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INTERNATIONAL MINERALS CORPORATION

NI 43-101 TECHNICAL REPORT

"UPDATE TO FEASIBILITY STUDY ON THE GOLDFIELD PROPERTY, NEVADA, USA"

Effective Date of Resource Estimate and Update to Feasibility Study: 17 June, 2013
Signature Date: 25 July, 2013

Sam Shoemaker Jr., Reg. Mem. SME.
Richard Gowans, P.Eng.
Christopher Jacobs, CEng, MIMMM.
R. Mohan Srivastava, P.Geo.

SUITE 900 - 390 BAY STREET, TORONTO ONTARIO, CANADA M5H 2Y2
Telephone (1) (416) 362-5135 Fax (1) (416) 362 5763

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1.0 SUMMARY

The Goldfield property (the Property) is located in the historic Goldfield Mining District of Goldfield, Nevada, USA (the Goldfield Mining District) approximately 30 miles south of Tonopah, Nevada, adjacent to the town of Goldfield in Esmeralda County. The Property hosts three currently-known gold deposits: Goldfield Main, McMahon Ridge and Gemfield. The Property is 100%-owned by International Minerals Corporation (IMZ), a Yukon Territory registered company based in Scottsdale, Arizona and listed on the Toronto (TSX) and Swiss (SIX) stock exchanges (trading symbol: IMZ).

The Gemfield Project (the Project) currently comprises the development of Gemfield as an open-pit heap leach operation. IMZ completed a Feasibility Study on the potential production of gold and silver from the Gemfield deposit in 2012. Subsequently, updates were made to the feasibility study in 2013 and Micon International Limited (Micon) has been retained to prepare a Technical Report in accordance with Canadian National Instrument 43-101 (NI 43-101) to support the disclosure of the results of the updates to the Feasibility Study (the Updated Study).

The Updated Study is based on the proposed open-pit mining and heap leach processing of the mineral reserves of the Project at a rate of 8,250 dry short tons per day (T/d) to produce a gold and silver doré product on-site. The updated mineral resource estimate on which the Updated Study is based is disclosed in this report and was prepared by R. Mohan Srivastava, P.Geo., an independent consultant.

The Qualified Persons (QP) who prepared this NI 43-101 Technical Report are listed below. All of the QPs are Independent of IMZ:

- R. Mohan Srivastava, P.Geo., independent consultant.
- Sam Shoemaker Reg. Mem. SME, Micon International Limited.
- Richard Gowans, P.Eng., Micon International Limited.
- Christopher Jacobs, CEng, MIMMM., Micon International Limited.

All currency amounts are stated in United States dollars ($).

Richard Gowans is responsible for preparing and supervising the preparation of the Technical Report.

1.1 PROPERTY DESCRIPTION

The Property is located approximately 30 miles south of Tonopah, Nevada and 174 miles northwest of Las Vegas, Nevada, adjacent to the town of Goldfield. The majority of the Property is within Esmeralda County, with the remaining portion located in Nye County. The site location is shown in Figure 1.1.

Figure 1.1
Goldfield Property Site Location

US Highway 95 runs (north-south) through the western portion of the Property with the majority of IMZ's land holdings located to the east of the highway. A major geographic feature on the Property is Columbia Mountain, located in the west-central part of the Property.

The Property is located within portions of 56 sections of land within various Townships and Ranges and comprises 568 patented and 1,017 unpatented mining claims, either owned or controlled by IMZ.

The Goldfield Mining District was the site of intensive mining after the initial discovery in 1902 until 1919, when the Goldfield Consolidated Mill closed. Since that time, operations have been sporadic and relatively small in scale. Remnants of the early mining activities include numerous head frames, mine shafts and waste-rock dumps widely scattered around the Property.

1.2 HISTORY

Goldfield was one of the world's great gold camps and one of the last historic gold rushes in the western USA. Although typical of many of the Nevada "boom and bust" gold camps, what made Goldfield unique was the extremely high-grade nature of the material mined. From 1906 to 1910 Goldfield was the largest city in Nevada.

Modern gold production has been confined to the Goldfield Main area, extending from the southern part of Columbia Mountain in the north to the Red King Shaft located approximately one mile to the south. Production figures during this time are incomplete; however the Nevada Bureau of Mines reports 28,400 ounces of gold were produced during this period.

Property ownership in the Goldfield Mining District has historically been complicated and fragmented. However, over the past decade and a half, IMZ and its predecessors have been successful in consolidating the largest land position ever held by a single owner in the Goldfield Mining District.

The list of companies and individuals involved in the Goldfield Mining District is extensive. A partial list of the companies that have done work at Goldfield since the 1970s includes Cordex, Hanna Mining, Noranda, Utah International, Cyprus, Southern Pacific Land, Blackhawk Mines, Westley Mines, Transwestern Mining, Newmont, Meridian, Echo Bay, AMAX, Dexter, American Pacific, Red Rock, Crown, Santa Fe, American Resources, Kennecott, Cameco, Rea Gold, North, Romarco, Geochem Mines, Lode Star Gold, and Metallic Ventures.

On February 26, 2010, IMZ acquired the Canadian public company, Metallic Ventures Gold Inc., including its three Nevada operating subsidiaries: Metallic Ventures (U.S.) Inc., (with its royalty interest in Barrick's Ruby Hill gold mine in central Nevada); Metallic Nevada Inc. (with its Converse gold property in north-central Nevada); and Metallic Goldfield Inc., which controlled the majority of the historic Goldfield Mining District.

1.3 GEOLOGY AND MINERALIZATION

The oldest known rock unit found in the Goldfield Mining District is the Ordovician Palmetto Formation. The Palmetto Formation mainly consists of black siliceous shale and argillite, but also contains minor amounts of limestone. Jurassic granitic to granodioritic batholithic rocks intrude the Palmetto Formation. These rocks have been dated at 170 million years old (Ma).

During Oligocene to Early Miocene times, for a period of approximately 14 million years, the Goldfield mining district became a centre of volcanic activity, hydrothermal alteration and gold deposition. By the Late Miocene, volcanic activity in the district had diminished, with volcanism in the Goldfield Mining District limited to minor amounts of tuff deposited in

a subsiding, lacustrine environment. By the end of the Miocene, flood basalts were extruded over the region

Different models for the origins of the structures in the Goldfield Mining District have been proposed. One of the earliest proposed a re-activated caldera model whereby caldera collapse during the Early Miocene resulted in the creation of a "ring structure", followed by left-lateral movement along northeast trending faults (Cross Fault system), the latter movement resulted in the creation of low-angle (shingle) faults, one of which is the main mineralized structures at the Goldfield Main deposit.

The ore bodies in the Goldfield Mining District generally occur within silicified hydrothermal alteration zones. Both historically and presently, these zones are often referred to as "ledges". The siliceous ledges were created during multiple hydrothermal alteration events. The events began with an early "acid" event (pH<2) which resulted in the partial to pervasive dissolution of the host rock, creating a "vuggy" host. Progressively less acid solutions later flooded the surrounding rock and deposited quartz together with alunite, barite and pyrite (pH =3-4).

The Gemfield, McMahon Ridge, and Goldfield Main deposits are structurally controlled, volcanic-hosted, epithermal gold deposits of the high-sulfidation, quartz-alunite type. Other examples of the deposit type include Paradise Peak (Nevada, USA), Summitville (Colorado, USA), Pierina and Yanacocha (Peru), Nansatsu (Japan), El Indio (Chile), Temora (New South Wales, Australia), Pueblo Viejo (Dominican Republic), Chinkuashih (Taiwan), Rodalquilar (Spain), Lepanto and Nalesbitan (Philippines).

1.4 IMZ EXPLORATION

Drilling using both reverse circulation (RC) and core drilling methods started in May 2010 and continued until June 2013. As of the date of this report, 541 RC drill-holes totaling 335,645 ft and 77 HQ core holes totaling 46,657 ft have been completed. Table 1.1 shows the breakdown for the areas drilled at the Property.

Table 1.1
Summary of IMZ Drilling Completed in the Goldfield Mining District

Area	RC Holes	Footage (ft)	Core Holes	Footage (ft)	Total Holes	Total Footage (ft)
Goldfield Main	174	146,895	15	10,463	189	157,358
Gemfield	214	102,250	54	33,083	268	135,333
McMahon	26	13,750	8	3,112	34	16,862
Reconnaissance	105	60,715	0	0	105	60,715
Monitor Well	22	12,035	0	0	22	12,035
Total	**541**	**335,645**	**77**	**46,657**	**618**	**382,302**

Initially, RC drilling was started in the main Goldfield Main deposit, concentrating on the shallow east-dipping Columbia Mtn.-Red Top fault. Close to half of all of the recorded

historic production of 4 million ounces of gold has been produced from this shallow-dipping structure. IMZ drilling shows that lower-grade gold mineralization, not of interest to past operators, continues at depth from the historic surface open pits to the east for a distance of approximately 1,500 ft and along strike for 3,000 ft.

Drilling at the Gemfield deposit was concentrated around the edges of the known Gemfield resource. Drilling on the west side of Gemfield, in the deeper extension of the deposit, did intercept gold mineralization. However, much of this mineralization was sulfide and refractory in nature. Drill programs on the southeast side of Gemfield have encountered zones of low grade (0.3-0.4 ppm gold) oxidized Sandstorm Rhyolite. To date no mineral resource has been estimated for these possible extensions.

At McMahon Ridge, drilling was undertaken to provide samples for metallurgical studies and pit slope geotechnical assessment.

A number of reconnaissance exploration targets were drill tested. The most encouraging results came from the Florence mine area at Goldfield Main. Strongly anomalous gold grades averaging around 0.5 ppm over widths of 100 ft were encountered.

Detail geologic mapping, soil sampling over large portions of the Goldfield District along with PIMA clay alteration studies continue at this time to help define future drill targets in the district.

1.5 MINERAL RESOURCE ESTIMATE

The mineral resource estimate for the Gemfield deposit has been updated from 3D block models in which grades have been interpolated from drill-hole data using ordinary kriging within separate geological and statistical domains.

Table 1.2 summarizes the current estimate of the mineral resources at the Gemfield deposit at a cut-off grade of 0.25 g/t gold (Au).

Table 1.2
Mineral Resource Estimate for the Gemfield Deposit, at a Cut-off Grade of 0.25 g/t Au (June 17, 2013)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	15.50	1.05	524,000
Indicated	9.10	0.54	157,000
Measured + Indicated	**24.59**	**0.86**	**681,000**
Inferred	1.08	0.52	18,000

Notes:

1. *Numbers are rounded to reflect the precision of a resource estimate.*
2. *The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.*

The resource estimates for the Goldfield Main and the McMahon Ridge deposits have not been updated. For these two deposits, the resources reported in the 43-101 report filed on SEDAR on August 31st, 2012 (with an effective date of July 17th, 2012) remain current.

1.6 MINERAL RESERVE ESTIMATE

As of June 17, 2013, the mineral reserves for the Gemfield deposit are summarized in Table 1.3 and Table 1.4. There are no mineral reserves estimated for the Goldfield Main and the McMahon Ridge deposits. These mineral reserves are included within the mineral resources summarized in Section 1.5 of this report.

Table 1.3
Mineral Reserves for the Gemfield Deposit (June 17, 2013), Metric Units

Reserve Category	Tonnes	Au (g/t)	Contained Ounces
Proven	13,751,000	1.1	493,000
Probable	3,509,000	0.7	74,000
Total Proven + Probable Reserves	**17,260,000**	**1.0**	**567,000**

Table 1.4
Mineral Reserves for the Gemfield Deposit (June 17, 2013), Imperial Units

Reserve Category	Tons	Au (oz/ton)	Contained Ounces
Proven	15,158,000	0.0325	493,000
Probable	3,868,000	0.0191	74,000
Total Proven + Probable Reserves	**19,026,000**	**0.0298**	**567,000**

Notes

1. *Numbers are rounded to reflect the precision of the estimate*
2. *The Mineral reserves were estimated using a gold price of $1,450 per ounce with an average cut-off grade of 0.0066 oz/ton (approximately 0.25 g/t).*
3. *Micon is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant factors that will materially affect the validity of this reserve estimate.*
4. *CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) define a 'Proven Mineral Reserve as "the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study", and define a 'Probable Mineral Reserve' as the "the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study."*
5. *These estimated ore reserves have an effective date of June 17, 2013.*

1.7 MINING

The Updated Study assumes only the development of the Gemfield deposit at this time.

Standard mining technology will be utilized to create an open pit with approximate dimensions of 3,000 feet north-south by 1,900 feet east-west and a maximum depth of 550 feet below current ground level. A second smaller open pit is also developed with approximate dimensions of 1,000 feet north-south by 1,100 feet east-west and a maximum depth of 220 feet below current ground level. The mine was evaluated assuming that IMZ will purchase and operate its own mining fleet with a maintenance and repair contract (MARC) on all mobile equipment.

Micon was retained by IMZ to complete new pit optimizations in order to determine the optimal economic pit shape for the new Gemfield mineral resource model. A number of scenarios were considered and completed in order to determine the final conceptual pit shell that was ultimately selected. This pit shell was then used for feasibility-level detailed phased pit designs and production scheduling.

A mine production schedule was prepared using Maptek's Chronos scheduling software. Ore selection was based on a fixed cut-off of 0.0066 ounces per ton (oz/T) gold (0.25 g/t gold) and calculated using updated mine optimization parameters.

Four pit shells selected from the Whittle optimization were used to guide the design of five distinct pit phases. Three phases were developed in the Gemfield Main pit while two phases are developed in the Gemfield Southeast pit. Primary access for each pit phase was designed from the north to minimize haul cycle times. The tonnage and grade tabulations by pit phase are summarized in Table 1.5.

Table 1.5
Gemfield Mineral Reserve Summary by Pit Phase

Pit	Phase	Proven + Probable Ore Tons			QAL	Waste Rock	Total	Stripping
		Tons	Au oz/T	Rec'd Au oz	Tons	Tons	Tons	Ratio
Main	1	2,630,000	0.0398	86,500	705,000	2,149,000	5,484,000	1.09
Main	2	6,146,000	0.0333	175,200	1,532,000	10,989,000	18,667,000	2.04
Main	3	8,464,000	0.0255	186,400	1,213,000	19,987,000	29,664,000	2.50
Main	**All**	**17,240,000**	**0.0305**	**448,100**	**3,450,000**	**33,125,000**	**53,815,000**	**2.12**
Southeast	1	352,000	0.0385	10,900	321,000	688,000	1,361,000	2.87
Southeast	2	1,434,000	0.0197	24,300	505,000	2,675,000	4,614,000	2.22
Southeast	**All**	**1,786,000**	**0.0234**	**35,200**	**826,000**	**3,363,000**	**5,975,000**	**2.35**
Total	**All**	**19,026,000**	**0.0298**	**483,300**	**4,276,000**	**36,488,000**	**59,790,000**	**2.14**

1.8 METALLURGICAL TESTING AND PROCESSING

Various sources of information address mineral processing and metallurgical testing at the Property. They include:

- Results from historical testing by previous owners.

- A metallurgical overview report dated April 1995 by American Resource Corporation (ARC) which discusses their heap leach operations treating feed from Goldfield Main in the early to mid-1990s.

- Results from testing by IMZ.

Based on all of the information available it has been decided to focus on the development of Gemfield as a heap leach operation. Metallurgical sampling and testing has been carried out on material selected from core-holes that are considered to be well distributed throughout, and spatially representative of, the Gemfield deposit and it has been concluded that:

- Gemfield is amenable to heap leaching but mining should be limited to the oxidized rhyolite (i.e. from above the redox boundary established in geological modeling).

- The Gemfield mineralization is not particularly hard but becomes quite abrasive in zones of higher silicification.

- At Gemfield the highest gold grades are found in the highly silicified zones ("ledges") which require finer crushing to ensure adequate gold particle liberation. Conversely, at a constant crush size, recovery declines with increased silicification (i.e. increased gold grade).

- Gold recovery at grades typically anticipated from the Gemfield pit (nominal cut off and average grades of 0.01 and 0.03 oz/T respectively) are relatively insensitive to crush sizes between 100% passing 0.5 inches and possibly as high as 100% passing 2 inches.

- 100% passing 1.0 inch leach feed has been selected for design purposes. At this crush size, ultimate gold recoveries in column tests at grades generally anticipated from the Gemfield pit ranged between the low 70%'s and the mid-90%'s. Load permeability testing on column tails indicates that at 100% passing 1.0 inch the ore could be stacked on the leach pad up to 300 ft compared to the 200 ft considered for engineering design.

The model proposed for gold recovery versus head grade has been used for mine modeling to develop pit production schedules and economic analysis of the Project and results in a life-of-mine (LOM) average metallurgical gold recovery of 85.2%.

The process flowsheet comprises three-stage crushing and screening, heap leaching, carbon adsorption, stripping and regeneration followed by electrowinning and smelting to produce doré bars (gold and silver with minor impurities) for shipment offsite to third party refiners. Crushing and pad loading will operate on day shift only 12 hours per day, seven days per week. Leaching and gold recovery will operate on two shifts, 24 hours a day, seven days a week.

1.9 PROJECT INFRASTRUCTURE

Onsite ancillary facilities include various infrastructure buildings, power supply, fuel storage and distribution, water storage and distribution and sanitary sewage systems. Offsite ancillary facilities include realignment to the west of approximately 2.5 miles of US Highway 95.

1.10 ENVIRONMENTAL STUDIES

In general the environmental liabilities attached to the Goldfield land package are those associated with working within a historic mining district.

The environmental study work undertaken so far includes development of the Environmental Design Criteria, Environmental Baseline Studies, and initial permitting for the Project.

The Environmental Design Criteria was prepared as a technical memorandum. This document defines the environmental aspects relating to soil, water, air and noise that will be considered and implemented during design, development and closure of the proposed Project.

An environmental baseline study was undertaken in 2003/2004 on the Project area. An additional baseline study program was implemented in 2012 and 2013 to collect environmental, natural resource and socio-economic data required to support the completion of the federal and state agency permitting and approval program, and the environmental documentation process that will be required under National Environmental Policy Act (NEPA) for the proposed mining project.

Mine closure and reclamation for the project will be conducted in accordance with BLM and State of Nevada regulations and guidelines and best management practices. The closure and reclamation plan for the Project will include the open pit, waste rock dump and heap leach pad and other ancillary facilities.

The permitting strategy identifies and addresses the various local, state, federal and international environmental and social requirements and standards applicable to the Project.

1.11 CAPITAL COSTS

Capital expenditures and capitalized pre-production operating costs for the Project are summarized as initial and sustaining costs in Table 1.6. The estimates are expressed in first quarter 2013 US dollars, without escalation. The expected accuracy of the estimates is ±15%.

Table 1.6
Capital Cost Summary

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Permitting	600	-
Mining	20,253	4,825
Processing & Infrastructure	71,391	8,376
Indirect Costs	10,290	-752
Owner's Costs	34,332	-
Contingency	14,151	193
Total	**151,017**	**12,642**

Pre-production capital expenditures are estimated to total $151.0 million, including $20.3 million for mining, $71.4 million for processing and infrastructure, $10.3 million for indirect costs, and $34.3 million for owner's costs (including $20.9 million for realignment of US Highway 95), and contingencies totaling $14.1 million.

1.12 OPERATING COSTS

A summary of the life-of-mine (LOM) operating costs is presented in Table 1.7.

Table 1.7
Summary of Life-of-Mine Operating Costs

Area	Life-of-mine Cost ($ 000)	Unit Cost $/Ton ore treated	Unit Cost $/tonne ore treated
Permitting cost (pre-production total)	2,000	0.11	0.12
Permitting cost (30% capitalized)	(600)	-0.03	-0.03
Sub-total Permitting cost (expensed)	**1,400**	**0.07**	**0.08**
Drilling and Blasting	28,128	1.48	1.63
Loading and Hauling	56,568	2.97	3.28
Support Equipment	24,569	1.29	1.42
Mine G & A	9,124	0.48	0.53
Less Capitalized OPEX	(3,686)	-0.19	-0.21
Sub-total Mining	**114,703**	**6.03**	**6.65**
Operating Labor	21,173	1.11	1.23
Maintenance Labor	8,522	0.45	0.49
Power	10,773	0.57	0.62
Reagents and Consumables	43,351	2.28	2.51
Maintenance Spares	6,548	0.34	0.38
Truck Haulage to Leach Pad	5,560	0.29	0.32
Loaders and Dozers	3,284	0.17	0.19
Sub-total Processing	**99,210**	**5.21**	**5.75**
Labor	31,654	0.83	0.92
Power	2,183	0.14	0.15
General Expenses	8,955	0.56	0.62
Maintenance materials	6,452	0.20	0.22
Sub-total General and Administrative	**49,244**	**1.73**	**1.91**
Total Operating Costs	**248,214**	**13.05**	**14.38**

The LOM average unit operating cost totals $13.05/T leached over the LOM period, comprising $6.03/T for mining, $5.21/T for processing, and $1.80/T for general and administrative (G&A) costs, including permitting and environmental expenses. Adding bullion transport and refining costs gives an estimated cash operating cost of $524/oz gold produced. The net smelter return (NSR) royalty payable to third parties and the Nevada Net Proceeds of Minerals (NPOM) tax bring total cash cost estimates to $612/oz of gold produced.

1.13 ECONOMIC ANALYSIS

Micon has prepared its assessment of the Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined.

Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, first quarter 2013 dollars, i.e. without provision for escalation or inflation.

The 3-year trailing average price at the end of May, 2013 for gold was $1,546/oz, however, in the economic evaluation, a more conservative price of $1,350/oz was selected, being close to the spot price prevailing at the time of writing of this report. This price was then applied consistently throughout the operating period of the LOM.

US federal income tax and NPOM tax have been allowed for in the cash flow model. Regular and alternative minimum tax (AMT) were taken into consideration for federal tax.

Depreciation is applied on a unit of production basis and a 15% depletion allowance is taken where applicable. Net operating loss carry forwards related to the Property of $36.5 million are also taken into account.

A 5% NSR royalty, payable to third parties, has been provided for in the cash flow model on 100% of the production from Gemfield.

The LOM base case project cash flow is presented in Figure 1.2 and Table 1.8.

Figure 1.2
Life-of-Mine Cash Flows

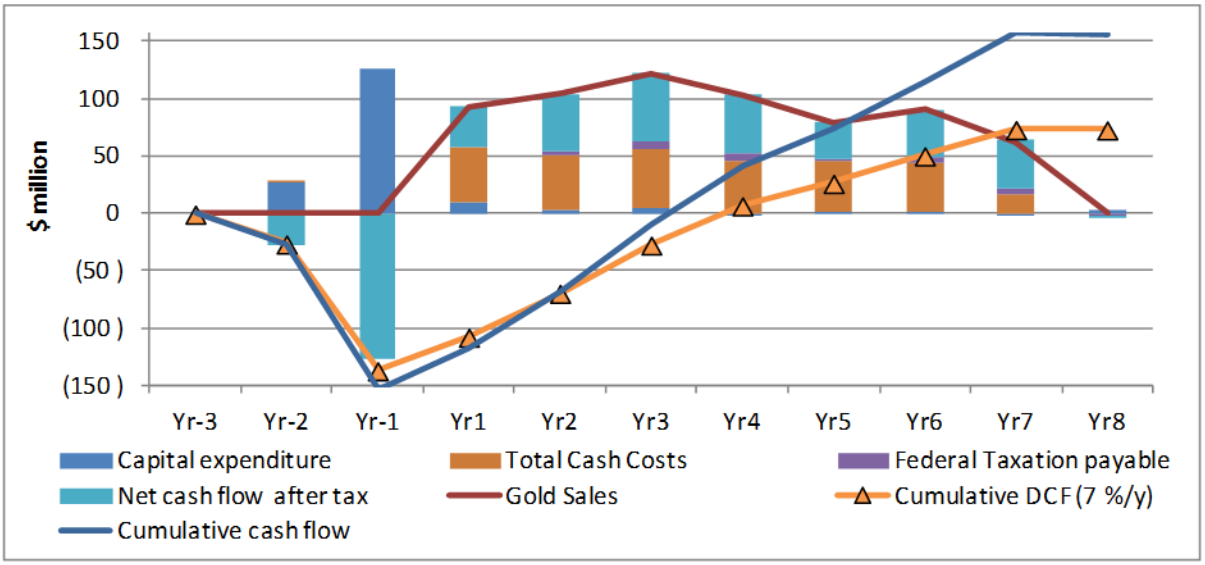


Table 1.8
Life-of-Mine Cash Flow Summary

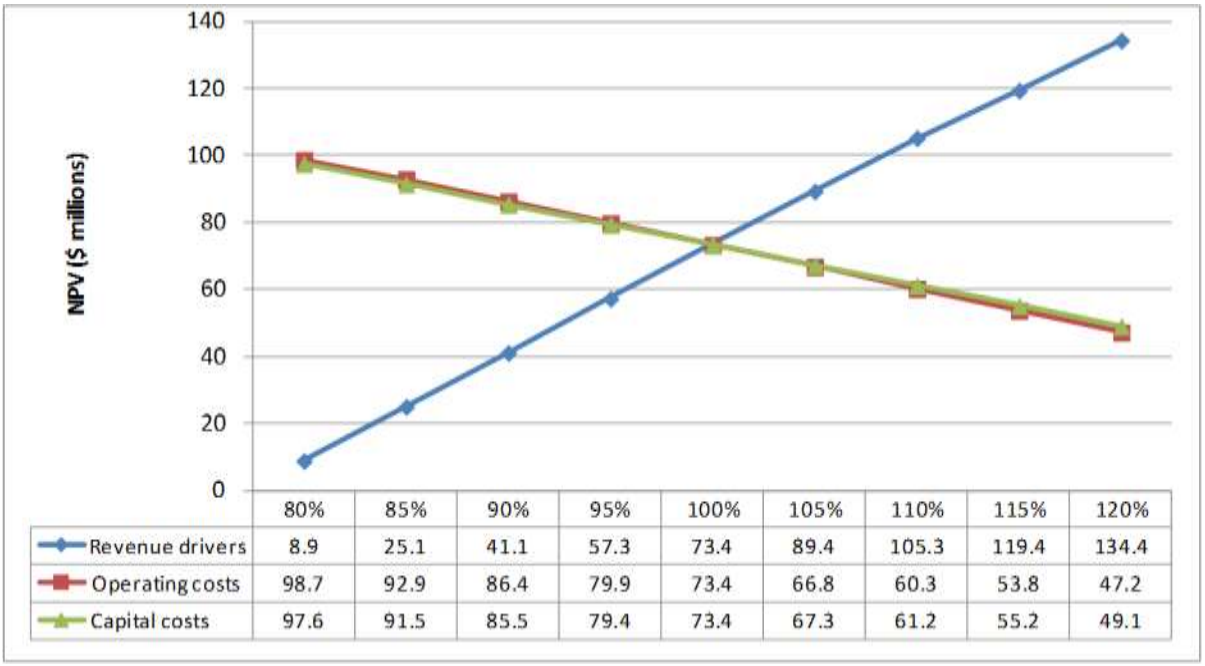
	LOM total ($ 000)	$/Ton Processed	$/tonne Processed	$/oz Au Recovered
Gold Revenue	**651,824**	**34.26**	**37.77**	**1350.00**
Permitting & Environmental	1,400	0.07	0.08	2.90
Mining costs	114,703	6.03	6.65	237.56
Processing costs	99,210	5.21	5.75	205.48
General & Administrative costs	32,901	1.73	1.91	68.14
Direct site operating costs	**248,214**	**13.05**	**14.38**	**514.08**
Silver credit	-	-	-	-
Bullion delivery	1,640	0.09	0.10	3.40
Refining charges	2,414	0.13	0.14	5.00
Insurance	652	0.03	0.04	1.35
Cash operating costs	**252,920**	**13.29**	**14.65**	**523.83**
Nevada NPOM	10,067	0.53	0.58	20.85
NSR Royalty	32,356	1.70	1.87	67.01
Total Cash Costs	**295,343**	**15.52**	**17.11**	**611.69**
EBITDA	**356,481**	**18.74**	**20.65**	**738.31**
Capital expenditure	171,597	9.02	9.94	355.40
Net cash flow (before tax)	**184,884**	**9.72**	**10.71**	**382.92**
Federal Taxation	28,854	1.52	1.67	59.76
Net cash flow (after tax)	**156,030**	**8.20**	**9.04**	**323.16**

The sensitivity of the project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs

was tested over a range of 20% above and below base case values. The results show that the project is most sensitive to revenue factors, with an adverse change of 20% reducing the Net Present Value at a 7% discount rate (NPV_7) from $73.4 million to $8.9 million. The impact of changes in capital and operating costs are almost identical: adverse changes of 20% reduce NPV_7 to $49.1 million and $47.2 million, respectively.

In Micon's analysis, applying an increase of more than 30% in both capital and operating costs simultaneously would be required to reduce NPV_7 to below zero. Figure 1.3 shows the results of changes in each factor separately.

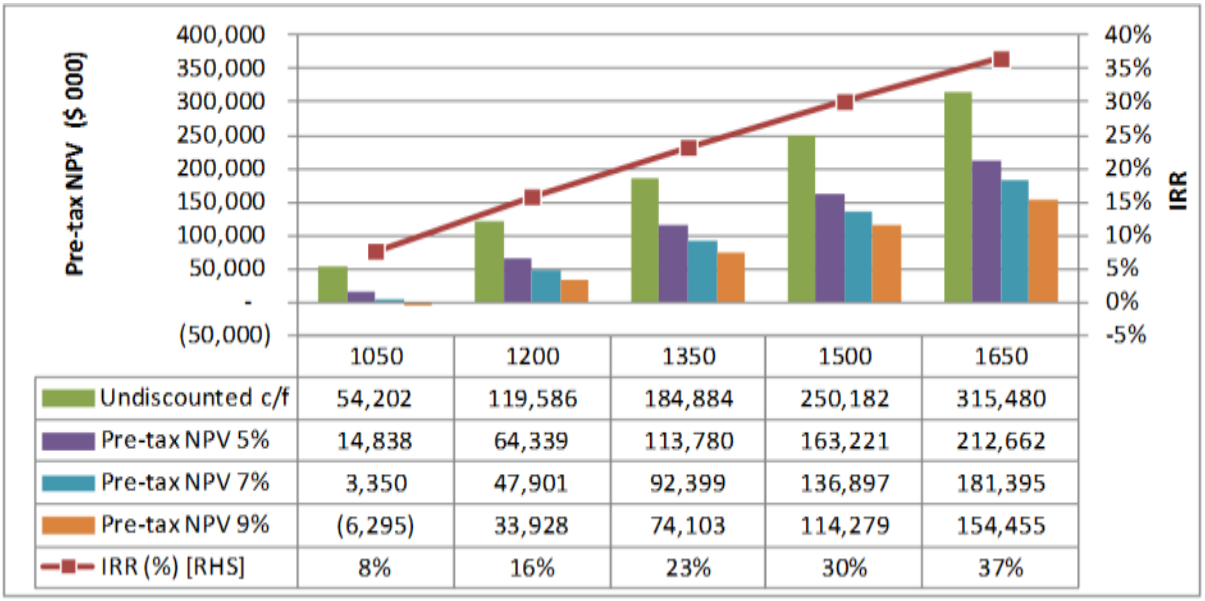
Figure 1.3
Sensitivity Diagram



	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue drivers	8.9	25.1	41.1	57.3	73.4	89.4	105.3	119.4	134.4
Operating costs	98.7	92.9	86.4	79.9	73.4	66.8	60.3	53.8	47.2
Capital costs	97.6	91.5	85.5	79.4	73.4	67.3	61.2	55.2	49.1

The sensitivity of the project to variation in gold price was tested using values of $1,050/oz, $1,200/oz, $1,350/oz (base case), $1,500/oz and $1,650/oz over the life-of-mine period, as shown in Figure 1.4.

Figure 1.4
Sensitivity to Metal Prices



	1050	1200	1350	1500	1650
Undiscounted c/f	54,202	119,586	184,884	250,182	315,480
Pre-tax NPV 5%	14,838	64,339	113,780	163,221	212,662
Pre-tax NPV 7%	3,350	47,901	92,399	136,897	181,395
Pre-tax NPV 9%	(6,295)	33,928	74,103	114,279	154,455
IRR (%) [RHS]	8%	16%	23%	30%	37%

1.14 UPDATED STUDY CONCLUSIONS

The results of the Updated Study are summarized in Table 1.9.

Table 1.9
Summary of the Updated Study Base Case Results (Imperial Units)

Item	Unit	Value
Total LOM leach feed production	Tons (000s)	19,025
Total LOM waste production	Tons (000s)	40,789
Average gold grade	oz/T	0.0298
Average gold process recovery	%	85.2
Total LOM gold production	oz (000s)	483.3
Annual gold production (average)	oz (000s)	76.5
LOM	Years	6.3
Pre-production capital cost	$ millions	151.0
Sustaining and closure capital	$ millions	20.6
LOM on-site operating cost	$ millions	248.2
LOM cash operating cost	$/T leach feed	13.05
Average base case gold price	$/oz	1,350
LOM gross gold sales	$ millions	651.8
LOM off-site costs and Nevada NPOM	$ millions	14.8
LOM NSR royalties	$ millions	32.4
LOM net revenue	$ millions	356.5
Project cash flow before tax	$ millions	184.9
Pre-tax NPV @ 7.0% discount rate	$ millions	92.4
Pre-tax NPV@ 5.0 % discount rate	$ millions	113.8
Pre-tax NPV@ 9.0 % discount rate	$ millions	74.1
Project cash flow after tax	$ millions	156.0

Item	Unit	Value
After tax NPV @ 7.0% discount rate	$ millions	73.4
After-tax NPV@ 5.0 % discount rate	$ millions	92.4
After-tax NPV@ 9.0 % discount rate	$ millions	57.1
Pre-tax IRR	%	23.2
After-tax IRR	%	20.4

Micon concludes that this Updated Study demonstrates the viability of the Project as proposed and that further development of the Project is warranted.

1.15 RECOMMENDATIONS

Following the completion of the Updated Study, it is recommended that IMZ continue to advance the development of the Project into final detailed engineering and construction. It is also recommended to complete the necessary environmental and permitting work required on the Property.

It is also recommended to continue with the development of the Goldfield Main and McMahon Ridge deposits, including additional exploration, geological modeling and metallurgical testing.

1.15.1 Budget

The following costs (Table 1.10) have been estimated for the continued development of the Project:

Table 1.10
Gemfield Project Budget

Activity	Estimate ($ 000))
Detailed engineering and procurement	9,057
Environmental and permitting	2,000
IMZ dedicated project support staff	1,500
Total	**12,557**

Other costs budgeted over the next two years for the development of the Goldfield Main, McMahon Ridge deposits and other exploration targets on the Property includes $1 million for exploration and $250,000 for metallurgical investigations.

2.0 INTRODUCTION

The Goldfield property (the Property) is located in the historic mining district of Goldfield, Nevada, USA (the Goldfield Mining District), approximately 30 miles south of Tonopah, Nevada, adjacent to the town of Goldfield in Esmeralda County.

The Property hosts three currently-known gold deposits: Goldfield Main, McMahon Ridge and Gemfield.

At this time, only the Gemfield gold deposit is being considered for development as an open-pit heap leach operation (the Project). The other two deposits are still being evaluated by IMZ.

At the request of International Minerals Corporation (IMZ), Micon International Limited (Micon) has been retained to prepare a Technical Report in accordance with Canadian National Instrument 43-101 (NI 43-101) to support the disclosure of the results of an update to the Feasibility Study (the Updated Study) on the potential production of gold from the Project.

A press release was issued by IMZ on 17 June, 2013, announcing the results of the Updated Study. This release can be accessed from IMZ's web site at http://www.intlminerals.com.

IMZ is a Yukon Territory registered company based in Scottsdale, Arizona and listed on the Toronto (TSX) and Swiss (SIX) stock exchanges (trading symbol: IMZ). IMZ has interests in gold and silver properties located in Peru and the USA. The Property is 100% owned by IMZ through its wholly-owned Nevada subsidiary, Metallic Goldfield, Inc.

2.1 TERMS OF REFERENCE

The Goldfield property has been the subject of four previous independent Technical Reports. These were entitled:

- "Technical Report on the Goldfield Project, Esmeralda County, Nevada USA" by Mine Development Associates (MDA), dated September 30, 2002 (MDA, 2002).

- "Technical Review of the Goldfield Project in Esmeralda and Nye Counties, Western Nevada USA" by Watts, Griffis and McOuat (WGM), dated July 12, 2005 (WGM, 2005).

- "Amended Preliminary Assessment Gemfield and McMahon Ridge Deposits Goldfield Mining District, Nevada" by AMEC, dated 15 March, 2007 (AMEC, 2007).

- "Feasibility Study on the Goldfield Property, Nevada, USA" by Micon, dated 17 July, 2012 (Micon, 2012).

These reports were filled on the Canadian System for Electronic Document Analysis and Retrieval (SEDAR).

The Updated Study for the Project (the Gemfield deposit) is based on the following:

- The most recent mineral resource estimate prepared by R. Mohan Srivastava, P.Geo., an independent consultant.

- An open-pit mine design, schedule and cost estimate prepared by Sam Shoemaker EngReg. Mem. SME, Principal Mining Engineer with Micon.

- Geotechnical testing and assessment of the open pit slopes by Golder Associates Inc Inc. (Golder).

- Interpretation of metallurgical testwork undertaken at McClelland Laboratories Inc. (MLI) and Kappes, Cassiday & Associates (KCA) by Tony Brown, IMZ's metallurgical consultant, and Richard Gowans P. Eng., President and Principal Metallurgist of Micon.

- Heap leach pad design and cost estimate by SRK Consulting (U.S.) Inc. (SRK) under the supervision of John Cooper PE, Principal Consultant.

- Geochemical waste rock and ore characterization study by SRK.

- Hydrogeological studies and preliminary dewatering estimation by SRK.

- Design, engineering and cost estimation by M3 Engineering, Micon, Atkins, IMZ and SRK, reviewed by Richard Gowans P.Eng.

- A report on environmental considerations by Larry Gorell Environmental Manager, Nevada with IMZ.

- Financial model prepared by Chris Jacobs C.Eng., CIMMM., Vice President at Micon.

2.2 QUALIFIED PERSONS

The Qualified Persons who prepared this NI 43-101 Technical Report are:

- Sam Shoemaker EngReg. Mem. SME.
- Richard Gowans, P.Eng.
- Christopher Jacobs, CEng, MIMMM
- R. Mohan Srivastava, P.Geo.

R. Mohan Srivastava visited the site between 29 and 31 January, 2005, and between 29 and September, 2010.

Sam Shoemaker visited the site on 15 February, 2013. Richard Gowans and Chris Jacobs have not visited the site.

Richard Gowans was responsible for preparing and supervising the preparation of the Technical Report.

2.3 USE OF REPORT

This report is intended to be used by IMZ subject to the terms and conditions of its agreement with Micon. That agreement permits IMZ to file this report as an NI 43-101 Technical Report with the Canadian Securities Administrators (CSA) pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

The conclusions and recommendations in this report reflect the authors' best judgment in light of the information available to them at the time of writing. The authors and Micon reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

This report includes technical information that requires subsequent calculations or estimates to derive sub-totals, totals and weighted averages. Such calculations or estimations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Micon does not consider them to be material.

2.4 INDEPENDENCE

Micon does not have nor has it previously had any material interest in IMZ or related entities or interests. The relationship with IMZ is solely a professional association between the client and the independent consultant. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.5 UNITS, CURRENCY AND ABBREVIATIONS

All currency amounts are stated in United States dollars ($). Units are generally expressed in the United States system of measurement where one short ton (T) contains 2,000 pounds (lb) and the US gallon measures 3.79 litres, although in the financial evaluation section of the report metric tonnes (t) are used. Precious metal values are given in troy ounces (oz) and grade as ounces per short ton (oz/T). In some cases grades are shown in grams per metric tonne (g/t). Precious metal grades may be expressed in parts per million (ppm) or parts per billion (ppb).

The following conversions are used in the preparation of this report:

- 1 troy ounce = 31.1035 g.
- 1 pound = 14.5833 Troy oz. (= 16 ounces avoirdupois).
- 1 ton = 2,000 lb.
- 1 tonne = 2204.627 lb.
- 1 foot = 0.3048 m.

Table 2.1 is a list of the abbreviations used throughout this report.

Table 2.1
List of Abbreviations

Term	Abbreviation
Above sea level	asl
Acceleration due to gravity	*g*
Acid base accounting	ABA
Acid rock drainage	ARD
Acid Rock Drainage and Metal Leaching	ARDML
Acre-foot(feet)	acre-ft
Acre-feet per year	acre-ft/y
Adsorption, desorption, recovery	ADR
Alternative minimum tax	AMT
American Resources Corporation Inc.	ARC
American Society for Testing of Materials	ASTM
Atomic absorption	AA
Benchmark Six	B6
Bureau of Land Management	BLM
Bureau of Mining Regulation and Reclamation	BMRR
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian Securities Administrators	CSA
Capital asset pricing model	CAPM
Carbon-in-leach	CIL
Carbon-in –pulp	CIP
CDN Resource Laboratories	CDN
Centimetre(s)	cm
Controlled Source Audio Magneto Tellurics	CSAMT
Copper	Cu
Cubic foot (feet)	ft^3
Cubic foot (feet) per minute	ft^3/min
Cubic metre(s)	m^3
Cubic metre(s) per hour	m^3/h
Cubic metre(s) per year	m^3/y
Cubic yard(s)	yd^3
Day	d
Decommissioning Services, LLC	DSL
Degree(s)	^{o}
Degrees Celsius	^oC
Degrees Fahrenheit	^oF
Diamond drill-hole	DDH

Term	Abbreviation
Earnings Before Interest, Tax, Depreciation and Amortization	EBITDA
East Dipping Alteration zone	EDA
Enterprise Optimization	EO
Environmental Assessment	EA
Environmental Design Criteria	EDC
Environmental Impact Statement	EIS
Feasibility Study	FS
Fire assay	FA
Foot(feet)	ft
Gallon (US)	gal
Gallons (US) per day	gal/day
Gallons (US) per minute	gal/min or gpm
Gallons (US) per minute per square foot	gpm/ft^2
General and Administration	G&A
Gold	Au
Goldfield historical database	GHDB
Gram(s)	g
Grams per litre	g/L
Grams per metric tonne	g/t
Heap Leach Drain-down Estimator	HLDE
Hectare(s)	ha
High density polyethylene	HDPE
High Voltage	HV
Horizontal wells	HW
Horsepower	HP
Hour(s)	h
Inch(es)	in
Induced Polarization	IP
Inductively Coupled Plasma	ICP
Inductively Coupled Plasma - mass spectrometry	ICP-MS
Inductively Coupled Plasma - atomic emission spectrometry	ICP-AES
Initial Public Offering	IPO
Internal rate of return	IRR
International Electrotechnical Commission	IEC
International Minerals Corporation	IMZ
International Society of Rock Mechanics	ISRM
International Organization for Standardization	ISO
Kappes, Cassiday & Associates	KCA
Kennecott Minerals Corporation	KMC
Kilogram(s)	kg
Kilograms per tonne	kg/t
Kilometre(s)	km
Kilovolt(s)	kV
Kilowatt	kW
Kilowatt hour(s)	kWh
Laboratory Information Management System	LIMS
Life-of-mine	LOM
Litre(s)	L
Litres per hour	L/h
Litres per hour per square metre	$L/h/m^2$
Litres per second	L/s

Term	Abbreviation
Main mineralized horizon	MMH
Maintenance and repair contract	MARC
Material take-off	MTO
McClelland Laboratories Inc	MLI
Megawatt(s)	MW
Memorandum of Understanding	MOU
Metallic Ventures Gold Inc	MVG
Metric tonnes (2,204.6 lb)	t
Metre(s)	m
Metres per second	m/s
Micron(s)	microns
Milligrams	mg
Milligrams per litre	mg/L
Millimetre(s)	mm
Millimetres per year	mm/y
Million	M
Million gallons	Mgal
Million ounces	Moz
Million pounds	Mlb
Million short tons	MT
Million metric tonnes	Mt
Million years old	Ma
Mine Development Associates	MDA
Mineralized ledge horizon	MLH
Minute(s)	min
Molybdenum	Mo
Motor control centre	MCC
National Environmental Policy Act	NEPA
National Instrument 43-101	NI 43-101
Net present value	NPV
Net smelter return	NSR
Nevada Department of Transportation	NDOT
Nevada Division of Environmental Protection	NDEP
Nevada Net Proceeds of Minerals	NPOM
Nevada Standardized Reclamation Cost Estimator	SRCE
Non-acid generating	NAG
Non-government organization	NGO
Not applicable	n.a.
Ounce(s) (troy ounce)	oz
Ounces per short ton	oz/T
Parts per million	ppm
Parts per billion	ppb
Plan of Operation	POO
Public Land Survey System	PLSS
Rock mass rating	RMR
Rock quality determination	RQD
Potentially acid generating	PAG
Pound(s)	lb
Process Fluid Cost Estimator	PFCE
Preliminary Economic Assessment	PEA
Quality assurance	QA

Term	Abbreviation
Quality control	QC
Reverse circulation	RC
Romarco Nevada Goldfield Inc.	RNG
Run-of-mine	ROM
Second(s)	s or ″
Semi-autogenous grinding	SAG
Short ton, ton	T
Short tons per day	T/d
Short tons per year	T/y
Square foot(feet)	ft^2
Square metre(s)	m^2
Square kilometre(s)	km^2
SRK Consulting (U.S.) Inc	SRK
Swiss Stock Exchange	SIX
System for Electronic Document Analysis and Retrieval	SEDAR
Thousandth of an inch	mil
Thousand short tons	kT
Thousand metric tonnes	kt
Toronto Stock Exchange	TSX
United States dollar(s)	$
United States Forest Service	USFS
Vertical wells	VW
Water pollution control permit	WPCP
Watts, Griffis and McOuat	WGM
Weak acid dissociable	WAD
Weight percent	wt%
Weighted average cost of capital	WACC
Whittle Consulting Pty	WCL
Year(s)	y

3.0 RELIANCE ON OTHER EXPERTS

The qualified persons have not carried out any independent exploration work, drilled any holes or carried out any sampling and assaying on the Property, other than examining/verifying mineralization in samples. While exercising all reasonable diligence in checking, confirming and testing it, the authors of this report have relied upon IMZ's presentation of data on the Property and the findings of its consultants in formulating their opinion.

The status of the mining claims under which IMZ holds title to the mineral rights for the Property has not been investigated or confirmed by the Micon, and Micon offers no legal opinion as to the validity of the mineral titles claimed. The description of the Property, and ownership thereof, as set out in this report, is provided for general information purposes only.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION AND AREA

International Minerals (IMZ), through its wholly-owned Nevada subsidiary, Metallic Goldfield Inc., owns or controls approximately 22,000 acres (over 34 square miles) in and around the historic Goldfield Mining District of Nevada. The Property is located approximately 30 miles south of Tonopah, Nevada and 174 miles northwest of Las Vegas, Nevada, adjacent to the town of Goldfield. The majority of the Property is within Esmeralda County, with the remaining portion located in Nye County.

The Property is located at approximately 37º42' N latitude and 117º14' W longitude. The site location is shown in Figure 4.1.

State Highway 95 runs (north-south) through the western portion of the Property with the majority of IMZ's land holdings located to the east of the highway. A major geographic feature on the Property is Columbia Mountain, located in the west-central part of the Property.

The Property is located within portions of 56 sections of land within the following Townships and Ranges: Township 2 South, Range 42 East; Township 3 South, Range 42 East; Township 2 South, Range 43 East; Township 3 South, Range 43 East, using the Public Land Survey System, (PLSS). Map coverage of the area is on the 7.5 minute USGS Goldfield Nevada topographic quadrangle map.

4.1.1 Land Status

As is often the case with historic mining districts, land ownership in the Goldfield Mining District is somewhat complex. Many of the claims were staked in the early 1900s. Since that time, many claims have been passed through generations of families, with ownership often divided among heirs. The sheer number of individual property owners has been a serious impediment in recent years to the re-start of commercial mining activities in the area. Over the past decade and a half, IMZ and its predecessors have been successful in assembling the largest property position ever controlled by a single company in the Goldfield Mining District.

The Property comprises 568 patented and 1,017 unpatented mining claims, either owned or controlled by IMZ and is summarized in Table 4.1.

Table 4.1
Land Status Summary

Claims	Owned	Leased	Total
Patented claims	464	104	568
Unpatented claims	1,005	12	1,017
Total	1,469	116	1,585

Figure 4.1
Goldfield Property Site Location

4.1.2 Property Maintenance Payments

The unpatented mining claims that IMZ owns or controls in the Goldfield Mining District are subject to all the rights and obligations associated with claims located on federal lands under the US Mining Law of 1872. The unpatented claims have the rights vested to them to develop the underlying minerals and utilize the surface in such a manner as to benefit the development of the resource. The unpatented mining claims are subject to annual

maintenance fees paid to the Bureau of Land Management (BLM) and annual filing fees paid to the counties.

The patented mining claims are lands to which title has been passed from public to private ownership through the mineral claims patenting process. Thus patented claims are subject to annual county real estate taxes. Some of the claims controlled by IMZ are held by lease and thus subject to annual payments (as either pre-production minimum royalties or as lease payments). The numerous individual leases have variable terms. The fees, taxes and other payments for both unpatented and patented claims have consistently been maintained in a manner to ensure the claims remain valid.

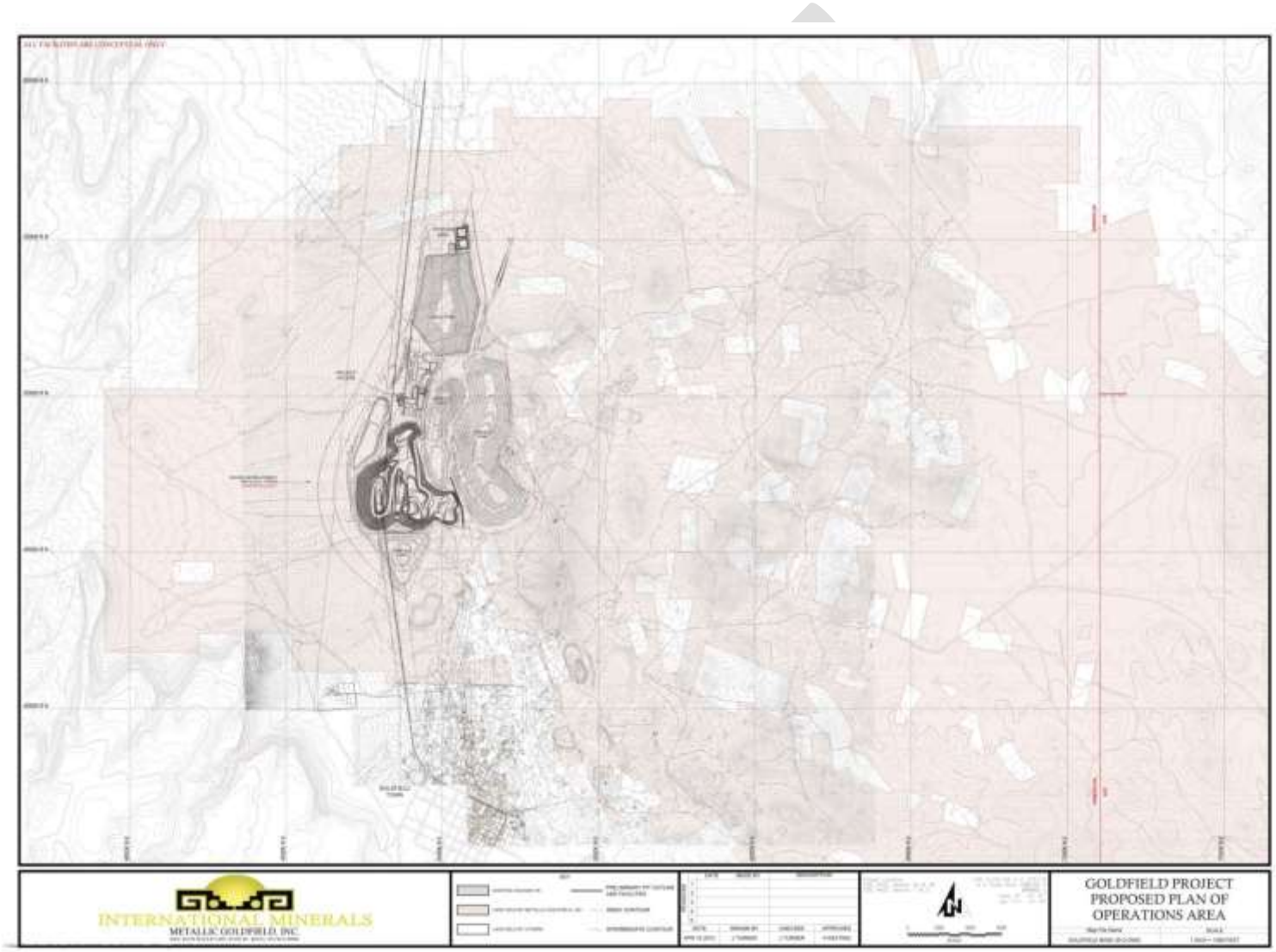
4.2 PROPERTY DESCRIPTION

The Goldfield Mining District was the site of intensive mining after the initial discovery in 1902 until 1919 when the Goldfield Consolidated Mill closed. Since that time, operations have been sporadic and relatively small in scale. Remnants of the early mining activities include numerous head frames, mine shafts and waste-rock dumps widely scattered around the Property. The foundations of the Goldfield Consolidated Mill are still evident on the western flank of Columbia Mountain. The mill tailings, from the Goldfield Consolidated Mill, were discharged by gravity and deposited on the flat areas below the mill.

A Property map showing the land ownership and claims in the area controlled by IMZ is presented as Figure 4.2.

Exploitation work during the 1980s and 1990s consisted of several companies performing limited re-processing of the historic dumps and tailings and limited mining by open pit methods. Pits excavated included the Adams, Red Top, Jumbo and the Combination. In general, ore mined was stacked on leach pads and the metal was recovered by heap leaching. The resulting features are three open pits, a small pit that was backfilled, a re-contoured waste dump, a decommissioned heap leach pad, process ponds, and several buildings. Several exploration drill programs have been carried out to locate and expand the mineral inventory on the Property. As required under the terms of the exploration Plan of Operations (POO) filed with the Bureau of Reclamation, site reclamation was conducted concurrently with this exploration drilling resulting in minimal environmental impact.

Figure 4.2
Goldfield property Site Map



4.3 PROPERTY ROYALTIES AND PAYMENTS

The applicable net smelter return (NSR) royalties for the Property are summarized in Table 4.2. A summary of the annual maintenance payments is presented in Table 4.3. There are no federal or state royalty payments.

Table 4.2
Summary of Applicable Royalties

% Royalty	Number of Claims	Comments
1.0% NSR	3	
2.5% NSR	83	
3.0% NSR	37	
3-5% NSR	43	Sliding scale based on grade
4.0% NSR	160	This royalty is capped at $1.4 million
4.0% NSR	35	This royalty reduced to 3% NSR once above cap of $1.4 million
3-5% NSR	81	5% Based on current Au price (This royalty, covering the Gemfield deposit, is sliding scale based on gold price: <$300 = 3%; >$300<$400 = 4%; >$500 = 5%)
5.0% NSR	20	
5.0% NPR	2	
5-7.5% NSR	3	Sliding scale based on ore value
10.0% NPR	5	
No Royalty	1113	
Total Claims:	**1585**	

Table 4.3
Summary of Annual Property Payment Obligations

Item	2009	2010	2011	2012	2013
Goldfield leases	$108,250	$120,250	$120,250	$134,450	$139,700
BLM Fees	$126,375	$141,540	$141,540	$142,380	$142,380
County Notice of Intent fees	$10,768	$10,768	$10,768	$10,831	$10,831
County affidavit of mining claims held in Nevada fee [1]	-	$49,548	-	-	-
Real property taxes	$15,646	$15,593	$15,593	$16,249	$16,302
Total Goldfield Property Holding Costs	**$261,039**	**$337,699**	**$288,151**	**$303,910**	**$309,213**

[1] *2010 one-time Nevada State mining tax. (This fee was rescinded and monies paid were refunded by the state)*

4.4 ENVIRONMENTAL LIABILITIES

In general the environmental liabilities attached to the Property are those associated with working within a historic mining district.

There are numerous open shafts scattered throughout the Property as a result of intensive mining during the 1900s. Nevada state law requires all open shafts be fenced to prevent access to areas where "Abandoned Mine Hazards" exist. All identified shafts have been fenced, backfilled, or otherwise remediated to comply with state law. The open pits located on the Property have a relatively small footprint and barriers have been placed to limit access to these pits.

The foundations of the historic Goldfield Consolidated Mill, which operated for eleven years starting in 1908, are still standing today.

Infrastructure associated with mining and processing activities which occurred during the 1980s and 1990s (including a heap leach pad, process ponds, maintenance shop, warehouse and processing plant building) has been removed and/or remediated (with exception of the building which housed the adsorption, desorption and recovery (ADR) plant). Reclamation responsibility for the heap rests with Decommissioning Services, LLC (DSL), an affiliate of Kappes, Cassiday & Associates (KCA) of Reno, Nevada. The reclamation work on this heap leach facility is basically completed, but has not yet been formally accepted by the state of Nevada.

4.5 PERMITS REQUIRED AND IN FORCE

The permits currently in place at the Property are those required for exploration drilling only. An exploration POO was submitted and approved for the ongoing and projected drilling programs. The plan limits the amount of land that can be disturbed as the program progresses to a maximum of 23.8 acres at any time. This limit is maintained by concurrent reclamation of drill sites as the drilling program progresses. All permits are active and in good standing for the completion of the planned exploration drilling program.

The BLM will be the lead regulatory agency for the overall mine and surface disturbing activities while the BMRR would be responsible for ore processing permitting and approval process. Coordination between IMZ, BLM and the BMRR would ensure all required state, federal and local agency permits and approvals are obtained for the Project.

Permitting of the Gemfield Project would entail acquiring the following major permits: Esmeralda County Special Use Permit; BLM Plan of Operations (including reclamation and closure plans); BLM Environmental Impact Statement (EIS); Nevada Water Pollution Control Permit; Nevada Reclamation Permit; Nevada Air Quality Permit; Nevada Water Discharge Permit and various SF. 299 Right-of-Way applications.

A number of additional permits would also be required, but the above will require the most significant effort, time and cost. The information that has been gathered and developed for the above-listed permits will be used in the completion of any additional permits. All other permits will be phased such that they are received in the timeliest manner.

Estimated timing of the permitting process is shown in Figure 4.3.

Figure 4.3
Gemfield Project Permitting Schedule

Gemfield Project NEPA Process	Mar-13	Jun-13	Sep-13	Dec-13	Mar-14	Jun-14	Sep-14	Dec-14	Mar-14	Jun-14	Sep-14	Dec-14
	2013				2014				2015			
Base line studies												
Submit plan of operations												
Completeness review by BLM												
EIS contractor selection												
Publish Notice of Intenet in Federal Register												
Publish Scoping Meetings												
Preparation of Draft EIS												
Publish notice of Draft EIS in Federal Register												
Draft EIS public meetings/comments												
Address comments for Final EIS												
Publish notice of Final EIS in Federal Register												
EIS Record of Decision												
Construction												

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS TO THE PROPERTY

The Property is located approximately midway between Las Vegas and Reno, Nevada along a main north-south transportation route in Nevada, US Highway 95. Goldfield is approximately 250 miles southeast of Reno and 174 miles northwest of Las Vegas. Goldfield has been the county seat of Esmeralda County since 1907. The county covers an area of 3,589 square miles.

5.2 LOCAL POPULATION CENTRES AND INFRASTRUCTURE

Esmeralda County is sparsely populated with about 40% of the population residing in Goldfield. The other main population centres in Esmeralda County are Silver Peak, approximately 26 miles west, and Dyer/Fish Lake Valley, approximately 100 miles to the west. In 2010, the population of Goldfield was 268 and the entire population of Esmeralda County was 783. The county is designated as rural by the United States Census Bureau.

The nearest community of significant size to Goldfield is Tonopah, located about 30 miles to the north of Goldfield (30 minutes by car). Tonopah is the county seat of Nye County, has a population of approximately 2,600 and is a full service community. Trained labor, accommodations, most required commercial services and educational and medical facilities are available in Tonopah.

Problems with transportation of materials, equipment and supplies into and out of Goldfield are not anticipated. A major north-south route, US Highway 95, traverses Goldfield and connects Las Vegas and Reno. Both these cities serve as major transportation hubs to all points in the United States. A network of gravel and dirt roads provide good access to various parts of the Property.

Major airports are located in both Las Vegas and Reno. A small airport is located a short distance east of Tonopah. This airport is not serviced by any commercial air carrier and can be used only for private or charter flights.

5.3 TOPOGRAPHY, FLORA AND FAUNA

The topography of the area comprises broad valleys and alluvial fans, low lying mesas and foothills. The foothills and mesas are rocky with drainages and topographic low areas in between. The area lies within the southern-most extreme of the Tonopah Section of the Great Basin, and the northern border of the Mohave Desert. The maximum topographic relief on the Property is 1,424 ft at elevations between 5,440 ft asl (Red Top open pit) and 6,864 ft asl (Diamondfield).

Vegetation in the area is sparse, consisting of sagebrush, salt desert shrub, Joshua trees, and grasses. There are no federally listed Threatened or Endangered plants on the Project site. One BLM Special Status Sensitive plant species, Eastwood's Milkweed *(Asclepias eastwoodiana)*, is known to occur within the vicinity of the Project. However, it is not found within the direct disturbance area. Wildlife on the Property includes rabbit, quail, dove, and wild burros.

5.4 CLIMATE AND LENGTH OF OPERATING SEASON

The Property is situated in the high desert region of the Basin and Range physiographic province in the Western USA. Precipitation averages 5.8 inches per year, primarily derived from snow and summer thunderstorms. There are warm summers and generally mild winters. However, overnight freezing conditions are common during winter. The mean annual temperature is 51°F.

5.5 SUFFICIENCY OF SURFACE LAND AREA

IMZ has a substantial land position in the Goldfield Mining District, comprising over 34 square miles. IMZ has sufficient property under control for its planned mining and processing operations.

6.0 HISTORY

6.1 PRIOR OWNERSHIP TO THE PROPERTY AND OWNERSHIP CHANGES

Property ownership in the Goldfield Mining District has historically been complicated and fragmented. However, over the past decade and a half, IMZ and its predecessors have been successful in consolidating the largest land position ever held by a single owner in the Goldfield Mining District.

The list of companies and individuals involved with work in the Goldfield Mining District is extensive. A partial list of the companies that have done work on the Property since the 1970s includes Cordex, Hanna Mining, Noranda, Utah International, Cyprus, Southern Pacific Land, Blackhawk Mines, Westley Mines, Transwestern Mining, Newmont, Meridian, Echo Bay, AMAX, Dexter, American Pacific, Red Rock, Crown, Santa Fe, American Resources, Kennecott, Cameco, Rea Gold, North, Romarco, Geochem Mines, Lode Star Gold, and Metallic Ventures.

On February 26, 2010, IMZ acquired the Canadian public company, Metallic Ventures Gold Inc., including its three Nevada operating subsidiaries: Metallic Ventures (U.S.) Inc., (with its royalty interest in Barrick's Ruby Hill gold mine in central Nevada); Metallic Nevada Inc. (with its Converse gold property in north-central Nevada); and Metallic Goldfield Inc., which controlled the majority of the historic Goldfield Mining District.

Hereafter, the parent company and subsidiaries of Metallic Ventures Gold Inc. will be referred to as MVG.

6.2 RESULTS OF EXPLORATION AND DEVELOPMENT WORK OF PREVIOUS OWNERS

In the early 1990s American Resources Corporation (ARC) began mining and processing activities in the Goldfield Mining District. Ore was mined from the Red Top, Combination and Jumbo pits and stacked on a heap leach pad. In 1995 ARC reported that 532,379 tons grading 0.044 oz/T Au was mined from the Red Top and Jumbo pits. In 1998 the parent company of ARC, Rea Gold Corporation, declared bankruptcy. Following bankruptcy, the property interest and reclamation responsibilities were acquired by Decommissioning Services LLC (DSL), a Reno, Nevada private company.

In 1999 Romarco Nevada Goldfield, Inc. (RNG), a wholly-owned subsidiary of Romarco Minerals Inc. (Romarco), obtained a mining sublease, lease and option to purchase agreement from DSL for claims including the Goldfield Main and McMahon deposits. RNG conducted exploration work on these properties until 2001.

In March 2001 MVG purchased all issued and outstanding shares in RNG. MVG conducted several exploration drilling programs, updated the mineral resources and began preliminary engineering work directed toward production from the Property.

In August 2002 MVG acquired the claims covering the Gemfield deposit from Newmont Capital Limited.

In early 2010 IMZ purchased all issued and outstanding shares of MVG and assumed control of its assets, including the Property. Shortly after acquiring MVG, IMZ began a comprehensive drill exploration program to expand the mineral resources on the Property.

Geophysical surveys have been carried out at various locations on the Property since 1980, including Induced Polarization-resistivity (IP) and Controlled-Source Audio-Frequency Magneto Telluric (CSAMT). Both methods aid in the identification of intense silicification (locally known as ledges) that is often associated with gold mineralization in the Goldfield Mining District.

6.3 HISTORY AND PRODUCTION FROM PROPERTY

Goldfield was one of the world's great gold camps and also one of the last historic gold rushes in the western USA. Although typical of many of the Nevada "boom and bust" gold camps, what made Goldfield unique was the extremely high-grade nature of the material mined. From 1906 to 1910 Goldfield was the largest city in Nevada.

Production records are incomplete and generally difficult to reconcile. The exact amount of modern era production and the production results are not known. A number of companies have attempted to detail production figures and the following paragraph summarizes the results of these investigations.

In 1902, a Shoshone Indian named Tom Fisherman went to Tonopah with some gold-bearing samples he had found about 25 miles south of the site near what is now known as Columbia Mountain. The first claims were staked by two prospectors, William Marsh and Harry Stimler, in December of 1902 and early 1903 on the northern part of Columbia Mountain.

In 1903 the Combination group of claims and the Florence group of claims were located south of Columbia Mountain, in the present day location of the Goldfield Main district. The Jumbo and Red Top mines were located on the Florence group of claims. Over the next six years these mines produced gold worth $1,250,000, with the gold price then at $20.67 per troy ounce. During this time period the Tennessee and Berkeley claims were located and these later became the location of the rich Mohawk mine.

The Goldfield Mining District was formally organized on October 20, 1903 and the initial town site of Goldfield was laid out on October 24, 1903. The first ore was shipped from the Combination No. 2 claim in November 1903. A major gold rush ensued. In just the first six weeks of 1904 Goldfield grew from 400 to 1,000 residents. Reportedly the Mohawk mine produced gold worth $5,000,000 in the first 106 days of mining operations. An indication of the ore grade produced at Goldfield is the records of 1904 showing production of over 110,000 ounces of gold from 8,000 tons at an average grade of 13.75 oz/T.

In September 1905, the Tonopah and Goldfield railroad was completed. Goldfield had a population of about 8,000 residents and was still growing. By 1905, a dozen mines in town had produced gold with a value of nearly $7,000,000, with the Florence mine at the top of the list with production valued at $1,848,000.

Late in 1907 the majority of the mines in Goldfield District were acquired by the Goldfield Consolidated Mines Company. By 1910, the population of Goldfield had risen to 20,000 and the mines had a record year with gold production valued at over $11.2 million. However, by 1919, Goldfield Consolidated had closed the last of their mills and moved-out the equipment.

Minor production continued from leasing operations through 1926. Leasing was a system in which claim owners leased out small portions of their holdings on an annual basis. Between 1927 and 1937 about 3.1 million tons of tailings from the Goldfield Consolidated mill were reprocessed. The recorded production was 160,800 ounces of gold recovered at an average grade of 0.05 oz/T Au.

Several mining companies worked and explored the area between 1935 and 1951; however, production was relatively minor. Newmont reportedly produced approximately 17,000 ounces of gold from about 30,000 tons of ore mined underground from the “Newmont Lode” between 1948 and 1951.

Beginning in 1970, Blackhawk leached 60,000 tons of tailings grading 0.078 oz/T Au, (recovering 75% of the gold). From 1979 to 1981, Blackhawk also mined and heap leached ore from the Adams pit and some of the Goldfield Main area dumps. Transwestern Mining Company (also known as Trafalgar Mines) leached 62,900 tons of mixed dump and tailings, achieving 61% gold recovery. Dexter mined 357,000 tons at 0.058 oz/T Au of material from the Main district in the Red Top pit, during the period 1986 to 1988. Red Rock reprocessed 285,000 tons of waste dumps in 1989 but apparently only 149,000 tons grading 0.078 oz/T gold were properly agglomerated and a total of 7,500 ounces of gold were recovered from the dump leaching operation, yielding a recovery of 65%.

Modern gold production has so far been confined to the Goldfield Main area, extending from the southern part of Columbia Mountain in the north to the Red King Shaft located approximately one mile to the south.

Production figures for the district since 1990 are incomplete. The Nevada Bureau of Mines reports only 28,400 ounces of gold were produced during the 1980s and 1990s.

Heap leach ore was extracted by ARC from the Red Top, Combination and Jumbo open pits during the early 1990s. American Pacific Minerals reported in 1995 that 532,379 tons grading 0.044 oz/T Au were mined from ARC’s pits but no official figures are known.

6.4 HISTORIC MINERAL RESOURCE ESTIMATES

Historic and on-going exploration programs have defined and expanded the mineral resources in the three currently-known gold deposits within the Property: Gemfield (approximately 1.5 miles north of the town of Goldfield); McMahon Ridge (approximately 3.5 miles north-east of the town of Goldfield); and Goldfield Main (approximately 0.5 miles east-north-east of the town of Goldfield). Table 6.1 provides a summary of the recent resource estimates.

Table 6.1
Historic Goldfield Resource Estimates

Author	Year	Cut-off Grade (Au oz/T)	Tons (Thousand)	Grade (oz/T)	Contained Gold (oz)	Classification
McMahon Ridge						
Mine Development Associates	2002	0.01	2,439	0.035	85,400	Indicated
Watts Griffis and McOuat [2]	2005	0.01	8,200	0.035	285,000	Measured and Indicated
		0.01	171	0.019	3,000	Inferred
AMEC	2007	Note [1]	4,138	0.042	175,500	Measured and Indicated
		Note [1]	172	0.038	6,500	Inferred
Srivastava	2012	0.3 [3]	5,510 [3]	1.34 [3]	238,000	Measured and Indicated
		0.3 [3]	110 [3]	1.09 [3]	4,000	Inferred
Gemfield						
Mine Development Associates	2002	0.01	14,320	0.027	393,100	Indicated
		0.01	8,110	0.021	166,700	Inferred
Watts Griffis and McOuat [2]	2005	0.01	16,853	0.032	541,000	Measured and Indicated
		0.01	1,001	0.022	22,000	Inferred
AMEC	2007	Note [1]	12,459	0.031	387,600	Measured and Indicated
		Note [1]	88	0.116	10,200	Inferred
Srivastava	2012	0.3 [3]	17,030 [3]	1.05 [3]	574,000	Measured and Indicated
		0.3 [3]	4,170 [3]	0.55 [3]	74,000	Inferred
Goldfield Main						
Mine Development Associates	2002	0.01	23,410	0.031	720,300	Indicated
		0.01	10,239	0.024	247,000	Inferred
Watts Griffis and McOuat [2]	2005	0.01	6,651	0.036	242,000	Measured and Indicated
		0.01	2,129	0.038	80,000	Inferred
AMEC	2007	Note [1]	919	0.049	45,000	Measured and Indicated
		Note [1]	177	0.040	7,100	Inferred
Srivastava	2012	0.4 [3]	8,550 [3]	1.53 [3]	421,000	Measured and Indicated
		0.4 [3]	6,590 [3]	1.70 [3]	360,000	Inferred

Notes

1 *Resources were defined by inclusion of Measured, Indicated and Inferred Resources within a Lerchs Grossmann pit shell using a gold price of $500/oz, variable gold recoveries by metallurgical types and operating costs of $1.24/T ore mined, $0.98/T waste mined, $2.51/T ore processed, and $0.61/T G&A.*

2 *Resources were estimated by Metallic Ventures Gold Inc. but externally audited by Watts, Griffis and McOuat Ltd. (WGM, 2005).*

3 *2012 resource estimates published in metric units. Grade in g/t and tonnage in metric tonnes.*

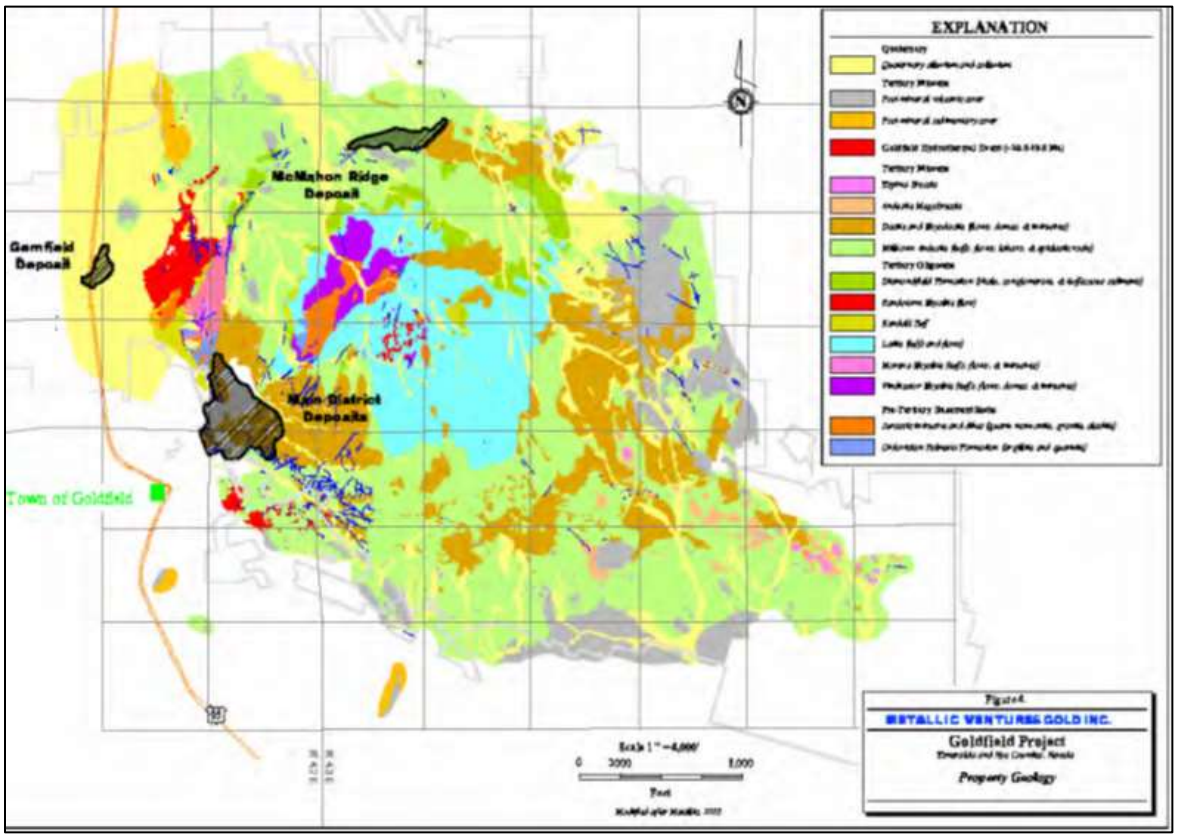
7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 PRE-MINERAL STRATIGRAPHY

The oldest known rock unit found in the Goldfield Mining District is the Ordovician Palmetto Formation. The Palmetto Formation mainly consists of black siliceous shale and argillite, but also contains minor amounts of limestone.

Jurassic granitic to granodioritic batholithic rocks intrude the Palmetto Formation. These have been dated at 170 million years old (Ma) by Ashley and Silberman (Ashley, 1976). Both the Palmetto and the granites are exposed in the western part of the district, on Columbia Mountain; in the central part of the district, on Vindicator Mountain and in drill core at both Gemfield and Goldfield Main (Figure 7.1).

Figure 7.1
Property Geology of the Goldfield Mining District



Source, (WGM, 2005)

During Oligocene to Early Miocene times, for a period of approximately 14 million years, the Goldfield mining district became a centre of volcanic activity, hydrothermal alteration and gold deposition. By the Late Miocene, volcanic activity in the district had diminished, with volcanism in the Goldfield Mining District limited to minor amounts of tuff deposited in

a subsiding, lacustrine environment. By the end of the Miocene, flood basalts were extruded over the region (Table 7.1).

Table 7.1
Goldfield Mining District Stratigraphic Units

Period	Unit	Symbol	Age (Ma)
Quaternary	Alluvium	Qal	
Pliocene	Thirsty Canyon tuff, Spearhead member	Tts	6.7-7.5
Late Miocene	Basalt of Black Cap Mountain	Tb	10.4-12.0
	Siebert Fm; lake sediments and gravels	Ts	
	Siebert Fm; silicic air-fall tuff	Tst	12.7-16.0
Early Miocene	**Goldfield Hydrothermal Events**		**19.6-21.0**
	Chispa Andesite; dykes, plugs and flows	Tc	20.8
	Rhyolite of Wildhorse Spring		20.3-21.2
	Tuff of Chispa Hill	Tct	21.1
	Espina Breccia	Te	22.2
	Dacite flow domes*	Td	19.8-23.2
	Milltown Andesite tuff*	Tmat	
	Milltown Andesite*	Tma	21.5
Late Oligocene	Sandstorm Rhyolite; flow dome*	Tsr	28.8
	Morena Rhyolite; ash-flow tuff	Tmr	24.4-31.0
	Kendall Tuff*	Tkt	31.1-33.2
	Diamondfield Fm; moat sediments*	Tdf	
Early Oligocene	Vindicator Rhyolite; ash-flow tuff and flow breccia	Tvr	33.0
	Latite tuff	Tlt	
	Latite to quartz latite; flows	Tl	33.5-33.6
	Rhyolite dykes and tuff		
Jurassic	Granite to Quartz Monzonite	Jgr	170
Ordovician	Palmetto Fm; black argillite, siltstones, quartzite, minor limestone	Op	

Source: Radiometric ages (Ashley and Silberman, 1976)

The pre-mineral volcanic rocks in the Goldfield Mining District were deposited during two eras, each characterized by distinct magmatic assemblages: Oligocene (33.6 Ma - 28.8 Ma), predominantly rhyodacitic; and Early Miocene (23.2 Ma - 19.8 Ma), predominantly andesitic/dacitic. Gold mineralization was emplaced during the waning stages of andesite/dacites deposition (20.0 Ma - 21.2 Ma).

A geological model for the formation of the Oligocene-Early Miocene volcanic rocks, which incorporates a caldera, has been proposed by Ashley (Ashley, R.P., 1974). He based his model mainly on the localization of the Goldfield Main and McMahon Ridge deposits (Gemfield had not, as yet, been discovered) along an annular, normal fault system (ring fracture) between the two deposits. His caldera model incorporated two compositionally-distinct source magmas (rhyodacitic and andesitic/dacitic), whose extrusion was separated by a period of at least 5 million years. As such, his model was at odds with the accepted models for caldera formation which require rapid extrusion and collapse followed by hydrothermal alteration and mineralization. Ashley did, however, point out evidence for an annular structural control (ring fracture) for the younger dacitic domes as a basis for including them in the formation of the caldera, but this could be the result of erosion of the centrally-uplifted

portions of the caldera, giving an appearance of circular control; or, if real, it could be the result of re-activation of earlier caldera structures. The evidence for the formation of a caldera strictly during the Oligocene is presented as follows.

The Oligocene volcanic succession at Goldfield is characterized by dykes, flows, ash-flow tuffs and breccias of rhyolitic to dacitic compositions. These rocks are distributed into two distinct areas, implying two different eruptive centres. Age dating by Ashley has shown that these eruptive centres were active during two distinct periods, separated by a period of collapse and lacustrine sedimentation:

- Early Oligocene, centre of the district: early rhyolite dykes and tuff, latites/quartz latites (33.5-33.6 Ma) and Vindicator Rhyolite (33.0 Ma).

- Late Oligocene, northern half of the district: lacustrine sedimentation (Diamondfield Formation).

- Late Oligocene, western part of the district: Kendall Tuff (31.1-33.2 Ma), Morena Rhyolite ash-flow tuffs (24.4-31.1Ma) and Sandstorm Rhyolite (28.8 Ma).

All of the late Oligocene (western) rhyodacitic volcanic units are localized within a semi-circular arc along the northwestern quadrant of the district, implying a semi-circular structural control, possibly a ring structure.

The above relationships are consistent with known caldera features in other parts of the world. The small size of the Goldfield caldera (diameter of approximately 2 miles) is consistent with the small volume of material ejected during Early Oligocene volcanism.

Following a hiatus of approximately five million years, andesitic lavas of the Milltown Andesite were erupted during the Early Miocene. Related dacitic flow domes were also emplaced during the same period, together with local units (Espina Breccia, tuff of Chispa Hills, Chispa andesite and rhyolite of Wildhorse Springs). Although stratigraphic relationships between these units are poorly understood, overlapping radiometric ages suggest that they were roughly contemporaneous (Table 7.1).

The andesites/dacites are typical of similar successions, in other parts of the world, which are thought to have been deposited during periods of relatively quiet structural activity. Berger (Berger, 2007), however, postulated that the eruption of the andesites/dacites were related to the eastward migration of strike-slip faults and related extensional stepping, based on age-dating by Vikre (Berger, 2007).

The Vikre dates, however, are for hydrothermal alunites, which would make them post-volcanic. Ashley's age dating of Early Miocene volcanic rocks from the same three zones do not show any systematic progression in ages (Table 7.2).

Table 7.2
Early Miocene Andesite/Dacite Ages in the Goldfield Mining District (after Ashley, 1976)

Stages (Berger, 2005)	Ages of Volcanic Stages (Berger, 2005)	Areas Sampled (Ashley, 1976)	Radiometric Ages Ma (Ashley, 1976)
Stage 1	22 Ma	Chispa Hills	20.8±0.4
Stage 2	21 Ma	NE of Vindicator Mtn	21.5±0.5
Stage 3	20-21 Ma	Espina Hill	20.8±0.7
			21.6±0.5
			20.5±1.6
			21.2±0.4
			23.2±1.6
			19.8±1.4
			20.1±2.0
			21.4±0.4

7.2 POST-MINERAL STRATIGRAPHY

The third period of volcanism in the Goldfield Mining District occurred during the Late Miocene after hydrothermal alteration and gold deposition. This period is characterized by rhyolitic tuffs and lacustrine sedimentary rocks (Siebert Formation) which were deposited in local basins.

The Siebert Tuff is the most common post-mineral unit in the Property area and is generally composed of volcanic ash, gravel and scattered beds of diatomaceous earth. In the Goldfield Main area, the basal beds of the Siebert consist of sedimentary breccias, probably colluvium, composed of locally derived altered porphyritic dacite, rhyodacite, and Milltown Andesite.

The fourth period of volcanism in the Goldfield Mining District consists of basalt flows that have yielded ages ranging from about 10.4 to 14 Ma (Armstrong, 1972). Such flows cap the Siebert on Myers and Mira Mountains, south of Goldfield, and patches of basalt are also exposed on all sides of the Goldfield Hills at lower elevations in the pediment areas.

The fifth and final period of volcanism in the Goldfield Mining District occurred following a hiatus of approximately 3.6 million years. This consists of the Spearhead Member of the Thirsty Canyon Tuff, dated at 6.8-6.7 Ma (Ashley, 1976), which outcrops on the flanks of the Goldfield Hills on the western, southern and eastern portions of the district. Several other local units, very young in age, occur in the general vicinity of Goldfield, including the Pozo Formation, which lies between the Siebert and the Spearhead Member, the Rabbit Spring Formation, which lies between the Spearhead and the Malpais Basalt. Gravels of Quaternary age overlie all of the previous units.

7.3 STRUCTURE

Five populations of faults have been identified in the Goldfield Mining District:

1. An arc-like structure (ring fracture), which begins at the Goldfield Main deposit (the "Combination-Florence" structure), strikes northerly along the east side of Columbia Mountain, continues north-easterly and easterly through the Adams mine and the McMahon Ridge deposit, respectively. The ring structure hosts gold mineralization at both Goldfield Main and McMahon Ridge. Drilling in both areas, as well as at Columbia Mountain, show this fault to be normal, with sense-of-movement towards the centre of the district. Searls (1948) observed a normal, vertical displacement of up to 200 ft on this fault. At Columbia Mountain, vertical displacements of 300 ft have been recorded by previous workers.

2. A north-striking, shallow, east-dipping, structure hosts the major portion of the mineralization at Goldfield Main (Main Mineralized Horizon or MMH) and continues northward to Columbia Mountain. Its age relationship to the adjacent "ring structure" has not been proven. Detailed mapping in the Combination pit, however, shows that they are distinct structures. Sense-of-movement has not been determined on this shallow-dipping structure.

3. Three west-northwest fault sets which are regional in scale; one tangential to the "ring" structure at Goldfield Main; a second at Tom Keane; and a third in the northern part of the district at Black Butte and Belmont. The sense-of-movement on these structures has not been determined, except at the south-eastern extremity of the district, near the Tom Keane occurrence.

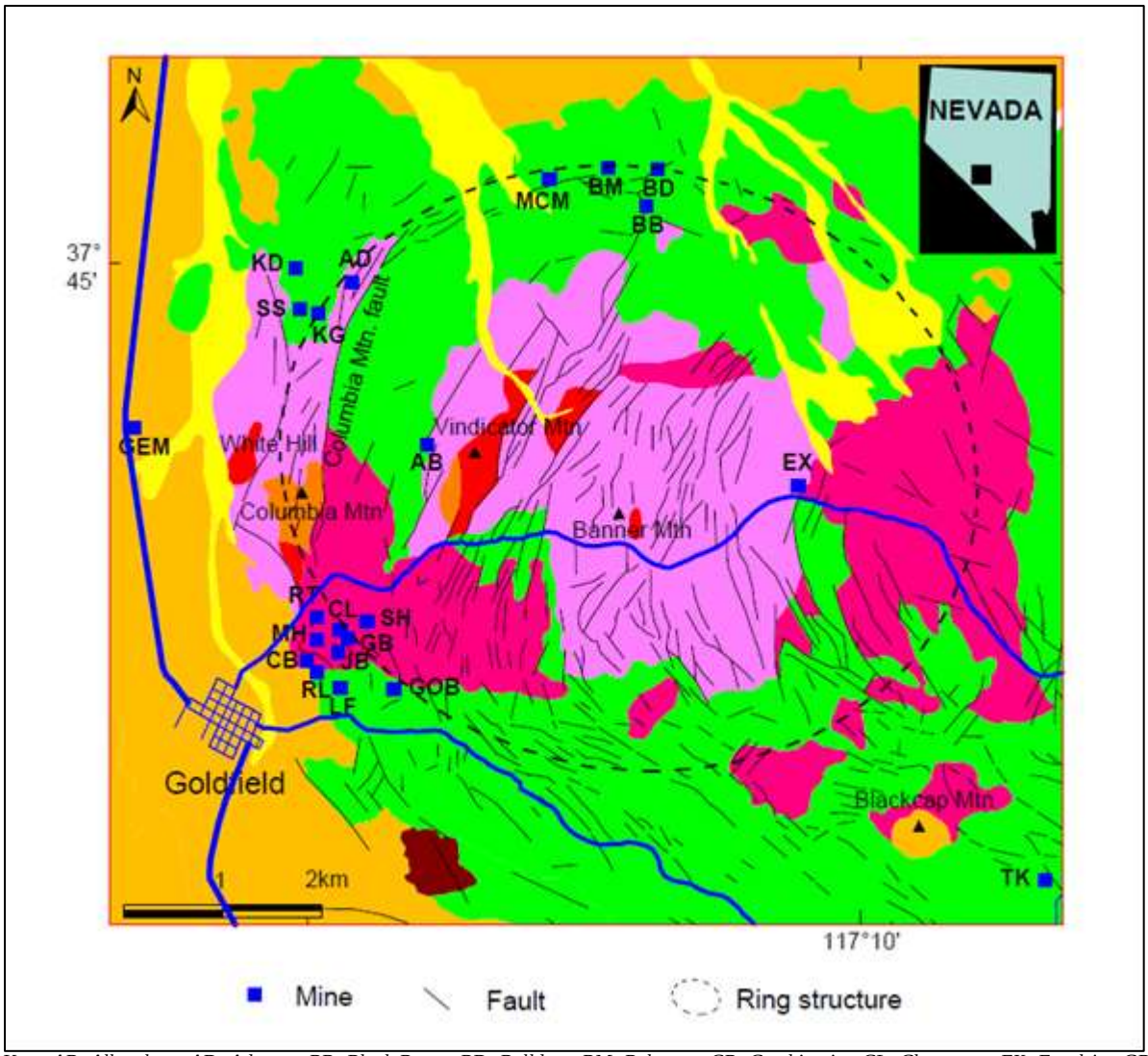
4. Northeasterly structures, which cut both the Goldfield Main and Vindicator Mountain areas. Detailed mapping of these structures by both IMZ and MVG, in the vicinity of the Combination pit (the Cross Fault) and towards Vindicator Mountain, together with recent drilling by IMZ, indicate that these structures are south-side-down, normal or normal-oblique.

5. Steep, normal, north-northeast trending faults at the Gemfield deposit, which form a horst.

A simplified structural map of the Goldfield Mining District is presented in Figure 7.2.

Different models for the origins of the structures in the Goldfield Mining District have been proposed. The earliest was by Ashley who commented on the similarity of the Goldfield area to a caldera (Ashley, 1974). He proposed a re-activated caldera model whereby caldera collapse during the Early Miocene (i.e. post-Milltown andesite) resulted in the creation of a "ring structure", followed by left-lateral movement along northeast trending faults (Cross Fault system), the latter movement resulted in the creation of low-angle (shingle) faults, one of which is the main mineralized structures at the Goldfield Main deposit.

Figure 7.2
Simplified Structural Map of the Goldfield Mining District



Key: AB: Alhambra AD: Adams BB: Black Butte BD: Bulldog BM: Belmont CB: Combination CL: Clermont EX: Excelsior GB: Grizzly Bear GOB: Gold Bar JB: Jumbo KD: Kendall KG: Kruger LF: Little Florence MCM: McMahon Ridge MH: Mohawk RL: Reilly RT: Red Top SH: Spearhead SS: Sandstorm TK: Tom Keane.

Recent work in the Goldfield Mining District, however, shows that a caldera likely formed much earlier than the Early Miocene, during the Oligocene, as evidenced by the emplacement of moat sediments of the Diamondfield Formation. The second problem with the Ashley model is that the "left-lateral" movement on the north-easterly trending faults is "apparent" and not "actual". Recent studies have shown that the west end of the same fault exhibits apparent right-lateral movement of the same dacite flow unit. This feature, together with the funnel-shape of the displaced dacite flow unit, determined by recent drilling, can only be explained by south-side-down normal or oblique-normal movement. Down-dropping

of the Main Mineralized Horizon (MMH) across the same fault implies that the movement was post-mineral.

More recent structural proposals have focused on right-lateral (Walker Lane type) movements as regional controls for mineralized structures in the Goldfield district. There are a few difficulties with these models, the most important being a lack of empirical evidence for strike-slip movement. To date, only one such observation has been made in the Goldfield district, in the Tom Keane area, located at the southeastern extremity of the district (Figure 7.2)

The Walker Lane structural models are summarized as follows.

Swager (1997) proposed a model whereby right-lateral (Walker Lane) movement on the Combination-Florence fault, coupled with left-lateral movement on the Cross Fault, resulted in the creation of the low-angle, MMH structure which hosts the Goldfield Main Deposit. The two problems with this model have already been mentioned: a lack of empirical evidence for right-lateral movement on the Combination-Florence fault (drill-hole data indicate normal, east-side-down movement for this fault) and evidence indicating that the Cross Fault is a south-side-down normal fault (and not a left-lateral fault). Swager's model also requires the rotation of east-dipping, dacite flows towards the west. However, drilling by IMZ on the down-dip extension to the MMH clearly indicates that the dacite is a funnel-shaped intrusive dome with no evidence for any kind of rotation.

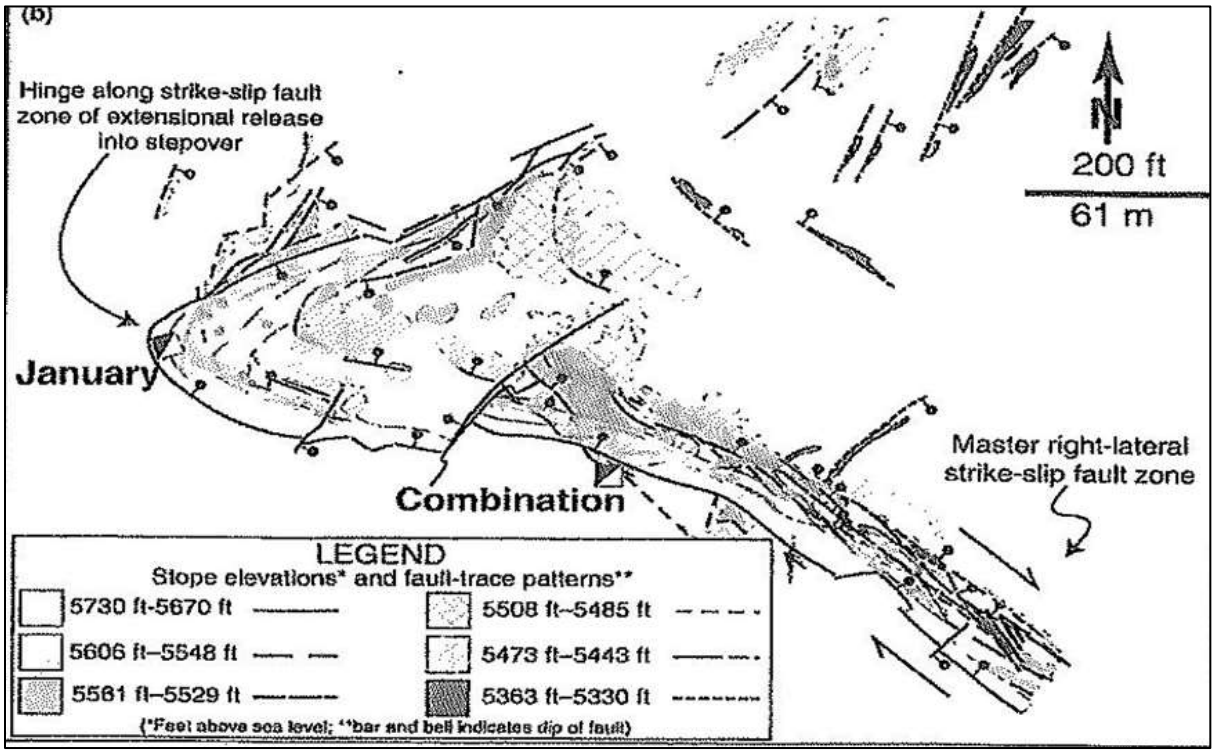
More recently, Berger (2007), proposed a strike-slip, releasing bend model for the structures in the Goldfield Mining district, whereby northwestern trending, strike-slip faults (e.g. Combination-Florence) are coupled with northeastern trending, normal-oblique faults (e.g. Cross Fault) at synformal bends. His hypothesis is mainly based on data from the Combination-January pit area (Figure 7.3).

Berger also postulates that a series of these strike-slip releasing bends migrated across the district in an easterly fashion through time.

As with the previous strike-slip model, the assumption of strike-slip movement on the Combination-Florence and other northwest trending faults has not yet been sufficiently demonstrated by field evidence. A second problem with this hypothesis is that the relationship between the Combination and Cross Faults, which Berger indicates as being essentially continuous in space and time. Surface mapping at Goldfield Main, however, shows that these faults are separate in both space and time, with the northeast set being younger and displacing both the northwest set and the MMH mineralization.

The fault movement data shown in Berger's 2007 paper does indicate a small number of northwest trending, shallow-plunging (<45°), fault striations. The location of these striations, however, is unknown.

Figure 7.3
Strike-Slip, Releasing-Bend Model for the Goldfield Main Structures (after Berger, 2007)



7.4 ALTERATION

Potassium-argon dating of hypogene alunite and potassium-mica from hydrothermally altered rocks yields mineralization ages of 19.6 Ma to 21.6 Ma (Ashley 1976), which is in good agreement with the age of mineralization established by potassium-argon dating of unaltered pre-mineralization and post-mineralization volcanic units.

Most of the Early Miocene volcanic rocks in the district exhibit some degree of hydrothermal alteration, with the exception of the porphyritic dacite on Tognoni Mountain, the Chispa Andesite (20.8 Ma), and the rhyolite of Wildhorse Spring. These three units appear to be unaltered and together with the radiometric ages obtained from alunites, sericites and apatites obtained by Ashley, date the mineralization at Goldfield at around 20 Ma.

The hydrothermal alteration in the Goldfield Mining District is fairly typical of high-sulfidation, gold hydrothermal systems found throughout the world. The hydrothermal fluids in these types of systems are emplaced episodically and follow a pattern of decreasing sulfidation state, decreasing acidity and increasing gold content over time.

In the areas studied by IMZ geologists (Goldfield Main, Gemfield and McMahon Ridge), the following scenario of alteration/mineralization events were observed.

The earliest hydrothermal fluids were highly acidic (pH<2) and followed fractures and faults created by pre-mineral structures. This event resulted in the partial corrosion of the original rock, forming a vuggy texture. As the hydrothermal fluids became progressively less acid (pH 3-4) due to dilution, they deposited an acid-sulfate assemblage consisting of quartz, alunite, barite, kaolinite and pyrite, with gold, mainly as silica-flooded zones surrounding the vuggy rocks. Outward from this silica-flooded zone, the host rocks were altered to an assemblage of intermediate clays (smectite-illite) and chlorite, forming a broad argillic-propylitic zone. Other workers have noted a gradual increase in smectite and decrease in illite with increasing distance from the silica-flooded zones, which is consistent with a decreasing temperature gradient. Very low levels of gold are associated with this early alteration.

The gold-bearing hydrothermal events (two were noted in core samples from the Gemfield deposit) cross-cut the early hydrothermal assemblages and consist of white chalcedonic quartz with clear, druzy terminations, dickite, pyrophyllite (indicating a pH of about 4), and fine disseminations of pyrite, chalcopyrite, sphalerite, enargite-luzonite, famatinite, tetrahedrite-tennantite, bismuthinite, goldfieldite, hessite, argentite and electrum in various combinations. The mineralization is either in the form of finely-banded veinlets (indicative of boiling), as breccia fragment coatings, or coating vuggy cavities.

Post-mineral, low-temperature veinlets of smectite (indicating temperatures of <160 ºC) were observed in a few localities.

Alunite of supergene origin is common in the Goldfield Mining District. Dating of supergene alunite samples yields imprecise ages in the range 12 to 9 Ma that probably record the first exposure of their hydrothermally-altered host rocks to oxidizing conditions (Ashley, 1976).

7.5 MINERALIZATION

The ore bodies in the Goldfield Mining District generally occur within silicified hydrothermal alteration zones. Both historically and presently, these zones are often referred to as “ledges”. The siliceous ledges were created during multiple hydrothermal alteration events. The events began with an early “acid” event (pH<2) which resulted in the partial to pervasive dissolution of the host rock and creating a “vuggy” host. Progressively less acid solutions later flooded the surrounding rock and deposited quartz together with alunite, barite and pyrite (pH of 3-4).

Two hydrothermal pulses have been identified. The main gold-bearing event, which was volumetrically-low compared to the initial event, deposited white, chalcedonic quartz. In addition dickite and pyrophyllite were deposited in narrow veinlets and vugs in the previously-silicified rock. Fine-grained disseminations and bands of enargite, famatinite, bismuthinite and pyrite, with gold tellurides, were also deposited with the quartz. Spatially, the gold-bearing quartz veins and disseminations are usually superimposed on, or adjacent to, early silica-flooded zones.

The high-grade sulfide-sulfosalt-gold bearing rocks generally form as veins, vugs, and open cavities in fractured and brecciated parts of the fault zones. Many mineralized bodies have a medial seam of high-grade material, historically termed a "stope streak", surrounded by low grade. Spectacular high-grade material often consists of breccia fragments coated with several layers containing quartz, pyrite, famatinite, tetrahedrite-tennantite, bismuthinite, goldfieldite and gold in various combinations. Sphalerite or tellurides also occur in the crusts of some high-grade zones and gold is often clearly visible in one or more of the layers. In typical mineralization, pyrite is the most abundant sulfur-bearing mineral followed by famatinite. Famatinite often encloses subordinate tetrahedrite-tennantite, sometimes minor bismuthinite and a few tiny specks of gold.

Junyoung Sung (University of Utah, M.Sc. Thesis, 2005) defined three stages of mineralization in the Goldfield Mining District:

- Stage I: (pre-mineralization): quartz-alunite-kaolinite-barite-pyrite.
- Stage II (main mineralizing stage):
  - Sub-stages IIa-IId: chalcopyrite, enargite-famatinite, mohite, tetrahedrite and goldfieldite.
  - Sub-stage IIe: bismuthinite.
  - Sub-stages IIf: argentite and hessite.
- Stage III (supergene weathering): chlorargyrite, native silver.

Three distinct structural zones in the Goldfield Mining District gold mineralization have been recognized, and are typified by the Goldfield Main, Gemfield and McMahon Ridge deposits.

7.5.1 Gemfield

The Gemfield deposit is hosted by the Sandstorm Rhyolite which is composed of strongly flow-banded, often glassy, but generally devitrified, porphyritic rhyolite.

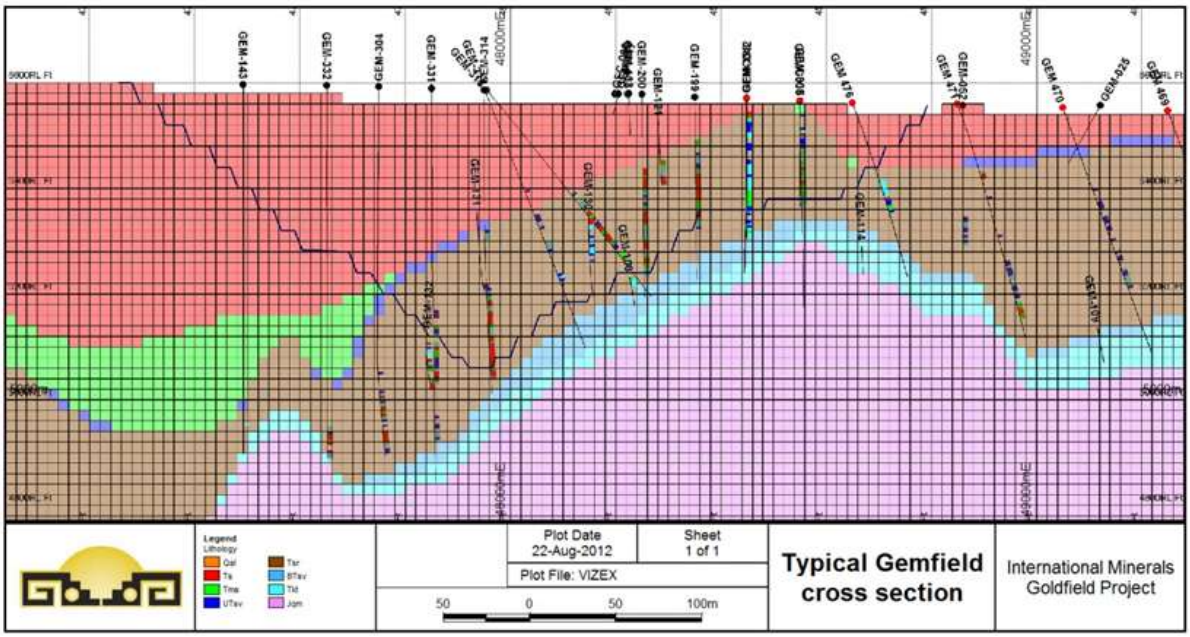
Northeast trending normal structures, which contain most of the higher-grade mineralization, acted as "feeders" from which mineralization spread along shallow-dipping, flow contacts in the Sandstorm Rhyolite resulting in the lower grade, generally stratabound mineralization.

Lava flows of the Sandstorm Rhyolite are almost always hydrothermally altered, including propylitic, argillic, and intense silicification. The widespread distribution of hydrothermal alteration in the rhyolite is due to the highly permeable character of portions of the flow-banded volcanic stratigraphy. Formations above and below the rhyolite are generally weakly altered and unmineralized.

The Gemfield deposit has a known strike length of approximately 2,400 ft and is 1,200 ft wide at its widest point. The depth of gold mineralization beneath barren alluvial cover varies from about 10 ft in the northeast part of the deposit where the Sandstorm Rhyolite has been removed by erosion, to a maximum depth of nearly 700 ft at the southwest margin of the deposit. A representative section of the geology and mineralization is provided in Figure 7.4.

The Gemfield deposit is fault-bounded on at least three sides: east, west, and south.

Figure 7.4
Mineralization Style of the Gemfield Deposit (Cross-Section)



Key: Qal: Aluvium, Ts: Siebert tuff, Tma: Milltown andesite, UTsv: Upper Vitrophyre sandstorm ryolite, Tsr: Sandstorm rhyolite, STsv: Lower Vitrophyre sandstorm ryolite Tkt: Kendall tuff, Jqm: Quartz Monzonite.

7.5.2 Goldfield Main

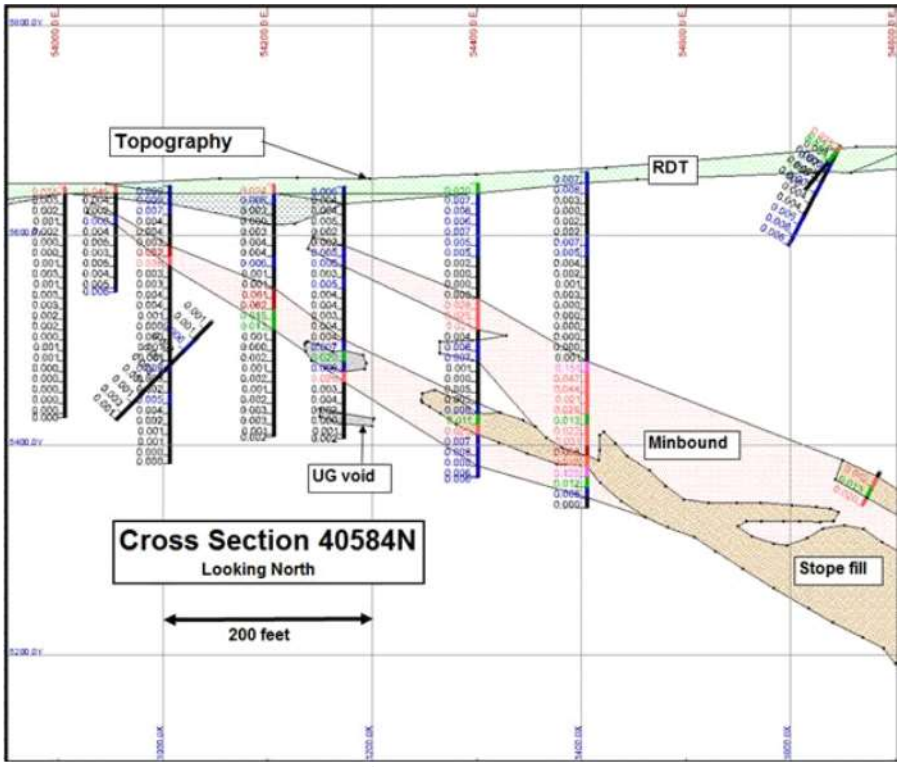
At Goldfield Main, the majority of the gold mineralization occurs within a moderate to shallow, east-dipping, fault known as the Main Mineralized Horizon (MMH). The MMH is exposed in both the Red Top and Combination pits, where it consists of a series of parallel faults, forming a mineralized zone between 100 ft and 200 ft thick. In both pits, the MMH structure dips east at 35º to 55º east.

With increasing depth to the east, the MMH dips at lower and lower angles, 30º or less in the lower levels of the Mohawk and the Clermont mine, and about 20º in the Jumbo Extension and Grizzly Bear mines. According to some authors, the MMH has also been mined in lower levels of the Merger and Jumbo mines. Searls (1948) observed a normal, vertical displacement of up to 200 ft on this fault. Further north, at Columbia Mountain, vertical displacements of 300 ft have been recorded by previous workers.

The Goldfield Main deposit has a strike length of 3,000 ft and a down-dip length of up to 1,500 ft. The average thickness is in the order of 100 ft.

A representative cross section of the Goldfield Main deposit is provided in Figure 7.5.

Figure 7.5
Simplified Sections Showing Mineralization Styles at the Goldfield Main Deposit



Note: Minbound is the approximate limit of in-situ mineralization. RDT is recent dump material, mostly historic tailings.

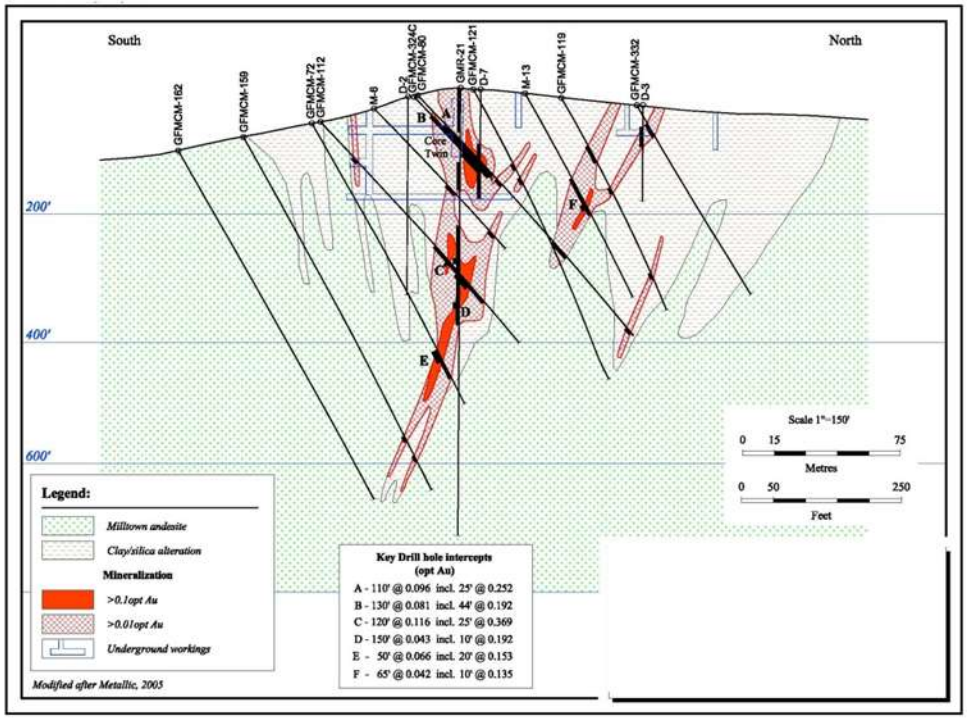
7.5.3 McMahon Ridge

At McMahon Ridge the gold mineralization is controlled by the main ring structure which strikes easterly and dips steeply to the south. Gold mineralization is hosted principally by the Milltown Andesite consisting of tuffs, flows and lahars. Mineralization in deeper drill-hole intercepts is hosted by tuffaceous sedimentary rocks and black shales of the Diamondfield Formation.

Hydrothermal alteration ranges from weak propylitic and argillic in unmineralized rocks to strong argillic and intense silicification in the mineralized zones. Mineralization is generally structurally controlled with the possible exception being the apparent stratabound character of mineralization in the Diamondfield Formation.

A representative section of geology and mineralization is provided in Figure 7.6.

Figure 7.6
Simplified Sections Showing Mineralization Styles at the McMahon Ridge Deposit



8.0 DEPOSIT TYPES

The Gemfield, McMahon Ridge, and Goldfield Main deposits are structurally controlled, volcanic-hosted, epithermal gold deposits of the high-sulfidation, quartz-alunite type. Other examples of the deposit type include Paradise Peak (Nevada, USA), Summitville (Colorado, USA), Pierina and Yanacocha (Peru), Nansatsu (Japan), El Indio (Chile), Temora (New South Wales, Australia), Pueblo Viejo (Dominican Republic), Chinkuashih (Taiwan), Rodalquilar (Spain), Lepanto and Nalesbitan (Philippines).

High-sulfidation systems of this type are commonly found in extensional and transtensional settings in continental margin and oceanic arc settings. They commonly occur in zones with high-level magmatic emplacements where andesitic stratovolcanoes and other volcanic edifices are constructed above plutons. High-sulfidation epithermal systems have been shown to overlie, and be genetically related to, porphyry copper systems within subvolcanic stocks. Other settings include calderas, flow-dome complexes and maars (the latter being rare).

Host rocks are typically volcanic pyroclastic and flow rocks, commonly andesite to dacite in composition, and their subvolcanic intrusive equivalents. Permeable sedimentary intervolcanic units can also be sites of mineralization.

Mineralization typically forms in vuggy bodies, veins, stockworks and breccias. Commonly, irregular deposit shapes are determined by host rock permeability and the geometry of ore-controlling structures. Multiple, cross-cutting composite veins and breccias are common.

The most common minerals in high-sulfidation deposits are pyrite, marcasite, enargite/luzonite, famatinite, chalcocite, covellite, bornite, gold, and electrum. Deposits can also contain chalcopyrite, sphalerite, tetrahedrite/tennantite, galena, marcasite, arsenopyrite, silver sulfosalts, and tellurides including goldfieldite.

Most of the production from the Goldfield Mining District has come from composite silicified zones consisting of an early quartz-alunite-pyrite phase, with low levels of gold; followed by a later, gold-bearing quartz-dickite pyrophyllite phase. In the Goldfield Mining District, these deposits are locally referred to as "ledges," which generally consist of one or more, shallow to steeply dipping, vein-like, irregular, silicified zones.

9.0 EXPLORATION

The more recent exploration programs undertaken at the Property are discussed in this section of the Technical Report.

9.1 METALLIC VENTURES GOLD INC. (MVG) EXPLORATION PROGRAMS

9.1.1 MVG Exploration Program – 2001

MVG began exploration in the Goldfield Mining District in 2001 and the southeastern part was the initial target. Exploration work done during this time included geological mapping and geochemical sampling. The results of this program identified six previously unrecognized drill targets. Each of these drill targets occurred along a major east-southeast trending structural zone from the Goldfield Main Deposit.

Other work included:

- A detailed mapping and geochemical sampling program in the Jumbo open pit area.
- A district-scale geological mapping program from the McMahon Ridge-Black Butte area eastward along the northeast extension of the main regional ring fracture system.
- A detailed geological mapping program in a number of other areas, mainly in the south-eastern part of the district.
- The completion of approximately two line-miles of soil gas geochemistry in the north-western part of the district.

9.1.2 MVG Exploration Program – 2002

In 2002 MVG completed 203 exploration drill-holes for a total of 76,072 ft. The program was a combination of 73,655 ft of reverse circulation (RC) and 2,417 ft of core drilling and was designed to investigate the McMahon Ridge, Goldfield Main, and Gemfield Deposits.

A total of 155 RC and three core drill-holes were completed in the McMahon Ridge deposit. 12 RC and five core drill-holes were completed in the Goldfield Main deposit. The drilling at Goldfield Main was centred in the area between the Red Top and Combination open pits. At Gemfield, 28 core drill-holes were completed in order to twin (duplicate) existing RC holes to confirm confidence in the RC data. Assay results indicated good correlation.

Drilling at McMahon Ridge was designed as a two-phased program. Phase I was infill drilling between existing drilling to test the continuity of the deposit at a nominal drill-hole spacing of 100 ft. Phase II consisted of a combination of additional infill and step-out drilling. The Phase II program added 3,400 ft of strike length to the McMahon Ridge deposit, increasing the identified strike length to 4,800 ft.

9.1.3 MVG Exploration Program – 2003-04

A total of 449 drill-holes were drilled for a total 213,736 ft from March 2003 to March 2004. The goal of the program was in-fill drilling at Goldfield Main and McMahon Ridge, metallurgical sampling, limited deep drilling at Goldfield Main, condemnation drilling, and testing of identified geologic targets.

A total of 373 drill-holes were completed from March through December, 2003 for a total of 166,666 ft.

In the McMahon Ridge deposit 54 RC holes and four core drill-holes were completed. The RC drilling was designed to, and was successful in, increasing the known mineralized area. The core drill-holes were sited to obtain material for metallurgical testing and were located to twin existing RC drill-holes. Assay values correlated well for the core and RC drill-holes.

At the Gemfield deposit, 187 RC and six core drill-holes were completed. The RC program was designed to reduce nominal spacing to about 100 ft and to provide close-spaced data for the development of a geologic model. The drilling located additional high-grade mineralization and better defined the geometry of the central "ledge". In addition several splays and offshoots of high-grade mineralization on the northeast and southwest edges of the deposit were identified. The core drill-holes again were designed to provide material for metallurgical testwork and to twin existing RC drill-holes. Condemnation drilling completed during 2003 consisted of 28 RC drill-holes and were sited to test areas tentatively selected for construction of surface facilities, such as proposed waste rock dumps and process plant locations.

A total of 94 RC drill-holes were completed in 2003 to test previously-identified geologic targets. 10 drill-holes were completed at the Tom Keane prospect in the extreme southeast of the Goldfield Property. This area had seen a small amount of historic production and had been identified as a potential target for additional mineralization. Encouraging results from this program included drill-hole TK-5 with 145 ft grading 0.03 oz/T of gold and TK-6 with 75 ft with an average gold grade of 0.084 oz/T.

A total of 76 RC holes were drilled from January through March 2004 for a total of 47,070 ft. An additional 25 RC holes were drilled at McMahon Ridge as infill drilling.

MVG constructed a new geologic model with the information resulting from the 2003/2004 drill program. Condemnation drilling continued in 2004 with 39 condemnation holes drilled in the area of the Adams Mine, which is located between the McMahon Ridge and Gemfield deposits. 12 RC holes were drilled in the area of the Goldfield Main area and were designed to test the deep potential at Goldfield Main.

9.1.4 MVG Exploration Program – 2005

In 2005 MVG completed 14 RC holes for a total of 6,300 ft to explore extensions of the mineralization in six areas outside of the Gemfield deposit. Results of this program indicated that there was potential for the discovery of additional gold mineralization beyond the limits of that currently defined.

9.1.5 MVG Exploration Program – 2006

A limited drilling program was undertaken by MVG in June, 2006. Ten RC drill-holes for a total of 2,380 ft were completed within the Gemfield deposit. It was intended to more clearly define the continuity of gold mineralization in a near-surface mineralized block at Gemfield for use in a preliminary economic assessment (PEA) report. Results were as expected with shallow, high-grade, near-surface intercepts reported.

9.2 IMZ EXPLORATION PROGRAM – 2010-2013

Following IMZ's February 2010 acquisition of MVG, IMZ initiated a RC drilling program in April 2010 to target the Main Mineralized Horizon (MMH) at the Goldfield Main deposit.

By January 2012, IMZ had completed 174 RC drill-holes, totaling 146,895 ft at the Goldfield Main deposit.

RC drilling was also carried out in 2011 at four targets outside of the three known deposits. These drill targets were: Florence, Kendall, Tailings and North Gemfield. By January, 2012, a total of 58 RC drill-holes, totaling 26,405 ft had been completed.

Core drilling (including both twinning of RC and metallurgical drill-holes) was initiated at the end of 2010 at the Gemfield, McMahon Ridge and Goldfield Main deposits. By January 2012, 54 drill-holes, totaling 29,288 ft, had been completed at Gemfield; eight drill-holes, totaling 3,112 ft. at McMahon Ridge; and 15 drill-holes, totaling 10,463 ft, at Goldfield Main.

Detailed geologic mapping, soil sampling over large portions of the Property along with PIMA clay alteration studies were also carried out to help define future drill targets.

10.0 DRILLING

A description of the historic drilling conducted on the Property is provided in Section 9.

10.1 SUMMARY OF PRE-ROMARCO DRILLING

Many exploration drill campaigns in and around the Property were undertaken prior to 1999. Records of this earlier drilling are only partially complete. Table 10.1 lists the drill program dates, companies, drill-hole numbers and other information.

Table 10.1
List of Drilling Campaigns Prior to 1999

First Hole	Last Hole	Company	Date Drilled	Drill Type
5000	5216	Noranda	1980-1982	Reverse Circulation
798	7924	Pacific Gold	1979	Rotary (?)
A-1	A-14	Cordex	1975	Down-the-hole-hammer (?)
B-1	B-5	Red Rock	1992	Reverse Circulation (?)
C-1	C-5	Red Rock	1992	Reverse Circulation (?)
D-1	D-12	Cordex	1975	Rotary
DR-1	DR-5	(?)	(?)	Core
Bonz-1	Bonz-15	Western Minerals Exploration Co.	1981	Reverse Circulation (?)
CGF-1	CGF-22	Cameco	1993-1994	Reverse Circulation, CGF-19 core
CGF-23	CGF-39	Cameco/Granges	1995	Reverse Circulation
EG-1	EG-6	Santa Fe	1985-1986	Reverse Circulation
GB-1	GB-10	Meridian Precious Metals	1982	Reverse Circulation
GD-1	GD-3	Kennecott	1995	Reverse Circulation
GE-1	GE-3	Noranda	1981	Rock bit/Core
GEM-2	GEM-164	Kennecott & Franco Nevada	1994-1997(?)	Reverse Circulation
GF-01	GF-3	Meridian	1982	Reverse Circulation
GF-1	GF-11	Hanna Costal Mining	1979	Rotary/Core
GF-91-1	GF-91-8	Crown Resources/Red Rock?	1991	Reverse Circulation
GFC-001	GFC-009	Kennecott	1994-1995	Core
GK-1	GK-44	Kennecott	1994-1996	Reverse Circulation
GFMCMC-1		North	1997	Core
GMR-1	GMR-34	Kennecott	1994-1995	Reverse Circulation
GMW-1	GMW-20	Kennecott	1994-1995	Reverse Circulation
JV-1	JV-69	Dexter	1986-1987	Reverse Circulation
K-1	K-3	Red Rock	1992	Reverse Circulation
Linda-1	Linda-32	National Energy Corp.	1973	Rotary
M-1	M-17	Red Rock Mining/ARC	1989-1993	Reverse Circulation
P-1	P-129	Westley	1985-1987	Reverse Circulation
PM-1	PM-29	Santa Fe	1984-1985	Reverse Circulation
R-1	R-322	Red Rock Mining	1988-1991	Reverse Circulation
R-323	R-456	ARC	1992-1994	Reverse Circulation
R-458	R-461	American Pacific Minerals (ARC)/(Kennecott)	1994	Reverse Circulation
R-462	R-553	American Pacific Minerals (ARC)	1994-1995	Reverse Circulation
RK-533	RK-545	Kennecott	1994-1995	Reverse Circulation
SK-1	SK-21	Western Minerals Exploration Co.	1981-1982	Reverse Circulation (?)
SSK-1	SSK-8	Red Rock	1992	Reverse Circulation (?)
T-83-1	T-83-6	Noranda	1983	Reverse Circulation
TC-5	TC-9	Western Minerals Exploration Co.	1982	Reverse Circulation (?)
TF-92-1-R	TF-92-11-R	Noranda/Cameco	1992	Reverse Circulation
TK-2		(?)	(?)	(?)
UCC-1	UCC-4	Utah Construction	1966	Core

First Hole	Last Hole	Company	Date Drilled	Drill Type
UNK-01	UNK-20	(?)	(?)	(?)
USGS-1	USGS-3	USGS	1968-1969	Core
VE-1	VE-4	Meridian Precious Metals	1982	Reverse Circulation
VE90-4	VE90-9	Crown Resources	1990	Reverse Circulation
WC-1	WC-20	Westley Explorations	1985-1986	Reverse Circulation
GFBC-1		North	1997	Core
GFMGC-1		North	1997	Core
GFMAINC-1		North	1997	Core
VE4-1	VE4-5	Crown Resources(?)	1984	Reverse Circulation(?)
VE9-1	VE9-3	Crown Resources(?)	1989	Reverse Circulation
D9-1	D9-2	Crown Resources	1989	Reverse Circulation
ND1	ND6	Pacific Gold	1979	Rotary
PH1	PHxx	Santa Fe	1984	Reverse Circulation(?)
VE9-1	VE9-2	Crown Resources	1989	Reverse Circulation(?)
C-1	C-15	Placer Dome	1985-1986	Reverse Circulation(?)
RT-5		Placer Dome	1986	Reverse Circulation(?)
AP-1	AP-13	Westley Explorations	1986	Air Track
82MP-1	82MP-2	Meridian Precious Metals	1982	Reverse Circulation(?)
TG80-1	TG80-20	Western Minerals Exploration Co.	1980	Rotary(?)
TG81-21	TG81-132	Western Minerals Exploration Co.	1981	Rotary(?)
GMF-1	GMF-10	Westley Explorations Inc.	1985	Reverse Circulation
ARG-1	ARG-93	Argonaut Company/Blue Bull Mining Co.	1981	Air Track
PG-94-1		Cameco	1994	Reverse Circulation

10.2 ROMARCO MINERALS INC. (ROMARCO) – 1999

Romarco completed 33,865 ft of RC drilling in 77 drill-holes during a single campaign from June to August 1999. All were drilled by the Eklund Drilling Company of Elko, Nevada (Eklund). Three Romarco geologists supervised all drilling operations on a hole-by-hole basis during the entire program.

Sampling of drill cuttings and first-pass geologic chip logging was undertaken on site during the actual drilling process. Later, more detailed chip logging was done at Romarco's Goldfield office where a binocular microscope and infrared spectrometer were used to acquire additional information regarding host and wall rock lithology, hydrothermal alteration, and mineralogical detail.

Generally drill-hole depths during this phase of drilling were relatively shallow (average 440 ft); therefore much of the drilling was dry.

Representative sample splits over five-foot intervals were taken using a Gilson adjustable sample splitter placed beneath the cyclone sample collector attached to the drill rig. Where ground water and/or bad drilling conditions were encountered, water and drilling fluids were injected into the hole to stabilize the hole and to improve fluid circulation and sample recovery. A rotary wet splitter was used to reduce the wet sample size down to about 5 to 10 kg per split. Wet samples were collected beneath the wet splitter in 20 inch by 24 inch polypropylene fabric bags to allow for water filtration, thereby minimizing the loss of fine solids. The bags were placed in a 5-gallon plastic bucket and hung beneath the sample outflow port at the base of the wet splitter during sample collection.

Duplicate samples were collected at the drill rig by taking two equal splits of the recovered sample cuttings collected during the 5 ft long drill runs. The duplicate samples were used for assay checks and/or for back-up samples in case the primary sample split was unavailable for assay due to accident, loss, or vandalism. By the end of the program, the duplicate sample set had only been used for select check assays, and all primary assay samples reached the laboratory without incident and in good condition.

10.3 ROMARCO - 2000

Romarco completed 8,055 ft of RC drilling in 15 holes with an average depth of 537 ft from July to August, 2000. All of the holes were drilled by Eklund. All drilling operations were continuously supervised by two Romarco geologists.

Standard procedures regarding geologic chip logging, drill sample handling, temporary on-site sample storage, and transportation to a contract analytical laboratory were carried out in essentially the same manner as described above for the 1999 campaign, with one exception. Because several of the holes were deeper during this program, unusually high water flows (>10 gpm) were sometimes encountered (e.g. GFBK-84: 1,160 ft and GFBK-89: 965 ft at the "East Goldfield" target area). In cases where the volume of water flow was greater than could be contained in a 20 in. by 24 in. sample bag and supporting 5-gallon bucket, the bag and bucket were placed in a large rubber pan for the purpose of collecting fines from the stream of water that overflowed from the sample bag. In cases where sample slurry overflow was deemed imminent, and prior to actual slurry overflow, a flocculating agent was added to the slurry in the sample bag (and eventually to the contents in the overflow pan) for the purpose of settling fines from the sample slurry. The slurry in the overflow pan was then allowed to settle, clear water was later decanted and discarded, and the dewatered sample fines were added to the sample bag. In some cases, particularly in clay zones with slow drill penetration rates, some sample fines were observed to overflow the second catchment basin, but no effort was made to contain this material.

ALS Chemex was the primary analytical laboratory used during the 2000 drill program.

Sample preparation procedures again included drying when necessary, crushing the entire sample, splitting, and pulverizing to the same standards as described above for the 1999 drill campaign.

10.4 MVG – 2002

10.4.1 Phase I

Between March and May, 2002, four drill rigs completed 22,220 ft in 61 drill-holes. The program included 19,800 ft of RC and 2,420 ft of core drilling.

The handling procedures for RC samples during this campaign varied only slightly from those for the Romarco 1999 and 2000 drilling programs. In this case, because most of the

drilling was restricted to only two geographic locations, the samples were stored at each drill site by the drill contractor and then later loaded on site by ALS Chemex employees and transported to the Elko sample preparation facility.

Two RC rigs were operating simultaneously and therefore the sample load required a minimum of two lab pick-ups per ten-day drill period. For the purpose of sample security, the final sample pickup during a ten-day drill period always collected all remaining samples from the project area so that no sample was left on site during a four-day work break.

HQ-sized drill core was stored in 10 ft capacity waxed-cardboard core boxes on site until the hole was completed. Upon drill-hole completion, the core boxes were transported to the Merger shaft sample storage facility for geologic logging and sampling. Except for a few days at the end of the program, the core drilling operation coincided with the RC drill program in terms of period of operation, but the core drilling operation involved two 12-hour shifts per day; hence, the core stored on site was rarely left unattended.

During the geologic logging procedure, the whole core was first photographed and then placed on wooden benches for core recovery measurements and geologic review. The photographs of the core were initially taken using a single lens reflex camera and conventional color film. Four-by-six inch color photographs of the core were kept in a file in the MVG Reno office. Duplicate digital photographs were prepared by the commercial photo lab during the film development and printing process by scanning the original film negatives. These photographs were stored on CD media in the MVG Reno office.

The HQ-size drill core was sampled by taking a ½ split of the original whole core sample. A ½ split of core was taken by one of the following three splitting techniques.

1. In the case of competent core material, the core was sawn using one of two on-site electric motorized diamond-bladed saws.

2. If the core material was extremely hard, such as typical silica ledge material, the core was split using a hand-operated hydraulic core splitter. In this case, the core samples are first etched on two sides with a shallow pass of the diamond-bladed saw and then these cuts were used to align and secure the jaw blades of the hydraulic core splitter.

3. In the third case, where very soft clay-rich and/or very broken and/or friable core must be split, a steel paint scraper was used for splitting the core.

Finally, bagged half splits of the HQ-size core was transported to ALS Chemex in Elko, Nevada, and the second ½ core split was stored in a secure steel container at the Merger shaft sample storage facility.

10.4.2 Phase II

Procedures for phase II, regarding geologic chip logging, drill sample collection, sample handling, temporary sample storage, and sample transportation to ALS Chemex were essentially the same as those described above for the 2002 phase I drill campaign with one exception. A set of standard samples were prepared by ALS Chemex from material obtained from the Goldfield Main District. These standards were to be included with future primary drill sample shipments for the purpose of laboratory checks.

10.5 IMZ – 2010-2013

Drilling using both RC and core drilling methods started in May 2010 and continued until June 2013. As of the date of this report, a total of 541 RC rotary drill-holes totaling 335,645 ft and 77 HQ core holes totaling 46,657 ft have been completed. Table 10.2 shows the breakdown for the areas drilled at the Property by IMZ.

Table 10.2
Summary of IMZ Drilling Completed in the Goldfield Mining District

Area	RC Holes	Footage (ft)	Core Holes	Footage (ft)	Total Holes	Total Footage (ft)
Goldfield Main	174	146,895	15	10,463	189	157,358
Gemfield	214	102,250	54	33,083	268	135,333
McMahon	26	13,750	8	3,112	34	16,862
Reconnaissance	105	60,715	0	0	105	60,715
Monitor Well	22	12,035	0	0	22	12,035
Total	**541**	**335,645**	**77**	**46657.3**	**618**	**382,302**

Initially, RC drilling was started in the main Goldfield Main deposit concentrating on the shallow east-dipping Columbia Mtn.-Red Top fault. Close to half of all of the recorded historic production of 4 million ounces gold has been produced from this shallow-dipping structure. Between May 2010 and January 2012, a total of 174 RC drill-holes totaling 146,895 ft and 15 core holes totaling 10,463 ft were completed This drilling shown that lower grade gold mineralization, not of interest to past operators, continues at depth from the historic surface open pits to the east for a distance of 1,500 ft and along strike for 3,000 ft.

Drilling at the Gemfield deposit was concentrated around the edges of the known Gemfield resource. This program commenced in January 2011 and to date a total of 214 RC drill-holes totaling 102,250 ft and 54 HQ core holes totaling 33,083 ft have been completed. Exploration on the west side of Gemfield, in the deeper roots of the deposit, did intercept gold mineralization. However, much of this mineralization was sulfide and refractory in nature. Drill programs on the southeast side of Gemfield have encountered zones of low grade (0.3 to 0.4 ppm Au) oxidized Sandstorm Rhyolite. Most of the core drilling at Gemfield was used for metallurgical studies, twinning potential contaminated RC drill-holes, pit slope studies, hydrology packer test studies and drilling in the deeper portions of the Gemfield gold system.

At McMahon Ridge, 8 core holes totaling 3,112 ft were used to provide samples for metallurgical studies and pit slope geotechnical assessment. Reconnaissance RC drilling west of McMahon Ridge totaled 12,050 ft from 23 RC drill-holes. Results received from this drill program have not encountered any significant mineralization.

A number of reconnaissance exploration targets were drill tested with 105 RC drill-holes totaling 60,715 ft.

Detailed geologic mapping, soil sampling over large portions of the Property along with PIMA clay alteration studies were also carried out to help define future drill targets.

10.6 SUMMARY OF SIGNIFICANT RESULTS

Table 10.3 shows some significant assay results from the IMZ drilling programs at the various known deposits on the Property.

Table 10.3
Significant Assay Results from IMZ Drilling Programs on the Property

Area	Drill-hole	Hole Type	Depth (ft)	Intersection from-to (ft)	Intercept (ft)	Gold Value (oz/T)	Gold Value g/t
Goldfield Main	2010008	RC	600	115-130	15	0.0992	3.4
			600	170-180	10	0.1283	4.4
	2010015	RC	785	550-670	120	0.0467	1.6
	2010017	RC	680	450-565	115	0.0933	3.2
				450-495	45	0.1838	6.3
	2010021	RC	680	545-595	50	0.0671	2.3
	2010024	RC	450	255-330	75	0.035	1.2
	2010027	RC	655	590-615	25	0.1954	6.7
	2010032	RC	600	260-320	60	0.2742	9.4
				260-280	20	0.7933	27.2
	2010033	RC	540	240-260	20	0.1954	6.7
	2010035	RC	715	635-655	20	0.4463	15.3
	2010048	RC	885	620-780	160	0.0496	1.7
	2010049	RC	825	25-40	15	0.4229	14.5
	2010055	RC	845	125-145	20	0.0788	2.7
				585-600	15	0.8254	28.3
	2010067	RC	840	475-520	45	0.2129	7.3
	2010074	RC	750	380-450	70	0.0642	2.2
	2010085	RC	685	510-545	35	0.0942	3.23
	2010087	RC	645	555-585	30	0.0613	2.1
	2010089	RC	740	500-605	105	0.0379	1.3
	2010108	RC	900	130-150	20	0.245	8.4
	2010111	RC	780	710-780	70	0.1225	4.2
	2010112	RC	860	730-860	130	0.07	2.4
	2011127	RC	1080	980-1010	30	0.0467	1.6
	2011143	RC	1200	0-20	20	0.0583	2
Gemfield	GMD004C	Core	554	405-445	40	0.11	3.77
	GEM 434	RC	200	105-190	25.9	0.0788	2.7
	GEM 443	RC	220	85-164	24.4	0.1983	6.8
	GEMC025	Core	929	236-248	12.2	0.0904	3.1
	GEMC026	Core	806.5	541-666	38.1	0.0613	2.1
	GEMR005	RC	665	355-400	45	0.0379	1.3
	GEMR008	RC	825	575-625	50	0.07	2.4
	GEM 492	RC	400	145-190	45	0.0376	1.29
	GEM 493	RC	500	120-185	65	0.1418	4.86
	GEM 515	RC	300	80-125	45	0.5209	17.86
	GEM 520	RC	400	220-260	40	0.0324	1.11
	GEM 531	RC	400	160-220	60	0.0318	1.09
	GEM 534	RC	345	30-85	55	0.1068	3.66
McMahon Ridge	GEMC050	Core	899	550-610	60	0.0499	1.71
	MCM002C	Core	353	80-85	5	0.3316	11.37
	MCM003C	Core	603	220-240	20	0.2138	7.33
				244-289	45	0.0863	2.96
	MCM004C	Core	401	230-401	171	0.2663	9.13
	MCM005C	Core	603.5	120-220	100	1.4919	51.15
				332-450	118	0.2669	9.15
Other	KEN003	RC	300	20-80	60	0.0671	2.3

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 CORE/RC CHIP HANDLING AND LOGGING PROTOCOLS

11.1.1 RC Drilling

Procedures for chip handling and logging during the pre-Romarco RC drilling campaigns are not well documented. It is assumed that standard practices and procedures normal to the industry at that time were employed. Eklund Drilling of Elko, Nevada was the main drill contractor during that period. Eklund was, and still is, a well-established drilling contractor with over thirty years of experience drilling gold deposits in Nevada.

RC chip logging and handling procedures were similar for all MVG and IMZ drilling programs. The initial 20 ft of each hole was generally drilled dry. State of Nevada environmental regulations for dust suppression mandate drilling with water injection for the remainder of each drill-hole. Wet RC cuttings were split at the drill rig using a rotary wet splitter. A geologist was assigned to each rig to ensure samples did not overflow the collection bucket. Overflow of the collection bucket would result in the loss of fines from the sample.

The main difference in sampling procedures before and after IMZ acquired MVG was for samples collected where high ground water flow was encountered: pre-IMZ, the overflow was collected in an oversized rubber tub and a flocculent was used to settle the fines. Once the fines had settled the water was decanted from the overflow tub and the fines added to the sample; IMZ drilling did not recover the overflow of fines.

Geologists also collected a representative portion of the chip sample from the reject material for each sample and placed these in covered plastic chip trays. These chips were then logged by the geologist. Lithology, alteration and recovery were logged as the interval was drilled. Plastic chip trays were subsequently stored in a locked IMZ facility at Goldfield, Nevada.

11.1.2 Core Drilling

Core logging and handling procedures were similar for all MVG and IMZ drilling programs. Company geologists first digitally photographed the drill core; the core was then logged for alteration, mineralization, lithology and core recovery. Sampling was performed after completion of the logging.

Relevant information was recorded on paper log sheets by geologists. Subsequent to the core logging the information was transferred into Excel and then imported into a digital database. The digital database enables the use of the recorded information in geologic modeling programs to aid in interpretation of the data. The data transfer from the log sheets to Excel was done by hand entry.

11.2 SAMPLE HANDLING AND SECURITY

11.2.1 Romarco and MVG (1999-2006)

Sample handling and security procedures initiated under Romarco in 1999 continued through to 2006 under MVG. Romarco/MVG personnel were responsible for the drilling and both used similar procedures for sampling and security.

RC samples were collected at the drill rig by a sampling technician. Samples were placed in heavy duty plastic bins and were transported to the storage yard. The samples were stored in the yard until pick-up by personnel from the commercial assay laboratory.

The core drill samples were placed in boxes by the drill contractor. The full boxes were collected on a daily basis and transported to the storage yard. Samples were stored under lock and key at the former Merger Mine site in the Goldfield Main area.

There was little opportunity for anyone to tamper undetected with the samples at any step in the shipping, preparation and assaying process.

All samples, whether RC chips or core were logged, split and packaged on site, prior to being shipped to the lab for assay.

11.2.2 IMZ (2010-2013)

IMZ uses similar sampling and security controls as discussed above.

11.3 SAMPLE PREPARATION AND ASSAYING

11.3.1 Romarco and MVG (1999-2006)

Romarco and MVG used American Assay Labs based in Reno, Nevada, an ISO/IEC 17025 certified facility. ALS Chemex based in Reno, Nevada an ISO 9002 certified facility was also used. The analytical procedures these companies followed are described in the following paragraphs.

The wet samples were oven-dried as the first step in the process. Individual samples were passed through a jaw crusher to produce a nominal minus 10 mesh size, which was then passed through a Jones riffle splitter to obtain a 200 to 400 g split for pulverization. The individual samples were pulverized to 90% passing -150 mesh (106 microns) using a ring grinder. Barren rock was subsequently run through both the jaw crusher and pulveriser between samples to prevent cross-contamination between samples.

A 30 g representative pulp sub-sample was analyzed for gold by fire assay with an atomic absorption (AA) finish with a 5 ppb lower detection limit.

Samples returning grades exceeding 10 g/t Au were re-assayed using the gravimetric assay method as it is more accurate for higher-grade gold samples. Gold-mineralized intervals exceeding about 100 ppb Au were subsequently assayed for silver by aqua regia digestion and AA analysis. In some cases, samples from mineralized intervals were later analyzed using an Inductively Coupled Plasma (ICP) multi-element analysis method.

Silver analyses were done using an aqua regia digestion and the atomic absorption analysis method with a 0.2 ppm lower detection limit. Check samples for this drill campaign were prepared from select sample rejects.

11.3.2 IMZ (2010-2013)

IMZ uses Inspectorate Labs based in Reno, Nevada as the primary laboratory and American Assay Labs in Reno as a check laboratory. Both are internationally recognized and certified to ISO 9002 or ISO/IEC 17025 standards.

11.3.2.1 Sampling Preparation

Samples received at Inspectorate are documented and organized within the laboratory. The Laboratory Information Management System (LIMS) allocates a work order to each set of samples arriving at the laboratory using bar coding and scanning technology.

Once the samples are received and sorted, they are placed in a dryer for 24 hr. The rock and drill-hole samples are dried at 110 ºC, while the soil and stream sediment samples are dried at 60ºC. The dried samples are then prepared for analyses. These samples are pulverized to >85% passing 200 Mesh (75 microns) to reduce the mean grain size, thereby homogenizing the samples. The Inspectorate code for this process is SP-RX-2K. Pulps and coarse rejects from the prepared samples are returned to IMZ and stored in a secured warehouse in Reno, for future reference.

11.3.2.2 Assaying

For both RC and core samples, the prepared sample pulps were analyzed for gold by fire assay using 30 g of sub-sample and are analyzed by atomic absorption spectrophotometry (AA) finish (Inspectorate code Au-1AT-AA). Samples that assayed in excess of 5 g/t Au are re-analyzed with a gravimetric finish to ensure a more accurate result at the higher grades (Inspectorate code Au-1AT-GV).

In addition to the methods outlined above, the prepared sample pulps from selected samples are also analyzed by a semi-quantitative multi-element ICP method. The Inspectorate code for this is 30-AR-TR. The pulps are digested using aqua regia and read on ICP-AES for the 30 elements in the package.

The sample rejects are returned to IMZ and stored in a secured warehouse in Reno, for future reference.

11.4 ASSAY QUALITY ASSURANCE AND QUALITY CONTROL

11.4.1 Quality Control Coverage

11.4.2 Romarco and MVG (1999-2006)

The quality control (QC) data available for assays from campaigns prior to the Romarco exploration programs has not been evaluated. AMEC reviewed the QC data available in the MVG offices prior to the acquisition by IMZ. These data were predominantly generated from drill information acquired during the MVG drilling campaigns. The results were in the form of reported AA duplicates on selected samples in each report and from a check assay program that re-submitted coarse reject samples corresponding to mineralized intercepts. These intervals were identified by MVG geologists (AMEC, 2007).

11.4.3 IMZ (2010-2013)

IMZ uses a comprehensive protocol based on standards, blanks and duplicates for quality control for the current drilling program. The standards are used to monitor accuracy and the blanks are used to monitor contamination.

IMZ uses a head office staff geologist to monitor and implement the QA/QC program and the results of the standards and blanks for deviations from the accepted values. All anomalous results are reported to IMZ's Exploration Manager (based in Reno) for resolution with the laboratory.

IMZ utilizes five standards, four of which were purchased from CDN Resource Laboratories (CDN), and one which was purchased from Rocklabs Ltd. The gold standards range from 0.61 g/t to 6.42g/t.

The samples from the Property are shipped to IMZ's Reno storage facility where the standards and blanks are inserted and the entire sample shipment prepared for delivery to the Inspectorate laboratory in Reno. Either a standard or a blank is inserted every ten samples, with standards and blanks being used alternately. The position of the QA/QC samples is determined in the field during logging and sample tags are reserved for QA/QC purposes so that the sample numbering of the standards and blanks are seamlessly integrated with the samples.

Standards are monitored using control charts similar to the ones shown in Figure 11.1, Figure 11.2 and Figure 11.3. The solid magenta line on the control chart shows the reference value for the standard; the red points show the assay values reported by the lab; and the red lines define the acceptable range for each standard.

Figure 11.1
Goldfield Control Graph (Shewhart) Standard CDN-GS-2G

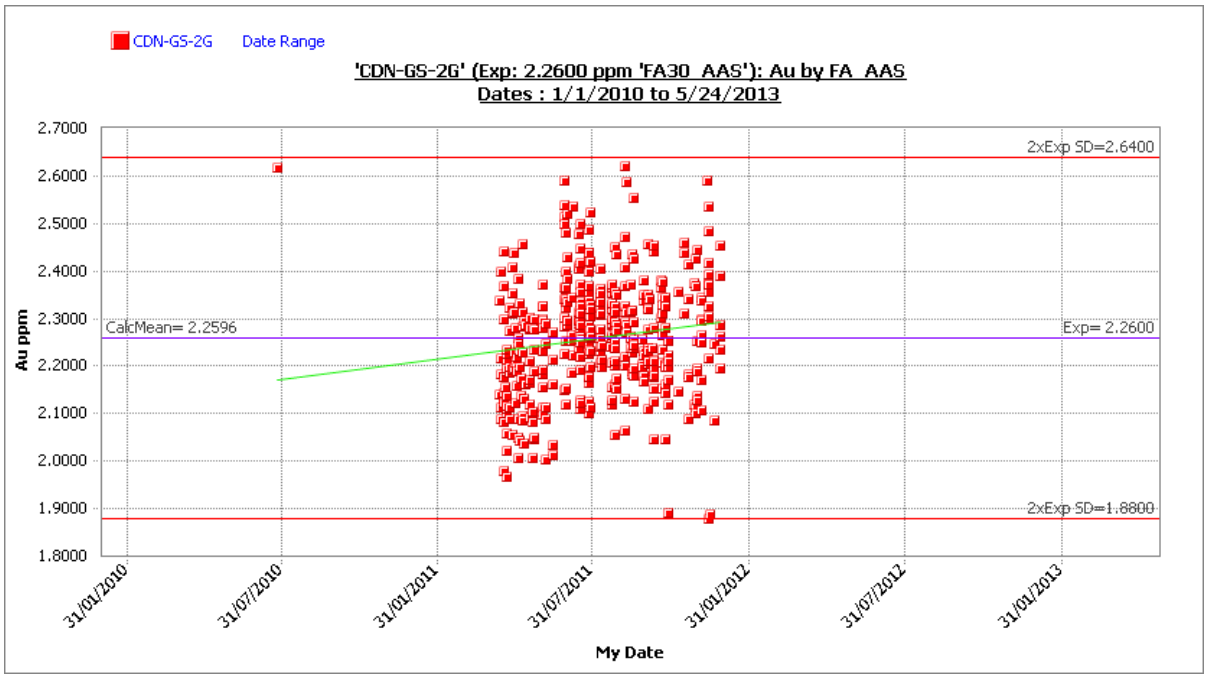


Figure 11.2
Goldfield Control Graph (Shewhart) Standard CDN-GS-1F

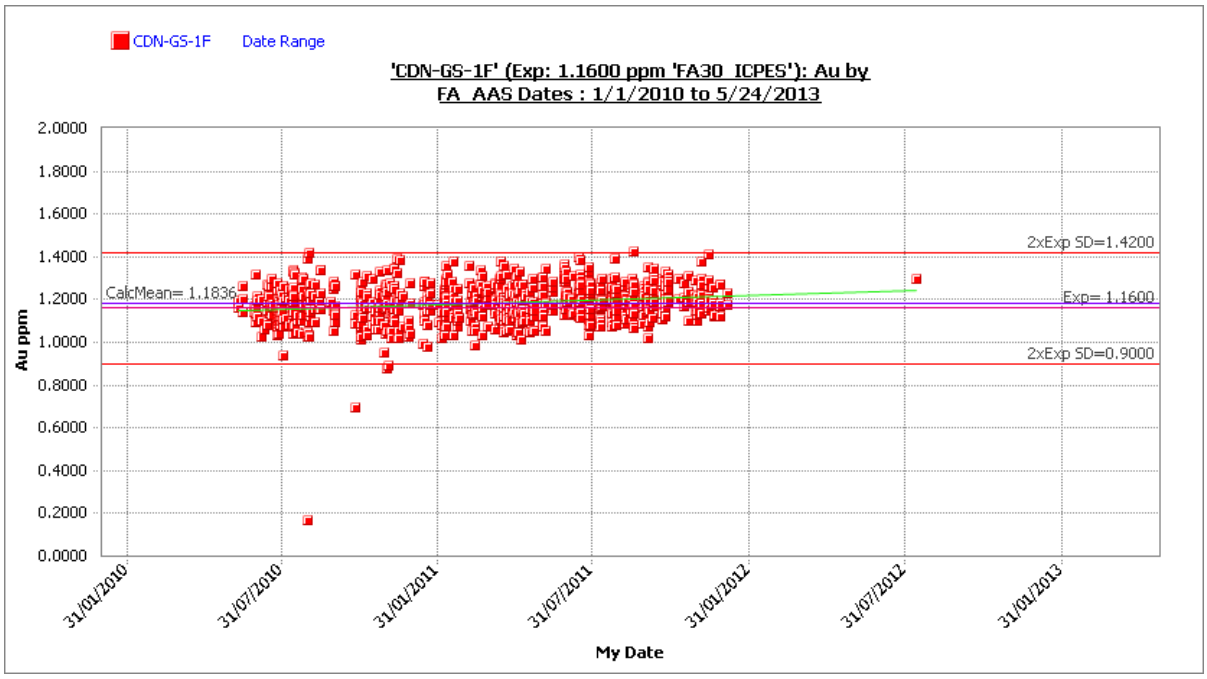
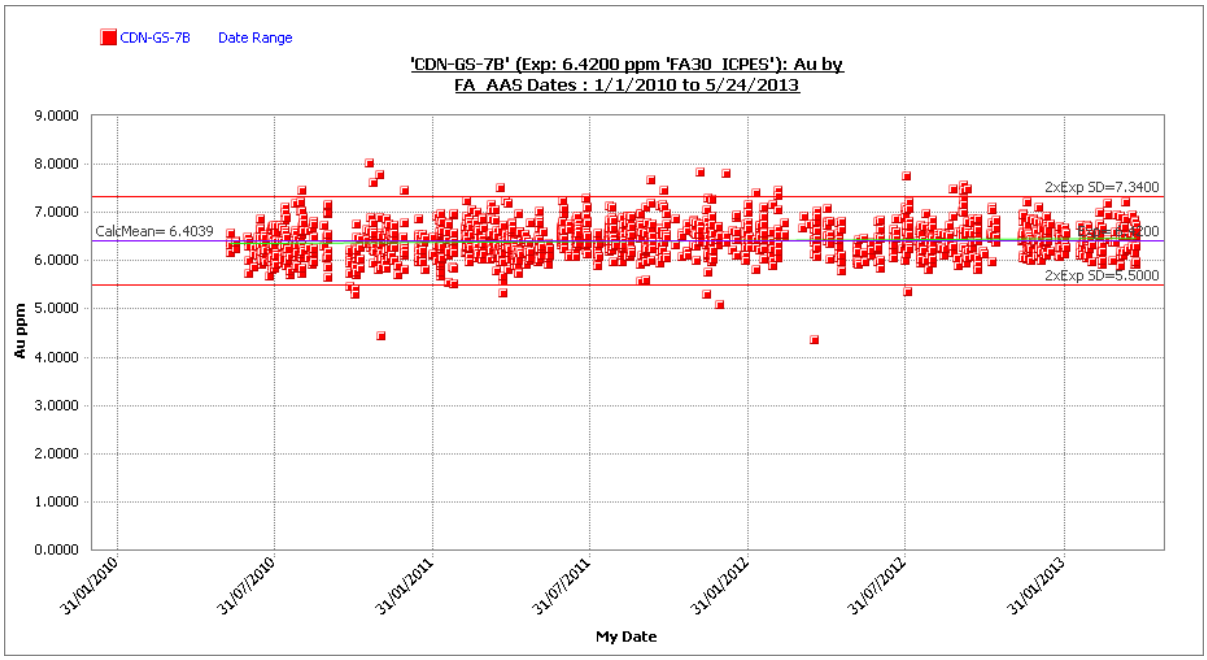


Figure 11.3
Goldfield Control Graph (Shewhart) Standard CDN GS 7B



The QA/QC database records the specific action taken on each failure. In all cases, discussions with the laboratory and a review of the other assays in the same batch led to a decision that no samples needed to be re-analyzed. A summary of the statics relating to the gold standard analyses is presented in Table 11.1.

Table 11.1
Summary Statistics for Gold Standards

Au Standard(s)			No. of Samples	% of Total Samples
Standard Code	Value	SD		73,691
CDN-BL-6-7-9-10	0	0	1449	1.97
CDN-GS-1F	1.16	0.13	903	1.23
CDN-GS-1H	0.97	0.11	292	0.40
CDN-GS-2F	2.16	0.24	468	0.64
CDN-GS-2G	2.26	0.19	427	0.58
CDN-GS-2L	2.34	0.05	200	0.27
CDN-GS-2J	2.36	0.2	329	0.45
CDN-GS-7B	6.42	0.46	1428	1.94
CDN-GS-P7E	0.77	0.03	231	0.31
SE44-SE58-SE68	0.61	0.06	1474	2.00

IMZ uses two types of duplicate check analyses, pulp duplicates and reject duplicates (see Figure 11.4 and Figure 11.5). Approximately 15% of samples that assayed above 0.3 ppm

gold are sent for duplicate analysis. The pulp duplicates are sent to a second lab; the reject duplicates are re-assayed at the lab that performed the original analysis.

Figure 11.4
Goldfield Pulp Duplicates

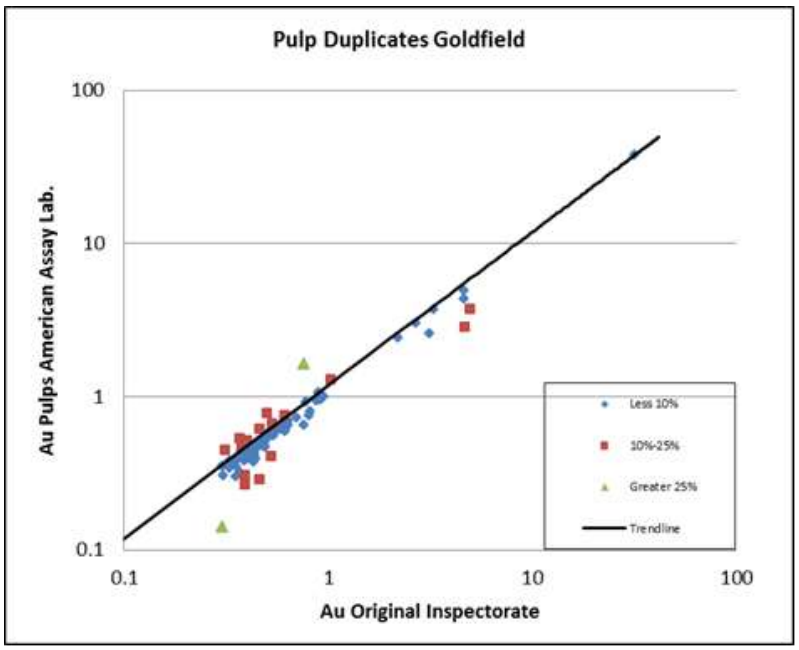
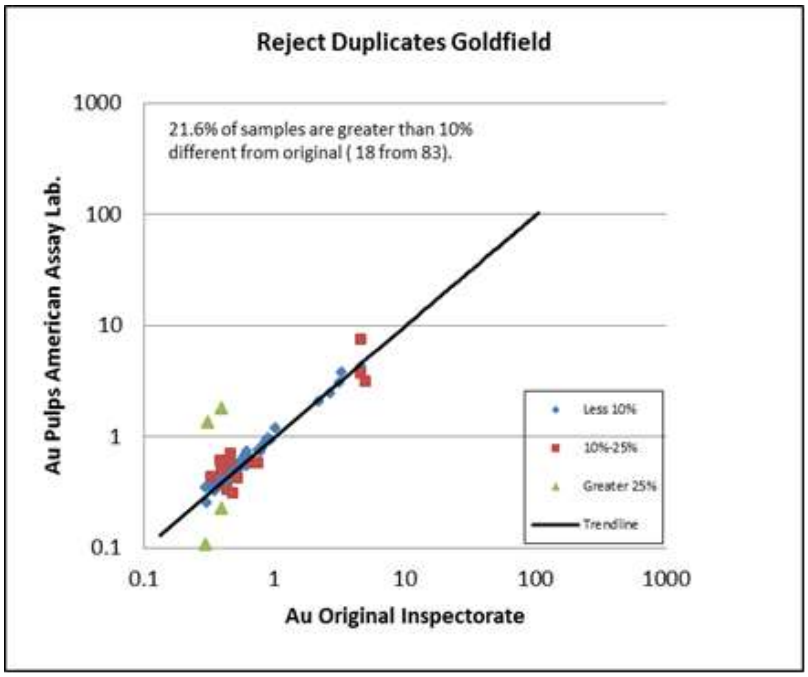


Figure 11.5
Goldfield Reject Duplicates



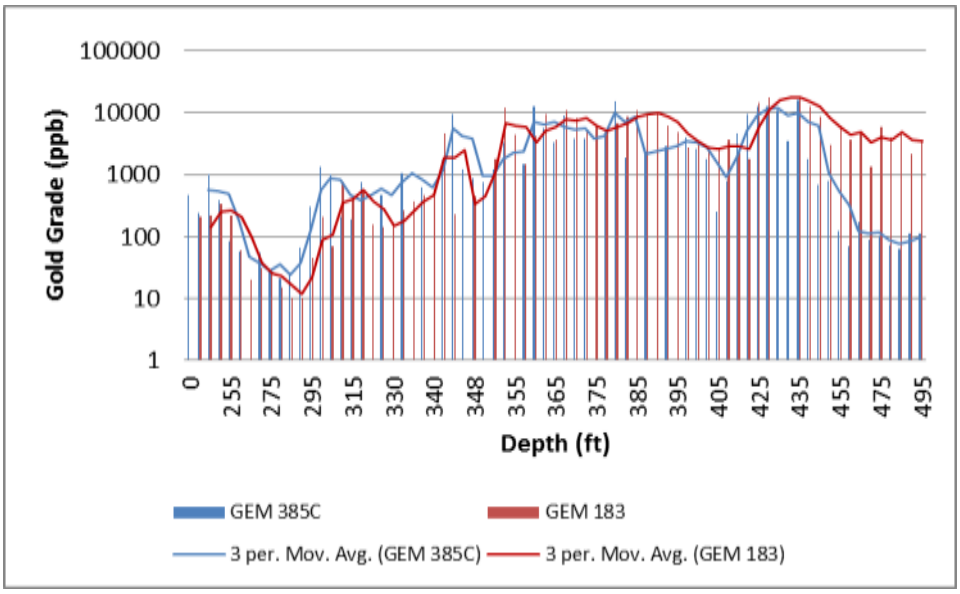
11.5 TWIN DRILL-HOLES (CORE AND RC)

11.5.1 Romarco and MVG (1999-2006)

MVG, prior to acquisition by IMZ, had twinned several RC drill-holes with core drill-holes. In its report, WGM noted that for over the entire mineralized zone the RC and core drill-hole assays were, on average, within 2% of each other for the Gemfield deposit, and within 4% of each other for McMahon Ridge deposit (WGM, 2005). An example of one of these twin-hole comparisons is presented in Figure 11.6.

The results showed that the RC drill-hole grades were usually higher by about 10%. The reason for this difference is not known but should be considered as added risk to the resource estimation process. It may be a consequence of either selective gold loss from core-drilling, or gold enrichment in RC cuttings, or a combination of both. Selecting high-grade RC drill-holes to twin with core-holes may exaggerate the effect if mineralization occurs in lenses that have dimensions that are less than or similar to the distance separating the holes.

Figure 11.6
Core to RC Drill-Hole Twin Comparison (GEM 385C vs GEM 183)



MVG reported that these differences were reviewed and most of the differences in gold assay between twin holes generally corresponded to changes in the geologic environment within each drill-hole. The PQ size core size used in the MVG drilling program in 2003 resulted in core holes that were very straight (i.e. little deviation) when compared to the twin RC hole. The twin RC holes showed significant drift (deflection) at depth. Thus the drill-hole separation distance increases between twinned drill intervals with increasing drill-hole depth. As the drill-hole separation increases, geological variations are more likely to occur. This is especially true in the case of mineralized zones in bonanza gold systems such as those in the Goldfield District.

11.5.2 IMZ (2010-2011)

After carefully reviewing the geologic cross sections, there were four suspected RC contaminated drill-holes, especially in the deeper and higher grade portion of the Gemfield deposit. These four RC holes were twinned with HQ core and IMZ noticed the same effect observed by MVG - that the higher-grade silicified (ledge) material in the RC drill-holes was causing down-hole contamination. As a result, only the gold zones intercepted in the core holes were used in the geologic cross sections in these areas. The comparisons showed that the gold grades were similar in the RC and core holes but the thickness of the gold zones were highly exaggerated in the RC drill-holes.

11.6 CONCLUSIONS

In the opinion of the QP responsible for resource estimation, the assay data have an accuracy and precision adequate for resource estimation purposes.

12.0 DATA VERIFICATION

IMZ verified the Goldfield historical database (GHDB) when the Property was first acquired and on two occasions since then, when new data became available. Each of these data verification exercises consisted of checks of the entries in the electronic database against original assay certificates from the laboratories.

In 2009 the review of the GHDB focused on 132 drill-holes from the Gemfield deposit and 187 drill-holes from McMahon Ridge deposit. The procedure consisted of:

- Ensuring that 100% of the drill-holes had been loaded into the database.
- Identifying the percentage of geochemically anomalous samples in the records (over 300 ppb in gold).
- Randomly selecting 5% of the total of geochemically-anomalous records, checking that the correct units were used. The reason for this check was that the database records values in g/t (ppm) as well as in ppb, while the majority of the original certificates had gold values in ppb, but some are reported in ppm. This check confirms that there has been no inadvertent mistake in the units of measurement between the certificates and the database.
- A record-by-record check of the selected 5% to confirm the grade values, creating a database of every difference. For each record the original gold ppb data value found in the Access database was compared to the Au ppb data found in the assay certificates and the difference and percentage difference were calculated.
- Once these results were reviewed, “AU_PPB Original” data were plotted versus “AU_PPB IMZ” checked data in a scatter plot.
- The scatter plot was used to check the accuracy of the data.

In 2010, the same method was used to analyze the GHDB; this exercise focused on 130 Gemfield drill-holes and 226 Goldfield Main drill-holes.

The method used in 2011 consisting in checking 20% of the historical holes that had not been previously checked. A total of 480 drill-holes from all three deposit areas were checked.

These verification exercises revealed that approximately 95% of the database records matched exactly with the original assay certificates. Of the 5% where discrepancies were detected, the vast majority of these were differences between a value that carried decimal places and a rounded value. For the very few cases where the database value was wrong (usually missing, when a value actually existed), the database was corrected.

In the opinion of the QP, the verification of the database has been thorough and has been appropriately updated when new data have been entered. The resulting electronic database is reliable for the purposes of resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 SUMMARY

Various sources of information address mineral processing and metallurgical testing at the Property. They include:

- Results from previous testing by others.
- A metallurgical overview report dated April 1995 by American Resource Corporation (ARC) which discusses their heap leach operations treating feed from Goldfield Main in the early to mid-1990s.
- Results from testing by IMZ.

13.2 PREVIOUS TESTING

A review of information in MVG's records revealed that older files generally consisted of poorly documented testing, inadequately identified samples and informal handwritten notes. These have been largely ignored, but comments on more recent reports produced by recognized independent testing facilities are summarized as follows.

A more detailed description of the previous testing can be found in the NI 43-101 Technical Report "Feasibility Study on the Goldfield Property, Nevada, USA" by Micon, with an effective date of 17 July, 2012 (Micon, 2012).

13.3 CURRENT TESTING BY IMZ

13.3.1 Gemfield

Drill core was collected from core-holes GEM C001, 002, 003, 004 and 005 representing material from what was then described as Gemfield East and West (now the Gemfield deposit) and assembled into four composites (low, medium, high and very high-grade), for column testing at 100% passing 1.5 inches and 0.5 inches. Based on previous work the composites were agglomerated with a limited quantity of cement and allowed to cure prior to the onset of irrigation. No blinding or ponding was noted as the tests progressed.

Results are summarized in Table 13.1 and show good response to column leaching at either crush size, with higher recoveries, cyanide and lime consumption at the finer size.

The locations of the drill-holes used for this phase of testwork are shown in Figure 13.1.

Table 13.1
Summary: Gemfield Column Leach Test Results

Sample	Feed Size P_{100} (mm)	Leach Time (days)	Feed Grade Au (g/t)	Au Recovery (%)	NaCN Consumed (kg/t)	Cement Added (kg/t)
Low grade	38	86	0.41	82.9	1.34	2.5
Low grade	12.5	86	0.38	86.8	1.78	5.0
Medium grade	38	86	1.10	85.5	1.20	2.5
Medium grade	12.5	86	1.16	89.7	1.79	5.0
High grade	38	86	5.01	73.5	2.13	2.5
High grade	12.5	86	4.07	85.0	2.24	5.0
Very high grade	12.5	86	7.44	90.2	3.49	5.0

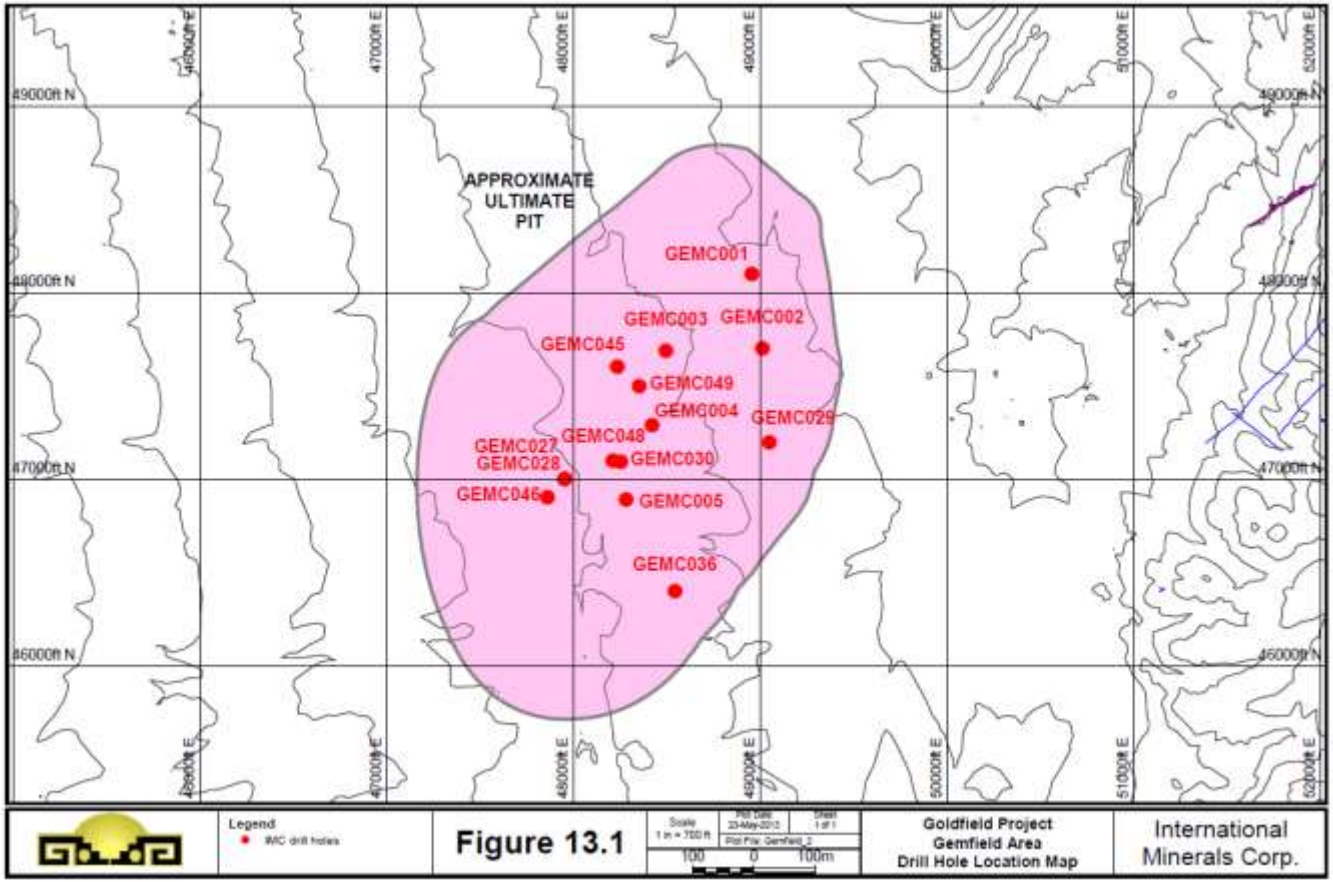
Bottle roll testing at 80% passing 75 and 45 microns (the latter also with carbon-in-leach (CIL) testing) on splits of the column composites, as well as individual variability samples taken from elsewhere on the Gemfield deposit, averaged gold recoveries in the range of low to mid 90% for gold, low to mid 50% for silver, relatively low cyanide and lime consumption, and no evidence of preg-robbing.

Column residues (non-rinsed, air dried) were submitted to Knight Piésold (KP) in Elko, Nevada, for load-permeability testing. Results indicated that minus 1.5 and 0.5 inch residues could both be stacked on the leach pad significantly higher than the 200 ft established for the original Gemfield Feasibility Study design purposes.

As the geology of the Gemfield deposit has become better understood and modeled by IMZ, it is apparent that gold grade increases significantly with the degree of silicification in the oxidized rhyolite.

Drill results were classified into four levels of increasing silicification, numbered 1 thru 4, and additional testwork was focused on their different metallurgical characteristics.

Figure 13.1
Gemfield Deposit – Location of Drill-holes Used for Phase 3, 4 and 5 of the Metallurgical Testwork



Drill core was collected from core-holes GEM C027, 028, 029 and 030 and assembled into 14 composites for column testing, representing a range of gold grades within the various silicification classes (plus two additional composites classified as non-silicified). The locations of the drill-holes used for this phase of testwork are shown in Figure 13.1.

All the composites were column leached at 100% passing 0.5 inches, corresponding to the base case crush size selected for the Feasibility Study. Selective re-compositing of excess material was used to set up a limited number of parallel tests on coarser-sized material. The composites were not agglomerated prior to the onset of irrigation. No blinding or ponding was noted as the tests progressed.

Column test results are summarized in Table 13.2 and generally show good response to column leaching at all the crush sizes tested. Gold recoveries tended to decrease and cyanide consumption increase with increased silicification, but lime consumption was relatively low for all samples.

Table 13.2
Summary: Column Leach Test Results

Sample	Feed Size P_{100} (mm)	Leach Time (days)	Feed Grade Au (g/t)	Au Recovery (%)	NaCN Consumed (kg/t)	Lime Added (kg/t)
4-2+3+4	38	48	3.38	59.2	1.88	0.5
1-2+3	38	48	0.44	95.5	0.97	0.5
1-2+3	25	48	0.52	94.2	1.69	0.5
0-1	12.5	48	0.52	96.2	1.44	0.5
0-2	12.5	48	<0.06	>83.3	1.41	0.5
1-1	12.5	48	2.73	92.7	1.66	0.5
1-2	12.5	48	0.78	94.9	1.48	0.5
1-3	12.5	48	0.42	92.9	1.63	0.6
1-4	12.5	48	0.17	88.2	1.86	0.5
2-1	12.5	48	0.90	92.2	1.81	0.4
2-2	12.5	48	0.65	92.3	1.61	0.4
2-3	12.5	48	<0.24	>95.8	1.56	0.4
3-1	12.5	48	3.37	85.5	1.91	0.4
3-2	12.5	48	1.53	83.7	1.91	0.4
3-3	12.5	48	0.51	92.2	2.68	0.4
4-1	12.5	48	14.84	57.5	2.53	0.4
4-2	12.5	48	7.09	39.5	5.58	0.5
4-3	12.5	48	3.99	71.2	2.88	0.5
4-4	12.5	48	1.01	72.3	3.66	0.5

Leached residues (non-rinsed, air dried) from the columns were submitted to KP for load-permeability testing and confirmed earlier results. Material crushed to 100% passing 0.5 inches can be stacked on the leach pad to 300 ft and still maintain adequate permeability. This compares to the 200 ft considered for the Updated Study.

Bottle roll testing at 80% passing 75 and 45 microns, (the latter with CIL) was carried out on core samples collected from below the oxide/sulfide (redox) boundary. Gold recoveries were significantly lower than those encountered in oxidized material, ranging between the low 10's and mid-20%'s, confirming the need to restrict mining to the oxidized zones of the Gemfield deposit.

Random drill-core samples selected to represent increasing levels of silicification were submitted to Phillips Enterprises, LLC, in Golden Colorado (Phillips) to determine crushing and abrasion indices. Bond crushing indices were quite moderate, ranging between 5.5 and 6.8 kWh/T. Abrasion indices ranged between 0.0498 to 0.5404.

For the most recent testing, drill-core was collected from holes GEM C001, 002, 003, 004, 005, 036, 045, 046, 048 and 049 (drill-holes locations are shown in Figure 13.1) and assembled into four composites (low, medium, high and very high-grade) for column testing at 100% passing one and 0.5 inches. Lime was added to the composites prior to loading the columns based on bottle roll test results. The samples were not agglomerated but no blinding or ponding was noted as the tests progressed.

Results are summarized in Table 13.3 and show good response to column leaching at either crush size. As previously concluded from earlier testing, except for very high grade samples (generally in excess of 3ppm gold) recovery appears relatively insensitive to crush size in the range tested (100% passing 1.5 to 0.5 inches). Recoveries fit the model proposed in the original Feasibility Study. Leaching was essentially complete in all tests at or before the point of 2.5 tons of solution per ton of ore previously established as the limit for primary leaching.

Based on these and all prior test results it was concluded that 100% passing one inch should be used as the basis for plant design.

Table 13.3
Summary: Column Test Results

Sample Name	Calculated Head Grade Au ppm	Crush Size P100 mm	Terminal Tons Solution Per Ton of Ore (t:t)	Gold Recovery %	
				2.5 t:t	Terminal
P-1 Low Grade	0.40	25.0	2.6	91.7	92.5
P-2 Medium Grade	0.78	25.0	2.7	85.9	85.9
P-3 High Grade	1.79	25.0	2.6	82.5	82.7
P-4 Very High Grade	8.10	25.0	4.1	67.0	67.9
P-5 Low Grade	0.43	12.5	2.0	N/A	86.0
P-6 Medium Grade	0.63	12.5	2.0	N/A	90.5
P-7 High Grade	1.76	12.5	2.8	81.8	81.8

13.3.2 McMahon Ridge

A more detailed description of the IMZ testwork at McMahon Ridge can be found in the NI 43-101 Technical Report “Feasibility Study on the Goldfield Property, Nevada, USA” by Micon, with an effective date of 17 July, 2012 (Micon, 2012).

The conclusions can be found in Section 13.4 of this report.

13.3.3 Goldfield Main

A more detailed description of the IMZ testwork at Goldfield Main can be found in the NI 43-101 Technical Report "Feasibility Study on the Goldfield Property, Nevada, USA" by Micon, with an effective date of 17 July, 2012 (Micon, 2012).

The conclusions derived from the testwork can be found in Section 13.4 of this report.

13.4 CONCLUSIONS

Based on all the information available to date (including operational problems addressed in the ARC Report) it has been concluded that Goldfield Main is not a candidate for heap leaching. Similarly McMahon Ridge requires further investigation but seems to be more suited to a milling operation. The decision has therefore been made to focus exclusively on the Gemfield deposit for development as a heap leach operation:

- Gemfield is amenable to heap leaching but mining should be limited to the oxidized rhyolite (i.e. from above the redox boundary established in geological modeling).

- Load permeability testing on column tails indicates that at 100% passing 0.5 inches the ore could be stacked on the leach pad up to 300 ft compared to the 200 ft considered for the original Feasibility Study.

- The Gemfield mineralization is not particularly hard but becomes quite abrasive in zones of higher silicification.

- The highest gold grades are found in the highly silicified zones ("ledges") which require finer crushing to ensure adequate gold particle liberation at higher grades. Conversely, at a constant crush size, recovery declines with increased silicification (i.e. increased gold grade).

- Gold recovery at grades typically anticipated from the Gemfield pit (nominal cut-off and average grades of 0.01 and 0.03 oz/T respectively) are relatively insensitive to crush sizes between 100% passing 0.5 inches and possibly as high as 100% passing 2 inches.

- 100% passing 1.0 inch leach feed represents a reasonable compromise upon which to base engineering design. At this crush size, ultimate gold recoveries in column testing at grades generally anticipated from the Gemfield pit ranged between the low 70%'s and the mid-90's.

Results from IMZ testing, along with data from the earlier KCA columns were submitted to Mohan Srivastava of Toronto, Canada, for statistical analysis. On the premise that gold recovery appears relatively insensitive to crush size (in the range of head grades anticipated

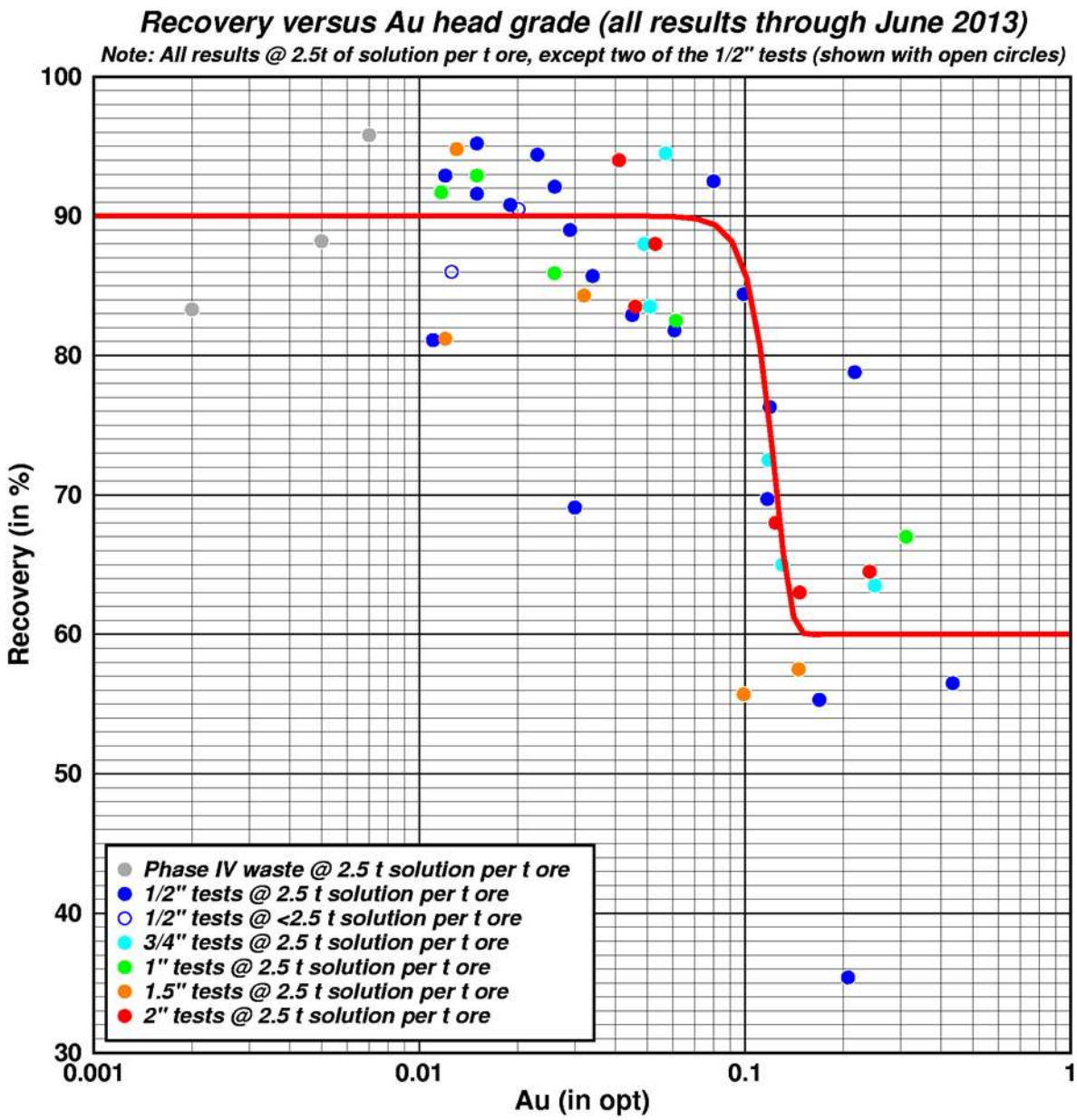
from the Gemfield pit) Srivastava evaluated data from all tests on minus one inch material or less.

Test recoveries were limited to a solution to ore ratio of 2.5 to 1, the same as used for primary leaching in engineering design. Analysis of the end-of-test results with longer irrigation cycles indicates incremental recovery might be achieved in production due to "trickle down" leaching by solutions from subsequent lifts placed on top of primary leached feed.

For the purposes of this study, projected gold recovery (in oxidized rhyolite) has been based on gold head grade. The graph shown in Figure 13.2 presents the model proposed by Srivastava for gold recovery versus head grade. This has been used for mine modeling in the Updated Study to develop pit production schedules and economic analysis of the project and results in a life-of-mine (LOM) average metallurgical gold recovery of 85.2%.

The process flowsheet selected for project development comprises three-stage crushing and screening, heap leaching, carbon adsorption, stripping and regeneration followed by electrowinning and smelting to produce doré bars (gold and silver with minor impurities) for shipment offsite to third party refiners. Crushing and pad loading will operate on day shift only 12 hours per day, seven days per week. Leaching and gold recovery will operate on two shifts, 24 hours a day, seven days a week).

Figure 13.2
Srivastava Model – Gold Recovery vs Grade



14.0 MINERAL RESOURCE ESTIMATES

14.1 OVERVIEW

14.1.1 Resource Estimation Procedures

The mineral resource estimates for all three deposits on the Property (Gemfield, Goldfield Main and McMahon Ridge) have all been estimated from 3D block models in which grades have been interpolated from drill-hole data using ordinary kriging within separate geological and statistical domains. Broadly speaking, there are two domains within each deposit: a strongly mineralized domain and a weakly mineralized domain. Each domain has its own estimation parameters, with the most important changes being in the variogram model parameters and the capping levels.

In each deposit, the estimation procedure customizes the local direction of maximum continuity so that it is aligned with the direction of maximum continuity identified from geological cross-sections. In the case of Gemfield and Goldfield Main, the direction of maximum continuity is sub-horizontal, and is captured by using a coordinate system that follows the undulations of a mineralized zone whose geometry is clearly identified on the geological cross sections. At Gemfield, this mineralized zone is a silicified band within the Sandstorm Rhyolite that generally dips to the west, but that locally undulates and that is offset by normal faults. In Goldfield Main, the mineralized zone is an east-dipping alteration zone (the MMH) whose dip flattens at depth.

At McMahon Ridge, the mineralization is interpreted as steeply-dipping but the orientation of the mineralized lenses does not lend itself to the same method as the one used for Gemfield and Goldfield Main. In particular, there are several sections on which there appears to be two sets of mineralized structure, one dipping to the north and the other dipping to the south, creating an “X” pattern. For this deposit, the local variations in the direction of maximum continuity were captured by creating a block model of the strike and dip of the mineralization, interpolating the orientations drawn on geological cross-sections.

In all three block models, gold grades were interpolated directly from drill-holes assay data; no compositing was performed. After ordinary kriging weights were calculated for the individual assays, these weights were multiplied by the assay length, and renormalized to sum to one. This length-weighting and renormalization procedure ensures that assay length is taken into account, and that short-length/high-grade sample intervals from core holes do not unduly influence the grade estimates. Generally speaking, the vast majority of the drill-holes sample intervals are the 5 ft intervals that come from the RC drill-holes. So the variation in sample length is minor, and the direct use of assays, rather than composites, permits greater fidelity to the original sample intervals; grades are not needlessly blurred across visible geological contacts.

In all three block models, assay grades are capped at levels where the upper tail of the distribution starts to break up, as determined from cumulative probability plots of gold on a logarithmic scale.

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The resource estimation for each of the three block models included an analysis of the possibility of down-hole contamination in RC samples. RC assays were not used for resource estimation wherever drill logs included commentary on down-hole contamination, and/or where geological interpretations raised serious doubts about the validity of RC assays, particularly in intervals with high grades in clayey material that lies immediately below silicified material. RC intervals that were deemed to be contaminated were removed from the database used for resource estimation, and these intervals were treated as having missing values (i.e. not as zeros).

All three block models use search ellipsoids that are aligned with the principal axes of the variogram model, with the three radiuses of the ellipsoid being set to the ranges of the variogram model. All three block models use octant searches, with the maximum number of samples per octant being set to four. The minimum number of samples required for estimation was one; meaning that every block that had even one sample within the range of the variogram received a grade estimate.

14.1.2 Classification

All of the mineral resources at Goldfield have been classified in accordance with the Definition Standards of the Canadian Institute of Mining and Metallurgy:

*A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough to confirm both geological and grade continuity.*

*An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

*An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling*

and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes.

Classification for each of the block models (Gemfield, Goldfield Main and McMahon Ridge) takes into account the number of drill-holes used in the estimation, and the number of octants that had data; blocks whose grade estimate is based on only one sample, or on assays from only one drill-hole, are always classified as "inferred". "Indicated" resources require samples from more than one drill-hole, and within 2/3 of the range of the variogram. "Measured" resources require samples from several drill-holes, and within 1/3 of the range of the variogram.

14.1.3 Location Information: Local Coordinates, Topography and Down-Hole Surveys

The local coordinate system used throughout the Property is an imperial coordinate system known as GOLD2008. It assigns 50000E, 50000N to the survey monument designated as Columbia-1950 and uses 5,066 ft (in NAVD88) as the elevation plane onto which horizontal coordinates are projected. Srivastava (2012) provides linear equations that can be used to convert between the GOLD2008 coordinates and UTM coordinates.

A high-resolution topography model, with 2 ft contour intervals, was developed in 2008 by Aerographics from aerial photography and 105 ground control points surveyed in the GOLD2008 local coordinate system.

For resource estimation purposes, the Aerographics topography was locally updated to incorporate drill-hole collars that were not available as control points in 2008. This was done by extracting the 2 ft contours, building a digital elevation model, calculating residuals (i.e. differences between drill-hole collars and topography), gridding the residuals using minimum curvature interpolation, and then adding the estimated residuals to the original topography. With this procedure, the locally adjusted topography within the resource study area exactly matches the drill-hole collars. The residuals calculated for this procedure were checked to ensure that there were no anomalous values that might reflect horizontal or vertical location errors. For the several hundred new collars in all three deposit areas, only one required a database correction; in all other cases, the surveyed collar elevation and the elevation from the Aerographics topography agreed to within ±5 ft.

During the development of cross-sectional interpretations, it became apparent that a few drill-holes had incorrect down-hole survey data in the electronic database. These were checked against original paper records, and corrected.

14.1.4 Location and Configuration of Block Models

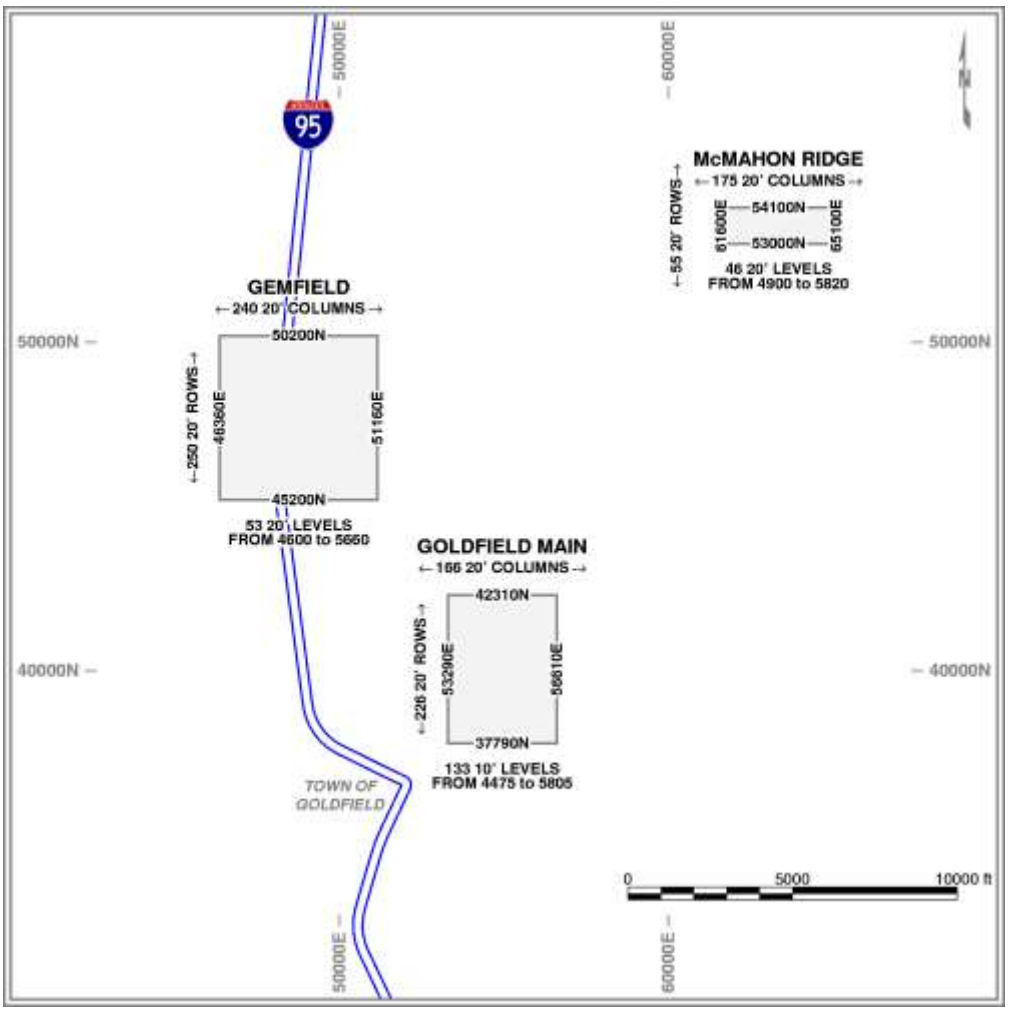
Table 14.1 and Figure 14.1 provide the location and configuration of the three block models that cover the three main mineralized areas on the Property. The columns and rows of each

block model are aligned with the north-south and east-west axes of the GOLD2008 coordinate system (which are, for all practical purposes, also the north-south and east-west axes of the UTM coordinate system); there is no rotation of the block model.

Table 14.1
Block Model Configuration Parameters

Parameter	Gemfield	Goldfield Main	McMahon Ridge
Western edge	46,360E	53,290E	61,600E
Eastern edge	51,160E	56,610E	65,100E
Southern edge	45,200N	37,790N	53,000N
Northern edge	50,200N	42,310N	54,100N
Bottom elevation	4,600 ft	4,475 ft	4,900 ft
Top elevation	5,660 ft	5,805 ft	5,820 ft
Block size	20 ft × 20 ft × 20 ft	20 ft × 20 ft × 10 ft	20 ft × 20 ft × 20 ft
Number of blocks	240 × 250 × 53	166 × 226 × 133	175 × 55 × 46

Figure 14.1
Location of the Resource Block Models Sub-Areas on the Property



14.1.5 Units

All of the block models use a combination of imperial and metric units. Imperial units are used for coordinates and for the block sizes (always a multiple of 5 ft in each direction) and are also used for tonnage factors. Grades are interpolated in metric units (grams per metric tonne) since these are the units used by the lab to report the grades. In the final block models that have been archived, all of the estimates are presented in imperial units; in the case of the estimated grades, these are reported both in imperial units (troy ounces per short ton) and in metric units (grams per metric tonne). The header record of each file identifies the units used for each field. “Tons” are imperial short tons; “tonnes” are metric tonnes; “ounces” are troy ounces.

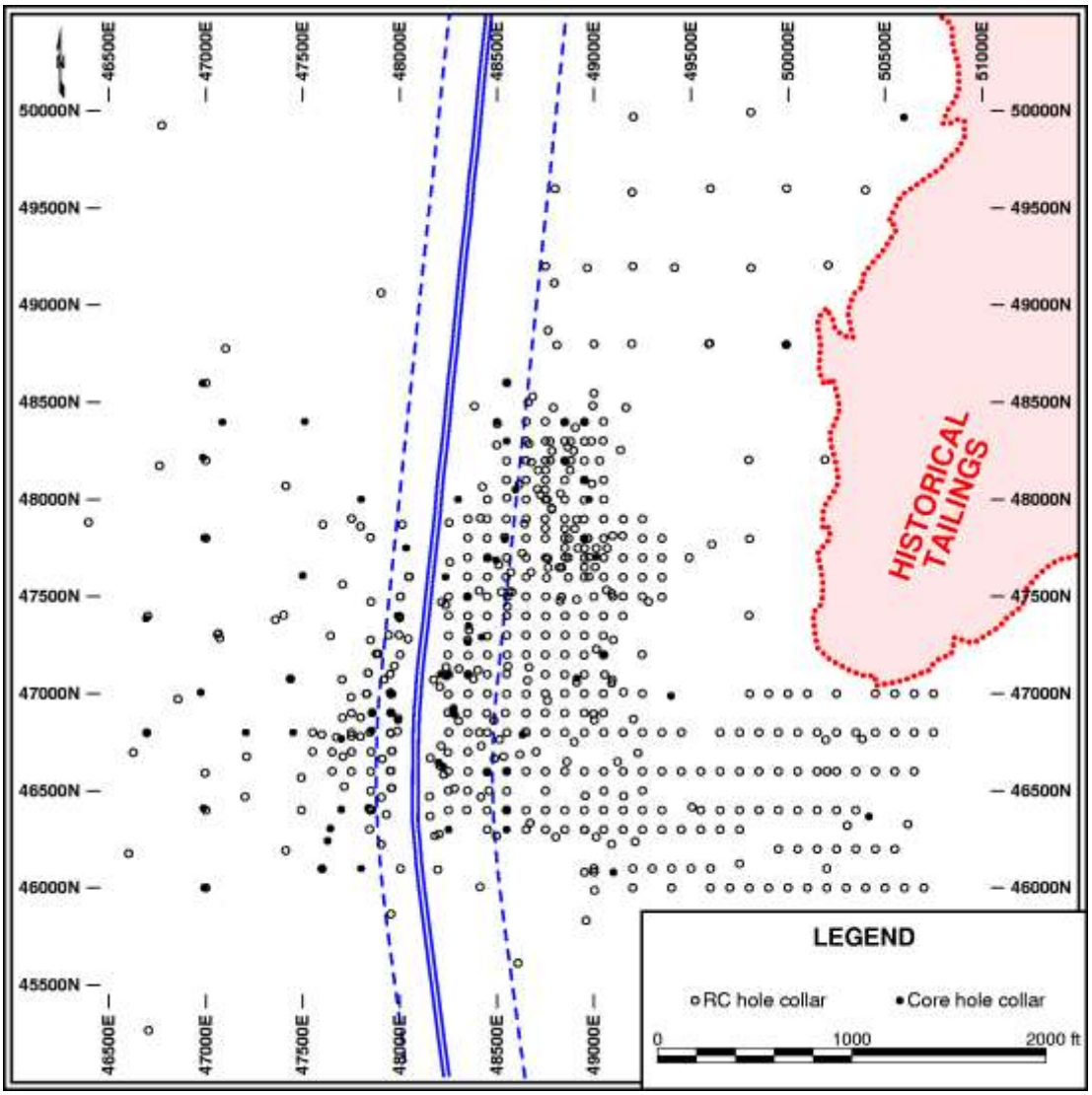
14.2 GEMFIELD

14.2.1 Gemfield Data

14.2.1.1 Drill-holes

The current resource estimate for the Gemfield deposit is based on all drill-holes whose assays were available by mid-January 2013: a total of 655 core and RC drill-holes totaling approximately 94,500 m. Figure 14.2 shows the collars of these drill holes. Most of the additional drilling since the previous resource estimate is in the region east of the Highway 95 and south-southeast of the historical tailings.

Figure 14.2
Collar Locations of Holes Used For January 2013 Resource Estimation of the Gemfield Deposit.



For the purpose of resource estimation, the only available Gemfield drill-holes that were not included are those drill-holes that were drilled within the footprint of the historical tailings. The reasons for excluding these drill-holes in their entirety are:

1) Although some of these drill-holes do penetrate through the base of the tailings, and provide assay in the underlying rock, the exact base of the tailings is hard to determine. It is hard to distinguish easily between Quaternary alluvium and tailings that have been compacted and weathered for nearly 100 years, and whose clays often have cemented the grains.

2) The historical tailings pile contains some erratic high grade material.

3) There may be down-hole contamination, with gold in the alluvium intervals being sourced from the tailings above.

4) Above the central part of the Gemfield deposit, the alluvium is essentially unmineralized.

For all of the blocks that lie beneath the footprint of the historical tailings, the Gemfield resource block model has grades of zero.

14.2.1.2 Assays

Of the 50,121 gold assays available in mid-January 2013, all were used for resource estimation, with the exception of those within the footprint of the historical tailings and those from RC intervals where there was evidence of down-hole contamination. At this cut-off date for the assay database, the last holes that provided assay information were GEM 599 in the RC series and GEMC-053 in the core series.

Except as noted below, for sample intervals where duplicates existed, the assay used for resource estimation purposes was the first assay. All of the duplicates were checked and in no case was there a significant difference between the first assay and any of the duplicates.

In three recent core holes (GEMC-028, GEMC-029 and GEMC-030) there are seven intervals for which the first assay, the AA-finish assay, was reported as “>10 g/t”. Because a gravimetric assay was available for each of these seven intervals, the gravimetric assay was used for resource estimation in these seven cases.

Assays below detection limit were set to half the detection limit. Unsampled intervals were treated as missing (and not as having a grade of zero).

14.2.1.3 Density

The dry bulk densities used for calculating tonnages in the resource block model are those used in previous Gemfield deposit resource block models, as reported by AMEC (2007) and are summarized in Table 14.2.

In April 2012, Tetra-Tech measured dry bulk densities of 27 samples obtained from core holes. These included 13 samples from the Sandstorm Rhyolite, the main host of the mineralization at the Gemfield deposit, and 14 samples from unmineralized formations whose densities had not previously been measured, but that would have to be stripped in an open pit. Since the ranges of densities measured by Tetra-Tech were consistent with the values used in previous studies, no changes were made to the dry bulk densities used previously.

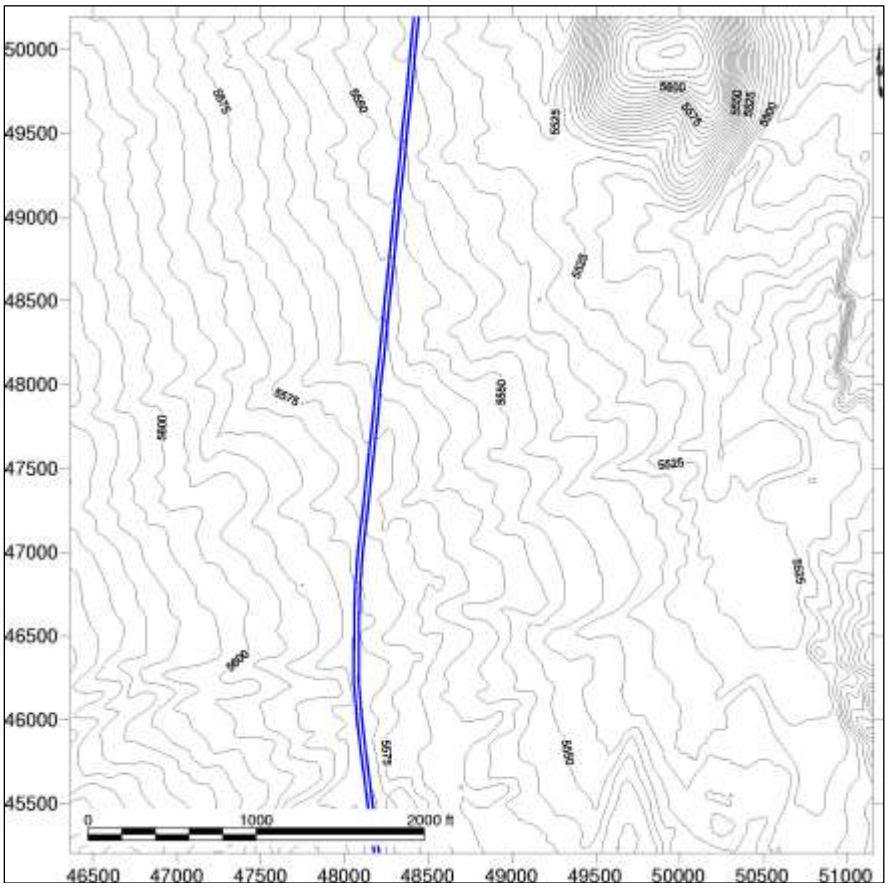
Table 14.2
Dry Bulk Densities Used For the Gemfield Resource Block Model

Formation	Comments	Dry Bulk Density (t/m^3)	Tonnage Factor (ft^3/T)
Alluvium (Qal)	Waste	1.89	16.95
Siebert (Ts)	Waste	2.05	15.67
Milltown Andesite (Tma)	Waste	2.05	15.67
Upper Vitrophyre (UTsv)	Sometimes ore	2.04	15.75
Silicified Sandstorm Rhyolite (Tsr)	Usually ore	2.37	13.50
Unsilicified Sandstorm Rhyolite (Tsr)	Sometimes ore	2.21	14.50
Lower Vitrophyre (LTsv)	Sometimes ore	2.04	15.75
Kendall Tuff (Tkt)	Waste	2.05	15.67
Quartz Monzonite (Jqm)	Waste	2.05	15.67

14.2.1.4 Topography

Figure 14.3 shows the topography above the Gemfield deposit, after the minor adjustment that integrates new drill-hole collars into the Aerographics topography.

Figure 14.3
Topography above the Gemfield Deposit



The region is essentially flat, with an alluvial plain that straddles the state highway that traverses the Gemfield deposit, draining to the east. Across the Gemfield deposit, the elevation changes from approximately 5,525 ft asl in the east to 5,600 ft asl in the west.

The peak in the northeast of the study area is a minor peak near Columbia Mountain whose summit serves as the 50000E, 50000N reference point for the GOLD2008 local coordinate system.

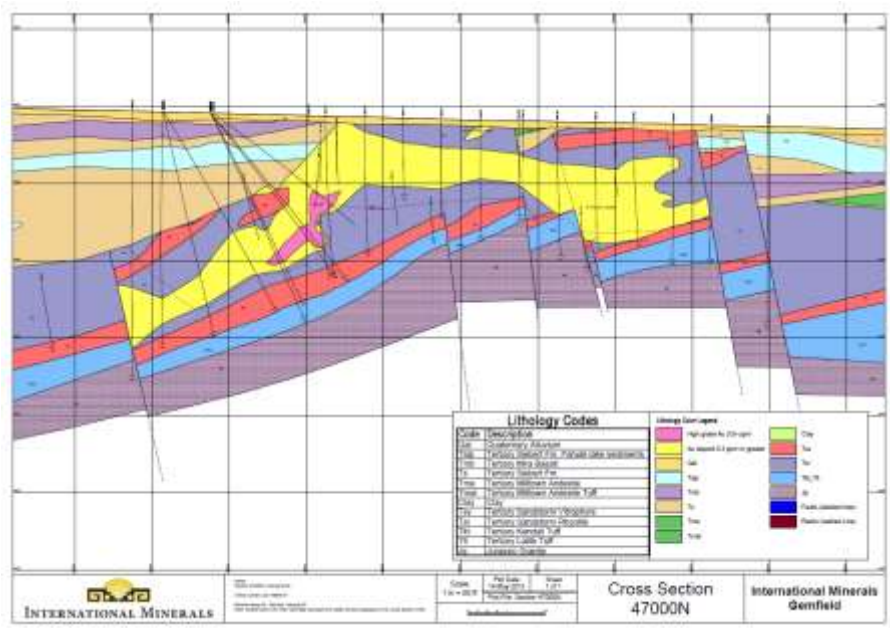
14.2.2 Gemfield Data Analysis

14.2.2.1 Geological Controls on Mineralization

Figure 14.4 shows a typical cross-section through the mineralization at Gemfield. Virtually all of the gold mineralization at grades reaching economically viable levels lies in the Sandstorm Rhyolite, specifically within an undulating band that contains the strongly silicified material referred to locally as "ledge". On Figure 14.4, the band that contains the strong gold mineralization, and the vast majority of the strongly silicified Sandstorm Rhyolite, is shown in yellow. In the discussion that follows, this band is referred to as the "mineralized ledge horizon" (MLH).

Although the MLH extends, in places, into the vitrophyres that lie immediately above and below the Sandstorm Rhyolite, bringing minor amounts of gold into the Upper and Lower Vitrophyres, the vast majority (well over 98%) of the gold lies in the Sandstorm Rhyolite.

Figure 14.4
Geological Interpretation Through the Gemfield Deposit on Cross-Section 47000N, Looking North

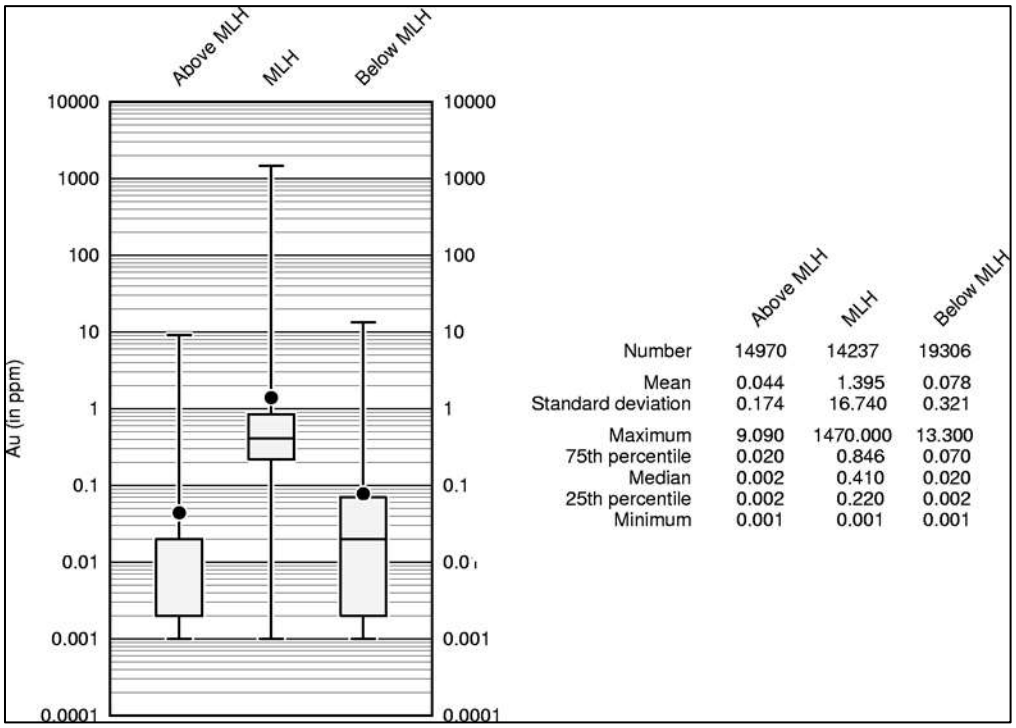


Although the MLH is, broadly speaking, relatively flat-lying at Gemfield, it has significant local undulations. In places, particularly at the edges of horst-and-graben fault blocks, the dip of the MLH can appear vertical. Elsewhere, particularly to the west, the MLH appears to follow the westerly dip of the Tertiary formations.

The location of the MLH within the Sandstorm Rhyolite also changes across the deposit. Sometimes it lies at the top of the formation, reaching into the Upper Vitrophyre; elsewhere, it lies at the bottom of the formation, reaching into the Lower Vitrophyre.

Figure 14.5 shows side-by-side boxplots of the distribution of gold assays above, within and below the MLH. The only significant mineralization lies within the MLH. Although the gold grades below the MLH are noticeably higher than they are above the MLH, they very rarely reach levels that are economically viable.

Figure 14.5
Side-by-Side Boxplots of Distribution of Gold Assays in Different Geological Domains at the Gemfield Deposit



	Above MLH	MLH	Below MLH
Number	14970	14237	19306
Mean	0.044	1.395	0.078
Standard deviation	0.174	16.740	0.321
Maximum	9.090	1470.000	13.300
75th percentile	0.020	0.846	0.070
Median	0.002	0.410	0.020
25th percentile	0.002	0.220	0.002
Minimum	0.001	0.001	0.001

In the data analysis that follows, and in the resource estimation, the Gemfield deposit was separated into two geological and statistical domains: the MLH and the non-MLH (both above and below the mineralized ledge horizon).

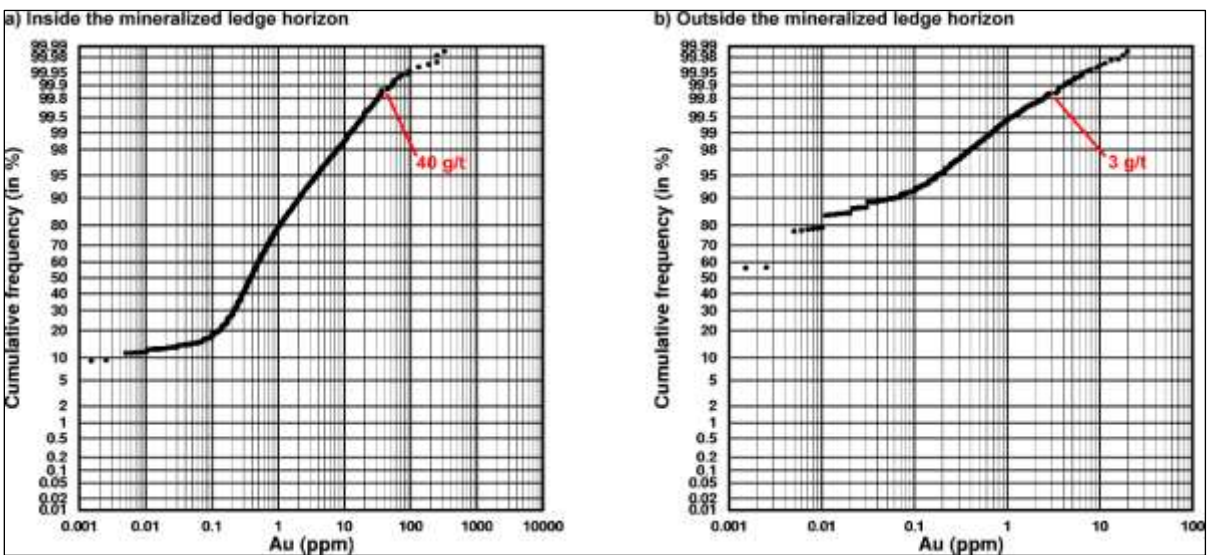
As seen in the example of Figure 14.4, the MLH can generally be treated as an undulating horizon that has an upper and lower contact. In places, the geological interpretations show more complexity: lenses of non-silicified rhyolite within the main band, as well as forks and

splays. For the purposes of resource estimation, the MLH was digitized as a band with an upper and lower contact. Where the geological interpretation of the MLH shows more complexity, folding back on itself, the contacts were digitized to include the non-silicified waste material. Where the geological interpretation shows a "stair step" along a normal fault, the contact was digitized along the fault trace. With this slight simplification of the geometry of the MLH, the mineralized domain includes some minor amounts of non-silicified ledge material that was not shown in yellow in the original geological cross-sections. The effect of this assumption is conservative in the resource estimates: it brings small amounts of low-grade waste material into the domain that carries the majority of the gold mineralization.

14.2.2.2 Grade Capping

Figure 14.6 shows cumulative probability plots (on a logarithmic scale) of the gold assays in the two domains. In both domains, the high-grade tail is continuous through the 99th percentile of the distribution but has a slight break near the 99.8th percentile. For the gold assays that lie within the MLH, this break occurs near 40 g/t; for those outside the MLH, this break occurs near 3 g/t. Within the MLH, all gold assays above 40 g/t were capped at 40 g/t; outside the MLH, all gold assays above 3 g/t were capped at 3 g/t.

Figure 14.6
Cumulative Probability Plots of Gold Assays in the Two Domains at the Gemfield Deposit, With Capping Levels.



14.2.2.3 Variograms

Figure 14.7 shows correlogram maps of gold grades within each domain. These figures show the horizontal variation of the gold grades in all horizontal directions. The center of each plot, with zero separation in the east-west and north-south directions, shows extremely good correlation at very short distances (the red indicates a correlation close to 1). As pairs of

samples are further apart, the correlation generally decreases, eventually reaching zero (the blue/black colours).

There is a clear direction of maximum continuity in both domains: the N25°E direction that is roughly parallel to the major normal faults in Gemfield. The magenta line shows the ellipse formed by the ranges of correlation in the horizontal direction. In the MLH, the range of correlation is 175 ft parallel to the major faults, and approximately 100 ft perpendicular to the major faults. Outside the MLH, the ranges are longer: 250 ft parallel to the major faults, and approximately 150 ft perpendicular to them.

Figure 14.7
Correlogram Maps of Gold in the Two Domains at the Gemfield Deposit

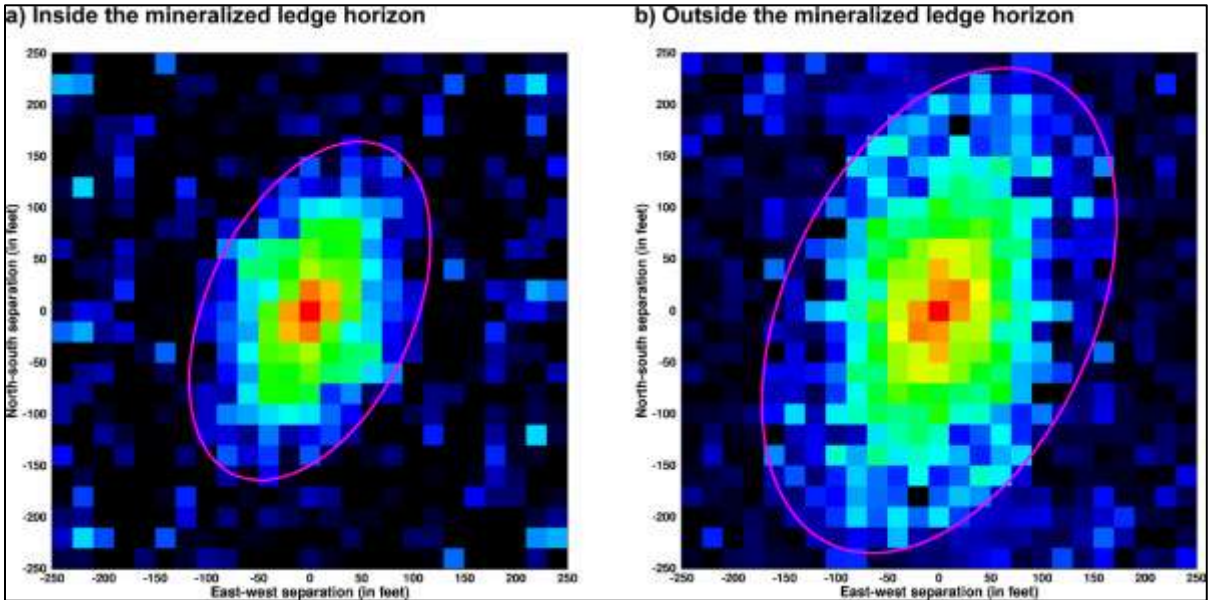
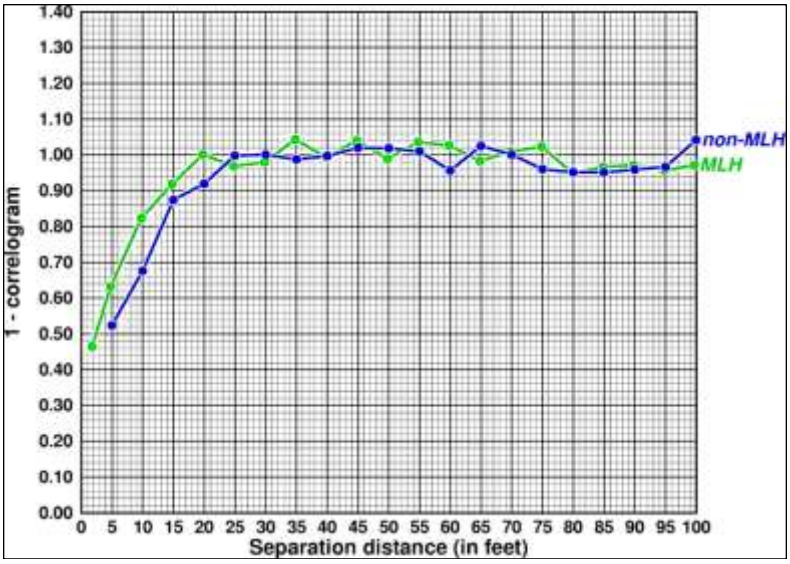


Figure 14.8 shows down-hole variograms of the gold grades in the two Gemfield domains. With most of the Gemfield holes being vertical, these down-hole variograms show the continuity of gold in the vertical direction. The relative nugget effect in each domain is about 30% of the sill in both domains. As with the horizontal ranges of correlation, the vertical range is slightly shorter inside the mineralized ledge horizon: approximately 20 ft inside the MLH and 25 ft in the much lower-grade material outside the MLH.

Figure 14.8
Down-Hole Variograms of Gold in the Two Domains at Gemfield.



14.2.3 Gemfield Resource Estimation

14.2.3.1 Resource Blocks and Sub-Blocks

With the level of selectivity inside the planned open pit being approximately a 20 ft cube, the block size for the resource block model was 20 ft × 20 ft × 20ft. In order to capture the details of the undulations in the mineralized ledge horizon and the rock formation contacts, sub-blocks were used. Each 20 ft × 20 ft × 20 ft resource block was split into sub-blocks if the entire block contained more than one rock formation. The 20ft dimension was sub-divided into two 10 ft blocks in each direction; if these 10 ft sub-blocks contained more than one rock formation they were further sub-divided into 5 ft sub-blocks. At the 5 ft scale, the block was assigned the rock formation code of the formation that constituted the largest tonnage within the sub-block. This sub-blocking affects the assignment of dry bulk density and tonnage. The tonnage and grade of the sub-blocks were estimated and these were then combined to get the total tonnage and average grade of each 20 ft × 20 ft × 20 ft block.

14.2.3.2 Estimation of Tonnage

The tonnage of each sub-block was calculated by multiplying the sub-block volume by the dry bulk density for the rock type assigned to the sub-block, using the dry bulk densities given in Table 14.2.

Rock type contacts were digitized from the geological cross-sections, and simplified in the same manner as the mineralized ledge horizon: all the contacts are represented as elevation grids, with minor fold-backs removed and with the contact running down faults where these serve as the formation boundary. These “layer-caked” elevation surfaces were clipped against

each other successively, from top to bottom, so that the top of an older, deeper formation never crosses above the bottom of a younger, shallower formation.

The top and bottom of the silicified Sandstorm Rhyolite were assumed to be the same as the top and bottom of the MLH (the yellow band on Figure 14.4). There are some minor amounts of silicified rhyolite that fall outside the MLH and, within the MLH, there are minor amounts of rhyolite that are not strongly silicified. Tonnage estimates could be improved if the geological logs permitted logging of the silicification intensity. Since the current and historical formats for the descriptive logs do not allow this to be properly coded, the most reliable data for the assessment of the density variations within the rhyolite are the geological interpretations of the mineralized ledge horizon. With the density constants for rhyolite (see Table 14.2) having been rounded to the nearest decimal place, the uncertainty on the exact location of the silicified / unsilicified contact within the rhyolite has very little impact on the resource estimates. Any error in the tonnage estimates is globally very minor, likely less than 2% for the blocks that contribute to measure and indicated resources within the pit.

At the ground surface, the topographic surface (Figure 14.3) was used to clip the block model. Sub-blocks with their centroids above the topography had their tonnages set to zero. With no historical mining activity at Gemfield, there was no need to account for historical production voids beneath the ground surface.

The tonnage of each 20 ft × 20 ft × 20 ft resource block is the sum of the tonnages of the individual sub-blocks within it.

14.2.3.3 Estimation of gold grade

Hard domain boundaries

The gold grade for each sub-block was estimated by interpolating the nearby drill-holes assays using ordinary kriging using hard domain boundaries. If the centroid of the sub-block falls within the MLH domain, the sub-block was treated as belonging to the MLH and was estimated from surrounding MLH assays. If the centroid of the sub-block falls within the non-MLH domain, the sub-block was treated as belonging to the non-MLH domain and was estimated from surrounding non-MLH assays.

No estimates below base of oxidation

Since metallurgical test work has consistently shown poor recoveries for non-oxide material at Gemfield, all sub-blocks with centroids below the base-of-oxidation (redox) surface were assigned gold grades of zero. The base-of-oxidation surface was digitized from the geological cross-sections and interpolated between sections to produce a continuous surface that completely spans the resource block model area.

Use of stratigraphic vertical coordinate

A stratigraphic coordinate was used instead of elevation. Within the MLH, this is the distance above/below the centreline of the MLH band. Above the MLH, the stratigraphic coordinate was the distance above the top of the MLH band; and below the MLH, it was the distance below the bottom of the MLH band.

Replacing the elevation with a stratigraphic coordinate ensures that the interpolation follows the undulations in the MLH band. Using Figure 14.4 as an example, the direction of maximum continuity has a dip of about 45° to the west on the western side of section 47000N but has a shallow dip to the east on the eastern side of the same section; in the middle, it steepens (near a fault) and then flattens to nearly horizontal. The use of a stratigraphic coordinate captures all of these variations in the direction of maximum continuity since the centreline of the MLH follows the undulations.

Variogram model

Table 14.3 gives the variogram model parameters used for kriging. The variogram models are all single-structure exponential models, with a relative nugget effect of 30% of the sill. The long and intermediate ranges are in the plane of the MLH; the shortest range is perpendicular to the MLH.

Down-hole variograms (Figure 14.8) were used to determine the nugget effect, the shape and the vertical range of the variogram model; correlogram maps (Figure 14.7) were used to determine the directions of maximum and intermediate continuity and the ranges in these directions. Because of the use of a stratigraphic vertical coordinate, the directions of maximum and intermediate continuity are not strictly horizontal but, instead, follow the undulations of the mineralized ledge band.

Table 14.3
Variogram Model Parameters Used For Grade Interpolation in the Gemfield Deposit Resource Block Model

Parameter	Inside MLH	Outside MLH
Nugget effect (C0)	0.3	0.3
Height of first structure (C1)	0.7	0.7
Total sill (C0+C1)	1.0	1.0
Shape	Exponential	Exponential
Maximum range (N25°E, parallel to MLH)	175 ft	250 ft
Intermediate range (N65°W, parallel to MLH)	100 ft	150 ft
Minimum range (perpendicular to MLH)	20 ft	25 ft

Search strategy

An octant search was used within an ellipsoid that was the same as the ellipsoid defined by the variogram ranges. Within each octant, the closest four assays were retained for

estimation. The maximum number of nearby samples that could be used for the estimation of any sub-block was 32; the minimum was 1.

Reblocking to 20 ft × 20 ft × 20 ft

The average gold grade for a 20 ft × 20 ft × 20 ft block is a tonnage-weighted average of the gold grades in all of the sub-blocks that fall within the parent block. Sub-blocks that have no grade estimate (usually because they fall below the base-of-oxide surface) were assigned a grade of zero before reblocking to the 20 ft × 20 ft × 20 ft scale used for reporting purposes.

14.2.3.4 Silver

The previous resource block model for Gemfield (used in the original Feasibility Study) contained silver grade estimates. Since many older drill-holes do not have silver assays, and previous economic analysis established that silver contributes less than 2% to gross revenue, the current block model in the Updated Study does not include estimates of silver grades.

14.2.4 Gemfield Resource Classification

Resource classification was done in a two-step procedure. In the first step, each sub-block was assigned a 1/2/3 code according to the following criteria:

- Sub-blocks with samples within ⅓ of the variogram range, with at least four octants containing data from at least four different drill-holes were assigned a value of 1.
- Sub-blocks with samples within ⅔ of the variogram range, with at least four octants containing data from at least two different drill-holes were assigned a value of 2.
- Sub-blocks with samples within the variogram range were assigned a value of 3.

The resulting sub-block codes were then spatially averaged to produce values between 1 and 3 for each 20 ft × 20 ft × 20 ft block. All blocks outside the MLH were classified as Inferred, regardless of their average code. Within the MLH, blocks with an average code less than 1.5 were classified as Measured; those with average codes between 1.5 and 2.5 were classified as Indicated; and those with average codes above 2.5 were classified as Inferred.

Since sub-blocks below the base-of-oxidation surface were left unestimated, the classified resources extend down to the base-of-oxidation surface, but not beyond it.

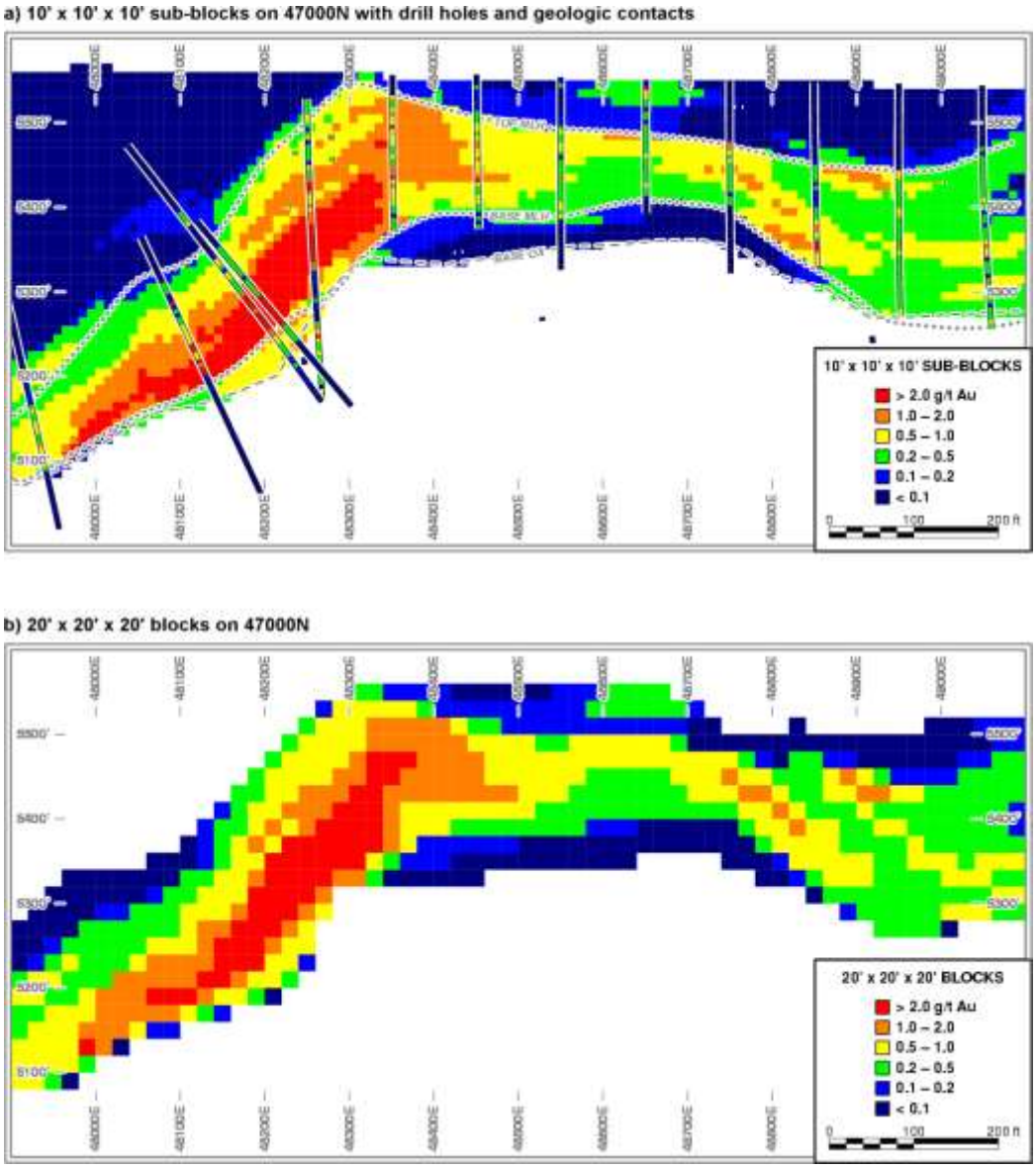
14.2.5 Checks of Gemfield Resource Block Model

14.2.5.1 Visual Checks

The resource block model was checked visually by overlaying drill-holes data on cross-sections through the block model, along with the geologic surfaces used to guide the grade

interpolation: the top and bottom of the MLH, and the base-of-oxidation. Figure 14.9 shows an example of this type of visual check.

Figure 14.9
Visual Checks on North Facing Cross Section 47000N of Gemfield Block Model.

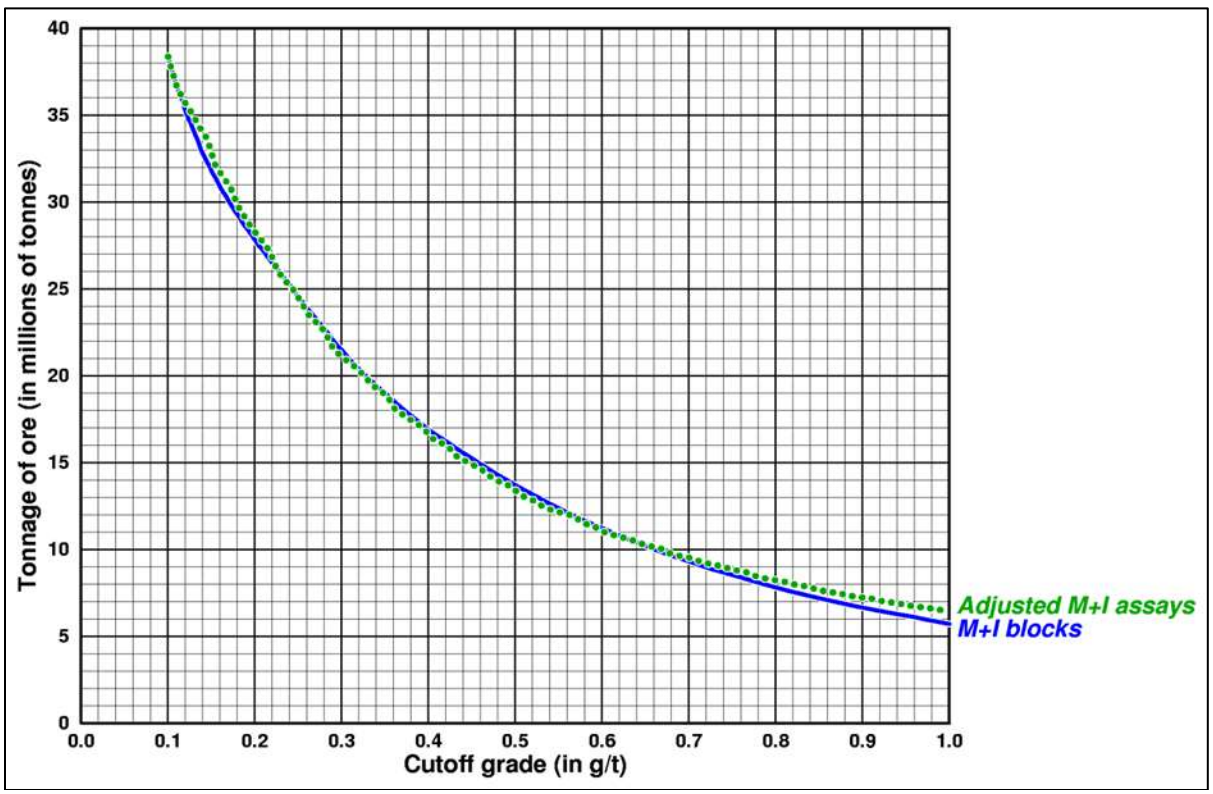


As shown in Figure 14.9, the resource block model was checked both at the scale of the 20 ft blocks, as well as at the scale of the sub-blocks, since the honouring of the drill-hole data and the geological contacts was easier to assess at the finer scale.

14.2.5.2 Statistical Checks

The distribution of estimated gold grades for the 20 ft × 20 ft × 20 ft blocks was compared to the distribution predicted from a change-of-support procedure (Parker, 1979) that reduces the variance of the assay distribution, using the variogram model to determine the variance reduction factor. Figure 14.10 shows, as a solid line, the grade-tonnage curve for the Measured and Indicated blocks in the model and, as a dashed line, the grade-tonnage curve predicted by the indirect lognormal change-of-support procedure applied to the assays that fall within the Measured and Indicated blocks. The variogram model predicts that the gold grades of 20 ft cubes should have a standard deviation 53% lower than that of the assays. The dotted line in Figure 14.10 comes directly from the length-weighted assay distribution, with each assay value moved toward the average by the amount needed to reduce the standard deviation by the 53% predicted by the variogram model. The close agreement between the two curves indicates that the degree of smoothing in the estimates is appropriate for the intended level of mining selectivity and that the block model does a good job of reflecting the grade variability that the mining operation will experience

Figure 14.10
Comparison of Grade-Tonnage Curves from the M+I Blocks in the Gemfield Deposit Block Model and from Change-of-Support Prediction Applied Directly to the Assays That Fall Within Those Blocks.



14.2.5.3 Software Checks

The resource block model was build using in-house Benchmark Six (B6) software developed by R. Mohan Srivastava, the QP for resources, for the specific purpose of enabling the interpolation to be done in a stratigraphic coordinate system that follows the undulations and faulting in the Sandstorm Rhyolite.

A similar procedure was implemented in Micromine, a commercial software package that includes resource estimation and that has the necessary tools for manipulating surfaces, calculating stratigraphic coordinates and using these to guide interpolation.

For the measured and indicated blocks within the main mineralized horizon, the Sandstorm Rhyolite, the B6 and Micromine tonnages and grades agreed to within ±5% at the reporting cut-off of 0.25 g/t gold.

14.2.6 Gemfield Mineral Resource Estimate

Table 14.4 summarizes the current estimate of the mineral resources at Gemfield, using 20 ft × 20 ft × 20 ft blocks and a cut-off of 0.25 g/t gold.

Table 14.4
Mineral Resource Estimate for the Gemfield Deposit, at a Cut-off Grade of 0.25 g/t Au (June 17, 2013)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	15.50	1.05	524,000
Indicated	9.10	0.54	157,000
Measured + Indicated	**24.59**	**0.86**	**681,000**
Inferred	1.08	0.52	18,000

Notes:
1. *Numbers are rounded to reflect the precision of a resource estimate.*
2. *The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.*

14.2.6.1 Sensitivity to reporting cut-off

The mineral resources for Gemfield have been reported at 0.25 g/t gold, a slightly lower cut-off than was used for the previous resource statement in July 2012. As discussed in the following section on reserves, the most recent economic analysis indicates that the economic break-even cut-off is slightly lower than 0.25 g/t. Table 14.5 shows the sensitivity of Gemfield's Measured+Indicated resources to small changes in the cut-off grade, below and above the 0.25 g/t cut-off used as a base case.

Changes of approximately ±15% in the cut-off grade cause changes of approximately ±13% in tonnage, ±9% in gold grade, and ±4% in metal content of the Gemfield resource.

Table 14.5
Effect of Changes in Cut-off Grade on Measured+Indicated Mineral Resources for the Gemfield Deposit.

Sensitivity	Cut-off Grade Au (g/t)	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Lower case	0.20	27.85	0.79	705,000
Base case	**0.25**	**24.59**	**0.86**	**681,000**
Upper case	0.30	21.48	0.95	654,000

14.3 GOLDFIELD MAIN

14.3.1 Goldfield Main Data

A more detailed description of the Goldfield Main resource estimate can be found in the NI 43-101 Technical Report "Feasibility Study on the Goldfield Property, Nevada, USA" by Micon with an effective date of 17 July, 2012 (Micon, 2012). All of the technical information and discussion of the resource estimation procedures, assumptions and parameters provided in Section 14.3 of the 2012 report remains current.

14.3.2 Goldfield Main Mineral Resource Estimate

Table 14.6 summarizes the current estimate of the mineral resources at Goldfield Main, at a cut-off of 0.4 g/t Au.

Table 14.6
Mineral Resource Estimate for the Goldfield Main Deposit
at a Cut-off Grade of 0.4 g/t Au (February 1, 2011)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	0	0	0
Indicated	8.55	1.53	421,000
Inferred	6.59	1.70	360,000

Notes:
1. *Numbers are rounded to reflect the precision of a resource estimate.*
2. *The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.*

14.4 MCMAHON RIDGE

A more detailed description of the McMahon Ridge resource estimate can be found in the NI 43-101 Technical Report "Feasibility Study on the Goldfield Property, Nevada, USA" by Micon with an effective date of 17 July, 2012 (Micon, 2012). All of the technical information and discussion of the resource estimation procedures, assumptions and parameters provided in Section 14.4 of the 2012 report remains current.

14.4.1 McMahon Ridge Mineral Resource Estimate

Table 14.7 summarizes the current estimate of the mineral resources at McMahon Ridge, at a cut-off grade of 0.3 g/t Au.

Table 14.7
Mineral Resource Estimate for McMahon Ridge, at a Cut-off Grade of 0.3 g/t Au (July 17, 2012)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	0	0	0
Indicated	5.51	1.34	238,000
Inferred	0.11	1.09	4,000

Notes:
1. *Numbers are rounded to reflect the precision of a resource estimate.*
2. *The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.*

14.5 TOTAL MINERAL RESOURCES FOR GOLDFIELD PROJECT

Table 14.8 combines the information from Tables 14.4, 14.6 and 14.7 to provide a project-wide summary of the current total mineral resources for the Property.

Table 14.8
Total Mineral Resources for the Goldfield Property (as of June 17, 2013)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	15.50	1.1	524,000
Indicated	23.16	1.1	816,000
Measured + Indicated	**38.66**	**1.1**	**1,340,000**
Inferred	7.78	1.5	382,000

Notes:
1. *Different reporting cut-off grades are used in each deposit: 0.25 g/t in Gemfield, 0.4 g/t in Goldfield Main and 0.30 g/t McMahon Ridge. The tonnage-weighted average cut-off grade for Measured and Indicated Resources is 0.29 g/t.*
2. *Numbers are rounded to reflect the precision of a resource estimate.*
3. *The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.*

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that would materially affect this mineral resource estimate. The only cautionary note, provided as a footnote to each resource table in this section, is that the contained metal estimates are subject to factors such as mining losses and dilution, as well as to process recovery losses. These are taken into account in the calculation of mineral reserves in the following section.

15.0 MINERAL RESERVE ESTIMATES

As of June 17, 2013, the mineral reserves for the Gemfield deposit are summarized in Table 15.1. There are no reserves estimated for the Goldfield Main and the McMahon Ridge deposits. These mineral reserve estimates are included within the mineral resource estimates discussed in Section 14 of this report. The parameters for optimization and subsequent mine design work supporting these reserves are described more fully in Section 16 below.

Table 15.1
Mineral Reserve Estimates for the Gemfield Deposit (June 17, 2013)

Reserve Category	Tons	Au (oz/ton)	Contained Ounces
Proven	15,158,000	0.0325	493,000
Probable	3,868,000	0.0191	74,000
Total Proven + Probable Reserves	**19,026,000**	**0.0298**	**567,000**

Notes:

1. *Numbers are rounded to reflect the precision of the estimate.*
2. *The Mineral reserves were estimated using a gold price of $1,450 per ounce with an average cut-off grade of 0.0066 oz/ton.*
3. *Micon is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant factors that will materially affect the validity of this reserve estimate.*
4. *CIM Standards on Mineral Resources and Reserves Definitions and Guidelines (1) define a 'Proven Mineral Reserve as "the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study", and define a 'Probable Mineral Reserve' as the "the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study."*
5. *These estimated ore reserves have an effective date of June 17, 2013.*

The mineral reserves presented in Table 15.1 were estimated by Sam Shoemaker, Jr., Registered Member-SME. Mr Shoemaker is a QP as defined in NI 43-101 and is independent of IMZ.

The Gemfield resource model was developed using a 20 ft x 20 ft x 20 ft parent block with 5 ft x 5 ft x 5 ft sub-blocks. This resource model was then exported into the Whittle open pit optimization software. This block size exported (20 ft x 20 ft x 20 ft) was selected as a suitable basis for pit optimization, mine planning, and reserve estimation because it represented a smallest mining unit (SMU) and best balanced the objectives to (a) account for mining dilution and losses while still maintaining reasonable fidelity to the geological interpretation and (b) reflect practicable mining selectivity based on the equipment envisioned for this project.

The mine plan developed in this report is based on Measured and Indicated Resources. The ultimate pit limit is based on the results of a Whittle Optimization completed by Micon. Sulfide material and all inferred resource blocks were not considered as potential revenue-generating blocks in the optimization.

A fixed economic cut-off was applied on a block-by-block basis to select blocks for processing using the parameters summarized in Section 16 below. Gold recovery was coded into the underlying resource model and was used to determine the recoverable gold ounces on a block by block basis. Blocks that were in the Measured or Indicated categories, with a gold grade greater than 0.0066 oz/T gold, were sent to the crusher. Other than a small amount of crusher feed developed during the pre-production mining period, no ore material is stockpiled and all ore material mined is assumed to be sent directly to the crusher.

16.0 MINING METHODS

The Updated Study assumes only production from the Gemfield deposit at this time.

Standard mining technology will be utilized to create an open pit with approximate dimensions of 3,000 feet north-south by 1,900 feet east-west and a maximum depth of 550 feet below current ground level. A second smaller open pit is also developed with approximate dimensions of 1,000 feet north-south by 1,100 feet east-west and a maximum depth of 220 feet below current ground level. Four pit shells selected from the Whittle open pit optimization discussed below were used to guide the design of five distinct phases as shown in Figures 16.4 to 16.8.

IMZ will have responsibility for site preparation, haul road construction and maintenance, production drilling and blasting, the excavation and haulage of ore and waste, management of ore stockpiles, oversize breakage, pit dewatering and haul road maintenance. IMZ will provide the open pit equipment, operator training, supervision, pit technical support services, mine consumables, and the pit operations and maintenance facilities. Mine maintenance personnel will be supplied by the manufacturer, per the terms and conditions of a standard maintenance and repair (MARC) agreement. Specialized contractors will provide explosives storage on-site. Explosives, blasting agents, fuel and other consumables will be provided by established suppliers.

16.1 MINE DESIGN

The ultimate pit limit for the Gemfield deposit was selected based on Whittle open pit optimizations. The pit will be developed using five distinct phases designed to approximate an optimal extraction sequence. The phase designs are based on slope design parameters and benching configurations provided by Golder Associates (Golder). A mine production schedule was prepared by Micon using Maptek's Chronos scheduling software.

16.1.1 Whittle Pit Optimization

Micon was retained by IMZ to complete new pit optimizations in order to determine the optimal economic pit configuration for the new Gemfield mineral resource model. To accomplish this task, Micon used the GEMCOM Whittle open pit optimization software. This software uses the industry standard Lerchs-Grossmann algorithm to determine an optimal pit shape using various economic, geotechnical, and metallurgical parameters. A number of scenarios were considered and completed in order to determine the final conceptual pit shell that was ultimately selected. This pit shell was then used for feasibility-level detailed phased pit designs and production scheduling.

Mine planning slopes for the Enterprise Optimization (EO) were based on recommendations provided by Golder Associates (Golder, 2011), as summarized in Table 16.1.

Table 16.1
Gemfield Pit Slope Design Recommendations

Geotechnical Unit	Max Overall (°)	Max Inter-Ramp (°)
Siebert Formation	40	45[1,2]
Milltown Andesite	45	45[1]
Upper Vitrophyre	45	45[1]
Sandstorm Rhyolite	45	45[1]
Lower Vitrophyre	30	30[1]

1. *Maximum recommended inter-ramp slope angles assume effectively depressurized slope. Reduce design slope by 5° if reasonable amount of residual pore pressure is anticipated under an active dewatering program.*
2. *Maximum slope height of 200-ft for recommended inter-ramp slope angles*

Using the detailed mine planning results from the original Feasibility Study, basic operating cost assumptions were developed for the Whittle open pit optimization. The previous study assumed a production rate of 6,600 tons per day as a basis. The Updated Study will be using a production rate of 8,250 tons per day and the costs were updated as required. These parameters are summarized in Table 16.2.

Table 16.2
Optimization Parameters

Item	Cost	Units
Ore	$1.91	$/Ton Ore
Waste	$1.62	$/Ton Waste Rock
Qal (Alluvium)	$1.14	$/Ton Qal
Incremental	$0.0272	$/Ton/Bench
Centre of Mass	5420	Elevation
Exit Bench	5520	Elevation
Process	$5.14	$/Ton Ore
G&A	$1.93	$/Ton Ore
NSR Royalty	5.0%	---
Price	$1,450.00	$/Troy Ounce
Selling	$10.00	$/Troy Ounce

Initial Whittle optimizations were completed using an early version of the Gemfield resource model (February 7, 2013). This early work allowed a better understanding of the changes to the resource model from the previous models and its impact on the optimal pit shape. This early work included 12 scenarios and examined various combinations of mining restrictions and minimum mining widths. Once the final Gemfield resource model was available (March 12, 2013), this earlier work allowed Micon to quickly determine the optimal pit shape for design and production scheduling using the new resource model.

During the initial Whittle optimization runs, it was determined that the new resource model would extend into the east drainage diversion ditch. A trade-off study, to determine if restricting the optimization to avoid this drainage ditch or to allow it, was completed. It was found that by restricting the pit optimization to avoid this drainage ditch the overall project

economics were improved and a restriction was placed in the resource model to not allow the pit optimization to mine in this area.

For the final Whittle open pit optimization runs, four major scenarios were examined:

- ***GEM 13*** – Base economics, Measured and Indicated resources only, east pit lobe restricted, minimum mining width of 100 feet, no mining allowed of the lower Vitrophyre unit, and no slope adjustment for ramps.

- ***GEM 14*** – Base economics, Measured and Indicated resources only, east pit lobe restricted, minimum mining width of 100 feet, no mining allowed of the lower Vitrophyre unit, and slope adjustments for ramps.

- ***GEM 15*** – Base economics, Measured and Indicated resources only, east pit lobe restricted, minimum mining width of 100 feet, no mining allowed of the lower Vitrophyre unit, slope adjustments for ramps and a 100 foot set back from the plan of operation limits.

- ***GEM 16*** – Base economics, Measured and Indicated resources only, east pit lobe restricted, re-blocked to a 100 feet by 100 feet by 20 feet block size to account for a minimum mining width of 100 feet, no mining allowed of the lower Vitrophyre unit, slope adjustments for ramps and a 100 foot set back from the plan of operation limits.

Scenario GEM 13 was examined primarily to evaluate the maximum pit extents without regard for ramps in the overall slopes. This scenario was generated as a check against the remaining three scenarios.

Scenario GEM 14 was found to intrude on the plan of operations limits (POO) and a limit establishing a 100 foot set-back from the POO was established in later scenarios. GEM 14 was then abandoned.

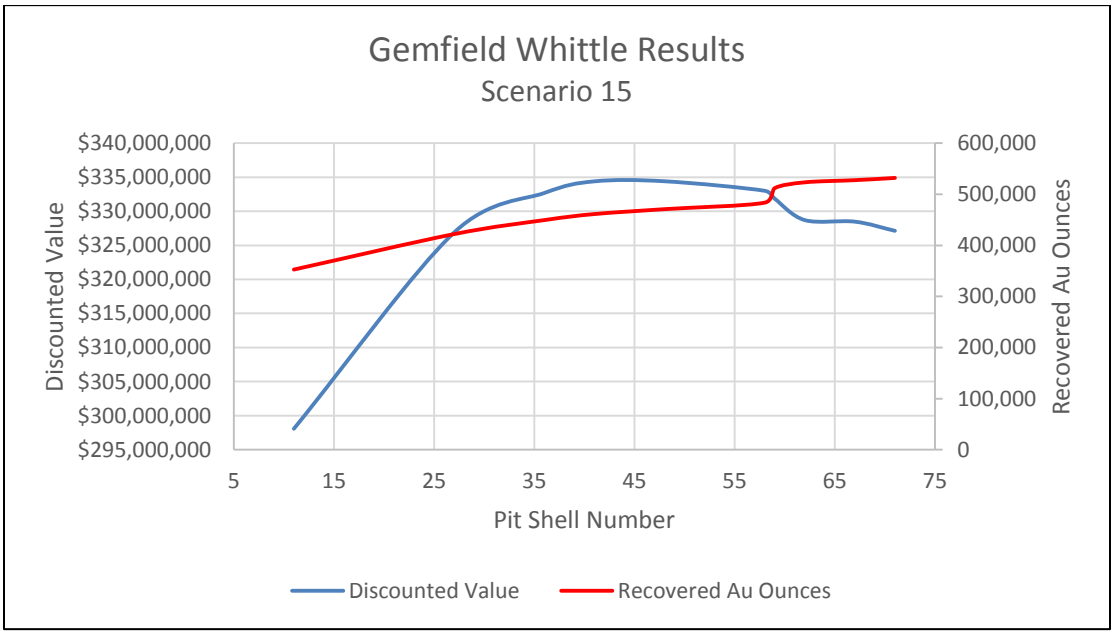
Scenario GEM 16 examined using a re-blocked model to account for a minimum mining width. This coupled with the other limitations in the optimization severely restricted the mining footprint and as a result GEM 16 was also abandoned.

Scenario GEM 15 was selected for detailed design and production scheduling. This scenario applied actual operational constraints in order to ensure that the final pit shell selected would remain within the operational plan of operations, not impact the east drainage ditch, and avoid both high sulfide materials and the lower Vitrophyre unit. Additionally, the overall slopes were reduced in order to account for ramps in the final design. An attempt was made through re-blocking to apply a minimum mining width (scenario GEM 16) which does not generate a workable pit shell. Minimum mining width was applied instead through the Whittle optimization by the use of a floating template that provided the required minimum mining width of 100 feet in each direction. With the exception of some minor issues with

drop cuts at the bottom of the pit, this approach worked well and resulted in a very workable pit shell for the final design.

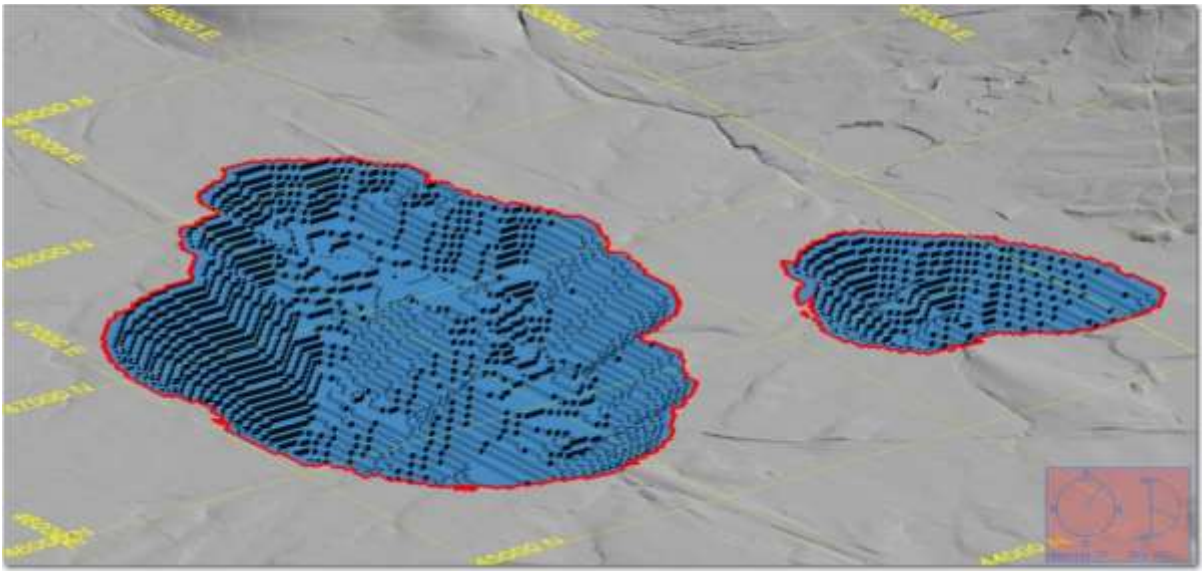
Based on this analysis, inflection points were selected where significant increases in tonnages were identified as the user specified case. In the case of scenario 15, pit shells were selected at pit shell numbers 6, 11, 36, 41, 48, 58, 62, 67, and 71 where pit shell 71 is the shell where the revenue factor is equal to one. This production scenario is shown as the specified case on the green line in Figure 3.1 above. Pit shell 71 is the maximum optimal pit at a full revenue factor of 1. A material balanced production schedule was then run for each of these pits reducing each schedule by the previous run pit shell number (the first schedule would use all 9 phases, the second would use 8 phases dropping off shell 71 and so on). This approach allows the economic impact of waste stripping (in this case, pit shells above pit number 58 have significant cost due to increased waste movement) to be examined at various material movement inflection points. Figure 16.1 shows these results graphically.

Figure 16.1
Scenario GEM 15, Graph of Results



Based on these results, pit shell 58 was selected for detailed design and production scheduling. This scenario was selected as a compromise between the best discounted value (without capital) and mine life. Selecting a pit near revenue factor 1 resulted in a significant increase in waste stripping with a minimal increase in recovered gold ounces and a decrease in discounted value. Pits below shell 41 also resulted in reduced discounted value and significantly reduced recovered gold ounces. Pit shell 48 achieved the highest discounted value with the selected pit 58 being about $1.4 million less. Based on this analysis, the best fit for mine life and discounted value was achieved at pit shell 58. Figure 16.2 shows the overall shape of the selected pit shell.

Figure 16.2
Gemfield Whittle Pit Shell 58



16.1.2 Design Criteria

Geotechnical Assumptions

Different mining phases were designed in accordance with the recommended bench configurations provided by Golder Associates as tabulated in Table 16.3. Double benching of 20 ft production benches were determined to be feasible in all geologic units except the Lower Vitrophyre. In the final optimization run, blocks within the Lower Vitrophyre were assigned higher mining costs to prevent pit walls from intersecting this unit. Ramps and pit walls were also carefully adjusted as needed to avoid exposure of the Lower Vitrophyre.

Table 16.3
Gemfield Bench Configuration Criteria

Geotechnical Unit	Inter-Ramp Slope (°)	Bench Height (ft)	Bench Face Angle (°)	Berm Width (ft)
Siebert Formation	44	40	70	25
Milltown Andesite	45	40	70	25
Upper Vitrophyre	45	40	70	25
Sandstorm Rhyolite - Footwall/End Walls	45	40	70	25
Sandstorm Rhyolite - Hanging Wall	45	40	60	17
Lower Vitrophyre	30	20	60	23

Haul Road Design Parameters

Two-way haul roads, 60-ft wide at a 10% grade, were designed in most cases where higher traffic may require extra width for safe and efficient passing of trucks. To maximize ore recovery at depth, the final benches of each pit floor were designed with single-lane access and steeper grades ranging from 12% to 15%. Safety berms were designed in accordance with the recommendations provided by Golder. Where haul roads intersect the highwall safety berm, the full safety berm was added to maintain efficient access onto berms as may be required to keep haul roads free of loose rock. Figure 16.3 shows the final pit and waste storage area configuration.

16.1.3 Phase Designs

Four pit shells selected from the Whittle optimization were used to guide the design of five distinct pit phases as shown in Figures 16.4, 16.5, 16.6, 16.7, and 16.8. Three phases were developed in the Gemfield pit while two phases are developed in the Gemfield Southeast pit. Primary access for each pit phase was designed from the north to minimize haul cycle times. The tonnage and grade tabulations by pit phase are summarized below in Table 16.4.

Table 16.4
Gemfield Reserve Summary by Pit Phase

Pit	Phase	Proven + Probable Ore Tons			Alluvium	Waste Rock	Total	Stripping
		Tons	Au oz/T	Recovered Au oz	Tons	Tons	Tons	Ratio
Main	1	2,630,000	0.0398	86,500	705,000	2,149,000	5,484,000	1.09
Main	2	6,146,000	0.0333	175,200	1,532,000	10,989,000	18,667,000	2.04
Main	3	8,464,000	0.0255	186,400	1,213,000	19,987,000	29,664,000	2.50
Main	**All**	**17,240,000**	**0.0305**	**448,100**	**3,450,000**	**33,125,000**	**53,815,000**	**2.12**
Southeast	1	352,000	0.0385	10,900	321,000	688,000	1,361,000	2.87
Southeast	2	1,434,000	0.0197	24,300	505,000	2,675,000	4,614,000	2.22
Southeast	**All**	**1,786,000**	**0.0234**	**35,200**	**826,000**	**3,363,000**	**5,975,000**	**2.35**
All	**All**	**19,026,000**	**0.0298**	**483,300**	**4,276,000**	**36,488,000**	**59,790,000**	**2.14**

Gemfield Pit Phased Designs

The main Gemfield pit is made up of three pit phases; Phase 1 or the ‘starter’ pit; Phase 2 an intermediate phase design to allow balanced ore production while the final phase is developed; and Phase 3, the final pit phase. The small Phase 1 starter pit was designed to maintain a minimum set-back of 250 feet from the existing highway to facilitate alluvium earthworks and blasting during the final months of the highway realignment. Waste rock from this pit phase will be used to construct the crusher road and the first lift of the rock stockpile. Once the Highway relocation is complete, development of the Phase 2 can begin in tandem with the Phase 1 development. Phase 3 completes the final development of the main Gemfield pit.

Southeast Pit Phased Designs

The Gemfield Southeast pit is made up of two phases, developing a minor extension of the deposit located to the southeast of the main pit. The first phase of this pit develops a high-grade core and the final phase extends to the final pit limits. Overall, this is a fairly low-grade pit and, with the exception of the Phase 1 pit, it is mostly developed towards the end of the Gemfield mine life.

16.1.4 Stockpile Designs

The rock and alluvium stockpile was designed with a 36° angle of repose, based on the assumption that the quantity of fines or plastic materials incorporated would be minimal. To facilitate reclamation to final out slopes of 2.5(H):1(V), the rock stockpile was designed in 40-foot lifts with setbacks of 45 feet. A swell factor of 25% was applied to the Siebert Formation and alluvium and a swell factor of 40% was applied to all other bedrock materials.

An estimated 4.3 million tons of alluvium will be removed from the pit area. Alluvium not used during construction and not stored in various small stockpiles will be stored in the rock stockpile. Alluvium removal required for the heap leach pad and other plant facilities will also be stockpiled in the alluvium/growth medium storage area for use in future reclamation and plant closure.

During the pre-production period, waste developed will be used as needed for road and ramp construction to provide access to the crusher and heap leach areas. Waste not used in construction will be sent to the waste stockpile area.

Figure 16.3
Gemfield Life-of-Mine Pit and Stockpile Designs

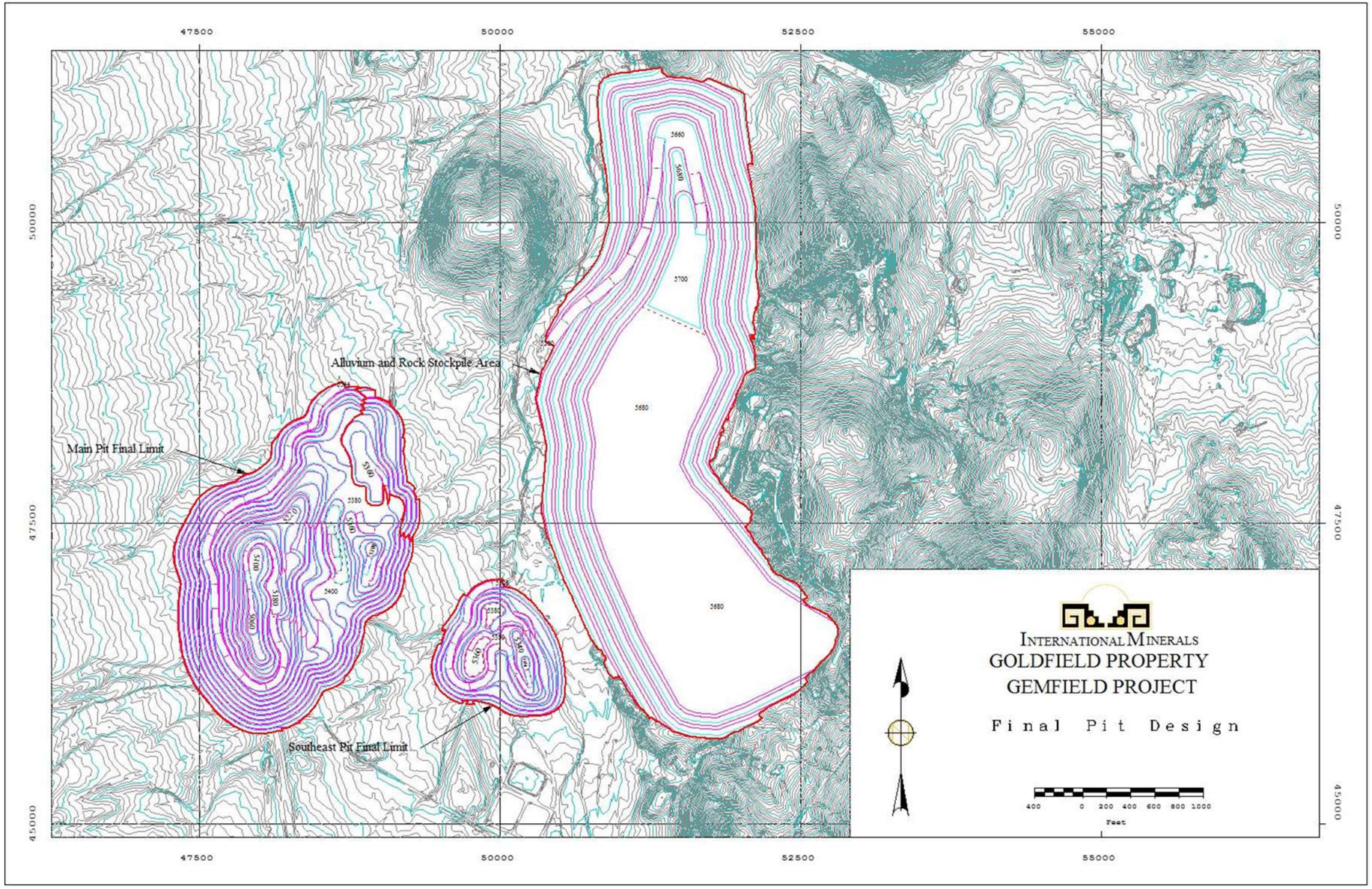


Figure 16.4
Gemfield Main Pit Phase 1 Design

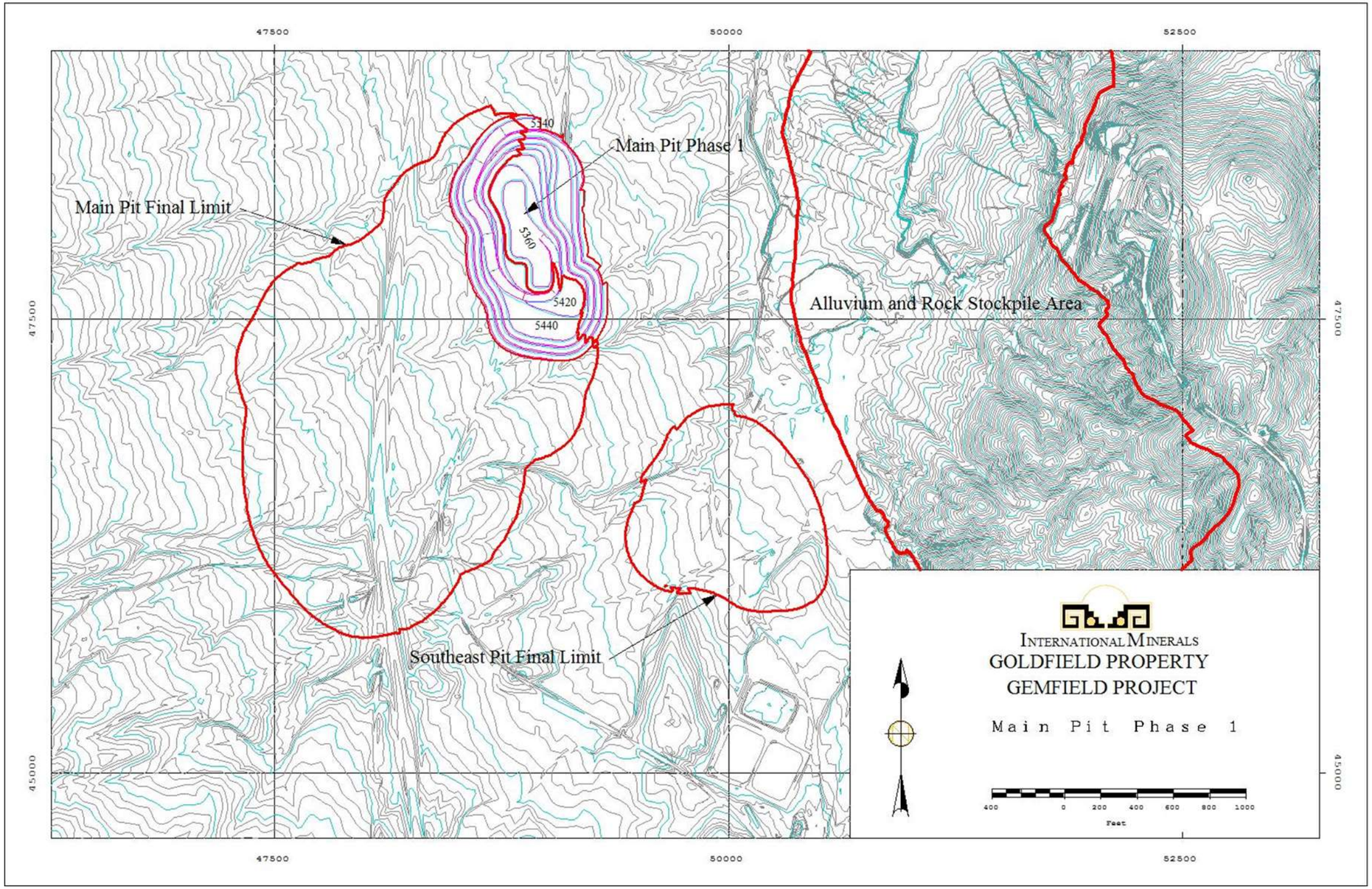


Figure 16.5
Gemfield Main Pit Phase 2 Design

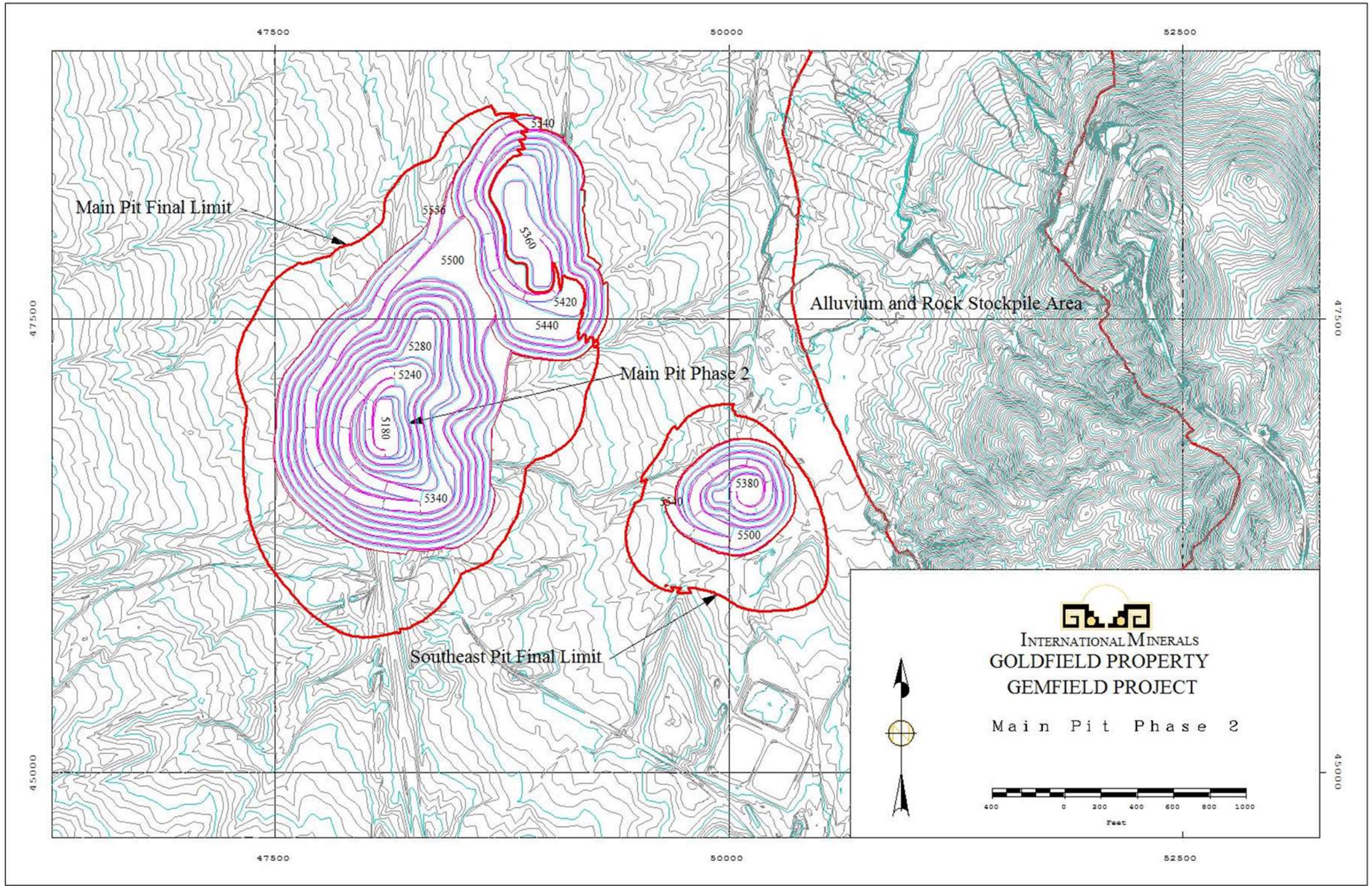


Figure 16.6
Gemfield Main Pit Phase 3 Design

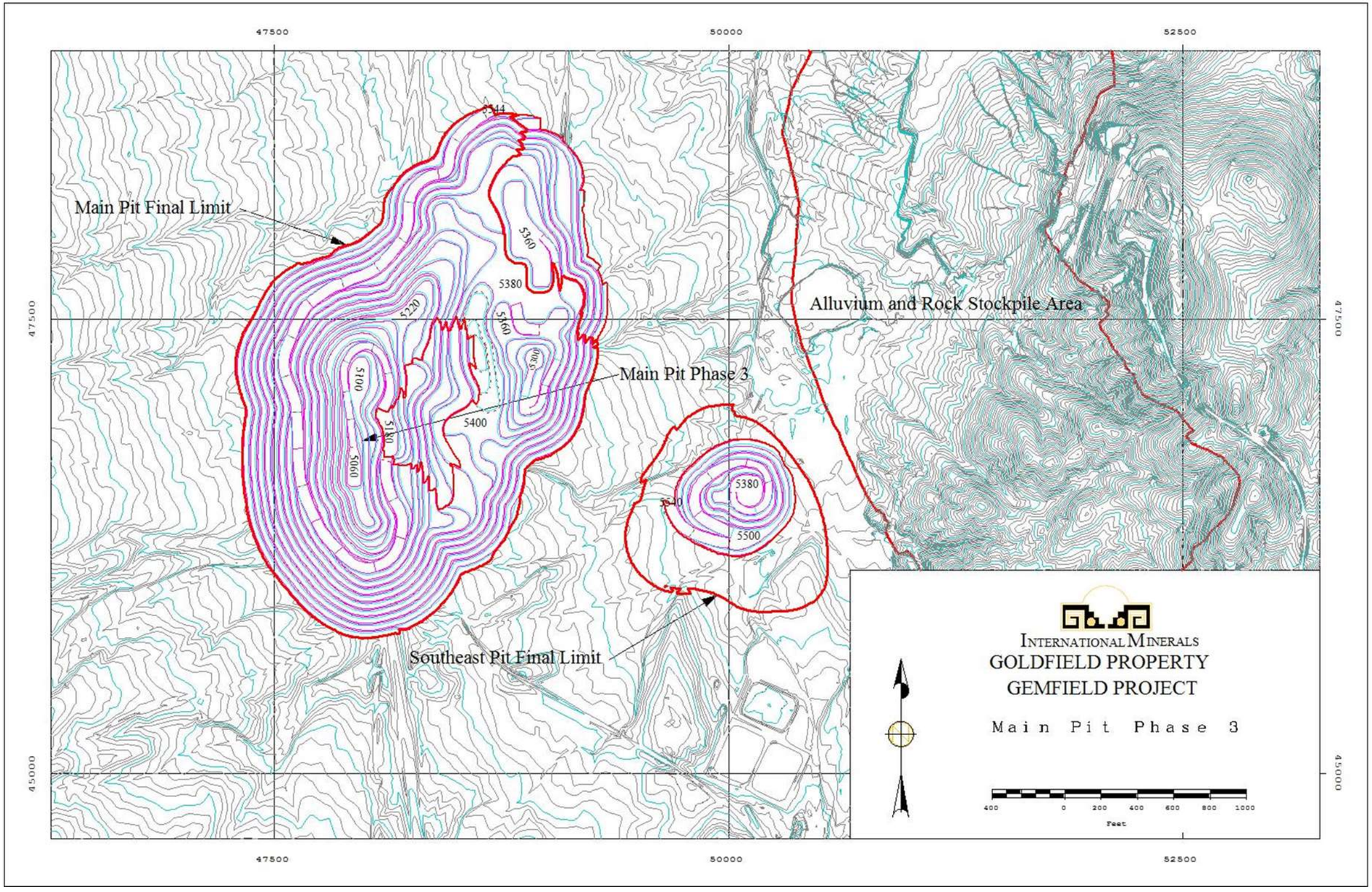


Figure 16.7
Gemfield Southeast Pit Phase 1 Design

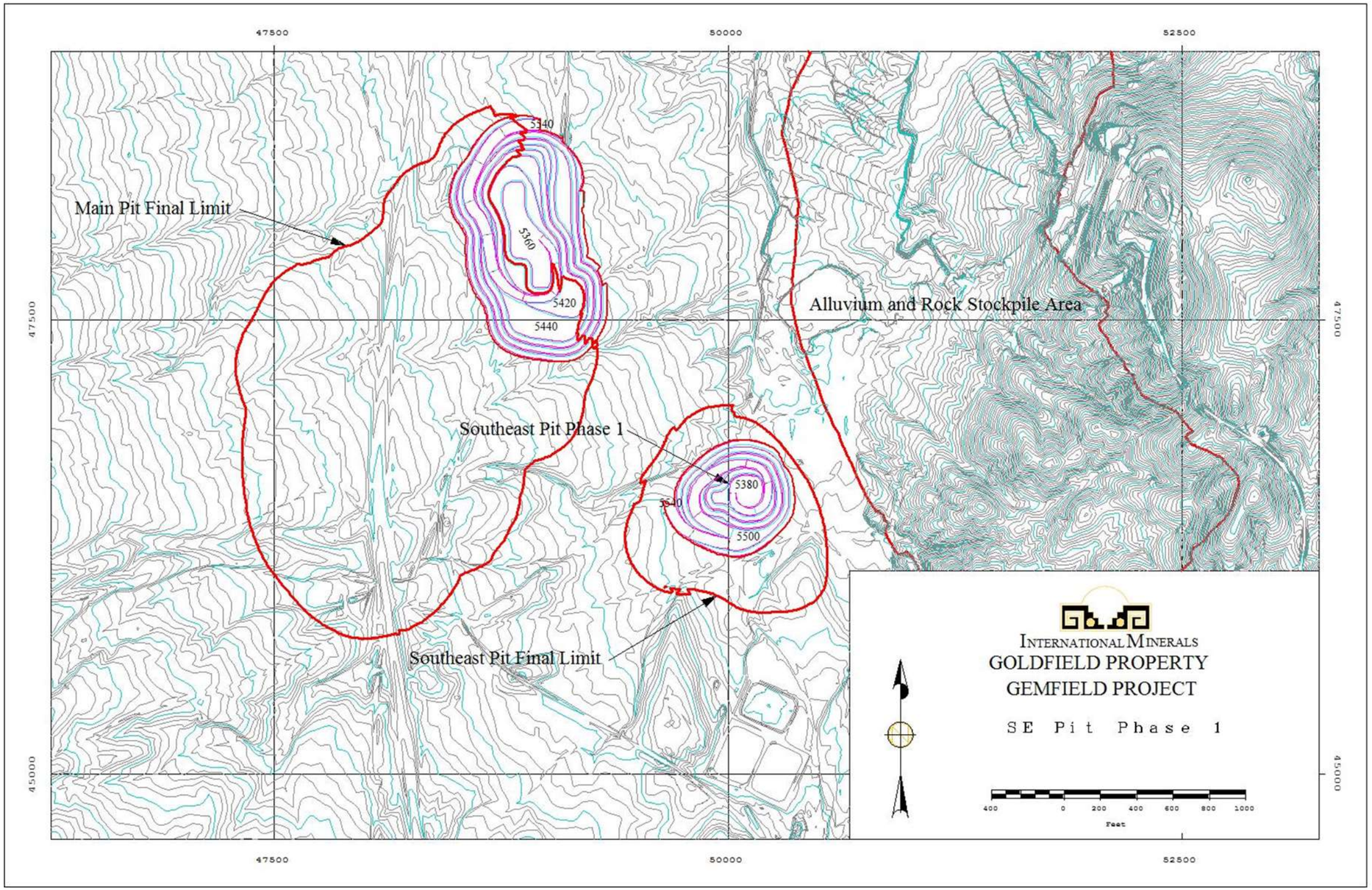
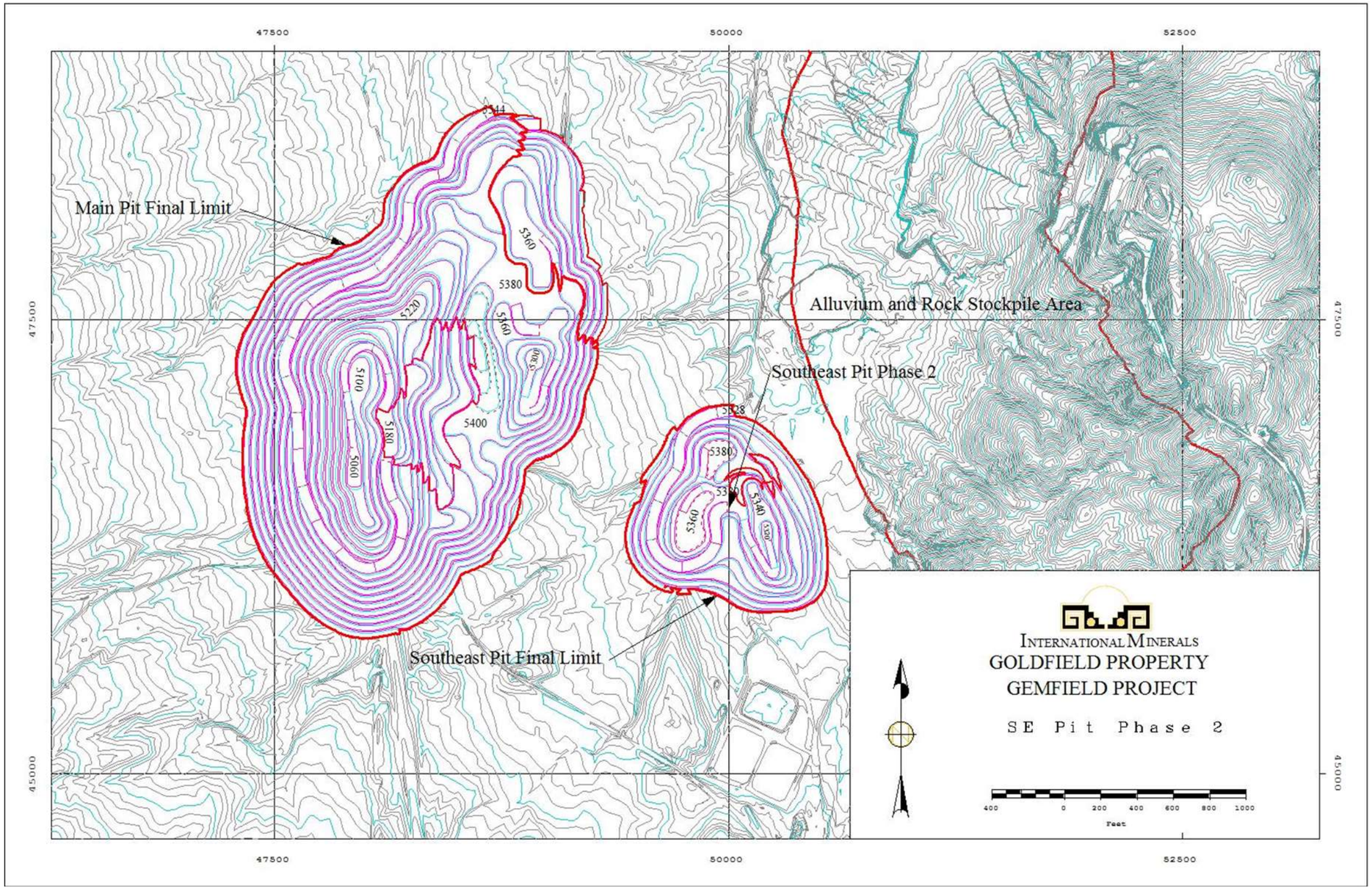


Figure 16.8
Gemfield Southeast Pit Phase 2 Design



16.2 MINE PRODUCTION SCHEDULE

A mine production schedule was prepared using Maptek's Chronos scheduling software. Ore selection was based on a fixed economic cut-off of 0.0066 ounces per ton gold and calculated using the parameters shown in Table 16.2. The schedule was prepared on the following basis:

- ***Period 1*** – This is a three-month period for pre-production waste stripping. For scheduling purposes this is assumed to be the third quarter of 2015.

- ***Periods 2, 3, and 4*** – These three periods are monthly during the initial start-up. Scheduling by month during this first quarter of production ensured that the crusher could be fed during this period. For scheduling purposes this is assumed to be the fourth quarter of 2015.

- ***Periods 5 through 12*** – These are quarterly periods through the end of the first two full years of production. For scheduling purposes this is assumed to be the quarters for the years 2016 and 2017.

- ***Periods 13 through 17*** – These are annual production schedules for the remainder of the mine life. For scheduling purposes this is assumed to be years 2018 through 2022. Please note that the mine operation during the last period (quarter 17 or year 2022) occurs only during the month of January when the current mineral reserve is fully exhausted.

Other than a small amount of leach material mined during the pre-production period (period 1), all ore material is assumed to be directly fed into the crusher. For those times when the crusher is down or not available, it is assumed that the mine will switch to waste production as required. The detailed production schedule is presented in Table 16.5.

Table 16.5
Mine Production Schedule for the Gemfield Deposit Mineral Reserve (June 17, 2013)

Description-	Yr-1 2015	Yr-2 2016	Yr-3 2017	Yr-4 2018	Yr-5 2019	Yr-6 2020	Yr-7 2021	Yr-8 2022	Total
Ore to Stockpile									
Tons	105,750	-	-	-	-	-	-	-	105,750
Au OPT	0.0522								0.0522
Recovered Au OPT	0.0384								0.0384
Stockpile to Crusher									
Tons	105,750	-	-	-	-	-	-	-	105,750
Au OPT	0.0522								0.0522
Recovered Au OPT	0.0384								0.0384
Ore to Crusher									
Tons	653,250	3,019,500	3,011,250	3,011,250	3,011,250	3,011,250	3,011,249	189,936	18,918,935
Au oz/t	0.0429	0.0301	0.0330	0.0371	0.0223	0.0197	0.0308	0.0640	0.0297
Recovered Au oz/t	0.0335	0.0260	0.0281	0.0315	0.0201	0.0177	0.0253	0.0489	0.0253

Description-	Yr-1 2015	Yr-2 2016	Yr-3 2017	Yr-4 2018	Yr-5 2019	Yr-6 2020	Yr-7 2021	Yr-8 2022	Total
Total Crusher Feed									
Tons	759,000	3,019,500	3,011,250	3,011,250	3,011,250	3,011,250	3,011,249	189,936	19,024,685
Au OPT	0.0442	0.0301	0.0330	0.0371	0.0223	0.0197	0.0308	0.0640	0.0298
Recovered Au oz/t	0.0342	0.0260	0.0281	0.0315	0.0201	0.0177	0.0253	0.0489	0.0254
Recovered Au Ounces	25,900	78,400	84,600	94,900	60,400	53,400	76,300	9,300	483,200
Waste Tons									
QAL Tons	1,799,159	1,373,025	597,099	2,576	0	0	505,402	0	4,277,262
Rock Tons	2,369,881	7,305,975	6,931,026	7,467,359	4,485,880	4,950,878	2,947,736	27,952	36,486,685
Ore to Stockpile	105,750	-	-	-	-	-	-	-	105,750
Total Waste Tons	4,274,790	8,679,000	7,528,125	7,469,935	4,485,880	4,950,878	3,453,138	27,952	40,869,697
Total Material Movement	4,928,040	11,698,500	10,539,375	10,481,185	7,497,130	7,962,128	6,464,387	217,888	59,788,633
Tons Per Day	27,378	32,051	28,875	28,716	20,540	21,814	17,711	7,029	24,902
Strip Ratio	6.49	2.87	2.50	2.48	1.49	1.64	1.15	0.15	2.15

16.2.1 Operating & Manpower Assumptions

The open pit will operate 24 hours a day and seven days a week on a 12-hour shift basis. Crushing and pad loading will operate on day shift only. Ore hauls will primarily run in conjunction with the crushing schedule on day shift. Waste stripping operations will run two shifts a day. Blasting operations will be devoted to day shift only. Productivity estimates were based on an assumed mechanical availability of 85% and utilization of available hours varied to reflect seasonal usage of equipment where appropriate.

A standard day-shift blasting crew will be required while three rotating labor crews will be scheduled to operate production equipment.

16.2.2 Preproduction Development

Approximately 2.5 miles of new highway will be constructed to the west of the existing US Highway 95. During the final stages of this Highway realignment, a small starter pit (Main Pit Phase 1) will be developed to provide waste rock material required to construct the main crusher haul road. Any ore encountered in the upper benches will be stockpiled temporarily until crushing commences.

16.2.3 Mine Waste Management

The rock and alluvium stockpile was designed with a 36° angle of repose, based on the assumption that the quantity of fines or plastic materials incorporated would be minimal. To facilitate reclamation to final out slopes of 2.5(H):1(V), the rock stockpile was designed in 40-foot lifts with setbacks of 45 feet. A swell factor of 25% was applied to the Siebert Formation and alluvium, and a swell factor of 40% was applied to all other bedrock materials.

An estimated 4.3 million tons of alluvium will be removed from the pit area. Alluvium not used during construction and not stored in various small stockpiles will be stored in the rock stockpile. Alluvium removal required for the heap leach pad and other plant facilities will

also be stockpiled in the alluvium/growth medium storage area for use in future reclamation and plant closure.

16.3 MINE EQUIPMENT SELECTION

The Gemfield deposit will be developed using standard open pit technology, scaled appropriately for the size of the operation. All mobile mining equipment is assumed to be owner-operated under a MARC contract. The required mining equipment is shown below in Table 16.6.

Table 16.6
Required Mining Equipment for the Gemfield Project

Description	Manufacturer & Model	Units Required	Capital Cost
Haul Truck	Cat 772	9	$7,088,760
Wheel Loader	Cat 990H	3	$3,091,680
Production Drill	Atlas Copco DM45HP	3	$4,295,580
Wheel Dozer	Cat 824H	1	$815,476
Track Dozer	Cat D8T	1	$815,670
Track Dozer	Cat D9T	1	$997,935
Motor Grader	Cat 14M	1	$452,300
Water Truck	Cat 772	1	$873,450
IT FEL/Tire Handler	Komatsu WA600 IT FEL	1	$986,389
Backhoe Loader w/Forks	Komatsu WB146	1	$147,825
Fuel/Lube Truck	Fuel/Lube Truck	1	$413,589
ANFO/Slurry Truck	ANFO/Slurry Truck	1	$118,650
Mechanics Truck	Mechanics Truck	1	$78,750
Welding Truck	Welding Truck	1	$78,750
Crane (25 Ton)	---	1	$136,000
Pick Up Trucks	---	9	$302,400
Tractor/Trailer	---	1	$327,600
Portable Light Towers	Amida AL4000	9	$121,500
Mine Planning/Survey Equipment	---		$250,000
Total Mine Capital			**$21,392,304**

16.3.1 Drilling and Blasting

Drilling demands will be met with three Atlas Copco DM45HP production drills. Blasting volumes are based on a 6.75 inch blasthole diameter and bench height of 20 ft. Drill spacing was adapted based on the assumed material properties for each geologic unit encountered. Conceptual drill pattern and powder factor assumptions are summarized in Table 16.7. Sub-drilling is estimated at 3 ft with stemming columns of approximately 11 ft. Productivity estimations were based on a mechanical availability of 85%, a utilization rate of 80%, an assumed re-drill rate of 5%, and an average set-up and sampling time of five minutes per hole. Revisions to these assumed parameters may be required as warranted by the availability of new geotechnical information and site conditions encountered in the field once the mine goes into operation.

Unit costs for explosives were based on a quote received from Southwest Energy LLC (Tucson, AZ), a joint venture with Orica. Chain of custody and delivery costs were based on monthly deliveries of bulk explosives, cartridges and detonators. Four 60 ton silos will be located on site providing adequate supply for 5 to 8 weeks, depending on production demands.

Grade control will be overseen by the mine geologist. Drilling operating costs already include the cost of sample collection as it is assumed cuttings from drill-holes will be collected by the drill operators, who will subsequently deliver samples to the on-site assay laboratory during shift change.

Table 16.7
Drill and Blast Pattern Assumptions

Rock Type	Tonnage Factor (ft^3/T)	Powder Factor† (lb/yd^3)	Powder Factor (lb/T)	Drill Pattern Spacing B x S	Penetration Rate (ft/hr)
Alluvium	16.950	0.00	0.00	n/a	n/a
Siebert Formation	15.672	0.40	0.23	23.0’ x 23.0’	132
Mira Basalt	15.672	1.30	0.75	12.5’ x 12.5’	88
Milltown Andesite	15.672	1.00	0.58	14.5’ x 14.5’	128
Upper Vitrophyre	15.750	1.00	0.58	14.5’ x 14.5’	128
Rhyolite - Weak Silicification	14.500	0.75	0.40	16.5’ x 16.5’	128
Rhyolite - Strong Silicification	13.500	1.20	0.60	13.0’ x 13.0’	84
Lower Vitrophyre	15.750	1.00	0.58	n/a	128
Kendall Tuff	15.672	1.00	0.58	n/a	122
Jurassic Quartz Monzonite	15.672	1.00	0.58	n/a	84

16.3.2 Loading and Hauling

A fleet of two to three Cat 990H wheel loaders and nine Cat 772 haul trucks will be required. Ore hauls will be reserved for day shift to coincide with the crushing operations. A small run-of-mine ore pad will be maintained with a loader in the event of daily logistical delays. No stockpiling is planned for the operation and all ore mined is assumed to feed directly into the crusher.

Trucking requirements were estimated using Maptek’s Vulcan Haulage Profile software program. Using performance data provided by the manufacturer, the Haulage Profile program automatically generates highly detailed cycle time estimates for a given haulage profile, taking into full account anticipated acceleration and deceleration, as well as any user-applied speed limits based on safety considerations. This program outputs the results for each block contained within the block model. Data calculated and stored include the total block productivity time (time required to excavate and move the block to its final destination) in minutes, total cycle time in minutes, and total one-way haul in feet. The block data is used later during schedule optimization as a constraint to level out haulage equipment requirements.

16.3.3 Support Equipment

An auxiliary fleet of dozers, graders, water trucks and other support equipment will be required. One Cat D9T track dozer will be staged in the active pit while the other Cat D8T track dozer will be staged at the waste rock stockpile. The Cat D9T track dozer will be devoted to alluvium removal and stockpiling in the early years of the project. The Cat 824H wheel dozer will remain available as a mobile unit capable of handling miscellaneous tasks as required.

16.3.4 Manpower

Total estimated manpower requirements for the mine are shown below in Table 16.8. Micon notes that grade control samples are assumed to be collected by the driller as part of the move and set-up period between drill-holes. A Chief Mine Engineer is not included in the mine G&A since this position is assumed to be filled out of the corporate office in Reno.

Table 16.8
Mine Manpower Requirements

Description	1 2015	2 2016	3 2017	4 2018	5 2019	6 2020	7 2021	8 2022	Total
Pit Supervision									
Mine Superintendent/Chief Mine Engineer	1	1	1	1	1	1	1	1	1
Pit Foreman	4	4	4	4	4	4	4	4	4
Clerk	1	1	1	1	1	1	1	1	1
Technical Services									
Mine Engineer	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1
Geologist	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1
Surveyor Technician	1	1	1	1	1	1	1	1	1
Technician	1	1	1	1	1	1	1	1	1
Drilling and Blasting									
Driller	8	12	12	8	8	8	8	4	9
Lead Blaster	1	1	1	1	1	1	1	1	1
Blaster's Helper	3	3	3	3	3	3	3	3	3
Load and Haul Operations									
Loader Operator	8	12	8	8	8	8	8	4	8
Haul Truck Operator	28	36	36	36	24	32	24	12	29
Wheel/Tracked Dozers	5	5	4	4	4	4	4	4	5
Other Support Equipment	8	8	8	8	8	8	8	8	8
Mine Labor	4	4	4	4	4	4	4		4
Total Labor Manpower Requirements	**77**	**93**	**88**	**84**	**72**	**80**	**72**	**52**	**79**

17.0 RECOVERY METHODS

The process flowsheet selected for the Project comprises three stage crushing and screening, heap leaching, carbon adsorption, stripping and regeneration followed by electrowinning and smelting to produce doré bars (gold and silver with minor impurities) for shipment offsite to third party refiners. Crushing and pad loading will operate on day shift only 12 hours per day, seven days per week. Leaching and gold recovery will operate on two shifts, 24 hours a day, seven days a week.

A conceptual site plan showing the site facilities is presented in Figure 17.1.

17.1 PROCESS DESCRIPTION

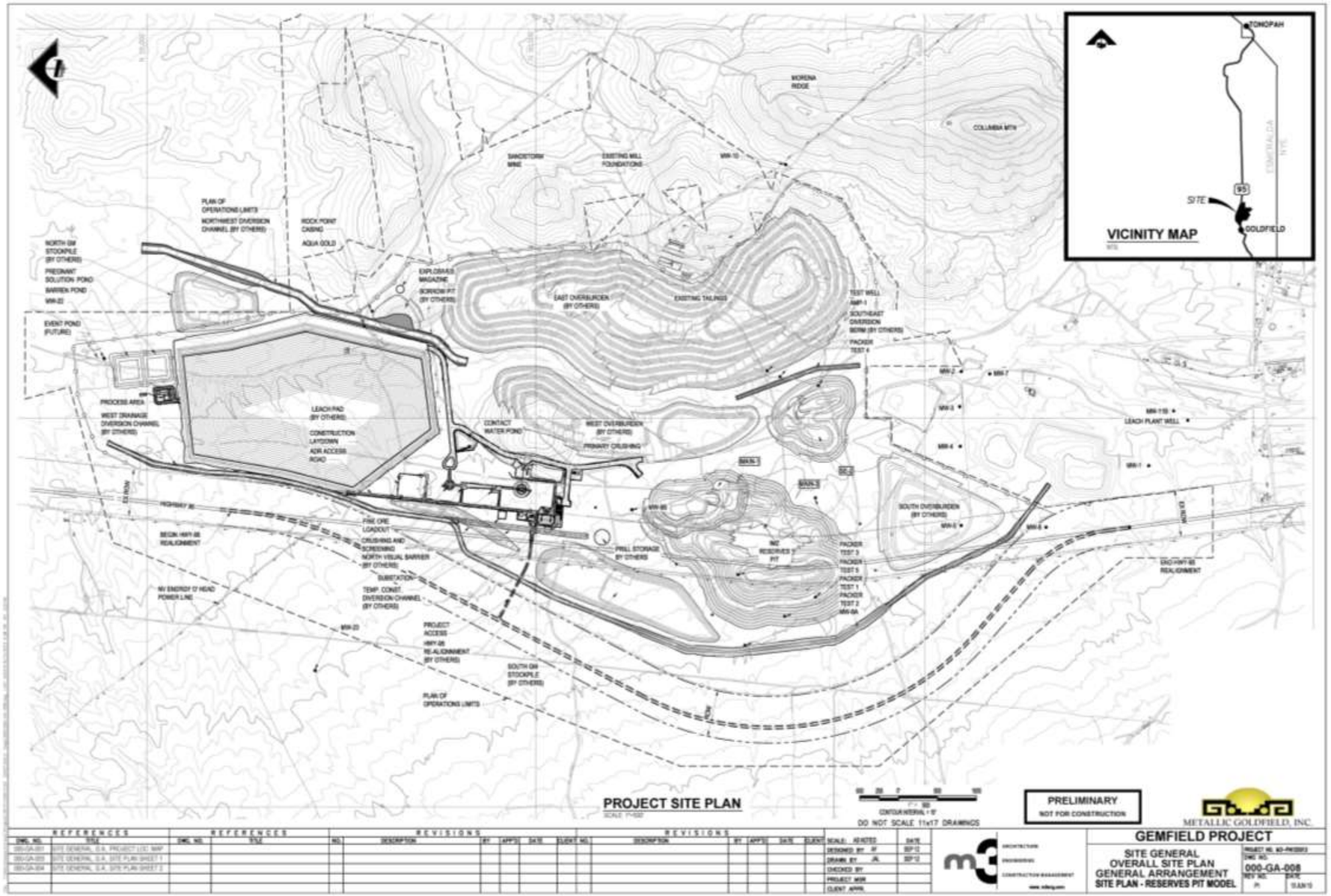
The three stage crushing and screening plant will produce leach feed, 100% passing one inch at the rate of 8,250 dry T/d. The crushing and screening facilities will operate on a schedule of one 12-hour shift per day, 7 days per week. Ore will be hauled from the pits and direct dumped into the primary crusher feed hopper.

Tertiary screen undersize (final product, 100% passing one inch) will be conveyed to an uncovered conical load-out pile adjacent to the leach pad. A remotely operated “clamshell” dump gate will reclaim crushed ore onto a high speed load-out conveyor designed to fill a 50-ton capacity dump truck in one minute. Pebble lime will be added at a predetermined rate to the load-out pile feed conveyor. Dump trucks will transport crushed ore to the leach pad.

Leach pads, ponds, pumps, solution distribution and collection piping and associated storm-water diversion systems were designed by SRK (Elko and Reno, Nevada). The leach pad will be constructed in multiple phases but will ultimately cover an area of approximately 4 million square feet. The pad will be designed to meet State of Nevada Division of the Environment requirements consisting of a base of low permeability soil beneath an 80-mil thick HDPE geomembrane covered with crushed over-liner, which will contain the solution collection piping resting on the geomembrane. Ore will be placed in 25 ft high lifts on the leach pad to an ultimate height of 200 ft. Heap faces will be contoured with catch benches to provide an overall operating slope of 2.5:1.

Due to the region’s low rainfall and high evaporation rate, control of the project water balance will be an important factor in leach operations. To minimize evaporative losses leach solutions will be applied to the heap with “drippers”, which will also be buried to minimize possible freezing problems in the winter. Testwork has indicated that the ore leaches rapidly and a single-stage leach system will be designed to deliver 2.5 tons of sodium cyanide (NaCN) solution per ton of ore (approximately 90 days at a solution application rate of 0.005 gpm/ft^2) for primary leaching.

Figure 17.1
Gemfield Project Conceptual Site Plan



Leach effluent solutions will be collected in a common launder system at the toe of the heap and flow by gravity to the pregnant solution pond. Pregnant solution will be pumped to the carbon adsorption plant. Cyanide will be added to barren solution discharged from the carbon columns in the barren solution tank before it from where it is returned to leaching. The barren solution tank overflows to the barren solution pond as does the pregnant solution pond in the event of significant rainstorms. In this way the barren pond functions as the event containment pond and will retain pad area runoff until it can be returned to circuit in place of raw water.

The pregnant and barren ponds will be double-lined and provided with intermediate leak detection monitoring systems. The ponds will be fenced and netting and/or bird balls will be provided for wildfowl protection.

The process plant and solution ponds will be located at the northern edge of the leach pad. Leaching and gold recovery circuits will be monitored and controlled from an operator's station located in the process building.

The high-security refinery area (electrowinning and smelting) will be located adjacent to and contiguous with the process building. Carbon adsorption will consist of a single train of five up-flow columns. Loaded carbon will be advanced to acid washing with dilute hydrochloric acid followed by rinsing with raw water and stripped of the contained metal (elution) in a pressure stripping system. Spent acid wash solution and acidified rinse water will be neutralized with caustic soda and discharged to the barren pond. Spent eluate will be periodically discharged to the adsorption circuit.

Gold and silver will be electrowon from loaded eluate and deposited on stainless steel wool cathodes, rinsed-off, decanted and collected as "sludge" which will be dried and smelted on-site to produce doré bars for shipment to third party refiners.

Stripped carbon will be thermally regenerated in a gas fired rotary kiln operating at 1,200 ºF and returned to carbon adsorption.

In accordance with environmental regulations regarding the possible presence of mercury at mining operations, the carbon stripping, regeneration, electrowinning cathode and sludge handling and smelting circuits will be provided with mercury vapor capture equipment

Bulk handling systems will be provided for the receipt, storage, mixing and distribution of sodium hydroxide, sodium cyanide, hydrochloric acid, anti-scalant, activated carbon and propane.

17.2 PROCESS DESIGN CRITERIA

Base case process design was developed from experience at similar projects coupled with metallurgical test results discussed in Section 13 of this Technical Report. Key criteria used to size major items of equipment for this study are included in Table 17.1.

Table 17.1
Summary of Key Process Design Criteria

Criterion	Units	Value
Crushing and Screening		
Crusher/screening plant operating schedule	h/day , day/week	12 , 7
Primary crusher availability	%	80
Secondary and tertiary availability	%	85
Final crushed product size	inch	0.5
Design gold feed grade	oz/T	0.032
Design silver feed grade	oz/T	0.146
Leaching		
Lift height	ft	25
Ultimate heap height	ft	200
Irrigation rate	gpm/ft^2	0.005
Primary leach cycle	days	90
NaCN consumption	lb/T of feed	1.05
Lime consumption	lb/T of feed	2.3
Gold Recovery Plant		
Adsorption column sizing	gpm/ft^2	25
Carbon loading (Au+Ag)	oz/T	100
Carbon stripping capacity	T/day	2.5
Metallurgical gold recovery	%	85
Metallurgical silver recovery	%	5
Nominal daily (annual) gold production	oz	210 (76,787)
Nominal daily (annual) silver production	oz	60 (21,982)

Note: As discussed in Section 13.0 analysis of test results received suggest that average gold recoveries will be slightly higher than originally projected. The slight increase in production can easily be handled by the current process design.

17.3 WATER BALANCE

The raw water requirement is projected at approximately 425 gpm. This water will be sourced from dewatering wells installed around the perimeter of the ultimate pit. A raw water distribution system will deliver raw water as required for drilling, dust control, process makeup, cooling, and reagents. An onsite purification plant will provide potable water.

18.0 PROPERTY INFRASTRUCTURE

Onsite ancillary facilities include various infrastructure buildings, power supply, fuel storage and distribution, water storage and distribution and sanitary sewage systems. Offsite ancillary facilities include realignment to the west of approximately 2-1/2 miles of US Highway 95. Figure 17.1 shows the location of the main ancillary infrastructure.

18.1 BUILDINGS

Plant site buildings will include the following modular and engineered structures:

- The administration building is a single-storey modular structure approximately 144 ft long by 60 ft wide. This structure will house all of the management, administrative and engineering staff.

- The security/safety building is a single-storey, modular structure approximately 38 ft long by 24 ft wide. It will be used to control access to the site and incorporate a first aid room and an ambulance garage.

- The laboratory consists of a single-storey, modular structure approximately 72 ft long by 60 ft wide. It will contain all the sample preparation and analytical facilities required to process both mine and process plant samples.

- The truck and plant maintenance shop/warehouse is an engineered steel structure approximately 80 ft long by 80 ft wide. It will incorporate one high-bay for truck and loader maintenance and two low-bay areas for small vehicle repair and general maintenance together with a warehouse area and office space for the maintenance staff. A tank farm containing all engine and hydraulic oils, antifreeze and waste fluids will be located outside but adjacent to the building.

18.2 POWER SUPPLY

Primary power source will be sourced from the existing 60kV NV Energy power line which traverses the project site. This power source will feed a new plant substation where the voltage will be reduced to 13.8 kV for in-plant distribution via overhead power lines to several unit substations where the voltage will be further reduced to serve the needs of the process, administrative and maintenance facilities.

18.3 FUEL STORAGE AND DISTRIBUTION

Storage tanks will be provided for gasoline and both off-road and on-road diesel fuel. Storage capacities are one 1,000 gallon gasoline tank, one 1,000 gallon on-road diesel tank and two 25,000 gallon off-road diesel tanks. These tanks will be installed above-ground inside a

concrete containment area to contain leaks and will incorporate both bulk and independent vehicle dispensing equipment.

18.4 WATER SYSTEMS

Although no significant dewatering activities are envisaged for the first 2 years of mining, approximately 425 gpm of raw water will be required for project operations. This water will be sourced from six wells installed primarily to the west of the ultimate pit limit.

A raw water storage tank (capacity 360,000 gallons) will be provided on site. Raw water distribution pumps will deliver raw water as required for dust control, process makeup, cooling, and reagents.

The bottom 20% of the raw water storage tank (72,000 gallons) will be reserved for fire protection, serviced by dedicated firewater protection pumps and hydrants.

It is expected that approximately 5 gpm of potable water will be required. This will be provided from an on-site purification plant.

18.5 SANITARY SEWAGE SYSTEMS

Sanitary sewage will be collected and treated in two sanitary waste systems each system consisting of a septic tank and sanitary leach field. One system will be located adjacent to the ADR plant area and the second will service the ancillary building area.

18.6 US HIGHWAY 95 REALIGNMENT

The existing US highway 95 currently traverses the planned Gemfield open pit and will therefore need to be realigned. Approximately 2-1/2 miles of new Highway will be constructed to the west of the existing road. It will consist of a two lane highway, providing Class II access road to the project. The realigned road section will be sited such that there will be a minimum distance of 1,000 feet from the future pit so as to avoid the necessity of road closures during blasting operations. In addition to relocating the road several utilities in the existing Right-Of-Way, including a 12.5 kV NV Energy power line, a section of the town of Gemfield well-water supply line, together with a booster pump station and two fiber optic cables owned by AT&T and Nevada Hospitals Association respectively, will also be reconstructed in the new Right-Of-Way. This realignment project is being coordinated by Atkins North America Inc. (Reno, Nevada) under contract to IMZ.

19.0 MARKET STUDIES AND CONTRACTS

There are no relevant marketing contracts pertaining to the Property.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section provides an overview of environmental and permitting work undertaken and requirements for the Property. The work undertaken included development of the Environmental Design Criteria, Environmental Baseline Studies, and initial permitting for the Project.

20.1 ENVIRONMENTAL DESIGN CRITERIA

The Environmental Design Criteria (EDC) was prepared as a technical memorandum by SRK, in consultation with IMZ. The EDC defines the environmental aspects relating to soil, water, air and noise that will be considered and implemented during design, development and closure of the proposed Project.

The relevant design criteria (by category) for environmental protection focus on engineering design as well as operational requirements. Also included in the EDC are design considerations (based on qualitative requirements) that IMZ considered using during the preparation of the Updated Study.

20.2 ENVIRONMENTAL BASELINE CONDITIONS

An initial environmental baseline study program was undertaken by MVG in 2003/2004 on approximately 696 acres as part of an Environmental Assessment (EA) for what was then named the Gemfield Exploration Project. Upon consultation with the BLM, an additional baseline study program was implemented to collect environmental, natural resource and socio-economic data required to support the completion of the federal and state agency permitting and approval program, and the environmental documentation process that will be required under NEPA for the proposed mine at Gemfield.

The topography of the area comprises broad valleys and alluvial fans, low-lying mesas and foothills. The foothills and mesas are rocky with drainages and topographic low areas in between. The area lies within the southernmost extreme of the Tonopah Section of the Great Basin and the northern border of the Mohave Desert.

The climate is arid with average annual precipitation in the area of approximately 6.4 inches, most of which falls in the form of winter snow and/or spring rain. Summers are typically dry and hot, with occasional summer thunderstorms and average daytime temperatures in the 90 to 100ºF range. Elevations in the Project area range from approximately 5,400 to 6,000 ft. The Property has been mined both historically and recently, leaving much of the area disturbed, except for the area directly above the Gemfield deposit, which is beneath a cover of alluvium.

Elements to be included in the Environmental Baseline Study comprise the following:

- Geology.
- Soils.
- Climate/Air.
- Cultural Resources.
- Paleontology.
- Vegetation.
- Range.
- Noxious Weeds.
- Invasive, non-native Species.
- Wildlife.
- Threatened, Endangered, and Special Status Species.
- Native American Religious Concerns.
- Land Use.
- Visual Resources.
- Noise.
- Recreation and Wilderness.
- Water Resources.
- Hydrogeology.
- Hydrology.
- Geochemistry (potential Acid Rock Drainage and Metal Leaching (ARDML)).
- Socioeconomics.
- Environmental Justice.

The following critical elements of environment are likely neither present nor affected by the proposed project:

- Areas of Critical Environmental Concern.
- Farm Lands (prime or unique).
- Floodplains.
- Wetlands/Riparian Zones.
- Wild and Scenic Rivers.
- Wilderness.
- Forestry.

As of June 2013 the following baseline studies have been completed:

- A Class III Cultural Resource Inventory for Three Parcels of Land Near Goldfield, Esmeralda County, Nevada. BLM Report No. CRR-6-2260. WCRM 2004.
- The Last Gold Rush: Class III Inventory of the Goldfield Mining District, Esmeralda County, Nevada. BLM report No. BLM6-2687-1(P).
- Goldfield Project: Baseline Socioeconomic and Environmental Justice Conditions. Blankenship Consulting, LLC. 2012.
- Rock Characterization and Handling Plan for the Gemfield Project, SRK, 2013.

- Numerical Groundwater-Flow Model for the Gemfield Project, SRK 2013.
- Paleontological Baseline Technical Report for the Goldfield Mining Project In Esmeralda County, Nevada. PaleoResource Consultants, 2012.
- Goldfield Project Wildlife Surveys. Wildlife Resource Consultants, 2012.
- Goldfield Project Botanical Resource and Range Condition Specialist Field Report. International Minerals and Goldfields (U.S.) Inc. Joan Reynolds Botanical Consultants, 2012.
- Goldfield Project - 189 Acre TES Plant Survey Botanical Field Report, Joan Reynolds Botanical Consultants, 2013.
- Small Mammal Trapping Survey, Wildlife Resource Consultants, 2013.
- Goldfield Project, Golden Eagle Nesting Surveys, Wildlife Resource Consultants, 2013.
- Biological Surveys and Management Recommendations for Abandoned Mined on Lands owned by Metallic Goldfield, Inc., Nevada.
- Waters of The United States Jurisdictional Determination, Gemfield Project – Metallic Goldfield, Inc. Esmeralda County, Nevada, JBR Environmental Consultants. 2013.
- Gemfield Project Blasting Vibration Study. Tierra Group International, 2013.
- Monitoring Protocol and Quality Assurance Project Plan for Conducting Ambient Air Quality and Meteorological Monitoring for the Proposed Gemfield Gold Mine. JBR Environmental Consultants, 2013.

20.3 CLOSURE

Mine closure and reclamation for the Project will be conducted in accordance with BLM and State of Nevada regulations and guidelines and best practices. A closure and reclamation plan will be developed for the Project and will include the open pit, waste rock dump and heap leach pad and other ancillary facilities. This plan is also required for the permitting of the Project. A closure bond, which is jointly held by the State of Nevada and the BLM, will be required to be posted prior to permit approval.

After operations cease, solution in the heap leach pad will be allowed to drain until the rate of flow from this facility can be passively managed through evaporation from the ponds or a combination of evaporation and infiltration. The barren, pregnant, and storm ponds will be backfilled and the double-lined barren and pregnant ponds converted to evaporation cells for heap drain-down.

The berm around the perimeter of the pit will be extended and a pit lake will be allowed to develop at closure. Pits can sometimes be excluded from requirements to backfill and/or re-contour at closure under the Nevada mining regulations. However, if possible, backfilling some of the pits can sometimes reduce long-term liabilities. This is an option that will be evaluated by IMZ during future studies.
Waste rock and heap leach facilities will be capped with a 2 ft cover and vegetation established that will comply with Nevada regulations. Reclamation works will be completed

in one year. Reclamation monitoring has been assumed for three years for vegetation and ten years for water.

Closure costing for the Project has been carried out using the Nevada Standardized Reclamation Cost Estimator (SRCE) model version 1.1.2, Heap Leach Drain-down Estimator (HLDE) version 1.2, Process Fluid Cost Estimator (PFCE), and 2011 mob-demob cost estimator, which are published in the Nevada Division of Environmental Protection (NDEP) website. The well closure cost estimate was carried out with the well abandonment sheet of the public domain version of the SRCE.

The closure cost associated with the Project is estimated to be $7.36 million.

20.4 PERMITTING

The permitting strategy identifies and addresses the various local, state, federal and international environmental and social requirements and standards applicable to the project. These include the statutory requirements, stakeholder interests, and environmental, social and economic aspects.

The strategy fulfills requirements or standards of the following:

- Local (Esmeralda County land use, environmental and safety requirements).
- Federal (BLM and other federal agencies with regulatory jurisdiction).
- State (Nevada) environmental and safety requirements.
- Canadian securities and CIM requirements for disclosure of liabilities associated with environmental aspects, including but not limited to, closure and reclamation.
- Financial institutions.
- The Board of Directors of IMZ.

A multi-agency regulatory process will be completed to obtain all required federal, state, and local agency permits and approvals necessary to construct, operate, and ultimately close the project.

Nevada permits are issued by the NDEP Bureau of Mining Regulation and Reclamation (BMRR). Three primary branches of BMRR include the Regulation Branch that issues water pollution control permits (WPCP), the Reclamation Branch that issues reclamation permits and the Closure Branch that reviews and approves closure plans. The latter branch is involved in an advisory role during project permitting and it reviews, approves, and administers formal final closure plans at the end of mine life. Under current NDEP administrative procedures, the Regulation Branch also issues permits for dewatering for mine

projects, in lieu of issuing a discharge permit by the Bureau of Water Pollution Control, in order to consolidate and streamline the permitting process for mines into one Bureau.

A review of the permits requirements indicates that the following primary permits/actions will be required:

- BLM Plan of Operations (including reclamation, closure, and bond estimate).
- BLM Environmental Impact Statement.
- NDEP Water Pollution Control Permit.
- NDEP Reclamation Permit.
- NDEP Air Quality Operating Permit.
- NDEP Mercury Air Quality Operating Permit to Construct.

A number of other permits will be required, but the above will require the most time, effort, and cost. Planned timing for the other permits will be such that all permits are received well before the proposed construction start date.

The following are the main permits required from the State of Nevada:

- Nevada Air Quality Operating Permit.
- Nevada Water Pollution Control Permit.
- Nevada Reclamation Permit.
- Nevada Water rights.
- Hazardous Material Storage Permit.
- Other Permits.

20.4.1 Plan of Operations/Reclamation Permit Application

The NEPA process is initiated by the submission of a Plan of Operations or POO to the BLM who will review the document for completeness and to identify potential issues. The POO was submitted to the BLM on July 10. This review will take at least 30 days. The same document can be submitted to NDEP as an application for a Reclamation Permit.

The POO includes basic information concerning the company and its principal contact, detailed maps showing all project components: size of areas where surface resources will be disturbed; information sufficient to describe either the entire operation proposed or reasonably foreseeable operations and how they would be conducted, including the nature and location of proposed structures and facilities; the type and condition of existing and proposed means of access, the means of transportation used or to be used, and the estimated period during which the project activities will take place.

The POO includes description of:

- Highway 95 realignment.
- Open Pit.

- Material handling.
- Crushing and Process Facilities.
- Heap leach.
- Process Ponds.
- Water Supply Ponds.
- Onsite/offsite Roads.
- Other Land Disturbance.
- Water Supply.
- Pipelines.
- Chemical Storage Facilities.
- Fuel Storage Facilities (Tanks and Pipelines).
- Surface Drainage Facilities.
- Electrical Generation/Transmission Facilities.
- Noise.
- Light.

Reclamation is a major part of the POO and must describe the reclamation measures that will take place and be used to prepare the reclamation cost estimate for bonding.

Because the NEPA analysis will be performed on the proposed POO, potential impacts have already been identified during the Feasibility Study and the Updated Study and mitigation incorporated in the design. For example, potentially sensitive cultural sites have been avoided in the rerouting of US Highway 95 and location of the waste dump.

20.4.2 NEPA Analysis

After the POO is reviewed and accepted, BLM is required to evaluate the potential impacts from the proposed project in accordance with NEPA. These impacts are analyzed through the preparation of an EA and/or an EIS. For the Project, it has been determined that an EIS is necessary.

During the NEPA process impacts for project construction, operation and closure will be evaluated. These include, at a minimum, the following:

- Air Quality.
- Cultural Resources.
- Environmental Justice and Socio-Economic Values.
- Wildlife.
- Native American Religious concerns.
- Vegetation, including Special Status species and Non-Native or Invasive species.
- Geochemistry.
- Water quality.
- Paleontology.
- Visual Resources.

The permitting schedule is very much affected by the context of the mine project, including the environmental elements in the project area, technical factors specific to the project, local community support, the level of public support/controversy, BLM workloads, BLM practices, BLM and NDEP staff assigned to the project, the EIS contractor that is selected by BLM, other NEPA analyses being conducted at the time, the political context etc.

IMZ is in process of developing a proactive, integrated, project-based management plan to meet all concerns, rather than separate, reactive, non-integrated plan. The management plan will be structured according to the context of the Project during any given phase and updated continuously during the life of the Project.

Successful permitting is as much a result of regulatory and community engagement, communications, transparency and superior technical quality as it is with meeting and fulfilling explicit statutory and regulatory requirements.

The IMZ technical team believes the Project could be permitted within 18 months from the acceptance of the POO for the following reasons:

- Meetings/communications with the BLM in 2012. The BLM advised IMZ of a fast-tracking mechanism in place for projects in Nevada and development of IMZ's Project is being targeted as one of the early ones under this new permitting mechanism. The BLM has also instituted a cost recovery program for review of projects that requires that the project proponent/owner pay for BLM staff time and expenses dedicated to the project. The exact cost of this will not be known until the process is initiated and will change depending upon the complexity of the environmental review and time to complete the project.

- The lack of many critical natural resources and environmental issues on the Property related to designated Areas of Critical Environmental Concern, farm lands, forests and rangelands, wild and scenic rivers, wilderness, wetlands, jurisdictional waters subject to Clean Water Act, riparian habitat, prime recreational areas and public opposition.

- Favorable site characteristics include the geology and rock types associated with the mine, the lack of meteoric water (approximately 6.5 inches of annual precipitation) and limited and manageable surface water, all of which make the Project favorable with regard to management of mine waste rock to prevent acid rock drainage and leaching of metals of concern.

- Local attitude to mining. Goldfield is a historic mining area and the local population is familiar with mining activities. Interaction with the community indicates a fair level of support with the local community. Even though there are active NGO's in Nevada, diligent engagement of the local community and stakeholders is a necessity and can help offset the inevitable state-level and national anti-mining sentiment.

- Many members of the IMZ project team have been involved with development of mining projects in both the western USA, including Nevada, and internationally, including several high profile mining projects. Their understanding of stakeholder and community concerns, technical issues, regulatory process, and government officials will be beneficial to guide the Project through the permit process.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

Capital expenditures and capitalized pre-production operating costs are summarized as initial and sustaining costs in Table 21.1. The estimates are expressed in first quarter 2012 US dollars, without escalation. The accuracy of the estimates is ±15%.

Table 21.1
Capital Cost Summary

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Permitting	600	-
Mining	20,253	4,825
Processing & Infrastructure	71,391	8,376
Indirect Costs	10,290	-752
Owner's Costs	34,332	-
Contingency	14,151	193
Total	**151,017**	**12,642**

An additional LOM amount of $7.9 million has been estimated to cover Project closure.

21.1.1 Mining

The initial capital cost estimate for mining comprises fleet purchase costs, as shown in Table 21.2. Sustaining capital comprises additions to the mining fleet in Years 1 and 2.

Table 21.2
Capital Cost Summary – Mining

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Mining Equipment	16,567	4,825
Capitalized pre-production	3,686	
Total	**20,253**	**4,825**

21.1.2 Process Plant & Infrastructure

The capital estimate components for the processing plant and infrastructure are shown in Table 21.3. On-going maintenance is covered by operating costs. The only sustaining capital is for an extension to the heap leach pad, which takes place in years 3 and 5 and the installation of drainage channels in years 1 and 2.

Table 21.3
Process Plant & Infrastructure Estimate

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
General Site	6,500	-
Primary Crushing	4,456	-
Coarse Ore stockpile	3,574	-
Secondary/Tertiary Crushing & S/Pile	16,088	-
Heap Leach	8,207	-
Ponds	2,581	-
ADR	10,423	-
Cyanide Storage	520	-
Refinery	915	-
Water Systems	4,918	-
Main Substation	1,918	-
Overhead Power Transmission	297	-
Reagents	724	-724
Ancillary Facilities	5,165	-
Laboratory/Assay Lab.	2,139	-
Mobilization	684	-
Taxes	2,282	-
Heap leach pad extension	-	6,611
Drainage diversion channels	-	2,490
Total	**71,391**	**8,376**

21.1.3 Indirect Capital

The indirect capital cost estimate is shown in Table 21.4.

Table 21.4
Indirect Capital Cost Estimate

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Engineering	4,238	-
Construction Management	3,565	-
Project Controls, Accounting	425	-
Project Services	182	-
Site CM Facilities	143	-
EPCM Fee	500	-
Vendor Support/Commissioning	370	-
Operating Spares	289	-289
Commissioning Spares	116	-
Capital Spares	463	-463
Total	**10,290**	**-752**

Due to its specialist nature, the heap leach pad and ponds have been separated from the rest of the process plant and infrastructure for the project, so the engineering and procurement, and construction management (EPCM) of these aspects have been estimated separately.

Indirect costs also include strategic spares and a provision for freight charges on capital equipment. No indirect sustaining capital is provided for in the estimate.

21.1.4 Owner's Costs

The Owner's capital cost estimate is shown in Table 21.5.

Table 21.5
Indirect Capital Cost Estimate

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
US Highway 95 realignment	20,918	-
Power Supply	1,291	-
Other	12,122	-
Total	**34,332**	**-**

"Other" costs include the Owner's technical team for project and technical management salaries and expenses as well as costs of operational personnel during construction, commissioning and start-up.

21.1.5 Capital Contingency

In addition to the above, contingency is provided in the capital estimate as shown in Table 21.6.

Table 21.6
Initial Capital Cost Summary – Contingency

Area	Initial Capital Cost ($ 000)	Sustaining Capital Cost ($ 000)
Mining contingency	663	193
Plant and Infrastructure contingency	8,803	-
Owner's costs contingency	4,685	-
Total	**14,151**	**193**

21.2 OPERATING COSTS

Estimated cash operating costs over the life of the project are summarized in Table 21.7.

Table 21.7
Summary of Life-of-Mine Operating Costs

Area	Life-of-mine Cost ($ 000)	Unit Cost $/T ore treated
Permitting cost (pre-production total)	2,000	-
Permitting cost (30% capitalized)	(600)	-
Sub-total Permitting cost (expensed)	**1,400**	**0.07**
Drilling and Blasting	28,128	1.48
Loading and Hauling	56,568	2.97
Support Equipment	24,569	1.29
Mine G & A	9,124	0.48
Less Capitalized OPEX	(3,686)	-0.19
Sub-total Mining	**114,703**	**6.03**
Operating Labor	21,173	1.11
Maintenance Labor	8,522	0.45
Power	10,773	0.57
Reagents and Consumables	43,351	2.28
Maintenance Spares	6,548	0.34
Truck Haulage to Leach Pad	5,560	0.29
Loaders and Dozers	3,284	0.17
Sub-total Processing	**99,210**	**5.21**
Labor	31,654	0.83
Power	2,183	0.14
General Expenses	8,955	0.56
Maintenance materials	6,452	0.20
Sub-total General and Administrative	**49,244**	**1.73**
Total Operating Costs	**248,214**	**13.05**

21.2.1 Permitting

Permitting costs of $2.0 million are incurred during the pre-production period. In terms of tax regulations, 30% of this cost is capitalized, with the balance of $1.4 million being treated as an operating expense, which creates a loss carry-forward into the subsequent operating period.

21.2.2 Mine Operating Costs

Mine operating costs are inclusive of operating and maintenance labor, spares and consumables for the Owner's fleet of production and support equipment.

21.2.3 Processing Operating Costs

Processing costs include crushing run-of-mine material, loading and hauling of crushed ore to the leach pad, heap construction, reagents and process consumables, power, operating and maintenance labor and spare parts.

21.2.4 General and Administration Costs

General and Administration costs include site management, supervisory and technical staff, power, office running costs, and environmental management. Corporate overheads are excluded.

22.0 ECONOMIC ANALYSIS

22.1 BASIS OF EVALUATION

Micon has prepared its assessment of the Gemfield Project on the basis of a discounted cash flow model, from which Net Present Value (NPV), Internal Rate of Return (IRR), payback and other measures of project viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the weighted cost of capital invested.

The objective of the study is to determine the viability of the proposed open pit mine and heap leaching process plant to exploit the Project. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV to be made. The sensitivity of this NPV to changes in the base case assumptions is then examined.

22.2 MACRO-ECONOMIC ASSUMPTIONS

22.2.1 Exchange Rate and Inflation

All results are expressed in United States dollars. Cost estimates and other inputs to the cash flow model for the project have been prepared using constant, first quarter 2013 money terms, i.e. without provision for escalation or inflation.

22.2.2 Cost of Equity Capital

The cash flow projections used for the valuation have been prepared on an all-equity basis.

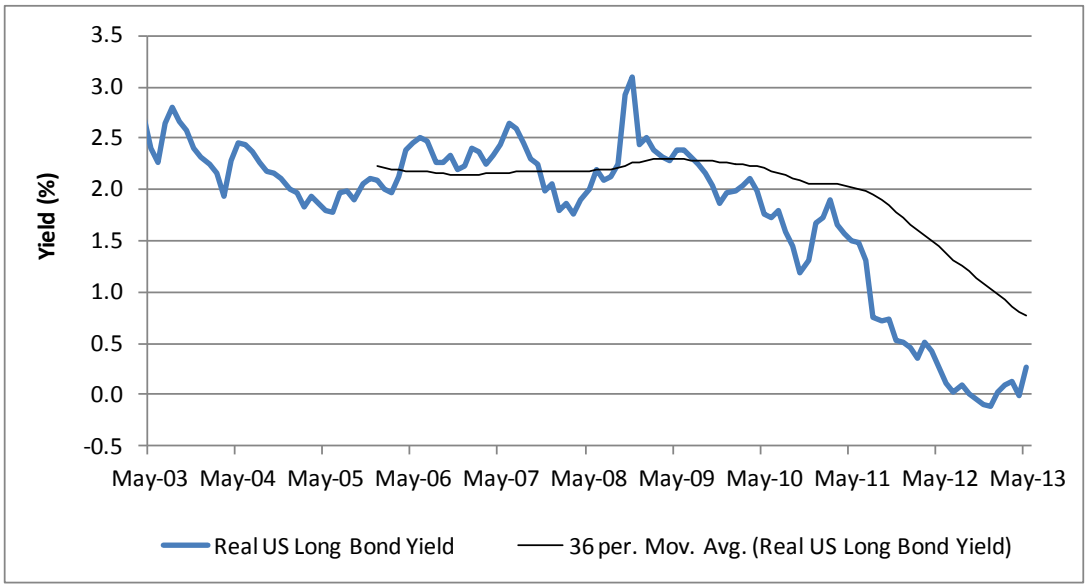
In order to estimate the NPV of the cash flows forecast for the Project, an appropriate discount factor must be applied which represents the cost of equity capital (CEC) imposed on the project by the capital markets. This being the case, CEC is equal to the market cost of equity, and can be determined using the Capital Asset Pricing Model (CAPM):

$$E(R_i) = R_f + \beta_i(\mathrm{E}(\mathrm{R_m}) - \mathrm{R_f})$$

Where $E(R_i)$ is the expected return, or the cost of equity. R_f is the risk-free rate (usually taken to be the real rate on long-term government bonds), $E(R_m)$-R_f is the market premium for equity, commonly estimated to be around 5%, and beta (β) is the volatility of the returns for the relevant sector of the market compared to the market as a whole.

Figure 22.1 illustrates the real return on US bonds computed by Federal Reserve, taken as a proxy for the risk-free interest rate. Recently, this has dropped from around 2.0% to less than 0.5%. Nevertheless, it is generally accepted that using a long-term average rate will give a more reliable estimate of the cost of equity. Micon has therefore used a value of 2.0% for the risk free rate, close to the real rate of return averaged over 10 years.

Figure 22.1
Real Return on US Bonds
(source: US Federal Reserve)



Taking beta for this sector of the equity market to be in the range 0.7 (among larger gold producers) to 1.3 (typical for the mining sector), CAPM gives a cost of equity for the Project of between 5% and 9%, as shown in Table 22.1. Micon has taken a figure in the middle of this range as its base case, and provides the results at alternative rates of discount for comparative purposes.

Table 22.1
Cost of Equity Capital

Range	Lower	Middle	Upper
Risk Free Rate (%)	1.5	2.0	2.5
Market Premium for equity (%)	5.0	5.0	5.0
Beta	0.7	1.0	1.3
Cost of equity (%)	**5.0**	**7.0**	**9.0**

22.2.3 Expected Gold Prices

Figure 22.2 shows the monthly average gold price over the past ten years, together with the 3-year trailing average. At the end of May, 2013, the three-year trailing average price was $1,546/oz gold. However, in the economic evaluation, a more conservative price of $1,350/oz was selected, being close to the spot price prevailing at the time of the economic analysis of the Project. This metal price was then applied consistently throughout the operating period of the LOM of the Project.

Silver was not attributed any value in the economic evaluation, notwithstanding that small amounts of silver are expected to be present in the doré produced at Gemfield.

Figure 22.2
Monthly Average Gold Price over 10 years)
(source: Kitco.com)

For comparison, Micon also evaluated the project with gold at $1,050, $1,200, $1,500 and $1,650/oz. As part of its sensitivity analysis, Micon also tested a range of prices 20% above and below the base case value.

22.2.4 Taxation Regime

US federal income tax and Nevada Net Proceeds of Minerals (NPOM) tax have been allowed for in the cash flow model. Regular and alternative minimum tax (AMT) was taken into consideration for federal tax.

Depreciation is applied on a unit of production basis and a 15% depletion allowance is taken, where applicable. Net operating loss carry-forwards related to the Property of $36.5 million are also taken into account.

22.2.5 Royalty

A 5% NSR royalty, payable to third parties, has been provided for in the cash flow model on 100% of the production from Gemfield. There is currently no federal or state royalty.

22.3 TECHNICAL ASSUMPTIONS

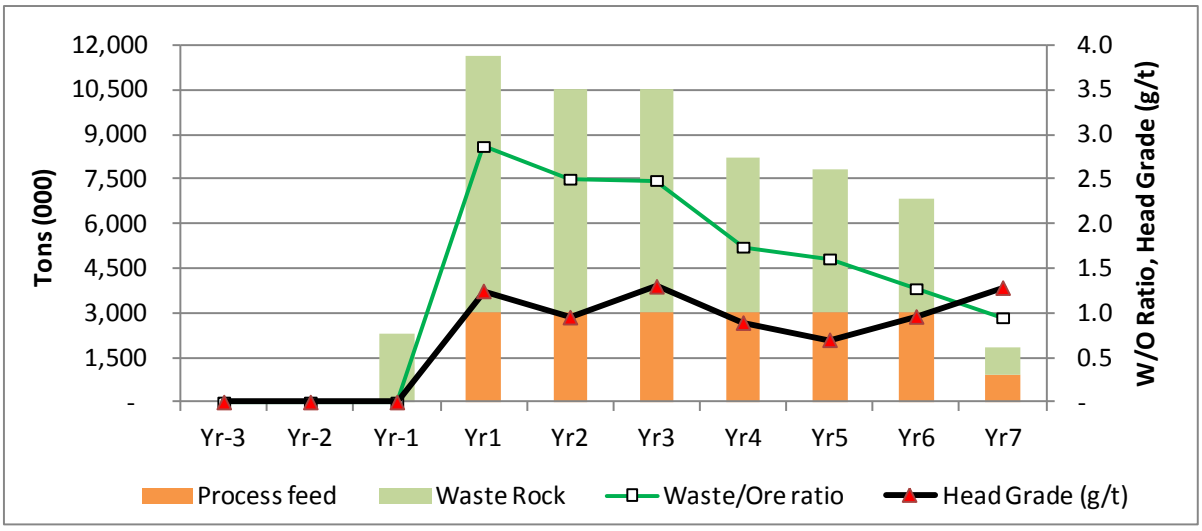
The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the base case cash flow model. These inputs to the model are summarized below. The measures used in the study are short tons and troy ounces. Where applicable, the metric equivalents of these are given here for reference only.

22.3.1 Mine Production Schedule

The mine plan was prepared on a quarterly basis and consolidated into annual periods for the purposes of the economic evaluation.

Waste mining is scheduled to start one quarter-period earlier than ore mining, with approximately 2.3 million tons of waste rock mined prior to the first ore being placed on the heap. Figure 22.3 shows the annual mining of ore and waste rock mining of all types, including alluvium, as well as ore grade (expressed in g/t Au) and the waste-to-ore ratio.

Figure 22.3
Annual Mining Production Schedule
(Showing Head Grade in grams per tonne (g/t))



Process plant throughput is held steady over the LOM period at 3.01 million tons annually. Except for 105,750 tons re-handled from a temporary stockpile in Year 1, all material processed is direct (run-of-mine) feed to the crusher.

A gold recovery of 85.2% is forecast over the LOM. The timing of sales is adjusted to reflect an inventory of recoverable gold in the leach pad equivalent to a 90 day production cycle.

Silver is present in the ore and a small amount will be recovered into the doré product. However, silver is not expected to make a significant contribution to project economics and is not used in the economic evaluation of the Project.

22.3.2 Selling Expenses

Doré transport charges of $5,000 per shipment, based on a weekly schedule, have been allowed for. Insurance charges of 0.1% of doré value are also provided.

Refining charges are estimated at $5.00/oz gold, based on similar mining projects elsewhere in the USA and internationally. Payability was estimated to be 99.9% of gold in the doré.

22.3.3 Closure Costs

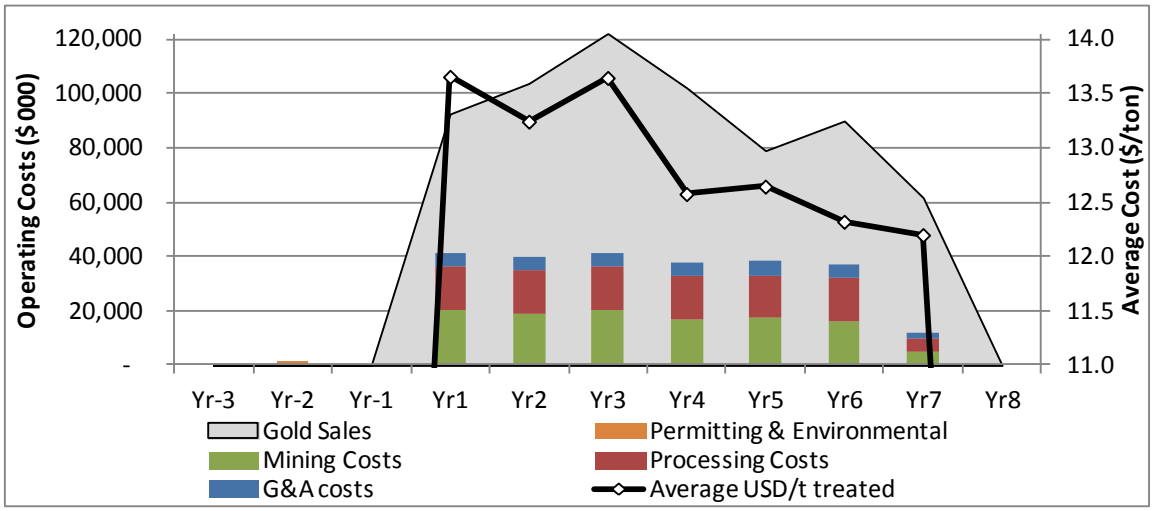
Estimated closure costs for the project are $7.36 million. Although this cost will only be incurred at the end of the Project life, the cash flow model assumes that a bond will be lodged for 25% of this amount (i.e. $1.84 million) upon Project start-up, and that the balance of the closure costs will be secured by way of insurance or similar financial instrument at an annual cost of 1.5% of that amount (i.e., $82,771 each year until closure). The closure costs are off-set by a refund in income tax in respect of the post-closure losses incurred.

22.3.4 Operating Costs

Direct operating costs average $13.05/T treated over the LOM period, comprising $6.03/T for mining, $5.21/T for processing, and $1.80/T for general and administrative (G&A) costs, including permitting and environmental expenses.

Figure 22.4 shows these expenditures over the LOM period, compared to the net sales revenue, showing the strong margin maintained over the LOM period in the base case. Over the LOM period, unit operating costs trend downward from over $13.50/T to around $12/T as the volume of waste rock mined reduces annually.

Figure 22.4
Cash Operating Costs



22.3.5 Capital Costs

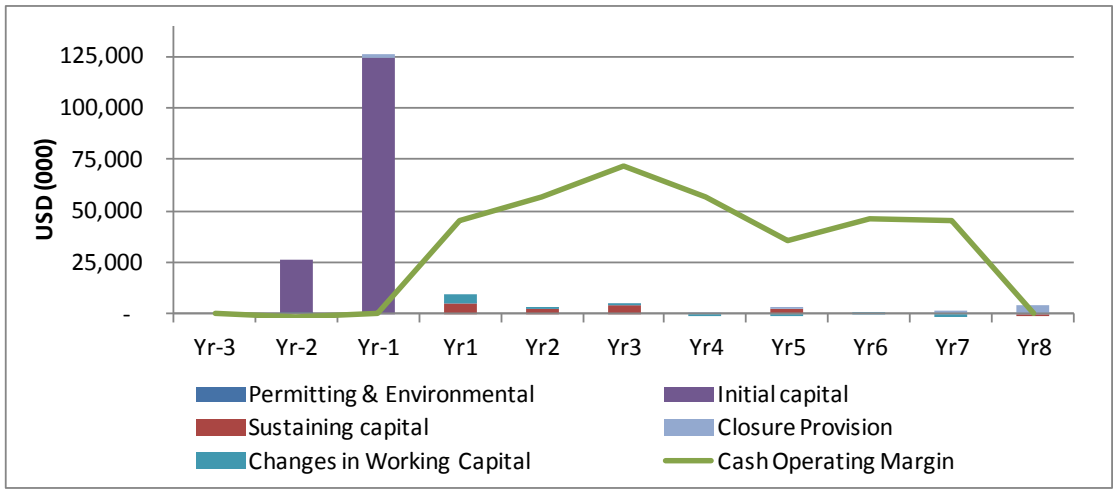
Pre-production capital expenditures are estimated to total $151.0 million, including $20.3 million for mining, $71.4 million for processing and infrastructure, $10.3 million for

indirect costs, and $34.3 million for owner's costs (including $20.9 million for realignment of US Highway 95), and contingencies totaling $14.1 million.

Working capital has been estimated to include 90 days product inventory in the heap leach circuit and 15 days receivables from dispatch of doré. Stores provision is for 30 days of consumables and spares inventory, less 30 days accounts payable. An average of $4.2 million of working capital is required over the LOM period.

Figure 22.5 compares annual capital expenditures over the preproduction and LOM periods with the project's cash operating margin.

Figure 22.5
Capital Expenditures



22.3.6 Base Case Cash Flow

The LOM base case project cash flow is presented in Figure 22.6 and Table 22.2. The application of gold revenue to each main area of usage is shown in Figure 22.7. Annual cash flows are presented in Table 22.3.

Direct site operating costs equate to $13.05/T. Adding bullion transport and refining costs gives an estimated cash operating cost of $524/oz gold produced. The NSR royalty and Nevada Net Proceeds of Minerals (NPOM) tax bring total cash cost estimates to $612/oz of gold produced.

Figure 22.6
Life-of-Mine Cash Flows

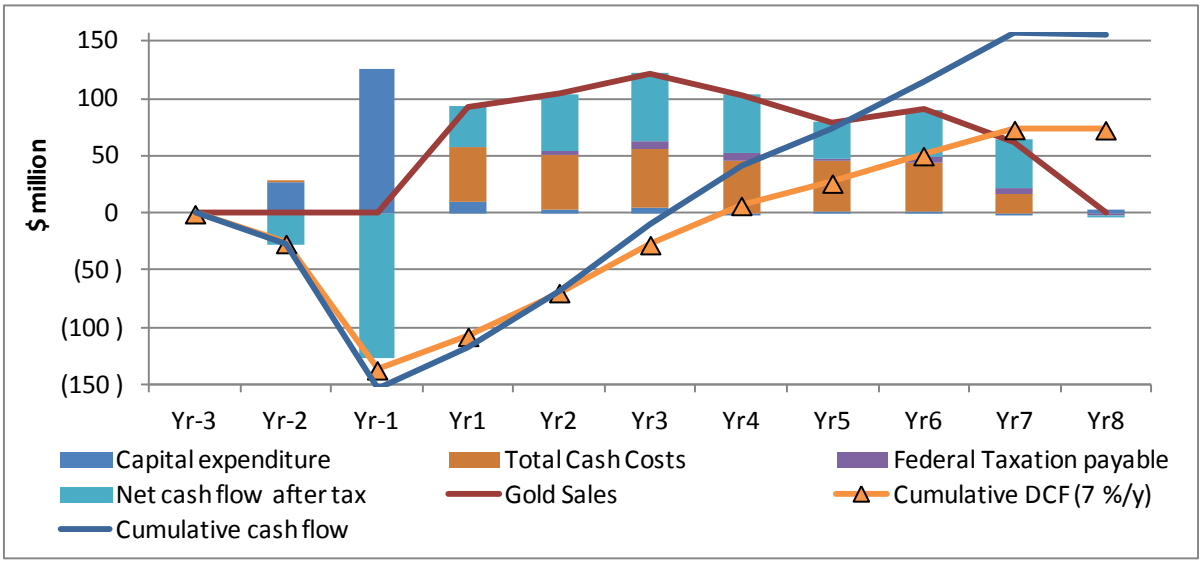


Table 22.2
Life-of-Mine Cash Flow Summary

	LOM total ($ 000)	$/T Processed	$/tonne Processed	$/oz Au Recovered
Gold Revenue	**651,824**	**34.26**	**37.77**	**1350.00**
Permitting & Environmental	1,400	0.07	0.08	2.90
Mining costs	114,703	6.03	6.65	237.56
Processing costs	99,210	5.21	5.75	205.48
General & Administrative costs	32,901	1.73	1.91	68.14
Direct site operating costs	**248,214**	**13.05**	**14.38**	**514.08**
Silver credit	-	-	-	-
Bullion delivery	1,640	0.09	0.10	3.40
Refining charges	2,414	0.13	0.14	5.00
Insurance	652	0.03	0.04	1.35
Cash operating costs	**252,920**	**13.29**	**14.65**	**523.83**
Nevada NPOM	10,067	0.53	0.58	20.85
Royalty	32,356	1.70	1.87	67.01
Total Cash Costs	**295,343**	**15.52**	**17.11**	**611.69**
EBITDA	**356,481**	**18.74**	**20.65**	**738.31**
Capital expenditure	171,597	9.02	9.94	355.40
Net cash flow (before tax)	**184,884**	**9.72**	**10.71**	**382.92**
Federal Taxation	28,854	1.52	1.67	59.76
Net cash flow (after tax)	**156,030**	**8.20**	**9.04**	**323.16**

Figure 22.7
Breakdown of Gold Revenues by Usage

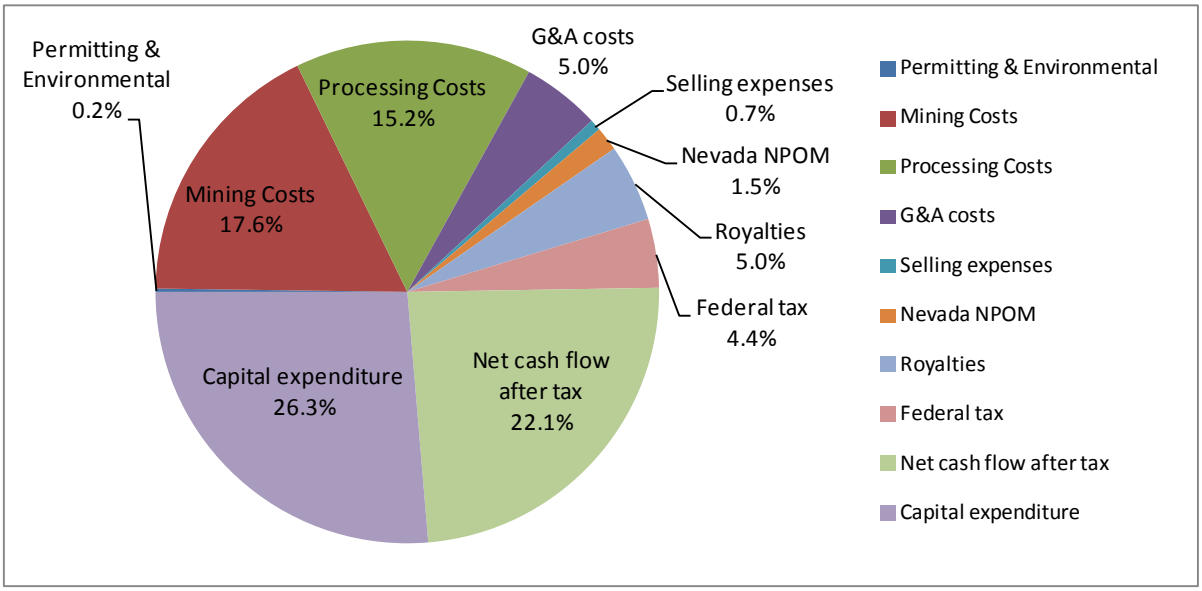


Table 22.3
Base Case Life of Mine Annual Cash Flow

		Yr end 30 Sept:	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Mining Schedule	ALL SHORT TONS												
Direct feed	000t	18,919			-	2,914	3,011	3,017	3,011	3,011	3,011	943	-
Stockpiled	000t	106			106	-	-	-	-	-	-	-	-
Process feed	000t	**19,025**			**-**	**3,020**	**3,011**	**3,017**	**3,011**	**3,011**	**3,011**	**943**	**-**
Waste Rock	000t	40,764			2,294	8,657	7,528	7,500	5,232	4,835	3,828	891	-
Total Mined	000t	59,789			2,294	11,676	10,539	10,517	8,243	7,846	6,839	1,834	-
Waste/Ore ratio	t/t	2.14			-	2.87	2.50	2.49	1.74	1.61	1.27	0.95	-
Ore treated	Leached (000 t)	**19,025**	**-**	**-**	**-**	**3,019.5**	**3,011.2**	**3,017.4**	**3,011.2**	**3,011.3**	**3,011.2**	**942.7**	**-**
	Au grade (oz/t)	0.0298				0.0364	0.0278	0.0379	0.0260	0.0204	0.0280	0.0374	-
	Au content (000 oz)	566.9				109.8	83.6	114.3	78.3	61.3	84.3	35.3	-
	Recovery (%)	85.2	-	-	-	82.95	86.68	84.49	88.18	89.96	83.75	80.37	-
	Au yield (000 oz)	483.3				91.0	72.5	96.6	69.0	55.2	70.6	28.4	-
Gold inventory	Au (000 oz)	90 days				22.4	17.9	23.8	17.0	13.6	17.4	-	-
Gold sales (gross)	Au (000 oz)	483.3				68.7	77.0	90.6	75.8	58.6	66.8	45.8	-
Gold payability	%	99.9%	-	-	-	99.9	99.9	99.9	99.9	99.9	99.9	99.9	-
Gold Sales (payable)	Au (000 oz)	482.8	-	-	-	68.6	76.9	90.5	75.7	58.6	66.7	45.7	-
Revenue	Gold Sales	651,824	-	-	-	92,598	103,868	122,235	102,253	79,078	90,047	61,745	-
Operating Costs	Permitting & Environmental	1,400	-	1,400	-	-	-	-	-	-	-	-	-
	Mining Costs	114,703	-	-	-	20,360	19,042	20,310	17,035	17,242	15,974	4,740	-
	Processing Costs	99,210	-	-	-	15,661	15,631	15,653	15,631	15,631	15,924	5,080	-
	G&A costs	32,901	-	-	-	5,209	5,201	5,207	5,201	5,201	5,201	1,679	-
	Direct Operating Costs	248,214	-	1,400	-	41,229	39,874	41,171	37,867	38,074	37,100	11,499	-
less	By-product credit	-	-	-	-	-	-	-	-	-	-	-	-
add	Bullion delivery	1,640	-	-	-	260	260	260	260	260	260	80	-
	Refining charges	2,414	-	-	-	343	385	453	379	293	334	229	-
	Insurance	652	-	-	-	93	104	122	102	79	90	62	-
	Cash Operating Costs	252,920	-	1,400	-	41,925	40,623	42,006	38,608	38,706	37,783	11,870	-
add	Production Tax (NPOM)	10,067	-	-	-	1,195	1,624	2,115	1,651	748	1,249	1,485	-
	Royalty	32,356	-	-	-	4,595	5,156	6,070	5,076	3,922	4,468	3,069	-
	Total Cash Costs	295,343	-	1,400	-	47,715	47,403	50,191	45,334	43,376	43,501	16,423	-
Cash Operating Margin		356,481	-	(1,400)	-	44,883	56,465	72,045	56,919	35,702	46,546	45,321	-
Capital Expenditure	Permitting & Environmental	600	-	600	-	-	-	-	-	-	-	-	-
	Initial capital	150,417	-	25,900	124,517	-	-	-	-	-	-	-	-
	Sustaining capital	12,643	-	-	-	5,072	2,436	4,348	-	2,263	-	-	(1,476)
	Closure Provision	7,937	-	-	1,839	83	83	83	83	83	83	1,554	4,047
	Changes in Working Capital	-	-	-	-	3,938	448	713	(776)	(908)	434	(3,849)	-
	Total Capital Expenditure	171,597	-	26,500	126,356	9,093	2,967	5,144	(693)	1,437	517	(2,294)	2,570
Net cash flow before tax		184,884	-	(27,900)	(126,356)	35,790	53,498	66,900	57,612	34,265	46,029	47,616	(2,570)
Federal Taxation payable		28,854	-	-	-	-	3,373	8,038	6,274	2,841	4,747	5,641	(2,060)
Net cash flow after tax		156,030	-	(27,900)	(126,356)	35,790	50,126	58,862	51,338	31,424	41,283	41,974	(510)
Total production cost	[Tot Cash Cost +Depr+Closure]	315,922	-	1,400	1,839	52,870	49,922	54,622	45,417	45,722	43,583	17,977	2,570
"All-In Cost"	[Tot Cash Cost +Tot Capex]	466,940	-	27,900	126,356	56,808	50,369	55,335	44,641	44,813	44,018	14,129	2,570
"Complete Cost"	[Tot Cash Cost +Tot Capex+Tax]	495,793	-	27,900	126,356	56,808	53,742	63,373	50,914	47,654	48,764	19,770	510
Cumulative cash flow			-	(27,900)	(154,257)	(118,466)	(68,341)	(9,478)	41,860	73,284	114,566	156,541	156,030
Payback period on undiscounted cash flow (years)		3.2				1.00	1.00	1.00	0.18	-	-	-	-
Discounted Cash Flow (7 %/y)		73,361	-	(26,075)	(110,365)	29,216	38,241	41,968	34,209	19,569	24,027	22,831	(259)
Cumulative DCF (7 %/y)			-	(26,075)	(136,440)	(107,224)	(68,983)	(27,015)	7,194	26,763	50,789	73,621	73,361
Payback period on discounted cash flow (years)		3.8				1.00	1.00	1.00	0.79	-	-	-	-
Capital expenditure		171,597	-	26,500	126,356	9,093	2,967	5,144	(693)	1,437	517	(2,294)	2,570
Ave Revenue per tonne treated		15.52	-	-	-	15.80	15.74	16.63	15.05	14.40	14.45	17.42	-
Ave Cost per tonne treated		13.05	-	-	-	13.65	13.24	13.64	12.58	12.64	12.32	12.20	-
Operating Margin		16.0%	0.0%	0.0%	0.0%	13.6%	15.9%	18.0%	16.5%	12.2%	14.7%	30.0%	0.0%
Cash Operating Cost per ounce payable gold		523.83	-	-	-	611.23	527.99	463.92	509.72	660.78	566.46	259.53	-

22.3.7 Base Case Evaluation

The Project demonstrates an undiscounted pay back of 3.2 years, or approximately 3.8 years discounted at 7.0%, leaving a production tail of just over 2 years. The base case generates an IRR of 23.2% before tax and 20.4% after tax. At a discount rate of 7.0%, the net present value (NPV_7) of the cash flow is $92.4 million before tax and $73.4 million after tax.

Table 22.4 presents the results at comparative discount rates of 5%/year, 7%/year and 9%/year.

Table 22.4
Base Case Cash Flow Evaluation

$ million	LOM Total	Discounted at 5%/y	Base Case Discounted at 7%/y	Discounted at 9%/y	IRR (%)
Gold Revenue	**651,824**	**494,467**	**445,354**	**402,379**	
Permitting & Environmental	1,400	**1,333**	**1,308**	**1,284**	
Mining costs	114,703	**88,000**	**79,591**	**72,200**	
Processing costs	99,210	**75,477**	**68,049**	**61,541**	
General & Administrative costs	32,901	**25,040**	**22,578**	**20,422**	
Direct site operating costs	**248,214**	**189,850**	**171,528**	**155,447**	
Silver credit	-				
Bullion delivery	1,640	**1,249**	**1,126**	**1,019**	
Refining charges	2,414	**1,831**	**1,649**	**1,490**	
Insurance	652	**494**	**445**	**402**	
Cash operating costs	**252,920**	**193,424**	**174,749**	**158,358**	
Nevada NPOM	10,067	**7,592**	**6,823**	**6,152**	
Royalty	32,356	**24,545**	**22,107**	**19,973**	
Total Cash Costs	**295,343**	**225,561**	**203,678**	**184,484**	
EBITDA	**356,481**	**268,906**	**241,676**	**217,895**	
Capital expenditure	171,597	**155,126**	**149,277**	**143,791**	
Net cash flow (before tax)	**184,884**	**113,780**	**92,399**	**74,103**	**23.2**
Taxation (including NPOM)	28,854	**21,358**	**19,038**	**17,018**	
Net cash flow (after tax)	**156,030**	**92,422**	**73,361**	**57,085**	**20.4**

22.4 SENSITIVITY STUDY

22.4.1 Capital, Operating Costs and Revenue Sensitivity

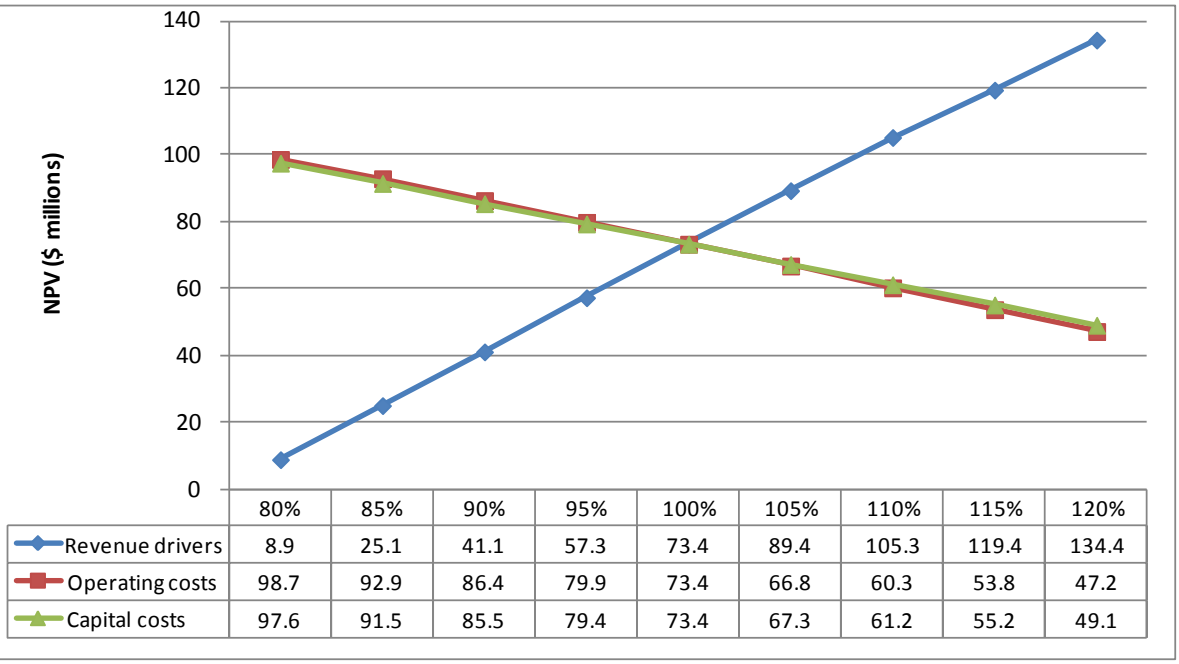
The sensitivity of the project returns to changes in all revenue factors (including grades, recoveries, prices and exchange rate assumptions) together with capital and operating costs was tested over a range of 20% above and below base case values. The results show that the project is most sensitive to revenue factors, with an adverse change of 20% reducing NPV_7 from $73.4 million to $8.9 million. The impact of changes in capital and operating costs are

almost identical: adverse changes of 20% reduce NPV_7 to $49.1 million and $47.2 million, respectively.

In Micon's analysis, applying an increase of more than 30% in both capital and operating costs simultaneously would be required to reduce NPV_7 to below zero.

Figure 22.8 shows the results of changes in each factor separately.

Figure 22.8
Sensitivity Diagram



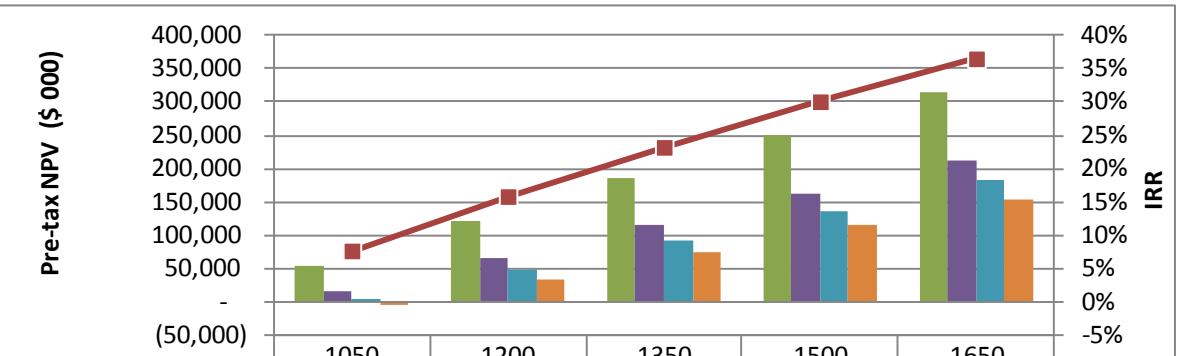
	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue drivers	8.9	25.1	41.1	57.3	73.4	89.4	105.3	119.4	134.4
Operating costs	98.7	92.9	86.4	79.9	73.4	66.8	60.3	53.8	47.2
Capital costs	97.6	91.5	85.5	79.4	73.4	67.3	61.2	55.2	49.1

22.4.2 Metal Price and Discount Rate Assumptions

The sensitivity of the project to variation in gold price was tested using values of $1,050/oz, $1,200/oz, $1,350/oz (base case), $1,500/oz and $1,650/oz over the life-of-mine period, as shown in Figure 22.9.

The base case price of $1,350/oz approximates the spot price at the time of Micon's economic evaluation estimate and is below the three-year trailing average price of $1,546/oz at the end of May, 2013, At $1,050/oz gold the project approaches an economic breakeven (zero NPV) using the base-case annual discount rate of 7%.

Figure 22.9
Sensitivity to Metal Prices



	1050	1200	1350	1500	1650
Undiscounted c/f	54,202	119,586	184,884	250,182	315,480
Pre-tax NPV 5%	14,838	64,339	113,780	163,221	212,662
Pre-tax NPV 7%	3,350	47,901	92,399	136,897	181,395
Pre-tax NPV 9%	(6,295)	33,928	74,103	114,279	154,455
IRR (%) [RHS]	8%	16%	23%	30%	37%

22.5 CONCLUSION

Micon concludes that this Updated Study demonstrates the viability of the Gemfield Project as proposed and that further development is warranted.

23.0 ADJACENT PROPERTIES

IMZ controls most of the minerals rights for the Property. The other significant group of mineral claims and patents are owned by Lode-Star Gold Inc., who controls an area adjacent to the town of Goldfield (which currently is subject to an option agreement with Corazon Gold Corp.). In addition, there are some minor claims in the area of the Combination Pit, where IMZ and Lode-Star Gold share ownership. Multiple small claims within and around IMZ property have individual owners.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT IMPLEMENTATION

A comprehensive project schedule was developed by IMZ and Micon. This schedule has been derived from information developed during execution of the Feasibility Study and the Updated Study for the Project. It reflects a phased approach to the engineering, permitting, procurement, construction (EPCM) and commissioning of the Project.

The final approval of the EIS is the critical issue for all on-site development and construction activities at Gemfield.

Although the Project has been evaluated on a 100% equity basis, IMZ may fund the project from a combination of working capital and debt. Due to the market capitalization of IMZ, the existing cash-flow from its Peruvian projects - the Pallancata silver mine and the anticipated cash-flow (in 2015) from the Inmaculada gold mine - IMZ anticipates that raising the finance necessary to implement the development and construction of the Project should not present a significant challenge.

A summary of project milestones presented in Table 24.1.

Table 24.1
Summary of Project Milestones

Item	Date (mmm-yy)
Submit Plan of Operations	Jul-13
Plan of Operations accepted	Aug-13
Basic engineering - complete	May-13
Long lead item purchased	Jan-14
Start EIS process	Aug-13
Record of Decision	Dec-14
Detailed engineering complete	Dec-14
Construction complete	Dec-15
First gold pour	Feb-15

25.0 INTERPRETATION AND CONCLUSIONS

The Goldfield property (the Property) is located in the historic Goldfield Mining District of Goldfield, Nevada, USA (the Goldfield Mining District) approximately 30 miles south of Tonopah, Nevada, adjacent to the town of Goldfield in Esmeralda County. The Property includes three known mineralized deposits, Goldfield Main, McMahon Ridge and Gemfield. Mineral resources for all three of these deposits have been updated.

The Gemfield Project (the Project) currently comprises the development of Gemfield as an open-pit heap leach operation and IMZ has completed an update of the Feasibility Study (the Updated Study) on the potential production of gold and silver from this deposit. The Updated Study is based on the proposed open-pit mining and heap leach processing of the mineral reserves at a rate of 8,250 dry short tons per day (T/d) to produce a gold and silver doré product on-site.

25.1 MINERAL RESOURCE ESTIMATE

The mineral resource estimates for the Gemfield deposit was updated from 3D block models in which grades have been interpolated from drill-hole data using ordinary kriging within separate geological and statistical domains.

Table 25.1 summarizes the current estimate of the mineral resources at the Gemfield deposit at a cut-off grade of 0.25 g/t Au.

Table 25.1
Mineral Resource Estimate for the Gemfield Deposit, at a Cut-off Grade of 0.25 g/t Au (June 17th, 2013)

Classification	Metric Tonnes (Mt)	Au (g/t)	Au Ounces
Measured	15.50	1.05	524,000
Indicated	9.10	0.54	157,000
Measured + Indicated	**24.59**	**0.86**	**681,000**
Inferred	1.08	0.52	18,000

Notes:
3. Numbers are rounded to reflect the precision of a resource estimate.
4. The contained metal estimates remain subject to factors such as mining dilution and losses and, process recovery losses.

25.2 GOLDFIELD UPDATED STUDY

The Updated Study is based on the proposed open-pit mining of the Gemfield deposit and heap leach processing of the mineral reserves at a rate of 8,250 dry short tons per day (T/d) to produce a gold doré product on-site.

The results of the Updated Study comprise the following:

- Total proven and probable reserves of 19.0 million T grading 0.0298 oz/T containing 567,000 oz of gold.
- Target nominal annual heap leach mill feed rate of 8,250 T/y.
- Life-of-mine (LOM) waste-to-leach feed ratio of 2.14.
- The life of the operating mine will be approximately six years.
- Conventional open-pit mining techniques and gold heap leach processing technology will be used to produce a gold doré product.
- Estimated LOM gold recovery of 85.2%.
- Estimated LOM gold production of 483,000 oz and an average annual production of approximately 76,000 oz.
- Access to site will be via the nearby US Highway 95.
- Electrical power will be provided from the existing power line which traverses the project site, on-site generation will be for emergency purposes only.

The results of the Updated Study are summarized in Table 25.2. All dollars are in United States Dollars.

Table 25.2
Summary of the Updated Study Base Case Results

Item	Unit	Value
Total life-of-mine leach feed production	T (000s)	19,025
Total life-of-mine waste production	T (000s)	40,789
Average gold grade	oz/T	0.0298
Average gold process recovery	%	85.2
Total life-of-mine gold production	oz (000s)	483.3
Annual gold production (average)	oz (000s)	76.5
Life of the mine	Years	6.3
Pre-production capital cost	$ millions	151.0
Sustaining and closure capital	$ millions	20.6
LOM on-site operating cost	$ millions	248.2
LOM cash operating cost	$/T leach feed	13.05
Average base case gold price	$/oz	1,350
LOM gross gold sales	$ millions	651.8
LOM off-site costs and Nevada NPOM	$ millions	14.8
LOM royalties	$ millions	32.4
LOM net revenue	$ millions	356.5
Project cash flow before tax	$ millions	184.9
Pre-tax NPV @ 7.0% discount rate	$ millions	92.4
Pre-tax NPV@ 5.0 % discount rate	$ millions	113.8
Pre-tax NPV@ 9.0 % discount rate	$ millions	74.1

Item	Unit	Value
Project cash flow after tax	$ millions	156.0
After tax NPV @ 7.0% discount rate	$ millions	73.4
After-tax NPV@ 5.0 % discount rate	$ millions	92.4
After-tax NPV@ 9.0 % discount rate	$ millions	57.1
Pre-tax IRR	%	23.2
After-tax IRR	%	20.4

Micon concludes that this Updated Study demonstrates the viability of the project as proposed and that further development is warranted.

26.0 RECOMMENDATIONS

Following the completion of the Updated Study, it is recommended that IMZ continue the development of the Gemfield Project into detailed engineering and construction. It is also recommended that all the necessary work to finalize the project environmental acceptance and permitting be completed.

It is also recommended to continue with the development of the Goldfield Main and McMahon Ridge deposits, including exploration, geological modeling and metallurgical testing.

26.1 BUDGET

The following costs have been estimated for the continued development of the Project:

Table 26.1
Gemfield Project Budget

Activity	Estimate ($ 000))
Detailed engineering and procurement	9,057
Environmental and permitting	2,000
IMZ dedicated project support staff	1,500
Total	**12,557**

Other costs budgeted over the next two years for the development of the Goldfield Main, McMahon Ridge deposits and other exploration targets on the Property includes $1 million for exploration and $250,000 for metallurgical investigations.

Micon concludes that the work items and cost budgets are reasonable

27.0 DATE AND SIGNATURE PAGE

R. Mohan Srivastava *{signed and sealed}*

R. Mohan Srivastava, P.Geo.
Signing Date: 25 July, 2013

"Sam Shoemaker" *{Signed and sealed}*

Sam J. Shoemaker, Reg.Mem.SME
Signing Date: 25 July, 2013

"Richard Gowans" *{Signed and sealed}*

Richard Gowans, P.Eng.
Micon International Limited
Signing Date: 25 July, 2013

"Christopher Jacobs" *{Signed and sealed}*

Christopher Jacobs, CEng, MIMMM
Micon International Limited
Signing Date: 25 July, 2013

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29.0 CERTIFICATES

CERTIFICATE OF AUTHOR
Richard M. Gowans

As a co-author of this report entitled "NI 43-101 Technical Report, Update to Feasibility Study on the Goldfield Property, Nevada, USA", with an effective date of 17 June, 2013 (the "Technical Report"), I Richard M. Gowans, P. Eng., do hereby certify that:

1. I am employed by, and carried out this assignment for:

   Micon International Limited
   Suite 900, 390 Bay Street
   Toronto, Ontario,
   M5H 2Y2
   tel. (416) 362-5135 fax (416) 362-5763
   e-mail: rgowans@micon-international.com

2. I hold the following academic qualifications:

   B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K., 1980

3. I am a registered Professional Engineer of Ontario (membership number 90529389); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked as an extractive metallurgist in the minerals industry for over 30 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and design of numerous metallurgical testwork programs and metallurgical processing plants.

6. I have not visited the Property.

7. I am responsible for the preparation of Sections 1.0 to 6.0, 13.0, 17.0 to 21.0, and 23.0 to 28.0 of this Technical Report.

8. I am independent of International Minerals Corp., as defined in Section 1.5 of NI 43-101.

9. I was an author of a previous NI 43-101 Technical Report on the project with an effective date of 17 July, 2012.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 25rd day of July, 2013

"Richard M. Gowans" {signed and sealed}

Richard M. Gowans, P.Eng.

CERTIFICATE OF AUTHOR
Christopher Jacobs

As a co-author of this report entitled "NI 43-101 Technical Report, Update to Feasibility Study on the Goldfield Property, Nevada, USA", with an effective date of 17 June, 2013 (the "Technical Report"), I, Christopher Jacobs, do hereby certify that:

1. I am employed by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135 email: cjacobs@micon-international.com

2. I hold the following academic qualifications:
   B.Sc. (Hons) Geochemistry, University of Reading, 1980;
   M.B.A., Gordon Institute of Business Science, University of Pretoria, 2004.

3. I am a Chartered Engineer registered with the Engineering Council of the U.K.
   (registration number 369178);

   Also, I am a professional member in good standing of: The Institute of Materials, Minerals and Mining; and The Canadian Institute of Mining, Metallurgy and Petroleum (Member);

4. I have worked in the minerals industry for 30 years; my work experience includes 10 years as an exploration and mining geologist on gold, platinum, copper/nickel and chromite deposits; 10 years as a technical/operations manager in both open-pit and underground mines; 3 years as strategic (mine) planning manager and the remainder as an independent consultant when I have worked on a variety of deposits including gold and silver;

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101;

6. I have not visited the Property;

7. I am responsible for the preparation of Section 22.0 of this Technical Report.

8. I am independent of International Minerals Corp., as defined in Section 1.5 of NI 43-101;

9. I was an author of a previous NI 43-101 Technical Report on the project with an effective date of 17 July, 2012

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument;

11. As of the date of this certificate to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 25rd day of July, 2013.

"Christopher Jacobs" {signed and sealed}

Christopher Jacobs, CEng, MIMMM

CERTIFICATE OF AUTHOR
R. Mohan Srivastava

As a co-author of this report entitled "NI 43-101 Technical Report, Update to Feasibility Study on the Goldfield Property, Nevada, USA", with an effective date of 17 June, 2013 (the "Technical Report"), I , R. Mohan Srivastava, P. Geo., do hereby certify that:

1. I am employed by, and carried out this assignment for:

   Benchmark Six Inc.
   Suite 1121, 120 Eglinton Avenue East
   Toronto, Ontario
   M4P 1E2

   tel. (416) 322-2857, fax (416) 322-5075
   e-mail: MoSrivastava@fssconsultants.ca

2. I hold the following academic qualifications:

   B.Sc., Earth Sciences, Massachusetts Institute of Technology (1979)
   M.Sc., Applied Earth Sciences (Geostatistics), Stanford University (1987)

3. I am a registered Professional Geologist of Ontario (Membership Number 0547).

4. I have worked as a geostatistician in the minerals industry for over 30 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the calculation, review and supervision of mineral resource estimates for base and precious metals deposits, including porphyry-style gold/copper deposits.

6. I visited the Goldfield property on January 29th to 31st, 2005, and again on September 29th and 30th, 2010.

7. I am responsible for Sections 7.0 to 12.0 and 14.0 of the Technical Report.

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.5 of NI 43-101.

9. From 2004-2009, I assisted Metallic Ventures Inc. with their resource estimation studies for the. Gemfield and McMahon Ridge deposits. In 2005, I assisted Watts Griffis and McOuat with its technical review of the resource estimates done by Metallic Ventures Inc. Since 2009, I have assisted International Minerals Inc. with its evaluation of the Goldfield property acquisition, and with subsequent geological, resource estimation studies on the Property, as well as statistical evaluation of data from metallurgical test work

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

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

    Dated this 25rd day of July, 2013.

    "R. Mohan Srivastava" {signed and sealed}

    R. Mohan Srivastava, P.Geo.

CERTIFICATE OF AUTHOR
Sam Shoemaker, Jr., Reg.Mem.SME

As a co-author of this report entitled "NI 43-101 Technical Report, Update to Feasibility Study on the Goldfield Property, Nevada, USA", with an effective date of 17 June, 2013 (the "Technical Report"), I, Sam Shoemaker, do hereby certify that:

1. I am the Principal of Shoemaker Mining Services Inc., 109 Canberra Street, Gwinn, Michigan 49841, USA, and carried out this assignment as an Associate of Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, tel. (416) 362-5135, fax (416) 362-5763, e-mail sshoemaker@micon-international.com.

2. I hold the following academic qualifications:

   B.Sc., Mine Engineering, Montana College of Mineral Science and Technology, 1982

3. I am a registered of the member of the Society for Mining, Metallurgy, and Exploration, Inc. (Member Number 2941320); as well, I am a member in good standing of other technical associations and societies, including the Australasian Institute of Mining and Metallurgy (Member Number 229733

4. I have worked as a mining engineer in the minerals industry for over 30 years.

5. I have read NI 43-101 and Form 43-101F1 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 10 years as a mining engineer with Cleveland Cliffs Inc. and 18 years with other mining companies where I was responsible for completing geologic models, reserve estimates, economic analysis, slope designs, pit optimization, pit design, long term scheduling, short term scheduling and reserve validation.

6. I visited the Property on 15 February, 2013.

7. I am responsible for the preparation of Sections 15.0 and 16.0 of the Technical Report.

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

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

    Dated this 25rd day of July, 2013.

    "Sam Shoemaker, Jr." {Signed}

    Sam Shoemaker, Jr., B.Sc., Reg.Mem.SME.