TECHNICAL REPORT

TURNER GOLD RESOURCE
and
PRELIMINARY ECONOMIC ASSESSMENT

Josephine County, Oregon, USA

Prepared for:

Green Park Capital Corp.

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1.0 SUMMARY

This Technical Report summarizes the results of a resource estimate and a Preliminary Economic Assessment (PEA) for the Turner Gold Project located in southwestern Oregon, USA. This document was prepared for Green Park Capital Corp. for the purpose of a qualifying transaction with the TSXV.

The Turner Gold Project is held by Green Park Capital Corp who entered into a letter of intent to acquire all of the shares of Josephine Mining Corp. (JMC) on March 26, 2010. See section 4.1.

This technical work was completed by several companies and individuals. Their responsibilities and the qualified persons are listed in Section 2.0 (Introduction).

The Turner Gold Project is located approximately 40 miles southwest of Grants Pass, Oregon. The deposit is a massive sulfide deposit that can be potentially exploited by underground mining methods to produce economic concentrations of gold, copper, zinc, silver, and potentially cobalt. Most of the mineralization is amenable to flotation to produce three concentrates: 1) a copper concentrate, 2) a zinc concentrate and 3) a gold concentrate.

The Turner Gold Project deposit is contained within three patented mining claims which total about 60 acres. An additional 264.55 acres of contiguous private land is controlled by Green Park Capital Corp. through Josephine Mining Corp.

The deposit can be classified as a volcanogenic massive sulfide deposit of the “Cyprus” model. The Turner Gold deposit is ophiolite hosted and is associated with sea floor volcanism and extensional tectonics. There are three zones of mineralization: UHZ, MUZ, and MLZ that appear as semi-tabular bodies that strike 130 degrees (southeast) and dip 35 degrees to the northeast. The mine plan does not currently plan for production from the UHZ so within this text the terms MUZ and MLZ are often referred to as Upper and Lower zones respectively. The strike length of the mineralization as it is currently understood is about 1000 to 1500 ft with a down dip extent of about 800 to 1000 ft.

The mineral resource was developed based on historic drilling that was completed by several companies during the 1980’s. The assay information that was recorded by hand on paper logs were keypunched into an electronic data base for assembly of a block model.

As part of this project, 44 core samples were recovered from the core shed located in O’Brien, Oregon for independent check assay. The results of those assays confirm the presence of gold, copper, silver, zinc, and cobalt. IMC holds the opinion that these recent check assays provide sufficient confidence that mineral resources can be defined in the indicated and inferred categories.

The Turner Gold deposit is currently envisioned to be mined using the Avoca underground mining method to produce 1,250 tons per day of ore to a flotation concentrator. Initial estimates of mining, process, and overhead costs were applied along with initial estimates of process and mining recovery to establish an estimate of mineral resources that have reasonable expectation of economic extraction. The corresponding resources are contained within geometries that can be produced by the selected stoping method and they exceed an estimated $42/ton NSR cutoff.

Table 1-1 summarizes the mineral resources at the Turner Gold Project.

Table 1-1
Turner Gold Deposit
Mineral Resources

Mineral Resource at Metal Prices, $900/oz Gold, $2.00/lb Copper, $12.50/oz Silver, $0.65/lb Zinc														
Category		Cutoff NSR/t	Short Ktons	NSR $/ton	Gold oz/ton	Copper %	Silver oz/ton	Zinc %	Cobalt %	Contained KOzs Gold	Contained KLbs Cu	Contained KOzs Silver	Contained KLbs Zn
Undiluted Indicated		$42.00	2,447	92.88	0.090	1.25	0.31	2.65	0.047				
Mining Recovery	90%		2,202	92.88	0.090	1.25	0.31	2.65	0.047				
Mining Dilution	10%		220	42.26	0.049	0.50	0.16	0.79	0.038				
Recov+Diluted Indicated			2,422	88.27	0.086	1.18	0.30	2.48	0.046	209	57,245	718	120,169
Undiluted Inferred		$42.00	2,084	86.40	0.088	0.99	0.64	2.78	0.036				
Mining Recovery	90%		1,876	86.40	0.088	0.99	0.64	2.78	0.036				
Mining Dilution	10%		188	42.26	0.049	0.50	0.16	0.79	0.038				
Recov+Diluted Inferred			2,064	82.38	0.084	0.94	0.59	2.60	0.036	174	38,991	1,223	107,290

Notes:
Undiluted calculations are from the block model at the $42.00/ton NSR Cutoff
Undiluted calculations require each block to have 4 neighbors above cutoff grade
Dilution grade based on the grade of material surrounding the undiluted tabulation, at a $5.00/ton NSR Cutoff

The resources were estimated by John Marek P.E. of Independent Mining Consultants, Inc. John Marek is a qualified person under NI43-101 for the definition of resources at this type of project. John Marek is independent of the issuer as defined in Section 1.4 if NI43-101.

The PEA mine plan includes both indicated mineral resources and inferred mineral resources. A component of the combined categories is included within the PEA mine plan. Not all of the stated mineral resources are included within the PEA plan. The plan did not include some potentially higher cost ores in the UHZ that are likely oxidized and removed some of the smaller (albeit potentially minable) zones when establishing the mine plan. The PEA mine plan is based on the AVOCA underground mining method.

This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized. IMC holds the opinion that the additional drilling as outlined in the recommendations chapter will add confidence, and could potentially add resources.

Table 1-2 summarizes the PEA mine plan and production schedule including mining dilution and mining recovery. Table 1-2 includes components of both Indicated Mineral Resources and Inferred Mineral Resources. Figure 1-1 is an illustration of the mine development and general deposit layout.

Table 1-2
Mine Production Schedule, Turner Gold Project

Diluted Minable Material Feed to the Process Facility									
Material	Year of Production								
and Grade	1	2	3	4	5	6	7	8	Total
Ore (short tons)	421,246	455,000	455,000	455,000	455,000	455,000	455,000	434,512	3,585,758
Gold (oz/ston)	0.091	0.089	0.089	0.089	0.089	0.089	0.089	0.086	0.089
Copper (%)	1.34	1.22	1.22	1.22	1.22	1.22	1.22	1.10	1.22
Silver (oz/ston)	0.52	0.50	0.50	0.50	0.50	0.50	0.50	0.48	0.50
Zinc (%)	2.48	2.86	2.86	2.86	2.86	2.86	2.86	3.22	2.86
Cobalt (%)	0.040	0.038	0.038	0.040	0.038	0.038	0.038	0.036	0.038
Waste (short tons)	265,228	50,024	50,024	50,024	50,024	50,024	50,024	27,697	593,069

tab18-1.xls

The table includes both indicated and inferred category mineral resources
The cutoff grade for ore on the table is $50.00 NSR per ton

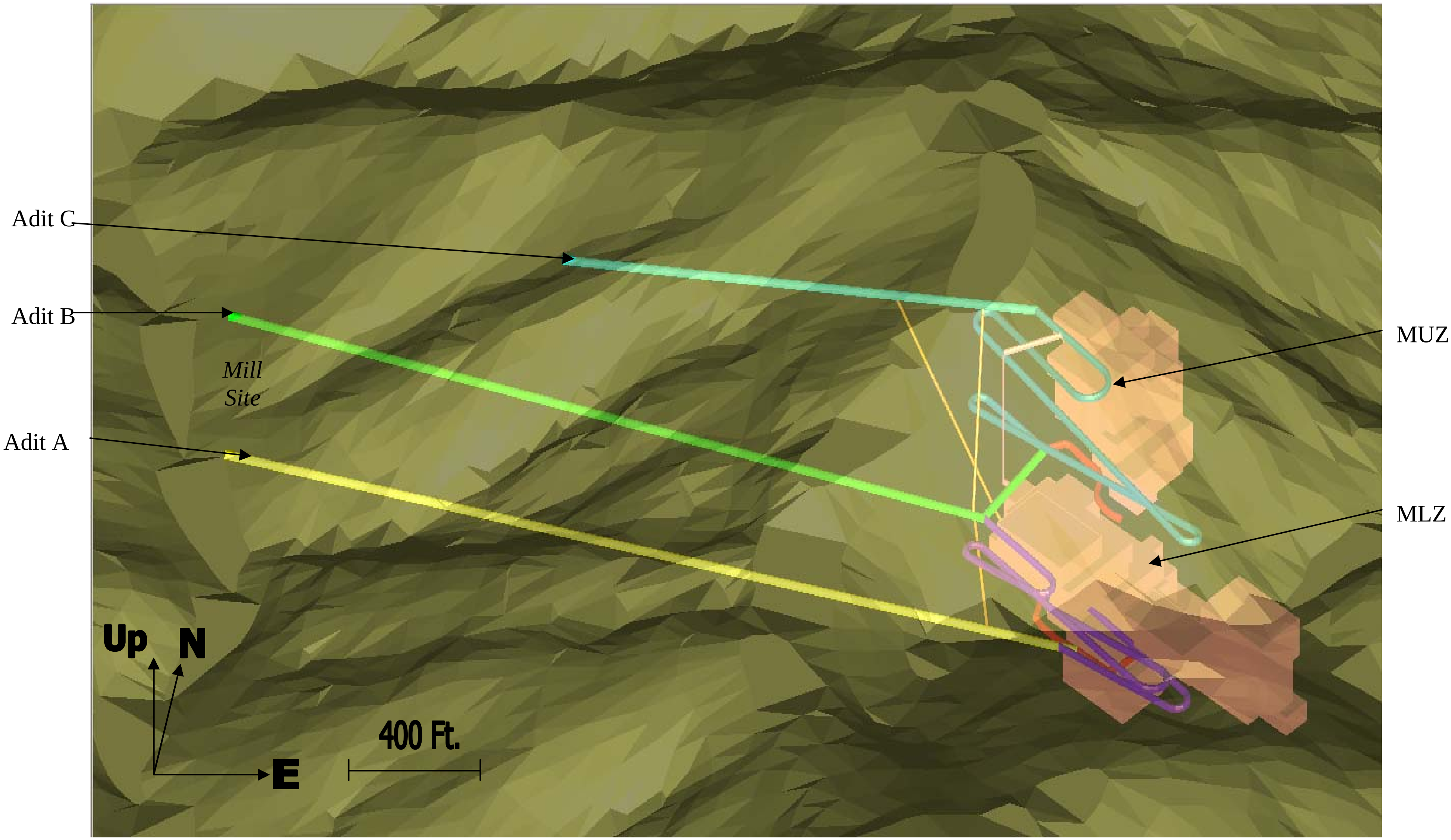
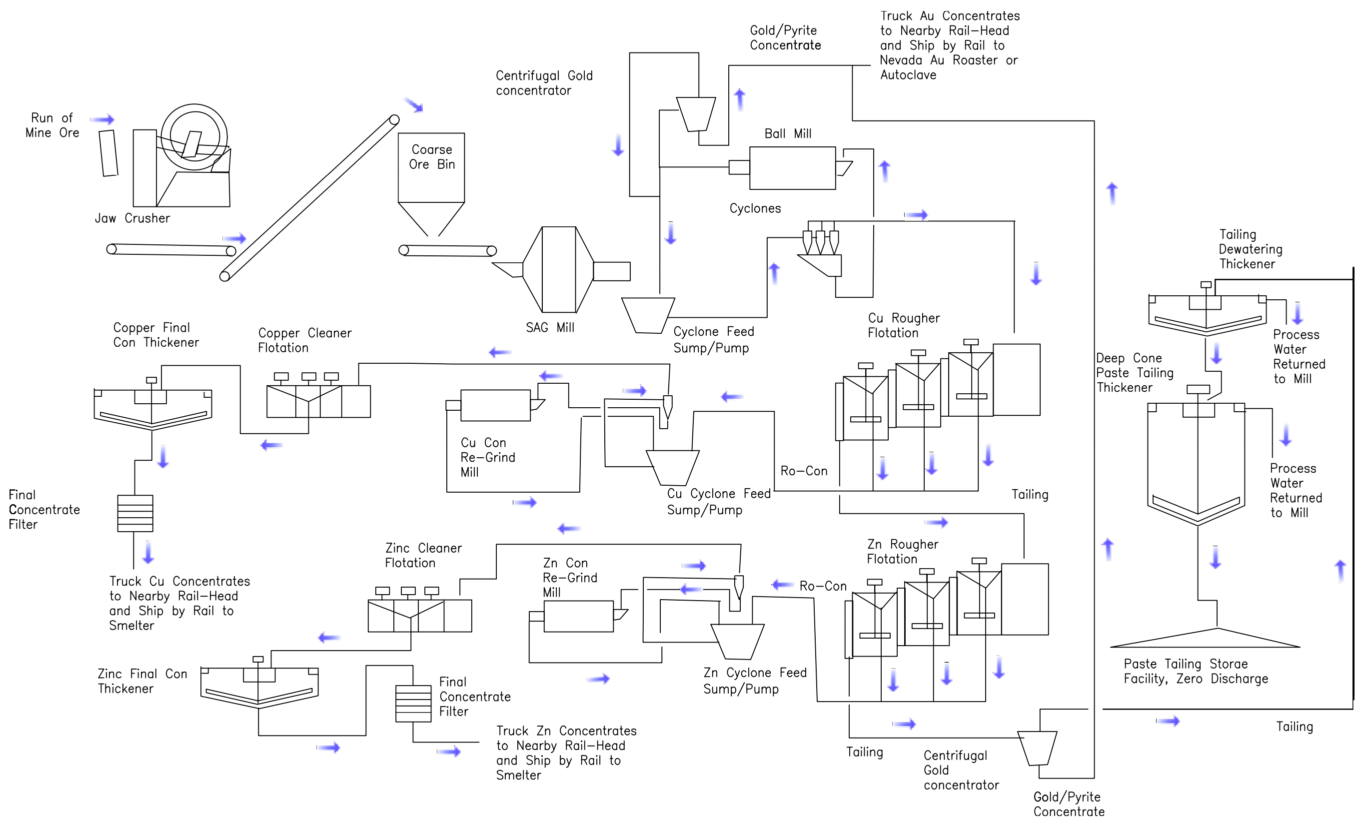


Figure 1-1
Three D View of Development Adits and Minable Zones
Looking North-Northwest

Figure 1-2 is a flow sheet of the mill as currently contemplated.

Figure 1-2



The ore is planned to be delivered to the process facility by truck haulage out of the access adits and ramps. The process plant will crush the ore and utilize a SAG mill followed by a ball mill to reduce the ore for copper and zinc flotation. The target flotation size is a P_{80} of 55 microns.

Mine and Process Operating Costs as developed in Section 18 are summarized below:

Major Cost Category	Cost per Ton Ore
Mining	$27.45
Mill Operations	$14.68
Copper Concentrate Smelting and Refining	$4.25
Zinc Concentrate RLE and Refining	$9.22
Precious Metal Refining	$0.11
Support, Facilities, G&A, Shipping	$6.61
Total	$62.32

Project Capital Costs are developed in Section 18 and are summarized below:

Major Cost Category	Cost in Millions USD
Mining Equipment	$ 9.76
Initial Mine Development	$ 7.56
Process Plant and Infrastructure	$34.36
Owner's Cost	$ 5.17
Total Project Capital Estimate	$56.85

The PEA economic analysis utilized the above costs as well as metal prices of:

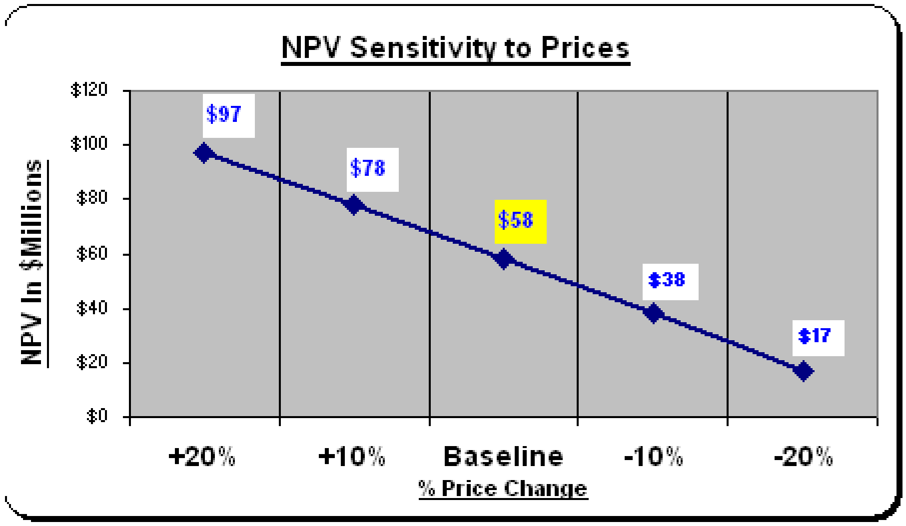
Base Case Metal Prices for PEA	
Gold	$900 /troy oz
Copper	$2.00 /lb
Zinc	$0.65 /lb
Silver	$12.50 /troy oz

No economic benefit was applied to cobalt.

The base case economic analysis (Table 18-14) indicates that the project's NPV at 8% discount rate is $58.5 million, Internal Rate of Return (IRR) of 32.2% and a payback period of 2.6 years from beginning of production. (This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized.

Price sensitivity to project's NPV is shown in the Figure 1-3 below. This chart illustrates different NPV values at the base case prices as well as +/-10% and +/-20% changes in metals' prices.

Figure 1-3 Price Sensitivity



The cost sensitivity charts below indicates that the NPV of the project is much less sensitive to the changes in both Initial Capital cost and Operating cost relative to price.

Figure 1-4 Initial Capital Cost Sensitivity

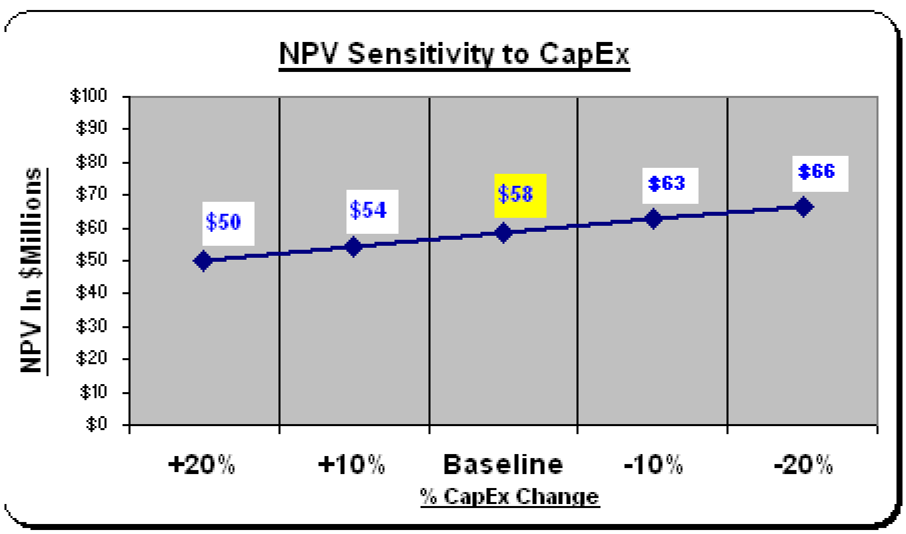
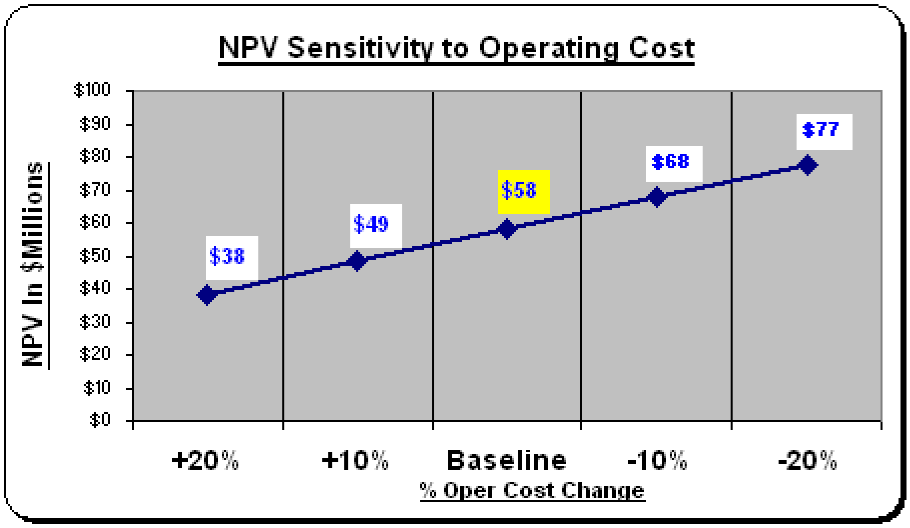


Figure 1-5 Operating Cost Sensitivity



The results of the PEA indicate that the Turner Gold project has the potential to become an economic producer of gold, copper, silver and zinc in the form of three concentrates of: 1) copper, 2) zinc, and 3 gold. (This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized.
The historic drilling, geologic information, check assay verification provide support for IMC to form the opinion that the resources at Turner Gold are as stated on Table 1-1.

There is high potential to add a few hundred thousand tons to the resource tonnage at the Turner Gold deposit as there are significant areas, particularly in the lower zone (MLZ), where drilling has not found the limits of the mineralization.

Based on the known information provided to date, JBR (Environmental Consultants, See Section 2.0) sees no environmental issues that would prevent the permitting of the proposed operations. Although JBR currently does not see any permitting issues that would prevent the operation of the proposed Turner Gold Mine, JBR cannot predict all the concerns or issues the permitting agencies may have with the proposed project during the permitting process, nor can JBR control how long the agencies will take to issue the necessary permits. At this time, quantification of all the environmental impacts of the proposed facilities and operations is not possible. A better understanding of these will be developed during the permitting process.

There is potential to increase metal recoveries, particularly for precious metals, with newer technologies introduced to processing in recent years. Gravity concentration methods and non-cyanide leaching of gold and silver from copper sulfides, pyrite and arsenopyrite concentrates are a few processes of merit to investigate. Production of a separate cobalt/pyrite concentrate may also be practical given the advances in fine grinding methods in recent years.

This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized.

2.0 INTRODUCTION

This Mineral Resource and Preliminary Economic Assessment for the Turner Gold Project was prepared for Green Park Capital Corp. for the purpose of a qualifying transaction with the TSXV. The following team of consulting firms prepared this report:

Company and Person	Summarized Responsibility
Michael D. Strickler, LithoLogic Resources LLC	Geology and History
James J. Moore	Process and Infrastructure
Srikant Annavarapu, Master Geotech Services, LLC	Mine Planning
Brian Buck, Catherine Clark, Jonathan Williams of JBR Environmental Consultants, Inc.	Environmental, Permitting
John Marek, Independent Mining Consultants, Inc.	Resources and Report Assembly

The above group worked together as a team and each provided a qualified person for this Technical Report under the definitions of NI43-101. John Marek acted as the primary author of the Technical Report.

Green Park Capital Corp. entered into a letter of intent on March 26, 2010 to acquire all of the shares of Josephine Mining Corp. (See section 4.1).

The Turner Gold project is a massive sulfide deposit that is potentially amenable to small tonnage underground mining. The project is located in southwestern Oregon. The deposit is contained on 3 patented claims of approximately 60 acres. An additional 264.55 acres of contiguous private land is controlled by Green Park.

This work was started in July of 2009 and this final Technical Report completed in early November 2009.

Historic drill data was obtained from paper drill logs that were on file under the control of JMC. Independent Mining Consultants, Inc. (IMC) personnel keypunched the drill hole information into computer files for use in the generation of the computer based block model and mineral resource estimate.

The Turner Gold Project has also been referred to historically as the Turner-Albright project. Most of the drilling for the project was completed in the 1980's by several different companies including: AmSelco, Baretta, Noranda, Rayrock and AUR/Lupine. A number of historic reports have been prepared that were of value as background in the development of this report. Those reports are listed in the reference section of this Technical Report.

John Marek, Mike Strickler, and Catherine Clark visited the property on September 2, 2009 in the company of JMC management. Srikant Annavarapu visited the property during September 3 - 4, 2009. Jim Moore visited the property on October 6, 2009 and

Brian Buck visited the property on June 3, 2009. With the exception of Catherine Clark, all parties visited the core shed to review the condition of the drill core. All qualified persons reviewed the core, toured the property, visited potential infrastructure sites.

This report is in English units. Tons are short tons of 2000 lbs. Ktons means 1000 short tons. Precious metal grades for gold and silver are presented in troy ounces per short ton. Base metal grades for copper, zinc, and cobalt are in percent by weight.

3.0 RELIANCE ON OTHER EXPERTS

This Technical Report was assembled by the team of consultants as outlined in Section 2.0. Each was responsible for specific chapters in this report. Final assembly of the report was accomplished by John Marek of Independent Mining Consultants, Inc. who also acted as the primary author of the Technical Report.

The chapter responsibilities are summarized below:

Qualified Person	Section Responsibilities
Michael Strickler, LithoLogic Resources LLC	Sections 5 through 13
James J. Moore, P.E.	Section 16, 18.2, 18.4, 18.5
Srikant Annavarapu, Master Geotech Services, LLC	Sections 18.1 and 18.4.1
Brian Buck, of JBR Environmental Consultants, Inc.	Section 18.3
John Marek, Independent Mining Consultants, Inc.	Sections 1, 2, 3, 4, 14, 15, 17, 19, 20, 21

Independent Mining Consultants, Inc, and the consultants listed above have not verified or audited the property ownership as outlined in Section 4.0. The authors have relied on the opinion of Legal Council to JMC as evidenced in the letter provided by Duane Wm. Schultz, P. C. attorney and counselor at law (his office is in Grants Pass, Oregon) regarding the land status in a letter to JMC dated October 21, 2009. James J. Moore, (Qualified Person) has reviewed the letter and concurs with its overall assessment that JMC has the right to continue exploration and development of the property following the laws of Oregon and the United States.

Where possible, the authors have confirmed information provided by JMC or previous authors by comparison against other data sources or by field observation.

IMC has not reviewed the environmental situation at the property. IMC has assumed that any operating permit and reclamation requirements are properly accounted for in the information provided by JBR and JMC and that any potential future operations will not be prejudiced by environmental, permitting, or related constraints.

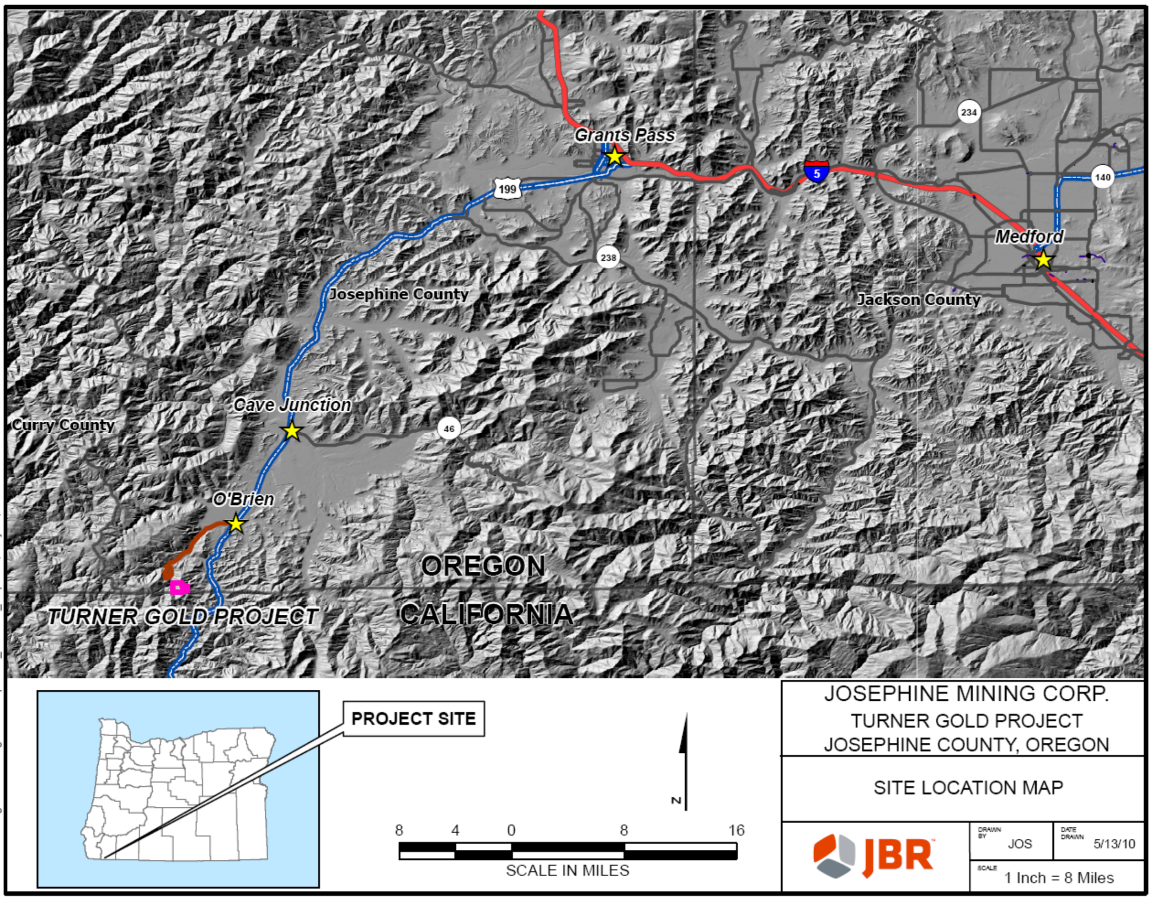
IMC has not audited the process plant or tailing design within this document and has relied on the opinions and expertise of fellow author James J. Moore.

JBR Environmental Consultants, Inc. (JBR) currently does not see any permitting issues that would prevent the operation of the proposed Turner Gold Mine, but JBR cannot predict all the concerns or issues the permitting agencies may have with the proposed project during the permitting process, nor can JBR control how long the agencies will actually take to eventually issue the necessary permits. At this time, quantification of all the environmental impacts of the proposed facilities and operations is not possible. A better understanding of these will be developed during the permitting process.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Turner Gold deposit is situated in southern Josephine County, Oregon, immediately north of the California border and approximately 2 miles west of Highway 199 (see Fig. 4-1). It is located approximately forty miles southwest of Grants Pass, Oregon; the county seat, located on Interstate-5.

Turner Gold: Figure 4-1
Location Map: Regional



The property consists of three patented mining claims (approximately sixty acres), which contain the deposit as currently defined. An additional 264.55 acres of contiguous private land is controlled under option to purchase by Josephine Mining Corp. (JMC) and adjoins the patented claims to the west (see Fig. 4-2). Under option, JMC also controls title to 1.0 acres in O'Brien. Turner is the only asset held by JMC at the time of this writing. (Shown below in Figure 4-2 is the claim map for the project.)

Three (3) unpatented mining claims cover locatable Federal lands adjacent to the JMC holdings to the north and east. (See Table 4-1 for a summary of all land holdings.)

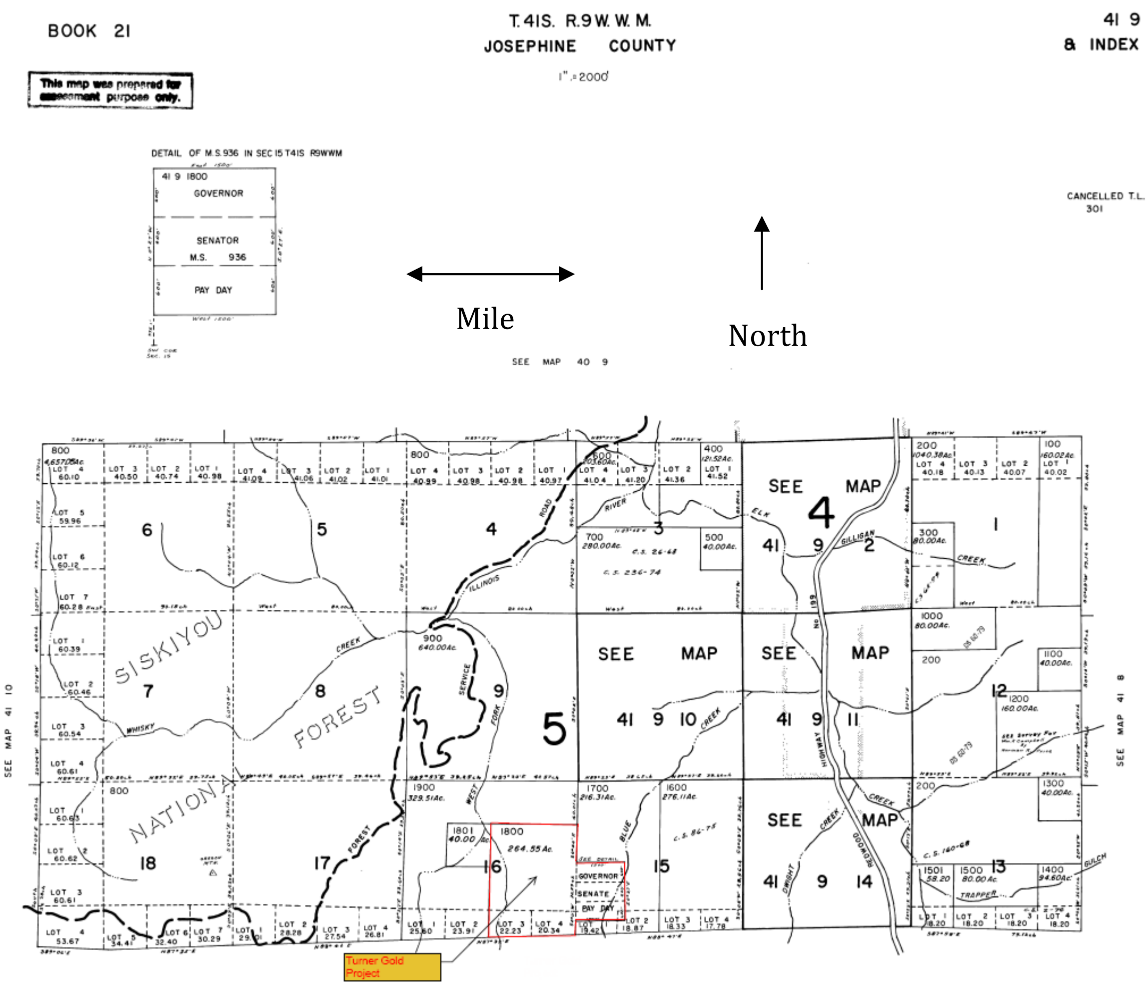


Figure 4-2
Claim Map with Sections

Table 4-1
Legal Description of Land Position
As provided by Duane WM. Schultz, P.C. (21 October 2009)

Government Lots 3 and 4; the South Half of the Northeast Quarter; and North Half of the Southeast Quarter, all in Section 16, Township 41 South, Range 9 West of the Willamette Meridian, Josephine County, Oregon.

Also, U.S. Mineral Survey No. 936 being those certain patented mining claims formerly known as the Governor, Senator and Pay Day Lode Mining Claims, Patent No. 1194083, dated April 2, 1959 as the same appears of record of Josephine County Deed Records in Volume 200, Pages 154 and 173.

Including that certain easement created by Warranty Deed recorded January 25, 1974, in Volume 297, Page 267, Josephine County Deed Records, for road right of way, 60 feet in width, in the Southeast Quarter of the Northwest Quarter of Section 16, Township 41 South, Range 9 West, Willamette Meridian, Josephine County, Oregon, the centerline of which is described as follows:

> Beginning at a point on the North line of the Southeast Quarter of the Northwest Quarter of said Section 16 which bears South 18°53’40” West 1395.13 feet from the Northeast corner of the Northwest Quarter of said Section 16; thence South 26°33’ East 26.42 feet; thence South 40°51’ East 249.05 feet; thence South 1°36’ East 305.99 feet; thence South 39°40’30” East 269.68 feet; thence South 71°02’feet East 101.32 feet to the East line of the Northwest Quarter of said Section 16.

In addition to the project location, there is an additional piece of property located in O’Brien, Oregon and is described as follows:

> Beginning at a point where the East line of the Redwood Highway (40 feet from centerline) intersects the North line of the Northeast Quarter of Section 25, Township 40 South, Range 9 West of the Willamette Meridian, Josephine County, Oregon, said point of beginning bears South 89°02’46” West, a distance of 852.22 feet from the Northwest corner of said Section 25; thence North 89°02’46” East, along the North line of said Section 25, a distance of 183.91 feet to the Northeast corner of the Northwest Quarter of the Northeast Quarter of the Northeast Quarter of said Section 25; thence South 00°36’29” East, along the East line of said Northwest Quarter of the Northeast Quarter of the Northeast Quarter of said Section 25 a distance of 483.50 feet; thence North 75°00’37” West a distance of 414.68 feet to the East line of the Redwood Highway (40 feet from centerline); thence North 29°30’46” East, along the East line of said Redwood Highway, a distance of 430.00 feet to the point of beginning.

Table 4-1, Continued
Legal Description of Land Position
As provided by Duane WM. Schultz, P.C. (21 October 2009)

EXCEPTING THEREFROM all of PARCEL 1 of PARTITION PLAN NO. 1994-20, located in the Northeast Quarter of the Northeast Quarter of Section 25, Township 40 South, Range 9 West, Willamette Meridian, Josephine County, Oregon.

Adjacent to the project location are three unpatented lode claims as described below.

The unpatented lode Mining Claims are all situated in Waldo Mining district, Section 15, Township 415, Range 9W, W.M., Josephine County, Oregon, and more particularly described as follows:

Tab 99-2 Fraction ORMC 154245
Tab 99-3 Fraction ORMC 154246
Tab 99-4 Fraction ORMC 154247

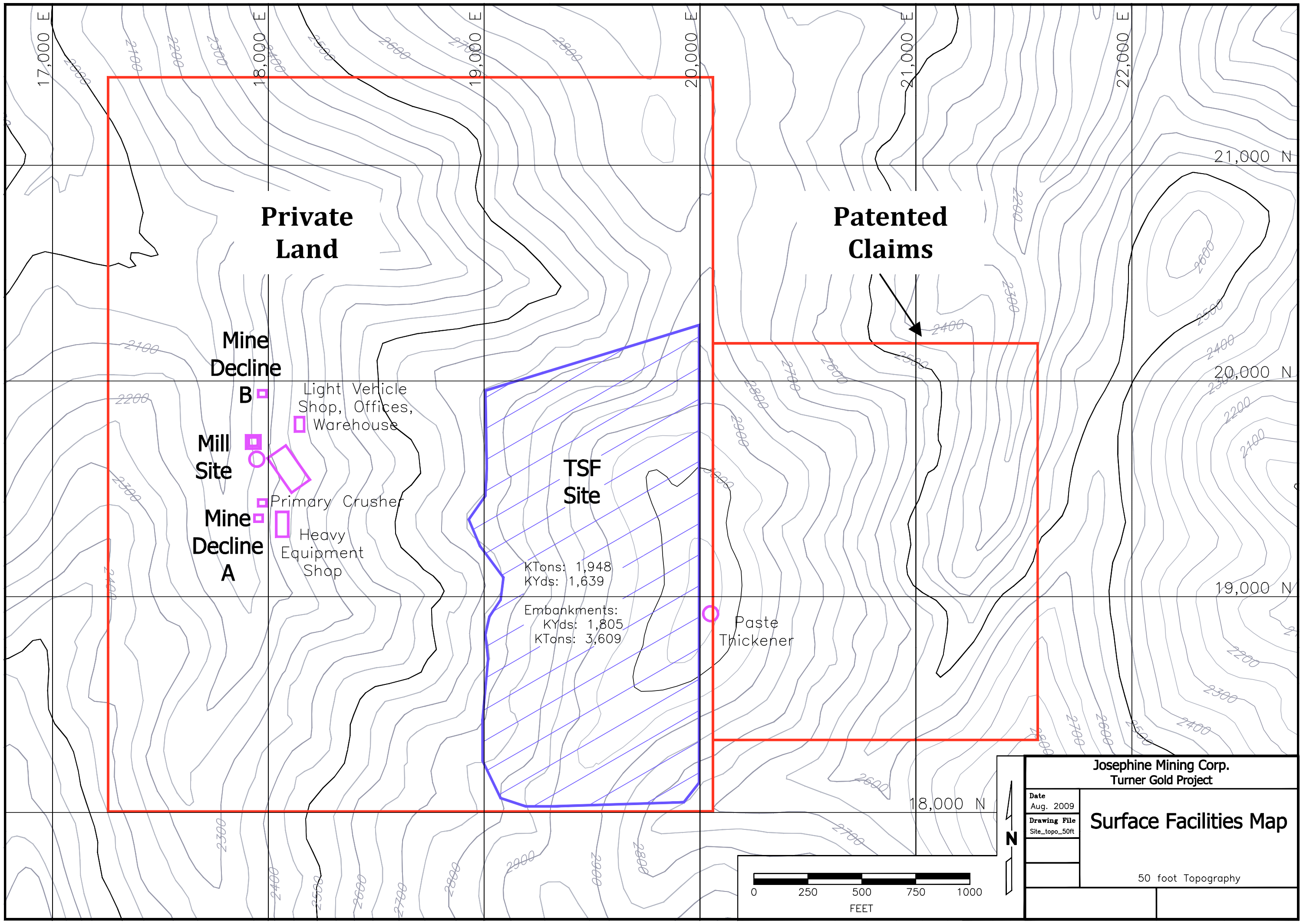


Figure 4-3
Approximate Property Boundaries

4.1 Option to Purchase Agreement with General Moly, Inc.

Green Park Capital Corp. entered into a letter of intent on March 26th, 2010 to acquire all of the shares of Josephine Mining Corp. under a Capital Pool Company (CPC) qualifying transaction. The obligations that Green Park Capital will have upon acquiring the property will be as per those obligations of Josephine Mining Corp under the “Option to Purchase” agreement.

The terms of the proposed agreement (the "Qualifying Transaction" or "QT") whereby Green Park Capital Corp. (the "CPC") will acquire, all of the issued and outstanding shares of Josephine Mining Corp are as follows: The CPC will accordingly acquire beneficial ownership of a certain “Option to Purchase” agreement as described below and subsequently the mineral claims (the "Claims") and fee land to be acquired under such agreement by Josephine Mining Corp. and to be registered with the applicable mining recording offices and Josephine County, Oregon in the name of Josephine Mining Corp. or a wholly owned subsidiary of CPC upon completion of all payments and obligations per the “Option to Purchase.” It is anticipated that the QT will constitute a reverse takeover of the CPC by Josephine Mining Corp..

Green Park Capital agrees to be responsible for maintaining the property and any water rights and claims, paying all taxes, and progressing the development program defined in this technical document. Green Park Capital, or its subsidiary under the reverse takeover contemplated as described above, will make all scheduled payments to General Moly as described below in the “Option to Purchase” agreement as long as the agreement is in effect.

On June 26th, 2009, an “Option to Purchase” agreement was executed by General Moly, Inc. (property owners) and Josephine Mining Corp. Josephine Mining Corp. has an exclusive right to purchase the property. The terms of this agreement are as follows:

Josephine Mining Corp. was to pay, in consideration of the agreement, $100,000.00 upon execution of the agreement. This was completed on 6/25/2009. This payment gives Josephine Mining Corp. the right to enter and occupy the property for a period of eighteen months from the execution date of the agreement. There is an option to extend the agreement for an additional twelve months at the eighteen month point with an additional payment of $300,000.00. This would extend the option to a total of thirty months.

The outright purchase price for the property is $2,000,000.00. The option payments are applied against the total purchase price. The balance remaining after the option payments are made is $1,600,000.00 and is due at the earliest of either, December 26, 2011 (30 months after agreement execution) or receipt by Josephine Mining Corp. of all permits and/or approvals necessary to commence mining operations plus three months from the date of permit/approval. Josephine Mining Corp. has the right to execute the option to purchase any time on or before 5:00PM Pacific Time 912 days after the execution of the agreement. If Josephine Mining Corp. does not exercise the option, General Moly, Inc. retains all previous payments received.

Sixty day notice of the intention to exercise the option is to be made in writing. Closing will occur sixty days after receipt of said notice or at a mutually agreed upon time.

Per the terms of the agreement, General Moly, Inc. will provide all data and information in its possession, control and/or ownership with respect to the property and the mineral potential of the property for Josephine Mining Corp. to copy at its own expense. Josephine Mining Corp. is responsible for maintaining the property and any water rights and claims and paying all taxes.

In addition to the purchase price, Josephine Mining Corp agrees to pay General Moly, Inc. a production royalty or net smelter return. Josephine Mining Corp. has agreed to pay 1.5% net smelter return on mineral products mined and produced from the property and sold by Josephine Mining Corp.

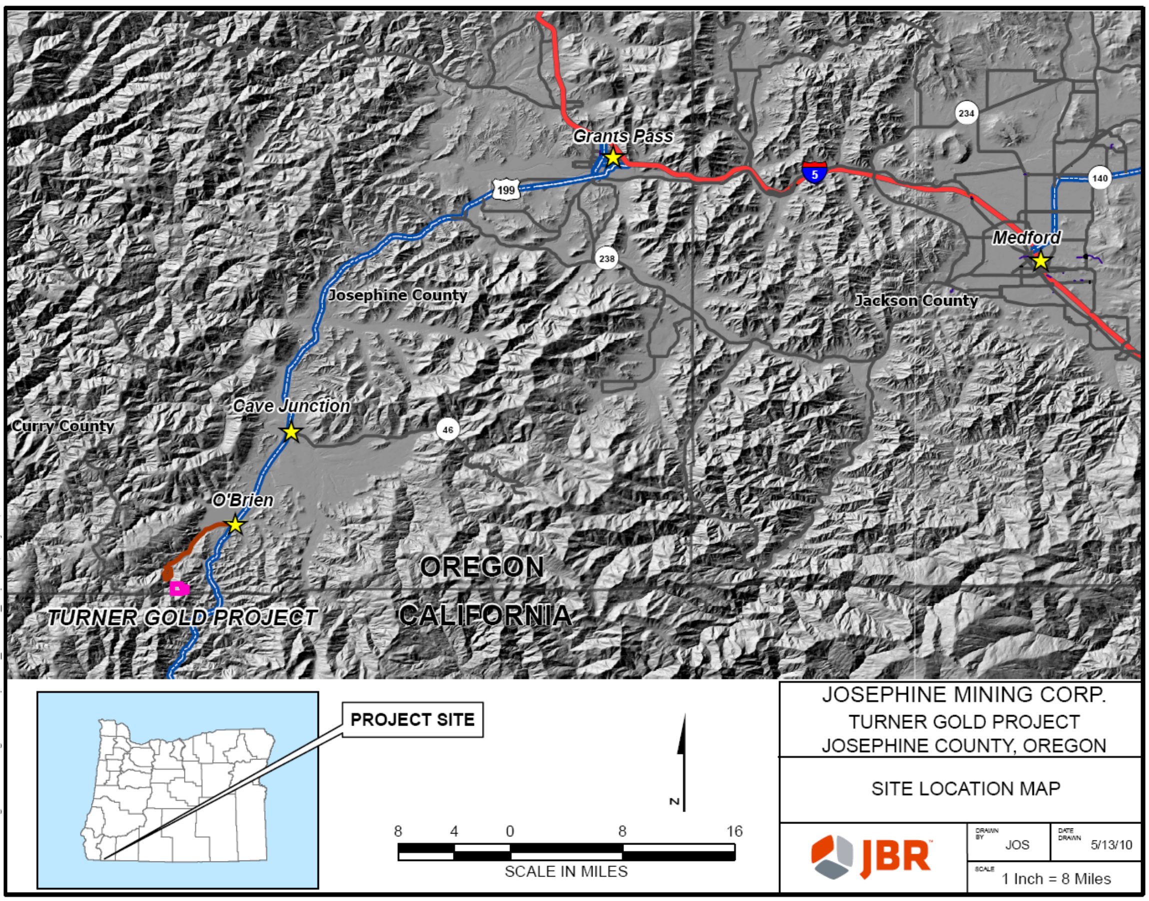
Josephine Mining Corp. has the option to terminate the agreement by letting the option expire or with thirty day written notice.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

Access to the Turner Gold deposit is via Lone Mountain Road, which joins with U.S. Highway 199 in O'Brien, Oregon, approximately forty miles southwest of Grants Pass (see Figure 5-1). From O'Brien, the Lone Mountain Road parallels the West Fork of the Illinois River to the turnoff to the property, a distance of approximately six miles. From there, an extensive system of access and drill roads provides year-round entry to most portions of the deposit by two-wheel drive and/or four-wheel drive vehicles.

Turner Gold: Figure 5-1
Location Map: Local



5.2 Climate

Regional rainfall during the wet season (generally November through May) can be quite heavy, with seasonal totals in excess of 100" possible in the project area. Snowfall is common above 3000', and can last from December through April. Storms come in groups, with weeks of clear weather common between systems. Summers are hot and dry. Temperatures above 100°F are possible from July through mid-September.

There are no climatic conditions that should cause the project great operational difficulty. The greatest climatic issue will be managing storm waters that will result from excessive rainfall at intermittent times during the life of the deposit; however, this is a common area of concern at many mine sites and should be manageable with proper controls.

5.3 Local Resources

The local resources would seem to be primed for a project of this magnitude. Populations in both the Rogue and Illinois Valleys are expanding; however, the demise of the timber industry, coupled with challenging economic times, has resulted in a region, historically based upon primary industry, which is eager for some form of economic stimulus.

According to the Oregon State University Population Center, the 2008 populations of communities near the Turner Project were as follows:

	Population
Cave Junction	1,730
Grants Pass	32,260
Medford	76,850

O'Brien is quite small and almost the entire population would be un-incorporated county residents. The total population of Josephine County is approximately 85,000, including Cave Junction and Grants Pass.

5.4 Infrastructure

Much of the basic infrastructure is largely in-place for exploration and development. A paved highway runs six miles northeast of the project, and good access exists throughout the Turner Project itself. Interstate-5, forty miles northeast in Grants Pass, is the major north-south highway linking the metropolitan centers of the western United States, from Seattle near the Canadian border to the Los Angeles - San Diego megalopolis in the south. Grants Pass also marks the location of the closest railhead. Coos Bay, Oregon, approximately 140 miles north of the California border, is the nearest deep-water port (see Fig. 4-1).

Water for exploration has been taken from flooded historic mine workings and/or Blue Creek, a small surface drainage that runs through the eastern end of the project area. Water for mining and processing could be obtained from wells, planned to be situated at Turner, and located at the western edge of the private lands adjoining the deposit.

Power is available from the main transmission line that connects Southwest Oregon to the coast, and parallels Lone Mountain Road from Highway 199. At closest approach, the transmission lines are approximately one mile west of the proposed surface facilities.

There is a small, older core facility in O'Brien (approximately 2,500 square feet), also currently under option by JMC, which has been used during previous exploration programs for the logging of drill core, sample preparation, and office space. This facility is currently filled with all core salvaged from prior drilling on the property. Any future activities will require the development of additional facilities.

5.5 Physiography

Relief at the Turner Gold deposit is moderate to locally steep, with elevations ranging from 1900' to 3100' above sea level. The private lands have been heavily logged on several occasions, and thick stands of brush and second-growth timber now cover those portions of the property that are underlain by volcanic or sedimentary members of the local ophiolite stratigraphy. Areas underlain by peridotite and/or serpentinite, generally to the west and north of the deposit, are commonly sparse of vegetation, with little or no significant timber resources.

6.0 HISTORY

Tonnage and grade estimates within this section are indicative of historic work. They do not conform to the definitions within NI43-101 and are presented as part of the historical prospective of the deposit.

Mineralization associated with the Turner Gold deposit (historically known as the Mammoth Mine, and later as the Turner-Albright) was originally located in the late 1800s. Early efforts concentrated on developing the gold potential of several discontinuous gossan outcrops located on the ridge with sporadic exploration and limited development continuing through the 1930s. Several short crosscuts driven at the base of the oxide horizon encountered mineralization that was of sufficient grade to allow three claims to be patented in 1959 (Senator, Governor, and Payday; see Fig. 4-1).

Exploration targeting the primary sulfides began in the 1950s with a one-year program by Granby International. Local geologist Lloyd Frizzell (Associated Geologists, Grants Pass, Oregon) continued intermittent exploration throughout the 1960s and early 70s with several programs consisting of churn and shallow diamond core drilling, and an initial Induced Polarization geophysical survey.

A two-year drilling program (2947.4 feet) by American Selco (1974/75) explored the potential of the 'South Zone' gossans, and resulted in an estimated drill-indicated resource of 150,000 tons of sulfide ore averaging 1.70% copper and 0.03 oz/ton gold across an eight-foot wide zone of highly siliceous basaltic breccias*. Evidence of a larger mineralized body north of the 'South Zone' was indicated by an Induced Polarization geophysical survey and several short diamond drill holes.

Savanna Resources/Baretta Mines Ltd. of Calgary, Alberta, Canada, obtained an option upon the termination of the American Selco program. Through 1981, Baretta conducted the first coordinated exploration of the Turner deposit itself, as well as the identification and initial exploration of favorable units to the south and southwest. A total of thirty diamond core holes, with an aggregate length of 35,498.1 feet, were completed on the patented ground, and resulted in the initial definition of the Main Upper Zone (MUZ), Main Lower Zone (MLZ), as well as indications of the Upper High-grade Zone (UHZ). At the close of the Baretta program, Turner was estimated to contain drill-indicated in-place mineralization of 1.7 million tons averaging 0.113 oz/ton gold, with additional values in copper, zinc, silver, and cobalt*.

Subsequent programs by Noranda Exploration, Inc. (1982) and Rayrock Resources Limited (1983/84) continued to refine both the geologic and structural characteristics of the deposit, utilizing a variety of methods including: diamond core drilling and sampling, surface mapping, geochemistry, and surface and down-hole geophysics. Initial attempts

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*A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; the issuer is not treating the historical estimate as current mineral resources or mineral reserves as defined in sections 1.2 and 1.3 of this Instrument; and this historical estimate should not be relied upon.

to define the metallurgical characteristics of the deposit were also begun during these programs. A one-season program by Aur Resources (1989) represents the last round of active exploration and drilling on the deposit.

Idaho General Mines, Inc. gained an interest in the property in 2004 through a stock arrangement with Savanna Resources, Ltd. No exploration activities, other than claim consolidation and maintenance, have occurred since acquisition.

General Moly, Inc. obtained the property during the transfer of assets from Idaho General Mines, Inc. to General Moly, Inc. The JMC option from General Moly was outlined in Section 4.0.

Estimates of mineralization for the Turner Gold deposit have been calculated many times by many companies, utilizing a variety of methods. At least two companies, Noranda and Rayrock, as well as several independent studies, performed preliminary metallurgical testing.

In addition to direct exploration targeted on defining the Turner's resource potential, a number of independent data reviews have been completed by various interested parties, and for varied purpose. These include studies by Marubeni (1988), R.L. Russell (1988), and Cominco (1990). A number of ongoing, but generally disconnected studies were also undertaken by various members of the intellectual community and branches of the U.S. government. These efforts included a team of geologists, marine geologists, and geochemists from the U.S. Geological Survey who studied the deposit during the mid-1980s to determine its similarities to active seafloor hydrothermal systems, and the U.S. Bureau of Mines (also in the 1980s), who initiated a limited mineralogical study of the cobalt-bearing sulfide body.

Please refer to Table 11-1 for a summary of exploration drilling efforts at Turner Gold.

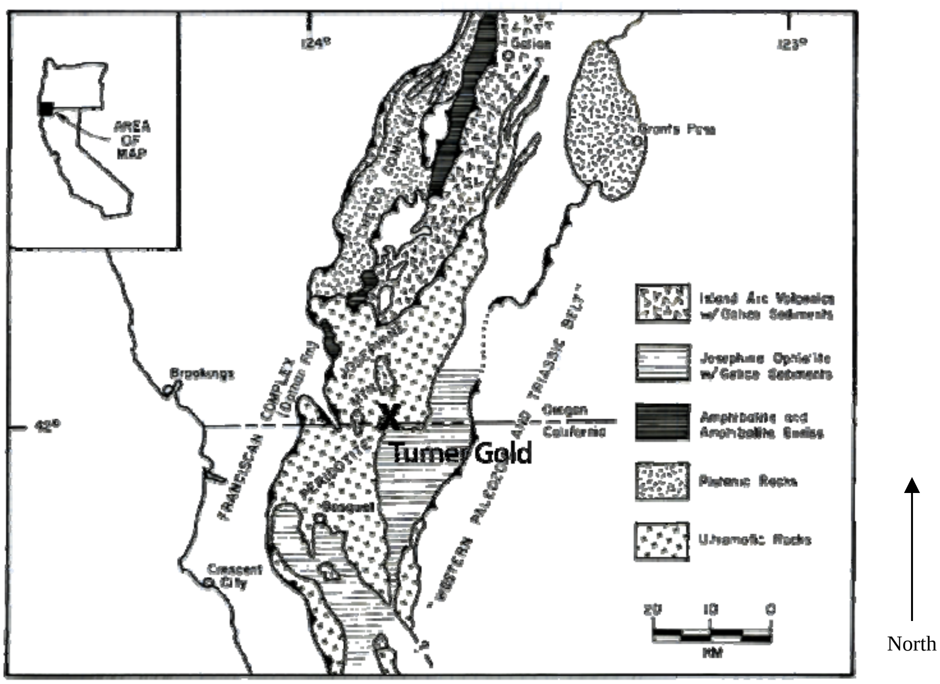
7.0 GEOLOGIC SETTING

An understanding of the geologic setting of the Turner Gold property has evolved with the years, thanks to contributions from the many geologists who have been involved with the deposit.

7.1 Regional Geologic Setting

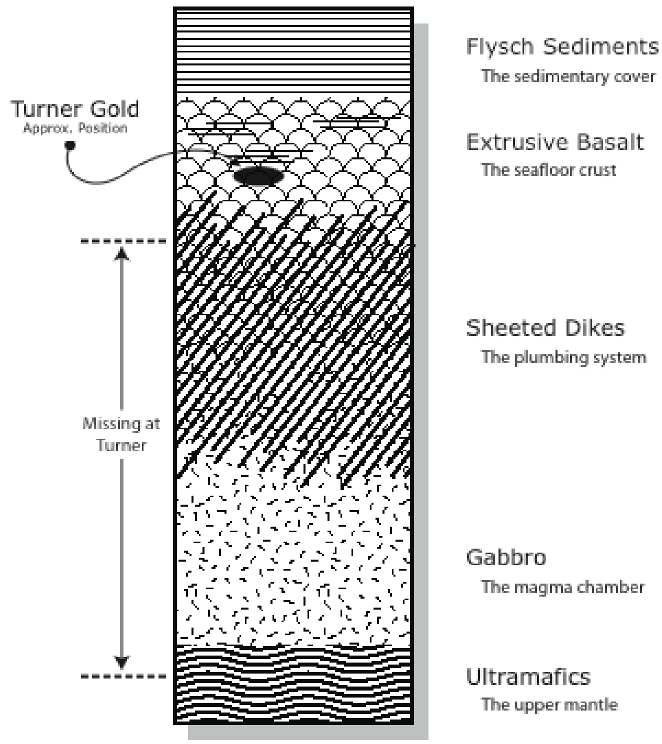
The Turner Gold deposit occurs in the Western Jurassic Belt (WJB) of the Klamath Mountains geomorphic province (see Fig. 7-1). The lithologies and age relationships within the Klamaths indicate repeated accretion, beginning in the early to middle Paleozoic and continuing through the Mesozoic, of ophiolitic and island arc terranes, with their associated sedimentary units, to the leading western edge of the North American plate. The WJB is in thrust contact with a similar suite of late Paleozoic and Triassic ophiolitic/arc units to the east, and is under-thrust from the west by the late Jurassic to Cretaceous Franciscan (Dothan) mélange.

Turner Gold: Figure 7-1
Western Jurassic Belt



A prominent feature of the WJB in southwestern Oregon is the Josephine Ophiolite; a preserved section of seafloor crust dated at 157 million years (mybp). Regionally, the Josephine Ophiolite trends NNE with a steep SE dip, and is essentially complete, with the discontinuous preservation of all major lithologies associated with classic ophiolite stratigraphy (see Figure 7-2).

Turner Gold: Fig.7-2
Idealized Ophiolite Stratigraphy



Precious and base metal mineralization is widespread in the region and consists of several varied genetic types. In addition to Turner, a number of other massive to semi-massive sulfide deposits have been identified. It is probable that several of these may be volcanogenic, and associated with ophiolitic rocks (Monumental, Fall Creek, Iron Hat, Babcock, Queen of Bronze/Cowboy Group), while others appear to be related to more felsic terranes (Almeda, Goff, Silver Peak, Yankee Silver Lode). Numerous high-grade precious and base metal deposits, commonly associated with mafic to felsic intrusive events, occur throughout the Klamath Mountains. Both vein and high-grade gold 'pockets' have eroded to form locally rich placer deposits, many of which have been extensively worked since the 1850s by methods ranging from pick and shovel to large-scale hydraulic mining.

7.2 Local Geologic Setting

The Turner Gold deposit is situated near the base of the extrusive pillow lavas and flows of the Josephine Ophiolite (Fig. 7-2), several hundreds of feet above their gradational lower contact with the sheeted dike sequence. In the immediate vicinity of Turner, the majority of ophiolite-related lithologies that are generally found stratigraphically below the extrusives are missing due to oblique post-ophiolitic low-angle faulting which has juxtaposed the uppermost portion of the extrusive/sheeted dike transition zone against serpentinized mantle peridotite. Compared with the total section as exposed south of Turner, up to five thousand feet or more of the ophiolite stratigraphy may be missing, including the middle and lower sheeted dikes, the entire massive and cumulate gabbro sequence, and an unknown thickness of mantle peridotite (Figure 7-2).

With the exception of scattered mafic dikes that occur within major shears in the ultramafics, all lithologies currently exposed in the vicinity of the deposit are interpreted to be associated with the primary development of the Josephine Ophiolite. See Figure 7-3 for a summary of the surface geology at the Turner Gold deposit. A brief description of the major units identified at Turner (from drill core and/or surface mapping) includes:

7.2.1 Basalt

Extrusive volcanic rocks exposed at Turner generally consist of basaltic flows, pillows, and hyaloclastites, and commonly contain plagioclase, clinopyroxene and/or iron titanium phenocrysts. Feldspar microlites and/or calcite veinlets and amygdules occur locally, and individual units may be locally vesicular. Well-developed pillow structures are evident, both in outcrop and drill core. Minor to locally intense alteration occurs, consisting of prehnite/pumpellyite, chlorite, sphene, and albite (+/- silica, hematite, and epidote), with increased alteration being localized within and adjacent to zones of shearing and faulting.

7.2.1.1 Mafic lava series

Work by Robert Zierenberg of the U. S. Geological Survey has defined a second extrusive member of limited extent that is apparently restricted to the mineralized horizon(s). This unit, which consists of glassy fragments of a relatively primitive mafic magma, has not been identified as flows or pillows (see 7.2.4 Basin Floor Rubble, below). The rock typically exhibits phenocrysts of olivine and/or chromium spinel (with occasional plagioclase) in a groundmass of glass and radiating clusters of quenched pyroxene.

7.2.2 Gabbro

Originally interpreted as an intrusive by American Selco, the term gabbro (as applied at Turner Gold) includes mafic igneous rocks with diabasic to micro-gabbroic (locally gabbroic) textures, and containing plagioclase and/or pyroxene phenocrysts in a generally fine-grained groundmass. There is no compelling evidence to date that supports an intrusive origin for the unit, and the gabbro is interpreted to represent coarse-grained members of the dominant plagioclase-bearing lava series that occur within the cores of thick extrusive basalt flows and/or pillows.

7.2.3 Mudstone

Turner mudstones include very fine-grained chemical and/or clastic sedimentary units, locally cherty, that occur as definable horizons three inches to six feet thick. Turner muds are also found as minor accumulations around pillows, and as infillings between flows. Color varies from red (hematitic) to green, brown, grey, and black (carbonaceous). Green and grey mudstones are often macroscopically indistinguishable from silicified basaltic gouge in drill core. Measurements of bedding from outcrop, as well as sub-surface structural calculations from 3-points, indicate a regular NNE strike to the units (sub-parallel to the regional trend of the ophiolite); however, dips vary from 30° SE to nearly vertical.

Composition of individual clasts can be difficult to determine; however, local variations in the silica content of the sediments support an exhalative or biogenic source for at least a portion of the material. Radiolarian tests, observed in a siliceous mudstone at the southern edge of the deposit, supported the regional dating of the ophiolite.

Relatively thin mudstone beds commonly cap the exhalative horizons, and appear to be laterally more extensive than the sulfide bodies themselves. At least two, and possibly three additional mudstone horizons have been identified that are not known to be associated with sulfide mineralization.

7.2.4 Basin Floor Rubble (BFR)

From an examination of textures associated with the sulfide bodies, it is apparent that a large portion of the deposit occurs as a replacement of brecciated fragments of basalt, with variable quantities of chert. The Basin Floor Rubble (BFR) represents a varying thickness, locally approaching several hundred feet, of brecciated basalt that covered the original depositional basin prior to the onset of hydrothermal activity and the venting of the sulfide horizons. The majority of the semi-massive sulfides, as well as a large portion of the massive sulfide horizon, may occur within highly altered portions of this unit. Intense alteration within this section of the Turner stratigraphy obscures the composition of many of the fragments; however, it is apparent, from petrologic studies by the United States Geological Survey, that clasts of the mafic lava series form a large portion of the unit, with clasts of the regionally dominant plagioclase-bearing lava being generally restricted to the base of the rubble pile.

7.2.5 Talus Deposits

High angle faulting associated with the formation of Turner resulted in several moderate to high relief pre- and post-mineral fault scarps in the original depositional basin. Brecciation and erosion led to the accumulation of talus deposits at the base of these structures. Individual talus piles can include fragments of basalt, mudstone, chert, and sulfides, with minor amounts of gabbro.

7.2.6 Sheeted Dikes

Ophiolitic sheeted dikes are characterized by sub-parallel diabasic dikes, and are interpreted to represent the conduits for the magma which supplied the overlying extrusive flows and pillows. The upper and lower contacts of the unit as a whole are commonly gradational. The upper transition zone with the extrusive lavas is composed of diabasic dikes with a downward decreasing proportion of basaltic 'screens,' while the lower contact zone with the intrusive gabbro is characterized by extremely erratic and confusing diabasic/gabbroic textural variations.

Due to faulting which has removed much of the base of the ophiolite, only the uppermost portion of the extrusive/dike transition zone remains at Turner. This section of the stratigraphy is poorly exposed, and has only been identified in several drill holes in the northwestern portion of the deposit, and in extensively weathered outcrops in fault contact with serpentinite. Individual dike margins are marked by chill zones up to 1cm across, and are often brecciated. Moderate to locally intense epidote alteration is common. Textures within the cores of individual dikes and the enclosing basaltic screens are often indistinguishable, which makes identification of this transition zone extremely difficult in outcrop, where the chill and/or breccia margins are generally obscured by surface weathering.

7.2.7 Ultramafics

Partially to completely serpentinized mantle peridotite outcrops immediately west of Turner, and presumably exists at depth within the footwall of the deposit. Where observed (surface exposure, and in a few drilled intercepts at the northern end of the deposit), all contacts are structural, and represent major zones of crustal shearing.

Lithologic variation within the ultramafics is the rule, and the unit as a whole has been subjected to intense but varying levels of internal alteration, shearing, and faulting. The ultramafics are highly magnetic relative to other ophiolitic members in the vicinity, and can be readily located by their magnetic signature and distinctive vegetative pattern.

The proposed mine plan places the surface facilities and adit portals within this unit. In addition, the proposed decline(s) to access the deposit will penetrate the ultramafics for 2-3 thousand feet prior to faulting into the extrusive mafic rocks that host Turner.

7.3 Structure

The majority of the known sulfides at Turner Gold occur within three vertically stacked horizons, representing two, and possibly three, separate time-stratigraphic horizons. They have been designated the Upper High-grade Zone (UHZ), the Main Upper Zone (MUZ) and the Main Lower Zone (MLZ) (see Figures 9-1 through 9-4). Three generations of faulting were also recognized during the Baretta program (pre-mineral, post-mineral, and emplacement), and have remained relatively unchanged by subsequent workers.

A series of pre- and post-mineral high-angle northwest-trending normal faults has been partially defined (termed the F-series faults). At least five separate structures (F-1 thru F-5) have been identified, and there is evidence for additional sub-parallel faulting south of the deposit. Measurements in outcrop and correlations between drilled intercepts indicate that the F-series faults strike roughly N60°W, with a dip of 65° to 85° to the northeast.

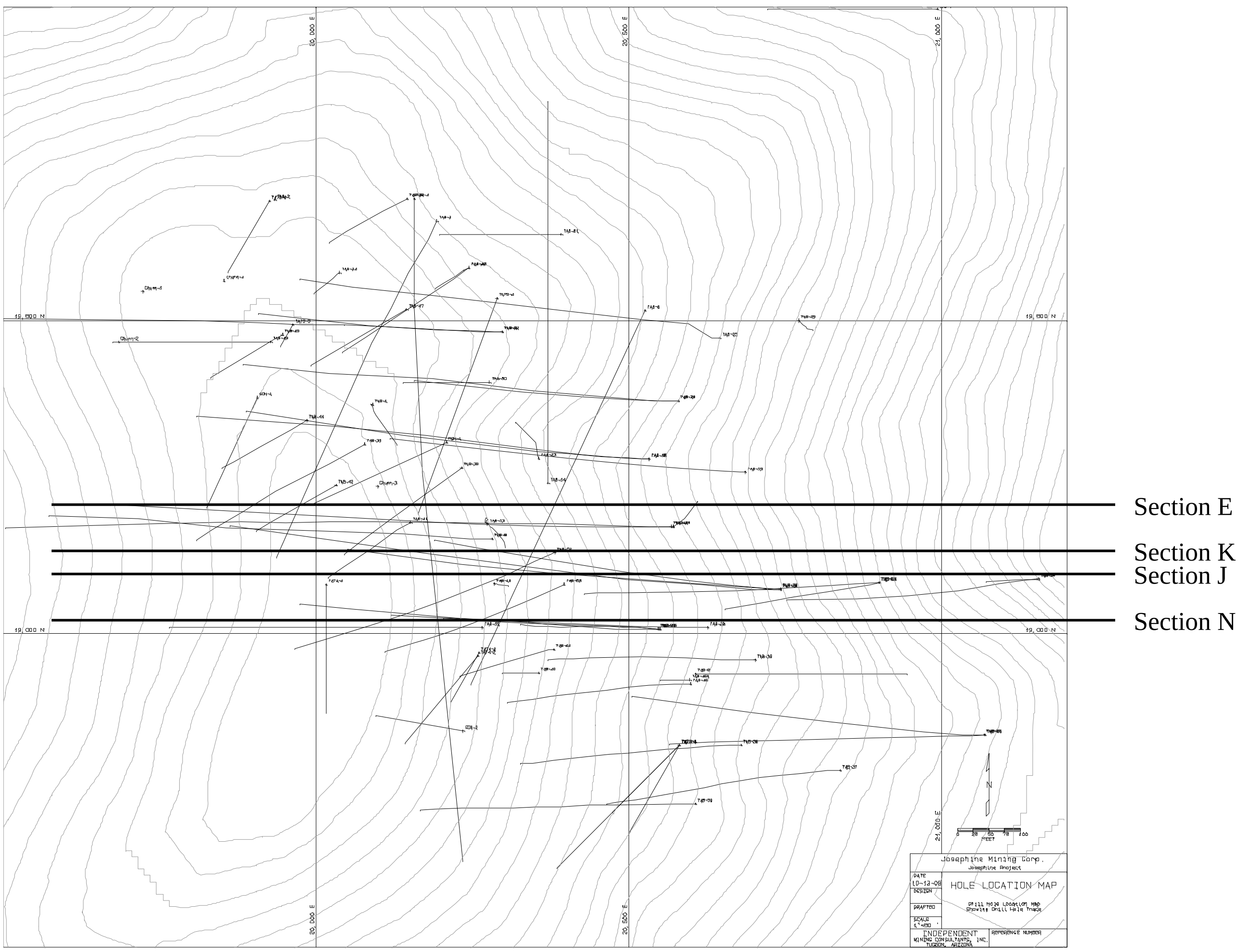
The southernmost mapped structure, F-1, is interpreted to have controlled the movements of the primary mineralizing fluids, and was the focus of the initial work by American Selco. While there is no persuasive evidence to indicate that other F-series faults pre-date the mineralization, the possibility of hydrothermal penetration and/or pre-mineral movement along some or all of the remaining F-series structures cannot be ruled out.

Post-mineral movement along the F-series faults disrupted the stratigraphy following formation of the sulfide horizon(s). This appears to have resulted in the down-dropping of the deposit to the northeast, and the dislocation of the MUZ and MLZ into somewhat discrete fault-bounded blocks; however, in many cases the original thickness of the disrupted sulfide horizon was greater than the displacement along the fault, so that when observed in drill core, a readily discernible lithology change may not be apparent across the structure.

A later series of low-angle east-west trending post-mineral reverse faults is indicated. Timing of the R-series faulting is unknown, but it is possible that these structures were associated with the emplacement of the Josephine Ophiolite along the continental margin, as well as with the faulting and removal of the lower portions of the ophiolite in the vicinity of Turner.

Three R-series faults have been tentatively identified to date (R-1, R-2 and R-3). Three-point structural calculations indicate that these structures strike generally east-west and have a very shallow northern dip (+/- 20°). The major impact appears to have been along R-1, where an apparent 300 to 600 feet of displacement may have resulted in the dislocation of a single sulfide horizon into the MUZ and MLZ. It is important to note that, as currently defined, the R-series faults cut and displace the F-series faults, complicating any attempt to reconstruct the configuration of the original depositional basin, as well as the current geometry of the deposit.

Turner Gold: Figure 7-3
Surface Geology



500 Ft Grid Shown

8.0 DEPOSIT TYPES

In general terms, the Turner Gold deposit is consistent in type and form with volcanogenic massive sulfide deposits of the "Cyprus" model. The following discussion summarizes gross features and similarities, as well as a few notable variations from strict adherence to a classic Cyprus-type deposit:

1) The Turner Gold deposit is ophiolite hosted, and occurs intimately associated with seafloor volcanism and extensional tectonics. Mineralization is structurally controlled, and is restricted to the lower portions of the extrusive lava series immediately above the extrusive/sheeted dike transition zone.

2) Several features common to Cyprus-type deposits, including umbers and ochres, have not been identified in detail at Turner. Iron-poor locally siliceous mudstones occur at the Turner deposit in the same relative stratigraphic position as the Cyprus ochres.

3) The largest portion of the known sulfide mineralization at Turner is interpreted to be the result of large-scale replacement of basaltic breccia. The original depositional basin contained several hundred feet of basaltic rubble that is compositionally different from the regionally dominant plagioclase-bearing lava series that forms both the footwall and hangingwall of the deposit. The highly permeable nature of the breccias had the effect of dissipating the mineralizing fluids into this clastic horizon prior to venting on the seafloor, with only a minimal percentage of the hydrothermal fluids actually reaching the rock-water interface to form exhalative sulfides.

4) A true 'silica stockwork' zone, in which hydrothermally altered flows and pillows stratigraphically below the massive horizon represents the feeder system for the overlying exhalative sulfides, has not been recognized at Turner. This stratigraphic position is represented by hydrothermal penetration and replacement within the BFR.

5) The sulfide bodies at Turner Gold have anomalously high gold values, as compared with typical Cyprus-type deposits.

9.0 MINERALIZATION

The majority of the mineralization at Turner Gold occurs within three vertically stacked horizons, representing two, and possibly three, separate time-stratigraphic horizons. They have been designated the Upper High-grade Zone (UHZ), Main Upper Zone (MUZ) and Main Lower Zone (MLZ) (see Figures 9-1 through 9-4). Identified sulfide minerals include pyrite (+/- marcasite), sphalerite, chalcopyrite, and linnaeite, with trace amounts of tetrahedrite, stannite, galena, and pyrrhotite.

As historically defined, the sulfide bodies at Turner are composed of three interrelated and transitional types of mineralization:

1) Massive sulfide horizons containing >50% total sulfide content
2) Semi-massive sulfide horizons containing 20% to 50% total sulfides, that are generally more distal and represent partial sulfide and silica replacement within the BFR (Basin Floor Rubble)
3) Mineralized basalt, containing decreasing quantities of disseminated and stringer sulfide enrichment and occurring at even greater distances from the main hydrothermal sources

Potentially economic portions of the deposit are generally restricted to the massive and semi-massive horizons, but are not necessarily restricted to those areas containing the greatest percentage of sulfides. Where exposed at the surface, all three units oxidize to form prominent gossans, marking the up-dip western limits of the MUZ and UHZ.

Massive sulfide horizons at Turner appear to have been formed by a combination of seafloor exhalative processes, and/or the extensive alteration and replacement of basaltic breccia within the BFR. Evidence of brecciation within the massive horizons commonly increases down-section, with ghosts of replaced basaltic clasts grading into mineralized rock with definable basalt and chert fragments. From the observed percentage of basaltic ghosts and fragments, it is apparent that large portions of the massive horizons are the result of partial to complete replacement within the BFR; however, the uppermost portions of the massive horizons may exhibit fragmental textures, and it is possible that these may in part represent collapsed chimney structures built by sulfide-rich fluids venting directly onto the seafloor. In addition, several small worm casts were tentatively identified by the USGS, supporting a probable exhalative source for the uppermost portion of the deposit. The origin of any given portion of the massive horizon (i.e. exhalative or partial to complete replacement) may be difficult to determine, and it is often impossible to define the original rock-water interface.

At Turner, semi-massive sulfides, containing 20% to 50% primary sulfides, represent a conformable transition from essentially complete replacement of basaltic breccias to weakly mineralized flows, pillows, and hyaloclastites. The contact between the semi-massive and massive sulfides (as well as with the more distal mineralized basalt) is gradational, and the actual boundary is somewhat irregular and arbitrary. The semi-massive sulfides are almost certainly the result of penetration and replacement within the

BFR, and are characterized by silica flooding of the breccias, with the addition of pyrite (+/- marcasite), chalcopyrite, sphalerite, and accessory sulfide minerals. Hydrothermal penetration of the breccia pile resulted in substantial alteration of the original rock (silica + sulfides + chlorite + albite). From a study of partially altered fragments (R. Zirenberg, USGS), it is apparent that the majority of the clasts are related to the mafic lava series (Section 7.2.1.1). The degree of mineralization and the economic value of the semi-massive sulfides are both somewhat erratic. This may be in part due to the original configuration of the rubble pile; with areas of higher mineralization reflecting increased fluid penetration along avenues of greater permeability.

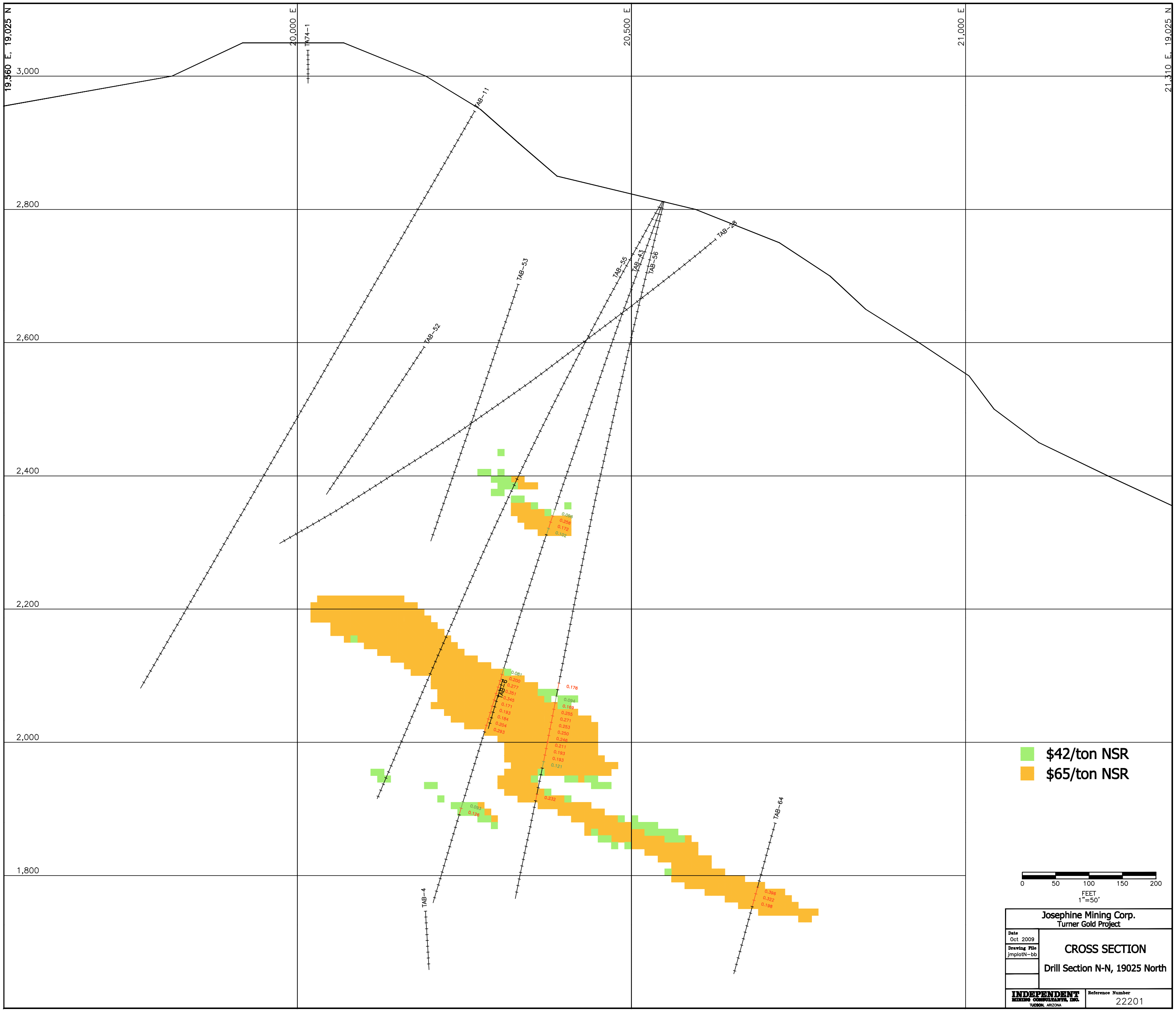
Mineralized basalt includes that portion of the volcanic breccias (and flows) which were subject to alteration by hydrothermal fluids, but which contain a total primary sulfide content of less than 20%. Re-logging of selected drill core by the USGS identified fragments of the regionally dominant plagioclase bearing lavas, as well as clasts of the mafic lava series. It is also evident that mineralization within flow units, as opposed to being restricted to altered breccias, occurs to a limited extent. The mineralized basalts, which are generally of lower economic grade, are interpreted to represent the most distal effects of the mineralizing fluids.

While assumed contributions from multiple vent sources and extensive post-mineral faulting complicate any study of primary zonation, it appears that the original metal distribution resulted in copper/gold rich centers at depth within the BFR and/or proximal to the vents, with zinc/silver, and pyrite with cobalt zones occurring with increasing distance from the sources of the mineralizing fluids.

Limited thin and polished section work by the USGS, the Bureau of Mines, and others, indicates that the metallurgical characteristics of the deposit are complex. Fine-grained chalcopyrite and sphalerite are tightly inter-grown with pyrite and each other. Gold occurs as discrete micron sized blebs within chalcopyrite (and, to a limited extent, sphalerite) and pyrite. This gold/pyrite association results in low to locally moderate gold values (0.02 to 0.07 oz./ton) in the distal pyrite 'halo,' in the absence of significant base metal credits.

Figures 9-1 through 9-4 illustrate east-west cross sections looking north through the Turner Gold deposit. The block model that is described in Section 17 was used as the basis to illustrate the geometry of the mineralization. The yellow and green block outlines reflect Net Smelter Return (NSR) cutoff grades of $65/ton and $42/ton respectively as illustrations of potentially economic mineralization. The drill hole traces are also shown on the sections. The red and green color codes on the drill hole composite intervals also reflect the $65 and $42/ton NSR cutoffs on Sections 9-1 through 9-4.

Figure 9-3 (Section J-J') provides the clearest illustration of the three zones of the deposit and their relative positions to each other. The sections are illustrated in plan view on Figure 7-3.



$42/ton NSR
$65/ton NSR
Josephine Mining Corp.
Turner Gold Project
CROSS SECTION
Drill Section N-N, 19025 North
Oct 2009
22201

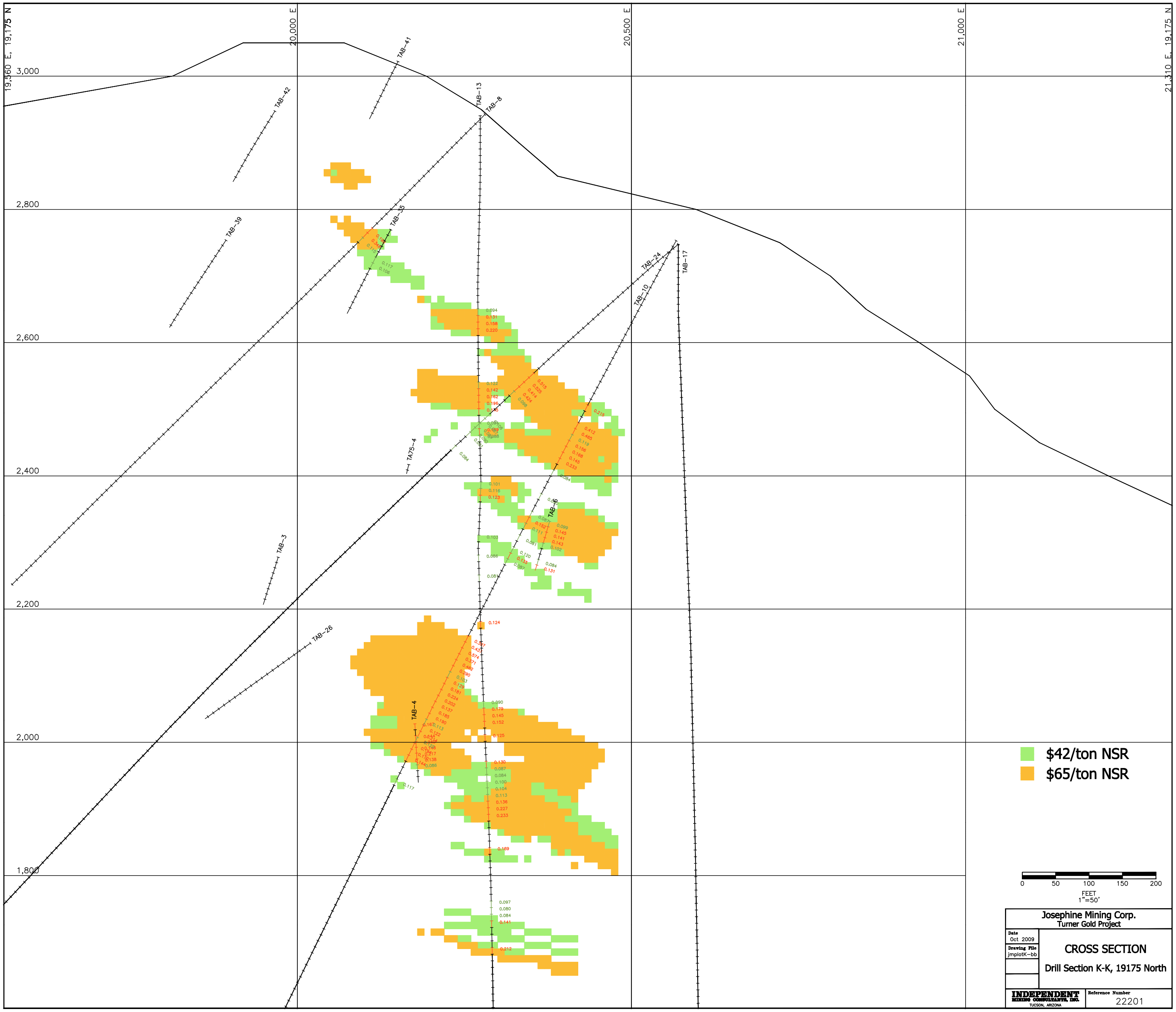


Figure 9-2
Section K-K'

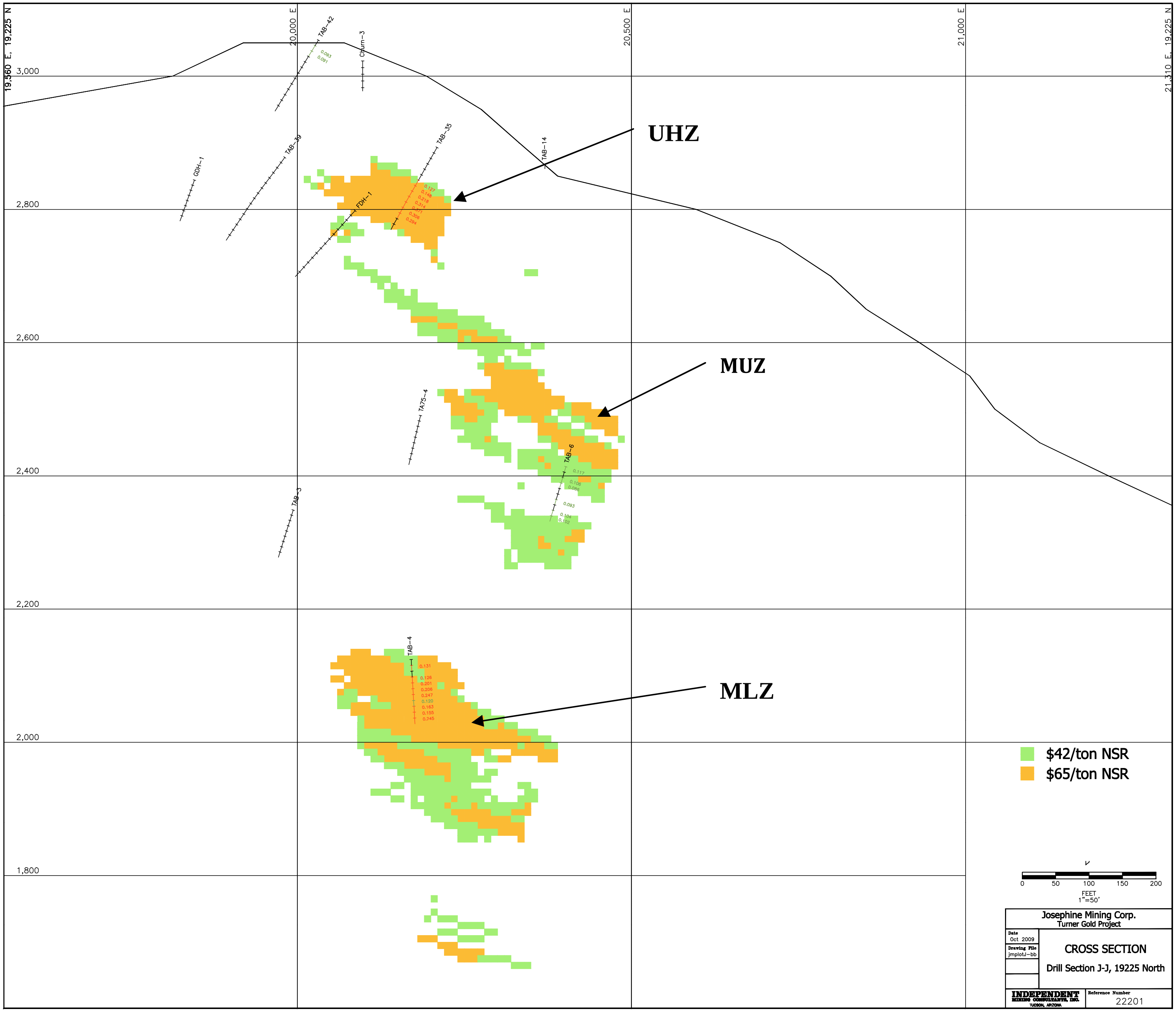


Figure 9-3
Cross Section J-J'

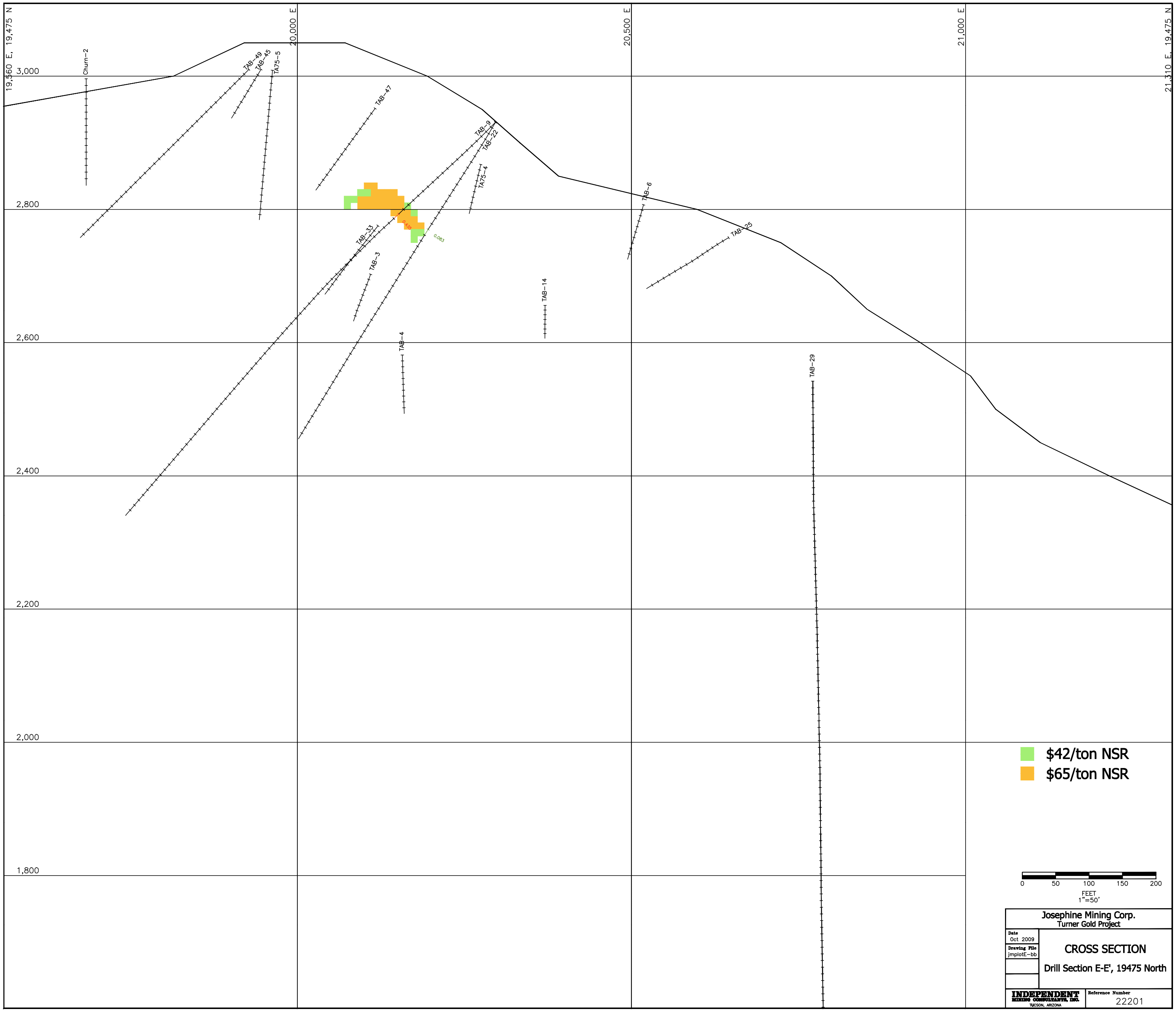


Figure 9-4
Cross Section E-E'

10.0 EXPLORATION

Exploration of the Turner Gold deposit has spanned many decades, and represents the efforts of numerous companies and individuals. A wide variety of techniques have been employed, including:

1) Surface and underground mapping and sampling
2) Drilling (primarily core)
3) Geochemistry (soil, stream, and down-hole)
4) Surface, down-hole, and airborne geophysics, including induced polarization, resistivity, pulse-EM, and magnetometer
5) Cross, long, and plan sections
6) Physical and conceptual three-dimensional modeling.

A significant portion of past work has focused on drilling to explore and define the economic potential of the property. Please refer to Section 11.0 (Drilling) and Table 11-1 for a summary of known drilling to date on the Turner Gold deposit. Section 6.0 (History) also summarizes much of the work done in the past by previous workers.

Future exploration will focus on confirming existing resource and expanding it further.

11.0 DRILLING

Historical

Seven (7) exploration companies have drilled a total of 64,112.2 feet in 84 separate holes on the Turner Gold deposit (see Figure 11-1 and Table 11-1). Of this total, 3,256.5 feet in 43 holes penetrated potentially ore-grade material.

With few exceptions, core drilling at Turner has been relatively straightforward, with minimal loss of recovery. Core size varied with project and hole, ranging from HQ to AX. Casing and reducing, and/or cementing, has worked well in the past, and fewer than five holes were abandoned due to drilling problems.

Turner Gold: Table 11-1
Drilling History by Company

1	Core Hole	Granby	1957/58	GDH-1
4	Churn Holes	Loyd Frizzell	1960's	Churn-1 to 4
2	Core Holes	Loyd Frizzell		FDH-1 and 2
9	DDH Holes	AmSelco	1974-1975	TA74 –1 to 4
				TA75-1 to 5
30	DDH Holes	Baretta	1980-1981	TAB-1 to 30
18	DDH Holes	Noranda	1982	TAB-32 to 48
13	DDH Holes	Rayrock	1983 - 1986	TAB-49 to 61
7	DDH Holes	Lupine-AUR	1989	TAB-62 to 68

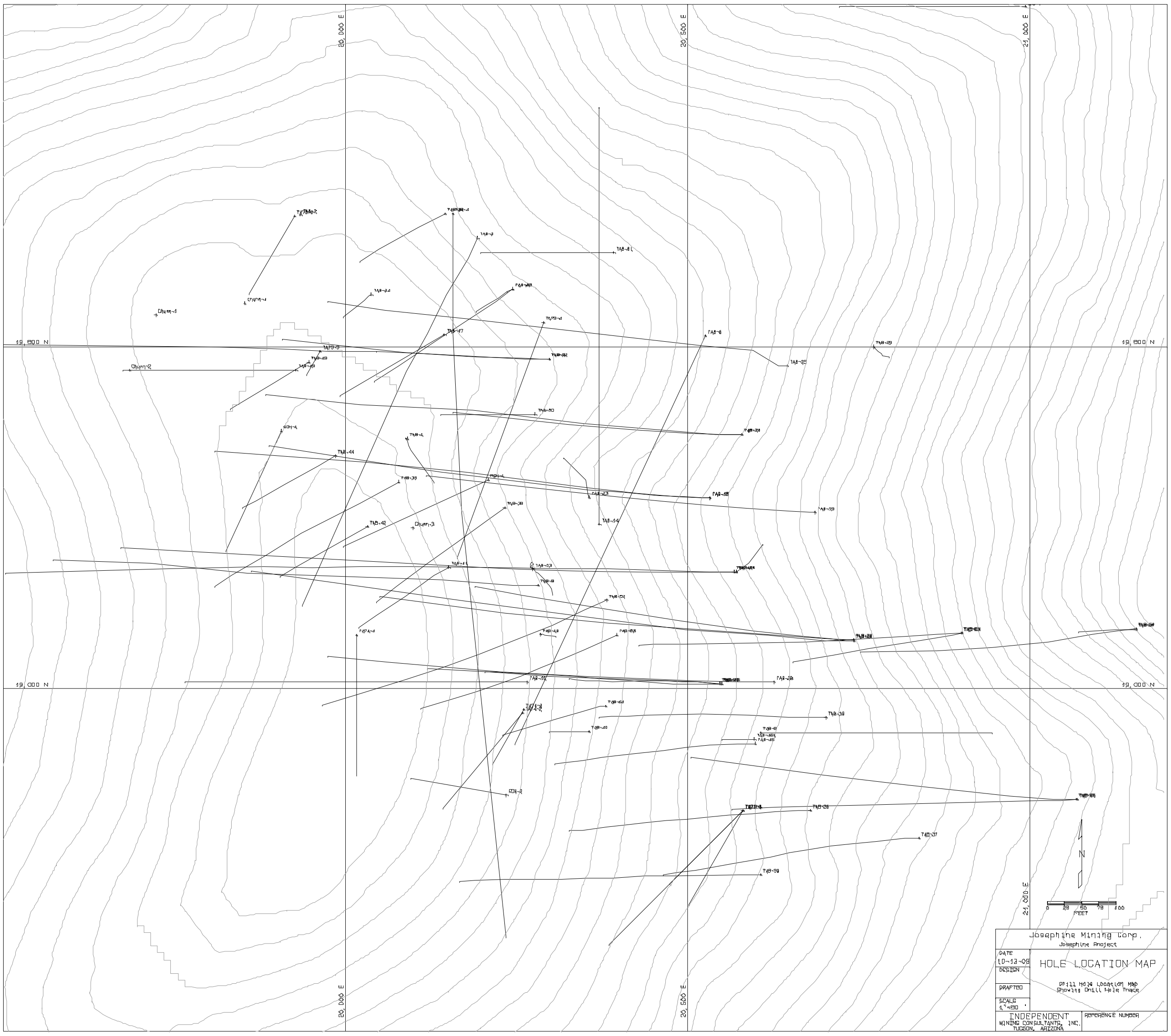
84 known total holes

A minimum of nine (9) drilling companies and fourteen (14) different geologists have been involved and/or responsible for drilling and core logging duties at the Turner Gold deposit. Table 11-2 is a summary of drillers and geologists known to have worked on the project.

Turner Gold: Table 11-2
Core drilling, logging, and sampling, by project

Project	Dates	Drilling Contractors	Logging / Sampling
Granby	1954/55	Unknown	Jan Haney (re-log 1984)
Frizzell	1960s	Shannon Drilling	Lloyd Frizzell
AmSelco	1974/75	Fran-Berg Drilling Co.	John Prochnau Les Bradshaw
Baretta	1980/81	Kay-Way Drilling	Geoff Garcia Charlotte Garcia Jim Haight Ace Parker Gary McLean Jan Haney Mike Strickler
Noranda	1982/83	Ruen Drilling	Roger Kuhns Jan Haney Mike Strickler
Rayrock	1983/85	Heli-Core Diamond Drilling D. H. Tift Diamond Drilling S.D.S Drilling Co.	Karen Comstock Mike Strickler Jan Haney
Aur/Lupin	1989	Advance Diamond Drilling	Perry and/or Bidwell
USGS	1982/84		Rob Zirenberg

Turner Gold: Figure 11-1
Plan view: Drill Hole Location Map
And Trace of the Holes



500 ft Grid Shown

Future

Future plans include twelve additional holes to confirm and expand the resource.

12.0 SAMPLING METHOD AND APPROACH

Resource estimates at Turner Gold have historically been based upon drilled intercepts. Nearly five thousand samples have been taken over the course of the project, and have been handled by no fewer than fourteen (14) geologists using nine (9) separate analytical labs (see Tables 11-2 and 13-1).

Sampling has commonly run concurrently with core logging, with sample intervals being determined by the geologist performing the core logging duties. In general, samples have been cut at five-foot intervals through the heart of the mineralized intercepts; however, partial-length samples are common at the top and bottom of an intercept, as well as randomly throughout the deposit, as determined by lithology, or the purpose of the geologist logging the core.

Sampled intervals were commonly run for gold, silver, copper, zinc, and cobalt. Recovered core was either split or sawn (depending on the program), with (as was common) one half of the split being returned to the boxes, and the remaining half submitted to a lab for analysis.

The presence of marcasite in portions of the deposit was first noted during the Noranda program, when core from the UHZ (TAB-33) began to oxidize and decrepitate several months after logging and sampling. In the interest of retaining fresh, unaltered sulfide material for metallurgical testing, selected intervals drilled during the Noranda and Rayrock programs were quartered, with one portion (quarter or half) being returned immediately to the box, and one quarter sent for assay. The remaining half (or quarter) was encased in plastic, flooded with nitrogen gas in order to displace the oxygen, and sealed.

The resource statements within this document are based upon assays collected by diamond drilling. The assay information was obtained from historic paper drill logs on file. A verification process was completed by IMC in an effort to validate the historic information. This process is described in Section 14.0.

13.0 SAMPLE PREPARATION ANALYSIS AND SECURITY

Estimates of mineralized tonnage and grade at Turner Gold have historically been based upon drilled intercepts. Approximately five thousand samples have been taken over the course of the project and were processed by no fewer than nine separate analytical labs (see Table 13-1).

Turner Gold: Table 13-1
Analytical Labs, By Company

Project	Dates	Assayer(s)
Granby	1954/55	Unknown
Frizzell	1960s	Union Assay Office, Inc.
AmSelco	1974/75	Rocky Mtn. Geochemical
Baretta	1980/81	Metallurgical Labs, Inc. Hunter Mining Lab, Inc. Bondar-Clegg Hoagland
Noranda	1982/83	Lakefield Cone Geochemical
Rayrock	1983/85	Min-En Labs Rocky Mtn. Geochemical Hunter Mining Lab, Inc.
Aur/Lupine	1989	Unknown

Certification credentials of the above assay laboratories are unknown as the work was completed in the past. Some of laboratories have been acquired, or closed in the interim.

Sample security varied by project and individual. In general, sampled intervals were determined and marked by the geologist during the logging process. For the bulk of the drilling (Baretta, Noranda, and Rayrock), all splitting and/or cutting occurred in same room in the O'Brien core shack, thereby minimizing the risk of disruption of the core during transport. The splitter/cutter would be responsible for bagging and tagging the analytical samples, and returning the saved portion to the original core box. The writer (M. Strickler) remembers no incident that would significantly impact the validity of the historic results.

Limited check assays were collected and run by several companies, including a suite of samples completed by Noranda from two of their initial holes (TAB-33 and TAB-35).

J. Marek (IMC) collected a suite of forty-four drilled intervals on September 2, 2009. All samples were obtained from material remaining in the O'Brien core shack, and were considered representative of variations in original program, lithology, and reported grade. The intent was to verify the original assay results. Please see Section 14.0 for a summary of results.

14.0 DATA VERIFICATION

All of the information that is available for the Turner Gold deposit is historic. No new drilling or assay information has been collected by JMC other than the 44 sample check program that was implemented for this study. It should be noted that additional drilling is planned by JMC at the time of this writing.

The drill hole survey, down hole survey, and assay data was provided to IMC in the form of historic paper drill hole logs. No electronic data base of the original information was available. IMC staff reviewed the historic reports to understand the historic drill programs and searched the extensive paper files for corresponding drill logs, surveys, and assays for the drill holes referenced in text.

Once the paper documents were found, they were keypunched into an electronic drill hole data base that established the basis for the assembly of a deposit block model.

The data verification process that IMC has applied to the project consists of the following tasks:

1) Verification of the data entry of the original assays by a complete, second entry of the data and cross check between both entries.
2) Practical check on collar coordinates and down hole surveys
3) Approximate field confirmation of drill hole locations relative to the available historic maps.
4) Confirmation of available core for selected drill holes in the core storage shed in O'Brien, Oregon.
5) Collection of 44 samples of the remaining half of split core from the existing drill core. These samples were sent to ALS, Reno for independent check assay.

Each of the above items will be discussed briefly in the following paragraphs

As a result of the verification work summarized in this Section, IMC has formed the opinion that the historic data base is sufficient for the development of indicated and inferred mineral resources. Sufficient uncertainties exist that measured resources should not be supported by this data set.

14.1 Data Entry Verification

The electronic drill hole data base was assembled by IMC personnel by hand entry of the information from the historic paper drill logs that were found in the project archives as obtained by Josephine Mining Corp.

The drill hole logs were in multiple formats which sometimes changed within a single hole. Assay units were reported in both percent and ppm for base metals, and in troy ounces per short ton and ppm for precious metals. Assay units sometimes were changed within the log of a single hole. These format and units issues complicated the data entry process.

In order to confirm proper interpretation of the historic drill logs, IMC staff entered the data twice. Two different persons entered the data, separately from each other, without knowledge of the entry results or progress of the other.

The resulting two data bases were compared interval by interval. Discrepancies between the two entry sets were resolved by an ore reserve professional with substantial experience in this type of data.

The final resolved data set resulted in a single set of assay data for each drill hole and all assayed intervals. The final units used for the data base, block model, and mineral resource estimate are:

Units for Data Base and Block Model

Gold:	Troy ounces per short ton
Silver:	Troy ounces per short ton
Copper:	Percent by weight
Zinc:	Percent by weight
Cobalt:	Percent by weight

IMC did not discover a description of the historic methods used for sample preparation or assay for the data base. Procedures certainly changed over time from drill program to drill program as summarized in Section 13.

In summary, the resulting electronic data base is as reliable as the original paper drill logs. However, the only independent verification of the assay values on the paper logs is the completion of the independent check assays.

14.2 Practical Check on Collar Coordinates and Down Hole Surveys

Drill hole collar coordinates were entered into the computer data base from the collar coordinate information provided on the drill logs. Down hole surveys, where available, were entered from paper records from the historic archives.

Historic drill location maps were available from the historic work completed by Rayrock Mines. Those maps showed the drill hole locations versus topography and drill road locations.

The IMC electronic data base was compared to the historic maps by plotting a drill hole location map of the IMC entered data. Drill hole locations in the assembled data base were confirmed within a few feet of the historic maps that were available for checking.

Drill hole collar coordinates for TAB-60 were never found in the paper record. Consequently the drill hole was not utilized in this model development and mineral resource estimate. Subsequent effort found the hole located on a historic map. However, the collar location was not measured from the map and TAB-60 was not used in the data base.

Of the 84 known drill holes, down hole survey information was found for 57 drill holes. Many of the down hole survey paper files indicated that a Tropari downhole instrument was used to obtain the drill hole inclinations. Scans of down hole survey information generally indicated reliable results although there were a few observations where there had been errors reading the quadrant compass of the Tropari.

A Tropari utilizes a magnetic compass down hole. Consequently, it is impacted by iron in the rock mass and drill hole casings and drill pipe. Expectation is that the holes were surveyed without casing or drill pipe in the holes, however there is no available confirmation of that assumption.

Occasional drill hole ore intercepts on section showed substantial offsets when compared with nearby holes. IMC holds the opinion that these offsets may be a function of variation in the down hole survey information rather than geologic occurrence. As a result of this uncertainty, IMC has formed the opinion that there is no measured category mineralization within the Turner Deposit at this time.

TAB-6 was one of the longer drill holes in the project. It is 1146 feet deep and was drilled at a bearing of 205 degrees with a 55 degree plunge. IMC could not find down hole survey information for this drill hole. Consequently, the entire hole was assumed to be on the coordinate of the collar orientation.

The MUZ intercepts of TAB-6 were consistent with other holes nearby; however, the lower portion (MLZ) of TAB-6 did not correlate with any nearby drill holes. Consequently, assay information in TAB-6 below the 689 ft depth was not used in the estimation of the block model or mineral resources as the location of the deep assays are highly in question.

14.3 Field Confirmation of Drill Hole Locations

The major drill campaigns at the Turner project marked the drill hole collars with either steel pipe or PVC pipe with the drill hole ID clearly marked. This would have provided a good opportunity to check the survey location with modern GPS equipment.

However, the timber on the site has been harvested since the drill hole collars were marked and many of the collar monuments have been disturbed from their original locations.

During the IMC site visit, a number of the PVC markers were found within a few feet of the mapped location and collar coordinate information. These had been disturbed to some degree, but the locations of the monuments were found to be consistent with the historic maps. IMC personnel were able to locate at least 10 holes without extensive effort.

During a separate effort, IMC selected 12 holes for coordinate verification by JBR personnel during one of their site visits. IMC was not aware of the disturbance by timbering when generating this list of holes. Of the 12 holes on the IMC generated list, 5 were located based on the presence of the drill hole monument in the same manner as those found by IMC. A few were in their original positions, but most were disturbed to some degree but located within a few feet of their historic recorded position.

The JBR work with a GPS on site indicates that there may be a local rotation “of a couple of degrees” to the original project survey grid. This is consistent with notes on historic Baretta exploration maps that state that the project grid is rotated 2 degrees, 13 minutes, and 18 seconds off of true north.

In summary, a high level check of the survey grid and all existing monumentation is recommended prior to more advance feasibility work at Turner.

14.4 Confirmation of Available Core

IMC developed a list of 50 samples for check assay that were located throughout the deposit and that represent the multiple drill programs. This list of 50 samples was spread between 27 different drill holes that covered the range of dates, drill programs, and drilled areas of the deposit.

Drill core from 25 of the 27 selected holes were found. Drill core from TA75-1 and TAB-3 was not found in the core shed. There were occasional core boxes missing from the 25 holes that were found in the core shed. In one case in drill hole TAB-33, the core box was found, but the remaining half core was missing as it had already been sampled by previous investigators.

Considering the number of drill programs and the time since drilling ceased, the existence of the majority of a requested sample set should be considered to be a reasonable verification that the core was indeed drilled and does indeed exist.

Comparisons of the original rock type log with the observed material in the core boxes was reliable.

14.5 Independent Assay

In order to gain some confidence in the historic paper record of assays, IMC established a program to collect 50 samples of the remaining half core for independent assay. The selection list for these 50 samples were from 27 drill holes and covered a range of grades, locations, and drilling programs. The percentage of samples from each drill program reflects the percentage of drilling from that program in the data base.

The sample selection list was generated with the following criteria:

1) All assay intervals above 0.10 oz/ton equivalent gold were identified (Approximately $50.00 NSR/ton).
2) The percentage of drilling from each drill program in the list above was identified and that ratio used to set the number of samples to pull from each drill program.
3) The grade range of sample was selected somewhat randomly in order to have the sample population that reflects the overall grade distribution of the original data base above the 0.10 oz/ton gold equivalent cutoff.
4) The list of 50 samples was developed without prior knowledge of the core availability or condition. JMC personnel were not aware of the list until the IMC site visit when the samples were collected.
5) Mike Strickler and JMC management personnel helped find and buck core boxes. However, all sample collection, labeling and bagging was completed by IMC personnel in order to assure independent collection of the sample.

After diligent search of the core shed, 44 samples from 25 drill holes were identified, labeled and bagged for shipment to the assay lab.

The samples were transported by truck from O'Brien to the ALS Laboratory Group of ALS Minerals (ALS) lab in Reno, Nevada by M. Price, Vice President of Operations for JMC. IMC specified the sample preparation, and assay procedures to be applied to the samples. ALS assay reports were sent to IMC and JMC independently by the ALS lab so JMC did not see the results prior to IMC or handle the results prior to receipt by IMC.

ALS Minerals have ISO9001:2000 and ISO 17025 certifications.

The sample preparation and assay methods selected by IMC and confirmed by JMC were as follows:

1) Log samples into the ALS lab tracking system
2) For samples with solid pieces of core, complete a specific gravity test with ALS procedure: OA-GRA08. This method uses water immersion to measure specific gravity. Only solid samples capable of immersion are planned for testing.
3) ALS: PREP-31, Crush entire sample to 2mm, split 250 gram, and pulp the split using a ring-and-puck pulverizer to obtain 85% passing 75 micron (200 mesh).
4) ALS: Au-AA25, fire gold assay with AA finish using a 30 gm aliquot.
5) ALS: ICP61, four acid digestion with ICP finish for silver, copper, and zinc.

Figures 14-1 through 14-5 illustrate the XY plots of the original assay versus the independent check assays completed on the IMC sample set. Basic statistics and the results of a Paired-T hypothesis test are also presented on the figures.

Figure 14-1 summarizes the check assay results for gold. There are 3 samples with a wide variance on the graph, but the results are generally unbiased with the check assays having similar grade to the original data base assays. The Paired-T statistic is smaller than 2.00 meaning that the differences between the paired data are essentially unbiased.

Figure 14-2 summarizes the check assay results for copper. The copper results show that 6 samples returned check assay values that were significantly lower than the historic recorded information. The mean of the original data is 1.37% copper and the mean of the check data is 1.10% copper. The Paired-T test is slightly higher than 2.00 meaning that the differences between the paired samples do not quite meet a 95% confidence for representing the same mean.

Typically copper is one of the easier elements to replicate in check assay. The samples that were out of tolerance for copper were not the same samples that were out of tolerance for gold or the other metals, meaning that the odds of a sample swap or miss-naming are minimal. One can note that the 4 to 6 samples that are out of tolerance are all relatively high grade (above 2.00% copper). This may indicate that the original or the check lab may have had issues reporting high value samples from the analysis method that was applied. All but one of the 6 were Baretta sourced information. If one removes the two samples that have an original assay of about 5.0% copper, the Paired-T test would indicate that the two data sets are similar with better than 95% confidence.

The uncertainty of the copper result raises concern and is one of the reasons that IMC holds the opinion that the data base does not support the estimation of measured category mineral resources.

Table 14.3 summarizes the results for silver. In summary, there are a number of samples where the original sample reported zero or low value and the check assay has reported a substantially higher value. As with copper, the Paired-T test indicates that the two data sets cannot be interpreted as similar with 95% confidence. However, in this case, it appears that the original data is conservatively low valued.

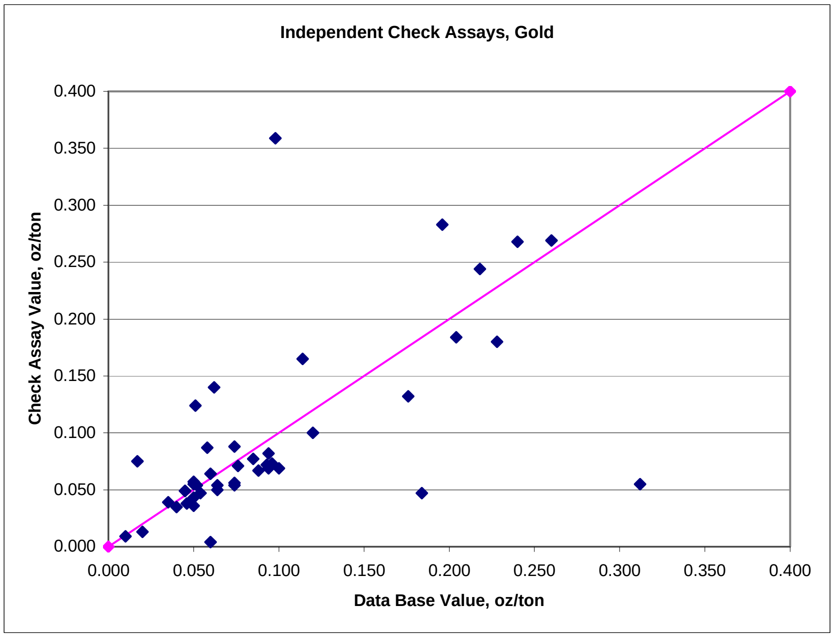
Table 14-4 summarizes the results for zinc. The zinc results appear generally reliable in that the check assays report similar results as the original values.

Table 14-5 summarizes the results for cobalt. There are two or three samples with some variance, but the means are similar, and the Paired-T test indicates that the differences between samples are sufficiently small that the two data sets can be considered similar with 95% confidence.

The results of the check assay work raise questions regarding copper. The issues associated with silver are not as critical because silver is not as significant economically as copper.

Since the check samples were split core, the tests reflect the impacts of sample preparation as well as assay procedure. The low values in the copper check results should trigger a more extensive data acquisition and check program before more advanced analysis is applied to the Turner deposit. A drill program will be required to confirm the deposit and add confidence for future analysis.

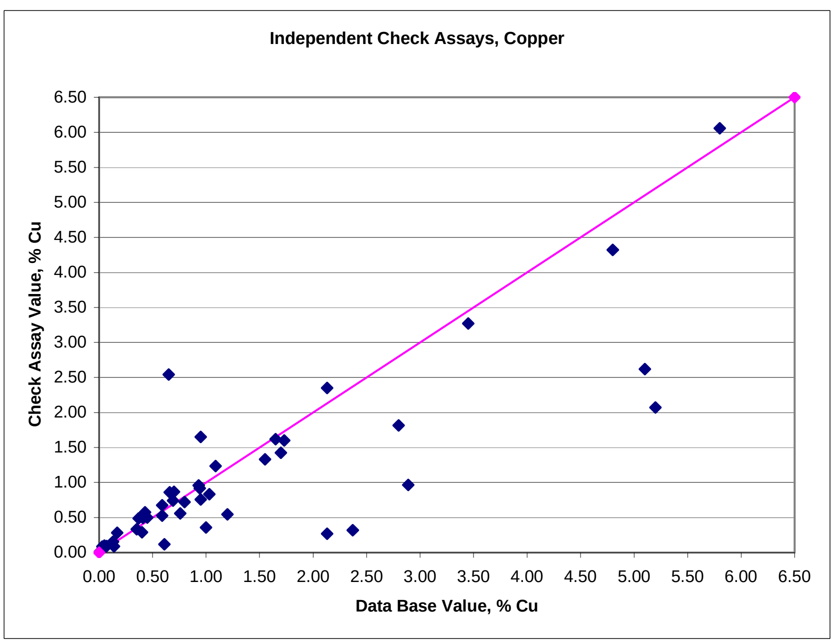
Figure 14-1
Independent Check Assay Program Results for Gold



Number of Samples	44
Data Base Mean	0.097 oz/ton
Data Base Std Dev	0.072
Check Assay Mean	0.095 oz/ton
Check Assay Std Dev	0.080
Paired-T Statistic	0.214

Line with a Slope of 1

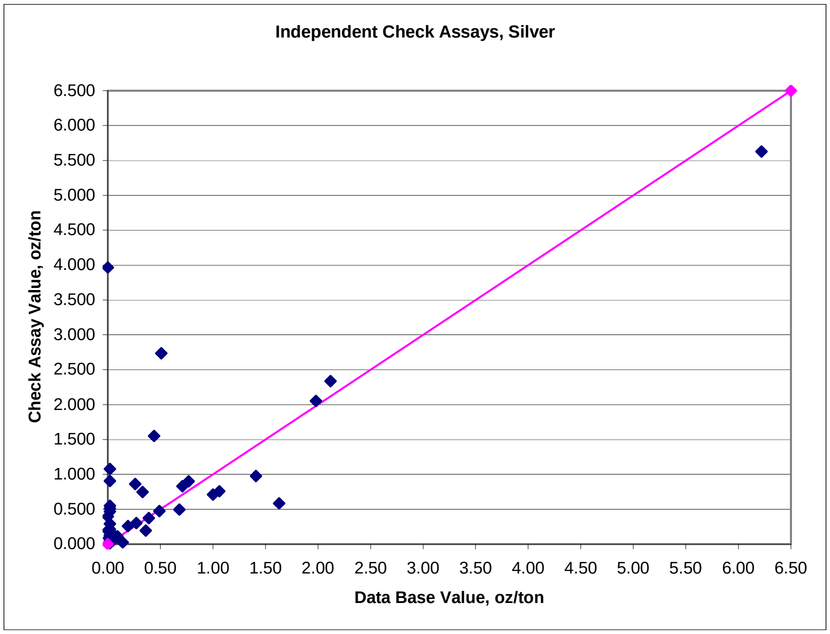
Figure 14-2
Independent Check Assay Program Results for Copper



Number of Samples	44
Data Base Mean	1.374 %
Data Base Std Dev	1.478
Check Assay Mean	1.100 %
Check Assay Std Dev	1.201
Paired-T Statistic	2.138

Line with a Slope of 1

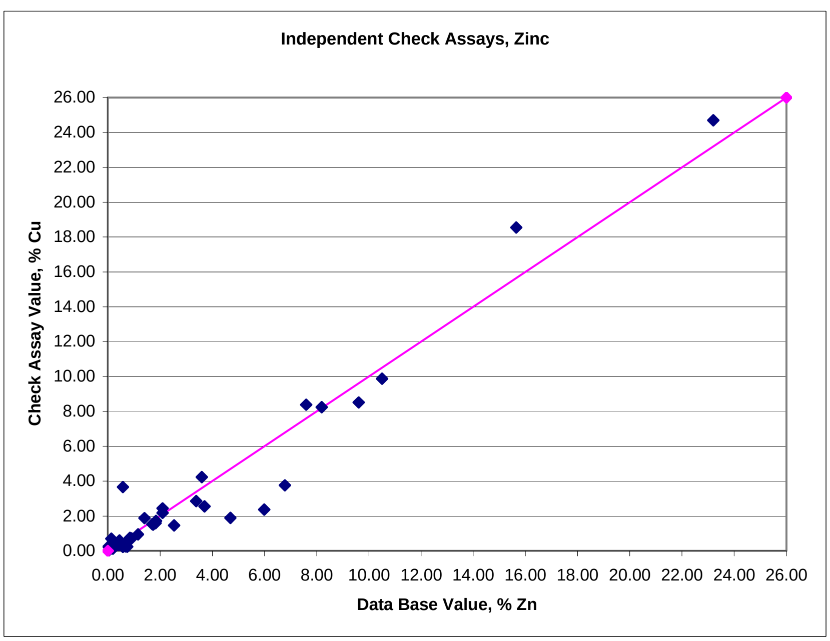
Figure 14-3
Independent Check Assay Program Results for Silver



Number of Samples	44	
Data Base Mean	0.488	oz/ton
Data Base Std Dev	1.037	
Check Assay Mean	0.739	oz/ton
Check Assay Std Dev	1.101	
Paired-T Statistic	2.202	

Line with a Slope of 1

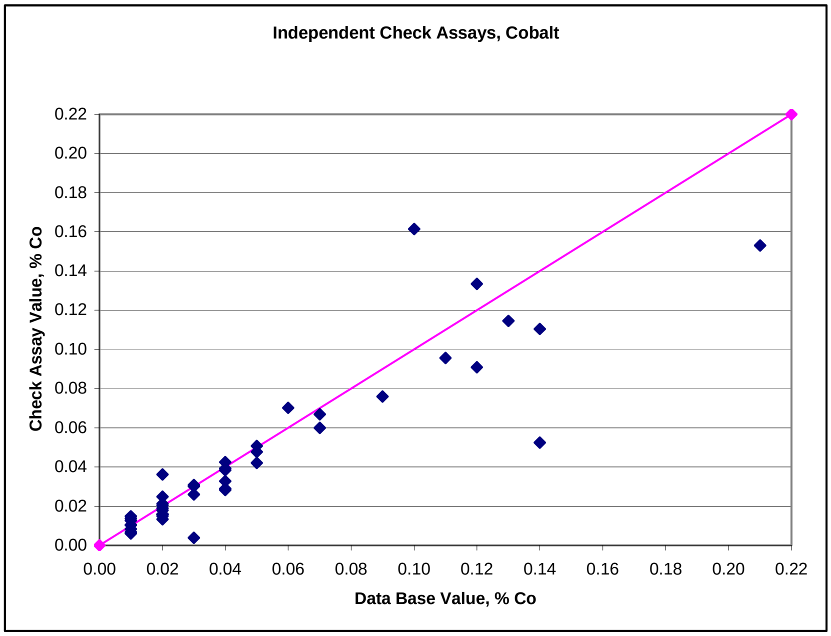
Figure 14-4
Independent Check Assay Program Results for Zinc



Number of Samples	44	
Data Base Mean	2.904	%
Data Base Std Dev	4.620	
Check Assay Mean	2.774	%
Check Assay Std Dev	4.880	
Paired-T Statistic	0.751	

Line with a Slope of 1

Figure 14-5
Independent Check Assay Program Results for Cobalt



Number of Samples	44	
Data Base Mean	0.049	%
Data Base Std Dev	0.046	
Check Assay Mean	0.044	%
Check Assay Std Dev	0.040	
Paired-T Statistic	1.831	

Line with a Slope of 1

14.6 Density Tests

The 44 sample check program was used to obtain a validation of the rock density. Previous work by R.L. Russell during 1988 had applied a rock density of 9 cu ft / ton. However, IMC was not able to find numeric backup for that assumption within the paper records.

Of the 44 samples submitted to ALS for check assay, 38 contained samples that were sufficiently intact to permit immersion in water using the ALS test protocol OA-GRA08. Samples were weighed in air. They were then weighed after immersion in water. No coatings were applied to the samples for this test.

The test results are summarized on Table 14-1. The average of all the values tested was 9.211 cubic feet per ton or about 2.5% lighter than the previous estimates by Russell. IMC observations of core during the site visit found that there were voids within the BFR and DBF rock units as well as zones of significant fracturing in the deposit. Therefore IMC reduced the measured density by another 2.5% to reflect that observation in the field.

The final outcome was an overall 5% reduction in bulk density from the previous work in 1988.

Table 14-1
Turner Gold
Density Test Results

IMC Sample No.	HoleID	From	To	East	North	Elev	Rock	Weight Air gm	Weight Water gm	SpG
JOE-15	TAB-15	390.0	395.0	20,332.0	19,293.3	2,410.3	Basalt	399.0	283.5	3.45
JOE-25	TAB-30	748.8	750.0	20,195.1	19,116.2	2,152.1	Basalt	637.7	489.4	4.30
JOE-35	TAB-56	910.0	915.0	20,358.2	19,012.0	1,919.6	Basalt	404.5	288.9	3.50
JOE-8	TAB-10	750.0	755.0	20,219.0	19,176.9	2,085.9	Basin Floor Rubble	457.2	336.7	3.79
JOE-9	TAB-10	820.0	825.0	20,187.8	19,178.5	2,023.3	Basin Floor Rubble	498.0	349.6	3.36
JOE-28	TAB-13	415.0	420.0	20,271.5	19,184.3	2,523.4	Basin Floor Rubble	450.8	329.9	3.73
JOE-27	TAB-13	635.0	640.0	20,271.5	19,181.0	2,303.6	Basin Floor Rubble	401.9	282.2	3.36
JOE-12	TAB-13	970.0	975.0	20,282.8	19,167.6	1,969.1	Basin Floor Rubble	488.0	350.6	3.55
JOE-11	TAB-13	1055.0	1060.0	20,286.5	19,164.2	1,884.3	Basin Floor Rubble	486.0	327.6	3.07
JOE-14	TAB-15	835.0	840.0	20,092.2	19,323.3	2,036.7	Basin Floor Rubble	593.6	426.5	3.55
JOE-31	TAB-18	435.0	440.0	20,287.6	19,080.1	2,508.9	Basin Floor Rubble	393.3	275.9	3.35
JOE-30	TAB-18	505.0	510.0	20,288.3	19,080.1	2,438.9	Basin Floor Rubble	352.8	229.0	2.85
JOE-23	TAB-18	970.0	975.0	20,296.1	19,078.2	1,974.0	Basin Floor Rubble	436.4	304.5	3.31
JOE-32	TAB-18	1045.0	1050.0	20,297.4	19,078.0	1,899.0	Basin Floor Rubble	349.5	230.5	2.94
JOE-40	TAB-24	390.0	395.0	20,279.2	19,176.8	2,484.5	Basin Floor Rubble	518.7	367.6	3.43
JOE-21	TAB-27	755.0	760.0	20,342.4	19,122.9	2,021.4	Basin Floor Rubble	617.4	434.5	3.38
JOE-22	TAB-27	825.0	830.0	20,301.3	19,129.4	1,965.1	Basin Floor Rubble	493.8	349.3	3.42
JOE-43	TAB-4	970.0	975.0	20,173.8	19,222.1	2,069.4	Basin Floor Rubble	640.9	480.5	4.00
JOE-38	TAB-4	1060.0	1065.0	20,178.2	19,179.4	1,990.3	Basin Floor Rubble	437.9	305.9	3.32
JOE-7	TAB-10	290.0	292.0	20,429.7	19,170.9	2,496.5	Chert	550.8	366.4	2.99
JOE-5	TAB-10	1025.0	1030.0	20,097.9	19,183.2	1,839.2	Debri Flow	738.0	504.8	3.16
JOE-44	TAB-35	185.0	190.0	20,145.7	19,202.8	2,779.9	Debri Flow	482.1	319.6	2.97
JOE-29	TAB-36	675.0	680.0	20,460.2	18,807.4	2,061.1	Debri Flow	317.9	203.8	2.79
JOE-16	TAB-22	195.0	200.0	20,192.0	19,483.9	2,763.6	Massive Sulfides	557.8	418.6	4.01
JOE-4	TAB-23	430.0	435.0	20,352.0	19,295.6	2,432.8	Massive Sulfides	698.1	509.1	3.69
JOE-41	TAB-26	460.0	465.0	20,363.6	19,103.2	2,398.2	Massive Sulfides	521.3	393.4	4.08
JOE-26	TAB-30	445.0	450.0	20,421.0	19,093.1	2,350.9	Massive Sulfides	685.7	526.3	4.30
JOE-24	TAB-30	815.0	820.0	20,143.3	19,122.5	2,108.3	Massive Sulfides	557.3	404.5	3.65
JOE-39	TAB-41	167.5	170.0	20,079.2	19,130.5	2,876.8	Massive Sulfides	484.1	359.3	3.88
JOE-36	TAB-43	750.0	755.0	20,305.8	19,020.3	2,099.4	Massive Sulfides	476.6	365.2	4.28
JOE-37	TAB-43	815.0	820.0	20,286.8	19,021.3	2,037.3	Massive Sulfides	513.2	361.1	3.37
JOE-33	TAB-56	790.0	795.0	20,381.6	19,011.1	2,037.3	Massive Sulfides	611.7	458.6	4.00
JOE-34	TAB-56	830.0	835.0	20,373.9	19,011.3	1,998.0	Massive Sulfides	472.2	335.0	3.44
JOE-3	TAB-23	580.0	585.0	20,351.0	19,300.7	2,282.9	Massive Sulfides	629.6	436.7	3.26
JOE-2	TA75-3	220.0	225.0	19,861.2	19,579.5	2,794.7	Unknown	354.8	229.0	2.82
JOE-19	TAB-63	780.0	785.5	20,494.7	19,066.2	1,872.7	Unknown	481.2	332.0	3.23
JOE-18	TAB-63	880.0	885.0	20,444.8	19,064.8	1,786.3	Unknown	385.4	254.5	2.94
JOE-20	TAB-65	677.5	679.0	20,650.2	18,880.7	2,011.7	Unknown	302.3	219.3	3.64

Average SpG in Cubic Feet per Ton = 9.212 cubic feet/ton
IMC reduced Avg density by 2.8% to reflect Voids = **9.474 cubic feet/ton for resource**

Average SpG = 3.478
Number of Samples = 38
Standard Devation = 0.419
Maximum = 4.30
Minimum = 2.79

File" '/assays/DensityWork12Oct09.xls

15.0 ADJACENT PROPERTIES

There are currently no active mineral projects that are close to the Turner Deposit. Other massive sulfide mineralization has been identified at historic locations in the Klamath Mountains. As noted in Section 7.0, those other properties include: Monumental, Fall Creek, Iron Hat, Queen of Bronze/Cowboy Group, Almeda, Goff, Silver Peak, and Yankee Silver Lode.

There has been some interest in ultramafic hosted nickel mineralization some distance north and west of the Turner deposit. IMC is not aware of any current activity in that area.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section provides a discussion of the metallurgical test work used as the basis for the process design. This work focuses on the metallurgy testing performed by Lakefield Research in January1990 for AUR Resources Inc., the mineral lease holder at the time. This work was chosen because it represented the most detailed flotation investigation performed, by a considerable margin, compared to earlier work by others. It must be noted the USBM performed significant metallurgy testing on the ore, also. The USBM work was focused on whole ore autoclave leaching in an acid environment and not on flotation. The earlier flotation studies are also briefly described in Tables below.

A generalized process flow diagram is presented in Figure 16-1.

The nominal design was 1,355 t/d at 92% availability, operating seven days per week producing three separate concentrates; one each of copper, zinc and gold.

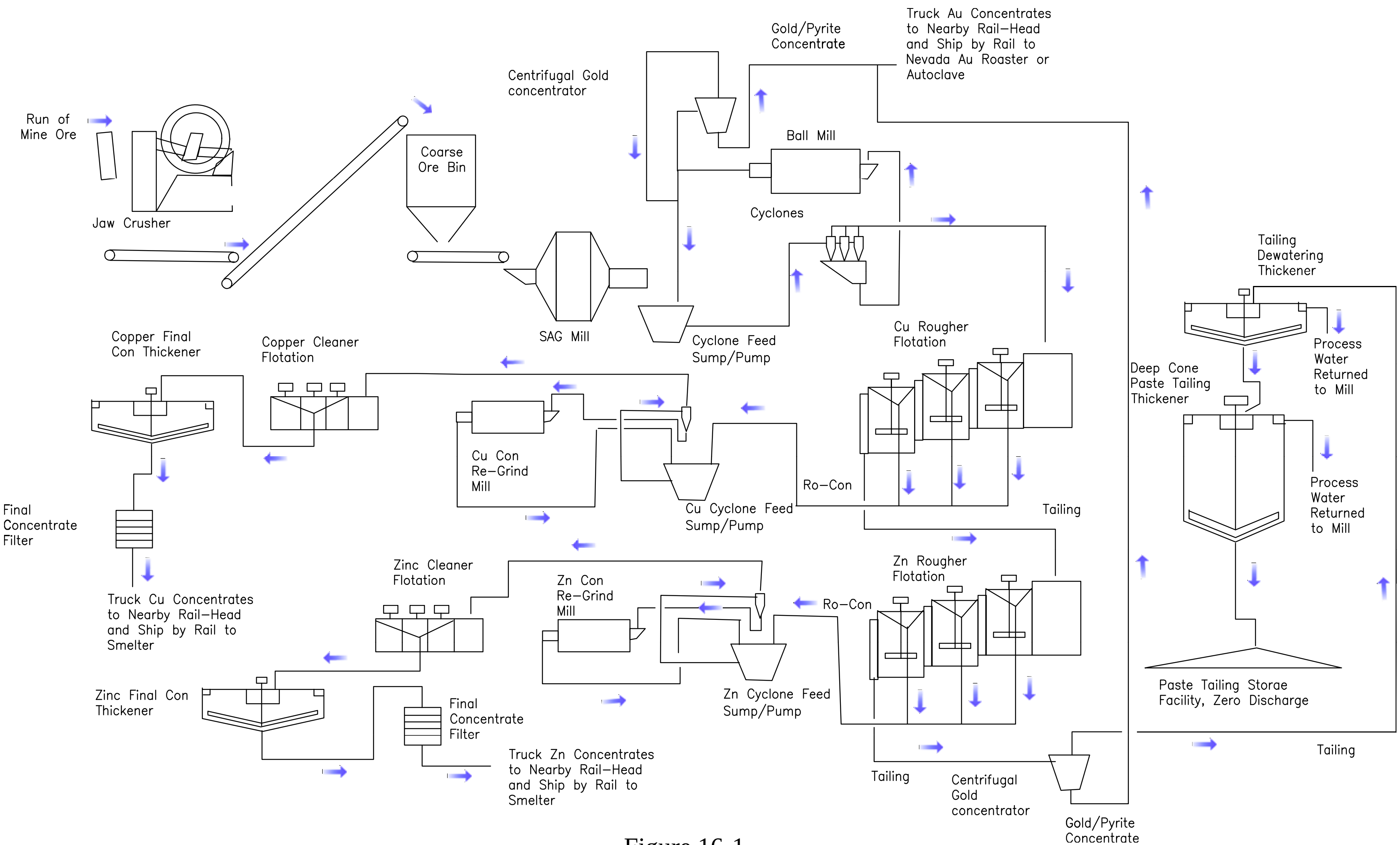


Figure 16-1
Process Flow Diagram

16.1 Grinding Area

Two sample composites were prepared for bond work index measurement from the drill core from the AUR/Lupine holes. A composite of the upper zone was prepared from the following four zones:

Zone	Interval (ft)
High Zn	298-355
High Au/Cu	355-375
Low Cu/Zn	375-480
High Cu	480-590

A composite of the lower zone was prepared from the following interval: 797-876. The terms upper and lower zones generally reflect the MUZ and MLZ respectively as discussed in Section 9.0.

The head analyses of these two composites are illustrated below.

Element	Lower Zone	Upper Zone
Copper, %	1.85, 1.71	0.97, 1.00
Zinc, %	3.20, 3.21	2.23, 2.27
Cobalt, %	0.009, 0.012	0.062, 0.070
Lead, %	0.030	0.006
Sulfur, %	19.5	38.1
Iron, %	19.3	35.0
Gold, g/t	4.80, 3.92	4.47, 6.36
Silver, g/t	15.6, 17.2	27.2, 25.9
Specific Gravity	2.95	3.33

It was not possible from the historical records to identify which drill holes the samples were taken from. The head assays are close to the assays for the minable estimates for the Lower and Upper Zones of ore as illustrated in Section 18.1. Thus, the material is a reasonable approximation from an assay point of view of the ore to be mined in each zone.

The Bond work index (kWh/metric ton) for each composite was determined in standard Bond ball mill closed circuit grindability tests and the results are reported below:

	Lower Zone	Upper Zone
Bond work index	16.5	11.5
Screen size, microns	74	74
Net undersize produced per revolution, grams	0.85	1.41
Product K_{80}, microns	56	60
Feed K_{80}, microns	2670	2670

The process facility design basis is 1,355 tons/day (tpd) and includes a 92% plant availability factor. The mine plans are consistent with this design basis. This section describes the grinding circuit and its design basis.

Design of Circuit

The grinding test results for the Lower Zone were applied to predict the primary grinding circuit. As much as 71% of the mineral resource is located in the Lower Zone and applying the grind test results form the Lower Zone will generate in a somewhat more conservative design.

The plant design was based on application of Hogg & Fuerstenau model for SAG mill power estimation and application of Bond's Law for estimation of conventional ball mill grinding capacity. The design also included some assumptions based on the author's experience regarding jaw crusher product size, practical product size from the SAG mill and probable Upper Zone rougher concentrate/cleaner tail regrind product size. The circuit design was a conventional SAB (with 1 ball mill).

1) A single SAG mill of 13 ft diameter and 5.67 EGL ft length connected to one 400 hp fixed speed motor
2) One ball mill at 12 ft diameter and 17.5 EGL ft length connected to one 1,300 hp fixed speed motor.
3) Copper circuit regrind mill treating rougher concentrate and potentially tailing recycle from cleaner flotation, one ball mill at 7 ft diameter and 11.5 ft EGL length connected to one 200 hp fixed speed motor.
4) Zinc circuit regrind mill treating rougher concentrate and potentially tailing recycle from cleaner flotation, one ball mill at 8 ft diameter and 11.0 ft EGL length connected to one 300 hp fixed speed motor.

The SAG mill will be outfitted with 50 mm (2 inch) grates and will discharge onto a trommel with a 3 mm screen aperture (0.12 inch).

Table 16-1: History of Grinding Tests

Test Description	Purpose	Performed By	Date	# of Tests	Results
Bond Ball Mill Tests	Prefeasibilty primary grinding study by AUR Resources	Lakefield Research	1990	2	Bond Mill Work Indices: Upper Zone 11.5 kWh/tonne, Lower Zone 16.5 kWh/tonne
Copper Cleaner Regrind Particle Size Analysis	Prefeasibility rougher con regrind study by AUR Resources	Lakefield Research	1990	1	Lower Zone, Cu rougher concentrate, product K_{80}, 19 micron

Additional future primary grinding test work on ore samples representative of the Lower Zone and Upper Zone is recommended to insure the primary grinding circuit is not over or under designed. Additional regrinding study is required to determine optimum regrind product size (K_{80}) for copper and zinc rougher concentrates from both the Upper and Lower Ore Zones.

16.2 Flotation Area

Design values reported here are based on the Aur Resources flotation testing reported by Lakefield Research in January 1990. The purpose of this test program was to provide prefeasibility level information regarding selective flotation producing separate copper and zinc concentrates from the Upper and Lower Ore Zones. Other metallurgy testing studies are briefly reported on in the Table 16-6 below. The other studies were scoping level studies and often evaluated cyanide leaching of tailings and whole ore, which is not applicable for this preliminary economic assessment.

The flow sheet developed for Turner was based on sequential flotation circuits for producing copper and zinc concentrates. Circuit designs were based on the author's interpretation of the Lakefield Research open circuit metallurgy testing results on Lower Zone Ore factoring in the effects of recycling cleaner tail streams. There was no lock cycle testing performed to simulate a commercial circuit. Also, the number of cleaner circuits was reduced from 4 to 2 by using modern Outotec tank cell technology. Additionally, gravity concentration circuits were added in the primary grinding circuit and on the zinc rougher tailings because these type circuits represent newer technology that had developed in the interim since 1990 and was reported to be successfully applied at commercially operating or operated volcanic massive sulfide deposits (see References). An estimate of 15% of the gold was assumed recovered to a gravity concentrate comprising mainly pyrite.

A summary of the Lakefield results used in the flotation circuit development are shown in Table 16-2 below.

Table 16-2: Selected Flotation Results for Flow Sheet Development

Test No.	Product	Wt. %	Assay, %, g/t			% Distribution		
			Cu	Zn	Au	Cu	Zn	Au
6	Cu Ro Conct.	12.01	12.9	10.7	19	88.0	40.7	53.8
24	Cu 1st Cleaner Conct.	6.54	21.5	8.7	30.9	82.0	19.3	44.3
24	Cu 1st Cleaner Scav Conct.	0.89	2.93	23.1	10.7	1.5	14.9	2.1
24	Cu Conct	4.54	28.2	4.8	37	74.7	7.4	43.2
13	Zn Ro Conct.	14.29	1.67	17.6	6.43	13.5	81.4	25.2
26	Zn 1st Cleaner Conct.	3.90	2.26	37.8	7.98	5.0	48.9	8.3
26	Zn 1st Cleaner Scav. Conct.	0.85	1.46	3.86	3.4	0.7	1.1	0.8
26	Zn Conct.	2.73	2.68	52.6	5.56	4.2	47.8	4.0

Table 16-3: Interpreted Results for Copper Circuit

Stream	Cu Stage Rec. %	Copper Grade %	Au Stage Rec. %	Au Grade g/t
Rougher	88.0	12.9	53.8	19.0
1st Cleaner	93.1	21.5	82.0	30.9
2nd Cleaner	85.0	28.2	80.0	40.7

Final Recovery Cu	86.9%
Final Recovery Au	52.6%

Table 16-4: Final Copper Concentrate Assay

	% Cu	% Zn	oz/t Au	oz/t Ag	% Fe	% S	% Insol
Cu Final Concentrate Grade	28.2	4.80	1.31	3.0	29.1	33.1	5.9

Table 16-5: Interpreted Results for Zinc Circuit

Stream	Zn Stage Rec. %	Zn Grade %	Au Stage Rec. %	Au Grade g/t
Rougher	81.4	17.6	25.2	25.2
1st Cleaner	60.4	35.0	33.0	24.0
2nd Cleaner	59.0	53.0	15.9	2.2

Final Recovery Zn	75.1%
Final Recovery Au	4.3%

Table 16-6: Final Zinc Concentrate Assay

	% Zn	% Cu	oz/t Au	oz/t Ag	% Fe	% S	% Insol
Zn Final Concentrate Grade	53.0	2.68	0.07	4.2	6.9	32.8	6.2

Table 16-7: Summary of Salable Metal Recovery

Metal	Percent Recovery			
	Copper Con	Zinc Con	Gold Con	Total
Gold	52.6	4.3	15.0	71.9
Silver	30.9	34.7	17.6	83.2
Copper	86.9	NA	3.9*	90.8
Zinc	NA	75.1	NA	75.1

*Economic modeling assumes no payment for copper in gold concentrate at this time.

Recoveries illustrated in Table 16-7 above are based on the interpreted recoveries and concentrates grades from the 1990 Lakefield test work and from other plant experience. Silver recovery was calculated from the silver grade in each concentrate and the estimated dry concentrate weights produced. Current assumption on the gold gravity concentrate is that only silver and gold will be paid for by the gold roaster or autoclave. If the gold concentrate is instead shipped to a copper smelter, copper would be paid for, also, based on the terms of the smelting contract.

Laboratory flotation times for each copper and zinc flotation stage shown on Table 16-2 were used to estimate the Outotec tank flotation cell size. These times were factored up by a factor of two times and then factored again with a 15% air volume factor.

Table 16-8: Flotation Design Criteria

Type of Flotation	**Lab Float Time**	**Plant Float Time, min.**	**Pulp Flow Rate, ft^3/min.**	**ft^3 Required**	**Cell Size, ft^3**	**No. of Cells**
Cu Rougher	15.5	36	80	2,909	565	5
Cu 1st Cleaner	13	31	20	623	177	4
Cu 1st Cleaner Scavenger	6	14	11	152	177	1
Cu 2nd Cleaner	10	24	12	281	106	3
Zn Rougher	18	42	75	3,193	565	6
Zn 1st Cleaner	12	28	35	990	565	2
Zn 1st Cleaner Scavenger	6	14	33	463	177	3
Zn 2nd Cleaner	8	19	23	434	106	4

Table 16-9: Gravity Gold Concentrate Assay

	% Zn	% Cu	Oz/t Au	Oz/t Ag	% Fe	% S	% Insol
Gravity Au Concentrate Grade	3.4	3.3	1.1	4.4	41.2	46.9	3.2

Table 16-10: History of Flotation Tests

Test Description	**Purpose**	**Performed By**	**Date**	**# of Tests**	**Results**
Selective copper flotation followed by reactivation of the zinc tails and flotation of the zinc	Scoping Level Met Data	Dawson Met Labs	1981	3	Copper recovery on MLZ 65% at Cu grade 14%, 38% Cu recovery on MUZ at 9% Cu grade.
Selective copper flotation followed by reactivation of the zinc tails and flotation of the zinc	Scoping Level Met Data	Dawson Met Labs	1982	3	Two open cycle tests and one locked cycle test on combined MUZ/MLZ sample. Results of locked cycle were 79% Cu recovery at grade of 22% Cu, 73% Zn recovery at Zn grade of 45% and 60% Au recovery (bulk to copper con).
Selective copper flotation followed by reactivation of the zinc tails and flotation of the zinc	Scoping Level Met Data	Lakefield Research	1983	1	One open cycle test. Results 54% copper Cu recovery projected to 80% and Au 40% if closed circuit recycle factored, Copper grade was 25% and Zn grade 3.1%. No Zn con produced.
Whole Ore Roasting & Pressure Oxidation Leaching	Prefeasibility Level Testing	United States Bureau of Mines	1987-1988	2	Roasting followed by atmospheric leaching in 50 g/l sulfuric acid leached 71% of Co, 91% of the Cu and 88% of the Zn. 100% of the gold was leached with cyanide after 72 hr. Acidic pressure oxidation leached 94% of Co, 99% of the Cu, and 98% of the Zn, cyanide leached 88% of Au and Ag.
Selective copper flotation followed by reactivation of the zinc tails and flotation of the zinc	Prefeasibility Level Testing	Lakefield Research	1990	28	Open cycle tests. Best results on copper were 90% recovery at the rougher stage and 28.2% Cu con grade with 4.8% Zn. Best results on zinc were 88% recovery at rougher stage and a con grade of 53% Zn with 2.7% Cu. Gold recovery to cons at the best Cu and Zn con grades was 57%.

Additional testing on Upper and Lower Ore is recommended because of the uncertainty regarding the representativeness of past flotation samples. The samples should be selected by ore horizons to investigate potential variation in flotation response with depth, which is clearly indicated by the Lakefield Research work of 1990. Lower Zone Ore has clearly performed better throughout the historical testing. Other than the Lakefield work, definition of where the Upper Zone ended and the Lower Zone began was not clearly defined in the sample composites. In the case of the Lakefield work of

1990, 300 feet of core was combined into one sample to represent the Upper Zone. Thus, the poorly performing ore at one elevation could have affected much better performing ore at another level. The UHZ portion of the Upper Zone is naturally oxidized and may have had a negative impact on flotation test results.

Some if not all of the pyrite/marcasite ore has been observed to oxidize rapidly. This effect was readily seen in old core samples. To counter this effect and more closely simulate actual mining and milling operations where the time between mining and milling is often less than 24 hours, freezing or vacuum sealing of future coarse ore rejects selected for met testing should be performed as soon as they are produced at the assay lab and these samples should remain in this condition until execution of laboratory grinding and flotation tests begin on each sample.

Investigation of bulk flotation as well as selective flotation should be investigated. Bulk flotation may decrease primary grinding equipment capital and operating. Bulk flotation concentrate would then be reground to liberate copper from zinc and then followed with selective flotation. Regrinding of copper rougher concentrate and zinc rougher concentrate would follow before cleaning.

Thorough study of regrind product size is required to optimize metal recoveries and production of salable concentrate grades. Work at Lakefield Research in 1990 suggested K_{80} values of less than 20 microns were needed. It is quite possible to rapidly evaluate particle sizes less than 10 microns with modern particle size equipment in the laboratory and commercially. Rougher concentrate regrind and probably regrind of some cleaner tailing are the key to producing optimal recoveries and concentrate grades.

Evaluate centrifugal gravity recovery of gold in a pyrite concentrate. This method has proven successful at other concentrators around the world and at specific concentrators treating massive sulfide ores (see references). Proving tests should be performed on the feed ore, copper rougher tailing and the zinc rougher tailing.

Evaluation of bulk concentrate processing by hydrometallurgical methods is recommended as a means of increasing base and precious metal recoveries should Knelson type gravity concentrator recovery of gold produce poor results.

Hydrometallurgical treatment of copper concentrates to remove zinc so they are salable to copper smelters may need to be evaluated in the event of encountering difficulties separating zinc from copper concentrate.

16.3 Analytical Procedures for Process Testing

Records in this regard were limited. Dawson Metallurgical Labs provided Certificates of Analysis from two different labs for their studies in 1981 and 1982. The labs were:

Western Analytical Inc.
2417 South 2700 West
Salt Lake City, Utah 84119
(801) 973-9238

Union Assay Office, Inc.
P.O. Box 1528
Salt Lake City, Utah 84110
(801) 363-3302

17.0 MINERAL RESOURCE ESTIMATE

The mineral resources estimate for the Turner Gold project was developed based on a computer generated block model of the deposit. The block model utilized the historic drill hole data and geologic information that was obtained from the project archives. The mineral resource presented later in this section meets the criteria for reasonable expectation of economic extraction in that the stated material is contained within potentially minable shapes based on reasonable economic cutoff grades.

The steps that were used to generate the model and mineral resource statement are summarized below:

Data Base

1) The data base was assembled from historic drill logs and geologic cross sections that were provided by JMC from the project archives.
2) All available drill hole data was plotted on several sets of cross sections at various orientations through the deposit.
3) Many drill holes have long intervals that were not assayed. IMC made a judgment regarding the unassayed intervals to establish them as: 1) zero grade or, 2) no-assay intervals.
4) Drill hole data was composited to nominal 10ft down hole lengths prior to block grade estimation.

Model Assembly

5) Statistical populations were evaluated relative to mapped and interpreted structures.
6) Block grade assignments were established using conventional statistical methods bounded by grade and structural boundaries.
7) Rock density was assigned based on recent test work requested by IMC.
8) Classification codes established.

Mineral Resource

1) Mining, processing, smelting and refining costs were estimated based on knowledge of the project and recent costs from other projects.
2) Mining and process recoveries were applied based on the mine plan and process plant design and testing.
3) A potential economic cutoff grade was established to guide stope layout.
4) Material contained within approximate minable (stope) boundaries was tabulated to reflect potential resources.

The model and mineral resource estimates will be summarized in the following sub-sections.

17.1 Data Base

The drill hole data base was assembled from the historic paper drill hole logs and assay certificates that were found within the project archives. Section 14 on Data Verification summarized the process used by IMC to enter and verify the data entry.

Geologic and structural information was extracted from historic cross sections developed for Rayrock Mines Inc. by the qualified author Mike Strickler. This information was stored in a data base by IMC.

The paper logs and the resulting drill hole data base contains long runs of drill intervals without assay. There are 25 of the 84 drill holes that do not have any assay information of any kind. IMC reviewed each hole and each interval without assay to establish a method of treatment on a drill hole by drill hole basis.

The following outline summarizes the amount of available drilling and the amount of assay data available for estimation of model grades.

Summary of Available Turner Gold Project Drill Hole Data

Total Holes Referenced
- 84 holes
- 4871 intervals
- 64,129 ft drilling

- Holes with Survey and Assay >0 (Drill holes Found)
  - 57 holes
  - 4511 intervals
  - 51,877 ft drilling

- Holes with Survey and at Least One Assay > 0.10 EqAu = $53 NSR
  - 42 holes
  - 3795 intervals
  - 41,286 ft drilling

- Holes with Survey and Assay > 0.10 EqAu = $53 NSR that are contained in the Modeled Mineralized Zones
  - 41 holes in Ore
  - 641 Assays
  - 3,080 ft of Drilling

The last illustration at the $53/ton NSR cutoff was not a sort applied to the data prior to model assembly. It is provided as an indication of the amount of ore intercept assay that is available to estimate block grades and to illustrate the component of the drilling that was actually assayed.

Geologic information within the old drill logs was difficult to read and interpret. The geologic recording practices that were applied predated many of the techniques that have become common with the application of computer based data bases. The rock type descriptions within the logs were lengthy discussions of minerals present, rock fabric, alteration, and texture. In many cases, the rock type was logged as the ore type “Massive Sulfide” rather than as the protolith. This is likely because the sulfide alteration obliterated the original protolith.

Fortunately, a series of east-west cross sections through the drill hole data were developed by Rayrock Mines Inc. during 1984. One of the primary geologists involved with the development of these sections was Mike Strickler, one of the fellow qualified person’s contributing to this report.

The drill hole sections indicated the rock type, visual percentage of sulfides, and structural indications of shearing or faulting on the drill hole trace. IMC staff measured the rock type boundaries and structure codes from the drill hole traces with a scale and stored that information within a data base. The rock type representations on these sections were the best consistent set of data that IMC was able to find during the archive search.

The drawback to the IMC section measurements is that the apparent depth on section can differ from the true depth when drill holes are not precisely on section. Consequently, one must understand that the rock type coding within the data base is approximate and does reflect the specific coding of individual assay intervals. The coding is however accurate to a few feet within any given drill hole.

The structural information was also entered into the data base to provide an approximate basis for geotechnical judgments regarding mining method for the PEA. IMC found nine drill holes with RQD information within the paper archives. All had been drilled by Noranda. That information was entered into another data base so that typical or average RQD values by rock type or structure coding could be developed from this data.

<u>Intervals without Assay</u>

As noted above there were 25 holes where assay information was not found. Some were old holes such as the Churn holes. Others may have been used for process testing rather than assay. In many cases, IMC was simply not able to find the assay information within the paper files.

In addition, there are large segments of the assayed drill holes that were not assayed. IMC generated a listing of these holes and then located them on the Rayrock sections or on plotted overlays to the Rayrock sections.

A copy of the assay data was stored in a second assay field in the IMC data base. The original data from the drill logs were coded with a flag or code for “No Assay”. IMC then made the judgment that many of those intervals should be considered as zero valued assay. The second copy of the data was changed to a value of 0.0 to reflect the barren rock type or zone. There was no modification to the original data field in the data base.

In many cases, IMC assumed that long intervals in a drill hole with no assay were likely based on the logging geologist's opinion that there were no sulfides and consequently, no assay values. In those cases, IMC changed the working field to zero (0.0).

Other drill holes without assay, particularly some of the early holes in the upper ore zone appeared on the Rayrock sections with notes that they contained observed sulfides. In these cases, IMC left the drill hole coded as "No Assay". Blocks in these areas would be estimated using surrounding holes.

The selection assignment of "No Assay" versus "Zero Assay" was based on the judgment of the ore reserves Qualified Person. Since this is a judgment call, there could be alternative interpretations. Since the determination is open to interpretation, there is further support to the lack of measured category mineralization at Josephine at this time.

Data Base Composites

Once the "zero" versus "no assay" decision was made, IMC calculated down hole composites of 10 ft length. The length of the composite was selected to match the block size that was in turn guided by the potential mining methods and drill hole spacing.

Within the composite process, composites that were less than 5ft long were coded as "no assay". The compositing was applied to the data copies that were added by IMC to incorporate the zero versus no assay decision. For reference these variables were coded with names like: au_use, cu_use, ag_use, zn_use, etc.

Further statistical analysis of the project utilized the drill hole composites.

17.2 Block Model Assembly

Geologic and Structural Interpretation

The rock types information from the Rayrock cross sections was assigned to the 10 ft drill hole composites in order to understand the grade distribution by rock type. Figure 17-1 summarizes the basic statistics for equivalent gold, gold, and copper by rock type. The equivalent gold calculation on the table is intended to summarize the combined value of copper, zinc, silver, and gold, into the value of equivalent gold. The equation used for equivalent gold is shown at the bottom of the figure and is a preliminary calculation intended to understand the distribution of values within the deposit.

Figure 17-1 indicates that the majority of the ore is contained within the following host units: BFR = Basin Floor Rubble, DBF = Debris Flow, Sulf = Massive Sulfide. Other units are generally low grade or barren.

Given more drill hole data with reliable survey and precise rock type coding, future model construction should endeavor to develop three dimensional rock type geometries for assignment to the block model. The complexity of the rock boundaries, the spacing between drilling, and the uncertainties in the data base did warrant the detailed effort for three dimensional interpretation at this time.

Observation of the cross sections often showed abrupt grade changes at the top of the mineralization and somewhat more disseminated grade distributions at the bottom of the deposit. Therefore, a requirement for the model was to reflect those abrupt and disseminated grade boundaries where they exist.

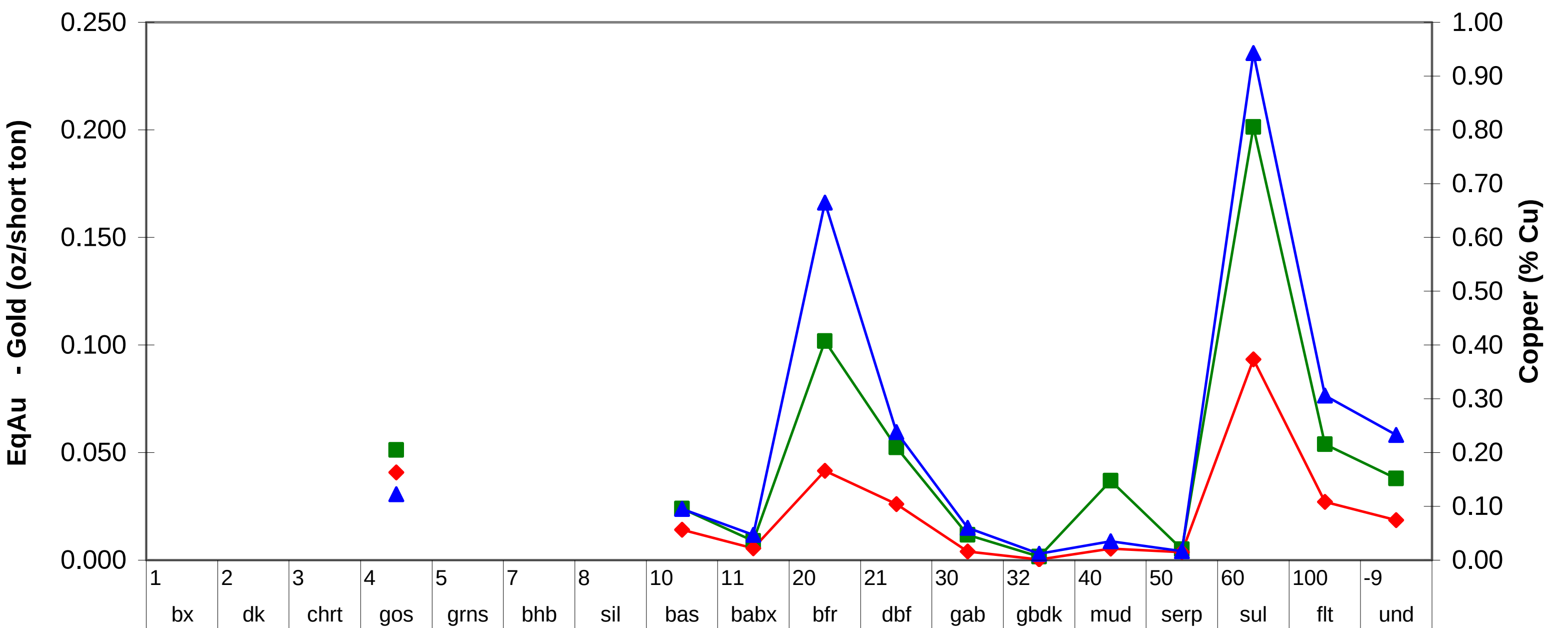
Figure 17-2 summarizes a cumulative frequency plot of the calculated equivalent gold using the equation at the bottom of Figure 17-1. The graph indicates a change in grade that centers on about 0.04 oz/ton equivalent gold. There is also the indication of a high grade population above about 0.5 oz/ton equivalent gold in the upper zone.

The upper and lower zones of the deposit will be summarized on the following sub-section.

Studies of cross sections also indicated that a value of about 0.04 oz/ton was within the range of distinct boundaries between barren assay intervals and well mineralized ore intervals.

The significance of the 0.04 oz/t equivalent grade brake was utilized to develop hard boundaries between mineralized and un-mineralized rock. The discussion of the procedure follows later in this section.

Figure 17-1, Turner Albright 10 Foot Down Hole Composites



	1 bx	2 dk	3 chrt	4 gos	5 grns	7 bhb	8 sil	10 bas	11 babx	20 bfr	21 dbf	30 gab	32 gbdk	40 mud	50 serp	60 sul	100 flt	-9 und
EqAu				0.051				0.024	0.009	0.102	0.052	0.012	0.002	0.037	0.005	0.201	0.054	0.038
Gold				0.041				0.014	0.005	0.042	0.026	0.004	0.000	0.005	0.004	0.093	0.027	0.019
Copper				0.122				0.095	0.047	0.665	0.238	0.060	0.012	0.035	0.017	0.943	0.306	0.233
Number	0	0	0	7	0	0	0	397	85	346	165	92	12	12	8	203	17	581

Rayrock Rock Codes

EqAu = Au+ (Cu x 0.0623) + (Ag x 0.0052) + (Zn x 0.0161)

krige/CompStats.xls

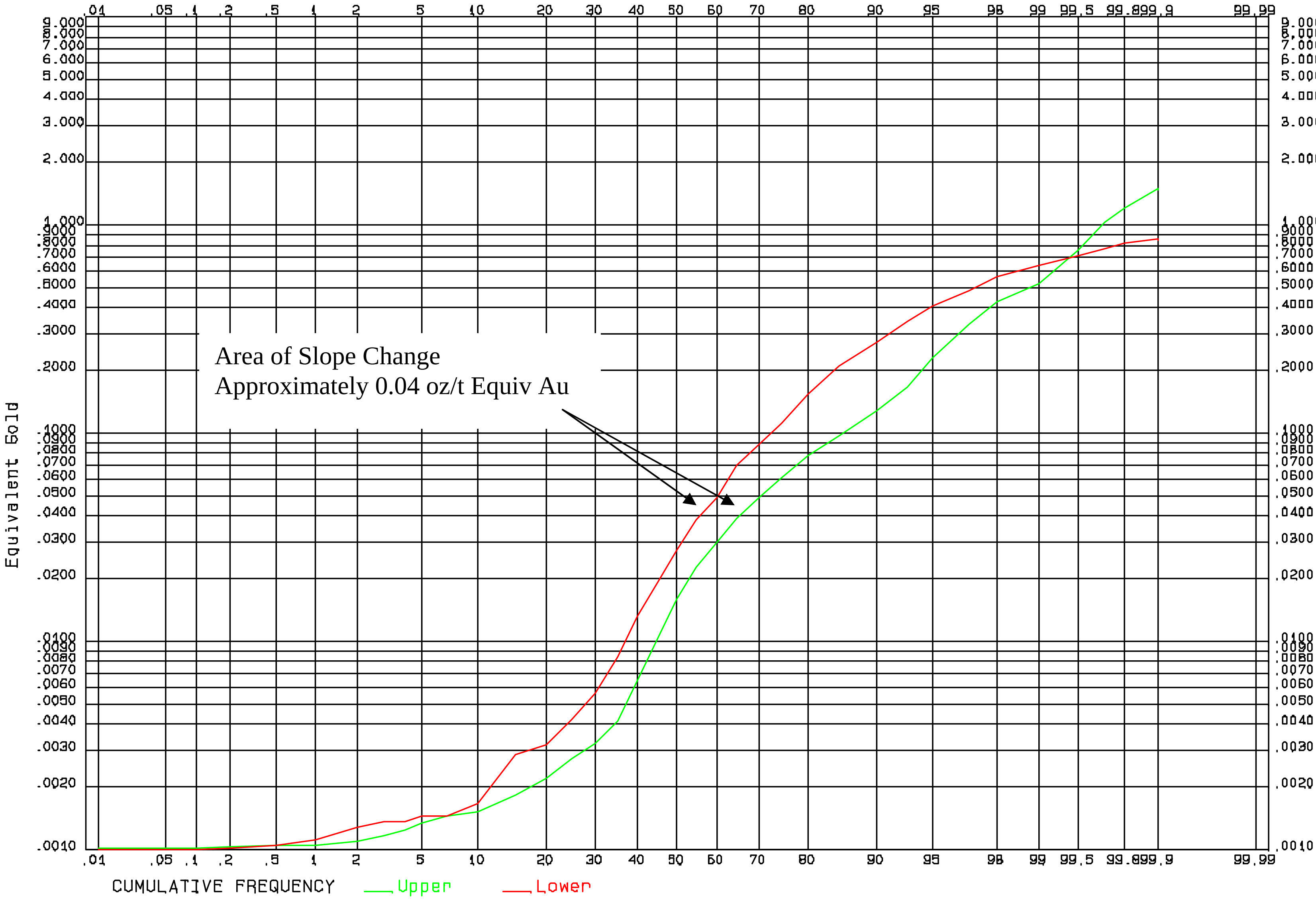


Figure 17-2
Cumulative Frequency Plot
Equivalent Gold, Upper and Lower Zones

Previous work on the Turner Gold Project has identified a number of fault structures that cross the deposit. The predominate faults have been referenced as the “F” series and the “R” series faults. The F faults are interpreted to strike northwest (N60W) and dip steeply at 65 to 85 degrees to the northeast. The historic interpretations result in five “F” faults numbered 1 to 5 from south to north.

The R Faults are interpreted to strike roughly east-west with shallow northerly dip of about 20 degrees. Historic work has interpreted from 1 to 3 faults of the R series numbered from 1 to 3 from the top down.

The Rayrock work by Strickler potentially interprets that the deposit has been separated in to UHZ and MUZ versus the MLZ by post-mineral displacement of the R-1 fault. The Rayrock data included a surface geology map that located the named F and R series faults on topography. IMC combined that data along with the structural coding on the east – west Rayrock sections to interpret a set of F and R faults in 3 dimensions. The IMC interpretations simplified each of the faults to simple planes that were a best fit to the drill data and surface intercepts of each fault.

IMC generated block assignments of four F series faults and three R series faults. The 10 ft drill composite data was assigned the code for the F and R fault block that contained each composite.

Statistical analysis of the fault boundaries were completed by comparing composite grades on opposite sides of each of the 20 resulting fault blocks. This analysis indicated that the IMC interpreted F faults were not boundaries to mineralization. The R1 fault could potentially be a boundary, but the statistical analysis was not clear indicating that there was related grade mineralization on opposite sides of the R fault.

The composite data was next color coded and studied using software that allowed IMC to rotate and review the data as a cloud in three dimensional space. That effort provided a strong indication that the break between the upper and lower deposits was along an orientation of 310 degrees strike (northwest) with a dip of 35 degrees to the north east.

Although roughly parallel in strike to the F faults, the break between the deposits has a dip that is substantially more shallow that the F fault interpretations.

For convenience, IMC has named this break in the deposit as the J Fault. However, there is no immediate evidence that this deposit break is a fault. It may reflect two rubble zones that have been mineralized independently, or it could reflect a structural offset of a single deposit.

Figure 17-3 is a three dimensional presentation of the 10 ft composite data. The two colors of yellow and red reflect grades of 0.04 and 0.10 oz/t equivalent gold. Figure 17.3 is about 5 degrees off of the dip direction and about 5 degrees off of the strike of the two deposits. The plot looks northeast and illustrates the break between the upper and lower zones.

Figure 17-4 is also a 3 dimensional plot looking horizontally along the strike of the break on a bearing of 310 degrees. This is not a single cross section but a look through all of the data in the strike orientation. The upper and lower zone break can be clearly seen on the figure.

As a result of the new boundary interpretation, IMC combined the geologic components of the deposit into simplified zones for similar statistical treatment.

> The UHZ and the MUZ have been combined into an "Upper Zone"
> The MLZ is referred in the statistical analysis as the "Lower Zone"

The above terminology is not necessarily inconsistent with previous naming conventions, but reflects that block model statistical treatment that follows.

The block model and composite data was assigned a code to indicate location relative to the deposit break (J Fault). Blocks and composites above the boundary received codes of 100. Blocks and composites below the boundary received codes of 200.

This break provided the primary basis for geostatistics and orebody zoning to follow.

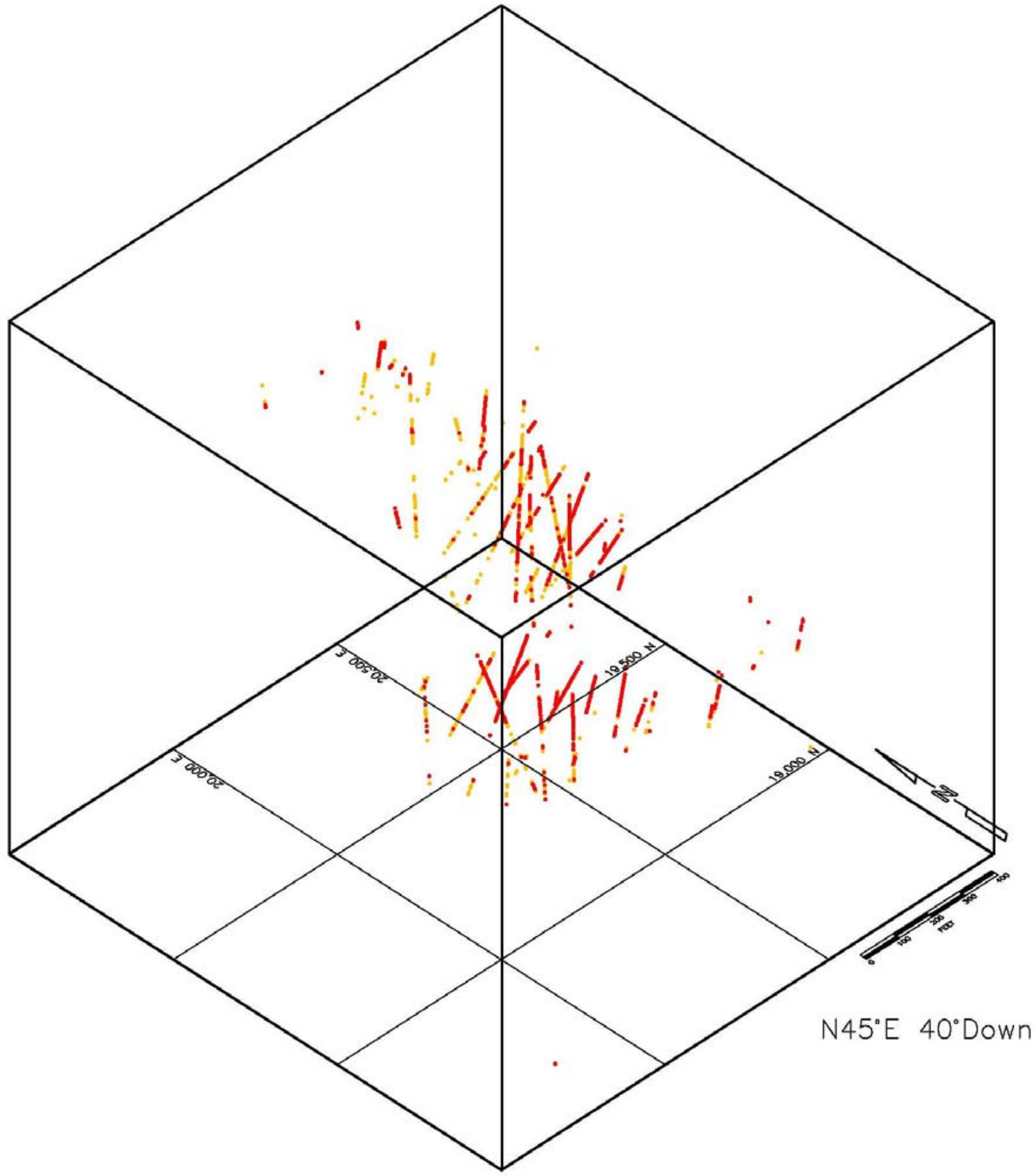


Figure 17-3

10 ft Drill Composites, Orange = 0.04, Red = 0.10 oz/ton Equiv Gold

Look 45 Degree Bearing, Look Down 40 Degrees
Nearly Down Dip
Indicates Break Between Upper and Lower Deposits

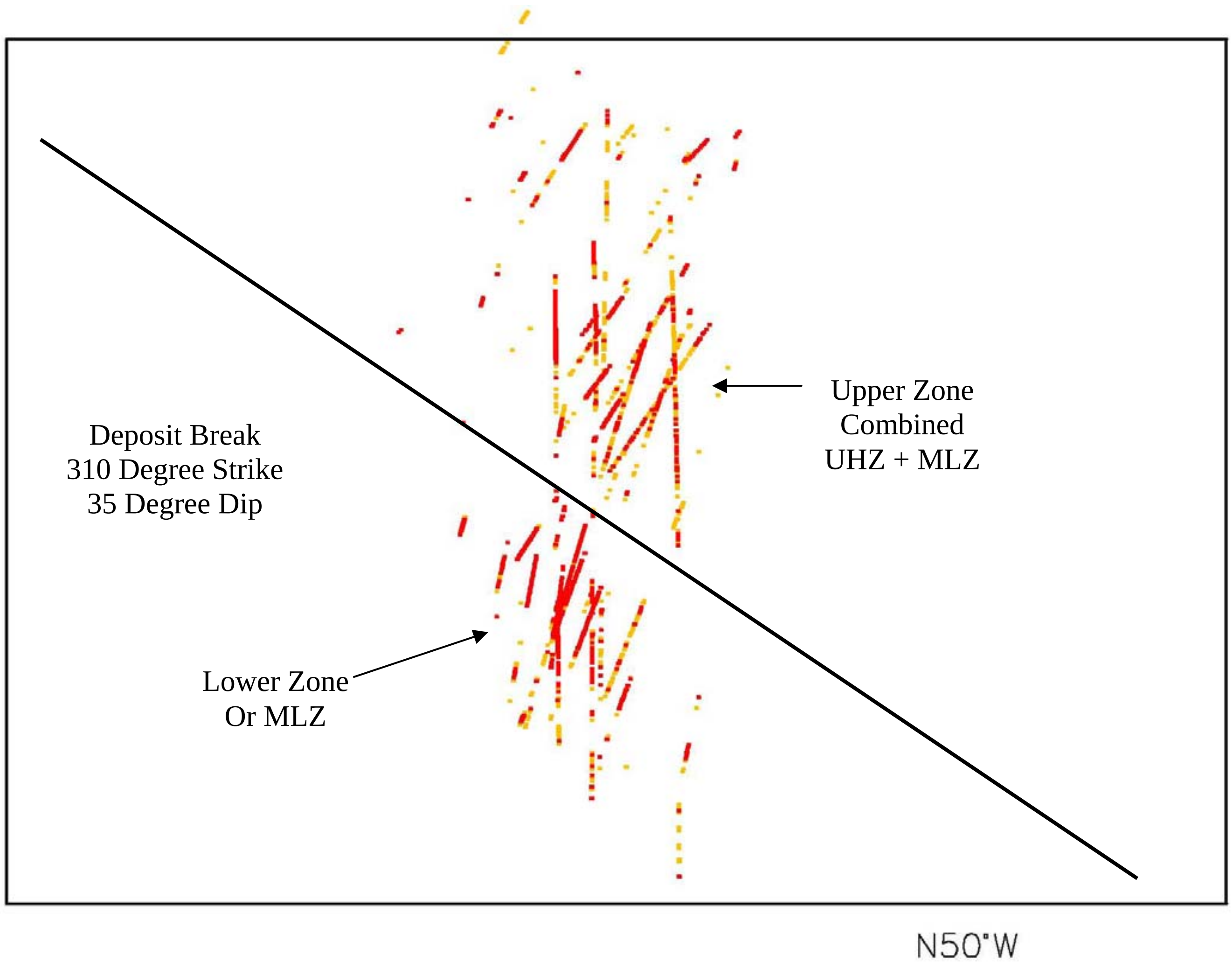


Figure 17-4

10 ft Drill Composites, Orange = 0.04, Red = 0.10 oz/ton Equiv Gold

Look Toward Bearing of 310 Degrees,
Flat Plunge, Along Strike

Indicates Break Between Upper and Lower Zones of the Deposit
1500 ft Horizontal Axis

Deposit Statistics and Variography

The so called J Fault boundary that is illustrated in the previous cross section was used to separate the deposit into upper and lower divisions. This is consistent with much of the previous work. The only change is the orientation of the boundary.

The 0.04 oz/ton equivalent gold cutoff was utilized to further segregate the deposit. That grade boundary was used to separate the mineralized zones from the surrounding barren material. In addition to the change in statistical population that is inferred from Figure 17-2 and the observed abrupt changes in grade, there was a further practical reason for applying a grade boundary to the deposit.

Mine planning cutoffs were expected to be in the range of about $50.00 NSR/ton which is around 0.10 oz/ton equivalent gold. A grade limitation within the model that is somewhat lower than mining cutoffs limits the amount of tonnage over estimation that can occur when conventional unbounded grade estimation techniques are applied.

The assignment of the 0.04 oz/ton equivalent grade boundary to the model will be discussed in the next few pages. Once that coding was available, the composites contained in each zone were coded with the grade zone and structural (upper and lower) zone of the deposit.

Table 17-1 is a summary of the basic statistics of the 10 ft composites for each of the economic metals in the deposit.

Table 17-1

Composite Statistics, Josephine Mining Project
All Assayed Composites

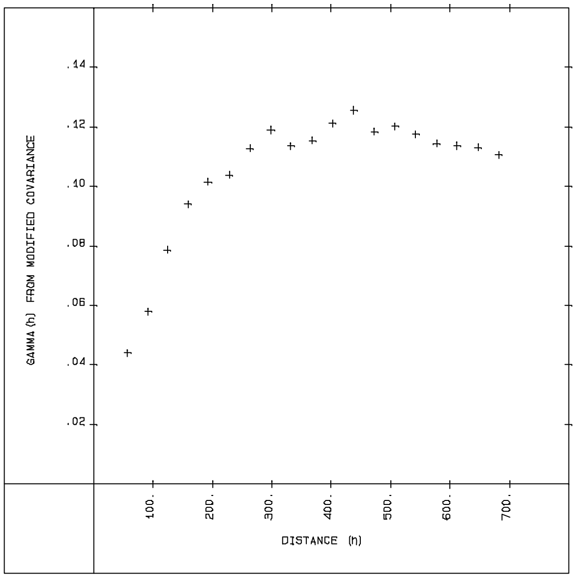
Structural Sone	Grade Zone	Number	Gold, oz/ton			Silver, oz/ton			Copper %			Zinc, %		
			Mean	Std Dev	Max	Mean	Std Dev	Max	Mean	Std Dev	Max	Mean	Std Dev	Max
Upper Zone	Mineralized	380	0.076	0.092	0.815	0.225	0.437	3.26	0.744	1.524	14.50	1.33	2.048	12.33
	Low Grade	591	0.006	0.008	0.052	0.079	0.101	1.04	0.045	0.066	0.38	0.10	0.170	1.39
Lower Zone	Mineralized	238	0.070	0.056	0.392	0.415	0.784	4.40	1.148	1.562	11.84	2.87	5.231	30.15
	Low Grade	250	0.006	0.007	0.035	0.059	0.070	0.37	0.044	0.067	0.52	0.09	0.167	1.36

Variography was completed in two stages on the deposit: 1) indicators based on 0.04 oz/t equivalent gold, and 2) grade variograms within the defined indicator mineral zones.

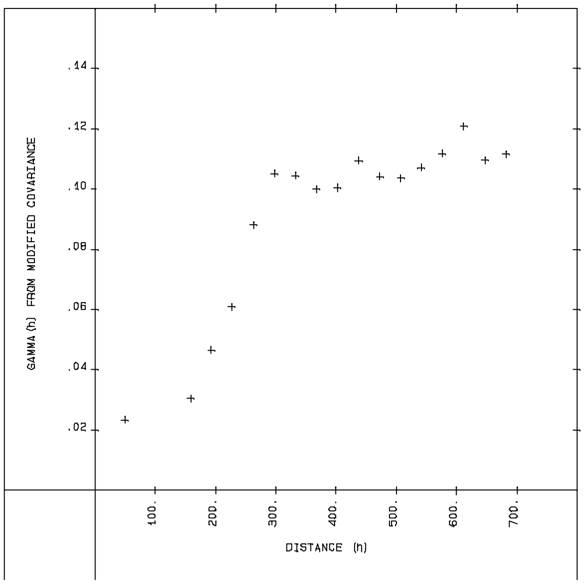
Indicators were used to understand the continuity of the mineralized zones. In this case, the indicators are values of 0 and 1 that represent composites less than 0.04 equivalent (0) versus those greater than 0.04 oz/ton equivalent (1).

Figures 17-5 and 17-6 illustrate indicator variograms for the upper and lower zones of the deposit.

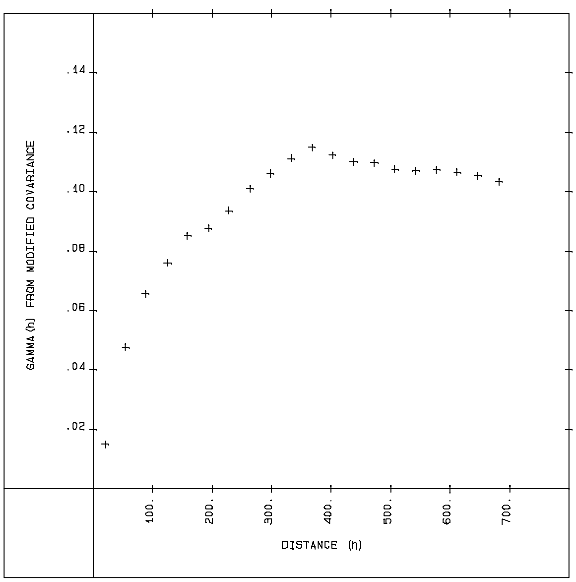
Figure 17-5
Indicator Variograms
Gamm(h) from Modified Covariance
Equivalent Gold at 0.04 oz/t Discriminator
Upper Zone



130 Bearing,
0 Plunge

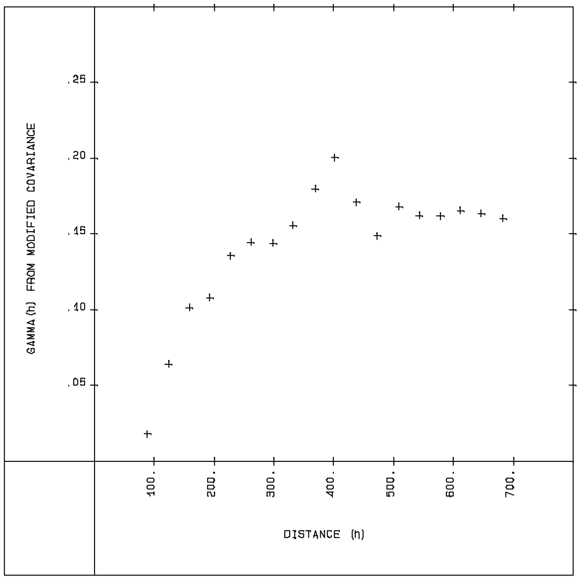


40 Bearing
35 Plunge

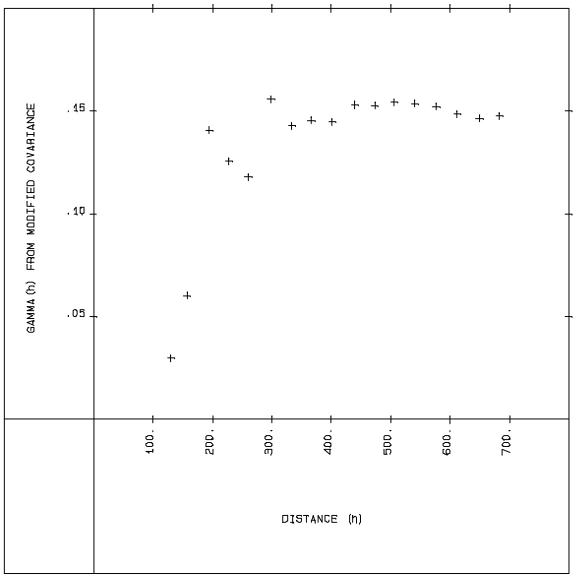


220 Bearing
55 Plunge

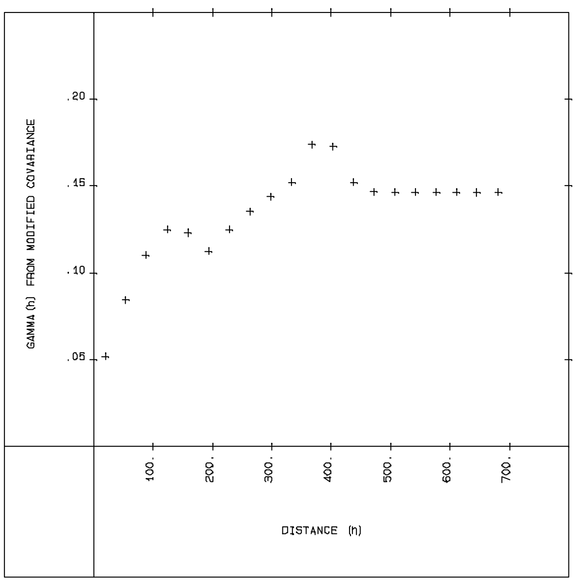
Figure 17-6
Indicator Variograms
Gamm(h) from Modified Covariance
Equivalent Gold at 0.04 oz/t Discriminator
Lower Zone



115 Bearing
10 Plunge



18 Bearing
33 Plunge



259 Bearing
77 Plunge

Indicator kriging was used to assign a code to every block in the model that indicated it had better than a 50% probability of exceeding the 0.04 oz/ton equivalent discriminator. The 0 and 1 values determined by the 0.04 oz/ton equivalent gold discriminator are used as input to ordinary linear kriging. The resulting value for each block can be interpreted to represent the probability that the entire block has grade above the discriminator value.

IMC contoured the kriging results at the 0.50 (50:50 probability) level. Blocks above 0.50 were assigned a code of 1, the remaining blocks retained a code of 0.0. The 1.0 coded blocks provided an indication of the blocks with potentially interesting grade.

Table 17-2 summarizes the selected kriging parameters.

Table 17-2
Variogram and Kriging Parameters

Orientation, Degrees			Variogram Ranges ft			Variogram Model		Search Distances ft		
Bearing	Plunge	Rotation	Major	Semi	Minor	Nugget	Total Sill	Major	Semi	Minor
Upper Zone										
130	0	35 Dn North	150	150	75	0.05	1.00	150	150	75
Lower Zone										
115	10 Dn SE	33 Dn North	150	125	75	0.05	1.00	150	125	75

Indicator Kriging Composite Requirements: Mininum = 2, Maximum = 10, Maximum per Hole = 3
Grade Kriging Composite Requirements: Mininum = 2, Maximum = 10, Maximum per Hole = 3

tab17-2.xls

Once the block codes were assigned based on the 0.04 oz/ton equivalent discriminator, block grades for the individual metals were assigned inside of those defined grade contour zones.

The grades were assigned by ordinary linear kriging inside of the 0.04 grade contours. The same variogram parameters were used for grade estimation as summarized on Table 17-2. Each metal was estimated separately: gold, copper, silver, zinc, and cobalt. Once assigned to the blocks, cross sections and level maps were plotted to check the outcome. The figures presented in Section 9.0 regarding mineralization are examples of the drill hole to block grade comparisons. A result of the indicator process was the development of abrupt grade boundaries within the model just as observed in the drill hole data.

<u>Density Assignment</u>

A default density of 9.474 cubic feet to the ton was assigned to every block in the model. Thirty eight (38) density tests were completed as part of the data verification process that was summarized in Section 14.0. Those samples were targeted at ore grade zones and covered the range of rock types, and elevations within the deposit. The average specific gravity of all 38 samples was 3.478 (9.23 cu ft/ton). This result is a 2.5% reduction in density from the average values previously used by R.L. Russell in 1988.

IMC further reduced the test results by another 2.5% to reflect the voids that were observed during the site visit within the Basin Fill Rubble, and Debris Flow Rock types. The resulting bulk density is 9.474 cubic feet per ton or 211 lbs per cubic foot.

Classification Codes

Due to the uncertainties in the data base that were presented earlier, IMC has made the judgment that there is no measured category mineralization at the Turner Gold deposit. Additional data will be required to confirm the historic information in order to consider the assignment of measured category in the future.

The following criteria was used to assign the codes of inferred, and indicated to the Turner Gold model. The grade kriging for copper was used as the basis for classification although any of the metals could be used since there are identical numbers of composites for each metal inside the 0.04 discriminator zones.

If the block was inside of the 0.04 oz/ton discriminator zone and,
Copper Grade was assigned: Then Code = 3 = Inferred

If the block was inside of the 0.04 oz/ton discriminator zone and,
Copper Grade was assigned and,
Kriged Standard deviation < 0.90 and,
Number of composites = 9 or 10 (3 holes) Then Code = 2 = Indicated

17.3 Mineral Resource

The mineral resource was developed based on the block model and highly preliminary estimates of mining and processing costs in order to establish the component of the mineralization that has reasonable expectation for economic extraction.

Each block in the model was assigned a Net Smelter Return (NSR) value based on estimated metal prices, process recoveries and smelter terms. An NSR cutoff for resource was then developed based on the estimated mining and process costs. Since the Turner Gold deposit is a polymetallic deposit, the treatment of both copper and zinc concentrates is a significant component of the project operating costs.

IMC utilized the concentrate grades and process recoveries that were developed within the R.L. Russell Feasibility Study of 1988 as the initial guide to estimating concentrate treatment costs. Recent smelter terms from the IMC files were then applied to the concentrate grade information as provided within the Russell report. It should be noted that alternative cost structures were developed within the PEA discussion to following in Section 18.0. However, the information in this section was used to guide the development of the mineral resource statement.

Table 17-3 presents the information used by IMC to establish the NSR value for each block in the model. The calculation of NSR as well as the equivalent gold or equivalent copper grade that would result from these estimates is also included. The calculations of equivalent and NSR on Table 17-3 differ from the initial gold equivalent calculation that was used for model assembly because there was more knowledge available for the planning values than was available for the model assembly.

The estimated cutoff grade is also shown on Table 17-3. It includes the estimated stope mining cost, milling cost and G&A for comparison against the calculated NSR values within each block.

Metal prices have been estimated by IMC at $2.00/lb copper, $900/troy ounce gold, $12.50/troy ounce silver, and $0.65/lb zinc. These prices generally reflect the end of April 2009 spot prices for the quoted metals. All are less than the spot prices during the time this report was being written in October 2009. All but the gold price are less than the 3 year backward average. The $900 gold price is a good approximation to the 60% historic and 40% future average as of October 2009.

As a result of the calculation on Table 17-3, a cutoff grade of $42/ton NSR was applied to the calculation of mineral resources for the Turner Gold deposit.

Table 17-3
Turner Gold Deposit
Cutoff Grade and NSR Calculations

Metal Prices Based on April 2009 Spots						
Metals	Metal		Avg TCRC with RLR Conc and IMC Costs			
	Price	unit	Cost	unit	Mill Recov	Smelt Recov
Copper	$2.00	/lb	$0.367	/lb	79.0%	95.2%
Gold	$900.00	/ troy oz	$2.00	/ troy oz	61.0%	97.0%
Silver	$12.50	/ troy oz	$0.15	/ troy oz	28.6%	77.0%
Zinc	$0.65	/lb	$0.225	/lb	62.8%	95.0%

Equivalent Multipliers for Each Metal

	Copper %	Gold oz/t	Silver oz/t	Zinc %
NSR=	$24.569	$531.347	$2.720	$5.069
Cu Eq	1.0000	21.6266	0.1107	0.2063
Gold Eq	0.0462	1.0000	0.0051	0.0095

NSR = Copper x $24.569 + Gold x $531.347 + Silver x $2.720 +Zinc x $5.069

NSR Cutoffs

	$28.72	Mining	From S. Annavarapu
	$6.42	Milling	From J. Moore
Internal Cutoff	$35.14	/ton	
	$6.70	G&A = $2,000,000/year at 299 kt/yr	
Breakeven Cutoff	**$41.84**	**NSR Cutoff**	
	0.079	EqAu Cutoff	

TCRC Support Notes

Assume Ore Head Grades from RLR Report, 1988		
Copper	1.52	%
Zinc	3.78	%
Copper Concentrate		
Copper Concentrate Grade	21%	From RLR
Zinc Recovery to Copper Con	7.6%	From RLR
Copper Smelting Charges		
Per Ton of Concentrate	$72.73	/ton concentrate
Per Lb of Recovered Copper	$0.080	/lb Recovered Copper
Concentrate Freight	$36.36	/ton concentrate
Zinc Grade in Copper Con Based on RLR Head Grades	5.02	%
Zinc Penalty in Copper Con		
For each 1% over 2%	$1.82	/ton concentrate
Refining Gold	$2.00	/troy ounce
Refining Silver	$0.15	/troy ounce
Copper TCRC per Saleable Pound	**$0.367**	/lb Salebale Copper
Zinc Concentrate		
Zinc Grade of Zinc Concentrate	51%	From RLR
Copper Recovery to Zinc Con	3.0%	No Penalty
Gold Recovery to Zinc Con	10.5%	Not Payable
Silver Recovery to Zinc Con	35.5%	Not Payable
Zinc Smelting Charges		
Per Ton of Concentrate	$181.82	/ton con
Concentrate Freight	$36.36	/ton concentrate
Zinc TCRC per Saleable Pound	**$0.225**	/lb Salebale Zinc

p22201/econ/Equiv_30Sep09

IMC developed a preliminary tabulation of all blocks in the deposit with grade above \$42/ton NSR. However, that tabulation includes isolated blocks that could not be incorporated into a minable stope geometry.

In order to establish continuous geometries of mineralization that could potentially be mined. A requirement was added that each block above cutoff be surrounded by four other blocks that are also above cutoff.

The calculation of neighboring blocks above cutoff was established based on a simple assumption that each block could have a maximum of 6 neighbors (north, south, east, west, above, and below). The number of those that were above the \$42/ton NSR cutoff was then counted.

The judgment of four neighboring blocks above cutoff was established such that the single rows or columns of blocks could not be considered as potentially minable.

The undiluted tabulation from the block model was then utilized as the basis to apply estimated mining recovery and dilution so that the resulting statement of mineral resources does include reasonable approximations of mining recovery and dilution.

Cobalt is reported because it was assayed. However, there has been no economic benefit applied to contained cobalt in the determination of resources or within the PEA.

Table 17-4 summarizes the statement of mineral resources.

Table 17-4
Turner Gold Deposit
Mineral Resources

Mineral Resource at Metal Prices, \$900/oz Gold, \$2.00/lb Copper, \$12.50/oz Silver, \$0.65/lb Zinc														
Category		Cutoff NSR/t	Short Ktons	NSR \$/ton	Gold oz/ton	Copper %	Silver oz/ton	Zinc %	Cobalt %	Contained KOzs Gold	Contained KLbs Cu	Contained KOzs Silver	Contained KLbs Zn
Undiluted Indicated		\$42.00	2,447	92.88	0.090	1.25	0.31	2.65	0.047				
Mining Recovery	90%		2,202	92.88	0.090	1.25	0.31	2.65	0.047				
Mining Dilution	10%		220	42.26	0.049	0.50	0.16	0.79	0.038				
Recov+Diluted Indicated			2,422	88.27	0.086	1.18	0.30	2.48	0.046	209	57,245	718	120,169
Undiluted Inferred		\$42.00	2,084	86.40	0.088	0.99	0.64	2.78	0.036				
Mining Recovery	90%		1,876	86.40	0.088	0.99	0.64	2.78	0.036				
Mining Dilution	10%		188	42.26	0.049	0.50	0.16	0.79	0.038				
Recov+Diluted Inferred			2,064	82.38	0.084	0.94	0.59	2.60	0.036	174	38,991	1,223	107,290

Notes:
Undiluted calculations are from the block model at the \$42.00/ton NSR Cutoff
Undiluted calculations require each block to have 4 neighbors above cutoff grade
Dilution grade based on the grade of material surrounding the undiluted tabulation, at a \$5.00/ton NSR Cutoff

The mineral resources on Table 17-4 were developed by John Marek, P.E., President of Independent Mining Consultants, Inc., a Qualified Person for development of resources on this type of deposit as defined within NI43-101.

John Marek is independent of the issuer as defined in Section 1.4 of NI43-101.

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

A component of the Inferred Mineral Resources has been included within the economic evaluation of the Preliminary Economic Assessment.

The preliminary economic assessment is preliminary in nature and it includes inferred mineral resources that are too speculative geologically to have economic consideration applied to that would enable them to be categorized as mineral reserves and there is no certainty that the preliminary economic assessment will be realized.

18.0 OTHER RELEVANT DATA AND INFORMATION

Within this section, the mine plan, process facility, and infrastructure will be discussed such that was integrated into the Preliminary Economic Assessment (PEA). The estimated capital and operating cost for each of the project areas will be presented followed by a cash flow analysis for the PEA.

Inferred category material was included within the mine plan, process plan, and economic analysis within this PEA. The entire mineral resource as stated in Section 17 was no included within the mine and process plan. However, a significant component (85%) of the combined indicated mineral resource and inferred mineral resource were included in the PEA analysis.

18.1 Mine Plan and Mining Method

18.1.1 Introduction

The total estimated mineable material in the Turner Gold PEA mine plan amounts to 3.586 million diluted short tons at average grades of: 0.089 oz/ton Gold, 1.22% Copper, 0.50 oz/ton Silver, 2.86% Zinc, and 0.038% Cobalt. The cutoff grade used to estimate the minable material was $50.00 /ton Net Smelter Return (NSR). The minable material is a combination of indicated mineral resources and inferred mineral resources. The projected mine life is 8 years at the proposed production rate of 1,250 short tons per day, with 8 months of preproduction underground mine development. Table 18-1 summarizes the mine production schedule for the PEA.

This minable material estimate does not include the material above the 2600 ft level, which is a part of the Upper High-grade Zone (UHZ) referenced in Section 9.0. This material amounts to about 0.42 million tons and was not included in the minable material estimate for the following reasons:

1) The zone is currently interpreted as smaller volumes that would require additional underground development to produce. A judgment was applied that they may not be economic, and
2) Portions of the UHZ are oxidized and would likely not respond well to the flotation process as currently envisioned.

This study describes mining of the two ore zones at Turner Gold, namely the MUZ and MLZ. Since a large portion of the ore zones lies above the proposed location of the processing plant, the main access to the mine will be through three adits as shown in Figure 18.1. From these adits, dedicated spiral footwall ramps have been provided at each orebody for moving men, equipment, and supplies underground. The mine design has been developed to allow flexible access to both the MLZ and MUZ.

Table 18-1

Turner Gold Mine Production Schedule

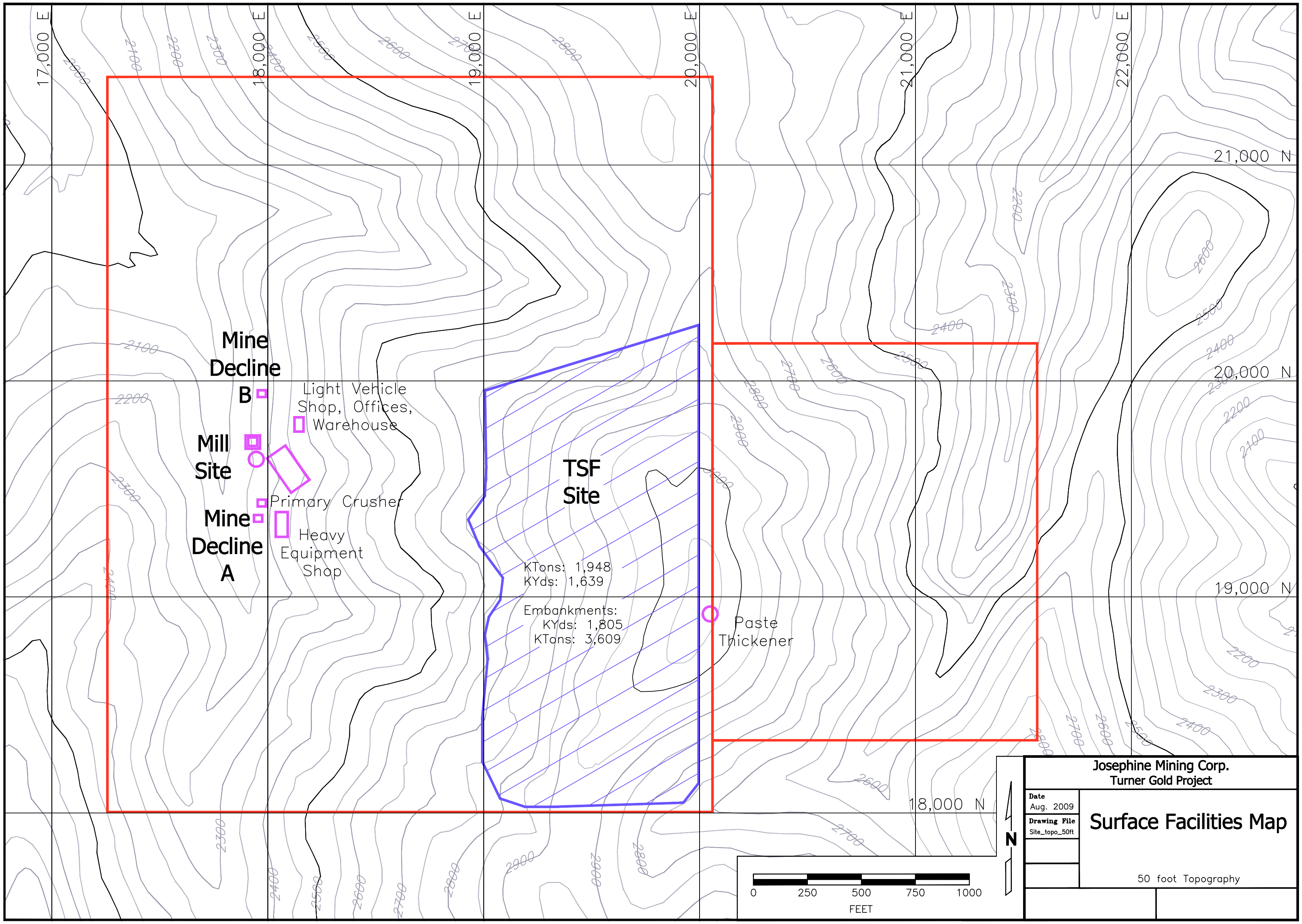
Undiluted Material in Stopes as Reported from the Model									
Year	1	2	3	4	5	6	7	8	Total
Waste (short tons)	265,228	50,024	50,024	50,024	50,024	50,024	50,024	27,697	593,069
Ore (short tons)	421,246	455,000	455,000	455,000	455,000	455,000	455,000	434,512	3,585,758
Gold (oz/ston)	0.096	0.093	0.093	0.093	0.093	0.093	0.093	0.090	0.093
Copper (%)	1.43	1.30	1.30	1.30	1.30	1.30	1.30	1.17	1.30
Silver (oz/ston)	0.56	0.54	0.54	0.54	0.54	0.54	0.54	0.52	0.54
Zinc (%)	2.67	3.09	3.09	3.09	3.09	3.09	3.09	3.49	3.09
Cobalt (%)	0.040	0.038	0.038	0.040	0.038	0.038	0.038	0.036	0.038

Apply 90% Mining Recovery

Apply 10% Dilution at the following Average Dilution Grades

Gold (oz/ston)	0.049
Copper (%)	0.50
Silver (oz/ston)	0.16
Zinc (%)	0.79
Cobalt (%)	0.038

Diluted Minable Material Feed to the Process Facility									
Year	1	2	3	4	5	6	7	8	Total
Ore (short tons)	421,246	455,000	455,000	455,000	455,000	455,000	455,000	434,512	3,585,758
Gold (oz/ston)	0.091	0.089	0.089	0.089	0.089	0.089	0.089	0.086	0.089
Copper (%)	1.34	1.22	1.22	1.22	1.22	1.22	1.22	1.10	1.22
Silver (oz/ston)	0.52	0.50	0.50	0.50	0.50	0.50	0.50	0.48	0.50
Zinc (%)	2.48	2.86	2.86	2.86	2.86	2.86	2.86	3.22	2.86
Cobalt (%)	0.040	0.038	0.038	0.040	0.038	0.038	0.038	0.036	0.038



17,000 E
18,000 E
19,000 E
20,000 E
21,000 E
22,000 E
21,000 N
20,000 N
19,000 N
18,000 N
Mine Decline B
Light Vehicle Shop, Offices, Warehouse
Mill Site
Primary Crusher
Mine Decline A
Heavy Equipment Shop
TSF Site
KTons: 1,948
KYds: 1,639
Embankments:
KYds: 1,805
KTons: 3,609
Paste Thickener
N
0 250 500 750 1000
FEET
Josephine Mining Corp.
Turner Gold Project
Date
Aug. 2009
Drawing File
Site_topo_50ft
Surface Facilities Map
50 foot Topography

Figure 18-1

Ore from the stopes will be hauled by LHDs (load-haul-dump underground mining vehicle) to one of the ore passes developed for each of the two ore zones. The ore passes will lead to a haulage level from where the ore will be collected and hauled out to the surface by a set of underground trucks. The ore passes will provide 1,700 short tons surge capacity for underground production. Waste rock will be hauled to the surface via Adits A and B, where it will be crushed and prepared for inclusion in the backfill mix. (See Figure 18-8)

A modified AVOCA method of end-slicing with concurrent backfill will be the primary stoping method used for all areas of the MUZ and MLZ where the orebody is more than 44 ft wide. Where the width is less than 44 feet, the end-slicing method will be used with delayed backfill. Both methods are similar in operation thereby reducing the chance of errors in execution. This method allows some selective recovery of ore within the orebody.

All stopes will be mined from hangwall to footwall and will be backfilled within the sequence to provide support to the hangwall and sidewalls of the excavated stopes and to limit the extent of sidewall exposed at any time. Backfill will consist of deslimed tailings from the concentrator mixed with adequate amounts of aggregate and cement to ensure that the emplaced backfill will be free-standing, especially after consolidation. The backfill plant will be located near Adit A.

18.1.2 Mine Production Rate and Mine Life

The mine production rate is based on supplying the mill with 8,750 short tons per week of ore. The mill will operate seven days per week with an availability of 92%. The mine will operate 7 days per week, 365 days per year. The average mine production rate will be 1,250 short tons per day.

Taylor’s Rule of Thumb suggests 1128 t/d for 3.586 million tons of potentially mineable material

Taylor’s formula is:

$$\text{Optimum Production Rate} = \frac{5 \times (\text{Potentially Mineable Material})^{0.75}}{(\text{Production Days per year})}$$

Mine operating life is estimated at 8 years at 1,250 tons per day, with 8 months of pre-production underground development.

The AVOCA method will require an average advance of 7.5 ft per day to meet the target of 1,250 t/d. Each round will generate approximately 420 tons based on an average stope width of 15 ft and a bench height of 35 ft. Additional muck will be provided from development headings in ore to the extent of 410 tons to provide a total of 1,250 tons daily. For effective operation of the AVOCA method, filling of two stopes will be conducted in sequence.

The mine productivity per manshift is calculated as the target production divided by the number of direct and indirect personnel, including supervision. The productivity per total manshift includes all personnel related to mining, including technical services, maintenance and the mine manager and are shown in Table 18-2.

Table 18-2: Underground Mine Productivity Planned at Turner Gold Project

	Manpower	Tons/manshift
Mine productivity	48	26.04
Total productivity	55	22.72

8.1.3 Geotechnical Evaluation

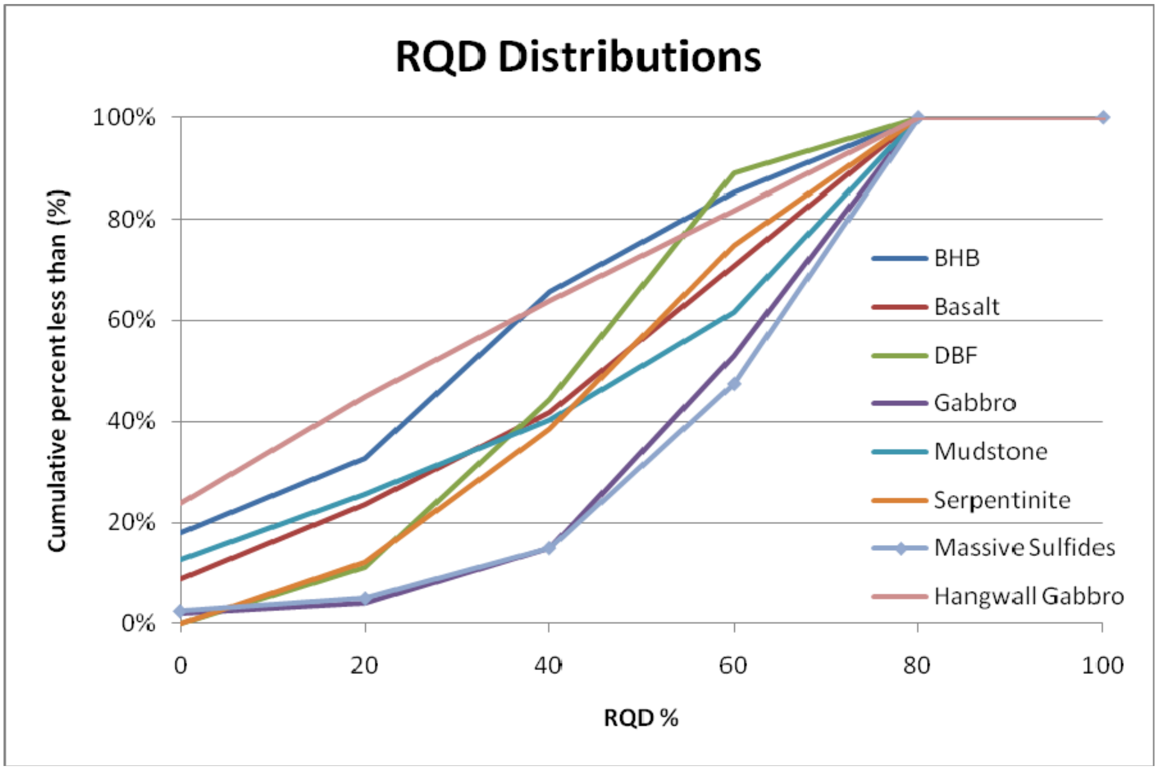
Based on a physical examination of the core from several drill holes through the deposit and RQD records from some of the holes drilled by Noranda, the two ore zones are located within a weak to moderate strength Massive Sulfide with an average RQD of 60%. Mudstone and Gabbro units in the hangwall side are of lower strength with an average RQD of 40-50% while the footwall consists of Basalt and Gabbro units with an average RQD of 50%. Surface outcrops show the presence of at least three sets of joints of varying degrees of weathering.

The RQD data collected by Noranda for the different rock types from 9 holes within the area is plotted in Figure 18-2. These results represent ground conditions that can be described as Poor to Fair. The physical examination of the core showed that there are several zones of sheared or broken ground and marcasite was present in many sections.

Based on the above, the excavations are expected to require moderate ground support in the form of split sets, with some areas requiring wire mesh. Due to the presence of marcasite, excavations with a life exceeding 6 months will require shotcrete support. Large excavations will require more detailed ground support design.

Slickenslides were observed within the hangwall and footwall units and are likely to increase ground support requirement for long-term excavations. Additional geotechnical information will need to be collected in the next phase to identify suitable locations for long term excavations in the footwall.

Figure 18.2 RQD Distributions at Turner Gold Project by Rock Type



18.1.4 Mining Method

Several factors were taken into account when selecting the appropriate mining method for the Turner Gold Project, including:

- The general shape and thickness of the orebody,
- Dip angle of the orebody,
- Ground conditions within the orebody and wall rocks

Based on the available geotechnical and geological information, the preferred mining methods for this type of deposit are open-pit mining, square-set stoping and cut-and-fill. Open-pit mining would require large scale surface disturbance in this area and is not recommended. Square-set stoping would require the use of large quantities of timber and is not amenable to mechanization. While a traditional cut-and-fill operation is suitable, the large spans and moderate to weak rock would require that post-pillars be used to break the spans. This would reduce the recovery of mineral resource from the deposit.

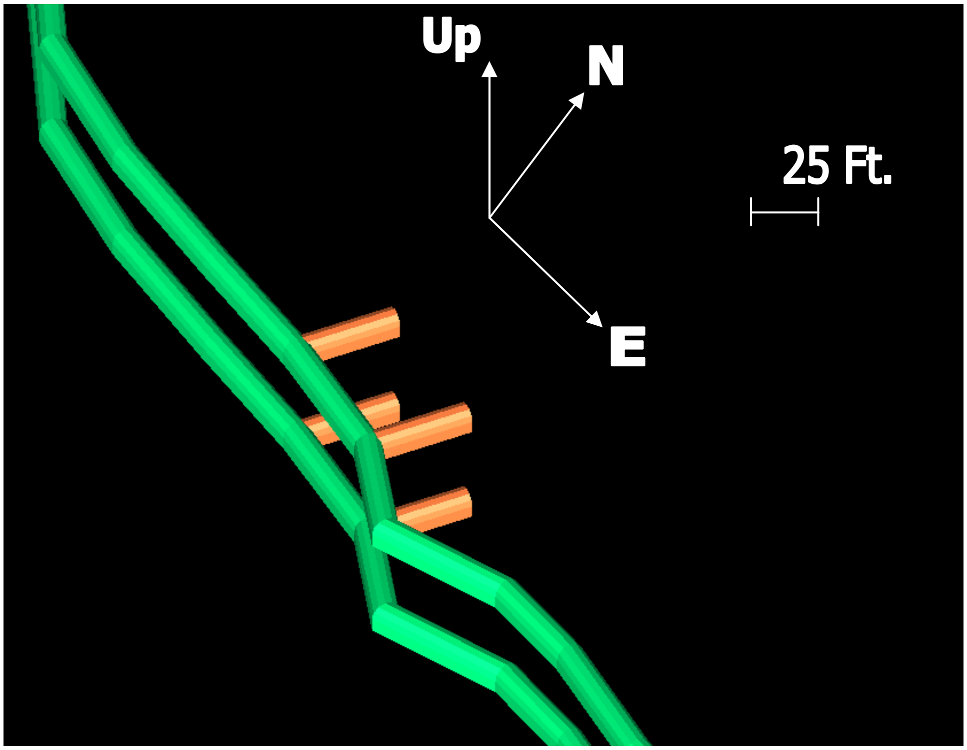
The modified Avoca method would allow for increasing the recovery of the resource while maintaining controlled open spans within the deposit. However, in-stope dilution of 7-10% can be expected for ore zone widths of 200 ft. The method also allows for a higher degree of mechanization and reduced manpower and supervision costs. The modified Avoca method also allows for sharing the drilling and extraction levels within a vertical series of stopes. Scheduling of stoping operations is less critical in this method since the drilling, mucking and filling operations are conducted from different accesses to the stope.

The modified Avoca method will be used as the primary stoping method at the Turner Gold project for this PEA.

<u>Description of Avoca Method</u>

From the footwall ramp, 12 ft by 12 ft access drifts will be developed parallel to the ore zone at 50 ft vertical intervals (Figure 18-3) and 12 ft by 12 ft access crosscuts will be developed at 75 ft centers to the footwall of the ore zone. The crosscuts will be advanced to the hangwall side of the ore zone to a width of 15 ft within the ore zone. The crosscut will be advanced to a distance of 20 ft into the hangwall and then connected to the adjacent crosscut (Figure 18-4).

Figure 18-3: Development of Access Drifts and Stope Access Crosscuts



At the lower level, 50 ft below the upper level, a similar layout of drifts and crosscuts will be developed as extraction drifts. Once both the upper and lower loops are completed, a 6 ft wide slot will be developed between the two levels at the hangwall side using a production drill drilling 1-5/8 in. vertical holes (Figure 18-5). The blasted material will be mucked from the lower level using a 5 cu. yd. remote operated LHD, which will muck into the footwall ore pass.

The production drill will continue to drill vertical holes parallel to the slot and two rings will be blasted at a time in the stope (Figure 18-6) while the LHD clears the muck at the lower level. Since the mine area is expected to be dry, ANFO explosives will be used for blasting, initiated with dynamite primers and non-electric detonators. Each blast will produce 420 tons which will be cleared by the LHD to the ore pass within the shift.

Figure 18-4: Development of Stope Drifts and Hanging Wall Connecting Drift

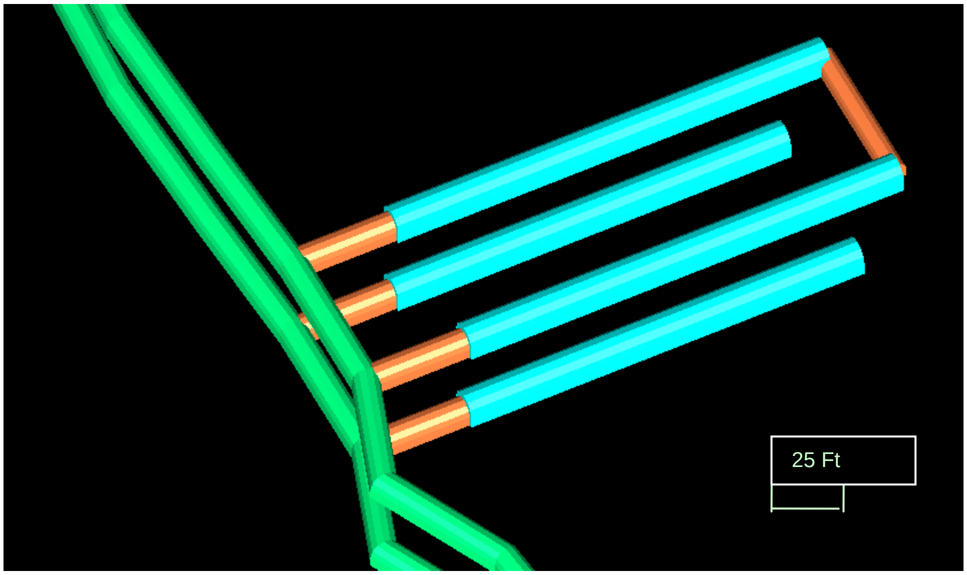
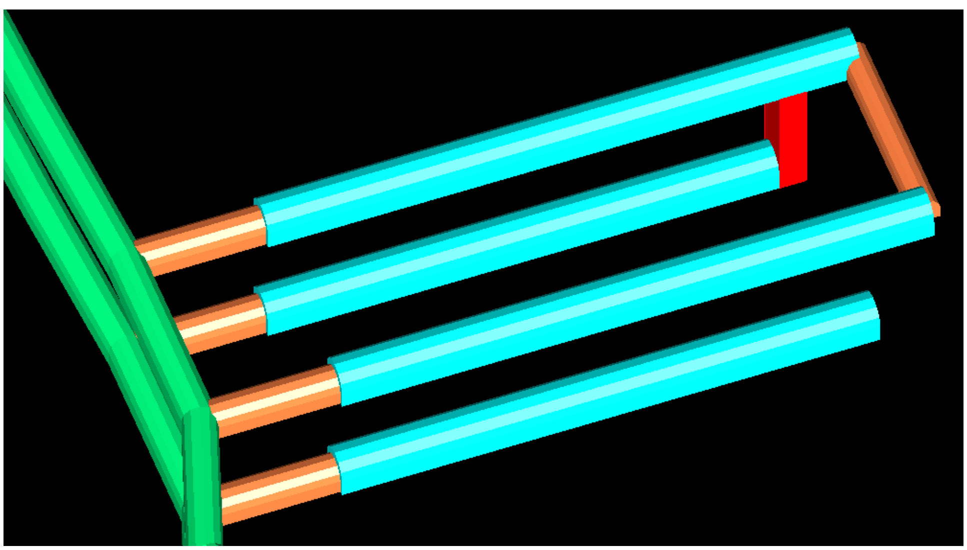


Figure 18-5: Blasting the Slot in the Hanging Wall Side of the Stope



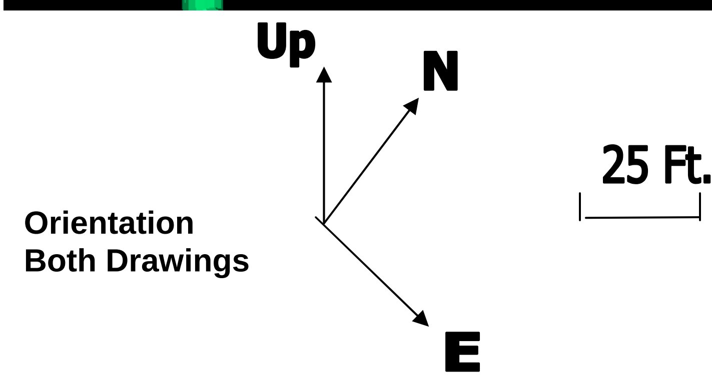
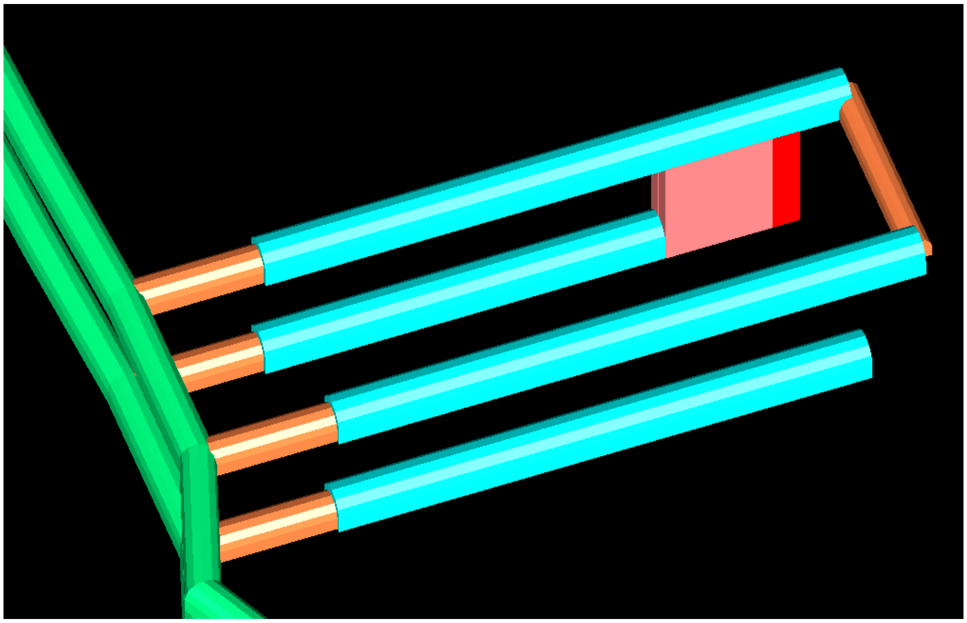
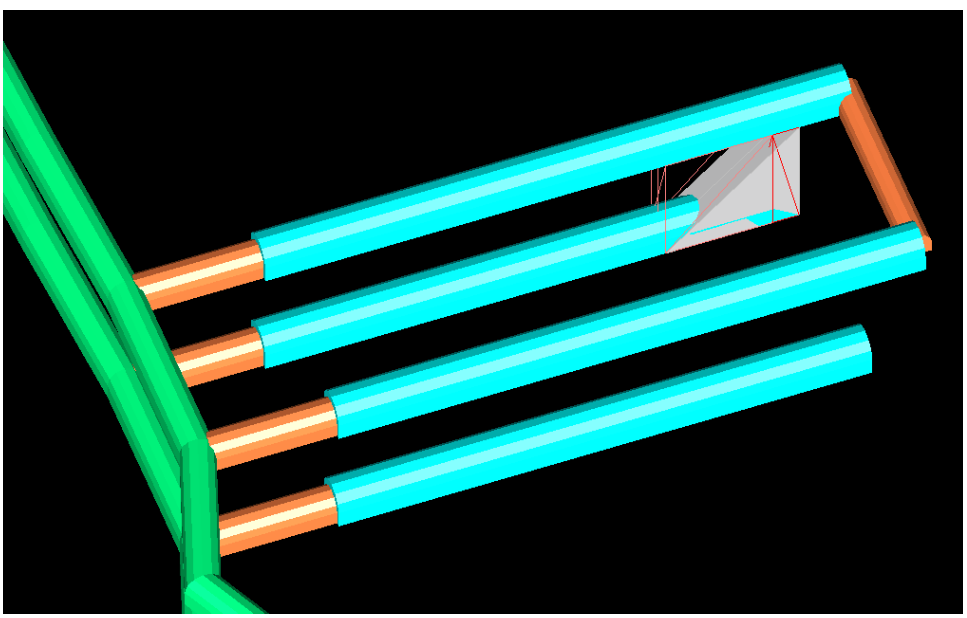
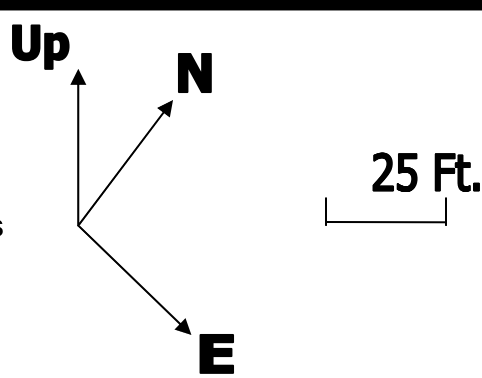


Figure 18-6: Production Blasting from Hanging Wall to Footwall in the Stope



18-7: Backfilling of Stope from Upper Hanging Wall Drift



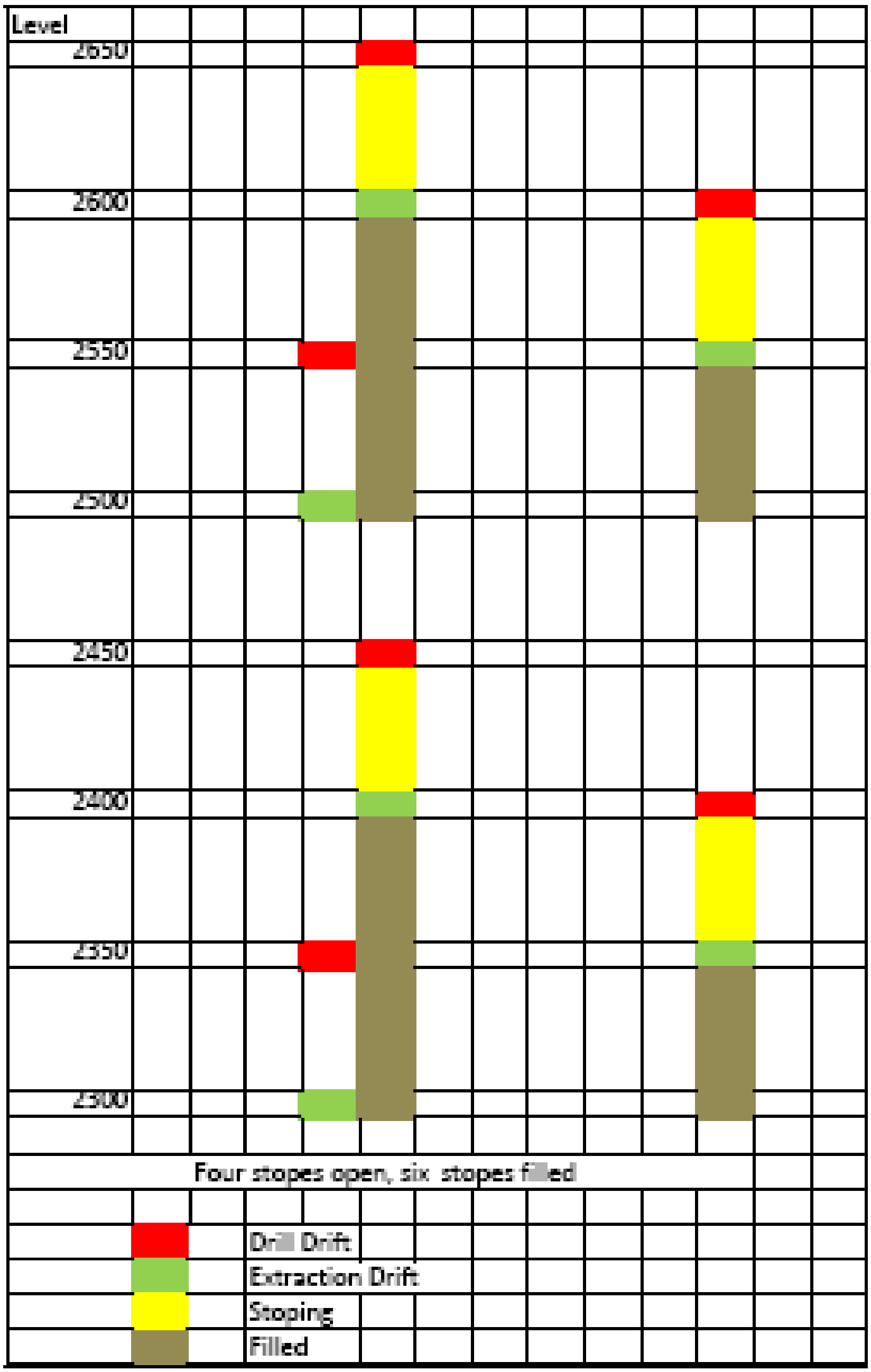


Orientation
Both Drawings

Backfilling will be started from the upper level drift on the hangwall side and continued in sequence with the advance of stoping operations (Figure 18-7). The backfill will consist of about 25-30% deslimed tailings (+200 mesh), coarse aggregate from the crushing of development waste rock, 8-10% cement and additional aggregate and sand from a surface basalt quarry within the mining area.

Once the entire stope is backfilled and the backfill has consolidated, the top drill drift can be used as the bottom extraction drift for the next cut in the sequence (Figure 18-8).

Figure 18-8
Proposed Stoping Sequence



18.1.5 Mining Schedule

The modified Avoca method requires three accesses to each stope block

- A drill drift at the upper level from where production drilling will be conducted
- A bottom extraction drift from where the LHD will remove the broken muck
- A fill drift on the hangwall side of the upper level from where the backfill will be poured into the excavated stope.

The mining schedule will sequence the operations so that production uniformity is maintained. Adequate advance will be maintained within the development headings so that stoping areas are always available for drilling, blasting, mucking and filling operations. Since the backfill will be placed from the hangwall side, there will be little dependence of the schedule on the strengthening of the backfill, though it will need about 8-10 days to be consolidated and about 21-28 days for it to develop full strength capacity. Production operations will be scheduled to account for the strengthening of the poured backfill and to ensure that the exposed sidewalls do not exceed 50 ft in length.

The presence of marcasite within the orebody will restrict the length of time that an excavation can be kept open. From current indications, excavations can be expected to be stable without extensive weathering for about 3-6 months. However, this should be confirmed with testing on the weathering rates of the marcasite. Additional ground support measures such as campaign shotcreting may be recommended in case the weathering rate is faster than currently anticipated.

All lateral and ramp waste development will be performed by two boom mining jumbos. Split sets of 6-8 ft length will be installed as the primary ground support using the same jumbos. All permanent Load-haul-dump units (LHDs) will muck broken rock to a remuck bay before loading into articulated haul trucks, which will haul the waste to a crushing plant at the surface to prepare the aggregate required to be added to the backfill. Single heading development rates are estimated at 30 ft per day for the adits and 24 ft per day for the ramps and access drifts.

18.1.6 Mine Access

The high value blocks of ore within the identified resources lie above the 1900 level in the MLZ and the 2300 level in the MUZ. Two adits are therefore proposed from the 2150 ft elevation near the proposed plant site to the 1900 ft and 2300 ft elevation. The MLZ Adit will be developed as a decline from a portal at Site A shown on Figure 18.9 to the 1900 ft elevation close to the MLZ ore zone. The MUZ adit will be developed as an incline from a portal at Site B to the 2300 ft elevation close to the MUZ ore zone. An additional adit will be developed at the 2550 ft elevation to provide access to the top of the MUZ and to serve as an exhaust ventilation opening at the top of the ore zones. This adit can also be a main access to the UHP pods if they are deemed to be economical to mine at a later stage.

The three adits will allow access to the top and bottom of the two ore zones. In addition, footwall ramps will need to be developed between the levels to allow access to the stoping levels. The footwall ramps will be 13 ft wide by 13 ft high and will be developed at a grade of -15%. From the footwall ramps, stope access drifts will be developed at 50 ft intervals as part of the pre-production development.

18.1.7 Preproduction Development

Over a period of 8 months, the underground mine pre-production development will include the items listed in Table 18-3. The main adits and ramps will be developed using mining crews who will continue to work on other mine development and stoping work once the adits and ramps are completed. During the pre-production phase, crews will be working 7 days a week. The ventilation raises and ore passes will be developed using a raise boring contractor, since these activities will be one time only.

Development of four stopes will be included in the pre-production development with 1440 ft of 10 ft by 10 ft drifts in waste and 2000 ft of 15 ft by 15 ft drifts in ore.

Table 18-3: Preproduction Development

Development	Size	Length	Number	Total	Rock
Adit A to 1900 level	15 ft wide x 15 ft high	2324 ft	1	2324 ft	Waste
Adit B to 2300 level	15 ft wide x 15 ft high	2410 ft	1	2410 ft	Waste
Adit C to 2600 level	15 ft wide x 15 ft high	1483 ft	1	1483 ft	Waste
MUZ Footwall Ramp (2300-2600)	13 ft wide x 13 ft high	1138 ft	1	3000 ft	Waste
MLZ Footwall Ramp (1900-2300)	13 ft wide x 13 ft high	560 ft	1	4000 ft	Waste
Ventilation Raise (1900-2600)	8 ft dial	400 ft	1	700 ft	Waste
Ore pass	6 ft dia	680 ft	2	800 ft	Waste
Access drifts	13ft wide x 13 ft high	500 ft	4	2000 ft	Waste
Stope development in waste	13 ft wide x 13 ft high	180 ft	8	1440 ft	Waste
Stope development in ore	15 ft wide x 15 ft high	250 ft	8	2000 ft	Ore

18.1.8 Mine Development

The main working levels will be developed at 50 ft intervals. Access drifts will be developed parallel to the ore zone at each working level from the footwall ramp. The stope access crosscuts will be developed from the access drifts to the ore zone. Since adjacent stopes cannot be mined at the same time, the stope access drifts will be developed as 10 ft by 10 ft drifts at a spacing of 75 ft along strike. These drifts will be developed from the footwall to the hangwall side and then looped back to connect on the hangwall side. The completion of the hangwall drifts will aid ventilation.

Stope development will include the items shown in Table 18-4.

Table 18-4: Stope Development

Stope access cross-cuts	13 ft wide x 13 ft high	30 ft	8	240 ft	Waste
Stope drifts	15 ft wide x 15 ft high	250 ft	8	2000 ft	Ore
Hangwall drifts	13 ft wide x 13 ft high	100 ft	4	400 ft	Waste

18.1.9 Backfill

Backfill will be required to provide working floor and for sidewall stability in the modified Avoca stopes. The backfill will be comprised of deslimed tailings, cement, coarse aggregate and sand and will be mixed at a backfill mixer plant close to the thickener. The final design of the backfill plant will need to be undertaken based on the quality and quantities of available inputs to the mix and the strength requirement from the backfill. The working floor will be critical to minimize paste backfill dilution and maximize productivity of the production LHDs.

18.1.10 Material Handling

Waste rock generated during the pre-production phase will be hauled to a development rock dump close to the portals of the adits and it is expected that most of this waste rock can be crushed in the surface crushing plant and used as aggregate for backfill when stoping operations start.

Ore from stopes will be mucked using 5 cu. yd. LHDs with tele-operation capability (for operating in the open areas of the Avoca stopes) and will be hauled to the central ore pass. The ore will be transferred to the 2,500 level and the 1,900 level from where the ore will be loaded into 30 ton underground haul trucks and hauled to the surface through Adit A or Adit B. The ore passes will provide a surge capacity of about 1,700 tons, if required.

Development waste rock will be loaded into articulated haul trucks by LHDs and hauled to surface crushing plant for generating coarse aggregate for backfill. Waste rock from within the stopes will be stored in a remuck bay and moved out to the surface crushing plant either by LHDs or by the underground haul trucks.

The backfill will be transported to the operating stopes by a dedicated 30 ton underground haul truck with an ejector bucket which will rear-dump into the stopes from hangwall side. The backfill will then be leveled by a development LHD.

18.1.11 Mine Equipment

The Turner Gold Project will be mined using medium sized underground mining equipment including 1-boom and 2-boom drill jumbos, LHDs and underground haul trucks. Most of the equipment will be diesel powered and be mobile, but the drill jumbos, ventilation fans and pumps will require electrical power. The compressor can be either electric or diesel powered and will provide the necessary compressed air for the operation of pneumatic equipment within the mine.

Table 18-5 lists the underground development, production, and service equipment required for the operation of the project. Some of the equipment will require line power and the power requirements are also estimated in the table.

Table 18-5 Mine Equipment Requirement

Equipment	No of units	Operating voltage	Total Line Power Required
Battery Charger - 40 lamp unit	1	110 V	1 KW
Drifter	6		
Drifter Feed	6		
1-boom Jumbo	2	440 V	120 KW
2-boom Jumbo	2	440 V	250 KW
5 cu yd LHD	3		
30-ton UG Truck	5		
ANFO Loader	1		
Lube Truck	1		
Main fan	1	440 V	360 KW
Auxiliary Fans	4	220 V	100 KW
Shotcreters	1		
Fresh water pumps	3	110 V	90 KW
Compressors	1	440 V	200 KW
Exploration drills	1	220 V	45 KW
Scissorlift	1		
Flatbed truck	2		
Grader	1		

18.1.12 Mine Exploration

Mine exploration from underground will be required early in the mine life. During preproduction, access to the orebody from the MUZ and MLZ adits will be used for exploration drilling of the lower portions and the footwall sides of the ore zones. The MLZ is currently open down dip and will require further exploration drilling. Delineation drilling will be required for grade control.

18.1.13 Mine Services

<u>Ventilation</u>

The adits at 2,150 level will serve as primary intakes to the MLZ and MUZ areas of the Turner Gold Project. The intake air will be coursed though the ramps and the access drifts to the working areas and will be exhausted to the exhaust raise on the hangwall side. Adit C will be used as the primary exhaust at full production and will be linked to the main exhaust raise and will be equipped with a main exhaust fan. The total ventilation requirements for the Turner Gold Project will be around 350,000 cfm.

Adequate ventilation of the stopes is necessary since diesel fumes from the operating equipment and blasting smoke must be exhausted. Ventilation intake will be through the footwall side of the access drifts at each level and will be exhausted through the hangwall excavations to the exhaust raise. Air will be blocked from entering unused areas. The principal of single-pass ventilation (one use) will be followed.

Auxiliary fans will be used on production levels to provide approximately 20,000 cfm for one LHD or 40,000 cfm for two LHDs.

Water and Mine Drainage

Groundwater inflows and water from development and production drilling will be collected in underground sumps. A permanent sump will be required at the lowest point on each ramp. Water from the lowest point of the MUZ footwall ramp will be coursed through delivery lines in Adit B to the surface water treatment facility. A submersible electric pump installed at the bottom of the MLZ ramp will be used to pump water from the 1,650 level to the surface treatment facility at the 2,150 ft level.

Compressed Air

Compressed air will be required for the following:

- Development and production jumbo drilling
- Explosive loading
- Cleaning or dewatering blast holes with blowpipes
- Shotcreting

Mobile air compressors will be located on surface at the Adit A Portal and Adit B Portal. Compressed air will be distributed via steel piping with other mine services suspended in the upper corners of development and stope headings. An 8-in diameter pipe will be required in the main ramps, with 4-in to 2-in diameter pipes in secondary headings and stopes.

Explosives Storage

ANFO will be the bulk explosive for underground production and development. During pre-production there will be blasting at anytime for the development headings. After the pre-production period all blasting will be at the end of each shift. All personnel underground will be required to be in a designated Safe Work Area during blasting.

Cap and powder magazines will be located near the portals of the two main adits. The cap and explosive magazines will be installed approximately 100 ft apart and have sufficient storage for one week of explosives. Transport of explosives underground will be by an underground flatbed logistics truck.

Supplies and Personnel Transportation

Flatbed diesel-powered utility vehicles will move supplies including drill parts, explosives, and other consumables from surface to underground work areas. Supervisors, engineers, geologists, surveyors, mechanics, and electricians will share small diesel powered vehicles to travel to working areas within the mine.

Maintenance

Preventive maintenance encompasses all activities that prolong the life of equipment and reduce premature failures. Management of the preventive maintenance program will be implemented early in the mine life. Maintenance personnel underground will perform preventative and corrective maintenance work including adjustments, lubrication, and refueling.

All major repair and maintenance on mining equipment including drills, loaders, and trucks will be performed on surface in the heavy vehicle workshop located between the mine dry and the concentrator plant. The maintenance planner on-site will develop maintenance schedules.

Fuel Storage and Distribution

The storage fuel tanks will be installed on a concrete pad with environmentally approved containment with concrete berms to prevent contamination in the event of a spillage. All bulk lubricants for operations will be stored in approved containment areas.

Mine trucks hauling ore and waste rock will be refueled on surface. A lube-fuel truck with a 1,000 US gallon tank will fuel LHD units, drills, and other underground diesel equipment not reporting to the surface each shift.

18.1.14 Mine Operating Expenses

The operating expenses for the Turner Gold Project have been developed using the cost models in CostMine 2009 and have been modified to reflect the average wages in the local area based on Labor Market Information from the Oregon Employment Department. The wages for the different categories of workers include 35% burden and 5% overtime.

	$/Ton
Equipment Operation	$ 7.64
Hourly Wages	$ 7.50
Salaried Employees	$ 2.78
Supplies	$ 7.45
Sundries	$ 2.08
Total	$27.45

The manpower required for this project includes 18 salaried personnel and 48 hourly employees. The category of manpower and the corresponding wage rates (including burden for all employees and overtime for hourly employees) used for the calculation of the mine operating expenses are shown in Table 18-6 and Table 18-7.

Table 18-6: Salaried Employees

Category	Number	Rate/yr	Rate/ton
Managers	1	$128,600.00	$0.29
Superintendents	1	$104,300.00	$0.23
Foremen	4	$87,600.00	$0.78
Engineers	1	$97,700.00	$0.22
Geologists	1	$95,000.00	$0.21
Shift bosses	4	$74,900.00	$0.67
Technicians	3	$58,700.00	$0.39
	15		$2.78

Table 18-7: Hourly Employees per day

Category	Number	Rate/hr	Rate/ton
Stope Miners	6	$28.69	$1.10
Development Miners	6	$28.69	$1.10
Equipment Operators	6	$28.69	$1.10
Support Miners	2	$21.05	$0.27
Diamond driller	1	$23.99	$0.15
Backfill plant operators	2	$21.05	$0.27
Electricians	2	$25.20	$0.32
Mechanics	3	$25.20	$0.48
Maintenance workers	4	$21.05	$0.54
Helpers	6	$21.05	$0.81
Underground laborers	6	$21.05	$0.81
Surface laborers	4	$21.05	$0.54
Total	48		$7.50

The cost of materials are based on the cost models from CostMine 2009 and the estimates of time and materials required for the Avoca method have been estimated using standard underground mine design procedures.

18.1.15 Mine Capital Expenses

Capital expenses related to the mine for the Turner Gold Project include the purchase of equipment (as shown in Table 18-5) and the cost of capital development of infrastructure (as shown in Table 18-3) Administrative overheads for the equipment have been based on expected usage of the equipment over the life of the project. While one set of equipment can be expected to service the mine till the end of the mine life, it is prudent to plan for some additional equipment as replacements after 7 years.

	Mine Capital, USD
Equipment purchase	$ 9,764,815
Capital Development	$ 7,558,588
Total	$17,323,403

18.1.16 Mine Production Plan

A preliminary production plan for the Turner Gold Project is shown in Table 18-1. Average grades for each year have been estimated based on the areas which can be mined during the year, with high grade areas being targeted in the early years of the mining operations. The quantity of waste rock excavated per year is also shown, most of which will be crushed at the surface crushing plant and mixed as aggregate in the backfill. The total backfill requirement will be 250,000 short tons per year at full production and will include deslimed tailings (30% of total tailings production), development waste from the mine (30,720 short tons) and surface quarried rock.

18.2 Process

The process facilities layout, showing the various process and maintenance buildings and the tailing storage facility is presented on Figure 18-1. Figure 18-9 presents the flow sheet again for convenience. A brief description of the facility operations is as follows:

Crushing

The primary jaw crusher will be located outside the declines near the mill. 30-ton underground trucks will deliver an average of 52 tons per hour (1,250 per day average) of ROM ore to the jaw crusher. A belt feeder will transfer crushed ore to a coarse ore conveyor belt. The coarse ore conveyor will then transfer ore to the coarse ore storage bin located in the concentrator building. The dust emissions produced during the crushing and subsequent ore transfer will be collected with a cartridge filter dust collector.

Coarse Ore Storage Bin

Crushed ore will be conveyed from the crusher to the coarse ore storage bin. The bin will have approximately 1,500 tons of live capacity, or 1 day concentrator feed. Material will be taken from the storage bin through two draw holes by a belt conveyor supplying the Semi-Autogenous Grinding (SAG) mill feed conveyor.

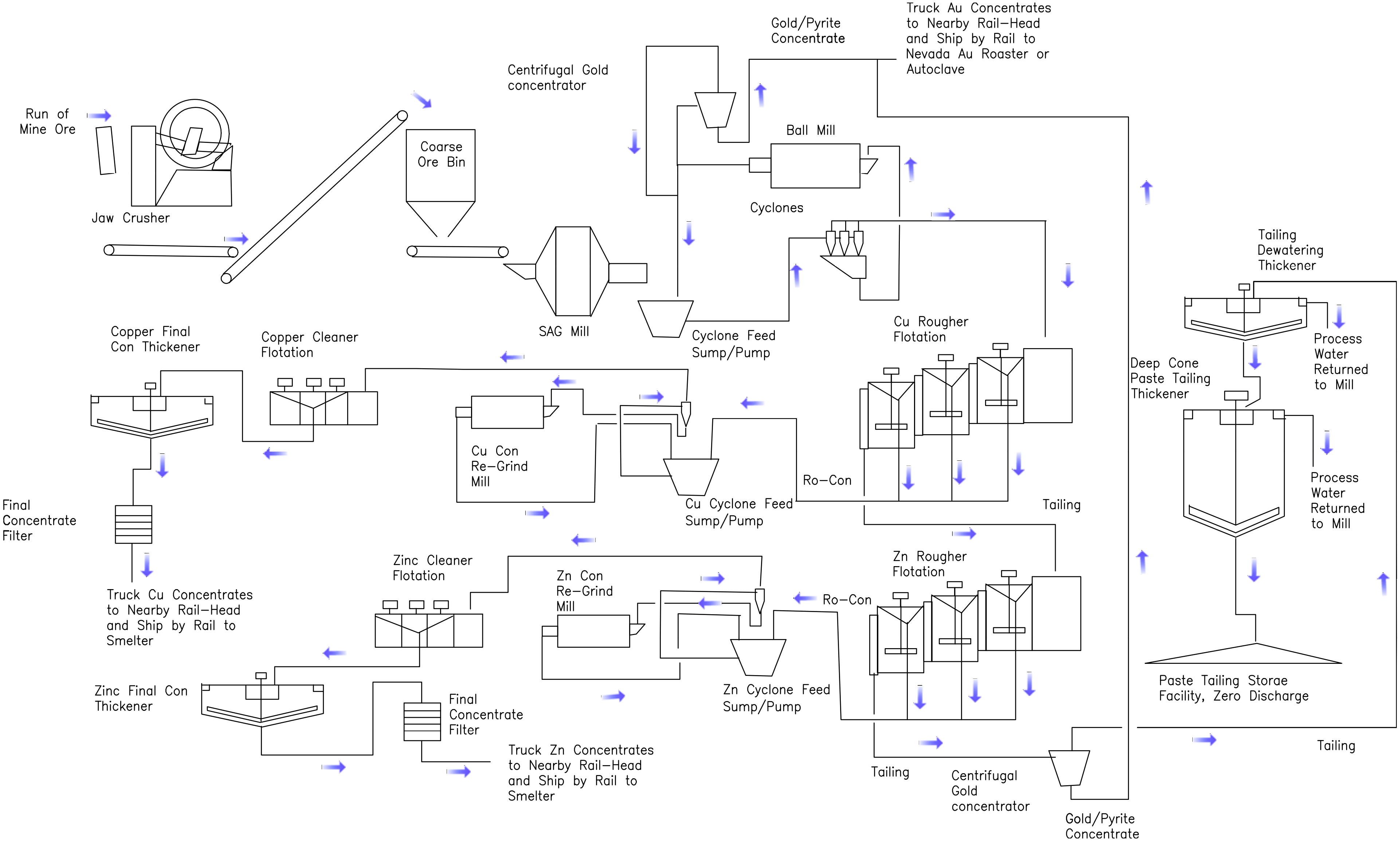


Figure 18-9: Process Flow Sheet

Grinding

The concentrator steel building will enclose the SAG mill, ball mill, and all associated equipment. The grinding circuit will reduce the primary crushed ore from the jaw crusher P_{80} of 100,000 microns to the target flotation feed size of P_{80} 55 microns. The SAG mill will be designed to operate at an average throughput rate of 56 tons per hour; however, the actual throughput will vary due to the variability of ore hardness. Primary grinding will be carried-out in one stage using one 13 foot diameter x 5.67 foot EGL SAG mill (400 hp drive) with oversize from the trommel screen recycling to the feed end of the mill by conveyor belts. Transfer size from the SAG will be T_{80} 1000 microns.

Secondary grinding will be performed in one 12 foot diameter x 17.5 foot EGL ball mill (1,300 hp drive). The ball mill will operate in a closed circuit with a hydro-cyclone classifier. Properly sized material (hydro-cyclone overflow) will flow to the flotation circuit by gravity. Oversized material slurry in the hydro-cyclone underflow will be sent back through the ball mill. A portion, one-third, of the underflow will be processed through a centrifugal type gravity concentrating circuit to recover coarser gold bearing pyrite particles into a gold concentrate. Tailing from the gravity circuit will flow back to the ball mill. These grinding and gravity concentrating process will be wet processes and, therefore, negligible dust emissions will be generated.

Flotation and Regrind

The flotation and regrind operations will be housed in the same concentrator steel building, 100 feet by 250 feet, as the grinding circuit. There will be two flotation/regrind circuits. One circuit will produce a copper concentrate for off-site smelting and refining. The other will produce a zinc concentrate for off-site refining.

Cyclone overflow from the grinding circuit will be conditioned in a stirred aeration tank followed by a second conditioning tank where flotation reagents (collector, frother and depressants) will be added to float the copper minerals and depress the pyrite. The conditioned slurry will gravity flow into the rougher flotation circuit consisting of five 565-cu.ft. tank type flotation cells in series configuration. Copper rougher concentrate will advance by gravity to a cyclone feed cell for the copper regrind circuit where it will mix with the discharge of the copper regrind ball mill.

The cyclone feed slurry will be pumped to a cyclone classifier. Properly sized overflow slurry, P_{80} 8-15 micron, will flow by gravity to the copper cleaner flotation circuits where additional gangue sulfides (pyrite and sphalerite) will be removed to upgrade the copper concentrate to 28% copper. Underflow from the cyclone will flow by gravity to the feed chute to the copper regrind mill (7 foot diameter x 11.5 foot EGL ball mill with 200 hp drive). Additional flotation reagents will be added to the copper cleaner circuits to collect and float the copper minerals and depress the non-copper minerals. The copper cleaner circuit will consist of a first cleaner circuit (four 177-cu.ft tank cells in series), a first cleaner scavenger circuit (one 177-cu.ft. tank cell) and a second cleaner circuit (three 106-cu.ft. tank cells in series). The second cleaner circuit will upgrade copper concentrates from the first cleaner circuit and the scavenger cleaner circuit. Final copper concentrate will be produced in the second cleaner circuit and will flow by gravity to the final copper concentrate thickener (16.5 foot diameter). Thickened final copper concentrate will be pumped to a dewatering filter. Tailing from the second cleaner will be recycled to the first cleaner and/or the copper re-grind cyclone feed cell. Tailing from the first cleaner scavenger may be discharged to the rougher flotation tailing, sent to the re-grind cyclone feed cell or fed back into the copper rougher circuit depending on the copper mass flow in this tailing.

Thickened final copper concentrate will be pumped to a vertical packed type pressure filter, located in the concentrate building, which utilizes a final air blow for final dewatering of the copper concentrate. Concentrate will be dewatered to less than 10% water and conveyed to a copper concentrate storage bin. Concentrate from this bin will be loaded into a end dump tractor trailer by conveyor belt and shipped to a rail head for transfer to a rail car for shipment to the copper smelter and refinery.

Copper rougher flotation tailing will advance by gravity to the zinc rougher flotation circuit for recovery of the zinc minerals. Slurry will flow into a stirred conditioning tank where reagents will be added to reactivate the zinc minerals (copper sulfate) and float the zinc minerals (collector, frother and depressant for sulfide gangue). The conditioned slurry will gravity flow into the rougher flotation circuit consisting of six 565-cu.ft. tank type flotation cells in series configuration. Zinc rougher concentrate will advance by gravity to a cyclone feed cell for the zinc regrind circuit where it will mix with the discharge of the zinc regrind ball mill.

The cyclone feed slurry will be pumped to a cyclone classifier. Properly sized overflow slurry, P_{80} 8-15 micron, will flow by gravity to the cleaner flotation circuit where additional gangue sulfides (pyrite) will be removed to upgrade the zinc concentrate to 53% zinc. Underflow from the cyclone will flow back to the regrind mill (8 foot diameter x 11.0 foot EGL ball mill with 300 hp drive). Additional flotation reagents will be added to the zinc cleaner circuits to collect and float the zinc minerals and depress the non-zinc minerals. The zinc cleaner circuit will consist of a first cleaner circuit (two 565-cu.ft tank cells in series), a first cleaner scavenger circuit (three 177-cu.ft. tank cell) and a second cleaner circuit (four 106-cu.ft. tank cells in series). The second zinc cleaner circuit will upgrade concentrates from the first cleaner circuit and the scavenger cleaner circuit. Final zinc concentrate will be produced in the second cleaner circuit and will flow by gravity to the final zinc concentrate thickener (16.5 foot diameter. Thickened final zinc concentrate will be pumped to a dewatering filter. Tailing from the second cleaner will be recycled to the first cleaner and/or the zinc re-grind cyclone feed cell. Tailing from the first cleaner scavenger may be discharged to the zinc rougher flotation tailing, sent to the cyclone feed re-grind cell or fed back into the zinc rougher circuit depending on the zinc mass flow in this tailing.

Thickened final zinc concentrate will be pumped to a vertical packed type pressure filter, located in the concentrator building, which utilizes a final air blow for final dewatering of the zinc concentrate. Concentrate will be dewatered to less than 10% water and conveyed to a zinc concentrate storage bin. Concentrate from this bin will be loaded into an end dump tractor trailer by conveyor belt and shipped to a rail head for transfer to a rail car for shipment to the zinc refinery.

Zinc rougher flotation tailing will advance by gravity to the centrifugal type gravity concentrating circuit to recover gold bearing pyrite particles into a gold concentrate. This concentrate will be mixed with the gold concentrate from the aforementioned primary ball mill grinding circuit, stored in supper sacks for truck and rail shipment to a gold autoclave or roaster in Nevada.

Reagents

Process reagents will be stored in tanks, shipping barrels or shipping totes in a central area inside the concentrator building. Reagents will be pumped and distributed to their respective usage points. Some reagents, such as flocculants, will be diluted in water before using them.

Tailings Cycloning, Thickening and Underflow Pumping

Tailings from the gravity concentrator circuit for the zinc rougher tailings will flow by gravity to a pump box where they will be pumped to a single stage cyclone. At the cyclone a sand/slime split will be made to provide coarse tailing for the underground cemented back fill program. Sands will be slurried, stored in an agitated stock tank and pumped to the underground mine back fill operation as required. The stock tank will have 18 hours of storage. The fines (cyclone overflow) will flow by gravity to the tailing dewatering thickener located adjacent to the concentrator building.

One, above ground high capacity tailing dewatering thickener (40 foot diameter), will remove water from the zinc rougher flotation tailings. Clarified water will overflow to a water tank for reuse. The thickened underflow slurry (approximately 55% solids by weight) will be pumped to the paste tailing deep cone thickener (25 feet in diameter) located on top of the hill in Figure 18-1. Under flow (70-77% solids by weight) from the paste tailing thickener will be pumped to dozed out storage cells located in the upper portion of the hill as illustrated in Figure 18-1. Clarified water will over flow the deep cone thickener to a water tank and drain by gravity to the concentrator for reuse in the process.

The paste tailing as placed will probably have no free water so the only free water to collect will come from rainfall. Once a cell is filled with paste tailing the cell will be isolated from further tailing flow and filling of the next available cell will begin. Paste tailing at other mines demonstrate an encapsulating nature and form at crust rapidly allowing access to the surface within a few weeks of stopping deposition. This condition will allow access to the surface for timely placement of growth media, from the stockpiles, and addition of native plant seed and grass seed for concurrent reclamation of the tailing site. Additional geochemical studies are required on the Turner Project paste tailing to determine if the cells will need clay amended compacted soil liner or membrane liner or no liner at all. Additional details for the paste tailing storage facility are mentioned in chapter 18.4.3.

18.2.1 Process Plant Operation Costs

Operating cost for the concentrator, off-site smelting and refining and supporting facilities and shipping is summarized in this section.

The concentrator operating cost per year is $14.68 per ton of processed ore. Mill process operating cost includes crushing and conveying, grinding and classification, flotation and regrind, concentrate thickening, filtration, tailing disposal, tailing area concurrent reclamation and mill ancillary services.

Off-site custom concentrate fees are as follows. Operating cost per year for copper concentrate smelting and refining averages $ 4.25 per ton of mill ore. Operating cost per year for zinc concentrate roast-leach-electrowinning and refining averages $9.22 per ton of mill ore. Operating cost per year for gold concentrate, gold autoclaving and processing are covered in the metal recovery paid for (80%). The operating cost for precious metal refining of gold and silver is $0.11 per ton of ore.

The operating cost for the supporting facilities, G&A and shipping is $6.61 per ton of milled ore. The supporting facilities include laboratory costs at $0.87/ton. The shipping cost, $3.08/ton was estimated based on purchase of end dump trucks, rehabilitation of nearby rail load out facility, purchase of small loader, purchase of sufficient number of rail cars, fuel for the trucks and hiring of truck drivers. The shipping cost covers the delivery of material to the custom treatment facilities. The general and administrative costs, $2.66/ton includes safety and environmental, accounting, human resources and permitting.

The process direct operating cost estimate by cost center is shown in Table 18-8 below. All costs are estimated in 2009 US dollars.

Table 18-8: Summary of Processing Costs

Item	Annual Cost$	Per Ton Ore Processed
Mill Operations	$6,679,400	$14.68
Supporting Facilities (Lab)	$396,396	$0.87
General and Administrative	$1,208,480	$2.66
Zinc Concentrate Shipping	$624,880	$1.37
Copper Concentrate Shipping	$579,692	$1.27
Gold Concentrate Shipping	$197,501	$0.43
Cu Concentrate Smelting	$1,259,459	$2.77
Copper Refining	$675,335	$1.48
Gold Refining	$41,323	$0.09
Silver Refining	$8,119	$0.02
Zn Concentrate RLE and Refining	$4,196,424	$9.22
Total	$15,867,009	$34.87

18.2.2 Process Plant Capital Costs

The total capital costs are $39.5 million. For purposes of the analysis, it is assumed that the initial capital will be expended in the 3 years prior to start up..

Table 18-9: Summary of Process Plant Capital Costs

Area	Description	Cost
State Permits	Water Pollution Control Permit, Air Permit and Various Permits	$400,000
Engineering	Detailed Engineering for Concentrator and Infrastructure	$3,669,497
Surface Facilities and Equipment	Concentrator, Warehouse, Heavy Equipment Shop, Substation, Office	$30,287,020
Owner’s Cost	Salaries and Wages, Insurance, Legal Fees, Travel, Consultants, Etc.	$5,167,992
Total		$39,524,509

All figures are in 2009 dollars. There is no process sustaining capital cost for the project because it is a relatively short life project and these are accounted for in the maintenance operating cost estimates.

18.3 Environmental

18.3.1 Permits and Approvals

The Turner Gold Mine is currently controlled by JMC through an option to purchase agreement with the owner of the patented mine claims, General Moly, Inc. The proposed Turner Gold Mine falls under Federal, State, and local agency purviews with respect to environmental permits and approvals. These agencies include the Josephine County Planning Department, Oregon Department of Geology and Mineral Industries (DOGAMI), Oregon Department of Environmental Quality (DEQ), Oregon Water Resources Department (WRD), Oregon Department of Forestry (ODF), Oregon Department of Fish and Wildlife (ODFW), Oregon Division of State Lands (DSL), U.S. Fish and Wildlife Service (USFWS), and U.S. Army Corps of Engineers (Corps).

Review of the project area resources and refinement of the proposed action will lead to the identification of the pertinent regulatory agencies, regulations, and necessary authorizations that will be required for construction and operations associated with the project. JMC has retained the services of JBR Environmental Consultants, Inc., (JBR) Medford, Oregon, to assist with the environmental permitting of the proposed facilities. JMC and JBR personnel met with DOGAMI personnel on June 3 and September 3, 2009. JMC and JBR personnel met with DOGAMI and DEQ personnel on October 5, 2009. JMC and JBR personnel met with BLM personnel on October 23, 2009. JMC and JBR personnel conducted a site tour for DOGAMI and ODFW personnel on October 26, 2009. Based on information known to date and discussions held during the above-referenced meetings, the following permits will be required for the Turner Gold Mine:

<u>Oregon Mine Land Regulation and Reclamation – DOGAMI Operating Permit</u>

The Turner Gold Project development and operations will be regulated by DOGAMI Division 35 Oregon Mined Land Reclamation Act (Applicable to Coal and Metal-Bearing Ores Operations) and implementation of the project will require a DOGAMI Operating and Reclamation Permit.

An Operating Permit is required for mining operations that have an activity level that exceeds one acre and/or 5,000 cubic yards of new disturbance in any 12-month period, unless the excavated material stays on the property. In addition to baseline environmental data provided in support of obtaining the Operating and Reclamation Permit, the permit application to DOGAMI will contain the following major sections:

- Physical Description of the Ore Body
- Site Clearing and Construction
- Construction Schedule
- Description of the Underground Operations, Including Production Schedule
- Metallurgical Process Description
- Process Facilities Description

- Tailings Impoundment Facility
- Topsoil Salvage and Storage
- Development Rock Use and Storage
- Drainage and Sediment Control
- Ancillary Facilities
- Visual Screening
- Water Management Plan
- Environmental Consequences and Mitigation
- Spill Prevention and Countermeasure Plan
- Chemical Handling Procedures
- Other site specific items determined through resource evaluation
- Work Force

The DOGAMI Operating and Reclamation Permit is anticipated to be issued in the first quarter of 2012. This schedule is premised on the following: construction of access roads for monitoring well installation and monitoring well installation completed by September 2010, supplemental corehole drilling for geochemical and mine planning data completed by September 2010, completion of a detailed mine plan by May 2011, and DOGAMI agreeing to initiate review of the Operating and Reclamation Permit referencing interim baseline surveys for select resources (e.g. groundwater).

Under section 632-035-0025 of DOGAMI Division 35 rules, DOGAMI, in consultation with DEQ, may require a bond of up to $100,000 per acre of new surface disturbance.

Additional permits (in addition to DOGAMI Operating and Reclamation Permit) required for the proposed project are summarized in Table 18-10.

Table 18-10
Anticipated Permits and Granting Agencies

Granting Agency	Permit/Approval
Oregon Water Resources Department (WRD)	Water Use Permit
Oregon Department of Environmental Quality (DEQ)	WPCF Permit if there is a wastewater discharge
Oregon Department of Environmental Quality (DEQ)	NPDES storm water discharge general permit (1200-Z)
Oregon Department of Environmental Quality (DEQ)	NPDES general construction Permit (1200-C)
Oregon Department of Environmental Quality (DEQ)	Solid waste permit or permit exemption
Oregon Department of Environmental Quality (DEQ)	Air Quality Operating Permit or Permit Exemption if discharge less than 10 tons per year
Oregon Department of Environmental Quality (DEQ)	Spill Prevention and Countermeasure Plan (SPCC) Permit
Oregon Department of Environmental Quality (DEQ)	On-Site Wastewater Treatment System Permit (Septic System)
Oregon Division of State Lands (DSL)	Material Fill or Removal in Waterways Permit to Remove Material from or Place Material in Waterway
U.S. Army Corps of Engineers	Nationwide or Individual Permit
Oregon Department of Forestry (ODF)	Notifying the State of Work Planned on Forest Lands
Josephine County	Site Plan/Use Permit

It is anticipated that the above-referenced permits can be obtained in the first quarter of 2012. Baseline studies to support the permits listed above were initiated in October 2009 and the majority of the baseline data collection is anticipated to be completed by Spring 2011. The groundwater monitoring effort is anticipated to be completed in the third or fourth quarter of 2011, although an interim groundwater baseline survey is anticipated to be completed in the first or second quarter of 2011. Surveys proposed to be performed include the following:

- General vegetation community characterization and threatened and endangered species vegetation survey
- General wildlife habitat characterization and threatened and endangered wildlife species survey
- Seep and Spring Survey
- Wetland and Waters of the U.S. Survey
- Surface Water Investigation, sampling, and quarterly monitoring
- Groundwater Investigation, monthly water level monitoring, and quarterly sampling
- Geochemistry Analysis
- Soils Assessment

18.3.2 Acid Rock Drainage (ARD) Evaluation

A baseline geochemical survey will be conducted at the Turner Gold site to characterize the ARD potential and metals leachability of waste rock and paste tailing so that operations and reclamation minimize and/or mitigate for potential impacts. Specific objectives include the following:

- Characterize developmental waste rock along the anticipated sloped declines into the lower areas of the ore bodies using acid-base accounting (ABA) methods.
- Characterize non-ore rock units:
    - left in place (in situ) or exposed
    - <u>encountered</u> during mining or mine development. These rocks would all be included in the cemented backfill and would be evaluated as such.
- Characterize paste tailing.
- Characterize paste tailing embankment rock sources.
- Evaluate the geochemical data and prepare a baseline report.

The analytical laboratory testing described below is premised on chemical characterization of new core collected during the supplemental core hole drill program, and makes no use of older core except to guide sample selection for the geochemical baseline survey. Development waste rock along the declines will be characterized using a combination of surface samples (serpentinites and basalts) in conjunction with confirmation samples collected during decline construction.

Table 18-11 summarizes the number of samples, sample types, and proposed laboratory analyses for the geochemical baseline survey. Two types of composite samples will be generated using fresh core. Short composite samples will be developed from approximately 10 to 20 subsamples representing approximately 30-foot intervals, and long composite samples will be developed from approximately 30 subsamples representing 100-foot intervals.

The samples will be prepared by crushing to a uniform size (particles approximately 1/3rd-inch in diameter). Following homogenization and blending, splits of the composites will be submitted to a laboratory for analysis that may include all of the following chemical characterization analyses and stoichometric calculations: total sulfur, sulfur speciation, acid generating potential (AGP), acid neutralizing potential (ANP), net neutralization potential (NNP), and metals leachability using the synthetic precipitation leachate procedure (SPLP). The remainders of the crushed composite samples may be submitted for bulk mineralogy analyses and column-based neutralization kinetic studies with laboratory-generated ARD.

While the specific laboratory for analytical testing has not yet been selected, two laboratories for consideration based on their technical capabilities and services are Cantest Laboratory in Burnaby, British Columbia, and SVL Analytical of Kellogg, Idaho.

Data collected during the geochemical baseline survey will be evaluated and presented in a baseline survey report. The report will include descriptions of the following: corehole drilling methods, sample selection, sample preparation, analytical results, evaluation of the potential environmental impacts of the different rock units, and recommendations for additional work, if warranted.

Table 18-11
Proposed Analytical Laboratory Analyses
Geochemistry Baseline Survey

Sample Type	Short Composite (30 feet)	Long Composite (100 feet)	Subsample Preparation	Paste pH	Total Sulfur	Sulfur Speciation	ANP	Whole Rock Major Metals	Whole Rock Trace Metals	Mineralogy	Kinetic Tests	SPLP Leachability (fluids)
Declines												
Basalt		3	30	3	3	3	3		3	3	3	3
Sheeted Dikes		3	20	3	3	3	3		3	3	3	3
Serpentinite		3	30	3	3	3	3		3	3	3	3
Non-Ore Rocks Likely To Be Encountered												
Massive Sulfide Horizons	2		20	2	2	2	2	2	2	1	2	2
Silica Stockwork Stringer Zone	3		30	3	3	3	3	3	3	1	3	3
Mineralized Basalt	2		20	2	2	2	2	2	2	1	2	2
Basin Floor Rubble	1		10	1	1	1	1	1	1		1	1
Talus	1		10	1	1	1	1	1	1		1	1
Non-Ore Rocks That May Be Encountered												
Gabbro	1		10	1	1	1	1		1			1
Mudstones	1		10	1	1	1	1		1			1
Black Massive Chlorite	1		10	1	1	1	1		1			1
Tailings												
Composite Tails with Pyrite (underground)			2	2	2	2	2	2	2	2	1	2
Composite Tails without Pyrite (aboveground)			2	2	2	2	2	2	2	2		2
Totals	12	9	204	25	25	25	25	13	25	16	19	25

Notes:
ANP = Acid Neutralizing Potential
SPLP = Synthetic Precipitation Leachate Procedure

18.3.3 Surface Water and Groundwater Baseline Surveys

It is anticipated that the greatest level of environmental scrutiny by regulatory agencies will be placed on surface water and groundwater surveys. Accordingly, detailed baseline surveys to characterize both surface water and groundwater will be performed. The surface water baseline survey was initiated in October 2009, and it is anticipated that the groundwater baseline survey will be initiated later in the fourth quarter of 2009. The surface water resources of the project area will be defined by the watershed and subwatersheds of the area. These contributing areas will delineate the on-site versus off-site surface water management requirements. The surface water baseline survey will address all existing surface waters, including seeps, springs, and adits, ephemeral and perennial watercourses, precipitation and stormwater runoff estimates, and monitoring and analysis of these waters.

The groundwater resources at the site will be characterized by completing the following tasks: installation and development of 10 groundwater monitoring wells; aquifer testing to define hydrogeologic parameters; monthly monitoring of groundwater and surface water elevations at staff gauges proximal to nested monitoring well pairs; quarterly monitoring of groundwater quality for one year, including sample collection and analyses; and data evaluation and reporting. Proposed monitoring well locations and depths are illustrated on Figure 18-10. It should be noted that both surface water monitoring and groundwater monitoring will continue during and after mining operations to evaluate potential impacts to these resources.

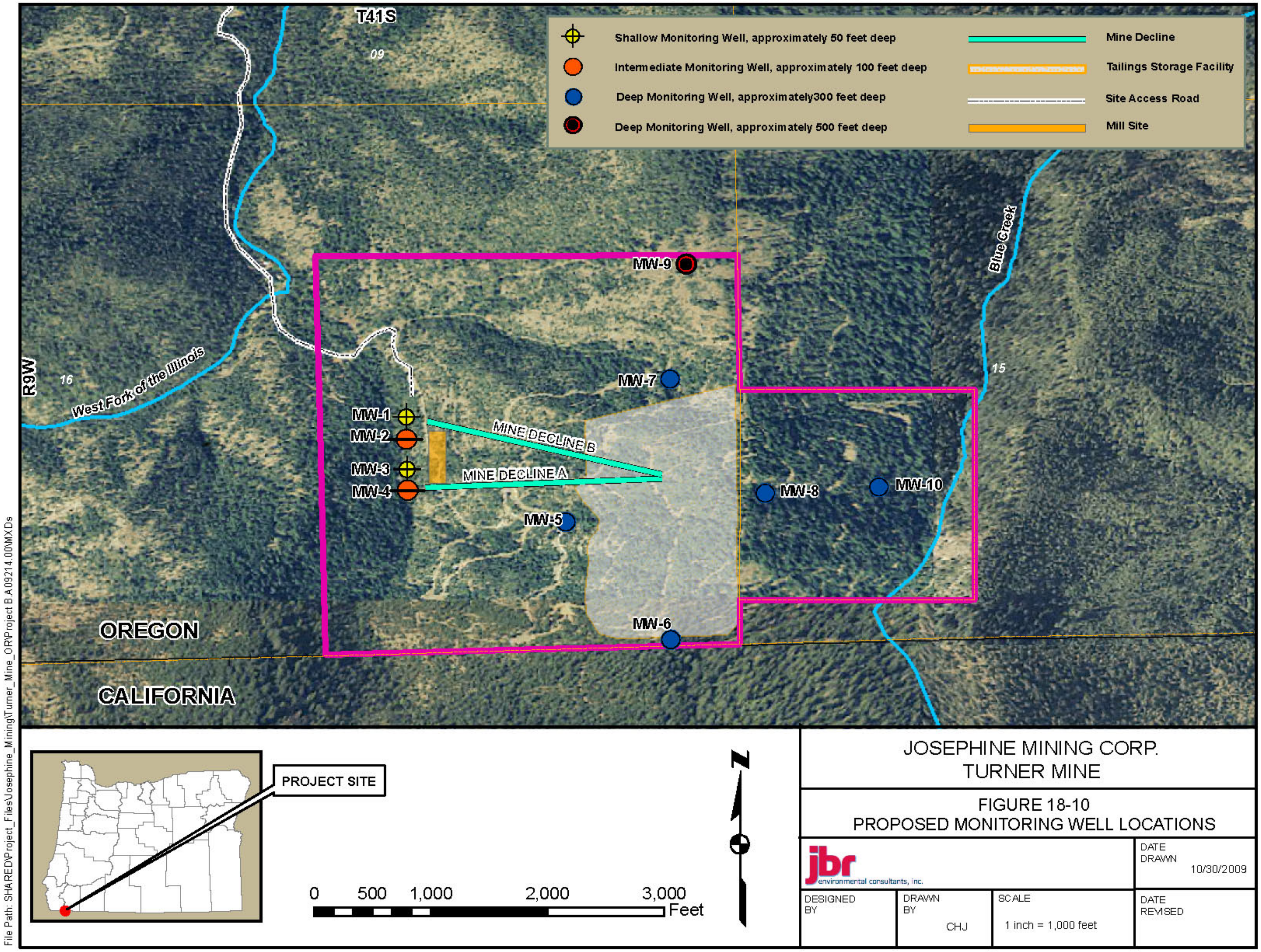


FIGURE 18-10
PROPOSED MONITORING WELL LOCATIONS

18.3.4 Permitting Risk

Submittal of completed permit applications to agencies is anticipated to occur second or third quarter 2011 with permit approvals obtained in the first quarter of 2012.

Of the resources listed above, surface water and groundwater will receive the most scrutiny due to DOGAMI and DEQ concerns regarding potential ARD impacts. Taking into consideration the proposed mine plan, data collected during the surface water and groundwater baseline surveys will be evaluated to address the following issues:

- Groundwater elevations may be affected by water supply pumping and/or mine dewatering. Some seeps and springs may be permanently or temporarily impacted by this pumping, which could affect existing water rights.
- Surface water and groundwater quality may be impacted by mining operations, including placement of both waste rock and tailings.

Should adverse impacts to surface water or groundwater quality be recognized during permitting or operations, appropriate mitigation measures will be implemented to minimize or eliminate these impacts.

The existing access road crosses a small parcel of Bureau of Land Management (BLM) land through an existing BLM right of way. A potential power line route to connect the mine property to an existing electric transmission line crosses County and U.S. Forest Service (USFS) property. Communications and/or meetings with the BLM, USFS, County, and the local power provider will be held to refine plans for these facilities located off the mine property.

Based on the known information provided to date, JBR sees no environmental issues that would prevent the permitting of the proposed operations. Although JBR currently does not see any permitting issues that would prevent the operation of the proposed Turner Gold Mine, JBR cannot predict all the concerns or issues the permitting agencies may have with the proposed project during the permitting process, nor can JBR control how long the agencies will take to issue the necessary permits. At this time, quantification of all the environmental impacts of the proposed facilities and operations is not possible. A better understanding of these will be developed during the permitting process.

18.3.5 Workforce

JMC expects to employ a total of 102 people, year-round during peak operations. Construction is expected to commence during second quarter 2012, subject to receiving all necessary permit approvals. Employees are expected to be hired from the local communities of O’Brien, Cave Junction, Kerby, Selma, and Grants Pass, Oregon (termed the Illinois River Valley). Unemployment within the State of Oregon is currently estimated at 11.9 percent and unemployment within Josephine County is estimated at 14.9 percent (Illinois Valley News, 2009). Therefore, obtaining a workforce for the proposed project is not anticipated to be a concern and increased demands for housing or public/social services is not anticipated.

18.4 Infrastructure Facilities, Description and Costs

18.4.1 Waste Rock Management Facility

Waste rock generated during the pre-production phase will be hauled to a development rock dump close to the portals of the adits and it is expected that most of this waste rock can be crushed in the surface crushing plant and used as aggregate for backfill when stoping operations start.

Development waste rock will be loaded into articulated haul trucks by LHDs and hauled to the surface crushing plant for generating coarse aggregate for backfill. Waste rock from within the stopes will be stored in a re-muck bay and moved out to the surface crushing plant either by LHDs or by the underground haul trucks.

A preliminary production plan for the Turner Gold Project was shown in Table 18-1. The quantity of waste rock excavated per year is shown, most of which will be crushed at the surface crushing plant and mixed as aggregate in the backfill. The total backfill requirement per year will be 350,000 tons at full production and will include de-slimed tailing (30% of total tailings production), development waste from the mine (50,024 tons) and surface quarried rock.

18.4.2 Tailing Management Facility

The thickened underflow slurry (approximately 55% solids by weight) from the concentrator tailing dewatering thickener will be pumped to the paste tailing deep cone thickener (25 feet in diameter) located on top of the hill. Under flow (70-77% solids by weight) from the paste tailing thickener will be pumped to dozed out storage cells (25-30 feet deep) located in the upper portion of the hill as illustrated on Figure 18-1. Clarified water will over flow the deep cone thickener to a water tank and drain by gravity to the concentrator for reuse in the process.

The paste tailing as placed will probably have no free water so the only free water to collect will come from rainfall. Once a cell is filled with paste tailing the cell will be isolated from further tailing flow and filling of the next available cell will begin. Paste tailing at other mines demonstrate an encapsulating nature and form at crust rapidly allowing access to the surface within a few weeks of stopping deposition. This condition will allow access to the surface for timely placement of growth media, from the stockpiles, and addition of native plant seed and grass seed for concurrent reclamation of the tailing site. Additional geochemical studies are required on the Turner Project paste tailing to determine if the cells will need clay amended compacted soil liner or membrane liner or no liner at all.

It is expected that the tailing storage facility will be a zero discharge facility in keeping with the rest of the process and mine facilities, which are designed to be zero discharge facilities. Based on site visits and interviews with those that drilled the deposit the indications are that excess water beyond what is needed for mining and processing the ore will not be available from water seeping from the underground workings. In all likelihood two small ground water wells (one a standby) will be needed to balance out the process consumptive use of water. Future geo-hydrology drilling and testing in the underground deposit area will be undertaken to check this assumption.

Construction of the paste tailing storage facility will occur in such a manner to minimize meteoric rain runoff from the active tailing deposition areas. Again viewing the process layout on Figure 18-1, construction will start downhill from the paste thickener at the 2650 foot contour in the southwest section, near grid coordinates 19,000 East and 18,000 North approximately, near the southern project boundary (Oregon/California state line). Paste tailing cells would be built along this contour in a northern direction for approximately 300 feet. A permanent lined collection channel several feet below the cells to the west and up the south side would be built before start-up to collect any rain runoff from the under construction and pre-reclaimed tailing storage surface areas. Runoff in this channel would collect in a lined pond that has an overflow pipeline that will carry the water back to the tailing dewatering thickener at the concentrator. A temporary clean water diversion would be built up slope, east, of the cells on the 2650 contour to take runoff away from the storage area and divert it around to the north side of the storage facility and back into the natural drainage. Once 300 feet of cells are built out to the north on contour 2650, new construction of cells would begin up slope from these cells and again on contour and head north again as before. The southern permanent collection channel would be extended eastward to contain the new cells. A new temporary rain water runoff diversion channel would be built up slope to divert rain runoff back into the natural drainage. This sequence would repeat itself until just before the crest of the hill to the east is reached, approximately 2900 contour.

In preparation of a new 300 section of paste tailing storage, to the north of the existing one, starting again at the 2650 contour, the westward permanent collection channel would be extended on contour to the north. The sequence described above will repeat.

The first section of paste tailing storage, when reclaimed, will have its surface contoured during reclamation in such a manner that rain running off its surface will be diverted from the new storage area under construction and into the natural drainage. The clean water diversion upslope and east of the reclaimed area will be reclaimed. The permanent collection channel to the west of the reclaimed tailing storage area would be reclaimed. As new sections of tailing storage are reclaimed this process will repeat itself. Planned geochemical and hydrological evaluations will determine if additional tailing storage reclamation activities would be required, such as inclusion of a membrane between the top of the paste tailing and the growth media cap.

18.4.3 Mine Access Road

The most probable mine access for construction and operation would be from the Lone Mountain Road illustrated on Figure 18-11 that runs southwest from the town of O'Brien to just north of the mine site. An existing BLM logging road connects the Lone Mountain Road to the mine site and JMC has the right to use this road by an existing agreement with the BLM. There would need to be some upgrades to both roads to make them suitable for the expected traffic of tractor trailers hauling concentrate out and supplies in. Road upgrades would be widening the road in places, improving drainage ditches, providing vehicle turn outs to allow traffic to pass, widening and lengthening turns into some of the bridges and perhaps improving one bridge. The roads and bridges have been used in the past by large logging trucks and during construction of the Pacific Corp. power line that is also visible on the map below.

Most employees would be picked up at a parking area in O'Brien by company operated vans and transported to the site and transported back out at the end of shift the same way. This arrangement would limit traffic on the road to the few large trucks mentioned above and to company owned pickups that management would travel in and the occasional salesperson's vehicle.

An alternate access from the east is also shown on the Figure 18-11. This road can be accessed from State Route 199 and is shorter in distance than the Lone Mountain Road described above. Significantly more improvements are required compared to the access described above and elevation changes are far greater.

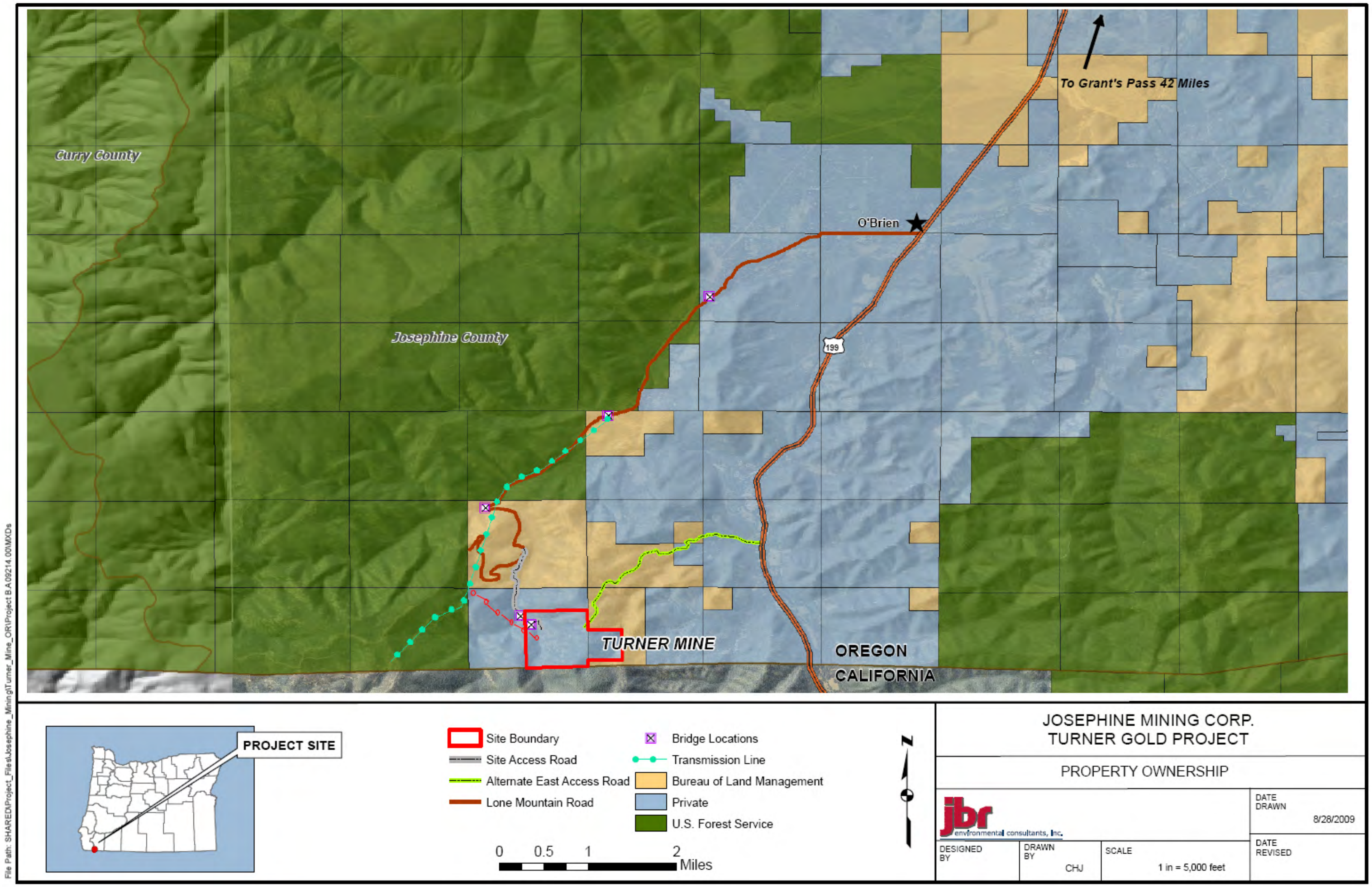


Figure 18-11: Access and Power Lines

18.4.4 Water Supply

Water supply will have three components; well water, tailing storage area run-off and water seepage from underground mine workings. The rainfall component is an intermittent source and therefore cannot be counted on for a continuous basis. Water supply from well water (one well operating and one on standby) will be designed to fulfill all requirements and include a 25 percent safety factor. Average continuous requirements are 81 gpm, thus with a 25 percent safety factor this becomes 101 gpm. Water for underground mine equipment is partially reduced because approximately 80% is recovered by sump pumps in the mine for re-use in the mill process. Table 18-12 shows the average water balance for the project.

Table 18-12: Project Water Balance

Water Inputs	**gpm**	**Water Outputs**	**gpm**
Rainfall	26	Underground Mine Equipment	80
Fresh Water from Wells	55	Potable Water Systems	5
Water Recovery from Underground Mine Usage	65	Retention in the TSF Solids	60
		Evaporation	1
Total	**145**	**Total**	**145**

The location of the two wells for water supply is shown in Figure 18-10. They are located in the northeast portion of the project area designated MW-9. There is a 0.75 million gallon overflow pond located in the mill site area that is used for collection of rain run-off on the TSF and mill site areas and for any overflows from thickener water tanks. As water is collected in this pond it will be pumped back into the tailing thickener. During these instances well water pumping will cease until the pond is returned to its emergency containment level.

Under Oregon law, water needs exceeding 5,000 gallons per day require a water right. Accordingly, JMC will be applying for a water right from the Oregon Water Rights Division (WRD) under Oregon Administrative Rules (OAR), Chapter 690, Division 310. Hydrogeologic data collected as part of the groundwater monitoring plan will be used to support the water rights application. It should be noted that in Oregon, a water right for mine dewatering is not needed. To assist in obtaining the water right, JMC has retained Ms. Martha Pagel, formerly the Director of the Oregon WRD, and currently practicing law with the firm Schwabe, Williamson, and Wyatt in Salem, Oregon.

18.4.5 Power Supply

Connected power load to the mine is approximately 3.0 MW (consumptive load 2.4 MW) placing the load as a Schedule 48 type load (non-residential greater than 1000 kW) in the Pacific Corp. schedule definitions for Oregon. Pacific Corp would supply power to the mine site by its line shown on Figure 18-11 which is 0.75 miles from the proposed mill and mine sites. A drop line to a transformer located on private ground right under the power line would be installed and a line routed to the mine substation; denoted on Figure 18-11 by the red circles. The transformer would change the Pacific Corp. line voltage of 115 kV to the mine supply voltage of 13.8 kV.

The estimated power cost per kWh obtained from Pacific Corp. tariff schedules dated October 1, 2009 was $0.045 for Schedule 48 load. Annual power costs are estimated at $1.36 million ($3.0/ton ore).

18.5 PEA Cash Flow Analysis

The Turner Gold project economics were performed using a discounted cash flow approach. Costs are in constant 2009 US dollars with no provisions for escalation. Annual cash flow projections were estimated over the life of the mine based on capital expenditures, production costs, and sales revenue.

This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized. IMC holds the opinion that the additional drilling as outlined in the recommendations chapter will add confidence, and could potentially add resources.

Economic Start Date and Life of the Project

The beginning date for the economic analysis is 3 years prior to start of production. The economic model's fiscal years are associated with mine's beginning of production (October 2012). For instance, Year 1 in economic model represents the time period from October 2012 to September 2013. Years prior to beginning of production are abbreviated as Year -1, Year -2, and Year -3. The project's financials returns are estimated in fourth quarter of 2009 (Year -3).

The life of the mine is 8 years for mining and milling operations.

Exchange Rate

All values are expressed in U.S. dollars unless otherwise noted. Although it is acknowledged that the project may be subject to future exchange rate risk, no attempt to account for this risk has been made in the analysis, primarily due to the inability to accurately forecast future exposures.

Date of Estimate

Estimates are in 3rd Quarter 2009 dollars for all equipment, materials, contract labor, and services.

Revenue

Annual revenue is determined by applying estimated gold, copper, zinc and silver prices to their respective production. Gold, copper, zinc and silver will be sold from the mine through custom concentrate contracts with smelters and refiners. Metal prices were selected based on recent long range price projections from producers, equity groups and lenders. Sensitivity analysis on the bottom of this section demonstrates financial returns at wide range of gold, copper, zinc, and silver prices.

Table 18-13: Base Case Metal Pricing

Gold/troy oz	$900
Copper/lb	$2.00
Zinc/lb	$0.65
Silver/troy oz	$12.50

Total revenue from the sale of metals over the life of mine is $440 million.

18.5.1 Capital Expenditures

Initial Capital

The total initial capital of new construction is estimated at $56.8 million. The initial capital is shown as a cash outflow prior to commissioning of the plant (see CapEx summary below). The base case financial indicators have been determined with 100% equity financing of the initial capital. Any acquisition cost or expenditures prior to October 2009 have been treated as “sunk” cost and have not been included in the analysis.

Sustaining Capital

Due to the short duration of the mining operation, 8 years, no sustaining capital will be scheduled.

Working Capital

Working capital for accounts receivables will vary by year depending on sales revenue with a delay of 60 days before receipt of payments. Working capital for plant consumable inventory is estimated in year 1 in the amount of $1.5 million. Working capital for accounts payable was based on operating cost assuming a 30 day account payable period. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

Salvage Value

No allowance for salvage value has been made at the end of the mine life.

18.5.2 Total Cash Cost

Operating Cost

The average Total Cash Operating Cost over the 8 years life of the mine (LOM) is estimated to be $62.32 per short ton of ore processed. The Total Cash Operating Cost includes mine operations, concentrator operations, concentrate shipping costs, custom smelter and refinery charges, selling costs, supporting facilities, and G&A.

Note: This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves and there is no certainty that this preliminary assessment will be realized.

Total Cash Cost

Total Cash Cost includes Total Operating Cost plus Royalties and Reclamation expenses. The average Total Cash Cost over the LOM is estimated to be $64.55 per short ton of ore processed.

Royalties and Acquisition Costs

An initial acquisition payment to GMI of $100,000 was made in June 2009. Two additional payments are due in the next 30 months from June 2009. A payment of $300,000 is due 18 months from June 2009 and a final payment of $1,600,000 is due when permits are issued or 30 months from June 2009, whichever is earlier. This cost is not included in the economic model. Property ownership would transfer to JMC at that time with the final payment.

Production royalties are owed to the Owner (GMI) through-out the life of the mine. The production royalty is calculated based on 1½ percent (1.5%) of the Net Smelter Returns. Net Smelter Returns is the gross amount received by JMC from any smelter, refinery, hydrometallurgical treatment facility, etc. for payment for the mineral products mined from the property and sold and delivered, less allowable deductions. Allowable deductions would be: shipping; sales taxes (sales, use, severance, etc.); purchaser's smelting, refining and other treatment charges or costs; and representation, assaying, and umpire costs and fees.

Total production royalty payments over the life of the mine are expected to be $5.7 million.

Reclamation

Most of the reclamation effort is on the surface storage for paste tailing. It is expected that concurrent reclamation will occur over the life of the mine for the tailing facility and during the year after mining stops. Reclamation of the mill site, infrastructure and mine adits will occur during the year after mining and milling are concluded. During the first year of operation the reclamation cost is estimated at $300,000 and $600,000 per year thereafter. During the year after operations cease reclamation costs are estimated at $2.3 million. The total reclamation cost over the project life is estimated at $6.8 million. The $4.5 million difference between the end of project reclamation costs and the total reclamation cost is included as a subset of the milling costs.

The base case assumes that JMC will be required to self-finance the initial reclamation bond in Year -2. The reclamation bond's value was estimated at $1.2M. This cost will be recovered the year after operation ceases.

18.5.3 Total Production Cost

Total Production Cost is the Total Cash Cost plus the depreciation:

Depreciation

Depreciation of all asset classes, except owners cost and underground development, is calculated by double declining balance depreciation over 8 years starting from the first year of production; last year was adjusted for a write-off. Starting from Q4 2010 (after estimated Decision to Proceed), the owners cost and the underground development cost were accounted for using 70/30 rule (70% of total costs were expensed the year incurred and the remaining 30% were amortized over 5 years).

Project Financing

It is assumed the project will be 100% equity financed.

Depletion

The percentage depletion method was used in the evaluation (gold, copper and silver at 15% and zinc at 22%). It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. The gross income from the property is defined as metal revenues minus downstream cost from the mining property (smelting, refining and transportation). Taxable income is defined as gross income minus operating expenses, overhead expenses, and depreciation.

Federal Income Tax

Taxable income for income tax purposes will be defined as mineral revenues minus operating expenses, royalty, reclamation and closure expense, depreciation and depletion. The income tax rate for federal taxes is 35%.

Net Income after Tax

Net Income after Tax amounts to $119 million over life of the mine.

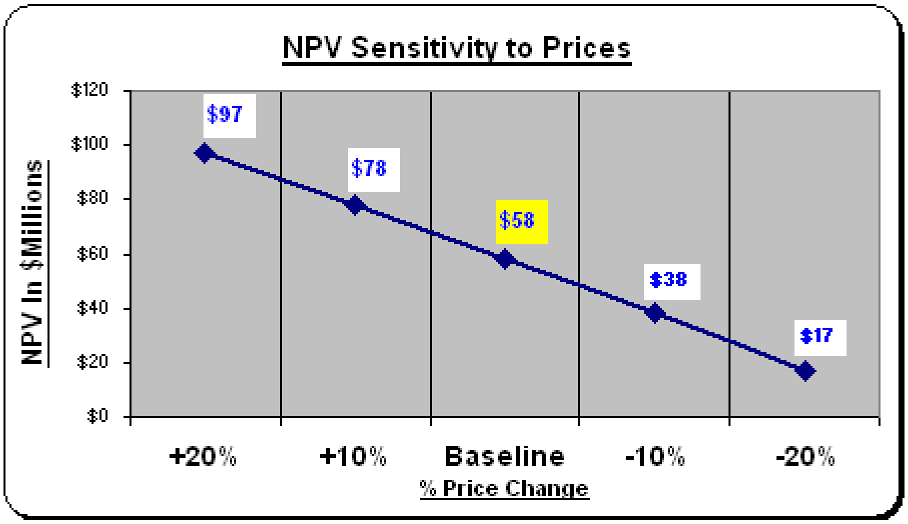
Net Present Value, Internal Rate of Return, Payback

The base case economic analysis (Table 18-14) indicates that the project's NPV at 8% discount rate is $58.5 million, Internal Rate of Return (IRR) of 32.2% and a payback period of 2.6 years from beginning of production.

18.5.4 Sensitivity Analysis

Price sensitivity to project's NPV is shown in the Figure 18-12 below. This chart illustrates different NPV values at the base case prices as well as +/-10% and +/-20% changes in metals' prices.

Figure 18-12 Price Sensitivity



The cost sensitivity charts below indicates that the NPV of the project is much less sensitive to the changes in both Initial Capital cost and Operating cost relative to price.

Figure 18-13 Initial Capital Cost Sensitivity

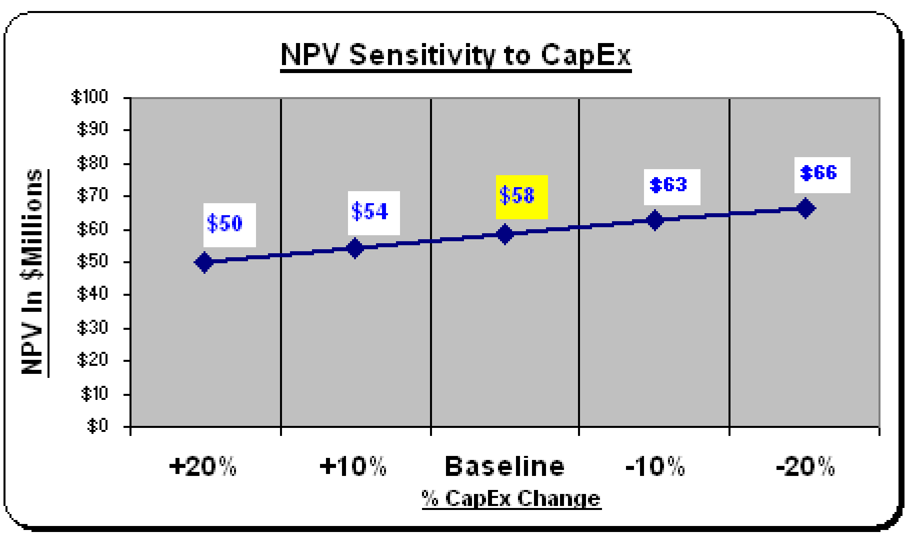


Figure 18-14 Operating Cost Sensitivity

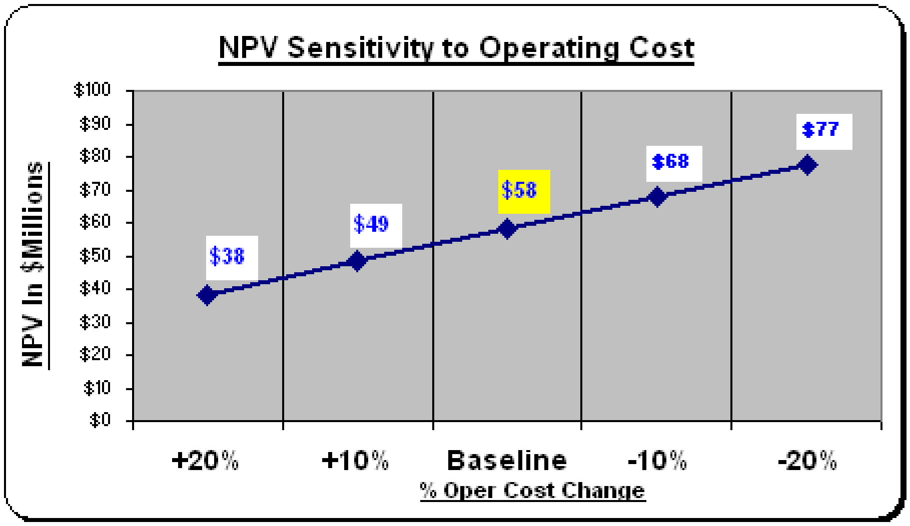


Table 18-14 follows and describes base case annual economics and projected after tax cash flows.

Table 18-14, Base Case Cash Flow Statement

Econ Analysis

11/11/2009

TURNER GOLD			-3	-2	-1	1	2	3	4	5	6	7	8	9	
	Starting Oct 1,		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	TOTAL
	Base Unit	Used Unit													
Volume															
Total Tons Mined (K, short tons)						421.2	455.0	455.0	455.0	455.0	455.0	455.0	434.5		3,585.8
Total Tons Mined (K, metric tons)						382.2	412.8	412.8	412.8	412.8	412.8	412.8	394.2	0.0	3,253.0
Total Tons Mined (M, metric tons)						0.382	0.413	0.413	0.413	0.413	0.413	0.413	0.394	0.000	3.253
Copper Grade						1.340%	1.220%	1.220%	1.220%	1.220%	1.220%	1.220%	1.100%		
Zinc Grade						2.480%	2.860%	2.860%	2.860%	2.860%	2.860%	2.860%	3.220%		
Gold Grade, oz/ short ton						0.0910	0.0890	0.0890	0.0890	0.0890	0.0890	0.0890	0.0860		
Gold Grade, oz/ metric tonne	0.09					0.100	0.098	0.098	0.098	0.098	0.098	0.098	0.095	0.000	
Silver Grade, oz/ short ton						0.520	0.500	0.500	0.500	0.500	0.500	0.500	0.480		
Silver Grade, oz/ metric tonne	0.52					0.573	0.551	0.551	0.551	0.551	0.551	0.551	0.529	0.000	
Total Tons Milled (m tons)						0.382	0.413	0.413	0.413	0.413	0.413	0.413	0.394	0.000	3.253
Copper Price per lb	$2.00					22.6	22.2	22.2	22.2	22.2	22.2	22.2	19.1	0.0	175
Zinc Price pr lb	$0.65					13.6	16.9	16.9	16.9	16.9	16.9	16.9	18.2	0.0	133
Gold Price per oz	$900.00					34.5	36.4	36.4	36.4	36.4	36.4	36.4	33.6	0.0	287
Silver Price per oz	$12.50					2.7	2.8	2.8	2.8	2.8	2.8	2.8	2.6	0.0	22
Gross Revenue, $Million						73.4	78.4	78.4	78.4	78.4	78.4	78.4	73.5	0.0	617
Equivalent Gold Grade (g/tonne)						6.64	6.56	6.56	6.56	6.56	6.56	6.56	6.45	0.00	
Copper Rec to Copper Con (%)	86.9%														
Gold Recovery to Copper Con (%)	52.6%														
Silver Rec to Copper Con (%)	30.9%														
Copper in Conc (k tons)					0	4.5	4.4	4.4	4.4	4.4	4.4	4.4	3.8	0.0	34
Copper Concentrate (k tons)	28.2%	28.2%			0	15.8	15.5	15.5	15.5	15.5	15.5	15.5	13.4	0.0	122
Zinc Rec to Zinc Con (%)	75.1%														
Gold Recovery to Zinc Con (%)	4.3%														
Silver Rec to Zinc Con (%)	34.7%														
Zinc Concentrate (ktons)					0	13.4	16.7	16.7	16.7	16.7	16.7	16.7	18.0	0.0	132
Zinc in Conc. (ktons)	53.0%					7.1	8.9	8.9	8.9	8.9	8.9	8.9	9.5	0.0	70
Zn Deduct (%)	0.0%					0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0
Payable Zn Recovery (%)	95.0%					95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	11
Payable Zinc (ktons)						6.8	8.4	8.4	8.4	8.4	8.4	8.4	9.1	0.0	66
Zinc RLE Cost (k$)	$0.226					$3,369	$4,196	$4,196	$4,196	$4,196	$4,196	$4,196	$4,512	$0	$33,059
Gold in Cu Conc. (oz)						20,163	21,300	21,300	21,300	21,300	21,300	21,300	19,656	-	167,621
Gold in Cu Conc. (g)	31.1					627,081	662,442	662,442	662,442	662,442	662,442	662,442	611,289	-	5,213,018
Deduct (g)	0					-	-	-	-	-	-	-	-	-	0
Payable Recovery (g)	97.0%					608,268	642,568	642,568	642,568	642,568	642,568	642,568	592,950	-	5,056,628
Payable Gold (oz)	31.1					19,558	20,661	20,661	20,661	20,661	20,661	20,661	19,066	-	162,593
Gold Refining Cost (K$)	$2.00					$ 39	$ 41	$ 41	$ 41	$ 41	$ 41	$ 41	$ 38	$ -	
Gold Insurance Cost (K$)	0.0%					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Silver in Cu Conc. (oz)						67,686	70,298	70,298	70,298	70,298	70,298	70,298	64,447	-	553,918
Silver in Cu Conc. (g)	31.1					2,105,029	2,186,252	2,186,252	2,186,252	2,186,252	2,186,252	2,186,252	2,004,296	-	17,226,838
Deduct (g)	0.00					-	-	-	-	-	-	-	-	-	0
Payable Recovery (g)	77.0%					1,620,872	1,683,414	1,683,414	1,683,414	1,683,414	1,683,414	1,683,414	1,543,308	-	13,264,665
Payable Silver (oz)	31.1					52,118	54,129	54,129	54,129	54,129	54,129	54,129	49,624	-	426,517
Silver Refining Cost (K$)	$0.15					$ 8	$ 8	$ 8	$ 8	$ 8	$ 8	$ 8	$ 7	$ -	
Gold in Zn Conc. (oz)						1,648	1,741	1,741	1,741	1,741	1,741	1,741	1,607	-	13,703
Gold in Zn Conc. (g)	31.1					51,263	54,154	54,154	54,154	54,154	54,154	54,154	49,972	-	426,159
Deduct (g)	1.0					13,429	16,728	16,728	16,728	16,728	16,728	16,728	17,985	-	131,782
Payable Recovery (g)	80.0%					30,267	29,941	29,941	29,941	29,941	29,941	29,941	25,589	-	235,502
Payable Gold (oz)	31.1					973	963	963	963	963	963	963	823	-	7,572
Gold Refining Cost (K$)	$0.00					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	

Note: This [illegible] is no certainty th[illegible]

Econ Analysis

11/11/2009

	Starting Oct 1,		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	TOTAL
	Base Unit	Used Unit													
Gold Insurance Cost (K$)	0.0%					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Silver in Zn Conc. (oz)						76,010	78,943	78,943	78,943	78,943	78,943	78,943	72,372	-	622,037
Silver in Zn Conc. (g)	31.1					2,363,899	2,455,112	2,455,112	2,455,112	2,455,112	2,455,112	2,455,112	2,250,779	-	19,345,349
Deduct (g)	93.3					1,252,949	1,560,714	1,560,714	1,560,714	1,560,714	1,560,714	1,560,714	1,678,045	-	12,295,278
Payable Recovery (g)	70%					777,665	626,078	626,078	626,078	626,078	626,078	626,078	400,914	-	4,935,050
Payable Silver (oz)	31.1					25,005	20,131	20,131	20,131	20,131	20,131	20,131	12,891	-	158,683
Silver Refining cost (K$)	$0.00					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Zn Con Shipping Cost($/ton)(K$)	$33.62					$ 502	$ 625	$ 625	$ 625	$ 625	$ 625	$ 625	$ 672	$ -	$4,922.8
Gold Rec to Gravity Gold Con 1 (%)	8%														
Gravity Gold Con 1(oz)						3,067	3,240	3,240	3,240	3,240	3,240	3,240	2,989	0	25,494
Gold Rec to Gravity Gold Con 2 (%)	7%														
Gravity Gold Con 2 (oz)						2,683	2,835	2,835	2,835	2,835	2,835	2,835	2,616	0	22,307
% Pay at Gold Refinery On Grav. Au	80.0%														
Payable Gold (oz)						4,600	4,859	4,859	4,859	4,859	4,859	4,859	4,484	0	38,241
Gold Refining Cost (K$)	$0.00					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Tons of Au Con						5,227	5,522	5,522	5,522	5,522	5,522	5,522	5,096	-	43,455
Shipping cost (K$)	$33.62	6.0%				$ 187	$ 198	$ 198	$ 198	$ 198	$ 198	$ 198	$ 182	$ -	1,554
Gold Insurance Cost (K$)	0.0%					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Silver Rec to Gold Con %	17.6%														
Silver in Au Conc. (oz)						38,552	40,040	40,040	40,040	40,040	40,040	40,040	36,708	0	315,500
Silver in Au Conc. (g)	31.1					1,198,981	1,245,244	1,245,244	1,245,244	1,245,244	1,245,244	1,245,244	1,141,606	-	9,812,050
Deduct (g)	-					-	-	-	-	-	-	-	-	-	0
Payable Recovery (g)	80%					959,185	996,195	996,195	996,195	996,195	996,195	996,195	913,284	-	7,849,640
Payable Silver (oz)	31.1					30,842	32,032	32,032	32,032	32,032	32,032	32,032	29,366	-	252,400
Silver Refining cost (K$)	$0.00					$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	0
Total Salable Silver(K oz)						108	106	106	106	106	106	106	92	0	838
Total Salable Gold (K oz)						25	26	26	26	26	26	26	24	0	208
Total Salable Zinc (Mlbs)						14.9	18.6	18.6	18.6	18.6	18.6	18.6	20.0	0.0	146
Smelter Cu Deduct, %, K tons	1%					0.045	0.044	0.044	0.044	0.044	0.044	0.044	0.038	0.000	0.345
Smelter payable Cu, %, K tons	96.5%					4.2	4.2	4.2	4.2	4.2	4.2	4.2	3.6	0.0	33.2
Smelter charge ($/ton of conc), K$	$81.16					$1,281	$1,259	$1,259	$1,259	$1,259	$1,259	$1,259	$1,084	$0	$9,922
% Moisture in Con	10%														
Shipping charge ($/ton conc), K$	$33.62					$589	$580	$580	$580	$580	$580	$580	$499	$0	$4,567
Refining, $/lb cu, K$	$0.070					$687	$675	$675	$675	$675	$675	$675	$581	$0	$5,320
Equivalent Copper Units (% Cu)						4.4%	4.3%	4.3%	4.3%	4.3%	4.3%	4.3%	4.2%	0.0%	0.3
Equivaluent Gold (K oz)						58.2	62.0	62.0	62.0	62.0	62.0	62.0	57.9	0.0	488
Equivaluent Copper Units (M Lb)						26.2	27.9	27.9	27.9	27.9	27.9	27.9	26.0	0.0	220
Revenue		1.0													
Zinc, $/lb, $Million	$0.65	$0.65				$10	$12	$12	$12	$12	$12	$12	$13	$0	$95
Silver, $/oz, $Million	$12.50	$12.50	payable	71.38%		$1	$1	$1	$1	$1	$1	$1	$1	$0	$11
Gold, $/oz, $Million	$900.00	$900.00	recovery			$23	$24	$24	$24	$24	$24	$24	$22	$0	$188
Copper, $/lb, $Million	$2.00	$2.00				$19	$19	$19	$19	$19	$19	$19	$16	$0	$146
Total						**$52**	**$56**	**$56**	**$56**	**$56**	**$56**	**$56**	**$52**	**$0**	**$440**
Value per ton						137	135	135	135	135	135	135	132	0	
Eq Oz gold per year	62,049														
Operating Cost	Per Short Ton	1.0													
Mining Costs	$27.45	$27.45				$11.6	$12.5	$12.5	$12.5	$12.5	$12.5	$12.5	$11.9	$0.0	$98
Milling Cost	$14.68	$14.68				$6.2	$6.7	$6.7	$6.7	$6.7	$6.7	$6.7	$6.4	$0.0	$53

Note: This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves and there is no certainty that this preliminary assessment will be realized.

Econ Analysis

11/11/2009

	Starting Oct 1,		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	TOTAL
	Base Unit	Used Unit													
Supporting Facilities (Lab)	$0.87	$0.87				$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.0	$3
G&A	$2.656	$2.66				$1.1	$1.2	$1.2	$1.2	$1.2	$1.2	$1.2	$1.2	$0.0	$10
Copper & Zinc Concentrates															
Zinc con shipping						$0.5	$0.6	$0.6	$0.6	$0.6	$0.6	$0.6	$0.7	$0.0	$5
Cu Con Shipping						$0.6	$0.6	$0.6	$0.6	$0.6	$0.6	$0.6	$0.5	$0.0	$5
Au Con Shipping						$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.0	$2
Smelter Charge						$1.3	$1.3	$1.3	$1.3	$1.3	$1.3	$1.3	$1.1	$0.0	$10
Copper Refining						$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.6	$0.0	$5
Gold Refining cost						$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0
Silver Refining Cost						$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0
Zinc RLE						$3.4	$4.2	$4.2	$4.2	$4.2	$4.2	$4.2	$4.5	$0.0	$33
Other						$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0
Total		$62.32		$0.0	$0.0	$25.9	$28.4	$28.4	$28.4	$28.4	$28.4	$28.4	$27.4	$0.0	$223.5
		1.0													
Gold & Silver Credit	0.0%	0.0%				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Operating Margin						**$26**	**$27**	**$27**	**$27**	**$27**	**$27**	**$27**	**$25**	**$0**	**$216**
GM%						51%	49%	49%	49%	49%	49%	49%	47%	0%	49%
		1.0													
Reclamation Cost	**$2.30**	**$2.30**				$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	**$2.3**	**$2.3**
		1.0													
Pre-prod mine dev & OC (70%)	$8.19	$8.19		**$1**	**$7**										**$8**
		1.0													
Production Royalty	1.5%	1.5%				($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	$0.0	($5.7)
Other	0.0%	0.0%				$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0
Total Royalty Cost	1.5%	1.5%				($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	($0.7)	$0.0	($5.7)
TOTAL CASH COSTS				**$1**	**$7**	**$27**	**$29**	**$29**	**$29**	**$29**	**$29**	**$29**	**$28**	**$2**	**$240**
Depreciation Basis	8.0	8.0				$34	$25	$19	$14	$11	$8	$6	($0)		
Depreciation Expense (w/o pre-prod-n)	$45.2	$45.2				($11)	($8)	($6)	($5)	($4)	($3)	($2)	($6)		**($45)**
Depreciation Exp (30% pre prod-n)	$3.5	$3.5				($0.7)	($0.7)	($0.7)	($0.7)	($0.7)					**($3.5)**
Total Depreciation	$48.7	$48.7		$0.0	$0.0	($12.0)	($9.2)	($7.1)	($5.5)	($4.3)	($2.7)	($2.0)	($6.0)	$0.0	**($48.7)**
		1.0													
Required Capital	$56.8	$56.8													
Interest Expense	0%	0%				$0	$0	$0	$0	$0					0%
Income Before Tax				**($1)**	**($7)**	**$14**	**$18**	**$20**	**$21**	**$22**	**$24**	**$25**	**$18**	**($2)**	**$151**
Depletion Basis	16%	16%				16%	17%	17%	17%	17%	17%	17%	17%		
Depletion at 50% Rule	50%	50%				$7	$9	$10	$11	$12	$12	$12	$9		**$83**
Depletion @ 15%, 22%						$7	$8	$8	$8	$8	$8	$8	$7		**$62**
Depletion Used for Tax Calc						$7	$8	$8	$8	$8	$8	$8	$7		
Tax Loss Carry Forward Applied	($8.2)			$0	($1)	($8)	($2)	$0	$0	$0	$0	$0	$0		
Tax Loss Carry Forward Balance						($2)									
		1.0													
Estimated Tax Expense	35%	35%		$0	$0	$0	($3)	($4)	($5)	($5)	($6)	($6)	($4)	$0	**($32)**
Net Income (M$)				**($1)**	**($7)**	**$14**	**$15**	**$16**	**$17**	**$17**	**$18**	**$19**	**$14**	**($2)**	**$119**
Add-back: Depreciation						$12	$9	$7	$5	$4	$3	$2	$6	$0	**$49**
Annual CapEx Allocation		1.0	3%	13%	77%	8%									
Less: Initial Capital (excl. Pre-develp)	$48.7	$48.7	($1.4)	($6.1)	($37.3)	($3.9)									($49)
Less: Pre-Strip		$0		$0.0	$0.0										$0
		1.0													

Note: This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves and there is no certainty that this preliminary assessment will be realized.

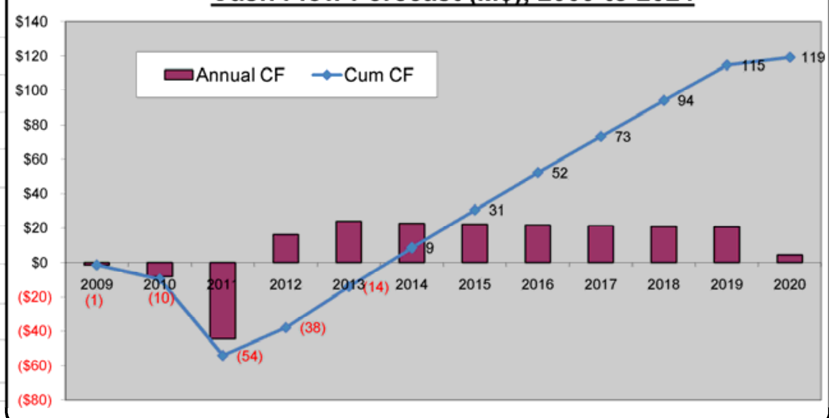
Econ Analysis

11/11/2009

	Starting Oct 1,		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	TOTAL
	Base Unit	Used Unit													
Less: Sustaining Capital	$0.00	$0.00				$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0
Change in Working Capital		1.0													
Account Receivable	45	45				($6)	($0)	$0	$0	$0	$0	$0	$0	$6	$0
Account Payable	30	30				$2	$0	$0	$0	$0	$0	$0	($0)	($2)	$0
Initial Inventory Build-up	$1.5	$1.5				($2)	$0	$0	$0	$0	$0	$0	$0	$2	$0
		1.0													
Reclamation Bond Requirement	$1.2	$1.2		($1.2)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1.2	$0
Annual CF Before Acquisition Cost (M$)			($1)	($8)	($45)	$16	$24	$23	$22	$22	$21	$21	$21	$5	$119
Cum CF Before Acquisition Cost			($1)	($10)	($54)	($38)	($14)	$9	$31	$52	$73	$94	$115	$119	$853
		1.0													
Less: Acquisition Cost (M$)	$0.0	$0.0	$0.0	$0.0	$0.0										
Annual CF After Acquisition Cost (M$)			($1)	($8)	($45)	$16	$24	$23	$22	$22	$21	$21	$21	$5	$119
Cum CF After Acquisition Cost			($1)	($10)	($54)	($38)	($14)	$9	$31	$52	$73	$94	$115	$119	
Payback Years						1.0	1.0	0.6	-	-	-	-	-	-	

Economic Results			2009	2010	2011
NPV (M$)	8%	8%	$58.5	$64.6	$78.7
IRR %			32.2%		
Payback Years			2.6		
PV Ratio (NPV / CapEx)			1.2		

Cash Flow Forecast (M$), 2009 to 2024



Note: This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves and there is no certainty that this preliminary assessment will be realized.

19.0 INTERPRETATION AND CONCLUSIONS

The estimation of resources and the development of a PEA were completed for Green Park Capital Corp. for the purpose of a qualifying transaction with the TSXV. The results of the PEA indicate that the Turner Gold project has the potential to become an economic producer of gold, copper, silver and zinc in the form of three concentrates for shipment to a copper smelter, zinc refinery and gold roaster or autoclave facility.

The historic drilling, geologic information, and recent check assay verification provide support for IMC to form the opinion that the data density and data reliability are sufficient to establish the estimate of mineral resources at Turner Gold as stated on Table 1-1 and Table 17-4.

There is potential to add resource tonnage to the Turner Gold deposit as there are significant areas, particularly in the lower zone (MLZ), where drilling has not found the limits of the mineralization. The additions could be in the range of 100,000's of tons.

Based on the known information provided to date, JBR sees no environmental issues that would prevent the permitting of the proposed operations. After review of the laws of the State of Oregon and the planned project, this project should apply under DOGAMI Division 35 Oregon Mined Land Reclamation Act. Although JBR currently does not see any permitting issues that would prevent the operation of the proposed Turner Gold Mine, JBR cannot predict all the concerns or issues the permitting agencies may have with the proposed project during the permitting process, nor can JBR control how long the agencies will take to issue the necessary permits. At this time, quantification of all the environmental impacts of the proposed facilities and operations is not possible. A better understanding of these will be developed during the permitting process.

There is potential to increase metal recoveries, particularly for precious metals, with newer technologies introduced to processing in recent years. Gravity concentration methods and non-cyanide leaching of gold and silver from copper sulfides, pyrite and arsenopyrite concentrates are a few processes of merit to investigate. Production of a separate cobalt/pyrite concentrate may also be practical given the advances in fine grinding methods in recent years.

This preliminary assessment includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized.

20.0 RECOMMENDATIONS

The results of the PEA indicate that the Turner Gold project has the potential to become an economic producer of gold, copper, silver and zinc in the form of three concentrates for shipment to a copper smelter, zinc refinery and gold roaster or autoclave facility. However, more information will be required to move the project forward to a prefeasibility study.

IMC recommends a step wise approach where additional information should be gathered and its resulting impact on the project evaluated prior to commitment to additional phases of work.

IMC has recommended an initial drill program of 12 diamond drill holes that will add confidence, and potentially add tonnage to the Turner Gold Deposit. These holes will provide information for a broad range of topics at Turner in addition to geology and assay information. The details of this drilling project are shown below.

Table 20-1: Recommended Drilling Project

Hole ID	Mine Grid Coordinates			Bearing Degrees	Plunge Degrees	Depth, ft	$Cost per Foot	Total $Cost	Target Description
	East, ft	North, ft	Elev., ft						
RM-1	20208.5	19305.8	2932.5	220	70	1000	$ 97	$ 97,000	1) Improve Confidence 2) Potentially Add Resources
RM-2	20208.5	19305.8	2932.5	220	80	1000	$ 97	$ 97,000	1) Improve Confidence 2) Potentially Add Resources
RM-3	20208.5	19305.8	2932.5	40	85	1200	$ 97	$ 116,000	1) Improve Confidence 2) Potentially Add Resources
RM-4	20031.7	19236.7	3054.1	vertical		1250	$ 97	$ 121,000	1) Improve Confidence 2) Potentially Add Resources
RM-5	20370.8	19239.9	2868.1	220	45	750	$ 97	$ 73,000	1) Improve Confidence 2) Potentially Add Resources
RM-6	20370.8	19239.9	2868.1	220	75	1100	$ 97	$ 107,000	1) Improve Confidence 2) Potentially Add Resources
RM-7	20370.8	19239.9	2868.1	vertical		1200	$ 97	$ 116,000	1) Improve Confidence 2) Potentially Add Resources
RM-8	20532.2	19278.5	2747.5	215	60	1000	$ 97	$ 97,000	1) Improve Confidence 2) Potentially Add Resources
RM-9	20532.2	19278.5	2747.5	215	75	1000	$ 97	$ 97,000	1) Improve Confidence 2) Potentially Add Resources
RM-10	20016.1	19078.7	3067.1	vertical		850	$ 97	$ 83,000	1) Potentially Add Resources
RM-11	20685.9	19257.4	2658.9	200	65	1000	$ 97	$ 97,000	1) Potentially Add Resources
RM-12	20685.9	19257.4	2658.9	200	80	1000	$ 97	$ 97,000	1) Potentially Add Resources
							Total	**$ 1,198,000**	

Acid rock characterization studies are planned on the drilling information outlined by JBR in Table 18-11. As a result of the ARD (acid rock drainage) testing, rates of weathering and mechanical breakdown for the pyrite-marcasite ores should be determined to address mine geotechnical concerns

Once the additional drill hole information is obtained, IMC holds the opinion that a three dimensional interpretation of rock type should be developed based on both old and new drilling data.

Process testing on new core should address the following items:

Additional flotation tests (lock cycle)
Freeze samples or limit oxidation of pyrite marcasite
Evaluate bulk flotation
A thorough study of regrind product size is required
Evaluate centrifugal gravity recovery of gold in a pyrite concentrate
Evaluate bulk concentrate processing by hydrometallurgical methods

The additional drilling should apply a highly accurate down hole survey method such as a Maxi-bore unit. Geotechnical data should be logged along with the geologic logging process. Some geotechnical testing will also be required on the new core.

The proposed budget for the additional drilling, analysis of the drill results and above mentioned studies is:$1.5 million USD. Josephine Mining Corp. currently plans to implement the drill program during the third quarter of 2010.

21.0 REFERENCES

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22.0 DATE AND AUTHORS CERTIFICATES

The Original Date of this Technical Report was: 16 November 2009.
The report was revised on 17 May 2010.

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

1. I am currently employed as the President and a Senior Mining Engineer by:

   Independent Mining Consultants, Inc.
   3560 E. Gas Road
   Tucson, Arizona, USA 85714

2. I graduated with the following degrees from the Colorado School of Mines
   Bachelors of Science, Mineral Engineering – Physics 1974
   Masters of Science, Mining Engineering 1976

3. I am a Registered Professional Mining Engineer in the State of Arizona USA
   Registration # 12772
   I am a Registered Professional Engineer in the State of Colorado USA
   Registration # 16191

   I am a Registered member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

4. I have worked as a Mining Engineer for a total of 34 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 1, 2, 3, 4, 14, 15, 17, 19, 20, and 21 of the report titled Technical Report, Turner Gold Resource and Preliminary Economic Assessment, dated 16 November 2009. I visited the Turner Gold project during the period of September 1 - 3, 2009.

7. Independent Mining Consultants, Inc., and this author have not worked on the Turner Gold Project prior to this report.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

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

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

Dated 20th day of May, 2010.

John M. Marek P.E.

CERTIFICATE OF QUALIFIED PERSON

I, James J. Moore hereby certify that:

1. I am currently engaged as a consultant on the Turner Gold Project. My address is:
   413 E. Avenida Sierra Madre
   Gilbert, AZ 85296

2. I graduated with the following degree from the Colorado School of Mines
   Bachelors of Science, Metallurgical Engineering – 1978

3. I am a Registered Professional Engineer in the State of Colorado USA
   Registration # 24529
   I am a professional member of AIME through SME.

4. I have worked as a Metallurgical Engineer for a total of 32 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 16, 18.2, 18.4.2, 18.4.3, 18.4.4, 18.4.5 and 18.51 of the report titled Technical Report, Turner Gold Resource and Preliminary Economic Assessment, dated 16 November 2009. I visited the Turner Gold project during the period of October 5-6, 2009.

7. Independent Mining Consultants, Inc., and this author have not worked on the Josephine project prior to this report.

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

9. I am not independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

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

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

Dated 19th day of May, 2010.

James J. Moore

James J. Moore, P.E.

Master Geotech Services LLC

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Phone: +1-520-762-6736
e-mail: srikant@mgstucson.com

CERTIFICATE OF QUALIFIED PERSON

I, Srikant Annavarapu P.E. do hereby certify that:

1. I am currently employed as the President and Geotechnical Engineer by:

   Master Geotech Services, LLC
   9193 N Desert Earth Place
   Tucson, Arizona, 85743 USA

2. I graduated with the following degrees

   Bachelors of Technology, Mining Engineering (1980) from Indian Institute of Technology, Kharagpur, West Bengal, India.

   Masters of Science, Mining Engineering (1998) from University of Arizona, Tucson, Arizona, USA.

3. I am a Registered Professional Geological Engineer in the State of Arizona USA Registration #36554

   I am a Registered member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

4. I have worked as a Mining Engineer for a total of 29 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 18.1 and 18.4.1 of the report titled Turner Gold Project NI43-101 Technical Report, Preliminary Economic Assessment, dated 16 November, 2009. I visited the Turner Gold project on 3 September, 2009.

7. Master Geotech Services LLC and this author have not worked on the Turner Gold project prior to this report.

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

9. I am independent of the issuer 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 to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

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

Dated 21st day of May, 2010.

Srikant Annavarapu P.E.

Registered Professional Engineer (Geological)
CERTIFICATE NO.
36554
SRIKANT
ANNAVARAPU
Date Signed
ARIZONA U.S.A.

Expires 09/30/2010

CERTIFICATE OF QUALIFIED PERSON

I, Michael D. Strickler, R.P.G. do hereby certify that:

1. I am currently the Single Member of:
   LithoLogic Resources, LLC
   P.O. Box 369, Selma, OR 97538

2. I graduated with the following degrees:
   Bachelor of Arts, Earth Science, California State University at Fullerton; 1975
   Master of Education (Geology), Southern Oregon University; 1996

3. I am a Registered Professional Geologist (#G869) in the State of Oregon, USA.

4. I have worked as a Professional Geologist for a total of 34 years since graduation.

5. I have read the definition of "qualified person" as set out in National Instrument 43-101 ("NI43-101"), and certify that by reason of my education and relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 5 through 13, inclusive, of the report titled Technical Report, Turner Gold Resource and Preliminary Economic Assessment, dated 16 November, 2009.

7. This author first became involved with the Turner Gold project during the American Selco program of 1975/1976, and has been involved with the majority of the exploration efforts since that time; including Baretta, Noranda, Rayrock, and others.

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

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

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

Dated 20th day of May, 2010.

Michael D. Strickler, R.P.G.

JBR

creating solutions for today's environment

CERTIFICATE OF QUALIFIED PERSON

I, Brian W. Buck, P.G., do hereby certify that:

1. I am currently employed as a Principal and Geologist by:
   JBR Environmental Consultants, Inc.
   8160 S. Highland Drive
   Sandy, UT 84093

2. I graduated with the following degrees:
   M.S., Geological Engineering, University of Utah, Salt Lake City, Utah, 1976
   B.S., Geology, University of Wisconsin, Madison, Wisconsin, 1973

3. I have the following Registrations/Certifications:
   Registered Professional Geologist, State of Utah, 2003
   Certified Environmental Manager, State of Nevada, 1994
   Registered Professional Geologist, State of Wyoming, 1992
   Registered Environmental Assessor, State of California, 1989

4. I have worked as a Geologist and Environmental Professional for a total of 33 years since 1976.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 18.3.1, 18.3.2, 18.3.3, 18.3.4, and 18.3.5 of the report titled *Technical Report, Turner Gold Resource and Preliminary Economic Assessment*, dated November 16, 2009. I visited the Josephine project site on June 3, 2009.

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

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

9. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and to my knowledge, the environmental studies and permitting information in the Technical Report have been prepared in compliance with that instrument and form.

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

Dated 21st day of May, 2010.

Brian W. Buck, P.G.