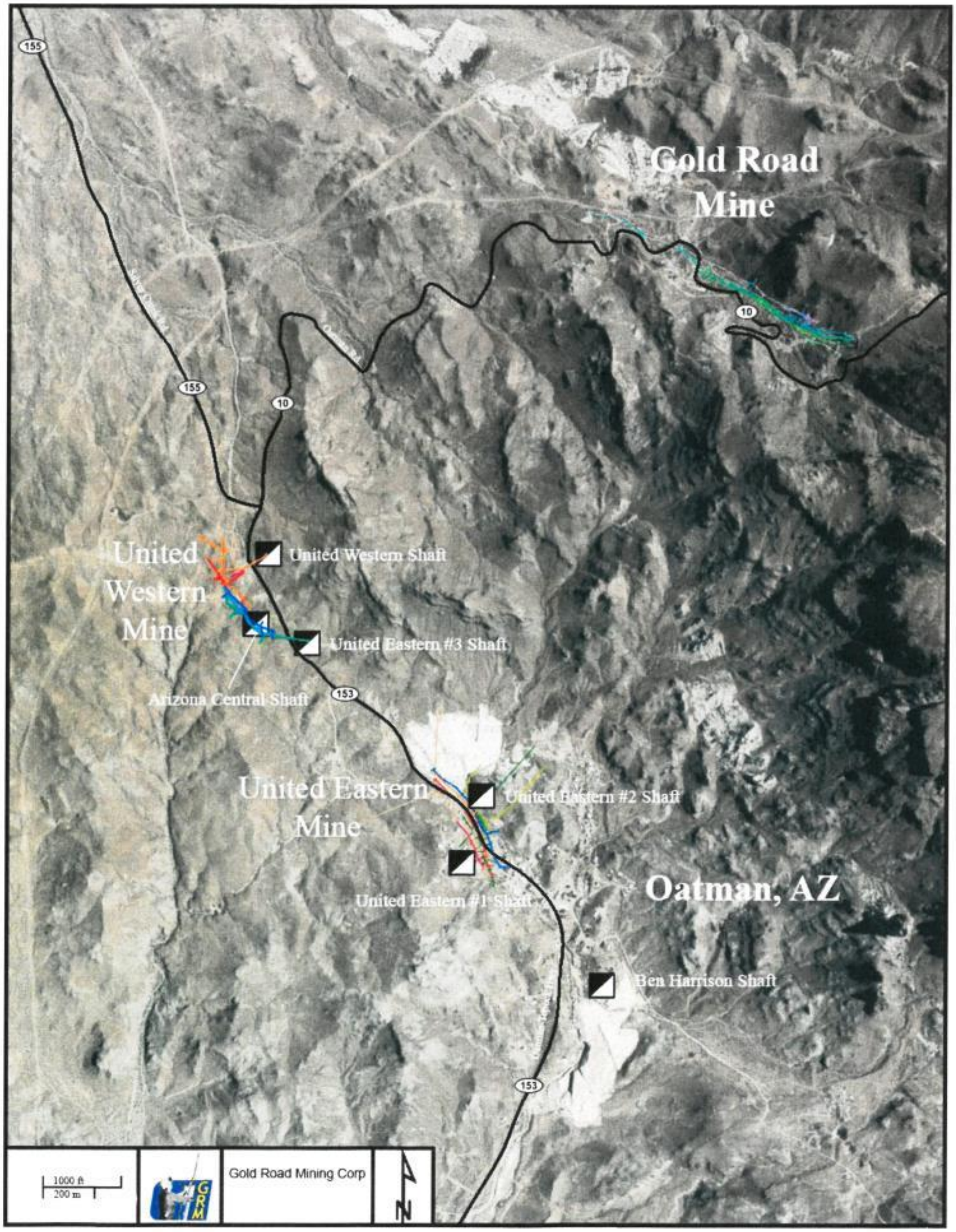
The Gold Road property is in northern Arizona within the Oatman mining district which has the largest gold production of any mining district in Arizona.

4.1 Location

The Gold Road property (Property) is 15 miles (24 km) northeast of the Arizona-California-Nevada border and 25 miles (40 km) southwest of Kingman, Arizona. The property is located in sections 2, 10, 11, 12, 13, 14, & 15 Township 19 N, Range 20 W and in sections 7, 16, 17, & 18 Township 19 N, Range 19 W. The Gold Road Mine portal is located at 35°02'43.3 N and 114°22'31.5 W at an elevation of 3000 ft. (914 m) above sea level. Old U.S. Highway 66 crosses the property within 350 ft (107m) of the Gold Road Mine Portal. (Figure 4-1 and Figure 4-2.)

Figure 4-1 General Location Map of the Gold Road Mine

Figure 4-2 Regional Location Map of the Gold Road Mine



4.2 Property Description

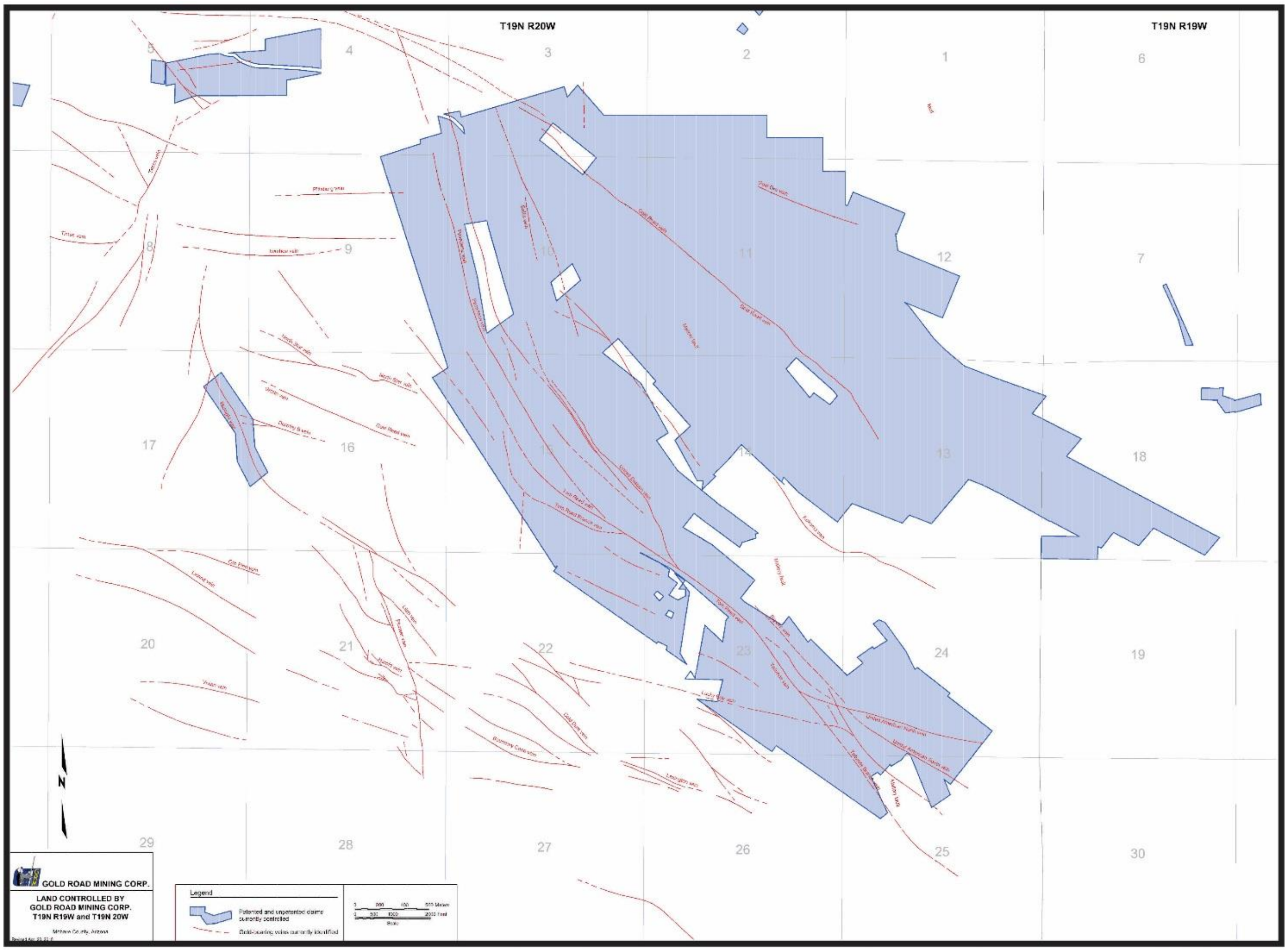
The Gold Road property fully owned or controlled by Para Resources Inc. (Company) consists of 21 patented claims (299 acres), 4 patented millsite claims (20 acres or 8.08 ha) and 82 unpatented claims (1,525 acres). The Company also has under lease an additional 31 patented claims (466 acres) from Cruski Mines. Total acreage owned or controlled by the Company for the Gold Road property is 2,290. In addition to the Gold Road property, Para Resources, through Gold Road Corporation controls an additional 2846 acres of land that includes 76 patented claims and 74 unpatented claims within the Oatman Mining District. Table 4-1 summarizes the Company's land holding in the Oatman district and Figure 4-3 show the outline of the total claim block along with the veins that have been exploited historically.

Table 4-1 Gold Road Corp. Land Holdings (owned or controlled)

Land Holdings	Patented Claims	Patented Acres	Unpatented Claims	Unpatented Acres	Total Acres
Gold Road	21	299	82	1525.1	1824.1
Cruski Acquisition	31	466	0	0	466
La Questa Acquisition	2	33	2	25	58
Claims staked (TrUe Vein)	0	0	72	1739.2	1739.2
Anderson Acquisition	74	1049	0	0	1049
Total	**128**	**1847**	**156**	**3289.3**	**5136.3**

All of the patented claims have been surveyed by a U.S. Mineral Surveyor as a condition of the patent. The unpatented claims were surveyed by a registered surveyor or a handheld GPS by a claim staker in order to establish corners and boundaries.

Figure 4-3 Gold Road Mine Corp. Controlled Claim Map



4.2.1 Claim Holding Costs

The annual holding cost unpatented for the 82 unpatented mining claims is $155 per claim for a total of $12.792 payable to the Bureau of Land Management (BLM). The annual Mohave County filing fee for holding the unpatented claims is approximately $930. The annual holding costs for the Cruski claims are $8,000. Holding costs for the 48 claims to be recorded and filed will be approximately $7,930.

4.2.2 Company Interests and Encumbrances

The Gold Road Mine property was sold by Mohave Desert Minerals LLC (a Nevada LLC) to the Gold Road Mining Corp (a Nevada Corporation). Gold Road Mining Corp is in turn 88% owned by Z79 Gold (USA) Corp (a Nevada Corporation); 6% owned by Four C Resources LLC (a Colorado LLC); 6% owned by Bauer Resources LLC (a Colorado LLC). The Z79 Gold (USA) Corp is in turn owned by Para Resources Inc. (a Canadian TSX.V Company).

The 134 patented and unpatented claims that the Company owns or controls and are the subject of this report are subject to obligations and royalties as described below.

The Gold Road Property was purchased from Mohave Desert Minerals LLC for a total of $7,000,000 payable annually in $1 million dollar payments. The initial payment of $1 million has been made and the remaining 6 million is included in the cash flow model. This purchase included only The Gold Road and Cruski claims. All other claims controlled by the Company are under separate agreements and have no impact on this PEA. If there is a default in the annual payment the entirety of the property reverts to the seller. The property has a

NSR royalty of 2% on all gold production derived from the Gold Road Mine and a 1% NSR on any toll milling of ores from outside properties. The Cruski royalties total 3 %.

The property is subject to the Arizona severance tax. The Arizona severance net tax on all mining operations is 2.5% of a modified net income. Currently with the mine shut down the property taxes is approximately $13,000 per year. Once the mine goes back into operation the property taxes will increase.

4.2.3 Environmental Liabilities

Current environmental liability is limited to a $37,319 cash Arizona State Mine Inspector Bond and a cash Arizona State APP bond totaling $81,603. The current surface disturbance is on patented mining claims which are not subject to federal reclamation regulation. The State of Arizona does not have a mined land reclamation act and the counties in Arizona have no jurisdiction to regulate mining activities. A recent third party review of recent water quality compliance sampling shows no recent water quality violations.

4.3 Permitting

The Company stated all necessary permits for mining and production are in place. The environmental issues including permitting, bonding, and closure plans and costs will be discussed in more detail in Section 20 of this report. The most recent permit amendment allows for toll milling at the Gold Road Mill of mineralized material similar to those of the Gold Road Mine.

Current Permits:

- Aquifer Protection Permit (APP) 2015 – Permit No. 102805
- Air Quality Control Permit – No. 65238 as amended LTF No. 67979
- Permit to Appropriate Public Water of The State of Arizona – Permit No. 33-96287-000
- Nationwide Permit 404 – File No. 930128500 (Clean Water Act)
- EPA NPDES Storm Water Discharge Permit – Permit No. AZCN68776
- NPDES Construction Storm Water Permit – Permit No. AZCN68776
- Mining Safety and Health Administration Mine Identification # 02-02620

5. Accessibility, Climate, Infrastructure, and Physiography

The mine is in an area that has long history of mining and is close to major highways providing easy access and a good infrastructure. Further because the mine has been operation intermittently for more than a century, the mine infrastructure is in place and operational.

5.1 Accessibility

The Property is easily accessed by paved roads from the cities of Kingman and Bullhead City. The Mine is approximately 25 miles (40 km) southwest of Kingman via historic U.S. Route 66 (Oatman Highway) and is approximately fourteen (14) miles (23 km) southeast of Bullhead City via the Oatman Highway and Boundary Cone road. A Mohave County-maintained gravel road (Silver Creek Road) serves as an alternate access route from Bullhead City.

5.2 Climate

The area climate is arid, typical of the northeastern Mojave Desert. Summers are generally very hot and dry, with occasional monsoonal moisture and thunderstorms. The winter months are substantially cooler and generally breezy. Overnight temperatures range from lows near freezing in December and January to 85°F (29C) in July and August. Daytime highs average around 60°F (16°C) in the winter to 110°F (43°C) in the summer months. Precipitation averages less than ten (10) inches (254mm) per year and is usually received during brief summer thunderstorms.

Vegetation is dominated by thorny scrub brush and cacti. The most abundant scrub brush plants are creosote, mesquite, ocotillo and crucifixion thorn. Barrel, prickly pear and cholla cacti are common. There are no trees on the Gold Road Property, but wildflowers and desert grasses may bloom during periods of monsoonal thunderstorm activity.

5.3 Infrastructure

The Property is approximately halfway between the cities of Kingman, AZ (population 27,600) and Bullhead City, AZ (population 49,800), which are major commercial centers for northwestern Arizona and are the two largest cities in Mohave County. These cities are capable of supplying most of the labor, equipment and/or service needs for an operating mine. Kingman, as the county seat, has been a commercial center for past mining operations including the Gold Road Mine, Mineral Park copper mine and the Portland gold mine.

Electrical power is supplied to a sub-station at the Gold Road Mine by UniSouce Energy. Two transcontinental natural gas pipelines, operated by Transwestern Gas Pipeline Company and Questar Pipeline Company, cross The Property.

Both Kingman and Bullhead City have airports capable of handling commercial and passenger air services. Kingman is also served by the Burlington Northern Santa Fe Railroad and is a major transportation hub on U.S. Interstate Highway 40 (I-40).

A significant labor force is available in the area and due to the current lull in the industry; the re-hiring of a percentage of past employees to staff the operation is possible.

Potable water is pumped out of multiple water wells on the Property, while the inflow of ground and rainwater into the mine supports the demand of the mill and mine alike.

5.4 Physiography

The San Francisco mining district lies mainly on the western slope of the Black Mountains of northwestern Arizona. The western slope consists of steep, rugged peaks and incised canyons. The eastern slope is much less rugged and more gently sloping. Precambrian basement rocks underlie a thick package of Tertiary volcanic rocks in the Oatman area. Elevations range from 2,000 (610 m) to 4,500 ft (1,372 m) above sea level. The portal of The Mine's decline is at approximately 3,000 ft above sea level.

Silver Creek, an ephemeral wash, is the main drainage within the Property boundary. Silver Creek flows northwest and is a tributary to the Colorado River.

6. History

Information for this section was taken from Guilinger (2017).

The Gold Road vein was discovered in 1900 and the town of Gold Road was founded. Originally, production from the Gold Road vein was by the Gold Road Mining and Exploration Company. The United States Smelting Refining and Mining Co (USSR&M Co.) acquired the property in 1911 and operated until 1919. The town of Oatman was established in 1912 to service the operations in the district. Prior to 1912 the Gold Road Mine was served by a small community at the mine site. Lessees operated Gold Road mine until 1922 and then intermittently until 1937. By that time, with the increased price of gold, USSR&M Co. built a new mill, rehabilitated the mine and developed additional reserves by underground exploration. USSR&M Co-operated the mine successfully until closure in 1942 by Presidential Order L-208 suspending non-essential industry in favor of the World War II war effort. The Gold Road Mill was dismantled and moved to Bayard, New Mexico to process zinc ore not affected by L-208. As of the 1942 closure, the Gold Road Mine is credited with production of some 612,000 ounces of gold from 1,690,000 tons of ore.

Analysis conducted after the Second World War found that, due to high labor costs, Gold Road Mine was not a profitable endeavor. Twenty years later the owner, USSR&M Co, became insolvent. Subsequent survivor companies held the property but never consolidated sufficient ownership to allow reopening of the mine.

In 1993 Addwest Minerals consolidated ownership of the claims covering the Gold Road vein system. After acquiring the necessary permits, a new processing plant was built, a 6,200 foot long decline was driven to access the lower levels using rubber tired mining equipment and reserves sufficient to resume production in 1995 were developed. From March 1995 through June 1998, Addwest produced and sold 87,624 fine ounces of gold from 381,878 tons of ore milled (average grade 0.23 opt). Mohave Desert Minerals acquired the mine in 2010 and the mine re-started in July of 2010 and operated until early 2015 producing 40,470 ounces from 293,305 tons of ore mined and 395,571 tons of tailings processed. The tailings processed consisted of the French Tails (located on the Gold Road Property) and tailings purchased for a $1.00 per ton from the United Eastern Mine. Shown in Table 6-1 is the historic production for Gold Road. Figure 6-1 shows the areas mined (in red) at Gold Road. The area mined shown in red represents approximately 62% of the total area developed by workings.

Table 6-1 Historical Gold Production of the Gold Road Mine

Time Period	Owner/Operator	Average Grade (Opt)	Tons	Ounces Produced
1900-1911	Gold Road Mining	0.6	327,165	196,229
1912-1916	USSRM	0.37	500,104	185,033
1922-1923	Lessees	0.6	61,317	36,734
1926	Lessees	0.63	2,847	1,794
1935-1942	USSRM	0.24	800,000	192,000
1996-1998	Addwest Minerals	0.23	381,878	87,624
2010-2015	Mohave Desert Minerals	0.16	293,305	46,626
Totals		**0.32 (9.92 g/t)**	**2,366,616**	**746,040**

6.1 Historical Resources

The following discussion reports resources that are considered historical by the NI 43-101 reporting standards. They do not conform to NI 43-101 reporting standards and should not be relied upon or interpreted as such. A QP has not done sufficient work to classify the historical estimates as the current mineral resources and the Company is not treating the historical estimates as current mineral resources. They are presented here for background information and as an example of the productivity potential of the mine.

Table 6-2 contains the historical resources for the Gold Road Mine. Figure 6-2 illustrates a long section of Gold Road showing the locations of historical resource blocks. To date, insufficient work has been completed to classify historical estimates as current mineral resources. This information is presented here for informational purposes only.

Table 6-2 Historical Resource Blocks

Blocks	Within 50 ft of Samples			50 to 100 ft from Samples			100 to 200 ft from Samples		
	Tons	Grade*	Ounces	Tons	Grade*	Ounces	Tons	Grade*	Ounces
W	23,185	0.294	6,816	23,185	0.294	6,816	47,187	0.295	13,920
C	67,558	0.237	16,041	27,037	0.205	5,543	54,564	0.205	11,186
E	39,067	0.201	7,852	24,598	0.209	5,141	27,306	0.246	6,717
FE	5,778	0.212	1,225	5,778	0.212	1,225	11,556	0.212	2,450
RT	5,694	0.303	1,725	6,426	0.209	1,343	2,586	0.208	538
Total	**141,282**	**0.238**	**33,630**	**87,024**	**0.231**	**20,068**	**143,199**	**0.243**	**34,811**
Grams/ton		**8.1**			**7.9**			**8.3**	

*ounces per short ton

There is over a century of sampling data for the Gold Road mine. The sample locations and results of most of the samples have been plotted on maps and most have been digitized and are captured on spreadsheets. The quality of these samples is unknown The samples used in the Gold Road historical resource estimates consist of channel samples, stope samples and core samples. The normal channel sample spacing along the development workings was five (5) ft (1.5 m) along strike with the sample being taken perpendicular to the strike. Vein widths were measured at all sample locations. Stope samples were collected every ten (10) ft (3.0 m) as grab samples from the muck pile. Grab samples also were collected from the haul trucks and these, in aggregate, represented the estimate of shipped grade.

To attempt to establish the quality of historical sampling an underground sampling program has been recommended to confirm historically reported grades and vein widths by duplicating as closely as possible the locations of the historical sampling. As the locations are not known exactly, large lengths of drift will be sampled and the geostatistics of the historical sampling and the new sampling that will be will be compared. If it can be demonstrated the two sample populations are similar, the QP will accept the historical sampling as of sufficient quality to estimate as a minimum inferred resources. As an additional check on the quality of the historical assays, reconciliation of stope production records and historical sampling will be reviewed to assist in the effort to confirm the historical sampling is of sufficient quality to be use in a resource estimation consistent with industry standards. Should this effort be successful, an NI 43-101 mineral resource will be estimated. The results of this estimation are reported in Section 14 of this report.

Figure 6-1 Historical Long Section Showing Stope Outlines

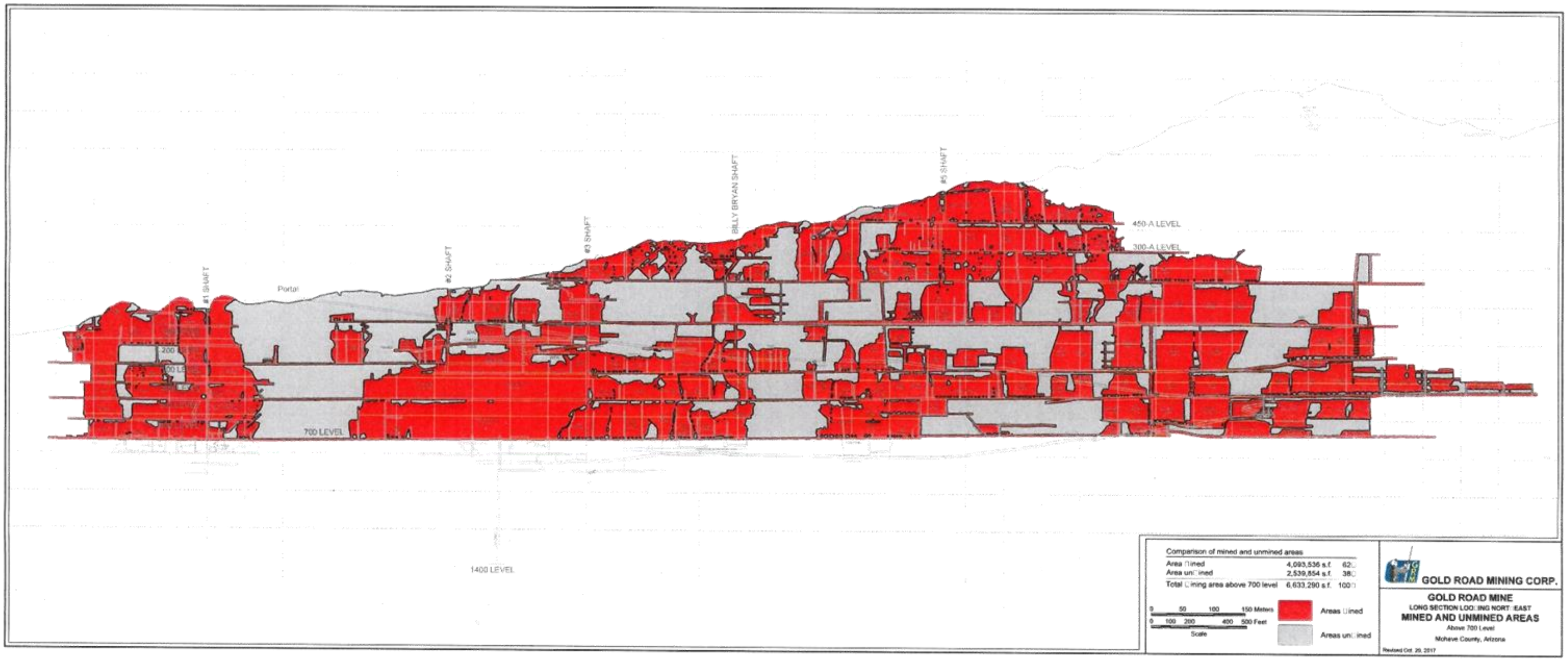
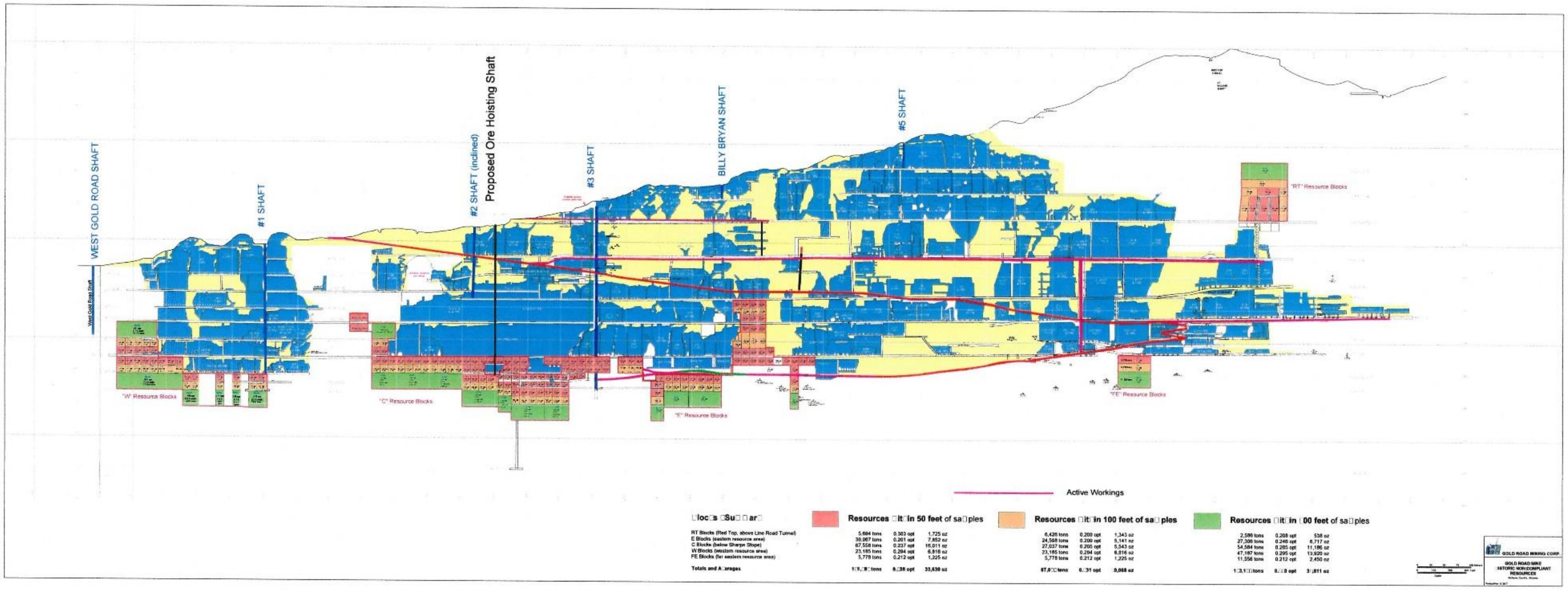


Figure 6-2 Historical Resource Blocks Long Section



7. Geology

The following section was taken from the unpublished report "Technical Report on the 2005-2007 Exploration Program at the Gold Road Mine Property" written by George F. Klemmick and Guilinger (2017). The Gold Road Mine is within the Oatman district.

7.1 Regional Geology

The Oatman mining district lies in northwestern Arizona on the western flank of the Black Mountains, a fault-bounded range situated near the eastern edge of the Basin and Range Province. The Black Mountains are composed of a sequence of rhyolitic to basaltic Tertiary volcanic rocks which rest unconformably on Proterozoic-age metamorphic basement rocks. The volcanic sequence is late Oligocene to early Miocene in age (30-15 Ma) and is related to regional extensional tectonism. The volcanic sequence consists of flows, tuffs and agglomerates which accumulated to a total thickness of up to 5,000 ft (1,524 m). A small number of intrusive stocks, plugs and dikes of Tertiary age are present in the Oatman area and these have been tentatively correlated with equivalent extrusive units. The Proterozoic basement is locally exposed along the lower western flanks of the Black Range and consists of metavolcanic and metasedimentary schist and gneiss and metamorphosed granitic rocks.

Regionally, the volcanic units have a N20°W strike and a 10° – 35° easterly dip, attributed by some workers to regional tilting and, by others, to a "central volcanic edifice at Oatman or related to late magmatic doming". Exploration work in the 1980's suggested that the east-dipping volcanic units are due to rotation along a west-dipping, low-angle detachment fault near the Precambrian-Tertiary contact. This fault has not been identified in the field at Oatman, but a detachment fault exposed near Union Pass (about 15 miles (24km) north) and extending northward to Lake Mead, may project beneath the Oatman volcanic sequence.

The volcanic sequence in the Oatman region is cut by northwest-trending, moderate- to high-angle normal faults of moderate displacement, generally 300-600 ft (91-183 m). Faults displace all volcanic and plutonic units at Oatman, with the exception of the youngest basalt flows. Dominantly up-to-the-northeast movement on these faults has helped to elevate the area after the cessation of volcanic activity. These faults and associated fractures host the most important epithermal gold-bearing vein deposits at Oatman. The faults are especially closely-spaced and numerous around the Oatman area, where they host valuable gold deposits (Figure 7-1).

7.2 District Geology

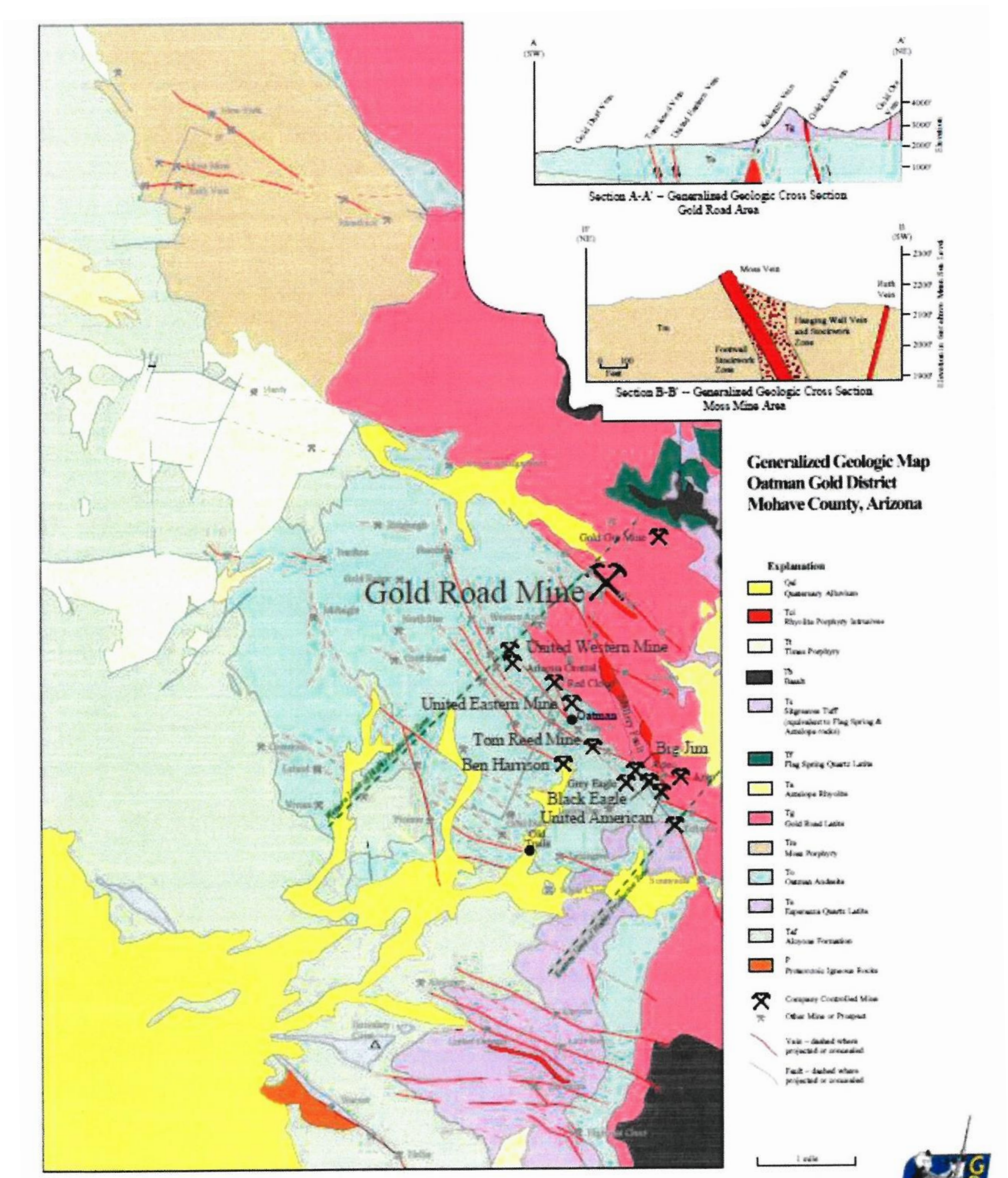
The Oatman mining district is dominated by Tertiary volcanic rocks representing at least four major cycles of late Oligocene to early Miocene volcanism. These cycles of volcanism are represented by the Lower Volcanics series, the Middle Volcanics series, the Upper Volcanics series and younger basalt-dominated volcanism (Figure 7-2). Rock units range from basaltic to rhyolitic in composition, but the bulk of the volcanic sequence consists of alkalic to subalkalic, intermediate rocks with latitic to andesitic compositions.

An eruptive center for at least some of the volcanic rocks is inferred to be near the town of Oatman. This inference is based on:

- The high concentration of rhyolite, dacite and latite dikes and plugs near Oatman
- The presence of two epizonal to hypabyssal plutons within a two-mile radius
- The rapid thinning of the volcanic sequence away from the town.

Closely-spaced northwest- to north-northwest-trending normal faults of moderate displacement cut the volcanic sequence and host the important gold-bearing epithermal veins of the district. The mineralized veins generally have a quartz-calcite-adularia-gold (electrum) mineralogy. Two of the important vein-hosting structures, the Gold Road vein system and the Tom Reed-United Eastern vein system, have accounted for about 90% of the total gold production in the Oatman mining district. At least twenty additional structures have been mapped in the area. They remain poorly explored but highly prospective (Figure 7-1).

Figure 7-1 Geological Map of the Oatman Area

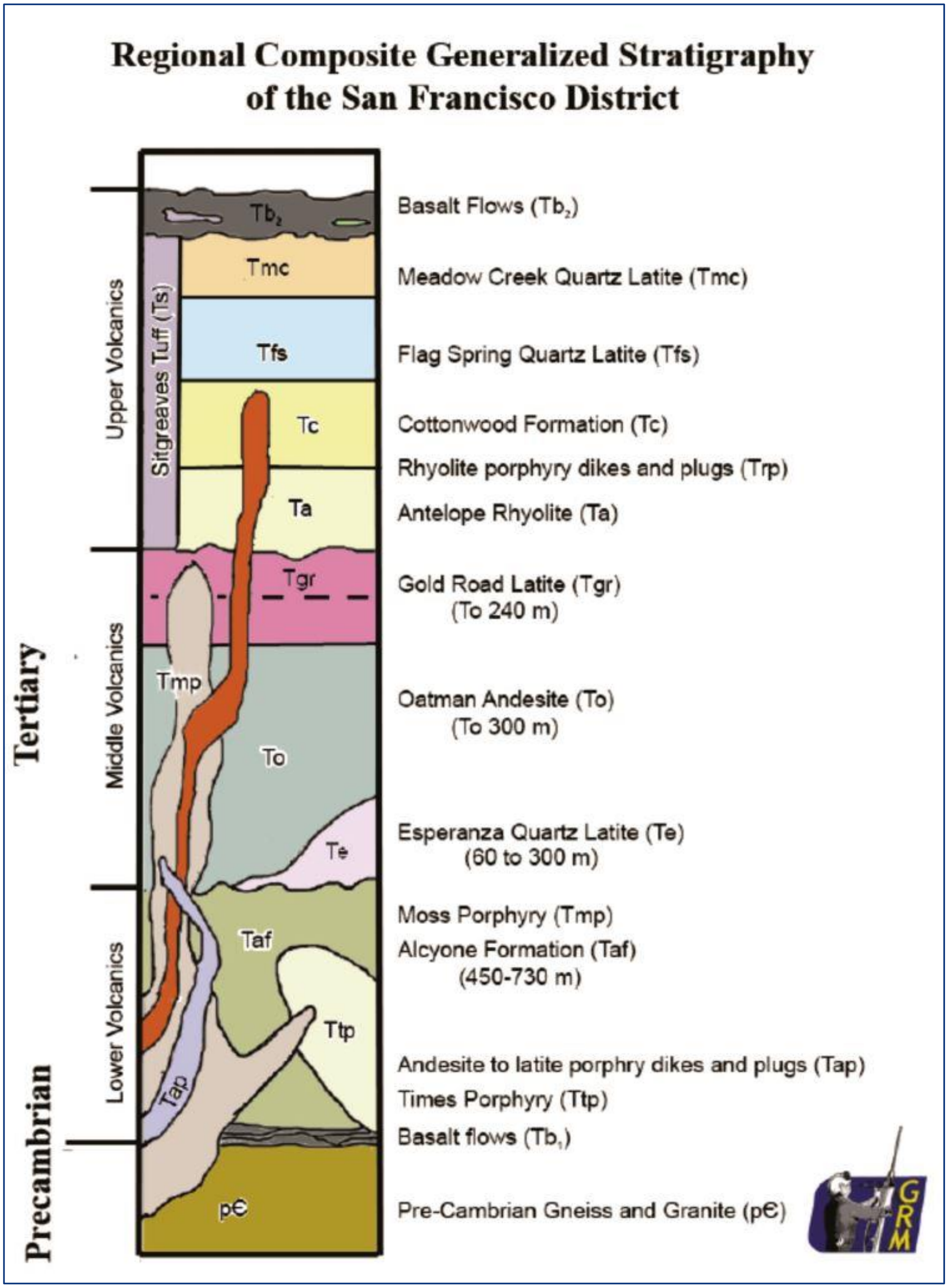


Generalized Geologic Map of the Oatman Gold District, Arizona

7.2.1 Stratigraphy

Detailed descriptions of the rock units exposed in the Oatman district are provided below and a stratigraphic column is shown in Figure 7-2.

Figure 7-2 Regional Composite Generalized Stratigraphy



Proterozoic Basement (pЄ)

Proterozoic basement rocks are inferred to underlie the entire Oatman district, but basement outcrops are rare. Only two exposures of the Proterozoic basement have been mapped in the Oatman mining district. One, near Boundary Cone, consists of a dark gray to brown, medium- to coarse-grained, biotite-rich diorite which has been partially silicified and oxidized. The other exposure consists of highly weathered and decomposed biotite gneisses and schists which have been intruded by a light colored, coarse-grained, equigranular granite.

Pre-Alcyone Formation Basaltic Rocks (Tb1)

At the Boundary Cone locality described above, Proterozoic basement is unconformably overlain by a thin series of basaltic sandstones and conglomerates of probable pre-Eocene age. These basaltic sedimentary rocks are in turn conformably overlain by thin basalt flows. This exposure possibly establishes a pre-Alcyone cycle of volcanism.

Lower Volcanics Series (Alcyone Formation)

The Alcyone Formation represents volcanic and volcaniclastic rock units that were deposited into a basin during the first cycle of volcanism at Oatman. Activity began sometime between the Eocene and lower Miocene and is represented by the eruption of a thick and aerially extensive sequence of welded trachyte crystal tuff. This tuff is overlain by landslide breccia deposited as inward-directed and inward-thinning tongues. A second eruption deposited more trachyte welded tuff in local depressions on the basement floor. Sedimentary tuff breccia was then deposited in the basin. Sedimentation continued but was gradually replaced by quartz latite volcanism in the form of lava flows and breccias filling the basin. Only minor amounts of gold have been produced from veins hosted by rock units of the Lower Volcanics series.

Middle Volcanics Series

After a period of erosion, a second cycle of volcanism occurred at Oatman. This eruption cycle is represented mostly by lava flows that range from dacite to basalt in composition, which together may have formed a broad shield volcano centered near the town of Oatman. These rocks are termed the Middle Volcanics series and they are middle Miocene in age. Rocks of the Middle Volcanics series host the most productive deposits in the Oatman mining district. The Middle Volcanic series is made up of the following units:

- Esperanza quartz latite (Te) - a very fine-grained, purplish-brown lava flow, or flows, that contain a few small phenocrysts of plagioclase and sparkling, elongate blades of biotite. The unit is encountered only along the southern edge of the Oatman mining district and is not seen in the Property area. It is also a relatively thin unit at 60 to 300 ft thick (18-91 m).
- Oatman latite (To) –hosts many of the important gold-bearing veins of the district including at the Gold Road Mine, the United Western property and possibly at the Gold Ore property. The Oatman latite is a porphyritic to rarely phaneritic, medium- to fine-grained, black to gray rock with abundant phenocrysts of plagioclase and pyroxene. It locally displays flow-breccia texture. Biotite is notably absent. Flow units average 40-50 ft (12-16 m) thick and may have vesicular flow tops and tuff and flow breccia sub units. The total thickness of the Oatman latite varies tremendously over relatively short distances. About five miles (8 km) south of Oatman, the Oatman latite is 300 ft (91 m) thick. Under Oatman it may be as thick as 2,200 ft (670 m). The Oatman latite tends to erode more easily than the overlying Gold Road latite.
- Gold Road latite (Tgr) - hosts important gold-bearing veins, especially at the Gold Road Mine and the Gold Ore property. The Gold Road latite is series of biotite and pyroxene-rich lava flows which rest conformably upon the Oatman latite. It is a resistant unit and commonly forms high, steep cliffs east of the town of Oatman and in the Gold Road Mine area. Flows of the lower Gold Road latite are composed of variable amounts of conspicuous biotite, pyroxene, plagioclase and quartz phenocrysts in a very fine groundmass. The flows of the lower unit are 30-100 ft (9-30 m) thick and commonly have vesicular tops and brecciated bases. They are generally light gray to reddish-brown in color. They are similar to the lower Gold Road latite flows, except that they are lighter in color, quartz is conspicuously absent, biotite is

less common and K_2O contents are higher. Individual flows of the upper Gold Road latite can be 150-200 ft (46-61 m) thick. The thickness of the Gold Road latite varies tremendously over short distances, similar to the Oatman latite.

Upper Volcanics Series

Following a brief period of faulting and erosion after deposition of the Middle Volcanic series, a third cycle of volcanism and volcaniclastic deposition began in upper Miocene time. This third cycle represents the Upper Volcanic series. Volcanic flows and tuffs of the Upper Volcanic series are latitic to rhyolitic in composition and were deposited on the flank of the faulted and partly eroded shield volcano. Flows and tuffs were deposited into basins and interfingered with the basin sediments. Significant gold deposits have so far not been identified in rocks of the Upper Volcanic series.

Olivine Basalt (Tb2)

After the cessation of Upper Volcanics series volcanism, there was a sustained period of erosion represented by an unconformity, followed by a period of basalt volcanism. These younger basalts inundated the area; they occur on both sides of the Black Mountains and cap the highest peaks of the range. The basalts are dark grey to black, aphanitic and studded with red-brown grains of weathered olivine. Flows are commonly 40-50 ft (12-16 m) ft thick and frequently have scoriaceous tops. The basalts accumulated to a thickness of at least 1,000 ft (305 m).

Intrusive Rocks

Below is a brief summary of the intrusive rocks in the Oatman area.

- Moss Porphyry (Tmp) – The Moss Porphyry is a north-northwest elongate, concentrically zoned stock, 2 miles by 4 miles (3 km by 6 km) in extent, which intrudes the Alcyone Formation, Oatman latite and Gold Road latite. It is located several miles northwest of the Property (Figure 7-1). The Moss Porphyry has an outer monzodiorite border, an inner porphyritic tonalite to quartz monzonite margin and a central tonalite-granodiorite core. The Moss Porphyry is considered to be a subalkalic to slightly alkalic intrusive equivalent to the Gold Road latite based on bulk chemical compositions and similar age dates. The historic Moss gold deposit is hosted by the Moss Porphyry.
- Times Porphyry (Ttp) – The Times Porphyry is an intrusive body of granite to alkali granite exposed in the western Oatman mining district, 3-4 miles (4.5-6.5 km) northwest of the town of Oatman. It is not present on the Property. Based on these age dates and similar bulk chemical compositions, it is believed that the Times porphyry is an intrusive equivalent to the Antelope quartz latite and Cottonwood Formation flows of the Upper Volcanics series.
- Rhyolite porphyry dikes and plugs (Trp) - Numerous dikes and plugs of white to cream-colored rhyolite porphyry occur throughout the Oatman district. They are characterized by a very fine-grained, equigranular groundmass of quartz and potassium feldspar and small, partially resorbed phenocrysts of quartz. The rhyolite porphyries are considered to be feeders for flow units contained in the Upper Volcanics series. There appears to be a strong spatial association of these rhyolite porphyry dikes to important gold-bearing veins in the district, including those at the Tom Reed, United Eastern, United Western, Gold Road and Gold Ore mines.
- Diabase dikes and sills - Several large diabase dikes and sills occur along the eastern flank of the Black Mountains along old Route 66. These diabase intrusions have mineralogy and textures similar to the olivine basalt flows described above. The dikes are most likely the feeders for some of the olivine basalt lava flows. A few lamprophyre dikes are scattered around the Oatman mining district also, especially around the Moss mine.

7.2.2 Structure

The major structures which host important gold mineralization in the Oatman district form a roughly radial pattern outward and southeast from an area centered near the Oatman Amalgamated prospect (Figure 7-1). This area may be near the center of a three-mile diameter (5 km) circular feature defined by a concentric fracture and joint set, inwardly-dipping faults and dikes and lineaments detected using Landsat satellite imagery and high altitude aerial photographs. These may reflect concentric fractures developed during the ascent or descent of magma within a near-surface magma chamber.

7.2.3 Mineral and Alteration

The gold-bearing mineralized bodies in the Oatman mining district are tabular to lens-shaped quartz+calcite+adularia veins localized along northwest- to north-northwest-trending faults and fractures. The structures typically dip steeply north with a few exceptions, notably the Gold Ore and Moss veins and several structures in the far southern portion of the district. Movement on these structures has been pre-, syn- and post-mineralization. Sense of motion on these faults is generally normal, with the dip-slip component greater than the lateral or strike-slip component. Gold-bearing veins typically occupy dilatant zones that have formed by relatively minor right lateral slip along gently curving fault planes. Generally, economically important gold mineralization has been found where fault-plane dilations have a concave-to-the-north or northeast curvature. Due to the differing mechanical properties of the formations, vein mineralization hosted by the Gold Road latite tends to occur as tight, tabular, fissure veins, whereas mineralization in the Oatman latite typically occurs as braided stringers or stockwork veins of complex geometry. Almost without exception, the most important gold mines in the district are from veins hosted by either the Oatman latite or Gold Road latite. In 2006, however, AMI discovered disseminated gold mineralization hosted by altered units within the Sitgreaves Tuff which assayed greater than 1 g/tonne (0.032 opt).

The major ore bodies on the Tom Reed-United Eastern vein system are typical of epithermal vein mineralization in that they have sharp tops and bottoms of bonanza grade in the vertical dimension and they may occur in periodic fashion along the host structure in the horizontal dimension. The total vertical extent of mineralization on this vein system is about 1,300 ft (396 m), with the bottom of mineralization occurring at the elevation of 1,500 ft (457 m) above sea level. The individual lodes averaged 425 ft (130 m) in length, 575 ft (175 m) in height and about 15 ft (4.5 m) in width. Between the mineralized quartz veins (lodes), the fault is barren and usually consists of clay gouge with some thin, uneconomic calcite and/or quartz stingers.

At the Gold Road Mine, the vein system is exposed on the surface for about 1.5 miles (2.4 km) and the ore-grade segment is nearly continuous for about 1 mile (1.6 km). The Gold Road Mine has been mined in the vertical dimension down to the elevation of 2,200 ft (671 m) above sea level. Mining has extracted ore from the Gold Road vein system for a horizontal distance of 7,000 ft (2,133 m) and for a vertical distance of 1,450 ft (442 m). Individual lodes on the Gold Road vein structure are up to 2,100 ft (640 m) in length, 620 ft (190 m) in height and vary in width from 3-7 ft (1-2 m) within the Gold Road latite and up to 23 ft (7 m) within the Oatman latite. Between the lodes, the Gold Road structure is also barren and usually consists of gouge with some thin, uneconomic calcite and/or quartz stingers.

Based partly on previous reports and exploration programs (which included detailed geologic mapping, geochemical analysis and petrographic studies), a distinctive alteration pattern is emerging. At the Property, potassium metasomatic, usually directly associated with gold deposition, typically overprints and is later than lower-pH alteration assemblages such as phyllic or illitic alteration. Potassic alteration appears to have not been fully recognized by workers studying the vein systems at Oatman in the past. Geochemical analyses of surface and underground rock chip samples and drill samples from drilling programs at the Gold Ore and United Western mines, suggest that potassium metasomatism is a significant component of the alteration and mineralization assemblages.

Wide phyllic or illitic alteration blooms occur locally in Oatman latite and Upper Volcanics series units, especially in the hanging wall of mineralized vein structures. These alteration types are much more spatially restricted in the Gold Road latite, although they are still plainly evident. Potassic alteration mineral assemblages typically grade outward from vein margins in both the hanging and footwalls for up to 20 ft (6 m) at the Gold Road Mine. Silicification is common directly adjacent to productive veins and usually extends outward from the vein for a few ft. Propylitic alteration is ubiquitous in the district and is not a useful guide to ore. Argillic and advanced argillic alteration also occurs in the Oatman district, but they appear to be associated mostly with retrograde processes or weathering of the lower-pH alteration assemblages rather than with hypogene gold mineralization.

7.3 Property Geology

The mineralization at the Gold Road Mine consists of quartz-calcite-adularia veins within the northwest-trending Gold Road fault zone. The fault zone can be over 150 ft (46 m) wide and quartz vein(s) may occupy one or more strands within the structure. Vein strands usually occupy the footwall, hanging wall or a central portion of the structure, but strands may occur in two or all three of these positions within the same area. Where the fault zone is narrow (such as areas within the Gold Road latite) vein material may occupy the entire structure.

The “main” Gold Road vein occupies a strong fault fissure, typical of the district. This structure was formed by several separate movements before, during and after gold mineralization. The strike of the sinuous vein varies from N50°W to N66°W and generally dips 65° to 85° to the northeast, though locally the vein can be vertical or dip steeply to the southwest. The vein system crops out continuously for about 7,500 ft (2,286 m) on the Property (Figure 7-3), including a segment that is in ore grade mineralization on the surface for over a mile (1,524 m). Most of the ore has been mined in wide lenses within dilatant zones of the vein structure. The wider dilatant zones of the vein may be related to areas of north- to northeast-curving concavity along the sinuous normal fault.

The character of the Gold Road vein varies considerably along its vertical extent. In the lowest stratigraphic exposures, the vein consists of fine-grained chalcedonic and banded quartz in a braided or complex stockwork vein system up to 30 ft (9 m) wide hosted in Oatman latite. At stratigraphically higher exposures where the vein is hosted in Gold Road latite, the vein is a tabular, fissure-like body typically 3-7 ft (0.9-2.1 m) wide. Historically, high-grade ore has only been mined from the Gold Road vein within Oatman latite and Gold Road latite. At the highest exposed levels, the Gold Road vein cuts lithologies of the Upper Volcanics series (Sitgreaves Tuff and Antelope quartz latite). This upper zone generally coincides with the Red Top prospect area. Mineralization at Red Top consists of very fine grained chalcedonic quartz and siliceous sinter deposits reflective of the uppermost levels of a low-sulfidation epithermal system, near the paleosurface or paleo-water table. Quartz mineralization at Red Top is several inches to a foot (0.3 m) wide and does not contain significant gold mineralization. Although the character of Gold Road vein varies depending on host rock, the changes in the vein at formation contacts are not sharp and abrupt. There is a gradual narrowing of the vein system upward.

The mineralogy of the Gold Road vein is typical of the district. The deposit is interpreted to be a low-sulfidation, epithermal vein deposit. The vein consists mostly of quartz with local concentrations of calcite and adularia. At least five major stages of quartz deposition are present in the vein. The last two stages of quartz, which consist of pale green to deep honey yellow, fine-grained chalcedonic silica and breccia, appear to have accompanied most of the ore grade gold in the vein.

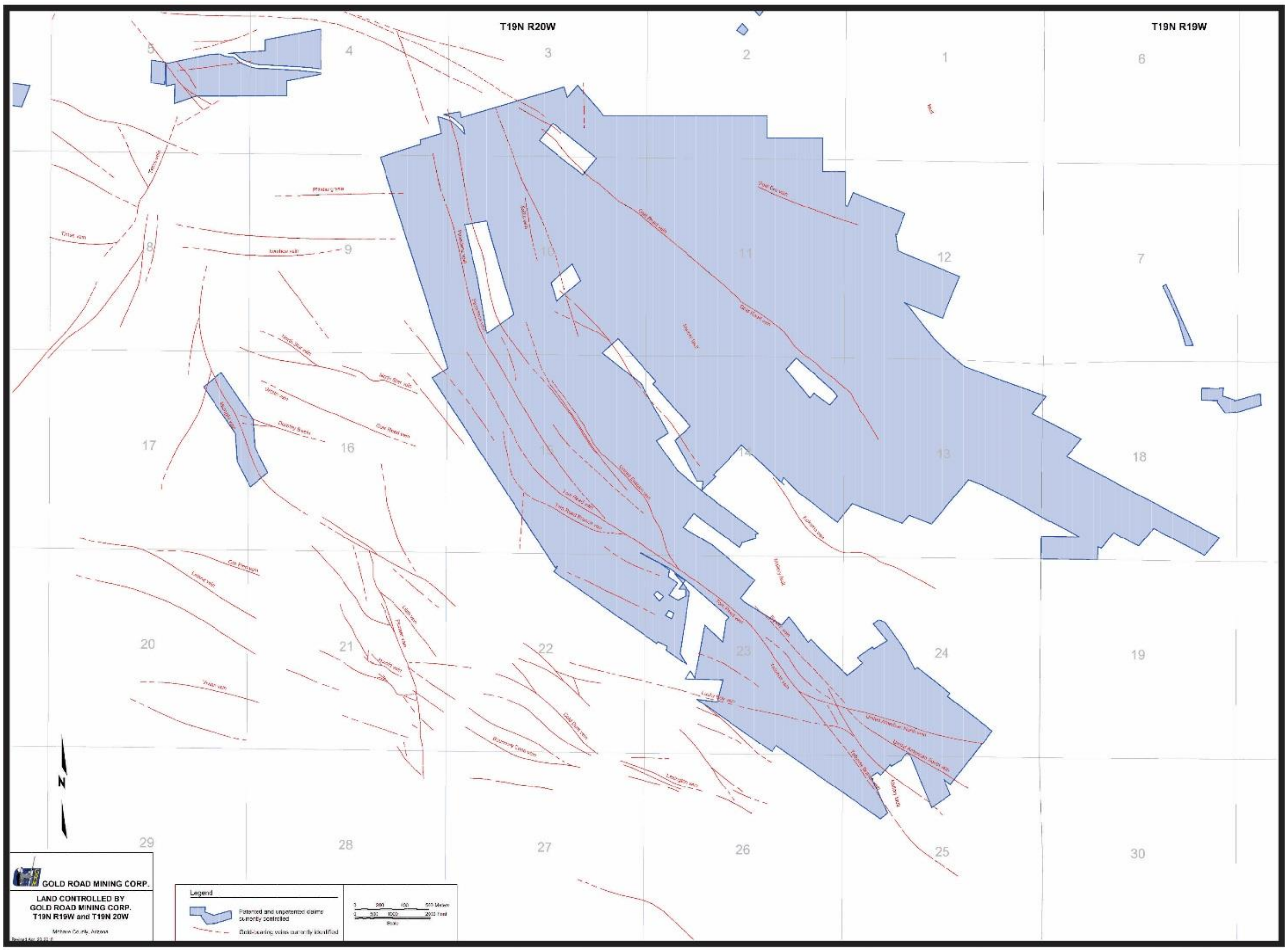
Gold is the most valuable constituent of the ore at the Gold Road Mine. Silver occurs with the gold in a ratio of approximately one part gold to one part silver on average. Sulfides are rare in the ore, but do occur in higher grade shoots. Fluorite is locally present. High-grade ore frequently contains very fine-grained free gold, copper sulfides or carbonates and, rarely, cerargyrite (AgCl) in thin quartz bands. Most gold is microscopic.

Individual ore shoots within the Gold Road vein system can range in size from 50,000 to 500,000 tons and range in width from 3 to 30 ft (1-9 m). They can extend from 100 to 1,600 ft (30 to 488 m) along strike and from 200 to over 1,000 ft (60 to 300+ m) down dip. To date, mining has extracted ore from the Gold Road vein over a horizontal distance of approximately 7,000 ft (2,133 m) and a vertical range of 1,400 ft (426 m).

The 500 and 700 Levels of the Gold Road Mine were stoped continuously for 1,000 ft (304 m) along the vein in the Number 3 shaft area.

The deepest ore-grade gold mineralization so far encountered at the Gold Road Mine was sampled on the 800 Level, which corresponds to an elevation of 2,200 ft (670 m) above sea level. The other principal mines in the district, the United Eastern and the Tom Reed, were productive between 1,500 and 2,800 ft (457 to 853 m) above sea level. If the Gold Road and Tom Reed-United Eastern vein systems are part of the same mineralizing

Figure 7-3 Vein Geology and Gold Road Mine Corp. Claims



event and the bottom of mineralization (or boiling zone interface) is similar, the Gold Road Mine may have an additional 700 vertical ft (213 m) of potentially mineralized ground to be explored. Diamond drilling successfully discovered high-grade gold mineralization down-dip on the Gold Road vein, but more drilling needs to be completed to fully realize the potential in this direction.

A layer in the upper portion of the Gold Road latite, approximately 200 ft thick (61 m) and known informally as the "Red Sill", has historically been the upper limit of mining (Figure 7-7c). This layer is reported to dip 25° to the southeast, but this has not been verified. Rock units that overlie the Red Sill (principally the Sitgreaves Tuff) have been sparsely prospected in the past, but a raise on the eastern end of the Line Road Tunnel appears to have fully penetrated the Red Sill and encountered ore-grade mineralization on the other side of the structure. If this is correct, it would suggest good exploration potential in this area.

An inferred post-mineral fault offsets the Gold Road vein at the northwestern end of the mine where the workings abruptly end. High-grade stopes bound the fault to the east. Ten reverse-circulation (RC) drill holes and five surface trenches in this area attempted to locate the inferred offset portion of the vein. The drilled area is termed the Gold Road Vein-Northwest Extension prospect. The trenches and the first five drill holes were successful in locating the Gold Road vein and local moderate-grade gold was discovered. The last five drill holes were unsuccessful in locating the vein structure and were characterized by wide zones of unmineralized phyllic to illitic alteration. While post-mineral faulting probably plays a role, the phyllic alteration bloom may be a more significant factor. This area could represent a transition zone between a higher temperature (?), lower-pH, high-sulfidation hydrothermal system such as the AJ prospect (which is on-trend with the Gold Road vein system) and a lower temperature (?), near neutral-pH, low-sulfidation hydrothermal system such as the quartz-calcite-adularia-electrum epithermal veins of the Gold Road Mine.

Mineralization at the Gold Road Mine is inferred to be the remains of a fossil geothermal system. Siliceous sinter deposits, with visible ripples marks and plant casts (?), are present at the Red Top area. Alteration at Red Top also includes hypogene argillic alteration and silicification which overprints weak potassic alteration. This assemblage is contained within a slightly larger phyllic altered envelope. With increasing depth on the Gold Road vein system, potassic alteration becomes the dominant alteration type, with only relict patches of illitic- and phyllic- altered wall rock. With further increase in depth, the potassic alteration halo becomes limited to a narrow zone directly adjacent to the vein and illitic and phyllic alteration assemblages become larger and the more pervasive. In deeper, unexplored areas of the Gold Road vein system, illitic alteration (magnesium-rich) may become the dominant alteration type, similar to what is found at the Tom Reed-United Eastern vein system.

The ore and gangue mineralogy of the San Francisco (Oatman) vein deposits is remarkably simple. The only ore mineral is electrum, which generally assays around 650-800 fine. Base metal sulfides or sulfosalts are rare or completely absent from the veins; however, trace amounts of pyrite, chalcopyrite, sphalerite, galena and marcasite have been noted from veins in the district. Pyrite is more common in altered wall rocks adjacent to the veins. Major hypogene gangue minerals are white to multi-colored chalcedonic quartz, calcite, adularia, chlorite and minor fluorite, sericite and corrensite, a chlorite-group mineral. Quartz may take on a pale lime-green to dark honey-yellow color in high-grade gold mineralized zones due to chlorite and corrensite inclusions. Smith (1984) noted that electrum from the Gold Road Mine contained up to 0.15% tellurium. Supergene gangue minerals include minor gypsum, kaolinite, alunite, pyrolusite, psilomelane, hematite, limonite and rare malachite, azurite, and wulfenite. Supergene enrichment of the Oatman ores is extremely rare, but some wire gold and cerargyrite have been reported from near-surface oxidized zones at several mines in the district.

8. Deposit Types

The ore deposits of the district are low-sulfidation, quartz-calcite-adularia-electrum epithermal veins, with associated quartz stockwork veining and silicified breccias, hosted by Tertiary volcanic rocks. Bonanza gold grades are locally encountered. Typically, bonanza mineralization occurs as discreet ore shoots within the larger quartz body or lode. The district is of the type usually associated with extensional tectonic regimes, alkali-rich host rocks and restricted vertical ranges of mineralization. The vertical range of ore deposition is bounded by paleo-boiling zone interfaces, which constitute the “bottom” of ore and paleosurfaces or paleo-water tables, which form the “top” of the ore. Individual gold-mineralized quartz bodies may be separated from each other by barren fault gouge or breccias zones. Occasionally, this deposit type may grade upward into near-surface, hot springs-related gold-silver deposits characterized by siliceous sinter or opaline deposits. Similar district deposits include Bodie, California; Guanajuato, Tayoltita and Pachuca Real del Monte, Mexico.

9. Exploration

The exploration conducted on the Gold Road mine area has been over the history of the mine has been drifting on the vein and sampling along the face and back. Production sampling in stopes was also completed. Beyond the sampling maps, there is little information about exploration programs prior to 2005. From Q4 2005 to Q1 2007 Addwest Mineral Inc. (AMI) carried out a systematic, multi-faceted exploration program on the Gold Road property.

The program focused on understanding the subtle features of gold mineralization in the Oatman district through detailed geologic mapping, geochemical sampling and analyses (from surface and underground), drilling, and petrological and mineralogical studies. Detailed geologic mapping, mostly at 1:2400 scale, was completed over targeted areas of The Property. Five hundred rock chip samples were collected and submitted for multi-element geochemical analysis. The samples were collected from surface outcrop, accessible underground exposures, float, and surface trenches. 24 reverse-circulation (RC) drill holes and one (1) diamond core hole were completed, totaling 20,611 ft. From the drilling, 2,733 samples were submitted for geochemical analysis. In addition, 26 select samples were analyzed petrographically and by X-ray powder diffraction (XRD) methods to aid in alteration and mineralization studies. The results of the drilling will be discussed in Section 10 of this report.

10. Drilling

RPM has been provided no documentation of any drilling prior to Addwest Minerals Inc's (AMI) work in the early 1990s. The following discussion was provided by Guilinger (2017).

Starting in 2005 AMI drilled 24 reverse-circulation (RC) drill holes and 1 diamond core hole were completed, totaling 20,611 ft.

The drilling program explored the northwestern extension of the Gold Road vein system immediately northwest of the terminus of the Shaft #1 workings, which was among the highest grade stopes in the history of the mine. Addwest was successful in locating the Gold Road vein system, and hanging wall and footwall vein splays were encountered in most of these holes. Sheeted quartz vein zones up to 30 ft in true thickness were encountered in several holes, and generally occurring between and adjacent to the hanging and footwall massive quartz veins. Despite the presence of the large quartz zones, no ore grade gold was encountered.

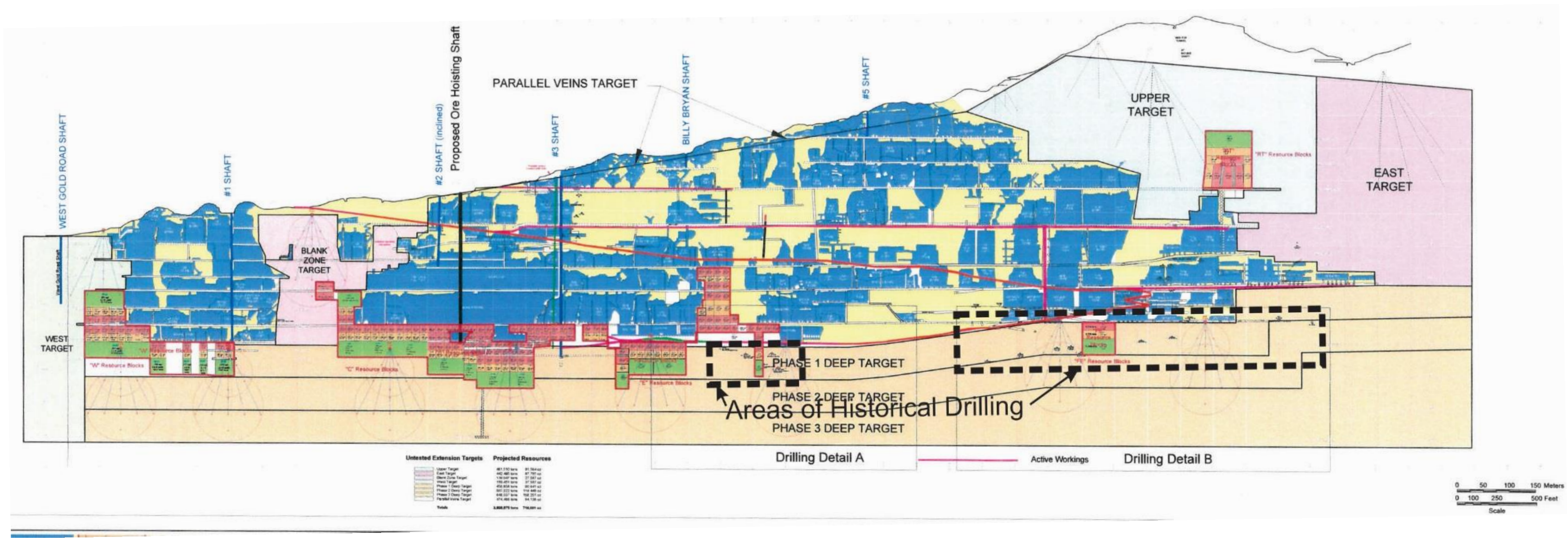
Underground core drilling took place at many sites within the mine at various times in the recent history of development and production. The drill results of those sites not subsequently mined are listed in the following Table 10-1 and shown on map Figure 10-1. It should also be noted that all of these holes are within area that is below the 900 level and into an area of excellent potential. The reported historical drill intercepts serve as a strong confirmation that both the vein and associated gold mineralization continue to depth. Note that the holes shown on the figure are the pierce point locations where the vein was intersected by the drilling.

No additional drilling has been completed since the 2005-2007 drill program.

Table 10-1 Underground Drill Results

Hole Number	Vein Interval ft (m)	Gold Grade oz/t (gms/t)
DD-32-1B	3.0 (0.91)	0.85 (26.36)
DD-32-3	3.0 (0.91)	0.06 (1.86)
DD-32-4	8.8 (2.68)	0.23 (7.13)
DD-32-5	6.0 (1.83)	0.02 (0.62)
DD-32-6	1.0 (0.30)	0.14 (4.34)
DD-900-2	1.0 (0.30)	0.11 (3.41)
DD-900-3	8.5 (2.59)	0.01 (0.31)
DD-900-4	3.5 (1.10)	0.19 (5.89)
DD-900-9	2.1 (0.64)	0.93 (28.84)
DD-700-10	3.5 (1.07)	1.08 (34.69)
DD-850-1	0.7 (0.21)	0.04 (1.41)
DD-850-2	4.0 (1.22)	0.26 (8.46)

Figure 10-1 Gold Road Historical Underground Drilling



11. Sample Preparation, Analysis, and Security

As most of the activity and mining at Gold Road was pre NI 43-101 requirements, there has been little documentation of the any of the work. In 2005 AMI began working on the property and the work they did is documented and appears to meet some of the standards established by CIM. Any work done prior to AMI's involvement is not documented at all.

11.1 Addwest Minerals Work

The AMI samples for exploration were collected from RC drilling under the supervision of a QP. The same QA/QC protocols were used for all soil, rock and drill samples. The drill samples were collected from a rotary splitter on the drill rig and sealed and packaged for shipment to a lab. AMI used three labs, ALS Chemex (Chemex), Sparks, Nevada, USA; Acme Analytical Laboratories Ltd. (Acme), Vancouver, British Columbia, Canada; or American Assay Laboratories Inc. (AAL), Sparks, Nevada, USA.

All samples collected within the 2005-2007 period were analyzed for gold using an industry standard, 1 assay-ton (30g) fire assay charge, or occasionally a 50g charge if high-grade gold was suspected, with an atomic absorption (A.A.) or gravimetric finish. Results were reported in parts per million (ppm). Trace element geochemistry was performed using standard ICP-AES (Inductively Coupled Plasma-Atomic Emission Spectrometry) methods beginning with a four- acid, near-total digestion of the sample matrix. A 27-element ICP-AES analysis package was chosen, which included the most important and useful trace elements such as silver, antimony, arsenic, beryllium, molybdenum, potassium, etc. Samples with geochemical content greater than the detection limit for the ICP-AES method were re-analyzed using standard assay methods for that particular element.

Since all the geochemical labs used by AMI during the 2005-2007 period are ISO-certified, they regularly employed in-house internal checks in the form of blank and standard samples inserted at frequent and regular intervals in the assay run, to ensure precision and accuracy. AMI also instituted additional quality control (QC) procedures including the insertion of standard and blank samples, manufactured by and acquired from independent labs, into the shipped sample batches at regular intervals. AMI also instituted routine check assaying programs and completed frequent re-analysis of pulps and coarse reject duplicates.

11.2 Current Sample Protocols

All the sampling and drilling data available for the Gold Road property are considered historical. While the AMI information is well documented and while well done and in part consistent with current industry standards, there are critical elements of the program that do not meet current industry standards. There are many places currently that are not accessible for sampling due to the mine being closed for several years. Also there are stope sampling records along with stope production records from all generations of activity. The objective of the new sampling will be to verify the older sampling is of sufficient quality to be used in a PEA resource estimation

The author has developed a protocol that will verify the historical data. The protocol will include:

- Sampling along drifts that have historical channel samples taken every 5 ft across the width of the vein.
- Sampling along long lengths of the vein on multiple levels and on drifts that were sampled before 1942 and after 1990.
- Compare the results of the two generations of samples using accepted geostatistical methods that would be employed when comparing twin drill holes.
- For the current sampling program two assays will be carried out, a fire assay and a cyanide assay. This is necessary because the majority of samples collected after 1990 were only assayed using cyanide. Below is a description of the cyanide soluble protocol used by the Gold Road mine lab during this period. This protocol was developed to replicate the recoveries in the mill. The protocol is:

- Mix 1.9 gms of cyanide to 1,000ml of water and Ph balance to 10.5 – 10.8 by adding caustic.
- Use 10 gms sample ground to 200 mesh and mix with 30 ml cyanide solution to make a 3 to 1 dilution in sample tube.
- Shake tubes and put on hot shake at 77°F.
- Shake by hand every hour for the first 4 hours and let it shake on the shaker table for a total shake of 8 hours.
- Let settle or filter the sample and read on AA.

In addition to the historical sampling, reconciliation of stope samples and drift samples with production records will be carried out. Reconciliation has been reviewed by other reports by Watts, Griffith, and McQuat (WGM) (1996) and Behre Dolbear (1997). Table 11-1 shows the reconciliation of selected stopes that were mined by AMI.

Table 11-1 AMI Reconciliation for Selected Stopes

Stope	Reserve Tons	Reserve Grade	Tons Mined	Mined Grade	Reserve ounces	Mined Ounces	% Tons Extracted	% Ounces Extracted
1	32,693	0.296	41,864	0.362	9,677	15,155	128%	157%
2	21,690	0.625	24,108	0.621	13,556	14,971	111%	110%
6	13,667	0.274	10,970	0.280	3,654	3,021	80%	83%
37	14,372	0.172	7,219	0.122	2,472	881	50%	36%
Total	**82,422**		**84,161**		**29,359**	**34,028**	**102%**	**116%**

This table incorporates a mine dilution of 24.6% at a grade of 0.84 gms Au/ton. This was the reported dilution for shrink stopes historically. While the overall tons mined are within 2% of the estimate, the ounces produced are 16% higher than the resource estimate. In more detail, the results are typical for a narrow vein, high grade gold mine with the reconciliation between reserves and production being of individual stopes being highly variable. It is important to look not only at the stope by stope variability but the variability with a larger sample size.

11.2.1 Sampling Description

In order to verify the historical sampling to use for the estimation resources that meet the industry standards as defined by CIM, RPM has developed a protocol to establish the veracity of the historical channel samples. The protocol recognized the fact that the exact location along the drift is not known and the type of sample (face, sill or back sample) is not known.

The verification sampling is as follows:

- A length of drift that is accessible is identified for sampling.
- Sample sites are marked every ten ft along the drift.
- The sampling surface is cleaned using compressed air.
- Samples are taken from the sill or the back using a chipping hammer.
  - Sample the entire exposed vein width. If the exposed vein width is greater than 4 ft, the sample may be split into two samples split into two equal length samples.
  - Samples weigh 2 to 3 kilograms.
- The samples are bagged and marked with a unique number and placed in a plastic bucket.
- Once the bucket is full, it is sealed and prepared for shipment.
- The sample length is measured and the sample is described.

- Once the samples were all collected and placed in buckets, they were shipped to ALS Chemex via USPS.

11.2.2 Security

The sample collection was carried out by samplers supervised by a QP. Once the samples were collected, they were marked with a unique number and placed in five gallon buckets. When the buckets were filled, they were sealed and brought to the surface under the supervision of the QP and placed in a locked storage facility. Once the sampling was completed, the samples were shipped to ALS Chemex using USPS. The samples were under the direct control of the QP from collection until delivery to the USPS.

11.2.3 Sample Preparation

Once ALS received the samples they were crushed, split and pulverized (Prep Code PUL 31) according to ALS protocols. From each sample two splits of 30 grams each were taken to for fire assay and cyanide soluble assay for gold and silver.

11.2.4 Analysis

Samples were analyzed by ALS for gold and silver using both fire assay and cyanide soluble assay techniques. The fire assay code used was Au-ICP21. The code for the cyanide soluble is Au-AA13. It will be necessary to compare the cyanide soluble assays from ALS with those reported by the Gold Road lab. The protocols are different and while comparison between the two may be approximated, the difference in protocols showed a low bias of the cyanide soluble gold of about 25% based on the verification sampling. The Gold Road lab assay protocols are discussed above in Section 11.2. The ALS protocols as described to the author over the phone from Mary Dougherty of ALS in Reno are as follows:

- A 30 gram sample is used for the assay
- A 0.25% cyanide solution is added along with a 0.05% NaOH solution to the sample
- The sample is leached for one hour at room temperature while being shaken
- The sample is then centrifuged to separate the solids and the gold content of the liquid is read with an AA machine.

As can be readily seen, the ALS protocols vary in sample size, cyanide strength, and leach time. Each of these factors may impact the final result and make direct comparison difficult. However comparing the results should establish a relationship between the two protocols and can be related to the fire assay results of the verification sampling.

12. Data Verification

The data available to be used to estimate resources are historical assays from channel samples taken during mining. These were generally face or back samples across the vein as the headings were advanced. The majority of the samples were taken every 5 ft along the drift. The author has had verification samples taken along two levels, the 700 Level and 840 Level. All verification sampling was supervised by a QP.

The historical sampling on the 700 Level was completed prior to 1942 and records indicate it was fire assayed in the mine lab. The historical sampling on the 840 Level was after 1996 and the assays are all cyanide soluble assays completed by the Gold Road lab using the protocols discussed in Section 11.2.

The verification samples were taken every ten ft along the drift either as a sill sample or a back sample across the exposed vein. Because the exact locations of the historical samples are known only from sample locations on a drift map, the author felt that sampling every 10 ft and then comparing the verification sample against a sample taken at approximately the same location and comparing with samples 5 ft on either side of the original sample location should provide sufficient information to assess the quality of the historical sampling. The comparison between the historical sampling and the verification sampling will be similar to that used to compare twinned drill holes; i.e., compare widths and composite assays and not individual assays.

12.1 Comparison Sampling 700 Level

The historical sampling on the 700 Level was pre-1942. The verification sampling was a back sample if the back was reachable or a sill sample if the back was too high to be accessible without assistance from ladders or equipment. All sill samples were taken between the rails and ties. All verification sampling was submitted with blanks and standards but no duplicates.

Tables 12-1 and 12-2 summarize the results of the sampling for both the original and verification sampling. Table 12-1 shows the intervals ≥ 0.10 oz. gold per ton and ≥ 20 ft in length for the verification sampling. Table 12-2 shows the intervals ≥ 0.10 oz. gold per ton and ≥ 20 ft in length for the original sampling. Figure 12-1 visual summary of Table 12-1 and 12-2. The 700 Level is dry and no water was encountered.

Thirty nine samples were taken covering a distance of 390 ft. If multiple samples were taken at one sample location, the results were composited using length weighted compositing. The verification sampling was compared to the 3 original sample closest to the verification sample location which where composited using length weighted compositing. The average of the fire assays for the 39 verification samples was 0.227 oz. Au per ton. The average of the corresponding original sampling was 0.180 oz. Au per ton.

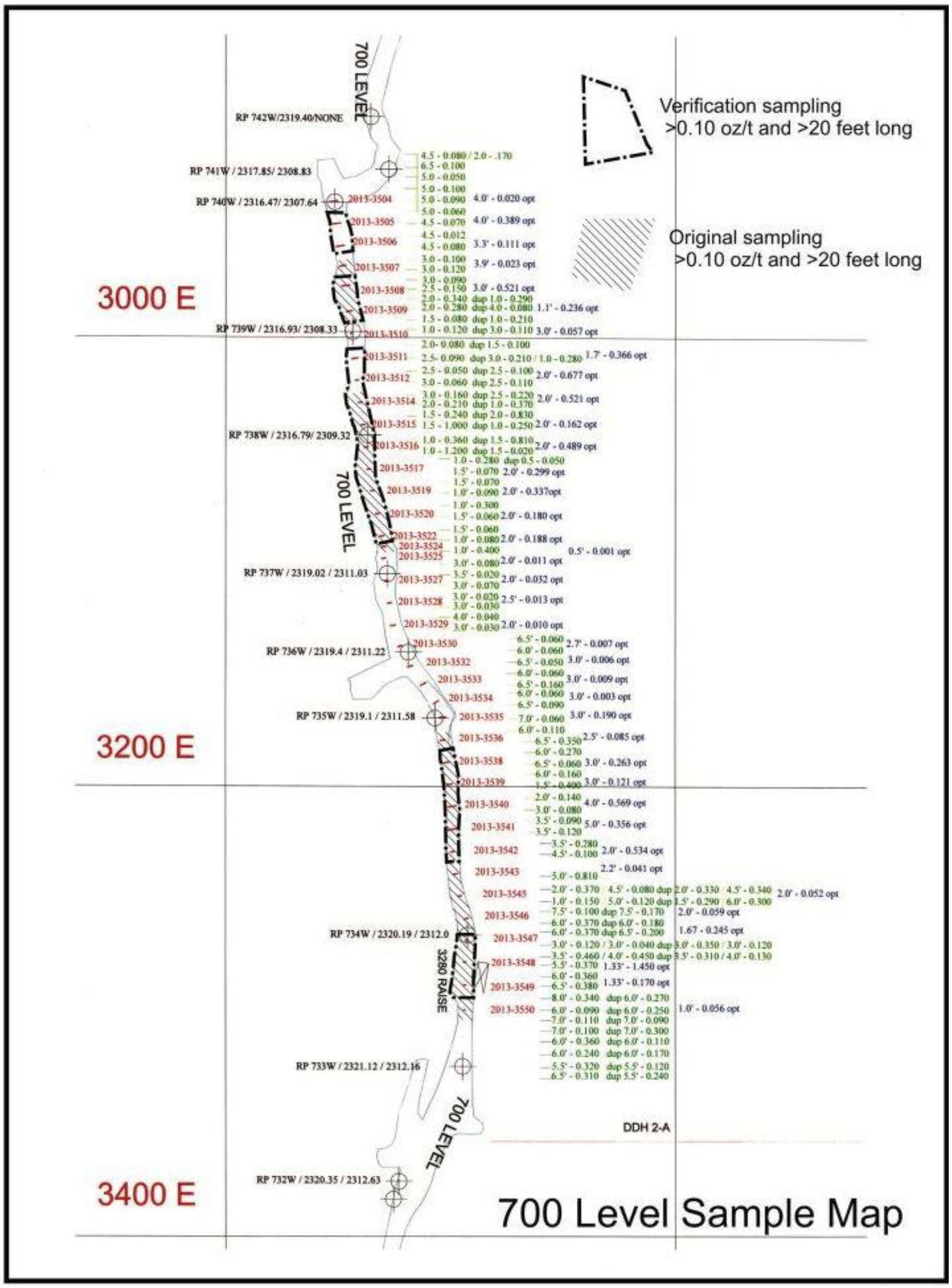
Table 12-1 Intervals >0.1 oz Au for Verification Sampling 700 Level

Interval					
From	To	Length Ft	Verification CN	Original FA	Verification FA
3505	3506	20	0.281	0.05	0.25
3508	3509	20	0.418	0.21	0.378
3511	3522	90	0.298	0.23	0.366
3538	3542	50	0.356	0.21	0.368
3547	3549	30	0.521	0.32	0.622

Table 12-2 Intervals >0.1 oz Au for Original Sampling 700 Level

Interval From	To	Length Ft	Verification CN	Original FA	Verification FA
3507	3509	30	0.289	0.18	0.26
3514	3524	80	0.229	0.26	0.272
3536	3550	125	0.281	0.26	0.308

Figure 12-1 700 Level Sample Map



It is the opinion of the author the verification sampling confirms the following:

- There is significant gold in the vein on the 700 Level as shown by both the historical sampling and the verification sampling.
- The verification sampling on a sample to sample basis does not correlate well with the original sampling both in terms of gold assays and location but over the entire sample length, the correlation is generally good.
- Although the number of samples is limited, the verification sampling indicates the original sampling may be biased low as can be seen in Tables 12-1 and 12-2.
- The verification sampling supports the use of the historical sampling for estimating a resource. Because the historical sampling appears to be biased low, an estimate using the historical sampling should be a conservative estimate.

12.2 Comparison Sampling 840 Level

The historical sampling on the 840 Level was completed by AMI in the late 1990s. The historical sampling was face sampling on 5 foot intervals. The verification sampling was sill samples collected across the vein width every 10 ft along the drift. These samples were collected using a chipping hammer to cut the samples. There was some water in places on the sill that may have affected the verification sampling.

The historical sampling was assayed by the mine lab. The assays were for cyanide soluble using the protocol described in Section 11.2 of this report. The verification sampling was assayed by ALS using both fire assay with an AA finish and cyanide soluble using the protocol described in Section 11.2.4.

Tables 12-3 and 12-4 summarize the results of the sampling for both the original and verification sampling. Table 12-3 shows the intervals ≥ 0.10 oz. gold per ton and ≥ 20 ft in length for the verification sampling. Table 12-4 shows the intervals ≥ 0.10 oz. gold per ton and ≥ 20 ft in length for the original sampling. Figure 12-2, a summary of Table 12-3 and 12-4, shows the layout of the sampling and both the historical results and the verification results for the cyanide assays along with a delineation of the mineralized zones.

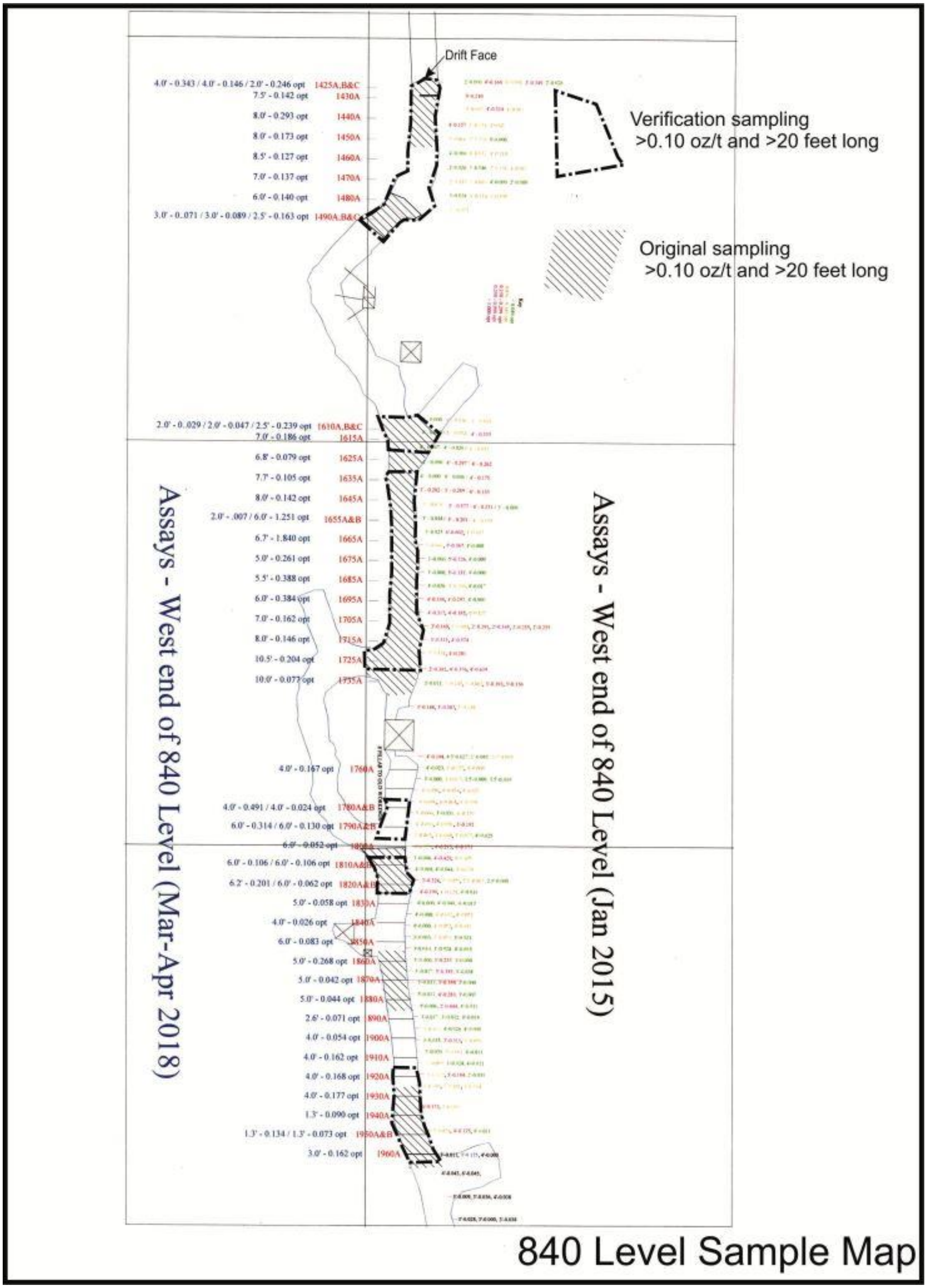
Table 12-3 Intervals >0.1 oz Au for Verification Sampling 840 Level

Interval					
From	To	Length Ft	Verification CN	Original CN	Verification FA
1425	1615	85	0.139	0.129	0.197
1635	1725	100	0.408	0.38	0.539
1780	1790	20	0.163	0.085	0.264
1810	1820	20	0.113	0.179	0.116
1920	1960	50	0.146	0.119	0.185

Table 12-4 Intervals >0.1 oz Au for Original Sampling 840 Level

Interval					
From	To	Length Ft	Verification CN	Original CN	Verification FA
1425	1450	40	0.175	0.146	0.235
1480	1735	160	0.293	0.296	0.394
1800	1820	30	0.091	0.168	0.106
1860	1880	30	0.14	0.182	0.163
1930	1960	40	0.139	0.126	0.146

Figure 12-2 840 Level Sample Map



Forty-three samples were taken covering a sampling distance of 440 ft. As can be seen in Figure 12-2 there is a gap in sampling where the vein is not exposed. If multiple samples were taken at one sample location, the results were composited using length weighted compositing. The verification sampling was compared to the 3 original sample closest to the verification sample location which where composited using length weighted compositing. The average of the cyanide soluble assays for the 43 verification samples was 0.181 oz. Au per ton. The average of the corresponding original sampling was 0.186 oz. Au per ton. The average for the fire assays was 0.243 oz. Au per ton. The average of all samples ≥ 0.10 oz. Au per ton was 0.229 oz. Au per ton for the verification sampling and 0.231 oz. Au per ton for the original sampling, and 0.30 oz. Au per ton for the fire assays.

It is the opinion of the author the verification sampling confirms the following:

- There is significant gold in the vein on the 840 Level as shown by both the historical sampling and the verification sampling.
- The verification sampling on a sample to sample basis does not correlate well with the original sampling both in terms of gold assays and location but over the entire sample length, the correlation is good.
- Although the number of samples is limited, there is evidence based on the original cyanide assays and the cyanide verification assays as compared to the fire assays and understanding the mill recovers 95% of the gold of a fire assay, that both the original and verification cyanide assays understate the recoverable gold. As some of the assays on which the resource is based are cyanide soluble assays from the mine lab, the author considers the resource estimates to be conservative.
- The verification sampling verifies the historical sampling and supports the use of the historical sampling for estimating a resource.

12.3 Independent Verification Sampling

As an additional check on the historical sampling and the verification sampling, RPM collected 5 independent duplicate samples on the 840 Level. Table 12-5 gives the results of this sampling demonstrating the presence of strong gold mineralization,

Table 12-5 Independent Verification Samples

Original Sample		RPM Sample	
Sample No.	**FA oz/t**	**Sample no**	**FA oz/t**
1645	0.16	RPM 1	0.21
1665	2.03	RPM 2	2.79
1685	0.43	RPM 3	0.58
1705	0.18	RPM 4	0.19
1725	0.22	RPM 5	0.16

12.4 QA/QC for Verification Sampling

Standards and blanks were included in the verification sampling. There were also a set of duplicate sample pulps that were run through the ALS lab. No samples were sent to a second lab.

12.4.1 Standards and Blanks

The standards and blank material was all obtained from MEG, Inc. in Reno, Nevada. Para obtained three Reference Standards and one preparation blank. They are:

- MEG-Au.12.27 (2.9 ppm Au + 607 ppm Ag)
- MEG-Au.09.07 (10.1 ppm Au)

- MEG-S107013X (27 ppm Au)
- MEG-PrepBlank (<0.003 ppm Au, Carbonate Matrix)

Table 12-6 shows the results of the Reference Standard assays that were submitted with the sample batches.

Table 12-6 Reference Standard Results

	MEG-Au.12.27		MEG-Au.09.07		MEG-S107013X	
	Cert Value	**Assay Value**	**Cert Value**	**Assay Value**	**Cert Value**	**Assay Value**
	ppm	ppm	ppm	ppm	ppm	ppm
	2.9	3.5	10.1	10.6	27	26.9
	2.9	2.62	10.1	10.45	27	26.8
	2.9	2.93	10.1	10.45	27	26.7
	2.9	3.08	10.1	10.4	27	25.9
	2.9	2.59	10.1	10.2		
			10.1	10.3		
Average	**2.9**	**2.94**	**10.1**	**10.4**	**27**	**26.6**
STD Dev	**0.258**	**0.33**	**0.355**	**0.12**	**0.699**	**0.40**
95% Confidence	**2.415 to 3.446**		**9.4 to 10.8**		**25.5 to 28.3**	

A review of Table 12-6 shows that the results for Standard MEG Au 12.27 show one sample outside of the 95% confidence limit but within 3 three standard deviations for the mean. The remaining samples are all well within 2 standard deviations of the mean. All these results were deemed acceptable. For Standard MEG-Au.09.07 all samples fall within 2 standard deviations of the mean but all the sample are higher than the mean. All these results were deemed acceptable. The results for Standard MEG-S107013X all are within 2 standard deviations of the mean and all samples were below the certified value. These results are deemed acceptable

A total of 11 preparation blanks were submitted with the samples and all returned value of <.05 ppm. There is no evidence of any contamination during sample prep.

12.4.2 Duplicate Sampling

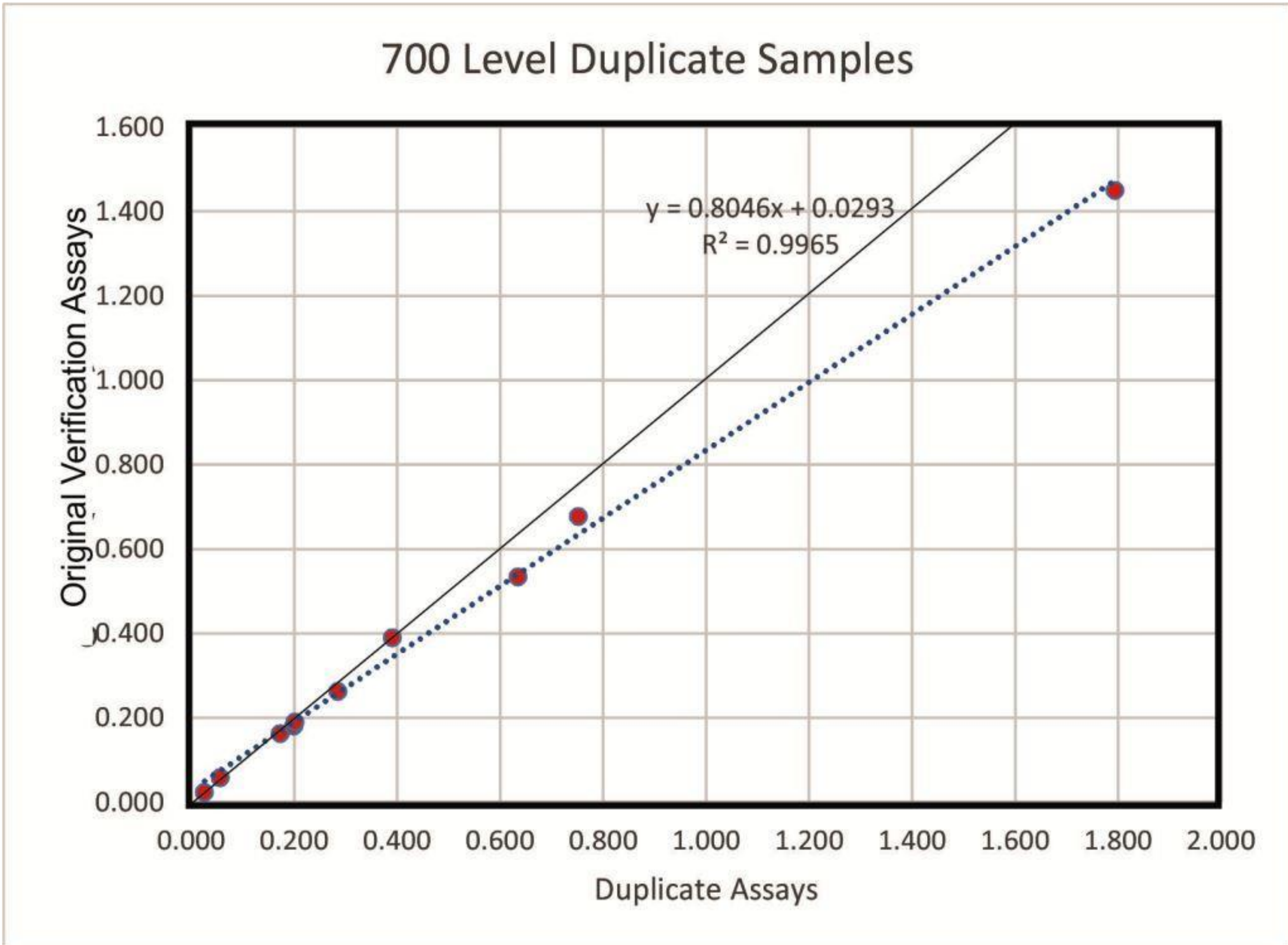
A total of 11 sample pulps along with two blanks and two standards were submitted to the primary lab for assay checks. Figure 12-3 show the results of the duplicate assays. These results were deemed acceptable.

12.5 Verification Conclusions

The Author opines the historical sample results which includes samples collected prior to 1942 which were fire assayed, and the samples collected after 1996, which were assayed by the mine lab using cyanide soluble assay protocols have been verified as being of sufficient quality to be used in estimating a resource for the Gold Road Mine.

Further, the Author opines the pre-1942 samples may be biased low and the use of cyanide soluble assays does not reflect the total gold in the samples when compared to the fire assays. The average gold solubilized by the cyanide is about 76% of the gold in the fire assays. The mill is expected to recover 95% of the gold delivered to the processing plant. The cyanide soluble assays also under estimate the potential gold available and the resource estimation is considered conservative.

Figure 12-3 Duplicate Sample Analysis



13. Mineral Processing and Metallurgical Testing

The gold in the Gold Road ore is present as very-fine, well-dispersed electrum containing about 75% gold. The principal gangue mineral is chalcedonic quartz. Minor gangue minerals are calcite and chlorite. Trace amounts of pyrite and base-metal sulfides are sometimes present.

Gold Road ore was processed from 1937 to 1941 in a 400-ton/day-capacity plant that incorporated counter-current decantation (CCD) and Merrill-Crowe precipitation. During this period the plant processed about 650,000 tons ore grading about 0.20 ounces gold/ton. The plant was closed during the Second World War and eventually scrapped. Production data for the old plant is shown in Table 13-1. The grind was 80% passing 200-mesh (75 µm) and the gold recovery was about 92%.

Testwork conducted in 1993, prior to the construction of the present 500-ton/day-capacity carbon-in-leach (CIL) plant, is shown in Table 13-2. This testwork was conducted on three different ore samples (with grades of 0.33-, 0.35-, and 1.47-ounces gold/ton) at two different cyanide strengths (1.00- and 0.25-grams NaCN/liter) and at two different grinds (80% passing 53- and 37-µm). Based on the testwork, the average recovery, using 0.25-grams NaCN/liter and a grind of 80% passing 45-µm, would be expected to be about 93.5%. The testwork also showed that the ore is exceptionally hard with an average Work Index of about 25 kWh/ton.

Operating data for the present plant is shown in Table 13-3. For the first two years of operation, from 1996 to 1998, the plant processed close to 400,000 tons of good-grade (0.23 ounces gold/ton) Gold Road ore. Thereafter, up to the present time, the plant has processed about 700,000 tons of a combination of low-grade Gold Road ore and marginal-grade tailings from nearby mines. When processing good-grade Gold Road ore the plant matched the 1993 testwork results with a gold recovery of 95%. The cyanide strength used in the plant is about half that of the testwork, at 0.25-lb NaCN/ton solution; cyanide and lime consumptions have been about 1 lb/ton ore and 2 lb/ton ore respectively.

Table 13-1 Old Plant Production Data

Year	Ore Source	Plant Circuit	Grind (80% passing, mesh)	Quantity (tons)	Grade (opt)	Recovery (percent)
1937	Gold Road Mine	CCD & Merrill-Crowe	200	100,303	0.205	92.5
1938	Gold Road Mine	CCD & Merrill-Crowe	200	131,412	0.207	91.5
1939	Gold Road Mine	CCD & Merrill-Crowe	200	130,636	0.194	89.5
1940	Gold Road Mine	CCD & Merrill-Crowe	200	144,442	0.185	92.5
1941	Gold Road Mine	CCD & Merrill-Crowe	200	147,419	0.212	91.7
Total				**654,212**		

Table 13-2 Testwork Conducted by IC Technologies Inc. in 1993

Parameter	Units	Composite # 1	Composite # 2	High Grade
Ore grade	opt	0.33	0.35	1.47
Ball mill work index	kWh/ton	21.8	25.5	
Leaching 24 hours, 50% solids, 1 g/L NaCN				
Grind P_{80} of 270 mesh (53 µm)				
Recovery	percent	90.1	93.0	
Cyanide consumption	lb/ton	1.18	1.21	
Grind P_{80} of 400 mesh (37 µm)				
Recovery	percent	95.9	95.1	
Cyanide consumption	lb/ton	1.44	1.58	
Leaching 24 hours, 50% solids, 0.25 g/L NaCN				
Grind P_{80} of 270 mesh (53 µm)				
Recovery	percent			96.8
Cyanide consumption	lb/ton			1.07
Grind P_{80} of 400 mesh (37 µm)				
Recovery	percent			97.6
Cyanide consumption	lb/ton			1.01

Table 13-3 New Plant Production Data

Year	Ore Source	Plant Circuit	Grind (80% passing, mesh)	Quantity (tons)	Grade (opt)	Recovery (percent)
1996-1998	Gold Road Mine	CIP	325	381,878	0.229	96.1*
2010	Gold Road Mine			59,857	0.084	
	French Tails			2,359	0.004	
	Combined	CIP	325	62,216	0.083	91.0
2011	Gold Road Mine			111,695	0.090	
	French Tails			12,411	0.040	
	Combined	CIP	325	124,106	0.085	93.7
2012	Gold Road Mine			66,016	0.103	
	French + United Eastern Tails			85,877	0.043	
	Combined	CIP	325	151,893	0.069	83.1
2013	Gold Road Mine			15,585	0.371	
	United Eastern Tails			151,678	0.038	
	Combined	CIP	325	167,263	0.070	81.2
2014	Gold Road Mine			35,363	0.123	
	United Eastern Tails			140,273	0.029	
	Combined	CIP	325	175,636	0.048	80.2
2015	Gold Road Mine			4,788	0.140	
	French Tails			2,975	0.036	
	Combined	CIP	325	7,763	0.101	89.0
2016	Coyote Mine, Nevada	CIP	325	7,145	0.079	93.5
Total				**1,077,900**		

*1998 recover

14. Mineral Resource Estimates

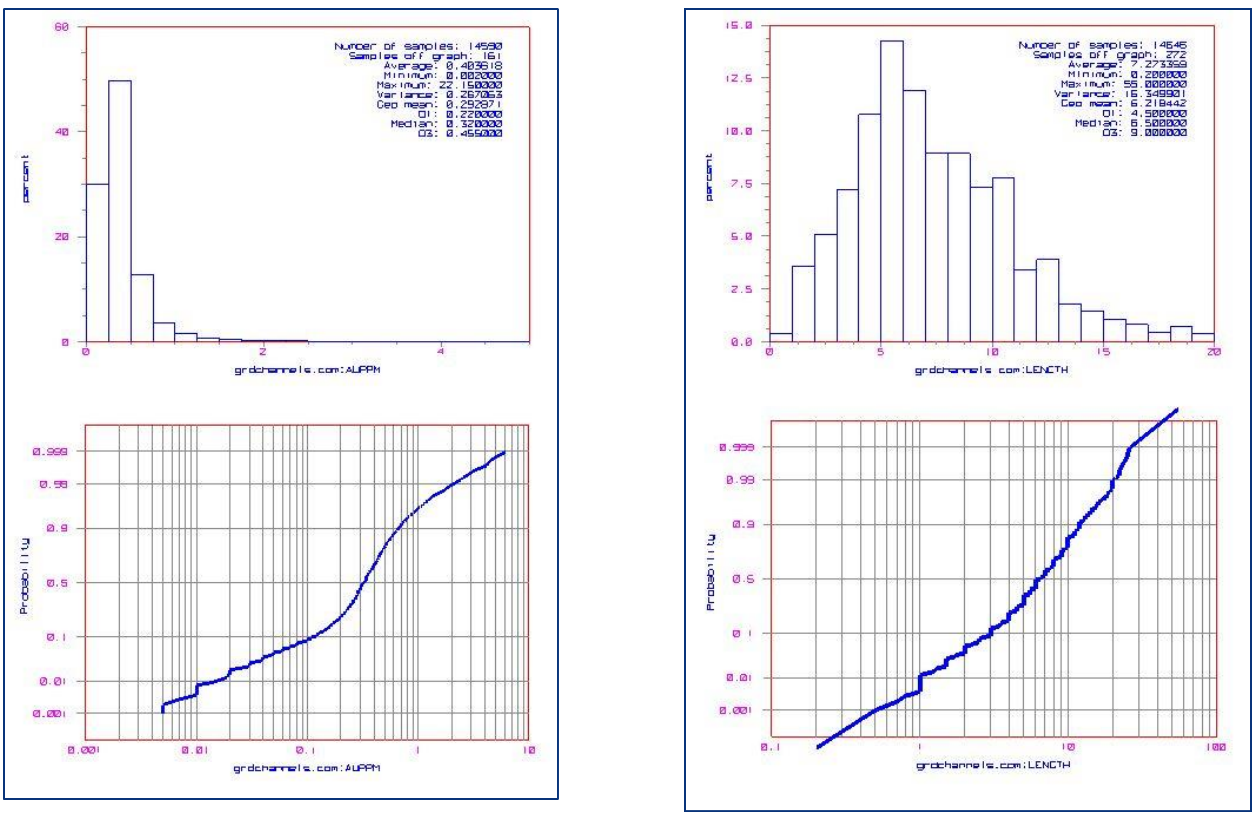
14.1.1 Database

Resources were supported by 14,768 channels and 16 drill holes. The database contains 19,400 assay data excluding the 16 drill holes which were not used in the resource estimate. The channel samples were taken within the mined stopes and the development workings.

Channels are spaced every five ft along the drifts. Drifts are every 100 ft in the lower part of the mine. Some stopes were sampled in a grid pattern of five ft by five ft. Samples were located with true x and z coordinates while the north coordinate was projected to an east – west strike but keeping the differential in order to preserve the structural dip. Accuracy of the coordinates is unknown.

RPM explored the grade distribution with histograms, log-probabilistic charts (Figure 14-1) and directional clouds (Figure 14-2). No domains have been defined along the vein. The current sample database has some missing stopes and also is clustered in the high grade zones. The partial sample data, however, shows a trend of the product grade by length in the vertical and the strike directions (Figure 14-2).

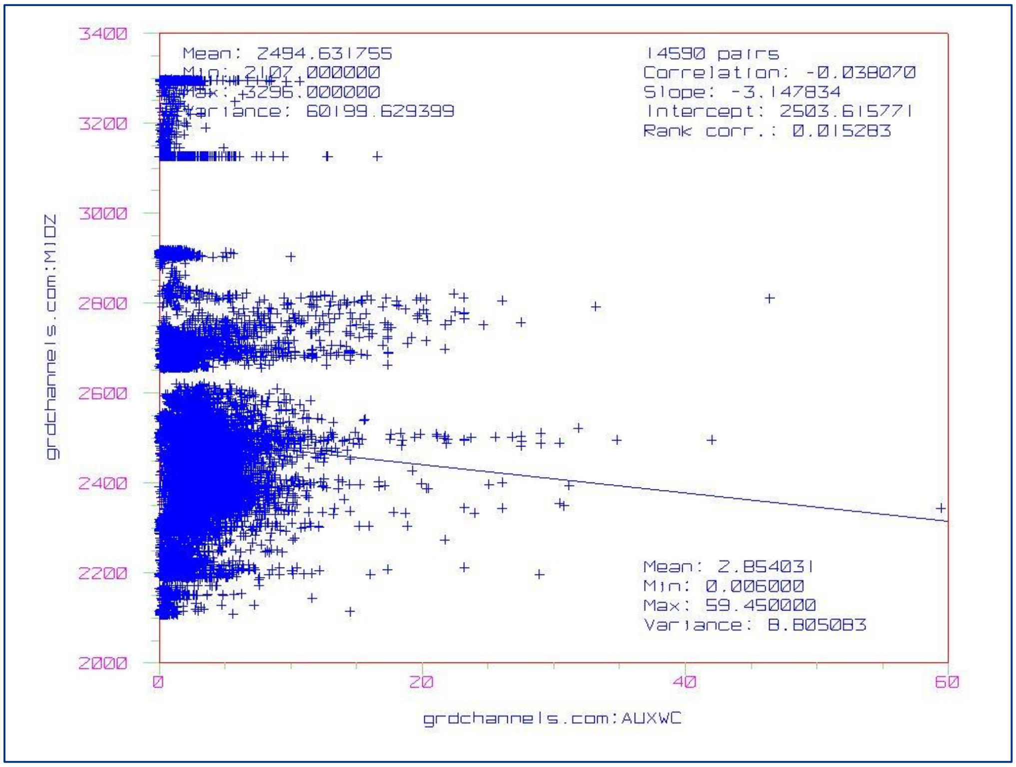
Figure 14-1 Histogram and Log-Prob of Gold and Thickness

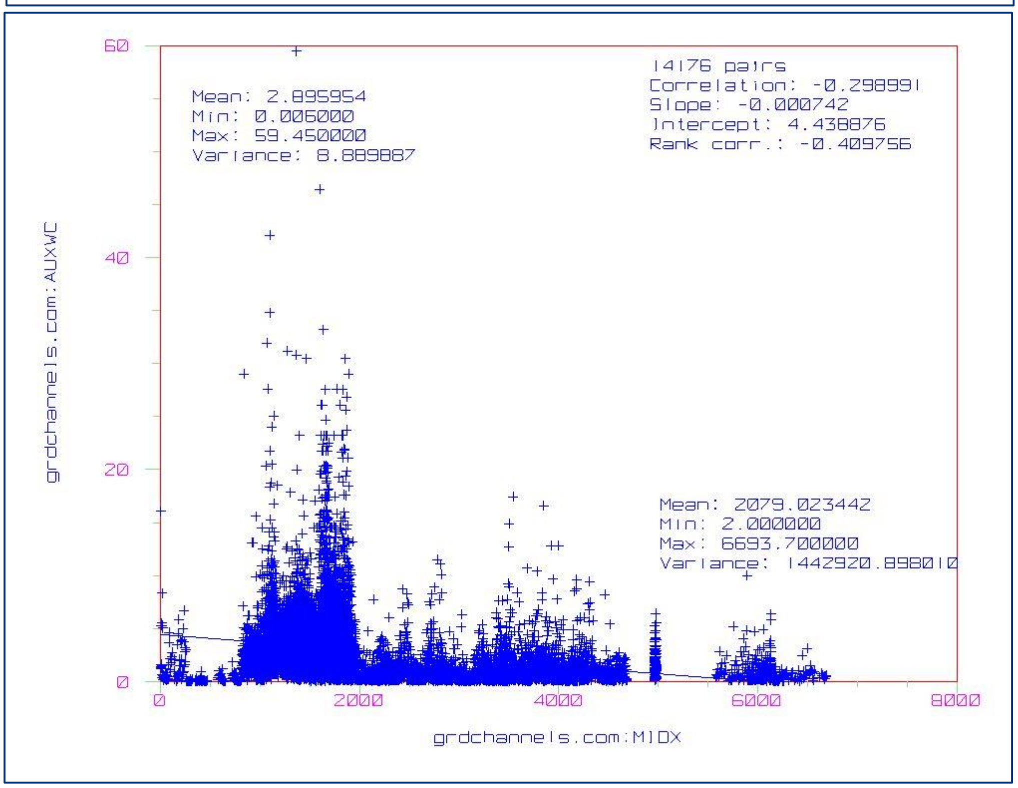


Length of the channel samples were assumed as the thickness of the mineralized vein. Historically it known the veins are often wider than the development drifts so in some cases the vein may be wider than the sample length, which in the opinion of the Author, makes the resource estimate potentially conservative.

Few extreme high grade samples were capped at 2.9 opt Au.

Figure 14-2 Trend Charts of Gold – Thickness Product





14.1.2 Compositing

Samples were composited using the method of Interval selection. This method determines composite intervals based on assay values and attempts to produce the longest possible composite interval, while maintaining a composite value above a certain ore/waste cut-off.

A minimum of one foot composite length with a grade above 0.1 (opt Au) was utilized to generate the mineralized intervals. Samples less than 0.1 opt Au (waste) between higher grade samples (higher than 0.1 opt Au) were included as dilution. Waste samples outside of the mineralized intervals were kept as separate composites but finally excluded of the estimation process.

14.1.3 Block Modelling

No 3D interpretation solids were built to delineate the volume of the vein; instead the thickness of the mineralized structure was estimated based on the composite lengths.

The block model was orientated west – east following the projected sample locations in a vertical plane. A block model of 50 ft x 25 ft x 10 ft was created to estimate the thickness and the product of Au - thickness. Au (opt) and tonnage of ore inside each block were calculated based on thickness, length and height of the block and a tonnage factor (inverse of density) of 13.5 ft^3/t. True length of the mineralized structure was corrected by the factor of sin (41°) which is the angle with respect to the real average orientation of vein.

14.2 Estimation Parameters

Gold - thickness product and thickness estimations were carried out using two-passes of ordinary kriging (OK) in the 50 x 25 x 10 ft blocks.

The continuity model used to assign weights in the OK equation was obtained through the 2D correlogram model (Figure 14-3) with a nugget effect of 0.25 and effective ranges around 150 ft.

Sample configuration for estimation was calibrated by doing cross validation where a maximum of composites used were tested at 20, 32, 40, 48, 60 and 100 composites. Although correlation between true value and estimates was high in all the cases the variance decreased from 20 to 40 maximum composites being roughly constant with more than 40 composites. Therefore, minimum and maximum composites for the first pass were respectively set at 12 and 40, and they were respectively reduced to 6 and 12 in the second pass. Octants restriction was used for the first pass.

First pass search distances cover the length of the blocks in order to select the channel samples within it, i.e. 50 ft, and a vertical distance of 200 ft to make sure the vertical continuity will be respected. Second pass extends the horizontal searching distances to 200 ft.

RPM validated the estimates by comparing them against inform data (Figure 14-4), detailed visual inspections of the long-sections and swath plots (Figures 14-5 and 14-6) to validate that spatial trends are honored.

RPM executed validations of the estimated grades using swath plots comparing all the OK estimates with respect to the mean of the six closest composites observing global and local estimates were unbiased.

Figure 14-3 Varmap and Directional Correlograms

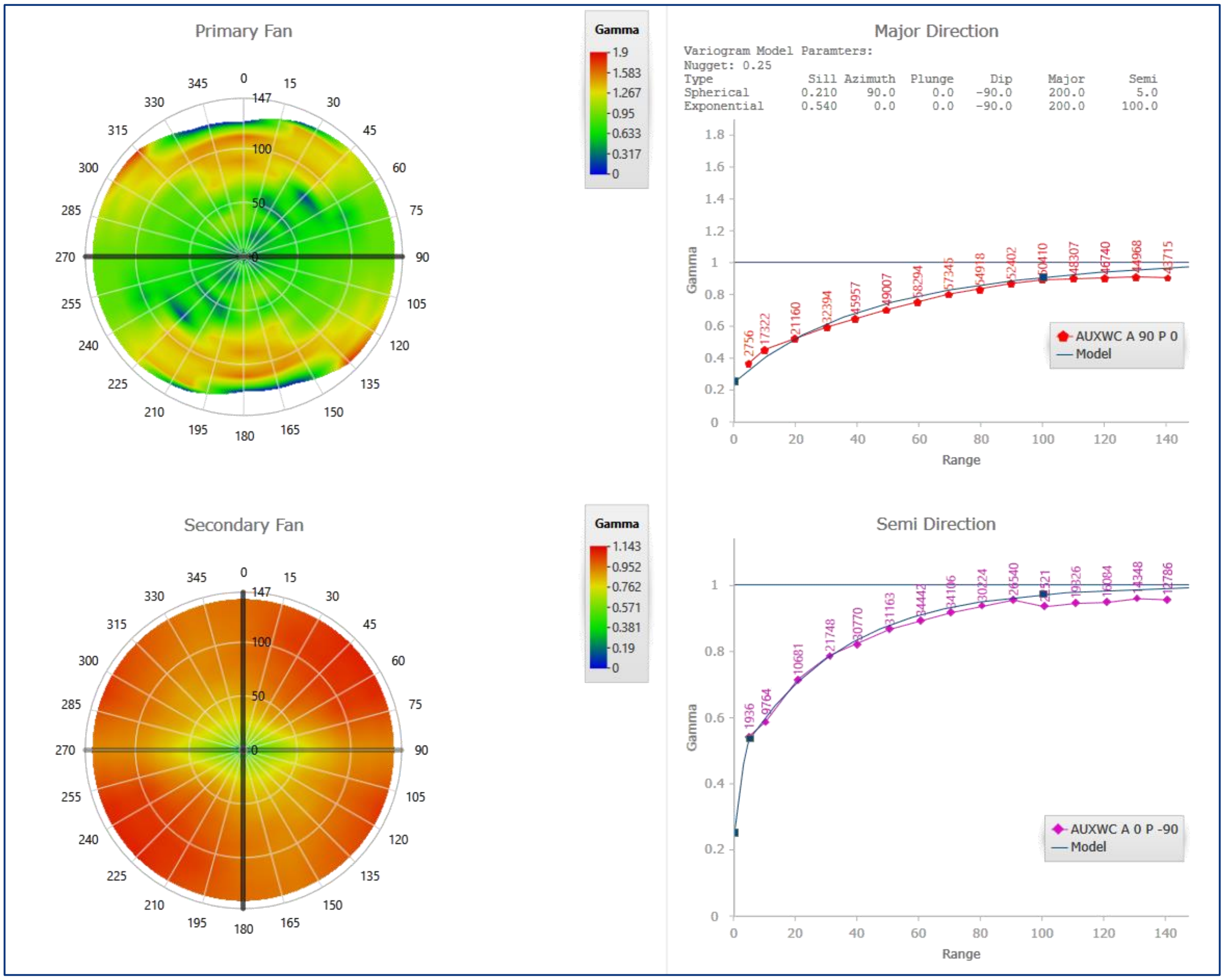


Figure 14-4 Grade – thickness product and thickness estimates vs. mean scatter plots

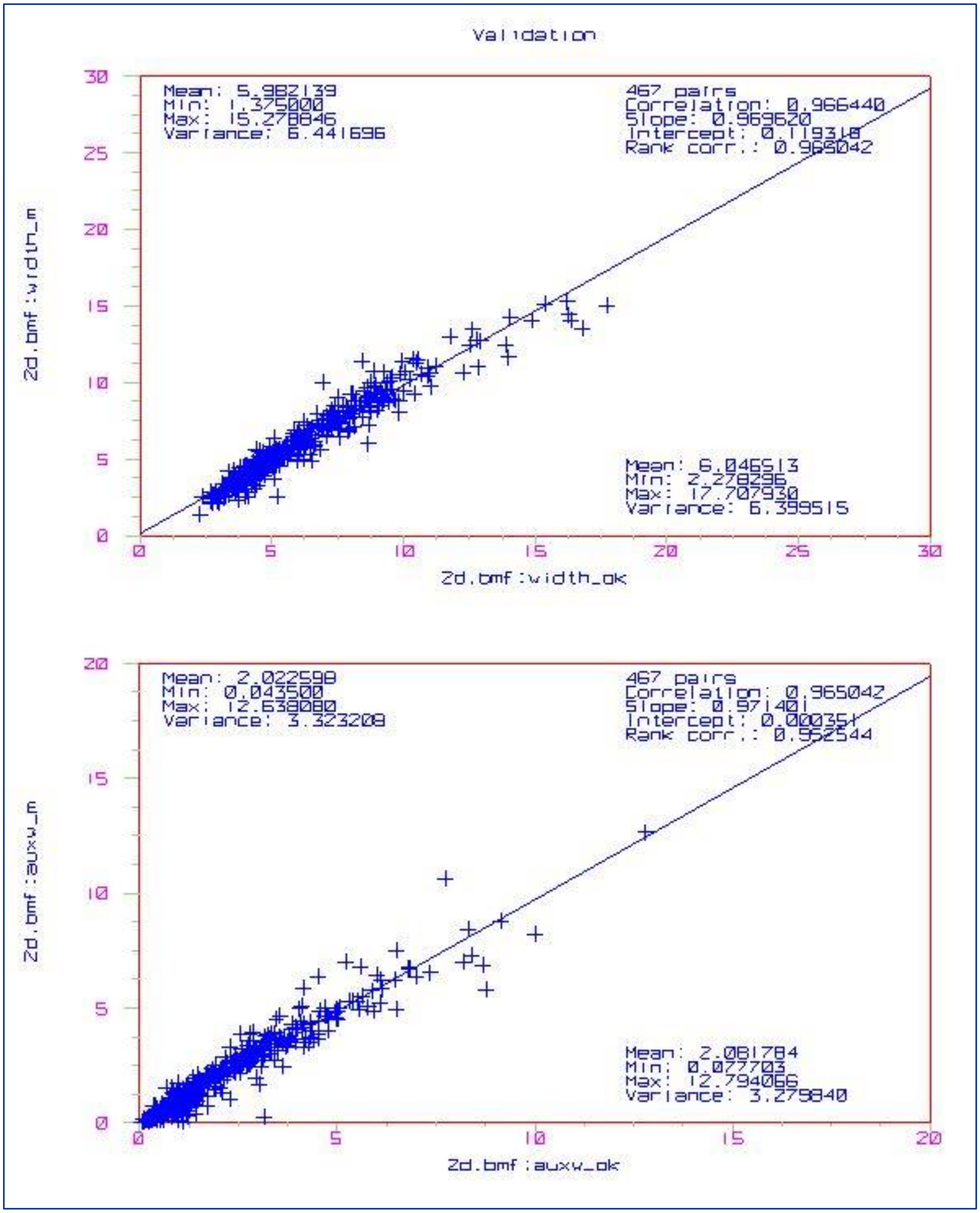


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Figure 14-5 Swath Plot Elevation

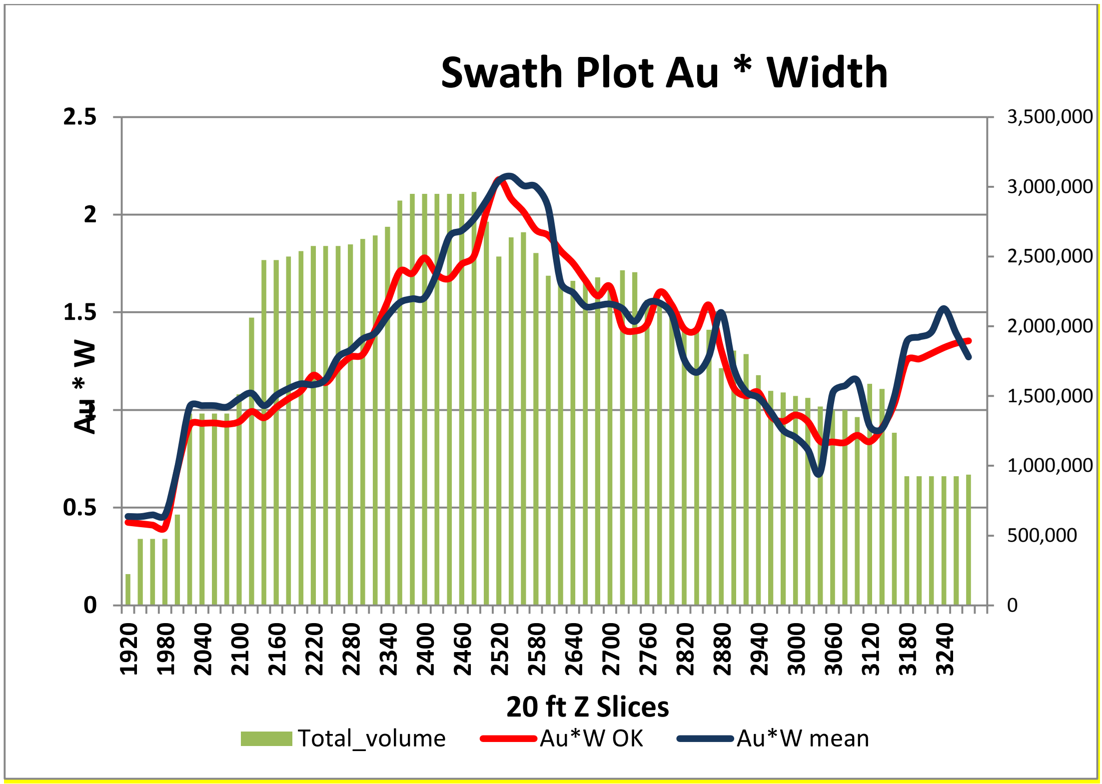
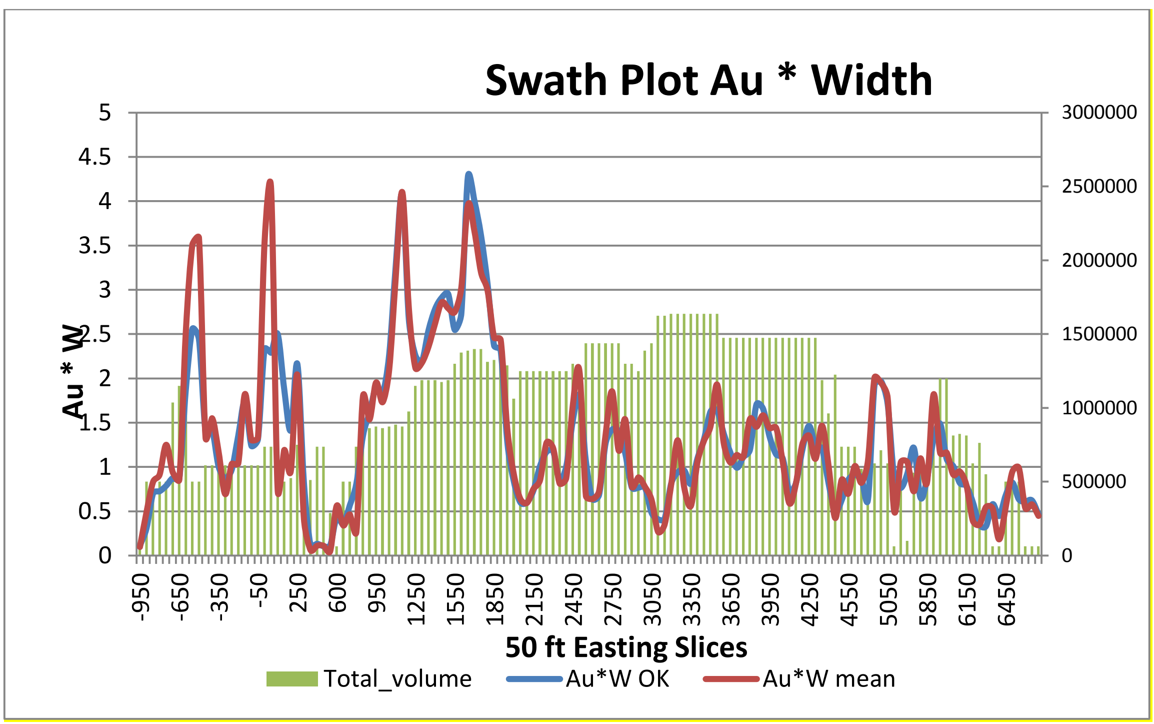


Figure 14-6 Swath Plot Strike Direction



14.3 Resource Classification

Because no QA/QC, 3D sample location and geologic model and most of the blocks were extrapolated, all the resources were defined as Inferred in this stage.

In order to upgrade Inferred resources it is necessary to validate historical sampling and build a three-dimensional model to constrain the grade estimation.

Additionally, creating a probabilistic model is recommended to assess uncertainty and support the classification.

14.4 Resource Statement

All the Inferred mineral resources are reported using a 0.1 opt Au cutoff, which is roughly the current economic cutoff. In order to meet the international requirement of reasonable prospect for eventual economic extraction, the mineral resources quoted in this report are constrained within a maximum vertical distance of 200 ft. from a drift (Table 14-1).

Table 14-1 Gold Road Mineral Resources at 0.1 opt Au Cutoff

Distance (ft)	Au (opt)	Tons	Ounces
<50	0.23	160,000	36,200
50-100	0.21	268,000	56,600
100-200	0.22	550,000	121,000
Total	**0.22**	**978,000**	**214,000**

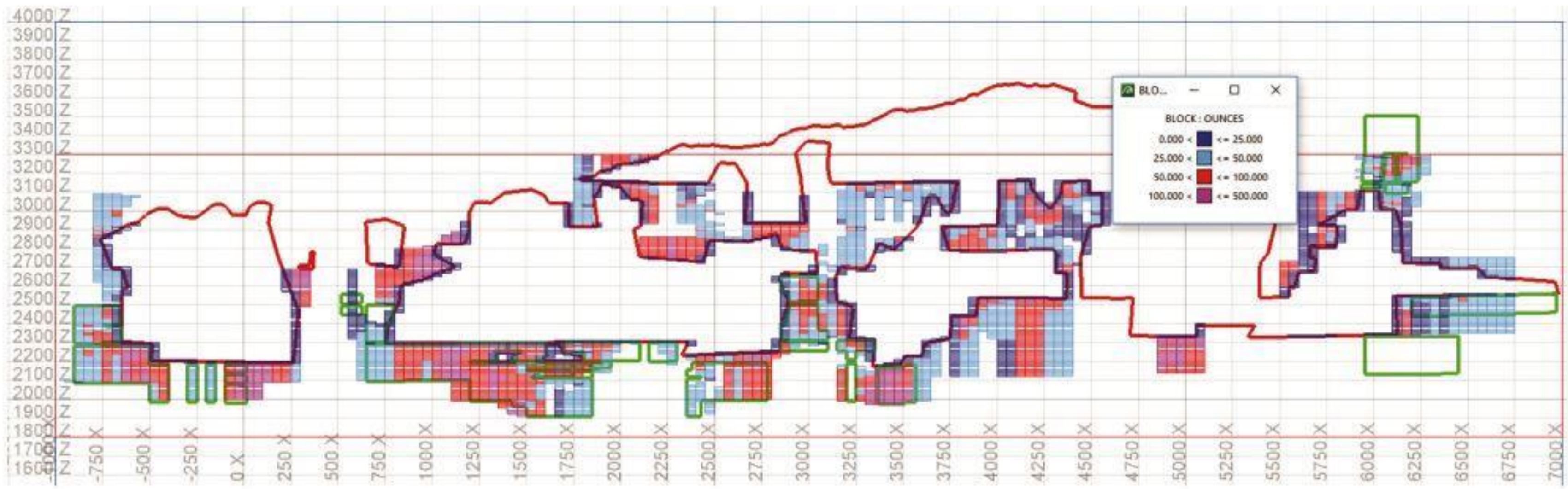
Figure 14-7 shows the distribution of the resource ounces within the mine.

14.5 RPM Comments

RPM has calculated mineral resources using linear estimation for PEA purposes. However, RPM believes that a probabilistic model would be more adequate to define resources in Gold Road.

Any resources that fell within the boundaries of the mined stopes was removed from the resource base and only this resources in areas not previously mined are included in the resource estimate.

Figure 14-7 Distribution of Ounces within the Gold Road Mine



15. Mineral Reserve Estimates

There are no reserve estimates as this PEA is preliminary in nature and is based on 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.

The PEA is based on Inferred Resources defined by the results of historical sampling that has been verified by a sampling program specifically designed for this PEA. The QP has assumed that the current sampling program has verified the historical sampling to a level that the historical results can be used to define Inferred Resources. While some of the inferred resources are bounded on at least one side by mine workings and geostatistical studies along with the sampling support the continuity of the mineralization, the Author feels that there is enough uncertainty in the historical sampling assay values that only Inferred Resources can be defined. The author also assumed the resources defined would be accessible to exploitation by the mining methods proposed. No detailed mine plan has been developed.

This PEA provides the basis for further studies of a more detailed nature. Any future studies should address the detailed mine plan to better define those Inferred Resources that can economically be accessed for mining. The mining method proposed (Alimak) is different than the mining method used historically (Shrink Stope). Mining costs are based on estimates from a mining contractor experienced in Alimak mining. Processing costs are based on historical processing costs updated to today. Capital costs are estimated based on the results of the site visit and benchmarking with similar operations. The costs to raise bore a shaft are a quote from a contractor. The author would anticipate these costs would change little with further studies with the possible exception of development costs to access some of the Inferred Resources.

16. Mining Methods

The following section describes the proposed mining method for the Gold Road gold mine near Oatman, AZ. The underground mine historically used Shrinkage Stoping Mining (SSM). During the most recent mining phase haulage of the ore and waste to the surface was with underground trucks. RPMGlobal reviewed the continued use of SSM for the Gold Road Mine and has also reviewed an alternative mining method known as Raise Access Mining. Raise Access Mining (RAM) incorporates an Alimak style raise climber to develop the access raise in ore from the sill level to the top level. Once the raise is established the climber is used to drill and blast horizontal production holes from the bottom of the ore block up in retreat. The ore is loaded from the bottom sill level using a mucker loading into nearby muck bay, a truck for the haul up the decline or to the proposed new shaft loadout. Gold Road used a form of RAM in a test stope in the past with mixed results. RPM advises that the use of RAM is the preferred method and has been used in the PEA as part of the restart and full mining method for the mine. RAM provides a safer mining process, with limited, to no exposure of the miners to unsupported ground, as well as lower operating cost. Both mining methods are discussed in the following sections of the PEA.

RPM also reviewed alternative haulage methods using the current truck haulage decline (11,000 ft one way) as well as a truck/shaft ore haulage scenario. As expected the truck/shaft method was the preferred alternative due to costs and efficiency. The RAM mining method and the truck/shaft haulage scenario are the preferred methods for the PEA.

16.1 Mine Design Criteria

The evaluation of the characteristics of the Gold Road vein and waste rock are summarized as follows.

- Steeply dipping with an average dip of +75°.
- Consistent vein widths ranging from 2.0 ft. to 8.0 ft. with an average of 4.7 ft.
- High grade deposit with good continuity with an expected diluted grade of 0.19 ounces gold per ton.
- Over 1,070,000 tons of diluted resources suitable for mining.
- The overall vertical extent of the deposit has been mined from the surface to the current depth of 900 ft with consistent gold grades and good ore and waste rock conditions.
- The overall known strike length of the mineralized zone is over 6,000 ft.
- There has been over 2.4M tons of ore mined (overall stated gold grade – 0.32 opT) from the historic operations (last operations from 2010 to 2015 – 293,300 tons). Historical mining methods were mostly shrinkage stope mining.
- The mine has historically not been backfilled with any waste rock or tailing material although future plans are to gob fill development waste rock in the historic workings.
- Currently existing develop on the 900 level and above have opened up mine areas that can be readily put into operation.
- Historic mining and processing infrastructure are currently in place and in good working order.
- There are no known historic deleterious minerals or radiation emitting minerals associated with the mineralized material or waste rock material.
- The density of the ore and waste is 13.5 ft^3/ton

16.2 Mining Method Evaluation and Selection

The restart of the Gold Road mine is currently under review by Para Resources. The historic operations, which utilized shrinkage stope mining, were reviewed with added consideration to operating areas that could be potentially improved. The excellent ground conditions and steeply dipping ore offered the availability of alternative mining methods that were suggested. Historically long hole stoping and a modified Alimak raise mining method were used with less than favorable results, higher than expected dilution, higher operating costs and less productivity. The RPM review and discussion for the SSM and the RAM methods mining method is as follows.

16.3 Selected Mining Method

RPM is of the opinion that RAM should be the preferred underground mining method for the following reasons;

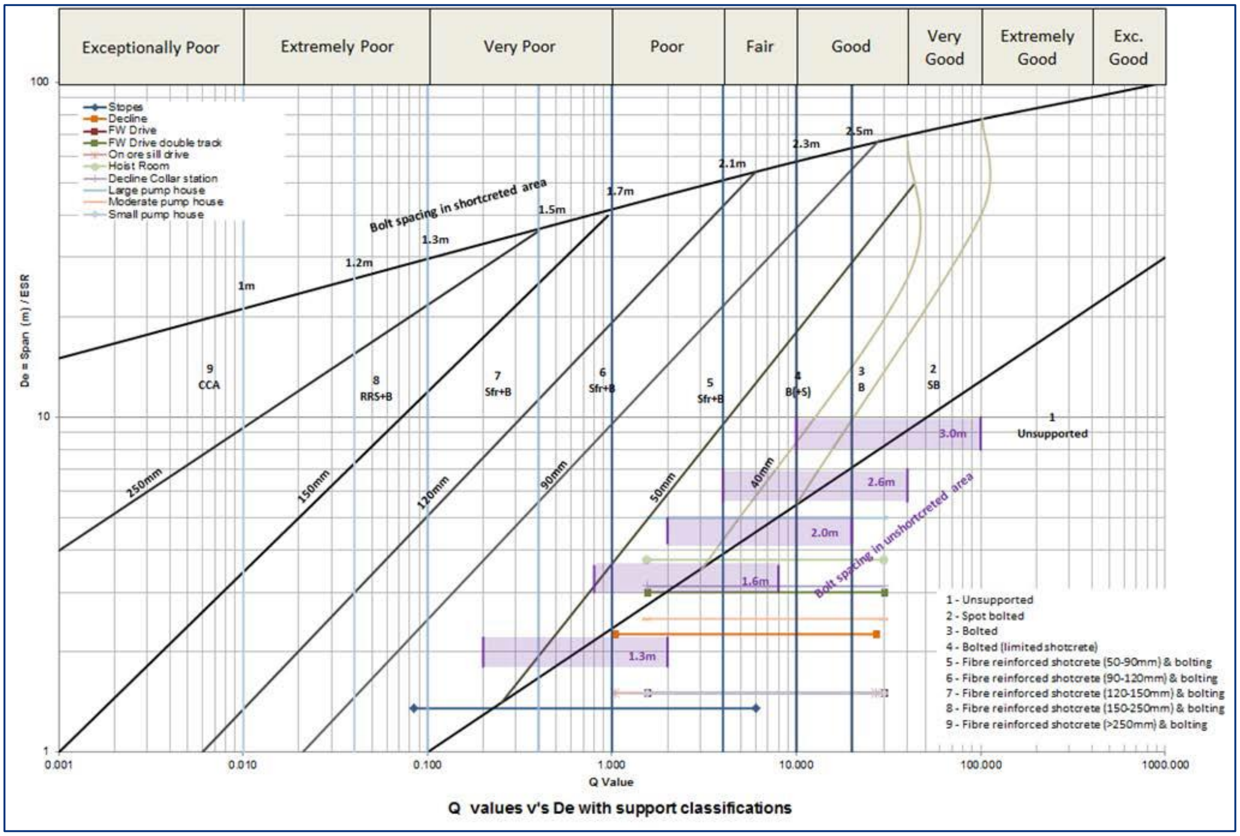
- Limited to no exposure of the miners to unsupported ground due to the protection offered by the raise climber
- Limited ground control costs – only required during the raise mining phase
- Reduced development costs since the raise is developed in ore.

The typical RAM stope will be a total of 60 ft. along strike, 200 ft high along dip and have an average mining width of 5.5 ft. This typical ore block represents 4,900 tons of material with a nominal gold grade of 0.19 opt using the calculated dilution factor of 14%.

16.4 Geotechnical Design

The historical operations were reviewed as part of the PEA. The mine openings had been developed during previous operations and show little to no deterioration or sloughing. The drifts were secured using mats and friction bolts and are holding up well. There are a few localized areas that require minimal scaling down and cleanup. It is expected that these type of ground conditions will be encountered as the mine is reopened and production starts. The proposed detail core drilling program will be used to confirm the geotechnical design with additional data such as RQD's information reviewed. Figure 16-1 is a graphical summary of the Q values used to empirically indicate the support requirements for a drift or stope design. The Gold Road determination is in the fair to good category with mats and friction bolting required in the waste drifts. The raise development in ore will also need to a minimum of 4 ft. friction bolts and mats as required with special attention to the hanging wall as the raise advances. The use of cable bolts may be required due to the long open span (200 ft) along the dip in the raise.

Figure 16-1 Q Value Graph



16.5 Mine Hydrogeology

The mine area in general is desert and there is relatively little meteoric water available from seepage into the historic mine workings. The underground areas have a small amount of seepage water (25 gpm) that is collected and pumped to the surface for use in processing and other surface requirements.

The lack of porosity in the waste rock and ore leads to a low permeability and low storage capacity for the mine. Predicted inflows are expected to be low throughout the LOM although dewatering the mine will be necessary and a preliminary dewatering and pumping strategy has been developed for the PEA.

16.6 Cut Off Grade Calculation

The economic cutoff gold grade was determined for the Gold Road project using a gold price of US$1,200 per ounce. Table 16-1 summarizes the parameters used to develop the breakeven cutoff grade of 0.091 ounces gold per ton.

Table 16-1 Breakeven Cutoff Grade

Area	Detail	Cost (US$/ton)
Direct Mining		$9.49
Ore Transportation	Truck & Shaft	$9.00
Fixed Cost		
	Gold Road	4.19
	Contractor	36.63
Total Mining		64.90
Milling		27.06
G & A		5.89
Total Cost	US$/ton	$97.85
Cut Off Grade	Gold oz. per ton	0.091
Revenue		
Gold Price	US$/oz.	US$1,200
NSR (97.5%)		97.5%
Recovery		95%
Net Value	**US$/ton**	**US$101.00**

16.7 Mine Design

A detailed mine design will be required in subsequent review and engineering based on additional information gained from the core drilling program that is recommended by RPM. The preliminary mine design that was completed as part of the PEA is based on a review of the previous mining operations, information from GRM personnel as well as RPM experience with similar mining operations. The steep dip, good footwall and hanging wall strength characteristics, the requirement to limit dilution lend itself to using the shrinkage stope mining method. Other mining methods such as long-hole stoping and RAM mining were reviewed as well as shrink stopes. RPM has recommended that Raise Access Mining is the preferred alternative for Gold Road using an experienced contractor.

16.7.1 Shrink Stope Mining

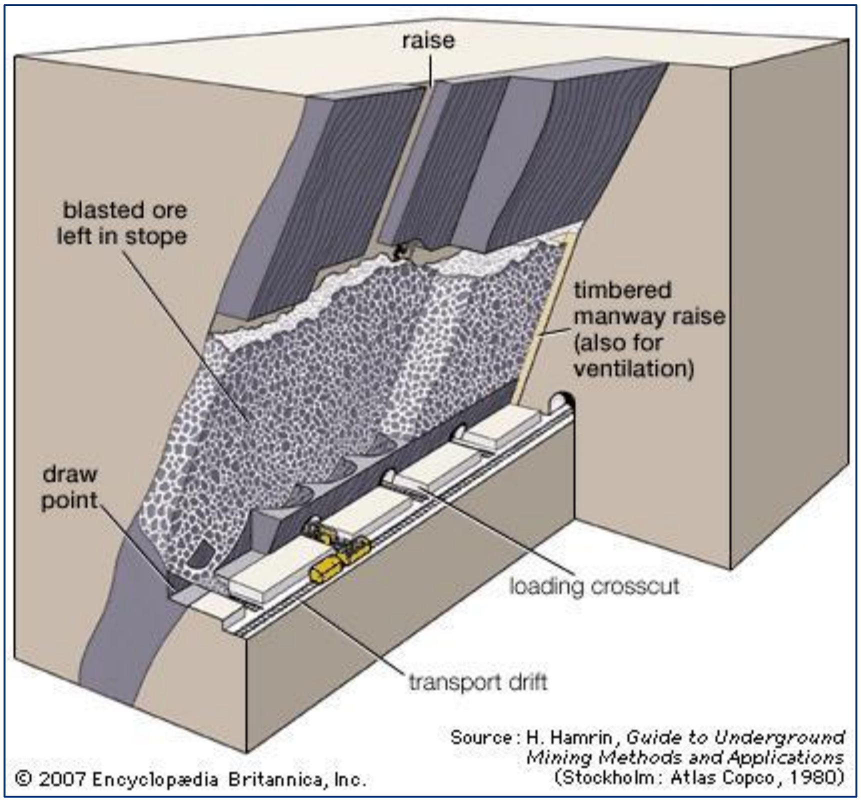
Shrink stope mining has been used throughout the world for over 150 years. Figure 16-2 is a typical shrink stope showing the different parts of the mining process. The ore is accessed from a sill level on the bottom of the design stope that is driven along the strike and a top cut level that is historically driven along strike 200 ft. above the sill level in the case of the Gold Road Mine.

16.7.2 Conventional Shrink Stope at Gold Road

A review of continuing to use conventional shrink stope mining was completed by RPM. The method is well known and has been used historically for many years including at the Gold Road mine. The following design criteria was used to estimate the productivity and estimated costs to be used for mine design planning as well as comparison to other mining methods.

Figure 16-2 is an illustration of a conventional shrink stope. The assumption is that the development, in waste, required to access the stope area is in place and will not be part of these costs. These development costs as well as all other costs would need to be determined and included in a full mine plan and cash flow model.

Figure 16-2 Conventional Shrink Stope Mining



- The initial timbered raise started on the sill cut and is developed along the vein vertically to the top cut approximately 200 ft. above. The raise is used for access and ventilation during the production mining sequence. Each round of advance is mapped and sampled for grade control.
- The initial sill cut (5 ft. wide by 9 ft. high) is along the strike of the vein (assume – 4.7 ft. wide) and is nominally 200 ft. along the strike length. Normally cross cuts from the access drift in the footwall are driven on 40 ft. centers to allow an LHD to muck out the shrink stope ore as required.
- The typical design sequence includes taking a back stope in the ore - 8 ft. high along the entire length of the stope. Once the back stope is blasted the ore is drawn out via the cross cuts as well as smoothed out with a slusher in the stope. The new stope back is bolted and drilled for the next 8 ft. deep back stope round. The bolting sequence is critical to ensure safe working conditions for the miners.
- There will be a conventional timber raise taken on either side of the stope up to the top cut 50 ft. (floor to floor) above the sill cut. The raises are in ore and are 9.0 ft. wide by 9.0 ft. along the strike.
- The remaining shrinkage stope is mined on a normal basis with jacklegs and ANFO explosives. Ore is drawn down as required to make room for the next cut across the stope.
- The estimated cost per ton of ore is US$119.00 and has an efficiency factor of calculated tons per man shift of 10. Using a nominal grade of 0.25 ounce of gold per ton the cost per ounce of gold is estimated to be US$479.

The mine design for the Gold Road restart would include drifting along the vein, taking out the back stope and establishing a concrete sill pillar with air actuated truck chutes for direct loading of the ore into the 20 ton haul trucks. This concept is still in the preliminary design phase. There will be a series of truck load chutes (every 20 ft.) installed in the sill/backstope area. The chutes will be prefabricated with compressor air actuated cylinders, placed with concreted forms installed along the full length of the sill cut. Concrete will be delivered via a slick line from the surface and pumped to a thickness of 6 ft. above the back of the sill. The forms will have a conical shape in the concrete that feeds the chute area eliminating the potential for any dead bedded ore left behind. The use of truck chutes allows for the complete recovery of the ore as compared to installation of the chutes in the ore sill pillar.

16.7.3 Raise Access Mining (RAM)

Raise Access Mining is a relatively new method used to mine vertical narrow vein areas in a safe and efficient manner.

The typical RAM stope will be a total of 60 ft. along strike, 200 ft. high along dip and have an average vein width of 5.5 ft. This typical ore block represents 4,900 tons of material with a nominal gold grade of 0.19 opt using the calculated dilution factor of 14%.

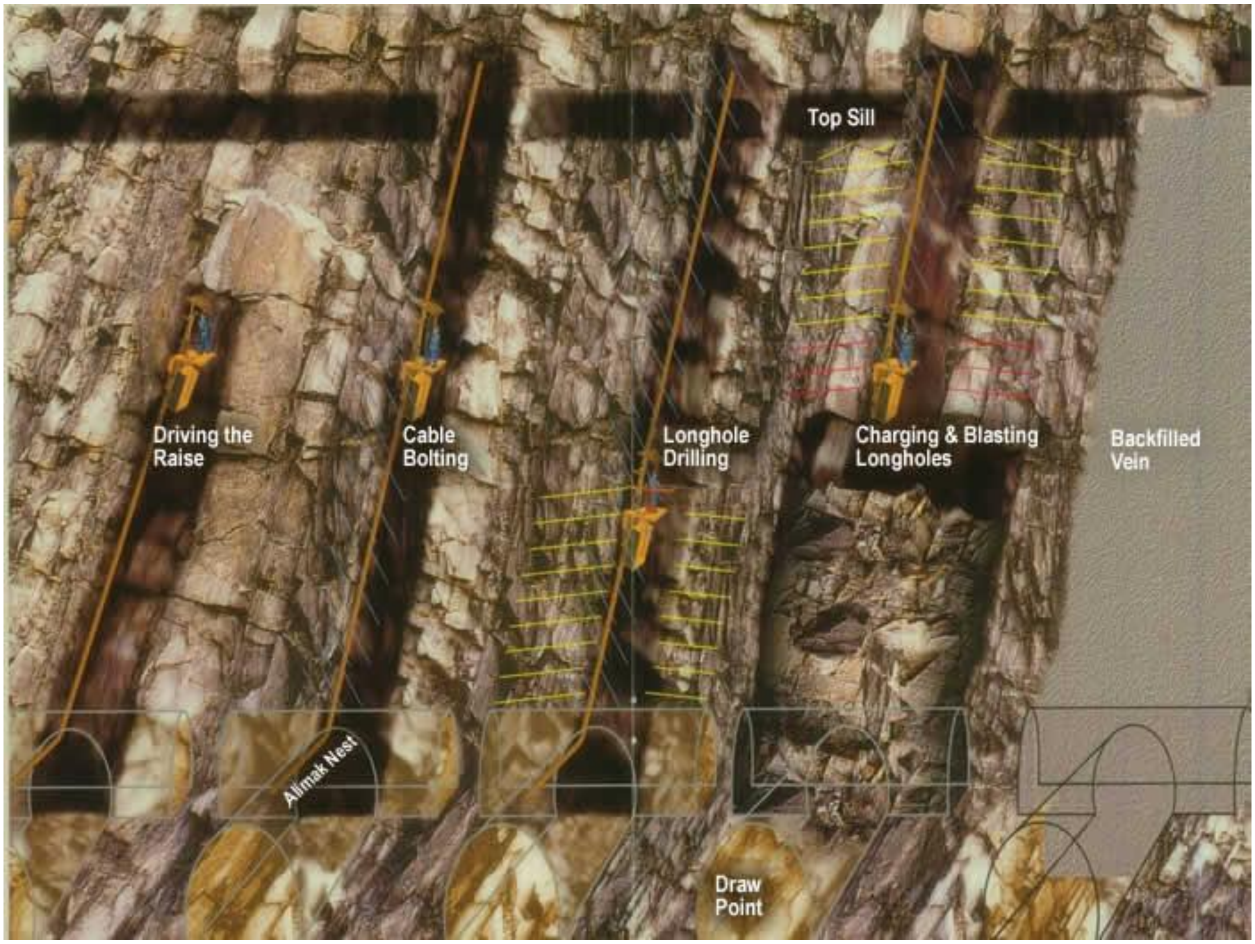
RAM, also known as Alimak Mining has been developed over the past decade as a safe, cost effective and proven technology to allow production mining in narrow veins with a limited amount of dilution.

The advantages of the mining method are:

- Less development with greater advance along the dip of the ore thereby minimizing the number of sill levels and associated lateral development cost.
- Limited, to no exposure to unsupported ground for the miners.
- Faster access to ore production.
- Better ground control and lower costs since the only area that requires ground control is directly within the Alimak raise area.
- Lower unit costs than other methods.
- Capital costs for Alimak machines have dropped to an estimated US$500,000 per machine.
- One of the disadvantages is that experienced Alimak miners demand a premium wage although with more machines in use the number of experienced miners is increasing.

Figure 16-3 demonstrates the mining sequence that is summarized as follows:

Figure 16-3 Alimak Raise Mining



- Development of the bottom sill and haulage, establish the draw point and Alimak climber nest, and top sill access.
- Raise (9.0 ft. by 9.0 ft.) development is continued vertically along the dip of the vein until the floor of the top sill is reached. Once the climber nest is established on the top sill all access is from the top down (see Figure 16-3).
- Production is from the raise climber platform drilling near horizontal long holes (approximately 25 ft. each way) along the strike of the vein. A series of these holes is charged and blasted with the ore mucked out from the draw point on the bottom sill. The only ground control required is during the initial development of the bottom and top sill development and the Alimak raise.

Figure 16-4 RAM Raise

As shown in Figure 16-5 the ore falls down the stope and is mucked at the draw point with a conventional diesel LHD into nearby muck bays or trucks as available.

Figure 16-5 RAM Ore Removal

Figure 16-6 shows the mucking sequence from the bottom of the stope with the use of conventional LHD machines since the ore flows out of the bottom access point. During the final cleanout of the stope a remote control mucker is used to reach all of the ore in the stope. The mucking sequence is independent of the Alimak production sequences other than there has to be enough room between the area to be blasted and the top of the previously broken ore.

Figure 16-6 RAM Mucking Sequence

- The method allows for near 100% recovery of the ore since there is no support pillar required in good ground conditions. Once the stope is cleaned out a 10 ft. thick bottom pillar of concrete is poured to allow safe mining from the next stope below the current stope. Concrete is delivered via slick lines from the surface.
- The estimated cost per ton of ore is US$64.90 and has an efficiency factor of 10.5 tons per man shift. Using a nominal example grade of 0.25 ounce of gold per ton, the cost per ounce of gold is US$290. The estimate includes the 100 ft. of development drift (12 ft. by 12 ft.) in waste that is required to access the stope.

16.8 Summary

Table 16-2 is a summary of the estimated costs and other details from the mining method review and is presented for illustrative comparison only.

Table 16-2 Mining Method Summary

Description	Units	Shrink Stope	RAM Mining
Ore	Tons	1,600	2,222
Waste	Tons	533	747
Ore Grade	Opt	0.25	0.25
Gold Mined	oz.	400	444
Total OPEX	US$	$191,858	$129,185
Unit OPEX	US$/oz.	$479.64	$290.67
	US$/ton-ore	$119.91	$58.13
Efficiency	Man shifts	160	211
	Tons per MS	10	10.5

16.9 Mine Access

The Gold Road mine currently has an 11,000 ft. long decline that is used for access. The restart of the mine operations will continue to use this access for the ore/waste requirements as well as men and material for the underground operation. Gold Road is considering the installation of a new shaft that would be used to hoist the ore/waste requirements for the 500 tpd operation. The new shaft is discussed in Section 16.18.

16.10 Development Plan

The mine development plan was based on the new RAM mine operations with access headings required for the Raise climber stopes. Additional waste development is required to access the new level in the mine below the 900 level. All of the waste headings are 12 ft. by 12 ft. and will be developed by the contractor. This development is considered to be sustaining capital for the LOM.

16.11 Capital - Lateral Design

The capital development is considered development on the levels required to access the ore zones. A critical part of the development design will be the results of the detailed core drilling program. The normal sequence will be to drift on the vein with routine ore control and mapping. As the vein is defined in more detail and the detailed drilling confirms the potential location of a stope, or sequence of stopes, a run around drift will be developed in the hanging wall. The run around stope will leave a 30 ft. pillar of waste rock between the new stope and run around stope. The access drifts will be on 60 ft. centers to the strike of the vein and perpendicular to the strike. The raise climber nest is established as well as a muck bay.

16.12 Capital - Vertical Design

The vertical design is relatively simple since the new shaft is the only vertical development required. All other access and ventilation raises have been developed during the previous mining cycle. New vertical development for access and ventilation are provided by the production mining sequence since the stopes are left open. Most of these stopes will be blocked off from access with a few strategically placed stopes fitted with ladders and landing to provide a secondary method of egress and part of the ventilation loop.

16.13 Development summary

Table 16-3 is a summary of the development requirements for the LOM of the Gold Road mine.

Table 16-3 Development

Development	Description	Units	Amount
Lateral	Waste heading 12 ft. by 12 ft. as required	ft.	41,300
Vertical	New shaft – 8 ft. diameter	ft.	900

16.14 Mine Dilution and Ore Recovery

Dilution was calculated for both mining methods using the following methodology with dilution calculated using the following formula;

$$\text{Dilution (\%)} = \frac{[\text{Planned (tons with 0-grade)} + \text{Unplanned (tons with 0-grade)}]}{[\text{mineralized material (tons with grade)} + [\text{Plan (tons with 0-grade)} + \text{Unplanned (tons with 0-grade)}]}$$

- Mineralized material are the tons from mine stope based on the block model
- The planned dilution is 0.25 ft. on either side of the mineralized vein – total 0.5 ft.
- The unplanned dilution is an additional 0.15 ft. on either side of the mineralized vein.

- The density for mineralized material or waste is 13.5 ft^3/ton
- The full dilution used in the Gold Road mine plan is 14%.

16.15 Mining Manpower

The Gold Road underground mine will use a contractor to perform all of the mining requirements to produce the 500 tons per day production goal including development. The crews will work a rotating shift with three crews working 10 hr. shifts. Table 16-4 Mining Manpower is a summary of the proposed crew make up. The summary includes an estimate of the salary and hourly requirements. The overall costs for contractor were US$4.33/ton for management and US$37.88/ton for the hourly although these costs are considered fixed costs in the cash flow model. Hourly wages ranged from US$45.00 for experienced miners to US$20.00/hr. for entry level personnel. All costs include a 40% burden to account for benefits and required taxes.

Table 16-4 Gold Road Mining Manpower Summary

Area Gold Road	**Position**	**Number**	**Total**
	General Manager	1	
	Environmental	1	
	Engineer	1	
	Accounting	2	**5**
Contractor			
Salary	Project Manager	1	
	Engineer	1	
	Shift Foreman	3	**5**
Hourly			
	Miners	14	
	Equipment Operators	6	
	Hoist man	3	
	Laborers	6	
	Electricians	3	
	Mechanics	3	**35**
Total Gold Road			**5**
Total Contractor			**40**
Total			**45**

16.16 Mining Equipment

The underground mining equipment requirements were reviewed based on the RPM alternative of using Raise Climber Mining (RCM). The mine will require the following equipment supplied by the contractor except for the raise climbers.

- Underground haul trucks – 20t (3).
- LHD – 6.7 tonnes – (3)
- Small single or two jumbos drills for development headings - (2)
- Raise Climbers - (3)
- Ancillary equipment including jacklegs for ground control and long hole drills for the raise climbers

16.17 Mine Operations

The mine operations consist of the development and production requirements to meet the 500 tpd (174,000 tons per year) design objective.

16.18 Haulage – Truck / Shaft

The Gold Road operations current use the 11,000 ft. one-way decline to access the mine. All men and materials use the 12 ft. by 12 ft. decline. The restart of the mine will use two 20 tonne haul trucks (plus one spare) supplied by the contractor. The trucks will continue to be used throughout the LOM although will only haul to the new shaft loadout pocket when it is ready in Year 2. The development of a shaft to be used for ore and waste only has been reviewed and developed by Gold Road. The plan is to use a raise bore to construct a new 8 ft. diameter shaft with the capability to move ore and waste – there will not be any men or material moved in the shaft.

Gold Road has negotiated and reviewed a budgetary contract to lease a headframe that would be installed over the new shaft. The headframe is 70 ft. high has a 685 hp motor and all of the required control circuits. The drum diameter is 96 in by 59.5 in wide. The rope size is 7/8 in and has a 3.5 ton capacity skip. The new shaft will be 900 ft. deep plus sufficient room to include a loadout pocket and sump at the bottom. The overall installed cost for the raise, hoist and equipment is US$1.9 M. The shaft haulage is expected to have an operating cost that is less than 50% of the historic truck haulage costs.

The new shaft is developed by using a contractor to drill a pilot hole and then back ream the new 8 ft. diameter shaft. Cuttings developed from the raise bore will be on the 900 level and will be trucked to the surface to be placed on the waste dump. The raise bore process is expected to take 2-3 months. After successful completion of the raise bore project the new shaft is fitted with the new guides and support. A working deck is used to complete the ground control and shaft infrastructure requirements including installing the rope guides. The shaft project is expected to take 10 months and cost an estimated US$1.9 M. Ongoing lease payments for the headframe will be made by Gold Road for the LOM.

16.19 Drilling and Blasting

Drilling and blasting will be used for the 12 ft. by 12 ft. conventional development headings. The headings are drilled with a small two boom jumbo approximately 11 ft. deep drill holes. Conventional ANFO explosives will be used with Nonel blasting caps. The drilling and blasting in the RAM stopes will use stoppers for the upholes in the raise and longhole drills with sectional steel for the production holes drilled from the raise climber. ANFO and Nonels with electric caps for the initial detonation will be used in the raise and for production holes.

16.20 Ground Control

The ground conditions for the Gold Road mine have been historically very good. Ground control in the drift includes mats and 6 ft. long friction bolts placed on a 3 ft. by 3 ft. patterns. Ground control in the production raise is mostly in the hanging wall and includes 4 ft. to 6 ft. long friction bolts placed on a 3 ft. by 3 ft. pattern. Ground control in the production stope area is not required since miners never enter these areas.

16.21 Mine Services

The required mine services are discussed below and include ventilation, power, water, pumping and other services. The mine will also construct refuge facilities on the different levels that will provide fresh air, water, provisions, communications for the miners in case of an emergency situation. Communications throughout the mine will be supplied with a leaky feeder system to be able to track and communicate with the operations. A small underground shop facility will also be set up to provide limited maintenance services without going to the surface.

16.21.1 Ventilation

Mine ventilation for underground mines is especially important to be able to provide a safe and healthy environment for the miners and provide for efficient equipment performance. The preliminary mine ventilation system design should be used as a guide with additional design details developed during the actual startup process. The ventilation systems was designed to remove and dilute diesel fumes, clear blasting fumes and provide some air cooling effect to the underground mine. Mine Safety and Health Administration (MSHA) provides regulations and guideline that are required to be met by the mine management/operator. In general the minimum air velocity in any airway/drift is 100 ft/min. The other guideline is for the volume of air moved per horsepower for the diesel equipment. The factor is 100 cfm/hp. The preliminary air quantity requirement is calculated based on the number of underground diesel equipment units, rated horsepower and quantity, with an additional factor for utilization. Table 16-5 is a summary of the estimated airflow required. As noted most of the air quantity requirement is based on the number of trucks and loaders in use and is estimated to be 172,500 cfm.

Table 16-5 Mine Air Quantity Requirements

Equipment	Motor Power (kw)	Units	Total kw	Utilization	Adjusted Total kw	Total Airflow Required (m3/sec)	Total Airflow Required (cfm)
Haul Truck – TH320	240	3	720	75 %	540	43.2	91,540
Sandvik LH 307	160	3	480	80%	384	30.7	65,100
Other Equipment	50	1	50	100%	50	6.0	12,710
Other Facilities	25	1	25	100%	25	1.5	3,178
Total					**1,000**	**81.4**	**172,520**

From the required air quantity the ventilation fan power is developed. Overall fan power is estimated to be 165 hp with 75 hp required for the exhaust fan located near the top of the decline and three additional 30 hp fans located in the mine.

16.21.2 Power

Electrical power was installed during previous operations and is in good working order. There are mobile power centers that will be rebuild and installed in strategic locations based on the mining activity. The underground mine equipment that requires power will be the expected equipment supplied by the contractor including the jumbo drill. The three raise climbers will also require power. The permanent power requirements will be for the underground pumping and ventilation fan requirements. The new hoist will require sufficient power for the hoisting operations. The new line will be a surface line on poles from the existing surface transformer.

16.21.3 Water

The mine makes a relatively small amount of water (approximately 25 gpm). The water is collected using ditches and smaller pumps that pump to sumps located on the different levels. The sumps have two sides with a dirty water side and a clean water side. There are two pump systems on the clean water side, one for the drill and wash down water that is piped to the working areas and the other pump that pumps the water to the surface to be used in the mill. Backup pumps are installed for both sets of pumps.

16.21.4 Pumping

As noted above there is relative small amount of water inflows from the mining operation. Water is collected and water that is not used for the drilling is pumped to the surface. The recommendation is to have the raise bore contractor (or others) drill at least three strategically placed bore holes from the surface. The holes are fitted with steel pipe to be used as required for pump columns.

16.21.5 Compressed Air

The Gold Road mine currently has compressed air equipment on the surface. The compressed air requirement is relatively limited since the jumbos are electric/hydraulic with the stopper drills and jacklegs requiring compressed air. Part of the reopening plan will be to rebuild and upgrade these machines. The underground compressed air line will also be rebuilt as required. One of the new boreholes will be fitted with a steel airline and used to supply compressed air as required to the production headings. As the mine expands, the use of portable underground air compressors is recommended. Atlas Copco has skid mounted units specifically designed for this purpose.

16.22 Operations Management

In discussions with Gold Road the use of a capable and experienced underground mining contractor is recommended by RPM. The mining section of this report has been set up with that in mind although the mine plan, equipment and manpower requirements were developed as if Gold Road were going to operate the mine in order to set up the estimated costs. The contractor will supply the mining equipment, management and operators, Gold Road would supply the site management, engineering, environmental and accounting on site. The three raise climbers will be supplied by Gold Road as well as the new shaft, headframe and hoist. Gold Road has been in contact with a couple of different contractors and will be developing a request for proposal (RFP) and enter negotiations in the near future once the final decisions on the operations have been made by Gold Road.

17. Recovery Methods

The Gold Road ore-processing facility consists of a conventional 500-ton/day-capacity carbon-in-pulp (CIP) mill that was built in 1995. The mill has, from the time it was constructed up to the present time, processed about 1.1 million tons of ore, operating about one third of the time. It was last in operation in 2016. The mill incorporates two-stage crushing, two-stage grinding, 24-hour leaching, CIP adsorption, carbon-stripping, electrowinning, and refining. Tailings are pressure filtered and conveyed to a dry-stack tailings storage facility (TSF). The grind of the plant is 80% passing 325-mesh (45 µm) and the gold recovery is about 95%.

Principle process parameters are shown in Table 17-1 and a listing of principle process equipment is shown in Table 17-2. A flow diagram of the plant is shown in Figure 17-1 and a layout of the plant is shown in Figure 17-2.

Table 17-1 Principal Process Parameters

Parameter	Units	Value
Throughput Rate		
Annual (at full rate)	tons/year	180,000
Daily	tons/day	500
Ore Grade (average LOM)	ounce/ton	0.24
Gold Recovery	percent	95
Gold Production (at full rate)	ounces/year	40,000
Ore Physical Characteristics		
Work Index	kWh/ton	25
Abrasion Index	very abrasive	
Primary Crush Size	80 % passing, in.	3
Secondary Crush Size	80 % passing, in.	0.50
Primary Mill Grind Size	80% passing, µm	1,000
Secondary Mill Grind Size	80% passing, µm	45
Dewatering Parameters		
Thickening rate	ft^2/ton/day	0.5
Thickener u'flow density	% solids	50
Leach density	% solids	50
Filtering rate	ft^2/ton/day	7.5
Filter cake moisture content	% solids	83
Leach Parameters		
Retention Times		
Leaching	hours	24
CIP	hours	15
Detoxification	hours	3
pH		10.5
Cyanide strength		
Start of leach	lb/ton solution	0.25
End of leach	lb/ton solution	0.15
End of CIP	lb/ton solution	0.10
Post detoxification	ppm	12-15
Carbon concentrations		
Tank 1	grams/liter	45
Tanks 2-5	grams/liter	15
Carbon processing rate	tons/day	0.75
Gold loading on carbon	ounces gold/ton	150-175

Doré fineness		950
Employees		
Operation (at full rate)	number	22
Maintenance (at full rate)	number	3
Utilities Consumption		
Power	kWh/ton	65
Fresh Water (make-up)	gpm/ton	30
Grinding Ball Consumption		
Primary mill	lb/ton	2.00
Secondary mill	lb/ton	4.00
Reagent Consumptions		
Quicklime	lb/ton	0.60
Sodium cyanide	lb/ton	1.00
Flocculent	gallons/ton	0.02
Carbon	lb/ton	0.15
Copper sulfate	lb/ton	0.15
Hydrogen peroxide	lb/ton	3.00
Operating Cost	US$/ton	27
Capital Cost	US$ millions	0.5
Projected Start-Up	Date	Jan-19

Table 17-2 Principal Process Equipment

Item	Size	(hp ea)	Oper.	S'by.
Crushing System				
Run-of-Mine Pad	area with ~20,000-ton capacity		1	
Front-End Loader	Cat 950		1	
Dump hopper	50-ton capacity with 12- x 12-in grizzly		1	
Vibrating Feeder	3- x 14-ft	20	1	
Jaw Crusher	single-toggle, 24- x 36-inch	100	1	
Screen Feed Conveyors	24-inch	20	2	
Vibrating Screen	5- x 12-ft, double deck	15	1	
Cone Crusher	4-1/4-ft short head	200	1	
Mill Feed Bin Belt	24-inch	7.5	1	
Mill Feed Bin	600 ton live, 24-ft dia. x 45-ft	-	1	
Primary Mill				
Feeder	belt, 30-in x 20-ft, variable speed	10	1	
Lime Bin	30-ton silo (feed to rod mill feed belt)		1	
Rod Mill Feed Belt	24-inch	7.5	1	
Mill	8.5-ft dia. x 12-ft	400	1	
Secondary Mill				
Mill (with trommel screen)	overflow, 10.5-ft dia. x 15-ft	900	1	
Cyclone Feed Pump	5- x 5-inch, variable speed	50	1	
Cyclones	10-in	-	2	1
Trash Screen	3- x 6-ft, vibrating, 24-mesh	3	1	
Thickener	20-ft dia., high-rate	5	1	
Thick. U'flow/CIL Feed Pump	3- x 3-in, variable speed	3	1	
Air Compressor	for air clutch & general service	60	1	
Leaching System				

Leach Tanks	21.3-ft dia. x 28.5-ft high	30	3	
CIP Tanks	12-ft dia. x 14.3-ft high	7.5	5	
In-Pulp Carbon Screens	24-mesh, stationary w/air bubblers		5	
Carbon Advance System	air lifts		5	
Safety Screen	3- x 6-ft, vibrating, 24-mesh	3	1	
Detoxification Tank	12-ft dia. x 12-ft high	7.5	1	
Leach & CIP Air System	Use mine air compressors			
Tailings System				
Slurry Surge Tank	12-ft dia. x 20-ft high	5	1	
Filter Feed Pump	5- x 5-inch, 2-stage	75	1	
Filter	100 plate, Maytec	40	1	
Conveyors	36-in., 900- and 430-ft	20 & 10	2	
Bulldozer	Cat D7, rented		1	
Storage Area	Phase 1A: 1.7 M yd^3; 2.2 M tons		1	
Emergency Pond	Phase 1A: ~120- x 100- x 10-ft		1	
Carbon System				
Loaded Carbon Screen	3-ft wide x 6-ft long	3	1	
Loaded Carbon Wash Tank	4-ft dia. x 9.5-ft		1	
Stripping Column	3-ft dia. x 13.6-ft high (~1.1-ton carbon)	-	1	
Strip Solution Heater	133-kW, electrically heated	178	1	
Heat Exchanger			1	
Barren Carbon Screen No. 1	40-in dia., Kason	0.3	1	
Barren Carbon Screen No. 2	40-in dia., Kason	0.3	1	
Reactivation Kiln	2-ft dia. x 15-ft, electrically heated, 74.8-kW	100 & 2	1	
Carbon Holding Tank	5-ft dia. x 10-ft high, 45^{o} cone, 1.5-tons	-	1	
Carbon Fines Tank	8.5-ft dia. x 7.5-ft high, 1,600-gal	-	1	
Refinery				
Pregnant Solution Tank	6.5-ft dia. x 7.5-ft high	-	1	
Electrowinning Cell	750-amp, 0-6 volts	2	1	
Barren Solution Tank	6.5-ft dia. x 7.5-ft high	-	1	
Cathode Sludge Filter			1	
Cathode Sludge Dryer	3.8-kW	5	1	
Smelting Furnace	induction, crucible, 300-lb, 125-kW	168	1	

Figure 17-1 Process Flow Diagram

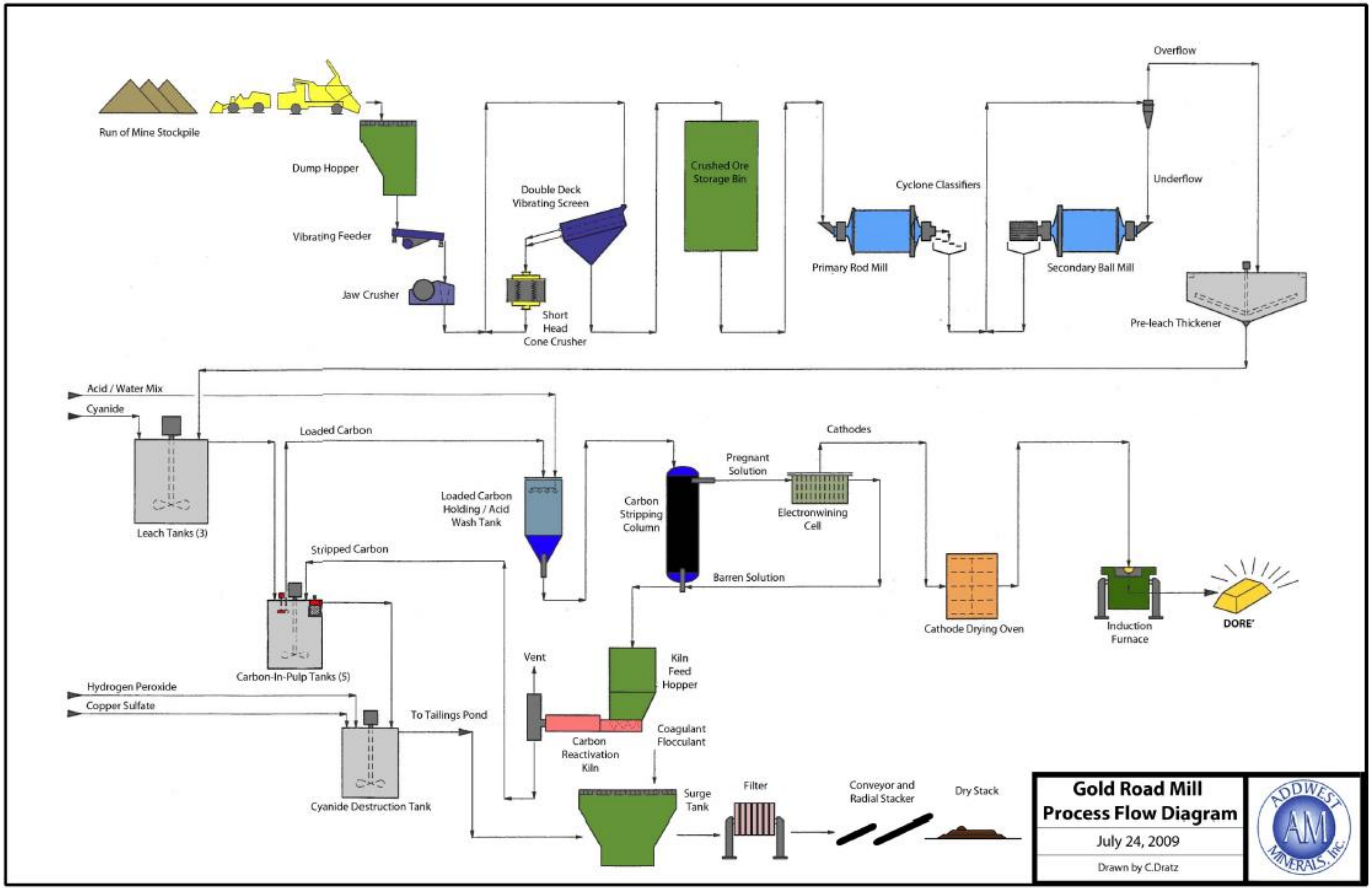


Figure 17-2 Plant Layout

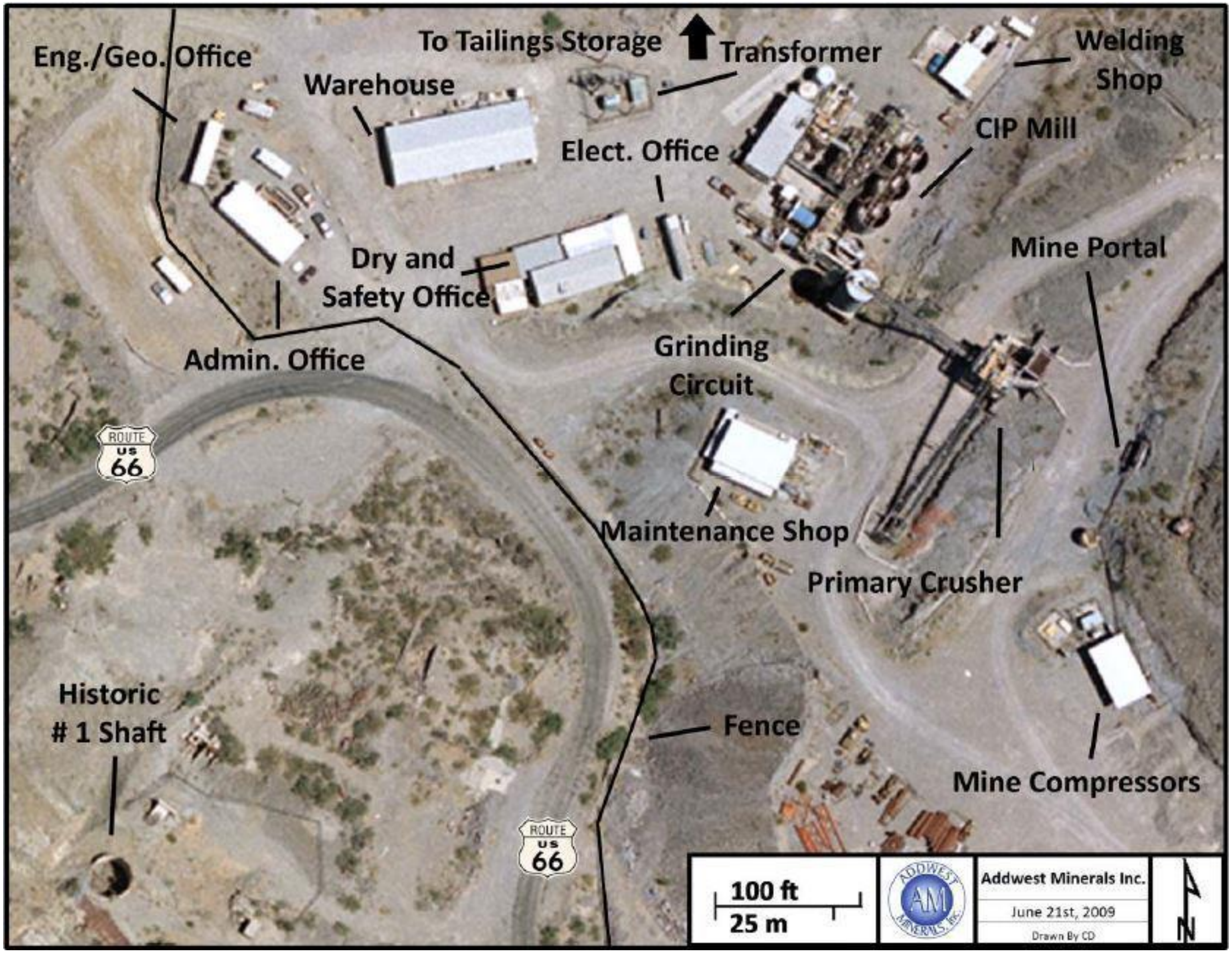
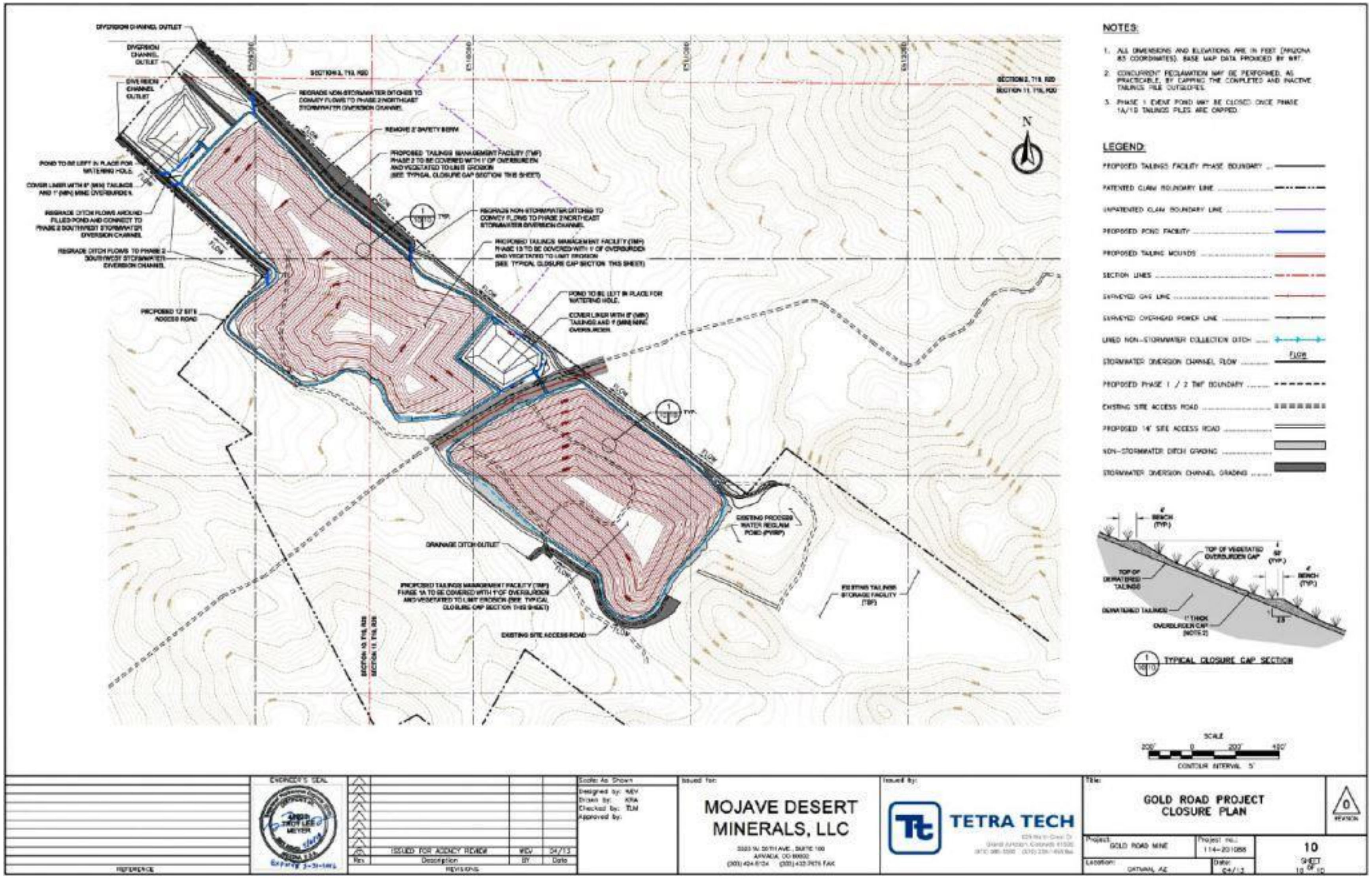


Figure 17-3 Tailings Storage Facility



Ore from the mine is either directly dumped in the primary-crusher dump hopper or reclaimed from the adjacent stockpile by a front-end loader and placed in the dump hopper. A vibrating feeder feeds the ore to a jaw crusher and the crushed product, in combination with the product of an adjoining cone crusher, is conveyed to a vibrating screen. Screen oversize feeds the cone crusher and screen undersize is conveyed to a mill feed bin.

The crushed ore is ground in two stages, firstly in a rod mill which is operated in open circuit and then in a ball mill which is operated in closed circuit with cyclones. The rod mill is more often operated as a ball mill rather than as a rod mill since the mill capacity and product size is similar when operated as a ball mill and it is more convenient to operate. Cyanide is added to the grinding circuit. The cyclone overflow is thickened and the thickener overflow flows to a process water tank to which tailings-reclaim water and makeup water are added to provide water for the grinding circuit. Thickener underflow is pumped to the leach circuit.

The leach circuit consists of three mechanically-stirred, air-sparged tanks in series providing 24 hours retention time. Leached slurry flows to a series of five mechanically-stirred CIP tanks with a total retention time of 15 hours. Stripped carbon is added to the last of the CIP tanks and is moved counter-current to the slurry flow using air lifts. The CIP tanks incorporate stationary screens. Slurry from the final CIP tank flows to a vibrating safety screen. Safety screen oversize is collected in an adjacent drum and the underflow flows to a mechanically-stirred cyanide-destruction tank. Hydrogen peroxide and copper sulfate are added to the cyanide-destruction tank to effect cyanide destruction. Slurry from the cyanide-destruction tank gravitates to a tailings filter plant.

Gold-loaded carbon from the first CIL tank flows to an acid-wash tank and then to a carbon-stripping column, both of which operate on a batch basis. Pregnant leach solution (PLS) from the stripping column is circulated through an electrowinning tank to recover the gold. Gold-loaded cathodes are periodically removed from the electrowinning tank and the gold washed from the cathodes. The gold-bearing sludge from the cathodes is dried in a cathode-sludge drying oven, then smelted in an induction furnace to produce doré metal.

Slurry from the cyanide-destruction tank flows to a tailings filter plant consisting of a tailings-slurry surge tank and plate-and-frame filter. The tailings filter plant is located about 500 ft northeast of the ore-processing plant. Tailings filter cake from the filter is conveyed to a dry-stack TSF. A drawing of the TSF is provided in Figure 17-3. Prior to 2010 about 0.4 million tons of tailings were placed as slurry in a conventional TSF located adjacent to the filter plant. In 2010 the tailings filter plant was installed and about 0.5 million tons of filtered tailings were placed on top of the slurry tailings, using portable conveyors and a stacker conveyor. In 2014 a semi-permanent conveyor was installed alongside the original TSF and placement of filtered tailings began in the Phase 1A area of the current TSF. A D7 tractor is used to distribute the tailings. Estimated capacity of the current TSF is 3.9 million tons. The quantity of tailings placed in the current TSF thus far amounts to about 0.2 million tons. Accordingly, available capacity of the current TSF is 3.7 million tons. Should additional TSF capacity be required in the future, the plan is to extend the TSF in the same direction northeastwards, following the axis of the existing and planned TSFs.

Filtrate from the tailings filter flows to a water-reclaim pond in the TSF area and is pumped from there to the process water tank.

The plant and TSF systems are in good condition, but some maintenance work is required prior to the restart. The maintenance work and estimated cost is discussed in Section 20 of this report.

18. Project Infrastructure

Infrastructure requirements for the Gold Road mine are minimal since it is close to established towns and has been in operation, albeit sporadically, for more than 100 years. The plant and mine decline adjoin an existing paved county road so there is need for an access road. Water requirement is minimal since the tailings are filtered and more than sufficient water is available from old shafts and from the mine. Grid power is supplied to the plant by the local utility. All the buildings required are already in place.

Figures 18-1 and 18-2 shows the general mine layout.

Figure 18-1 General Mine Layout

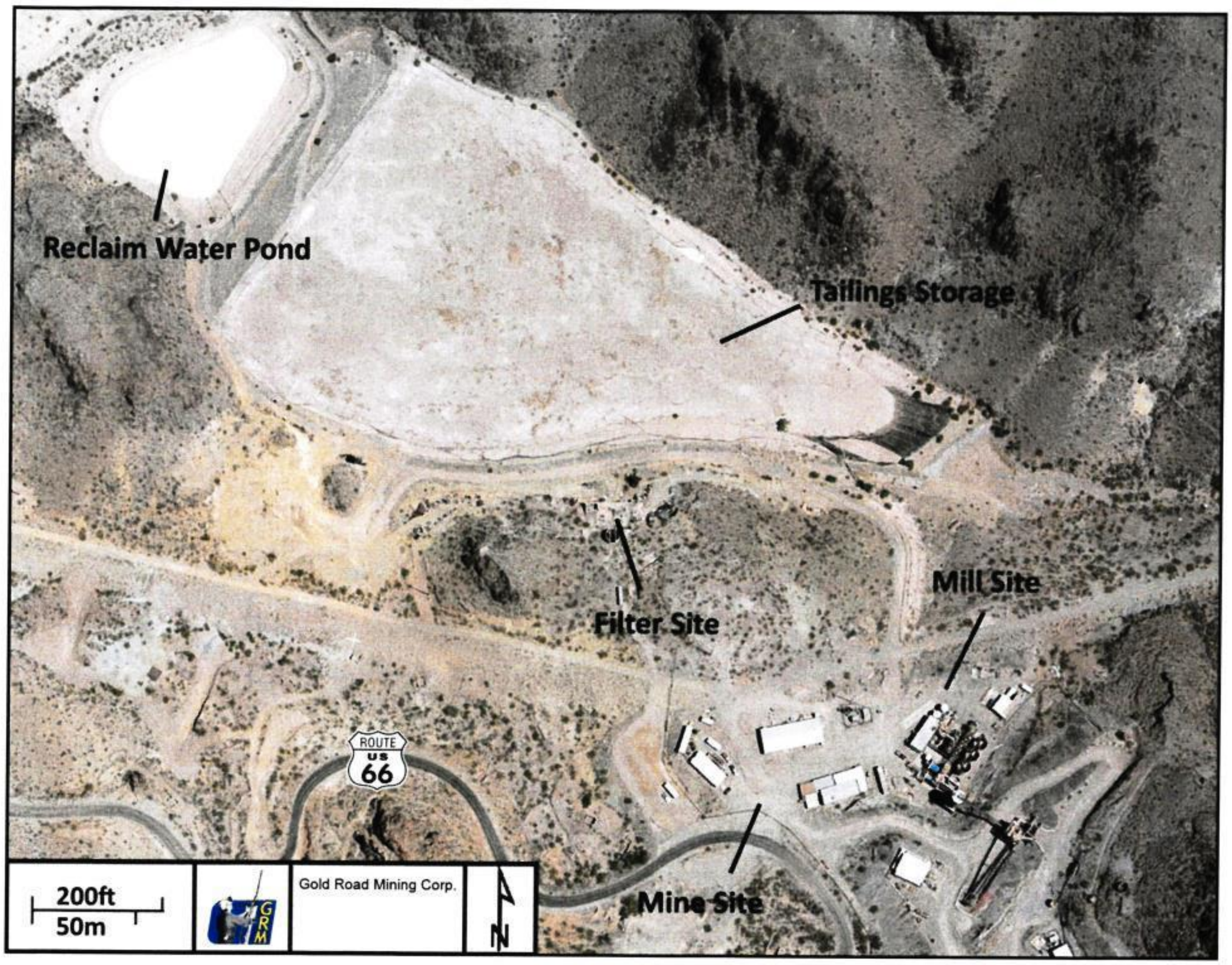
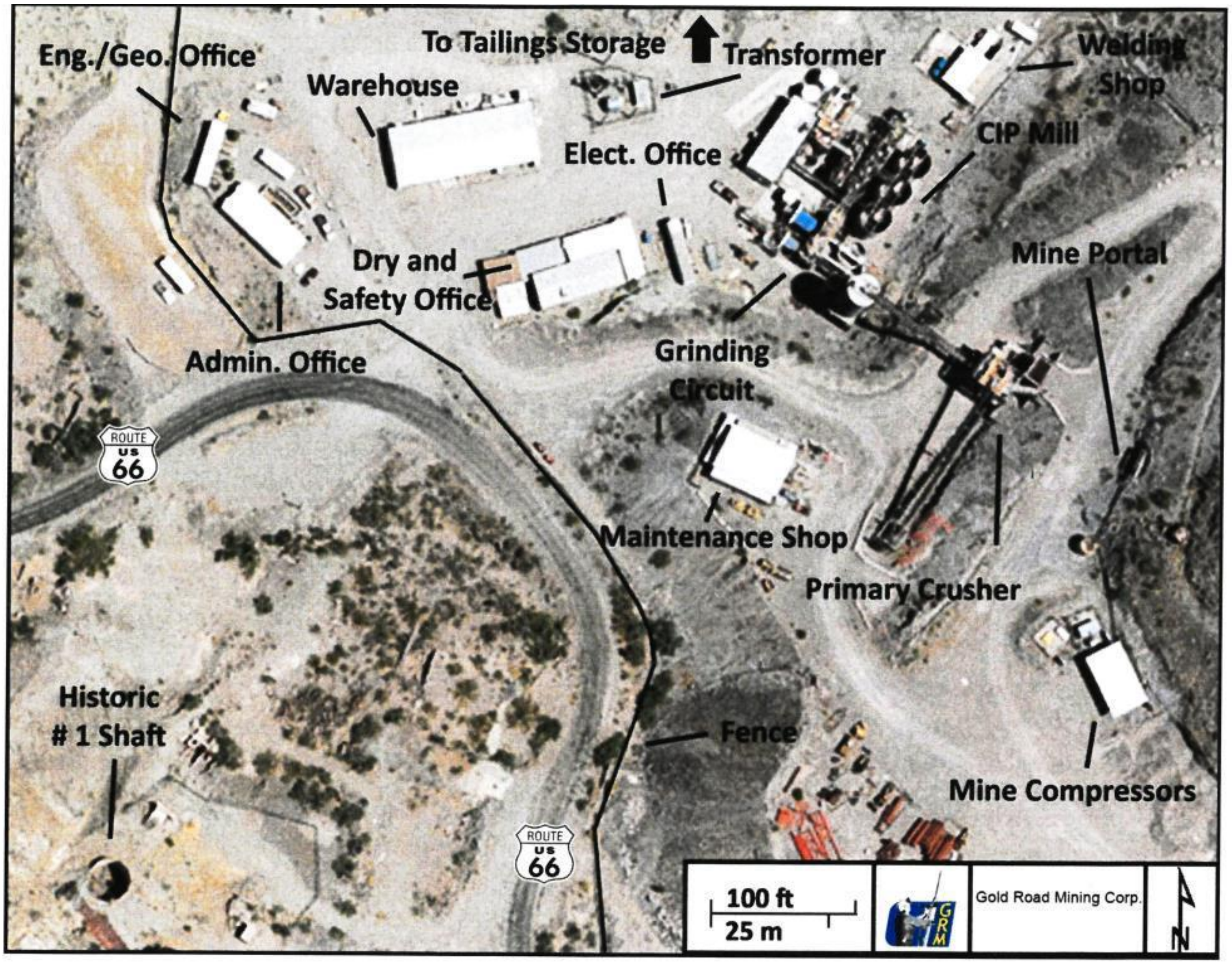


Figure 18-2 Layout of Mine Surface Facilities



A listing of the principal infrastructure facilities is shown in Table 18-1. The project site layout is shown in Figure 18-2. The layout figure shows the old tailings-storage facility (TSF), but does not include the current TSF. The infrastructure is in good condition and currently functional without need for repair or upgrading.

Table 18-1 Principal Infrastructure Facilities

Facility	Description
Roads	
Access	None required; mine and plant adjoin County Road 10 (US Route 66)
Internal	About 2 miles for access to TSF, water pumps
Water Supply	
Process	From pumps in old mine shafts, one at Oatman, the other near the mine
Domestic	From a well near the plant, used for all systems except drinking water
Water Storage	
Process	10-ft dia. x 12-ft high
Fresh	10-ft dia. x 8-ft high
Power Supply	
From grid	Unisource Energy Systems 65 kV line, use ~3 MW for plant and mine
Main transformer	Single unit, 65 kV to 4,160 V
Emergency	500 kW diesel, for plant leach and CIP tank agitators and thickener rakes
Buildings	
Change house/safety office	~40- x 100-ft
Mobile equipment shop	~40- x 60-ft
General office	~25- x 40-ft
Geology office	~10- x 20-ft
Warehouse/laboratory	40- x 100-ft, 14-ft eve height; 1/2 warehouse, 1/2 laboratory

The Property is easily accessed by paved roads from the cities of Kingman and Bullhead City. The Mine is approximately 25 miles (40 km) southwest of Kingman via historic U.S. Route 66 (Oatman Highway) and is approximately fourteen (14) miles (23 km) southeast of Bullhead City via the Oatman Highway and Boundary Cone road. A Mohave County-maintained gravel road (Silver Creek Road) serves as an alternate access route from Bullhead City.

Electrical power is supplied to a sub-station at the Gold Road Mine by UniSouce Energy at 69 kva. Two transcontinental natural gas pipelines, operated by Transwestern Gas Pipeline Company and Questar Pipeline Company, cross The Property. Both Kingman and Bullhead City have airports capable of handling commercial and passenger air services. Kingman is also served by the Burlington Northern Santa Fe Railroad and is a major transportation hub on U.S. Interstate Highway 40 (I-40).

A significant labor force is available in the area and due to the current lull in the industry; the re-hiring of a percentage of past employees to staff the mine is expected.

Potable water is pumped out of multiple water wells on The Property, while the inflow of ground and rainwater into the mine supports the demand of the mill and mine alike.

The property is a zero discharge facility.

Projected administrative personnel for the operation are shown in Table 18-2. The majority of the personnel costs have been accounted for in the mining costs due to management of the mine and contractor.

Table 18-2 Administrative Personnel Requirements

Personnel	Number
General Manager	1
Receptionist/Clerk	1
Accountant/Purchasing Agent	1
Warehouseman	1
Personnel & Security Manager	1
Environmental/Permitting Specialist	1
Total	**6**

Projected personnel for the ore-processing facilities are shown in Table 18-3. The plan is to run the plant for 14 days each month for the first six months of operation and then full-time thereafter. For the initial six month period, when the mill will be operated half the time, the mill operating and maintenance personnel will work 12-hour shifts per day for 14-days and then have 14 days off until the next work period.

Table 18-3 Ore-Processing Personnel Requirements

	Number	
Personnel	**250 tpd***	**500 tpd**
Operations		
Crusher operators	2	3
Grind & leach operators	2	4
Stripping operators	1	2
Filter/tailings operators	2	3
Laborers	1	1
Maintenance		
Mechanics	2	3
Electricians	1	2
Laborers		1
Supervision		
Superintendent	1	1
Foreman	1	1
Clerk	1	1
Assaying		
Technicians	3	3
Total	**17**	**25**

* Mill operated at 500 tpd for 14 days/month

19. Market Studies and Contracts

The only product of the operation is doré bullion. As in the past, the metal will be sold to an established refinery. Anticipated refining terms are as follows:

Treatment charge:	US$1.00/troy ounce shipped
Gold payment:	99.9%
Silver payment:	99.5%

The only anticipated contract of any consequence is that for mining.

20. Environmental Studies, Permitting and Social or Community Impact

The Gold Road mine is within an historical mining area within a state that rely on mining for a significant portion of their revenue. The Gold Road Mine and processing facility are on private ground and subject to only the environmental regulations of Arizona. The operation was active as late as 2016.

The Company stated all necessary permits for mining and production are in place. The most recent permit amendment allows for toll milling at the Gold Road Mill of mineralized material similar to those of the Gold Road Mine.

Current Permits:

- Aquifer Protection Permit (APP) 2015 – Permit No. 102805
- Air Quality Control Permit – No. 65238 as amended LTF No. 67979
- Permit to Appropriate Public Water of The State of Arizona – Permit No. 33-96287-000
- Nationwide Permit 404 – File No. 930128500 (Clean Water Act)
- EPA NPDES Storm Water Discharge Permit – Permit No. AZCN68776
- NPDES Construction Storm Water Permit – Permit No. AZCN68776
- Mining Safety and Health Administration Mine Identification # 02-02620

There are no reported issues related to adverse social or community impact since the mine has been in periodic production since 1994. During one of the shutdown periods in the early 2000's Gold Road was in operation as a tourist mine, restaurant and gift shop. The site (located on Route 66) was a very popular tourist destination until it was decided to shut down this activity and start up mining again. Between mining and tourism Gold Road has been a positive contributor to the surrounding communities of Oatman and Kingman in Mohave County Arizona.

21. Capital and Operating Costs

21.1 Summary

The following section summarizes the capital and operating costs for each major section for the Gold Road mine operations. The total LOM capital cost estimate is US$24.2M and includes US$5.7M in preproduction capital, US$6.45M in sustaining capital, US$6.0M detail core drilling and US$6.0M in property payments.

The total operating costs for the LOM is estimated to be US$110.4M with an average unit cost of US$99.45/ ton of mineralized material. The operating costs are:

- Mining (US$64.90/ton-ore)
- Processing cost (US$27.06/ton)
- General and Administrative cost (US$5.89/ton)
- Other (US$1.60/ton)

21.2 Capital Costs – Underground Mining

The total capital costs for the restart of the Gold Road Mine include US$5.4 M for the restart, preliminary development and shaft/hoist installation and purchase of the raise climbers completed in Year 1. All other development and other capital cost are considered to be sustaining capital for the remaining LOM.

21.2.1 Underground Development

The underground development is an ongoing process and will be completed by the contractor. The monthly requirements will ramp up to approximately 100 ft. of advance in waste per month. The advance requirements are based on the mine plan with advance in the hanging wall along-side the development drift advanced in the vein. The access drives over to the vein are designed to be 30 ft. long and are driven perpendicular to the vein. The access drift is developed to be able to store the raise climber in a suitable nest in the back.

21.2.2 Underground Equipment Capital

The restart plan assumes that an experienced mine contractor is hired to complete the restart and ramp up the production to the planned 500/tpd over an estimated 18 month period. The contractor will be responsible for the mobile equipment including loaders and haul trucks suitable for the mine plan. Gold Road will purchase the Raise Climbers for approximately US$500,000 each with one purchased in Year 1 and two in Year 2.

21.2.3 Main Production Shaft

The review of multiple ore/waste haulage scenarios was completed as part of haulage trade off study completed by RPM for Gold Road. The results of the study were that for the initial restart of the mine operation should use 20 ton underground haul trucks since the mine production could start in a relatively short time frame. The review also concluded that a shaft haulage scenario should be implemented as soon as possible due to the higher truck haulage costs expected with a one way haul distance of 11,000 ft. Two alternatives were reviewed for shaft conveyance including the rehabilitation of the existing No 3 shaft compared to the construction of a new shaft using a raise bore for the development of the 900 ft shaft. The new 8 ft diameter shaft will be used for hoisting ore/waste only with men and material accessing the mine from the existing decline. Table 21-1 is a summary of the cost for the preferred alternative of developing the 8 ft diameter shaft.

Table 21-1 New Shaft Estimated Cost (US$)

Area	Total	Q 1	Q 2	Q 3	Q 4
Hoist / Headframe	$500,000	$300,000		$50,000	$150,000
Freight	$40,000	20,000	20,000		
Concrete Pad / Collar	$50,000	$50,000			
Hoist house	$200,000		200,000		
Power	$50,000		50,000		
Pilot Hole	$165,000	75,000	90,000		
Raise bore	$550,000		375,000	175,000	
Material Handling	$86,000		60,000	26,000	
Final Construction	$220,000			160,000	60,000
Total	**$1,861,000**	**$445,000**	**$795,000**	**$411,000**	**$210,000**

- Gold Road has an opportunity to lease a headframe and hoist through a long term lease arrangement – after the first year the lease payments are part of the sustaining costs
- The hoist house is a suitable steel building that will house the hoist, control gear and power circuits
- The pilot hole is drilled by the contractor
- The back reaming process is required to create the 8 ft. diameter shaft performed by the contractor
- Material handling is for the haulage of the cuttings, from the reamer, to the surface by truck. There is the possibility to review placing this material in suitable old workings
- Final construction is the ground control and guide fitting process working from the top down of the new shaft. The new loading pocket will also be part of the final construction process.

21.2.4 Underground Infrastructure

The underground infrastructure is summarized in Table 21-3. These costs are relatively low since the mine was previously in operation.

Table 21-2 Underground Infrastructure

Area	Estimated Cost (US$ 000's)
Surface Requirements	
Refurbish air compressors	50
Upgrade Shop	20
Mine Rescue Station	60
Underground	
Ventilation	160
Pumps	75
Tractors	150
Upgrade Main haulage	150
Capitalized Labor	
Mechanics	30
Electricians	18
Miners	50
First Fills	
Drill supplies	70
Ground control & electrical	130
Total (US$ 000's)	**US$963**

21.3 Sustaining Mine Capital

Sustaining capital includes all of the estimated cost to maintain the mine operations at the required 500 tons per day level. The costs include waste development and are estimated to average US$1.16M per year for the LOM.

21.4 Capital Costs – Processing

The overall capital costs for the processing are estimated to be US$0.5M and are relatively low due to the excellent condition of the existing mill. Table 21-3 is a summary of the processing plant refurbishment project.

Table 21-3 Mine Operating Costs

Item	Cost (US$ K)
Crusher liners	15
Primary mill liners	20
Secondary mill liners	60
Secondary mill bolts with washers	8
Secondary mill motor rebuild	60
Electrical supplies	10
Cyclone spare	5
Thickener u'flow pump rebuild	10
Process water pump spare	10
CIP screens bolts and nuts	5
Reclaim water pump and spare	16
Filter cloths and spare pimp	55
Rubber lining agitators	5
Water sprays on chutes	1
Oil changes for crushers, mills, and filter	6
Fuel tank for FEL and truck	3
Miscellaneous supplies	10
Clean out emergency pond	20
Assay lab supplies	2
Bobcat	15
Mechanic labor (3 for 1,000 hours)	30
Electrical labor (2 for 600 hours)	18
Cleanup labor (2 for 80 hours)	2
Grinding balls	50
Reagents Cyanide	20
Carbon	20
Nitric acid	10
Copper sulfate	10
Refinery	5
Lifting the tailings conveyors (240 hours)	7
TOTAL	**US$508**

Estimate Date 3rd Q. 2017
Projected Accuracy -5/+20 %

21.5 Operating Costs

21.5.1 Mining

The underground mine operating costs were developed on a per foot of advance basis for the waste development. These estimates include contractor's costs and are included in the sustaining capital cost section above and are not part of the operating costs. The remaining operating costs, includes raise mining (in ore), production mining, contractor labor and management and Gold Road site management for the underground mine. Table 21-4 describes these unit costs and summarizes the estimated LOM weighted average cost per ton mined. These costs were used in the economic model described in section 22 of this report.

Table 21-4 Mine Operating Costs

Area	Category	US$ / ton ore	Estimated Annual US$000's	LOM Total US$000's
Fixed Cost				
	Gold Road			
	Management	$3.65	$635	$3,920
	Hoist Lease	$1.29	$224	1,380
	Contractor			
	Management	$4.33	$753	$4,648
	Labor	$37.88	$6,590	$40,668
Sub total		$47.14	$8,202	$50,616
Variable				
	Raise Stope Mining	$9.49	$1,652	$10,541
	Truck Haulage	$1.28	$223	$1,378
	Shaft Haulage	$8.42	$1,466	$9,373
Sub Total		**$19.20**	**$3,341**	**$20,616**
Total		**$66.34**	**$11,543**	**$72,057**
Total Tonnage	**1,110,274**			
Average Annual	**174,000**			

- Raise mining costs include the costs to develop the stope block in ore. The raise climber is used to develop the raise from the sill level to the top cut level nominally 200 ft. along the dip of the vein. The advance rate is 16 ft. per day. All of the material is considered ore.
- Production mining costs include the drilling and blasting costs for the ore within the stope accessed from the raise climber. There is no ground control cost due to the raise mining methodology
- Haulage Cost include the cost to load and haul the ore from the muck bay to the surface using 20 ton haul trucks and the existing decline during the initial restart. Once the shaft has been developed and is operational the ore/waste is transported using the haul trucks to the new shaft loadout on the 900 level. Underground ore/waste transfer to the loadout, hoisting and transfer costs of the ore to the surface stockpile area near the crusher at the mill have been estimated to be US$9.00/ton of material moved.
- Contractor Labor includes the costs for all of the labor underground with the three crew roster. The hourly crew includes; miners, mechanics and electricians as required.
- Contractor Management cost is for the contractor overhead with a site manager, three shift bosses, safety and accounting functions required by the contractor.
- Site management overhead includes the mine manager, engineering, surveyor, environmental and accounting allocations for the underground mine. There are no corporate allocations included in this estimate.

21.5.2 Processing

Operating costs for the processing are estimated to be US$27.06/ton of mineralized material. Table 21-5 is a summary of the expected processing operating cost.

Table 21-5 Processing Operating Cost

Commodity	Consumption Rate	Rate Basis	Commodity Cost	Cost Basis	Operating Cost (US$/ton)
Labor	25	Employees	70,000	US$/employee/year	9.72
Power	65	kWh/ton	0.095	US$/kWh	6.18
Maintenance Materials		fixed cost/ton	1.00	US$/ton	1.00
Miscellaneous		fixed cost/ton	0.50	US$/ton	0.50
Mobile Equipment					
Cat 950 FEL	12	hours/day	45.00	US$/hour	1.08
Cat D7 bulldozer (rental)	4	hours/day	50.00	US$/hour	0.40
Bobcat	4	hours/day	10.00	US$/hour	0.08
Truck crane (rental)	8	hours/month	30.00	US$/hour	0.02
Forklift	2	hours/day	10.00	US$/hour	0.04
Water truck	2	hours/day	10.00	US$/hour	0.04
Pickup	4	hours/day	10.00	US$/hour	0.08
Crusher Liners					
Jaw crusher	4	rebuilds/year	10,000	US$/rebuild	0.22
Cone crusher	4	sets/year	11,000	US$/set	0.24
Mill Liners					
Primary mill	0.75	set/year	140,000	US$/set	0.58
Secondary mill	1	set/year	90,000	US$/set	0.50
Grinding Balls					
Primary mill	2	lb/ton	1,000	US$/ton	1.00
Secondary mill	4	lb/ton	1,000	US$/ton	2.00
Reagents					
Quicklime	2.00	lb/ton	125	US$/ton	0.13
Cyanide	1.00	lb/ton	3,250	US$/ton	1.63
Flocculent	0.02	gallons/ton	5	US$/gallon	0.10
Carbon	0.15	lb/ton	2,500	US$/ton	0.19
Copper sulfate	0.15	lb/ton	3,250	US$/ton	0.24
Hydrogen peroxide	3.00	lb/ton	480	US$/ton	0.72
Other	0.50	lb/ton	600	US$/ton	0.15
Total					3.15
Filter Cloths	135	cloths/year	300.00	US$ each	0.23
TOTAL					**27.06**

Estimate Date Q3 2017
Projected Accuracy -5/+10 %

21.5.3 G&A Costs

The estimated G&A costs are expected to be approximately US$5.89/ton or US$1.03M per year. The expense costs including administration, environmental, taxes, licenses and fees. The costs for salaries have been included in the Mining Cost section due to the direct management requirement of the contractor as well as the other administrative duties. Table 21-6 is a breakdown of these estimated costs. The projected accuracy is -5/+20%. Other general costs were estimated to be US$250,000 per year or US$1.60/ton ore.

Table 21-6 G & A Operating Cost Estimate

Item	Annual Cost (US$ 000 year)	Unit Cost (US$/ton)
Administration		
Rentals and Leases	171	0.95
Postage and Shipping	59	0.33
Communications	56	0.31
Employee Relations	16	0.09
Public Relations	16	0.09
Insurance	63	0.35
Environmental		
Permitting	43	0.24
Audits	85	0.47
Taxes, Licenses, and Fees	551	3.06
TOTAL	**1,060**	**5.89**

Estimate Date Q3 2017
Projected Accuracy -5/+20%

22. Economic Analysis

The following section summarizes the economic analysis that was completed for the Gold Road Project. All currency is in US$ with a base case of Year 1 without any escalation beyond that date. For the net present value (NPV) estimation all net cash flows are discounted at 5% from the base date. The metal price selected was US$1,200 per ounce and is based on a consensus long term estimate. A regular tax rate of 21% for federal tax, 4.9% for Arizona tax and 2.5% for the Arizona severance tax rate was also applied. To calculate the net revenue there is a 2.0% Net Royalty and 0.5% net refining fee applied to the gross revenue from the sale of the doré.

22.1 Economic Analysis

The total capital costs for the restart of the Gold Road Mine include US$5.24M for the restart, preliminary development and shaft/hoist installation and purchase of the raise climbers completed in Year 1 and 2. All other development and other capital, is considered to be sustaining capital for the remaining LOM. Table 22-1 shows the key economic assumptions.

Table 22-1 Gold Road – Key Economic Assumptions and Results

Gold Road	Units	LOM Value
Mineralized Material	Tons	1,110,274
Gold - Mined Grade	Gold oz. per ton	0.19
Gold Recovery	%	95%
Payable Gold	oz.	203,569
Gold Price	US$/oz.	$1,200
Net Revenue	US$ 000's	$238,175
Capital Cost	US$ 000's	$5,744
Sustaining Capital	US$ 000's	$6,454
Total Capital	US$ 000's	$12,198
Total Operating Cost	US$ 000's	$110,362
Total All-in Sustaining Cost (AISC)	US$/oz. gold	$632.79
Total All in Cost	US$/oz. gold	$659.29
Payback Period	Year	1.5
Cumulative Net Cash flow	US$ 000's	$103,964
Pre Tax NPV @ 5 %	US$ 000's	$81,309
Pre Tax IRR	%	238%
Post Tax NPV @ 5 %	**US$ 000's**	**$56,739**
Post Tax IRR	**%**	**175%**

22.1.1 Economic Model

The following table (Table 22-2) is a summary of the annualized economic model for the Gold Road Project through the current LOM based on the Inferred resources of 977,784 tons diluted to a LOM amount of 1,110,300 tons of mineralized material.

Table 22-2 Gold Road Economic Model

Gold Road - Economic Model — *RPM - Resources Only*

500 tons per day
Raise - Stope Mining (RAM) with Contractor

	Tons	Grade (oz gold / ton)	Ounces
Inferred Resources	977,784	0.219	214,135
13.55% Dilution			
Mineable Resources	**1,110,274**	**0.193**	**214,135**

	Rate (US$/ton)	Units	Total	Year 1	2	3	4	5	6	7
Underground Mine										
Mineralized Material		**ton**	**1,110,274**	**68,875**	**134,850**	**174,000**	**174,000**	**174,000**	**174,000**	**210,549**
Cumulative		ton		68,875	203,725	377,725	551,725	725,725	899,725	1,110,274
Average Tons per day		Tpd	250	68,875						
			350		134,850					
			500			174,000	174,000	174,000	174,000	210,549
Waste		ton	364,063	34,438	67,425	87,000	1,200	87,000	87,000	-
Development - Lateral		ft	-			-	-	-		
Development - Vertical										
Raise		ft	-							
Shaft		ft	1,100	1,100						
Total Mill Feed		ton	1,110,274	68,875	134,850	174,000	174,000	174,000	174,000	210,549
Gold Grade		oz/ton	0.193	0.193	0.193	0.193	0.193	0.193	0.193	0.193
Overall Recovery		%	95%	95%	95%	95%	95%	95%	95%	95%
Total Payable Metal										
Gold		**ounces**	**203,569**	**12,628**	**24,725**	**31,903**	**31,903**	**31,903**	**31,903**	**38,604**
Metal Price		**US$/oz.**		**$ 1,200**						
Gross Revenue		**US$'000**	**$ 244,282**	**15,154**	**29,670**	**38,283**	**38,283**	**38,283**	**38,283**	**46,325**
Net Revenue (97.5% of Gross)										
Includes 2% Royalty and 0.5 % Refining Charge		**97.5%**	**$ 238,175**	**14,775**	**28,928**	**37,326**	**37,326**	**37,326**	**37,326**	**45,167**
			TRUE							
Total Costs	**US$/ton**		US$ 000's							
Gold Road										
Fixed Cost - Management			**3,920**	560	560	**560**	**560**	**560**	**560**	**560**
Hoist Lease			**1,530**	-	500	**250**	**240**	**180**	**180**	**180**
Contractor										
Fixed Cost										
Management			4,648	616	672	**672**	**672**	**672**	**672**	**672**
Labor			40,668	4,960	5,951	**5,951**	**5,951**	**5,951**	**5,951**	**5,951**
Variable										
Raise Stope Mining	$ 9.49	US$'000	10,541	654	1,280	**1,652**	**1,652**	**1,652**	**1,652**	**1,999**
Truck Haulage	$ 20.00		1,378	1,378	-	**-**	**-**	**-**	**-**	**-**
Shaft Haulage	$ 9.00		9,373	-	1,214	**1,566**	**1,566**	**1,566**	**1,566**	**1,895**
Total Mining		*TRUE*	**$ 72,057**	**$ 8,167**	**$ 10,177**	**$ 10,651**	**$ 10,641**	**$ 10,581**	**$ 10,581**	**$ 11,257**
US$ / ton - ore				**$ 119**	**$ 75**	**$ 61**	**$ 61**	**$ 61**	**$ 61**	**$ 53**
Processing	$ 27.06		30,043	1,864	3,649	**4,708**	**4,708**	**4,708**	**4,708**	**5,697**
G & A	$ 5.89	US$'000	6,540	406	794	**1,025**	**1,025**	**1,025**	**1,025**	**1,240**
Other Costs			1,722	210	252	**252**	**252**	**252**	**252**	**252**
Total Processing and G & A		*TRUE*	**$ 38,305**	**2,479**	**4,695**	**5,985**	**5,985**	**5,985**	**5,985**	**7,189**
Salvage Value										
Reclamation & Closure										
Total Operating Cost		*TRUE*	**$ 110,362**	**$ 10,646**	**$ 14,873**	**$ 16,637**	**$ 16,627**	**$ 16,567**	**$ 16,567**	**$ 18,447**
Total Operating Cost / ton - ore		**$ 99.40**		**$ 154.57**	**$ 110.29**	**$ 95.61**	**$ 95.56**	**$ 95.21**	**$ 95.21**	**$ 87.61**
Project Capital	***Use Contractor***									
New Shaft			1,894	1,894						
Mine Equipment	3 - Raise Climbers		1,500	1,500						
Restart			1,500	1,500						
Processing			500	500						
Sustaining Capital			-							
Development - Lateral		ft	6,700	**700**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**-**
	$ 750.00	Cost/ft		525	900	900	900	900	**900**	**-**
	$ 20.00	Cost / ton		149	256	256	256	256	**256**	**-**
	Total cost		**$ 6,454**	**674**	**1,156**	**1,156**	**1,156**	**1,156**	**1,156**	**-**
Development - Vertical	Included - Mining		-							
Development Drilling			6,000	2,400	1,200	1,200	1,200	-	-	**-**
Property Payment			6,000	1,000	1,000	1,000	1,000	1,000	1,000	**-**
Total Capital			**$ 23,849**	**$ 9,469**	**$ 3,356**	**$ 3,356**	**$ 3,356**	**$ 2,156**	**$ 2,156**	**$ -**
			TRUE	*TRUE*						
Undiscounted cash flow (pre tax)			**$ 103,964**	**$ (5,340)**	**$ 10,699**	**$ 17,334**	**$ 17,344**	**$ 18,604**	**$ 18,604**	**$ 26,720**
Cumulative		**$ 103,964**		**$ (5,340)**	**$ 5,359**	**$ 22,693**	**$ 40,037**	**$ 58,641**	**$ 77,244**	**$ 103,964**
Discounted Cash Flow	NPV-0%		103,964	*TRUE*						
	NPV-5%		81,309							
	NPV-8%		70,713							
	IRR %		238%							
Income Tax	21.00%		**$ 31,078**	**36**	**3,039**	**4,923**	**4,926**	**5,283**	**5,283**	**7,588**
Estimates for PEA level	4.90%		*TRUE*							
	2.50%									
	28.40%									
Undiscounted cash flow (Post tax)		*TRUE*	**$ 72,886**	**(5,376)**	**7,661**	**12,411**	**12,418**	**13,320**	**13,320**	**19,132**
Discounted Cash Flow	NPV-0%		$ 72,886							
	NPV-5%		$ 56,739							
	NPV-8%		$ 49,193							
	IRR %		175%							

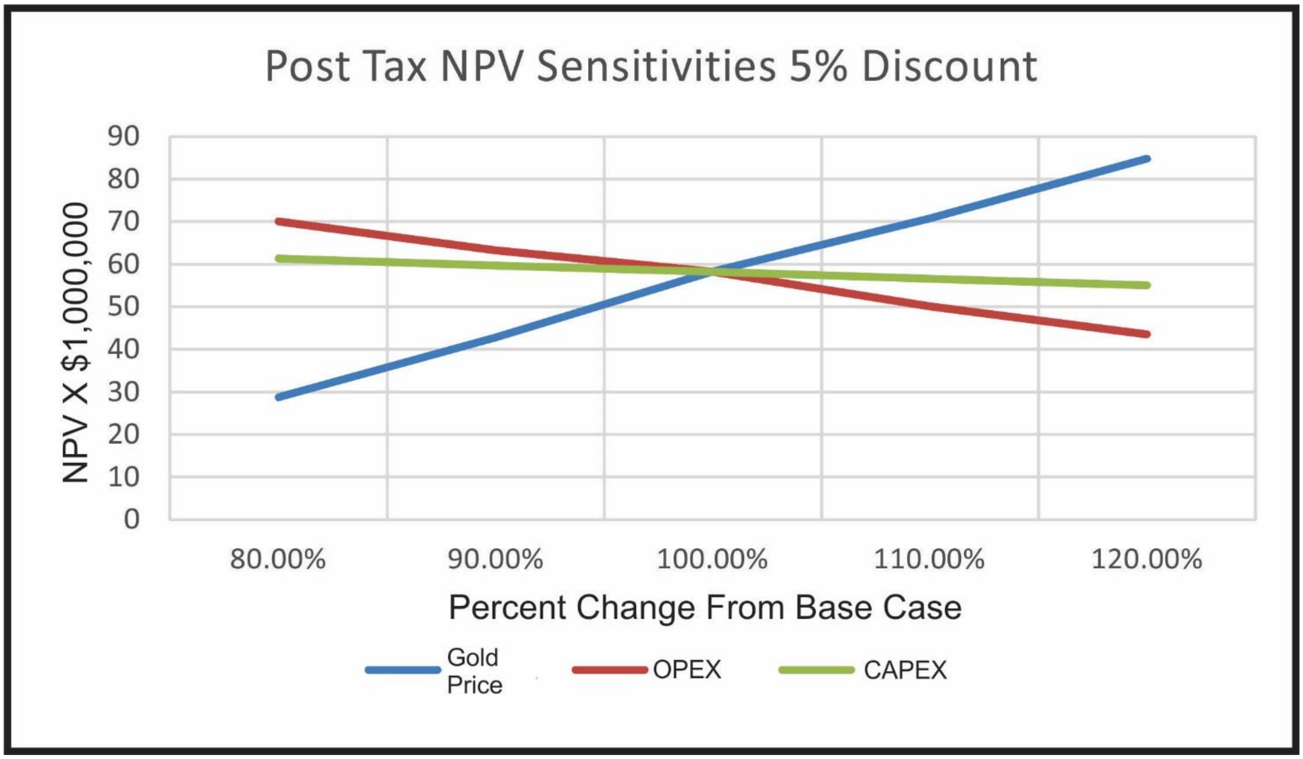
22.1.2 Sensitivity Analysis

RPMGlobal carried out a sensitivity analysis for the Gold Road project based on the economics as previously described. All of the sensitivities were carried out at the post tax NPV@5% discount rate. The sensitivities were carried out a 20% positive or negative change in gold price, operating costs and total capital costs. The results of the analysis show that the post-tax NPV@5% remains robust for the range of sensitivities. Only one variable was changed for each economic factor while the other two remained constant.

Table 22-3 Economic Sensitivity Analysis

	Percent Change from Base Case	**Gold Price US$**	**NPV at 5% Discount X $1,000,000**
Gold Price	Base Case	1,200	58.2
	-20.00%	960	28.7
	-10.00%	1,080	42.7
	10.00%	1,320	70.7
	20.00%	1,440	84.7
OPEX	Base Case	1,200	58.2
	-20.00%	1,200	70.0
	-10.00%	1,200	63.3
	10.00%	1,200	50.1
	20.00%	1,200	43.5
CAPEX	Base Case	1,200	58.2
	-20.00%	1,200	61.3
	-10.00%	1,200	59.7
	10.00%	1,200	56.6
	20.00%	1,200	55.0

Figure 22-1 Sensitivity Analysis Summary



The economic outcome of the project is most sensitive to the price of gold. The project is somewhat sensitive to OPEX while it is relatively insensitive to CAPEX.

22.1.3 Economic Assumptions

The following assumptions were used for the economic model review.

- The Overhead G&A does not include any corporate cost.
- The analysis was completed at the 100 % project level.
- Final property payments totalling US$6.0M have been included at US$1.0M per year.
- The full tax rate is 28.4% (Federal 21%, Arizona Income Tax 4.9%, Arizona Severance Tax 2.5%.
- Project financing is through 100% equity funding.
- All historic tax attributes have been ignored. There are no loss carry forwards. A full detailed tax review was not completed for this stage of the project. Subsequent project review should include these and other tax ramifications.
- Property Taxes were assumed to be a flat US$200,000 per year and are included in the G&A costs.
- There are no closure or reclamation costs included.
- There is no salvage value used in the model.
- All mining is through a contractor.

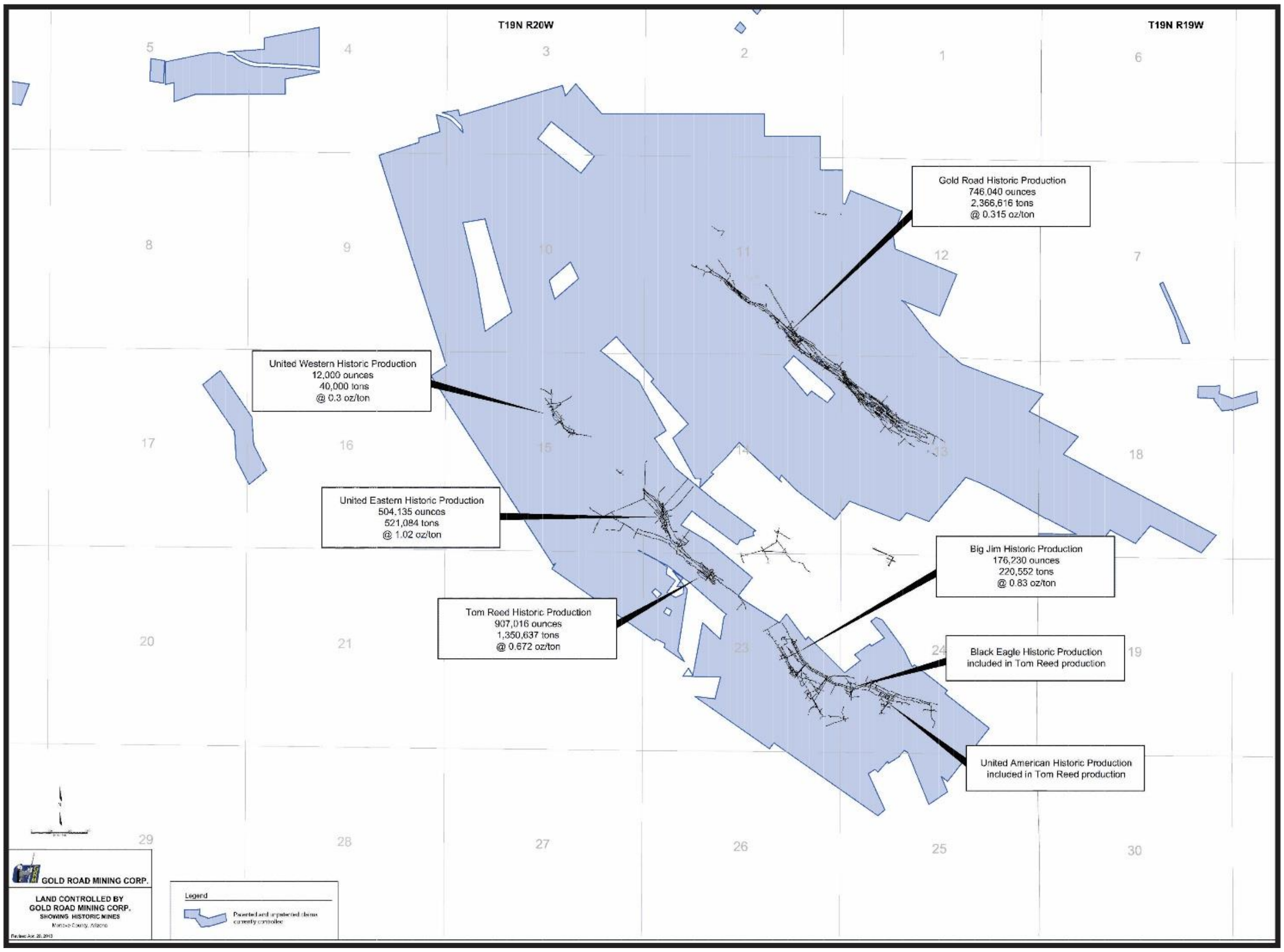
23. Adjacent Properties

The Oatman mining district has had long history and contains numerous mines that are all closely related to the same structural and hydrothermal settings. Table 23-1 shows the production for the district. Other than the Gold Road vein and mine, the most productive structure in the district was the Tom Reed/United Eastern vein system. Figure 23-1 shows the location of the mines in the district.

Table 23-1 Historic Production in the Oatman Gold District

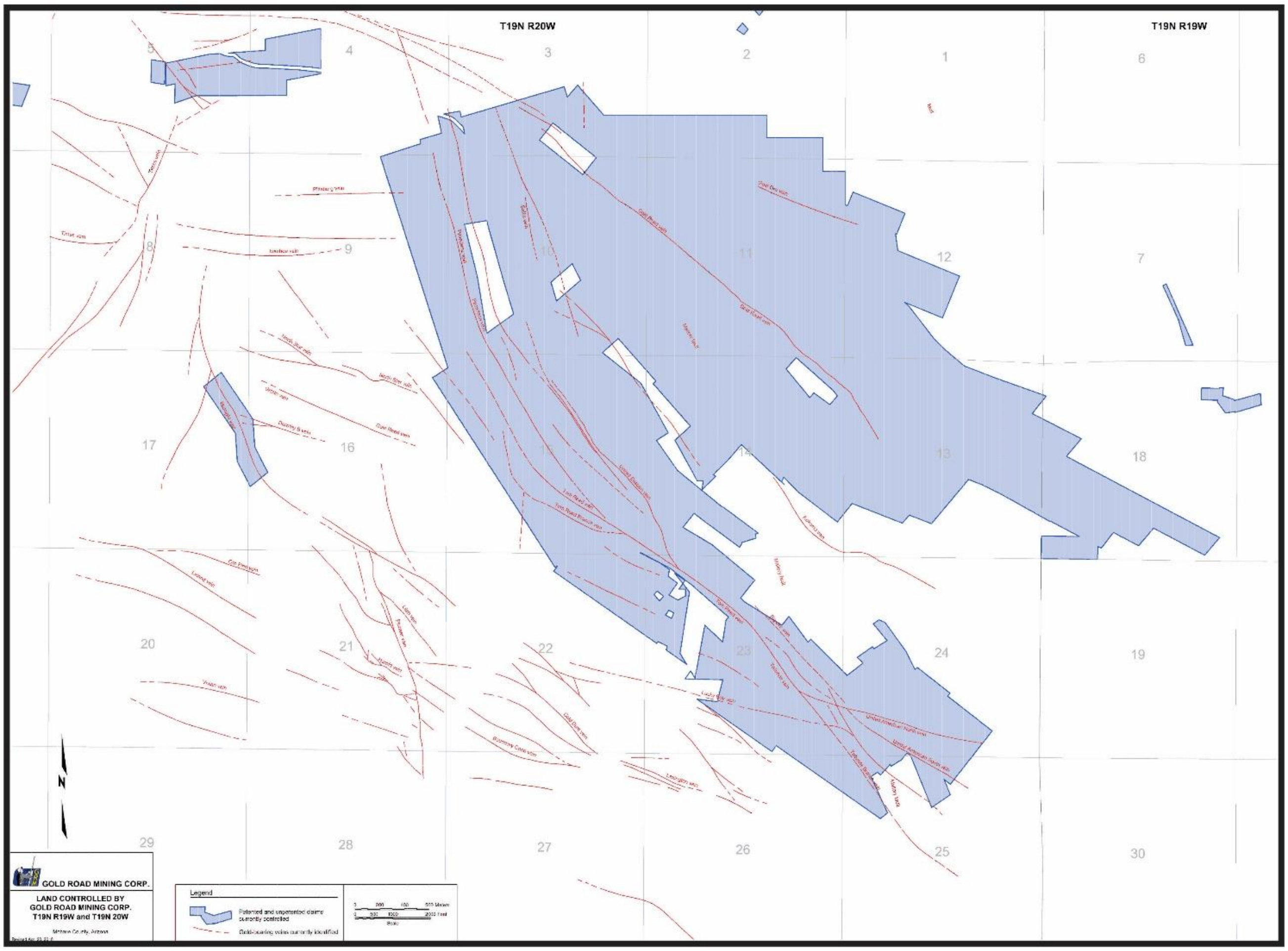
Mine	Production Period	Tons	Average Grade oz/t (g/t)	Gold Ounces Recovered
Gold Ore	1918-1932	12,931	0.58 (17.99)	7,500
Gold Road	1900-2015	2,366,616	0.315 (9.77)	746,040
United Western	1928-1940	40,000	0.30 (9.3)	12,000
United Eastern	1917-1923	550,000	1.12 (34.74)	616,000
Tom Reed/Tip Top	1915-1928	250,000	0.7 (21.71)	175,000
Ben Harrison	1897-1928	250,000	0.7 (21.71)	175,000
Big Jim/Aztec	1921-1924	500,000	0.75 (23.26)	176,230
Black Eagle	1920's	200,000	0.5 (15.51)	100,000
United American	1920's	140,000	0.5 (15.51)	70,000
Total		**4,309,547**	**0.48 (14.89)**	**2,077,770**

Figure 23-1 Location Map of Mines in the District



While the focus of the Company and this report is the Gold Road vein, the Company controls the ground that covers many of the major veins in the district including United Eastern, the United Western, and the Tom Reed/Tip Top. Figure 23-2 shows the outline of the Company's claim group and the location of the veins in the district.

Figure 23-2 Veins in the Oatman District



24. Other Relevant Data and Information

While the subject of this report is the Gold Road mine, the Company controls the claims that cover most of the major mines in the district. While the Company's focus in on the Gold Road mine, there is significant potential on the other veins in the system although access to them is more limited and establishing resources and reserves leading to production may take some time but once the Gold Road mine is in production, developing the other mines should be built into a long term plan.

24.1 Potential of the Gold Road Vein

The deepest ore-grade gold mineralization so far encountered at the Gold Road Mine was sampled on the 900 Level, which corresponds to an elevation of 2,100 ft (640 m) above sea level. The other principal mines in the district, the United Eastern and the Tom Reed, were productive between 1,500 and 2,800 ft (457 to 853 m) above sea level. If the Gold Road and Tom Reed-United Eastern vein systems are part of the same mineralizing event and the bottom of mineralization (or boiling zone interface) is similar, the Gold Road Mine may have an additional 600 vertical ft (180 m) of potentially mineralized ground to be explored. The areas of potential include down dip on the vein, lateral extensions of the vein, and areas of the vein above the 900 level that have not been adequately explored. Figure 24-1 shows these areas.

Underground core drilling took place at many sites within the mine at various times in the recent history of development and production. The drill results of those sites not subsequently mined are listed in the following Table 24-1 and shown in Figure 6-2 in the area in while below the 900 level. It should also be noted that all of these holes are within the Phase 1 exploration target area and serve as confirmation that both the vein and associated gold mineralization continue to below the level of the current workings. Note that the holes shown on the figure are the pierce point locations where the vein was intersected by the drilling.

Table 24-1 Underground Drilling Results

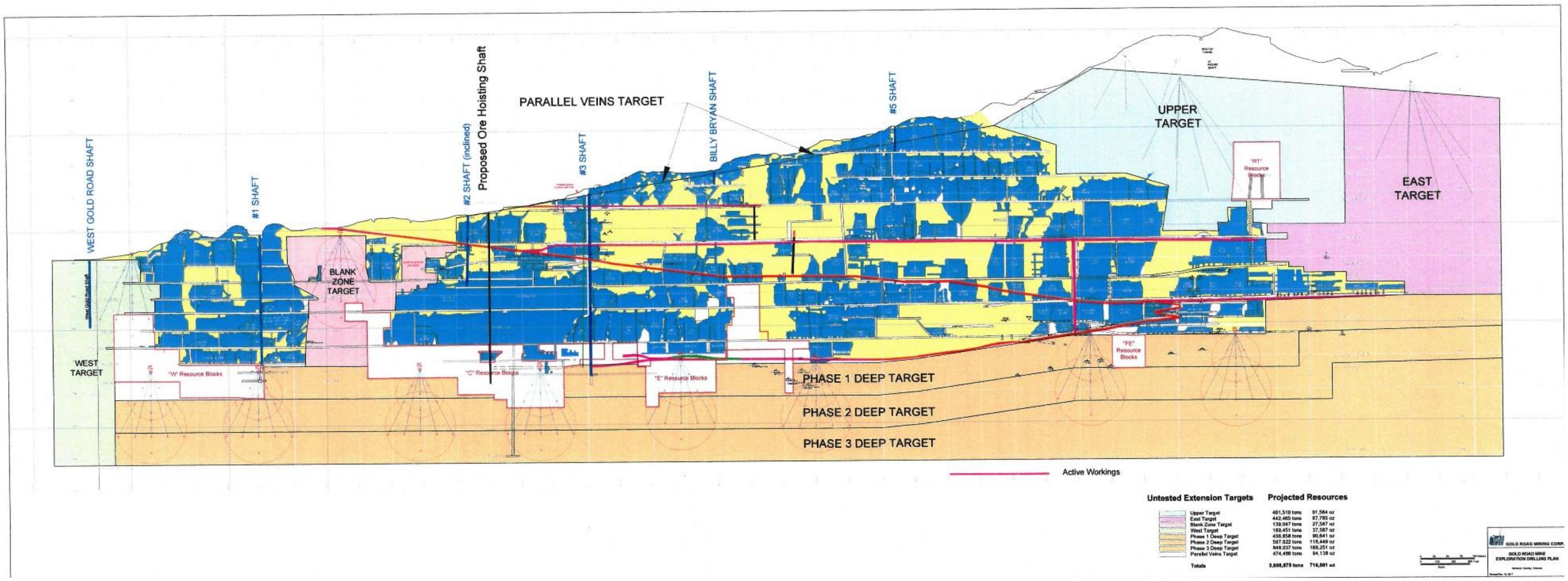
Hole Number	Vein Interval ft (m)	Gold Grade oz/t (gms/t)
DD-32-1B	3.0 (0.91)	0.85 (26.36)
DD-32-3	3.0 (0.91)	0.06 (1.86)
DD-32-4	8.8 (2.68)	0.23 (7.13)
DD-32-5	6.0 (1.83)	0.02 (0.62)
DD-32-6	1.0 (0.30)	0.14 (4.34)
DD-900-2	1.0 (0.30)	0.11 (3.41)
DD-900-3	8.5 (2.59)	0.01 (0.31)
DD-900-4	3.5 (1.10)	0.19 (5.89)
DD-900-9	2.1 (0.64)	0.93 (28.84)
DD-700-10	3.5 (1.07)	1.08 (34.69)
DD-850-1	0.7 (0.21)	0.04 (1.41)
DD-850-2	4.0 (1.22)	0.26 (8.46)

Guilinger (2017) states exploration potential for all the targets is 716,000 ounces of gold. The author of this report feels this estimate is aggressive and puts the potential from 446,000 ounces to 558,000 ounces at average grade of 0.20 to 0.25 ounces per ton.

24.2 Potential of the Remainder of the Company Claims

No data was reviewed to assess the potential of the additional veins within the claim block but the Company is currently reviewing data from these mines to determine the potential of the veins and to assess the operational issues of establishing access to these veins underground.

Figure 24-1 Areas of Exploration Potential



25. Interpretation and Conclusions

The Gold Road vein has been exploited intermittently since its discovery in the early 1900s. It one of several major gold producing veins in the Oatman Mining District which has produced over 2 million ounces of gold. The mine is accessed through a 11,000 foot long decline. The mine has been developed on 9 levels with vertical workings going to the 13th level. Mining has historically been by shrink stoping as other mining methods introduced excessive dilution. Processing has been done in a 500 tpd mill that with minimal work can be operational within three months. All infrastructure is in place and operational. All permits are approved and mining and processing can proceed.

Based on RPM's review, the following conclusions can be drawn:

- The geology is well understood.
- The resources are based on historical sampling of the drifts and stopes.
- The historical sampling has been verified by a sampling program that used industry standards for QA/QC.
- The resources were estimated using industry standard methodology.
- The processing is a CIL cyanide leach process that historically recovers 95% of the gold.
- The ore is very clean with no deleterious elements such as As, Hg, Se, Pb, and no sulfides to produce ADR.
- Mining over the years has produced a large amount of workings including several shafts (none of which are operational), several miles of drifts and a large number of stopes above the 700 Level.
- Mine access and ore haulage is via an 11,000 foot decline.
- Mining post 1996 has been a bit haphazard and has produced some areas within the mine that will likely require some modification to support mining at a level of 500 tpd.
- The decline will require some rehabilitation to facilitate ease of use.
- Exploration potential below the 900 Level is excellent with historical drill holes reporting ore grade intercepts and widths. These are not verified.
- The mill will require an expenditure of US$500,000 to become operational.
- The mine start-up will require an expenditure of approximately US$5.4 million to upgrade it to produce 500 tpd.

26. Recommendations

In order for the mine to be placed into operation, RPM makes these recommendations.

26.1 Exploration

There are sufficient resources that can be produced near term to supply the mill for 18 months to two years. It will be necessary to carry out an ongoing exploration program which will include drilling from underground and drifting of the vein on multiple levels to establish stope outlines.

Guilinger (2017) has put together a budget that is designed to define resources at a rate double that of production. Table 26-1 shows the budget for all the defined target areas. Figure 24-1 shows the areas of potential included in the budget.

Table 26-1 Exploration Target Summary and Multi-Year Budget

Area	Proposed Exploration	Cost
Deep Zone Phase 1	800 Feet of drifting; 9,000 ft of drilling down dip	$990,000
Deep Zone Phase 2	800 Feet of drifting; 10,000 ft of drilling down dip	$1,155,000
Deep Zone Phase 3	800 Feet of drifting; 13,600 ft of drilling down dip	$1,650,000
Upper Zone	9,000 ft of Drilling	$900,000
East Target	6,000 ft of drilling	$840,000
West Target	3,800 ft of drilling	$380,000
Blank Zone	2,700 ft of drilling	$270,000
Parallel Veins	50 percussion and 10 core holes	$425,000
	Total	**$6,610,000**

The drilling is designed to test the areas to establish the presence of the vein and mineralization. Underground resources are best defined by drifting on the vein and direct sampling of the vein once drilling has established the presence of the vein and mineralization. The budget is only an estimate of the level of expenditures and work required to define resources but will need to be detailed yearly to efficiently explore. It will be necessary to begin exploration immediately upon commencement of operations to ensure there are sufficient resources to feed the mill on an ongoing basis.

Exploration drilling should commence as soon as possible to ensure the framework for developing an ongoing mine plan is in place. Once the drilling has established the presence of the vein in an area, the mine plan should be developed to provide direct access to the vein.

RPM concurs with this exploration program.

26.2 Mining

Mining in the past has been by shrink stope. Pre-1942 mining used shafts to hoist the ore to the surface but in the post-1996 mining ore was hauled to the surface by trucks using the decline. This proved expensive and proved to be a bottleneck to the operation.

Based on this history, RPM would make the following recommendations:

- Engage a mining contractor to do all the development and mining.
- Change the mining method to Alimak mining as it is less expensive, and safer than shrink stoping.

- Use the decline for access and haulage initially but with the onset of production, begin to raise bore a shaft to use for moving ore from the mine to the surface.
- Once the shaft is in place only use the decline for moving men, equipment and supplies.

26.3 Processing

The mill is in generally good condition but will require some rehabilitation to get it functional including getting consumables to begin operations. RPM recommends the work on the mill and the ordering of supplies commence with the onset of underground exploration.

27. References

1. Gold Road Project Metallurgy and Historic Mill Summary. Document includes Metallurgical and Engineering Evaluation of Samples from the Gold Road Mine, April 29, 1993, performed for Addwest Minerals, Inc. by IC Technologies, Inc. Also includes Addwest Minerals, Inc., Gold Road Mine, Mohave, Arizona, Metallurgy, 1937-1942 Mill Operating Information and 1981 Metallurgical Testing

2. Gold Road Major Component Inventory (Excel spreadsheet)

3. Design Report Tailings Management Facility prepared for Mojave Desert Minerals, Inc., prepared by Tetra Tech, Inc., May 7, 2013

4. Mojave Desert Minerals, LLC, Cost Detail (Operating Cost Estimate)

5. GRM Monthly Production Summary (2010-2014) (Excel spreadsheet)

6. Kilborn Mill Drawings, 1994

7. Simple Mill Flowsheet

8. Start-Up Cost Estimate (Excel spreadsheet)

9. 2009 Technical Report on the Gold Road Mine prepared for Addwest Minerals, Inc., prepared by World Industrial Minerals, LLC, September 25, 2009

10. 2017 Technical Report on the Gold Road Mine prepared for Para Resources, Inc., prepared by World Industrial Minerals, LLC, November 30, 2017

28. Certificate of Qualified Person

Richard J. Kehmeier, C.P.G.
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I, Richard J. Kehmeier, do hereby certify that:

1. I, am a Chief Geologist of RPMGlobal USA, Inc. of 6251 Greenwood Plaza Blvd., Suite 275, Greenwood Village, Colorado 80111 United States. This certificate applies to the NI 43-101 Technical Report, Preliminary Economic Assessment of the Gold Road Mine, Arizona, USA, dated May 2, 2018, (the "Technical Report").

2. I graduated from Colorado School of Mines in 1970 with a Bachelor's degree in Geological Engineering studies. I have practiced my profession continuously since 1970.

3. I am designated by the American Institute of Professional Geologists as a Certified Professional Geologist

4. I have worked as a Geologist for a total of 48 years since my graduation from college and have been involved in the evaluation and/or operation of mineral properties for copper, gold, iron, lead, pyrite, silver, tin, tungsten, uranium, zinc, fluorite, perlite, and zircon in Argentina, Bolivia, Brazil, Canada, Chile, Costa Rica, Ecuador, Greenland, Guyana, Mexico, Nicaragua, Peru, the United States, and Venezuela.

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 responsible for the preparation of Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 15, 19, 20, 23, 24, 25, 26, and 27.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the Issuer in accordance with Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the "Technical Report" has been prepared in compliance with that instrument and form.

11. I consent to the filing of the "Technical Report" with any securities regulatory authority, stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Greenwood Village, CO, this 3rd day of May, 2018.

Richard J. Kehmeier, Certified Professional Geologist (#10879)

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I, Esteban Acuña, do hereby certify that:

1. I am a Principal Geologist of RPMGlobal USA, Inc. of 6251 Greenwood Plaza Blvd., Suite 275, Greenwood Village, Colorado 80111 United States. This certificate applies to the NI 43-101 Technical Report, Preliminary Economic Assessment of the Gold Road Mine, Arizona, USA, dated May 2, 2018, (the "Technical Report").

2. I graduated from Universidad de Concepción, Chile in 1995 with a bachelor degree in Geology studies. I have practiced my profession continuously since 1996.

3. I am designated by Chilean Review Committee of Resource and Reserve Competency as a Competent Person.

4. I have worked as a geologist for a total of 21 years since my graduation from college and have been involved in the evaluation and/or operation of mineral properties for copper, gold, molybdenum, iron, lead, silver, zinc, cobalt, niobium, platinum, and rare earth in Argentina, Brazil, Peru, Panama, Canada, Chile, Mexico, DRC, Botswana, Australia, Pakistan, South Africa and the United States.

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 responsible for the preparation of Sections 14.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the Issuer in accordance with Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the "Technical Report" has been prepared in compliance with that instrument and form.

11. I consent to the filing of the "Technical Report" with any securities regulatory authority, stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Denver, CO, this 3rd day of May, 2018.

____

Esteban Acuña

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I, David K. Young, do hereby certify that:

1. I am a Principal Mining Engineer of RPMGlobal USA, Inc. of 6251 Greenwood Plaza Blvd, Suite 275, Greenwood Village, Colorado, 80111, United States. This certificate applies to the NI 43-101 Technical Report, Preliminary Economic Assessment of the Gold Road Mine, Arizona, USA, dated May 2, 2018, (the “Technical Report”).

2. I graduated from the Colorado School of Mines in December of 1983, with a Bachelor of Science degree in Mining Engineering. I have practiced my profession continuously for 34 years since 1983.

3. I am designated by the State of Idaho as a Professional Engineer.

4. I have worked as a mining engineer for a total of 34 years since my graduation from college and have been involved in the evaluation and/or operation of mineral properties for copper, gold, iron, lead, lithium, potash, silver, zinc, in Argentina, Bolivia, Brazil, Bulgaria, Canada, Ecuador, Finland, French Guiana, Mexico, Peru, Russia, Tanzania, and the United States,

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 responsible for the preparation of Sections 16, 21, and 22.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the Issuer in accordance with Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the “Technical Report” has been prepared in compliance with that instrument and form.

11. I consent to the filing of the “Technical Report” with any securities regulatory authority, stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Greenwood Village, CO, this 3rd day of May, 2018.

David K. Young

David K. Young, Idaho Professional Engineer (#6517)

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I, Richard Addison, do hereby certify that:

1. I, am a Principal Process Engineer with RPMGlobal USA, Inc. of 6251 Greenwood Plaza Blvd., Suite 275, Greenwood Village, Colorado 80111 United States. This certificate applies to the NI 43-101 Technical Report, Preliminary Economic Assessment of the Gold Road Mine, Arizona, USA, dated May 2, 2018, (the "Technical Report").

2. I graduated from the Colorado School of Mines in 1968 with a M.Sc. degree in metallurgical engineering. I have practiced my profession continuously since 1964.

3. I am designated by the state of Nevada as a Professional Engineer.

4. I have worked as a metallurgical engineer for a total of 53 years since my graduation from college and have been involved in the evaluation and/or operation of mineral properties for copper, molybdenum, gold, silver, lead, zinc, iron, manganese, pyrite, tin, tungsten, uranium, niobium, bauxite, potash, phosphate, fluorite, perlite, gypsum, barite, and kaolin in Argentina, Australia, Bolivia, Botswana, Brazil, Canada, Chile, Colombia, Costa Rica, Dominican Republic, DRC, Ecuador, Ethiopia, Guinea, Ghana, Guyana, Haiti, Honduras, Japan, Kazakhstan, Mexico, Nicaragua, Peru, the Philippines, Portugal, Serbia, Sierra Leone, South Africa, Spain, Turkey, the United States, Venezuela, Zambia, and Zimbabwe.

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 responsible for the preparation of Sections 13, 17, 18, and 19, and parts of Sections 1 and 21.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the Issuer in accordance with Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the "Technical Report" has been prepared in compliance with that instrument and form.

11. I consent to the filing of the "Technical Report" with any securities regulatory authority, stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Greenwood Village, Colorado, this 3rd day of May, 2018.

R. Addison

Richard Addison, Nevada Professional Engineer (# 003198)

– END OF REPORT –