National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska, USA

Report Date: June 30, 2016
Effective Date: June 24, 2016

PREPARED FOR:

Gold Torrent Inc.
960 Broadway Avenue Suite 530
Boise, ID 83706

TO BE FILED BY:

Miranda Gold Corp.
15381 – 36th Avenue, South Surrey
British Columbia, Canada V3Z 0J5

PREPARED BY:

HARD ROCK CONSULTING, LLC

Hard Rock Consulting, LLC
7114 W. Jefferson Avenue Suite 308
Lakewood, CO 80235
HRC Project Number: 16-CSM-1001

Endorsed by QP(s):
Zachary J. Black, (HRC), SME-RM (No. 4156858RM)
J. J. Brown, P.G. (HRC), SME-RM (No. 4168244RM)
Erik Spiller, (HRC), SME-RM (No. 3051820RM)

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Gold Torrent Inc. ("GTOR") and Miranda Gold Corp. ("Miranda") by Hard Rock Consulting, LLC ("HRC"). The quality of information, conclusions, and estimates contained herein is consistent with the scope of HRC's services based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by GTOR and Miranda subject to the terms and conditions of their contract with HRC, which permits GTOR and Miranda to file this report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

CERTIFICATES OF QUALIFIED PERSONS

I, Zachary J. Black, SME-RM, do hereby certify that:

1. I am currently employed as Principal Resource Geologist by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 308
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of the University of Nevada, Reno with a Bachelor of Science in Geological Engineering, and have practiced my profession continuously since 2005.

3. I am a registered member of the Society of Mining and Metallurgy and Exploration (No. 4156858RM)

4. I have worked as a Geological Engineer/Resource Geologist for a total of ten years since my graduation from university; as an employee of a major mining company, a major engineering company, and as a consulting engineer with extensive experience in structurally controlled precious and base metal deposits.

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 personally inspected the Project November 12th through November 15th, 2014.

7. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report, Preliminary Feasibility Study for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska, USA," dated June 30, 2016, with an effective date of June 24, 2016, with specific responsibility for Sections 1.4 through 1.5, 9 through 12, and 14 of this report.

8. I have had no prior involvement with the property that is the subject of this Technical Report.

9. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 5th day of July, 2016.

"Signed" Zachary J. Black

/s/ Zachary J. Black
Signature of Qualified Person

Zachary J. Black, SME-RM
Printed name of Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

I, Jennifer J. Brown, P.G., do hereby certify that:

1. I am currently employed as Principal Geologist by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 308
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of the University of Montana and received a Bachelor of Arts degree in Geology in 1996.

3. I am a:
   - Licensed Professional Geologist in the State of Wyoming (PG-3719)
   - Registered Professional Geologist in the State of Idaho (PGL-1414)
   - Registered Member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (4168244RM)

4. I have worked as a geologist for a total of 19 years since graduation from the University of Montana, as an employee of various engineering and consulting firms and the U.S.D.A. Forest Service. I have more than 10 collective years of experience directly related to mining and or economic and saleable minerals exploration and resource development, including geotechnical exploration, geologic analysis and interpretation, resource evaluation, and technical reporting.

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 the report titled "National Instrument 43-101 Technical Report, Preliminary Feasibility Study for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska, USA," dated June 30, 2016, with an effective date of June 24, 2016, with specific responsibility for Sections 1.1 through 1.3, 2 through 8 and 20 of this report.

7. I have had no prior involvement with the property that is the subject of this Technical Report.

8. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

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

Dated this 5th day of July, 2016.

"Signed" Jennifer J. (J.J.) Brown

/s/ Jennifer J. Brown
Signature of Qualified Person

Jennifer J. (J.J.) Brown, SME-RM
Printed name of Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

I, Jeffery W. Choquette, P.E., do hereby certify that:

1. I am currently employed as Principal Engineer by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 308
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of Montana College of Mineral Science and Technology and received a Bachelor of Science degree in Mining Engineering in 1995

3. I am a:

   - Registered Professional Engineer in the State of Montana (No. 12265)
   - QP Member in Mining and Ore Reserves in good standing of the Mining and Metallurgical Society of America (No. 01425QP)

4. I have nineteen years of domestic and international experience in project development, resource and reserve modeling, mine operations, mine engineering, project evaluation, and financial analysis. I have worked for mining and exploration companies for fifteen years and as a consulting engineer for three and a half years. I have been involved in industrial minerals, base metals and precious metal mining projects in the United States, Canada, Mexico and South America.

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 the report titled “National Instrument 43-101 Technical Report, Preliminary Feasibility Study for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska, USA,” dated June 30, 2016, with an effective date of June 24, 2016, with specific responsibility for Sections 1.6 through 1.8, 15, 16, 18, 19 and 21 through 26 of this report.

7. I have had no prior involvement with the property that is the subject of this Technical Report.

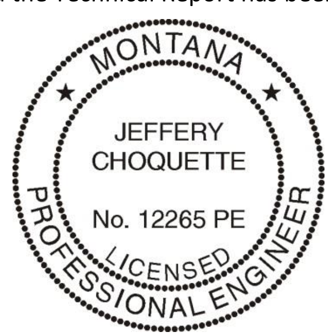
8. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

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

Dated this 5th day of July 2016.

“Signed” Jeffery W. Choquette



/s/ Jeffery W. Choquette
Signature of Qualified Person

Jeffery W. Choquette, P.E.
Printed name of Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

I, D. Erik Spiller, P.E., do hereby certify that:

1. I am a Principal of:

   Spiller Consultants, LLC
   1930 Denver West Drive, Ste. 713
   Golden, Colorado 80401 U.S.A.

   I am a graduate of Colorado School of Mines, and received a Bachelor of Science degree in Metallurgical Engineering in 1970.

2. I am a:

   - Qualified Professional Member of the Mining and Metallurgical Society of America (No. 01021QP)
   - Registered Member of the Society of Mining, Metallurgy and Exploration, Inc. (No. 3051820RM)

3. I have worked as a metallurgical engineer in the mineral resource industry for more than 40 years. During this career I held responsible positions in process research, process development, engineering, and senior management. In addition, I have served as an Adjunct instructor (22 years) and as an appointed Research Professor (9 years) in the Metallurgical and Materials Engineering Department at the Colorado School of Mines, where I lecture in mineral beneficiation and direct graduate students conducting metallurgical research in my area of expertise.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report, Preliminary Feasibility Study for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska, USA," dated June 30, 2016, with an effective date of June 24, 2016, with specific responsibility for Sections 13 and 17 of this report.

6. I have had no prior involvement with the property that is the subject of this Technical Report.

7. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 5th day of July, 2016.

"Signed" D. Erik Spiller, QP

/s/ D. Erik Spiller

D. Erik Spiller, Principal
Printed name of Qualified Person

D. Erik Spiller
01021QP
Mining & Metallurgical
Society of America

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LIST OF ACRONYMS

AA	Atomic Absorption
AES	Atomic Emission Spectrometry
Ag	Silver
Au	Gold
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CL	Control Limit
CV	Coefficient Variation
FSE	Frankfurt Stock Exchange
ft	Foot
g/t	Grammes per tonne
GTOR	Gold Torrent Inc..
HDPE	High Density Polyethylene
HP	Horsepower
HRC	Hard Rock Consulting
ID	Inverse Distance
in	Inches
IRR	Internal Rate of Return
LL	Lower Control Limit
MSO	Mineable Shape Optimizer
NN	Nearest Neighbor
NPV	Net Present Value
NYSE	New York Stock Exchange
OK	Ordinary Kriging
opt	Ounces per tonne
oz	ounce
QA/QC	Quality Assurance/Quality Control
SRM	Standard Reference Material
tpd	Tonnes per day
TSX	Toronto Stock Exchange

1. EXECUTIVE SUMMARY

1.1 Introduction

Gold Torrent Inc. ("GTOR") retained Hard Rock Consulting, LLC ("HRC") to prepare a Preliminary Feasibility Study ("PFS") for the Lucky Shot Project (the "Project") located in the Matanuska-Susitna Borough, Alaska, USA. This report, to be filed by Miranda Gold Corp. ("Miranda"), presents the results of the PFS and associated work completed by HRC, and is intended to fulfill the Standards of Disclosure for Mineral Projects according to Canadian National Instrument 43-101 ("NI 43-101"). This report is specific to the Lucky Shot Project, which includes mineral resources and mineral reserves from the Coleman and Lucky Shot areas, and was prepared in accordance with the requirements and guidelines set forth in Companion Policy 43-101CP and Form 43-101F1 (June 2011). The mineral resource and mineral reserve estimates presented herein are classified according to Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards - For Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. The results of the PFS are based in part on the mineral resource estimate reported in the previously filed NI 43-101 Technical Report, Mineral Resource Estimate for the Willow Creek Project, Matinuska-Susitna Borough, Alaska, dated March 24, 2016, with an effective date of February 1, 2016.

1.2 Property Description and Ownership

The Lucky Shot Project is located within the historic Willow Creek mining district of south central Alaska, roughly 40 km northeast of the town of Willow in the Matanuska-Susitna Borough, and about 80 km (50 miles) northeast of Anchorage. The district covers an area of around 82 km2 along the southwestern edge of the Talkeetna Mountains, centered at about N61°46'30" latitude and W149°20'30" longitude. The Project area is situated in the western portion of the district, and includes the historic Lucky Shot, War Baby, Coleman, and Murphy mines and prospects.

On November 5, 2014, GTOR entered an Exploration and Option to Enter Joint Venture Agreement between Gold Torrent and Miranda, which grants Gold Torrent the right to earn-in up to 70% ownership in the proposed joint venture. Miranda would hold the remaining 30%. The initial work commitment requires GTOR to expend $1,070,000 including to pay and perform all obligations of the lessee under the underlying mineral lease and pay all costs incurred on or for the benefit of the property for exploration and development work. Upon completion of the initial earn-in expenditure of $1,070,000 GTOR shall earn a 20% interest and the joint venture shall be formed as a limited liability company between Miranda and Gold Torrent. Miranda shall own 80%. The joint venture company shall be managed by GTOR and governed by a two-person management committee composed of one member from each party. Miranda shall contribute the AHI mineral lease to the joint venture and GTOR shall contribute the $1,070,000 in expenditures including the work product, reports, engineering and permitting results achieved.

Miranda and AHI, the property owner, entered into a mineral lease for the patented and unpatented mining claims and rights. Miranda paid an initial payment of $50,000 to AHI on November 15, 2013 followed by a subsequent payment of $100,000 to secure the first year lease. As part of its earn-in obligations under the November 5, 2014 Exploration and Option to Enter Joint Venture Agreement between Gold Torrent and

Miranda, Gold Torrent paid AHI $150,000 by the due date of January 15, 2015, as required under the lease document to secure a second year of the lease. The term of the lease is eighty years.

1.3 Geology and Mineralization

Rock types in the Project area are comprised of faulted Upper Cretaceous granite and the Willow Creek quartz diorite-tonalite (a phase of the Talkeetna Batholith), bordered to the south by the Jurassic Hatcher Pass Schist of the Peninsular terrane. The Willow Creek quartz diorite-tonalite is intruded by a variety of dikes with chilled margins ranging in composition from aplite to lamprophyre (Cooley, 2009).

Northwest striking and steeply (60-80°) dipping faults crosscut the entire Willow Creek intrusive. Less faulting occurs in the Hatcher Pass Schist. The faults have dextral and normal movement with displacement to the east creating an en echelon pattern of vein segments, dikes, and fault blocks. Gold mineralization at the Project is hosted in shallow north-dipping mesothermal veins within shears or reverse faults in the intrusive (Cooley, M., 2006). Quaternary cover is predominantly a product of glaciation with only minor re-working.

The Willow Creek mining district's only known economic mineral is gold contained in mesothermal veins within low angle shears in the Willow Creek quartz diorite-tonalite intrusive. The important lode gold-bearing shears strike 60-80° and dip 30-60° northerly. The argon isotope method dates sericite in vein associated alteration to approximately 66.9 to 65.6Ma.

Another group of structures trending nearly due north with dips from near horizontal up to 45° to the west are also significant, but are only near to the intersections with the more important east-northeast trending veins. The productive veins have coarsely crystalline quartz with minor pyrite, sphalerite and other sulfides, telluride and visible gold. Gold deposition is a late event and only very minor amounts of gold are occluded within sulfides.

1.4 Status of Exploration

Enserch actively explored the Lucky Shot mine and greater Willow Creek district from 1978 through 1985. Enserch drove a 1,500 ft. (457 m) exploration drift below the Lucky Shot mine and drilled 11 underground core holes totaling 10,364 ft. (3,159 m) during the 1984 field season. Additionally, Ensearch drilled 7 surface core holes in the Coleman zone totaling 4,881 ft. (1,488m).

FMM acquired the Project in 2004, and subsequently moved forward with well-planned helicopter supported drilling exploration from 2005 to 2010. FMM completed seven (7) NQ2 size core drill holes in 2005 and intercepted significant gold mineralization. The drilling tested and confirmed earlier Enserch drill intercepts in the Coleman area. The majority of seventy-three (73) core drill holes drilled in 2006 tested the central Coleman area with step-out fan drilling. Encouraging results from drilling in the Coleman and Murphy areas prompted additional drilling in 2007 in these areas. Drilling during 2007 was concentrated northeast and southwest up and down dip from the historic Coleman workings and extended mineralization to the west both above and below the historical Coleman workings. Other drilling verified Enserch intercepts down dip from the Lucky Shot workings and expanded the mineralization in the

Murphy area. In 2008 ten core drill holes were completed in the War Baby mine area. Two of ten drill holes intersected gold mineralization with the best intercept of 0.4 m @ 33.75 g/t gold.

In 2014, Miranda and GTOR geologists completed a 234-sample soil grid and collected rock samples of quartz vein rubble. Results strongly suggest that the vein system mined on Bullion Mountain within the Willow Creek mining district extends beyond the fault that bounded historic production. The highlight of the sample program is the discovery of three quartz vein sub-crops that assayed 1.48 oz, 0.50 oz and 0.53 oz Au/t (50.74 g, 17.05 g and 18.15 g/t Au).

In 2015, GTOR shipped 2,200 lbs. of ore material from the Lucky Shot area to Hazen Laboratories in Golden Colorado for metallurgical and geochemical analysis. Three sample splits of quartz + sulfide + carbonate from the Lucky Shot shear zone ranged from 25 to 26.2 g/t Au. Results from this testwork are included in Section 13 and in the appended Hazen report of 2016.

1.5 Mineral Resource Estimate

Resource geologist Zachary J. Black, SME-RM, of HRC is responsible for the mineral resource estimate presented here. Mr. Black is a Qualified Person as defined by NI 43-101, and is independent of GTOR. HRC estimated the mineral resource for the Project based on drillhole data constrained by geologic vein boundaries with an Inverse Distance Weighted ("ID") algorithm. Datamine Studio RM® V1.0.73.0 ("Datamine") software was used to complete the resource estimate in conjunction with Leapfrog Geo® V.3.0.0 ("Leapfrog"), which was used to produce the geologic model. The metals of interest at Willow Creek are gold and silver.

The Project is defined by veins within the Coleman, Lucky Shot, War Baby, and Murphy fault blocks. The mineral resource estimate is comprised of 10 veins. Eight veins from the Coleman area and 2 veins from the Lucky Shot area. The Murphy and War Baby areas are excluded from the mineral resource estimate due to insufficient data. The mineral resources have been estimated using 3-dimensional ("3D") block model.

HRC first constructed the geologic vein model using Leapfrog Geo Version 3.3.0 using a linear interpolation methodology and sample intervals. East/West oriented cross-sections were used to select intervals from each drillhole representing the vein material. Points representing the hanging wall and footwall contacts were extracted to interpolate hanging wall and footwall surfaces. These surfaces were used to delineate each vein solid (Figure 14-2). The surfaces were evaluated in 3-dimensions to ensure that both the down dip and along strike continuity was maintained throughout the model. Vein volumes in the area of historical production were removed by clipping the solid against an area mapped as having been exploited in the historical mine workings.

The 3D geologic solids were converted to block models using Datamine. Block model prototypes were created for each of the fault blocks. The model prototypes are rotated along strike and down dip and encompass the entire vein. A block size of 1.5m x 1.5m in the strike and dip directions was established to match the mining method being planned by GTOR. The blocks in the z direction were sub-blocked to the vein thickness.

Comparisons were made with ordinary kriging ("OK") and inverse distance-squared (ID2.5) methods. The ID2.5 method was selected for reporting due to better fit with drillhole data throughout the model. Gold grades were estimated in each vein by using incremental search ellipses to provide an estimation of the gold and silver grade within every block inside the vein solids. Grades outside of the defined veins were not estimated. A true thickness composite length weighted ID2.5 was used to estimate grade for all domains. Ordinary Krige ("OK") and Nearest Neighbor ("NN") models were run to serve as comparison with the estimated results from the ID2.5 method.

HRC used two methods to classify the mineral resources into measured, indicated, and inferred. For the Coleman fault block veins, measured resource are those blocks with at least one composite within an anisotropic distance of 15 x 6 meters of the block centroid. Indicated resources are those blocks with at least 2 composites within an anisotropic distance of 75 x 30 meters. Inferred resources are those blocks with at least 2 composites within an anisotropic distance of 150 x 60 meters.

For the Lucky Shot area veins, a polygonal method was used to classify resources into indicated and inferred. An exterior drillhole fence, was constructed using drillhole intercepts. An indicated boundary was generated by expanding 25 meters from the drillhole fence polyline, and 25m from drillhole intercepts. Indicated resources are those blocks inside the indicated boundary. Inferred resources are those blocks outside the indicated boundary. In cases where a single drillhole was used to estimate mineral resources, those blocks within a 15 x 6-meter search volume are classified as indicated.

The mineral resources for the Project as of February 1, 2016, are summarized in Table 1-1.

Table 1-1 Mineral Resource Statement for the Lucky Shot Project, Matanuska-Susitna Borough, Alaska Hard Rock Consulting, LLC, Effective Date of February 1, 2016

Classification	Tonnes (x1000)	Gold		Silver	
		g/t	oz	g/t	oz
Measured	57.9	26.8	49,900	2.5	4,700
Indicated	148.6	15.0	71,600	1.6	7,400
Measured + Indicated	**206.6**	**18.3**	**121,500**	**1.8**	**12,100**
Inferred	59.0	18.5	35,100	1.5	2,900

Note: Measured, Indicated and Inferred mineral classifications are assigned according to CIM Definition Standards. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability and there is no guarantee that mineral resources will be converted to mineral reserves. (1) The mineral resource estimate was prepared by HRC based on data and information available as of February 1, 2016. The 2016 Measured, Indicated and Inferred mineral resources are reported considering a base case estimate that applies a cutoff grade of 5 g/t Au based on the estimated operating costs, 80% gold and silver recoveries, and a $1,265/oz gold price.

1.6 Mineral Reserve Statement

Mr. Jeff Choquette, P.E., MMSA QP, of HRC is responsible for the mineral reserve estimate presented herein. Mr. Choquette is Qualified Person as defined by NI 43-101 and is independent of GTOR. The mineral reserve calculation for the Project was completed in accordance with NI 43-101 and is based on all data and information available as of May 31st, 2016. Stope designs for reporting the reserves were created utilizing the mineral resources presented in Section 14 of this report. An open stull stope mining method is planned to extract the Coleman and Lucky Shot deposits with random waste pillars. Ore is planned to be processed in a gravity concentration process plant capable of processing 200 tpd.

HRC utilized Datamine's Mineable Shape Optimizer ("MSO") program to generate the stopes for the reserve mine plan. The stopes were created based solely on Measured and Indicated mineral resources, which have demonstrated to be economically viable, including internal stope dilution above the calculated cutoff; therefore, Measured and Indicated mineral resources within the stopes have been converted to Proven and Probable mineral reserves as defined by NI 43-101. Inferred mineral resources are not considered as part of the reserve statement.

Dilution is applied to Measured and Indicated resource blocks depending on the mining method chosen. For blocks to be exploited using open stull stope mining methods, external dilution was applied in the amount of 20% at a grade of zero. Internal dilution is also applied based on any blocks that fall inside the stope shape but are below cutoff. A mining recovery is also applied to convert resources to reserves and is estimated at 90%. These factors resulted in an overall dilution factor of 31% for the Coleman vein and 56% for the Lucky Shot vein, with an overall average dilution of 34% for the reserves.

The mining breakeven cut-off grade was used to generate the stope designs in DataMine's MSO for defining the reserves. The estimated operating costs and mill recoveries developed for the PFS are used to calculate the reserve breakeven cut-off grade. A gold price of $1175/oz was chosen, which is close to the 200 day moving average of $1162/oz as of May 31st, 2016. The mine operating costs used in the cutoff calculation exclude the costs for development. The cut-off is stated as gold equivalent since the ratio between gold and silver is variable and both commodities are sold, although the silver is estimated to be fairly insignificant with only 0.12% of the revenue coming from the silver produced. The average cut-off grade used for the Project is 7.0 g/t AuEg.

The Proven and Probable mineral reserves for the Project as of May 31st, 2016 are summarized in Table 1-2. The reserves are exclusive of the mineral resources reported in Section 14 of this report.

Table 1-2 Proven and Probable Mineral Reserves, Effective Date May 31st, 2016

Classification	Area	Tonnes (x1000)	Gold		Silver		Dilution
			g/t	oz	g/t	oz	
Proven	Coleman	68.7	18.9	41,672	2.02	4,465	37%
	Lucky shot	0	0	0	0	0	0%
Total Proven		**68.7**	**18.9**	**41,672**	**2.02**	**4,465**	**37%**
Probable	Coleman	87.9	13.4	37,936	1.67	4,728	26%
	Lucky shot	17.8	13.8	7,897	0.09	51	56%
Total Probable		**105.7**	**13.5**	**45,833**	**1.41**	**4,779**	**31%**
Total Proven + Probable		**174.4**	**15.60**	**87,504**	**1.65**	**9,244**	**34%**

1. Reserve cut-off grades are based on a 7.0 g/t gold equivalent with 78.3:1 silver to gold ratio.
2. Metallurgical recoveries were estimated at 90.0% silver and 91.8% for gold.

3. Mining recoveries of 90% were applied.
4. Minimum mining widths were 0.8 meters.
5. Dilution factors averaged 34%. Dilution factors are calculated based on internal stope dilution calculations and external dilution factors of 20%.
6. Price assumptions are $15 per ounce for silver and $1,175 per ounce for gold.
7. Figures in table are rounded to reflect estimate precision; small differences generated by rounding are not material to estimates.

1.7 Conclusions

HRC is of the opinion that GTOR has thorough understanding of the geology of the Project, that GTOR is applying the appropriate deposit model for exploration, and that the potential for continued discovery of additional mineral resources at the Project is high.

HRC concludes that the sample preparation, security and analytical procedures are correct and adequate for the purpose of this technical report. The sample methods and density are appropriate and the samples are of sufficient quality to comprise a representative, unbiased database.

HRC has reviewed the check assay programs and believes the programs provide adequate confidence in the data. Samples that are associated with failures and the samples associated with erroneous blank samples have been reviewed. Errors have been justified as labeling errors or are infrequent. All of the samples associated with erroneous QA/QC results are reviewed prior to inclusion in the database.

HRC received original assay certificates in csv format for the drilling conducted from 2005 to 2009 in the current database. A random manual check of 10% of the database against the original certificates was conducted. The error rate within the database is considered to be less than 1% based on the number of samples spot checked. HRC is of the opinion that the data maintained within the database is acceptable for mineral resource estimation.

Significant historical reports and more recent testwork strongly support the conclusion that the mineralized material with contained gold at the Willow Creek Project will respond favorably to beneficiation by careful liberation and gravity concentration. Such a process minimizes the generation of slimes and presents size-classified material to the gravity concentration unit operations, commercially expected to be spiral concentrators and table concentrators. Gold recovery in excess of 90% while producing non-acid tailings has been demonstrated by test work and calculations. Further testing in support of a feasibility study and final engineering is recommended as follows:

- Confirm comminution energy requirements via conventional Bond testing
- Determine Bond abrasion index for engineering cost estimation
- Quantify liquid/solid separation, i.e., settling tests on various process streams
- Establish performance of commercial spirals
- Establish additional liberation information from gravity size/recovery testing
- Establish gravity recovery as function of feed grade (both upwards and downwards)
- Characterize gravity tailings

- Determine if further classification of minus 325 mesh ore will increase gold recovery
- Evaluate enhanced centrifugal gravity concentration of ore fines and tailings

1.8 Recommendations

Exploration of the Project has advanced to the consideration of a year round underground mining operation. GTOR has made the decision to advance the project into devleopment and production. Determining the metallurgical and mining parameters for the Project will be the key focus of continued development.

HRC's recommendations are intended to provide GTOR with a path toward development of the Project. Advancing the Project is not contingent upon positive results of the work program outlined in Table 1-3, though the engineering, permitting and environmental requirements necessary to bring the Project into development need to be assessed in order to understand any difficulties or costs that might impact the overall Project economics. The anticipated costs for the recommended scope of work are presented in Table 1-3.

Table 1-3 2016 Recommended Scope of Work for the Lucky Shot Project

Recommended Scope of Work	Expected Cost (US$)
Environmental Permitting Work	$150,000
Geotechnical Analysis	$20,000
Metallurgical Testwork	$75,000
Detailed Mine Design and Final Engineering	$250,000
Exploration Drilling	$100,000
Subtotal	$595,000
15% Contingency	$89,250
Total Budget	**$684,250**

2. INTRODUCTION

2.1 Issuer and Terms of Reference

Gold Torrent, Inc. ("GTOR") is an American based junior mining and exploration company focused on the development of high grade underground mining properties in North America. GTOR is an OTC Venture Marketplace listed company (OTCQB:GTOR), and is headquartered in Boise, Idaho. Miranda Gold Corp. ("Miranda") (TSX-V: MAD) is a gold exploration company active in Alaska and Colombia, whose emphasis is on generating gold exploration projects with world-class discovery potential. In 2014, GTOR completed an Exploration and Option to Enter Joint Venture Agreement (Agreement) with Miranda on Miranda's Willow Creek (herein referred to as "Lucky Shot") gold project in the Matanuska-Susitna Borough, Alaska.

GTOR has retained Hard Rock Consulting ("HRC") to complete a Preliminary Feasibility Study ("PFS") for the Lucky Shot Project (the "Project"), with specific regard to the Coleman and Lucky Shot portions of the Project area. The PFS is a comprehensive study of a range of options for the technical and economic viability of the Project based on the proposed mining and processing methods. This study includes a financial analysis based on reasonable assumptions of modifying factors and evaluation of other factors sufficient to determine if all or part of the mineral resource may be converted to mineral reserves. This report presents the results of HRC's efforts, and is intended to fulfill the Standards of Disclosure for Mineral Projects according to Canadian National Instrument 43-101 ("NI 43-101"), and is to be filed by GOTR's joint venture partner Miranda.

This report was prepared in accordance with the requirements and guidelines set forth in NI 43-101 Companion Policy 43-101CP and Form 43-101F1 (June 2011), and the mineral resources and reserves presented herein are classified according to Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards - For Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. The mineral resource and mineral reserve estimates reported here are based on all available technical data and information as of June 1, 2016.

2.2 Sources of Information

A portion of the information and technical data for this study was obtained from the following previously filed NI43-101 Technical Reports:

HRC (2016). NI43-101 Technical Report, Mineral Resource Estimate for the Willow Creek Project, Matanuska-Susitna Borough, Alaska: prepared for GTOR and Miranda Gold Corp., effective date February 1, 2016.

Linebarger, D., (2014). NI43-101 Technical Report on the Willow Creek Project, Matanuska-Susitna Borough, Alaska: prepared for Miranda Gold Corp., effective date July 1, 2014.

Stevens, D.L., (2010). Lucky Shot Project, Willow Creek mining district, South-Central, Alaska: NI43-101 technical report prepared for Harmony Gold Corp., report date January 15, 2010.

Stevens, D.L., (2010). Lucky Shot Project, Willow Creek mining district, South-Central, Alaska: NI43-101 technical report prepared for Full Metal Minerals Ltd., report date August 27, 2009.

The information contained in current report Sections 4 through 8 was largely presented in, and in some cases is excerpted directly from, the technical reports listed above. HRC has reviewed this material in detail, and finds the information contained herein to be factual and appropriate with regard to guidance provided by NI 43-101 and associated Form NI 43-101F1.

Additional information was requested from and provided by GTOR. With respect to Sections 9 through 13 of this report, the authors have relied in part on historical information including exploration reports, technical papers, sample descriptions, assay results, computer data, maps and drill logs generated by previous operators and associated third party consultants. The authors cannot guarantee the quality, completeness, or accuracy of historical information, nor its preparation in accordance with NI 43-101 standards. Historical documents and data sources used during the preparation of this report are cited in the text, as appropriate, and are summarized in report Section 27.

2.3 Qualified Persons and Personal Inspection

This report is endorsed by the following Qualified Persons, as defined by NI 43-101: Mr. Zachary Black, Ms. J.J. Brown, P.G., and Mr. Jeff Choquette, P.E., all of HRC, and Mr. Erik Spiller, of Spiller Consultants LLC.

Mr. Black, SME-RM, has 10 years of experience working on structurally controlled gold and silver resource and reserve estimate projects. Mr. Black completed the mineral resource estimate for the Project and is specifically responsible for Sections 1.4, 9 through 12, and 14 of this report.

Ms. Brown, P.G., SME-RM, has 19 years of professional experience as a consulting geologist and has contributed to numerous mineral resource projects, including more than twenty gold, silver, and polymetallic resources throughout the southwestern United States and South America over the past five years. Ms. Brown is specifically responsible for report Sections 2 through 8.

Mr. Spiller, QP is a metallurgical engineer with more than 40 years of domestic and international experience in process research, process development, engineering, and senior management of mining and mining service companies. Mr. Spiller has been involved in industrial minerals, base metals and precious metal mining projects around the world, and is responsible for the current report Section 13 and Section 17.

As a Qualified Person and representative of HRC, Mr. Black conducted an on-site inspection of the Willow Creek property from November 12th through November 15th of 2014. While on site, HRC reviewed with GTOR's personnel the available drill core from previous operators, the surface geology, underground vein exposures in the Coleman area, underground drilling stations in the Lucky Shot area, the general site layout, and possible locations for the processing facilities. HRC also reviewed historical paper exploration and operating results from Enserch, and met with GTOR's geologist to review the geologic concept being explored and to review the previous operators sampling methods and types, geologic and resource modeling, prior to inspecting the core storage area.

2.4 Units of Measure

Unless otherwise stated, all measurements reported here are metric, and currencies are expressed in constant 2016 U.S. dollars.

3. RELIANCE ON OTHER EXPERTS

HRC has fully relied upon and disclaims responsibility for information provided by GTOR regarding property ownership and mineral tenure for the Project. GTOR's industry professionals who contributed to this report include: Mr. Pete Parsley, registered Professional Geologist; Mr. Bruce Thorndycraft, M.S. Metallurgical Engineering; and, Mr. Bryan Bishop, B.S. Mining Engineering. Regarding environmental information, HRC has relied upon information provided herein by Mr. Robert Loeffler, Partner, Jade North, LLC., Anchorage, Alaska, and otherwise on narrative information supplied directly by GTOR. HRC has not reviewed the permitting requirements nor independently verified the permitting status or environmental liabilities associated with the Project, and also disclaims responsibility for that information, which is presented in current report Sections 4 and 20.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Project is located within the historic Willow Creek mining district of south central Alaska, roughly 40 km northeast of the town of Willow in the Matanuska-Susitna Borough, and about 80 km (50 miles) northeast of Anchorage (Figure 4-1). The district covers an area of around 82 km2 along the southwestern edge of the Talkeetna Mountains, centered at about N61°46'30" latitude and W149°20'30" longitude. The Project area is situated in the western portion of the district, and includes the historic Lucky Shot, War Baby, Coleman, and Murphy mines and prospects. The Project includes both private and state lands, and map coverage is provided by the USGS 1:250,000-scale Quadrangle for Anchorage, Alaska (USGS, 1962) and associated map sheets D-6, D-7, C-6 and C-7.

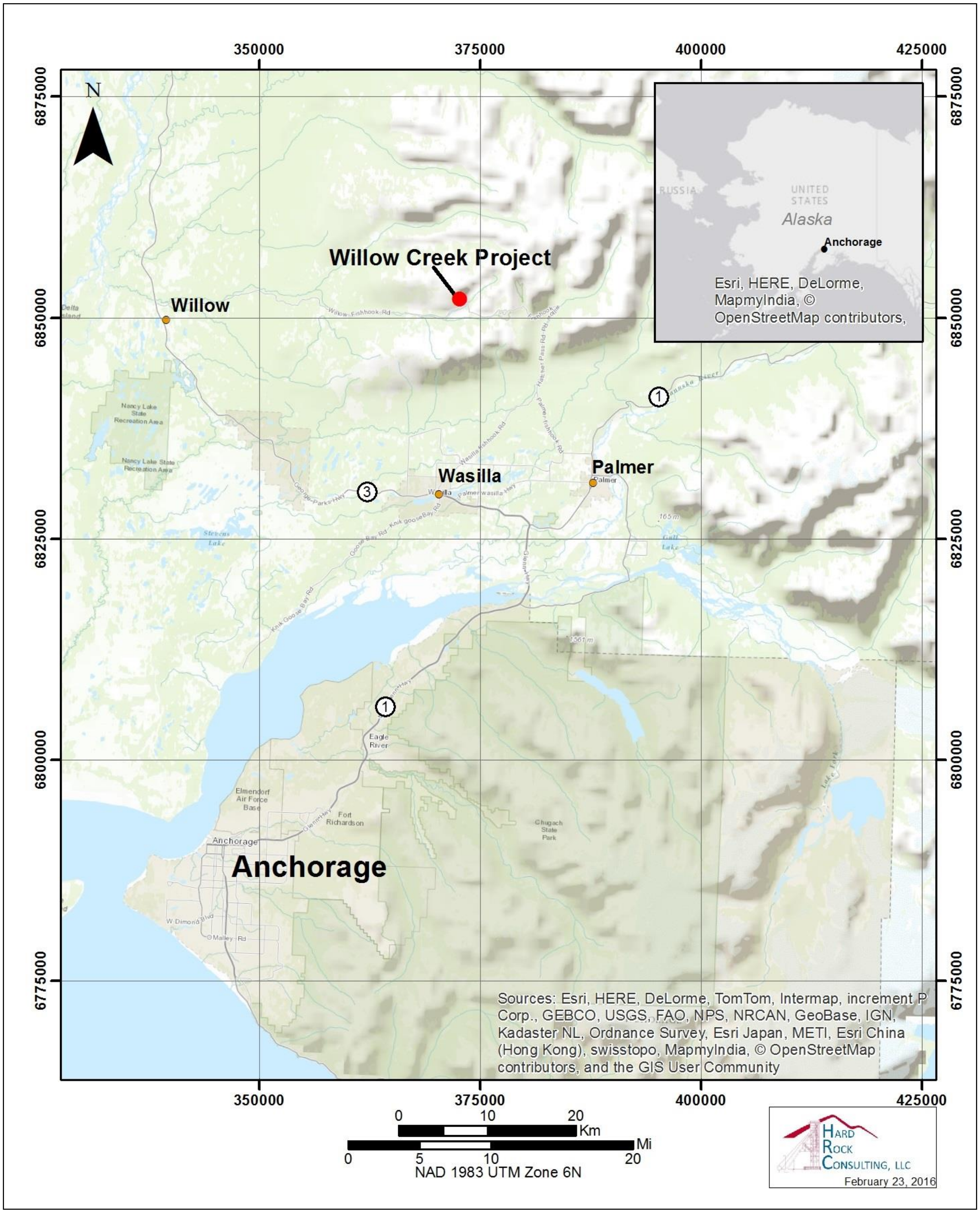


Figure 4-1 Project Location

The Project consists of 43 patented federal claims in two areas held by Alaska Hardrock Incorporated (AHI) and 62 contiguous 160-acre state mining claims also controlled by AHI. The claims cover approximately 4,299 hectares (10,623 acres) within 18 Sections of Townships 19 and 20 North, Ranges 1 East and 1 West (Figure 4-2). A complete list of the claims is provided in Appendix A.

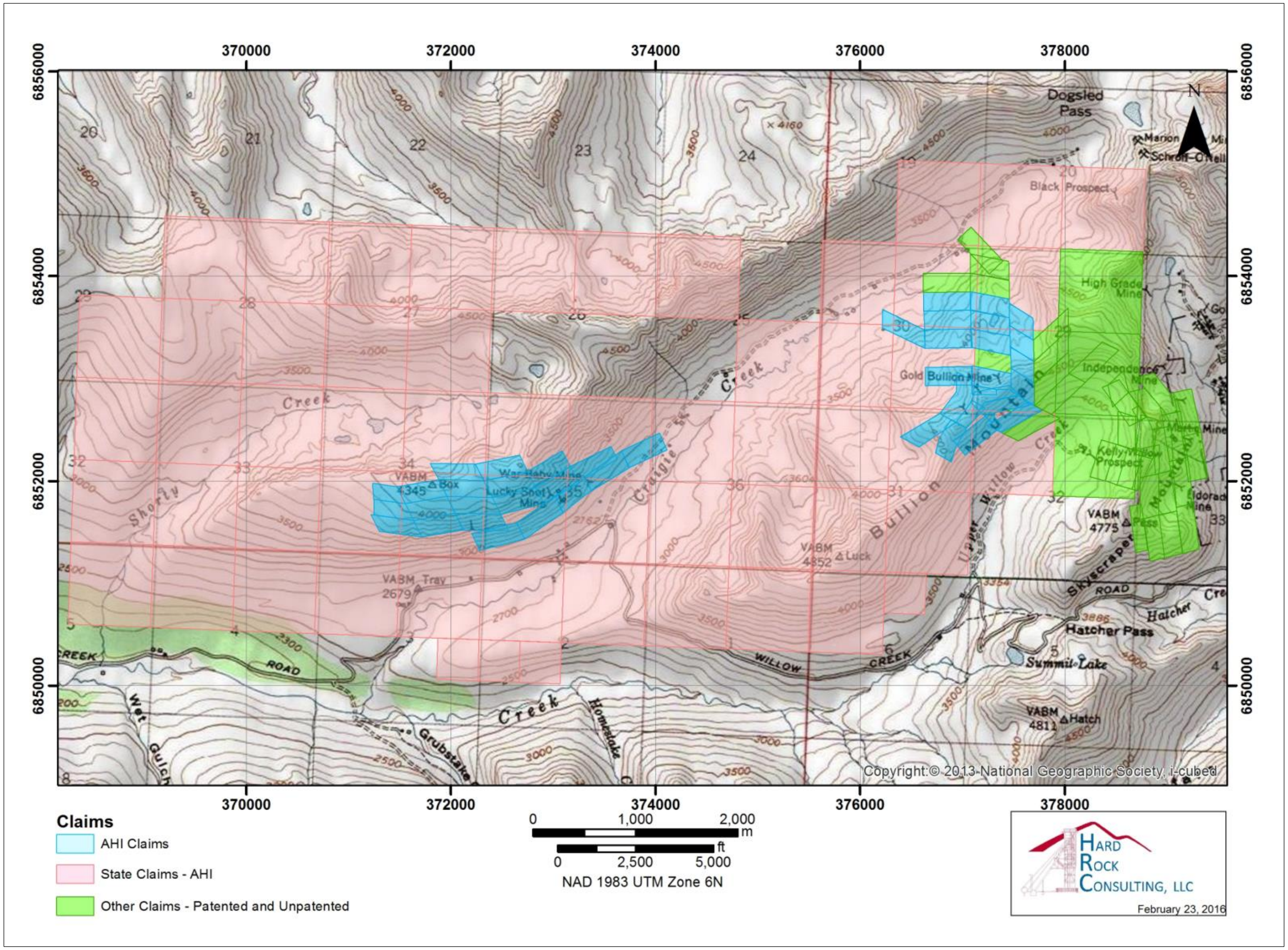


Figure 4-2 Lucky Shot Project Claim Map

4.2 Mineral Tenure, Agreements and Encumbrances

Miranda and AHI, the property owner, entered into a mineral lease for the patented and unpatented mining claims and rights. Miranda paid an initial payment of $50,000 to AHI on November 15, 2013 followed by a subsequent payment of $100,000 to secure the first year lease. As part of its earn-in obligations under the November 5, 2014 Exploration and Option to Enter Joint Venture Agreement between Gold Torrent and Miranda, Gold Torrent paid AHI $150,000 by each of the due dates of January 15, 2015 and January 15, 2016, as required under the lease document. The term of the lease is eighty years.

The lease is subject to annual payments of $150,000 and if production is achieved, a two (2) percent net smelter return royalty from production derived from patented lands and a four (4) percent royalty from state claims. The lease requires that the annual lease payment be paid until such time as a combination of lease payments and royalties exceeds $2.7 million, after which only royalty payments are required.

The November 5, 2014 Exploration and Option to Enter Joint Venture Agreement between GTOR and Miranda grants GTOR the right to earn-in up to 70% ownership in a proposed joint venture. Miranda would hold the remaining 30%. The initial work commitment requires GTOR to expend $1,070,000 including to pay and perform all obligations of the lessee under the underlying mineral lease and pay all costs incurred on or for the benefit of the property for exploration and development work. Upon completion of the initial earn-in expenditure of $1,070,000 GTOR shall earn a 20% interest and the joint venture shall be formed as a limited liability company between Miranda and Gold Torrent. Miranda shall own 80%. The joint venture company shall be managed by GTOR and governed by a two-person management committee composed of one member from each party. Miranda shall contribute the AHI mineral lease to the joint venture and GTOR shall contribute the $1,070,000 in expenditures including the work product, reports, engineering and permitting results achieved.

GTOR shall have the right to earn-in to an additional 25% ownership in the joint venture by expending and additional $2,440,000 within 24 months of the date of the Exploration and Option to Enter Joint Venture Agreement to increase its ownership to 45% and decrease Miranda's to 55%. The final earn-in period requires an additional $6,490,000 be expended on the property within 36 months of the date of the agreement to earn an additional 25% to increase GTOR's ownership in the joint venture to 70% and reduce Miranda's to 30%. Thereafter each party shall participate in any additional capital requirements of the Project on a ratable basis equal to their percentage ownership. The joint venture agreement shall be based on the Rocky Mountain Mineral Law Foundation Form 5 agreement which includes recognized provisions for accounting, taxes, dilution of an individual party's interests, dissolution of the venture and other commercial terms.

Upon earning to the 70% joint venture ownership level, GTOR is entitled to 90% of the Project cash flow until its first $10,000,000 of capital is repaid. If the initial capital required exceeds $10,000,000 to achieve cash flow both parties shall contribute their ratable share and GTOR shall be entitled to 80% of the cash flow thereafter until its capital contribution in excess of $10,000,000 is repaid. Once Gold Torrent's total capital contribution is repaid from 90% and 80% of the cash flow as set forth above the cash flow shall be shared 70% by Gold Torrent and 30% by Miranda.

4.3 Permits and Environmental Liabilities

There are no active permits issued for exploration or mining activities that are currently in place as of the date of this report. GTOR is reviewing past permitting efforts associated with the Project and is in discussion with contractors and advisors who are experienced with past permitting efforts. Previous property owner Full Metal Minerals ("FMM") had permits in place for exploration drilling and test mining and milling of 10,000 tons from the Coleman and Lucky Shot mines between 2005 and 2009. Authorization for simple exploration activities may be acquired without extensive environmental baseline work. All exploration permits for the Project are issued by the State of Alaska, not the federal government, and would not require public notice.

HRC knows of no current or potential future significant factors or risks that might affect access, title, or the right or ability to perform work on the Project.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access and Climate

General access to the Project is provided by State Highway 1 out of Anchorage, and the Parks Highway via the town of Wasilla. Access via Palmer is available only seasonally and requires navigating the steep switchback at Hatcher Pass, which is unsuitable for larger vehicles. Direct access to the Project area is provided by the Willow Creek-Hatcher Pass Road out of the town of Willow. Helicopter access is available from each of the local communities, but may be hampered by weather conditions.

The climate local to the Willow Project area is influenced by its proximity to Knik Arm/Cook Inlet and the Gulf of Alaska. Summer temperatures in Wasilla range from 7 to 20 °C (44 to 68 °F) and 7 to - 15 °C (4 to 43 °F) in winter on average. Precipitation averages 0.48 m (19 in.) annually and includes up to 1.62 m (64 in.) of snow. Heavy snowfalls are known to occur at nearby Hatcher Pass and can approach 6 m (20 ft.) in depth. Winds can approach 161 kph (87 knots) and average 7 kph (3.8 knots). Historic mining operations at the Lucky Shot mine were conducted in all weather conditions, but exploration activities are generally carried out during the summer months when the ground is free of snow.

5.2 Local Resources and Infrastructure

A large shop building, camp quarters, and mill structure are present at the southern extent of the Project area. The buildings date from 1981 and were part of Enserch Exploration's surface infrastructure. While the mill building no longer contains milling equipment, the structure is in good repair and is currently being used for equipment storage. The camp and shop are currently in use. All three facilities are located on private land surface. A historic tailings pond is located on state land surface just north of the shop/mill complex. The existing facilities rely on generated power, and satellite TV, internet, and cell phone service are available.

The nearest community is the town of Willow, 40 km (25 miles) to the west, with a population of about 2,000. The town of Willow offers few amenities, but supply centers are available in Wasilla, roughly 25 miles southwest of the Project site, and Palmer, about 20 miles to the south. A rail siding and access to the Alaska Railroad system is available in Wasilla, and the ports of Anchorage, Seward, and Whittier are accessible via road and railroad from Wasilla and Palmer.

Electrical power extends up the Willow Creek valley toward the site to within 16 km (10 miles). Currently GTOR is planning on operating the processing facility in the town of Willow for a readily available power source and more affordable land. Water for underground mining operations is obtained from mine water discharge and from a production well at the Enserch portal.

5.3 Physiography

The Project is located along the south-western margin of the Talkeetna Mountains, between the Chugach Mountains to the south and the Alaska Range to the north. The Talkeetna Mountains form a rough arc bordered to the west by the lower Susitna Valley, to the east by the Copper River Basin, and to the south by the Matanuska River. Southwest of the Willow Creek mining district, the Talkeetna Mountains gradually give way to the Susitna Valley. The south front of the Talkeetna Mountains descends steeply into the Matanuska Valley. The Talkeetna Mountains taper gradually to the north beyond a westward flowing portion of the Susitna River until the terrain rises again to form the Alaska Range.

The productive portion of the Willow Creek mining district lies above the tree line at around 550 m (1,800 ft.). Elevations range from 700 m (2,300 ft.) in the Willow Creek drainage to 1,600 m (5,200 ft.) just north of the Independence mine. Several peaks near the Project area are above 1,800 m (6,000 ft.) with elevations up to 2,440 m (8,000 ft.) farther north. The Talkeetna Mountains are deeply scarred by Quaternary glaciation (Ray, 1933, 1954). Typical U-shaped glacial valleys and higher, closely spaced glacial cirques are the dominant geomorphic features of the Willow Creek basin.

North flowing tributaries (Wet, Grubstake and Homestake Creeks) draining Bald Mountain ridge emerge from hanging valleys and drain into Willow Creek. Willow Creek is fed from the north by the south flowing Shorty Creek, and Craigie Creek, which is central to the Project.

6. HISTORY

Gold was first discovered at Willow Creek in 1897 within placers at Grubstake gulch. A small amount of placer production was won from the creek until 1906. The first lode-gold quartz vein was discovered in 1906 by Bob Hatcher and named the Skyscraper vein and later became part of the Independence mine. This discovery formed the nucleus of the Alaska Free Gold Mining Company who began to develop the area that later became known as the Independence mine. By 1909, four companies were actively developing the district (primarily at the Independence and Gold Bullion veins) and had installed several stamp mills. By 1912, three gold mines were in operation; the Gold Bullion, Alaska Free Gold and Gold Quartz (Brooks and others, 1913). Initial development in the district was hampered by the lack of access into the Talkeetna Mountains. Most mines operated for only three to five months during the summer season since they relied on the river system to supply power (Chapin, 1920). Early operations were also hampered by the lack of water available at higher elevations to operate mills (Capps, 1914). The first prospectors and miners landed by steamboat at the head of Knik Arm and had to transport supplies over difficult wagon trails into the Willow Creek district (Capps, 1914).

The Lucky Shot vein system on the western side of the district was not discovered until 1918. Production first began in 1919 at the War Baby mine and continued under various operators until 1936. At that time the War Baby and Lucky Shot mines were consolidated and operated by Willow Creek Mines, Inc. New management began to focus more on exploration and development in 1938. Willow Creek Mines discovered the faulted-off western segment of the Lucky Shot vein in 1939, known as the Coleman area. Mining of the Coleman area was continued until 1942 with limited development work due to labor shortages.

Prior owners and operators historically associated with the Project are summarized below:

- 1919 to 1921 - Bartholf & Company
- 1921 to 1928 - Willow Creek Mines, Ltd
- 1928 to 1939 - Willow Creek Mines, Pardners Mines
- 1939 to 1942 - Willow Creek Mines, Conwest Exploration
- Early 1940's to 1979 - Alaska Pacific Consolidated Mining Company
- 1979 to Early 1990's - Enserch Exploration, Alaska Gold, Solomon Resources
- Early 1990's - Present - Alaska Hardrock, Inc.
- 2004 to 2010 - Full Metal Minerals as leased from AHI
- 2012 - Miranda Gold Corporation
- 2014 - Gold Torrent, Inc./Miranda Gold Joint Venture

6.1 Historical Exploration

In the 1980's Enserch Exploration conducted an extensive exploration program which included soil sampling, drilling and underground exploration. The only available data from Enserch's exploration is the drillhole information, which is discussed in detail in report Sections 10 and 11.

FMM acquired the Project in 2004, and subsequently moved forward with well-planned helicopter supported drilling exploration from 2005 to 2010. FMM completed seven (7) NQ2 size core drill holes in 2005 and intercepted significant gold mineralization. The drilling tested and confirmed earlier Enserch drill intercepts in the Coleman area. The majority of seventy-three (73) core drill holes drilled in 2006 tested the central Coleman area with step-out fan drilling. Also, the Murphy area and the Nippon mine were drill tested in 2006. Encouraging results from drilling in the Coleman and Murphy areas prompted additional drilling in 2007 in these areas. Drilling during 2007 was concentrated northeast and southwest up and down dip from the historic Coleman workings and extended mineralization to the west both above and below the historical Coleman workings. Other drilling verified Enserch intercepts down dip from the Lucky Shot workings and expanded the mineralization in the Murphy area. In 2008 ten core drill holes were completed in the War Baby mine area. Two of ten drill holes intersected gold mineralization with the best intercept of 0.4 m @ 33.75 g/t gold.

In 2008 work focused on rehabilitation of the Coleman portal and underground access to mine a bulk sample from the Coleman resource area. A total of 42.6 m (140 ft.) of a planned 143 m (470 ft.) bulk sample access was completed.

In 2009 an additional 29 core drill holes were designed to precede possible commercial development of the property by FMM. This drilling was designed to infill the northeast trending upper "Golden Egg" mineralization. The "Golden Egg" was a small high-grade shoot without 43-101 compliant resources, only a rough estimate of size and grade potential for the purpose of designing drilling.

By the end of 2009, FMM was in financial distress and announced an option agreement with Harmony Gold. As part of the agreement Harmony Gold could earn a 60% interest in the Lucky Shot Property. Through 2010, Harmony continued permitting, geological studies, mine planning metallurgical work, and mill and tailings design and. On November 18, 2010, Harmony Gold and FMM terminated the Lucky Shot agreement. Harmony fulfilled its obligations under the option agreement, but due to various difficulties and a decision to focus on other projects, decided to terminate the agreement (FMM News Release, 2010).

6.2 Historical Production

Historical production from the Project is taken from anecdotal data summarized by various sources since production was halted due to World War II.

Gold was discovered in the Lucky Shot area in 1918, with subsequent mining from 1922 to 1942. Most of the historic production (1923 to 1928) came from the Hogan Stope at the Lucky Shot mine. During the 1922 to 1951 era, the Lucky Shot and War Baby mines produced about 252,000 ounces of gold (Stoll, 1997). War Baby, located just east of Lucky Shot, reported production from 1922 to 1927 with an average grade of 74.7 g/t (2.18 opt) from veins varying in thickness from 25.4 to 381 mm (1 to 15 in.). The average vein width in the Lucky Shot mine was 1.52 m (5 ft.) wide with workings extending to a depth of 152 m (500 ft.) down dip. In 1979, Enserch Exploration Inc. completed extensive activities inclusive of drifting, drilling (18holes), and geochemical sampling. A 100-ton-a-day mill was built in the early 1980's. Approximately 9,091 tonnes (10,000 tons) of material was processed from various sources. Into the 1990s, a few companies explored the property and mined another 1,000 tons (Hawley and Visconty, 1982b).

The principal gold-recovery period for the Gold Bullion was 1911 to 1927, milling 50,900 tonnes (56,000 tons). Toward the end of these years, reports indicate about 435 kg (14,000 ounces) were recovered by 1915.

Willow Creek Mines operated the Project until 1927, recovering about 1,866 kg (60,000 ounces). Plans were made to treat 3,636 tonnes (4,000 tons) of tailings in a 30-ton per day cyanide plant in 1914, in addition to continuing underground mining through the winter. In 1929, Willow Creek Mines used the Gold Bullion mill to process the Lucky Shot material. Later in 1938, leasers reworked some of the tailings with a cyanidation process. The property produced a rough total of 2,395 kg (77,000 ounces) of gold (Stoll, 1997; Capps, 1915; Grybeck, 2008).

6.3 Historical Mineral Resource and Mineral Reserve Estimates

Historical mineral resource and reserve estimates for the Project are largely unavailable. Charles S. Coleman reported probable reserves of 17,500 tons just after the Lucky Shot mine closed in 1942 (Stoll, 1997), but the methods used to arrive at this figure do not comply with current industry standards and the reported value is presented here for historical completeness only.

7. GEOLOGICAL SETTING AND MINERALIZATION

The following description of the geological setting for the Project is largely excerpted and modified from the technical report prepared by Linebarger (2014). HRC has reviewed the geologic data and information in detail, and finds the descriptions and interpretations provided herein reasonably accurate and suitable for use in this report.

7.1 Regional Geology

Oceanic plate subduction under the continental margin has dominated the Alaska geologic landscape for the last 150 million years, creating various metamorphic terranes. Many of the Middle Cretaceous and younger mineralized systems in southern Alaska are related to volcanic accretion and hydrothermal activity (Goldfarb, 1997). This includes the mesothermal veins in the Willow Creek mining district.

The Willow Creek mining district is located in the southern Peninsular terrane and the Alaskan part of the Wrangellia composite terrane at the southern end of the Talkeetna Mountains batholith (Goldfarb, 1997). The Jurassic Peninsular terrane consists of a well-stratified sequence of variably metamorphosed Paleozoic and Mesozoic volcanic and sedimentary rocks and a Jurassic granite batholith (Detterman and Reed, 1980; Jones and others, 1987; Nokleberg and others, in Plafker, G. and Berg, 1994). The Peninsular terrane occurs south of the Wrangellia terrane, north of the Chugach terrane, and is bounded by the Talkeetna thrust and West Fork fault to the north and by the Border Ranges fault to the south. See Figure 7-1. (Clift et al., 2005, after Trop et al., 2002).

An island-arc chain greater than 2,000 km long formed in the Jurassic period above a north-dipping subduction zone. This arc is represented by volcanic rocks of a distinct geochemical signature within the Peninsular terrane around the Anchorage area (Barker and Grantz, 1982). During the Late Jurassic to Early Cretaceous, the Talkeetna arc was subsequently accreted to southern Wrangellia and Alaska (Clift et al., 2005).

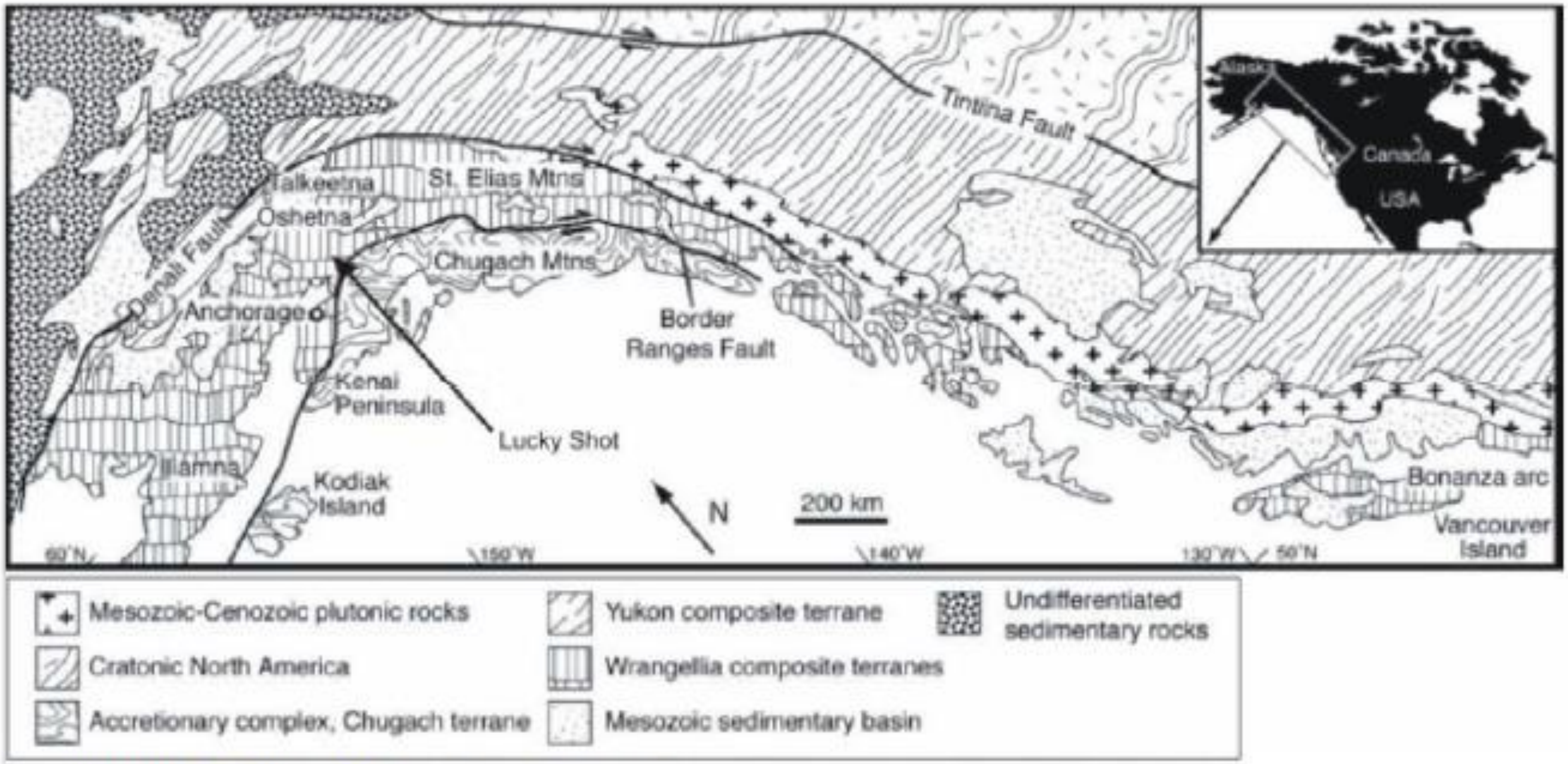
The Talkeetna arc is the tectonic feature defining the character of the Peninsular terrane. Defining lithologies of the Peninsular terrane are deep-level ultramafic-mafic assemblages of the Border Ranges, the Late Triassic and Early Jurassic Talkeetna Formation, and the Middle Jurassic part of the Alaska-Aleutian Range Batholith. The arc extends for over 1,500 km along the Alaska Peninsula and northeastward to the eastern Copper River basin (Plafker, 1994). Reverse faulting of older structures resulted from the accretion of the Talkeetna arc.

The Chugach terrane, an accretionary complex comprised of ultramafic oceanic crustal fragments and pelitic sediments, lies south of the Peninsular terrane and in bounded to the north by the Border Ranges fault margin. Reactivation of the Border Ranges fault by strike-slip movement affected the rocks on either side, developing a regional shear zone.

Local structures hosting gold bearing quartz veins in the Willow Creek area are synchronous with the high angle, dextral strike-slip movement of the Border Ranges fault (Cooley, 2006; Goldfarb, 1997).

During a brief period of extension in the middle-late Cretaceous the Willow Creek quartz diorite-tonalite body intruded along a shallow north-dipping crustal break. The intrusion provided gold-bearing fluids to occupy similar smaller scale north-dipping reverse fault zones within the intrusive.

The Kluane magmatic belt is an arc of late Cretaceous and early Paleogene magmatism (75-56 Ma) extending from southwestern to southeastern Alaska. Cretaceous plutons of the Kluane arc may have provided fluids and metals for the 66.9 to 65.6 Ma gold-bearing veins in the Willow Creek mining district (Harlen et al., 2003).



(Modified from Cooley, 2006, after Trope et al, 2002)

Figure 7-1 Regional Geology (Reprinted from Linebarger, 2014)

7.2 Local Geology

The following description of the geology local to the Project is based on historical literature and limited mapping completed by FMM. Rock types are comprised of faulted Upper Cretaceous granite and the Willow Creek quartz diorite-tonalite (a phase of the Talkeetna Batholith), bordered to the south by the Jurassic Hatcher Pass Schist of the Peninsular terrane. The Willow Creek quartz diorite-tonalite is intruded by a variety of dikes with chilled margins ranging in composition from aplite to lamprophyre (Cooley, 2009).

Northwest striking and steeply (60-80°) dipping faults crosscut the entire Willow Creek intrusive. Less faulting occurs in the Hatcher Pass Schist. The faults have dextral and normal movement with displacement to the east creating an en echelon pattern of vein segments, dikes, and fault blocks. Gold mineralization at the Project is hosted in shallow north-dipping mesothermal veins within shears or reverse faults in the intrusive (Cooley, M., 2006). Quaternary cover is predominantly a product of glaciation with only minor re-working.

Mapping completed in the Project area has defined the structural features related to the mineralization in the Coleman and Lucky Shot areas, and throughout the greater Willow Creek mining district (Figure 7-2).

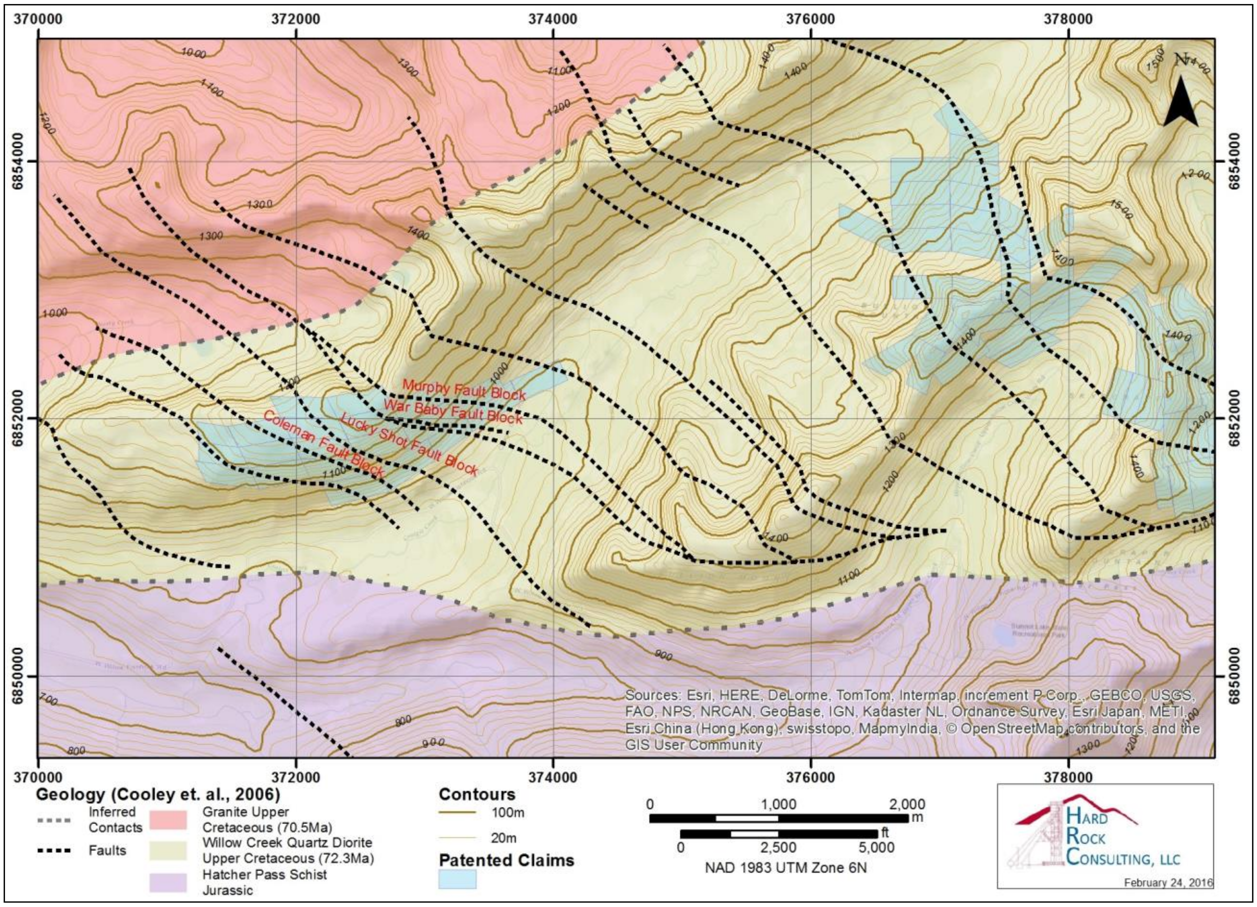


Figure 7-2 Property Geology

The structural fabric specific to the Project area was developed during the late Jurassic to early Cretaceous when the Talkeetna arc accreted to Alaska above a north-dipping subduction zone. When the Chugach terrane accreted to the south margin of the Peninsular terrane after the late Cretaceous, older suture zones between these terranes were reactivated with dextral strike-slip movement.

The Willow Creek quartz diorite-tonalite intrusive was emplaced during a brief period of extension on the Border Ranges fault. An age of 72.3Ma (late Cretaceous) has been is indicated for the Willow Creek quartz diorite using uranium – lead radioactive dating.

Subduction produced south-directed reverse-faulting in the overlying brittle Willow Creek diorite. Magmatic or metamorphic fluids moving along these faults formed the gold-quartz veins of the Willow Creek area. Post mineral, dextral strike-slip faults offset of the Willow Creek gold-quartz veins into en echelon segments.

The Project area includes east-northeast faults with northwest shallow dips hosting the productive veins, and N45-50°W faults with steep dips offsetting those veins. Refer to Figure 7-3. Minor narrow gneissic layering exists as local shear fabrics in the intrusive during crystallization. Other steep northerly dipping gneissic layers may be early dikes or veins emplaced and subsequently sheared while the intrusives were still semi-ductile before cooling (Cooley, 2006). The veins of the Lucky Shot mine and the Coleman zone are significantly displaced by major northwest faults and minor faults of variable orientation. The northwest faults produce a number of vein segments within fault blocks, the largest with a displacement of a few hundred meters. These northwest faults within the intrusive displace east-west vein segments en echelon to the south and east by dextral strike-slip and normal movement.

Bends and kinks in underground workings of the Lucky Shot and Coleman mines reflect dextral displacement of the vein along northwest faults (Cooley, 2006). Post-mineralization movement has caused shearing and brecciation within the plane of the vein, with subsequent healing by a later phase of quartz in places. This post mineralization movement has caused the veins to be recessively eroded, with only rare vein outcrops in the district.

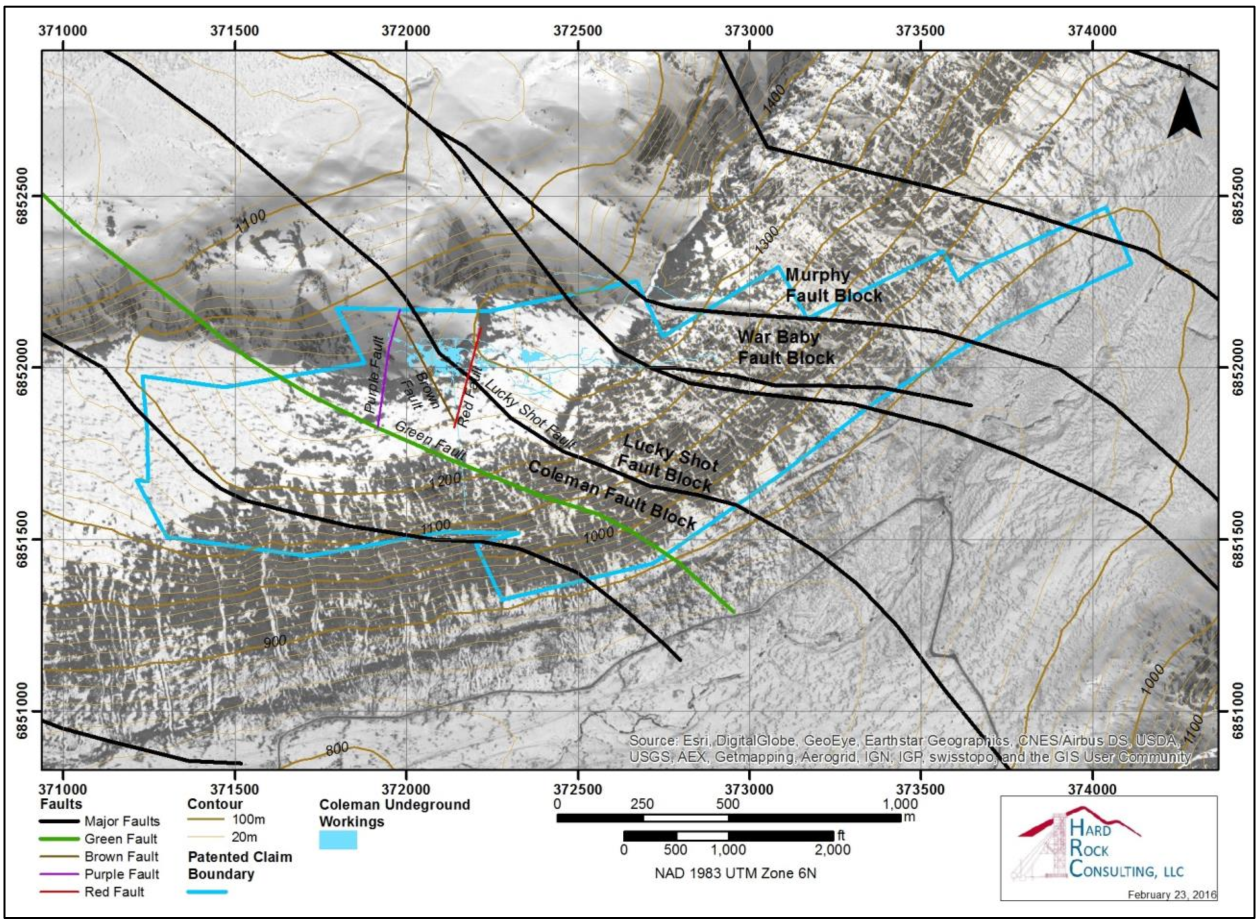


Figure 7-3 Structural Geology

Faults and foliation in the Jurassic Hatcher Pass Schist are not important to mineralization in the Project area. The schist foliation primarily dips northward at 36° but dips less than 20° near the southern contact of the Willow Creek intrusive. This may indicate a broad open fold with a sub-horizontal east northeast-trending axis (Cooley, 2006) within the metamorphic rocks.

Data derived from core drilling (2005 to 2009) was used to produce a sectional and three dimensional fault interpretation of the Coleman area. M. Cooley's Coleman fault study for FMM was also incorporated into this interpretation (Cooley, 2009).

The Coleman fault block is located to the west of the Lucky Shot fault block (Figure 7-3) and is separated by a fault with significant down dip and dextral offset on its east side. The Lucky Shot vein segment is located about 30 m (100 ft.) to the east of the Coleman vein segment. The fault to the southwest of the Lucky Shot fault has significant down dip movement and strikes approximately N49°W with a steep dip, offsetting a portion of the Coleman vein by about 20 m (66 ft.)

M. Cooley's Purple fault (Figure 7-3) in section shows about 6 m of offset to the east between drillholes C09- 169 and C06-41. This fault trends N29-34°E with a steep dip. It will be important to recognize this fault for any exploration taking place to the north of the Coleman area as it could impact future mining. The N34°W Brown fault and N17°E Red fault, both with probable steep dips and lateral offsets, appear as jogs in mapped drifts (Cooley, 2009).

Gold grades are commonly elevated where veins cross, splay, and merge. The age of mineralization by 40Ar/39Ar methods, using six samples of sericite from vein selvages, ranges from 66.9 to 65.6Ma (Harlan et al., 2003). Post mineralization faulting is thus younger than 65.6 Ma at a minimum.

The 3,884.9 square km (1,500 square mile) late Mesozoic Talkeetna batholith includes the Willow Creek quartz diorite on its southern border. The quartz diorite is cut by a variety of faults with displacements up to hundreds of meters. The major northwesterly trending northeast dipping faults cut the Willow Creek quartz diorite into four blocks with normal and dextral displacement. The faults offset gold producing veins up to 245 m (800 ft). In the workings, the fault damage zones can exceed 30 m (100 ft.) in width, with strong cataclastic textures and gouge (Ray, 1954). Cataclastic fabric with brittle failure indicates more than one movement on these structures. Horizontal movement on the major fault to the east of the War Baby is 183 m (600 ft.). Numerous northeast striking narrow faults are reported with displacements as much as 61 m (200 ft.). Field investigations and more recent mapping (Ray, 1954 and Cooley, 2009) indicate the joint and quartz-vein patterns in the intrusive do not extend into the Hatcher Pass Schist to the south.

The coarse grained biotite and hornblende-bearing quartz diorite-tonalite intrusion has a weak to moderately well-defined planar foliation of minerals interpreted to be primary intrusion fabric. The flow fabric has a general dip to the north-northeast at roughly 53°. A few narrow zones of gneissic layering are interpreted to represent minor local shear fabrics. In some areas, the gneissic layering could represent early dikes or veins intruded and sheared while the intrusive was semi-ductile before cooling. The gneissic zones dip steeply mainly to the north, except for a rare zones dipping southward.

The historically productive west-northwest trending gold-bearing veins in the Project area generally dip about 30° to the north but with steeper dips of 45° in the shallower part of the Lucky Shot vein segment. The gold-bearing veins probably occupy reverse faults related to north-directed subduction in the late Cretaceous. The pluton intruded at steeper angle than foliation in Hatcher Pass Schist. The intruding magma created a moderately north-dipping planar intrusive fabric. Continuing north-directed slab subduction may have created a sympathetic thrusting in the overlying quartz diorite-tonalite rocks. This thrusting would have created open brittle deformation through which gold-bearing fluids could flow to deposit quartz and metals.

7.3 Lithology

Important bedrock formations in the Project area include the Hatcher Pass schist and the Willow Creek quartz diorite-tonalite. Lesser glacial and alluvial deposits are also present. The following stratigraphic units are present within the Project area, and are described in order from Jurassic to Quaternary (oldest to youngest):

7.3.1 Jurassic Hatcher Pass Schist

The Jurassic Hatcher Pass pelitic schist underlying the southern part of the Project is chloritic and highly fissile. Locally, it is strongly folded. The schist contains dark gray quartz, muscovite, albite and chlorite, with minor chloritized garnet, biotite and tourmaline. The chloritization of the biotite and garnet suggests the schist was originally metamorphosed to amphibolite grade metamorphic facies in the Jurassic and later subjected to retrograde metamorphism in the Cretaceous (Silberman et al, 1978). Alternatively, other workers think the original metamorphic mineral assemblage was cordierite, garnet, muscovite, and chloritized biotite. This would indicate the rocks were metamorphosed from upper greenschist to lower amphibolite facies. The chloritic schist is silver-gray with strong foliation of abundant muscovite plates. Well-developed plagioclase porphyroblasts up to 5 mm (0.2 in.) across and black tourmaline needles up to 10 mm (0.4 in.) are common. More gneissic schist with more feldspar and less muscovite occurs in some places (Ray, 1954).

The schist likely originated as a deep water fine-grained mudstone deposited in the oceanic trench on the south margin of the Talkeetna arc. Fragments of oceanic crust were included with trench sediments during subduction as indicated by ultramafic enclaves comprising listwanite or serpentine in the schist of the Grubstake Gulch area (Cooley, 2006). The serpentinized ultramafic bodies in the Grubstake Gulch area have talc, chlorite, actinolite, tremolite, fuchsite and opaque minerals of the serpentine group (Albanese et al., 1983).

Dikes in the schist are rare. Small discontinuous pegmatite dikes are seen rarely. The pegmatite in schist is dissimilar to the pegmatite in the Willow Creek quartz diorite. Rare dikes with fine-grained phenocrysts of plagioclase, and lessor pyrite, quartz and hornblende are reported. The dikes are usually about 0.76 m (2.5 ft) or less in thickness and can be either along or crosscutting foliation (Capps, 1915).

The Hatcher Pass schist is considered to be Jurassic in age. Recent dating of detrital zircons collected from Grubstake Gulch in the south Project area had a significant population with a range in age from 160-210Ma. The schist may include sediment shed from the eroding Jurassic Talkeetna arc. Proterozoic age zircons in the Grubstake Gulch sample indicate sediment influx also from outside the Jurassic age Talkeetna arc rocks (Van Wyck and Norman, 2005).

7.3.2 Cretaceous Willow Creek Quartz Diorite-Tonalite Intrusive

The Willow Creek quartz diorite-tonalite is 74-73M, located near the southern margin of the larger Talkeetna batholith and hosts the mesothermal quartz veins of the Willow Creek mining district. The Willow Creek intrusive is bounded by 67-65Ma granite to the west (Csejtey et al., 1978, Madden-Mcquire et al, 1989). The intrusive is described as having primary flow structures and a gneissic texture near its southern boundary (Ray, 1954). The intrusive is medium grained except near its southern contact where finer grains may represent a chilled margin with the schist.

The intrusion is primarily comprised of plagioclase, quartz, biotite, and hornblende with the accessory minerals microcline, orthoclase, sphene, apatite, zircon, and magnetite. Two mineralogical variations are apparent: 1) conspicuous large crystals of hornblende with small plates of biotite, and 2) scattered large books of biotite and with small crystals of hornblende. These variations are gradational. Hornblende is coarser near the center of the intrusion. The quartz and plagioclase content are fairly uniform throughout. Feldspar tends to be more calcic and zoned within the central intrusive (Ray, 1954).

Plagioclase is modally more abundant (57-68%) followed by quartz (15-24%) in the tonalite. Quartz occurs interstitially between other minerals. Sometimes quartz replaces surrounding feldspar grains. A distinguishing feature of the quartz is the presence of microlites as hairline rows of minute gaseous or liquid inclusions showing slippage along small fractures. The most abundant mafic mineral is slightly bent biotite making up to 8-16% of the rock mass. The biotite alters to chlorite along cleavage planes.

Hornblende occurs as a primary mafic mineral from 3-13% by volume. Apatite and magnetite are common in hornblende. Traces of zircon and apatite are included in hornblende with epidote along hornblende cleavage planes. Hornblende can poikilitically occur in plagioclase and plagioclase poikilitically in hornblende. Magnetite is the most common accessory mineral, usually with biotite and hornblende. Magnetite is sometimes included in biotite and hornblende, and apatite is occasionally included in magnetite. The accessory minerals, microcline and orthoclase, compose less than 1 percent by volume.

Sphene is observed in mafic minerals and sporadically in plagioclase. The accessory minerals most common in the tonalite are sericite, chlorite, calcite, epidote, prehnite and leucoxene. Zircon is the least abundant accessory mineral occurring as inclusions in hornblende, biotite and magnetite. Calcite and epidote are common with seriticized plagioclase feldspar. Usually, biotite is altered to chlorite and rarely hornblende is chloritized. Leucoxene is seen bordering ilmenite or occurring as patches proximal to ilmenite (Ray, 1954).

7.3.3 Cretaceous Quartz Monzonite (Granite)

The Cretaceous 67-65 Ma granite is present in the western part of the district intruding the Willow Creek quartz diorite, (Csejtey et al., 1978, Madden-Mcquire et al., 1989). Also about 5.6 km (3.5 miles) northeast of the Lucky Shot mine, a small plug-like body of granite occurs on a ridge high point. The Willow Creek intrusive has granite dikes with widths up to 3 m (10 ft.). Generally, the granite is light- colored, mafic-poor, medium to fine-grained and with 7-16% anorthite plagioclase, quartz and potassium feldspar. The potash feldspar is usually microcline. Hornblende is commonly absent. The granite contains accessory biotite, muscovite, myrmekite, and microcline microperthite (Ray, 1954).

7.3.4 Tertiary Dikes

Dikes are variable in composition, not abundant, and difficult to map. Dike displacements are mapped in order to map faults in the area. The dikes, except for the diabase (diorite), are older than the post-ore faults in the Willow Creek area. The diabase dikes tend to follow post-ore faults and may be contemporaneous with them. Younger lamprophyre and diabase dikes tend to crosscut older aplite and pegmatite dikes. Other age relationships for dikes cannot be reliably established, but mafic dikes seem to be youngest. Lamprophyre dikes are cut by post-mineral faults in the Lucky Shot mine (Ray, 1954). The dike compositions are granite, lamprophyre, diabase (diorite), aplite and pegmatite, described below from youngest to oldest.

7.3.5 Granite

Granitic dikes are interpreted to be the youngest. They are coeval with the intrusion of a biotite muscovite granite pluton on the north side of the Willow Creek quartz diorite. Uranium-lead dates from the granite indicate an age of 70.5 Ma (Harlan et al., 2003). Granite near the gradational contact with the quartz diorite is steeply dipping and includes numerous xenoliths of quartz diorite (Cooley, 2006).

7.3.6 Diabase

The diabase is dense and black like the lamprophyre, however, the diabase is intensely sheared and lacks phenocrysts. Some, but not all, diabase dikes contain pyroxene. The diabase dikes have a width up to about 6.1 m (20 ft.) and can be traced up to a few hundred meters. They strike generally east to east northeast and dip steeply to the north dip. The most abundant mineral is laths of plagioclase feldspar. Original interstitial mafics are altered to green biotite, magnetite, chlorite and calcite. Rarely, feldspars are altered. Olivine may occur with chloritic alteration (Ray, 1954).

7.3.7 Lamprophyre

Lamprophyre trends northerly and dip southwest. Lamprophyre dikes are greenish-black, dense, and fine-grained, with hornblende phenocrysts. Lamprophyre is difficult to recognize in weathered outcrop. The dikes are generally parallel to southwest dipping joints in the intrusive. The dikes are highly continuous and offset by minor and major faults.

The lamprophyres are divided into two types. The first is porphyry with abundant twinned hornblende phenocrysts in a fine-grained groundmass of zoned plagioclase feldspar and minor small hornblende crystals. Its groundmass includes accessory minerals of sphene, biotite, calcite, deep chestnut-red rutile, chlorite, magnetite, and needles of hornblende microlites.

The second lamprophyre is coarser grained and mostly equigranular. The major minerals are well zoned feldspars, and hornblende altered to chlorite. The groundmass contains magnetite, chlorite, calcite, sphene, amphibole microlites, and lesser biotite, epidote, and zircon (Ray, 1954).

7.3.8 Aplite

Aplite dikes are common throughout the Willow Creek quartz diorite. The aplite follows southwest- dipping joints; strikes and dips are highly variable. The dikes are linear but with irregular widths, and can be traced for only about 30 m (100 ft.). The aplite can indicate minor fault offsets, but are not very useful to note major fault movement.

Aplite is strongly associated with pegmatite. They occur near each other and sometimes within the same dike with aplite as the border to a central pegmatite band or the reverse. Aplite is light-tan to pink with a fine groundmass. Most aplite dikes are less than 10 cm (4 in.) wide; a few are up to 152 cm (6 in.) wide. The dikes are composed of quartz, microcline, orthoclase and plagioclase. The plagioclase is commonly altered to sericite; the microcline unaltered. Rare biotite is altered to chlorite. Apatite, epidote, and a black opaque are common accessory minerals with secondary calcite along feldspar cleavage planes (Ray, 1954).

7.3.9 Pegmatite

The pegmatite has highly variable strike and dip changing within short distances. The pegmatite is typically traceable for less than 30 m (100 ft.) with variable widths tending to splay. Their widths are usually 5 cm (2 in.) to 122 cm (48 in.) in width crosscutting relationships indicate aplite and pegmatite are the oldest dike rocks in the Project area.

Coarse pink feldspar and quartz are the main mineral assemblage in pegmatite. Lesser coarse muscovite and biotite occurs with minor coarse black tourmaline needles and minor plagioclase. The radioactive minerals uraninite, cyrtolite, allanite and thorite are reported (Ray, 1954).

7.3.10 Quaternary Cover

The Quaternary cover includes alluvium, glacial debris, and talus (Ray, 1954). The glacial debris has very little remobilization.

7.4 Mineralization

The Willow Creek mining district's only known economic mineral is gold contained in mesothermal veins within low angle shears in the Willow Creek quartz diorite-tonalite intrusive. The important lode gold-bearing shears strike 60-80° and dip 30-60° northerly. The argon isotope method dates sericite in vein associated alteration to approximately 66.9 to 65.6Ma.

Another group of structures trending nearly due north with dips from near horizontal up to 45° to the west are also significant, but are only near to the intersections with the more important east-northeast trending veins. The productive veins have coarsely crystalline quartz with minor pyrite, sphalerite and other sulfides, telluride and visible gold. Gold deposition is a late event and only very minor amounts of gold are occluded within sulfides.

There are a number of small gold placers in the Willow Creek mining district with very minor development.

7.4.1 Coleman Veins

The Coleman zone contains two primary sub-parallel and some subsidiary gold-bearing quartz veins hosted by quartz diorite that strike approximately N83°W with a 25-35° dip to the north northwest. Relatively good continuity is demonstrated in the two veins. The veins can be separated by up to 20 m (66 ft.). They are located in proximity to the historic Coleman working. In the south-half of the area, the two veins merge and splay. Gold values may increase near these intersections. High gold values can occur anywhere in the system and are not restricted to any particular vein. All of the mineralized zones are sub-parallel; however, potential exists for veins in secondary or tertiary structural directions related the sense of shear of the primary structure hosting the veins. Figure 7-4 shows a 50-meter-thick cross section of the Coleman quartz vein system with drillhole intercepts.

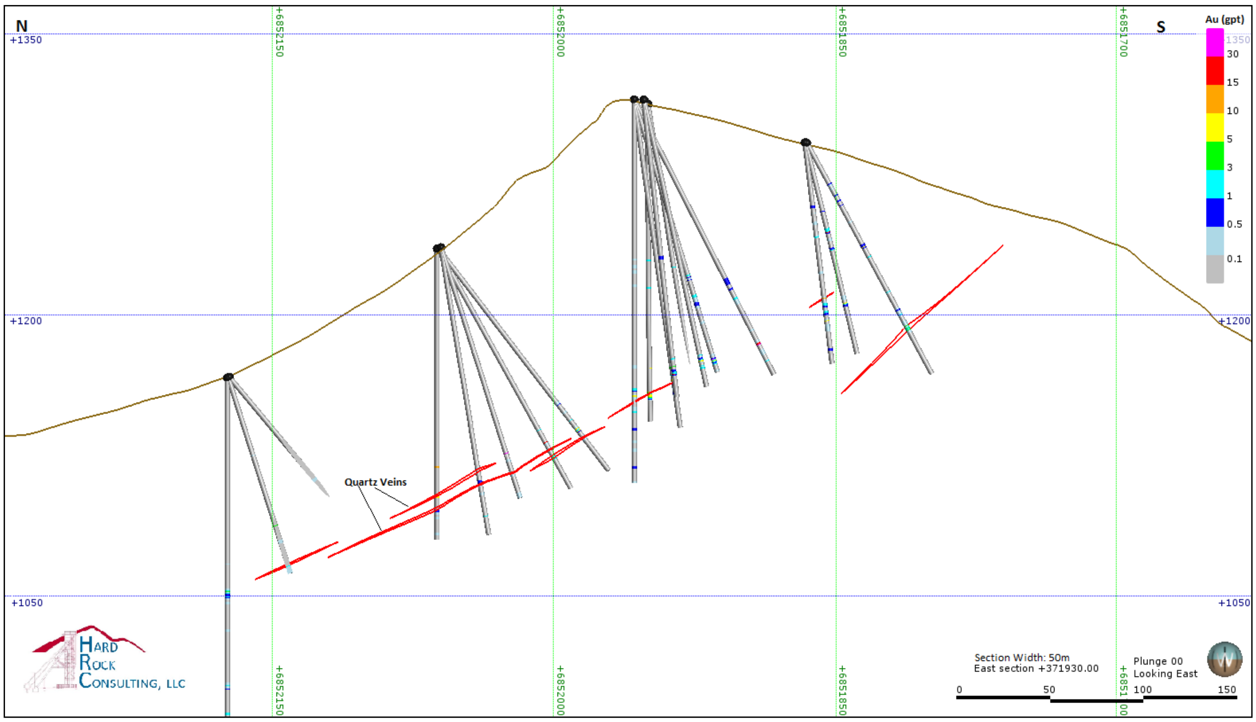


Figure 7-4 Cross Section (371930E)

Massive mesothermal quartz vein or veinlet packages with 2-3% metallic sulfides and telluride characterize the mineralized drillhole intercepts in the Coleman zone. Some zones include only one vein with a typical thickness of 0.5 m (1.64 ft.). More commonly a zone contains many quartz veins with individual widths of at least a centimeter. Disseminated visible gold, tetrahedrite and tellurides, pyrite, arsenopyrite and chalcopyrite are the primary minerals in the veins or near vein margins. Occasionally, banding and healed breccias are notable in the veins, and clay alteration or gouge. Veins and quartz diorite closer to surface have iron oxide, minor hematite, and trace malachite.

7.4.2 Coleman Footwall and Hanging Wall

The historic underground sample maps of the Coleman mine rarely distinguish the hanging wall from the footwall samples. This may be due to the relatively narrow average width, 0.83 m (2.8 ft.) of the veins.

Selective mining may have not been practiced such that the entire width may have been mined. The drill logs from 2005 to 2009 and the re-logging completed by Ms. Candace Dykeman in 2013 did not provide information to indicate strong differences between the hanging and footwalls of the veins.

7.4.2.1 Coleman Area Alteration

Review of the drill logs indicate that both hanging wall and footwall vein alteration envelopes are highly variable along strike and dip. Generally, most alteration envelopes are <14 m (46 ft.). The alteration includes chloritization, seritization, silicification, and argillization accompanied by disseminated pyrite and arsenopyrite. Increased gold within a vein does not appear to have a direct relationship to the intensity of alteration. Discontinuous veins outboard from the primary resource veins may have strong alteration envelopes. The alteration envelopes outside the veins often carry low gold grade of less than 3 g/t). Frequently wide halos of arsenic exceed the width of alteration envelopes.

8. DEPOSIT TYPES

The veins of the Project are mesothermal. This vein type was referred to as mesothermal deposits by Lindgren in 1933, orogenic metamorphic-hosted deposits by Bohlke (1981), and low-sulfide gold quartz veins by the U.S. Geological Survey's classification by Berger in 1986. Medium-grade facies metamorphic rocks host ninety percent of Alaska's lode gold production (Goldfarb, 1997).

Mesothermal veins are associated with linked networks of faults and low displacement shears of crustal scale shear systems in orogenic belts of Archean to Cenozoic age throughout the world. The Phanerozoic age mesothermal veins are related to crustal breaks characterized by dismembered ophiolite between diverse assemblages of island arc, subduction complexes and continental-margin clastic wedges. The Archean age veins are well age-constrained for the Superior Province and the Canadian Shield (2.68-2.67 Ga), and the Yilgarn Province, Western Australia (2.64-2.63 Ga) and are related to major transcrustal breaks within stable cratons of remnant terrane collisional boundaries (Ash, 1996).

Mesothermal veins are usually found in regional post-peak metamorphic regimes terranes of greenschist facies, and less commonly of sub-greenschist to granulite facies (Inverno, 2002). Related deformation indicates strain in brittle to ductile (plastic) regimes. Hydrothermal alteration and mineralization can occur during shear (Cox, 1999). Geothermal gradients during subduction events cause hydrated sedimentary sequences and volcanics to contribute hydrothermal fluids. The fluids migrate long distances at variable depths to form gold-bearing quartz veins (Groves, 1997).

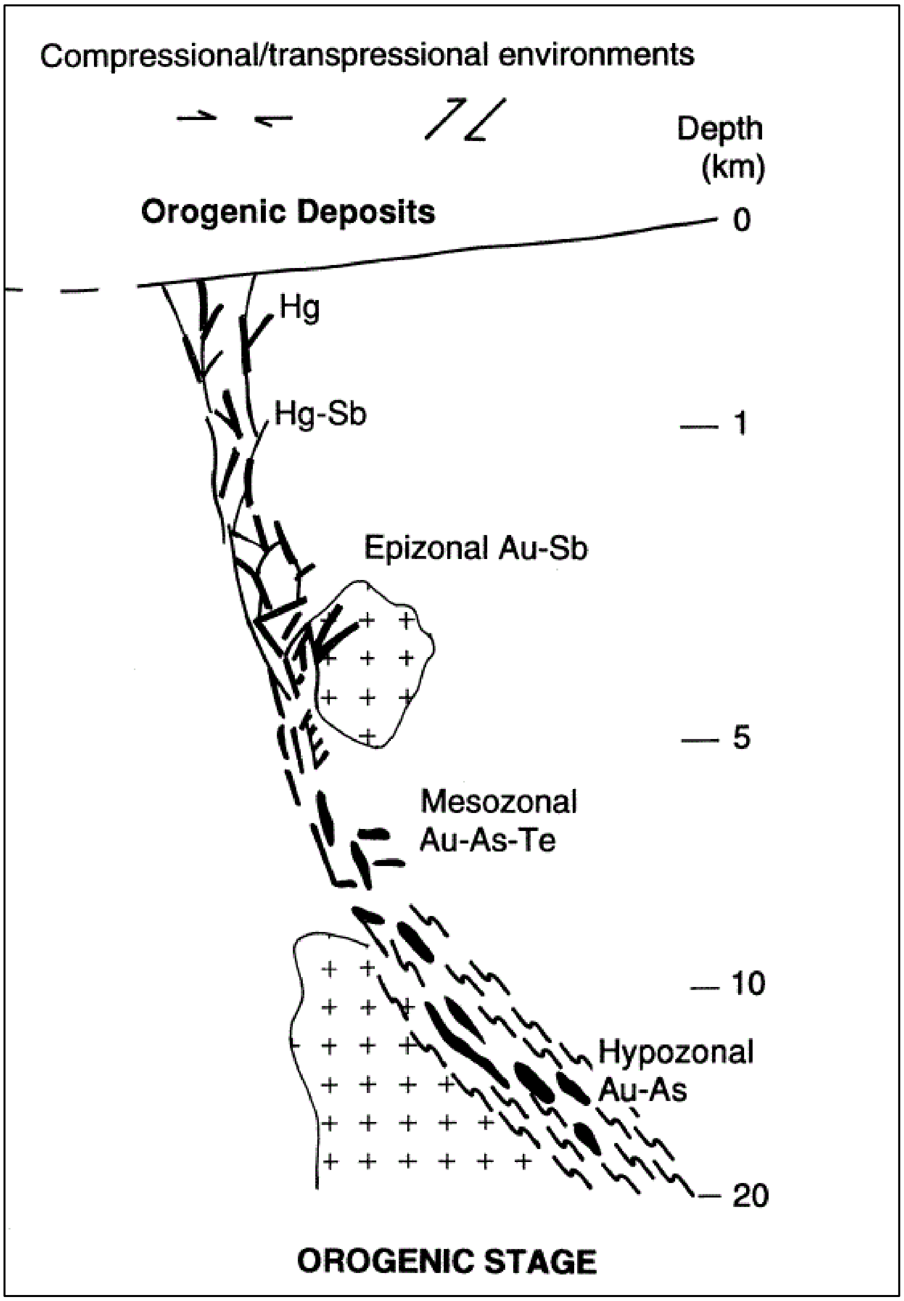


Figure 8-1 Schematic Geologic Cross Section of a Low-Sulfide Mesothermal Gold Deposit (Groves 1998)

These transcrustal breaks with highly connective shear systems can generate high, well dispersed fluid flow to scavenge metals from deeper crustal levels to develop large economic gold deposits at higher crustal levels (Cox, 1999). Vein arrays indicate the fluid responsible for mineralization and alteration is produced during episodic deformation events under supralithostatic pressure (cf. Cox et al., 1987).

Widespread laminated, banded, or crack-seal textures indicate fluid pressure fluctuations during deformational cycles. Episodic brittle or ductile slip events produce breccias and shear veins with variable alteration mineralogical assemblages in sub-horizontal extensional vein arrays. The vein displacements range from a few tens of centimeters to a few tens of meters (Cox, 1999).

Gold is typically in quartz veins and or disseminated in hydrothermally altered envelopes of faults and shears (Cox, 1999). Sometimes mineralization resides in alteration halos next to faults and shears rather than in the veins. This may result from sulfidation reactions in iron-rich host rocks causing gold to precipitate.

The predominant mineralization is quartz, native gold, pyrite, galena, sphalerite, chalcopyrite, arsenopyrite, and pyrrhotite. Locally tellurides, scheelite, bismuth, tetrahedrite, stibnite, molybdenite, and fluorite can occur (Berger, 1987). Silver, copper and antimony can be significant byproducts of mesothermal deposits (Ash, 1996). In many cases vein quartz is grayish or bluish due to the presence of fine-grained sulfides. Gangue mineralogy can include calcium, magnesium, and iron carbonate, tourmaline and graphite. Veins textures are usually massive or banded. Veins can occur in the form of discrete planes, saddle reefs, breccias, stockwork, or as anastomosing gashes and dilations. Generally, the mineralization lacks zoning or is consistent through the system (Berger, 1986; Ash, 1996).

Mesothermal vein alteration is dominated by quartz-sericite-carbonate-pyrite assemblages with minor siderite, ankerite, and albite. Silicification, pyritization and potassium metasomatism can occur within a meter from the mesothermal veins. This proximal alteration is enveloped with a broader halo of carbonate alteration including ferrous dolomite veinlets. Chromium mica, dolomite, talc, and siderite are alteration products in ultramafic wall rocks. Sericite, fuchsite, tourmaline, scheelite, and rutile are more common in granitic wall rocks (Cox, 1999; Ash, 1996; Berger, 1986).

Mineralization occurs during deformation at depths between 4 and 15 km (2.5 to 9.3 mi.) and temperatures between 250 to 450 °C. The deformation is in fluid pressure regimes of 1 to 3 kilobars and hydrothermal fluids of low salinity composed primarily water and carbon dioxide. Seismic activity provides episodic flow rates and fluid pressures in faults and shear zones producing gold deposits with multi-events.

Seismic reflection surveys in major goldfields indicate the mesothermal deposits are located in the hanging wall of large shear zones, and formed at mid-crustal depths beneath the greenstone sequences. Overpressured fluids are transported from depth to hanging wall structures in during seismic events (Cox, 1999).

Generally, the major fluid-chemical processes controlling gold precipitation in the mesothermal environment are fluid-rock reactions, phase separation due to fluid pressure fluctuation; and fluid mixing processes. Fluid-rock reactions such as sulfidation reactions most effectively control gold deposition as a result of fluid discharge from shears and faults into Fe-rich reactive host rocks. Gold grades may be poor where shears transect Fe-poor host rocks even though wall rocks are altered.

Seismic rupture events causing sudden decrease in fluid pressure can drive phase separation processes and gold deposition. This gold deposition is preferential to veins, particularly in severe pressure reduction zones like dilatant jogs. Fluid mixing processes especially between reduced wall-rock fluids and more oxidized water-carbon dioxide fluids within the faults produce gold vein deposition (Cox, 1999).

9. EXPLORATION

9.1 Exploration Work Completed by FMM

In 2006, underground mapping was completed in the Coleman workings and in the Enserch exploration adit. Surface mapping was done in the vicinity of the Lucky Shot mine. In 2007 widely spaced soil sampling in the West Coleman area was completed. The exploration activities of 2008 included ten core drillholes in the War Baby area and underground mapping, and sampling in the Coleman area underground workings and also mine dump sampling. In 2009 underground mapping and sampling was carried out in accessible workings and 29 infill drillholes were drilled in the Coleman area.

FMM completed rock and mine dump sampling during the exploration years 2005 to 2009. In 2005, 43 surface, underground, and trench samples were collected. These samples have sequential numbers from 8751 to 8786 and results for gold and multi-element ICP. The surface and underground rock sampling results are compiled with Sample ID, Location, Type, and Au in ppm. In 2007, rocks from the West Coleman 250 & 500 Lucky Shot levels, and dump samples from the War Baby dump were collected.

Three hundred and twelve (312) soil samples were collected from a grid in the west Coleman area (Figure 9-1). Forty-five dump and 23 rock samples were taken. The samples were assayed for gold and analyzed for multi-element by ICP by ALS Chemex of Fairbanks.

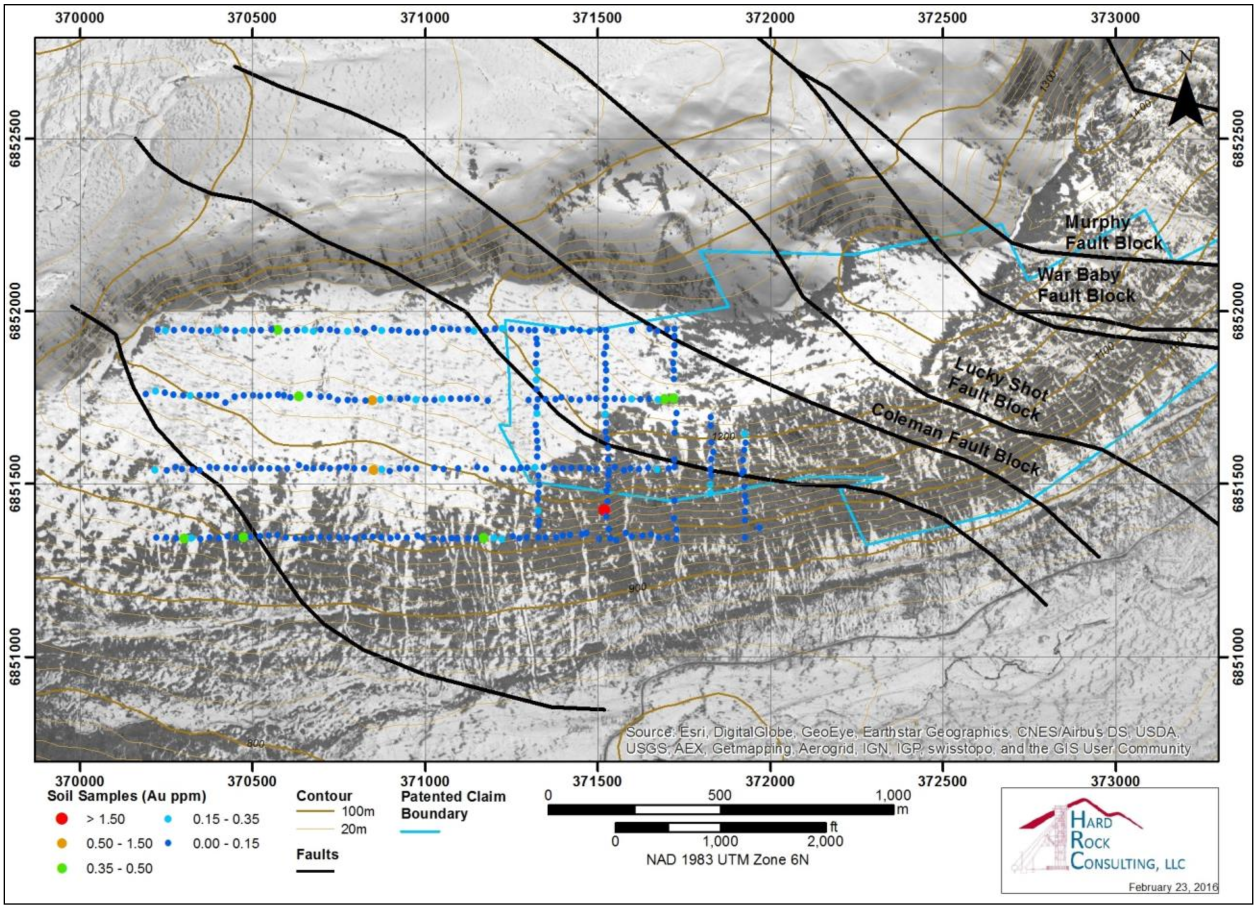


Figure 9-1 2007 West Coleman Soil Samples

The west Coleman soil grid values ranged from 0.025 to 4.48 ppm gold with an average of 0.139 ppm gold. The highest gold values are located at the east end of the grid. Forty-five samples were collected from the Lucky Shot 250 Level, Lucky Shot 500 Level, and the War Baby dumps. Twelve samples from the Lucky Shot 250 dump averaged 4.47 ppm Au. Twenty-three samples from the Lucky Shot 500 level dump averaged 3.68 ppm Au, and the 10 samples from the War Baby dump averaged 2.57 ppm Au.

9.2 Exploration Work Completed by Miranda and GTOR

In 2014, Miranda and GTOR geologists completed a 234-sample soil grid and collected rock samples of quartz vein rubble. Results strongly suggest that the vein system mined on Bullion Mountain within the Willow Creek mining district extends beyond the fault that bounded historic production. The highlight of the sample program is the discovery of three quartz vein sub-crops that assayed 1.48 oz, 0.50 oz and 0.53 oz Au/t (50.74 g, 17.05 g and 18.15 g/t Au).

The soil anomalies show discrete breaks from background and are defined as having gold values of greater than 0.100 ppm to 2.00 ppm. The samples were taken on a grid spaced 20 m X 200 m. Three or more linear anomalies are reflected consistently on 5 successive lines of the grid; the anomalies extend for 800 m and remain open. The three samples of quartz vein sub-crop within the anomalous gold in soil area indicate that the soil anomalies likely reflect new high-grade vein segments within the district. Visible gold was present in all of the vein sub-crops sampled.

In 2015, GTOR shipped 2,200 lbs. of ore material from the Lucky Shot area to Hazen Laboratories in Golden Colorado for metallurgical and geochemical analysis. Three sample splits of quartz + sulfide + carbonate from the Lucky Shot shear zone ranged from 25 to 26.2 g/t Au. Results from this testwork are included in Section 13 and in the appended Hazen report of 2016.

10. DRILLING

Since production was halted in 1942, only two operators have completed drilling programs on the Project. Enserch drilled 18 exploration holes from 1978 to 1985. FMM drilled 173 holes between 2005 and 2009. A summary of drillholes is presented in Appendix B.

10.1 Drilling Exploration Conducted by Enserch Exploration

The 1984 Enserch conducted a core drilling program in the Lucky Shot area under controlled conditions by a professional exploration company utilizing accepted North American practices. The drill program was observed by Mr. Scott Eubanks who is a professional miner and was present for the duration of underground activities. Drill procedures, core logging, and sample assaying were under the supervision of professional geologic personnel.

GTOR conducted personal interviews with Mr. Scott Eubanks on January 22nd and January 25th regarding the Enserch drill program at the Lucky Shot mine located approximately 25 miles east of Willow, Alaska. Mr. Eubanks is a professional miner and held supervisory underground mining positions during the Enserch drill program. The following information is corroborated by a combination of historical data sheets, field notes, corporate records, and underground field observations by Mr. Eubanks.

10.1.1 Lucky Shot Exploration

Enserch actively explored the Lucky Shot mine and greater Willow Creek district from 1978 through 1985. Enserch drove a 1,500 ft. (457 m) exploration drift below the Lucky Shot mine and drilled 11 underground core holes totaling 10,364 ft. (3,159 m) during the 1984 field season. Additionally, Ensearch drilled 7 surface core holes in the Coleman zone totaling 4,881 ft. (1,488m) (Table 10-1). A complete list of the significant intercepts is presented in Appendix C.

Table 10-1 Exploration Drilling Completed by Ensearch

Year	Type	Number	Total Meters	Total Feet
1984	Underground	11	3,159	10,364
1984	Surface	7	1,488	4,881

10.2 Drilling Methods

In early 1984, a drill station was set up at the end of the Enserch drift. A Longyear Hydacore Model 28 core drill was transported to the drill station in pieces. Core drilling utilized a BX diameter core barrel and 10 ft. steel rods. Samples were extracted by tripping out of the hole after 10 ft. runs. The drill holes were reported to be surveyed every 100 ft. All core was saved in waxed card board boxes which have been removed from the site. Drilling was completed both during day and night shifts under the supervision of geologists.

10.3 Drilling Exploration Conducted by FMM

Five drilling campaigns, totaling 173 core holes, were drilled by FMM from 2005 to 2009 in the Lucky Shot area. One hundred and seventy-three surface core holes (173) were completed in the vicinity of the Lucky Shot and Coleman areas. Table 10-2 shows holes drilled by FMM during the years 2005-2009 and excludes holes drilled on the Nippon prospect. Seven other core holes, N06-16 to N06-22, were drilled on the Nippon prospect quartz veins, and are not shown in Figure 10-1.

Table 10-2 Exploration Drilling Completed by Full Metal Minerals

Year	Number	Total Meters	Total Feet
2005	7	922	3,024
2006*	73	12,713	41,709
2007	54	13,303	43,646
2008	10	2,389	7,838
2009	29	4,776	15,670

* Excludes Nippon Prospect Holes

Nearly all of the holes drilled during the FMM campaigns have associated geotechnical and geologic logs, assay certificates (fire assay gold and multi-element ICP), and core photographs. Thirty-five (35) do not have down-hole surveys. Drillhole coordinates use UTM NAD83 Zone 6. All selected core samples of varying length from 2006 to 2009 were assayed for gold and analyzed for a multi-element suite with ICP completed by ALS Chemex, of Fairbanks. The 2005 core samples assayed for gold and analyzed for multi-element suites with ICP and were prepped by Alaska Labs and analyzed by BSI of Reno, Nevada. Samples with visible gold assayed during the years between 2006 and 2009 were re-assayed using the metallic screen method. However, in the year 2005, samples with coarse gold were not re-assayed using this method. Two samples from 2005 were eventually re-assayed with the metallic screen method during an audit.

During 2005 to 2009, Peak Exploration Ltd. of Yellowknife, Canada, was contracted by FMM to drill NQ2 (1.99" or 50.5 mm) size core on the Lucky Shot project. In 2005, the drill holes were surveyed with both an acid test for dip and an electronic 'Icefields' downhole survey tool to measure both dip and azimuth. A Brunton compass was used to set drill angles. Initially drill sites were located using a non-differentially corrected hand-held Global Positioning System device in Universal Trans-Mercator (UTM) NAD 83 Zone 6, with estimated position accuracy of 3- 5 m. When the 2006 to 2009 holes were completed, they were surveyed using a Leica TCR803 Total Station. The down hole surveys were done with a multi-shot FlexIT Smart Tool Drillhole Survey System which measures dip, azimuth and temperature.

Prism Helicopters of Wasilla, Alaska was contracted for support operations and used a Hughes 500D for site access from 2005 to 2009 and drill repositioning during 2008 to 2009. Northern Pioneer helicopters of Big Lake, Canada, used a Bell 204 to move the drills from 2005 to 2007.

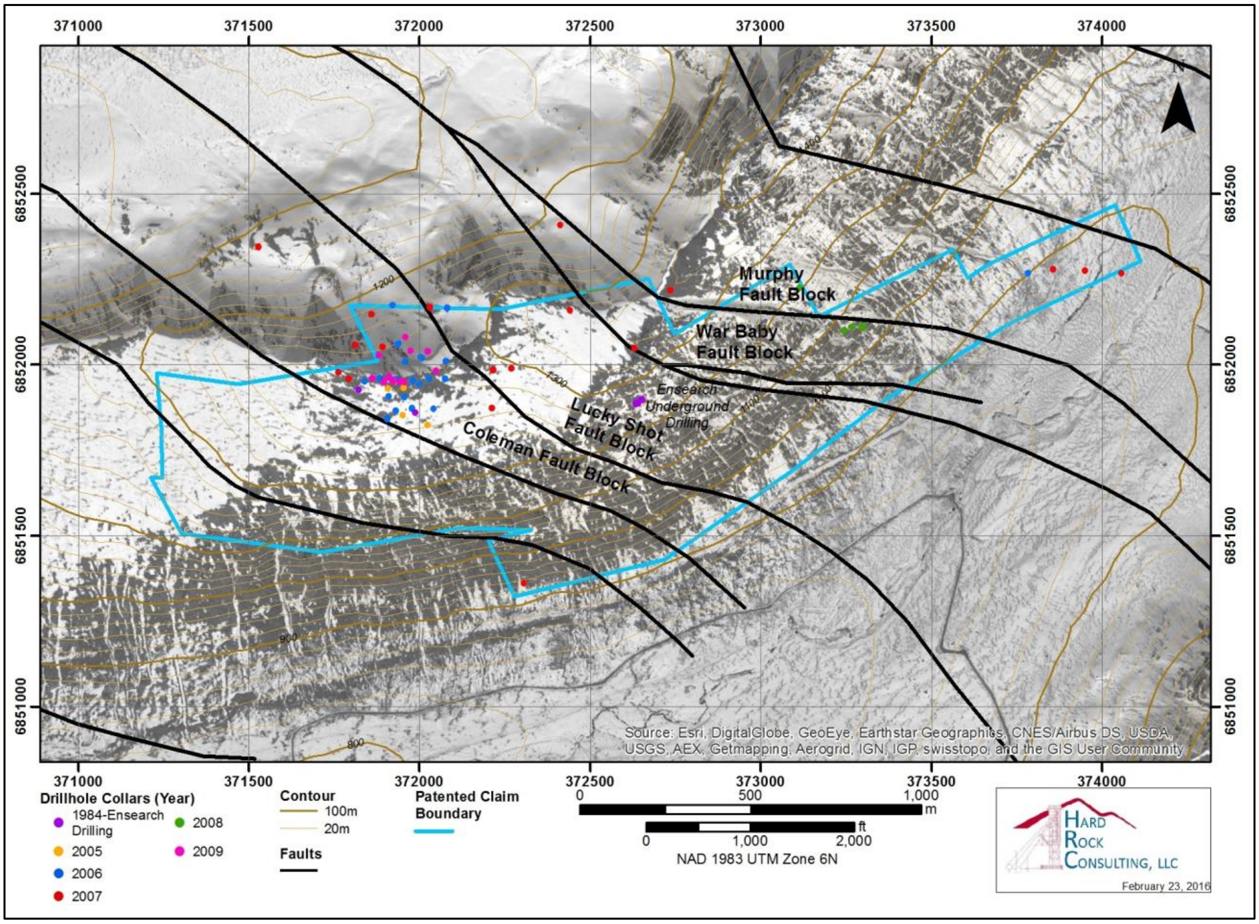


Figure 10-1 Project Drillhole Collars

10.3.1 2005 FMM Drilling Exploration

FMM drilled seven core holes (C05-08 to C05-14) totaling 920 m (3,020 ft.) in the Lucky Shot mine area during 2005 (Figure 10-2). The core holes were surveyed, logged, photographed, and selectively sampled in zones of interest. Peak Exploration Ltd of Yellowknife, Canada was the drilling contractor.

FMM targeted the western portion of the Lucky Shot vein system (Coleman area) located up dip about 200 m (650 ft.) from the historic Coleman workings. The drilling information indicated the vein strikes 277° and dips 24° to the north-northeast. All seven drill holes intersected the vein. The drilling intersected mineralized zones of up to 4.0 m (13.1 ft.) in long with values of up to 219.06 g/t Au (7 oz). The drilling showed that the system was open on the western edge and down dip. The eastern limit is defined by the Lucky Shot fault, representing the border between the Lucky Shot and Coleman workings (McLeod and Light, 2005). A complete list of the significant intercepts is presented in Appendix C.

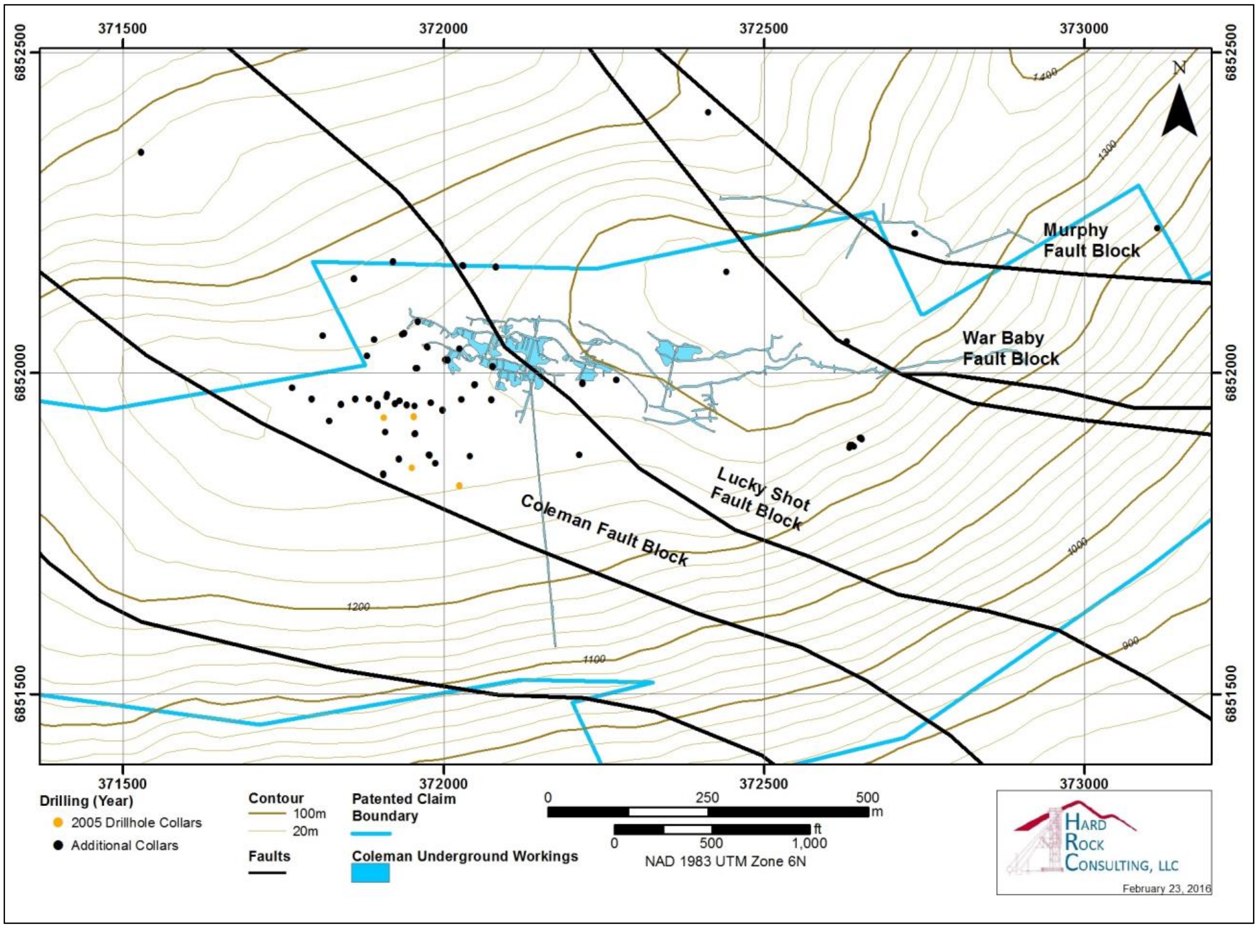


Figure 10-2 FMM 2005 Drilling

10.3.2 2006 FMM Drilling Exploration

Core holes C06-15 to C06-84, C06-86, and C06-89 to C06-90 were drilled by FMM at the Lucky Shot, Coleman and Murphy areas and N06-16 to N06-22 were drilled at the Nippon property. The 73 holes (not including the 7 holes drilled at Nippon) totaled 41,774.9 ft. (12736.3 m) of NQ2 size core (Figure 10-3). All core holes were surveyed, logged, photographed and selectively sampled in zones of interest. Peak Exploration Ltd. of Yellowknife, Canada, was the contractor for the drilling program conducted by FMM during 2006.

The Coleman vein, Lucky Shot vein, Murphy area and the Nippon vein were drilled during 2006. Three holes were drilled into the Murphy area, and the remainder were primarily drilled in the Coleman and the Lucky Shot mine areas. The drilling in the Coleman area was primary driven by the success of the 2005 drilling in the area 200 meters up-dip from the historic Coleman workings and west of the Lucky Shot fault. The 2006 drilling program had two main objectives:

1. Delineate the up and down-dip portions of the Coleman area and test along strike, and
2. Explore for a north-eastern faulted extension to the War Baby area (Murphy area).

The 2006 drilling proved to be a successful program. The Murphy area is faulted down and with dextral movement along the East fault relative to the War Baby mine. The drilling of the Murphy area extended the strike length of the Lucky Shot vein to over 1,800 meters, with continuous gold mineralization based on historic mining and recent drilling. Hole C06-89 drilled a mineralized structure 300 meters below the surface intersecting 0.4 m (down-hole length) of 19.3 g/t (0.56 opt) Au in the Murphy area. Late in the season, FMM initiated two core holes, C06-84 and C06-86, to test the anomalous gold intercepts reported by others. Weather conditions caused abandonment of the holes prior to reaching their target depth.

The Coleman fault block was delineated both up and down dip from the historic workings, and along strike to the west during 2006. The drill holes intended to pierce the mineralized zone at 23 m (75 ft.) centers. The vein appears to strike about 277° and dips 20-45° NE, local variations in dip and strike are commonly large. Widths of the zone varied from <1m to 4.5 m. Holes with the highest grade intercepts were concentrated within a NE plunging shoot, up dip of the historic Coleman workings.

The Nippon area was successfully drilled, intersecting the quartz veins of the prospect. Only minor quartz veining, alteration and low gold values were intercepted. The highest intercept encountered in the drill holes came from N06-18 from 222.9-223.32 m (down-hole length) @ 3.42 g/t Au (0.099 opt) Au (McLeod, 2006). A complete list of the significant intercepts is presented in Appendix C.

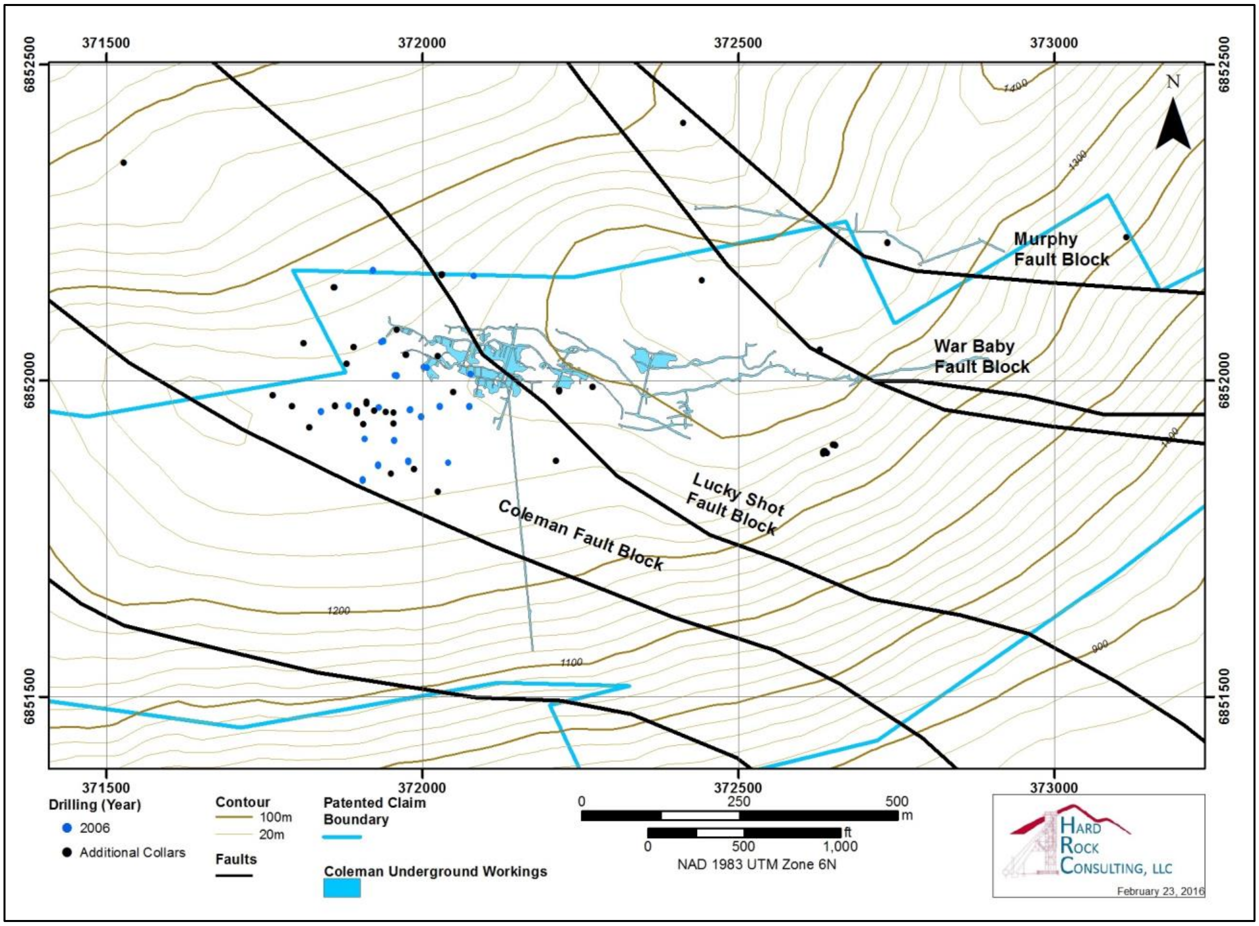


Figure 10-3 FMM 2006 Drilling

10.3.3 2007 FMM Drilling Exploration

In 2007, FMM drilled 54 core holes during 2007 totaling 13,794.1 meters (45,244.7 ft.), refer to Appendix B and Figure 10-4. The Murphy, Lucky Shot and Coleman areas were the targets for the drill holes. All core holes were surveyed, logged, photographed and selectively sampled in zones of interest.

Peak Exploration Ltd. of Yellowknife, Canada, was contracted by FMM to drill about 13,000 m (40,000 ft.) of NQ2 size core on the Lucky Shot.

The 2007 drill program had three goals at Lucky Shot:

1. Test an eastward extension of mineralization encountered in drill hole C06-89 in the Murphy area during the 2006 drilling program.
2. Extend the down dip extension of the Lucky Shot vein below historic underground workings.
3. Extend gold mineralization along strike to the west and, up and down dip in the Coleman area.

The success of the Murphy area drilling in 2006, dictated additional drilling. This was due to high grade gold values in the vein intercepts and similarities in the textures of the quartz veins to the Lucky Shot vein to the west. One of the five drill holes (C07-91 to C07-95) drilled in 2007 intersected a down-hole length

0.98 m @ 54.60 ppm Au (1.59 opt) in a fault zone with quartz veining including cataclastic textures and altered quartz diorite. All the drill holes intercepted the altered and veined fault zone in quartz diorite. The vein is similar in appearance to the veins of the Lucky Shot and Coleman areas.

FMM's 2007 drilling at the Lucky Shot vein was intended to verify and extend the known mineralization approximately 100 meters down dip from historic workings. The high grade was verified by FMM drilling of C06-84. This drill hole was re-entered and deepened and intersected the vein. This drilling indicated the vein to be about 2770 and dip 20-300 to the north. Drill holes C07-85, 104, 140, 143, 144, and 145 intersected quartz vein with cataclastic texture and visible gold. C07-143 contains a down-hole length 0.5 m @ 77.20 g/t (2.25 opt).

The drilling showed 250 m of continuity to the west and 150 m of continuity up and down dip from the historic workings with vein continuing to strike about 2770 with 20-450 dips NE, including common large local variations with strike and dip. Visible gold was in 19 of 42 drill holes with some holes containing more than one intercept with visible gold. Generally, a hanging wall fault marks the beginning of the Coleman zone. Below the fault, a brecciated (cataclastic) quartz diorite with gouge and one to two quartz veins in sericitization is common. The highest gold values with visible gold are usually in cataclastic zones with <5% to 20% total quartz included. A complete list of the significant intercepts is presented in Appendix C.

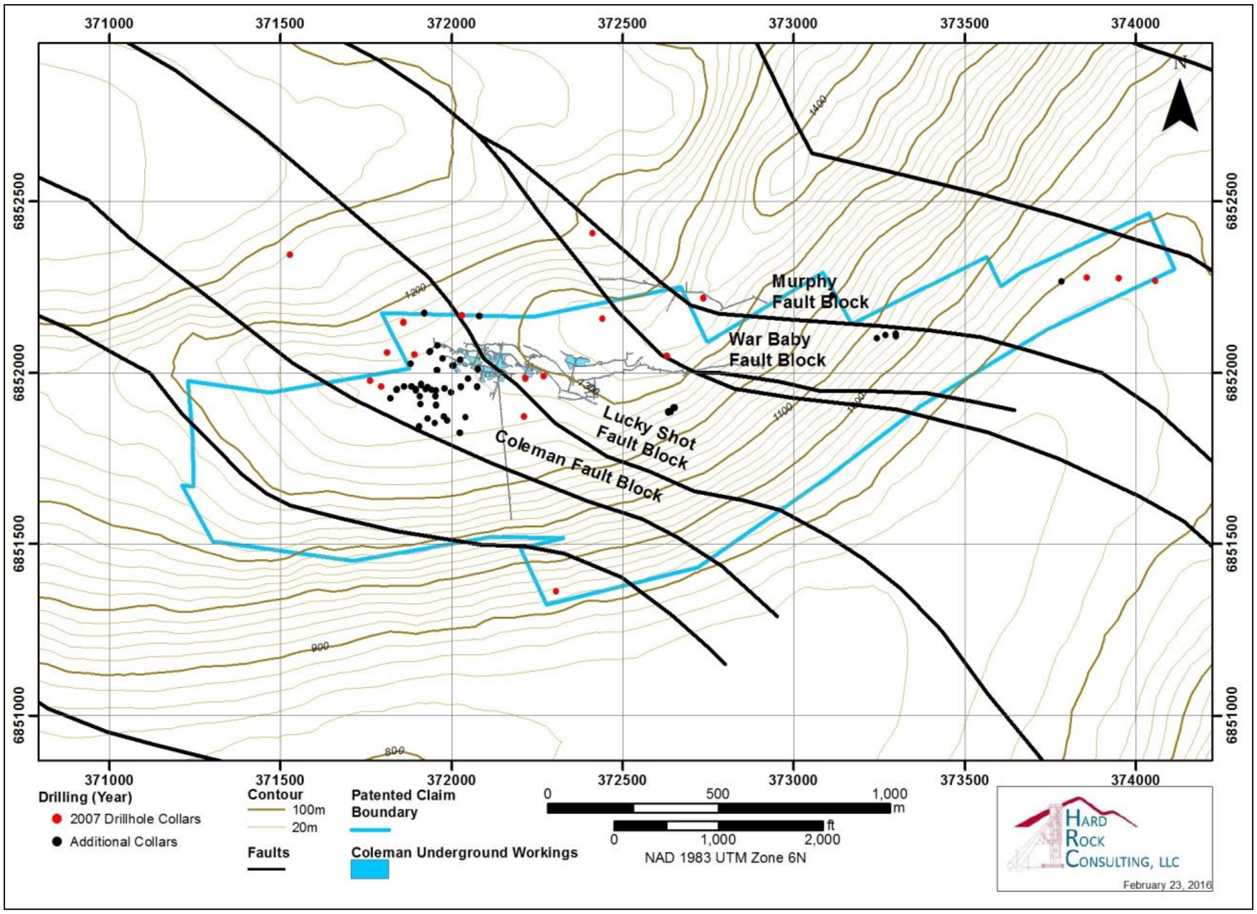


Figure 10-4 FMM 2007 Drilling

10.3.4 2008 FMM Drilling Exploration

The 2008 drill program at the Lucky Shot property was composed of ten surface drill holes, WB08-01 to WB08-10, located in the War Baby mine area, refer to Appendix B and Figure 10-5. Peak Exploration Ltd. of Yellowknife, Canada, was contracted by FMM to drill about 3,000 m (10,000 ft.) of NQ2 (1.99" or 50.5 mm) size core on the Lucky Shot project.

The total core hole footage drilled was 2,377 m (7,797 ft.). All core holes were surveyed, logged, (data in digital logs only), photographed and selectively sampled in zones of interest. No samples were re- assayed using the metallic screen method.

In 2008, the Lucky shot drilling program had three goals:

1. Identify near term working areas for future mining at the War Baby area of the Lucky Shot vein system, primarily by targeting fault extensions of mineralization missed by the historic operation, as well as down dip mineralization.
2. Infill areas of the Coleman shoot for stope definition (not completed).
3. Target near surface mineralization within the Murphy area (not completed).

Terrain proved challenging placing drill pads in most areas due to steep outcrops in this area. Multiple azimuths were used to test the vein in the War Baby area.

The War Baby area is located between the Lucky Shot and the Murphy areas. The War Baby area has been down-dropped with dextral movement relative to the Lucky Shot, and bounded between the post- mineral, steeply dipping and northwesterly striking Capps and East faults. Seven of FMM's ten drill holes drilled at War Baby, intersected anomalous gold values hosted in quartz veins with cataclastic textures.

Two of the drill holes intercepted old stopes. Five of the ten drill holes tested the down dip vein potential from the War Baby workings. The highest grades were intersected by holes WB08-01 and WB08-06 with down-hole intercepts of 0.4 m in length @ 33.075 ppm Au and 0.4 m in length @ 19.75 ppm Au respectively. A complete list of the significant intercepts is presented in Appendix C.

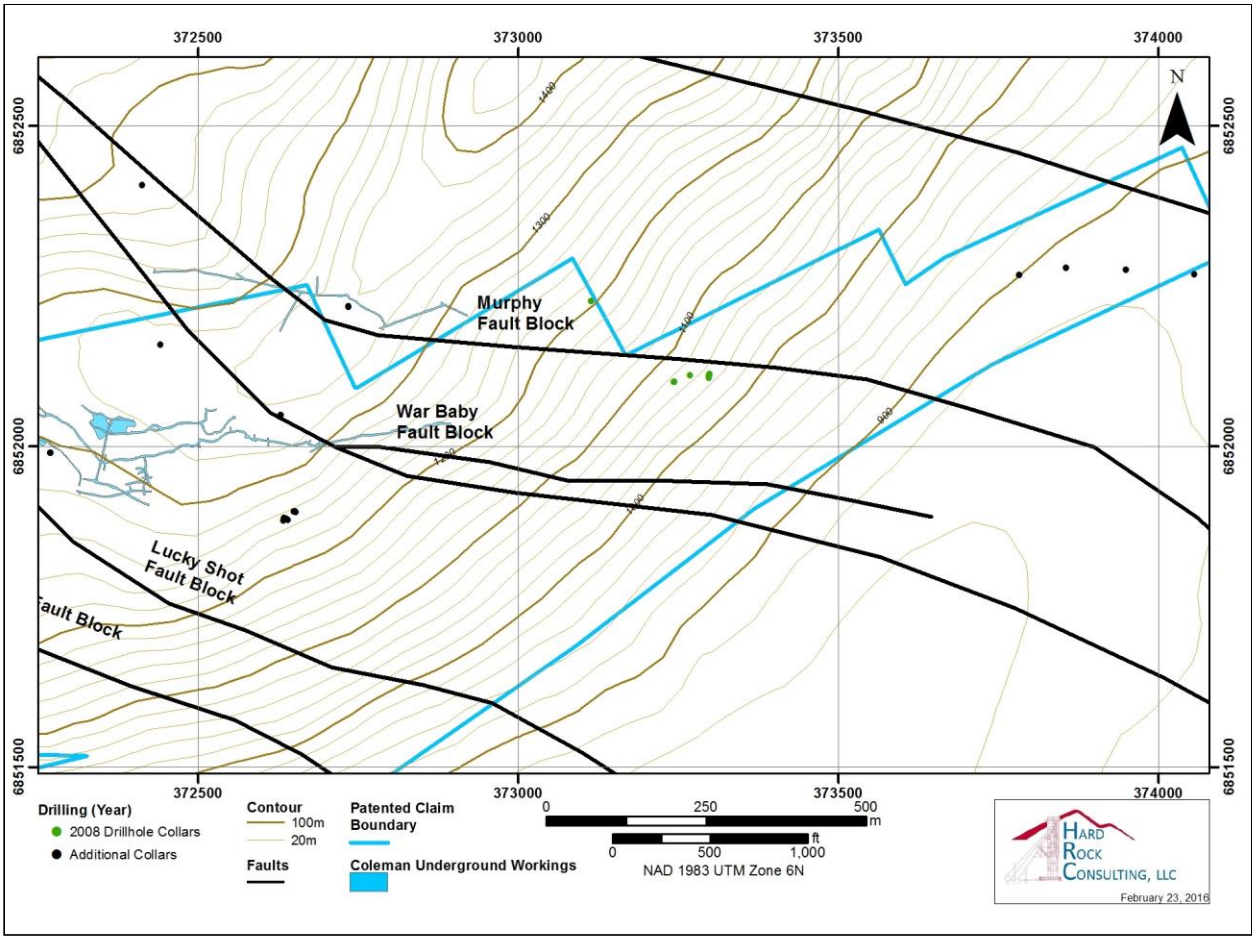


Figure 10-5 FMM 2008 Drilling

10.3.5 2009 FMM Drilling Exploration

During the 2009 drill program at the Lucky Shot property, 29 surface core holes were drilled (C09-147 to C09-175). The total core hole footage was 4,776 m (15,666 ft.), refer to Appendix B and Figure 10-6. All core holes were logged (original logs), surveyed, photographed and selectively sampled in zones of interest.

Peak Exploration Ltd. of Yellowknife, Canada, was contracted by FMM to drill NQ2 (1.99" or 50.5 mm) size core on the Lucky Shot project.

The drilling goal for the 2009 drilling was to infill drill along the NE-trending zone dubbed the "Golden Egg". This zone was revealed from a conceptual model developed from the 2005-2007 drill data for internal use in early 2008 and was meant to direct FMM towards "areas of opportunity" (Kirkham, email correspondence, March 6, 2009). The model suggested more drilling verification was needed before commercial development of the property, as recommended by Yukuskokon Professional Services (YKPS) of Wasilla, Alaska. YKPS was contracted to manage the Lucky Shot project.

YKPS recommended additional infill drilling along the NE-trending areas to verify the grade, thickness and orientation of the indicated shoots, and to add to the existing mineral inventory.

Objectives:

1. Drill 18 intercepts 1-1.5 meters in length averaging over 7 g/t Au.
2. Extend mineralization 50 m up and 200 m down plunge from the "Golden Egg" shoot.
3. Extend the northern shoot by 50 m up and 30 m down plunge from known mineralization.
4. Extend the down-dip extensions of both shoots at least 50 m below the historic underground stopes.

Significant results of the 2009 drilling program are in Table 10-3.

Table 10-3 Significant 2009 Drilling Intercepts

Drillhole Name	From (m)	To (m)	Interval (m)	Gold (g/t)
C09-152	154.2	155.6	1.4	55.5
C09-153	160.3	160.3	0.0	102.0
C09-158	121.6	122.2	0.6	58.2
C09-169	129.5	130.0	0.5	115.0
C09-171	130.7	131.1	0.4	249.0

During 2009, FMM entered into a joint venture agreement with Harmony Gold regarding the Lucky Shot project. As a result of the 2009 drilling, Harmony Gold and FMM planned an exploration program for 2010 to include:

1. Completing engineering studies, metallurgical test work and environmental studies;
2. Subject to a positive outcome, constructing underground access to the Coleman block;
3. Collect and process a bulk sample to confirm the continuity of gold mineralization within the Coleman property.

Subject to positive results from phases i) and ii), FMM and Harmony intended to commence phase iii) above in March 2011 (FMM Annual Report 2010). A complete list of the significant intercepts is presented in Appendix C.

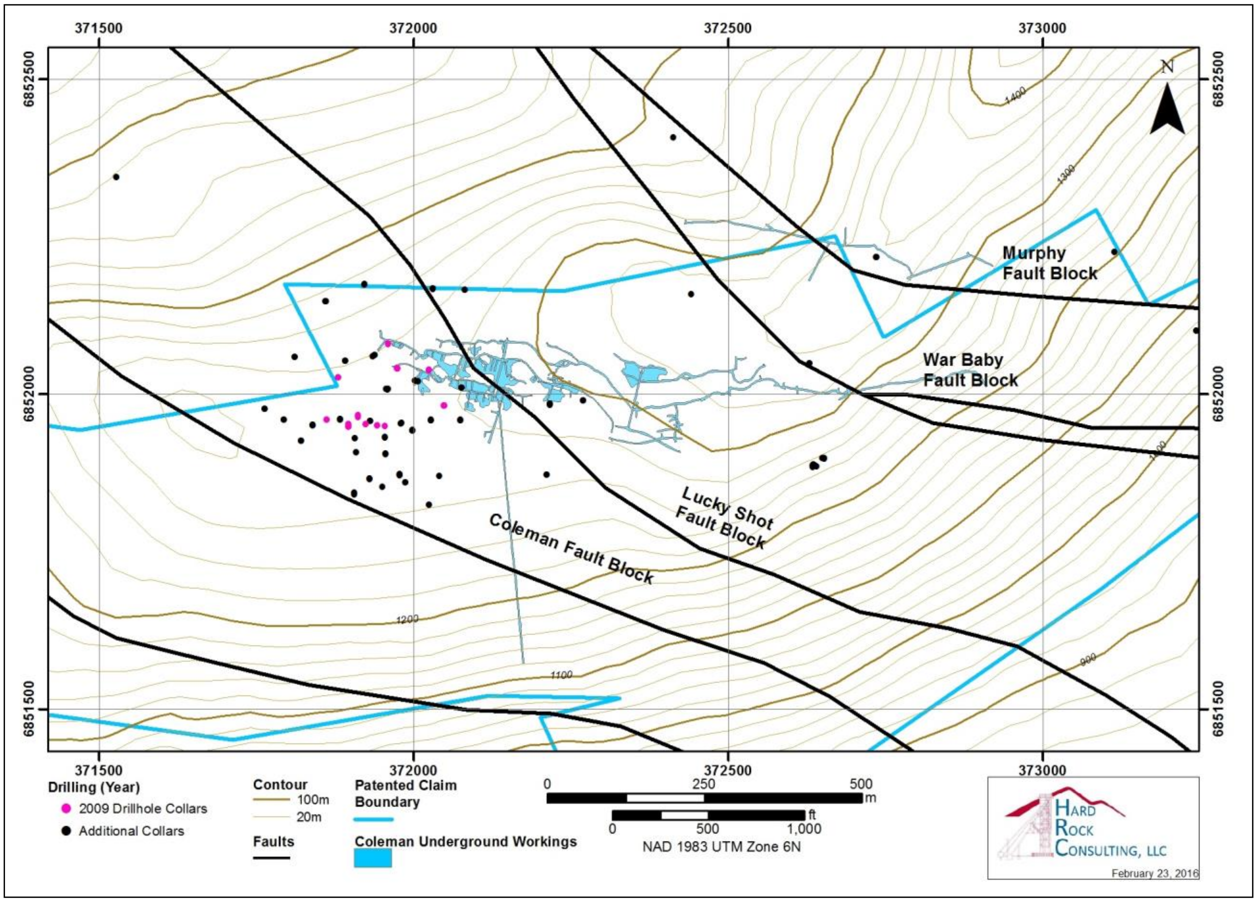


Figure 10-6 FMM 2009 Drilling

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Enserch Sample Preparation, Analysis, and Security

11.1.1 Enserch Core Sampling Procedure

Tonalite and granodiorite country rock were logged but not sampled for assay except where in contact with silica rich vein material or as xenoliths within the vein material. Assayed drill intercepts from the silica rich vein material (shear zone) are listed in Table 11-1.

Table 11-1 Assayed Drill Intercepts from Ensearch Exploration

Drillhole Name	Type	From (m)	To (m)	Length (m)	Gold (g/t)
LSB_1	Underground	196.10	196.77	0.67	8.211
LSB_1	Underground	205.06	205.12	0.06	1.213
LSB_1	Underground	214.70	215.56	0.86	4.977
LSB_1	Underground	229.86	230.29	0.43	1.337
LSB_2	Underground	358.84	360.21	1.37	3.328
LSB_3	Underground	310.74	313.18	2.44	1.182
LSB_4	Underground	222.77	223.50	0.73	7.123
LSB_5	Underground	186.44	187.08	0.64	1.12
LSB_5	Underground	189.43	190.44	1.01	4.386
LSB_6	Underground	253.80	254.17	0.37	1.991
LSB_6	Underground	254.47	254.65	0.18	3.235
LSB_6	Underground	272.75	273.05	0.30	2.83
LSB_7	Underground	217.17	217.54	0.37	3.67
LSA_1	Underground	206.23	207.14	0.91	3.577
LSA_2	Underground	267.94	268.50	0.56	0.747
LSA_3	Underground	177.92	178.38	0.46	3.888
LSA_3	Underground	193.72	193.81	0.09	15.241
LSA_4	Underground	151.59	151.74	0.15	55.986
LSA_4	Underground	155.99	156.60	0.61	5.505
CB1	Surface	No Reported Assay Intercepts			
CB2	Surface	No Reported Assay Intercepts			
CB3	Surface	No Reported Assay Intercepts			
CB4	Surface	No Reported Assay Intercepts			
COLE1	Surface	No Reported Assay Intercepts			
COLE2	Surface	No Reported Assay Intercepts			
COLE3	Surface	No Reported Assay Intercepts			

11.1.2 QA/QC and Assays

Enserch mobilized fire assay equipment to the Lucky Shot site for analysis of core samples as well as rock chip and soil samples from the Willow Creek mining district. Core samples from the 1984 program were assayed on site under the supervision of the company geologists and chemists. Standards and duplicates were included in the assay stream. Enserch did not produce any assay certificates but they did create a table of assay results. Enserch drill data was compiled into a database by FMM in 2007 for use in modeling software (file reference: enserch_ug_gh.str).

11.2 FMM Sample Preparation, Analysis, and Security

The drilling campaigns conducted by FMM were similar from year to year with sample logistics concerning Lab choice, except for 2005 and 2007. In 2005 Alaska Assay Labs prepped the samples and sent the pulps onto BSI-Inspectorate in Sparks, NV, whereas in 2007, Alaska Assay Labs prepped and analyzed the core samples. In 2006, 2008 and 2009, ALS Chemex prepped and analyzed the core samples in their Fairbanks, Alaska, prep lab and their analytical facility in Vancouver, B.C.

From 2005 to 2009, the core was split using an electric powered masonry saw with a diamond tipped blade. Half core splits and rock samples collected by FMM were placed into tied bags, and stored on site under FMM's supervision.

11.3 Sample Preparation

All diamond drill core from FMM’s drilling programs were logged, photographed, and tagged for sampling in the following manner:

Drill core was logged and sampled at the Project site under the supervision of the Project geologist. Altered, veined, or otherwise mineralized core were split with an electric powered masonry saw with a diamond tipped blade; other highly fractured intervals were split by hand or sawed. After splitting, the sample number is written on a poly bag and one half of the split core, along with a sample tag, is placed in the bag. The remaining ½ of the split core is returned to the core box prior to removing the next piece of core to be split. Sample bags are zip-tied shut and the individual bag weights recorded before submittal.

Drill core samples were stored in a locked facility on site. The remaining ½ of the split core stays in the core box. Core boxes remain onsite stored in open air conditions within the private property boundary maintained by Scott Eubanks.

In 2005 samples were delivered by FMM personnel to Alaska Earth Sciences (AES) offices in Anchorage who provided FMM with logistical support during the 2005 field season. AES then arranged for transportation of samples to Alaska Assay Laboratories facilities via Lynden transportation for prepping the samples followed by sending the samples to BSI in Sparks, NV. Alaska Assay Labs were also used for core sample prepping and analysis in 2007 via core sample transportation by Lynden Transportation Services. In 2006, 2008 and 2009, samples were delivered by FMM personnel to ALS Chemex Prep Lab Facility in Fairbanks, Alaska, for core sample preparation and analysis in Vancouver, B.C.

11.3.1 Laboratories

From 2005 to 2009, samples were crushed and/or pulverized using standard chrome steel jaws, rings, and pucks. In most cases, a 1 kg pulverized sample was then analyzed. In 2005, Alaska Assay Laboratories acted as a prep lab for BSI Inspectorate Laboratories based in Sparks, Nevada. Thirty-gram pulp samples were prepared by AAL and sent for analysis to BSI in Nevada. The 30 g pulps were then submitted for fire assay gold using lead collection with an atomic absorption finish and multi-element ICP. In 2007, core samples were prepped and analyzed by AAL in Fairbanks. The pulps were then submitted for fire assay gold using lead collection with a gravimetric finish and multi-element ICP. FMM was satisfied with the Quality Control procedures and results implemented by Alaska Assay Labs.

In 2006, 2008, and 2009, a 1 kg pulverized samples were forwarded to ALS Chemex Laboratories in Vancouver for analysis. The samples were then fire assayed for gold using a 50 g charge and with lead collection and gravimetric finish at ALS. No significant analytical issues were encountered. From 2005 to 2009, multi-elements were analyzed using standard aqua regia digestion with inductively coupled plasma (ICP) techniques. In 2006 to 2007, gold assays above 1,000 ppb were re-assayed using metallic screening techniques. In 2007, FMM submitted eighteen samples from nine drill holes for Metallic Screen analysis with ALS Chemex. The samples contained visible gold. Metallic Screen analysis determines the gold content of the coarse and fine fraction of the sample. The Metallic Screen procedure is as follows:

> 1 kg of the final prepared pulp is passed thru a 100 micron (Tyler 150 mesh) stainless steel screen. Any material remaining on the screen (+100 micron) is dried, weighed, and analyzed by fire assay with gravimetric finish and reported as the Au (+) fraction. The material passing thru the screen (-100 microns) is dried, homogenized and two (2) fifty (50) gram sub-samples are analyzed by fire assay with AAS finish (Au-AA26 and Au-AA26D). The average of these two AAS results are reported as the Au (-) fraction. Both values are then used in calculating the combined gold content ((±) combined).

Fifteen of the eighteen samples (83%) contain greater than 66% of gold in the coarse fraction. These results may indicate that a large proportion of the gold could be recovered by conventional gravity methods (McLeod, 2005, 2006, 2007, and 2008, and Stevens, 2009).

11.4 Quality Assurance/Quality Control Procedures

11.4.1 Blanks

FMM used sample blanks to identify contamination between samples during sample preparation and to evaluate the accuracy of the sampling program. YKPS completed an analysis of the blanks based upon the principle that blank material should not exceed three times the detection limit. A total of 722 check samples were analyzed under the blank sample program during the years 2005-2009. Since analytical methods varied throughout the drilling programs, three detection limits for the various analytical methods were chosen: 0.05, 0.1 and 0.005 g/t Au. Several items were noted by YKPS regarding the blanks as follows:

- FMM did not use "true" blanks during the 2005, 2006 and 2007 drilling campaigns;
- Blanks were made by FMM from pulverized quartz diorite core that was carefully chosen from previous drilling campaigns with the least amount of alteration or mineralization. However, the blanks are not "true" blanks and contained elevated gold values;
- Mislabeling issues were identified, especially in 2005 where standards were inserted accidently instead of blanks.

Utilizing non-commercial blanks can lead to elevated results, as is seen in the analysis completed by YKPS. This is likely due to the blank material being collected from rocks in close proximity to the resource. Utilizing a maximum limit of 0.15 g/t gold to evaluate the blanks submitted by FMM from 2005 – 2009 the total number of failures is reduced to approximately 6%.

Failures that exceed the established criteria for blanks should be immediately reviewed and the samples following the failure should be re-analyzed unless a rational justification for the failure supports the inclusion of the original results in the exploration database.

11.4.2 Standards

With the exception of 2005, FMM randomly inserted commercial standards that were purchased from Rocklabs Ltd of Auckland, New Zealand and CDN Resource Laboratories Ltd of Delta, B.C. Canada into the drill sample stream. FMM used a total of six commercial standards from 2006 to 2009. YKPS completed an analysis of the standards using industry accepted guidelines. The six standards are shown in Table 11-2.

Table 11-2 Standard Reference Material Values

2006 Standards				2007 - 2009 Standards			
ID	Gold Grade (g/t)			ID	Gold Grade (g/t)		
	Lab Grade	Lower Limit	Upper Limit		Lab Grade	Lower Limit	Upper Limit
Standard	8.543	7.689	9.397	CDN-GS-15A	14.83	14.22	15.44
Standard	2.604	2.344	2.864	CDN-GS-5C	4.74	5.02	4.46
Standard	0.583	0.525	0.641	CDN-GS-P5B	0.44	0.40	0.48

During the drilling programs, acceptable reference standards tolerances were set to plus or minus two standard deviations as the recommended acceptable deviation from the mean. YKPS used the values from "Certificates of Analysis" for the commercial standard material to set the boundaries for tolerances. The results of the audit indicated the standards had high failure rate when compared with the commercial mean value. The errors were not significant; however, it was found that there was high frequency of outliers that indicated recording or laboratory errors.

YKPS examined drill logs, core photos, reports, computer files and sample booklets were to check the outliers and to determine proper corrective action. This analysis confirmed that there was a problem with mislabeled samples in the field. There are numerous occurrences where the recorded reference material reflected values or sample weights more consistent with another standard or blank. It was determined that in order to resolve the outlier issue, re-assay of the suspect samples was necessary. FMM initiated a program of re-assaying selected intervals in order to resolve the problem. The results of this program were unavailable at the time of writing of this report. A summary of the errors within the six standards is summarized in Table 11-3 for the years 2005 to 2008.

The nature and results of the majority of these errors indicate a mislabeling of blanks as a standard sample. Table 11-3 shows the analysis of the standards for the years 2005 through 2008.

Table 11-3 Standard Analysis

Standard	Total Standards Analyzed	Number Passing	Number Failing	Percent Failed	Average % of Standard Value	Average % of Standard Value Passing	Average % of Standard Value Failing
G5-P5B	67	44	23	34%	101%	97%	107%
SE-19	22	20	2	9%	100%	100%	95%
SJ-22	116	111	5	4%	97%	98%	80%
GS-5C	82	41	41	50%	93%	97%	88%
SN-26	51	47	4	8%	92%	97%	35%
GS-15A	58	33	25	43%	96%	99%	92%
Total	396	296	100	25%	96%	98%	91%

It has been recommended that any future drilling programs on this property utilize the following:

1. Drilling campaigns need to include higher grade standards more in line with the average grade of the deposit
2. Provide more care in labeling and increased supervisory personnel when applying blanks and standards to the sample stream so that mislabeling is minimized

Table 11-3 indicates that of the 396 total standards analyzed 25% failed the pass/fail criterial set by YKPS. This failure rate is considered to be high, however in review of the standard analysis it appears that standards are performing consistently. It is likely that an analysis using the mean grade and standard deviation calculated from the results of the standards with more than 25 samples will yield better results.

11.4.3 Duplicates

FMM did not utilize any core duplicate analysis as part of their QA/QC program.

11.4.4 Check Assays

FMM submitted approximately 10% of all pulp samples to an umpire lab for analysis. A stringent guideline for a pass or failure of check and original assay pairs was utilized. The results indicated that there may be a minor bias from one laboratory to the next, but that there was in general no intentional or accidental bias in the pulp check assays. In general, a deviation of no more than 10% from the original assay is considered acceptable for a pulp re-assay; however, in the coarse gold system present at Project a larger deviation is considered acceptable. The umpire labs used for the pulp check sample campaign were as follows:

- ALS Chemex
- International Plasma
- Alaska Assay and BSI Inspectorate

11.5 Opinion on Adequacy

HRC concludes that the sample preparation, security and analytical procedures are correct and adequate for the purpose of this Technical Report. The sample methods and density are appropriate and the samples are of sufficient quality to comprise a representative, unbiased database.

12. DATA VERIFICATION

The mineral resource estimate presented in Section 14 of this report relies in part on the following information provided to HRC by GTOR:

- Discussions with GTOR personnel;
- Personal investigation of the Project and field office;
- Exploration drillhole database (dec20_dhd) received February 14, 2015;
- Enserch drillhole database received July 20, 2015;
- A limited audit of exploration work conducted;
- The preliminary "Technical Report On the Project Matanuska-Susitna Borough, Alaska" dated September 30, 2014 and authored by David Linebarger, RM SME,
- The recent "Technical Report, Mineral Resource Estimate for the Willow Creek Project, Matanuska-Susitna Borough, Alaska" prepared for GTOR and Miranda Gold Corp., effective date February 1, 2016; and
- Additional information obtained from historical reports and internal company reports.

12.1 DATABASE AUDIT

FMM's drillhole exploration database and additional Enserch drilling data were combined into a single database for mineral resource estimation. The following tasks were completed as part of HRC's database audit:

- Performed a mechanical audit of the database;
- Validated the geologic information compared to the paper logs; and
- Validated the assay values contained in the exploration database with assay certificates from BSi Inspectorate (2005), ALS Chemex (2006, 2007), and Alaska Assay Labs (2008);

HRC limited the audit to the rock-type, assay, drillhole collar, and survey data contained in the database.

12.1.1 Mechanical Audit

A mechanical audit of the combined database was completed using Leapfrog Geo® software. The database was checked for overlaps, gaps, total drillhole length inconsistencies, non-numeric assay values, and negative numbers. The following list of drill holes were missing information:

- No Assay Data
  - 07-135
  - WB08-08
- No Lithology Data
  - N06-16
  - N06-17

- N06-18
- N06-19
- N06-20
- N06-21
- N06-22
- LSB_1
- LSB_2
- LSB_3
- LSB_4
- LSB_5
- LSB_6
- LSB_7
- LSA_1
- LSA_2
- LSA_3
- LSA_4

A total of 198 drillholes were imported into Leapfrog for validation.

12.1.1.1 Overlaps

A single overlapping interval in the lithology table was identified. Drillhole C06-68 had a duplicate interval from 166.4 to 182.88. The error in the database was resolved by removing the duplicate interval.

12.1.1.2 Gaps, Non-numeric Assay Values, and Negative Numbers

The software reported missing intervals for each of the 5 elements analyzed. The non-positive numbers (-9) were assumed to be non-sampled intervals and were omitted from the dataset. All of the other non-positive values were assumed to be below detection limit values and were set to 0.001 for silver (g/t), iron (%), copper (g/t) and arsenic (g/t). Gold below detection limit values were set to 0.0025 g/t. No non-numeric assays were encountered in the audit. Table12-1 below summarizes the number of intervals imported, the number of missing intervals, the number of non-positive values and the number of valid assays for each element.

Table 12-1 Database Import Summary

Element	Missing	Non-Positive Values	Assay Values
Ag (g/t)	1,023	47	7,288
Au (g/t)	969	33	7,234
Cu (g/t)	1,021	2	7,236
Fe (%)	1,050	2	7,207
As (g/t)	1,021	6	7,236

12.1.2 Survey Data

The collar coordinate elevations were compared to the corresponding elevation from the surface triangulation. The drillhole collar elevations represent similar elevations to the corresponding topography surface and are considered adequate for use in the mineral resource estimation.

The majority of 198 drillholes audited in the database have each been surveyed down-the-hole. A total of 35 drillholes were not surveyed down-the-hole. These drillholes were evaluated on section and found to have similar locations for geologic and grade breaks as compared to the surrounding surveyed drillholes.

12.1.2.1 Table Depth Consistency

The survey, assay, and geology tables maximum sample depth was checked as compared to the maximum depth reported in the collar table for each drillhole. Six downhole survey records exceeded the maximum depth as reported in the collar file. Each recorded was reviewed and corrected as reported in Table 12-2.

Table 12-2 Maximum Depth Corrections

BHID	Depth	Error	Solution	Logic
C06-27	178.32	Exceeds max hole depth	Ignored	Duplicate Survey at 170.69
C06-43	280.43	Exceeds max hole depth	Ignored	Duplicate Survey at 277.37
C06-66	216.43	Exceeds max hole depth	Ignored	Duplicate Survey at 213.36
C06-89	333.85	Exceeds max hole depth	Ignored	Duplicate Survey at 324.31
C06-90	362.72	Exceeds max hole depth	Extended Hole	Azimuth Changed at 362.72
WB08-03	142.73	Exceeds max hole depth	Extended Hole	Azimuth Changed at 175.11

12.2 Certificates

HRC received original assay certificates in csv format for the drilling conducted from 2005 to 2009 in the current database. A random manual check of 10% of the database against the original certificates was conducted. The error rate within the database is considered to be less than 1% based on the number of samples spot checked.

12.3 Adequacy of Data

HRC has reviewed the check assay programs and believes the programs provide adequate confidence in the data. Samples that are associated with failures and the samples associated with erroneous blank samples have been reviewed. Errors have been justified as labeling errors or are infrequent. All of the samples associated with erroneous QA/QC results are reviewed prior to inclusion in the database.

The majority of drill cores and cuttings from drilling have been photographed. Drill logs have been digitally entered into exploration database organized and maintained in a Maptek Vulcan project. The split core and cutting trays are stored at the Project and are susceptible to the elements. HRC recommends that the drill core be moved to a better storage facility to ensure that the samples are not damaged or lost due to inclement conditions.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Mineralized material containing gold in the Willow Creek district is essentially free milling and therefore a major portion of the gold is expected to be recoverable by gravity concentration methods. Historical gold production in the district (see W.M. Stoll 1974 and USGS Field Reports) covering a number of mines operating on gold bearing materials of similar rock type (lithology) in the same vein system as GTOR's Project, typically recovered upwards to and in excess of 90% of the identified contained gold. The historical Lucky Shot mill and mine located at the Project site (1932-1943) reportedly mined ore containing in excess of 34.29 g/t (1 opt) and processed it in a crush, grind, gravity separation, flotation, and cyanidation circuit. It is reported that the operation recovered 85% of the contained gold in the simple gravity separation circuit ahead of the flotation and cyanide circuits.

The preponderance of information relative to the Project suggests that a reasonable expectation for gravity recovery of the contained gold is 91.8%. There is strong evidence to a support higher gold recovery that should be the subject of additional study.

This section will briefly summarize the relevant past metallurgical test work and historic production results from the following sources:

- 2016 Physical Beneficiation Experiments on Samples from the Willow Creek, Alaska, Gold Project, Revision 1 – by Hazen Research for Gold Torrent Inc.
- 2014 Modified Acid-Base Accounting and static Acid Rock Drainage tests by McClelland Laboratories for Gold Torrent Inc.
- 2008 Enhanced Gravity Recovery Gold Testing Conducted by Knelson Research and Technology Centre for Full Metals Minerals (FMM)
- 1981 Metallurgical Test Work by Hazen Research, for Enserch Exploration
- 1974 Report - Independence Gold Properties, Alaska, Past Operations, Future Potential by W.M. Stoll
- 1915 Willow Creek District Report by the United States Geologic Service Bulletin 607

Text taken directly from the reports as originally published, except for occasional edits for consistency, are shown in quotation marks in the following sections.

13.1 2016 Physical Beneficiation Experiments on Samples from the Willow Creek Alaska Gold Project by Hazen Research for Gold Torrent Inc.

Gold Torrent, Inc., contracted with Hazen Research in November 2015, to initiate gravity concentration studies on various samples they provided which originated from the Willow Creek Alaska Gold Project. The work included heavy-liquid gravity separation on two relatively small samples (a drill core section and a composite of various rock fragments). The most significant work was conducted on a much larger bulk sample comprising the contents of five 55 gal drums of rock hand collected from the Project's existing historic mine dumps and subsequently visually sorted. The June 2016 Hazen report is included in its entirety as Appendix D to this report.

The test work was initiated by Gold Torrent to further understanding of the ore's response to an evolved all gravity gold recovery process incorporating staged crushing to avoid creation of excessive fines, precise desliming and size-classification, and progressive and repeated gravity concentration to produce concentrates suitable for refining to gold dore. This process avoids the permitting and environmental issues that arise when flotation and cyanidation processes are employed.

The preliminary studies confirmed the amenability of liberated crushed and ground Willow Creek ore to gravity concentration. The initial heavy-liquid separations conducted on a small core section and on a small chip sample composite, both ground to P80 105 microns, recovered 71.5% and 95.4%, respectively, of the available gold in the plus 25 micron fraction. The sink concentrates were very high grade and suitable for refining at 1,670 and 23,600 g/t, respectively. The variability in results between the two materials can be expected when gold occurs in free form.

The focus of the Hazen work on the much larger bulk sample was beneficiation testing conducted on shaking tables that were considered surrogate substitutes for commercial scale spiral concentrators. The work produced very good beneficiation results on ore that was carefully crushed, deslimed, and size-classified by screening. As expected, the contained and liberated gold was readily recovered to concentrate.

Although somewhat limited in scope of testing, the shaking tables produced the following results:

- 48 x 65 mesh fraction, rougher-scavenger recovered 91.7 % of the gold in 62.2 wt%.
- 65 X 100 mesh fraction, rougher-scavenger recovered 97.0 % of the gold in 55.3 wt%.
- Minus 325 mesh fraction, rougher-scavenger recovered 73.1 % of the gold in 4.90 wt% (for the minus 325 mesh fraction, if more weight percent were taken to concentrate, it is expected that the gold recovery would be significantly higher).

The high gold recovery results confirmed that the ore is amenable to gravity concentration and suggests that the gold is well liberated at 65 mesh, although significant recovery into high grade concentrates can be obtained at somewhat coarser sizes. Gravity concentration on 4 x 8 mesh (coarse) ore did not produce free gold particles, although at times there may be liberated free gold at that size; therefore, the Project process flowsheet is designed to accommodate coarse ore free gold recovery.

In addition to the high gold recovery demonstrated by gravity concentration, other observations during the Hazen work led to further refinements of the process design.

- Relatively coarse liberation of the contained gold suggested that fine crushing could be used for preparation of the feed materials in advance of gravity concentration. This eliminates the need for grinding except possibly for processing minor middling streams (less than 1 tonne per day) produced in the gravity circuit.
- The observed rapid settling of size-classified material that would be processed by spirals and tables in the plant indicates that simple decantation of excess water will aid the operators in providing the correct feed pulp density to subsequent unit operations.

- Similar observations of relatively quick settling of the minus 325 mesh material is supportive of moderate to small sand and clay ponds.
- Observations made during the gravity separation testwork showed an easily discernable visual separation of the concentrate from barren tailings/quartz. This difference supports visual operator control of the separation. Assaying will be required to determine precise metalurgical balance data, but basic "ore versus waste" selection will not require assaying.

13.2 2014 Modified Acid-Base Accounting and Static Acid Rock Drainage Tests by McClelland Laboratories for Gold Torrent Inc.

In 2014, GTOR conducted tests on a sample of gold bearing material collected from FMM's Coleman-area drill core relating to the environmental impact of tailings material created by a proposed gravity-only gold recovery plant. The key flow sheet parameters of the tests were that the proposed gold recovery plant in question would use no chemicals, and would use staged screening and crushing followed by gravity concentration. GTOR provided McClelland Laboratories samples of low-grade, gold bearing drill core to obtain chemical analyses data by ICP for tailings produced in a proposed gravity mill. A summary of the work performed, as reported by McClelland is as follows:

"Mod ABA static ARD potential tests were conducted on Sample A (-35M) and Sample B (-35M) by SVL Analytical, Inc. to assess potential of the solids to generate or neutralize acid in natural weathering and oxidizing environment. A TCLP extraction test was conducted on Sample A (-35M) and Sample B (-35M) by SVL Analytical, Inc. to categorize the solids potentially as non-hazardous or hazardous material. Silicon and mercury analyses were conducted on Sample A (pulp) and Sample B (pulp) by ALS Chemex. Metallic Screen assays for gold and silver were conducted on Sample A (-35M) and Sample B (-35M). In addition, Mercury (by cold vapor atomic absorption spectroscopy) and silicon (by sodium peroxide fusion-ICP-AES) results were produced by ALS Chemex. "

The conclusions of the report were that the two samples tested do not demonstrate potential to generate acid, and would not be categorized as hazardous wastes using US EPA criteria.

13.3 2008 Enhanced Gravity Recovery Gold Testing Conducted by Knelson Research and Technology Centre for FMM

FMM designed a testing program to assess amenability of ores from the Project to gravity recovery methods; results of the testing program conducted by Knelson Research and Technology Centre was reported in 2008. "A total of 114 kg of diamond core sample rejects were shipped to Knelson labs in Langley BC where Enhanced Gravity Recovery Gold (EGRG) tests were performed on 3 composite samples. The EGRG test determines the gold available in the sample recoverable by gravity methods and the size of grind necessary for liberation. This test employs progressive size reduction with recovery of gold as it is liberated while minimizing over-grinding and smearing gold particles, thus the process generates a size-by-size characterization of the products and can be used in a proprietary mathematical model for plant simulation. A 40-50kg (100% passing 1.7mm) representative sample of open circuit feed or drill core is required; larger sample weights are used in the EGRG test to take some of the variability out of the results. In order to determine if different areas of the deposit exhibited different recovery characteristics, each composite was designed to represent a different portion of the resource." The following results were reported. Note that the gold head grades for all the composite samples tested were lower than the current 18.3 g/t (0.534 opt) average grade of the measured and indicated mineral resources for Gold Torrent's Project.

Table 13-1 Summary of Gravity Recovery for Gold

Sample	Gravity Recovery %	Head Grade g/t Au
Composite 1	68.2	4.7
Composite 2	68.5	4.7
Composite 3	78.3	7.9

The Knelson report, briefly summarized in Table 13-1 above, showed gold recoveries of 68-78% on ore composite samples at nominal grinds of P_{80} 62 to 96 micrometers. There appeared to be no significant difference in recovery based on the location of the sample within the resource. However, it appears that the higher feed grade of Composite 3 may have had an influence on the higher gold recovery reported for Composite 3.

13.4 1981 Metallurgical Test Work by Hazen Research for Enserch Exploration

"This test work was completed on two samples of gold bearing material, one designated high grade (1.31 opt Au (44.9 g/t)) and one designated low-grade (0.33 opt (11.3 g/t)). Mineralogical examination showed that the gold is present as native gold varying in size from 10 to 500 microns. The purpose of the test work was to develop a process flow sheet for gold recovery as a high-grade concentrate. The test work resulted in a process flow sheet where gold is recovered from ore as separate gravity and flotation concentrates." It should be noted that liberated native gold particles greater than 500 microns in size have been observed in drill core and rock samples by GTOR.

The gravity portion of the process flow sheet was reviewed for its stand-alone results. The Hazen test work on the gravity portion of the flow sheet from Table 5 in the report showed 80% recovery on a rougher table test on minus 20-mesh material. After the first stage shaking table separation was completed the concentrate material from the first stage was then re-tabled which increased the gold concentrate grade from 730.3 g/t (21.3 opt) to 2,914.3 g/t (85.0 opt).

13.5 1974 Report - Independence Gold Properties, Alaska, Past Operations, Future Potential by W.M. Stoll

This report was written for the nearby Independence Mine and Mill that is one of the nearest neighbors to GTOR's Project that mined on a known extension of the same regionally hosted mezothermal vein system. "The Independence was a mine that produced gold from the turn of the last century until 1943 when the mine was closed by the War Production Board. The gold-quartz veins within the Independence properties outcrop between 1,158 m and 1,463 m (3,800 and 4,800-foot) elevations over a vertical range of about 305 m (1,000 feet). Veins outcropping on the west end of the property at the head of Willow Creek also fall within this elevation intercept. Productive veins in the district have all yielded essentially free-milling gold quartz ore containing no metals other than gold in commercial concentrations. Following the discovery of a high-grade outcrop on the Skyscraper vein on Skyscraper Mountain in 1906, five profitable mining operations and several minor operations were undertaken over the ensuing 40 years. Four miles west of Independence, the Lucky Shot – War Baby mine was discovered in 1918 and was operated profitably by Willow Creek Mines from 1921 to 1930."

Of importance is an analysis of the historic production records included within this report. The "Independence mill operated to capacity for 24 hours/day for 95% of available time from 1938 to 1943, regularly achieving a recovery of between 97% and 98% of gold contained in the mill feed. In May 1941, a hand sorting plant was added to the mill, operating at a sorting rate of between 20% and 30%, with waste rejects averaging about 0.7 g/t (0.02 opt) with consequent upgrading of mill heads. Treatment of the vein material from Independence properties was efficient and simple. Two 42" x 60" Marcy ball mills each having a capacity of about 35 tpd in closed circuit with Dorr rake classifiers. On average 86% of the mill input gold content was recovered on simple gravity-flow amalgamation tables." The report contains a table (Table 13-2) showing historic gold production from 1936 through 1943. For the gravity-only tabled concentrates that exhibit a potential for 86% average recovery before amalgamation the material resulted in the following percentage recoveries when followed by amalgamation. From the table line "Oz. recovered as free gold":

Table 13-2 Summary of Historic Mill Gravity Recovery for Gold

Year	1938	1939	1940	1941	1942
% Au Recovery	89	88	88	87	86

13.6 1915 Willow Creek District Report by the United States Geologic Service Bulletin 607

This report has extensive writing on the general geology, economic geology, ore deposits, history, and historic producing mines of the Willow Creek District, Alaska. Of particular interest is the information provided under the heading Economic Geology, Mining Properties, Lucky Shot. This section refers to the historic mine and mill operation that was once located at GTOR's current Project. The gold production came from the War Baby mine that is a small part of the entire vein system at the site. The site mineral resources come from a single vein group that has been dissected into four areas due to three deep seated, cross cutting, near vertical faults. Moving from east to west along strike of the mezothermal vein these faults have created a stair step effect where the uppermost Coleman vein area is about 100 m (300 feet) above the adjacent Lucky Shot vein area which in turn is about 100 m (300 feet) above the next adjacent War Baby vein area which is about 300 m (900 feet) above the next adjacent Murphy vein area. The extension of this same mezothermal vein system exists throughout the immediate vicinity.

Of importance is the information on the milling practice used to achieve gold production from the Lucky Shot project. The mill was constructed in 1931 with a capacity of 35 tons per day. Mill feed gold grade averaged 83 g/t (2.42 opt). The Colorado School of Mines designed the mill flow sheet. While the entire flow sheet included gravity concentration followed by mercury amalgamation and cyanidation, of interest is the initial gravity portion of the mill flow sheet. This mill employed a jaw crusher and Marcy ball mill to create minus 85-mesh material that was then processed across a Gibson table. The total gold recovery from the Gibson table alone was reported as "about 85% of the gold is obtained in the top cut on the Gibson table. The top-cut concentrates are collected in buckets, and the gold is amalgamated once a day in a batea." The report confirms that gold recoveries of near 85% were achieved prior to any other further treatment such as mercury amalgamation and cyanidation.

13.7 Summary

The most current 2016 gravity concentration studies as well as results of historic metallurgical test work and records of past production for gold recovery support a gravity-only flow sheet with gold recovery in excess of 90% while producing non-acid tailings. Regarding the limited 2008 metallurgical test work performed by Knelson Laboratories it is speculated that adding a de-sliming step prior to gravity concentration to remove clay particles should allow for better separation of heavier particles from lighter ones by eliminating the binding effect clays have on particles and improving pulp flow rheology. It is also speculated that classifying the gravity concentration feed into discrete size-based streams will improve gold concentration performance.

It should also be recognized that the 2008 test work was performed on samples with head grade values less than one-third the expected grade to be delivered to the planned gold recovery plant at Willow Creek. There may be a correlation of increased gold recovery to increased head grade.

13.8 Recommendations

Significant historical reports and more recent testwork strongly support the conclusion that the mineralized material with contained gold at the Willow Creek Project will respond favorably to beneficiation by careful liberation and gravity concentration. Such a process minimizes the generation of slimes and presents size-classified material to the gravity concentration unit operations, commercially expected to be spiral concentrators and table concentrators. Gold recovery in excess of 90% while producing non-acid tailings can be expected. Further testing in support of a feasibility study and eventual engineering design is recommended as follows:

- Confirm comminution energy requirements via conventional Bond testing
- Determine Bond abrasion index for engineering cost estimation
- Quantify liquid/solid separation, i.e., settling tests on various process streams
- Establish performance of commercial spirals
- Establish additional liberation information from gravity size/recovery testing
- Establish gravity recovery as function of feed grade (both upwards and downwards)
- Characterize gravity tailings
- Determine if further classification of minus 325 mesh ore will increase gold recovery
- Evaluate enhanced centrifugal gravity concentration of ore fines and tailings.

14. MINERAL RESOURCE ESTIMATES

Resource geologist Zachary J. Black, SME-RM, of HRC is responsible for the mineral resource estimate presented here. Mr. Black is a Qualified Person as defined by NI 43-101, and is independent of GTOR and Miranda. HRC estimated the mineral resource for the Project based on drillhole data constrained by geologic vein boundaries with an Inverse Distance Weighted ("ID") algorithm. Datamine Studio RM® V1.0.73.0 ("Datamine") software was used to complete the resource estimate in conjunction with Leapfrog Geo® V.3.0.0 ("Leapfrog"), which was used to produce the geologic model. The metals of interest at Willow Creek are gold and silver.

The mineral resources reported here are classified as Measured, Indicated and Inferred in accordance with standards defined by Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards - For Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014. Classification of the resources reflects the relative confidence of the grade estimates.

14.1 Density

This section is taken directly from the technical report on the Project dated September 30, 2014 (Linebarger, 2014).

There are a total of 44 sample intervals in the drillhole database that have bulk density determinations taken during the 2006 drilling campaign. These were conducted by the weight in air verses water method. In the authors' opinion the lack of data is not critical. This is due primarily to the evenly spaced distribution of the density samples over the area that comprises the Coleman zone. The majority of the density samples are included in the more densely drilled zones. Additional bulk density determinations are pending. A proper bulk density program would be composed of samples taken from each drillhole, and in this case one sample from the hanging wall, vein and footwall.

The average value of the quartz in the 44 available samples is 2.65 t/m^3 and the average value of the tonalite (footwall and hanging wall) is 2.68 t/ m^3. Three samples of combined quartz and tonalite had an average value of 2.73 t/ m^3. Values for the quartz range from a minimum of 2.60 t/ m^3 to a maximum of 2.71 t/ m^3.

The mineral resources contained herein are assumed to be contained entirely within the quartz veins and have been assigned a density of 2.65 t/ m^3.

14.2 Methodology

The Project is defined by veins within the Coleman, Lucky Shot, War Baby, and Murphy fault blocks (Figure 14-1). The mineral resource estimate is comprised of 10 veins. Eight veins from the Coleman area and 2 veins from the Lucky Shot area. The Murphy and War Baby areas are excluded from the mineral resource estimate due to insufficient data. The mineral resources have been estimated using 3-dimensional ("3D") block model. Table 14-1 summarizes the veins by area.

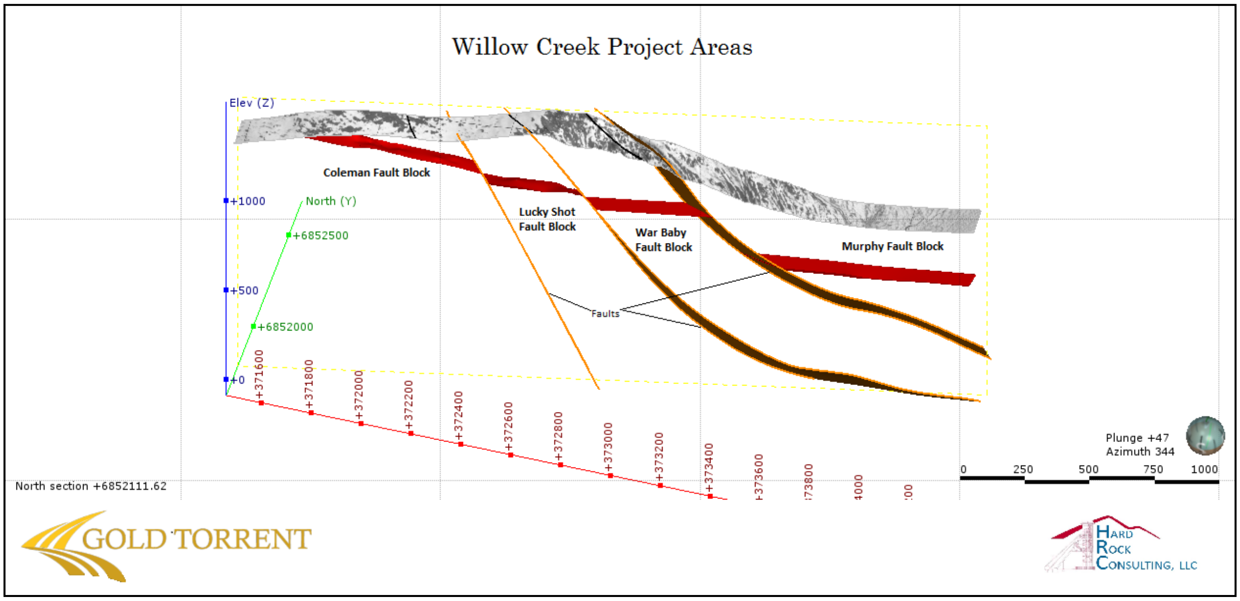


Figure 14-1 Geologic Concept showing Project Areas

Table 14-1 Summary of Estimated Veins by Area

Fault Block	Vein Name
Coleman	Vein 1
	Vein 2
	Vein 3
	Vein 4
	Vein 5
	Vein 7
	Vein 8
Lucky Shot	Footwall Vein
	Hangingwall Vein

14.3 3-Dimensional Block Model Method

14.3.1 Geologic Model

HRC constructed the vein model using Leapfrog Geo Version 3.3.0 using a linear interpolation methodology and sample intervals. East/West oriented cross-sections were used to select intervals from each drillhole representing the vein material. Points representing the hanging wall and footwall contacts were extracted to interpolate hanging wall and footwall surfaces. These surfaces were used to delineate each vein solid (Figure 14-2). The surfaces were evaluated in 3-dimensions to ensure that both the down dip and along strike continuity was maintained throughout the model. Vein volumes in the area of historical production were removed by clipping the solid against an area mapped as having been exploited in the historical mine workings.

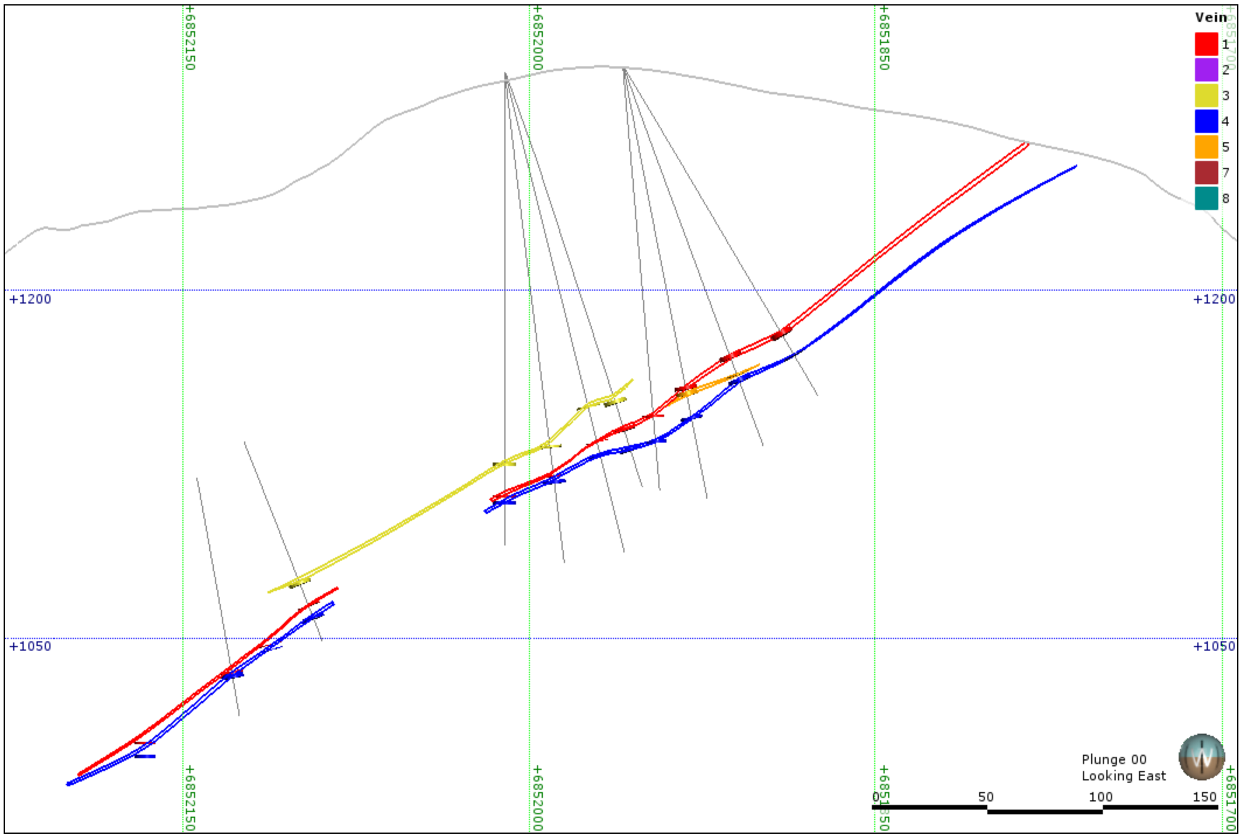


Figure 14-2 Vein Model East Section 372070 looking West

14.3.2 Block Model

The 3D geologic solids were converted to block models using Datamine. Block model prototypes were created for each of the fault blocks. The model prototypes are rotated along strike and down dip and encompass the entire vein. A block size of 1.5m x 1.5m in the strike and dip directions was established to match the mining method being planned by GTOR. The blocks in the z direction were sub-blocked to the vein thickness. A summary of the block model parameters is shown in Table 14-2. The volume, tonnage, and average statistics for sample length, silver, and gold are presented in Table 14-3.

Table 14-2 Willow Creek Block Model Parameters

Area	Origin			Rotation			Block Size			Number of Blocks			Maximum Extent		
	X	Y	Z	Z	X	Y	X	Y	Z	X	Y	Z	X	Y	Z
Coleman	371,610.0	6,851,718.6	1,221.3	0	29	0	1.5	1.5	100	495	360	1	372,352.5	6,852,258.6	1,321.3
Lucky Shot	371,995.0	6,851,667.3	1,060.2	0	26	0	1.5	1.5	200	640	620	1	372,955.0	6,852,597.3	1,260.2

Table 14-3 Vein Model Sample Statistics

Vein	Volume	Tonnage	Samples	Length	Length	Average		
	(m^3)	(tonne)	(n)	(m)	(%)	Interval Length	Au (g/t)	Ag (g/t)
Coleman Area								
Vein 1	153,610	407,067	178	121.69	1.59	0.877	14.026	1.336
Vein 2	5,296	14,034	29	17.09	0.22	0.687	39.292	2.437
Vein 3	37,769	100,088	47	43.97	0.58	1.199	2.147	0.312
Vein 4	120,800	320,120	175	121.57	1.59	0.899	11.788	1.112
Vein 5	7,509	19,899	17	12.57	0.16	0.964	5.74	0.525
Vein 7	7,382	19,561	23	15.78	0.21	0.845	5.033	0.543
Vein 8	50,467	133,738	21	14.67	0.19	0.987	3.834	0.611
Lucky Shot Area								
Footwall Vein	263,720	698,858	23	14.55	0.19	0.875	2.449	0.144
Hangingwall Vein	368,420	976,313	21	11.72	0.15	0.736	6.309	0.307

14.4 Compositing

The assays intervals used to define the hanging wall and footwall intercepts within each vein were composited into a single intercept. The true thickness was calculated using a vein dip of 28 degrees and a dip direction of N7E for the Coleman area. A vein dip of 26 degrees and a north dip direction was used to calculate composite true thickness for the Lucky Shot area. Composite true thickness statistics as well as descriptive statistics summarizing gold and silver are presented in Tables 14-4, 14-5, and 14-6.

Table 14-4 Composite True Thickness Statistics by Vein

Vein	Dip°	Dip Direction°	Minimum (m)	Maximum (m)	Mean (m)	Std. Dev.
Coleman Area						
Vein 1	28	7	0.32	3.65	1.03	0.65
Vein 2	28	7	0.36	1.85	0.83	0.48
Vein 3	28	7	0.38	3.45	1.04	0.56
Vein 4	28	7	0.33	4.00	1.01	0.62
Vein 5	28	7	0.31	1.49	0.81	0.39
Vein 7	28	7	0.33	2.00	0.82	0.42
Vein 8	28	7	0.50	3.00	1.62	0.82
Lucky Shot Area						
Footwall Vein	26	0	0.13	1.99	0.75	0.48
Hangingwall Vein	26	0	0.07	1.37	0.62	0.36

Table 14-5 Gold Summary Statistics within Veins

Vein	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area						
Vein 1	112	0.01	1206.3	19.62	114.23	5.82
Vein 2	19	0.00	236.8	31.68	56.38	1.78
Vein 3	40	0.00	27.7	2.75	4.88	1.78
Vein 4	113	0.00	358.0	11.67	37.99	3.26
Vein 5	15	0.16	154.5	11.16	38.32	3.44
Vein 7	18	0.45	71.6	6.94	15.87	2.29
Vein 8	9	0.03	11.7	4.50	3.90	0.87
Lucky Shot Area						
Footwall Vein	18	0.00	55.99	5.07	12.57	2.48
Hangingwall Vein	18	0.00	77.20	7.44	17.29	2.32

Table 14-6 Silver Summary Statistics within Veins

Vein	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area						
Vein 1	112	0.01	68.9	1.95	8.02	4.12
Vein 2	19	0.10	11.4	1.85	3.01	1.63
Vein 3	40	0.00	1.3	0.31	0.28	0.91
Vein 4	113	0.00	20.0	1.09	2.56	2.34
Vein 5	15	0.01	9.6	0.89	2.34	2.64
Vein 7	18	0.10	7.8	0.75	1.72	2.30
Vein 8	9	0.10	1.1	0.61	0.33	0.53
Lucky Shot Area						
Footwall Vein	18	0.00	0.30	0.06	0.09	1.46
Hangingwall Vein	18	0.00	0.85	0.09	0.20	2.19

14.5 Capping

Grade capping is the practice for replacing any statistical outliers with a maximum value from the assumed sampled distribution. This is done statistically to better understand the true mean of the sample population. The estimation of highly skewed grade distribution can be sensitive to the presence of even a few extreme values.

HRC utilized a log scale cumulative Frequency Plot ("CFP") of the assay data for both gold and silver to identify the presence of statistical outliers (Figures 14-3 and 14-4, respectively). Capping for gold and silver within each vein was determined from these plots. The final dataset for grade estimate in the block model consists of composites capped as presented in Table 14-7.

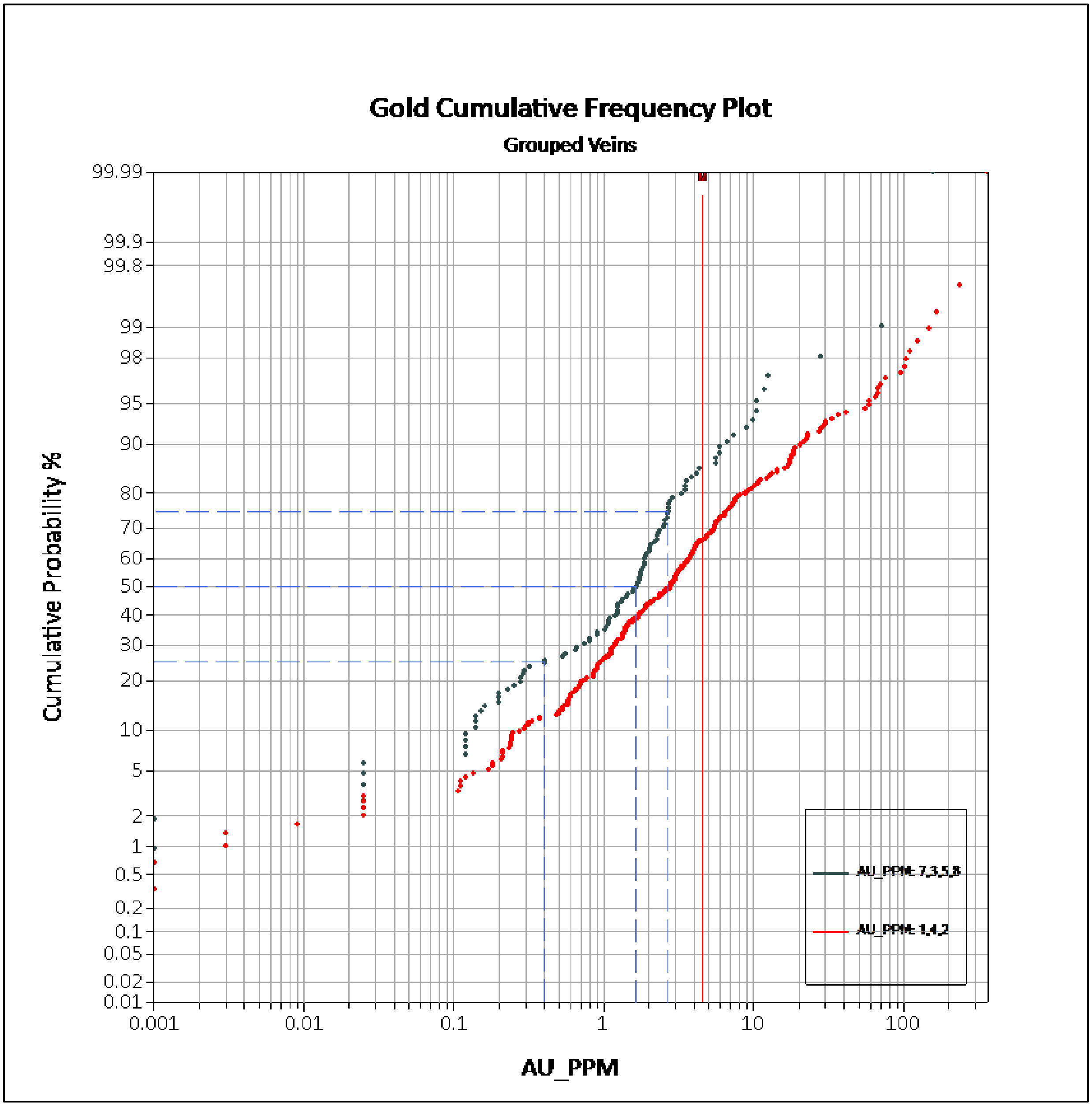


Figure 14-3 Gold Cumulative Frequency Plot

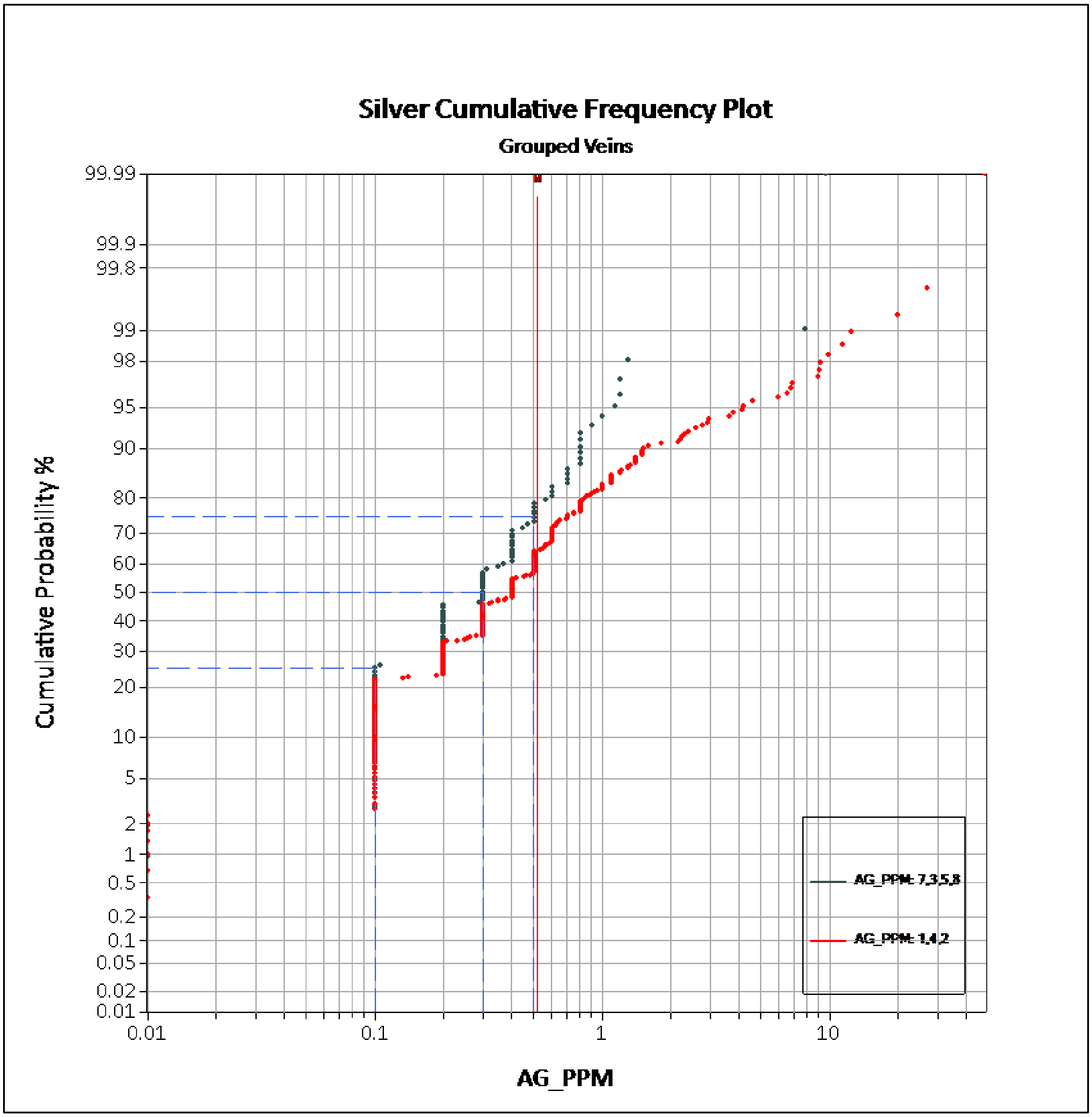


Figure 14-4 Silver Cumulative Frequency Plot

Table 14-7 Capping Values

Vein	Gold (g/t)	Silver (g/t)
Coleman Area		
Vein 1	160	40
Vein 2	160	40
Vein 3	30	2
Vein 4	160	40
Vein 5	30	2
Vein 7	30	2
Vein 8	30	2
Lucky Shot Area		
Footwall Vein	50	20
Hangingwall Vein	60	20

The descriptive statistics for the capped gold and silver composited data are presented in Tables 14-8 and 14-9, respectively.

Table 14-8 Capped Gold Summary Statistics within Veins

Vein	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area						
Vein 1	112	0.01	160.0	10.28	23.76	2.31
Vein 2	19	0.00	160.0	27.64	42.59	1.54
Vein 3	40	0.00	27.7	2.75	4.88	1.78
Vein 4	113	0.00	160.0	9.92	23.95	2.41
Vein 5	15	0.16	30.0	2.86	7.31	2.56
Vein 7	18	0.45	30.0	4.63	6.63	1.43
Vein 8	9	0.03	11.7	4.50	3.90	0.87
Lucky Shot Area						
Footwall Vein	18	0.00	50.00	4.74	11.22	2.37
Hangingwall Vein	18	0.00	60.00	6.49	13.46	2.07

Table 14-9 Capped Silver Summary Statistics within Veins

Vein	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area						
Vein 1	112	0.01	40.0	1.62	5.55	3.44
Vein 2	19	0.10	11.4	1.85	3.01	1.63
Vein 3	40	0.00	1.3	0.31	0.28	0.91
Vein 4	113	0.00	20.0	1.09	2.56	2.34
Vein 5	15	0.01	2.0	0.38	0.48	1.26
Vein 7	18	0.10	2.0	0.43	0.44	1.03
Vein 8	9	0.10	1.1	0.61	0.33	0.53
Lucky Shot Area						
Footwall Vein	18	0.00	0.30	0.06	0.09	1.46
Hangingwall Vein	18	0.00	0.85	0.09	0.20	2.19

14.6 Variography

A variography analysis was completed to establish spatial variability of gold and silver values in the deposit. Variography establishes the appropriate contribution that any specific composite should have when estimating a block volume value within a model. This is performed by comparing the orientation and distance used in the estimation to the variability of other samples of similar relative direction and distance.

Variography was analyzed using Snowden Supervisor Version 8.4. The continuity is established by analyzing variogram contour fans, in the horizontal, across-strike, and dip planes to determine the direction of maximum continuity within each plane. The subsequent variograms defining the maximum continuity were modeled with a spherical variogram (Figure 14-5). Variogram models representing Vein 1 were used to define the search volume used in the estimation of all veins. Tables 14-10 and 14-11 summarize the variogram parameters used for the analysis for gold and silver, respectively.

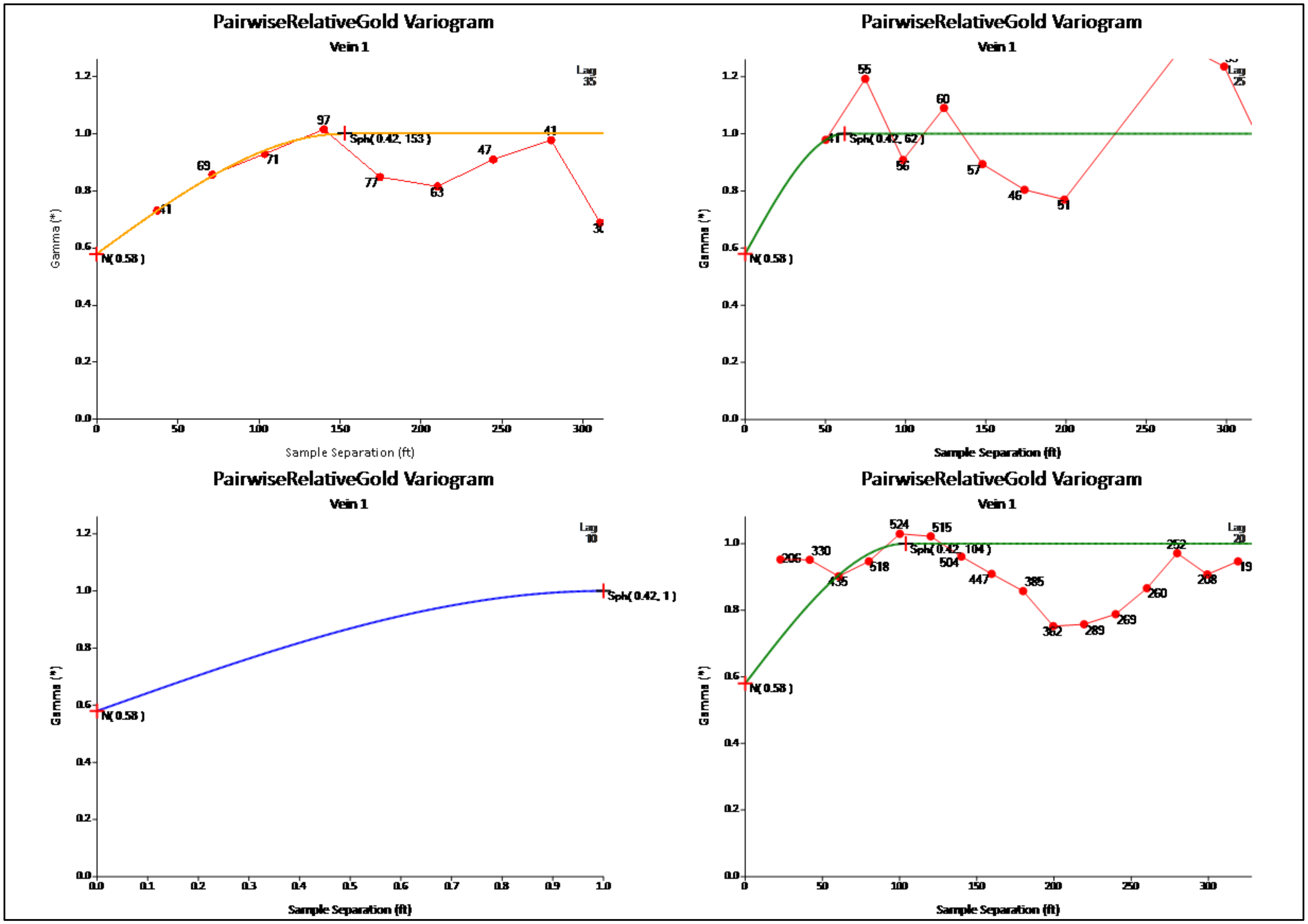


Figure 14-5 Vein 1 Gold Variogram

Table 14-10 Summary of Gold Variogram Parameters

Nugget (C0)		C1	
0.58		0.42	
Axis	**Range (meters)**	**Azimuth**	**Dip**
Z	153	15	-25
X'	62	285	0
Z'*	3	15	65

* Set to approximate vein thickness

Table 14-11 Summary of Silver Variogram Parameters

Nugget (C0)		C1	
0.27		0.73	
Axis	**Range (meters)**	**Azimuth**	**Dip**
Z	139	15	-25
X'	101	285	0
Z'*	3	15	65

* Set to approximate vein thickness

14.7 Estimation Parameters

Comparisons were made with ordinary kriging ("OK") and inverse distance-squared (ID2.5) methods. The ID2.5 method was selected for reporting due to better fit with drillhole data throughout the model. The search ellipse parameters used for estimation are shown in Table 14-12 below. These parameters feature a major axis orientation striking 80 degrees and dipping 30 degrees to the North.

Gold grades were estimated in each vein by using incremental search ellipses to provide an estimation of the gold and silver grade within every block inside the vein solids. Grades outside of the defined veins were not estimated. The search volumes were defined based on the practitioner's experience with similar resource estimates and the maximum continuity defined by the variogram from Vein 1.

A true thickness composite length weighted Inverse Distance to the power of 2.5 was used to estimate grade for all domains. Estimation parameters for each of the domains are presented in Table 14-12.

Table 14-12 Estimation Parameters

Coleman Area						
Metal	**Gold**			**Silver**		
No. of composites	**1st Pass**	**2nd Pass**	**3rd Pass**	**1st Pass**	**2nd Pass**	**3rd Pass**
Minimum	1	2	2	1	2	2
Maximum	3	5	6	3	5	6
Search Ellipsoid Distance						
Primary	15	75	150	15	75	150
Secondary	6	30	60	10	50	100
Tertiary	10	50	100	10	50	100
Lucky Shot Area						
Metal	**Gold**			**Silver**		
No. of composites	**1st Pass**	**2nd Pass**	**3rd Pass**	**1st Pass**	**2nd Pass**	**3rd Pass**
Minimum	1	2	2	1	2	2
Maximum	3	5	6	3	5	6
Search Ellipsoid Distance						
Primary	15	45	150	15	75	150
Secondary	6	18	60	10	50	100
Tertiary	10	30	100	10	50	100

14.8 Model Validation

The Willow Creek 3D models were validated by the following methods:

- Comparison of the global descriptive statistics from the Inverse Distance Weighting ("ID"), Ordinary Krige ("OK"), Nearest Neighbor ("NN"), and composite data;
- Comparison of CFPs from ID, OK, NN, and composite data; and
- Swath plots.

14.8.1 Comparison with Ordinary Krige and Nearest Neighbor Models

Ordinary Krige ("OK") and Nearest Neighbor ("NN") models were run to serve as comparison with the estimated results from the ID2.5 method. Descriptive statistics for the ID2.5 method along with those for the OK, NN, and drillhole composites for gold and silver are shown in Tables 14-13 and 14-14, respectively.

Table 14-13 Gold Model Descriptive Statistical Comparison

Vein	Model	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area							
Vein 1	Composites	112	0.01	160.0	10.28	23.76	2.31
	$ID^{2.5}$	59,245	0.01	160.0	4.82	11.44	2.37
	OK	59,245	0.01	160.0	5.27	11.64	2.21
	NN	59,245	0.01	160.0	4.71	13.98	2.97
Vein 2	Composites	19	0.00	160.0	27.64	42.59	1.54
	$ID^{2.5}$	3,564	0.00	160.0	24.03	31.36	1.30
	OK	3,564	0.00	160.0	24.94	29.65	1.19
	NN	3,564	0.00	160.0	24.34	35.54	1.46
Vein 3	Composites	40	0.00	27.7	2.75	4.88	1.78
	$ID^{2.5}$	21,833	0.00	27.7	2.88	3.35	1.17
	OK	21,833	0.00	27.7	3.40	3.60	1.06
	NN	21,833	0.00	27.7	2.91	5.20	1.79
Vein 4	Composites	113	0.00	160.0	9.92	23.95	2.41
	$ID^{2.5}$	59,564	0.00	160.0	7.32	13.72	1.87
	OK	59,564	0.00	160.0	7.10	13.04	1.84
	NN	59,564	0.00	160.0	6.96	17.39	2.50
Vein 5	Composites	15	0.16	30.0	2.86	7.31	2.56
	$ID^{2.5}$	5,193	0.16	30.0	1.60	3.52	2.20
	OK	5,193	0.16	30.0	2.02	3.70	1.83
	NN	5,193	0.16	30.0	1.83	5.85	3.19
Vein 7	Composites	18	0.45	30.0	4.63	6.63	1.43
	$ID^{2.5}$	5,166	0.45	30.0	5.25	5.29	1.01
	OK	5,166	0.45	30.0	6.09	5.50	0.90
	NN	5,166	0.45	30.0	6.46	9.19	1.42
Vein 8	Composites	9	0.03	11.7	4.50	3.90	0.87
	$ID^{2.5}$	11,953	0.03	11.7	4.19	2.34	0.56
	OK	11,953	0.03	11.7	4.77	2.39	0.50
	NN	11,953	0.03	11.7	3.63	3.24	0.89
Lucky Shot Area							
Footwall Vein	Composite	18	0.00	50.0	4.74	11.22	2.37
	$ID^{2.5}$	51,589	0.00	50.0	2.09	2.62	1.25
	OK	51,589	0.00	50.0	4.07	4.65	1.14
	NN	51,589	0.00	50.0	3.64	8.45	2.32
Hangingwall Vein	Composite	18	0.00	60.0	6.49	13.46	2.07
	$ID^{2.5}$	52,731	0.00	60.0	8.26	15.03	1.82
	OK	52,731	0.00	60.0	7.26	10.98	1.51
	NN	52,731	0.00	60.0	6.87	14.15	2.06

Table 14-14 Silver Model Descriptive Statistical Comparison

Vein	Model	Samples (n)	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Std. Dev.	CV
Coleman Area							
Vein 1	Composites	112	0.01	40.0	1.62	5.55	3.44
	$ID^{2.5}$	62,584	0.01	40.0	0.74	2.83	3.84
	OK	62,584	0.01	40.0	0.77	2.81	3.67
	NN	62,584	0.01	40.0	0.77	3.38	4.37
Vein 2	Composites	19	0.10	11.4	1.85	3.01	1.63
	$ID^{2.5}$	3,564	0.10	11.4	1.47	2.35	1.60
	OK	3,564	0.10	11.4	1.40	2.14	1.52
	NN	3,564	0.10	11.4	1.39	2.52	1.82
Vein 3	Composites	40	0.00	1.3	0.31	0.28	0.91
	$ID^{2.5}$	21,836	0.00	1.3	0.28	0.20	0.72
	OK	21,836	0.00	1.3	0.30	0.20	0.67
	NN	21,836	0.00	1.3	0.28	0.27	1.00
Vein 4	Composites	113	0.00	20.0	1.09	2.56	2.34
	$ID^{2.5}$	62,817	0.00	20.0	1.01	2.06	2.04
	OK	62,817	0.00	20.0	0.95	1.73	1.82
	NN	62,817	0.00	20.0	0.93	2.48	2.66
Vein 5	Composites	15	0.01	2.0	0.38	0.48	1.26
	$ID^{2.5}$	5,193	0.01	2.0	0.27	0.30	1.11
	OK	5,193	0.01	2.0	0.29	0.31	1.06
	NN	5,193	0.01	2.0	0.27	0.35	1.34
Vein 7	Composites	18	0.10	2.0	0.43	0.44	1.03
	$ID^{2.5}$	5,166	0.10	2.0	0.45	0.39	0.86
	OK	5,166	0.10	2.0	0.49	0.38	0.78
	NN	5,166	0.10	2.0	0.49	0.54	1.11
Vein 8	Composites	9	0.10	1.1	0.61	0.33	0.53
	$ID^{2.5}$	13,357	0.10	1.1	0.60	0.24	0.39
	OK	13,357	0.10	1.1	0.59	0.19	0.32
	NN	13,357	0.10	1.1	0.55	0.32	0.59
Lucky Shot Area							
Footwall Vein	Composite	18	0.00	0.3	0.06	0.09	1.46
	$ID^{2.5}$	65,296	0.00	0.3	0.06	0.05	0.97
	OK	65,296	0.00	0.3	0.04	0.04	1.00
	NN	65,296	0.00	0.3	0.05	0.07	1.48
Hangingwall Vein	Composite	18	0.00	0.8	0.09	0.20	2.19
	$ID^{2.5}$	66,222	0.00	0.8	0.12	0.20	1.69
	OK	66,222	0.00	0.8	0.08	0.13	1.67
	NN	66,222	0.00	0.8	0.10	0.25	2.39

The overall similarities of the statistical comparisons between the composites and models represent an appropriate amount of smoothing to account for the proposed narrow vein mining method with minimum dilution. The ID, OK, and NN models generally show similar means to the composites. This is based on the stopes having similar statistics to the composites in operation; however, this will need to be continually examined as additional data is made available. The statistical similarities are confirmed in the cumulative frequency plots of each of the models and drillhole composites (Figure 14-6).

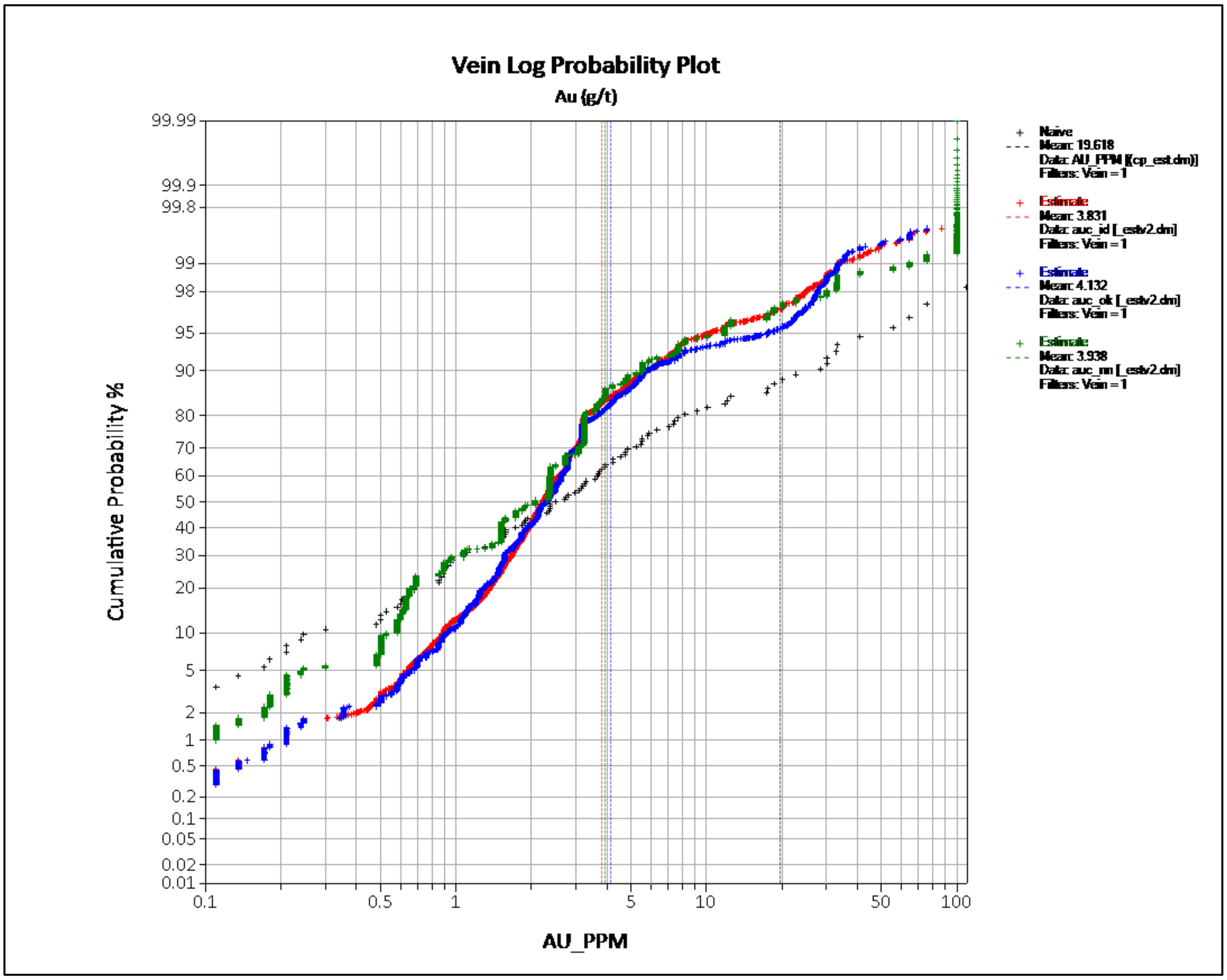


Figure 14-6 Cumulative Frequency Plot - Model Comparison (Composites – Black, NN – Green, ID – Red, OK – Blue)

14.8.2 Swath Plots

Swath plots were generated to compare average gold and silver grade in the composite samples, estimated gold and silver grade from ID2.5 method and the two validation model methods (OK and NN). The results from the ID2.5 model method, plus those for the validation OK model method are compared using the swath plot to the distribution derived from the NN model method.

Six swath plots are presented for Vein 1:

- Figure 14-7 shows average gold grade from west to east;
- Figure 14-8 shows average gold grade from south to north;
- Figure 14-9 shows average gold grade in the 1-meter benches, from bottom to top;
- Figure 14-10 shows average silver grade from west to east;
- Figure 14-11 shows average silver grade from south to north; and
- Figure 14-12 shows average silver grade in the 1-meter benches, from bottom to top.

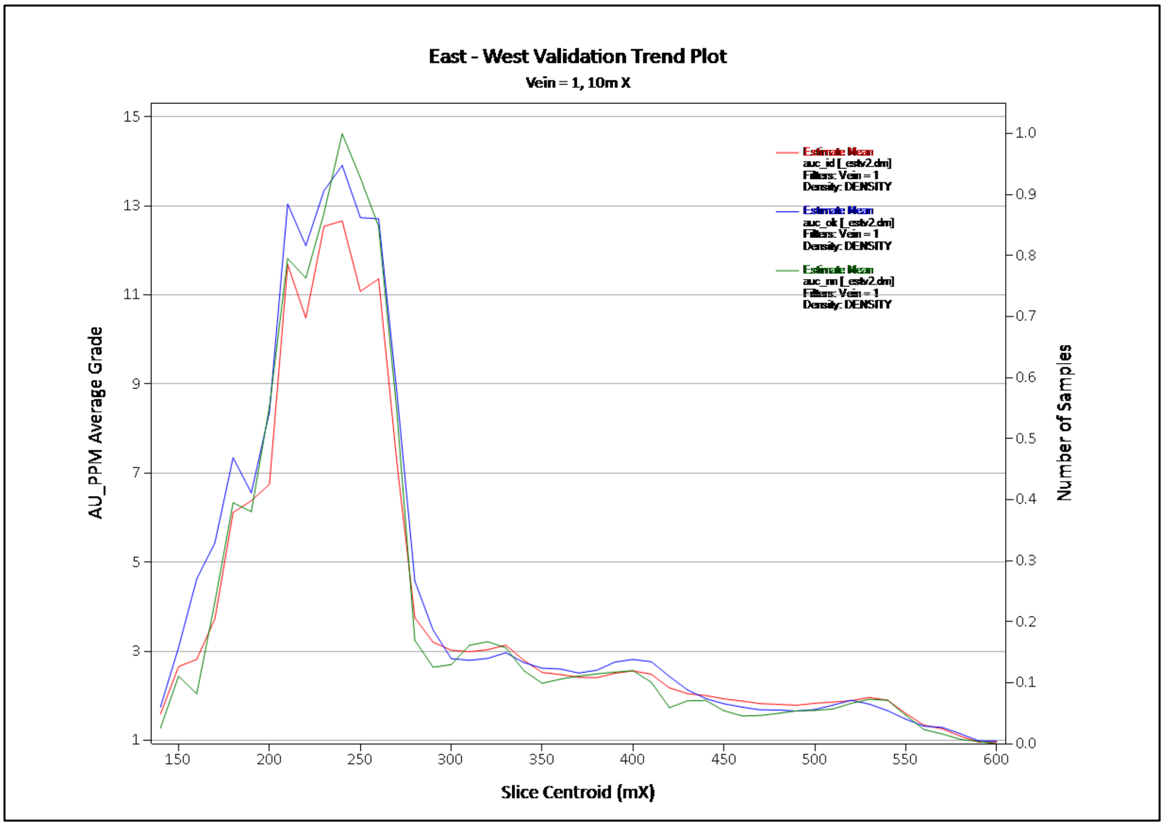


Figure 14-7 West/East Gold Swath Plot for Vein 1

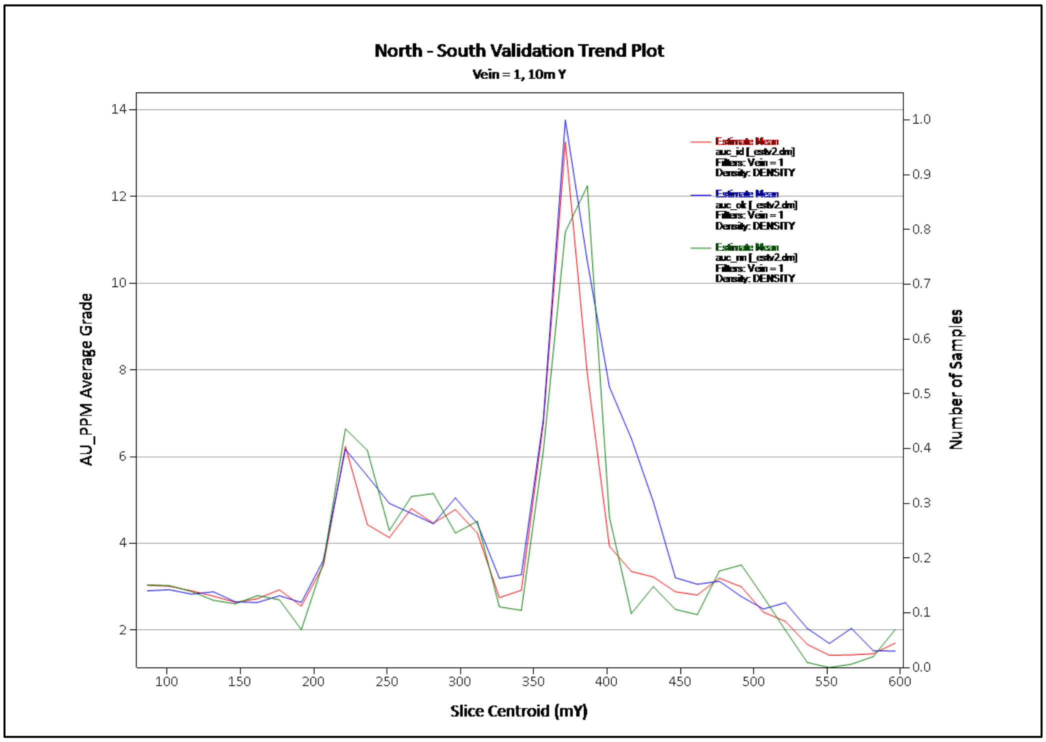


Figure 14-8 South/North Gold Swath Plot for Vein 1

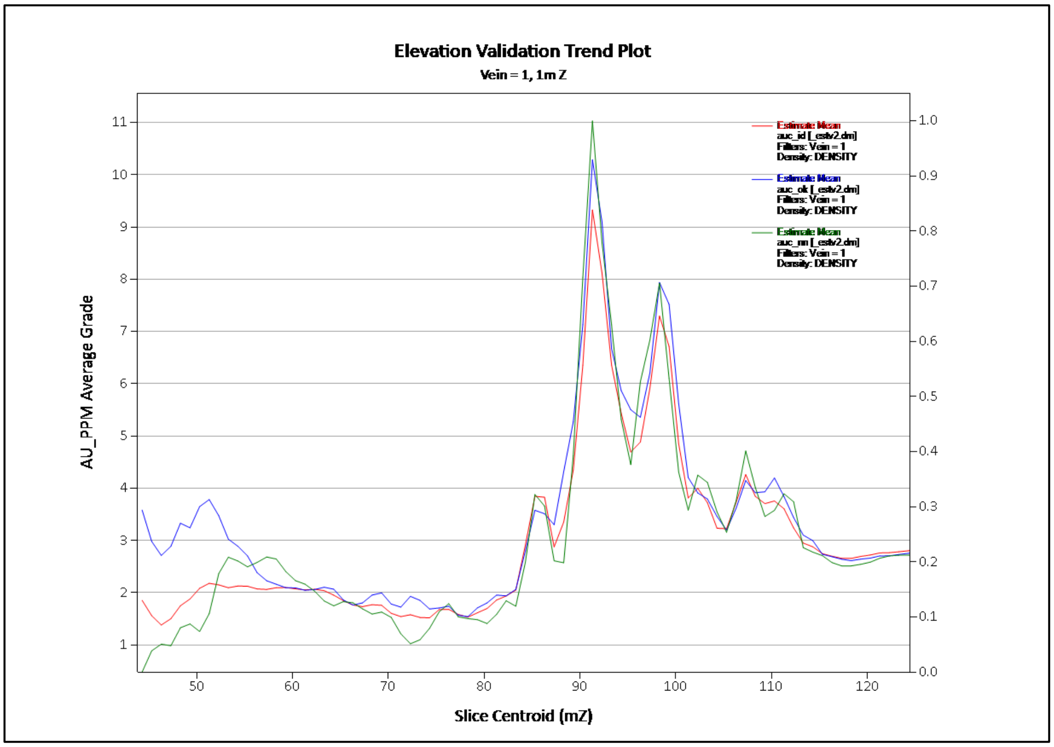


Figure 14-9 Elevation Gold Swath Plot for Vein 1

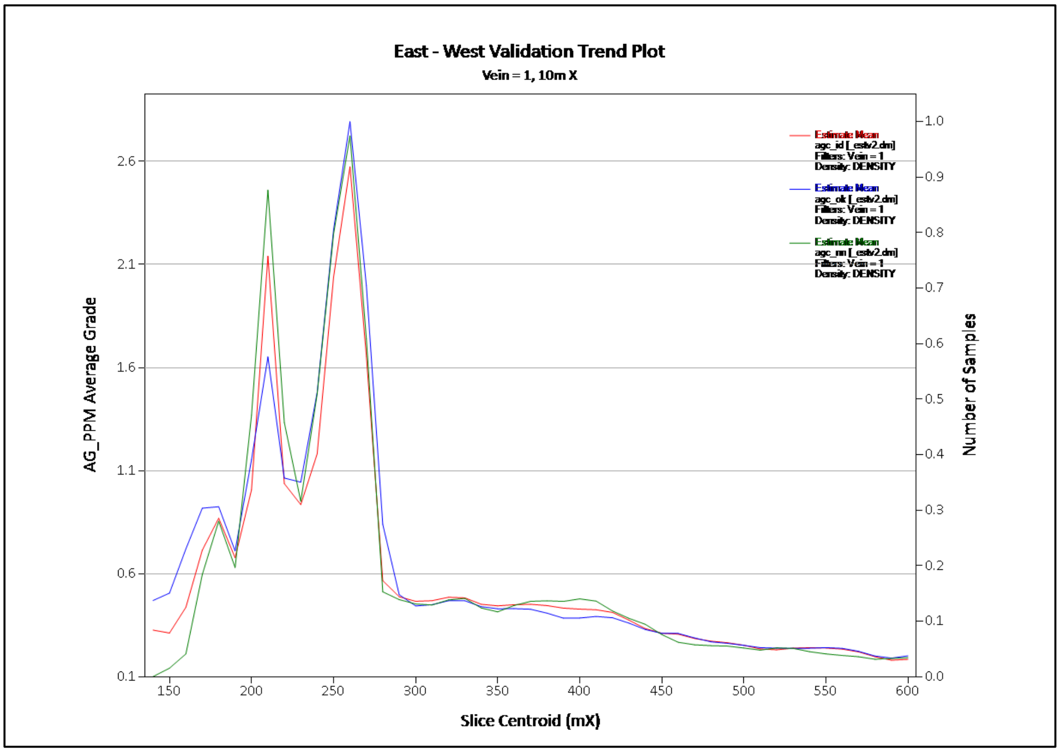


Figure 14-10 West/East Silver Swath Plot for Vein

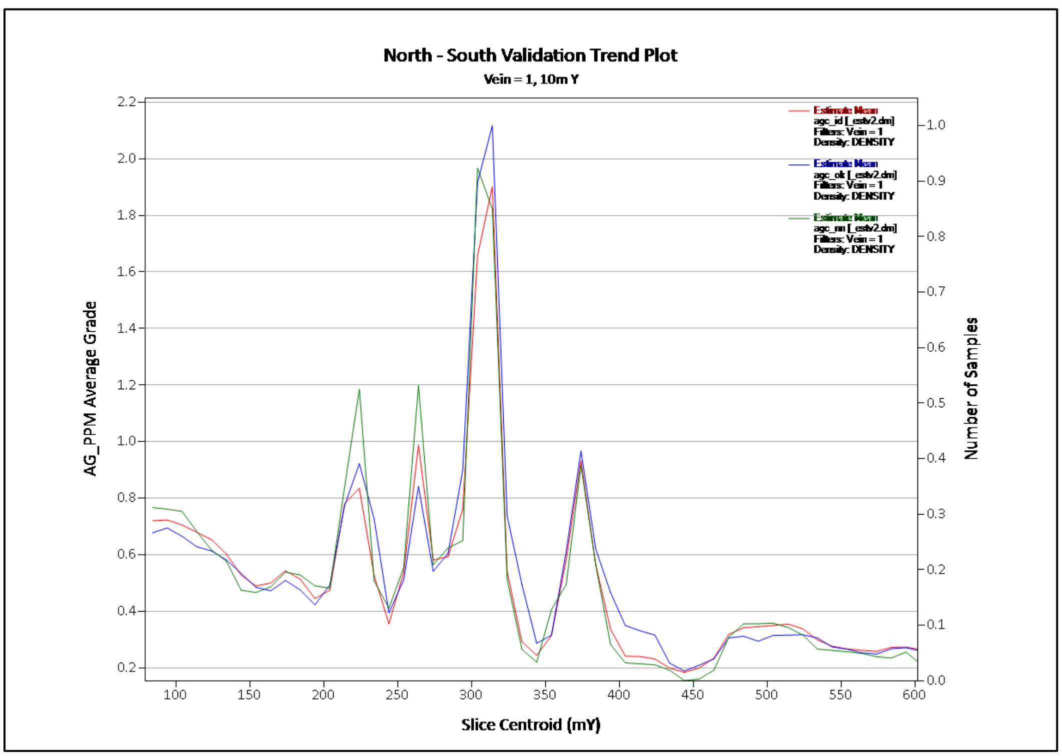


Figure 14-11 South/ North Silver Swath Plot for Vein 1

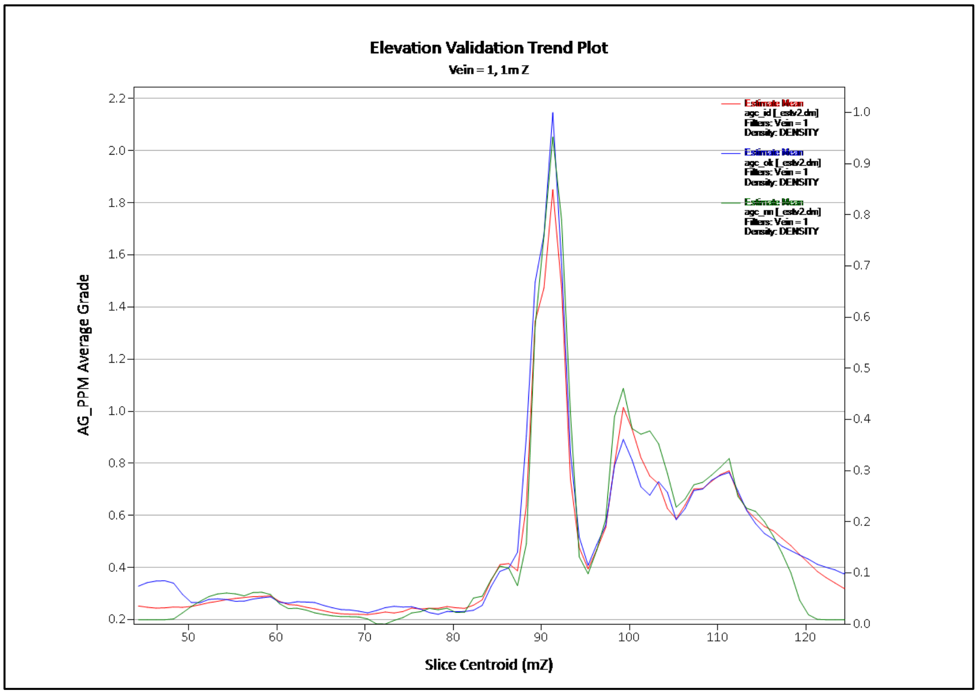


Figure 14-12 Elevation Silver Swath Plot for Vein 1

On a local scale, the NN model method does not provide a reliable estimate of grade, but on a much larger scale, it represents an unbiased estimation of the grade distribution based on the total data set. Therefore, if the ID2.5 model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the distribution of grade from the NN.

Overall, there is good correlation between the grade models and the composite data, although deviations occur near the edges of the deposit and in areas where the density of drilling is less and material is classified as Inferred resources.

14.9 Mineral Resource Classification

HRC used two methods to classify the mineral resources into measured, indicated, and inferred. For the Coleman fault block veins, measured resource are those blocks with at least one composite within an anisotropic distance of 15 x 6 meters of the block centroid. Indicated resources are those blocks with at least 2 composites within an anisotropic distance of 75 x 30 meters. Inferred resources are those blocks with at least 2 composites within an anisotropic distance of 150 x 60 meters.

For the Lucky Shot area veins, a polygonal method was used to classify resources into indicated and inferred. An exterior drillhole fence, was constructed using drillhole intercepts. An indicated boundary was generated by expanding 25 meters from the drillhole fence polyline, and 25m from drillhole intercepts. Indicated resources are those blocks inside the indicated boundary. Inferred resources are those blocks outside the indicated boundary. In cases where a single drillhole was used to estimate mineral resources, those blocks within a 15 x 6-meter search volume are classified as indicated. Figure 14-13 shows this resource classification method applied to the Footwall vein.

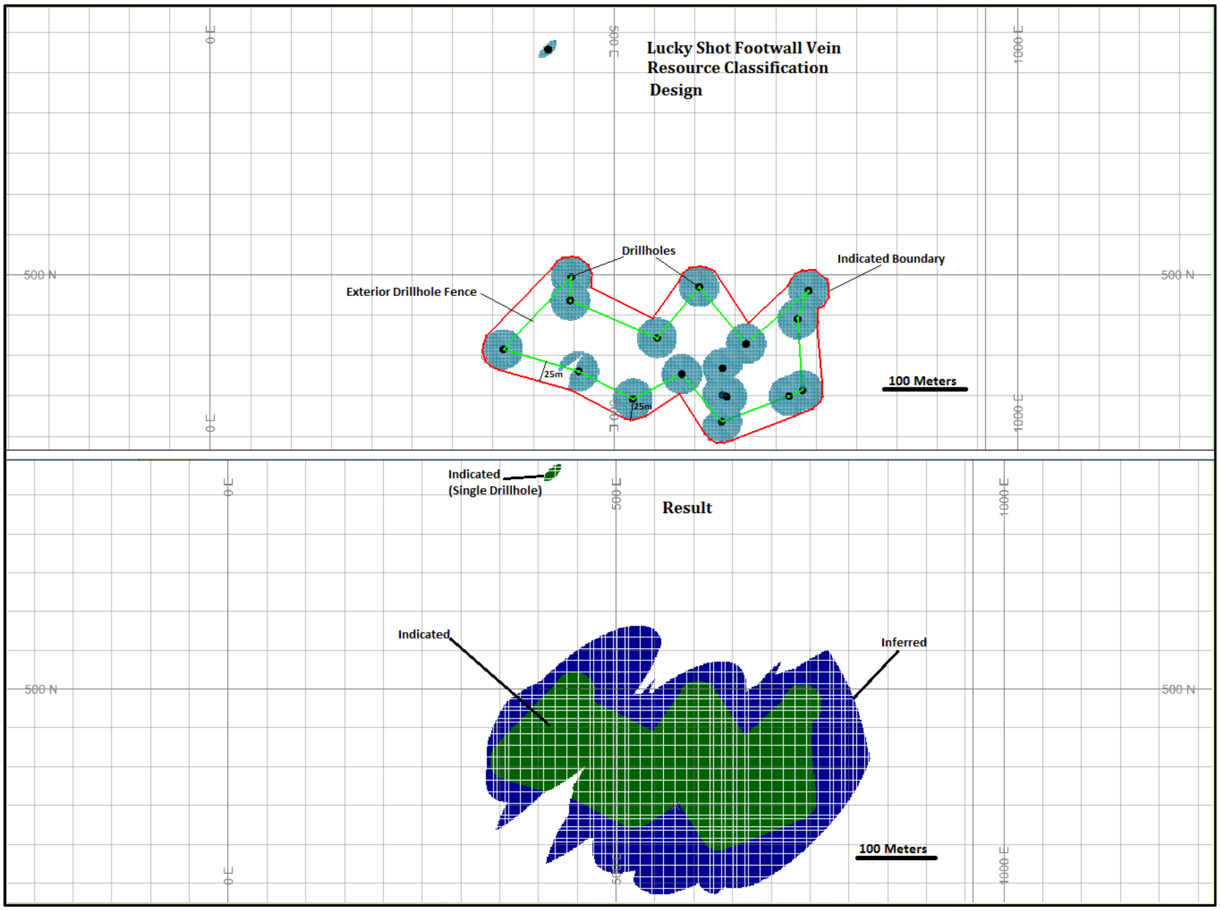


Figure 14-13 Lucky Shot Footwall Vein Mineral Resource Classification

14.10 Project Mineral Resource Estimate

The mineral resource estimate includes all analytical data obtained as of February 1, 2016. Mineral resources are not mineral reserves and may be materially affected by environmental, permitting, legal, socio-economic, political, or other factors.

Mineral resources are reported above a gold grade of 5 g/t, assuming a gold price of $1265 per ounce. HRC used a cutoff grade to test for reasonable prospects for economic extraction. Baseline assumptions for breakeven cutoff grade are based on the formula:

Cutoff Grade (gpt) = Operating Cost (per t) / Metal Price (per oz) / Metal Recovery (%)

Basis of Assumptions:

- Operating Cost (Underground Open Stope Mining): $150.00/tonne mining and processing cost
- Gold Price: $1265 /oz
- Gold Recovery (Gravity): 80%
- Cutoff grade = $150.00/tonne / ($1265/oz x 80%) x 31.10348 g/oz =4.1 g/t

Based on these assumptions, HRC considers that reporting resources at a 5.0 g/t cutoff constitutes reasonable prospects for economic extraction based on an underground open stope scenario with a gravity concentration recovery process. HRC cautions that economic viability can only be demonstrated through prefeasibility and feasibility studies.

14.10.1 Coleman area 3D Block Model Mineral Resource Estimate

The mineral resource for the Coleman area veins is presented in Table 14-15.

Table 14-15 Mineral Resource for the Coleman Area Veins, Effective Date of February 1, 2016 (HRC)

Vein	Classification	Tonnes (x1000)	Gold		Silver	
			g/t	oz	g/t	oz
Vein 1	Measured	19.3	23.8	14,700	3.22	2,000
	Indicated	28.0	12.2	11,000	2.01	1,800
	Measured + Indicated	**47.3**	**16.9**	**25,700**	**2.50**	**3,800**
	Inferred	1.2	9.6	400	0.28	0
Vein 2	Measured	5.8	46.5	8,600	2.82	500
	Indicated	4.7	23.5	3,500	1.32	200
	Measured + Indicated	**10.5**	**36.2**	**12,200**	**2.15**	**700**
	Inferred	0.0	17.1	0	0.45	0
Vein 3	Measured	1.8	13.9	800	0.59	0
	Indicated	8.4	8.8	2,400	0.31	100
	Measured + Indicated	10.3	9.7	**3,200**	**0.36**	**100**
	Inferred	2.9	5.8	500	0.42	0
Vein 4	Measured	26.4	28.3	24,100	2.37	2,000
	Indicated	69.4	17.2	38,400	2.17	4,800
	Measured + Indicated	95.8	20.3	**62,400**	**2.23**	**6,900**
	Inferred	22.5	11.9	8,600	3.31	2,400
Vein 5	Measured	0.4	21.6	200	1.27	0
	Indicated	0.8	7.7	200	0.79	0
	Measured + Indicated	1.1	12.1	**400**	**0.41**	**0**
	Inferred	-	-	-	-	-
Vein 7	Measured	2.0	11.1	700	0.74	0
	Indicated	3.3	9.2	1,000	0.65	100
	Measured + Indicated	5.4	9.9	**1,700**	**0.68**	**100**
	Inferred	0.1	5.9	0	0.65	0
Vein 8	Measured	2.2	9.7	700	0.86	100
	Indicated	10.3	7.3	2,400	0.88	300
	Measured + Indicated	12.6	7.7	**3,100**	**0.88**	**400**
	Inferred	6.8	5.4	1,200	0.65	100
Total	**Measured**	**57.9**	**26.8**	**49,900**	**2.52**	**4,700**
	Indicated	**125.0**	**14.6**	**58,900**	**1.82**	**7,300**
	Measured + Indicated	**182.9**	**18.5**	**108,800**	**2.04**	**12,000**
	Inferred	**33.6**	**9.9**	**10,700**	**2.40**	**2,600**

14.10.2 Lucky Shot Area 3D Block Model Mineral Resource Estimate

The mineral resource for the Lucky Shot area veins is presented in Table 14-16.

Table 14-16 Mineral Resource for the Lucky Shot Area Veins, Effective Date of February 1, 2016 (HRC)

Vein	Classification	Tonnes (x1000)	Gold		Silver	
			g/t	oz	g/t	oz
Footwall Vein	Indicated	9.9	7.92	2,500	0.02	0
	Inferred	4.3	6.65	900	0.02	0
Hangingwall Vein	Indicated	13.7	23.26	10,300	0.24	100
	Inferred	21.1	34.68	23,500	0.47	300
Total	**Indicated**	**23.6**	**16.82**	**12,800**	**0.15**	**100**
	Inferred	**25.4**	**29.93**	**24,400**	**0.39**	**300**

14.10.3 Willow Creek Mineral Resource Statement

The mineral resources for the Project as of February 1, 2016, are summarized in Table 14-17.

Table 14-17 Mineral Resource Statement for the Project, Matanuska-Susitna Borough, Alaska Hard Rock Consulting, LLC, Effective Date of February 1, 2016

Classification	Tonnes (x1000)	Gold		Silver	
		g/t	oz	g/t	oz
Measured	57.9	26.8	49,900	2.5	4,700
Indicated	148.6	15.0	71,600	1.6	7,400
Measured + Indicated	**206.6**	**18.3**	**121,500**	**1.8**	**12,100**
Inferred	59.0	18.5	35,100	1.5	2,900

Note: Measured, Indicated and Inferred mineral classifications are assigned according to CIM Definition Standards. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability and there is no guarantee that mineral resources will be converted to mineral reserves. (1) The mineral resource estimate was prepared by HRC based on data and information available as of February 1, 2016. The 2016 Measured, Indicated and Inferred mineral resources are reported considering a base case estimate that applies a cutoff grade of 5 g/t Au based on the estimated operating costs, historical recoveries, and a $1,265/oz gold price.

15. MINERAL RESERVE ESTIMATE

Mr. Jeff Choquette, P.E., MMSA QP, of HRC is responsible for the mineral reserve estimate presented herein. Mr. Choquette is Qualified Person as defined by NI 43-101 and is independent of GTOR. The mineral reserve calculation for the Project was completed in accordance with NI 43-101 and is based on all data and information available as of May 31st, 2016. Stope designs for reporting the reserves were created utilizing the mineral resources presented in Section 14 of this report. An open stull stope mining method is planned to extract the Coleman and Lucky Shot deposits with random waste pillars. Ore is planned to be processed in a gravity concentration process plant capable of processing 200 tpd.

15.1 CALCULATION PARAMETERS

HRC utilized Datamine's Mineable Shape Optimizer ("MSO") program to generate the stopes for the reserve mine plan. The parameters used to create the stopes are listed below:

- Cutoff Grade: 7 g/t AuEq
- Minimum Mining Width: 0.8 m.
- Full Stope Size: 9.14m W+ x 16.76m H
- Sub Stope Size: 4.57m W x 5.68m H
- External Stope Dilution: 20%
- Mine Ore Recovery Factor: 90%
- Silver Equivalent: 78.3:1 silver to gold
- Gold Price: US $1,175/oz
- Silver Price: US $15/oz
- Gold Recovery: 91.8%
- Silver Recovery: 90.0%

The stopes were created based solely on Measured and Indicated mineral resources, which have demonstrated to be economically viable, including internal stope dilution above the calculated cutoff; therefore, Measured and Indicated mineral resources within the stopes have been converted to Proven and Probable mineral reserves as defined by NI 43-101. Inferred mineral resources are not considered as part of the reserve statement.

15.1.1 Dilution

Dilution is applied to Measured and Indicated resource blocks depending on the mining method chosen. For blocks to be exploited using open stull tope mining methods, external dilution was applied in the amount of 20% at a grade of zero. Internal dilution is also applied based on any blocks that fall inside the stope shape but are below cutoff. A mining recovery is also applied to convert resources to reserves and is estimated at 90%. These factors resulted in an overall dilution factor of 31% for the Coleman vein and 56% for the Lucky Shot vein, with an overall average dilution of 34% for the reserves.

Dilution and mining recoveries are functions of many factors including workmanship, design, vein width, mining method, extraction, and transport. Currently there is no supporting documentation with which to validate these dilutions or mining recovery estimates. When production commences HRC recommends that individual dilution and recovery studies be performed on various veins and to refine the global estimates used for dilution and mining recovery.

15.1.2 Cutoff Grade

The mining breakeven cut-off grade was used to generate the stope designs in DataMine's MSO for defining the reserves. The estimated operating costs and mill recoveries developed for the PFS are used to calculate the reserve breakeven cut-off grade. A gold price of \$1175/oz was chosen, which is close to the 200 day moving average of \$1162/oz as of May 31st, 2016. The parameters used for the calculation are presented in Table 15-1. The mine operating costs used in the cutoff calculation exclude the costs for development.

The cut-off is stated as gold equivalent since the ratio between gold and silver is variable and both commodities are sold, although the silver is estimated to be fairly insignificant with only 0.12% of the revenue coming from the silver produced. The average cut-off grade used for the Project is 7.0 g/t AuEg.

Table 15-1 Mineral Reserve Breakeven Cutoff for Project

Reserve Cutoff		
Mining	\$/ore tonne	\$127.36
Processing	\$/ore tonne	\$75.14
G&A	\$/ore tonne	\$39.55
Recoveries	%	91.8%
Total cost	\$/ore tonne	\$242.05
Gold Selling Price	\$/oz	\$1,175.00
Cutoff Grade	**AuEq gpt**	7.0

15.2 Reserve Classification

Mineral reserves are derived from Measured and Indicated resources after applying the economic parameters as stated Section 15.1.2, utilizing Datamine's MSO program to generate stope designs for the reserve mine plan. Mineral reserves for the Project have been derived and classified according to the following criteria:

- Proven mineral reserves are the economically mineable part of the Measured mineral resource for which mining and processing / metallurgy information and other relevant factors demonstrate that economic extraction is feasible and have mine plan in place.
- Probable mineral reserves are the economically mineable part of the Indicated mineral resource for which mining and processing / metallurgy information and other relevant factors demonstrate that economic extraction is feasible and have mine plan in place.

15.3 Mineral Reserves

The Proven and Probable mineral reserves for the Project as of May 31st, 2016 are summarized in Table 15-2. The reserves are exclusive of the mineral resources reported in Section 14 of this report.

Table 15-2 Proven and Probable Mineral Reserves, Effective Date May 31st, 2016

Classification	Area	Tonnes (x1000)	Gold		Silver		Dilution
			g/t	oz	g/t	oz	
Proven	Coleman	68.7	18.9	41,672	2.02	4,465	37%
	Lucky shot	0	0	0	0	0	0%
Total Proven		**68.7**	**18.9**	**41,672**	**2.02**	**4,465**	**37%**
Probable	Coleman	87.9	13.4	37,936	1.67	4,728	26%
	Lucky shot	17.8	13.8	7,897	0.09	51	56%
Total Probable		**105.7**	**13.5**	**45,833**	**1.41**	**4,779**	**31%**
Total Proven + Probable		**174.4**	**15.60**	**87,504**	**1.65**	**9,244**	**34%**

1. Reserve cut-off grades are based on a 7.0 g/t gold equivalent with 78.3:1 silver to gold ratio.
2. Metallurgical recoveries were estimated at 90.0% silver and 91.8% for gold.
3. Mining recoveries of 90% were applied.
4. Minimum mining widths were 0.8 meters.
5. Dilution factors averaged 34%. Dilution factors are calculated based on internal stope dilution calculations and external dilution factors of 20%.
6. Price assumptions are $15 per ounce for silver and $1,175 per ounce for gold.
7. Figures in table are rounded to reflect estimate precision; small differences generated by rounding are not material to estimates.

15.4 Factors that may affect the Mineral Reserve Estimate

The process of mineral reserve estimation relies on technical information which requires subsequent calculations or estimates to derive sub-totals, totals and weighted averages. Such calculations or estimations inherently involve a degree of rounding and consequently introduce a margin of error. The QP does not consider these errors to be material to the reserve estimate.

The realization of the mineral reserves reported here is dependent on the successful implementation of the mining practices and open stull mining method suggested. If other mining methods are required, dilution may be hard to control on the narrow, shallow dipping veins. Although the Project has completed considerable baseline work for permitting, the final operating permits have not been issued. Mineral processing utilizing gravity separation reduces the potential for difficulty in obtaining the final operating permits compared to cyanide or floatation processing operations.

Other than the above, HRC is unaware of any legal, political, environmental or other risks that could materially affect the potential development of the mineral reserves.

16. MINING METHODS

16.1 Summary

The mine plan for the Project includes approximately 174,400 tonnes of ore grade material to be extracted by selective small scale underground mining in four years. The mine production schedule calls for the production of 200 tonnes per day of ore to be extracted from the mine, loaded into 10-tonne trucks and hauled from the mine site approximately 25 miles down to the Willow area to a mill site for processing. The overall mining dilution is expected to be approximately 34% from the underground mine using open stull stope mining.

The mineralized resource material will placed on a stockpile at the mill. A loader will be employed to feed the mill and gold recovery plant at two twelve-hour shifts, five days per week.

16.2 Geotechnical Parameters

Yukuskokon Professional Services (YKPS) completed a desktop geotechnical review of the Lucky Shot and Coleman areas in February of 2016, and produced a report on their findings and recommendations for the project as follows:

> "*The total number of drill holes and the drill hole density coupled with the geotechnical data in the areas of development should be looked at in depth considering fracture density, fracture fillings, and smoothness of the structural features. Additionally, geotechnical logging of the 2009 drill holes would strengthen the geotechnical database. Engineers could use such evaluation to estimate the maximum amount of open space for safe stope mining. YKPS recommends further detailed underground geotechnical mapping of faults, slips, joints, etc., with observations of gouge, fracture fill, alteration, frequency of structures, etc. to forecast similar structures or conditions in the development and the stopes during mining. Stress mapping and direct observation of active induced stress in the tunnels during mining would serve the operation to avoid risk to workers, especially in the stopes and around pillars and brows. If sufficient, the project could use this level of data collection, to develop a hazard map for a better understanding of the rock behavior as mine development approaches the vein, or as the mining stopes out the vein. During mining in situ stress measurements and rock bolt pull testing should be a component of the underground development and mining plan to maximize the safety of the operation.*"

HRC agrees with the conclusions presented by YKPS, and also recommends that further geotechnical evaluation be carried out in order to aid in stope design. HRC also recommends that an active data collection and review program be implemented and carried out through production.

Ground conditions used as the basis for assessing the mine design parameters are based on observation of the accessible mine workings at the Lucky Shot project and historical mine data and observations from the War Baby, Lucky Shot, Coleman and Independence mines.

16.2.1 Ground Support

General field observations of the existing historic conditions of the Enserch adit and the Coleman adit show that both were bolted with split set rock bolts. Split sets and wire mesh on 4- by 4-foot pattern with a 5 spot bolt pattern are adequate for most of the ground conditions that will be encountered in the planned drift and ramp development. The length of the split to be used will be either a 4-foot or 5-foot bolts. In areas of faulted ground, a combination of 6-foot split sets and/or 6-foot dywidag bolts will be used to control the fractured ground. When crossing the major offset faults, extra support will be installed in the form of a 3-piece drift set. The general reinforcement support observations are further supported by recommendations set forth in Figure 16-1.

The ground support that is planned for the stope areas will consist of 10- to 12-inch round or square timbers placed on 5-foot centers, along with random waste pillars. Access drifts, ramps, and escape way raises will be protected by pillars that will be two times the mining thickness. Stope areas that may experience heavy ground will also be supported by timber cribbing.

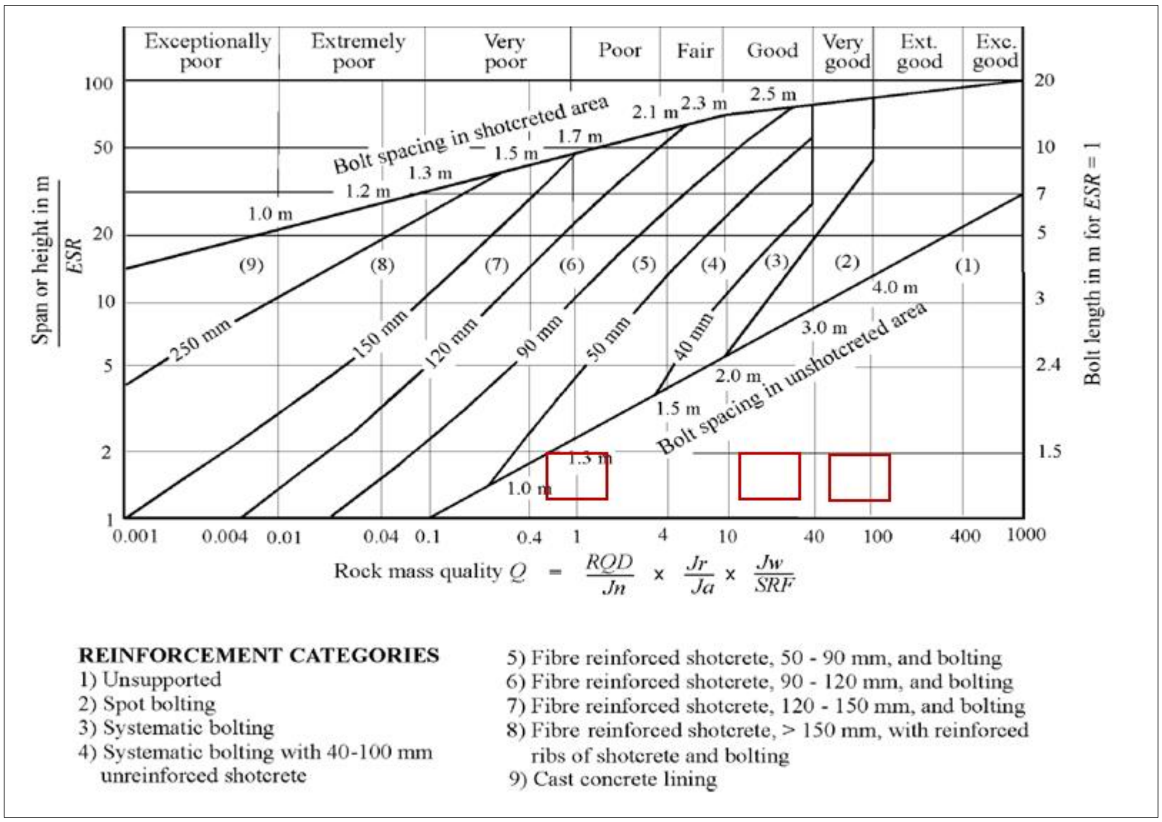


Figure 16-1 Ground Support Chart (Barton et. al., 1974)

16.3 Mining Method

Industry standard underground mining methods were reviewed together with the historical local mining method data to determine that open stull stope mining method will be used to extract the Coleman and Lucky Shot deposits with random waste pillars. The Lucky Shot area of the deposit will be used to test a modified version of open stull stoping. This method was selected considering the deposits' size, shapes, orientation, and dip angle. The modified version is designed to produce ore tonnes as quickly as possible while minimizing costs. Stoping operations will be conducted using jackleg drills for drilling and slusher loading of ore from ore blocks 60 meters wide by 100 to 141 meters long on dip. Slushers will be used to move the ore down raises to the main haulage level where it will be loaded into rail mounted ore cars and hauled outside.

The mining method proposed for the Lucky Shot mine project is a modified open stull stoping, a method that has been used at the copper deposits of upper Michigan and flat dipping gold, silver, and base metals veins/deposits of the western U.S. There are two methods of stope block access that will determine the stope layout: mechanized ramp, or raises driven on the vein. The general mine layout for access by a raise will consist of a vertical level spacing of 200 feet (60 meters) with 462 feet (141 meters) of on-vein mining. After the initial level has been developed on the vein structure, a series of raises, on 200-foot (61 meters) centers, will be driven up the vein dip to the next level. The ore block exposed by the raise will be 200 feet (61 meters) wide by 462 feet (141 meters) long with a minimum mining height of 2.5 feet (0.76 meters). Each raise ore block will then be divided into eight mining blocks to be extracted.

Open stull stoping was chosen for its flexibility in handling the low vein dip angles and the ability to leave waste in place as pillars. The method also minimizes the amount of dilution during mining by careful geological and management control of the mining, so an average external dilution of 20 percent is used and applied across all of the mining areas. Figure 16-2 shows an open stull stope at Commodore Mine in Creede, Colorado which is similar to the Coleman stopes.

Figure 16-2 Open Stull Stope at Commodore Mine in Creede, Colorado

The raises will be driven with a 5- by 8-foot or 10- by 8-foot cross section to the next level and sampled. On each side of the raise, four ore blocks will be marked out 115 feet long. At the bottom of the blocks, a scram on ore will be driven 100 feet to the edge of the block and sampled. A mining face 30 feet wide in a staggered layout will be started and carried up a 115 feet to the scram above. Each individual block of a staggered face is 5- by 5-feet square. Blasted ore is then scraped down from the face to the scram and scraped out to the raise. The raise scraper will then pull the ore down to the loading platform to load the ore cars. Ground support will be 8- to 12 -inch round or square timber placed on 5-foot centers and lagged to keep the blasted ore close to the face or waste pillars as the stope is being advanced. The advantage with a staggered face is a reduction in explosives and blast damage to the back. A staggered face also allows sampling on two faces for each 5-foot by 5-foot mine block.

The ramp access to the vein we will drive a mechanized ramp up the dip on the vein with mining occurring from small sub level drifts driven to the edge of the known resource block. Mining will start at the back end of the drift and will be carried up to the next sublevel. Each stope block will be 30 to 50 feet long and mined in a staggered configuration to reduce explosive usage and to minimize blast damage to the back. If the distance between sublevels is only 18 to 30 feet then three to four cuts will be taken that will average 50 feet long and will be mucked from the bottom sub level by a one cubic yard load-haul-dump (LHD). The LHD will haul the gold bearing ore material out to the ramp and load it into a 4-tonne underground mechanical haul truck. The haul truck will then haul the ore down ramp to a dump pocket. The dump pocket will then feed 5- tonne rail ore cars in an underground train that will haul the gold bearing rock to the portal. Figures 16-3 to 16-8 show the planned stoping layouts.

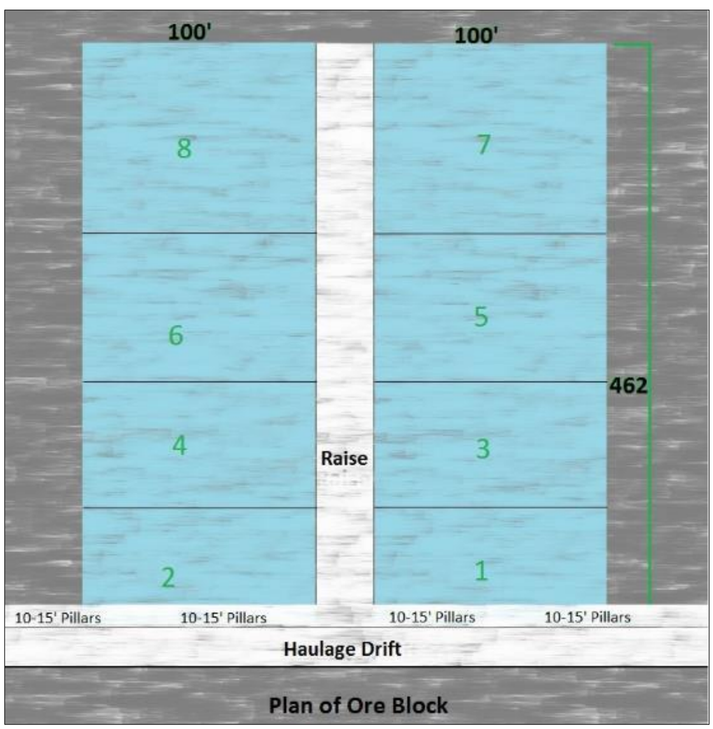


Figure 16-3 Ore block Developed by Raise

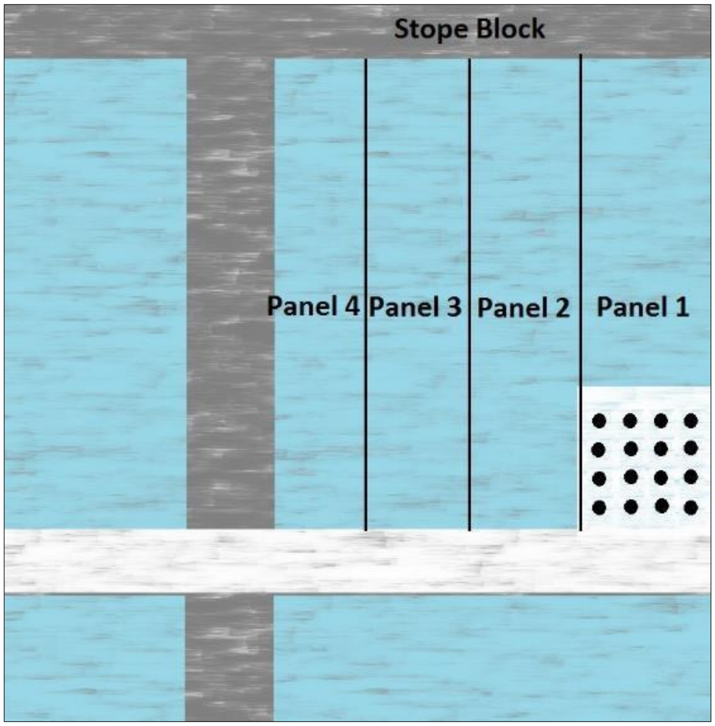


Figure 16-4 Stope Block Divided by Panels

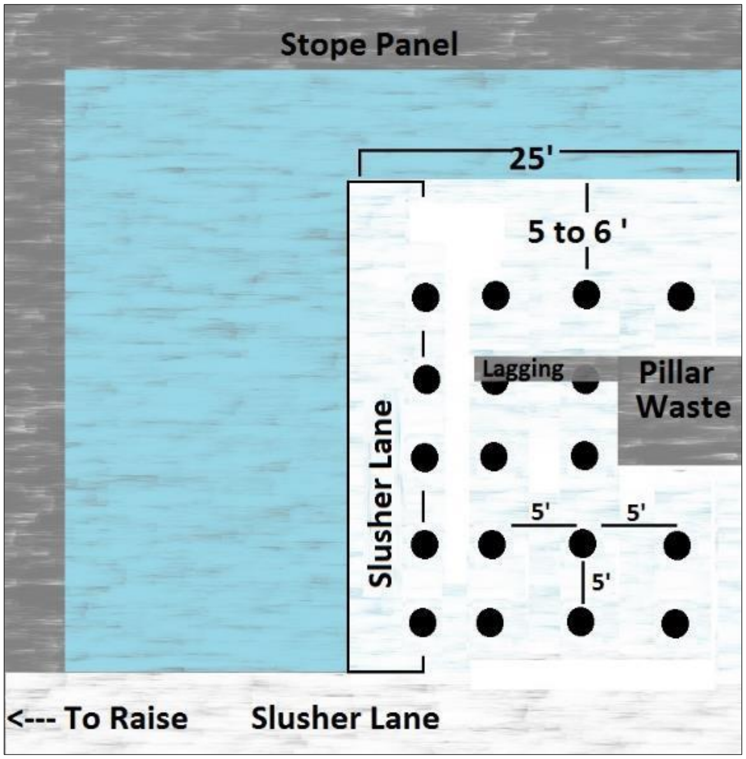


Figure 16-5 Stope Panel

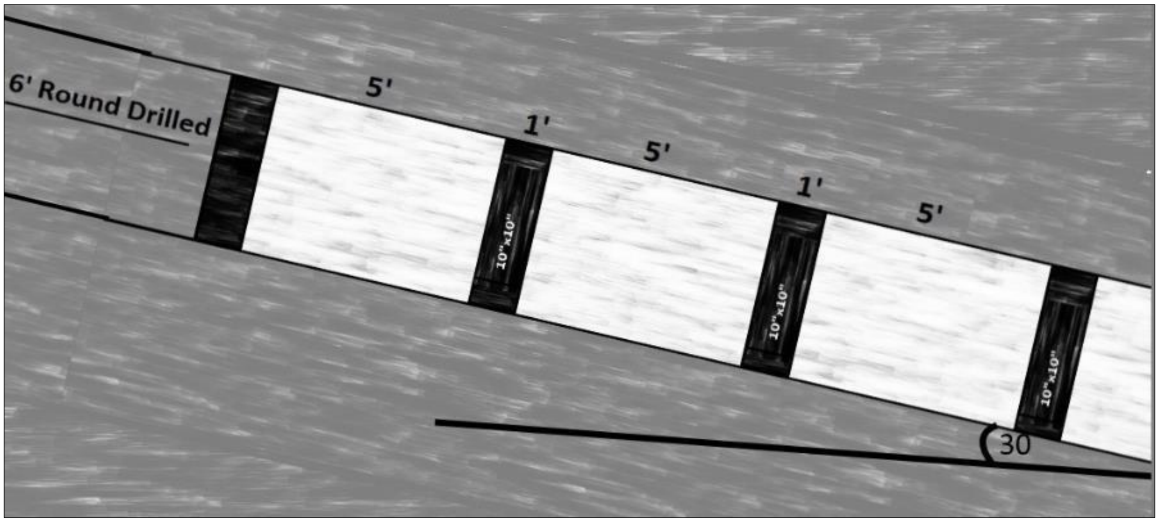


Figure 16-6 Cross Section of Stope Panel

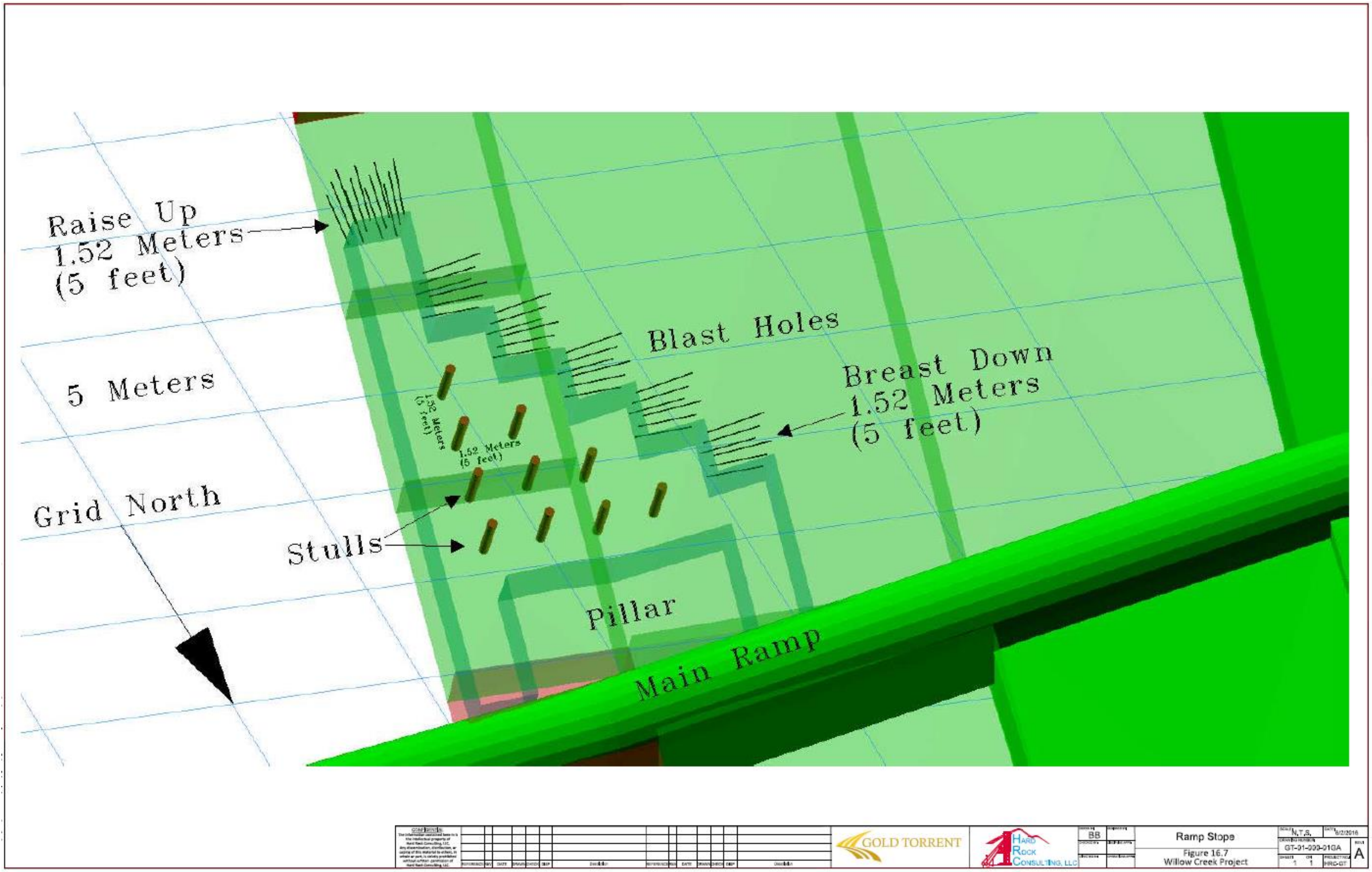


Figure 16-7 Ramp Stope Layout

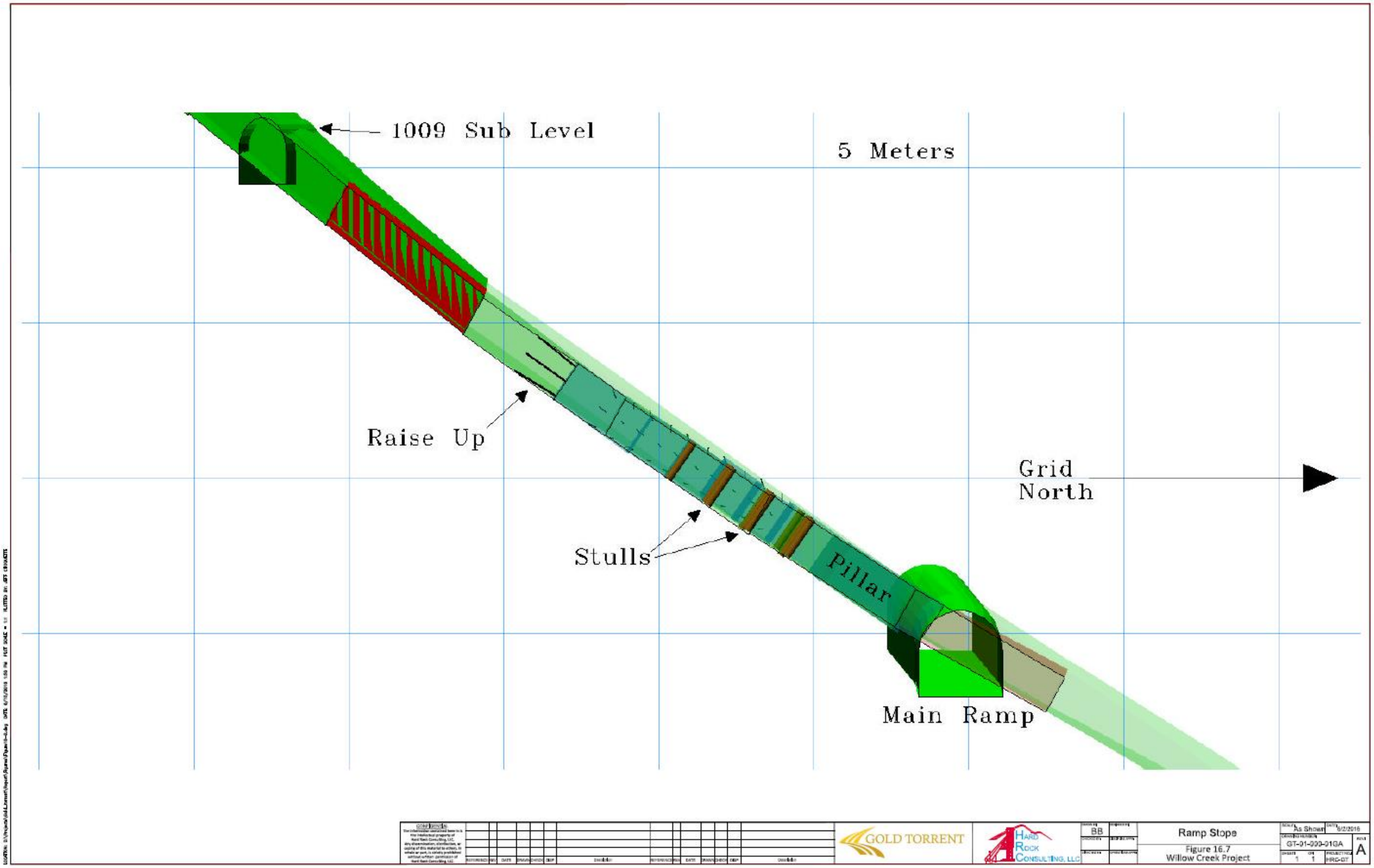


Figure 16-8 Ramp Stope Section

16.4 Access to Coleman from Lucky Shot via Enserch

Access to the Lucky Shot and Coleman block resources will be made by extending the existing Enserch adit workings back another 364 meters to intersect the down dip extension of the Lucky Shot vein. At the vein intersection a drift will be made along the face in an east-west direction on the Lucky Shot vein. The East drift will be given priority and driven 200 meters. At this point a raise will be driven up on the Lucky Shot vein and the 775 level will be driven 100 meters east where another raise will be developed to connect with the old 500 level. At the same time that the Enserch workings are being rehabbed, a crew will be working on the Lucky Shot 500 level rehabilitation. This initial phase of the development work is designed as an early secondary escape along with a ventilation circuit. At the same time the West haulage drift will be driven on the vein toward the Coleman resource block for 200 meters. At this point the main Coleman fault should be intersected and crossed. After crossing the main fault, the West drift will turn north and be extended roughly 55 meters until the down dip extension of the Coleman vein is intersected. At this point drifting will continue another 100 meters. After advancing 60 meters, a 5-meter crosscut will be driven and a ramp will be started at plus 15% that will be located within the vein and be driven up into the Coleman block resource. All drifting within the vein will likely produce mineralized material. After a review and tradeoff analysis of several Coleman access options, including Alamac raises, ore passes, aerial tramway, internal hoisting via inclined shaft, tunneling and conveying, and external roadways, the chosen approach is an internal ramp, which will allow access for wheel and tracked machinery. This method is preferred for lower overall cost, lower man count, and increased opportunity for mining.

Approximately 25,000 tonnes of waste material will be generated from the mine development will be placed in the waste dump located outside of the Enserch portal. Figure 16-9 shows the planned waste dump location.

Figure 16-9 Waste Dump at the Enserch Portal Site

16.5 Cut-off grade

The mining breakeven cut-off grade was used to generate the stope designs in DataMine's MSO for defining the reserves. The estimated operating costs and mill recoveries developed for the PFS are used to calculate the reserve breakeven cut-off grade. A gold price of $1175/oz. was chosen, which is close to the 200 day moving average of $1162/oz. as of May 31st, 2016. The parameters used for the calculation are presented in Table 15-1. The total operating cost per tonne of ore was estimated at $242.04/t and the gold recovery was estimated at 91.8%. The mine operating costs used in the cutoff calculation exclude the costs for development.

The cut-off is stated as gold equivalent since the ratio between gold and silver is variable and both commodities are sold, although the silver is estimated to be fairly insignificant with only 0.12% of the revenue coming from the silver produced. The average cut-off grade used for the Project is 7.0 g/t AuEg.

16.6 Production Schedule

The production schedule is based on the development rate assumptions shown in Table 16-1.

Table 16-1 Productivity Rates

Activity Type	Dimensions meters (ft.)	Rate meters (ft.)
Main Haulage	2.74 x2.74 (9x9)	4.57 m/d (15 ft./d)
Ramp	2.74 x2.74 (9x9)	4.57 m/d (15 ft./d)
Cross Cut	2.74 x2.74 (9x9)	3.35 m/d (11 ft./d)
Sub Level	1.52x2.13 (5x7)	3.35 m/d (11 ft./d)
Slusher Scram	1.22x2.13 (4x7)	3.35 m/d (11 ft./d)
Rail Drift	1.52x2.13 (5x7)	3.35 m/d (11 ft./d)
Raise	1.52x2.44 (5x8)	3.35 m/d (11 ft./d)
Rehabilitation of Old Workings		9.14 m/d (30 ft./d)

The mine operations schedule is based on 208 days/year 4 days/week, with two 10-hour shifts each working day. The shortened week is to minimize the amount of time that power is required to be generated for the mine. The production rate at full production is 200 tonnes per day with a 4-month ramp up period in the Lucky Shot mine.

Table 16-2 presents the annual mining schedule based on these assumptions. The project starts month one as January 1, 2017 (Year -1) with rehabilitation of the Enserch adit and the Lucky Shot mine 500 level. The initial plan is to spend the first year in rehab work and to connect the Enserch haulage level to the Lucky Shot 500 level to establish the secondary escape and ventilation. The initial ore mining will come from this developed area of the Lucky Shot vein system. During the trial mining development work will continue on the haulage level toward the Coleman block resource area of the mine.

Table 16-2 Annual Mining Schedule

Production Schedule	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Life-of-Mine
MINE PRODUCTION							
Tonnes Ore Mined	-	23,296	40,277	51,884	50,231	8,761	174,448
Ag, g/tn	-	1.88	1.50	1.86	1.59	0.80	1.65
Au, g/tn	-	15.01	13.48	16.31	17.56	11.49	15.60
Development Meters	2,012	2,195	834	764	791	-	6,596
Development Waste	21,820	44,687	10,994	6,551	6,790	-	90,843
Total Tonnes Mined	**21,820**	**67,983**	**51,271**	**58,435**	**57,020**	**8,761**	**265,291**

The mine development and production schedule is shown in Figure 16-10 and is colored coded by year.

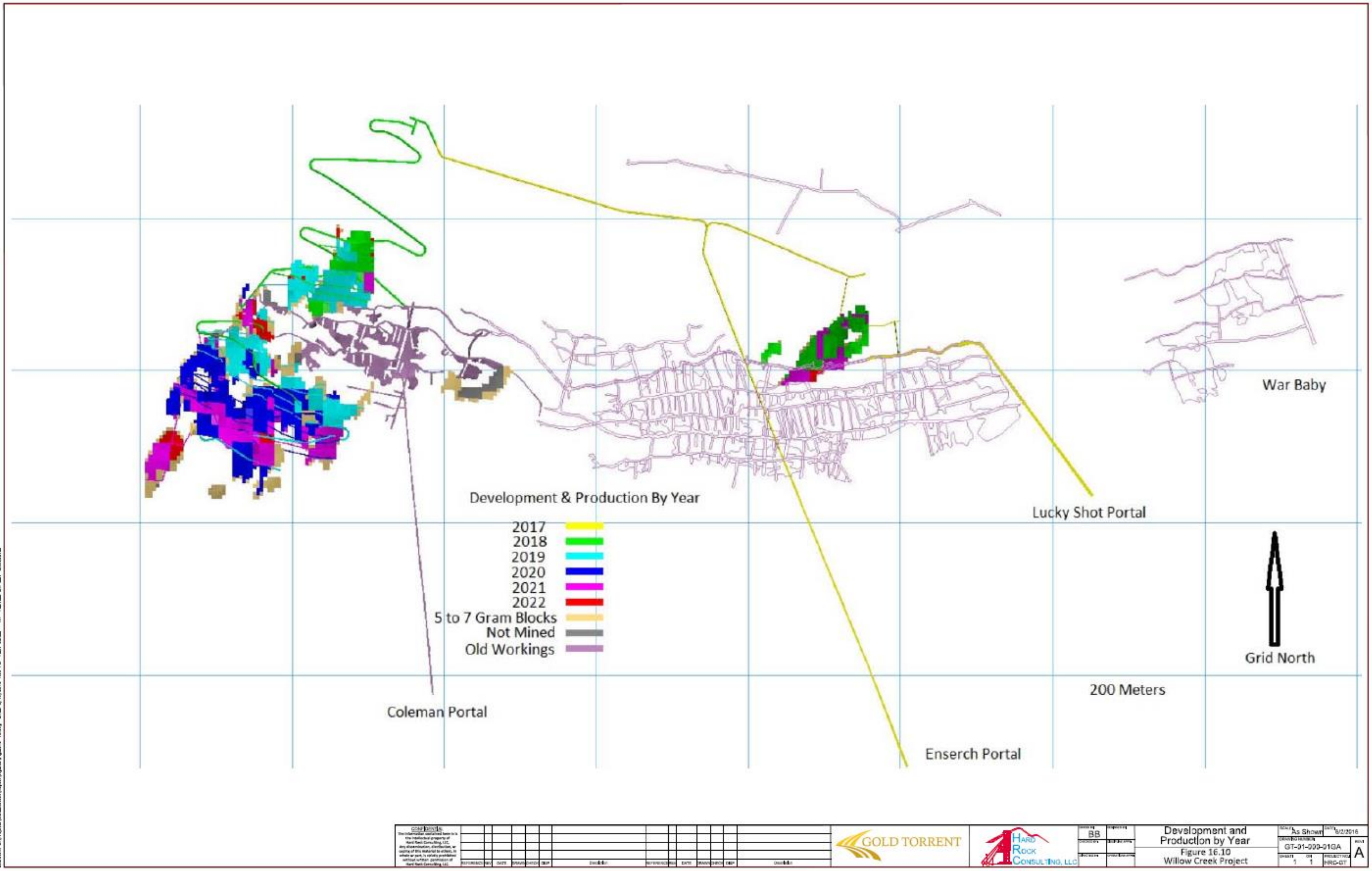


Figure 16-10 Mine Development and Production by Year

16.7 Lucky Shot Ventilation

The ventilation for the Lucky Shot project is divided into two phases: the development phase and the production phase. Several developments options were evaluated for access to the gold resources located underneath the Coleman mine block.

Option 1: All diesel mine ramp at plus 15% at 12- by 12-foot arched profile, with new portal located directly under the old Coleman mine portal at 3,000 feet elevation.

Option 2: Diesel ramp extending from the back end of the Enserch portal and ramping up to the bottom of the Coleman block resource. Enlarge Enserch tunnel to 12- by 12-foot profile with new 12- by 12-foot incline ramp at plus 15%.

Option 3: Extend Enserch tunnel back to down dip projection of the Lucky Shot vein, and follow the vein to underneath the Coleman mine block and then a ramp following the vein up to the resource area. An initial secondary escape is established on the East branch of vein development through a raise connection to the Lucky Shot 500 level.

Option 3 was selected because it establishes the secondary escape way within the first year and provides access to explore the vein system at a lower level.

All values used in the ventilation design are estimated based on take offs of the elevation of the mine from historical maps. When measurements are taken during the development phase the actual measurements will be used to refine the production ventilation circuit.

16.7.1 Development Phase

The Enserch tunnel will be rehabbed and ventilated using 24-inch steel ducting as the Enserch level is extended to the Lucky Shot vein. During the initial development a one cubic yard LHD will be used to muck out the face back to 60- to 100-cubic foot rail cars. The development fans selected are listed in Table 16-3 and Figure 16-11 shows the initial development layout.

Table 16-3 Development Fans

Mfr.	Model	RPM	HP	Units
Jet Air	231/4" x 17"	3600	30	3
Hurley	FC-24-18-324-6	3600	40	2

The development ventilation system is designed to deliver 12, 000 cfm, at 26 inches of water, at the end of 3,260 feet of ducting.

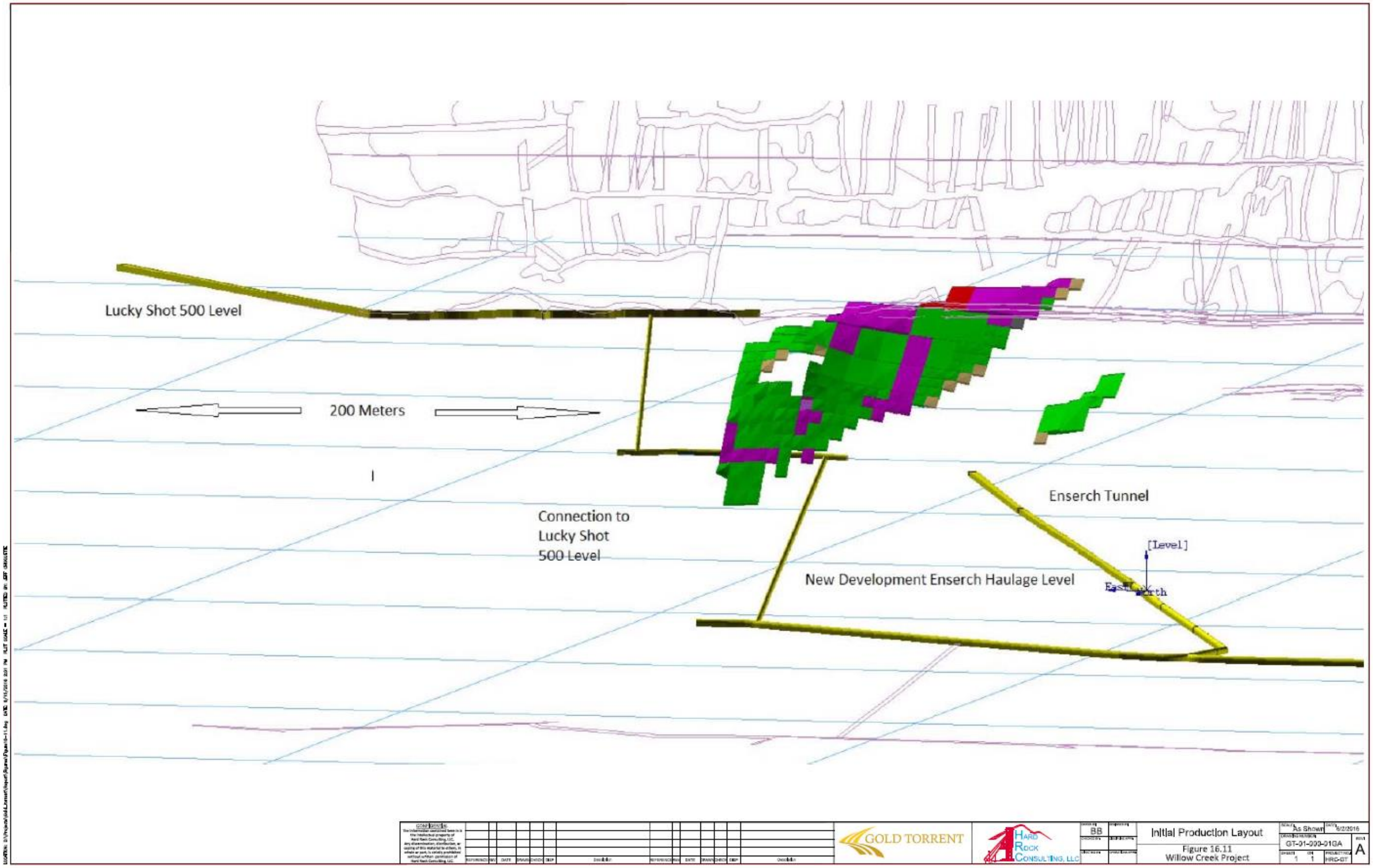


Figure 16-11 Lucky Shot Initial Production Layout

16.7.2 Production Ventilation Phase I

The production ventilation system is designed as an exhaust system to clear the mine quickly of DPM from diesel equipment and blasting fumes from both development headings and production stopes. The air will be pulled through the Enserch portal and up the raises to the 500 level and exhausted out the portal. The fan will be located 100 feet inside the 500 level portal in a bulkhead with air doors.

The total diesel equipment needed underground for the project is listed in Table 16-4.

Table 16-4 Equipment List

Equip.	Units	Hp	Airflow/Unit
1 yd. LHD	3	62	3,500 cfm
965-7	1	92.5	7,000 cfm
5 ton Loco	1	62	3,500 cfm
4 ton Loco	1	49	3,000 cfm
Mule	3	25	3,000 cfm

The natural ventilation point (NVP) was calculated from the data listed in Table 16-5.

Table 16-5 NVP Calculated Values

El. Enserch Portal 2966 ft.		
Atm. Press.	13.1384	psi
Bp.	26.7497	" hg
d	0.071	
El. Enserch 965 Level 3167 ft.		
Atm. Press.	13.1812	psi
Bp.	26.8368	" hg
d	0.0692	
El. LS 500 Level 3307 ft.		
Atm. Press.	13.0336	psi
Bp.	26.5364	" hg
d	0.0677	

The natural ventilation point for the Lucky Shot mine was calculated at Hn 1.1919 inches of water for the Enserch to the Lucky Shot 500 level. The calculated natural ventilation flow should be 32,772 cfm. This number will be checked against measured values collected from the mine site during construction start up.

The design mine head for the mine at 45,000 cfm is shown in Table 16-6.

Table 16-6 Calculated Mine Heads

	Hs	Hv	Ht
Portal	0.0255	0.0119	0.0374
Existing	1.2721	0.0331	1.3052
Ext 1	0.2725	0.0182	0.2907
Ext 2	0.1433	0.0182	0.1615
Ext 3	0.053	0.0182	0.0711
Vn 1	0.1005	0.0182	0.1187
Vn 2	0.1585	0.0182	0.1766
Vn 3	0.0776	0.0182	0.0958
Raise 1	0.5474	0.0518	0.5992
965 Level	1.4813	0.0973	1.5787
Raise 1	0.627	0.0518	0.6788
LS 500 Level	1.1664	0.0278	1.1942
	Total	**Mine Hs**	**5.9251**
		Mine Hv	**0.0278**
		Mine Ht	**5.9529**

From this calculated head, the mine horsepower can be calculated at 42 Hp. From this information, the mine fan selected is a Jet Air 60 Hp, model M-060-B, 42.5-inch by 21-inch by 1800 rpm fan for the 500 level exhaust.

16.7.3 Ventilation Phase II

During this phase of the Project, the drift to the east on the vein will be extended to underneath the Coleman resource block. The Enserch portal fan will be relocated underground and used for the drive east.

Table 16-7 shows the calculated mine head for the drive east. The air will enter the Enserch portal and be pulled to the vein split, where will be divided into 15,000 cfm directed to the west with 30,000 cfm will be directed East. This air will still go out through a connection off of the ramp to the Lucky Shot 500 Level fan. The calculated fan air hp. is at 34 hp. for this extension of the ventilation system. This is still within the operating range of the mine fan installed at the Lucky Shot 500 Level.

Table 16-7 Calculated Mine Heads East

	Hs	Hv	Ht
Portal	0.0255	0.0119	0.0374
Existing	1.2721	0.0331	1.3052
Ext 1	0.2725	0.0182	0.2907
Ext 2	0.1433	0.0182	0.1615
Ext 3	0.0534	0.0182	0.0716
Vn 1 East	0.2452	0.0182	0.2532
Fault	0.0388	0.0081	0.0469
Vn 3	0.0781	0.0081	0.0861
X-Cut	0.0195	0.0081	0.0276
Ramp 1	0.4867	0.0081	0.4948
Ramp to 500 Level	0.1168	0.0081	0.1248
LS 500 Level	0.8062	0.0129	0.8192
LS 500 Level	1.1621	0.0291	1.1912
	Total	**Mine Hs**	**4.7201**
		Mine Hv	**0.0129**
		Mine Ht	**4.733**

16.7.4 Completed Mine Ventilation System

At this point the mine ramp system under the Coleman resource block will be completed and connection made to the Coleman portal. The main fan will be relocated to this portal area with the mine still on exhaust (Figure 16-12).

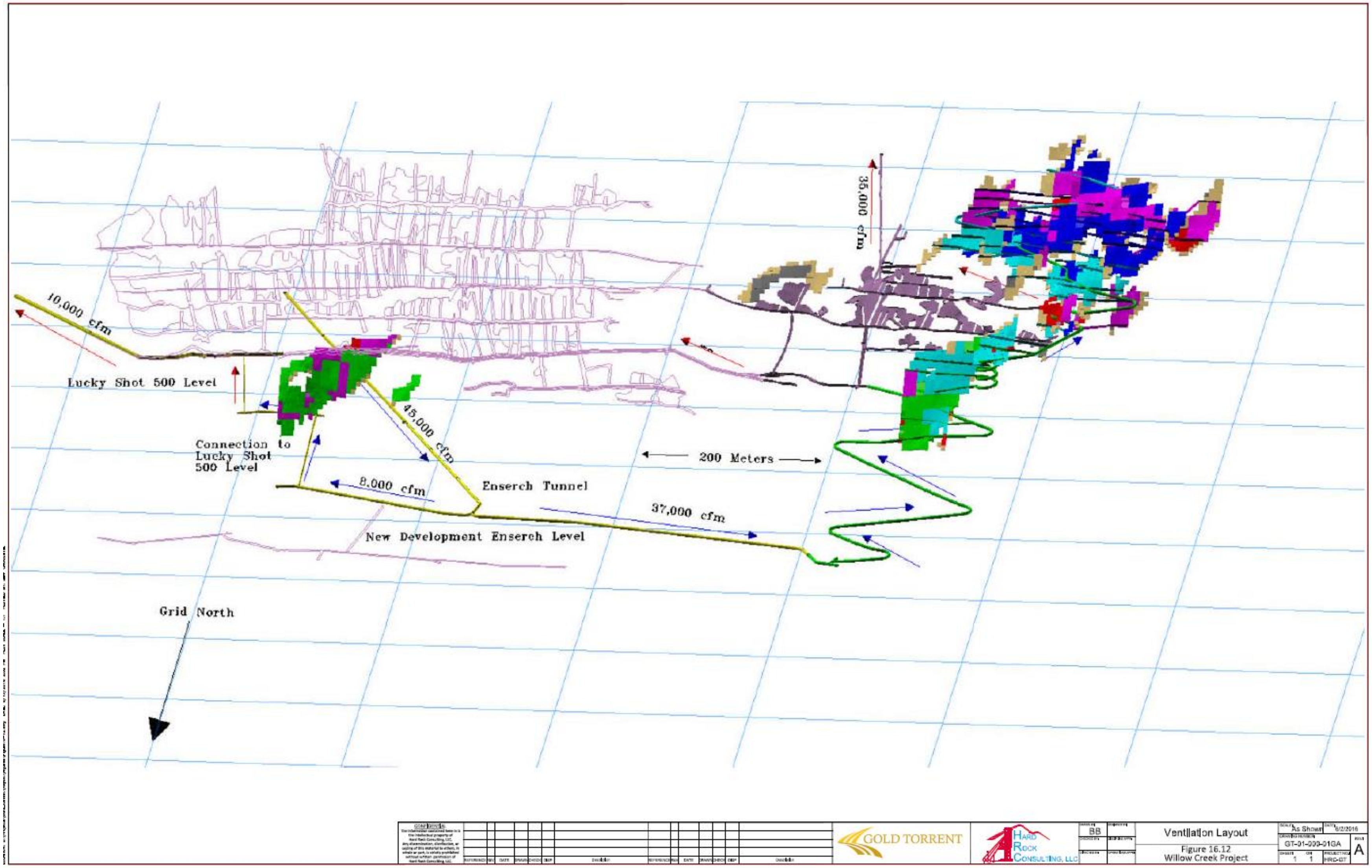


Figure 16-12 Ventilation Layout

Table 16-8 shows the calculated mine head for the completed mine ventilation system. From this the calculated air Hp. is 45 hp. for the finished mine layout.

Table 16-8 Calculated Mine Head Final

	Hs	Hv	Ht
Portal	0.0259	0.0119	0.0378
Existing	1.2936	0.0331	1.3268
Ext 1	0.2771	0.0182	0.2953
Ext 2	0.1458	0.0182	0.1639
Ext 3	0.0543	0.0182	0.0725
Vn 1 East	0.3792	0.0123	0.3915
Fault	0.06	0.0123	0.0723
Vn 3	0.1207	0.0123	0.133
X-Cut	0.0302	0.0123	0.0424
Ramp 1	0.7528	0.0123	0.7651
Ramp 2	0.4363	0.011	0.4473
Ramp to Coleman	0.1403	0.011	0.1513
Coleman	2.5497	0.0379	2.5876
	Total	**Mine Hs**	**6.266**
		Mine Hv	**0.011**
		Mine Ht	**6.277**

From this the final mine head, the exhaust fan that is currently installed on the LS 500 Level can be moved to the Coleman portal level of the mine.

Table 16-9 shows the calculated velocities for the mine along with the K factors used.

Table 16-9 Mine Velocities and K Factors

	V	K adj.
Portal	450 fpm	1.69E-08
Existing	750 fpm	1.69E-08
Ext 1 X-Cut	556 fpm	1.69E-08
Ext 2 X-Cut	556 fpm	1.69E-08
Ext 3 X-Cut	556 fpm	1.69E-08
Vn 1 Drift	457 fpm	1.86E-08
Fault	457 fpm	1.86E-08
Vn 3 Drift	457 fpm	1.86E-08
X-Cut	457 fpm	1.86E-08
Ramp 1	457 fpm	1.86E-08
Ramp 2	432 fpm	1.86E-08
Ramp to Coleman	432 fpm	1.69E-08
Coleman	828 fpm	1.86E-08
Vn 1	556 fpm	1.83E-08
Vn 2	556 fpm	1.83E-08
Vn 3	556 fpm	1.83E-08
Raise 1	938 fpm	1.66E-08
965 Level	1,286 fpm	1.79E-08
Raise 1	938 fpm	1.63E-08

16.7.5 Conclusions Regarding Ventilation

The mine ventilation system for the Lucky Shot project is currently based on values that are calculated; when actual values are collected the mine ventilation system will be reevaluated based on the collected data. The current calculations result in one 60 hp. Fan that is required to handle the desired flow rate of 45,000 cfm for the mine ventilation system. After mining is completed in the Lucky Shot vein area, the fan will have to be relocated to the Coleman portal for the remainder of the mine life.

16.8 Mining Equipment

The mining method will utilize slushers and jackleg drills to mine the rock. The ore will be loaded and hauled to the rail haulage level where diesel power locomotives will pull ore cars on track out of the mine. Table 16-10 is a list of underground equipment that is needed for the Project.

Table 16-10 Mining Equipment

Type	Number
Underground haul truck	1
1 Cu. yd. LHD	3
Kawasaki Mule	3
4 ton Diesel Locomotive	1
6 ton Diesel Locomotive	1
2 ton rocker dumps	5
64 Cu. Ft. Granby	10
Jack leg drills	18
12 B rail mucker	1
30 Hp. electric slusher	2
18 Hp. pneumatic slusher	4
Tuggers	3
Air Saws	6

16.9 Compressed Air and Electrical Services

When the mine is at full production the mine will be serviced by two 1,000 cfm capacity air compressors. Both compressors will be located underground in the Enserch adit, one just inside the portal and the other located further back by the underground shop. The air distribution network will consist of 8-inch main air lines, with 4-inch air lines serving the stoping areas.

The mine power requirements will be serviced by two 500 kw diesel generators at 480 volts. This voltage will be stepped up to 4160 volts for distribution underground, where it will be stepped back down to 480 volt power.

16.10 Manpower

The plan is to operate the mine on two 10 hour shifts 4 days per week. This schedule allows the mine to produce at 200 tonnes per day will minimizing the generating costs plus the labor costs. Table 16-11 shows the general crew mix for the mine.

Table 16-11 Mining Department Manpower

Manpower Summary		Min	Max	Average
Mining G&A				
Mine Superintendent	Mine G&A	1	1	1
Mine Foreman	Mine G&A	0	2	2
Mine Clerk	Mine G&A	0	1	1
Mine G&A		1	4	3
Development - See Development Sheet				
Miner 1	Development	0	2	2
Miner 2	Development	0	2	2
Miner 3	Development	0	2	2
Development		0	6	6
Sub Level Dev - See Development Sheet				
Miner 1	Raises & Levels	0	2	2
Miner 2	Raises & Levels	0	2	2
Raises		0	4	4
U/G Production				
Miner 1	UG Stoping	1	7	4
Miner 2	UG Stoping	1	7	4
Miner 3	UG Stoping	1	7	4
Stoping		3	21	13
Haulage				
Miner 3	Ore/Waste Transport	2	2	2
Miner 4	Ore/Waste Transport	2	2	2
Haulage		4	4	4
Mine Services				
U/G Helper - Supplies&Cleanup	Mine Services	0	2	2
Mine Maintenance				
Lead Mechanic	Mine Mntnc	1	1	1
Heavy Equipment Mechanic	Mine Mntnc	1	1	1
Mechanic Helper	Mine Mntnc	0	1	1
Mechanic/Welder	Mine Mntnc	0	1	1
Electrician	Mine Mntnc	1	1	1
Total Mine Maintenance		3	5	5
Total Mine Operations		15	45	32
Engineering				
Sr Mining Engineer	Engineering	1	1	1
Chief Surveyor	Engineering	1	1	1
Engineering		2	2	2
Geology & Grade Control				
Sr Geologist	Geology	1	1	1
Ore Control Geologist	Geology	0	1	1
Geology		1	2	2
	Total Mine Eng & Geo	2	4	4

16.11 Blasting and Explosives

After ore and waste material has been drilled by the jack leg drills ANFO or dynamite will be loaded into the holes. Ms excel nonel or handidet detonators will be used to set off each round. Development drift rounds will have the perimeter arch holes loaded with 500 grain detonating cord to create a smooth profile that will reduce blast damage and reduce the number of rock bolts required. Production blasting will only load

2 to 3 sticks of powder per hole in order to lightly blast the vein out while reducing excessive blast damage to the hanging wall. The explosives magazine will be located initially outside in one of two locations. Site 1 one is close to the War Baby waste dump and site 2 is in the area of the old yarder location.

17. RECOVERY METHODS

The gravity gold recovery plant includes the following processes: crushing and screening in closed circuit to minus 1/8-inch, desliming screw classifiers plus slimes processing, dual-jig and magnet separation, fine sizing screens yielding 10 process streams, spiral concentrators, middlings regrind and re-sizing, concentrating tables, and refining furnaces. Each process step rejects clean tailings while upgrading the remaining material into increasingly higher-grade ore. Final plant product is gold dore bullion. Figure 17-1 shows a simple line diagram of the plant flow sheet and Figures 17-2 through 17-4 show the plant and crusher layouts.

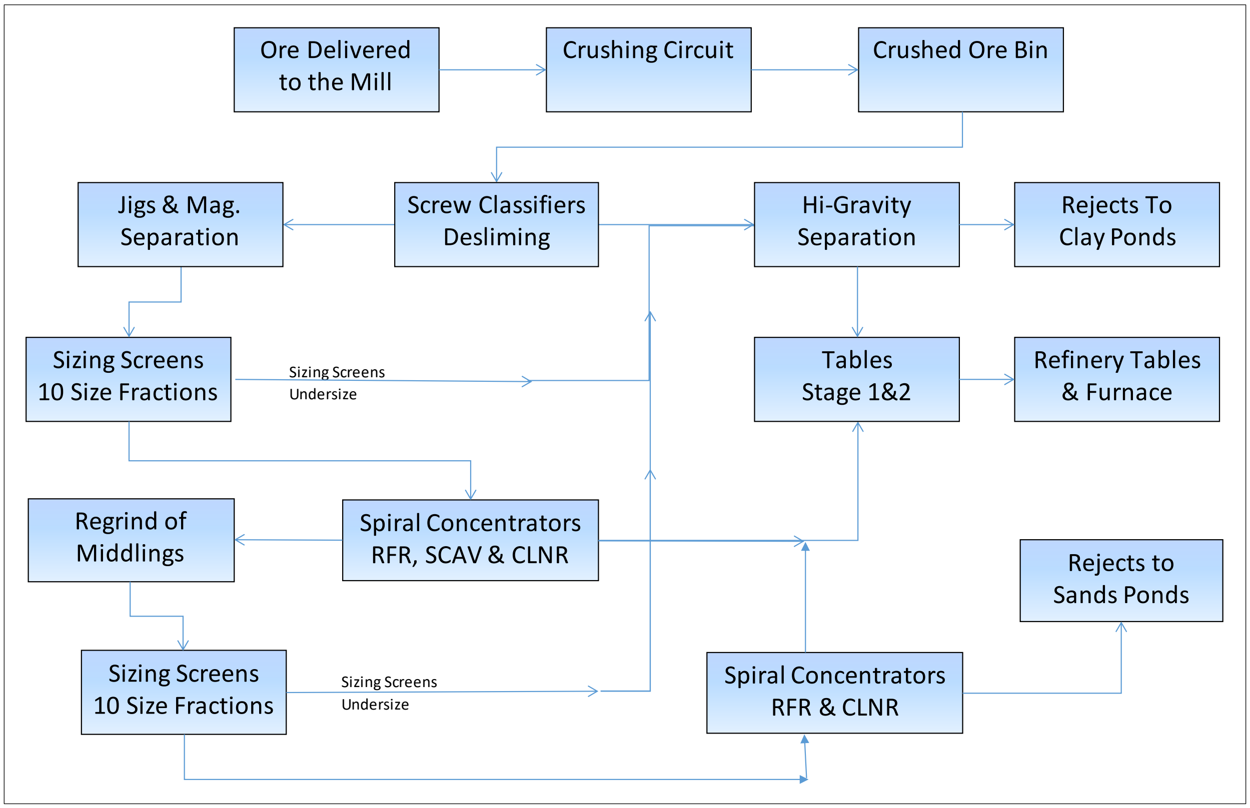


Figure 17-1 Simplified Flow Sheet Diagram

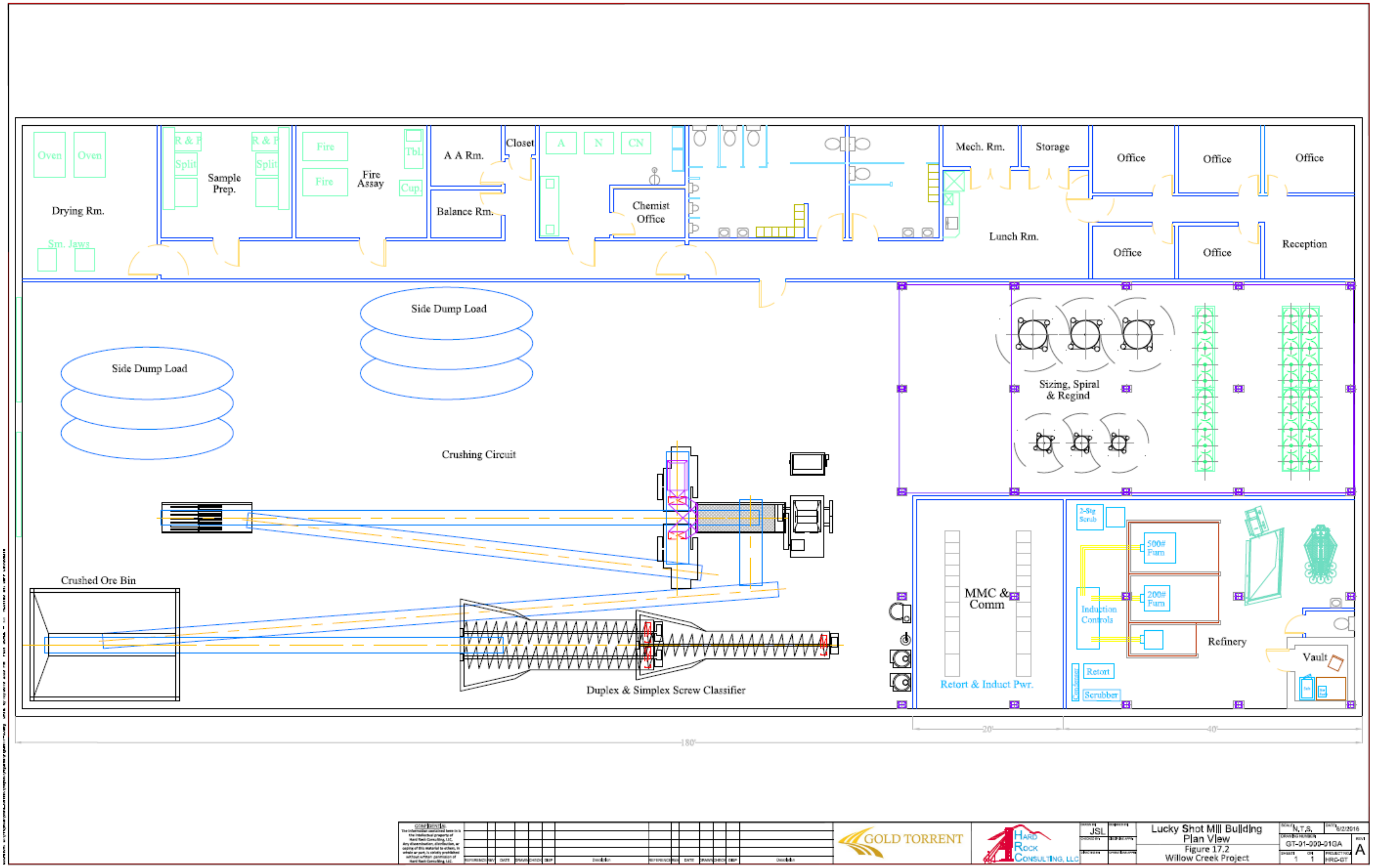


Figure 17-2 Mill Building Plan

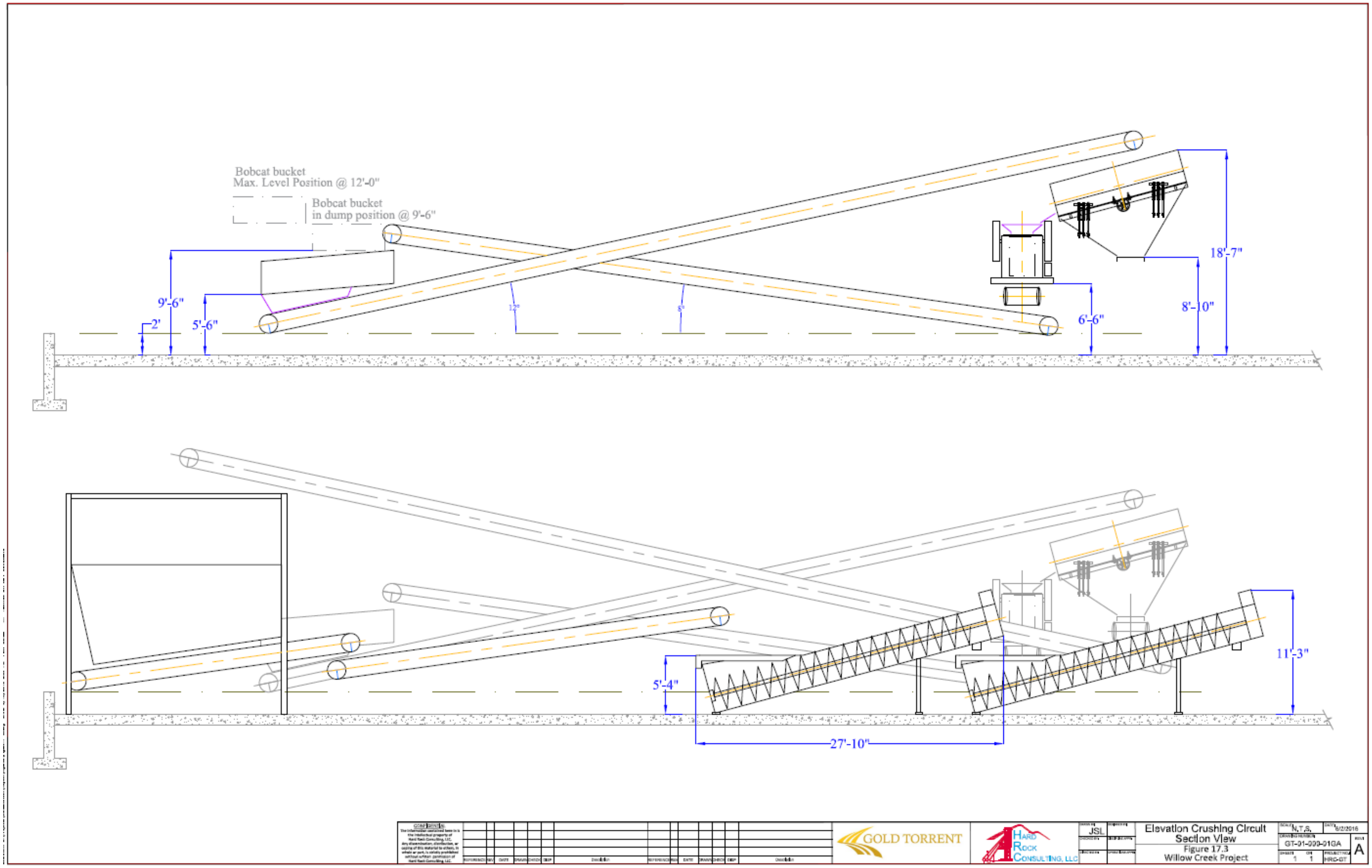


Figure 17-3 Crushing Plant Elevation

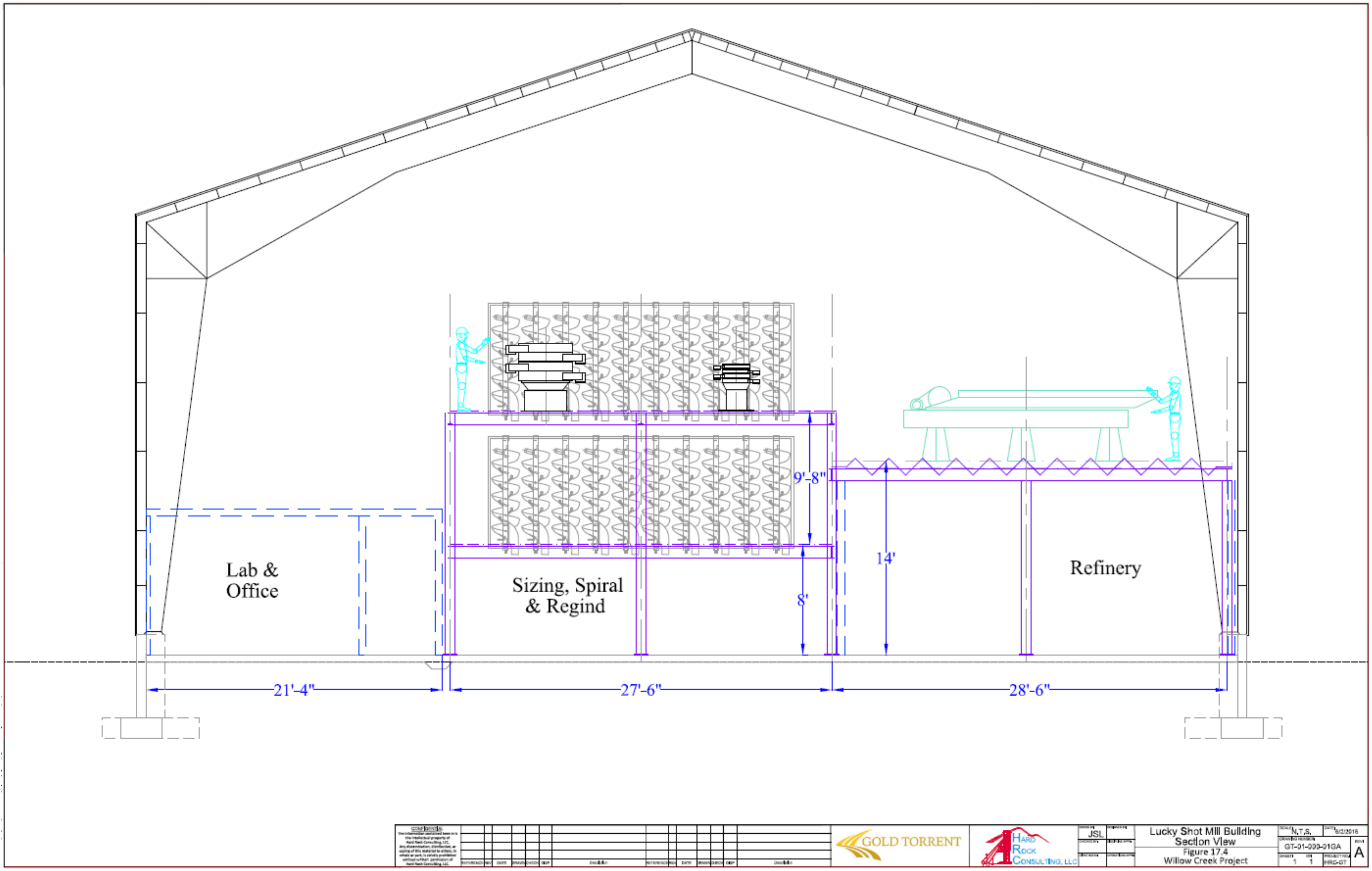


Figure 17-4 Mill Building Cross Section

The plant flow sheet and mass balance calculations are based on the mine's planned maximum output of 200 metric tonnes of ore per day with an average grade of 15.6 g/t Au. At the mine, the ore passes over a scalping screen to limit the maximum size of the run-of-mine ore shipped to the plant to 12 inches. The plant operates 24 hours per day, 5 days per week, 50 weeks per year, for a total of 250 days per year. The maximum ore received at the plant annually is 50,000 tonnes containing 780,000 grams of gold. The mass balance data is used to determine and refine the sizes and numbers of equipment and work force required.

As ore advances through the plant, screening and gravity separation causes barren waste rock to be continuously separated and rejected to tailings. The process reduces the balance of material that requires further treatment at each successive stage. As such, the size and capacity of subsequent unit operations is sized to handle the reduced tonnage for the downstream processing. The work force count is relatively static throughout the plant. Each successive process area raises the grade of the concentrate material and the requirement for operator attention increases. To optimize equipment and work force utilization, the plant process steps are routinely analyzed. Operator training is a vital part of the startup and ongoing operations to develop advanced skill levels and instill teamwork.

The plant site includes sand ponds and clay storage ponds for all rejected material. The site has adequate long-term storage capacity for the total mine plan ore tonnes, plus space for continued operations as exploration discovers additional ore.

17.1 Crushing Circuit

The crushing circuit includes three-stage crushing in a closed circuit with a triple-deck screen. The equipment is sized so that one 8-hour shift will crush the daily ore tonnes and store this material in the fine ore bin ("FOB"). The downstream concentration processes draw material from the FOB as required. For example, the crushing circuit could operate during the graveyard shift to ensure that the FOB is full at the beginning of each ore-processing day.

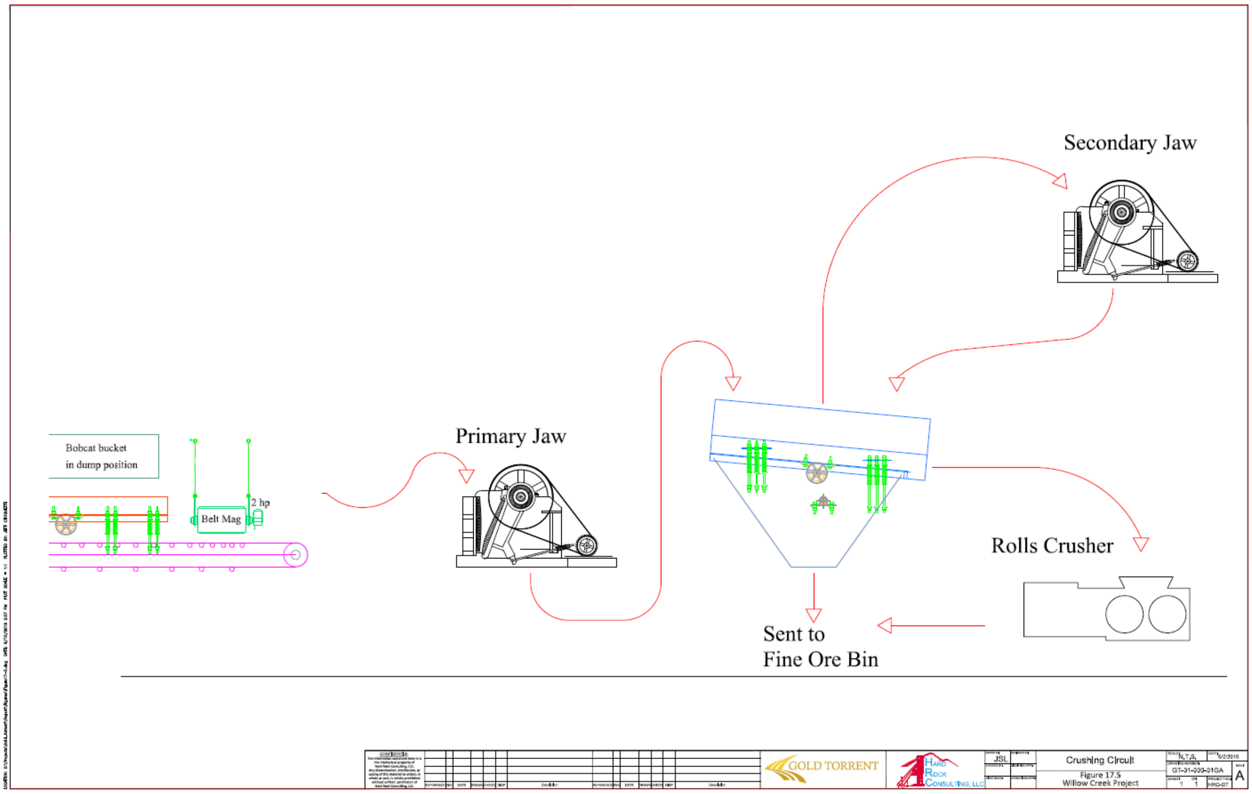


Figure 17-5 Crushing Circuit

Work force requirements for the crushing circuit are three operators working graveyard shift. Duties include a Bobcat operator to feed the primary jaw crusher; an operator to monitor and control the tramp steel magnets, crushers, screen, conveyors, and FOB; and one helper.

The crushing circuit is a dry process. The delivered run-of-mine ore contains approximately 7% moisture by weight. Additional water is added at key points in the dry circuits for dust control. The ore delivered to the FOB is limited to 13% moisture by weight.

Water pumped from the groundwater well and from the recycle area of the sand ponds, as required, is added to the remaining circuits.

17.2 Desliming Circuit

The desliming circuit removes all very fine slimes and clay material less than approximately 325 mesh size. The desliming circuit is fed by a variable-speed conveyor from the FOB. The circuit has two stages of screw classifiers with a stage-one duplex screw classifier and stage-two simplex screw classifier. Each of these classifiers is equipped with variable-speed drive motors to enable precise control. Each classifier will have clean process water pumped into the bottom of the classifying tub. The water flow is carefully metered through control valves to force the fine clay material to overflow the classifier tub. The desliming removes clay that would cause the slurry viscosity to be too high in the downstream circuits, hindering particle segregation based on specific gravity.

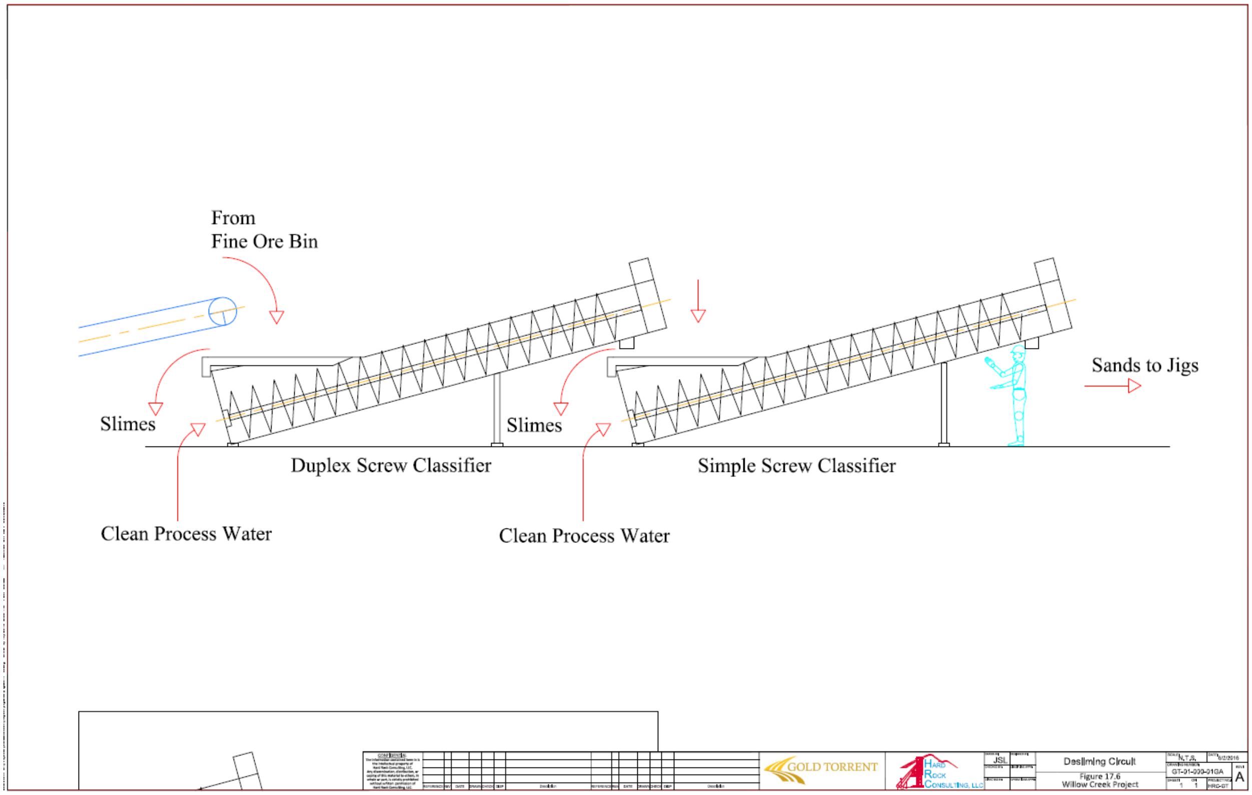


Figure 17-6 Desliming Circuit

17.3 Duplex Jigs Circuit

Ore from the mine is known to host bonanza grade stopes containing nuggets of gold. Visible coarse gold is quite common. In the plant, ore from the desliming circuit, sized to minus 1/8-inch, is passed through two 48-inch duplex jigs for the capture of coarse gold nuggets. These jig concentrates are delivered directly to the Refinery. The jig tails feed the remaining plant circuits.

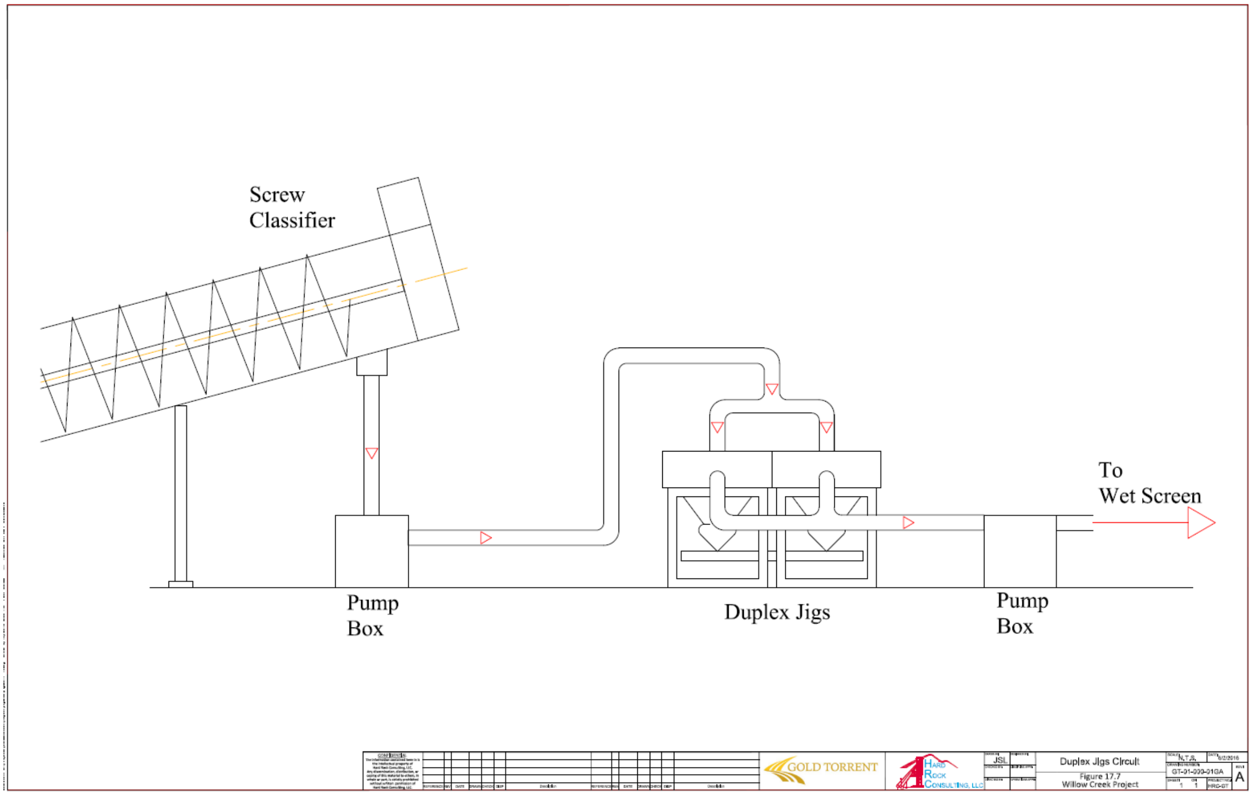


Figure 17-7 Duplex Jigs Circuit

17.4 Wet Screen and Rolls Crusher

A wet rolls crusher with a variable-speed drive provides a fourth stage of size reduction. This stage of crushing is fed with material passing a single-deck, 12-mesh polyurethane screen. The screen undersize is a slurry that contains all ore that has been crushed to minus 12-mesh. The plus 12-mesh screen oversize is wet (approximately 25% moisture), and the wet rolls crusher can accept this ore feed. About 25% of the tonnes produced from the desliming circuit will require wet rolls crushing. A variable-speed drive motor limits the amount of particle breakage. Material from the wet rolls crusher advances to magnetic separation, then 48-inch sizing screens, and then to the spiral concentrators.

The wet rolls crusher reduces minus 1/8-inch material to minus 12-mesh. Magnets are installed ahead of the rolls crusher to protect the rolls crusher from tramp metal.

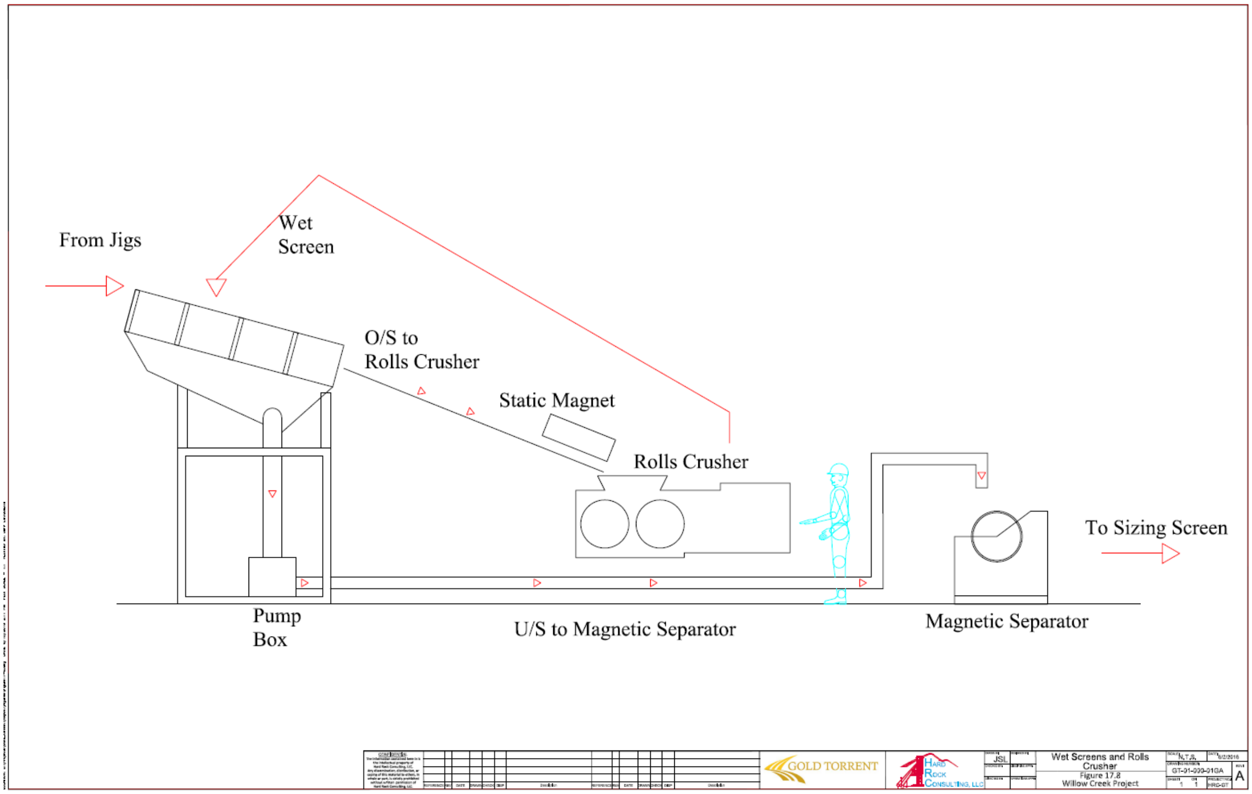


Figure 17-8 Wet Screen and Rolls Crusher

17.5 Slimes Processing

The slimes material that overflows the screw classifiers is fed to enhanced-gravity concentrators ("EGC") for recovery of gold. Hazen Research test data indicates that 4% by weight of the total feed to the desliming circuit is rejected as slimes. This material includes natural clays plus fines generated in the crushing circuit. The gold contained in the slimes is about 2% of the total contained gold.

Two EGC units operate in series to optimize gold recovery of the very fine gold in this material. The recovered material is sent to the refinery while the EGC tailings report to the slimes tailings ponds. Equipment for this area includes a vertical tank pump, cyclone, and EGC. The cyclone removes excess water to create the correct solids/density for feeding the EGC. Product batches from the EGC are upgraded on a Gemeni table in the refinery.

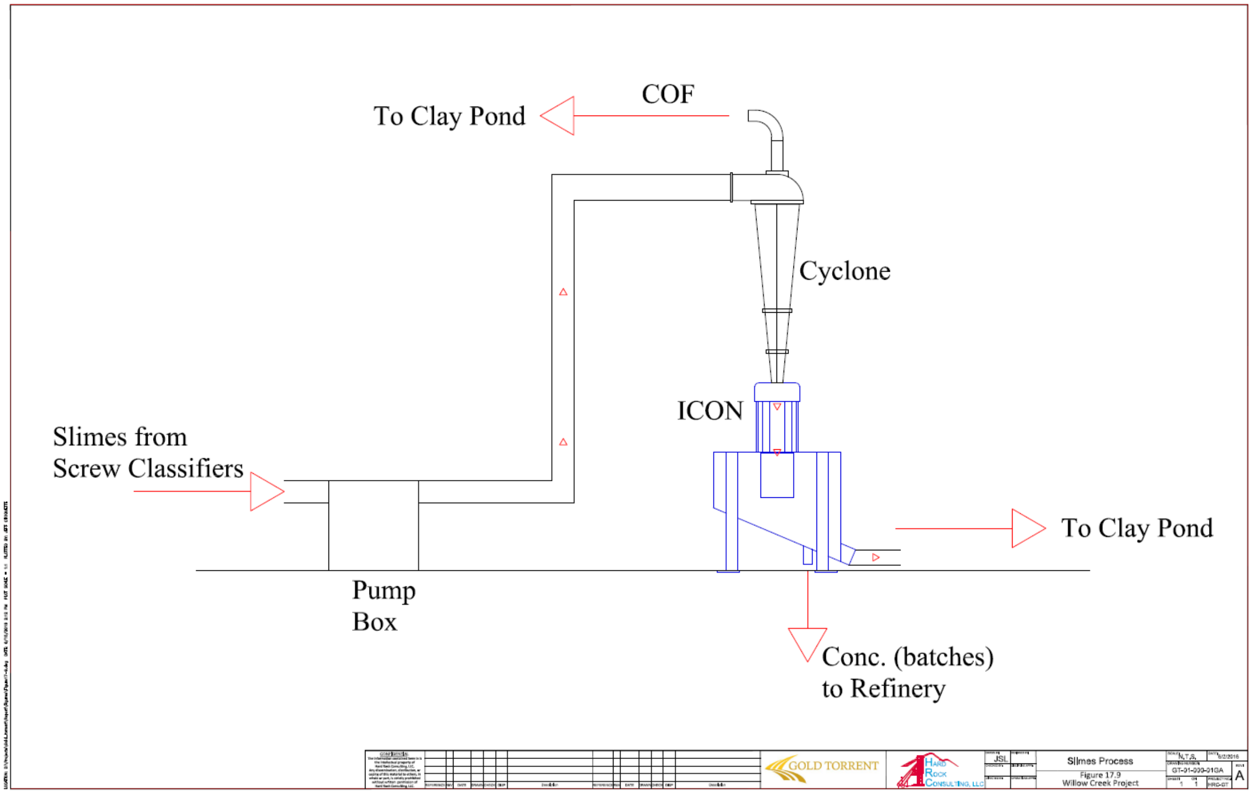


Figure 17-9 Slimes Process

17.6 Sizing Screens

Ore received from the desliming circuit and the wet rolls crusher is sent to three circular sizing screens. These are high-speed screens, 48 inches in diameter, with three decks each. Nine specific particle sizes streams are produced from the sizing screens. These nine streams are processed separately in the downstream spirals and tables. The last screen underflow creates a 10th stream, as an EGC feed.

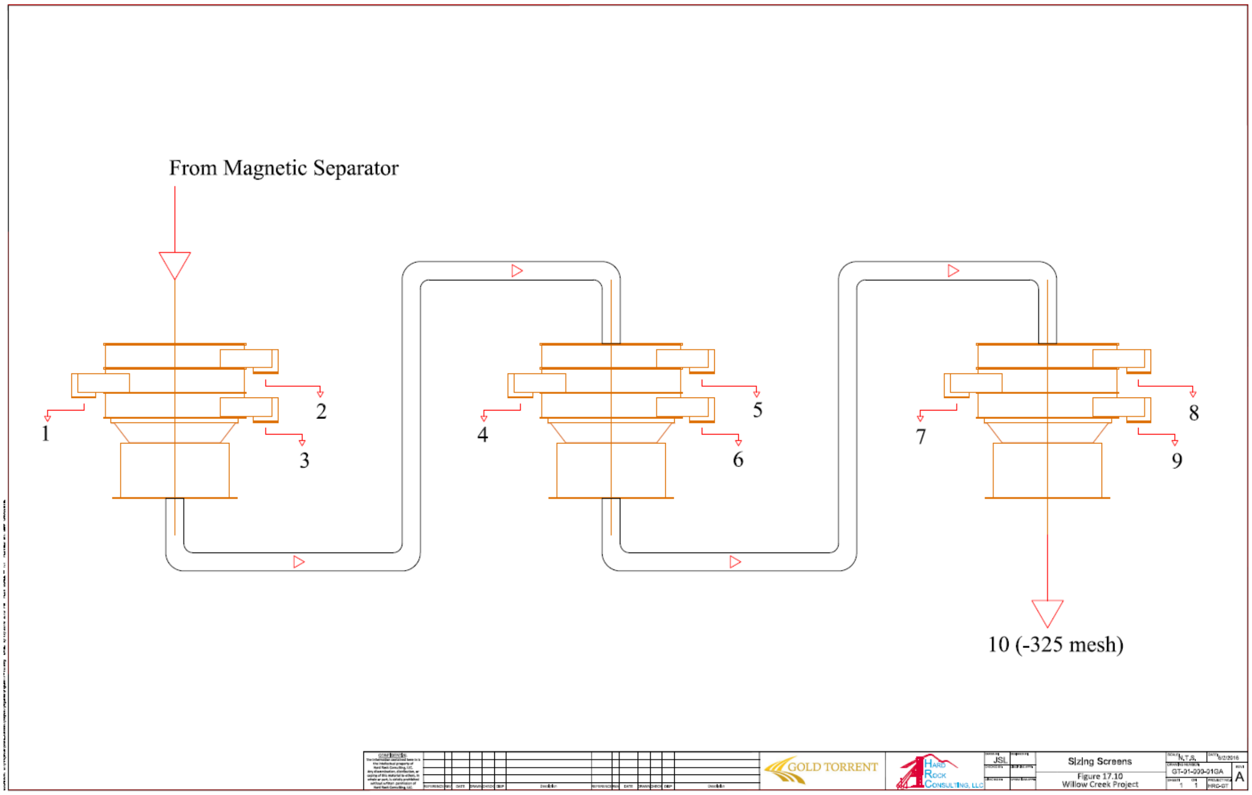


Figure 17-10 Sizing Screens

Together with desliming, the careful preparation of discrete particle sizes is a key criterion of the plant flow sheet design.

Recent Hazen test work demonstrated that the combination of careful desliming and screening led to excellent segregation of barren rock from the ore particles that contain either sulfides or gold. The water that moves the ore particles is extremely clear, since all slimes and clays have already been removed. Visual control of the spirals and tables is very effective under these conditions. The "colors" of all particles are obvious and allow segregation of four product streams. The largest stream is tailings, which are white to light cream in color. The next largest stream is middlings, which generally look like brown sands. The next, relatively small stream is black sands, which are predominately the sulfide minerals. The final small stream is gold, and looks like it, though some gold will be unliberated and combined with quartz. The black sands and the gold go directly to the refinery furnace.

17.7 Primary Spirals - Roughers

Sixty spiral concentrators are located on a deck above the floor of the mill. The floor has tanks for collecting the products from each spiral and pumps for delivering slurry flows to the feedboxes on top of each spiral.

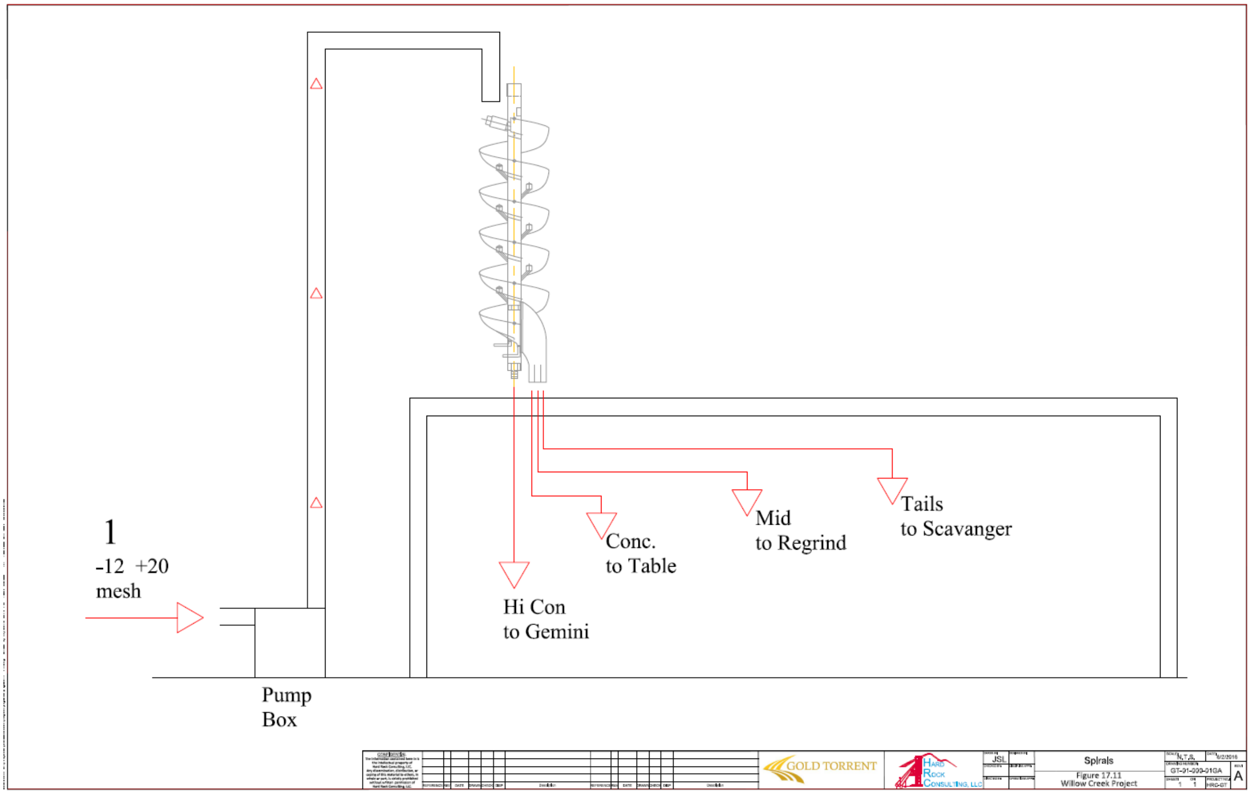


Figure 17-11 Spirals

The first concentrator group is the rougher spirals. Each rougher spiral creates a cleaner feed concentrate ("cons"), a middle material ("mids") and a scavenger feed ("tails"). The rougher tails discharge by gravity into a surge tank on the mill floor. From this tank the slurry is pumped to a scavenger spiral that processes the corresponding particle size. The scavenger spiral creates a mids and tails. The tails are waste rock that is pumped outdoors to the sands tailings pond.

The cons from the rougher is a small flow and is collected and stored until adequate tonnage is available to run a cleaner spiral. The cleaner spiral produces cleaner cons, mids and cleaner tails. The cleaner cons are feed for the tables circuit. The cleaner tails are feed for the cleaner scavenger.

The cleaner scavenger creates mids and tails. The tails are waste rock that is pumped outdoors to the sands tailings pond.

The mids created in each spiral are collected and re-processed. The first six size fractions are sent to a rolls crusher regrind circuit. The next three size fractions are processed by size fraction on smooth deck tables.

17.8 Regrind

The mids from the Primary Spirals circuit that process the first six size fractions are re-processed in the rolls crusher regrind circuit. These mids contain gold particles that are locked inside the particles of ore and waste rock. The rolls crusher creates the size reduction necessary to liberate gold and to minimize creation of fines.

After rolls crushing, the ore from these mids is passed over 24" diameter sizing screens to establish new size fractions. After regrinding in the rolls crusher, a high percentage of mids are in the fourth, fifth and sixth size fractions and no ore remains in the first three size fractions.

17.9 Secondary Spirals – Scavengers and Cleaners

The secondary spirals operate the same as the primary spirals; however, the mids in fourth through sixth size fractions are not returned to a regrind circuit but instead advance to the tables circuit for processing.

The work force for the complete spiral section is four individuals. Duties include a floor helper to control the pumps and ponds; a spiral/wet screen operator; and two spiral operators. The spiral operators are the senior operators and are responsible for setting the cut streams on the spirals and ensuring maximum recovery of the gold streams while minimizing gold losses to the tailings.

17.10 Tables Circuit

Deister and Gemeni tables, of various styles and sizes, process material received from the spiral circuits. The primary tables circuit includes three full-sized ribbed tables for processing ore in the first six size fractions, four full-sized smooth tables for processing ore in seventh through ninth size fractions, and 2 Gemeni 1000 tables for enhanced upgrading.

The ribbed tables yield three streams: cons, mids, and tails. The smooth tables produce two streams: cons and tails. The Gemeni table produces four streams: cons, black sands, brown sands, and tails.

The table circuit processes material in all nine size fractions, and streams exiting each table are kept segregated for varied downstream processing.

For the tables circuits, gold recovery is set at a 3-fold increase in the con grade grams per tonne over the feed grade grams per tonne. This concentration ratio is conservative. Feed rates are provided from data received from the table vendors.

17.11 Refinery

The refinery contains two Gemeni tables and three induction furnaces (one 500-pound, one 200-pound and one 50-pound steel capacity crucible). Two power packs control the three furnaces. A 75 kW power pack controls the two smaller furnaces and a 125 kW power pack controls the larger furnace. The furnaces produce gold dore bullion, the final plant product.

Three individuals are required to operate the refinery including a refinery engineer, flux/slag helper and a Gemeni 250 operator.

The refining engineer is responsible for all decision-making, record keeping, and controlling of the fluxing throughout the different furnaces. The large furnace requires different fluxes than the smaller furnaces.

The 200-pound furnace will require multiple fluxing when treating high-grade cons or cleaning up bars from the large furnace. The smallest furnace is used to create finished dore bars.

17.12 Mass Balance Consolidated Flow Chart

Table 17-1 is a consolidated summary of the ore tonnes and grade of gold at each process staged through the plant. Table 17-2 displays the tonnes and grade of gold delivered to the refinery. The expected final gold recovery is 91.8%.

Table 17-1 Calculated Tonnes, Grade and Recovery by Plant Circuit on a Daily Basis

Circuit Start	Start mt	Circuit Finish	Finish mt	Start g Au	Finish g Au	Start g/tn	Finish g/tn	Au %Rec by circuit
Ore rec'vd	200.0	Crushing	200.0	3120.0	3120.0	15.6	15.6	
Crushing	200.0	FineOreBin	200.0	3120.0	3120.0	15.6	15.6	100.0%
Classifiers	200.0	Ore to Jigs	192.0	3120.0	3057.6	15.6	15.9	98.0%
Cycl O/F	8.0	Clay Pond	1.0	62.4	3.1	7.8	3.1	
EGC 1 Feed	7.0	EGC 2 Feed	6.8	59.3	26.7	8.5	3.9	
EGC 1 Con	0.2	Refinery	0.2	32.6	32.6	163.1	163.1	
EGC 1 Tails	6.8	EGC 2 Feed	6.8	26.7	26.7	3.9	3.9	
EGC 2 Feed	6.8	Clay Pond	6.7	26.7	12.0	3.9	1.8	
EGC 2 Con	0.1	Refinery	0.1	14.7	14.7	146.9	146.9	75.8%
Jigs	192.0	Ore to Rolls	190.1	3057.6	2904.7	15.9	15.3	95.0%
Jigs Con	1.9	Refinery	1.9	152.9	152.9	79.6	79.6	100.0%
Wet Rolls	190.1	Sizing Screens	190.1	2904.7	2904.7	15.3	15.3	
Spirals	190.1	Table1	35.1	2904.7	2812.1	15.3	80.2	96.8%
Table1	35.1	Table2	22.9	2812.1	2742.6	80.2	119.6	97.5%
Table2	12.6	Refinery	8.8	1902.8	1872.6	150.8	212.7	98.4%
Scav	4.1	Recycle	1.0	47.4	40.3	11.6	39.3	85.0%
EGC 3&4	9.6	Refinery	1.2	124.3	107.4	13.0	90.5	86.4%
Gemeni	4.4	Refinery	2.6	1187.6	1163.8	271.8	443.9	98.0%
Refinery	9.3			2865.0			**Overall Recovery:**	91.8%

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Table 17-2 Calculated Tonnes and Grade Delivered to Refinery on a Daily Basis

				tonnes	gAu	gAu/t
Table 1	FS Refinery Furnace			0.82	489.4	598
Table 2	FS Refinery Furnace			1.27	904.2	712
Gem tables	Gem250 Cons	Refinery	Size 1-3	0.02	16.8	840
Gem tables	Gem250BlkSand	Refinery	Size 1-3	0.04	14.7	368
Gem tables	Gem250BrnSand	Refinery	Size 1-3	0.06	9.6	161
Gem tables	Gem1000 Cons	Refinery	Size 4-6	0.17	315.8	1898
Gem tables	Gem1000BlkSand	Refinery	Size 4-6	0.33	276.3	830
Gem tables	Gem1000BrnSand	Refinery	Size 4-6	0.50	181.6	364
Gem tables	Gem1000 Cons	Refinery	Size 7-9	0.25	142.5	569
Gem tables	Gem1000BlkSand	Refinery	Size 7-9	0.50	124.7	249
Gem tables	Gem1000BrnSand	Refinery	Size 7-9	0.75	81.9	109
ICON	Comb. EGC 1 & EGC 2 Conc		Slimes	1.45	47.3	33
ICON	Comb. EGC 1 & EGC 2 Conc		Size 7-9	1.19	107.4	90
Duplex Jig				1.92	152.9	80
	Total Refinery			**9.27**	**2865.0**	**309**

17.13 Laboratory

The laboratory is located at the plant site and includes sample prep, fire assay, wet lab AA and ICP equipment. The laboratory requires ten individuals for operation including: one manager/chemist, two sample receivers, four sample preparers, two fire assayers and one wet lab technician. The laboratory operates two shifts per day (day shift and swing shift). In addition to mill samples the laboratory will also supply the mine with assays for ore control, exploration and waste rock verification.

17.14 Work Force

Total gold recovery plant work force count is 31 for mill operations, 10 for the laboratory, and 5 security guards.

17.15 Power, Water, and Supplies

The gold recovery plant is located next to a Matsu Electric Authority power line. The plant is connected to the line via a small step down transformer and utilizes line power to operate the motors, pumps, crushers and other electrical demands. The step down transformer and associated connections are included in the project. Power costs for line power are estimated to be $0.085 per kWhr. Peak demand for the plant is 800 kw and total power consumed is between 900 and 1500 kW per hour.

Groundwater is abundant at the plant site and is supplied by a shallow well. Make up water will be reclaimed from the tailings and slimes settling ponds.

Process materials and supplies are oil and lube related to the servicing of the crusher equipment. The balance of service operating costs are for ongoing service and replacement of the numerous small pumps. Electrical repair supplies are used to service motors that drive the crushers, desliming equipment and tables.

17.16 Recovery Methods Conclusions

The gold recovery plant is a dynamic gravity recovery system that incorporates zero chemical reagents. The net results are clean tailings and will result in a relatively simple permitting process with associated government regulators.

The gold recovery system segregates ore into nine distinct size fractions. These size fractions are maintained throughout the spirals and table circuits. The flow sheet results in increased gold recovery within the spirals and tables circuits while reducing overall tonnage being processed. The splits flowing off the spirals and tables are managed visually by color.

18. PROJECT INFRASTRUCTURE

The infrastructure requirements for the Project are not extensive given that the mine has had past operations and the gold recovery plant will be located on the Parks Highway next to a transmission line. The mine also benefits from having access to Tract C, which is land and facilities that the Owner/Lessor has next to the mine. Tract C includes, among other things, a metal shop building and a metal mill building, one Cat 966 loader, man-camp for 40 people, Conex containers with new jackleg drills, slushers, mine fans and bagging, roof bolts, safety equipment, four diesel generators, 10,000 fuel storage tank, rail locomotives and 2-tonne cars, and a mill building that has storage areas and silo, crushers, and tanks.

The Project is accessed from Parks Highway via an existing road (Willow Fishhook road) that is paved for the first 11 miles and is improved gravel road for the remaining 13.5 miles. The Matsu Borough provides annual maintenance for the road year round for the paved portion, including snow removal and during the non-winter months for the non-paved portion. For the non-paved portion of road, the Project will keep the road open during the winter for the months starting late approximately November through approximately late April. The Project will employ a Cat 966 loader to remove snow and improve the road areas where required.

The road to the gold recovery plant from the mine from the Willow Fishhook Road turnoff is another 4.3 miles north on Parks Highway. Figure 18-1 shows the general layout of the mill area facalities and infrastructure.

18.1 Dumps

A waste rock dump will be constructed near the Enserch portal area for the disposal of mostly granite waste rock generated by the mine development process. The waste dump area can contain over 200,000 tonnes of waste rock stacked along the contour of the area. The waste dump will initially contain about 30,000 to 50,000 tonnes.

18.2 Stockpiles

Run-of-mine ore will be stockpiled at a loading area near the portal for loading into 10-tonne haul trucks. The stockpile will contain enough ore material to allow the mine and gold plant to operate at a normal schedule. A stockpile of run-of-mine ore will be maintained in the onsite existing mill building on Tract C. This stockpile will allow the mine to continue operating during possible weather events that may temporarily shut down the road access due to snow. A small stockpile of ore will be maintained at the gold recovery plant to accommodate surges and shortages in feed material due to weather and mining issues.

18.3 Tailings Disposal

The gold recovery plant will be disposing of two tailings products in two separate facilities: clay fines and quartz material in several size fractions. The clay fines tailings will be pumped from the gold recovery plant to the small, incised pond area. Since the quantities are small (5,000 tonnes per year) and the material is inert clay, the pond will not require lining and will be used to decant process water for recycling. Ultimate closure of the clay pond will occur when the pond is filled and a new pond is incised. The closure will include a cap of soil.

The quartz sands tailings disposal will occur in a larger pond that is also incised to contain several years of tailings production. These tails are also inert and do not generate any acid or trace elements. The tails will be pumped from the plant to the ponds where they will settle. The water will be decanted and recycled.

18.4 Power

The gold recovery plant will use line power from the Matsu Electric Coop. A three-phase line passes over the gold plant site and a small transformer will be installed to bring the power to the plant. The line power provides a low cost source of power for the gold recovery plant and requires only a few poles and line to connect it to the plant.

The mine will generate its own power using a 500 Kw diesel-fired generator set. The generator and a diesel storage tank will be located at the mine portal area. Additional power and diesel storage is also available from the 100 Kw generator package and a total of 15,000-gallon storage tanks included at the Tract C facilities.

18.5 Process Water

The plant site will have a ground water well to provide initial and make up water for the recovery plant. The process water sent to the tailings facilities will be recycled to be used again in the plant. Fresh water will be provided from the well to the plant washroom and other areas as required. A 40-man dry will be added to the large shop building included on Tract C. Tract C has fresh water wells and a septic system.

18.6 Rail Access

The gold recovery plant site is located very near a railroad siding that can be used for offloading materials during construction. The rail line passes the property several hundred meters from its southern border.

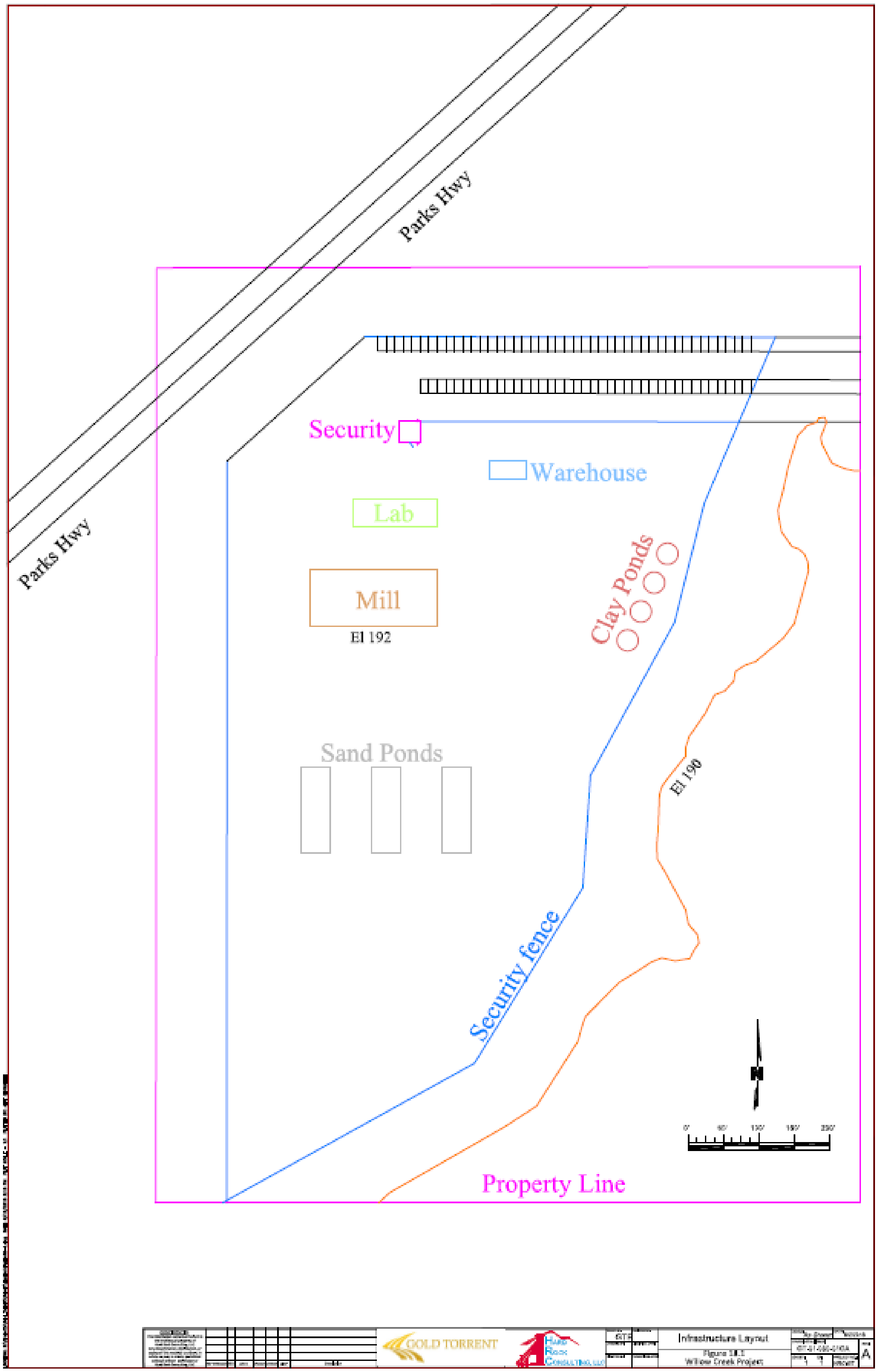

Figure 18-1 General Infrastructure Layout

19. MARKET STUDIES AND CONTRACTS

Gold and silver markets are stable, transparent, global markets serviced by well-known smelters and refiners located throughout the world. Silver and gold will be refined to 0.9999 or 0.99999 purity in the refinery, and, as such, are fungible commodities bought and sold universally. Therefore, no contracts have been negotiated at this stage of the Project. The Project does not have any forward sales of silver or gold, nor does it have any hedging programs in place at this time.

The gold-silver doré produced on site at the Project gold recovery plant can be transported to a number of reputable refiners that can improve the metal product into LME acceptable fineness and sizes for final sale. A long-established, dynamic, worldwide market exists for the buying and selling of gold and silver. It is reasonable to assume that the product from the Project will be salable.

A selling price of $1,175/oz. for gold and $15/oz. for silver has been used to develop this PFS. At the end of May 2016, the 18 month trailing average, as tabulated from public data from the website www.kitco.com, was $1176/oz. for gold and $15.68/oz. for silver. The closing spot price at that time was $1210/oz. for gold and $16.06/oz. for silver. The high closing spot gold price for the 12-month period was $1294, and the low was $1077. The three-year trailing average prices for gold and silver were $1,230 and $17.85 respectively, higher than the selling prices used in this PFS.

Figure 19-1 show the historical gold prices, and Figure 19-2 show the historical silver prices. There exists no method to predict future sales prices of gold and silver.

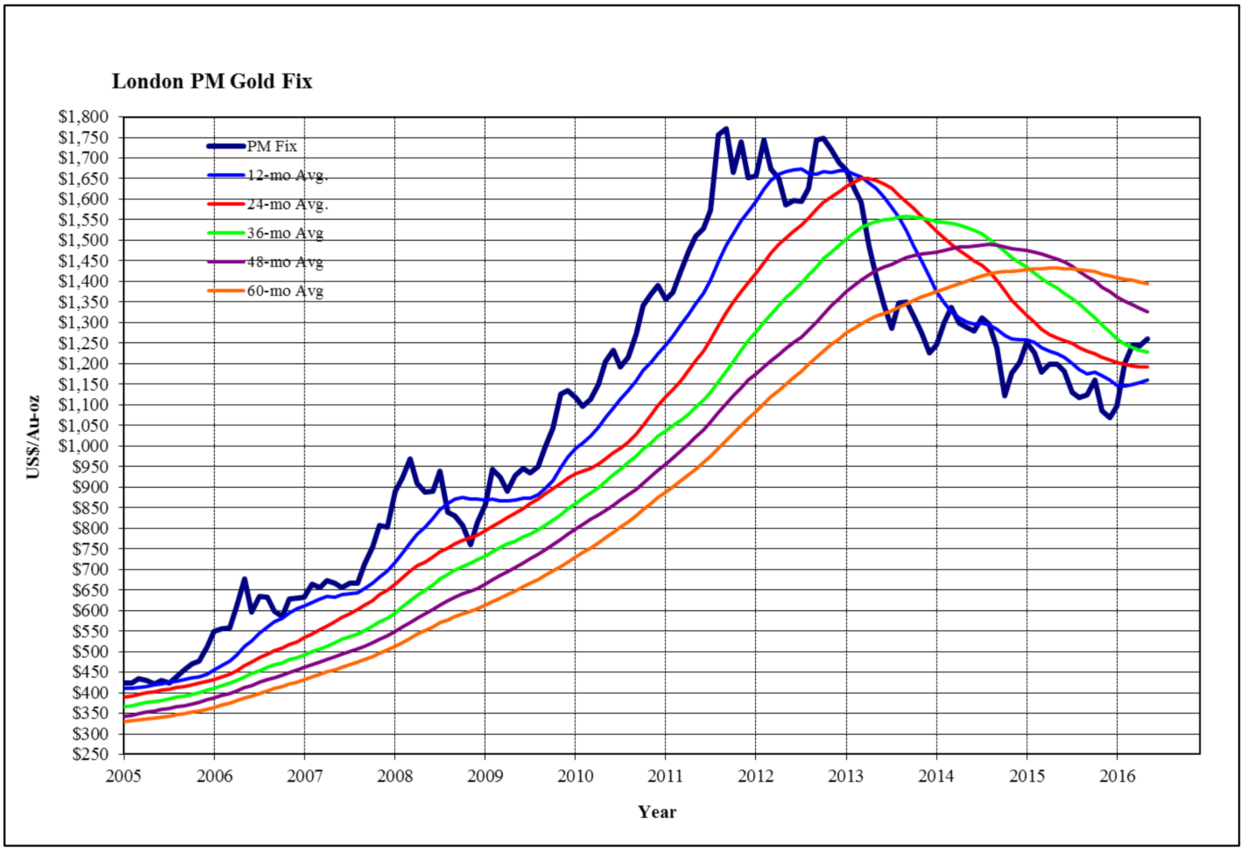

Figure 19-1 Historical Gold Prices

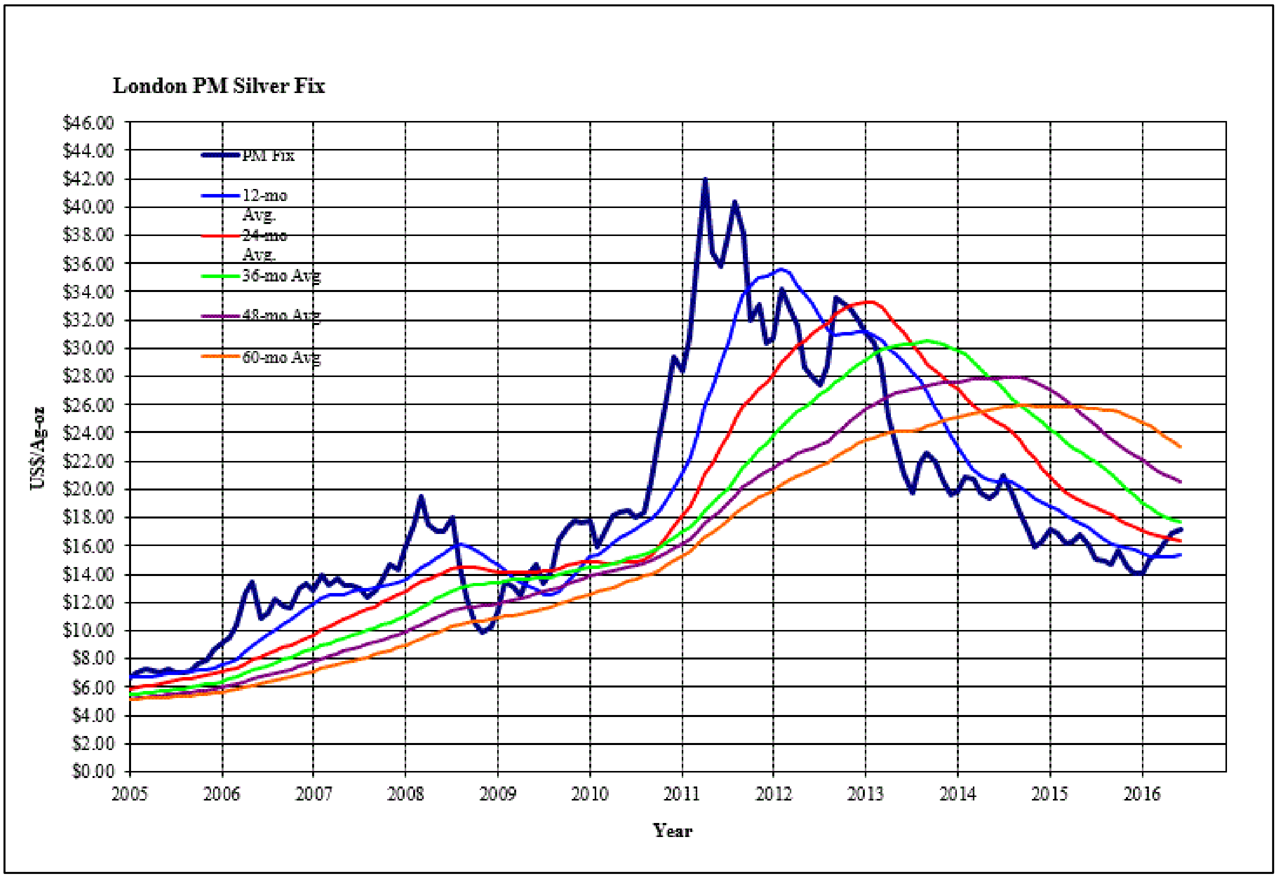

Figure 19-2 Historical Silver Prices

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The deposit at the Project has been the focus of underground exploration and mine planning at various times in the past. In 2008, the Coleman vein was permitted for underground exploration and extraction of a bulk sample. The Bulk Sample permit and preparation for mining at that time resulted in environmental studies to support the permitting and mine design process. There currently exists a wealth of data for beginning mine permitting.

The Project has more in common with a large placer mine than it does with the larger hard-rock projects that exist in Alaska. The ore will be processed using only gravity methods consistent with the testing and methods described elsewhere this report. Chemical processing is not proposed. The development rock and tailings are expected to be geochemically benign in that there is no acid-rock drainage potential, and little potential for metals leaching to create a discharge inconsistent with Alaska's water quality standards. The mine site is located on private land, and the processing site, though not yet purchased, is also expected to be located on private land. Neither the facilities at the mine site nor those at the processing site are expected to be located in areas requiring wetland fill or discharge.

The arrangement of the Project also decreases potential air impacts and the complexity of air permitting. Air quality issues at mines are typically caused by electrical generation and dust. Emissions from electric use at the gold plant processing facility, which are the majority of the Project's electric requirements, are eliminated because the processing facility will be connected to regional power grid. The only generation is at the mine site. Dust is minimized because the actual mining occurs underground, and the crushing and processing occur within a building. There may be some road-related dust issues due to the approximately a dozen round-trip journeys per day on the Willow Fishhook Road up to the mine. The dozen trips include mine workers who will be bussed to the mine, and 2 ore haul trucks conducting some 10 round trips per day. There will be more mine-related traffic on Parks Highway and on the paved, lower section of the Willow Fishhook Road because the gold plant processing facility is located on Parks Highway 4.3 miles north of the Willow Fishhook Road turnoff.

These location and design parameters significantly decrease potential environmental impacts of the Project, and in doing so, they significantly decrease permitting complexity. Without a surface discharge, the Project will not need an APDES (Alaska Pollution Discharge Elimination System) permit from DEC. With these parameters, there is limited federal involvement in the Project. Because there is no federal permit required, there is no formal document under the U.S. National Environmental Policy Act: no federal Environmental Impact Statement, or Environmental Assessment (though the state process investigates significant environmental issues). The Matanuska-Susitna Borough relies on the state to regulate precious metals mining, so no separate mining permit is required from local government. There are significant state permits required, but the fact that the Project is on private land and proposes gravity-only, chemical-free processing significantly decreases the expected complexity of state permits.

An aspect of the Project that potentially increases permitting complexity is the fact that it occurs within the Hatcher Pass area. Hatcher Pass is important for recreation: snow machining, skiing, and summer hiking and mountain biking. It will be important for the company and the agencies to work with the recreational users to ensure that the mine minimizes impacts to them. This may influence mine design, may increase the permitting timeline, and may require mitigation, such as providing alternative access for snowmachiners who currently use the road in winter.

20.2 Available Environmental Information

There is a significant amount of baseline environmental data available for the area of the mine site. Almost all of the data was gathered before June 2009. Some of the conclusions from that data must be confirmed by further sampling; however, the available data decreases the amount of additional sampling required. This section summarizes the available data.[1] The processing property has not been selected, and data for a prospective processing site is not summarized here.

20.2.1 Water Quality

A total of 32 water quality samples have been taken at 8 sites between June 2007 and March 2009: 12 samples spread among three locations on Craigie Creek, 12 samples at three locations on Willow Creek, 4 samples at a location on Shorty Creek, and 2 samples at a small streamlet near Independence Mine. Not all parameters were measured in all samples.

20.2.1.1 Surface Water Quality in Willow and Craigie Creeks

Willow and Craigie Creeks have a long history of historic mining. Tailings, including those contaminated with cyanide, were placed in the stream or floodplain when those practices were legal and typical. Decades have passed since those practices took place, and the water quality in both creeks is excellent and with some minor exceptions meets all of Alaska's water quality standards: average pH is slightly elevated (average of 8.55 which is greater than the maximum standard of 8.5). The few exceedances for metal concentrations appear to be one-time events, rather than systematic exceedance of Alaska's standards.

20.2.1.2 Adit Water Quality

A small discharge from the Enserch adit occurs year-round. While the flowrate was not measured, it was small enough that sampling technicians were forced to walk back into the adit to find a puddle large enough to get enough water for a sample. The Lucky Shot adit appears to discharge, at least to some extent, throughout the summer, though it may cease in the winter. The Coleman adit does not discharge all year. Rather it discharges only during spring run-off and significant rain events. Water from these three historic mine adits were sampled a total of eleven times between June 2007 and August 2008.

[1] Existing baseline information for land and water on the south side of Craigie Creek, the side away from the mine site, is not included in this summery. That information is generally not applicable to the current mine concepts, which do not affect land on the south side.

The water quality from the adits is similar to that of Craigie and Willow Creeks in that they appear to have a marginally high pH: the samples average a pH of 8.48. This value is similar to the waters of Craigie and Willow Creeks. There were a few minor metals exceedances, the most significant of which were two samples with a higher-than-standard concentration of arsenic from the Lucky Shot adit.

Arsenic concentrations appear to vary considerably by adit. Two of the three samples from the Lucky Short portal show very high concentrations, 669 and 661 ug/l. Concentrations form the Coleman adit vary from 11.3 to 80 ug/l. However, the four samples form the Enserch adit show consistently lower concentration with a small variance: from 10.2 to 13.6 ug/l. Arsenic is an unusual metal in that the drinking water standard is significantly lower than the standard for fish growth and propagation, the aquatic life standard. All but the two Lucky Shot adit samples have arsenic concentrations far below the aquatic life standard. All standards are greater than Alaska's drinking water standard of 10 ug/l. However, the four Enserch adit samples are only marginally higher than the drinking water standard. Therefore, it appears that the Enserch drainage will meet Alaska's most stringent water quality standard with only minor amounts of natural dilution. Drainage at the concentrations exhibited in the Enserch adit is not likely to be a significant concern. All other metals exceedances from adit samples were one-time events slightly above state standards.

20.2.1.3 Groundwater Quality

There are 14 wells drilled on the south side of Craigie Creek. Mine activities are not expected to occur on the south side, and so the water quality from those 14 sites are not summarized here. Two wells exist on the north side of the creek. These are just above the Willow Fishhook Road approximately downhill from the Coleman and Enserch adits.

The well downstream of the Enserch adit was sampled in November 2008 and March 2009. The well downstream of the Coleman adit was sampled in November 2008. More sampling is needed to draw firm conclusions from these two wells. Nevertheless, in the three samples, there was only one element that was beyond concentration for the Northside wells: mercury. Each of the three samples was beyond Alaska's water quality limit of 0.05 ug/l. The average of the three samples show 0.23 ug/l. The higher level of mercury is not apparent in the water discharging from the adits nor in the stream. Neither the ore nor waste rock contain significantly high levels of mercury. The mercury does not appear related to the mine site itself. All other parameters were within Alaska's standards.

20.2.2 Wetlands.

Full Metal Minerals commissioned a preliminary investigation of wetlands at the site. The investigation included some areas north of Craigie Creek below the Coleman and Enserch portals. While the information is not yet adequate for a determination by the U.S. Army Corps Engineers as to the extent of wetlands in the area (if any), it does provide an excellent first indication of the lack of wetlands for mine planning. The preliminary investigation, site inspection, and the lack of wetlands on the Matanuska-Susitna Borough's wetland maps help provide confidence in the lack of wetlands above the Willow Fishhook Road. There are likely to be wetlands in the Craigie Creek floodplain, but that area will not be affected by the mine.

20.2.3 Air Information

Hoefler Consulting Group (now owned by SLR Consulting) operated the Lucky Shot meteorological monitoring station located on the small ridge at the old millsite south of Craigie Creek and opposite the proposed mine site. The Lucky Shot Meteorological Monitoring Project was established to collect meteorological data that can be used to conduct the air dispersion modeling required for minor air permitting. The weather station location was approved by the Alaska Department of Environmental Conservation on May 8, 2009, and a Quality Assurance Project Plan was submitted in November 2009. Data collection began at the site on June 1, 2009 and ended on May 31, 2010. The meteorological monitoring station met all Prevention of Significant Deterioration monitoring conditions during the monitoring year.

20.2.4 Geochemical Data

Full Metal Minerals engaged Geochemica to review geochemical issues involving development rock from a prospective incline then planned from the Enserch portal up to the Coleman workings. Specifically, Geochemica analyzed: 1) the potential for the development rock produced from the adit to be acid-generating; and 2) what metals may be leachable from the development rock and at what sorts of characteristic levels.

The report concluded the following: "The FMM geochemical characterization program used stratified random sampling to obtain representative samples of the development rock that would be produced. The geochemical test work establishes that the rock will not be acid generating and, in fact, because of its benign mineralogy should be no more reactive than any of the tonalite that is exposed naturally near the Lucky Shot adit. Trace metal contents are low to very low in the rock, and because the rock will not be geochemically reactive, there should be no significant risk of metals release to the local waters. Exposure to the low levels of metals present in the solid rock will be de minimis because of FMM's plans for controlled management of the development rock." This report demonstrates the geochemically benign nature of development rock for the Project.

Similar evidence indicates the lack of acid-potential and low metals-leaching potential in the potential tailings. The best indication for the low acid-potential is provided by the 60+ years that the adit and workings of the historic mines in the vicinity have been exposed to air and water. The historic workings provide a realistic, full-scale test of acid potential.

There are no iron stains within the accessible portions of the Coleman and Enserch workings. The metal within the workings are rusty, but they show the normal deterioration of being open to air and water, not the accelerated deterioration characteristic of being exposed in an acidic environment. Further, the slow drainage from the adits have an average of pH of 8.48.

Laboratory results confirm the conclusions from the historic workings. In 2014, McClennand Laboratories, Inc. was contracted to analyze two samples of the ore. The acid base accounting results showed an AGP/ANP ratio of 2.61 and 3.06, for the two samples. Actual results of the tailings are likely have significantly less acid-generating potential because the proposed gravity processing will remove most of the sulfide minerals along with the gold. This processing will decrease or close to eliminate potentially the acid-generating elements from the tailings.

McClennand Laboratories also completed a TCLP test from two samples of ore for Arsenic, Barium, Cadmium, Chromium, Lead, Mercury, Selenium, and Silver. Results indicated that neither of the samples would be characterized as hazardous waste according to EPA RCRA criteria. While these criteria allow higher concentrations than does Alaska Water Quality Standards, the TCLP procedures uses water with pH < 2 which will accelerate leaching over a real-world situation with neutral water. Also, the results may over-estimate leaching because the proposed gravity processing will eliminate much of the most leachable elements, especially arsenic, from the tailings.

20.3 Permitting Requirements

This section describes required government authorizations and costs. To the extent practical, most of these authorizations are applied for, reviewed, and processed as a group by the agencies. Because they are processed as a group, most will be distributed to the public for comment together. The costs listed under each of these authorizations is the cost hiring a specialized consultant to prepare the permit application. General permitting costs – working with the agencies and the public, and preparing the remaining permits, is included in section 20.5 Summary of Permitting Costs. Costs that are a part of mine design and engineering are not included.

20.3.1 DNR Plan of Operations, and Reclamation Plan Approval.

These two authorizations are DNR's major authorizations for operation of the mine. The authorizations have considerable overlap.

The Plan of Operations applies only to state land. The vast majority of the proposed Project is on private land: patented mining claims at the mine itself, and private property at the processing site. Thus, the plan of operations will apply only to operations that spill over to public land at the mine site. If the mine proposes activities on state land, such as a trail for snow machines to compensate for plowing the road in winter, the plan would include these activities. The Plan of Operations approval balances the applicant's right to extract the minerals with effect on public resources. For state land, DNR has the authority under the plan of operations to stipulate changes in the design and operation of the mine to protect public resources.

The Reclamation Plan provides DNR authority to review operations to ensure that they comply with state's reclamation law, AS 27.19.20: "A mining operation shall be conducted in a manner that prevents unnecessary and undue degradation of land and water resources and the mining operation shall be reclaimed as contemporaneously as practical with the mining operation to leave the site in a stable condition." For hard rock mines, implementing DNR's authority under the law typically requires them to review the mine's plan of operations.

Both of these authorizations give DNR authority to stipulate mining operations to protect public resources (or the very similar prevention of "undue degradation of land and water resources"). While neither the plan of operation nor the reclamation plan have statutorily required public notice, DNR will likely distribute the draft authorizations for public comment for the Project.

The authorizations may consider the breadth of issues but the major issue is almost always water quality during and after mining. Because of the type of processing proposed and the generally benign nature of the ore and development rock, these may not be difficult issues.

20.3.1.1 Summary: DNR Plan of Operations and Reclamation Plan

- Agency: DNR Division of Mining, Land and Water
- Description: Review mine operations to protect public resources on state land; assure stable post-mining condition on public and private land.
- Environmental Background Information Required: A variety of studies may be required. However, the most important studies are the geochemistry and hydrologic studies, much of which has already been completed.

20.3.2 DNR/DEC Reclamation Bond

AS 27.19.040 directs DNR to require a Reclamation Bond: "an individual financial assurance in an amount not to exceed an amount reasonably necessary to ensure the faithful performance of the requirements of the approved reclamation plan." The Reclamation Bond is not a separate authorization. It is required by DNR under their Reclamation Plan authorities and by DEC under the authority of the solid waste permit. However, it is processed on a different schedule from the other authorizations, and so it is considered separately.

DNR and DEC jointly calculate the financial assurance necessary to reclaim the site and to complete any required post-mining water quality treatment (unlikely for this site), water quality monitoring, and site maintenance. DNR typically holds the bond for both agencies. The bond is usually processed after the remaining permits and may not require public notice.

20.3.2.1 Summary: Reclamation Bond

- Agency: DNR Division of Mining, Land and Water; DEC Division of Environmental Health
- Description: Required financial assurance to assure mine closure leaves the site in a stable state that meets water quality requirements.
- Background Information Required: None. Applicant submits calculations to support an estimate of the cost of reclamation and required monitoring. The agencies review and make a final determination of the estimated cost.

20.3.3 DNR Water Right or Temporary Water Use Authorization

A water right or temporary water use authorization from DNR is required before withdrawing a significant amount of surface or ground water. DNR conditions those permits to protect other water right holders (not likely to be a problem for the Project); other water users (also not likely to be a problem), or the environment which is typically fish habitat. A water right is a long-term or permanent property right to the water. A temporary water use authorization is for a use of less than 5 years. A temporary water use authorization does not convey a property right. Typically, a mine will require water rights for their permanent use of water, such as for processing, and temporary authorizations for some other uses, such as road construction, temporary dust control during construction, or similar uses.

A significant amount of water is defined in regulation (11 AAC 93.970) as more than 5,000 gallons per day from a single source; recurring use of more than 500 gallons per day for more than 10 days per year from a single source, or the non-consumptive use of more than 30,000 galls of water per day from a single source, or any water use that might adversely affect the water rights of other appropriators or the public interest.

A water right requires public notice but a temporary water use authorization does not. Some water will be needed at the mine site for drinking, and possibly for drilling. Therefore, the Project will require a water right for withdrawing groundwater from a well to supply these uses. Most of the water required for the processing site will be recycled. Water for start-up or make-up will be taken from a groundwater well from what appears to be a near-surface aquifer in an area not near other water users. The hydrologic investigation required for the DNR Plan of Operations or DEC Waste Disposal Permit will typically be adequate for water rights and water use authorizations for the Project.

20.3.3.1 Summary: Water Use Authorizations

- Agency: DNR Division of Mining, Land and Water
- Description: Water Right or Temporary Water Use Authorization
- Background Information Required: Hydrologic information gathered for other permits will likely be adequate for these authorizations.

20.3.4 Cultural Resources

The State Historic Preservation Office (SHPO) within DNR's Division of Parks and Outdoor Recreation has jurisdiction over effects to historic and cultural resources on state land.

SHPO staff work with state agencies during the early stages of project planning to protect cultural resources. They do this by providing information on the location of known sites and information from cultural resources surveys previously done in an area. If the potential to discover unknown sites is high, an on-the-ground survey may be recommended. When there are sites in a project area, SHPO staff determines National Register eligibility, how the project will affect sites, and methods to minimize or mitigate unavoidable damage.

The Willow Creek Mining District, including the Craigie Creek Valley was an important gold producing area in the first half of the 20th century. Photos of the area from the early 1900s show many buildings that are no longer visible. If there is significant disturbance on state land, SHPO may require cultural resources clearance (an on-the-ground survey) and possibly mitigation for any impacts to cultural or historic resources. Stipulations for the mitigation would most likely be applied to the DNR Plan of Operations.

20.3.4.1 Summary: Cultural Resources

- Agency: DNR Division of Parks and Outdoor Recreation, State Historic Preservation office; U.S. Army Corps of Engineers
- Description: Cultural Resources Clearance
- Background Information Required: If SHPO determines that there is likelihood of significant historic or cultural resources on state land that will be disturbed by the Project, they may require an on-the-ground investigation of known and potential sites. However, the fact that most of the Project is on private land decreases the likelihood and scope of such an investigation.

20.3.5 DEC Air Quality Permit

Air quality effects are regulated by the Alaska Department of Environmental Conservation, Division of Air Quality. The activities that typically trigger the need for an air permit for a mine of this type are electrical generation and crushing. The mine site is expected to have a 500 kW generator at the mine site with a similar generator for back-up. Both are not expected to be used at the same time. The power is used to power operations at the mine site. While DEC has not been approached for a final determination, it is unlikely that this level of power generation would require an air permit.

The processing site has a larger electrical requirement than the mine site. However, the processing facility requirements will be provided by connecting to Matanuska Electric Association grid.

With respect to activities at the processing location, DEC requires a minor air permit for activities that involve a rock crusher with a rated capacity of greater than five tons per hour. The processing facility is expected to have a crusher of that size or greater, though the crusher will be enclosed within a building. The processing facility will also include a furnace. The furnace will operate a few times per week. The crusher and the furnace will trigger the need for an air permit from DEC.

Typically mines have dust control measures on unpaved roadways. The mine is expected to require less than a dozen round-trips for personnel busses or mine trucks on the unpaved section of the road between the processing facility and the mine site. This level of traffic is likely to decrease the existence or intensity of best management practices for road-generated dust.

20.3.5.1 Summary: Air Quality Permit

- Agency: DEC Division of Air Quality
- Description: Air Quality Permit (minor permit)
- Background Information Required: None (However, a meteorological station with one-year's record is available at the mine site.)
- Other: Best management practices to control dust may also be required.

20.3.6 DEC Solid Waste Permit and Management Plan

The major issue with respect to the tailings and waste rock is the potential for water quality effects on surface or groundwater from the placement of tailings.

A solid waste permit is required for the tailings facility whether it a dry-stack tailings or disposed in a tailings lake. DEC has the authority under the Solid Waste Permit to require financial assurance from the company. This requirement overlaps DNR's authority to require a reclamation bond under its reclamation authorities. The bond is described previously in this section.

DEC also has the authority but not the mandate to require a solid waste permit for the placement of waste rock. DEC typically only requires a solid waste permit for waste rock if the rock has the potential to generate acid or significant metals leaching. If these do not occur, DEC may determine that a solid waste permit is unnecessary for the waste rock placement. With respect to development rock placed outside the Enserch adit, the relatively small volume, the benign nature of the rock, and the lack of potential for acid rock drainage or significant metals leaching make it unlikely that DEC will require a permit for the waste rock. DEC is required to distribute a solid waste permit for public comment.

20.3.6.1 Summary: Solid Waste Permit

- Agency: DEC Division of Environmental Health
- Description: Authorization for the placement of tailings (discretionary on the part of DEC for the placement of waste rock).
- Background Information Required: Understanding the potential water quality from the tailings and hydrologic investigation to characterize existing ground and surface water will be required. Some of this information is already available.
- Other: Monitoring will be required.

20.3.7 DEC Stormwater Pollution Prevention Plan

DEC requires a plan with best management practices to control stormwater. The Project qualifies for a DEC general permit on stormwater control: 2015 Multi-Sector General Permit for Storm Water Discharges Associated with Industrial Activity (2015 MSGP, AKR060000). The general permit requires that the operator have a Stormwater Pollution Prevention Plan (SWPPP) at the site that complies with the general permit and DEC requirements. To qualify for the general permit, the operator must submit a notice of intent to comply with the general permit to DEC and receive confirmation of coverage under the general permit.

There is an existing SWPPP for the mine site. It was written for the general permit that preceded the current permit, which became effective February 1, 2016.

The Project must update the SWPPP for the mine site, and prepare a new SWPPP for the processing site.

20.3.7.1 Summary: Solid Waste Permit

- Agency: DEC Division of Water
- Description: Prepare a Stormwater Pollution Prevention Plan for the mine site and processing site.
- Background Environmental Information Required: None.

20.3.8 EPA Spill Prevention, Control and Countermeasure Plan (SPCC Plan)

EPA requires that every facility that stores more than 1,320 gallons of fuel above ground must have a SPCC plan on-site for that facility. The plan is not submitted to EPA, but the operator must have the plan on-site and be following the plan. The plan is not "approved" by EPA and preparation of the plan is not considered a major federal action; therefore, this requirement does not trigger application of the National Environmental Policy Act. Put another way, this requirement does not require a federal Environmental Assessment or Environmental Impact Statement. These plans are frequently prepared by a specialized consultant.

20.3.8.1 Summary: SPCC Plan

- Agency: Not submitted for agency approval but required by the U.S. EPA
- Description: Spill Prevention, Control and Countermeasure Plan (SPCC Plan).
- Background Information Required: None.

20.3.9 Alaska Department of Transportation Maintenance and Road Use Agreement

The Willow Fishhook Road is a public road maintained by the Alaska Department of Transportation. The Project proposes to use road-legal cars and trucks on the road. Workers at the mine site will begin and end their work day at the processing facility and will be bussed to the mine site. There will be likely be either two 10-hour shifts or three 8-hour shifts per day. Therefore, there will be between 2 and 6 bus round trips per day (depending on the number of busses per shift). Ore will be transported roughly twice per day from the mine site to the processing facility. All told, there will be probably a dozen round-trips per day on the road between the processing facility and mine. That use does not require authorization from the Department. However, the road is closed and unmaintained in winter. To allow the Project to keep the road open, DOT will most likely require a road-use agreement that specifies the manner in which the Project will maintain the road and allow public access. It may also include a cost-sharing agreement by which the Project agrees to provide funding to maintain the road.

The road is used by recreational snowmobilers in the winter. They use the road to access the Hatcher Pass area, which is popular recreation destination. The Project may be required to provide alternative access for snow machines and possibly a parking area, so that the winter use of the road does not create a safety hazard or foreclose the snow machine access to the area.

20.3.9.1 Summary: Road Maintenance and Use Agreement

- Agency: Alaska Department of Transportation

- Description: This is not a permit. Rather, it is maintenance and potentially a funding agreement with DOT to keep the road open in the winter.
- Background Information Required: None.

20.3.10 U.S. Army Corps of Engineers Jurisdictional Wetlands Determination

The U.S. Army Corps of Engineers permit under Section 404 of the Clean Water Act requires an authorization (wetlands permit) before allowing discharge of fill into waters of the United States, including wetlands.

The Project is not anticipated to affect wetlands. However, the Project will be required to outline wetlands (or lack of wetlands) in the area to ensure that jurisdictional wetlands are not affected. The Corps provides detailed methodology for identification of wetlands under federal jurisdiction. The Project must provide the Corps with a map and supporting data so that the Corps can certify that the Project is outside of wetlands under their jurisdiction. The Corps does not distribute a jurisdictional determination for public comment.

20.3.10.1 Summary: Jurisdictional Wetlands Determination

- Agency: The Project expects that it will not place dredged or fill material into wetlands and therefore will not be subject to a U.S. Army Corps of Engineers Wetlands Permit
- Description: Prepare a wetlands analysis to allow the Corps to confirm that there are no dredged or fill material being placed in wetlands.
- Background Information Required: Wetlands map and supporting report.

20.3.11 U.S. Fish and Wildlife Service: Endangered Species Act, Bald Eagle Protection Act, and the Migratory Bird Treaty

The U.S. Fish and Wildlife Services oversees the endangered species act for upland species (i.e., not including the marine species under the jurisdiction of the National Marine Fisheries Service). The service has listed eight species in Alaska as threatened or endangered. None of their habitats exist in the Project area. Thus, the act has no application to the Project. The Bald Eagle Protection Act is not an authorization but a prohibition against affecting nests or harassing Bald Eagles. Unless there is an eagle nest on the processing property, there is little application of the act to the Project. The Migratory Bird Treat prohibits certain practices during certain time periods that may affect migratory birds. The Project will be required to comply with these acts, but there is not permit required, and the jurisdiction of the Fish and Wildlife Service is not a major federal action and does not trigger application of NEPA.

20.3.12 Matanuska-Susitna Borough

The Matanuska Susitna Borough is the local government with jurisdiction over the Project. The Borough does not regulate mining. Therefore, a permit from the Borough is not required.

20.4 Environmental Baseline Data: Hydrology

Water quality information is usually the single most important part of baseline data for mine permitting. Established locations are sampled during operation and for some period after mine closure. Typically, the agencies may require a year of pre-mining baseline. Sometimes some of that data is taken while construction or permitting is occurring. A significant amount of data exists for the mine site, but little or no data exists for the processing site. Usually, each water quality sample is tested for a full suite of water quality parameters to prepare a baseline, then a much smaller suite is analyzed based on the potential discharge and on the results of the first samples. For purposes of this study, the cost estimate includes a year of quarterly water quality data (i.e., four samples/year) for a full suite of parameters at the processing site, but a reduced suite at the mine site where data already exists.

At the mine site, the cost estimate includes sampling two established locations on Craigie Creek: one above and one below mining. It also includes sampling the two groundwater wells below the Coleman and Enserch Mine adits, and sampling discharge from the Enserch adit. The cost estimate includes four sampling episodes at the five sites (two on Craigie Creek, two wells, and the Enserch adit).

Forthe processing site, there are two sampling sites for each stream: one upstream one downstream of mining. For groundwater, there are two wells down gradient and one upgradient well of the plant – one in each of the two stream drainage basins with a year of quarterly sampling at seven locations (four surface water and three groundwater wells).

Table 20-1 Required Government Authorizations

Agency	Permit Title	Notes
DNR/DMLW	Plan of Operation/Reclamation Plan	General protection of public resources on state land and reclamation on public and private land.
DNR/DMLW & DEC/Env Health	Reclamation Bond	Financial assurance to ensure reclamation, provided to DNR before the project may is begin operations.
DNR/DMLW	Water Right or Temporary Water Use Authorization	Approval to withdraw and use water.
DNR/DPOR/SHPO	Cultural Resources	Documents and mitigates impacts to cultural resources on state land.
DEC/Division of Air Quality	Minor Air Permit	Assures air quality compliance at the processing facility.
DEC/Division of Environmental Health	Solid Waste Permit and Management Plan	Ensures water protection. Required before placement of tailings. Discretionary for placement of waste rock.
DEC/Division of Water	Stormwater Pollution Prevention Plan	Control over stormwater discharge.
U.S. EPA	Spill Prevention, Control and Countermeasure Plan	Required for fuel storage. Response plan.
DOT	Road Maintenance and Use Agreement	Maintenance and user agreement for keeping road open in winter
U.S. Army Corps of Engineers	Jurisdictional Wetlands Determination	Federal determination of the extent of wetlands. Project is not intending to discharge fill to wetlands.
U.S. Fish and Wildlife Service	ESA, Bald Eagle Protection Act; Migratory Bird Treaty	These are rules that apply throughout Alaska. No specific permit required.

20.5 Socioeconomic and Community Use Information

20.5.1 Community Use Information

The Hatcher Pass Management Area includes over 300,000 acres of the Talkeetna Mountains stretching from the Matanuska Valley north to the Kashwitna River drainage. The Project is located in a small part of the area along the Willow Fishhook Road. The Alaska Department of Natural Resources adopted a land-use plan for state land in the Management Area. The plan directs of DNR management of state land, but does not control activities on private land. DNR adopted the most recent plan in November 2010, and amended it in March 2012.

The plan describes the public's use of the area in general. "Other than mining, the Hatcher Pass area is primarily used for public recreation, which has increased significantly over the last 25 years since the initial plan was prepared. The open, rolling terrain, steep-walled valleys and jagged peaks provide a variety of recreational opportunities. Popular summer uses include sight-seeing, photography, hiking, river kayaking, mountaineering, hunting, fishing, horseback riding, hang-gliding, rock collection, off-road vehicle use in designated locations, and berry picking. Winter use includes various types of cross-country skiing, telemarking, boarding, sledding, and down-hill skiing, dog mushing, and snow-machining. Both summer and winter use has increased dramatically, but especially winter recreation. This is at least partly because of the area's deep powder snow, which comes early in the fall and remains late into spring, which makes for diverse skiing, boarding, and snow-machining opportunities and for an extended winter use period" (p.1-2).

The area of most concentrated use is away from the Project site, near Independence Mine Historic Park, on the east side of Hatcher Pass. Though people travel along the Willow Fishhook Road in summer, and snowmachine along the road in winter, the actual location of the Willow Creek Mine is not used for most of the recreation purposes due to steep terrain.

With respect to mining, the plan declares that "Current mining and reclamation authorities, both under statute and administrative code, are considered sufficient to deal with most aspects of mining development and reclamation and this plan does not recommend additional restrictions, with some few exceptions (p. 2 – 11)." The plan does provide that DNR's mining authorizations "should consider methods to minimize impact of the development on the scenic resources of Hatcher Pass."

The area around the Project minesite is located within the Craigie Creek Management Unit (a unit within the Hatcher Pass Plan). DNR designates state land in the unit for two uses, primarily: Public Recreation-Dispersed, and Minerals. The description of the unit includes a description of mining, "This unit is blanketed with mining claims that are still in operation today…Since 2000, there has been extensive exploration focusing on the Lucky Shot Prospect, between the old Lucky Shot Mine and Coleman adit high on the valley wall. Exploration in this area continues and given the extensive mining history, remaining mineral values, and extensive recent exploration, it is likely that mining will reoccur in the valley during the period of this plan" (p. 3 – 35).

The mineral activity has co-existed with recreation use: "The Craigie Creek management unit is popular for all types of recreation. The southern half of the Summit Lake Recreation Site is a prominent point of interest for non-motorized recreation in the summer. Non-motorized uses include hiking, paragliding, and back-country skiing in the winter. [The State Recreation site is near Hatcher Pass and away from the Project mine site.] Motorized recreation includes snowmachining in the winter. ATV/ORV use in the summer is popular in the Craigie Creek unit outside of this recreation site… This unit encompasses the former Lucky Shot, War Baby, and Gold Bullion mines. Remnants of Kellyville, the mining boomtown that housed area miners, are visible today" (p. 3-36).

20.5.2 20.5.2 Socioeconomic Information

The Project is within the Matanuska-Susitna Borough. The Borough is the fastest growing region in Alaska. According to the Alaska Department of Labor, the Borough grew 58% between 2002 and 2012. Further, the region within the borough[2] that includes the Project grew even faster: 79% during that period. See Table 20-2.

[2] The borough region that includes the project is calculated from US census districts: City of Houston, City of Wasilla, and four Census Designated Places: Willow, Big Lake, Meadow Lakes, and Knik-Fairview.

Table 20-2 Population Growth

Population Change 2002 - 2012	
Alaska	16.8%
Anchorage	14.8%
Mat-Su Borough	58.1%
Project Region	78.6%

Source: Alaska Department of Labor, Alaska Economic Trends. February 2013.

The 2014 population of the Borough was 93,843, approximately 84% white. Approximately 10% of houses in the Borough live below the poverty line. The mean commute to work for residents in the Project region was over a half hour. This is not surprising given the more than half of the Borough workforce that works outside the Borough. Thirty-one percent of the workforce travelled to Anchorage, 8% to the North Slope, and the remainder throughout Alaska. In general, wages are lower in the Borough than they are in Anchorage, but land prices and housing are significantly more affordable. That, and the availability of jobs, provide reasons that so much of the Mat-Su workforce commutes to Anchorage and elsewhere. And it is a reason why North Slope or other remote workers choose to live in the Borough. See Table 20-3.

Table 20-3 Selected Socioeconomic Characteristics

	Alaska	Mat-Su Borough	Project Region
Total population	728,300	93,843	39,303
Percent white	66%	84%	84%
Total households	251,678	31,104	13,499
Mean household income	$88,583	$84,838	$81,254
Percent below the poverty line	10%	10%	11%
Civilian unemployment rate	8%	10%	12%
Mean travel time to work (minutes)	19	34	36
Percent of workers who work in:			
Private industry	68%	71%	71%
Government	26%	22%	21%
Self-employed or unpaid family work	6%	8%	7%

Source: American Community Survey 2010-2014. US Bureau of the Census

The table shows that Project region within the borough had slightly lower mean household income, and slightly longer commuting times than elsewhere in the Borough. Approximately a fifth of workers in the area work for government.

The Project is likely to bring 75-100 new, good paying jobs to the Willow-Houston region of the Borough. This number is roughly a 1% increase in the civilian employment in that part of the Borough. Wages rates are expected to be greater than the average worker earnings in the Borough. In addition, assuming the employees are pulled from the region, the jobs will be available without a significant commute. Finally, the Project will pay a significant amount in property tax sales tax payments to Wasilla or Palmer, and property tax or other taxes paid by employees.

21. CAPITAL AND OPERATING COSTS

Capital costs for the PFS have been developed from detailed scope documents and front-end engineering that has been completed for the infrastructure areas by the Project development team and outside engineering firms. Major equipment has been priced competitively by vendors from detailed enquiry packages. Estimated costs for the major civil works were quantified from pre-feasibility level designs. The operating costs were determined based on HRC's industry knowledge and prior experience, Info Mine's Cost Mine Service, and actual costs provided by GTOR for supplies and consumables.

21.1 Capital Costs

21.1.1 Initial Capital

Initial Capital for this study includes items purchased in the first two years before production begins in year 1. The majority of the spending and construction is planned in the 18 months prior to production. The initial cost by area of the mine is shown below in the Table 21-1. Total initial capital with a 15% contingency is estimated at $13.556 million.

Table 21-1 Initial Capital Costs

Initial Capital	Total
Investment - Mine	$1,619,500
Investment - Primary Development	$4,313,816
Investment - Plant	$6,397,500
Investment - G&A	$20,000
Capital Indirects & Contingency	$1,205,550
Total Capital	**$13,556,366**

21.1.2 Initial Capital by Area

Initial mine equipment and mine infrastructure capital costs are presented in Table 21-2 by unit cost, units required and total cost. Table 21-3 lists the plant and G&A initial capital by unit cost, units required, and total cost. The production schedule shows production beginning in month 2 of year 1, so the costs for some of these units falling in month 1 of year 1 are not included in the summary costs presented in Table 21-1. Contingencies for all areas are included at 15%. A portion of the available mine equipment is quoted as used and is available in the area from a private party. The used equipment is noted in the description portion of the table. The plant circuit is all quoted as new except for the crushing circuit which is mostly used except for the rolls crusher and ore storage bin.

Table 21-2 Initial Mine Department Capital Costs

Area	Sub-Area	Description	Cost/Unit	Units	Initial Capital
Mine	Mining Equipment	Air Compressor Electric	$82,159	2	$164,318
Mine	Mining Equipment	Diesel Air Compressor - used	$7,250	2	$14,500
Mine	Mining Equipment	Heated Blower Desiccant Dryer	$45,000	2	$90,000
Mine	Mining Equipment	Generator/Compressor Building	$20,000	1	$20,000
Mine	Mining Equipment	Front End Loader	$125,000	1	$125,000
Mine	Mining Equipment	Generator 3456 455KW	$400,000	1	$400,000
Mine	Mining Equipment	Ore & Waste Dump	$75,000	1	$75,000
Mine	Mining Equipment	1 yard LHD - used	$110,000	2	$220,000
Mine	Mining Equipment	4 ton Diesel Locomotive - used	$75,000	1	$75,000
Mine	Mining Equipment	Mule	$16,000	1	$16,000
Mine	Mining Equipment	6 ton Diesel Locomotive - used	$95,000	1	$95,000
Mine	Mining Equipment	Rocker Dump - used	$1,700	5	$8,500
Mine	Mining Equipment	64 cu. Ft Granby - used	$2,500	10	$25,000
Mine	Mining Equipment	Slusher IR A5NNOH – used	$7,500	4	$30,000
Mine	Mining Equipment	Slusher B2F211 Joy 30 Hp elec. - used	$17,500	2	$35,000
Mine	Mining Equipment	Tuggers	$2,500	3	$7,500
Mine	Mining Equipment	Eimco 12B Overshot - used	$7,500	1	$7,500
Mine	Mining Equipment	Jackleg MW 83 or GD 83 - used	$1,200	18	$21,600
Mine	Mining Equipment	Stope Fan	$5,215	6	$31,290
Mine	Mining Equipment	Portal Fans	$5,635	2	$11,270
Mine	Mining Equipment	Portal Heaters	$40,000	2	$80,000
Mine	Mining Equipment	Portal and Stope Silencers	$1,765	16	$28,240
Mine	Mining Equipment	Portal and Stope Bells	$370	6	$2,220
Mine	Mining Equipment	Power Center	$84,500	2	$169,000
Mine	Mining Equipment	Main Mine Fan	$17,515	1	$17,515
Mine	Mining Equipment	Mine Water Pumps	$10,000	1	$10,000
Mine	Mining Equipment	Mine Dewatering pumps	$6,000	1	$6,000
Mine	Mine Infrastructure	Office	$150,000	1	$150,000
Mine	Mine Infrastructure	Safety	$50,000	1	$50,000
Indirects&Contingency	Contingency	Contingency Other		15%	$297,818
Total					**$2,283,271**

Table 21-3 Initial Plant and G&A Capital Costs

Area	Sub-Area	Description	Cost/Unit	Units	Initial Capital
Plant	Mill	Crushing Circuit Equipment - used	$605,515	1	$605,515
Plant	Mill	Wet Mill Circuit Equipment	$524,515	1	$524,515
Plant	Mill	Slimes Circuit Equipment	$40,090	1	$40,090
Plant	Mill	Sizing Circuit Equipment	$89,830	1	$89,830
Plant	Mill	Spirals Circuit Equipment	$108,445	1	$108,445
Plant	Mill	Tables Circuit Equipment	$341,469	1	$341,469
Plant	Mill	Refinery Circuit Equipment	$379,140	1	$379,140
Plant	Mill	Mill Building Equipment	$253,179	1	$253,179
Plant	Mill	Mill Direct Costs	$369,292	1	$369,292
Plant	Mill	Mill General Requirements	$49,000	1	$49,000
Plant	Mill	Mill Site work	$356,605	1	$356,605
Plant	Mill	Mill Concrete	$304,843	1	$304,843
Plant	Mill	Mill Materials	$110,144	1	$110,144
Plant	Mill	Mill Equipment	$221,481	1	$221,481
Plant	Mill	Mill Pre-Engineered Buildings	$610,666	1	$610,666
Plant	Mill	Mill Plumbing	$93,628	1	$93,628
Plant	Mill	Mill Heating and Air Conditioning	$44,282	1	$44,282
Plant	Mill	Mill Electrical	$680,850	1	$680,850
Plant	Mill	Mill Communications	$11,500	1	$11,500
Plant	Mill	Mill Electronic Safety and Security	$103,000	1	$103,000
Plant	Mill	Mill Community Meetings	$10,000	1	$10,000
Plant	Mill	Mill Insurance, Site Construction	$10,000	1	$10,000
Plant	Mill	Mill Permits & Fees	$38,500	1	$38,500
Plant	Mill	Mill Testing & Inspection	$31,000	1	$31,000
Plant	Mill	Mill Architect & Engineers	$43,000	1	$43,000
Plant	Mill	Lab equipment & building	$167,269	1	$167,269
Plant	Mill	Startup lab	$180,272	1	$180,272
Plant	Mill	Final engineering	$250,000	1	$250,000
Plant	Mill	Assaying 3rd party	$20,000	1	$20,000
Plant	Mill	lucky shot exploration	$100,000	1	$100,000
G&A	Infrastructure	School bus used	$25,000	1	$25,000
G&A	Infrastructure	Parking lots and signs	$20,000	1	$20,000
G&A	Infrastructure	pickups	$30,000	1	$30,000
G&A	Infrastructure	computers	$10,000	6	$60,000
Indirects&Contingency	Contingency	Contingency Other		15%	$942,377
Total					**$7,224,892**

21.1.3 Sustaining Capital

Sustaining capital costs are included for major mine equipment component rebuilds, plant equipment rebuilds, and additional mine equipment requirements. The sustaining capital requirements by year and area are presented in Table 21-4. Reclamation bonding costs and closure costs are estimated at $250,000 each, and are not included in the summary Table 21-4.

Table 21-4 Sustaining Capital and Financing Costs

Sustaining Capital	Year 1	Year 2	Year 3	Year 4	Total
Sustaining Capital - Mine	$960,200	$222,000	$0	$0	$1,182,200
Sustaining - Primary Development	$1,953,178	$261,936	$0	$0	$2,215,113
Sustaining Capital - Plant	$108,421	$259,137	$260,234	$239,703	$867,496
Sustaining Capital - G&A	$215,000	$0	$0	$0	$215,000
Sustaining Capital – Indirects & Contingency	$192,543	$72,171	$39,035	$35,956	$339,704
Total Sustaining Capital	**$3,429,342**	**$815,243**	**$299,270**	**$275,659**	**$4,819,513**

21.1.4 Development Costs

The primary development costs are included in the initial and sustaining capital cost estimates presented above. As production begins, the secondary development in the ore zones is included as an operating cost. The total life of mine development and overall costs, including labor, are presented in Table 12-5. Development costs are broken out and calculated by five discrete areas: rehab, drift with rail, open raise, ramp and ore drift.

Table 21-5 Development Requirements and Costs

	Year -1	Year 1	Year 2	Year 3	Year 4
Meters	2,012	2,195	834	764	791
$/m	$4,287.72	$2,166.20	$1,923.53	$1,628.30	$1,476.17

21.2 Operating Cost Estimates

Operating costs for the Project were developed from material and supply costs provided by GTOR, HRC's industry knowledge and prior experience, and Info Mine's Cost Mine Service for mining costs. The total operating cost summary per ton of ore and per ounce of gold is shown in Table 21-6.

Table 21-6 Total Operating Cost Summary

Operating Costs	$/oz. Au	$/tn ore
Total Mining	-$344.08	-$156.04
Total Processing	-$165.68	-$75.14
Total Site G & A	-$87.21	-$39.55
Cash Operating Costs	**-$596.97**	**-$270.73**
Royalties	-$67.14	-$30.45
Taxes	-$11.29	-$5.12
Total Cash Costs	**-$675.40**	**-$306.30**

21.2.1 Mine Operating Costs

Mine operating costs are calculated in detail by equipment, consumables, supplies, services and manpower requirements based on the mine schedule. Equipment costs are calculated based on required hours of operation to meet the production schedule and hourly costs for equipment components, supplies, consumables and manpower. Diesel costs were estimated at $2.15/gallon. Mine maintenance costs are principally based on manufacturer's recommendations, and component replacement and cost. HRC also used data from operating mines of similar size in developing the operating costs for the mine. The costs details by department and category over the life of the mine are shown in Table 21-7.

Table 21-7 Mine Operating Costs

Department	Category	Average Yearly Costs	Cost/ton ore mined	Cost/oz. Au Sold
Mine G&A	Labor & Benefits	$547,079	$10.94	$24.13
Mine G&A Total		**$547,079**	**$10.94**	**$24.13**
Secondary Development	Alloc	$1,243,957	$24.88	$54.86
Secondary Development Total		**$1,243,957**	**$24.88**	**$54.86**
UG Stoping	Alloc	$56,767	$1.14	$2.50
	Blasting Supplies	$684,785	$13.70	$30.20
	Labor & Benefits	$1,127,772	$22.56	$49.74
	Materials/Supplies	$1,417,369	$28.35	$62.51
	Service	$99,345	$1.99	$4.38
UG Stoping Total		**$3,386,038**	**$67.72**	**$149.33**
Mine Services	Alloc	$35,322	$0.71	$1.56
	Labor & Benefits	$118,095	$2.36	$5.21
	Materials/Supplies	$556,584	$11.13	$24.55
	Service	$167,098	$3.34	$7.37
Mine Services Total		**$877,099**	**$17.54**	**$38.68**
Ore/Waste Transport	Alloc	$37,447	$0.75	$1.65
	Labor & Benefits	$283,269	$5.67	$12.49
	Materials/Supplies	$85,327	$1.71	$3.76
	Service	$391,000	$7.82	$17.24
Ore/Waste Transport Total		**$797,043**	**$15.94**	**$35.15**
Engineering	Labor & Benefits	$398,705	$7.97	$17.58
Engineering Total		**$398,705**	**$7.97**	**$17.58**
Geology	Labor & Benefits	$266,275	$5.33	$11.74
Geology Total		**$266,275**	**$5.33**	**$11.74**
Mine Mntnc	Alloc	-$129,536	-$2.59	-$5.71
	Labor & Benefits	$602,099	$12.04	$26.55
Mine Mntnc Total		**$472,563**	**$9.45**	**$20.84**
Grand Total		**$7,988,760**	**$159.78**	**$352.31**

21.2.2 Plant Operating Costs

Processing costs were estimated based on equipment requirements built up in detail utilizing hourly costs for equipment components, supplies, consumables and manpower. Power was estimated at $0.085/Kwh based on current rates in the area. Due to the process utilizing gravity concentration, there are very few reagent and supply costs required, and most of the costs come from power, manpower and equipment maintenance. Table 12-8 shows the life of mine costs by department and cost category.

Table 21-8 Process Plant Operating Costs

Department	Category	Average Yearly Costs	Cost/ton ore mined	Cost/oz. Au Sold
Plant G&A	Energy	$82,121	$1.64	$3.62
	Fuel & Lubes	$13,189	$0.26	$0.58
	Labor & Benefits	$491,272	$9.83	$21.67
	Materials/Supplies	$1,672	$0.03	$0.07
	Services	$575	$0.01	$0.03
	Travel	$190	$0.00	$0.01
	Wear Parts	$678	$0.01	$0.03
Plant G&A Total		**$589,702**	**$11.79**	**$26.01**
Crushing	Alloc	$19,333	$0.39	$0.85
	Energy	$21,249	$0.42	$0.94
	Fuel & Lubes	$21,659	$0.43	$0.96
	Labor & Benefits	$436,849	$8.74	$19.27
	Wear Parts	$45,337	$0.91	$2.00
Crushing Total		**$544,427**	**$10.89**	**$24.01**
Desliming circuit	Alloc	$62,527	$1.25	$2.76
	Energy	$32,328	$0.65	$1.43
	Fuel & Lubes	$11,652	$0.23	$0.51
	Wear Parts	$34,139	$0.68	$1.51
Desliming circuit Total		**$140,646**	**$2.81**	**$6.20**
Sizing and Spirals Area	Alloc	$62,592	$1.25	$2.76
	Energy	$72,739	$1.45	$3.21
	Fuel & Lubes	$18,692	$0.37	$0.82
	Labor & Benefits	$266,205	$5.32	$11.74
	Wear Parts	$143,159	$2.86	$6.31
Sizing and Spirals Area Total		**$563,386**	**$11.27**	**$24.85**
Table Area	Alloc	$40,964	$0.82	$1.81
	Energy	$49,176	$0.98	$2.17
	Fuel & Lubes	$12,891	$0.26	$0.57
	Labor & Benefits	$279,856	$5.60	$12.34
	Wear Parts	$42,110	$0.84	$1.86
Table Area Total		**$424,996**	**$8.50**	**$18.74**
Tailings	Alloc	$11,673	$0.23	$0.51
	Energy	$15,937	$0.32	$0.70
	Fuel & Lubes	$13,299	$0.27	$0.59
	Labor & Benefits	$24,207	$0.48	$1.07
	Materials/Supplies	$1,230	$0.02	$0.05
	Wear Parts	$4,184	$0.08	$0.18
Tailings Total		**$70,530**	**$1.41**	**$3.11**

Table 12-8 Process Plant Operating Costs cont.

Department	Category	Average Yearly Costs	Cost/ton ore mined	Cost/oz. Au Sold
Refinery	Alloc	$215	$0.00	$0.01
	Energy	$15,345	$0.31	$0.68
	Labor & Benefits	$286,682	$5.73	$12.64
	Materials/Supplies	$7,094	$0.14	$0.31
	Services	$5,016	$0.10	$0.22
	Wear Parts	$258	$0.01	$0.01
Refinery Total		**$314,609**	**$6.29**	**$13.87**
Gravity Plant Mntnc	Alloc	-$197,304	-$3.95	-$8.70
	Fuel & Lubes	$5,228	$0.10	$0.23
	Labor & Benefits	$406,133	$8.12	$17.91
	Materials/Supplies	$658	$0.01	$0.03
	Services	$50	$0.00	$0.00
	Travel	$53	$0.00	$0.00
	Wear Parts	$269	$0.01	$0.01
Gravity Plant Mntnc Total		**$215,087**	**$4.30**	**$9.49**
Assay Lab	Labor & Benefits	$771,226	$15.42	$34.01
	Materials/Supplies	$30,686	$0.61	$1.35
Assay Lab Total		**$801,912**	**$16.04**	**$35.36**
Grand Total		**$3,665,294**	**$73.31**	**$161.64**

21.2.3 G&A Operating Costs

The G&A costs have been developed from HRC's knowledge and experience as well as data from similar size operations. The major G&A cost component is staff and labor, but G&A also covers such things as security, office equipment, heat and lighting, communications, overtime, property insurance, office supplies, computer system license fees, admin building maintenance, janitorial services, outside services and allowances for travel and meetings. Table 12-9 show the life of mine costs by department and cost category for the G&A department.

Table 21-9 G&A Operating Costs

Department	Category	Average Yearly Costs	Cost/ton ore mined	Cost/oz. Au Sold
Admin	Energy	$39,994	$0.80	$1.76
	Fuel & Lubes	$5,862	$0.12	$0.26
	Gen Costs	$496,185	$9.92	$21.88
	Labor & Benefits	$291,270	$5.83	$12.85
	Lease	$7,308	$0.15	$0.32
	Materials/Supplies	$18,601	$0.37	$0.82
	Services	$21,192	$0.42	$0.93
	Travel	$8,038	$0.16	$0.35
Admin Total		**$888,448**	**$17.77**	**$39.18**
Accounting	Gen Costs	$2,192	$0.04	$0.10
	Labor & Benefits	$233,331	$4.67	$10.29
	Materials/Supplies	$5,115	$0.10	$0.23
	Services	$1,462	$0.03	$0.06
	Travel	$6,577	$0.13	$0.29
Accounting Total		**$248,677**	**$4.97**	**$10.97**
Environmental	Gen Costs	$4,385	$0.09	$0.19
	Materials/Supplies	$20,973	$0.42	$0.92
	Services	$10,961	$0.22	$0.48
	Travel	$3,288	$0.07	$0.15
Environmental Total		**$39,607**	**$0.79**	**$1.75**
Human Relations	Gen Costs	$20,461	$0.41	$0.90
	Materials/Supplies	$731	$0.01	$0.03
	Services	$7,308	$0.15	$0.32
	Travel	$1,096	$0.02	$0.05
Human Relations Total		**$29,596**	**$0.59**	**$1.31**
Purchasing	Gen Costs	$731	$0.01	$0.03
	Labor & Benefits	$281,888	$5.64	$12.43
	Materials/Supplies	$4,385	$0.09	$0.19
	Services	$1,462	$0.03	$0.06
	Travel	$1,827	$0.04	$0.08
Purchasing Total		**$290,292**	**$5.81**	**$12.80**
Security & Safety	Gen Costs	$5,846	$0.12	$0.26
	Labor & Benefits	$403,509	$8.07	$17.79
	Lease	$731	$0.01	$0.03
	Materials/Supplies	$17,538	$0.35	$0.77
	Services	$2,192	$0.04	$0.10
	Travel	$2,923	$0.06	$0.13
Security & Safety Total		**$432,739**	**$8.65**	**$19.08**
Grand Total		**$1,929,359**	**$38.59**	**$85.09**

21.2.4 Labor

Operating labor rates and burdens percentages are based on estimates from Gold Torrent which are within industry averages. Staffing levels and rates for the life of the reserve are shown in Table 21-10. Overtime was estimated at 2.5% and payroll burdens were estimated at 32.5%.

Table 21-10 Manpower Requirements

Manpower Summary		Rates	Min	Max	Average
Mining G&A					
Mine Superintendent	Mine G&A	$125,000/yr.	1	1	1
Mine Foreman	Mine G&A	$75,000/yr.	0	2	2
Mine Clerk	Mine G&A	$19.23/hr.	0	1	1
Mine G&A			1	4	3
Development -					
Miner 1	Development	$30.00/hr.	0	2	2
Miner 2	Development	$25.00/hr.	0	2	2
Miner 3	Development	$22.50/hr.	0	2	2
Development			0	6	6
Sub Level Development					
Miner 1	Raises & Levels	$30.00/hr.	0	2	2
Miner 2	Raises & Levels	$25.00/hr.	0	2	2
Development			0	4	4
U/G Production					
Miner 1	UG Stoping	$30.00/hr.	1	7	4
Miner 2	UG Stoping	$25.00/hr.	1	7	4
Miner 3	UG Stoping	$22.50/hr.	1	7	4
Stoping			3	21	13
Haulage					
Miner 3	Ore/Waste Transport	$22.50/hr.	2	2	2
Miner 4	Ore/Waste Transport	$19.00/hr.	2	2	2
Haulage			4	4	4
Mine Services					
U/G Helper - Supplies Cleanup	Mine Services	$18.00/hr.	0	2	2
Mine Maintenance					
Lead Mechanic	Mine Mntnc	$35.00/hr.	1	1	1
Heavy Equipment Mechanic	Mine Mntnc	$30.00/hr.	1	1	1
Mechanic Helper	Mine Mntnc	$22.50/hr.	0	1	1
Mechanic/Welder	Mine Mntnc	$25.00/hr.	0	1	1
Electrician	Mine Mntnc	$30.00/hr.	1	1	1
Total Mine Maintenance			3	5	5
Total Mine Operations			15	45	32
Engineering					
Sr Mining Engineer	Engineering	$110,000/yr.	1	1	1
Chief Surveyor	Engineering	$90,000/yr.	1	1	1
Engineering			2	2	2
Geology & Grade Control					
Sr Geologist	Geology	$90,000/yr.	1	1	1
Ore Control Geologist	Geology	$80,000/yr.	0	1	1
Geology			1	2	2
Total Mine Eng. & Geo.			2	4	4

Table 21-10 Manpower Requirements (cont.)

Manpower Summary		Rates	Min	Max	Average
Processing Plant					
Salary Personnel					
Plant Superintendent	Plant G&A	$130,000/yr.	1	1	1
Met Engineer	Plant G&A	$90,000/yr.	1	1	1
Assay Lab Manager	Assay Lab	$100,000/yr.	1	1	1
Plant Salary			3	3	3
Hourly Personnel					
Met Technician	Plant G&A	$25.00/hr.	1	1	1
Loader crane deslime operator	Primary Crushing	$17.00/hr.	3	3	3
Screen crush convey operator	Primary Crushing	$20.00/hr.	1	1	1
Regrind wet size operator	Primary Crushing	$19.00/hr.	3	3	3
Operator	Spirals Area	$22.00/hr.	2	2	2
Regrind wet size operator	Spirals Area	$17.00/hr.	2	2	2
Clerk	Plant G&A	$17.00/hr.	1	1	1
Operators	Table Area	$22.00/hr.	1	1	1
Operator Assistant	Table Area	$19.00/hr.	1	1	1
Regrind wet size operator	Table Area	$17.00/hr.	1	1	1
Gemeni operator	Table Area	$24.00/hr.	1	1	1
Engineer	Refinery	$45.00/hr.	1	1	1
Operator	Refinery	$20.00/hr.	1	1	1
Regrind wet size operator	Refinery	$19.00/hr.	1	1	1
Assayer	Assay Lab	$28.33/hr.	2	3	3
Sample Prep	Assay Lab	$16.33/hr.	2	6	6
Mechanic	Gravity Plant Mntnc	$24.50/hr.	2	2	2
Mechanic Helper	Gravity Plant Mntnc	$20.00/hr.	2	2	2
Electrician	Gravity Plant Mntnc	$30.00/hr.	1	1	1
Plant Hourly			29	34	34
Total Processing Operations			32	37	37
General and Administrative					
Mine Manager	Admin	$150,000/yr.	1	1	1
Safety Superintendent	Security & Safety	$85,000/yr.	1	1	1
Controller	Accounting	$65,000/yr.	1	1	1
Purchasing Manager	Purchasing	$90,000/yr.	1	1	1
Payroll	Accounting	$20.00/hr.	0	1	1
Accounts Payable	Accounting	$18.00/hr.	0	1	1
Admin Assistant	Admin	$15.00/hr.	0	1	1
Safety/Security	Security & Safety	$20.00/hr.	0	4	3
Warehousemen	Purchasing	$20.00/hr.	1	2	2
Total G&A			6	13	11
Total Salary			7	14	13
Total Hourly			17	85	63
Total Property			24	99	76

22. ECONOMIC ANALYSIS

22.1 Summary

Information contained and certain statements made herein are considered forward-looking within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected.

The Project is planned to be an underground mining operation with milling and gravity separation processing of the ore completely contained on patented or privately held land surface, with an estimated production of 174,448 tonnes from its reserves, grading 15.6 gpt gold and 1.6 gpt silver. The operations are planned to run at a rate of 200 tonnes per day, five days per week, 350 days per year, with all mineralized material being crushed, milled and processed. Metal recoveries are estimated to average 91.8% for gold and about 90% for silver.

Economic analysis of the base case scenario for the Project uses a price of US$1,175/oz for gold, which is approximately equal to 18 month trailing average and the closing spot price at the end of May, 2016. The silver price for the base case is $15/oz of silver, which is slightly lower than the 18 month trailing average and more conservative than the closing spot price at the end of May 2016. The economic model shows an After-Tax Net Present Value @ 10% ("NPV-10") of $5.3 million using a 7.0 gpt Au mining cut-off grade, as well as an After-Tax Internal Rate of Return ("IRR") of 21.8%. Table 22-1 summarizes the projected Net Present Value, NPV-10; Internal Rate of Return, IRR; years of positive cash flows to repay the negative cash flow ("payback period"); multiple of positive cash flows compared to the maximum negative cash flow ("payback multiple") for the Project on both After-Tax and Before-Tax bases.

Table 22-1 Summary of Willow Creek Economic Results

Project Valuation Overview	After Tax	Before Tax
Net Cashflow (millions)	$14.69	$20.64
NPV @ 5.0%; (millions)	$9.17	$13.68
NPV @ 7.5%; (millions)	$7.05	$11.01
NPV @ 10.0%; (millions)	$5.28	$8.76
Internal Rate of Return	21.8%	28.1%
Payback Period, Years	1.88	1.75
Payback Multiple	1.91	2.27
Total Initial Capital (millions)	-$15.06	-$15.06
Max Neg. Cashflow (millions)	-$16.21	-$16.21

Table 22-2 summarizes the projected production schedule and cash flows. The economic evaluation and schedule is based on Proven and Probable reserves. Additional mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of additional estimated mineral resources will be converted into mineral reserves.

Table 22-2 Lucky Shot Project Schedule and Cash Flow

Note: All Dollars are in US	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Life-of-Mine
MINE PRODUCTION								
Tonnes Ore Mined			23,296	40,277	51,884	50,231	8,761	174,448
Ag, g/tn			1.88	1.50	1.86	1.59	0.80	1.65
Au, g/tn			15.01	13.48	16.31	17.56	11.49	15.60
Development Meters		2,012	2,195	834	764	791	-	6,596
Development Waste		21,820	44,687	10,994	6,551	6,790	-	90,843
Total Tonnes Mined		**21,820**	**67,983**	**51,271**	**58,435**	**57,020**	**8,761**	**265,291**
PROCESS PRODUCTION								
Tonnes Ore Processed			23,296	40,277	51,884	50,231	8,761	**174,448**
Ag, g/tn			1.88	1.50	1.86	1.59	0.80	1.65
Au, g/tn			15.01	13.48	16.31	17.56	11.49	15.60
Income Statement								
REVENUE								
Contained Oz Ag to Mill			1,407	1,939	3,103	2,569	226	9,245
Contained Oz Au to Mill			11,239	17,452	27,206	28,362	3,235	87,494
Saleable Oz Ag, post 93.0% Refinery credit			1,178	1,623	2,597	2,150	189	7,738
Saleable Oz Au, post 98.5% Refinery credit			10,162	15,781	24,600	25,645	2,925	79,114
Gross Revenue			$11,958,536	$ 18,567,079	$ 28,944,127	$ 30,165,616	$ 3,440,042	$ 93,075,400
Transportation and Refinery Charges			(17,011)	(26,106)	(40,796)	(41,694)	(4,672)	(130,278)
Net Refined Revenue			$11,941,525	$ 18,540,973	$ 28,903,331	$ 30,123,922	$ 3,435,370	$ 92,945,122
Royalties			(682,042)	(1,057,108)	(1,652,861)	(1,724,405)	(195,278)	(5,311,694)
Net Revenue			**$11,259,483**	**$ 17,483,865**	**$ 27,250,470**	**$ 28,399,518**	**$ 3,240,093**	**$ 87,633,428**
OPERATING EXPENSES								
Mining								
Mine G&A		-	(397,810)	(419,360)	(420,509)	(419,360)	(86,076)	(1,743,116)
Secondary Development		-	(848,726)	(1,079,789)	(1,243,357)	(1,168,252)	-	(4,340,125)
UG Stoping		-	(1,589,466)	(2,705,527)	(3,510,850)	(3,393,675)	(614,255)	(11,813,773)
Ore/Waste Transport		-	(456,545)	(651,192)	(771,108)	(753,469)	(148,542)	(2,780,856)
Mine Services		-	(594,606)	(603,348)	(604,708)	(603,348)	(153,748)	(2,559,758)
Mine Mntnc		-	(340,085)	(308,087)	(287,269)	(278,580)	(50,754)	(1,264,775)
Engineering		-	(265,000)	(265,000)	(265,726)	(265,000)	(65,342)	(1,126,068)
Geology		-	(206,119)	(225,250)	(225,867)	(225,250)	(46,538)	(929,025)
Contingency, @ 2.5%			(117,459)	(156,439)	(183,235)	(177,673)	(29,131)	(663,937)
Total Mining		**$ -**	**$ (4,815,816)**	**$ (6,413,993)**	**$ (7,512,630)**	**$ (7,284,607)**	**$(1,194,386)**	**$(27,221,433)**
Processing & ROM handling								
Plant G&A		-	(451,624)	(494,008)	(495,723)	(494,321)	(121,773)	(2,057,449)
Crushing		-	(417,387)	(456,127)	(457,376)	(456,127)	(112,470)	(1,899,487)
Desliming circuit		-	(107,827)	(117,835)	(118,158)	(117,835)	(29,055)	(490,710)
Sizing and Spirals Area		-	(431,922)	(472,011)	(473,304)	(472,011)	(116,386)	(1,965,634)
Table Area		-	(325,825)	(356,067)	(357,042)	(356,067)	(87,797)	(1,482,798)
Tailings		-	(54,072)	(59,090)	(59,252)	(59,090)	(14,570)	(246,075)
Assay Lab		-	(617,874)	(680,779)	(686,956)	(684,512)	(127,722)	(2,797,843)
Refinery		-	(241,207)	(263,577)	(264,272)	(263,577)	(65,026)	(1,097,660)
Gravity Plant Mntnc		-	(164,833)	(180,195)	(180,738)	(180,237)	(44,427)	(750,430)
Contingency, @ 2.5%		-	(70,314)	(76,992)	(77,321)	(77,094)	(17,981)	(319,702)
Total Processing		**$ -**	**$ (2,882,886)**	**$ (3,156,681)**	**$ (3,170,142)**	**$ (3,160,871)**	**$ (737,208)**	**$(13,107,787)**
Site General & Administration								
Admin		-	(725,775)	(732,746)	(734,754)	(732,746)	(173,744)	(3,099,766)
Human Relations		-	(24,300)	(24,300)	(24,367)	(24,300)	(5,992)	(103,258)
Security & Safety		-	(345,945)	(365,373)	(366,374)	(365,373)	(66,747)	(1,509,812)
Accounting		-	(198,152)	(207,380)	(207,948)	(207,380)	(46,764)	(867,625)
Purchasing		-	(240,524)	(240,524)	(241,183)	(240,524)	(50,063)	(1,012,818)
Environmental		-	(32,520)	(32,520)	(32,609)	(32,520)	(8,019)	(138,188)
Contingency, @ 2.5%			(39,180)	(40,071)	(40,181)	(40,071)	(8,783)	(168,286)
Total G&A		**$ -**	**$ (1,606,396)**	**$ (1,642,914)**	**$ (1,647,416)**	**$ (1,642,914)**	**$ (360,112)**	**$ (6,899,753)**
Property Tax		-	(144,131)	(132,541)	(101,878)	(59,070)	(11,022)	(448,642)
Alaska Mining License Tax		-	-	-	-	(444,393)	-	(444,393)
Severance Tax								
Cash Operating Costs		**$ -**	**$ (9,449,230)**	**$(11,346,130)**	**$(12,432,065)**	**$(12,591,856)**	**$(2,302,728)**	**$(48,122,008)**

Table 22-2 Lucky Shot Project Schedule and Cash Flow (cont.)

Table 22-2 Continued	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Life-of-Mine
Operating Margin	**-**	**-**	**1,810,253**	**6,137,735**	**14,818,405**	**15,807,662**	**937,365**	**39,511,419**
Development Deduction	(1,050,000)	(3,019,671)	(1,367,225)	(183,355)	-	-	-	(5,620,250)
Amortization	(45,000)	(174,415)	(233,010)	(240,868)	(240,868)	(240,868)	(1,233,650)	(2,408,679)
Depreciation	-	-	(1,637,363)	(2,873,726)	(2,781,571)	(2,489,409)	(2,064,880)	(11,846,950)
Reclamation Deduction	-	-	-	-	-	-	(500,000)	(500,000)
Income - before NOL & Depletion	**(1,095,000)**	**(3,194,086)**	**(1,427,345)**	**2,839,786**	**11,795,966**	**13,077,385**	**(2,861,165)**	**19,135,541**
Net Operating Loss Adjustment	1,095,000	3,194,086	1,427,345	(2,839,786)	(2,876,644)	-	2,861,165	2,861,165
Depletion	-	-	-	-	(4,087,570)	(4,259,928)	-	(8,347,498)
State Income Tax	-	-	-	-	(454,185)	(828,841)	25,461	(1,257,565)
Federal Income Tax	-	-	-	-	(1,693,027)	(3,086,110)	94,908	(4,684,229)
Taxable Income, less Tax	**$ -**	**$ -**	**$ -**	**$ -**	**$ 2,684,539**	**$ 4,902,506**	**$ 120,369**	**$ 7,707,413**
Cash Flow Calculation								
<u>Adjustments for Non Cash Items</u>								
Legacy Development Credit to Project	1,500,000							1,500,000
Development Deduction	1,050,000	3,019,671	1,367,225	183,355	-	-	-	5,620,250
Amortization	45,000	174,415	233,010	240,868	240,868	240,868	1,233,650	2,408,679
Depreciation/Reclamation/Salvage	-	-	1,637,363	2,873,726	2,781,571	2,489,409	2,564,880	12,346,950
Net Operating Loss Adjustment	(1,095,000)	(3,194,086)	(1,427,345)	2,839,786	2,876,644	-	(2,861,165)	(2,861,165)
Depletion	-	-	-	-	4,087,570	4,259,928	-	8,347,498
Total Adjust. for Non Cash Items	**$ 1,500,000**	**$ -**	**$ 1,810,253**	**$ 6,137,735**	**$ 9,986,654**	**$ 6,990,205**	**$ 937,365**	**$ 27,362,211**
<u>Capital</u>								
Investment - Mine	-	(1,619,500)						(1,619,500)
Investment - Primary Development	(1,500,000)	(4,313,816)						(5,813,816)
Investment - Plant	(1,839,800)	(4,557,700)	-					(6,397,500)
Investment - G&A	-	(20,000)	-					(20,000)
Capital Indirects & Contingency	(275,970)	(929,580)	-					(1,205,550)
Total Capital	**$(3,615,770)**	**$(11,440,596)**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$(15,056,366)**
Sustaining Capital - Mine			(960,200)	(222,000)	-	-	-	(1,182,200)
Sustaining - Primary Development			(1,953,178)	(261,936)	-	-	-	(2,215,113)
Sustaining Capital - Plant			(108,421)	(259,137)	(260,234)	(239,703)	-	(867,496)
Sustaining Capital - G&A			(215,000)	-	-	-	-	(215,000)
Sustaining Capital - Indirects& Contingency			(192,543)	(72,171)	(39,035)	(35,956)	-	(339,704)
Reclamation Closure Costs	(250,000)	-	-	-	-	-	(250,000)	(500,000)
Total Capital & Sustaining	**-3,865,770**	**-11,440,596**	**-3,429,342**	**-815,243**	**-299,270**	**-275,659**	**-250,000**	**$(20,375,879)**
Working capital		0	-787,000	0	0	0	787,000	-
Total Capital & Working Capital	**-3,865,770**	**-11,440,596**	**-4,216,342**	**-815,243**	**-299,270**	**-275,659**	**537,000**	**$(20,375,879)**
Beginning Cash	**0**	**-2,365,770**	**-13,806,366**	**-16,212,454**	**-10,889,963**	**1,481,961**	**13,099,012**	
Period Net Cash Flow	**-2,365,770**	**-11,440,596**	**-2,406,089**	**5,322,492**	**12,371,923**	**11,617,052**	**1,594,734**	**$ 14,693,746**
Ending Cash	**-2,365,770**	**-13,806,366**	**-16,212,454**	**-10,889,963**	**1,481,961**	**13,099,012**	**14,693,746**	**$ 14,693,746**

The projected total lifespan of the Project is 6.5 years: two years of pre-production and construction, and 4.5 years of full operations. Approximately 87,500 oz of gold and 9,200 oz of silver are projected to be mined, with 79,100 oz of gold and 7,700 oz of silver recovered and produced for sale. A total cash basis capital investment of $18.9 million, including contingency/sustaining capital/reclamation, is projected. Following the All-In-Sustaining-Cost ("AISC") guidelines, life-of-mine average base case Cash Operating Cost is projected to be $604/oz of gold sold, before credits for silver sales, and $602/oz of gold after silver credits. The AISC life-of-mine average base case Total Operating Cost (including royalties and production taxes), before credits for silver sales, is expected to be $677/oz and $675/oz after credits.

Table 22-3 Lucky Shot Project Total Operating Cost/ounce Gold & per tonne Ore

AISC Operating Costs	$/oz Au	$/tn ore
Total Mining	-$344.08	-$156.04
Total Processing	-$165.68	-$75.14
Total Site G & A	-$92.88	-$42.12
Transportation and Refining	-$1.65	-$0.75
Bi-product credit (silver)	$1.75	$0.79
Cash Operating Costs	**-$602.54**	**-$273.26**
Royalties	-$67.14	-$30.45
Production Taxes	-$5.62	-$2.55
Total Operating Costs	**-$675.29**	**-$306.25**

As previously mentioned, the gold price used in the economic evaluation (US$1,175/oz Au) is around the 18 month trailing average price and approximate spot price as of the end of May 2016. The silver price was chosen to be slightly more conservative than the 18 month trailing average price and approximate spot price as of the end of May 2016.

22.1.1 Taxes

State, local, and federal taxes, including income taxes and the Alaska Mining License Fee, have been considered in this study, and are included in the economic analysis.

22.1.2 Royalties

A 5.8 percent royalty, calculated on the gross proceeds less transportation and refining costs, has been included for all of the metal produced, as required by underlying agreements.

22.1.3 Corporate Income Taxes

United States and State corporate taxes have been considered in the economic analysis.

22.2 Economic Model

22.2.1 Basis of Evaluation

Mineral resources were incorporated in the model only if classified as Proven or Probable Reserves according to CIM definitions. A throughput of 200 tonnes per day five days per week is the base case processing rate, and operations and capital factors were developed from this basis. Recoveries for gold are expected to average 91.8% for the base case. A recovery 90% for the base case is expected for silver. Construction of the facilities is projected to begin in year -2, concluding at the end of year -1.

After-Tax cash flows were calculated on a yearly basis for the life of the base case. Federal, state and local taxes were considered for this evaluation.

The Project is projected to have robust economics at the base case gold and silver prices of $1,175/oz and $15.00/oz respectively. The expected sensitivities of the Net Present Value from variations in the discount rate have also been calculated on an After-Tax and a Before-Tax basis, as have the Payback Period, the Payback Multiple, and the Maximum Negative Cash Flow previously mentioned. Table 22-4 presents the summary economic results, on an After-Tax and a Before-Tax basis.

Table 22-4 Summary Projected Economic Results

Project Valuation Overview	After Tax	Before Tax
Net Cashflow (millions)	$14.69	$20.64
NPV @ 5.0%; (millions)	$9.17	$13.68
NPV @ 7.5%; (millions)	$7.05	$11.01
NPV @ 10.0%; (millions)	$5.28	$8.76
Internal Rate of Return	21.8%	28.1%
Payback Period, Years	1.88	1.75
Payback Multiple	1.91	2.27
Total Initial Capital (millions)	-$15.06	-$15.06
Max Neg. Cashflow (millions)	-$16.21	-$16.21

22.3 Sensitivity Analysis

22.3.1 Price

The Project, like almost all precious metals projects, is very responsive to changes in the price of its chief commodity, gold. From the base case, a change in the average gold price of US$50/oz Au (assuming that the silver price does not vary) would change the NPV-10 by 75%, or approximately $4.0 million (Figure 22-1).

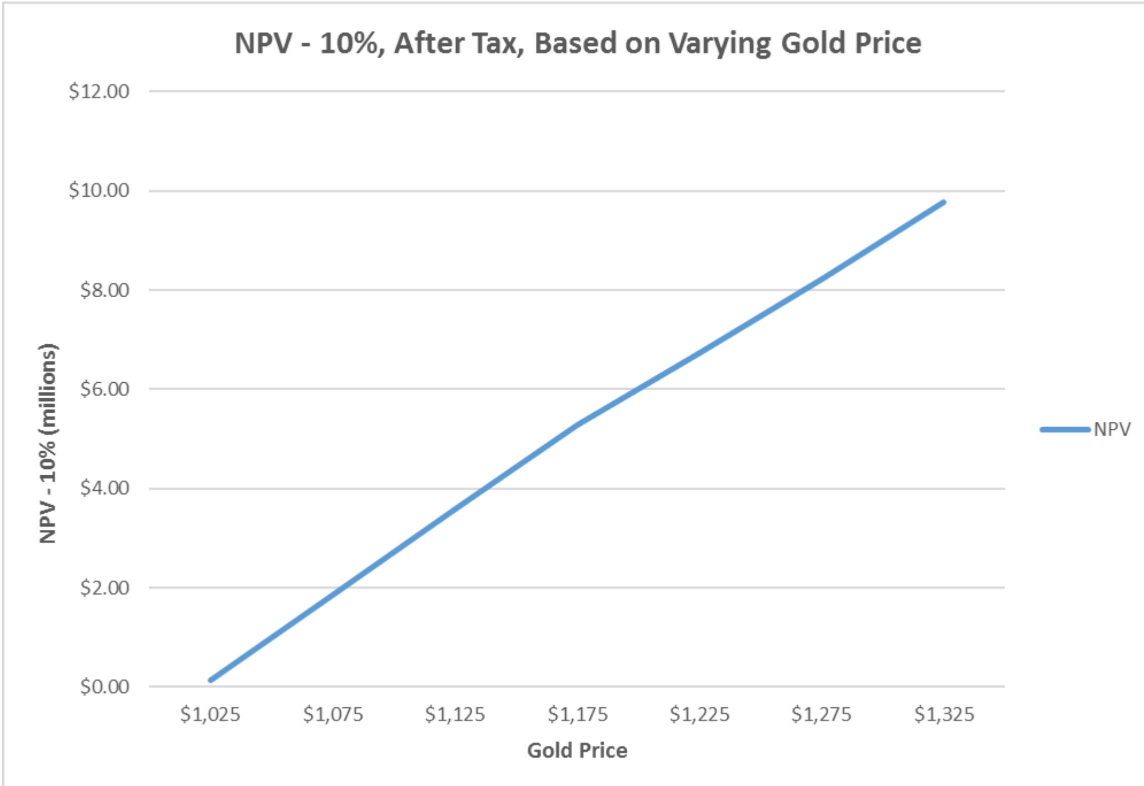

Figure 22-1 Project Gold Price Sensitivity Analysis

22.3.2 Cost and Recovery

The Project is quite sensitive to the cost of operations, incurring an approximately 20% decline in the NPV-5 for each increase of 10% in the operating costs. The Project is less sensitive to variances in the cost of capital, experiencing about 3.0% in decline in the NPV-5 for each increase of 10% in the capital costs, as shown in Figure 22-2.

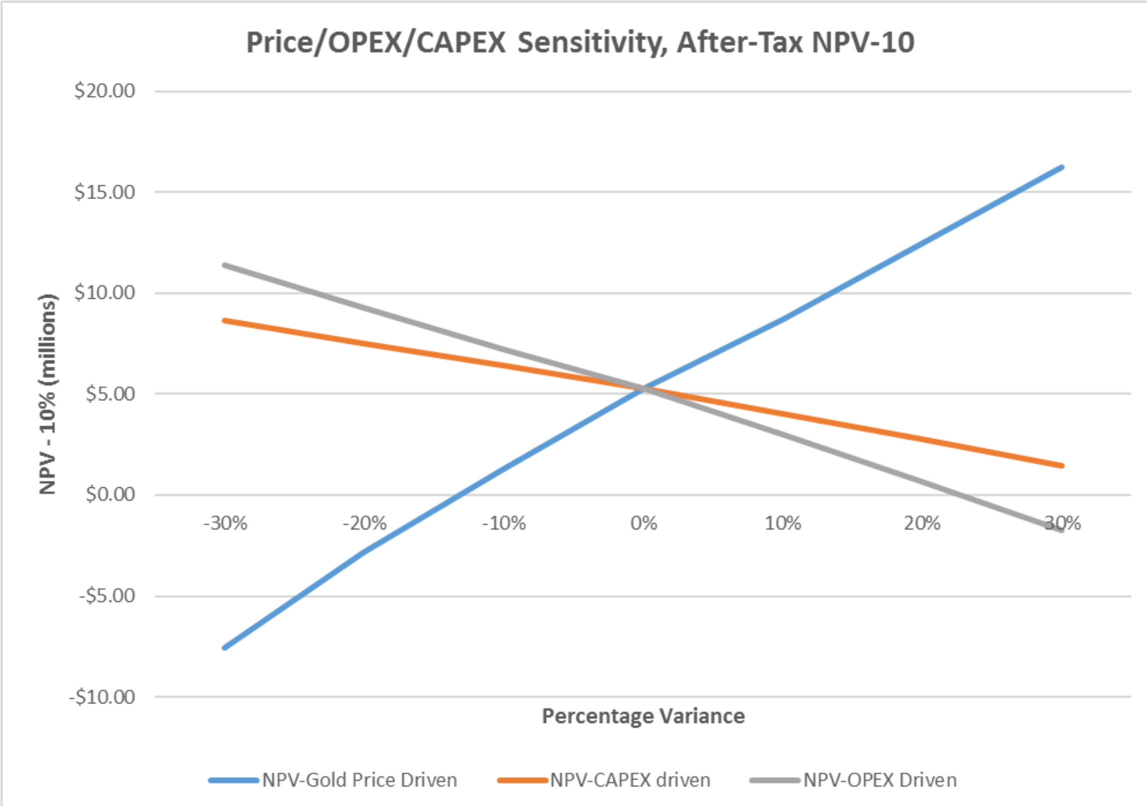

Figure 22-2 Project Operating Cost & Capital Cost Sensitivity Analysis

22.4 Conclusion

The Project would be economically viable based on the parameters considered in this study. The base case scenario produces approximately 79,100 salable ounces of gold and 7,700 salable ounces of silver over a 4.5-year period. The Project is most sensitive to the gold price and to operating costs, but not as sensitive to capital costs.

The base case economic analysis of the Project at a gold price of US$1,175/oz shows an After-Tax NPV-10 of $5.28 million using a 200 tonne/day crushing/grinding/gravity separation plant.

23. ADJACENT PROPERTIES

The Project is located within the historic Willow Creek mining district, which hosts a number of historically productive mines and in which mining has been carried out for nearly 100 years. While a majority of the past producers in the district are located on quartz veins similar or related to those in the Project area, there are no immediately adjacent properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Project.

24. OTHER RELEVANT DATA AND INFORMATION

This report summarizes all data and information material to the Project as of June 24, 2016. HRC knows of no other relevant technical or other data or information that might materially impact the interpretations and conclusions presented herein, nor of any additional information necessary to make the report more understandable or not misleading.

25. INTERPRETATION AND CONCLUSIONS

25.1 General Conclusions

HRC is of the opinion that GTOR has thorough understanding of the geology of the Project, that GTOR is applying the appropriate deposit model for exploration, and that the potential for continued discovery of additional mineral resources at the Project is high. Historic drilling and sampling indicate that mineralization occurs within the Project area beyond the resource area the defined in this report, specifically within the War Baby and Murphy areas.

The Project is not subject to any known environmental liabilities. As the state has a long history of mineral exploration, GTOR does not anticipate any barriers to access for work planned going forward. The preliminary assessment is based on the premise that the proposed Project processing facilities will be located in Willow, Matanuska-Susitna Borough, Alaska. The probable mine features such as underground workings, waste dumps, and infrastructure are all located on private land. At this time the only permits required for the Project are issued by the State of Alaska, not the federal government, and would not require public notice.

HRC concludes that the sample preparation, security and analytical procedures are correct and adequate for the purpose of this technical report. The sample methods and density are appropriate and the samples are of sufficient quality to comprise a representative, unbiased database.

HRC has reviewed the check assay programs and believes the programs provide adequate confidence in the data. Samples that are associated with failures and the samples associated with erroneous blank samples have been reviewed. Errors have been justified as labeling errors or are infrequent. All of the samples associated with erroneous QA/QC results are reviewed prior to inclusion in the database.

The majority of drill cores and cuttings from drilling have been photographed. Drill logs have been digitally entered into exploration database organized and maintained in a Maptek Vulcan project. The split core and cutting trays are stored at the Project and are susceptible to the elements. HRC recommends that the drill core be moved to a better storage facility to ensure that the samples are not damaged or lost due to inclement conditions.

HRC received original assay certificates in csv format for the drilling conducted from 2005 to 2009 in the current database. A random manual check of 10% of the database against the original certificates was conducted. The error rate within the database is considered to be less than 1% based on the number of samples spot checked. HRC is of the opinion that the data maintained within the database is acceptable for mineral resource estimation.

A total of 174 drillholes were used to model the veins present at the Project. Appendix B and Appendix C summarize the drillhole information and significant intercepts used as the basis for the mineral resource estimation. HRC finds that the density of data within the resource base is adequate for the mineral resource estimate. The mineral resource estimation is appropriate for the geology and underground mining method. Additional modeling should be conducted to define the alteration of the host rocks and to support further metallurgical testing. HRC concludes that the current mineral resource at the Project is sufficient to warrant continued planning and effort to explore, permit, and develop the Project.

The realization of the mineral reserves reported here is dependent on the successful implementation of the mining practices and open stull mining method suggested. If other mining methods are required, dilution may be hard to control on the narrow, shallow dipping veins. Although the Project has completed considerable baseline work for permitting, the final operating permits have not been issued. Mineral processing utilizing gravity separation reduces the potential for difficulty in obtaining the final operating permits compared to cyanide or floatation processing operations. Other than the above, HRC is unaware of any legal, political, environmental or other risks that could materially affect the potential development of the mineral reserves.

The Project would be economically viable based on the parameters considered in this study. The base case scenario produces approximately 79,100 salable ounces of gold and 7,700 salable ounces of silver over a 4.5-year period. The Project is most sensitive to the gold price and to operating costs, but not as sensitive to capital costs. The base case economic analysis of the Project at a gold price of US$1,175/oz shows an After-Tax NPV-10 of $5.28 million using a 200 tonne/day crushing/grinding/gravity separation plant.

26. RECOMMENDATIONS

Exploration of the Project has advanced to the consideration of a year-round underground mining operation. GTOR has made the decision to advance the project into devleopment and production. HRC offers the following specific recommendations with regard to advancing the Project to development:

- Continue efforts to obtain necessary environmental permits and clearances
- Conduct additional geotechnical studies as necessary for stope and detailed mine design
- Implement an active geotechnical data collection and review program to be carried out into production
- Complete additional metallurgical test work as necessary to finalize proposed processing methods
    - Confirm comminution energy requirements via conventional Bond testing
    - Determine Bond abrasion index for engineering cost estimation
    - Quantify liquid/solid separation, i.e., settling tests on various process streams
    - Establish performance of commercial spirals
    - Establish additional liberation information from gravity size/recovery testing
    - Establish gravity recovery as function of feed grade (both upwards and downwards)
    - Characterize gravity tailings
    - Determine if further classification of minus 325 mesh ore will increase gold recovery
    - Evaluate enhanced centrifugal gravity concentration of ore fines and tailings
- As development advances perform additional exploration drilling on the Lucky Shot area
- Design and carry out an exploration drilling campaign targeting the War Baby and Murphy portions of the Project area
- Complete final engineering on the processing plant and detailed mine design along with hard quotes for required equipment.

26.1 Budget

Table 26-1 summarizes the estimated cost of completing the individual tasks recommended above.

Table 26-1 Recommended Scope of Work and Estimated Cost

Recommended Scope of Work	Expected Cost (US$)
Environmental Permitting Work	$150,000
Geotechnical Analysis	$20,000
Metallurgical Testwork	$75,000
Detailed Mine Design and Final Engineering	$250,000
Exploration Drilling	$100,000
Subtotal	$595,000
15% Contingency	$89,250
Total Budget	**$684,250**

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APPENDIX A Project List of Patented Claims and State Claims

Patented Claims - AHI

The following claims are included in U.S. Mineral Surveys 960A, 1018, 1487, 2047, 2094, 2186A Palmer Recording District, State of Alaska; and are located within T20N, R1E., Seward Meridian, Sections 28, 29, 30, 31, 32, 33, 34, 35, and T19N, R1W, Section 2:

Table A - 1 Patented Claims

Claim	Name	Survey	Patent
1	Gold Dust No. 2	960A	478360
2	Golden Wonder	960A	478360
3	Golden Wonder No. 1	960A	478360
4	Gold Dust No. 2	960A	478360
5	Gold Dust No. 1	960A	478360
6	Gold Dust Fraction	960A	478360
7	Golden Eagle	1018	728475
8	Golden Eagle No. 1	1018	728475
9	Summit	1018	728475
10	Gold Nugget	1018	728475
11	Bird	1018	728475
12	Brassel Fraction Lode Claim	1487	1157928
13	War Baby No. One Claim	1487	1157928
14	War Baby No. Two Lode Claim	1487	1157928
15	Warbaby No. 3 Lode Claim	1487	1157928
16	War Baby No. 4 Lode Claim	1487	1157928
17	Lucky Shot Lode Claim No.1	1487	1157928
18	Lucky Shot Lode Claim No. 2	1487	1157928
19	Lucky Shot Lode Claim No. 3	1487	1157928
20	Lucky Shot No. 5 Load Claim	1487	1157928
21	War Eagle Fraction Lode Claim	1487	1157928
22	War Eagle No.1	1487	1157928
23	War Eagle No. 2	1487	1157928
24	War Eagle No. 3	1487	1157928
25	Lucky Shot Lode Claim	1487	1157928
26	Mary	2047	1127290
27	Black King No. 2	2094	1128877
28	Black King No. 3	2094	1128877
29	Black King No. 4	2094	1128877
30	Ready Bullion	2094	1128877
31	Ready Bullion No. 1	2094	1128877
32	Ready Bullion No. 2	2094	1128877
33	Ready Bullion Fraction	2094	1128877

Claim	Name	Survey	Patent
34	Early Cash	2094	1128877
35	Early Cash No. 1	2094	1128877
36	Lucky Gold Fraction	2094	1128877
37	Taylor Claim	2186A	1159173
38	Wilson Lode Claim	2186A	1159173
39	Madison Claim	2186A	1159173
40	Coolidge Lode Claim	2186A	1159173
41	War Eagle No. 4 Claim	2186A	1159173
42	War Eagle No. 5 Claim	2186A	1159173
43	War Eagle No. 6 Claim	2186A	1159173

State Claims - AHI

All State Claims are located in the Palmer Recording District within the Seward Meridian.

Table A - 2 State Claims

Claim	Name	ADL#	Township, Range, Section, Section/4			
1	LS 1	645931	S 20N 01W	34	SE	
2	LS 2	645932	S 20N 01W	35	SW	
3	LS 3	645933	S 20N 01W	35	SE	
4	LS 4	645934	S 20N 01W	36	SW	
5	LS 5	645935	S 20N 01W	36	SE	
6	LS 6	645936	S 20N 01 E	31	SW	
7	LS 7	645937	S 20N 01 E	31	SE	
9	LS 9	645939	S 20N 01 E	32	NW	
10	LS 10	645940	S 20N 01 E	31	NE	
11	LS 11	645941	S 20N 01 E	31	NW	
12	LS 12	645942	S 20N 01W	36	NE	
13	LS 13	645943	S 20N 01W	36	NW	
14	LS 14	645944	S 20N 01W	35	NE	
15	LS 15	645945	S 20N 01W	35	NW	
16	LS 16	645946	S 20N 01W	34	NE	
17	LS 17	645947	S 20N 01W	25	SE	
18	LS 18	645948	S 20N 01 E	30	SW	
19	LS 19	645949	S 20N 01 E	30	SE	
23	LS 23	645953	S 20N 01 E	29	NW	
24	LS 24	645954	S 20N 01 E	30	NE	
25	LS 25	645955	S 20N 01 E	19	SE	
26	LS 26	645956	S 20N 01 E	20	SW	
27	LS 27	645957	S 20N 01 E	20	SE	

Claim	Name	ADL#	Township, Range, Section, Section/4			
28	LS 28	650112	S 20N 01W	33	SE	
29	LS 29	650113	S 20N 01W	34	SW	
30	LS 30	650114	S 20N 01W	34	NW	
31	LS 31	650833	S 20N 01W	5	NE	
32	LS 32	650834	S 20N 01W	4	NW	
33	LS 33	650835	S 20N 01W	4	SW	
34	LS 34	650836	S 20N 01W	3	SE	
35	LS 35	650837	S 20N 01W	33	NE	
36	LS 36	650838	S 20N 01W	32	NW	
37	LS 37	650839	S 20N 01W	32	NE	
38	LS 38	650840	S 20N 01W	33	SE	
39	LS 39	650841	S 20N 01W	33	SW	
40	LS 40	650842	S 20N 01W	27	SE	
41	LS 41	650843	S 20N 01W	27	SW	
42	LS 42	650844	S 20N 01W	28	SE	
43	LS 43	650845	S 20N 01W	28	NW	
44	LS 44	650846	S 20N 01W	29	NE	
45	LS 45	650847	S 20N 01W	28	NW	
46	LS 46	650848	S 20N 01W	28	NE	
47	LS 47	650849	S 20N 01W	27	NW	
48	LS 48	650850	S 20N 01W	27	NW	
49	LS 49	650851	S 20N 01W	26	NW	
50	LS 50	650852	S 20N 01W	26	NE	
51	LS 51	650853	S 20N 01W	25	NW	
52	LS 53	650855	S 20N 01 E	30	NW	
53	LS 54	651699	S 19N 01W	3	NE	
54	LS 55	651700	S 19N 01W	2	NW	
55	LS 56	651701	S 19N 01W	2	NE	
56	LS 57	656100	S 19N 01W	1	NW	
57	LS 58	656101	S 19N 01W	1	NE	
58	LS 59	656102	S 19N 01E	6	NW	
59	LS 60	656103	S 19N 01E	6	NE	
60	LS 61	656104	S 19N 01W	3	SE	
61	LS 62	656105	S 19N 01W	2	SW	
62	LS 63	656106	S 19N 01W	2	SW	

APPENDIX B Drillhole Collar Locations by Year

Table B - 1 Collar Locations for 1984 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth°	Dip°	
CB1	Ensearch	371821.88	6851925.40	1311.68	148.74	0.0	90.0	Coleman
CB2	Ensearch	371821.88	6851925.40	1311.68	250.55	0.0	55.0	Coleman
CB3	Ensearch	371821.88	6851925.40	1311.68	230.12	300.0	45.0	Coleman
CB4	Ensearch	371821.88	6851925.40	1311.68	220.07	60.0	58.0	Coleman
COLE1	Ensearch	371987.20	6851859.46	1286.16	242.93	0.0	90.0	Coleman
COLE2	Ensearch	371987.20	6851859.46	1286.16	227.69	75.0	40.0	Coleman
COLE3	Ensearch	371987.20	6851859.46	1286.16	167.64	345.0	60.0	Coleman
LSA_1	Ensearch	372651.91	6851898.22	903.46	338.12	19.9	-3.4	Lucky Shot
LSA_2	Ensearch	372649.38	6851898.51	903.46	339.51	348.4	-3.2	Lucky Shot
LSA_3	Ensearch	372653.13	6851897.15	903.46	198.58	29.2	-30.1	Lucky Shot
LSA_4	Ensearch	372652.18	6851898.54	903.46	177.99	349.4	-33.7	Lucky Shot
LSB_1	Ensearch	372634.32	6851886.31	903.46	236.06	318.7	-26.3	Lucky Shot
LSB_2	Ensearch	372633.95	6851883.79	903.46	389.07	304.6	-10.5	Lucky Shot
LSB_3	Ensearch	372634.61	6851887.06	903.46	386.61	321.2	-6.5	Lucky Shot
LSB_4	Ensearch	372634.90	6851887.94	903.46	314.82	336.4	-14.0	Lucky Shot
LSB_5	Ensearch	372637.68	6851887.56	903.46	234.23	339.7	-24.1	Lucky Shot
LSB_6	Ensearch	372634.04	6851885.82	903.46	306.19	311.6	-15.9	Lucky Shot
LSB_7	Ensearch	372640.64	6851885.39	903.46	237.70	5.3	-12.9	Lucky Shot

Table B - 2 Collar Locations for 2005 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth°	Dip°	
C05-08	Full Metal Minerals	371954.23	6851931.45	1307.22	146.34	176.8	70.9	Coleman
C05-09	Full Metal Minerals	371954.18	6851931.99	1307.40	182.93	0.0	90.0	Coleman
C05-10	Full Metal Minerals	372024.78	6851823.97	1276.88	57.92	184.9	65.7	Coleman
C05-11	Full Metal Minerals	372024.76	6851824.32	1277.03	91.44	0.0	90.0	Coleman
C05-12	Full Metal Minerals	371907.07	6851930.34	1308.38	173.85	0.0	90.0	Coleman
C05-13	Full Metal Minerals	371907.07	6851930.34	1308.38	146.31	172.7	60.6	Coleman
C05-14	Full Metal Minerals	371950.96	6851852.58	1287.07	121.92	183.0	63.6	Coleman

Table B - 3 Collar Locations for 2006 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth°	Dip°	
C06-15	Full Metal Minerals	371883.69	6851959.45	1318.39	155.45	181.0	64.0	Coleman
C06-16	Full Metal Minerals	371883.69	6851959.45	1318.39	149.37	181.0	73.0	Coleman
C06-17	Full Metal Minerals	371883.83	6851960.03	1318.06	179.83	181.0	82.0	Coleman
C06-18	Full Metal Minerals	371883.79	6851960.28	1318.02	173.74	0.0	90.0	Coleman
C06-19	Full Metal Minerals	371931.27	6851957.26	1314.80	164.59	176.0	63.0	Coleman
C06-20	Full Metal Minerals	371931.27	6851957.26	1314.84	152.19	176.0	73.0	Coleman
C06-21	Full Metal Minerals	371931.27	6851957.26	1314.89	176.78	176.0	82.0	Coleman
C06-22	Full Metal Minerals	371931.27	6851957.26	1314.87	204.36	0.0	90.0	Coleman
C06-23	Full Metal Minerals	371980.58	6851953.98	1308.57	179.83	0.0	90.0	Coleman
C06-24	Full Metal Minerals	371980.52	6851953.76	1308.57	170.69	186.0	82.0	Coleman
C06-25	Full Metal Minerals	371980.49	6851953.55	1308.51	167.78	186.0	74.5	Coleman
C06-26	Full Metal Minerals	371980.43	6851953.22	1308.95	168.78	186.0	62.0	Coleman
C06-27	Full Metal Minerals	371998.31	6851942.80	1305.14	170.69	0.0	90.0	Coleman
C06-28	Full Metal Minerals	371998.31	6851942.53	1305.18	181.82	182.0	81.5	Coleman
C06-29	Full Metal Minerals	371998.32	6851942.37	1305.22	131.10	182.0	75.0	Coleman
C06-30	Full Metal Minerals	371998.27	6851942.21	1305.20	146.30	181.4	61.7	Coleman
C06-31	Full Metal Minerals	371998.22	6851941.90	1305.58	149.40	182.0	57.0	Coleman
C06-32	Full Metal Minerals	371840.00	6851950.93	1317.64	198.12	0.0	90.0	Coleman
C06-33	Full Metal Minerals	371839.87	6851950.62	1317.62	228.60	183.0	63.4	Coleman
C06-34	Full Metal Minerals	371840.00	6851950.31	1317.42	234.70	178.8	68.7	Coleman
C06-35	Full Metal Minerals	371840.07	6851951.41	1317.81	246.90	178.0	78.0	Coleman
C06-36	Full Metal Minerals	372027.80	6851959.02	1303.21	182.88	0.0	90.0	Coleman
C06-37	Full Metal Minerals	372027.78	6851958.86	1303.17	180.10	179.1	82.5	Coleman
C06-38	Full Metal Minerals	372027.76	6851958.72	1303.15	167.64	176.8	72.7	Coleman
C06-39	Full Metal Minerals	372027.75	6851958.60	1303.17	152.40	178.1	77.0	Coleman
C06-40	Full Metal Minerals	372027.67	6851958.40	1303.13	152.40	180.0	53.1	Coleman
C06-41	Full Metal Minerals	371908.67	6851907.71	1302.55	132.89	0.0	90.0	Coleman
C06-42	Full Metal Minerals	371908.67	6851907.55	1302.52	192.03	173.3	78.5	Coleman
C06-43	Full Metal Minerals	371921.64	6852174.32	1166.49	277.37	0.0	90.0	Coleman
C06-44	Full Metal Minerals	371921.73	6852173.28	1166.91	130.76	186.0	50.0	Coleman
C06-45	Full Metal Minerals	371921.68	6852173.81	1166.69	109.73	186.4	72.3	Coleman
C06-46	Full Metal Minerals	372007.04	6852020.27	1270.79	170.69	176.5	85.8	Coleman
C06-47	Full Metal Minerals	372002.43	6852020.93	1269.49	170.69	176.3	78.4	Coleman
C06-48	Full Metal Minerals	372002.43	6852020.93	1269.40	149.35	176.0	71.8	Coleman
C06-49	Full Metal Minerals	372007.06	6852019.87	1270.85	161.56	185.7	63.5	Coleman
C06-50	Full Metal Minerals	371930.29	6851866.23	1291.96	118.87	181.0	83.0	Coleman
C06-51	Full Metal Minerals	371930.28	6851865.76	1291.86	115.82	181.8	76.2	Coleman
C06-52	Full Metal Minerals	371930.22	6851865.17	1291.69	140.22	181.8	61.3	Coleman
C06-53	Full Metal Minerals	371977.59	6851872.69	1290.96	115.82	181.0	85.0	Coleman
C06-54	Full Metal Minerals	371977.56	6851872.33	1290.88	111.51	181.0	77.0	Coleman
C06-55	Full Metal Minerals	371977.66	6851871.97	1290.77	121.91	181.4	62.2	Coleman
C06-56	Full Metal Minerals	371977.65	6851871.08	1290.52	114.45	178.2	46.2	Coleman
C06-57	Full Metal Minerals	371906.03	6851843.24	1285.32	167.65	0.0	90.0	Coleman
C06-58	Full Metal Minerals	371905.99	6851842.51	1285.07	109.74	182.1	73.3	Coleman
C06-59	Full Metal Minerals	371906.04	6851840.99	1284.36	109.74	182.1	54.1	Coleman
C06-60	Full Metal Minerals	371959.26	6852007.77	1276.93	167.64	0.0	90.0	Coleman

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth˚	Dip˚	
C06-61	Full Metal Minerals	371956.92	6852007.83	1280.00	167.63	180.0	81.0	Coleman
C06-62	Full Metal Minerals	371956.92	6852007.83	1280.00	155.50	180.8	72.7	Coleman
C06-63	Full Metal Minerals	371959.19	6852007.30	1277.46	158.50	182.9	63.0	Coleman
C06-64	Full Metal Minerals	372076.85	6852010.32	1293.95	204.23	0.0	90.0	Coleman
C06-65	Full Metal Minerals	372076.86	6852010.17	1293.88	213.38	178.4	83.4	Coleman
C06-66	Full Metal Minerals	372076.78	6852010.05	1293.97	213.36	180.0	76.0	Coleman
C06-67	Full Metal Minerals	372076.79	6852009.88	1294.04	188.70	179.2	70.3	Coleman
C06-68	Full Metal Minerals	372074.81	6851959.04	1295.75	182.90	186.0	85.0	Coleman
C06-69	Full Metal Minerals	372074.83	6851958.78	1295.65	189.00	183.6	79.1	Coleman
C06-70	Full Metal Minerals	372074.79	6851958.53	1295.66	173.73	183.6	70.1	Coleman
C06-71	Full Metal Minerals	372074.79	6851958.25	1295.62	164.59	182.7	59.0	Coleman
C06-72	Full Metal Minerals	371938.44	6852062.44	1235.36	155.45	0.0	90.0	Coleman
C06-73	Full Metal Minerals	371935.52	6852060.33	1235.96	155.40	183.3	80.5	Coleman
C06-74	Full Metal Minerals	371935.52	6852060.33	1235.96	140.21	183.2	72.4	Coleman
C06-75	Full Metal Minerals	371938.39	6852061.95	1235.38	146.30	180.0	61.0	Coleman
C06-76	Full Metal Minerals	371938.37	6852061.77	1235.38	149.36	182.0	52.5	Coleman
C06-77	Full Metal Minerals	372081.89	6852165.17	1241.21	207.26	182.0	68.0	Coleman
C06-78	Full Metal Minerals	372081.92	6852165.65	1241.38	272.50	0.0	90.0	Coleman
C06-79	Full Metal Minerals	372081.90	6852165.38	1241.33	219.46	175.5	74.2	Coleman
C06-80	Full Metal Minerals	372081.91	6852165.52	1241.28	228.60	182.3	79.8	Coleman
C06-81	Full Metal Minerals	371955.37	6851905.76	1301.41	154.21	0.0	90.0	Coleman
C06-82	Full Metal Minerals	371955.32	6851905.04	1301.16	128.76	184.2	68.3	Coleman
C06-83	Full Metal Minerals	371955.29	6851904.54	1301.31	143.27	181.9	55.5	Coleman
C06-84	Full Metal Minerals	372629.23	6852048.98	1336.96	422.15	0.0	90.0	Lucky Shot
C06-86	Full Metal Minerals	372041.22	6851870.22	1284.00	36.59	0.0	90.0	Coleman
C06-89	Full Metal Minerals	373782.49	6852267.08	901.00	324.31	0.0	90.0	Murphy
C06-90	Full Metal Minerals	373782.49	6852267.08	901.00	362.72	186.1	84.4	Murphy
N06-16	Full Metal Minerals	377756.09	6852960.50	1317.98	262.14	0.0	90.0	Nippon
N06-17	Full Metal Minerals	377756.09	6852960.50	1317.98	204.23	130.1	77.0	Nippon
N06-18	Full Metal Minerals	377756.09	6852960.50	1317.98	292.61	88.7	53.2	Nippon
N06-19	Full Metal Minerals	377846.08	6852912.49	1280.76	246.90	0.0	90.0	Nippon
N06-20	Full Metal Minerals	377846.08	6852913.39	1280.76	182.90	89.4	70.1	Nippon
N06-21	Full Metal Minerals	377860.08	6852793.49	1236.26	128.33	0.0	90.0	Nippon
N06-22	Full Metal Minerals	377860.08	6852793.49	1236.00	109.73	93.8	69.0	Nippon

Table B - 4 Collar Locations for 2007 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth°	Dip°	
C07-85	Full Metal Minerals	372629.23	6852048.98	1336.96	379.63	180.0	85.0	Lucky Shot
C07-91	Full Metal Minerals	373856.01	6852278.05	874.49	316.99	0.0	90.0	Murphy
C07-92	Full Metal Minerals	373856.21	6852277.86	874.44	333.27	191.4	80.3	Murphy
C07-93	Full Metal Minerals	373949.72	6852275.61	870.16	304.04	0.0	90.0	Murphy
C07-94	Full Metal Minerals	373949.72	6852275.61	870.16	310.27	175.0	80.0	Murphy
C07-95	Full Metal Minerals	371794.33	6851959.32	1321.32	257.86	31.4	89.7	Coleman
C07-96	Full Metal Minerals	371794.33	6851959.32	1321.32	267.92	180.0	83.0	Coleman
C07-97	Full Metal Minerals	371794.34	6851959.28	1321.41	281.94	180.0	75.0	Coleman
C07-98	Full Metal Minerals	371794.32	6851959.35	1321.47	292.15	181.0	67.8	Coleman
C07-99	Full Metal Minerals	374055.80	6852268.23	878.66	313.64	181.0	88.5	Murphy
C07-100	Full Metal Minerals	371763.31	6851976.83	1322.62	227.53	0.0	90.0	Coleman
C07-101	Full Metal Minerals	371763.31	6851976.83	1322.62	231.65	180.0	80.0	Coleman
C07-102	Full Metal Minerals	371763.31	6851976.83	1322.62	259.08	180.0	75.0	Coleman
C07-103	Full Metal Minerals	371763.31	6851976.83	1322.62	265.18	179.0	69.5	Coleman
C07-104	Full Metal Minerals	372629.23	6852048.98	1336.96	373.99	181.0	80.5	Lucky Shot
C07-105	Full Metal Minerals	372216.77	6851984.72	1284.56	218.54	0.0	90.0	Coleman
C07-106	Full Metal Minerals	372216.74	6851984.59	1284.59	257.56	174.0	82.1	Coleman
C07-107	Full Metal Minerals	372216.53	6851983.60	1283.91	209.40	177.0	73.0	Coleman
C07-108	Full Metal Minerals	372216.79	6851984.36	1284.49	182.88	178.0	64.0	Coleman
C07-109	Full Metal Minerals	371891.53	6852052.61	1247.22	175.56	0.0	90.0	Coleman
C07-110	Full Metal Minerals	371891.53	6852052.61	1247.22	177.09	175.6	81.1	Coleman
C07-111	Full Metal Minerals	371891.53	6852052.61	1247.22	39.32	180.0	72.0	Coleman
C07-112	Full Metal Minerals	371891.53	6852052.61	1247.22	148.44	170.0	59.6	Coleman
C07-113	Full Metal Minerals	371891.53	6852052.61	1247.22	195.68	180.0	72.0	Coleman
C07-114	Full Metal Minerals	372216.52	6851982.81	1283.84	182.21	173.0	55.2	Coleman
C07-115	Full Metal Minerals	372212.03	6851872.51	1262.74	161.24	0.0	90.0	Coleman
C07-116	Full Metal Minerals	372212.03	6851872.51	1262.74	145.08	180.0	70.0	Coleman
C07-117	Full Metal Minerals	372212.03	6851872.51	1262.74	140.82	180.0	50.0	Coleman
C07-118	Full Metal Minerals	372269.84	6851989.37	1285.19	216.41	0.0	90.0	Coleman
C07-119	Full Metal Minerals	372269.84	6851989.37	1285.19	219.46	173.0	81.0	Coleman
C07-120	Full Metal Minerals	372269.84	6851989.37	1285.19	195.07	170.0	73.0	Coleman
C07-121	Full Metal Minerals	372269.84	6851989.37	1285.19	185.93	177.0	64.0	Coleman
C07-122	Full Metal Minerals	372269.84	6851989.37	1285.19	204.22	180.0	54.0	Coleman
C07-123	Full Metal Minerals	372031.25	6852167.50	1222.23	252.98	0.0	90.0	Coleman
C07-124	Full Metal Minerals	372030.42	6852167.71	1221.53	201.17	180.0	81.0	Coleman
C07-125	Full Metal Minerals	372030.42	6852167.71	1221.53	205.74	180.0	73.0	Coleman
C07-126	Full Metal Minerals	372030.42	6852167.71	1221.53	203.61	180.0	64.0	Coleman
C07-127	Full Metal Minerals	372031.15	6852166.61	1221.58	179.22	186.0	54.0	Coleman
C07-128	Full Metal Minerals	371811.87	6852058.48	1252.90	253.61	0.0	90.0	Coleman
C07-129	Full Metal Minerals	371811.87	6852058.48	1252.90	250.55	180.0	81.0	Coleman
C07-130	Full Metal Minerals	371811.87	6852058.48	1252.90	204.22	180.0	73.0	Coleman
C07-131	Full Metal Minerals	371811.87	6852058.48	1252.90	219.46	180.0	64.0	Coleman
C07-133	Full Metal Minerals	371860.67	6852146.86	1182.17	228.60	0.0	90.0	Coleman
C07-134	Full Metal Minerals	371860.57	6852147.43	1181.91	228.60	177.0	83.0	Coleman
C07-135	Full Metal Minerals	371860.54	6852147.04	1181.91	85.34	180.0	73.0	Coleman
C07-136	Full Metal Minerals	371860.56	6852146.93	1181.87	213.06	180.0	64.0	Coleman

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth˚	Dip˚	
C07-137	Full Metal Minerals	371860.46	6852147.05	1181.92	151.79	180.0	52.0	Coleman
C07-138	Full Metal Minerals	371528.23	6852344.57	1118.79	323.09	0.0	90.0	Coleman
C07-139	Full Metal Minerals	371528.05	6852344.23	1118.92	326.14	180.0	63.0	Coleman
C07-140	Full Metal Minerals	372441.80	6852158.04	1333.97	475.49	0.0	90.0	Lucky Shot
C07-143	Full Metal Minerals	372735.40	6852217.56	1353.82	457.20	180.0	63.0	Lucky Shot
C07-144	Full Metal Minerals	372735.61	6852217.84	1353.18	481.60	180.0	78.0	Lucky Shot
C07-145	Full Metal Minerals	372413.20	6852407.34	1188.57	396.20	0.0	90.0	Lucky Shot
C07-146	Full Metal Minerals	372306.27	6851361.28	914.26	295.66	0.0	90.0	Coleman

Table B - 5 Collar Locations for 2008 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth˚	Dip˚	
WB08-01	Full Metal Minerals	373298.34	6852110.87	1029.85	227.48	165.0	87.5	War Baby
WB08-02	Full Metal Minerals	373297.87	6852108.09	1029.40	152.40	171.4	75.5	War Baby
WB08-03	Full Metal Minerals	373297.93	6852106.98	1029.28	142.73	172.0	64.4	War Baby
WB08-04	Full Metal Minerals	373268.37	6852110.27	1051.70	143.26	0.0	90.0	War Baby
WB08-05	Full Metal Minerals	373298.56	6852111.90	1029.69	141.10	343.6	81.8	War Baby
WB08-06	Full Metal Minerals	373298.81	6852113.07	1029.60	207.26	349.1	64.0	War Baby
WB08-07	Full Metal Minerals	373243.63	6852100.45	1058.86	283.46	243.6	54.8	War Baby
WB08-08	Full Metal Minerals	373243.49	6852100.38	1059.09	368.81	245.2	46.9	War Baby
WB08-09	Full Metal Minerals	373114.00	6852226.00	1204.00	356.62	165.0	89.0	War Baby
WB08-10	Full Metal Minerals	373114.00	6852226.00	1204.00	365.76	157.8	86.3	War Baby

Table B - 6 Collar Locations for 2009 Drilling Campaign (Negative Dips Point Up)

Drillhole ID	Exploration Company	Coordinates NAD 1983 UTM Zone 6 North			Maximum Depth (m)	Collar Survey		Resource Area
		Easting	Northing	Elevation (m)		Azimuth°	Dip°	
C09-147	Full Metal Minerals	371911.60	6851963.00	1317.90	170.40	185.5	84.4	Coleman
C09-148	Full Metal Minerals	371912.30	6851966.50	1318.50	191.10	2.7	84.1	Coleman
C09-149	Full Metal Minerals	371862.30	6851959.01	1319.10	189.00	175.5	67.2	Coleman
C09-150	Full Metal Minerals	371862.30	6851959.40	1319.20	183.20	177.4	76.9	Coleman
C09-151	Full Metal Minerals	371862.40	6851959.70	1319.20	190.20	167.6	87.1	Coleman
C09-152	Full Metal Minerals	371954.80	6851948.90	1310.50	179.20	153.9	88.7	Coleman
C09-153	Full Metal Minerals	372048.60	6851981.60	1297.60	182.30	179.9	69.9	Coleman
C09-154	Full Metal Minerals	372048.70	6851981.70	1297.50	173.10	185.6	78.9	Coleman
C09-155	Full Metal Minerals	372048.80	6851982.00	1297.60	182.30	45.1	89.2	Coleman
C09-156	Full Metal Minerals	371880.70	6852026.70	1265.00	151.80	176.6	68.2	Coleman
C09-157	Full Metal Minerals	371880.70	6852026.70	1265.00	135.50	181.6	76.7	Coleman
C09-158	Full Metal Minerals	371880.70	6852026.70	1265.00	151.80	156.6	86.0	Coleman
C09-159	Full Metal Minerals	371974.20	6852040.70	1251.40	154.50	175.7	61.9	Coleman
C09-160	Full Metal Minerals	371974.20	6852040.75	1251.26	151.20	171.9	72.8	Coleman
C09-161	Full Metal Minerals	371975.06	6852040.50	1251.26	160.60	173.7	84.0	Coleman
C09-162	Full Metal Minerals	371959.60	6852079.70	1224.60	166.40	184.6	67.3	Coleman
C09-163	Full Metal Minerals	371959.40	6852080.20	1224.50	166.40	178.8	77.8	Coleman
C09-164	Full Metal Minerals	372024.90	6852038.00	1262.40	169.80	181.0	67.5	Coleman
C09-165	Full Metal Minerals	372024.80	6852038.20	1262.30	142.60	187.1	76.1	Coleman
C09-166	Full Metal Minerals	372024.80	6852038.40	1262.30	146.60	180.9	79.1	Coleman
C09-167	Full Metal Minerals	371923.90	6851952.00	1314.75	156.70	178.8	77.4	Coleman
C09-168	Full Metal Minerals	371923.90	6851952.20	1314.80	159.70	177.9	83.8	Coleman
C09-169	Full Metal Minerals	371896.43	6851948.40	1314.40	141.70	180.0	78.2	Coleman
C09-170	Full Metal Minerals	371896.45	6851948.21	1314.49	153.90	179.1	69.6	Coleman
C09-171	Full Metal Minerals	371896.50	6851948.60	1314.50	163.10	173.7	82.9	Coleman
C09-172	Full Metal Minerals	371896.50	6851951.90	1315.20	179.20	351.0	82.0	Coleman
C09-173	Full Metal Minerals	371896.80	6851951.80	1315.20	154.80	0.0	90.0	Coleman
C09-174	Full Metal Minerals	371942.20	6851949.90	1312.40	160.00	179.0	81.0	Coleman
C09-175	Full Metal Minerals	371942.10	6851950.20	1312.40	169.20	205.9	89.7	Coleman

APPENDIX C Drillhole Significant Intercepts by Vein

Table C - 1 Drillhole Composites used in Coleman Vein 01 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C05-08	128.68	129.24	121.37	15.80
C05-09	141.83	143.71	33.25	4.40
C05-10	40.54	42.98	3.25	0.75
C05-11	46.69	48.77	3.09	0.65
C05-12	132.66	133.40	1206.30	68.90
C05-14	69.19	70.71	0.17	0.20
C06-16	133.44	137.16	4.21	0.42
C06-17	139.65	140.12	1.92	0.60
C06-18	148.30	148.85	64.70	9.90
C06-19	145.60	146.40	22.73	3.75
C06-20	144.60	145.00	75.70	3.60
C06-21	150.75	151.30	7.39	1.00
C06-22	158.50	160.00	0.21	0.70
C06-23	158.67	160.86	4.75	0.67
C06-24	152.25	153.15	4.21	0.38
C06-25	144.57	146.19	2.31	0.38
C06-26	139.62	142.67	7.69	0.68
C06-27	149.91	150.71	2.75	0.40
C06-28	142.00	143.30	0.68	0.50
C06-30	132.50	132.82	3.15	0.50
C06-31	133.40	134.27	1.57	0.21
C06-36	157.40	159.47	11.80	0.87
C06-37	148.42	149.35	4.72	0.33
C06-38	142.05	142.95	1.57	0.30
C06-39	143.69	144.69	0.89	0.10
C06-40	137.95	138.38	0.60	0.20
C06-41	116.35	117.27	30.25	2.22
C06-43	113.35	114.22	2.34	0.48
C06-44	107.75	108.25	3.95	0.20
C06-45	105.60	106.00	1.07	0.10
C06-46	150.00	151.70	8.21	0.48
C06-47	143.80	145.30	0.03	0.20
C06-48	140.50	142.00	6.34	0.30
C06-49	139.50	141.90	5.23	0.63
C06-50	83.45	83.85	12.10	2.60
C06-53	95.59	96.00	2.98	0.60
C06-54	88.60	89.80	7.10	0.86
C06-55	84.00	85.00	0.50	0.10
C06-56	83.10	84.42	2.07	0.83
C06-60	149.80	151.20	7.46	0.10
C06-61	145.50	147.00	0.89	0.30
C06-62	138.60	139.00	33.00	48.40
C06-63	136.00	137.50	0.03	0.10
C06-64	182.00	183.50	1.11	0.30
C06-65	175.00	175.50	1.56	0.10
C06-66	164.00	164.40	5.90	0.50
C06-67	161.52	163.00	3.21	0.30
C06-68	150.00	151.00	1.08	0.01
C06-69	139.50	141.00	3.64	0.50
C06-70	131.50	133.30	0.65	0.01
C06-71	132.00	134.00	1.52	0.55
C06-72	132.00	133.00	5.53	0.30
C06-73	118.50	120.00	0.24	0.10
C06-74	115.00	115.50	108.50	4.10
C06-75	118.00	118.50	17.40	0.01

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-76	121.00	121.60	5.54	0.01
C06-77	190.50	191.00	0.18	0.10
C06-78	236.00	237.00	0.48	0.20
C06-79	201.32	201.72	12.55	0.50
C06-80	209.00	210.10	3.82	0.50
C06-81	123.00	124.50	0.85	0.20
C06-82	108.10	108.50	0.85	0.20
C06-83	110.00	110.50	10.10	1.10
C07-105	179.00	180.00	0.89	0.10
C07-106	152.00	153.00	0.63	0.20
C07-107	145.00	146.00	0.58	0.10
C07-108	139.00	142.50	2.38	0.28
C07-109	136.00	137.00	0.61	0.20
C07-110	121.50	122.00	18.60	0.80
C07-112	120.00	120.50	1.30	0.40
C07-113	118.00	119.00	0.11	0.10
C07-114	133.00	135.00	1.73	0.10
C07-115	42.50	43.50	2.37	0.50
C07-116	38.50	39.50	3.29	0.60
C07-117	32.00	33.00	2.72	0.20
C07-118	184.00	185.00	1.89	0.30
C07-119	170.00	171.00	1.83	0.30
C07-120	163.00	165.00	0.95	0.15
C07-121	147.00	148.00	0.21	0.10
C07-122	137.00	138.00	0.50	0.10
C07-123	198.00	199.00	0.69	0.10
C07-124	179.00	180.00	1.39	0.40
C07-125	172.00	174.00	2.37	0.38
C07-126	172.00	172.50	0.93	0.10
C07-136	107.00	107.50	4.53	0.10
C07-137	106.50	107.00	0.96	0.10
C09-147	152.60	153.00	1.87	1.50
C09-148	174.50	174.90	9.15	1.40
C09-151	140.20	140.70	5.81	0.60
C09-152	154.20	155.60	55.50	8.90
C09-153	144.80	146.50	0.87	0.10
C09-154	150.90	153.30	3.57	0.43
C09-155	166.60	167.00	4.85	0.50
C09-156	116.80	117.90	0.25	0.40
C09-157	119.60	120.50	1.47	0.10
C09-158	132.20	133.20	0.53	0.40
C09-159	128.90	130.10	3.70	0.26
C09-160	130.00	130.40	0.01	0.20
C09-161	138.30	139.30	3.93	0.80
C09-162	115.00	115.90	17.53	0.63
C09-163	124.30	124.70	0.14	0.10
C09-164	140.60	142.50	2.49	0.49
C09-165	142.10	142.60	1.58	0.50
C09-166	142.00	143.50	0.30	0.20
C09-167	142.20	142.90	5.45	1.00
C09-168	147.10	147.50	28.60	2.40
C09-169	131.30	131.80	1.21	0.30
C09-171	135.00	135.80	30.18	2.15
C09-172	159.10	159.90	3.73	1.00
C09-173	146.00	146.60	2.81	0.10
C09-174	144.20	144.60	41.00	2.90
C09-175	153.20	154.50	20.09	0.88

Table C - 2 Drillhole Composites used in Coleman Vein 02 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C05-12	126.49	127.10	0.01	0.30
C06-16	131.77	132.70	236.81	11.38
C06-17	139.25	139.65	5.52	0.50
C06-18	144.80	146.30	4.02	0.40
C06-21	144.00	144.75	3.52	0.30
C07-112	111.00	111.50	21.30	0.60
C07-113	114.50	115.50	17.68	2.25
C09-147	146.40	147.30	0.00	0.10
C09-148	164.00	164.50	16.80	0.40
C09-151	138.40	138.80	36.30	1.60
C09-156	114.80	115.70	0.99	0.40
C09-157	118.90	119.60	3.03	0.10
C09-158	121.60	122.20	58.20	1.40
C09-168	143.10	143.50	3.40	0.20
C09-169	129.50	131.30	37.73	2.24
C09-171	130.70	132.60	125.48	9.21
C09-172	154.80	155.60	9.76	0.30
C09-173	142.30	144.40	15.97	1.11
C09-175	140.80	141.20	5.43	2.30

Table C - 3 Drillhole Composites used in Coleman Vein 03 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C05-09	138.68	139.60	7.06	0.20
C06-22	154.03	155.55	2.62	0.20
C06-27	124.42	125.60	1.43	0.47
C06-28	117.11	117.51	1.84	0.20
C06-36	132.07	133.39	0.55	0.10
C06-37	127.02	127.72	1.22	0.40
C06-38	121.25	121.92	0.90	0.60
C06-39	124.76	125.71	1.00	0.29
C06-40	113.70	114.55	1.08	0.40
C06-44	91.75	92.25	1.41	0.10
C06-45	83.00	83.40	3.29	0.10
C06-46	134.00	134.63	2.02	1.20
C06-47	126.00	127.50	0.03	0.10
C06-49	118.50	120.00	0.28	0.20
C06-60	138.00	139.50	0.14	0.30
C06-61	136.00	137.00	0.15	0.10
C06-62	131.50	133.00	0.12	0.30
C06-63	133.00	134.50	0.40	0.10
C06-64	168.00	169.50	1.22	0.20
C06-65	162.00	163.00	1.77	0.10
C06-66	148.00	149.00	1.60	0.30
C06-67	149.00	150.64	1.87	0.40
C06-72	116.03	117.00	10.50	0.50
C06-75	110.50	111.00	1.90	0.01
C06-76	115.50	115.90	1.08	0.40
C06-77	179.50	181.00	0.12	0.20
C07-123	164.00	165.00	0.32	0.20
C07-124	157.00	157.50	8.89	1.30
C07-125	153.00	153.50	27.70	0.20
C07-126	151.50	152.50	0.14	0.10
C07-127	153.50	154.50	2.56	0.20
C09-152	149.40	150.20	1.09	0.30
C09-153	120.10	122.10	1.74	0.50
C09-154	130.90	134.50	4.51	0.75
C09-155	149.80	151.20	0.25	0.40
C09-159	109.60	111.20	0.80	0.10
C09-160	116.10	116.50	12.30	0.80
C09-161	121.80	122.80	4.12	0.20
C09-162	109.08	110.24	0.00	0.00
C09-163	109.72	110.68	0.00	0.00

Table C - 4 Drillhole Composites used in Coleman Vein 04 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C05-08	130.91	132.74	9.91	1.45
C05-09	143.71	146.00	36.06	2.73
C05-10	46.02	46.63	2.02	0.70
C05-11	50.90	53.11	2.16	0.40
C05-12	134.11	137.77	14.35	0.78
C05-13	128.93	129.54	1.16	0.30
C05-14	78.64	80.16	2.98	0.50
C06-16	144.73	145.13	0.59	0.20
C06-17	146.98	148.44	3.28	0.65
C06-18	153.30	153.70	1.69	9.00
C06-19	146.40	146.80	358.00	20.00
C06-20	145.40	146.50	5.67	0.63
C06-21	152.10	153.35	122.09	6.78
C06-22	161.45	162.00	0.27	0.10
C06-23	161.26	162.45	95.35	6.48
C06-24	153.15	154.55	2.19	0.32
C06-25	150.20	150.59	2.49	0.60
C06-26	145.29	145.69	0.73	0.40
C06-27	156.21	158.04	34.15	1.68
C06-28	150.90	151.65	1.21	0.10
C06-30	134.11	134.59	2.54	0.30
C06-31	135.04	135.57	0.77	0.40
C06-32	146.45	147.40	22.27	1.40
C06-36	167.22	167.94	7.12	0.69
C06-37	161.54	163.34	29.53	3.48
C06-38	154.61	155.07	4.83	0.60
C06-40	148.76	149.35	2.60	0.40
C06-41	119.20	120.98	3.19	0.10
C06-42	116.25	117.25	1.95	0.25
C06-43	116.86	117.27	1.13	0.40
C06-44	110.75	111.75	2.44	0.35
C06-45	108.20	108.60	0.30	0.20
C06-46	152.50	152.90	4.88	0.40
C06-47	146.12	146.50	3.80	0.30
C06-48	143.23	145.70	11.11	1.36
C06-49	144.90	146.40	3.49	0.50
C06-50	90.70	91.70	1.18	0.10
C06-51	87.90	88.90	3.76	0.35
C06-52	90.25	90.75	2.55	4.60
C06-53	98.00	99.50	1.27	0.27
C06-54	94.00	95.50	0.31	0.10
C06-55	85.00	86.00	0.18	0.30
C06-56	84.42	86.10	4.18	0.60
C06-60	152.55	155.15	18.38	0.49
C06-61	147.00	149.50	2.25	0.18
C06-62	140.00	141.38	2.89	0.20
C06-63	137.50	141.50	9.79	0.85
C06-64	185.00	186.50	2.99	0.20
C06-65	177.00	178.50	2.90	0.53
C06-66	169.50	170.82	1.39	0.37
C06-67	170.52	172.00	4.04	0.30
C06-68	161.00	162.13	17.40	1.10
C06-69	153.00	154.00	1.04	0.40
C06-70	142.50	144.00	0.37	0.80
C06-71	143.50	144.00	1.42	0.20
C06-72	139.00	140.50	0.69	0.01
C06-73	127.50	128.00	1.33	0.60

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-74	126.00	126.50	5.93	0.60
C06-75	122.50	123.00	1.11	0.40
C06-76	122.00	123.00	4.35	0.50
C06-77	195.50	197.00	10.55	0.50
C06-78	241.50	243.00	14.25	0.30
C06-79	203.50	204.00	2.05	0.20
C06-80	210.10	212.00	35.46	13.55
C06-81	129.15	131.00	82.59	5.14
C06-82	111.50	113.00	0.57	0.10
C06-83	117.50	118.00	0.53	0.30
C07-105	184.50	185.50	6.47	0.55
C07-106	171.00	171.50	9.26	0.50
C07-109	139.00	140.00	0.33	0.10
C07-110	130.00	131.00	1.33	0.40
C07-112	123.00	123.50	18.40	0.80
C07-113	123.00	124.00	1.80	0.10
C07-115	92.00	93.00	1.36	0.10
C07-118	188.50	190.00	8.18	0.43
C07-119	180.00	181.00	0.57	0.30
C07-120	175.00	176.00	0.67	0.10
C07-123	204.00	205.00	1.36	0.10
C07-124	189.00	191.00	1.71	0.10
C07-125	182.00	183.00	3.12	0.10
C07-126	173.25	173.85	22.90	1.30
C07-127	174.97	175.62	0.00	0.00
C07-128	116.00	116.50	2.30	0.60
C07-129	107.00	108.00	2.83	1.10
C07-136	110.00	110.50	0.11	0.10
C07-137	110.00	110.50	5.24	1.20
C09-147	155.20	156.50	0.21	0.10
C09-148	180.10	180.50	3.94	0.50
C09-149	141.60	142.40	0.58	0.10
C09-150	141.60	143.60	2.56	0.90
C09-151	150.30	151.40	0.00	0.10
C09-152	156.50	157.80	6.86	0.60
C09-153	160.40	161.30	102.00	4.20
C09-154	164.10	164.50	13.15	1.40
C09-155	170.50	171.40	1.12	0.30
C09-156	123.90	125.40	0.24	0.30
C09-157	125.34	126.35	0.00	0.00
C09-158	133.60	134.10	0.23	0.20
C09-159	132.00	133.10	0.11	0.10
C09-160	132.80	134.10	2.05	0.40
C09-161	141.90	142.30	0.29	0.30
C09-162	118.30	118.80	6.33	0.10
C09-163	129.60	130.50	18.47	1.81
C09-164	146.00	146.40	1.39	0.30
C09-167	144.90	145.30	1.17	0.30
C09-168	149.57	150.72	0.00	0.00
C09-169	136.00	136.40	0.24	0.20
C09-170	135.90	136.30	18.25	1.30
C09-171	136.80	138.00	1.32	0.20
C09-172	159.90	160.40	1.46	0.30
C09-173	147.60	148.20	0.76	0.10
C09-174	145.00	145.40	6.02	1.50
C09-175	154.90	156.50	6.14	0.85

Table C - 5 Drillhole Composites used in Coleman Vein 05 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-27	152.18	152.84	3.56	0.56
C06-28	144.50	145.75	0.74	0.10
C06-36	162.75	163.10	0.23	0.10
C06-37	156.10	156.56	1.03	0.70
C06-38	150.15	151.10	0.89	0.31
C06-39	151.06	151.46	154.50	9.60
C06-69	142.00	143.50	1.73	0.30
C06-70	140.50	141.00	0.20	0.01
C07-107	149.00	150.00	0.65	0.30
C07-108	148.50	150.00	0.20	0.10
C07-121	150.00	151.00	0.16	0.10
C07-122	147.00	148.00	0.29	0.10
C09-153	152.30	153.40	0.29	0.10
C09-154	158.00	158.50	1.19	0.40
C09-155	169.30	169.70	1.67	0.50

Table C - 6 Drillhole Composites used in Coleman Vein 07 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C05-12	145.39	146.00	0.45	0.20
C06-17	162.03	163.26	1.56	0.20
C06-22	166.20	167.13	2.55	0.10
C06-41	127.27	127.68	3.82	0.20
C06-75	126.50	128.50	1.91	0.15
C06-76	127.00	127.50	5.91	0.50
C07-109	170.00	171.00	2.00	0.10
C07-110	158.00	158.50	71.60	7.80
C07-112	134.00	134.50	2.74	0.20
C07-113	146.00	147.00	0.66	0.30
C09-147	165.60	166.70	5.91	0.45
C09-148	183.90	184.30	9.76	1.00
C09-150	163.30	163.80	2.30	0.40
C09-151	163.40	164.90	2.25	0.40
C09-156	139.00	139.40	0.53	0.40
C09-170	149.40	150.50	7.32	0.60
C09-171	159.20	160.00	2.36	0.10
C09-175	156.50	157.80	1.24	0.37

Table C - 7 Drillhole Composites used in Coleman Vein 08 Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-33	186.94	188.80	1.77	0.98
C06-34	192.58	193.89	11.71	1.14
C06-35	191.62	192.43	10.51	0.60
C06-52	110.75	113.75	2.33	0.63
C06-56	112.78	114.44	2.69	0.80
C06-57	121.71	123.00	6.60	0.70
C06-58	98.50	100.00	0.03	0.10
C06-59	86.25	89.25	2.85	0.35
C07-98	192.25	192.75	2.03	0.20

Table C - 8 Drillhole Composites used in Lucky Shot Hangingwall Vein Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-84	357.80	358.42	0.17	0.00
C07-104	322.79	324.22	1.90	0.85
C07-140	391.00	392.00	0.66	0.10
C07-143	400.75	401.25	77.20	0.00
C07-144	457.00	458.00	2.44	0.20
C07-145	371.00	372.50	3.32	0.23
C07-85	344.00	344.50	7.58	0.30
LSA_1	228.98	229.84	0.00	0.00
LSA_2	267.94	268.50	0.75	0.00
LSA_3	193.72	193.81	15.24	0.00
LSA_4	155.99	156.60	5.51	0.00
LSB_1	214.70	215.56	4.98	0.00
LSB_2	358.84	360.21	3.33	0.00
LSB_3	346.86	348.91	0.00	0.00
LSB_4	234.33	235.25	0.00	0.00
LSB_5	189.43	190.44	4.39	0.00
LSB_6	272.75	273.05	2.83	0.00
LSB_7	217.17	217.54	3.67	0.00

Table C - 9 Drillhole Composites used in Lucky Shot Footwall Vein Resource

Drillhole ID	From (m)	To (m)	Gold (g/t)	Silver (g/t)
C06-84	370.22	371.23	0.32	0.10
C07-104	339.00	339.49	3.30	0.20
C07-140	401.00	402.00	0.53	0.10
C07-143	411.00	413.00	0.36	0.10
C07-144	471.00	472.00	0.80	0.10
C07-145	387.50	388.00	2.67	0.30
C07-85	352.00	354.00	0.63	0.20
LSA_1	206.23	207.14	3.58	0.00
LSA_2	242.96	244.50	0.00	0.00
LSA_3	177.92	178.38	3.89	0.00
LSA_4	151.59	151.74	55.99	0.00
LSB_1	196.10	196.77	8.21	0.00
LSB_2	340.80	342.23	0.00	0.00
LSB_3	310.74	313.18	1.18	0.00
LSB_4	222.77	223.50	7.12	0.00
LSB_5	186.44	187.08	1.12	0.00
LSB_6	253.80	254.65	1.55	0.00
LSB_7	192.47	193.80	0.00	0.00

APPENDIX D Physical Beneficiation Experiments on Samples from the Willow Creek, Alaska Gold Project Revision 1