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HUAKAN INTERNATIONAL MINING INC.

NI 43-101 TECHNICAL REPORT

A PRELIMINARY ECONOMIC ASSESSMENT OF THE MAIN ZONE, J&L DEPOSIT REVELSTOKE B.C., CANADA

Effective Date of Resource Estimate: 16 May, 2011
Effective Date of Preliminary Economic Assessment: 24 April, 2012
Signature Date: 6 June, 2012

Tracy J. Armstrong, P.Geo.
Fred H. Brown, CPG, Pr.Sci.Nat.
Catherine A. Dreesbach, P.E.
Bogdan Damjanović, P.Eng.
Christopher Jacobs, CEng, MIMMM

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 J&L Property represents one of the largest undeveloped polymetallic deposits in British Columbia and is located 35 km north of Revelstoke. The property consists of 20 mineral tenure claims and 10 crown granted claims for a total of 3,051.73 ha. The J&L Property had been owned for many years by the T. Arnold estate and was optioned to Merit Mining Corp. (Merit) in 2007. In August, 2010, Merit exercised its option by advancing cash payments and share issuances, resulting in a 100% unencumbered interest in the J&L Property. In December, 2010, the company changed its name to Huakan International Mining Inc. (Huakan).

Micon International Limited (Micon) has been retained by Huakan to prepare a Technical Report in accordance with Canadian National Instrument (NI) 43-101 to support the disclosure of the results of the preliminary economic assessment (PEA) on the potential production of gold, silver, lead and zinc from the J&L Main Zone deposit.

The PEA is based on the proposed underground mining and carbon-in-leach processing of the mineral resources at a rate of 1,500 t/d to produce gold doré and saleable concentrate(s). The mineral resource estimate, on which the PEA is based, was previously disclosed in a technical report dated June 23, 2011 and filed on SEDAR.

1.1 HISTORY

The J&L area was first explored as early as 1865, when placer miners discovered gold in Carnes Creek. By 1896, two prospectors, Jim Kelley and Lee George, staked the first claims at the junction of Carnes and McKinnon Creeks, with the earliest work (1896-1900) carried out at the Roseberry mineral zone, 5 km northwest of where the J&L zone was later discovered. The property has been referred to as the J&L since its discovery by these two prospectors.

The Company entered into an option agreement dated April 13, 2007, whereby it may acquire a 100% undivided interest in the J&L Property in consideration for share issuances and cash payments totalling $10.79 million over a seven year period. In August, 2010, the Company exercised the option by advancing the cash and share issuances to acquire a 100% undivided interest in the J&L Property. There are no net smelter return (NSR) royalties owing to any parties.

1.2 GEOLOGY AND MINERALIZATION

The Property lies within the Selkirk Mountains near the north end of the Kootenay Arc, a complex belt of northwest-trending, east-dipping Neoproterozoic to Lower Paleozoic metasedimentary and metavolcanic miogeosynclinal rocks (Logan et al., 1997 A & B). The belt is characterized by tight to isoclinal folds and generally west-verging thrust faults. Greenschist grade regional metamorphism has affected most of the rocks in the area. Recent mapping by provincial government geologists has outlined the regional geology of the area.

The J&L Property contains two distinct deposits: the Main Zone and the Yellowjacket Zone. The Main Zone strikes north-west at about 330° and dips at about 50° to the northeast, and contains Au, Ag, Pb, and Zn. The Ag-Pb-Zn-bearing Yellowjacket is considered to be the remnant of a much larger, pre-existing, carbonate-hosted deposit which has subsequently been modified (remobilized, augmented and replaced) by the Main Zone structure and mineralizing system. Only the Main Zone deposit is considered in this PEA.

1.3 EXPLORATION

The J&L Property has been intermittently explored for over 100 years. The initial recorded mapping and prospecting was done by Dr. Gunning of the Geological Survey of Canada (GSC) in 1928. Other companies that have mapped and prospected the J&L include Piedmont Mines Ltd. (1929), Westairs Mines Ltd. (1963 to 1965), BP Selco (1981 to 1985), Equinox Resources Ltd. (1989) and Weymin Mining Corporation (1996 to 1997). The results of these mapping programs identified the surface trace of the Main Zone and several other parallel mineralized structures.

The Company, known as Merit Mining at the time, completed a 1,363-m, nine hole surface drill program on the Yellowjacket Zone in November 2007 with the objective of verifying historic drilling over a portion of the Yellowjacket deposit. The program successfully achieved this objective by intercepting multiple Ag-Pb-Zn zones similar in grade and width to previous drilling. The Yellowjacket Zone has no resources reported on in this technical report.

In November 2010, Merit/Huakan commenced a Phase I 2,000-m underground drill program with the objectives of verifying historic drilling and broadening the known resource in the Main Zone. This drilling was intended to provide more data for a mineral resource estimate compliant with the reporting requirements of NI 43-101. The program was expanded to 7,897 m and was completed by early February, 2011 with the completion of 60 BQTW (thin wall) core holes.

1.4 MINERAL RESOURCE ESTIMATE

The basis for the PEA is the mineral resource estimate prepared by P&E Engineering Inc. (P&E), which was dated June 23, 2011 and filed on SEDAR, as shown in Table 1.1 below.

Table 1.1
Main Zone Mineral Resource Estimate

Classification	Resource (Tonnes)	Au (g/t)	Au (oz)	Ag (g/t)	Ag (oz)	Pb (%)	Zn (%)
Measured	1,202,000	6.71	259,200	69.0	2,665,000	2.40	4.46
Indicated	1,165,700	6.92	259,200	64.9	2,432,000	2.01	3.86
Measured & Indicated	2,367,700	6.81	518,400	66.95	5,097,000	2.21	4.16
Inferred	4,538,100	5.19	757,500	67.8	9,888,000	2.16	2.99
Footwall Zone							
Classification	**Resource (tonnes)**	**Au (g/t)**	**Au (oz)**	**Ag (g/t)**	**Ag (oz)**	**Pb (%)**	**Zn (%)**
Inferred	292,800	4.54	42,700	49	461,900	0.91	0.73

1) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2) Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters. There is no guarantee that all or any part of a mineral resource can or will be converted into a mineral reserve.
3) The mineral resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
4) The following parameters were used to derive the NSR block model values:
   - -April 30/11 two year trailing average. metal prices of Au US$1,183/oz, Ag US$21/oz, Pb US$0.99/lb, Zn US$0.95/lb
   - -Exchange rate of US$0.95 = $1.00
   - -Process recoveries of Au 92%, Ag 88%, Pb 80%, Zn 72%
   - -Smelter payables of Au 96%, Ag 91%, Pb 95%, Zn 85%,
   - -Refining charges of Au US$15/oz, Ag US$0.50/oz
   - -Concentrate freight charges of $65/t and Smelter treatment charge of US185/t
   - -Mass pull of 5% and 8% concentrate moisture content.
5) The NSR cut-off of $110 per tonne was derived from $75/t mining, $25/t processing and $10/t G&A
6) The effective date of the mineral resource estimate is May 16, 2011.

Fred Brown, CPG, of P&E was the independent Qualified Person (QP) responsible for preparing the mineral resource estimate. The mineral resource estimate involved 3D modeling methods and parameters, and statistical and grade continuity analyses. Gemcom GEMS modeling software was used for the 3D block model and subsequent grade estimates. Grade capping was used to restrict the influence of statistical outliers during inverse distance squared (ID^2) interpolation of block grades. A total of 266 drill holes have been completed on the property. The resource estimate utilized a total of 396 bulk density measurements determined by on site by the Company, utilizing the wet immersion technique.

1.5 MINING

An underground mine plan developed by Micon comprises 1.09 million tonnes of measured resources at an average grade of 6.32 g/t Au, 65.91 g/t Ag, 4.16% Zn, and 2.28% Pb; 1.05 million tonnes of indicated resources at an average grade of 7.10 g/t Au, 66.08 g/t Ag, 3.94% Zn, and 2.04% Pb; and 2.76 million tonnes from inferred mineral resources at an average grade of 6.20 g/t Au, 72.43 g/t Ag, 3.24% Zn, and 2.22% Pb. The mine plan includes provision for mining losses and dilution. The PEA is based on a 1,500 t/d underground bulk mining operation, using ramp access. The main mining method considered is mechanized, longhole stoping with truck haulage and stope backfill using waste rock and process tailings.

1.6 METALLURGICAL TESTWORK AND PROCESSING

1.6.1 Metallurgical Testwork

Huakan has recently initiated metallurgical testing on the samples obtained from the 830 Drift on the J&L Property. This testwork includes heavy media separation (HMS), grinding, and several stages of flotation and pressure oxidation and bioleaching on a portion of the flotation products prior to cyanidation.

The results of Huakan's current testwork program were not available at the time this report was prepared. The PEA relies, therefore, on the results of extensive previous programs. The general thrust of the previous testwork has been to separate the base metals from the ore by selective flotation, and then recover the precious metals from the tails by some combination of additional flotation and hydrometallurgical processing (such as oxidation and cyanidation).

The majority of the testwork to date focused on producing saleable lead and zinc concentrates by various flotation separation techniques. In addition, some tests have been carried out on the refractory arsenopyrite concentrates. These have included Cashman (chloride leach), Redox (nitrate leach), batch roasting, pressure oxidation and biooxidation, followed by cyanidation, to recover the precious metals.

1.6.2 Recovery Methods

Two process flowsheet options were considered for this PEA and each includes an HMS circuit ahead of milling to remove 40% of the rock by weight, with estimated metal losses of 2%. The main difference between the two proposed options is whether a single lead concentrate or two concentrates, lead and zinc are produced for sale.

Each option involves a similar set of processing steps including crushing, HMS, grinding, flotation, oxidation, and cyanidation. The main source of assay data for this report is a study dated November 2, 2006, prepared on behalf of BacTech Mining Corporation (BacTech) by Process Research Associates Ltd. (PRA). This study used the closest flowsheet arrangement to the recommended program today, and was conducted on samples which had been pre-concentrated using heavy liquid separation similar to earlier flotation testwork conducted in 1998. The conceptual plant design for the single saleable concentrate is based on the results of testwork carried out by PRA on mineralization making up a composite sample with head grades of 8.1g/t Au, 72.4 g/t Ag, 3.2% Pb, and 4.8% Zn. For this PEA, Micon has estimated the life-of-mine (LOM) average feed grades as follows: 6.42 g/t Au, 69.8 g/t Au, 2.2% Pb, and 3.6% Zn.

Only a summary of the PRA report was available through Huakan. Thus, additional process parameters (including suitable HMS density, reagent consumption, et cetera) have been identified from a report dated October, 1998 report prepared by H.A. Simons (Simons). The Simons report was based on testwork conducted by PRA under the direction of Beattie Consulting Ltd.

1.7 INFRASTRUCTURE

There is a fully functional 40-man camp as well as a large shop and office facility located at the property. A preliminary review has been carried out of infrastructure required for the J&L Main Zone project, and cost allowances have been allocated. Existing buildings and facilities will be upgraded, as well as services, including water supply, electrical power supply and main substation upgrades. A more detailed investigation of the existing infrastructure is recommended to be carried out during the next stages of project development.

1.8 ENVIRONMENTAL STUDIES

Previous operators in the 1980's and early 1990's collected some background environmental data including water samples. Environmental baseline studies were initiated in 2007 by Merit/Huakan for the J&L Property, including meteorology, water quality, waste characterization, vegetation, wildlife, and fisheries studies.

Previous exploration activities generated PAG waste rock which is currently stored on a lined pad near the portal and being managed following provincial requirements. Ongoing costs with managing this material have not been included in the financial model.

The proposed project is expected to require a provincial environmental review under the British Columbia Environmental Assessment Act and a Screening level assessment under the Canadian Environmental Assessment Act. For economic scheduling purposes, this was estimated to take two years.

The J&L project falls within three First Nation territories, the Okanagan, Shuswap and Ktunaxa Nations. Huakan has initiated consultation in conjunction with exploration activities and it is understood that these activities will continue as the project moves forward.

Micon understands that all exploration permits are in place and maintained in compliance. Following good industry practice, it is assumed a social, environmental, health and safety management system will be implemented to meet the company's commitments to protect the environment and the health and safety of the workers and the surrounding communities. The management system and plans will monitor and maintain permit compliance and a social licence to operate.

1.9 CAPITAL COST ESTIMATE

A summary of the estimated pre-production capital costs are provided in Table 1.2.

Table 1.2
Summary of Project Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Mining	9,178	19,176	28,354
Process	97,490	9,451	106,941
Infrastructure	51,290	-	51,290
Indirects	18,714	-	18,714
EPCM	22,317	-	22,317
Contingency	49,747	-	49,747
Owner's	14,931	-	14,931
Closure	-	6,500	6,500
Total	**263,668**	**35,126**	**298,794**

The total estimated project life-of-mine capital expenditure amounts to $299 million comprised $264 million pre-production capital and $35 million sustaining capital, including closure costs.

1.10 OPERATING COST ESTIMATE

A summary of the LOM unit operating costs is presented in Table 1.3.

Table 1.3
LOM Average Operating Costs

Area	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Mining	287,307	58.67	313.93
Process	216,903	44.30	237.00
General and Administration	50,441	10.30	55.11
Total	**554,651**	**113.27**	**606.04**

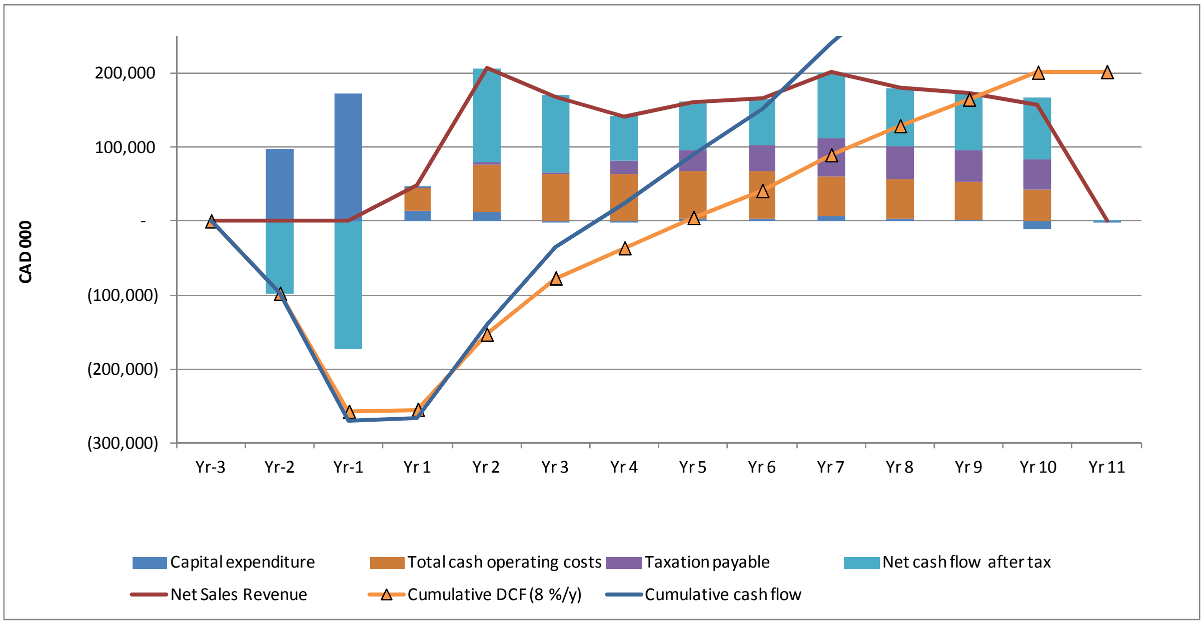
1.11 ECONOMIC ANALYSIS

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. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The base case evaluates to an IRR of 26% before tax and 21% after tax. At the selected discount rate of 8%, the net present value (NPV_8) of the cash flow is $344 million before tax and $202 million after tax.

Annual cash flows are presented in Figure 1.1.

Figure 1.1
Life-of-Mine Cash Flows



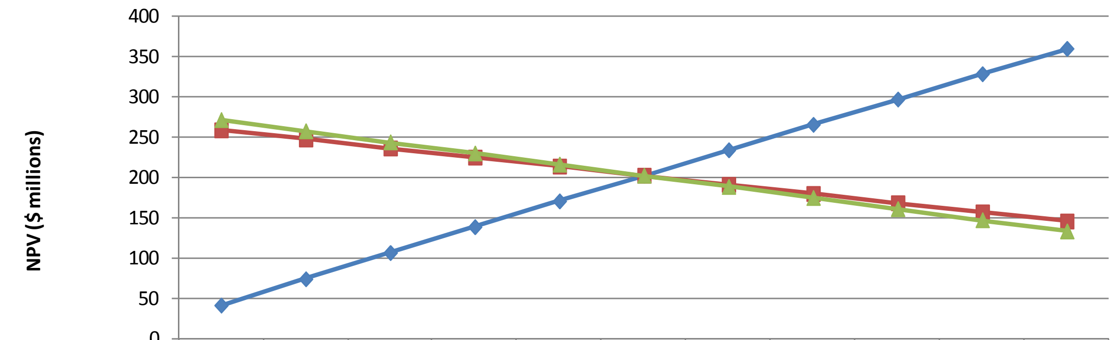
1.11.1 Sensitivity Analysis

The sensitivity of the project returns to changes in all revenue factors (including head grades, recoveries, prices and exchange rate assumptions), together with capital and operating costs, was tested over a range of 25% above and below base case values. Figure 1.2 shows the results of this analysis.

The results show that the project is most sensitive to revenue factors, with an adverse change of 25% reducing NPV_8 to \$41 million. The project is less sensitive to capital costs, with an adverse change of 25% required to reduce NPV_8 to \$133 million. Operating costs have the least impact on returns, with a 25% increase in cost reducing NPV_8 to \$145 million.

In further analysis, Micon notes that simultaneously applying an increase of more than 40% to both capital and operating costs simultaneously will be required in order to reduce NPV_8 to zero.

Figure 1.2
Sensitivity to Capital, Operating Costs and Revenue



	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Revenue drivers	41	73	106	138	170	202	233	265	296	328	359
Operating costs	258	247	236	224	213	202	190	179	168	156	145
Capital costs	271	257	243	229	216	202	188	174	160	147	133

1.12 RECOMMENDATIONS

Following the completion of the positive PEA study, it is recommended that Huakan advance the development of the J&L Property to the pre-feasibility stage and that baseline data collection and community and First Nations consultation continue in order to support potential permitting.

Huakan has carried out Phase I of its work on the J&L deposit. It is estimated that to complete a pre-feasibility study on the J&L deposit, including drilling, metallurgical testing, and engineering design will cost in the range of $24 million, spread over a further two phases. The next phase, Phase II, of the work program is estimated to cost $4.4 million and consist of 250 m of drifting, drill bays and 40 drillholes. Phase III is expected to cost $19.6 million and consist of 2,180 m of drifting, drill bay development and 47 new drillholes. Although no detailed budget has been completed for Phase III, this cost estimate is derived from industry experience with similar scale projects. A budget for Phase II has been compiled by Huakan and is shown in Table 1.4. Micon believes that both estimates are reasonable.

Table 1.4
J&L Phase II Development and Drilling Tracking Budget

Item	Cost ($)
Environmental Studies	40,000
First Nations Consultation	30,000
PAG Waste Pad Construction	54,000
Portal Shotcreting	55,300
Capital Expenditures	261,000
General Parts and Inventory	50,000
Rehabilitation/Slashing Underground Workings	406,800

Item	Cost ($)
Underground Development	935,200
Equipment Rental	20,000
Underground Diamond Drilling Program	1,072,000
Updating of Resource	15,000
Subtotal	**2,939,300**
Camp Costs	
Generator Fuel and Maintenance	560,700
Management/Supervision	105,000
Camp costs and water treatment	277,000
Safety Equipment and Training	26,000
Snow Removal and Routine Road Maintenance	92,000
Subtotal	**1,060,700**
Contingency	400,000
Total	**4,400,000**

2.0 INTRODUCTION

The J&L Property is located 35 km north of Revelstoke, British Columbia, Canada. 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 preliminary economic assessment (PEA) on the potential production of gold, silver, lead, and zinc from the J&L Main Zone deposit.

Huakan (as Merit) entered into an option agreement dated April 13, 2007, whereby could may acquire a 100% undivided interest in the J&L Property in consideration for share issuances and cash payments totalling $10.79 million over a seven-year period. In August, 2010, the company exercised the option by advancing the cash and share issuances to acquire a 100% undivided interest in the J&L Property. There are no NSR royalties owing to any parties.

A press release was issued by Huakan on April 24, 2012, announcing the results of this PEA. This release can be accessed from Huakan's web site at www.huakanmining.com.

2.1 TERMS OF REFERENCE

The J&L Property was the subject of a previous independent Technical Report entitled Technical Report and Resource Estimate, J&L Property, Revelstoke, British Columbia, Canada, by Fred Brown et al., of P&E Mining Consultants Inc. (P&E), dated June 23, 2011 (Brown, et al., 2011) and filed on the System for Electronic Document Analysis and Retrieval (SEDAR) on June 23, 2011.

An earlier independent Technical Report was prepared in May 2007 by David K. Makepeace, P.Eng., titled, J&L Property Technical (43-101) Report for Merit Mining Corp., (Makepeace, 2007).

The PEA study is based on the following:

- The most recent mineral resource estimate for the J&L Main Zone deposit, prepared by Fred Brown, CPG, P&E Mining Consultants Inc.

- A geomechanical assessment of the underground workings by Klohn Crippen Berger Limited (KCBL).

- An underground mine design, schedule, and cost estimate prepared by Catherine A. Dreesbach, P.E., Senior Mining Engineer with Micon.

- Interpretation of metallurgical testwork and hydrometallurgical aspects of mineral processing design by David Jones, Ph.D., Senior Associate Metallurgist with Micon.

- Interpretation of metallurgical testwork, process flowsheet, plant design, and cost estimate prepared by Bogdan Damjanović, P.Eng., Senior Metallurgist with Micon.

- Review of environmental considerations by Jenifer L. Hill, R.P.Bio., Senior Environmental Scientist with Micon.

- Financial model prepared by Christopher Jacobs, CEng, MIMMM., Project Manager and Vice President at Micon.

2.2 QUALIFIED PERSONS

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

- Tracy J. Armstrong, P.Geo.
- Fred Brown, CPG, Pr.Sci.Nat.
- Bogdan Damjanović, P.Eng.
- Catherine A. Dreesbach, P.E.
- Christopher Jacobs, CEng, MIMMM

Bogdan Damjanović, Catherine Dreesbach, and Christopher Jacobs visited the J&L Property on October 27, 2011. Fred Brown visited the J&L Property on December 17, 2010.

2.3 USE OF REPORT

This report is intended to be used by Huakan subject to the terms and conditions of its agreement with Micon. Huakan may 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.

2.4 UNITS, CURRENCY, AND ABBREVIATIONS

Unless otherwise stated all units used in this report are metric. Gold and silver assay values are reported in grams per tonne (g/t) unless ounces per tonne (oz/t) are specifically stated. Base metal assay values are given in percent (%). The Canadian dollar (CAD or $) is used throughout this report except in reference to metals prices which are given in United States dollars (US$). For the purposes of this report 1 CAD ($) = 1 US$.

The coordinate system used for the project grid is based upon a UTM grid coordinate system (NAD 83- Zone 11).

In this document, in addition to the definitions contained heretofore and hereinafter, unless the context otherwise requires, the following terms have the meanings set forth in Table 2.1.

Table 2.1
List of Abbreviations

Term	Abbreviation
Above Mean Sea Level	amsl
Acceleration Due to Gravity	g
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Adsorption, Desorption, Recovery	ADR
Arsenic	As
Atomic Absorption	AA
BQ Thin Wall (core size)	BQTW
British Columbia	BC
Calcium Carbonate	$CaCO_3$
Canadian Dollars	$, CAD
Canadian Dollars Per Tonne	$/t
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian National Instrument 43-101	NI 43-101
Canadian Securities Administrators	CSA
Capital Asset Pricing Model	CAPM
Carbon-in-Pulp	CIP
Carbon-in-Leach	CIL
CDN Resource Laboratories	CDN
Centimetre(s)	cm
Centimetres per year	cm/y
Cheni Gold Mines Ltd.	CGM
Copper	Cu
Cubic Metres	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
Days Per Year	d/y
Degree(s)	°
Degrees Celsius	°C
Diamond Drill Hole	DDH
Discounted Cash Flow	DCF
East	E
Electro-magnetic	EM
Foot(feet)	ft
General and Administration	G&A
Gold	Au
Gram(s)	g
Grams Per Cubic Centimetre	g/cm^3
Grams Per Litre	g/L

Term	Abbreviation
Grams Per Tonne	g/t
Heavy Media Separation	HMS
Hectare(s)	ha
Hour	h
Huakan International Mining Inc.	Huakan
Internal Rate of Return	IRR
Inverse Distance Squared	ID^2
Iron	Fe
Kilogram(s)	kg
Kilograms Per Square Metre Per Hour	$kg/m^2/h$
Kilometre(s)	km
Kilopascal(s)	kPa
Kilovolt Ampere(s)	kVA
Kilowatt-Hour	kWh
Kilowatt-Hour Per Tonne	kWh/t
Klohn Crippen Berger Limited	KCBL
Lead	Pb
Life-of-mine	LOM
Megawatt(s)	MW
Megawatthours per year	MWh/y
Merit Mining Corp.	Merit
Metal Leaching	ML
Metre(s)	m
Metres Above Mean Sea Level	m amsl
Metres Per Day	m/d
Methyl Isobutyl Carbinol	MIBC
Metric Tonnes Per Day	t/d
Micon International Limited	Micon
Micrometre(s), Micron(s)	µm
Milligrams Per Liter	mg/L
Millilitre(s)	mL
Millimetre(s)	mm
Million Cubic Metres	Mm^3
Million tonnes	Mt
Minute	min
Nearest Neighbour	NN
Net Present Value	NPV
Net Present Value at 8% per Year Discount Rate	NPV_8
Net Smelter Return	NSR
North	N
North American Datum	NAD
Oxygen	O_2
P&E Mining Consultants Inc.	P&E
Potentially Acid Generating	PAG
Pound	lb
Pounds Per Square Inch	PSI
Preliminary Economic Assessment	PEA
Pressure Oxidation	POX
Process Research Associates Ltd.	PRA
Qualified Person	QP
Quality Assurance/Quality Control	QA/QC

Term	Abbreviation
Run-of-Mine	ROM
Silver	Ag
Specific Gravity	SG
Square Metre(s)	m^2
Sulphur Dioxide	SO_2
Thousands	000s
Tonne(s)	t
Tonnes Per Cubic Metre	t/m^3
Tonnes Per Day	t/d
Tonnes Per Day Per Square Metre	$t/d/m^2$
Tonnes Per Hour	t/h
Tonnes Per Year	t/y
Treatment Charges/Refining Charges	TC/RC
Universal Transverse Mercator	UTM
Volt(s)	V
West	W
Zinc	Zn

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 J&L Property, other than examining/verifying mineralization on drill cores and visually examining the existing underground workings. Check-sampling and analysis was carried out by Fred Brown as part of his site visit in December, 2010 as described in Section 11.0. While exercising all reasonable diligence in checking, confirming and testing it, the authors of this report have relied upon Huakan's presentation of data for the J&L Property and the findings of its consultants in formulating their opinion.

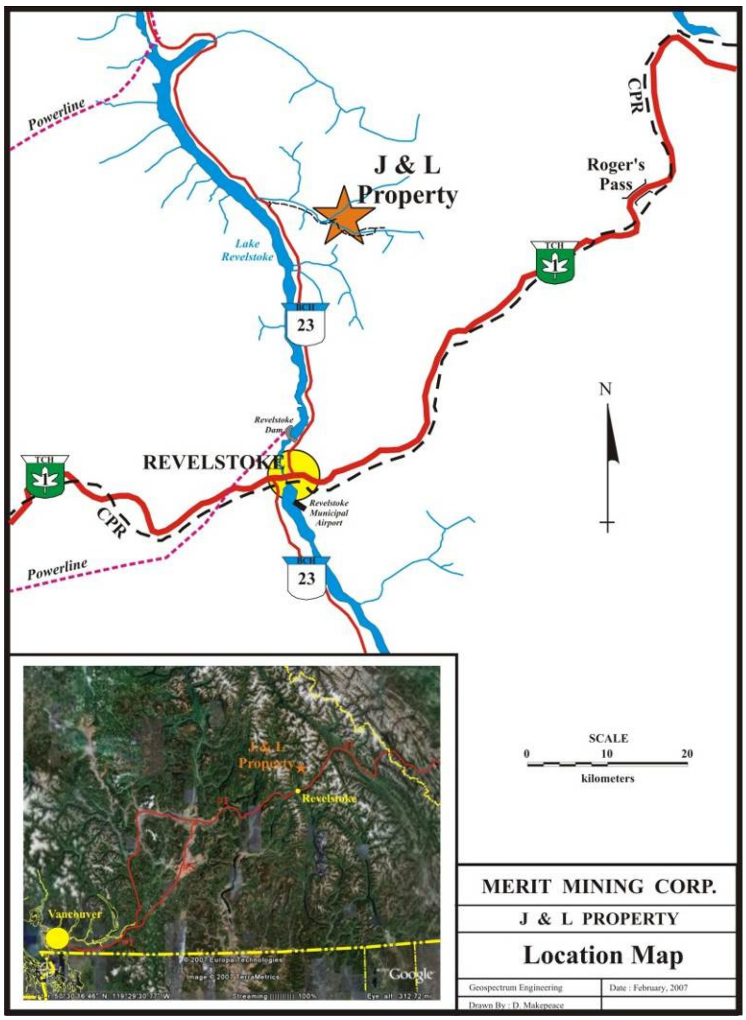
The status of the mining claims under which Huakan holds title to the mineral rights for the J&L Property has not been investigated or confirmed by the Micon, and Micon offers no legal opinion as to the validity of the mineral titles claimed. The description of the J&L Property, and ownership thereof, as set out in this report, is provided for general information purposes only. The existing environmental conditions, liabilities and remediation have been described where required by NI 43-101 regulations. These statements also are provided for information purposes only and Micon offers no opinion in this regard.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The J&L Property is located in southeastern British Columbia, approximately 35 km northeast of Revelstoke, BC. The Property is within the 082M-030 NTS map sheet. Most of the exploration activity to-date is centered at latitude 51° 17’ N, longitude 118° 08’ W (5681943 m N, 420960 m E, UTM NAD 83) (see Figure 4.1).

Figure 4.1
Regional Location Map



4.2 CLAIMS

There are two types of contiguous claims making up the J&L Property: 20 mineral claims and 10 crown granted claims. These mineral claims would cover 3,051.73 ha if there were no overlap of claims; however, there is some overlap. The mineral claims cover approximately

2,887.68 ha and the crown granted claims cover an additional 164.05 ha. The mineral claims are listed in Table 4.1, the crown grant claims in Table 4.2 and both are illustrated in Figure 4.2.

Table 4.1
J&L Mineral Claims

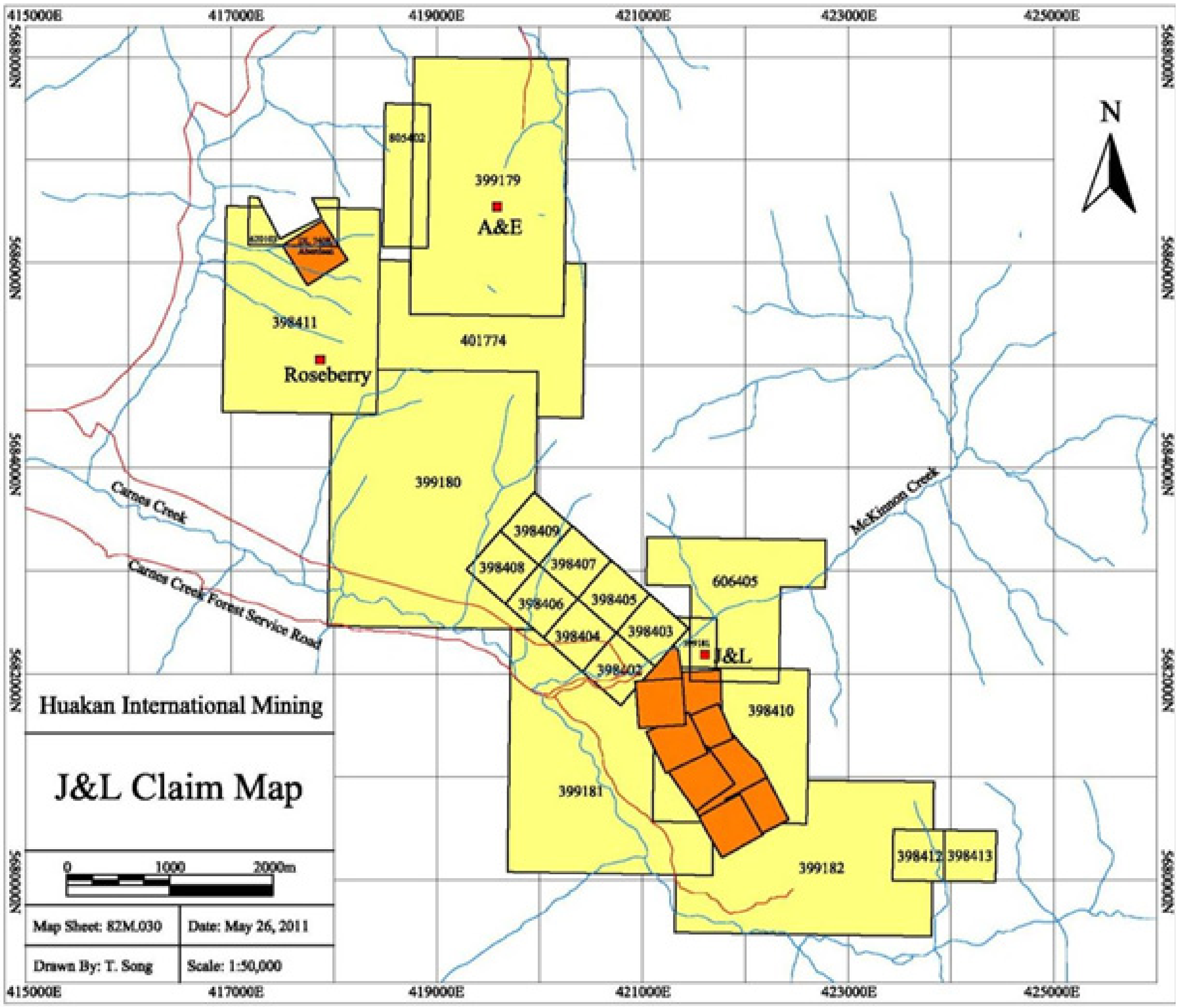
Tenure No.	Claim Name	Expiration Date	Mining Division
398402	J1	15/11/2017	Revelstoke
398403	J2	15/11/2017	Revelstoke
398404	J3	15/11/2017	Revelstoke
398405	J4	15/11/2017	Revelstoke
398406	J5	15/11/2017	Revelstoke
398407	J6	15/11/2017	Revelstoke
398408	J7	15/11/2017	Revelstoke
398409	J8	15/11/2017	Revelstoke
398410	J9	15/11/2017	Revelstoke
398411	J10	15/11/2017	Revelstoke
398412	J11	15/11/2017	Revelstoke
398413	J12	15/11/2017	Revelstoke
399179	Sage	15/11/2017	Revelstoke
399180	J13	15/11/2017	Revelstoke
399181	J14	15/11/2017	Revelstoke
399182	J15	15/11/2017	Revelstoke
401774	Brush	15/11/2017	Revelstoke
606405	Yellow Jacket	30/06/2012	Revelstoke
620103	Hardpan	30/06/2012	Revelstoke
805402	A & E - W	30/06/2012	Revelstoke

Claim status as of May 30, 2012.

Table 4.2
J&L Crown Granted Claims

Claim Number	Claim Name	Mining Division
L 14821	Goat Fraction	Revelstoke
L 14822	Goat No. 2 Fraction	Revelstoke
L14823	Goat No. 3 Fraction	Revelstoke
L 14824	Goat No. 4 Fraction	Revelstoke
L 14825	Goat No. 5 Fraction	Revelstoke
L 14826	Goat No. 6 Fraction	Revelstoke
L 14827	View Fraction	Revelstoke
L 14828	View No.2 Fraction	Revelstoke
L 14829	Creek Fraction	Revelstoke
L7408	Aberdeen	Revelstoke

Figure 4.2
Claim Map, J&L Property



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

Vehicle access to the area is via Provincial Highway 23, 35 km north of the town of Revelstoke, where Highway 23 intercepts the Carnes Creek Forest Service Road. The Property is then reached by travelling eastward 13 km along the Carnes Creek Forest Service Road before reaching the J&L mine camp. Travel time to camp is approximately 45 to 60 minutes from Revelstoke. The Forest Service Road is radio-controlled, but currently is not being used for logging activities. Due to lack of activity by logging companies, road maintenance and winter snow ploughing is undertaken wholly by Huakan. Helicopter access from Revelstoke takes approximately 15 minutes.

Road access within the property is via four-wheel drive or tracked vehicle. Several overgrown trails access the majority of the workings on the property but are slow and difficult.

5.2 CLIMATE

The summer weather is considered moderate with average temperatures between 16° to 30°C, with long stretches of sun and rain. The rain at times can be very heavy. The average precipitation is 65 cm/y. Winters are long and are characterized by heavy snowfalls (1 to 4 m) with cool temperatures (-15°C to +5°C). Snowfall typically occurs between October and May at higher elevations and between November and April at lower elevations.

Exploration and mining activities may be conducted year-round.

5.3 LOCAL RESOURCES

There is a large, skilled workforce of trades and technical professionals as well as equipment suppliers available throughout the region. The economy of Revelstoke is dependent on four primary sectors, forestry, tourism, transportation (mainly CP Rail) and public services.

5.4 INFRASTRUCTURE

The property has several adits and numerous trenches. Only two adits are accessible but are currently locked for safety requirements (830 Adit and 832 Adit). There is a fully functional 40-man camp as well as a large shop and office facility located in the immediate vicinity of the portal to the 832 Adit. Electric power is produced by on-site diesel generators and a satellite phone and internet system is in place. Selkirk Helicopters occasionally uses the Property as a re-fueling station for their operations. A skid-mounted fuel tank is located about 200 m from the 832 portal on the Forest Service Road.

The nearest population centre is the town of Revelstoke (population approx. 8,500), which is located approximately 35 km to the south of the Property. Revelstoke lies on the Trans-Canada Highway. The Canadian Pacific Railway runs through the town and a rail siding and load-out facility (owned by Huakan) is present on the eastern end of the town. A short asphalt airstrip on the south side of town can accommodate small charter planes and helicopters.

The Revelstoke Dam on the Columbia River is located 2 km north of Revelstoke and produces power for a large portion of British Columbia. There are no power lines running along Highway 23, although there is an underground telephone line.

There is a helicopter-accessible ski chalet located 5 km east of the J&L Property on the lower portion of the Durrand Glacier. It is used for heli-skiing in the winter and alpine hiking in the summer.

5.5 PHYSIOGRAPHY

The topography is characteristic of the Selkirk Mountains. The elevation ranges from 700 to 3,050 m amsl. The topographic relief is a result of recent alpine glaciation. Incised creeks such as McKinnon Creek created narrow valley floors while major creeks (e.g., Carnes Creek) exhibit a broader U-shaped appearance with potentially deep valley-bottom overburden. The talus covered slopes are steep, ranging from 28° to 40° while bedrock slopes range up to near vertical, depending on the lithology. All of these conditions make traversing the property hazardous and time-consuming.

Numerous avalanche chutes occur in the area. An avalanche chute occurs beside the 830 Adit portal which prompted the driving of the 832 trackless Drift. The 832 Drift allows safe year-round access to the underground network. Flat ground is limited on the property, but appears to be sufficient enough for a millsite, tailings and waste rock storage.

The main watercourse on the Property is Carnes Creek, which transects the area. Its source is the Durrand Glacier, which is east of the Property. McKinnon Creek is a tributary of Carnes Creek and is a more juvenile watercourse in which the flow volume can change rapidly. The area surrounding the intersection of McKinnon and Carnes Creeks has been the focus of the majority of the work over the life of the property.

Vegetation on the property changes from alder, devil's club, stinging nettles and deadfalls in the valley floor, through stands of cedar, hemlock and minor fir on the mountainsides, to sub-alpine to alpine plants at approximately 1,980 m elevation. The Carnes and Tumbledown Glaciers are immediately east of the Property boundary.

6.0 HISTORY

6.1 EARLY REGIONAL HISTORY

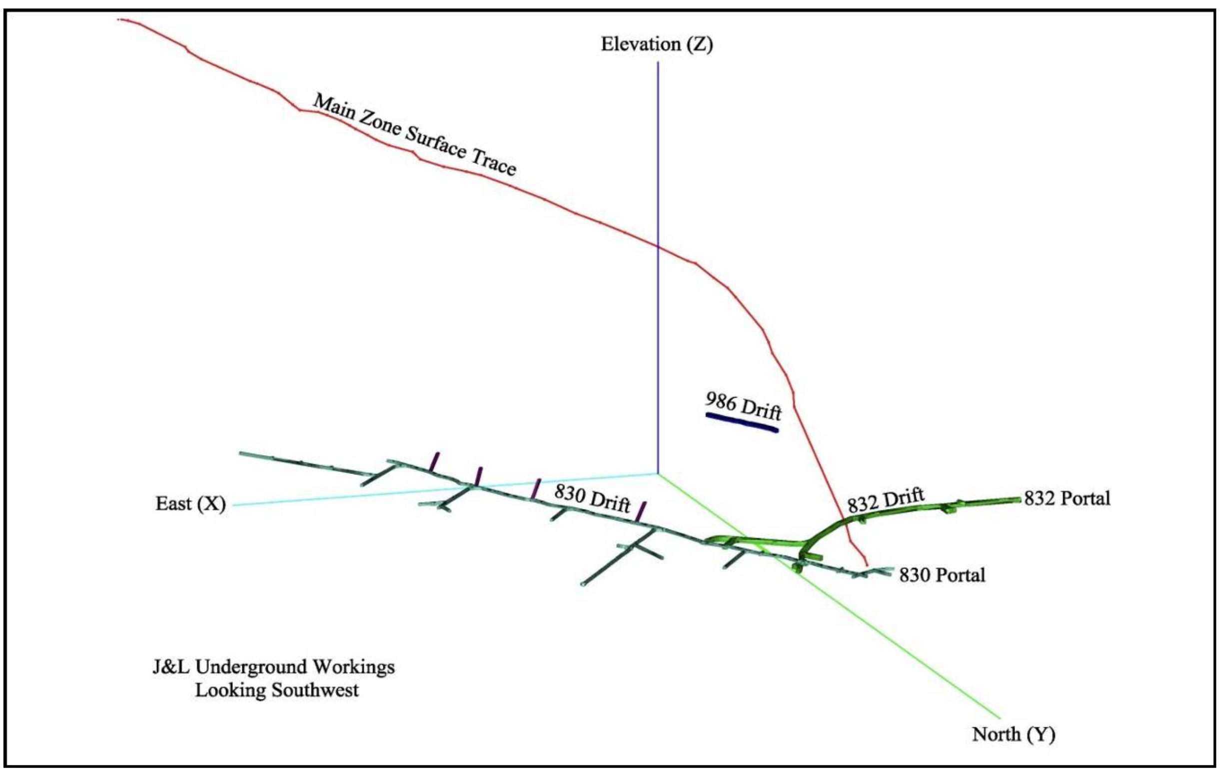
A summary of property activities is presented in Table 6.1. A schematic of the underground workings is presented in Figure 6.1 (not to scale).

Table 6.1
Summary of Historical Exploration

Year	Company	Exploration
1912	E. McBean	Collaring of the 986 level portal (91 m long) and 2 shallow shafts (each 46 m deep).
1924	E. McBean	Metallurgical tests were attempting to resolve problems due to the high arsenic content of the ore.
1924-1927	Porcupine Goldfields Development and Finance Company	43 m of underground drifting on two levels. In 1925, E McBean excavated 30 trenches and pits along the surface trace of the Main Zone on Goat Mountain. In 1926, 26 kilograms of Main Zone mineralized rock were shipped to the Department of Mines in Ottawa for metallurgical testing.
1928	Geological Survey of Canada	J&L area mapped under the direction of Dr. H. Gunning.
1934	T. Arnold	Crown-granted claims acquired.
1935	Raindor Gold Mines	Property optioned, 986 Level Adit extended a further 152 m on the Main Zone.
1946	Raindor Gold Mines	2 shafts deepened, to a total 117 m.
1952	Asarco	Several trenches were completed on the Main Zone.
1962-1967	Westair Mines Ltd.	The J&L, A&E and Roseberry prospects were optioned in 1962. 183 m of underground drilling completed
1965	Westair Mines Ltd.	A new portal (830 Level Adit) was collared to explore the Main Zone. Total length was 297 m. It is one of the major underground facilities on the Property. A road (12.4 km) was built into the property from the Big Bend road (now Highway 23).
1980	Pan American Minerals	Property leased from T. Arnold.
1981-1985	BP Minerals, Selco Division	Surface exploration program. 830 Level (Tracked) Adit extended an additional 1,333 m of drift and cross cuts. 64 underground drill holes advanced for 2,640 m.
1986-1987	Noranda Mines Ltd.	Completed metallurgical studies on the Main Zone.
1988	Pan American Minerals	830 Level (tracked) Adit extended an additional 250 m of drift and cross cuts and completed 4 raises totalling 120 m.
1988-1989	Equinox Resources Ltd.	Optioned property from Pan American. Advanced 32 underground drill holes totalling 2,985 m. A 270-t bulk sample from 3 "Take-Down-Backs" was collected for metallurgical studies.
1991	Cheni Gold Mines	Joins the joint-venture group with the discovery of the Yellowjacket deposit from 32 surface drill holes. Deposit lies in the hanging wall of the Main Zone. A new, trackless, 832 Adit (3.0 . x 3.5 m) that ran 170 m was collared.
1991	Equinox Resources Limited	Resource estimate calculated.
1996	Weymin Mining Corporation	3 surface boreholes were advanced for 503 m of drilling. A 120-t underground bulk sample was retrieved from the 830 Level for metallurgical studies from six sample locations.

Year	Company	Exploration
1996	H.A. Simons	Weymin Mining Corp. commissioned 2 detailed reports: Technical Review of the J&L Property and Project Opportunities for the J&L Property.
1998	H.A.Simons	Completed McKinnon Creek Property Scoping Study.
2004	BacTech Mining Corporation	Further metallurgical tests, engineering and environmental studies were conducted. A minor drilling program (2-3) boreholes was carried out. Due to the financial collapse of BacTech, the drilling details are not available.
2007	Merit Mining Corp.	Option to earn a 100% interest in the J&L property was acquired on April 13, 2007. A $10.8 million work plan approved for 2008. 40-man camp was installed and a shop/mine dry complex was completed and mining equipment was procured. 9 surface boreholes advanced totalling 1,363 m of drilling.
2008	Merit Mining Corp.	Rehabilitation of 832 drift was extended a further 550 m with the 5 by 5 m profile (to allow for 30 ton trucks) and connected to the 830 track Drift, approximately 310 m in from the 830 Adit portal. Tunnelling completed by September 2008 at which time the program was suspended due to financial constraints and a downturn in world metal prices.
2010	Merit Mining Corp.	Merit completed payments to earn 100% undivided interest in the J&L property. Mining activities resume to generate a NI-43-101 compliant resource.
2010	Huakan International Mining Inc.	Merit changed name to Huakan in December, 2010.

Figure 6.1
Underground Workings on the J&L Property



P&E, 2011 (not to scale).

6.2 HISTORIC DRILLING

Drilling on the J&L Property began in 1962 and various companies have undertaken drilling campaigns in the years since. A summary of historic drill programs is presented in Table 6.2.

Table 6.2
Summary of the Historic Drill Programs on the J&L Deposit

Year	Drillholes	Total (m)	Company
1962 to-1967	Underground	183	Westairs Mines Ltd.
1983 to 1984	65 underground drill holes	2,640	BP Selco Ltd.
1987 to 1988	20 underground drill holes	1,914	Pan American Minerals
1988 to 1989	32 underground drill holes	2,985	Equinox Resources Ltd.
1990 to 1991	50 underground drill holes 27 surface drill holes	13,889	Equinox Resources Ltd/ Cheni Gold Mines Ltd.
1997	3 underground drill holes	503	Weymin Mining Corp.
2006	2-3 underground holes	undisclosed	BacTech Mining Corp.
2007	9 surface drill holes	1,363	Merit Mining Corp.
2010-2011	60 underground drill holes	7,897	Merit/Huakan International Mining Inc

6.3 PREVIOUS RESOURCE ESTIMATES

An historic mineral resource estimate was prepared by Equinox Resources Ltd. in 1991 for the Main Zone; however, it is not discussed further since the resource could not be validated and has been superseded by an NI 43-101 compliant resource. The P&E mineral resource estimate, as of the date of this report, is considered to be the only current and valid estimate for the J&L Main Zone deposit that is verified by a Qualified Person.

In 1991, Equinox Resources Limited completed a historical resource estimate for the J&L deposit as shown in Table 6.3.

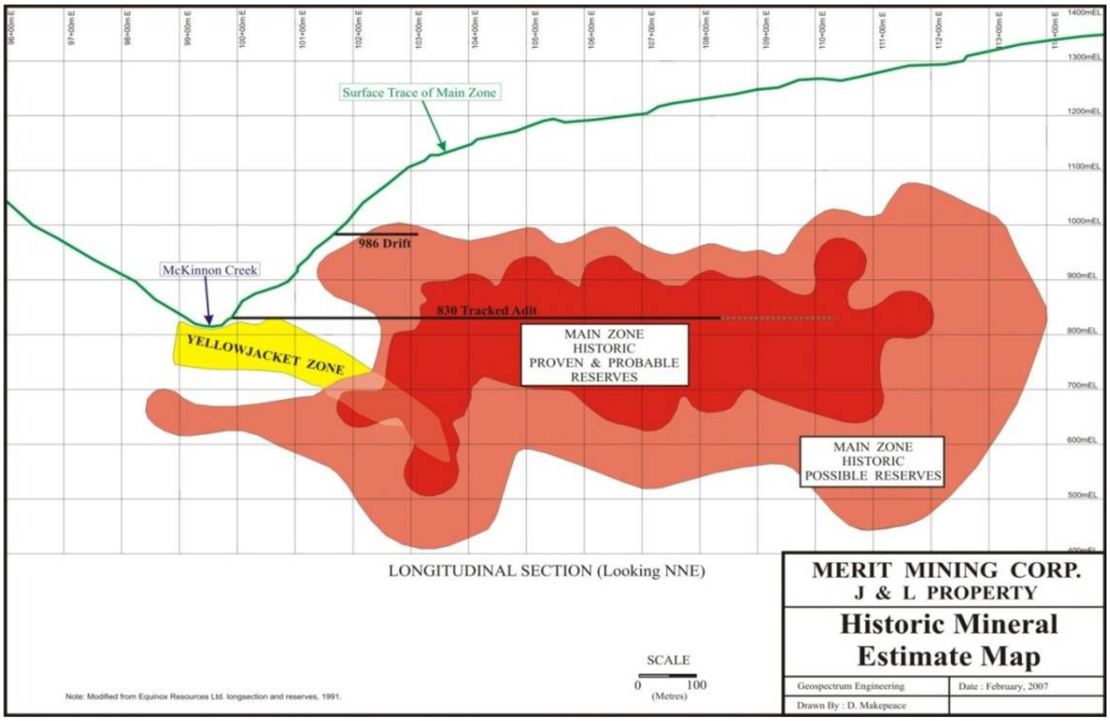
Table 6.3
Summary of the Historical Resource Estimates for the J&L Deposit

Zone	Type	Tonnage and Grade
Main Zone	Proven & Probable Ore Reserves	1.7 Mt grading 7.38 g Au/t, 75.9 g Ag/t, 2.64% Pb, 4.43% Zn.
Main Zone	Possible Ore Reserves	1.9 Mt grading 7.12 g Au/t, 85.5 g Ag/t, 3.32% Pb, 3.48% Zn.
Yellowjacket Zone	Probable Ore Reserves	693,000 t grading 52.3 g Ag/t, 2.45% Pb, 7.06% Zn.
Yellowjacket Zone	Possible Ore Reserves	337,000 t grading 53.1 g Ag/t, 2.50% Pb, 7.15% Zn.

The reader is cautioned that the above listed Resource and Reserve Estimates are dated prior to 2001 and are not compliant with the reporting requirements of NI 43-101. Estimates for the Main Zone have been superseded by the 2011 P&E resource estimate for the J&L Property dated 23 June, 2011, as described in Section 16 of this report.

In 2007, David Makepeace, P.Eng., authored a technical report on the J&L Property in which a mineral estimate map was created using the historic data to depict the non-compliant historic mineral resource (see Figure 6.2). Makepeace used a standard circular polygonal calculation, weighted average method for both zones.

Figure 6.2
Historical Mineral Estimate Map for the J&L Property



The 1998 scoping study by H.A. Simons provided analyses of six cases, exclusively on the Main Zone (i.e., the Yellowjacket Zone was not analyzed). The two favoured cases were:

- Base case at 1,000 t/d with all processing at McKinnon Creek;

- Base case at 1,500 t/d, grind, float and pressure oxidize at Goldstream, required a 4 Mt deposit.

The capital costs for the above scenarios above were $81.7 million and $115 million, respectively. At metal prices of US$ 350/oz Au, US$6/oz Ag, US$ 0.55/lb Zn and US$0.30/lb Pb, and a Canadian to United States dollar exchange rate of 0.70, the following key economic analyses for the two scenarios were:

- IRR of 13.8 and 18.0 respectively;
- Net Cash Flow of $75.7 million and $103.8 million, respectively;
- NPV @ 5% Discount Rate of $36.0m and $58.7m, respectively;
- Operating costs/tonne of $87/tonne and $64/tonne, respectively;
- Operating cost/oz Au Equivalent Recovered of $242/oz and $180/oz, respectively.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGY

The J&L Property is underlain by north to northwest striking, moderate to steeply east dipping metasediments and metavolcanic rocks of the Hamill and Lardeau Group and Badshot and Mohican Formation rocks. These units consist, for the most part, of sheared to intensely folded impure quartzites, quartz sericite to sericite to chlorite schists and phyllites, and grey banded to carbonaceous limestones.

A stratigraphic column displaying the age relationships of units is presented below and detailed in Figure 7.1.

Stratigraphic Column (after Logan, et al., 1996)

Lower Paleozoic
Lardeau Group
Jowett Formation - interlayered metavolcanic and non-carbonaceous marble
Micaceous Quartzite Unit
Index Formation - Greenstone and Black Phyllite)

Lower Cambrian
Badshot Formation - limestone/marble
Mohican Formation (quartzite, phyllite)

Neoproterozoic – Lower Cambrian

Hamill Group (quartzite, micaceous quartzite, phyllite)

7.1.1 Hamill Group

The Hamill Group rocks are predominantly interbedded medium brown to green-black sericitic and/or chloritic quartzite and phyllite with minor layers of argillite and graphite. This unit appears as the upper Hamill unit described by Logan et al. (1996), and is probably Lower Cambrian in age. Hamill group rocks form part of the footwall and hanging wall of the Main Zone deposit. The unit has a gradational upper contact with the Mohican/Badshot Formations.

7.1.2 Mohican Formation

The Mohican Formation is Lower Cambrian in age (Fritz et al., 1991). This unit is located at the eastern and southern boundary of the property. The eastern unit is in the hanging wall of the Main Zone. It is characterized as limonite-rich, sericitic chloritic calcareous phyllite and quartzite interlayered with narrow layers of marble. Logan describes the Mohican as a "transition between quartz-rich sediments of the Hamill Group and the carbonate-rich rocks of the Badshot Formation" (Logan et al., 1997A).

7.1.3 Badshot Formation

The Badshot Formation is the most visible and distinctive lithologic unit within the claims. It is Lower Cambrian in age. This white to grey, fine to medium-grained limestone/dolomite/marble varies in its silica content. The Yellowjacket Zone is totally contained within this unit. The higher silica content of the Yellowjacket Zone appears to be alteration specific to the Yellowjacket mineralizing system. The Main Zone crosscuts the Badshot Formation as observed in the 830 tracked Drift. Several diamond drill holes display elevated grades and widths where the Main Zone cross-cuts the Badshot Formation. Thin interlayers of black graphite are seen within the Badshot in the 832 Level portal.

7.1.4 Index Formation

The Index Formation can be subdivided into at least four units (i.e. black phyllite, marble, greenstone and quartz breccia), but only two have been identified on the Property.

The black phyllite unit is in the footwall of the Main Zone. Logan has also traced the unit in the northern portions of the claims around the A&E showings (see Section 7.5). The unit can be calcareous and graphitic and may contain minor marble and quartzite layers.

The greenstone units within the Property occur as a series of diorite sills. The diorite is composed predominantly of coarse-grained chlorite and plagioclase feldspar. The closest sill is approximately 500 m northwest of the intersection of the Main Zone with McKinnon Creek). Another diorite sill is immediately east of the Roseberry showing. A third sill is at the summit of Goat Mountain.

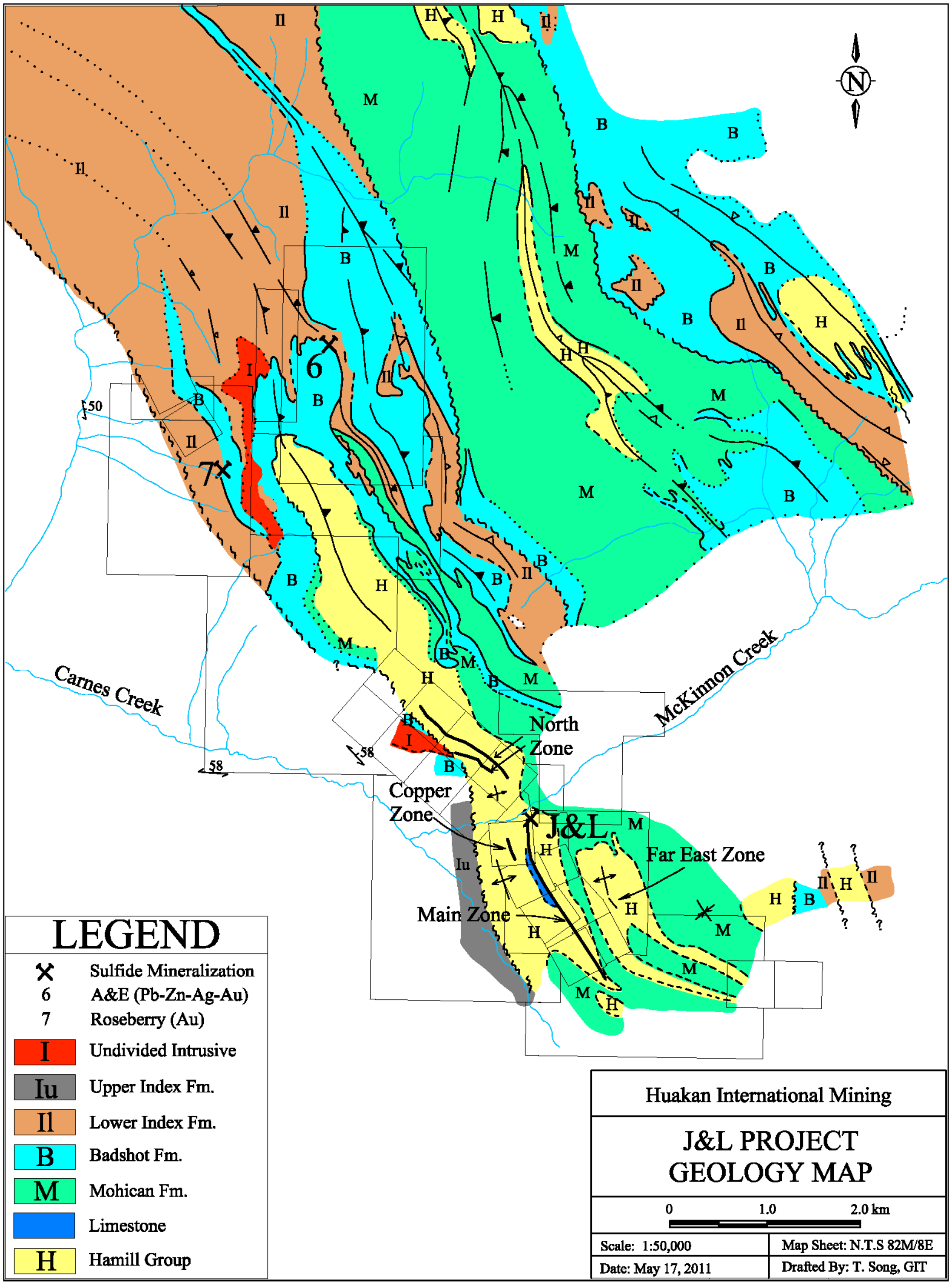
7.1.5 Micaceous Quartzite Unit

This unit is predominantly at the western edge of the Property and is well exposed along the Carnes Creek Forest Service Road. The unit is composed predominantly of quartzite to siliceous phyllite to quartz muscovite schist and may be loosely correlated to the Broadview Formation (Brown, 1991).

7.1.6 Jowett Formation

This unit is exposed in the first kilometre of the Carnes Creek Forest Service Road. It is an interlayered green metavolcanic and non-carbonaceous marble. This unit forms the hanging wall of the Columbia River Fault in the area of the claims.

Figure 7.1
J&L Local Geological Map



7.2 DEPOSIT GEOLOGY

Proximal to the Main Zone, the lithological assemblage consists of phyllite and schist (87%), limestone (8%), quartzite (5%), and rare dykes as shown by core logging from the 2010/2011 drill campaign.

The phyllite and schist units are moderately to well foliated, consisting of variable amounts of sericite, chlorite, and quartz. Chlorite, though in minor amounts, is considered the major contributor of the distinctive green hue in the units. Some banded sericite-chlorite-phyllite zones, ranging in width from 0.5 to 2.0 m, have a distinctive brownish hue due to the presence of fine-grained biotite or phlogopite. Although the phyllite is highly sheared and strongly foliated, feldspar phenocrysts are noted in the core indicating a possible mixing of a volcanic and/or sedimentary protolith.

There are two types of intercalated limestone seen in core proximal to the Main Zone, namely carbonaceous limestone and banded limestone. The interlayered beds vary in width from 1.0 to 20.0 m. The carbonaceous limestone units are fine to medium grained, dark grey to black in color, weakly to moderately foliated, and intensely jointed. The banded limestone units are light to medium grey in color with a medium-grained texture.

Quartzite beds consist of centimetre to metre thick intercalated layers of milky white quartz of varying purity. Minor amounts of sericite and/or chlorite are developed on foliation planes.

Late stage porphyritic intrusions are present as rare dykes. They are dark greyish green to brown in colour and medium-grained, composed of feldspar, quartz, and varying amounts of biotite. Only one dyke occurrence was seen in one drill hole from the 2010/2011 drill program. Its upper and lower contacts were sharp.

7.2.1 Structure

The dominant structure on the Property is a lithologic fabric that strikes north-westerly (striking about 330°) and dips (about 50°) toward the northeast. Near the southern edge of the J&L Crown-granted claims the lithology changes strike to a more east-west orientation (striking about 290°) and dipping northeast (about 40°). This change may be part of the Carnes Creek anticline (Logan et al., 1997A) or late stage deformation.

In general, the rocks in the area are faulted and intensely folded. One penetrative foliation has been imposed on all rock types and is the most readily recognizable feature. An early-stage foliation is still preserved and is best observed in silicified phyllite or quartz schist. However, most early-stage deformation features are only rarely preserved due to intense late-stage folding and strong shearing.

The Badshot Formation in the vicinity of the known deposits is recumbently overturned (Logan et al., 1997A). This rather ductile unit tends to flow under tectonic pressure and can form random boudinage structures that have become dispersed by the flowage. Some structural folding can be seen underground but is not excessive and is usually confined to the Main Zone wall rocks although the Main Zone mineralization is not affected. Limestone is much more ductile than argillite or quartzite, and only locally where completely enclosed by deformed limestone, is the mineralization affected by folding.

Two thrust sheets that strike north-westerly and dip to the east (Logan et al., 1997A) have been identified in the project area. Otherwise, faulting is almost exclusively confined to the Main Zone. Barely visible slips with a thin smear of gouge have been observed running along portions of the Main Zone. Occasionally, the heaviest of these faults can become randomly diverted into one of the walls, carrying the mineralized zone with it. Displacement along the faults is generally minor.

The Main Zone is a shear-hosted, sheeted, sulphide replacement deposit, which has overprinted a pre-existing carbonate hosted Ag-Pb-Zn deposit. The Main Zone appears to lie within a high angle thrust fault and crosscuts lithology along strike at a low angle. The shear zone is preferably developed near the contact between the limestone and phyllite or between quartz-rich schist and phyllite. Limestone tends to occur on the footwall of the mineralized zone along about half of the exposed underground strike length.

For much of the Main Zone exposed along strike in the underground workings, the zone is quite tabular with parallel sheeted massive and stringer sulphide bands but there are segments along strike where the banded massive sulphide units within the zone exhibit complex deformation textures. There are a number of indicators of shear sense, such as stretching lineation, rotated clasts, sheath folds, and asymmetric micro-folds. The asymmetrical folds indicate a dextral rotation.

The Ag-Pb-Zn-bearing Yellowjacket Zone is considered to be the remnant of a much larger pre-existing carbonate hosted deposit which has subsequently been modified (remobilized, augmented and replaced) by the Main Zone structure and mineralizing system.

7.3 MINERALIZATION

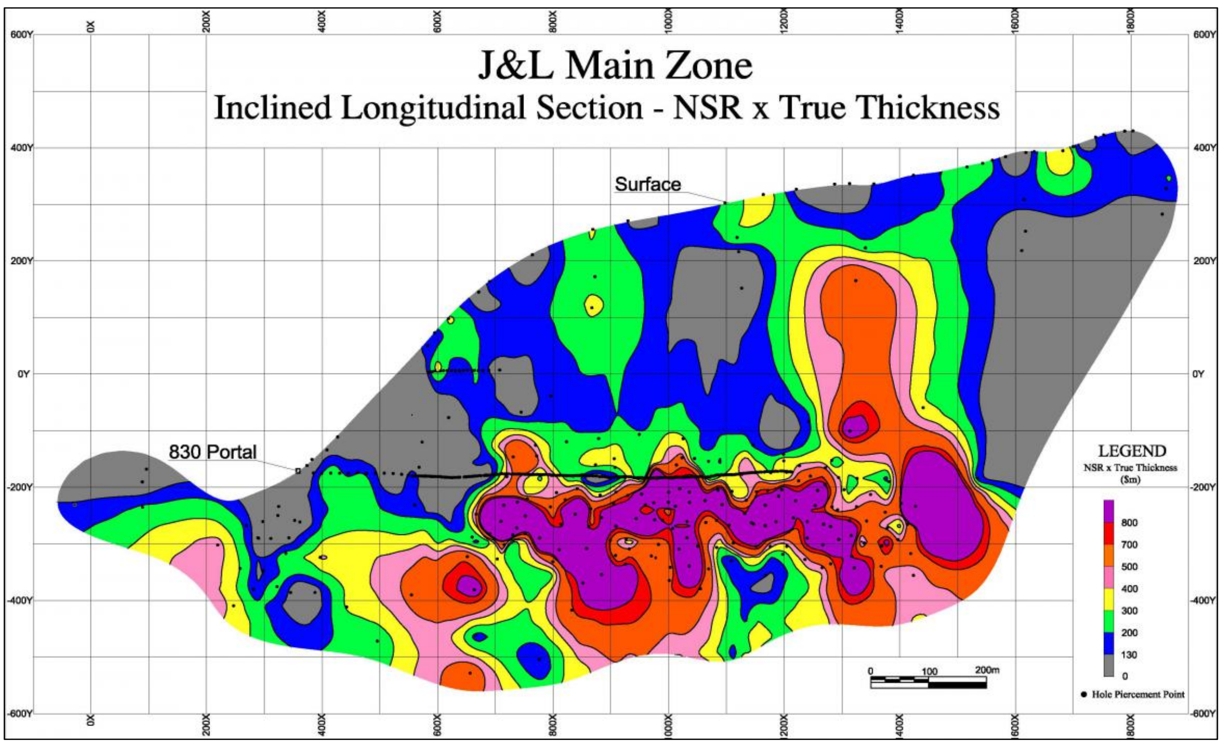
7.3.1 Main Zone Mineralization

The Main Zone is a structurally-controlled precious metal and polymetallic base metal sheeted sulphide (Au-Ag-Pb-Zn-As) deposit. The deposit has quite a reliable and predictable geometry. The zone is sheet-like or tabular with an average dip of 55° to the northeast. The zone of sheeted massive and stringer sulphide veining has an average width of 2.5 m but the sheeted sulphide veining can reach 15 m in true thickness. The continuity of the zone is broken by its absence in a few places within narrow sections as shown in Figure 7.2.

The deposit comes to surface with a surface strike trace of at least 3.34 km and a vertical extent of at least 0.8 km. It is speculated that the Main Zone is linked to the Roseberry Prospect and may also be linked to the former Mastodon Mine. This gives a collective distance of 9 km. Underground drifting has traced the deposit for 850 m (830 Level) and drilling has traced the deposit for 1,500 m along strike.

The Main Zone is composed of closely spaced bands of massive sulphides which frequently coalesce at its widest parts. Individual bands, which are generally tabular, may die out along strike over tens of metres but appear to resume in an adjacent band. Individual massive sulphide bands frequently range from 5 cm to 1 m thick. Sulphide minerals include pyrite, pyrrhotite, gold-bearing arsenopyrite, iron-rich sphalerite (blackjack), galena, tetrahedrite and trace chalcopyrite. There are also traces of silver-lead-antimony and lead-antimony sulphosalts. The banding ranges from predominantly arsenopyrite (high gold), to mixed arsenopyrite and massive sulphides, to massive sphalerite with no arsenic present.

Figure 7.2
Main Zone Inclined Longitudinal Section



Where the mineralization narrows, it is almost completely composed of arsenopyrite. As the zone of mineralization widens, the sulphide assemblage is more diverse where it is in contact with, or is completely enclosed by, limestone. Between mineralized bands, the host rock has been altered (sericite-quartz) and contains disseminated mineralization or thin massive to stringer sulphide streaks.

Three distinct types of mineralization have been noted. Type I mineralization is comprised of massive bands, lenses and sulphide stringers in a sericitic shear zone. Sulphides consist of medium to coarse grained pyrite, variously grain sized arsenopyrite, and fine-grained fracture-filled sphalerite and galena. Some coarse-grained pyrite and arsenopyrite display a brecciated texture. Type II mineralization is characterized by “milled” massive sulphide texture consisting of fine to coarse-grained, rounded to sub-rounded pyrite, arsenopyrite, quartz, and wall rock clasts in a very fine grained sulphide matrix. The matrix is composed of fine-grained pyrite, arsenopyrite, sphalerite, galena and quartz. Clasts derived from the host rock such as phyllite and schist contain sulphide stringers, which in part may represent Type I form of mineralization. This milled feature is interpreted as a mylonite texture developed by reworking within a structurally active shear zone. Milled sulphides carry high values of gold, silver, lead and zinc, and elevated mercury and antimony.

Type III mineralization consists of narrow stringers and fine to medium disseminations of principally sphalerite, with lesser amounts of galena and pyrite and very little arsenopyrite. Sphalerite is red to honey yellow in color, seeming to replace limestone. Although Type III mineralization can reach widths of 6-10 m, it appears to have limited extent both along the strike and vertically.

Gangue minerals include quartz, calcite, siderite, sericite, chlorite and graphite.

The wall rock in the hanging wall and footwall is mostly composed of sericite-chlorite-phyllite, quartz-sericite-chlorite-schist, and limestone. Phyllite and schist contain 1-5% pyrrhotite in the form of micro lenses on the foliation. An increase in pyrite development, concurrent with a sharp decrease in pyrrhotite, occurs in close proximity to the mineralized zone. Phyllite and schist are bleached due to sericitic alteration and silicification, resulting in apparent colour contrast between altered and unaltered rocks. Pervasive sericitization is extensively developed within the shear zone and its immediate hanging wall and footwall. The sericitic selvage ranges from 2 to 30 m wide. Marblization occurs immediately at the contact between limestone and the margins of the mineralized zone, varying in width from 0.1 to 1 m.

Pyrrhotite is disseminated ubiquitously throughout much of the non-mineralized rock in minor amounts. Trace amounts of chalcopyrite and pyrite are observed.

Sub-parallel intermittent footwall and hanging wall zones of similar Type I, II and III mineralization occur proximal to the Main Zone. The HM1 zone lies approximately 5 m into the hanging wall of the Main Zone. The HM2 zone lies approximately 8-10 m into the hanging wall of the Main Zone.

The FM1 zone lies approximately 2.5-10 m into the footwall of the Main Zone and has a degree of continuity. The FM2 zone lies approximately 15-20 m into the hanging wall of the Main Zone.

7.4 YELLOWJACKET ZONE MINERALIZATION

The Yellowjacket Zone is a stratabound carbonate-hosted, lead-zinc-silver deposit that is generally sub-parallel to the Main Zone. It is located approximately 30 m into the hanging wall of the Main Zone. The lead-zinc-silver mineralization is confined to multiple discrete zones related to siliceous carbonate units. The deposit does not outcrop but is defined only by drilling. Limited drilling (35 holes) has traced the deposit along strike for 500 m but remains open laterally in both directions and at depth. The deposit appears to pitch to the southeast at 30°.

The Yellowjacket deposit has no arsenic content. The mineralization is composed of patchy massive zinc-rich honey-coloured sphalerite (yellowjack) with minor medium-grained disseminated galena with elevated silver values. Other minerals include calcite, silica and minor sericite and siderite. Texturally, the mineralization can be foliated and/or laminated with sphalerite and galena running along cleavage surfaces. Other textures include brecciated or lacework patterns. Dolomite sections show discontinuous banding and are usually lower in grade.

The carbonate units hosting the Yellowjacket deposit may be located in the hinge of a recumbent isoclinal fold, fringed by phyllite and quartzite. The mineralization appears to thicken in the apparent fold hinge where darker coloured sphalerite and coarser and more abundant galena occurs. The Yellowjacket Zone is intensely silicified. Sericite has also been observed in core samples. Silicification also appears to intensify towards the apparent fold hinge. Fluorite is common in most mineralized sections, particularly near higher grade sections. Pyrite and pyrrhotite are present in low amounts.

7.5 OTHER MINERALIZATION

The Roseberry showing lies on the J&L Property (082M 091) 4.5 km to the northwest of the Main Zone (see Figure 7.1). The polymetallic (Cu-Zn-Pb-Ag-Au) vein-type showing lies just below the contact of Lardeau graphitic schists and Badshot Formation limestones. Although it has been known for almost a century, it has received only minor surface exploration due to its remote location. The mineralization is composed of coarse disseminated to semi-massive arsenopyrite in discontinuous quartz carbonate veins hosted by intensely sheared graphitic schist. The mineralization resembles the Main Zone mineralization. Chip sampling of the Roseberry showing returned values such as 15.03 g/t Au and 37.4 g/t Ag across 0.3 m.

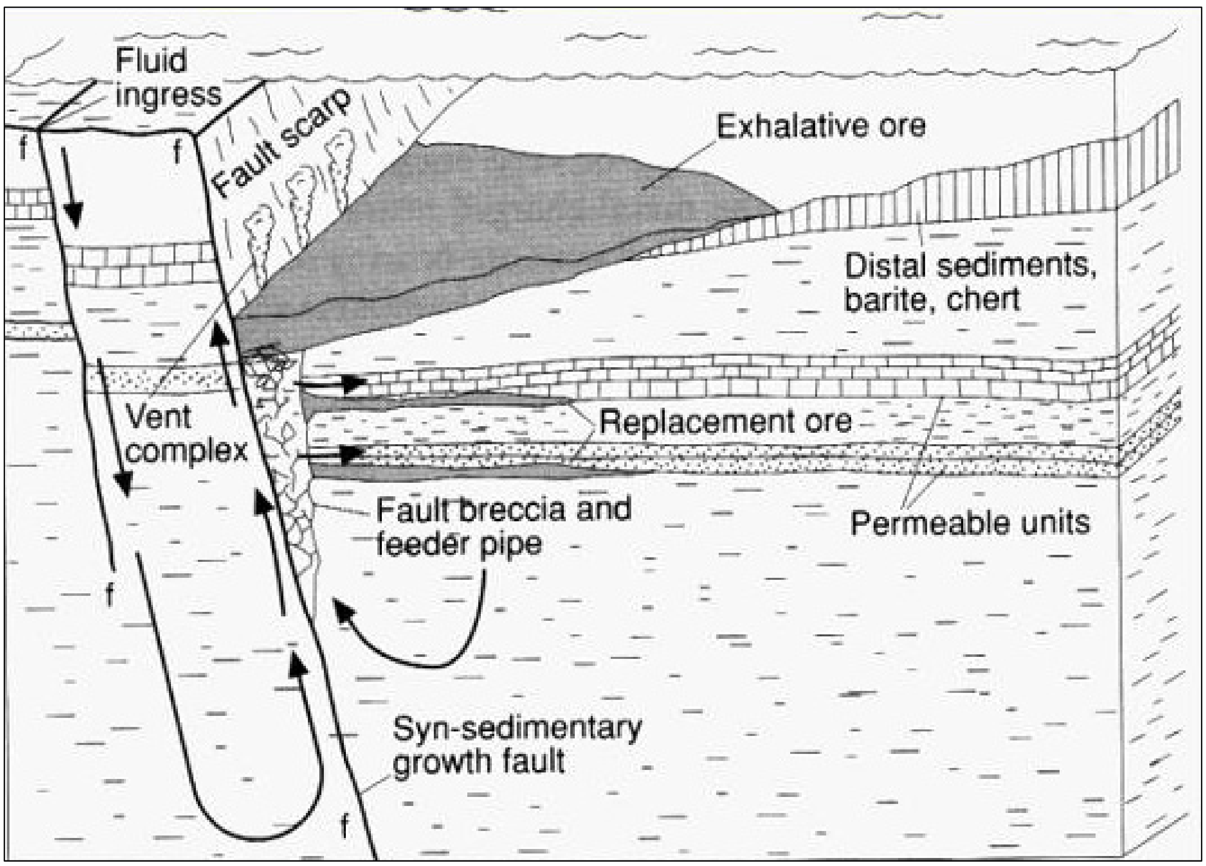
The A & E showing lies on the J&L Property (082M 099) 5 km north of the Main Zone and 2 km northeast of the Roseberry showing. The mineralization is related to sheared schistose zones with intense deformation and complex folding, interlayered with or in contact with limestone. This polymetallic (Ag-Pb-Zn-As) showing represents a series of three parallel mineralized zones similar to the Main Zone. One of the zones averaged 11.01 g/t Au, 356.7 g/t Ag, 10.75% Zn and 5.48% Pb from four muck samples. It is a narrow arsenical zone of massive sulphides. There are several hand-tooled short adits and surface showings that have traced the zone for 400 m along strike and 160 m vertically. It has not been drill tested at depth. The A & E prospect represents the potential for multiple parallel zones of mineralization.

The Copper Zone is located 100 to 150 m into the footwall of the Main Zone. It is a narrow stringer sulphide zone hosted by quartzites and chloritic phyllites and schists, and has been traced for 320 m horizontally and 90 m vertically. Although it does not appear to return economic grades at surface, the showings are leached and weathered. A chip sample by Equinox returned 3.55 g/t Au, 21.7 g/t Ag and 0.19% Cu over 1.0 m. This zone could be tested further by diamond drilling.

8.0 DEPOSIT TYPES

The J&L Property lies at the northern end of the Kootenay Arc which is known for its Irish-type carbonate hosted Zn-Pb, volcanogenic massive sulphide (VMS) and sedimentary exhalative (Sedex) deposits. The two deposits on the J&L Property are the Main Zone and the Yellowjacket Zone. An idealized generic Sedex-type deposit model is shown in Figure 8.1.

Figure 8.1
Generic Sedex-type Deposit Model



Wilkinson, 2007.

8.1 MAIN AND YELLOWJACKET ZONES

8.1.1 Main Zone

The Main Zone mineralization lies in an inferred shear zone, characterized by sheeted massive to semi-massive sulphide bands and stringers spatially associated with the Yellowjacket Zone.

The Main Zone mineralization cross-cuts lithologies at low angles and appears to represent a high angle thrust zone. The deposit is relatively tabular, continuous and predictable, much like a replacement vein in a broad sense. Surface exploration has traced the Main Zone for 3 km. The drilling to date demonstrates that the Main Zone continues for a strike length of at least 1.2 km and in the down dip direction for at least 800 m. There remains good potential for additional resources on the Main Zone, which remains open in the down dip direction and along strike to the northwest and southeast. In the respect to this size, the deposit fits a large structure. The Main Zone averages 2.5 m thick of sheeted sulphide veining but the sheeting can reach up to 15 m true thickness. It is a complex banded zinc-lead-silver-gold-arsenic deposit, partly having a close spatial relationship with limestone. Massive to semi-massive bands and stringers parallel or subparallel the dominant lithologic foliation, reflecting a strong structural domain, which is typical of fracture-fill deposits.

A long-standing oversimplification stated by previous workers that Main Zone occurs at the contact of footwall limestone with hanging wall phyllites. In fact, 2010/2011 drill program identified that the majority of Main Zone is more likely to occur at the base of the limestone unit in contact with footwall phyllites.

The close spatial relationship between mineralization and limestone can be interpreted for four reasons. Firstly the contact of limestone and phyllite is favourable for the development of a shear zone. Secondly the competency contrast between limestone and phyllite creates a favorable condition to allow dilation within the shear zone. Thirdly, limestone is a good structural seal. Fourthly, carbonaceous limestone is a good reducing agent in the process of gold precipitation.

Based on the detailed geological core logging, the mineralization is stronger when the shear zone cuts the phyllite unit rather than more schistose lithologies. This can be interpreted that the schist units are more competent than phyllite, so the hydraulic fracturing is more susceptible in phyllites than in schists at the same fluid pressure.

The Main Zone does not easily fit into a specific genetic model. Geologists who have worked on the Main Zone in the past have proposed a sedimentary exhalative model (Sedex) model, a volcanogenic massive sulphide model (VMS), a replacement model and a shear hosted model.

Huakan geologists interpret the Main Zone to be a shear hosted sheeted sulfide replacement deposit, which has overprinted the pre-existing carbonate hosted Ag-Pb-Zn Yellowjacket deposit and that the Yellowjacket is essentially a mere remnant of its former self. That much larger pre-existing carbonate hosted deposit has been cut and modified (remobilized, augmented and replaced) by the Main Zone structure and mineralizing system.

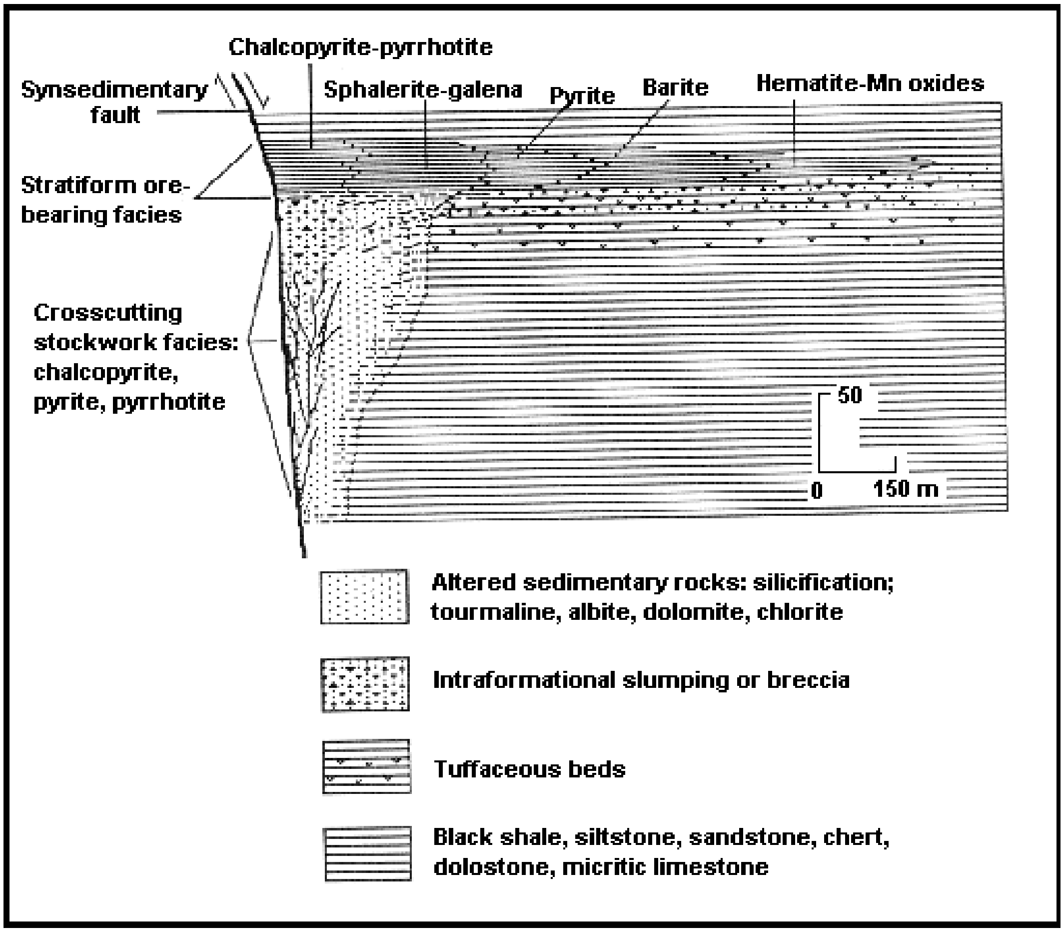
8.1.2 Yellowjacket Zone

It has been interpreted that the Yellowjacket Zone was first formed as a carbonate-hosted Zn-Pb deposit and was followed by the shear-hosted Main Zone. The Yellowjacket Zone has a close affinity to the Irish-type carbonate hosted Zn-Pb deposits which are characterized by:

- Active tectonics during sedimentation and some of the mineralization;
- Deposits are hosted by Carboniferous carbonates;
- Strong structural control seen in the deposits;
- Mineralization is stratabound with some local sections which cross cut stratigraphy;
- Mineralization textures are generally replacive and brecciated but locally banding is evident;
- Iron and magnesium carbonates seen in and around the mineralization;
- Zinc, lead, iron, copper and silver are known in the deposits and have some zoning laterally and vertically (see Figure 8.2);
- Isotopes point to two fluids being involved in the process, one hydrothermal and the other Carboniferous sea water,
- Fluid inclusions indicate that the temperature ranges from 100°C to 300°C.

Typically, the Yellowjacket deposit occurs about 30 m into the hanging wall (northeast) from and sub parallels the Main Zone, however, locally there is Yellowjacket type mineralization in the immediate hanging wall of the Main Zone as observed in the underground cross-cuts. The Yellowjacket deposit is stratiform, exclusively hosted within folded intensely silicified carbonate rocks. The Yellowjacket Zone is composed of a patchy massive honey-coloured (zinc-rich) sphalerite with minor disseminated galena and elevated silver values. The mineralization appears to be confined to favourable carbonate rocks that have been folded into a recumbent overturned anticline straddled by phyllite. Grade and thickness increases towards the hinge of the fold. Fluorite is present and barite is absent in the Yellowjacket, unlike the Irish-type deposits.

Figure 8.2
Diagrammatic Cross Section Showing Mineral Zoning in Sedimentary Exhalative Zn-Pb Deposits



Briskey, 1986.

9.0 EXPLORATION

9.1 PREVIOUS EXPLORATION

There remains good potential for additional resources in the Main Zone, which remains open in the down dip direction and along strike to the northwest and southeast. Extensive drilling has indicated a traceable continuous plane with virtually no fault offsets, cut-offs or fault drags zones. Exploration over the Property life has confirmed persistent vertical and horizontal continuity although there is reference to an element of improved grade in en-echelon series of northwest plunging lenses that strengthen with depth.

Details of historical exploration on the property can be found in Section 6.0, History.

9.1.1 Geology/Prospecting

The J&L deposit has been intermittently explored for over 100 years. The initial recorded mapping and prospecting was done by Dr. Gunning of the GSC in 1928. Other companies that have mapped and prospected the J&L deposit include Piedmont Mines Ltd. (1929), Westairs Mines Ltd. (1963 to 1965), BP Selco (1981 to 1985), Equinox Resources Ltd. (1989) and Weymin Mining Corporation (1996 to 1997). The results of these mapping programs identified the surface trace of the Main Zone and several other parallel mineralized structures. No geological mapping or prospecting has been carried out by Huakan on the J&L Property.

9.1.2 Geochemistry

Geochemical surveys were conducted on the J&L deposit by BP Selco (1981 to 1985), Equinox Resources Ltd. (1989) and Weymin Mining Corporation (1996 to 1997). Geochemical soil anomalies (Zn, Pb and As) identified the surface trace of the Main Zone and the Copper Queen Zone. No geochemical soil sampling surveys have been carried out by Huakan on the J&L Property.

9.1.3 Geophysics

The first geophysical survey on the property was a helicopter-based, input electromagnetic survey completed in May 1982 by Selco Inc. The center of the Questor survey was located at 51º22'N and 118º 15'N with a total of 699 km flown covering an area of 232 km^2. The main purpose of the survey was to delineate the structure which hosts the J&L zone and to trace any extension to the known mineralization. Approximately 22 anomalous responses were picked up from the Questor survey, 11 of which were found within the existing claim boundaries.

Nine of these responses were in potentially favorable geological settings and three were considered priority targets. No response was shown over the J&L deposit itself, although the survey did not adequately cover the extent of the known surface trace of the J&L Main Zone.

Due to incomplete cover of the Questor Input survey, a helicopter-based Dighem II electromagnetic survey was flown in August, 1982, covering a portion of the area covered by the earlier Questor survey. The Dighem II survey was carried out along 396 km of flight lines covering an area of 396 km^2. During the Dighem survey, only three new anomalies were recognized and comparisons were made between the Questor and Dighem surveys to delineate any anomaly commonalities between the two surveys. A weak response was detected approximately on strike with the J&L Main Zone, 1 km south of the McKinnon Creek valley.

During July, 1991 a program of ground transient Electromagnetic surveying was carried out on the J&L Property by Frontier Geosciences Inc. on behalf of Equinox Resources Ltd. The survey entailed 7.2 km of coverage testing 700 m of strike of the surface expression of the Main Zone. A base line was established along the northwest-southeast projected axis of the target and perpendicular lines were established every 100 m. A transmitter loop was laid out downslope and to the southwest of the survey grid.

Two EM anomalies typical of the proposed geological target were observed as a result of this survey. The stronger of the two anomalies occurred at the south-west corner of the north-west end of the survey grid near the valley bottom and appeared unrelated to the location/strike of the Main Zone. No geologic explanation was given for this anomaly, which was speculated to be caused by graphitic content as found in outcrop 400 m to the north-west of the anomaly.

The second anomaly was clearly attributed to the Main Zone. Responses to the target zone were observed on all lines surveyed across the main grid. The zone is typically evident as a strong anomaly, observed into the middle time channels. Although the overall response reflects a large, tabular zone, particularly on the more north-westerly lines, there is evidence which suggests the zone may contain multiple conductors.

No geophysical survey work has been carried out by Huakan on the J&L Property.

9.1.4 Trenching

There have been a total of 52 trenches excavated on the property from 1925 to 1982. Twenty of these trenches were dug by Westairs Mines Ltd. between 1962 and 1967. The trenches assisted in delineating the surface expression of the Main Zone over Goat Mountain. The primary documentation related to surface trenching on the J&L Property is found in the Summary Report on the J&L Option by R. Pegg for Selco Inc. In Pegg's report there is detailed documentation regarding the geological description of 26 individual closely-spaced surface trenches following approximately 1.2 km of strike along the Main Zone. This report also provides the assay data for each of the sampled mineralized trench intervals. Also included within Pegg's 1982 report are the descriptions of 11 mineral showings, mainly located between trench 24 and trench 26 at the south-east end of the surface expression of the Main Zone.

Micon is aware of 545 samples that were taken within the various trenches on the property.

No trenching work has been carried out by Huakan.

9.2 RECENT EXPLORATION

Huakan has not performed any exploration on J&L Property except underground drifting and drilling. Drilling was carried out from the existing mine workings during the fall of 2010 into the winter of 2011. The goal of the drilling was to increase the amount of indicated resources for the main zone. Details are provided in Section 10.1.

9.3 CURRENT EXPLORATION

Huakan has recently implemented Phase II of its exploration program, the goal of which is to increase measured and indicated resources to 1.35 Mt. Phase II is estimated to take approximately 8 months to complete.

9.3.1 Permitting

For the Phase II program, Huakan submitted an application for permit amendments to the government. The submittal includes design work for the relocation of the sulphide-bearing waste rock pile. This was a condition before any new tunneling was allowed. The government also requested that the 832 portal be shotcreted before resuming underground activities. The second portion of the submittal is to increase the permitted amount of new drifting. The permit was granted and Phase II is underway. The portal has been shotcreted. A lined PAG waste rock pad was constructed and the PAG waste rock generated from the 2008 tunnelling campaign has been re-located to the lined pad.

9.3.2 Drifting

An estimated four months will be required to drive 452 m of track drift, at a rate of 113 m/month.

9.3.3 Drilling

It is anticipated that drilling will take place after drifting has been completed. Drilling will require approximately two months, with two underground drill rigs completing a total of 6,700 m of drilling.

10.0 DRILLING

10.1 HUAKAN DIAMOND DRILLING 2007-2010

In November, 2007, Merit completed a 1,363 m, nine hole surface drill program on the Yellowjacket Zone which had the objective of verifying historic drilling over a portion of the Yellowjacket deposit. The program successfully achieved this objective by intercepting multiple zinc-lead-silver zones similar in grade and width to previous drilling.

In November, 2010, Merit/Huakan commenced Phase I of the exploration program, a 2,000 m underground drill program which had the objective of verifying historic drilling and broadening the known resource in the Main Zone. The program was expanded to 7,897 m and was completed by early February, 2011 with the completion of 60 BQTW (thin wall) core holes.

10.1.1 Collar Surveying

At the completion of both the 2007 surface and 2010/2011 underground drill programs, drillhole collar locations of all holes were marked and surveyed by B.C. professional land surveyors.

10.1.2 Downhole Surveying

During the 2007 surface diamond drill program downhole surveys were carried out using an Easy-Shot tool, taking measurements at the bottom and midway for the first three holes. Due to a defective tool, the final three drill holes were tested by acid tests at the bottom of each hole.

Downhole surveying in the 2010/2011 underground drill program utilized the FLEXIT SmartTool Drill Hole Survey system. Measurements were taken every 30 m down the hole, usually including a near-to-collar test as well as a near-to-bottom test. A declination factor of 17º was added to all azimuth readings taken during the downhole surveys to give true azimuth readings for this region of British Columbia. Other data collected were dip angles recorded at the various downhole reading sites as well as magnetic susceptibility. A small number of the azimuth readings gave erroneous results caused by strong magnetism, and those readings were therefore eliminated.

10.1.3 Core Recovery and Storage

Core recoveries throughout the 2010/2011 underground J&L drill program were normally over 90% and often over 95%. All drillcore from both the 2007 and 2010/2011 drill programs are securely stored on the property, near the camp facility.

10.1.4 Contractor

The 2007 diamond drill program was carried out by Elite Drilling Ltd. of Revelstoke, B.C. over the period October 23 to November 13, 2007.

DMAC Drilling Ltd. of Aldergrove, B.C. was the drilling contractor for the 2010/2011 program. Drilling took place between November 11, 2010 and January 31, 2011. Drilling was carried out on two 10-h shifts using two Hydracore drill rigs mounted on steel wheels, thus providing drill access to the tracked 830 main drift and cross-cuts.

10.1.5 Security Procedures

After taking custody of the drillcore, Huakan's geologists conduct an industry standard program of geological and geotechnical logging, photography, density measurements and core sampling. The core was logged in detail onto paper logs, and then entered into digital summary format.

In the author's opinion the core transfer procedures and security measures conducted and described by Huakan conform to standard industry practice

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following points describe the sampling procedures and steps taken during the 2010/2011 drill program:

- Core was first cleaned, organized and photographed;
- Geotechnical logging was undertaken by a trained technician;
- Core boxes were labelled using scribed aluminum tags;
- Core logging and sample selection was performed by the site geologists;
- In areas of Main Zone mineralization, sampling intervals were determined by similar sulphide abundance. Samples were generally 0.5 m long but rarely greater than 1.5 m long. A minor number of samples taken were less than 0.5 m in length, but not less than 0.25 m long;
- Sampling was carried out beyond the limits of the Main Zone sulphides both into barren hanging wall and footwall rocks;
- Every 18th, 19th and 20th sample tags were designated as a duplicate, standard and blank, respectively. The duplicate sample was a split of the sample preceding it (i.e., duplicate sample #18 is a 50% split of sample #17). Splitters retained the standards and blanks and placed the entire pouch of material into the labelled plastic sample bag in the corresponding tag order;
- Core was logged, sampled and stored on site. The logging geologist placed a colour crayon line along the desired sample cut to provide an even bisection of the core;
- The core was cut in half, bisecting fabric or vein material evenly;
- Technicians were instructed to place the same side of core back into the box and the other into a labelled clean plastic sample bag that was then sealed using a zap-strap;
- Sample bags were placed in address-labelled rice bags, sealed with plastic zap-straps and shipped from Revelstoke, B.C. by Greyhound Bus to Eco Tech Laboratory Ltd., Kamloops, B.C.;
- Sample shipment records were maintained. Records were also kept of sample preparation, analysis requested and the person intended to receive the results;
- Core sampling was carried out by use of a diamond blade core saw. The core sampler was highly experienced and sampling work was closely monitored by on-site core logging geologists;

- No core samples were taken by an employee, officer, director or associate of Huakan.

Analytical work for the 2010/2011 drill programs was carried out by Eco Tech Laboratory Ltd. (Eco Tech) of 100041 Dallas Drive, Kamloops, B.C. Huakan has archived all of the original assay certificates for both the 2007 and 2010/2011 drill programs.

Eco Tech is registered for ISO 9001:2008 by KIWA International (TGA-ZM-13-96-00) for the “provision of assay, geochemical and environmental analytical services”. Eco Tech also participates in the annual Canadian Certified Reference Materials Project (CCRMP) and Geostats Pty bi-annual round robin testing programs. The laboratory operates an extensive quality control/quality assurance program, which covers all stages of the analytical process from sample preparation through to sample digestion and instrumental finish and reporting.

Samples (minimum sample size 250-g) are catalogued and logged into the sample-tracking database. During the logging in process at the laboratory, samples are checked for spillage and general sample integrity. It is verified that samples match the sample shipment requisition provided by the clients. The samples are transferred into a drying oven and dried. Rock samples are crushed on a Terminator jaw crusher to -10 mesh ensuring that 70% passes through a Tyler 10 mesh screen. This is verified each batch.

Every 35 samples a re-split is taken using a riffle splitter to be tested to ensure the homogeneity of the crushed material. A 250-g sub-sample of the crushed material is pulverized on a ring mill pulverizer, each batch ensuring that 85% passes through a 200 mesh screen. The sub-sample is rolled, homogenized and bagged in a pre-numbered bag. A barren gravel blank is prepared before each job in the sample prep to be analyzed for trace contamination along with the actual samples.

For gold, a 30-g sample size is fire assayed along with certified reference materials using appropriate fluxes. The flux used is pre-mixed, purchased from Anachemia Science which contains Cookson Granular Litharge, (silver and gold free). The ratios are 66% litharge, 24% sodium carbonate, 2.7% borax, 7.3% silica. These charges may be adjusted with borax or silica based on the sample. Flux weight per fusion is 120g. Purified silver nitrate is used for inquartation. The resultant doré bead is parted and digested with aqua regia and then analyzed on an atomic absorption instrument (Perkin Elmer/Thermo S-Series AA instrument). Gold detection limit on AA is 0.03-100 g/t. Any gold samples over 100 g/t are run using a gravimetric analysis protocol. Each batch submitted is fire assayed as a batch.

Appropriate standards and repeat/re-split samples (quality control components) accompany the samples on the data sheet for quality control assessment. For 30 element ICP, a 0.5-g sample is digested with a 3:1:2 ($HCl:HNO_3:H_2O$) for 90 minutes in a water bath at 95°C. The sample is then diluted to 10 ml with water. All solutions used during the digestion process contain beryllium, which acts as an internal standard for the ICP run. The sample is analyzed on a Thermo Scientific IRIS Intrepid II XSP/iCAP 6000 Series ICP unit. Certified reference material is used to check the performance of the machine and to ensure that proper digestion occurred in the wet laboratory Quality control (QC) samples are run along with the client samples to ensure no machine drift or instrumentation issues occurred during the run procedure. Repeat samples (every batch of 10 or less) and re-splits (every batch of 35 or less) are also run to ensure proper weighing and digestion occurred. Results are printed along with accompanying quality control data (repeats, re-splits and standards). Any of the base metal elements that are over limit, Ag >100g/t; Cu, Pb, and Zn >10,000ppm) are run as an assay.

Appropriate standards and repeat/re-split samples (quality control components) accompany the samples on the data sheet. The digested solutions are made to volume with reverse osmosis water and allowed to settle. An aliquot of the sample is analyzed on a Perkin Elmer/Thermo S-Series AA instrument. (Detection limit 0.01% AA). Instrument calibration is done by verified synthetic standards, which have undergone the same digestion procedure as the samples. Standards used narrowly bracket the absorbance value of the sample for maximum precision.

It is the author's opinion that the sampling preparation, security and analytical procedures employed by Huakan were satisfactory.

12.0 DATA VERIFICATION

12.1 SITE VISIT AND INDEPENDENT SAMPLING

12.1.1 P&E Site Visit

The J&L Property was visited by Mr. Fred Brown, CPG, on December 17, 2010. Data verification sampling was done on diamond drill core, with 18 samples distributed in 18 holes collected for assay. These samples were collected from both the current drill program as well as from a number of the historic (1991 and earlier) drill holes. An attempt was made to sample intervals from a variety of low and high-grade material. The chosen sample intervals were then sampled by taking complete sections of the remaining half-split core. The samples were then documented, bagged, and sealed with packing tape and were delivered by Mr. Brown to ALS Minerals (formerly referred to as ALS Chemex), 2103 Dollarton Highway in North Vancouver for analysis.

ALS Minerals is the leading full-service provider of analytical geochemistry services for the global mining industry. With over 60 laboratories located in key mining districts on six continents, ALS Minerals maintains ISO 9001:2008 and ISO/IEC 17025:2005 certifications, provides clients with all internal quality control data, and maintains a library of detailed laboratory analytical methods required as the necessary documentation for 43-101 reporting.

The quality system is composed of:

- Certified Data Analysis: ISO 9001:2008 and ISO/IEC 17025:2005 accreditations: The ALS Quality Management System (QMS) complies with the requirements of International Standards ISO 9001:2008.

The North Vancouver analytical facility received accreditation to ISO/IEC 17025:2005 from the Standards Council of Canada (SCC) for the following methods:

- Fire Assay Au by atomic absorption (AA),
- Fire Assay Au and Ag by gravimetric finish,
- Fire Assay Au, Pt, and Pd by inductively coupled plasma (ICP),
- Aqua Regia Ag, Cu, Pb, Zn and Mo by AA,
- Four Acid Ag, Cu, Pb, Zn, Ni and Co by AA,
- Aqua Regia Multi-element by ICP and MS,
- Four Acid Multi-element by ICP and MS,
- Peroxide Fusion Multi-element by ICP.

At no time, prior to the time of sampling, were any employees or other associates of Huakan advised as to the location or identification of any of the samples to be collected.

A comparison of the P&E independent sample verification results versus the original assay results for gold, silver, lead and zinc can be seen in Figure 12.1 to Figure 12.4.

Figure 12.1
P&E Verification Sample Results for Gold

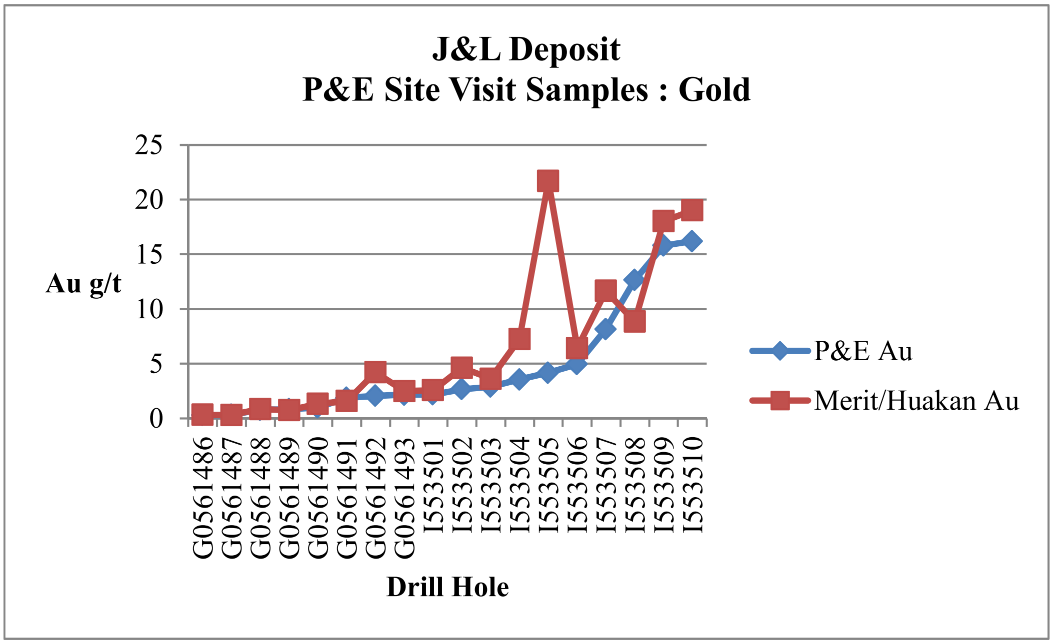


Figure 12.2
P&E Verification Sample Results for Silver

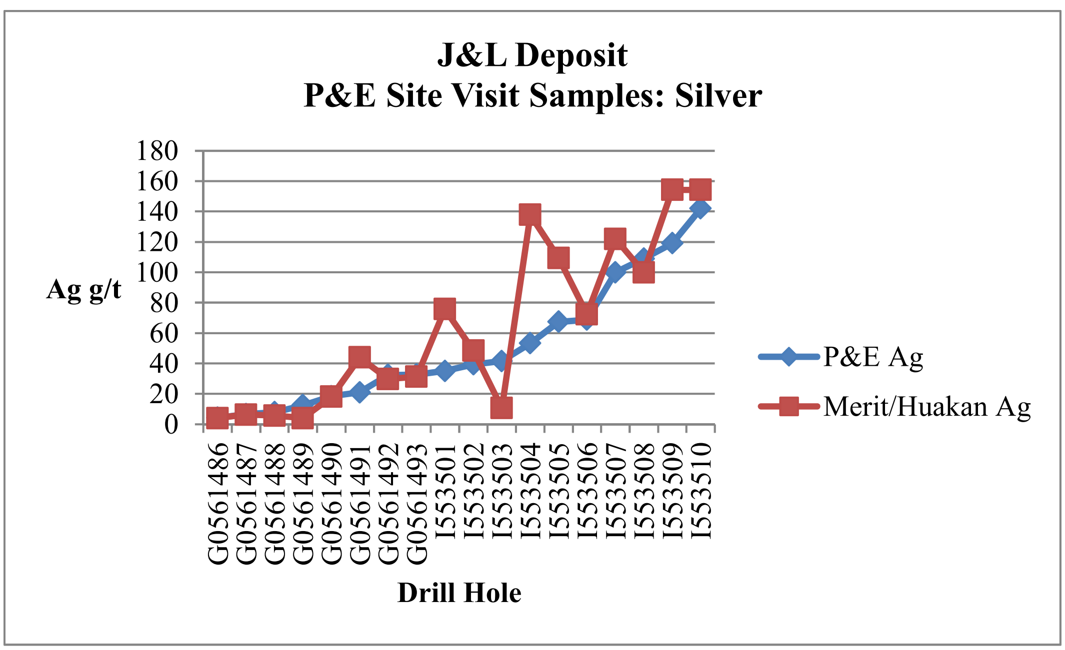


Figure 12.3
P&E Verification Sample Results for Silver

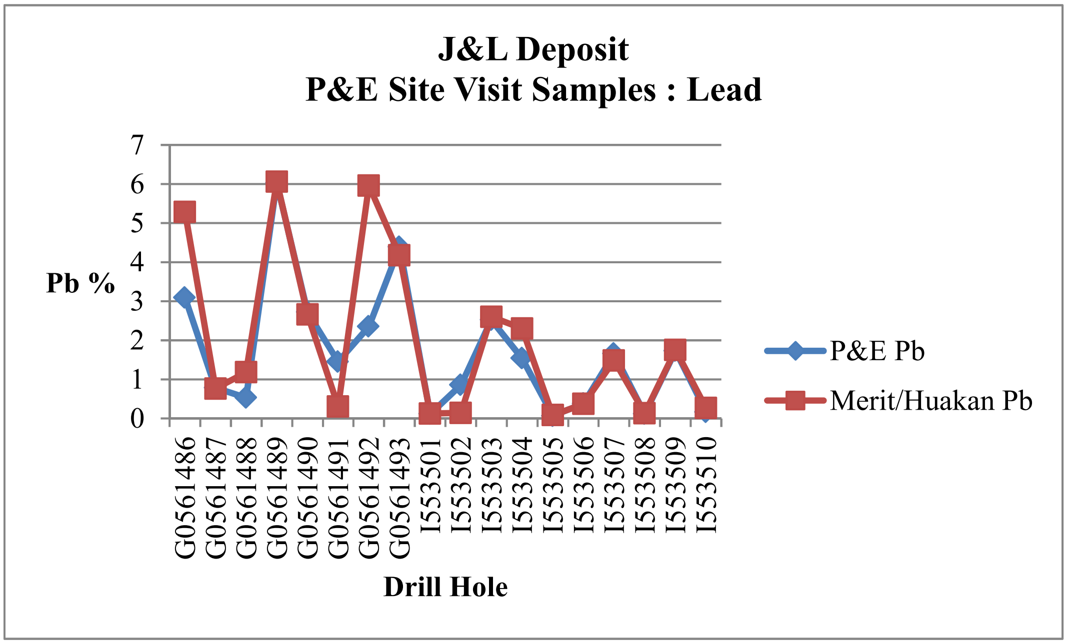
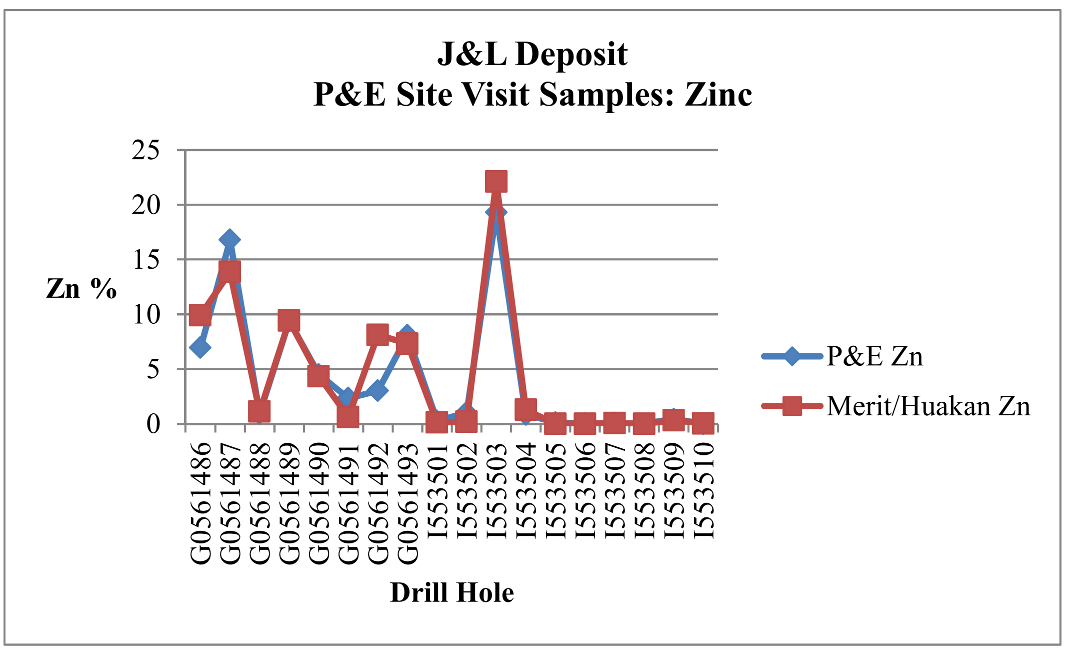


Figure 12.4
P&E Verification Sample Results for Zinc



12.1.2 Micon Site Visit

On October 27, 2011, Micon conducted a site visit to the J&L Property for the purposes of this PEA.

Micon's team at the site visit is listed below.

Christopher Jacobs, CEng, MIMMM	Project Manager/Economic Analysis
Catherine A. Dreesbach, P.E.	Senior Mining Engineer
Bogdan Damjanović, P.Eng.	Senior Metallurgist

Huakan was represented on site by Mr. Paul Cowley, VP Exploration, Mr. Fred Sveinson and Ms. Sandy Sveinson. Mr. David Willms, geological engineer with Klohn Crippen Berger Limited (KCBL), was present during the site visit, as part of KCBL's assessment of potential tailings storage sites.

The following are some of Micon's observations during the site visit:

Existing development at the J&L Property has been aimed at exposing ore for sampling and providing access for underground drilling. The 830 Adit, a track drift (8 ft by 8 ft) has been developed mostly in mineralization. Cross-cuts into the hanging wall provide drill cubbies, and a footwall cross-cut (15 ft by 15 ft) leads to the 832 Adit portal, which is currently used as the main access; the 830 portal lies too close to an avalanche chute for it to be used year-round. Some earlier development on the 990 m elevation is no longer accessible. Some raises mined in mineralized material from the 830 Level are blind.

Huakan has established an all-year camp, mobilized mining and maintenance personnel, and equipped a surface shop facility near the 832 Adit portal. Equipment on site includes EJC 430 ADTs (30 short tons) and EJC 210 (6 yd^3) scoop-trams.

There has been a significant amount of metallurgical testwork studies completed on the deposit to date. Huakan confirmed that it plans to conduct a comprehensive metallurgical study, managed by Frank Wright, P.Eng., independent metallurgical consultant to the Company, which will produce adequate quantities of material to test the above flowsheet and provide data for equipment sizing and metal recovery forecasts. Presently, this testwork is focused on the use of HMS to upgrade the mill feed.

A helicopter tour of the site gave a very useful overview of some of the potential sites for a tailings storage facility (TSF).

The proximity to Highway 23 and the town of Revelstoke are positive factors for the project. Re-alignment of the Forest Service road into the mine site is expected to be straight-forward.

12.2 HUAKAN QC PROGRAM

12.2.1 Reference Material

For the 2010/2011 drill programs, Huakan geologists inserted certified reference materials and blanks, which were obtained from CDN Resource Laboratories (CDN) of Langley, B.C. In addition, duplicate sampling was added to the sample stream.

The CDN standards were CDN-ME-7 and CDN-ME-11. Standards were inserted into the sample stream at a rate of 1 in 20 by the project geologists. Certified reference material is inserted regularly into batches of samples sent to the lab for analysis in order to monitor the accuracy (lack of bias) of the lab results.

The CDN-ME-7 had a total of 36 data points and CDN-ME-11 had a total of 19 data points. Both standards were certified for gold, silver, lead and zinc and both performed extremely well with all data points falling within the +/- two standard deviations from the mean.

12.2.2 Blanks

Huakan purchased blanks consisting of pulverized river rock (predominantly granite) from CDN for use in the 2010-2011 drilling programs. CDN's assaying of the blank material found it to contain <0.01 g/t Au.

Blanks were inserted into the sample stream at a rate of 1 in 20, for a total of 55 data points.

All data points for gold and silver were well below the upper threshold of three times the detection limit for the element in question, which was the upper threshold set for monitoring blank results. Lead returned an average value of 0.002% with a standard deviation of 0.0006. Zinc returned an average value of 0.005% with a standard deviation of 0.0005. All results indicate no contamination present at the analytical level.

12.2.3 Duplicate Sampling Program

For the 2010/2011 J&L drill program a duplicate core sampling procedure was implements in order to quantify precision (reproducibility) of analytical results at the field level.

For the purposes of this data verification, drill core duplicates were inserted into the sample stream at a rate of 1 in 20. A duplicate sample consisted of a 50% split of the numbered sample interval immediately preceding the duplicate sample. For example in a normal sampling stream, every 17th core sample provides the 50% split for the 18th or duplicate sample.

In addition, P&E examined the laboratory coarse reject duplicates and pulp duplicates for gold, silver, lead and zinc. The coarse reject data set contained on average 27 pairs, and the pulp data set contained 239 pairs for gold, 95 pairs for silver, 100 pairs for lead and 104 pairs for zinc.

Simple scatter graphs were plotted for the field, coarse reject and pulp duplicate pairs for the four metals. Precision improved steadily from the core duplicates through to the pulp duplicates. The precision at the pulp duplicate level for all four metals was excellent, with a 1:1 ratio.

12.2.4 Sampling and QC Recommendations

The QC program as implemented and monitored by Huakan has ensured a robust, accurate and precise database that is suitable for resource estimation. It is recommended however, to source a coarse material to be used as a blank, such as sterile drill core or landscaping material that is required to pass through all stages of sampling reduction. The current pulverized blank material only monitors contamination at the analytical level.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Huakan has recently initiated metallurgical testing on the core samples obtained from J&L Property. This testwork includes heavy media separation, grinding, several stages of flotation and pressure oxidation and bioleaching on a portion of the flotation products prior to cyanidation; however, these results were not available at the time this report was prepared; therefore this report has relied on previous test results, which are quite extensive The information in this section describes historical metallurgical testing performed by previous companies. Micon has not independently reviewed or assessed this testwork.

13.1 SUMMARY

Numerous mineralogical and metallurgical studies and reports have been carried out for the J&L Main Zone deposit. The metallurgy studies and assessments date from 1982 through to 2005 with particular effort made throughout the 1980's and 1990's in concert with exploration programs carried out by BP Canada (Selco) Ltd., Pan American Minerals Corporation, Equinox Resources Ltd., Cheni Gold Mines Ltd. (CGM) and Weymin Mining Corporation. The Weymin bulk sample work was substantial and was incorporated into the H.A. Simons 1998 scoping study. The most recent J&L report completed was for a metallurgical study carried out in 2005 on behalf of BacTech by PRA. The 1998 report by HA Simons and the 2006 report summary prepared by PRA are the primary sources of process data for this PEA.

The payable metals in the ore include Au, Ag, Zn and Pb, roughly in order of economic importance.

The general thrust of most testwork has been to try to separate the base metals by selective flotation, and then recover the precious metals from the tails by some combination of additional flotation together with hydrometallurgical process such as oxidation and cyanidation. A number of schemes for this have been proposed and some tested. It is possible to carry out an important pre-treatment of the ore by HMS so as to reduce the mass that must be treated in the mill.

The majority of the testwork to date has focused on producing saleable lead and zinc concentrates by various flotation separation techniques. In addition, some tests have been carried out on the refractory arsenopyrite concentrates where Cashman (chloride leach), Redox (nitrate leach), batch roasting, pressure oxidation and biooxidation techniques have been used, followed by cyanidation, to recover the precious metals.

The J&L deposit is a fine grained massive polymetallic mineralized body with complex mineralogy. Major minerals present include pyrite, arsenopyrite, galena, bournonite, freibergite, sphalerite and chalcopyrite. Preliminary test work indicated that the ore is amenable to heavy media separation (HMS) whereby up to half of the feed weight can be rejected with an estimated 2% losses of the metal values. The liberation size and specific gravity for heavy media separation have not yet been optimized. The ore does require very fine primary grinding (P_{80} =44 µm) followed by regrinding to achieve only partial liberation.

Gold appears in two modes. The majority (80%-90%) occurs as solid solution within the arsenopyrite matrix. The second mode of gold occurrence (10%-20%) is as fracture filling/veinlets in arsenopyrite or as coarse grains locked to gangue or other sulphides. This secondary gold occurrence liberates on grinding, floats with the lead concentrate and is cyanide leachable. The majority of the silver is present as freibergite and is in solid solution with the lead minerals. Impurities are present either due to mineralogical makeup, or as less than 10 µm locked particles. Metal recoveries to sulphide concentrates are in the order of 98%, however, high recoveries of lead and zinc are limited because of cross-contamination resulting from fine particle locking.

Gold and silver extraction from arsenopyrite concentrates treated by pressure oxidation and cyanidation were approximately 90% and 50%, respectively. Extractions for arsenopyrite concentrates treated by biooxidation and cyanidation were approximately 90% and 80%, respectively.

13.2 METALLURGICAL TESTWORK

In May 1991, Bacon Donaldson Laboratory carried out work commissioned by Cheni Gold Mines Inc. (CGM) that focused on separating of the pyrite from arsenopyrite concentrates in order to upgrade gold grade and reduce sulphide in arsenopyrite concentrates. Bacon Donaldson developed a novel flotation process for removing a pyrite concentrate before activation of the sphalerite. It was possible to achieve 60% pyrite rejection with an arsenic content of 1%. The pyrite contained between 5% and 10% of the gold in the ore. In the same test, an arsenopyrite concentrate containing 30.4% arsenic and 35.5 g/t gold was produced. Locked cycle testing of the process was carried out before all parameters were optimized resulting in significant zinc loss to the arsenopyrite concentrate.

In June, 1991, Bacon Donaldson carried out sink float separation on two samples of the Yellowjacket Zone for CGM. Bulk samples were crushed to -½-inch and screened at 4, 8 and 14 mesh. HMS was carried out on material with specific gravities of 2.7, 2.8 and 2.9. Tests indicate that 33% to 51% of the feed weight could be rejected at less than 8% of the lead and zinc values.

A flotation procedure developed in earlier test work by Bacon Donaldson was employed to produce lead, zinc, pyrite and arsenic concentrates. In this process, a lead rougher concentrate was floated first, followed by pyrite and zinc/arsenic. The lead and pyrite rougher concentrates were reground and cleaned. The zinc/arsenic rougher concentrate was reground and separated into a zinc concentrate and an arsenic concentrate.

The arsenic concentrate was pressure leached to oxidize the arsenopyrite and to release the associated gold. Nearly complete arsenopyrite oxidation was achieved when leached at 8% solids for 3 hours at 190°C with 100 PSO oxygen over-steam pressure, and with the addition of 5 g/L H_2SO_4, 1 g/L Fe^{2+} and 1 g/L Fe^{3+}. The sulphide sulphur content in the arsenic concentrate was reduced from 18.3% to 0.21% and 92.4% of the arsenic and 97.4% of the iron were fixed in the leach residue. Cyanidation of the residue extracted 96% of the gold at a sodium cyanide consumption of 3.77 kg/t of arsenic concentrate.

A pressure leach test was also performed on a rougher tail generated from a conventional Pb-Zn rougher float circuit. The rougher tail contained both pyrite and arsenopyrite. The pressure leach was aimed at selectively oxidizing the arsenopyrite to release the gold. While the pressure leach seemed to have oxidized the arsenopyrite, cyanidation of the residue only extracted 29.2% of the Au at a sodium cyanide consumption of 8.76 kg/t.

Further test work was recommended to increase the recovery of zinc to the zinc concentrate. Based on all of the historic test work, H.A. Simons used recoveries of 80.4% Pb, 72.0% Zn, 91.8% Au and 88.1% Ag in its 1998 scoping study.

In 2005, BacTech continued metallurgical testing on the same six samples extracted by Weymin Mining Corporation. The samples were tested for bulk density and Bond Work Index (test data not available). Flotation tests were initially done using the individual samples. The three flotation concentrates were a gold-bearing arsenopyrite/pyrite concentrate, a lead concentrate, and a zinc concentrate. A locked-cycle flotation test on three-product procedure returned acceptable recoveries and product grades. The results indicated that the total gold recovery obtained was 98.7%, of which 74.4% reported to the gold-bearing arsenopyrite/pyrite concentrate, 19.8% to the lead concentrate and 4.4% to the zinc concentrate. The grade of the arsenopyrite/pyrite concentrate was 18.9 g/t Au, 16.2 g/t Ag, 16.8% As and 37.5% Fe. The lead recovery was 79.7% in the lead concentrate. It had a grade of 45.1% Pb, 18.3% Zn, 2.2% As, 28.8 g/t Au and 1,028 g/t Ag. Approximately 80% of silver was recovered into the lead concentrate. The zinc concentrate recovered 73.6% Zn in a concentrate grade of 49.6% Zn, 5.1 g/t Au, 124.9 g/t Ag and 2.0% As. In addition, gold and silver recoveries in the zinc concentrate were 4.4% Au and 12.3% Ag. Heavy liquid separation tests were done on each individual sample as a possible method of rejecting waste rock and pre-concentrating flotation feed material. These tests indicated that using a heavy medium separation process at a specific gravity of 2.96 g/cm^3, significantly reduced the volume of material to be milled at a relatively low metal loss. The three-product flotation procedure was demonstrated to be a robust and a practical procedure, suitable for use in the pre-feasibility study. However, more testing will be required to optimize the procedure to reduce the zinc content in the final lead concentrate, the lead grade in the final zinc concentrate, and the arsenic content in both the lead and zinc concentrates.

Potential issues to be resolved include the potential for penalties in smelter contracts that may materially affect economics or render concentrates unsaleable:

- As levels in zinc concentrates;
- As and Sb levels in Pb concentrates;
- Hg levels in zinc concentrates;
- Zn levels in Pb concentrates.

13.3 FUTURE TESTWORK PROPOSAL

It has been proposed that Huakan should proceed with the testwork to produce two concentrates, along with a basic bioleach and a pressure oxidation test on the gold concentrate. The results of that program will allow two flowsheet arrangements to be considered further:

Option A – single saleable concentrate (lead concentrate) flowsheet comprising:

- HMS to reduce the quantity of feed to further processing;
- Flotation of lead concentrate for sale to smelter;
- Bulk sulphide flotation concentrate produced as feed to an autoclave;
- Total pressure oxidation of the bulk sulphide concentrate;
- Cyanidation of the pressure oxidation residue for gold recovery.

Option B – Two saleable concentrates (lead and zinc concentrates) flowsheet comprising:

- HMS to reduce the quantity of feed to further processing;
- Flotation of lead concentrate for sale to smelter;
- Flotation of a zinc concentrate for sale to a smelter;
- Asenopyrite/pyrite concentrate produced as feed to an autoclave;
- Total pressure oxidation of the asenopyrite/pyrite concentrate;
- Cyanidation of the pressure oxidation residue for gold recovery.

The two flowsheets are shown in Figure 13.1 and Figure 13.2, respectively. It should be noted that for the purpose of the economic analysis of the J&L project, only Option A was evaluated further.

Figure 13.1
Schematic Flowsheet Option A

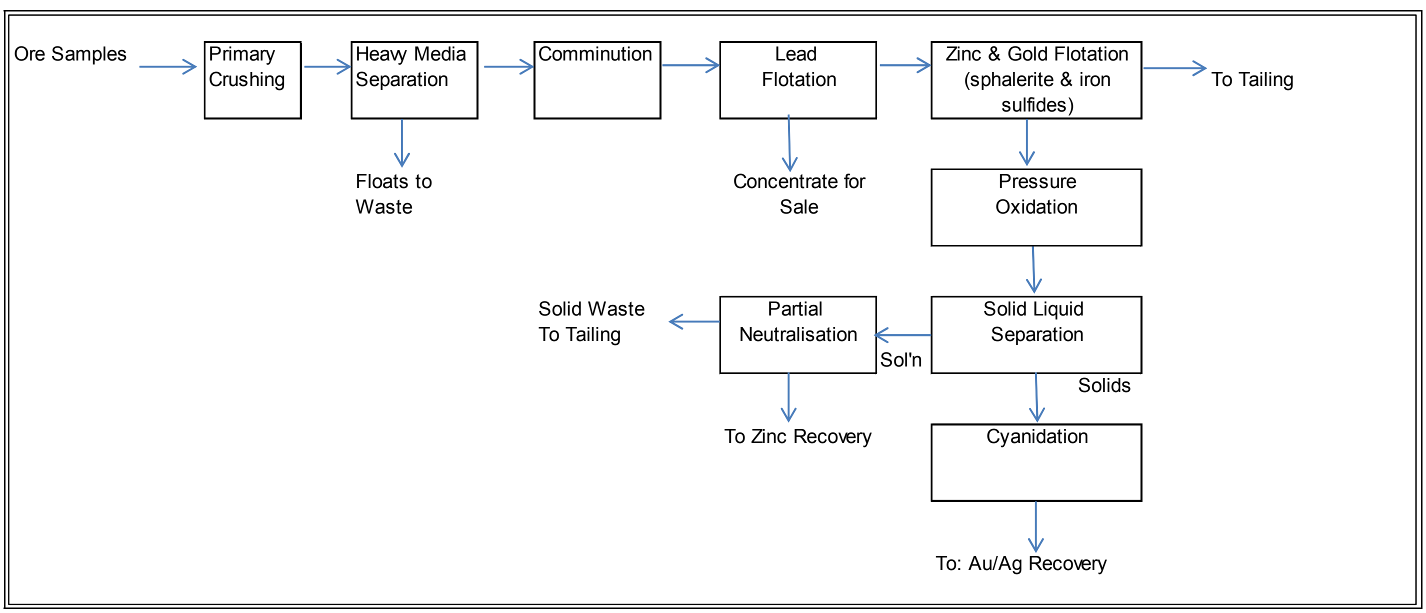
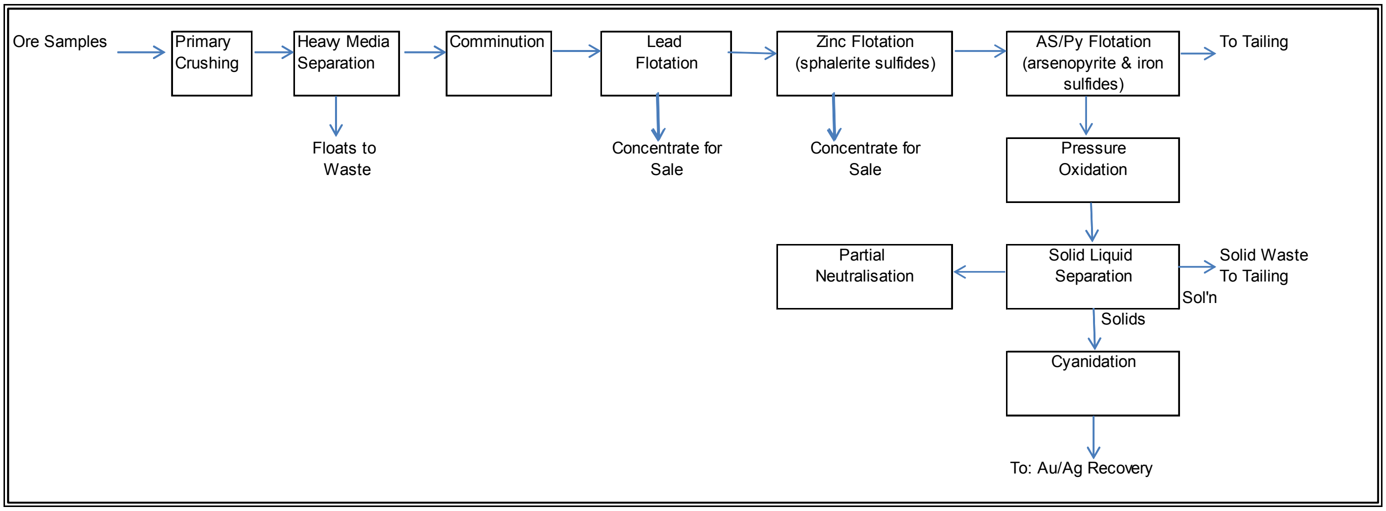


Figure 13.2
Schematic Flowsheet Option B



14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The mineral resource estimate presented herein is reported in accordance with NI 43-101 and has been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve. Confidence in the estimate of Inferred mineral resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent mineral resource estimates

All mineral resource estimation work reported herein was carried out by Fred Brown, M.Sc. (Eng), CPG, Pr.Sci.Nat., an independent Qualified Persons in terms of NI43-101, from information and data supplied by Huakan. The effective date of this estimate is May 16, 2011.

Mineral resource modeling and estimation were carried out using the commercially available Gemcom GEMS TM and Snowden Supervisor TM software programs.

14.2 DATA SUPPLIED

All sampling data were compiled by Huakan, which supplied a Microsoft Access^{TM} format database containing collar, survey, assay, specific gravity and lithology data, as well as a topographic surface and AutoCAD format wireframes of the underground workings. Huakan also supplied conceptual wireframe models of the Main Zone, Hanging Wall Zone and Footwall zone. All spatial data are relative to NAD 83 - Zone 11.

As implemented by P&E the database contains 537 records, encompassing surface trenches, underground chip sampling and drilling (summarized in Table 14.1). Of the 537 records, 29 records contained no associated assay data, were outside the project limits, or were incomplete, and therefore were not used for mineral resource estimation.

Table 14.1
Database Summary

Type	Record Count	Total Metres
Drilling	253	31,410.41
Underground Chip Sampling	223	529.15
Surface Trench Sampling	32	85.57
Not used	29	140.41

14.3 DATABASE VALIDATION

Industry standard validation checks were completed on the supplied databases. P&E typically validates a mineral resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields. Several minor out-of-sequence errors were detected and corrected. P&E believes that the supplied database is suitable for mineral resource estimation.

14.4 SPECIFIC GRAVITY

The supplied database contains a total of 396 specific gravity measurements. Huakan selected representative samples of dry halved drill core from within the Main Zone and the margins of the Main Zone. Huakan measured the dry weight of the drill core sample, and then determined the volume of displaced water from submerged drill core. Specific gravity was calculated from the ratio of the dry weight of the drillhole core to the weight of the displaced water. The supplied specific gravity measurements were used to estimate block density values (Table 14.2).

In addition, P&E collected eighteen bulk density measurements from drillhole core for verification purposes, which are in agreement with values reported by Huakan.

Table 14.2
Specific Gravity Values

	Marginal	Main Zone	P&E
Count	396	256	18
Minimum	2.61	2.66	2.76
Maximum	5.03	5	4.18
Average	3.31	3.52	3.36

14.5 ECONOMIC PARAMETERS

Based on the economic parameters listed in Table 14.3 net smelter return (NSR) value was calculated for individual assay values, which were used to construct economic mineralization domains, as well as block NSR grades. NSR values were calculated as:

NSR = (Pb% x \$17.46) + (Zn% x \$13.49) + (Ag g/t x \$0.56) + (Au g/t x \$34.91) - \$18.25

Table 14.3
J&L Economic Parameters

Element	Metal Price $US/lb or oz	Concentrate Recovery	Smelter Payable	Refining Chg. $US/lb or oz	Refining Chg. $C/lb or oz
Pb	$0.99	80%	95%	$0.00	$0.00
Zn	$0.95	72%	85%	$0.00	$0.00
Ag	$21.01	88%	91%	$0.50	$0.53
Au	$1,183	92%	96%	$15.00	$15.79

$C/$US	$0.950	
Concentration Ratio	20	Pb/Zn Blended
Smelter Treatment Charge $US/dmt	$185	Pb/Zn Blended
Concentrate Shipping Charge $C/tonne	$65	
Moisture Content	8%	

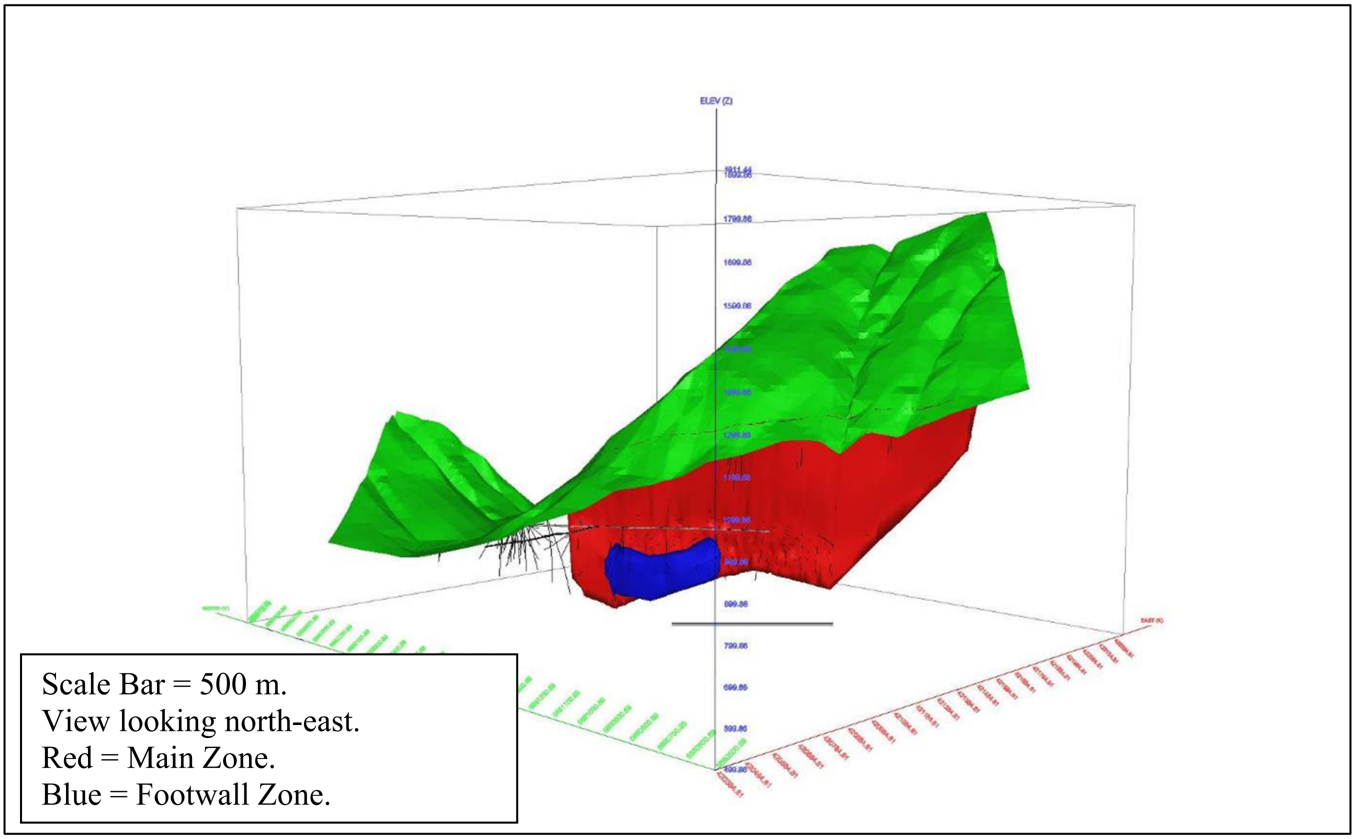
Element	Payable Metal $C/tonne/g or %
Pb	$17.46
Zn	$13.49
Ag	$0.56
Au	$34.91
TOTAL	**$66.42**

Local Ore Haulage Cost to Mill	$5.00
Smelter Treatment Charges	$9.74
Concentrate Shipping Charges	$3.51
TOTAL	**$18.25**
Mining Cost $C/t	$75.00
Processing Cost $C/t	$25.00
G&A Cost $C/t	$10.00
Cutoff $C/t	**$110.00**

14.6 DOMAIN MODELING

The Main Zone and Footwall Zone mineralization domains have been defined by Huakan along the primary structure, based on underground sampling, drilling, geological mapping and grade continuity. Based on the supplied interpretations, domain models were generated by P&E from successive polylines spaced every ten metres and oriented perpendicular to the trend of the mineralization. The outlines of the polylines were determined by the defined NSR cut-off of $110.00/t with demonstrated continuity along strike and down dip, and include low-grade material where necessary to maintain continuity between sections. All polyline vertices were snapped directly to drillhole assay intervals, in order to generate a true three-dimensional representation of the extent of the mineralization. Domain wireframes were then clipped above the topographic surface. The resulting domains were used for rock coding, statistical analysis and compositing limits. See Figure 14.1.

Figure 14.1
Isometric Projection of Mineral Resource Domains



14.7 COMPOSITING

Assay sample lengths for the Main Zone range from 0.03 m to 2.10 m, with an average sample length of 0.61 m. The mode of the sample interval length data is 0.50 m, and a compositing length of 0.50 m was therefore selected for use for mineral resource estimation.

Length-weighted composites were calculated for within the Main Zone and the Footwall Zone domains. The compositing process started at the first point of intersection between the drillhole and the domain intersected, and halted upon exit from the domain wireframe. The wireframes that represented the interpreted domains were also used to back-tag a rock code field into the drillhole workspace. Assays and composites were assigned a domain rock code value based on the domain wireframe that the interval midpoint fell within. A nominal gold grade of 0.001 g/t was used to populate a small number of un-sampled intervals. Composites that were less than 0.25 m in length were discarded so as to not introduce a short sample bias into the estimation process. The composite data were then exported to extraction files for grade estimation. Only assay values and underground channel samples were extracted for mineral resource estimation, and all trench samples were excluded.

14.8 EXPLORATORY DATA ANALYSIS

P&E generated summary statistics for the Main Zone composite data (Table 14.4) and the Footwall Zone composite data (Table 14.5). A total of 2,474 composites were generated for the Main Zone, and 79 for the Footwall Zone.

Table 14.4
Main Zone Composite Summary Statistics

	Length (m)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
Mean	0.49	5.05	51.64	1.81	3.33
Length-Weighted Mean		4.57	46.75	1.64	3.01
CV	0.08	1.48	1.34	1.46	1.38
Median	0.50	2.50	22.96	0.64	1.06
Mode	0.50	0.10	0.30	0.01	0.01
Standard Deviation	0.04	7.48	69.08	2.64	4.60
Sample Variance	0.00	55.94	4772.59	6.98	21.14
Kurtosis	17.88	88.94	7.54	6.54	5.26
Skewness	-4.27	6.15	2.37	2.37	2.00
Range	0.25	157.19	600.98	18.56	34.61
Minimum	0.25	0.00	0.00	0.00	0.00
Maximum	0.50	157.19	600.98	18.56	34.61
Count	2,474	2,474	2,474	2,474	2,474

Table 14.5
Footwall Zone Composite Summary Statistics

	Length (m)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
Mean	0.49	2.47	33.06	0.54	0.39
Length-Weighted Mean		2.51	33.51	0.53	0.38
CV	0.08	1.13	2.78	1.60	1.91
Median	0.50	1.91	11.27	0.19	0.11
Mode	0.50	1.06	3.77	0.00	0.01
Standard Deviation	0.04	2.78	91.84	0.86	0.75
Sample Variance	0.00	7.73	8434.61	0.74	0.57
Kurtosis	9.29	6.43	63.04	7.62	10.67
Skewness	-3.13	2.39	7.59	2.65	3.08
Range	0.20	13.34	796.47	4.66	4.30
Minimum	0.30	0.00	0.00	0.00	0.00
Maximum	0.50	13.34	796.47	4.66	4.30
Count	79	79	79	79	79

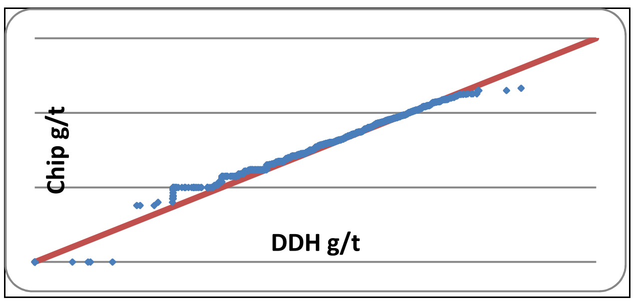
The correlation between grade elements was also examined for the Main Zone, indicating a high degree of correlation between Ag, Pb and Zn, and a moderate degree of correlation between Au and Ag as shown in Table 14.6.

Table 14.6
Main Zone Composite Correlation Matrix

	Ag	Au	Pb	Zn
Ag	1.00	0.43	0.85	0.60
Au	0.43	1.00	0.36	0.31
Pb	0.85	0.36	1.00	0.69
Zn	0.60	0.31	0.69	1.00

In addition, a comparison was made between underground chip sample values and drillhole assay values after compositing. The results indicate no significant bias between the two sample populations (Figure 14.2).

Figure 14.2
Main Zone QQ Plot for Drillhole vs. Chip Composites



14.9 TREATMENT OF EXTREME VALUES

The presence of high-grade outliers for the composite data was evaluated by a combination of decile analysis and review of probability plots. Decile analysis results indicate that minimal capping is required, with 22% of the mineral content contained in the upper decile and 7% in the upper percentile for Ag, 21% of the mineral content contained in the upper decile and 9% in the upper percentile for Au, 22% of the mineral content contained in the upper decile and 8% in the upper percentile for Pb, and 24% of the mineral content contained in the upper decile and 7% in the upper percentile for Zn (Figure 14.3). Minimal capping levels were therefore selected. Composite grades were capped to the selected threshold values prior to estimation (Table 14.7).

Figure 14.3
Decile Analysis Results

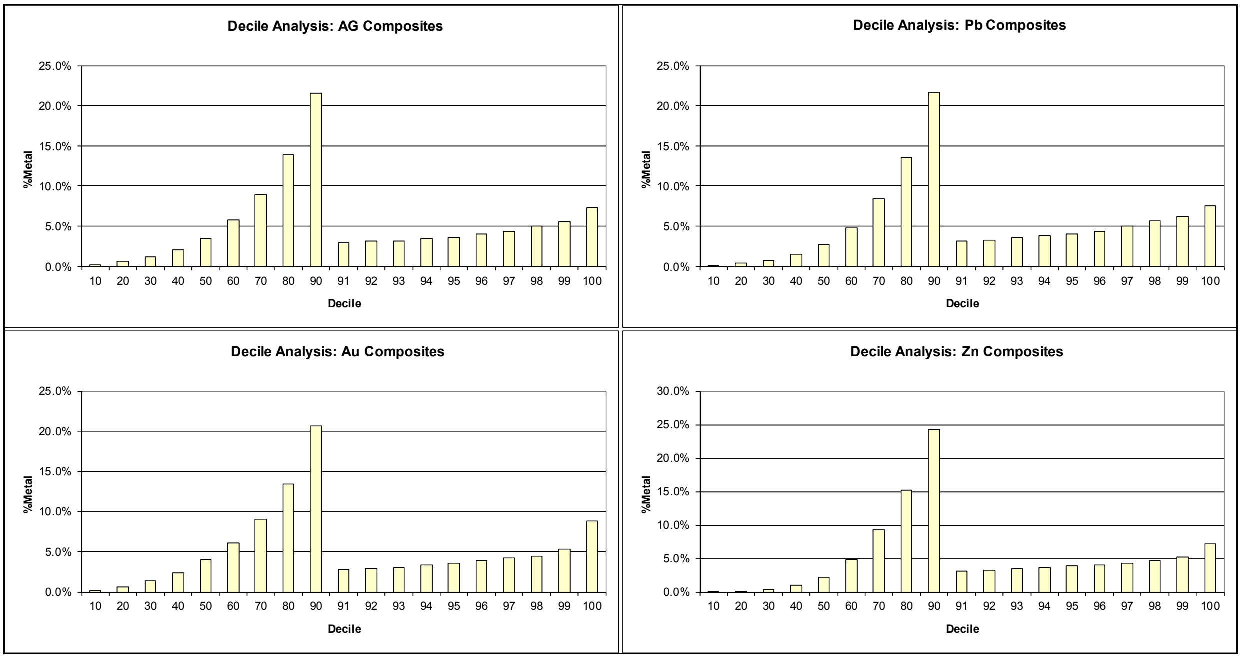


Table 14.7
Capping Thresholds

Element	Maximum	Threshold	Number Capped	Change in Metal
Ag (g/t)	600	460	2	0%
Au (g/t)	157.19	60 g/t	2	1%
Pb (%)	18.56	16%	6	0%
Zn (%)	34.61	34%	1	0%

14.10 CONTINUITY ANALYSIS

Domain-coded, composited sample data were used for continuity analysis. Strike orientations for the domains were modeled using the known geometry of the mineralization. Dip and dip plane orientations were modeled using orientations developed from variogram fans, which were assessed for geological reasonableness. Anisotropy was modeled with an average south-easterly strike and a north-easterly dip.

Based on the analysis of the resulting semi-variograms a strike distance of 60.0 m, a dip distance of 60.0 m, and a cross-dip distance of 20.0 m was selected as appropriate for mineral resource estimation. Continuity ellipses based on the observed ranges were then generated and used as the basis for estimation search ranges, distance calculations and mineral resource classification criteria.

14.11 BLOCK MODEL

A rotated block model was established across the property with the block model limits selected so as to cover the extent of the mineralized domains and the block size reflecting the generally narrow widths of the mineralized zones and the drill hole spacing (Table 14.8). The block model consists of separate models for estimated grades, rock code, percent, density and classification attributes and a calculated NSR block grade. A percent block model was used to accurately represent the volume and tonnage that was contained within the constraining grade domains. As a result, the mineral resource boundaries were properly represented by the percent model's capacity to measure infinitely variable inclusion percentages. The volume of the historical underground workings was deemed insignificant and was not depleted from the model.

Table 14.8
Block Model Setup

Dimension	Minimum	Maximum	Number	Size (m)
X	421,800.00	423,300.00	150	10
Y	5,680,100.00	5,683,100.00	300	10
Z	2,100.00	4,000.00	190	10
Rotation	-45.00			

14.12 ESTIMATION AND CLASSIFICATION

Block density values were calculated using a single pass. Anisotropic inverse distance squared (ID^2) linear weighting of between three and six bulk density values was used for the estimation of individual block density values.

Anisotropic inverse distance squared linear weighting of capped composite values was used for the estimation of block grades, with the anisotropy defined by the axes of the search ellipse.

A three-pass series of expanding search spheres with varying minimum sample requirements were used for sample selection, grade estimation and classification. Composite data used during grade estimation were restricted to samples located in their respective domains. Individual block grades were then used to calculate a NSR block model.

During the first pass, five to six composites from three or more drillholes or underground channel samples within a search ellipsoid defined by 50% of the observed continuity ranges were required for estimation.

During the second pass, three to six composites from two or more drillholes or underground channel samples within a search ellipsoid defined by 100% of the observed continuity ranges were required for estimation.

During the third pass, three to six composites from one or more drillholes or underground channel samples were required. The search ellipse was expanded to insure that all blocks within the defined domains were estimated.

Mineral resources were classified in accordance with guidelines established by the Canadian Institute of Mining, Metallurgy and Petroleum:

> Measured Mineral Resource: "A 'Measured Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape, and 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 drillholes that are spaced closely enough to confirm both geological and grade continuity."
>
> Indicated Mineral Resource: "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 drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed."
>
> Inferred Mineral Resource: "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 drillholes."

Huakan supplied detailed summaries of underground chip sampling, and believes that the information is of sufficient quality to justify the use of Measured resources (personal communication Paul Cowley). Based on the information supplied by Huakan, P&E therefore considered that there is sufficient drilling and sampling information, and that this information is of a sufficient quality, to support Measured, Indicated and Inferred classifications for the J&L Main Zone deposit.

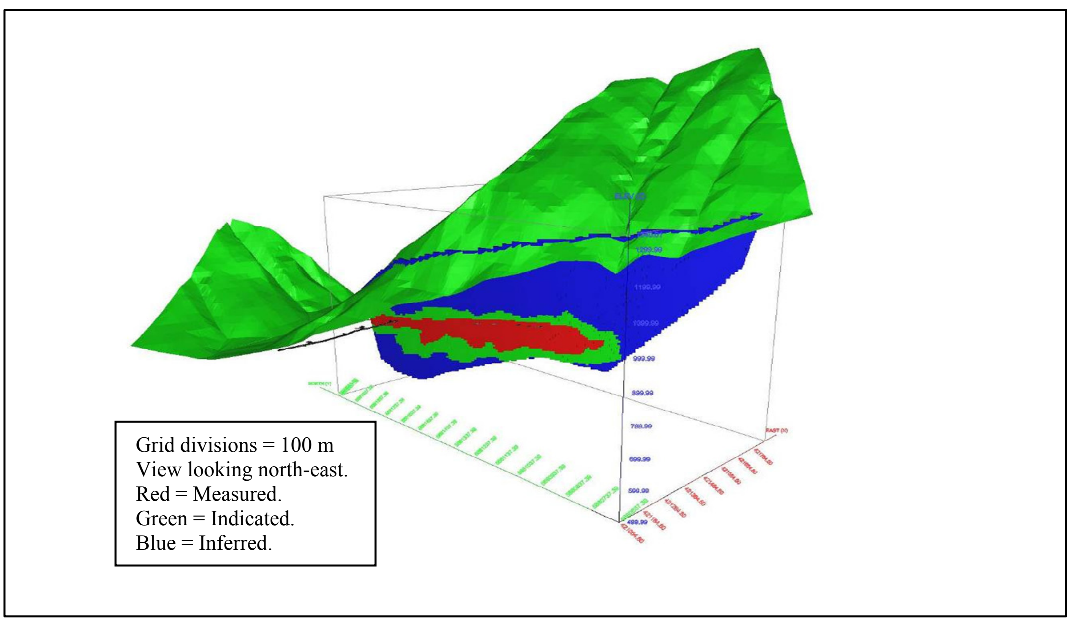
Resource classification was conducted by generating three-dimensional envelopes around those parts of the block model for which the drill hole data and grade estimates met certain criteria. The resulting classifications were iteratively refined to be geologically reasonable in order to prevent the generation of small, discontinuous areas of a higher confidence category being separated by a larger area of a lower confidence mineral resources.

Measured mineral resources were defined based on the results of the first pass, and then consolidated into a envelope digitized around the central area of blocks estimated during the first pass. This process downgraded isolated higher confidence blocks and combined the Measured mineral resources into a continuous unit.

Indicated resources were defined based on the results of the second pass, and then consolidated into a envelope digitized around the central area of blocks estimated during the second pass. This process downgraded isolated higher confidence blocks and combined the Indicated mineral resources into a continuous unit.

All remaining blocks estimated were classified as Inferred, including all blocks in the Footwall Zone (Figure 14.4).

Figure 14.4
Isometric Projection of Main Zone Block Classification



14.13 MINERAL RESOURCE ESTIMATE

The mineral resource estimate for the J&L Main Zone Deposit and Footwall Zone is reported at a NSR cut-off grade of \$110.00/t in Table 14.9, with an effective date of May 16, 2011.

Table 14.9
Mineral Resource Estimate

Main Zone							
Classification	**Resource (tonnes)**	**Au (g/t)**	**Au (oz)**	**Ag (g/t)**	**Ag (oz)**	**Pb (%)**	**Zn (%)**
Measured	1,202,000	6.71	259,200	69	2,664,600	2.4	4.46
Indicated	1,165,700	6.92	259,200	64.9	2,432,100	2.01	3.86
Measured & Indicated	2,367,700	6.81	518,400	66.95	5,096,700	2.21	4.16
Inferred	4,538,100	5.19	757,500	67.8	9,887,800	2.16	2.99
Footwall Zone							
Classification	**Resource (tonnes)**	**Au (g/t)**	**Au (oz)**	**Ag (g/t)**	**Ag (oz)**	**Pb (%)**	**Zn (%)**
Inferred	292,800	4.54	42,700	49	461,900	0.91	0.73

(1) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(2) Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters. There is no guarantee that all or any part of a mineral resource can or will be converted into a mineral reserve.
(3) The mineral resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
(4) The following parameters were used to derive the NSR block model values:

- April 30/11 US$ two year trailing avg. metal prices of Au US$1,183/oz, Ag US$21/oz, Pb US$0.99/lb, Zn US$0.95/lb
- Exchange rate of US$0.95US = CAD $1.00
- Process recoveries of Au 92%, Ag 88%, Pb 80%, Zn 72%
- Smelter payables of Au 96%, Ag 91%, Pb 95%, Zn 85%,
- Refining charges of Au US$15/oz, Ag US$0.50/oz
- Concentrate freight charges of C$65/t and Smelter treatment charge of US185/t
- Mass pull of 5% and 8% concentrate moisture content.

(5) The NSR cut-off of CAD $110 per tonne was derived from $75/t mining, $25/t processing and $10/t G&A.

14.14 VALIDATION

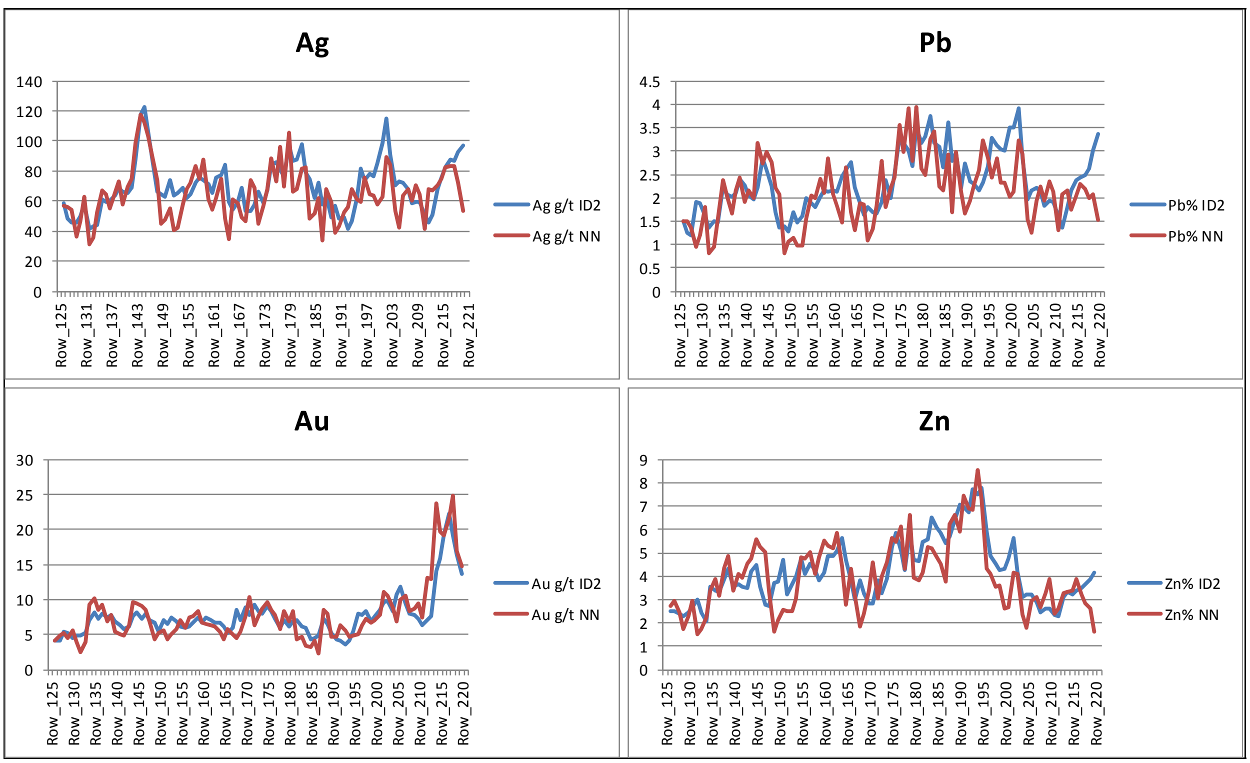
The block model was validated visually by the inspection of successive section lines in order to confirm that the block model correctly reflects the distribution of high-grade and low-grade samples. As a further check on the model the average model block grades were compared to the minimum de-clustered mean as well as the mean of the composite data (Table 14.10). No significant bias between the block model and the input data was noted.

Table 14.10
Main Zone Validation Statistics

Element	Model Mean	Composite Mean	De-clustered Mean	Length-Wt. Mean
Ag g/t	56.61	51.64	51.64	46.75
Au g/t	4.41	5.05	5.06	4.57
Pb%	1.78	1.81	1.81	1.64
Zn%	2.52	3.33	3.04	3.01

In addition, local trends were evaluated by comparing the ID^2 block estimates to a nearest neighbour estimate (NN) at zero cut-off along the strike of the Main Zone (Figure 14.5). In general the ID^2 block estimates are in good agreement with the NN estimates, and demonstrate no evidence of systematic bias in the model.

Figure 14.5
Main Zone Swath Plots



15.0 MINERAL RESERVE ESTIMATES

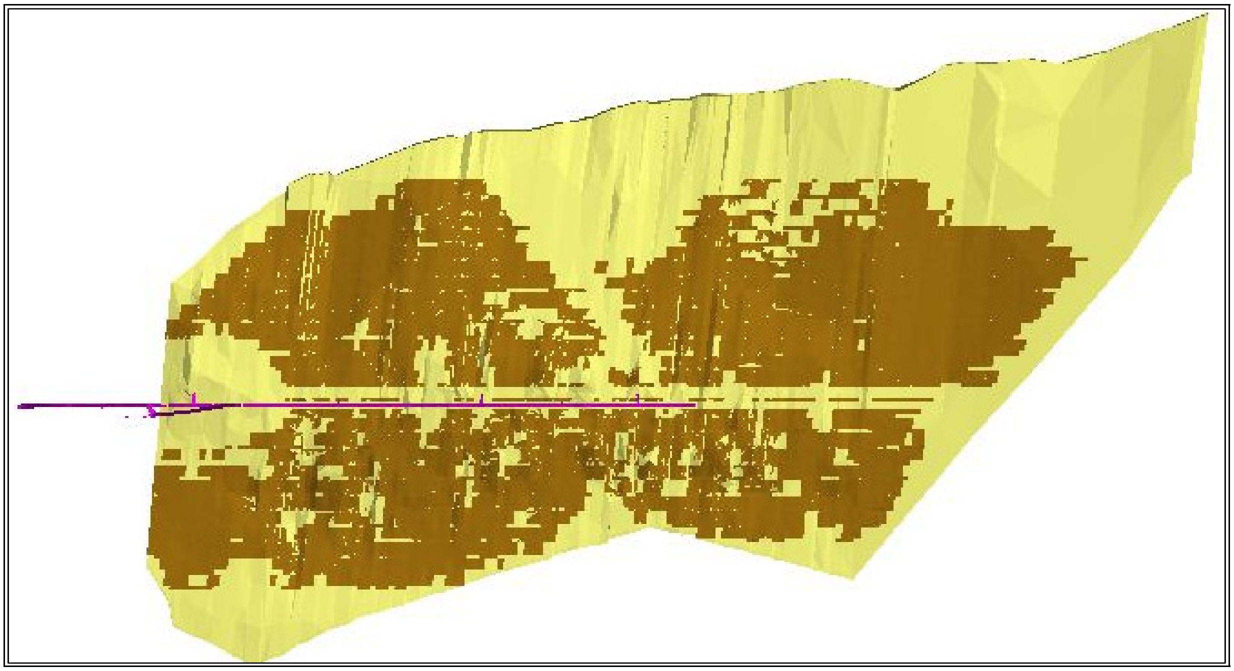
No mineral resources on the J&L Property can be classified as mineral reserves at this time.

16.0 MINING METHODS

16.1 SUMMARY

The potentially mineable portion of the J&L Main Zone mineral resource extends along the strike length of the deposit for approximately 1 km, and over 500 m vertically from the 1120 Level to the 600 Level. Figure 16.1 depicts the potentially mineable areas in tan. The currently known mineral resources, comprising the J&L Main Zone, are outlined in yellow in the figure, while the existing 830 Level development is shown in pink.

Figure 16.1
View Showing Portion of J&L Main Zone Included in the Mine Plan



The basis for the mine plan is a block model developed by P&E (Brown et al., 2011). Currently, there is an exploration drift on the 830 Level, accessed by the 832 Adit portal. There are also several other adits, none of which are anticipated to be used for mine traffic during mining operations. Portions of the mine plan, above and below the 830 Level, are referred to as the upper and lower mine, respectively.

Micon has developed a conceptual mine plan using Vulcan software to economically optimize resource extraction over the life-of-mine. The mine design is based on a net smelter return (NSR) cut-off value of $110/t. NSR was included as a variable in the P&E block model. The mine plan involves mechanized, longhole stoping with truck haulage, and backfill using cemented process tailings and development waste.

The production schedule was generated by Micon with the aid of iGantt software. At full production, just over 1,500 t/d of process feed will be mined and, during the first six years, about 1.24 million tonnes of development waste will be produced. The upper and lower mine will be developed simultaneously. As well as providing access to the resource for production purposes, development in the lower mine will establish a location from which further exploration drilling can proceed. Development in the upper mine will focus on preparing production headings to offset capital costs, including costs associated with the drilling program.

16.2 DEVELOPMENT LAYOUT

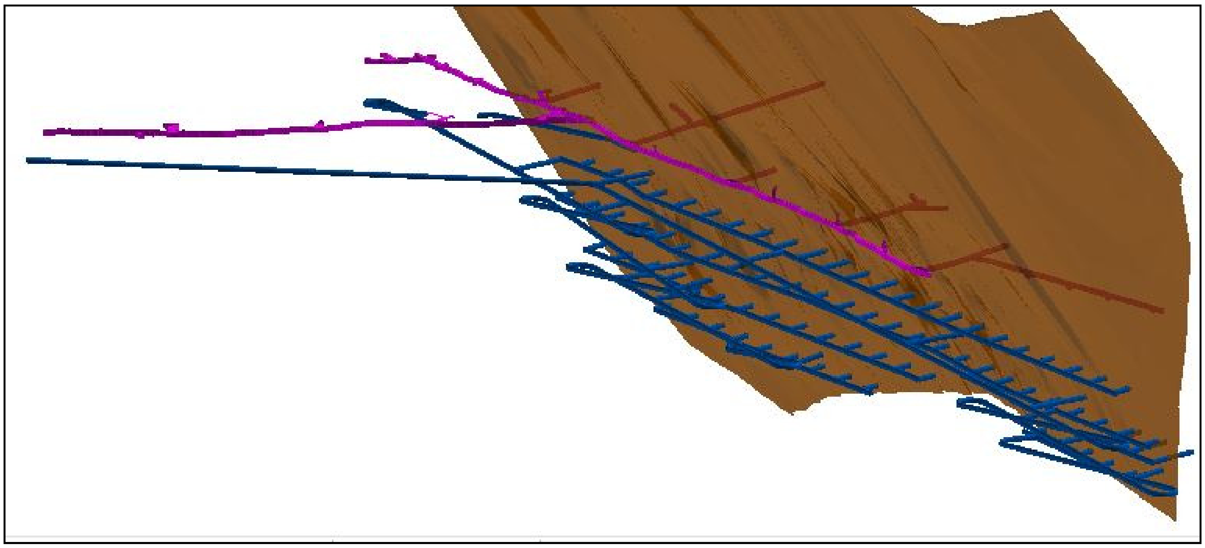
Initial waste development involves accessing both the upper and lower mine with ramps from the existing workings, as well as from a new portal. An incline, at a grade of 12%, will provide access to the upper mine from the existing 832 Adit, up to the 1120 Level.

The mine plan calls for collaring a new portal during the first year of development. This portal will increase mine productivity, as well as provide secondary egress from the mine. The location of the proposed portal is approximately 100 m south of the 832 Portal, and makes use of an existing road-cut above the PAG Waste Pile (location marked denoted by "X" in Figure 16.2). In the interest of minimizing ramp development and haulage distance, the elevation of the portal is matched as closely as possible to the surface facilities. Because of the steep terrain characteristic of the area, avalanche danger is also a consideration in choosing a suitable site for the portal. An avalanche chute can be seen in Figure 16.2, behind the proposed portal location. From this portal, the decline Ramp 1 will be driven with a gradient of -8% to access the lower mine.

Figure 16.2
Location of the Existing (832) and Proposed (Ramp 1) Portals

The priority for equipment resources dedicated to Ramp 1 is reaching and establishing the exploration drilling area at the southeast end of the 700 Level. Once this priority has been met, the ramp will be driven as far as the 660 Level. Access to the southeast end of the sublevels in the lower mine will be established from Ramp 1. A second decline, Ramp 2, will be driven from the existing 832 Drift at a gradient of -12%. Ramp 2 will access the northwest end of the lower mine, terminating at the 600 Level. Figure 16.3 depicts the conceptual layout for lower mine development. The view is from the south looking toward the north. Existing workings are shown in pink, proposed development in blue, and the mineralized zone in brown.

Figure 16.3
Schematic Showing Conceptual Plan for Waste Development in the Lower Mine



Access to the mine will be through the two portals: the existing 832 Adit portal and the proposed Ramp 1 Portal. The 832 Adit portal will initially be used for haulage, as well as transport of workers and materials. However, once breakthrough has occurred in the lower mine, connecting the two lower ramps, Ramp 1 will serve as the primary long-term haulage route for the lower mine. For this reason, Ramp 1 is designed with a milder grade. Ore transport from the upper mine will eventually require an ore pass in order to streamline mine traffic.

For the purpose of this PEA, all pre-production development, including access ramps and initial stope development, has been included as operating development, as shown in the schedule presented in Table 16.4, below. Accordingly, in Year one, a full year of operating development is planned, together with approximately six months of sill development and, in the first year, the equivalent at steady state of approximately two months of stope production. In subsequent years, a steady output at full rate is scheduled. Over the life-of-mine, approximately 22,850 m of waste development are planned, including ramps, drifts, and raises.

16.3 STOPING

Micon has designed the Main Zone stopes using the block model provided by P&E and third-party stope optimization software, supplied with the Vulcan software package. A cut-off value of $110/t for NSR and block model data (Brown et al., 2011) are input to the optimization model. Stope orientation is constrained by the geologic wireframe provided by P&E. Stope height is confined to 20-m sublevels, with 5-m high sill ore development on each level. Stope width is allowed to follow the width of the vein to a minimum of 2.5 m. Dilution is accounted for in the stope optimization process, and varies with the width of the vein, ranging between about 10% and 40%. The portion of the mineral resources included in the mine plan, with dilution taken into account, is shown in Table 16.1.

Table 16.1
Portion of the Mineral Resources included in the Mine Plan

Resource Class	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn(%)**
Measured	1,090,945	6.32	65.91	2.28	4.16
Indicated	1,046,490	7.10	66.08	2.04	3.94
Inferred	2,759,263	6.20	72.43	2.22	3.24

Longitudinally oriented stopes are planned, with cemented tailings and development waste as backfill. Mine sublevels, where drilling, blasting, and mucking will take place, are vertically spaced 20 m apart, with main haulage levels at 60-m vertical intervals. Sublevels are conservatively spaced to allow for gravity flow of the ore, given the intermediate dip of the mineralization (about 55^o, or less). Stopes are also designed to aid in gravity flow. On the haulage levels, crosscuts will be driven every 40 m along strike to access the muck.

To the extent possible, mining will progress in an overhand fashion with the lower blocks being mined first and progressively higher blocks being mined subsequently. The stoping cycle will include longhole drilling, blasting, mucking, and backfilling. Backfill plant construction will commence at the beginning of the first year, most likely on the 830 Level. Both development waste and cemented backfill will be used. Floats from the HMS plant will be available for use in backfill. PAG waste will report directly to the PAG Waste Pile and will not be used for backfilling. Combined sill development and stoping are expected to provide approximately 1,500 t/d of mill feed. Over the life-of-mine, sill development will contribute about 1.3 million tonnes and stoping will contribute about 3.6 million tonnes.

Stopes are expected to remain stable between mining and backfilling. Geomechanical evaluation by KCBL in 2007 (Klohn Crippen Berger Limited, 2007)was based on the CSIR Classification System:

> "The overall combined rating of the rock mass was 65/100, which is Class II rock and indicative of 'good rock.' The average stand-up time, for unsupported rock with a 4 m span is 6 months and the rock mass has high cohesion (>150 kPa) and a friction angle (40-45 degrees), again indicative of good quality rock."

16.4 UNDERGROUND OPERATIONS PERSONNEL

The proposed work schedule for underground operations consists of a day shift and a night shift, with 3 crews rotating every four days. The mine will operate 7 days per week. Management and technical staff will work day-shift only, 5 days per week. Table 16.2 summarizes the underground personnel requirements.

Table 16.2
Summary of Underground Operations Personnel

Description	Number
Shift Supervisors	2
Jumbo Drillers	12
Bolters	8
Longhole Drillers	4
Driller's Helpers	4
Mucker Operators	20
Truck Drivers	24
Anfo Truck/blasthole loading	8
Mine Labor	8
Mine Superintendent	1
Clerk	1
Chief Engineer	1
Mine Engineer	1
Surveyor	1
Survey Technicians	1
Chief Geologist	1
Geologist	1
Grade Control Sampler	1
Maintenance Superintendent	1
Lead Mechanic	1
Electrician	2
Welder	1
Maintenance Technician	1
Grader Operator	2
Drill Mechanic	2
Mechanic	4
Total	**113**

16.5 EQUIPMENT REQUIREMENTS

Current surface facilities include an insulated shop that is suitable for initial mine development activities (Figure 16.4). As production increases, an underground shop will need to be added to the maintenance facilities. Surface and underground laydown areas will also be needed for storage of consumables and other supplies. Basic mobile equipment needs for underground operations are listed in Table 16.3.

Figure 16.4
Existing Surface Maintenance Facilities

Table 16.3
Proposed Mining Equipment Fleet

Description	Number
3-Boom Jumbo Drill	1
2-Boom Jumbo Drill	2
Longhole Drill	1
Bolter	2
Haul Truck (20-Ton Capacity)	4
Haul Truck (30-Ton Capacity)	2
Remote LHD (6 CY Capacity)	2
LHD (6 CY Capacity)	2
LHD (3.5 CY Capacity)	1
Scissor Lift	1
Anfo Loader	2
Lube Truck	1
Crane Truck	1
Personnel Carrier	2
Grader	1
Utility Tractor	2
Total	27

Note: Some items are currently on site and are sunk costs.

Power demand for the underground mine is expected to be approximately 32,000 MWh/y. This includes power supply for the shop, yard, underground operations, pump station and compressor. A surface load centre will feed the shop, office, and other miscellaneous surface support facilities. Underground power supplied at 4,160-V will be distributed through a 1,000-kVA main mine load centre and several 500-kVA load centres. These will run the underground pump station, sumps, fans and other electrically-powered equipment.

16.6 SCHEDULE

Micon has used iGantt and Vulcan software to create the mine schedule. Development headings are prioritized to meet production goals, and mine production is scheduled such that four different areas of the mine can be mined independently. Higher productivity is realized by effectively dividing the mine into four separate production quadrants, opening multiple development and production headings simultaneously.

Sill ore production will commence in both the upper and lower mine during the first year. The first stopes will be taken in the upper mine during the second year of mining. As more production areas become available, production will increase to about 1,570 t/d by the third year. Production is expected to remain consistent throughout the mine life.

A summary of the annual mine development and production is shown in Table 16.4.

Table 16.4
Life-of-Mine Production Schedule

	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
Development	m	3,975	3,975	3,975	3,975	3,975	3,975	-	-	-	-	23,850
Development	t	206,700	206,700	206,700	206,700	206,700	206,700	-	-	-	-	1,240,200
Sills	t	92,000	182,500	182,500	183,000	182,500	182,500	171,150	111,776	-	-	1,287,926
Stoping	t	60,161	343,200	366,765	367,770	366,765	366,765	368,068	425,426	533,749	410,104	3,608,772
Prod (Sills+Stopes)	t	152,161	525,700	549,265	550,770	549,265	549,265	539,218	537,202	533,749	410,104	4,896,698
Au Grade	g/t	8.28	8.64	6.32	4.91	5.30	5.54	7.51	6.36	6.29	6.49	6.42
Ag Grade	g/t	63.22	70.20	58.51	60.95	80.62	81.95	69.80	72.91	69.03	64.68	69.78
Pb Grade	%	2.10	2.15	1.56	1.89	2.95	2.46	2.40	2.27	2.12	1.97	2.21
Zn Grade	%	3.21	2.99	2.31	3.18	4.50	4.20	4.58	3.89	3.56	3.19	3.61
NSR	$/t	386	400	293	263	324	321	387	337	325	322	334
Development Advance Rate (350 d/y)	m/d	11.36	11.36	11.36	11.36	11.36	11.36	-	-	-	-	
Development Rate (350 d/y)	t/d	590.57	590.57	590.57	590.57	590.57	590.57	-	-	-	-	
Production Rate (350 d/y)	t/d	434.75	1,502.00	1,569.33	1,573.63	1,569.33	1,569.33	1,540.62	1,534.86	1,525.00	1,171.73	

17.0 RECOVERY METHODS

17.1 SUMMARY

The process flowsheet options reviewed for this PEA study are shown in Figure 17.1 and Figure 17.2, respectively. Notable features are the HMS which will remove 40% of the rock by weight with negligible metal losses of 2%. The main difference between the two options is whether a single saleable concentrate (Pb) or two saleable concentrates (Pb and Zn) are produced.

Each option involves a similar set of processing steps including crushing, HMS, grinding, flotation, oxidation, and cyanidation. The main source of assay data for this report is a study dated November 2, 2006, prepared for BacTech by PRA. This study used the closest flowsheet arrangement to the recommended program today, and was conducted on samples which had been pre-concentrated using heavy liquid separation similar to earlier flotation testwork conducted in 1998. The conceptual plant design for the single saleable concentrate is based on the results of testwork carried out by PRA on ore types making up a composite sample with a head grade of 8.1g/t Au, 72.4 g/t Ag, 3.2% Pb, and 4.8% Zn. In comparison, Micon estimates the life-of-mine (LOM) average feed grades for this PEA as follows: 6.42 g/t Au, 69.8 g/t Au, 2.2% Pb, and 3.6% Zn.

Unfortunately, only the summary of this referenced PRA report was available through Huakan. Thus, additional process parameters (including suitable HMS density, reagent consumption, etc.) have been identified from the earlier October 1998 report prepared by H.A. Simons. The Simons report was based on testwork conducted by PRA under the direction of Beattie Consulting Ltd.

Table 17.1 summarizes the milling plant production.

Table 17.1
Production Summary for the Huakan's J&L Milling Plant

Item	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
Tonnes Processed Annually	t/y	540,000	540,000
Saleable Pb Concentrate Production, Average	t/y	19,680	19,680
Saleable Zn Concentrate Production, Average	t/y	-	25,210
Gold Production, LOM Average	oz/y	81,718	77,155
Silver Production, LOM Average	oz/y	115,756	42,741
Key Operating Parameters			
Operating Days Per Year		360	360
Mill Feed Rate	t/d	1,500	1,500
Chemical & Metallurgical Parameters			
Concentrate Grade, Pb	%	45	45
Concentrate Grade, Zn	%	-	53.5

Figure 17.1
Option A Concentrator Flowsheet

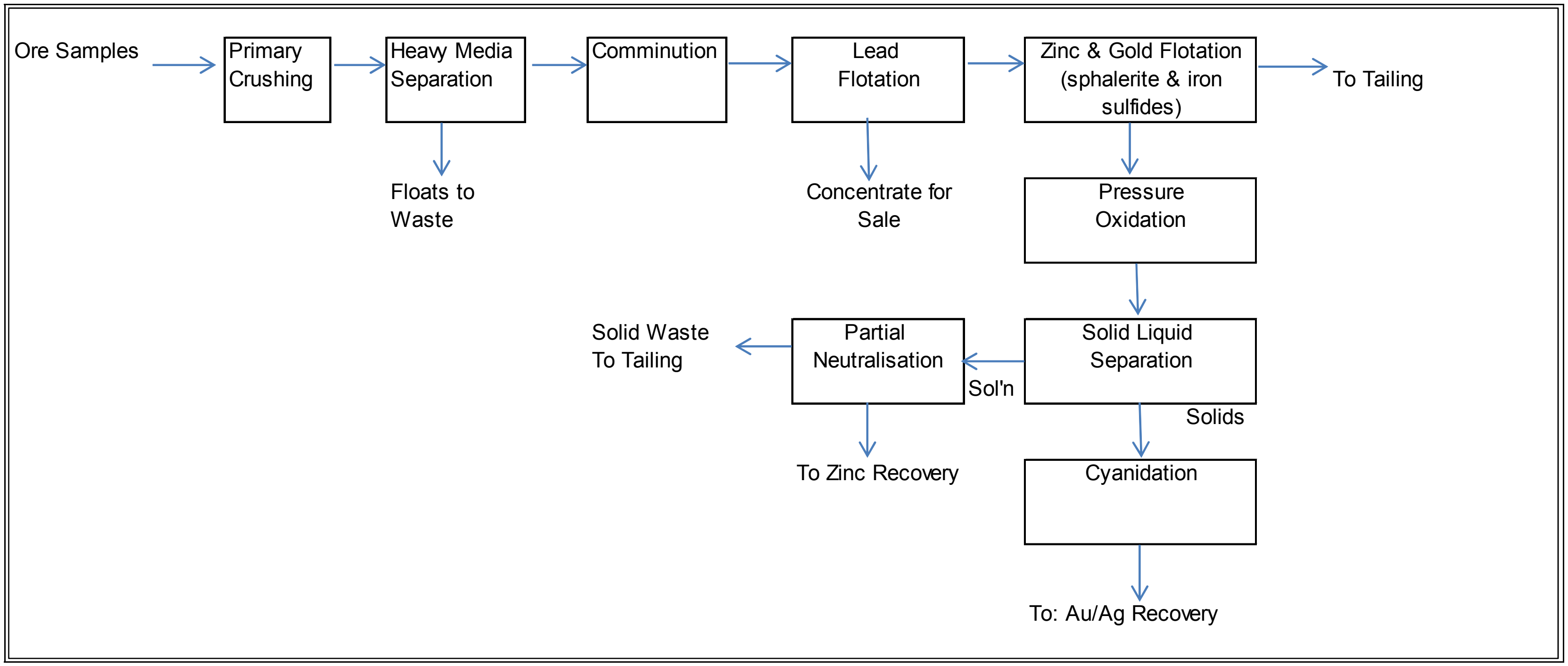
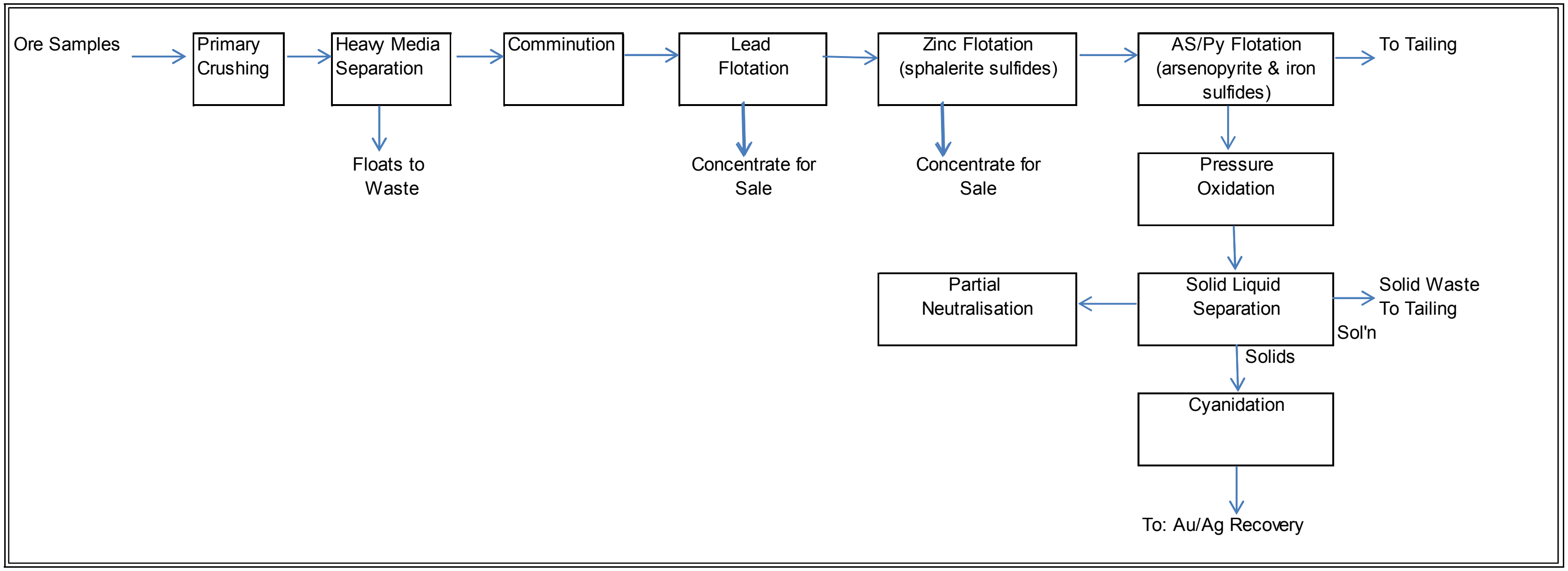


Figure 17.2
Option B Concentrator Flowsheet



17.2 PROCESS DESCRIPTION

Table 17.2 provides an overview of the process design criteria.

Table 17.2
Process Design Criteria

Description	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
General Feed Characteristics			
Annual Processing Rate	t/y	540,000	540,000
Operating Days Per Year	d/y	360	360
Daily Processing Rate	t/d	1,500	1,500
Average Feed Grades	%Pb	2.20	2.20
	%Zn	3.60	3.60
	g/t Au	6.42	6.42
	g/t Ag	69.78	69.78
Specific Gravity	t/m^3	3.36	3.36
Crushing			
Crusher Rate - Nominal	t/h	62.5	62.5
Crusher Operating Time	h/d	16	16
Crusher Availability	%	80	80
Crushing Rate - Design	t/h	117.2	117.2
Primary Crusher Type		Jaw	Jaw
Primary Crusher Size	mm	610 x 914	610 x 914
Primary Crusher Closed-Side Setting	mm	64	64
Secondary Crusher Type		Standard	Standard
Secondary Crusher Size	mm	914	914
Secondary Crusher Closed-Side Setting	mm	16	16
Secondary Screen Passing Size	mm	25	25
Crushed Ore Product P_{80}	mm	18.3	18.3
Heavy Media Separation			
Feed tonnage to Wet Screen - Nominal	t/d	1,500	1,500
Heavy Media Circuit Availability	%	90	90
Feed tonnage to Wet Screen – Design	t/d	1,667	1,667
Feed Rate to Wet Screen - Design	t/h	69.4	69.4
Wet Screen Mesh Size	mm	3.2	3.2
Percent Wet Screen Fines	%	20	20
Wet Screen Fines to Grinding	t/h	13.9	13.9
Feed Rate to Heavy Media Drum	t/h	55.6	55.6
HMS Float Sink (Product)	% wt. of	60	60
HMS Float Reject	% wt. of	40	40
HMS Float Reject	t/h	22	22
HMS Media Type		Ferrosilicon	Ferrosilicon
HMS Media Specific Gravity	t/m^3	2.9	2.9
Grinding			
Total Feed Tonnage to Grinding - Nominal	t/d	900	900
Mill Availability	%	90	90
Total Feed Tonnage to Grinding - Design	t/d	1000	1000
Feed Rate to Grinding - Design	t/h	41.7	41.7

Description	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
Rod Mill Feed F_{80}	micron	17,500	17,500
Rod Mill Discharge P_{80}	micron	800	800
Rod Mill Work Index	kWh/t	11	11
Rod Mill Unit Power Consumption	kWh/t	4.29	4.29
Rod Mill Installed Power	kW	239	239
Cyclone Overflow P_{80}	micron	65	65
Ball Mill Work Index	kWh/t	11.5	11.5
Ball Mill Unit Power Consumption	kWh/t	10.79	10.79
Ball Mill Installed Power	kW	524.4	524.4
$ZnSO_4.7H_2O$ to Grinding	g/t	1,000	1,000
$Na_2S_2O_5$ to Grinding	g/t	1,000	1,000
Lead Flotation and Regrinding			
Feed Rate to Lead Flotation – Design	t/h	41.7	41.7
Pb Flotation Feed Density	% solids	40	40
Pb Conditioning & Aeration Time	min	10	10
Pb Conditioning and Aeration pH	g/t	9.0-9.5	9.0-9.5
Lime to Pb Conditioning and Aeration	g/t	297	297
Sodium Ethyl Xanthate Collector to Pb Roughing	g/t	30	30
A242 Collector to Pb Roughing	g/t	5	5
MIBC Frother to Pb Roughing	g/t	9	9
Pb Rougher Retention Time	min	12	12
Pb Rougher Cell Volume	m^3	42.5	42.5
Pb Rougher Concentrate Tonnage	t/h	12.47	12.47
Pb Concentrate Regrind Work Index	kWh/t	22	22
Pb Concentrate Regrind COF P_{80}	micron	25	25
Pb Concentrate Regrind Unit Power	kWh/t	20.67	20.67
Pb Concentrate Regrind Installed Power	kW	261.9	261.9
Pb 1st Cleaner Retention Time	min	12	12
Pb 2nd Cleaner Retention Time	min	12	12
Pb 3rd Cleaner Retention Time	min	12	12
Pb 4th Cleaner Retention Time	min	12	12
Pb 5th Cleaner Retention Time	min	8	8
Lime to Pb 1st to 5th Cleaning	g/t	60	60
Sodium Ethyl Xanthate to Pb 1st to 5th Cleaning	g/t	20	20
MIBC to Pb 1st to 5th Cleaning	g/t	18	18
NACN to Pb 1st to 5th Cleaning	g/t	40	40
Lead Concentrate Mass Pull of HMS sink	%	6.13	6.13
Lead Concentrate Production – Annual	t/y	19,680	19,680
Lead Concentrate Production – Design	t/d	57	57
Lead Concentrate Grade	%Pb	45	45
Lead Concentrate Recovery	%Pb	78.0	78.0
Lead Tailings – Design	t/d	943	943
Zinc Flotation			
Feed Rate to Zinc Flotation - Design	t/h	N/A	39.3
Zn Rougher Feed Density	% solids	N/A	34.02
Zn Conditioning Time	min	N/A	5
$CuSO_4.5H2O$ to Zn Conditioning	g/t	N/A	1,000
SIPX to Zn Conditioning	g/t	N/A	110

Description	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
DF250 to Zn Conditioning	g/t	N/A	22
Zn Rougher Retention Time	min	N/A	36
Zn Rougher Concentrate Tonnage	t/h	N/A	21.64
Zn Concentrate Regrind Work Index	kWh/t	N/A	25
Zn Concentrate Regrind COF P_{80}	micron	N/A	25
Zn Concentrate Regrind Unit Power	kWh/t	N/A	22.41
Zn Concentrate Regrind Installed	kW	N/A	522
Lime to Zn Regrind	g/t	N/A	1,000
$CuSO_4.5H2O$ to Zn Regrind	g/t	N/A	350
SIPX to Zn 1st Cleaner	g/t	N/A	20
NaCN to Zn 1st Cleaner	g/t	N/A	30
MIBC to Zn 1st Cleaner	g/t	N/A	6
Zn 1st Cleaner	min	N/A	12
Lime to Zn 2nd to 4th Cleaner	g/t	N/A	744
NaCN to Zn 2nd to 4th Cleaner	g/t	N/A	30
SIPX to Zn 2nd to 4th Cleaner	g/t	N/A	5
MIBC to Zn 2nd to 4th Cleaner	g/t	N/A	19
Zn 2nd Cleaner Retention Time	min	N/A	12
Zn 3rd Cleaner Retention Time	min	N/A	10
Zn 4th Cleaner Retention Time	min	N/A	10
Zn Concentrate Mass Pull of HMS sink	%	N/A	7.78
Zn Concentrate Production – Annual	t/y	N/A	25,210
Zn Concentrate Production – Design	t/d	N/A	77.8
Zinc Tailings – Design	t/d	N/A	865
Zn Concentrate Grade	%Zn	N/A	54
Zn Concentrate Recovery	%Zn	N/A	72.1
Concentrate Dewatering			
Pb Concentrate Solids Feed Rate	t/h	2.38	2.38
Pb Concentrate Thickener Unit Area	t/d/m^2	1	1
Pb Concentrate Thickener Design Factor		1.54	1.54
Pb Concentrate Thickener Diameter	m	10.6	10.6
Pb Concentrate Unit Filtering Rate	kg/m^2/h	150	150
Pb Concentrate Filter Availability	%	80	80
Pb Concentrate Filter Design Factor		1.58	1.58
Pb Concentrate Pressure Filter Area	m^2	31	31
Zn Concentrate Solids Feed Rate	t/h	N/A	3.04
Zn Concentrate Thickener Unit Area	t/d/m^2	N/A	1
Zn Concentrate Thickener Design Factor		N/A	1.45
Zn Concentrate Thickener Diameter	m	N/A	11.6
Zn Concentrate Unit Filtering Rate	kg/m^2/h	N/A	150
Zn Concentrate Filter Availability	%	N/A	80
Zn Concentrate Filter Design Factor		N/A	1.38
Zn Concentrate Pressure Filter Area	m^2	N/A	35
Bulk Sulphide Flotation			
Feed rate to Bulk Sulphide Flotation- Design	t/d	943	865
Mass Pull to Flotation Product	% of feed	49.5	45.2
Bulk sulphide product – Design	t/d	467	391
Bulk sulphide production annual	t/y	151,238	126,678

Description	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
Bulk sulphide tailings annual	t/y	153,082	152,432
Pressure Oxidation			
Pressure Oxidation Feed Rate - Nominal	t/d	420	352
Pressure Oxidation Availability	%	90	90
Pressure Oxidation Feed Rate - Design	t/d	467	391
Pressure Oxidation Feed Rate	t/h	19.5	16.3
Oxygen Pressure, over steam	kPa	700	700
Pressure Oxidation Feed Pulp Density	% solids	13	13
Design retention Time	h	0.5	0.5
Autoclave Temperature	°C	220	220
Steam Pressure	kPa	2100	2100
Total Operating Pressure	kPa	2800	2800
Design Operating Pressure	kPa	3200	3200
Design Temperature	°C	235	235
Fill Factor, slurry volume % of total	%	72	72
Slurry Feed % solids	%	13	13
Slurry Feed Volume - Design	m^3/h	150	125
Autoclave Active volume required	m^3	75	62.5
Autoclave total volume inside lining	m^3	104	87
Number of compartments in Autoclave		5	5
Volume per compartment	m^3	20.8	17.4
ID of Autoclave inside lining	m	2.982	2.810
OD of Autoclave shell	m	3.282	3.110
Length of Autoclave shell	m	16.4	15.6
Oxygen Demand, t O_2/t feed	%	68	73
Oxygen Demand tonne/day – Design	t/d	322	286
Oxygen Demand tonne/day – Design	t/h	13.4	11.9
Oxygen usage efficiency in Autoclave	%	90	90
Oxygen requirements	t/h	14.9	13.2
Contingency on Oxygen plant design	%	15	15
Oxygen Plant design	t/h	17.1	15.2
Oxygen Plant design	t/d	412	365
Power Demand for Oxygen Plant	kWh/t O_2	400	400
Power Demand for Oxygen Plant	MW	6.9	6.1
Power for Agitators as % of O2 plant	%	15	15
Power for Agitators	MW	1.0	0.9
Power for other Hydromet Plant % of PO	%	20	20
Total Power required for Hydromet Plant	MW	9.5	8.4
Solids residue from POX filtration assumed	% of feed POX	150	160
Solids production from POX filter - Design	t/h	29.3	26.1
Cyanide Leaching and CIL			
Feed Rate to Cyanide Leaching - Nominal	t/h	26.4	23.5
Cyanide Leaching Availability	%	90	90
Feed Rate to Cyanide Leaching - Design	t/h	29.3	26.1
Leach Slurry Density	% solids	40	40
Leach Slurry Flow Rate	m^3/hr	59	47
Leach Residence Time	h	6	6
Number of Leach Tanks	#	6	6

Description	Units	Option A One Saleable Concentrate	Option B Two Saleable Concentrates
Dimensions of Leach Tanks	dia. x height, m	3.9 x 3.9	3.7 x 3.7
Cyanide Concentration	g/L	2	2
Cyanide Consumption	kg/t	6	6
Lime Consumption	kg/t	10	10
Effluent Treatment			
Feed Rate to Cyanide Destruction - Nominal	t/h	26.4	23.5
Effluent Treatment Availability	%	90	90
Feed Rate to Cyanide Destruction - Design	t/h	29.3	26.1
Slurry Density	% solids	35	35
Slurry Flow Rate	m^3/h	70	62
Cyanide Destruction Residence Time	h	2	2
No. Cyanide Destruction Reactor Tanks		3	3
Cyanide Destruction Tank Dimensions	dia. x height, m	4.0 X 4.5	3.5 x 4.0

17.2.1 Crushing

The ore is delivered by mine trucks to the plant and fed through a grizzly with 400 mm openings to the ROM feed hopper. The ore is extracted from the feed hopper by an apron feeder and fed to a jaw crusher with an opening of 18-20 mm. The crushed ore is discharged onto a conveyor which will transport the ore to the plant. Secondary Crusher product is 80% passing 18.3 mm.

17.2.2 Heavy Media Separation

HMS may be used to selectively remove non-sulphide gangue materials by applying an SG of 2.95-3.00. By removing the lower grade material prior to the grinding circuit, the HMS plant allows the mill feed tonnage to be increased without having to increase the grinding or flotation capacity.

The HMS plant is fed with ROM material from the storage bin by variable-speed drum feeders discharging onto conveyors, where the ore is measured by weightometer and tramp metal is removed by an electromagnet.

The separators are filled with a bath of ferrosilicon slurry at a specific gravity of 2.9. The low specific gravity waste floats on the bath, while the high specific gravity ore sinks. Approximately 40% of the mill feed is removed as float with less than 2% of the contained metals being lost.

Float is removed from the separators by paddles and laundered to drainage screens with drilled holes for medium recovery. The recovered medium is circulated back into the separator, while the float rock crosses washing screens (with drilled holes), where adhering medium is removed using water sprays. Washed float passes over a weightometer to a stockpile outside the plant. Float can be reclaimed by a loader and dump trucks and used for construction, reclamation, or backfill activities.

Sink is removed from the separators by bucket elevators and fed to single deck screens. Here the medium on the sink drains and is washed off. The sink is then transferred by conveyor to a tripper and into a live bin ahead of the rod mill.

17.2.3 Grinding

The HMS concentrate reporting to the grinding circuit, comprises 60% of the feed to the HMS circuit or 900 t/d of coarse sink. The primary grinding rod mill discharge product typically has a size passing of 800 µm.

The secondary grinding ball mill will operate in closed circuit with a cyclone, with the target cyclone overflow 80% size passing 65 µm.

17.2.4 Lead Flotation

The ball mill ore slurry discharge, at 40% solids, is fed to a conditioning tank, which then overflows to the lead rougher flotation cells. The conditioning tank also acts to buffer flotation from upsets in the sink/float and grinding areas.

The lead flotation circuit is a conventional lead-zinc differential flotation circuit. The initial lead flotation tails are cycloned, with the oversize returning to the ball mill and the fines reporting as feed to the lead scavenger circuit.

Aero242, Xanthate and cyanide are added to the roughers, cleaners and regrind circuit. MIBC (methyl isobutyl carbinol) is used as a frother, and lime is used as a pH modifier.

The concentrate from the rougher cells is fed to the cleaner cells, from which the final concentrate is sent to a pressure filter for dewatering.

17.2.5 Zinc Flotation

Lead rougher tailing, lead-scavenger tails, and zinc spill returns are conditioned with copper sulphate and lime in a conditioner. The overflow is treated with lime to raise the pH, and mixed with xanthate, Dowfroth 250 and MIBC, and pumped to the zinc roughers. The zinc rougher tailing is discarded from the circuit as final flotation tails. The final concentrate is pumped to the dewatering circuit.

The cleaner-scavenger tails are combined with the zinc rougher tails and pumped to the tailings pond.

17.2.6 Dewatering

Lead concentrate is thickened to 80% solids in a 10.6-m diameter thickener. Lead thickener overflow is recycled to the water reclaim tank. This reclaim water tank can serve as a head tank for the reclaim water pumps and provide the bulk of process water to the mill. It also serves as the water supply for the emergency diesel-powered fire pump.

Zinc concentrate is thickened to 70% solids, using an 11.6-m diameter thickener. Zinc thickener water is also returned to the reclaim water tank. A de-aeration sump is located immediately before the zinc thickener to promote breakdown of the froth and improve the effectiveness of flocculation and subsequent settling within the thickener.

Concentrates are filtered using pressure filters with an area of 31 m^2 for lead concentrate, and (for flowsheet option B) 35 m^2 for zinc concentrate.

17.2.7 Bulk Sulphide Flotation

For Option A, tails from lead flotation are run through bulk sulphide flotation to recover all remaining sulphides, including gold and silver bearing minerals, sphalerite, arsenopyrite, and pyrite.

The flotation concentrate is thickened, filtered, and sent to pressure oxidation.

For Option B, the same process is carried out on the zinc flotation tails.

17.2.8 Pressure Oxidation

For Options A and B, total pressure oxidation (PO) of the bulk sulphide concentrate is carried out in an autoclave, using oxygen produced in a dedicated plant on site. The resulting autoclave discharge slurry is neutralized using limestone. The PO residue is then filtered, to produce a solids feed to cyanidation.

For option A, zinc recovery from the neutralization filtrate is realized either by lime neutralization (to precipitate zinc hydroxide), sulphide precipitation, or Zn solvent extraction (ZnSX), to recover a saleable Zn product.

This PEA assumes precipitation of zinc hydroxide using lime. Filtrate from Zn recovery is neutralized with lime and discharged to tailings or recycled to PO.

For Option B, the same process is used, except that no zinc recovery circuit is needed.

17.2.9 Cyanidation

A six-stage carbon-in-leach (CIL) circuit is designed based on a retention time of 6 hours and a new carbon concentration of 20 g/L. Solids from filtration of the PO residue are slurried and fed to the train of six CIL tanks in series, where the slurry is leached using sodium cyanide in the presence of activated carbon. Each of the tanks has an in-tank screen to allow the slurry to flow by gravity through the CIL train, while retaining the carbon in the tanks. The carbon is transported counter-current to the slurry flow by means of carbon advance pumps. The loaded carbon from the first CIL tank is collected on a screen and is sent to the carbon stripping circuit. The slurry leaving the last CIL tank is passed over a safety screen to capture any carbon particles before it is sent to the cyanide destruction circuit.

17.2.10 Carbon Stripping/Carbon Reactivation/Gold Room

A carbon stripping/carbon reactivation circuit treats the loaded carbon to remove gold and silver, and to recycle the carbon to the CIL circuit.

The loaded carbon is acid-washed and then fed to a pressurized Zadra circuit to strip gold and silver from the carbon. The pregnant solution is then sent to two electrowinning cells to recover the gold and silver.

The carbon from the stripping circuit is fed to a diesel-fired rotary kiln to reactivate the carbon which is passed over a sizing screen prior to being recycled to the CIL circuit.

The stainless steel wool, containing the gold from the electrowinning cells, is fed to an induction furnace to produce gold doré bars for sale.

17.2.11 Cyanide Destruction

A cyanide destruction circuit is used to treat the slurry discharge from the CIL circuit.

The CIL discharge slurry is pumped to two agitator tanks in series, and SO_2 and air are added to destroy the cyanide before the slurry is pumped to the tailings disposal area.

17.3 PROCESSING PLANT LAYOUT

The planned location of the mill concentrator is shown in Figure 17.3. By locating the mill above the tailings storage facility, gravity can be used to assist the flow of the tailings slurry. Ore will be hauled in trucks from the mine to the crusher feed bin at the mill. Material is then processed through the plant with concentrate load out located on the north side of the mill, and tailings gravity fed from the west side of the mill into the proposed tailings storage facility.

17.4 PROCESSING PLANT PERSONNEL REQUIREMENTS

Table 17.3 summarizes the processing plant personnel requirements.

Figure 17.3
Proposed Processing Plant and Tailings Storage Facility Locations

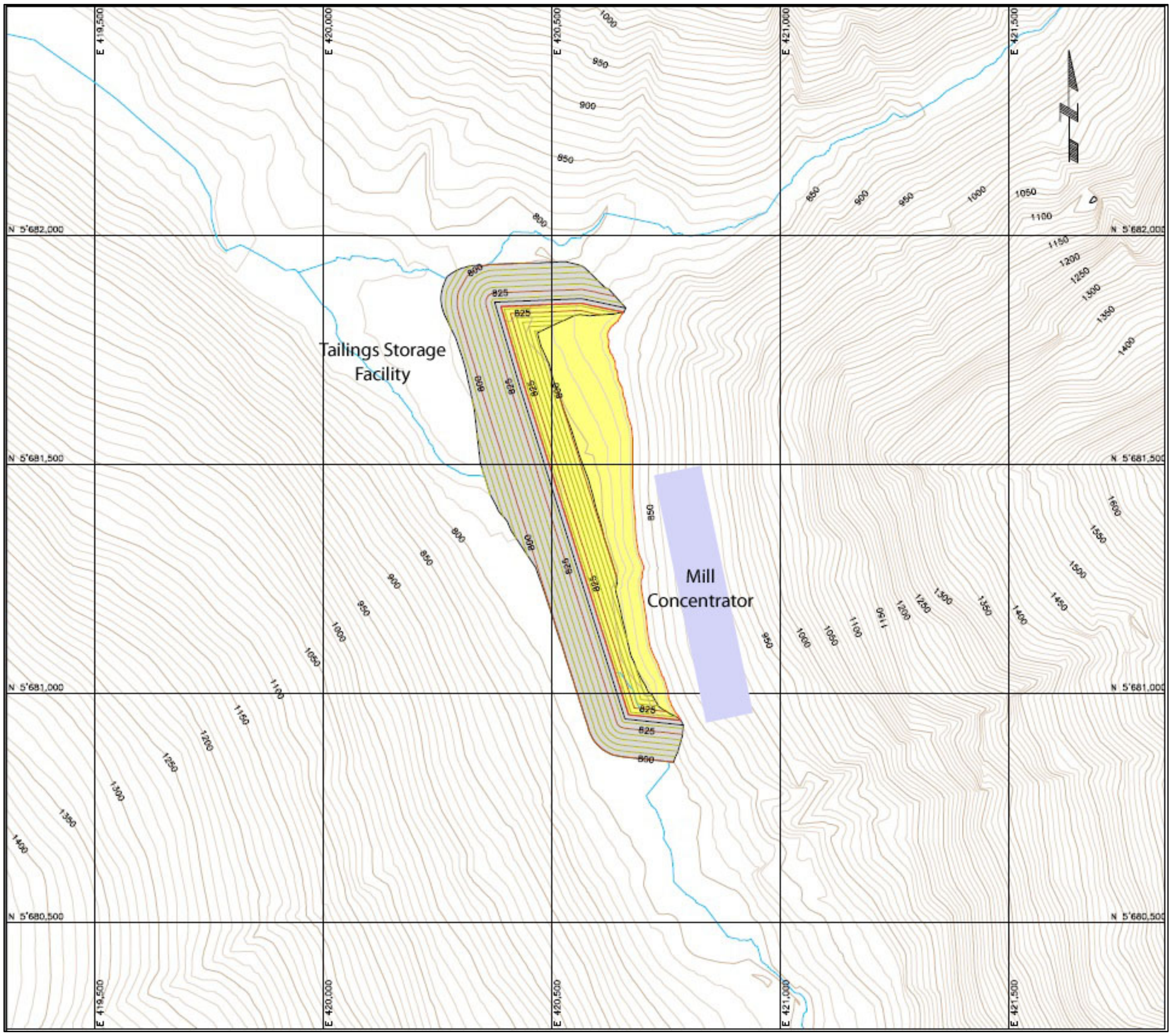


Table 17.3
Summary of Processing Plant Operations Personnel

Description	Number
Plant Superintendent	1
Metallurgist	1
Maintenance Superintendent	1
Chief Chemist	1
Maintenance Supervisor	1
Instrument Technician	1
Plant Shift Supervisor	4
Crusher Operator	4

Description	Number
Plant Operator	4
Control Room Operator	4
Operator Helper	4
Tailings Operator	2
Load out Operator	2
Electrician	2
Plant Mechanic	2
Sample Prep. Technician	2
Assay Technician	2
Autoclave Operators	8
Autoclave Maintenance/Instrumentation	2
Cyanidation Plant Operator	8
Oxygen Plant Operator	4
Day Labour Crew	4
Total	**64**

17.5 CONCLUSIONS

The capital and operating costs for the concentrator have been estimated on the basis of available data. Additional flowsheet development and pilot studies are required before design specifications can be finalized.

In Option A, it has been assumed that a zinc hydroxide precipitate will be produced. However, there is also the potential to produce a high grade Zn product by precipitating Zn from the leach solution as a sulphide or as a sulphate. It is recommended that this hydrometallurgical option should be explored further in future testwork.

Option B was not evaluated as Micon is doubtful that the quantity of Zn flotation concentrate that would be produced at J&L would find a market, considering its arsenic content.

Huakan should investigate further reducing the arsenic content of the lead concentrate, with a view to optimizing the trade-off between TC/RC penalties and lead recovery.

Based on Micon's preliminary hydrometallurgical calculations regarding the oxidation circuit design, the oxygen plant power requirements are likely to be significant, possibly exceeding the power requirements of the rest of the plant and mine site. This will be an important consideration in choice of supply and estimation of the infrastructure capital. However, it was also noted that the capital and/or operating costs of the oxygen plant may potentially be off-set by the production of other gases (for example, argon) as valuable by-products which are in high demand elsewhere. This opportunity should be further investigated during further project development.

Bioleach testwork has also been previously proposed. However, this process typically requires 7-d retention times, at relatively low pulp density. Given that mountainous terrain at the J&L site limits the amount of level area available, it may not be possible to accommodate the amount of bioleach tankage needed.

Micon suggests the plant layout should take advantage of the sloping terrain and use gravity-assisted process flows where possible. This could include locating the process plant further upstream along Carnes Creek and, if possible, above the selected site of the tailings storage facility (TSF) to minimize pumping costs.

18.0 PROJECT INFRASTRUCTURE

18.1 ACCESS

Vehicle access to the area is via Provincial Highway 23, 32 km north of the town of Revelstoke, where Highway 23 intercepts the Carnes Creek Forest Service Road. The Property is then reached by travelling eastward 13 km along the Carnes Creek Forest Service Road before reaching the J&L mine camp.

18.2 POWER SUPPLY

Electric power is currently produced by on-site diesel generators and a satellite phone and internet system is in place. In the PEA, a provision is made for power supply which includes an allowance for incoming transmission line and tie-in to Mica Dam or Revelstoke Utilities, on-site substation and emergency generators. The total power demand of the mine and mill is approximately 12.8 MW, and requires a substation capacity of approximately 16 MW.

18.3 WATER SUPPLY

Process water will be reclaimed from the tailings area. Mine water will be recycled and used underground for drilling, dust suppression, and maintenance needs. All mine water will report to a main sump underground.

18.4 TAILINGS DISPOSAL

Provision has been made in the PEA for the cost of constructing a tailings dam, based on a conceptual design prepared independently for Huakan in respect of one of several potential tailings storage sites identified for the project. The conceptual tailings dam consists of a 20 m high starter dam, transitioning to centerline construction. The maximum embankment height is 48 m, at an elevation of 833 m. The embankment is designed to contain a total fill volume of 3.3 Mm^3, with a final tailings elevation of 832.8 m. The footprint area is 205,977 m^2.

18.5 FUEL STORAGE

A diesel storage tank will be required at the mine yard and, as the mine continues to develop, underground diesel storage tanks will be located in the underground shop and other locations in the mine as needed.

Heavy fuel oil will be used for the carbon kilns; a storage tank and distribution pumps are required.

18.6 BUILDINGS

Existing buildings will be used for the new mine. The cafeteria, administration, warehouse/maintenance shop were confirmed to be in suitable condition. A new change facility is required and surface facilities will probably be expanded as operations ramp up.

For construction, the local operators will provide accommodation for their workers.

18.7 SEWAGE TREATMENT

The existing sewage treatment was not visited. It has been assumed that these facilities will need to be upgraded to accommodate operations ramp up.

18.8 FIRE PROTECTION

A fire protection system will need to be installed. Firewater pumps are provided in this study.

19.0 MARKET STUDIES AND CONTRACTS

There are no marketing studies or contracts relevant to the J&L Property.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES AND ISSUES

Previous operators in the 1980's and early 1990's collected some environmental data including water samples. Merit/Huakan commenced environmental baseline studies for the J&L Property in 2007, including meteorology, water quality, waste characterization, vegetation, wildlife, and fisheries studies.

20.1.1 Aquatic Resources

The J&L project site is located at the confluence of McKinnon and Carnes Creek. Carnes Creek then flows into the east side of Lake Revelstoke on the Columbia River. Lake Revelstoke is controlled at the south end by the Revelstoke Dam.

Water quality in Carnes Creek is typical of glacial fed streams (Table 20.1). It is moderately hard (average hardness 118 mg/L $CaCO_3$), has average pH of 8.0, average dissolved solids of 80 mg/L, and average total suspended solids of 35 mg/L with the highest suspended solids occurring during glacial melt in July and August. Total iron and aluminium occasionally exceed the water quality guidelines for the protection of aquatic life, but these metals are typically associated with streams with high suspended sediment loads.

Golder Associates Ltd. (Golder) conducted a fish assessment in the autumn of 2004 at the J&L project site to determine species presence, relative abundance, and distribution of fish species within Carnes Creek and McKinnon Creek watersheds. In addition, habitat within the streams was also assessed. Kokanee and bull trout were confirmed present in McKinnon and Carnes creeks. (Golder, 2008). There are no barriers on the creeks; however, both streams are glacial fed and have high turbidity and low temperatures which may be limiting factors for fish habitat quality (Golder, 2008a).

Table 20.1
Carnes Creek Water Quality

Parameter	Unit	Max	Min	Average	B.C. Aquatic Life Guidelines[1]
Physical Tests					
Hardness (as $CaCO_3$)	mg/L	181	95.6	116	
Conductivity	uS/cm	235	48.1	154	
pH		8.27	7.62	8.0	
Total Dissolved Solids	mg/L	125	11.5	80	
Total Suspended Solids	mg/L	100	3	32	
Turbidity	NTU	17.4	0.14	4.8	
Anions and Nutrients					
Ammonia as N	mg/L	2.9	0.005	0.30	pH and Temp. dependent
Acidity (as $CaCO_3$)	mg/L	80.7	1	11.7	
Alkalinity, Bicarbonate (as $CaCO_3$)	mg/L	107	2	74	
Alkalinity, Carbonate (as $CaCO_3$)	mg/L	2	2	2	

Parameter	Unit	Max	Min	Average	B.C. Aquatic Life Guidelines[1]
Alkalinity, Hydroxide (as $CaCO_3$)	mg/L	80.7	2	9.9	
Alkalinity, Total (as $CaCO_3$)	mg/L	107	0.005	74.1	low sensitivity to acid inputs
Bromide (Br)	mg/L	0.05	0.05	0.05	
Chloride (Cl)	mg/L	0.5	0.5	0.5	600 (max)
Fluoride (F)	mg/L	0.029	0.02	0.022	0.4 (@ hardness = 10 mg/L)
Sulfate (SO4)	mg/L	14.4	0.21	9.6	100 (max)
Nitrate and Nitrite as N	mg/L	0.739	0.067	0.27	10 (max)
Nitrate (as N)	mg/L	0.739	0.001	0.26	32.8 (max)
Nitrite (as N)	mg/L	0.0088	0.001	0.0019	0.06 (max)
Total Phosphate as P	mg/L	12.8	0.002	1.4	
Cyanides					
Cyanide, Weak Acid Dissociable	mg/L	0.005	0.005	0.005	0.005 (30-day avg)
Cyanide, Total	mg/L	0.005	0.005	0.005	
Total Metals					
Aluminum (Al)-Total	mg/L	0.777	0.005	0.169	0.1 (max dissolved)
Antimony (Sb)-Total	mg/L	0.0005	0.0005	0.0005	0.02
Arsenic (As)-Total	mg/L	0.0005	0.0005	0.0005	0.005 (max)
Barium (Ba)-Total	mg/L	0.042	0.02	0.022	5 (max – under review)
Beryllium (Be)-Total	mg/L	0.001	0.001	0.001	0.0053 (chronic-under review)
Boron (B)-Total	mg/L	0.1	0.1	0.1	1.2 (max)
Cadmium (Cd)-Total	mg/L	0.000017	0.000017	0.000017	0.00004
Calcium (Ca)-Total	mg/L	38.5	0.1	26.2	
Chromium (Cr)-Total	mg/L	0.0015	0.001	0.0011	0.001 (max Cr(VI)-under review)
Cobalt (Co)-Total	mg/L	0.00074	0.0003	0.0004	
Copper (Cu)-Total	mg/L	0.0016	0.001	0.0011	
Iron (Fe)-Total	mg/L	1.36	0.03	0.31	
Lead (Pb)-Total	mg/L	0.00089	0.0005	0.0006	0.003 (depending on hardness)
Lithium (Li)-Total	mg/L	0.005	0.005	0.005	
Magnesium (Mg)-Total	mg/L	7.15	0.1	4.6	
Manganese (Mn)-Total	mg/L	0.0298	0.0003	0.0094	
Mercury (Hg)-Total	mg/L	0.00002	0.00001	0.00002	
Molybdenum (Mo)-Total	mg/L	0.001	0.001	0.001	
Nickel (Ni)-Total	mg/L	0.0022	0.001	0.0012	0.065 (max, hardness dependent)
Potassium (K)-Total	mg/L	2	2	2	
Selenium (Se)-Total	mg/L	0.001	0.001	0.001	0.002 (average)
Silver (Ag)-Total	mg/L	0.00002	0.00002	0.00002	0.003 (max)
Sodium (Na)-Total	mg/L	2	2	2	
Thallium (Tl)-Total	mg/L	0.0002	0.0002	0.0002	0.003 (Ontario objective
Tin (Sn)-Total	mg/L	0.0005	0.0005	0.0005	
Titanium (Ti)-Total	mg/L	0.018	0.01	0.011	
Uranium (U)-Total	mg/L	0.00091	0.0002	0.0007	0.3 (max)
Vanadium (V)-Total	mg/L	0.001	0.001	0.001	0.006 (Ontario objective)
Zinc (Zn)-Total	mg/L	0.005	0.005	0.005	0.053 (hardness dependent)

[1] Statistics based on 10 samples, December, 2007 to August, 2008 and November, 2010. Guidelines from A Compendium of Working Water Quality Guidelines for British Columbia (Nagpal, Pommen, Swain, 2006) and individual parameter guidelines from the B.C. Ministry of Environment website (www.env.gov.bc.ca/wat/wq/BCguidelines).

20.1.2 Terrestrial Resources

The project site is located within the Interior Cedar-Hemlock Biogeoclimatic Zone (ICH) and within the very wet, cool variant subzone (ICHvk1). The area consists of areas of old growth western red cedar and western hemlock and areas of young planted spruce and Douglas fir with natural regeneration of red cedar, spruce and hemlock. Upper areas of the watershed, above 1,500 m, include the Engelmann spruce, sub-alpine fir (ESSF) biogeoclimatic zone and the Alpine Tundra (AT) zone at the peaks (Golder, 2008c). Vegetation surveys were completed in 2004 and 2008. Thirteen invasive plant species were noted during the 2008 vegetation surveys. Rare plants have the potential to occur in the area, but were not observed during the 2008 surveys (Golder, 2008c).

Golder conducted wildlife surveys in January, March and June, 2008. Results indicate the area is used by red squirrel, mouse, chipmunk, marten, beaver, coyote, moose, white-tailed deer, snowshoe hare, bear, pine siskin, pileated woodpecker, sapsucker, black capped chickadee, house and purple finch, red-breasted nuthatch, common raven (Golder, 2008b). Thirty one species of birds were recorded during the June 2008 breeding bird survey. The most common species included the dark-eyed junco, MacGillivray's warbler, winter wren and Swainson's thrush. Other species of note included nesting barred owls (Golder, 2008d). The western toad was observed during the June survey and is listed as a species of Special Concern (Golder, 2008d). Although no sign were found, there is the potential for fisher, caribou, and wolverine (Golder, 2008c).

20.2 WASTE AND WATER MANAGEMENT

20.2.1 Waste Characterization

MESH Environmental reported the following:

> "No evidence of existing acid generation could be delineated, despite exposure of the 832 Adit since development in 1991, indicating a substantial buffering potential in the deposit host rocks."
>
> "The database suggests that development rock with greater than 0.6% sulphur may have uncertain acid generating potential." "Arsenic was identified as a possible metal of environmental concern from the 832 Level Adit drainage water quality analyses. It was shown through shake flask extractions that noteworthy levels of arsenic were solubilized from a limestone sample containing approximately 300 ppm of solid-phase arsenic. A slightly higher solid-phase As cut-off level of 500 ppm was selected because the SFE are not representative of actual drainage and are designed to remove any solubility constraints" (MESH Environmental, 2007).

In September, 2008, Merit suspended operations at the J&L Property and kept the site on a care and maintenance status with on-site security until early December, 2008. Subsequently, mobile equipment was removed. In 2009, the PAG pile of waste rock was covered with overlapping panels of heavy duty non-rip nylon tarping as a temporary measure to minimize or eliminate precipitation onto the pile and thereby minimize possible oxidation effects. (Huakan 2011).

In October 2011, Huakan constructed a lined pad for PAG waste rock. In the spring of 2012 the existing PAG waste rock was re-located onto the lined PAG pad.

Huakan (2011) also reported:

> "Merit's 2008 development on the 832 drift generated 45,000 t of waste rock separated into two waste dumps based on their potential risks for ARD/ML. With the aid of an on-site portable low detection limit XRF machine, mining crews were able to determine the rock's sulfur and metal content on a round-by-round basis. When rounds were over "concern" threshold values, material was sent to a PAG/ML waste dump. When rounds were under "concern" threshold values, material was sent to a non PAG/ML waste dump. Three on-site kinetic test barrels loaded from the two waste dumps were established in September 2008. Two water samples were taken from the test buckets in 2010 before the system froze up for the winter. The kinetic test work sampling will continue in 2011 to provide in-situ ARD and metal leaching information which will ultimately aid in the final reclamation plan of the waste piles."

Sampling of the field barrels continued into 2011 and continued to show elevated sulphate, antimony, arsenic, cadmium, cobalt, and zinc. These results indicate that waste rock placement and waste rock runoff and seepage will need to be carefully planned and managed during operations and closure to minimize costs and long-term liabilities.

20.2.2 Water Management

Historic water quality samples were collected from the project area by other workers in 1984 and 1985. Merit began water quality sampling in December, 2007. Monthly drainage water quality testing of McKinnon and Carnes Creeks was discontinued in August 2008, but resumed at seven sites in November, 2010 (Huakan 2011).

Water will be managed during operations to maximize recycle and minimize discharges. Nonetheless, there may need to be a discharge. Any discharges will need to meet set permit criteria and be of a quality and quantity that will protect the receiving water quality for aquatic life. Seepage from the tailings and waste rock disposal areas are potential discharges that will require consideration in the water management design, monitoring program and management plans. An emergency response, spill contingency plan will be developed to help prevent and respond to accidental release of contaminants as is standard practice for mine operations.

20.2.3 Social and Environmental Management

Following good industry practice, it is assumed a social, environmental, health and safety management system will be implemented to meet the company's commitments to protect the environment and the health and safety of the workers and the surrounding communities. The management system and plans will monitor and maintain permit compliance and a social licence to operate.

For the purposes of the PEA, it is assumed that an operation of this size will employ one Environmental, Health and Safety (EHS) Manager with additional help coming from consultants and contractors as necessary. Annual environmental, social, health and safety operating costs are estimated at $500,000 in addition to the full time EHS Manager.

20.3 PERMITTING REQUIREMENTS

Micon understands that all exploration permits are in place and maintained in compliance; however, a legal opinion has not be obtained to confirm this and Micon offers no opinion on this.

The J&L project will exceed the threshold of 75,000 t/y set in the Reviewable Projects Regulations; therefore, the project will be subject to a provincial environmental review under the BC Environmental Assessment Act. At the production rate of less than 3,000 t/d, the project is likely subject to a Screening level assessment under the Canadian Environmental Assessment Act which will be harmonized with the BC environmental assessment process.

The time required for the environmental assessment depends on the complexity of the project and the requests for additional information. For the purposes of the PEA, it is assumed that permitting will take approximately two years. Permitting costs are included as indirect costs and are applied in the pre-production period.

Mine reclamation costs in British Columbia require financial assurance prior to construction.

In addition to the environmental assessment, the project will require a number of other permits which will likely include:

- Mines Act Permit from the B.C. Ministry of Energy, Mines, and Responsible for Housing;

- Explosives Magazine Storage and Use Permit from the B.C. Ministry of Energy, Mines, and Responsible for Housing;

- Water Licence and possible approvals or authorizations from the B.C. Ministry of Environment;

- Mining Lease from the B.C. Ministry of Forests, Lands and Natural Resource Operations;

- Possible Licence to Cut or Free Use Permit from the B.C. Ministry of Forests, Lands and Natural Resource Operations;

- Possible Navigable Waters Protection Act Authorization, Transport Canada for bridges.

At this stage, project designs are not expected to affect fish or fish habitat and therefore will not require a HADD (habitat alteration, disturbance or destruction) authorization under the Fisheries Act, or a Schedule 2 amendment under the Metal Mining Effluent Regulations. More time and resources will be needed if this assumption changes.

Similarly, it is currently assumed that the project will not affect critical wildlife habitat or protected species; therefore, no special provisions have been made for the Species at Risk Act or Migratory Birds Act.

20.4 SOCIAL AND COMMUNITY ASPECTS

The J&L project falls within three Nation territories, the Okanagan, Shuswap and Ktunaxa Nations. Table 20.2 presents the Nations and the associated bands that have been determined to have potential interest in the project. Consultations have taken place with the First Nations during the exploration phases and will continue with the goal of developing an agreement(s), as the project plans become more defined and the project moves closer to development.

Table 20.2
First Nations with Potential Interest in J&L Project

Nation	Band
Ktunaxa Nation Council (KNC) in Cranbrook.	Akisq'nuk First Nation in Windermere.
Shuswap Nation Tribal Council (SNTC) in Kamloops	Shuswap Indian Band in Inveremere. (Note - referrals are sent to Kinbasket Group of Companies with a cc to Chief and Council - Shuswap Indian Band.)
	Splatsin First Nation in Enderby
	Little Shuswap Indian Band in Chase
	Adams Lake Indian Band in Chase
	Neskonlith Indian Band in Chase
Okanagan Nation Alliance (ONA) in Westbank.	Okanagan Indian Band (OKIB) in Vernon.

20.5 MINE CLOSURE REQUIREMENTS

The object of the final reclamation is to return the disturbed area to as close to the original condition as possible. The wildlife species expected to re-inhabit the area are deer, black and grizzly bear, moose and a number of bird species that will use the area at various times of the year. The site area is considered prospective for future logging and mineral exploration as well as wildlife and occasional recreational use such as hunting. (Huakan, 2011).

"[The] Mine [was] put on care and maintenance in September, 2008; both portals gated and locked; explosives destroyed and explosives magazines removed; mining equipment stored and secured on site until December, then removed; buildings secured; and underground mine electrical utilities pulled out. Road access on Carnes Creek Access Road [was] gated (Huakan 2011)."

"During 2009 the PAG pile of waste rock generated in 2008 was covered with overlapping panels of heavy duty non-rip nylon tarping as a temporary measure to minimize or eliminate precipitation onto the pile and thereby minimize possible oxidation effects. In 2010 the pile and tarps were inspected and found to be stable with only minor tarp corner re-anchoring (Huakan 2011)."

In fall 2011, Huakan constructed a lined pad for PAG waste rock. In spring 2012, the existing PAG waste rock was re-located onto the lined PAG pad. A toe collection pond was excavated with seepage from the lined pad draining by PVC pipe to the collection pond. Huakan will collect leachate from the toe collection pond on a quarterly basis or more frequently depending on accumulations. Huakan does not expect much water accumulation. The samples will be analyzed for water quality and results will be submitted to the Forests, Lands and Natural Resources Operations Cranbrook office. If the analyses display acid generation or unacceptable metal leaching levels, Huakan will neutralize by adding lime before discharging. More preventative action will be taken depending on the severity of water quality to protect the receiving environment. Figure 20.1 shows the new lined PAG and non-PAG waste rock piles.

For the purposes of the PEA, it is estimated that the mine will require the following:

- Dismantling of the plant and ancillary facilities;
- Securing and closing off underground works;
- Re-contouring and capping of waste rock dumps;
- Closure and reclamation of the tailings impoundment;
- Scarification and revegetation of all other disturbed areas including the access road;
- Ongoing monitoring of the dam and reclamation.

Preliminary closure costs are estimated to be $6.5 million. This estimate is based on cost estimates from similar metal mines and scaled for an estimated 1,500 t/d underground mine with approximately 20 ha of disturbance, a 30% contingency and assumes that the waste rock and tailings impoundment closure is designed to avoid long-term water treatment. Mine reclamation costs in British Columbia require financial assurance prior to construction.

Figure 20.1
New Lined PAG and Uncovered Non-PAG Waste Rock Piles

Huakan, 2012.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The total estimated project life-of-mine capital expenditure amounts to $299 million comprised $264 million pre-production capital and $35 million sustaining capital, including closure costs. A summary of the estimated pre-production capital costs is provided in Table 21.1. All costs are stated in 1st quarter 2012 Canadian dollars. Accuracy is in the range of ±30%, typical for a PEA of this nature.

Table 21.1
Summary of Project Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Mining	9,178	19,176	28,354
Process	97,490	9,451	106,941
Infrastructure	51,290	-	51,290
Indirects	18,714	-	18,714
EPCM	22,317	-	22,317
Contingency	49,747	-	49,747
Owner's	14,931	-	14,931
Closure	-	6,500	6,500
Total	**263,668**	**35,126**	**298,794**

21.1.1 Mining Capital

The estimated mining capital costs, based on the annual equipment and development requirements, are summarized in Table 21.2.

Table 21.2
Summary of Mining Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Portal (Lump Sum)	230	-	230
Mobile Equipment (pre-prod.)	8,948	-	8,948
Backfill Plant	-	3,000	3,000
Main Ventilation Fan	-	140	140
Mobile Equipment (sust.)	-	16,036	16,036
Total Mining Capital	**9,178**	**19,176**	**28,354**

For the purpose of this PEA, all pre-production development, including access ramps and initial stope development, has been included as operating development in Year 1, as shown in the schedule presented earlier in this report, in Table 16.4.

21.1.2 Processing Capital

Initial capital costs, estimated at $97.5 million, are summarized in Table 21.3, which also reflects the provision of $6.4 million for sustaining capital in the plant of the LOM period.

Table 21.3
Processing Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Crushing plant	7,382	-	7,382
Grinding, incl. Regrind	8,473	-	8,473
Concentrator	5,219	-	5,219
De-watering	4,473	-	4,473
Common systems - Mill	8,946	-	8,946
Hydromet Area	30,000	-	30,000
Buildings, etc	7,997	-	7,997
Oxygen Plant	25,000	-	25,000
Sustaining		6,346	6,346
Total Processing Capital	**97,490**	**6,346**	**103,836**

21.1.3 Infrastructural Capital

Initial capital costs, estimated at $51.3 million as summarized in Table 21.4, include a power supply from the regional utility, tailings impoundment and storage, expansion of the existing camp facilities, and an on-site laboratory for grade control and environmental management.

Table 21.4
Infrastructural Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Camp expansion	1,500	-	1,500
Access Road upgrade	2,500	-	2,500
Site Roads	500	-	500
Tailings/Leach Residue Storage	16,830	-	16,830
Power Line	12,500	-	12,500
Power Distribution	8,200	-	8,200
Emergency Power	1,000	-	1,000
Fuel Storage & Distribution	150	-	150
Water Supply	850	-	850
Info & Communications Systems	250	-	250
Mobile Equipment - Surface	1,710	-	1,710
Admin Bldg, office equipt/furniture	1,250	-	1,250
Warehse, inventory control system	1,250	-	1,250
Gatehouse & Security	300	-	300
Assay Laboratory, incl. instruments	1,000	-	1,000
Reagent Storage	1,500	-	1,500
Sustaining		3,105	3,105
Total Infrastructural Capital	**51,290**	**3,105**	**54,395**

21.1.4 Indirect Capital Costs

Indirect capital costs are estimated at $18.7 million as summarized in Table 21.5.

Table 21.5
Indirect Capital Costs

Area	Pre-Production ($ 000)	Sustaining ($ 000)	LOM Capital Costs ($ 000)
Spare Parts	7,439	-	7,439
First Fills and Inventory	2,117	-	2,117
Freight, Transport and Insurance	744	-	744
Vendor Supervision	975	-	975
Basic Engineering	7,439	-	7,439
Total Indirect Capital	**18,714**	**-**	**18,714**

In addition to the above, the economic evaluation makes provision for the following charges which are applied in the pre-production period:

- $22.3 million for Engineering, Procurement and Construction Management (EPCM) fees in respect of the processing and infrastructural aspects of the project;

- $14.9 million owner's costs, to cover the costs of staffing, recruitment, training and subsistence during the pre-production period, as well as insurance, permitting and environmental monitoring costs.

21.1.5 Contingency

Contingency is a monetary provision in the estimate to cover additional costs that can be expected to be identified during the basic engineering and detailed design stages.

An allowance of 25% has been made in pre-production capital costs, excluding owners' costs and closure bonding, which amounts to $49.8 million. Micon considers this to be appropriate given the preliminary nature of this estimate.

21.1.6 Closure Costs

A provision of $6.5 million has been provided for mine closure and reclamation, within the total LOM sustaining capital provision of $35 million. For the purposes of economic evaluation, it has been assumed that this amount will be required for bonding, payable during the construction period.

Pre-operational testing costs were estimated by approximating the crew size and assigning an average labour rate over a typical start-up period based on information from similar projects.

Cost of initial fills is based on the estimated plant inventories including allowances for such items as specialty chemicals and reagents.

21.2 OPERATING COSTS

A summary of the LOM unit operating costs is presented in Table 21.6.

Table 21.6
LOM Average Unit Operating Costs

Area	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Mining	287,307	58.67	313.93
Process	216,903	44.30	237.00
General and Administration	50,441	10.30	55.11
Total	**554,651**	**113.27**	**606.04**

When by-product credits are taken into account, the average cost per ounce of payable gold falls to $172/oz, as presented in Table 21.7. By-product credits include silver in doré, silver and lead in concentrate, and zinc hydroxide.

Table 21.7
LOM Average Unit Operating Costs

Area	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Mining	287,307	58.67	313.93
Process	216,903	44.30	237.00
General and Administration	50,441	10.30	55.11
Less by product revenues	(397,203)	(32.15)	(434.00)
Total	**157,447**	**113.27**	**172.04**

21.2.1 Mining

The LOM mining operating costs are summarized in Table 21.8.

Table 21.8
LOM Average Mining Operating Costs

Description	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Development	**84,851**	**17.33**	**92.71**
Dev. Equipt Operation and Labour	46,142	9.42	50.42
Dev. Consumables	15,722	3.21	17.18
Power	3,422	0.70	3.74
Dev. Contingency	19,564	4.00	21.38
Production	**202,456**	**41.35**	**221.22**
Utility Equipment O & O	4,817	0.98	5.26
Operation and Labour	119,539	24.41	130.61
Backfill Plant Operation	29,380	6.00	32.10
Power	13,511	2.76	14.76
Ore Production Consumables	35,209	7.19	38.47
Mine Operating Costs Total	**287,307**	**58.67**	**313.93**

Mining consumables include supplies for ground control, drilling, blasting, utilities, and other mine services.

21.2.2 Process

The LOM process operating costs are summarized in Table 21.9.

Table 21.9
LOM Average Process Operating Costs

Description	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Labour	50,343	10.28	55.01
Electrical power	46,297	9.45	50.59
Process consumables	94,668	19.33	103.44
Maintenance	22,211	4.54	24.27
Concentrate handling	3,384	0.69	3.70
Process Operating Costs Total	**216,903**	**44.30**	**237.00**

Process labour costs include 64 people, including crews for maintenance and operation of the oxygen plant that supports operation of the autoclave.

Electrical power consumption comprises 20% of total process operating costs, based on an average demand of 8.1 MW for the processing facility and associated oxygen plant. The latter accounts for more than 75% of the processing power demand.

Process consumables include crushing and grinding media, ferrosilicon for the DMS plant, as well as reagents for the flotation, neutralisation, cyanidation, and cyanide destruction circuits.

A provision for maintenance spares is made at the rate of $2.5 million per year, in addition to the annual sustaining capital provision discussed above.

21.2.3 General and Administration

The LOM general and administrative operating costs are summarized in Table 21.10.

Table 21.10
LOM Average Process Operating Costs

Description	LOM Total ($ 000)	Average ($/t treated)	Average ($/oz Payable Gold)
Labour	11,357	2.32	12.41
Other	39,084	7.98	42.71
G&A Operating Costs Total	**50,441**	**10.30**	**55.11**

On an annualised basis, G&A costs amount to approximately $5.6 million per year, and includes provision for camp operating costs, labour, power, environmental, health and safety, security, office expenses, corporate overheads, and local taxes.

22.0 ECONOMIC ANALYSIS

22.1 BASIS OF EVALUATION

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. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the study was to establish the economic viability of the proposed development of an underground mining operation and processing for the production of a marketable lead concentrate, and gold and silver doré from the J&L deposit. 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, Inflation and Discounting

The project cash flow model, and all results derived from the model, is expressed in Canadian dollars ($). The assumption was made that parity with United States currency (US$) will be maintained over the project life.

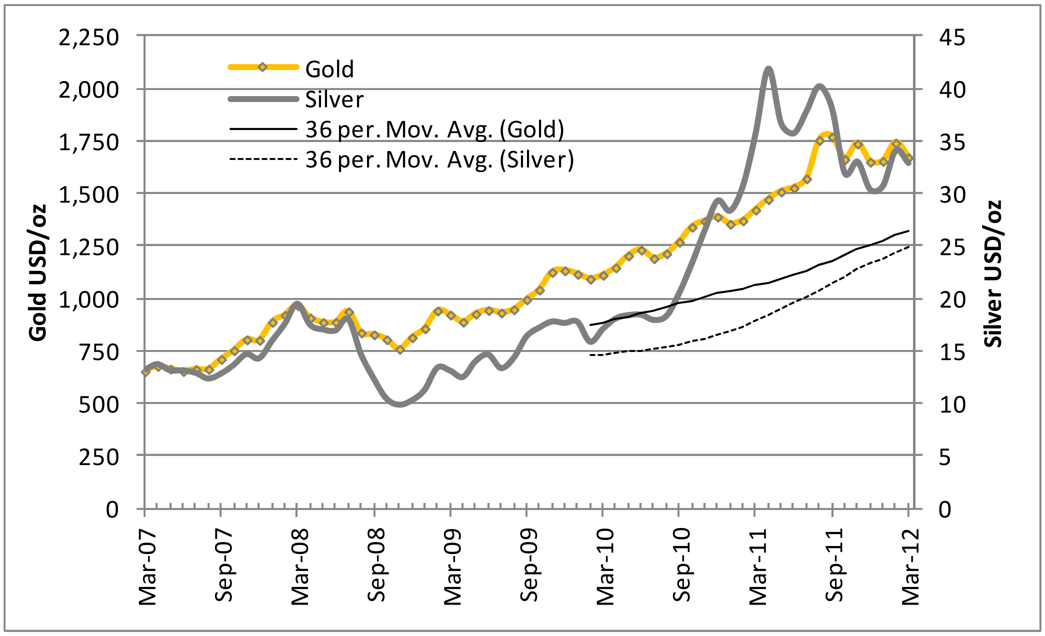
Constant, Q4-2011 money terms are used throughout, i.e., without provision for inflation. The cash flow is discounted to the period in which a decision is taken to proceed with project construction, two years prior to first production.

22.2.2 Expected Metal Prices

Figure 22.1 shows the monthly average price of gold and silver over the period 2007 to 2011 inclusive. During the period Q4-2010 to Q2-2011, silver prices rose briefly to more than $40/oz, giving rise to a gold/silver price ratio of as low as 35:1, compared to a longer term average of around 60:1. The price of silver has subsequently fallen back in line with this long term trend.

The base case has been evaluated using a gold price of $1,320/oz, equating to the three-year trailing average price at the end of March, 2012. The silver price of $22.00/oz assumes a 60:1 ratio of the gold and silver prices is maintained, notwithstanding a spike in the silver price during 2011. The study also uses the three-year trailing average prices for zinc and lead, being $0.98/lb and $0.94/lb, respectively. As part of its sensitivity analysis, Micon also tested a range of prices 25% above and below the base case values, and over a range of prices that include recent values.

Figure 22.1
Silver and Gold Prices 2007-2011



22.2.3 Weighted Average Cost of Capital

In order to find the NPV of the cash flows forecast for the project, an appropriate discount factor must be applied which represents the weighted average cost of capital (WACC) imposed on the project by the capital markets. The cash flow projections used for the valuation have been prepared on an all-equity basis. This being the case, WACC is equal to the market cost of equity, and can be determined using the Capital Asset Pricing Model (CAPM):

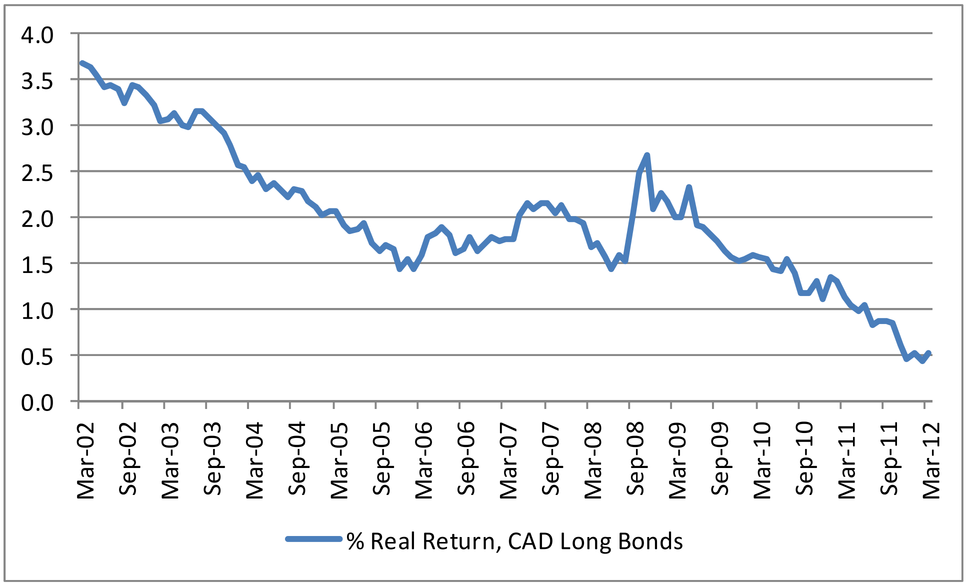
$$E(R_i) = R_f + \beta_i(E(R_m) - R_f)$$

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 22.2 illustrates the real return on CAD long bonds computed by the Bank of Canada, taken as a proxy for the risk-free interest rate. Over the past year, this has dropped from around 2.0% to 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.

Although principally a gold mine, the J&L project is expected to drive close to 40% of net revenue from sales of base metal concentrates. Taking beta for this sector of the equity market to be in the range 1.2 to 2.0, CAPM gives a cost of equity for the J&L project of between 8% and 12%. Micon has taken a figure at the lower end of this range as its base case, and provides the results at higher rates of discount for comparative purposes.

Figure 22.2
Real Return on Canadian Dollar Long Bonds 2002-2012



Bank of Canada.

22.2.4 Taxation Regime

The J&L project will be subject to Canadian federal income tax and British Columbia provincial mining and income taxes. These have been provided for in the cash flow model.

22.2.5 Royalty

Micon understands that the project is free of any royalty attributable to prior owners of the mineral title at the J&L Property, and none has been provided for in the cash flow model.

22.2.6 Marketing Costs

Provision has been made for the costs of concentrate transport and the refining of gold and silver contained in concentrates produced at the mine, at the rate of $5.00/oz and $0.50/oz respectively.

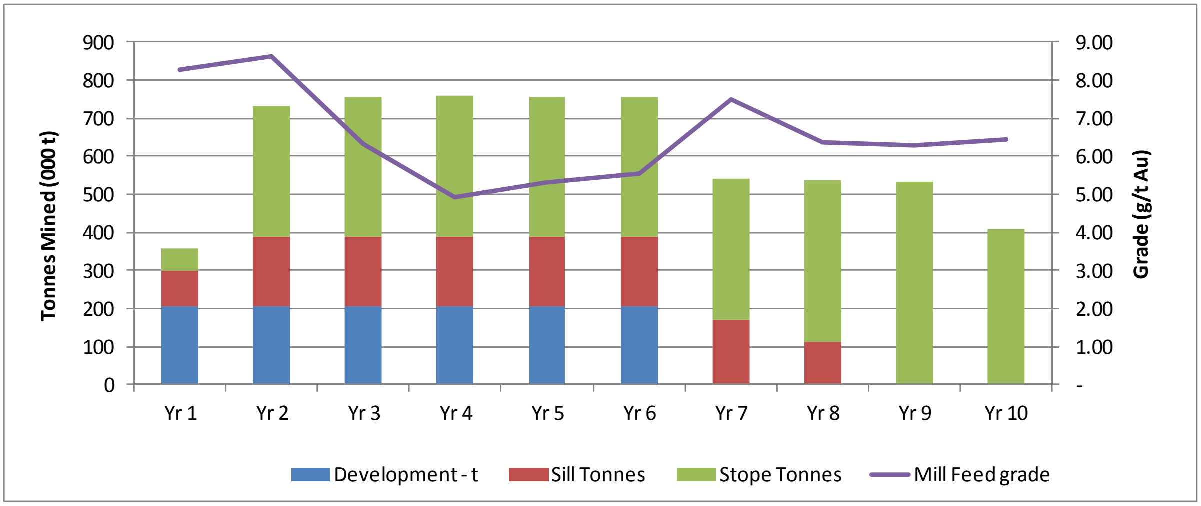
22.3 TECHNICAL ASSUMPTIONS

The technical parameters, production forecasts and estimates described earlier in this report are reflected in the base case cash flow model. These inputs to the model are summarised below.

22.3.1 Mine Production Schedule

Figure 22.3 shows the annual schedule for mining of process feed from the underground mine. The grade of process feed varies from 8.6 g/t to 4.9 g/t gold, and averages 6.4 g/t over the LOM period. The silver grade averages 70 g/t Ag, together with 2.2% Pb and 3.6% Zn.

Figure 22.3
Underground Mining Production Schedule



22.3.2 Stockpiling

Stockpiling of up to a maximum of 41,400 t of process feed has been taken into account in the production schedule. Stockpile tonnage peaks in Year 8 and is drawn down at the end of the LOM period in Year 11.

22.3.3 Processing Schedule

The annual tonnage of material processed ramps up from 135,000 t in Year 1 to steady state operations of 540,000 t/y processed in Years 2 to 11. Annual average gold, silver, lead and zinc grades of process feed are shown in Figure 22.4.

22.3.4 Product Sales

The production schedule allows for 60 days inventory of lead concentrate, and 15 days for gold and silver in doré for material locked-up in the processing circuit, as well as 21 days accounts receivable.

The breakdown of LOM revenues by product is shown in Figure 22.5.

Figure 22.4
LOM Processing - Grade Profile

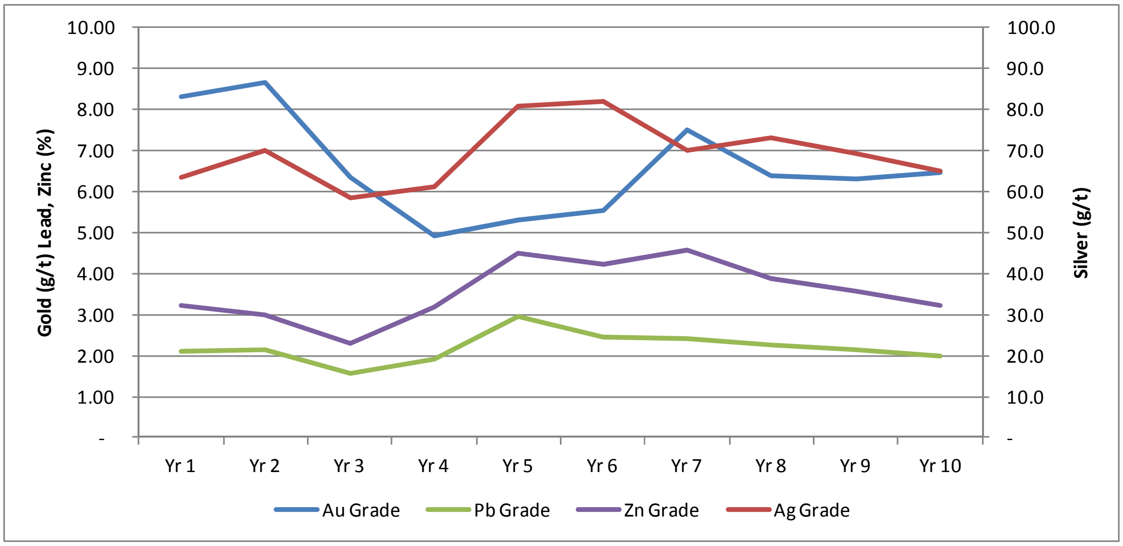
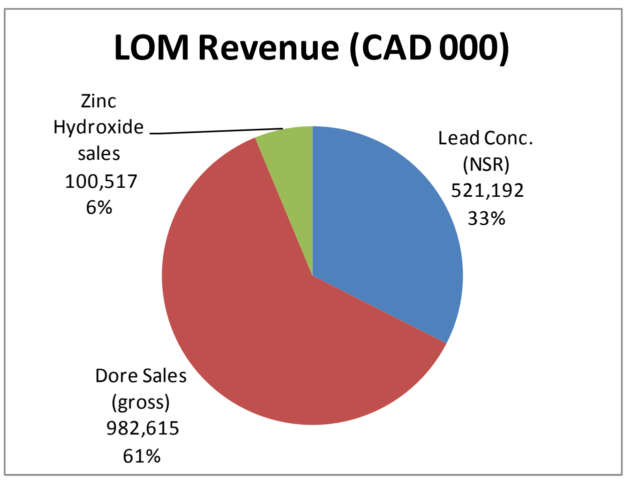


Figure 22.5
Breakdown of LOM Revenue

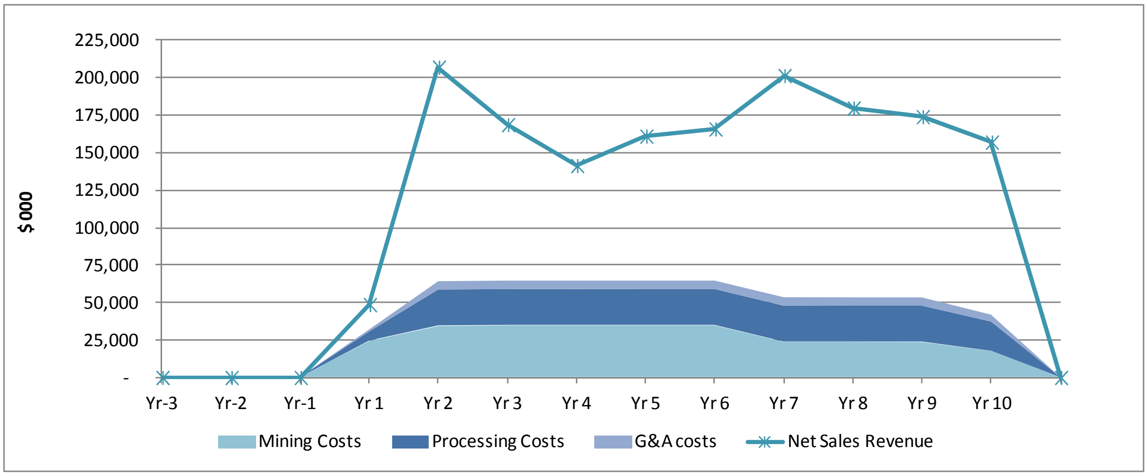


22.3.5 Operating Costs

Cash operating costs average $113.27/t treated over the LOM period, including $58.67/t mining, $44.30/t processing and $10.30/t for general and administrative costs. Mining costs assume an owner-operated fleet, including ancillary equipment, supervision and technical support. Stoping costs include a provision for backfill. Processing costs include labour, electrical power, process consumables, maintenance, and concentrate handling.

Figure 22.6 shows the operating expenditures over the LOM period. In the base case, costs equate to just 35% of net revenue, leaving an average LOM operating margin of 65%.

Figure 22.6
Cash Operating Costs



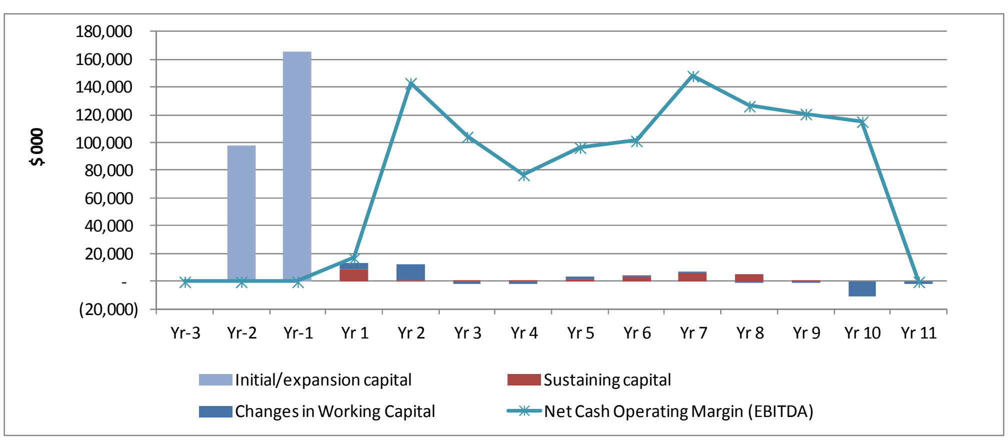
22.3.6 Capital Costs

Initial capital expenditures of $263.7 million include $9.2 million for mining, $97.5 million in the processing plant, $51.3 million for infrastructure, $41 million indirect costs, owner's costs of $14.9 million and a contingency of $49.8 million. In this evaluation, all pre-production development, including access ramps and initial stope development, has been included as operating development.

Sustaining capital of $35.1 million over the LOM period includes construction of a backfill plant in Year 1, replacement mobile equipment, plant sustaining capital and closure bonding.

Figure 22.7 compares annual capital expenditures over the preproduction and LOM periods with the project's cash operating margin.

Figure 22.7
Capital Expenditures



22.3.7 Project Cash Flow

The LOM base case project cash flow is summarized in Table 22.1

Table 22.1
Life-of-Mine Cash Flow Summary

	LOM Total ($ 000)	$/t Treated	$/oz Payable Gold
Net Sales Revenue	**1,604,324**	**327.63**	**1,060.86**
Mining Costs	287,307	58.67	200.28
Processing Costs	216,903	44.30	143.54
G&A costs	50,441	10.30	33.38
Total cash operating costs	**554,651**	**113.27**	**377.20**
Net Cash Operating Margin	1,049,673	214.36	683.66
Initial/expansion capital	263,668	53.85	274.67
Sustaining capital	35,126	7.17	32.49
Changes in Working Capital	-	-	5.85
Net cash flow before tax	**750,880**	**153.34**	**375.99**
Taxation payable	266,737	54.47	155.48
Net cash flow after tax	**484,143**	**98.87**	**220.51**

Annual cash flows are presented in Figure 22.8 and Table 22.2 (over).

Figure 22.8
Life-of-Mine Cash Flows

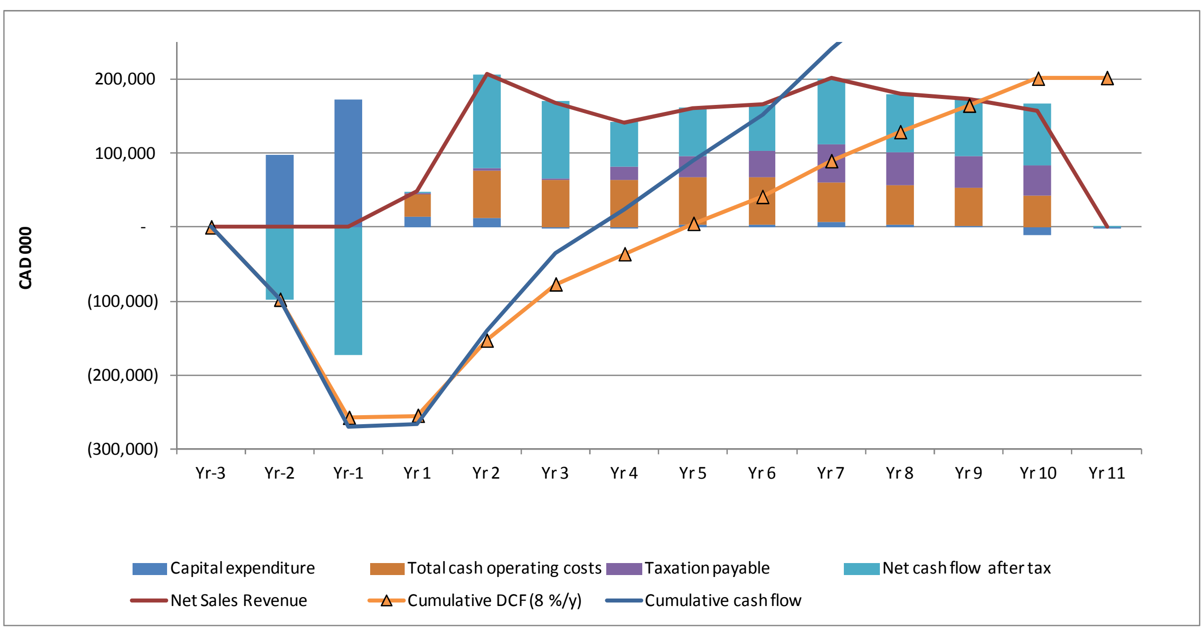


Table 22.2
Base Case Life of Mine Annual Cash Flow

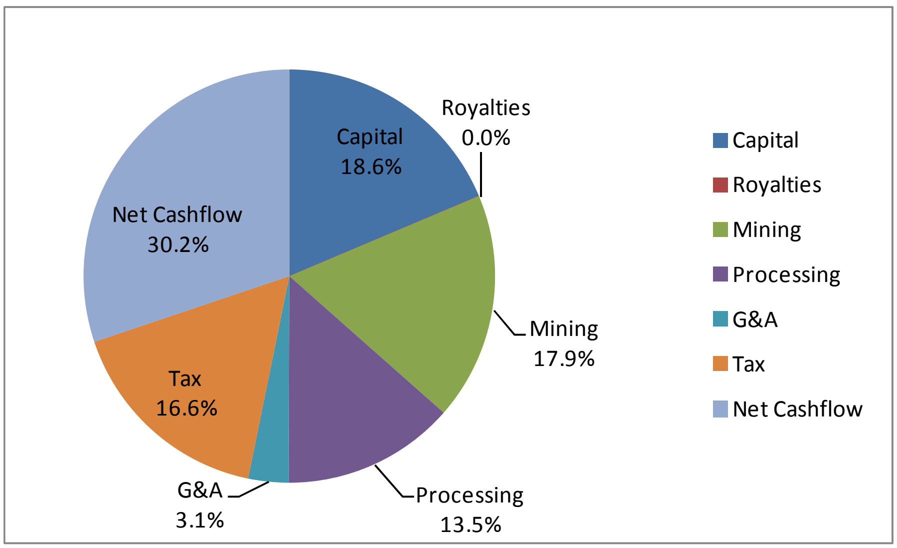
		TOTAL	Yr-3	Yr-2	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
Development - m	m	23,850				3,975	3,975	3,975	3,975	3,975	3,975	-	-	-	-				
Development - t	000 t	1,240				206.7	206.7	206.7	206.7	206.7	206.7	-	-	-	-				
Sill Tonnes	000 t	1,288				92.0	182.5	182.5	183.0	182.5	182.5	171.2	111.8	-	-				
Stope Tonnes	000 t	3,609				60.2	343.2	366.8	367.8	366.8	366.8	368.1	425.4	533.7	410.1				
Prod Tonnes (Sills + Stopes)	000 t	**4,896.7**				**152.2**	**525.7**	**549.3**	**550.8**	**549.3**	**549.3**	**539.2**	**537.2**	**533.7**	**410.1**				
HMS feed (ROM)	**000 t**	**4,896.7**				**135.0**	**540.0**	**540.0**	**540.0**	**540.0**	**540.0**	**540.0**	**540.0**	**540.0**	**441.7**	**-**	**-**	**-**	**-**
Lead concentrate	Conc sales (000 t)	187	-	-	-	4.1	17.6	15.5	17.2	26.0	23.8	22.6	21.5	20.1	18.5	-	-	-	-
Payable (oz, lbs)	Content - Au (000 oz)	188	-	-	-	5.58	24.38	21.68	16.64	16.93	17.79	23.22	21.17	20.36	20.41	-	-	-	-
	Content - Ag (000 oz)	7,789	-	-	-	162.12	750.60	743.80	745.22	951.99	1,005.99	884.08	890.87	857.64	796.41	-	-	-	-
	Content - Pb (000 lbs)	176,198	-	-	-	3,865	16,590	14,647	16,210	24,498	22,452	21,287	20,236	18,942	17,473	-	-	-	-
	Content - Zn (000 lbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dore production	Gold in dore (000 oz)	727	-	-	-	25.9	107.8	79.0	61.4	66.2	69.2	93.8	79.4	78.5	65.8	-	-	-	-
	Dore Silver sales (000 oz)	1,042	-	-	-	24.9	111.6	97.1	100.2	131.4	134.9	115.8	119.8	113.9	92.3	-	-	-	-
Zinc Hydroxide sales (net)	CAD 000	100,517	-	-	-	2,193	8,465	7,353	9,493	13,402	13,041	13,971	12,205	11,074	9,320	-	-	-	-
Lead Conc. (NSR)	CAD 000	521,192	-	-	-	13,153	58,201	53,344	47,683	57,512	58,565	62,383	59,207	56,645	54,500	-	-	-	-
Dore Sales (gross)	CAD 000	982,615	-	-	-	33,271	140,306	107,974	84,164	90,047	94,185	125,029	108,260	106,221	93,158	-	-	-	-
Zinc Hydroxide sales	CAD 000	100,517	-	-	-	2,193	8,465	7,353	9,493	13,402	13,041	13,971	12,205	11,074	9,320	-	-	-	-
Revenue	Gross Sales	1,604,324	-	-	-	48,617	206,972	168,671	141,340	160,961	165,791	201,384	179,672	173,940	156,977	-	-	-	-
less	Royalties	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Net Sales Revenue	1,604,324	-	-	-	48,617	206,972	168,671	141,340	160,961	165,791	201,384	179,672	173,940	156,977	-	-	-	-
Cash op. costs	Mining Costs	287,307	-	-	-	24,361	34,511	34,886	34,910	34,886	34,886	23,801	23,769	23,714	17,582	-	-	-	-
	Processing Costs	216,903	-	-	-	5,980	23,920	23,920	23,920	23,920	23,920	23,920	23,920	23,920	19,565	-	-	-	-
	G&A costs	50,441	-	-	-	1,391	5,563	5,563	5,563	5,563	5,563	5,563	5,563	5,563	4,550	-	-	-	-
	Total cash operating costs	554,651	-	-	-	31,732	63,993	64,369	64,393	64,369	64,369	53,283	53,251	53,196	41,697	-	-	-	-
Net Cash Operating Margin (EBITDA)		1,049,673	-	-	-	16,885	142,979	104,303	76,947	96,592	101,422	148,101	126,421	120,744	115,280	-	-	-	-
Capital Expenditure	Initial/expansion capital	263,668	-	97,799	165,868	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sustaining capital	28,626				8,575	1,050	1,050	1,050	1,959	3,082	5,710	5,100	1,050	-	-	-	-	-
	Closure Provision	6,500	-	-	6,500	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Changes in Working Capital	-	-	-	-	4,729	11,151	(2,203)	(1,592)	1,143	282	884	(1,269)	(339)	(10,655)	(2,131)	-	-	-
Net cash flow before tax	26%	750,880	-	(97,799)	(172,368)	3,581	130,778	105,456	77,489	93,490	98,058	141,506	122,590	120,033	125,935	2,131	-	-	-
Taxation payable		266,737	-	-	-	338	2,860	2,086	17,586	28,303	35,750	52,078	44,156	42,641	40,939	-	-	-	-
Net cash flow after tax	21%	484,143	-	(97,799)	(172,368)	3,243	127,918	103,370	59,903	65,187	62,308	89,429	78,434	77,392	84,996	2,131	-	-	-
Cumulative cash flow			-	(97,799)	(270,168)	(266,924)	(139,006)	(35,636)	24,266	89,453	151,761	241,190	319,624	397,016	482,012	484,143	484,143	484,143	484,143
Payback period on undiscounted cash flow (years)		4.6			1.00	1.00	1.00	1.00	0.59	-	-	-	-	-	-	-	-	-	-
Discounted Cash Flow (8 %/y)		201,809	-	(97,799)	(159,600)	2,781	101,546	75,980	40,769	41,079	36,356	48,316	39,237	35,847	36,454	846	-	-	-
Cumulative DCF (8 %/y)			-	(97,799)	(257,400)	(254,619)	(153,073)	(77,094)	(36,325)	4,754	41,110	89,426	128,662	164,510	200,963	201,809	201,809	201,809	201,809
Payback period on discounted cash flow (years)		5.9			1.00	1.00	1.00	1.00	1.00	0.88	-	-	-	-	-	-	-	-	-
Capital expenditure		298,794	-	97,799	172,368	13,304	12,201	(1,153)	(542)	3,102	3,363	6,594	3,831	711	(10,655)	(2,131)	-	-	-
Net Revenue per tonne treated		327.63	-	-	-	360.13	383.28	312.35	261.74	298.08	307.02	372.93	332.73	322.11	355.40	-	-	-	-
Ave Cost per tonne treated		113.27	-	-	-	235.05	118.51	119.20	119.25	119.20	119.20	98.67	98.61	98.51	94.40	-	-	-	-
Operating Margin		65%	0%	0%	0%	35%	69%	62%	54%	60%	61%	74%	70%	69%	73%	0%	0%	0%	0%

22.4 BASE CASE EVALUATION

At the end of Year 1, the project has a cumulative funding requirement of $270 million and in Year 1, working capital of $5 million is required prior to first revenue receipts, bringing the total funding requirement to $275 million. Thereafter, the project is forecast to be cash positive in each year of operation, generating a cumulative net cash flow of $751 million before tax, and $484 million after tax. On the undiscounted cash flow, payback occurs after 4.6 years. On the cash flow discounted at 8%, payback occurs after 5.9 years, and leaves a production tail of more than 5 years, based on the current mineral resources.

Figure 22.9 shows the principal LOM project cash flows as a percentage of net revenue:

Figure 22.9
LOM Project Cash Flows



Over the LOM period, the average cash operating cost equates to $113/t treated, or $377/oz payable gold. Taking into account by-product revenues for silver, lead and zinc totalling $434/oz gold reduces the cost of payable gold production to $172/oz.

The base case evaluates to an IRR of 26% before tax and 21% after tax. At the selected discount rate of 8%, the net present value (NPV_8) of the cash flow is $344 million before tax and $202 million after tax. The base case cash flow evaluation results are shown in Table 22.3. In Micon's opinion, the results demonstrate economic viability of the project base case, under the conditions described above.

It should be noted that this PEA is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that will enable them to be categorized as mineral reserves, and there is no certainty that the conclusions of the PEA will be realized.

Table 22.3
Base Case Cash Flow Evaluation

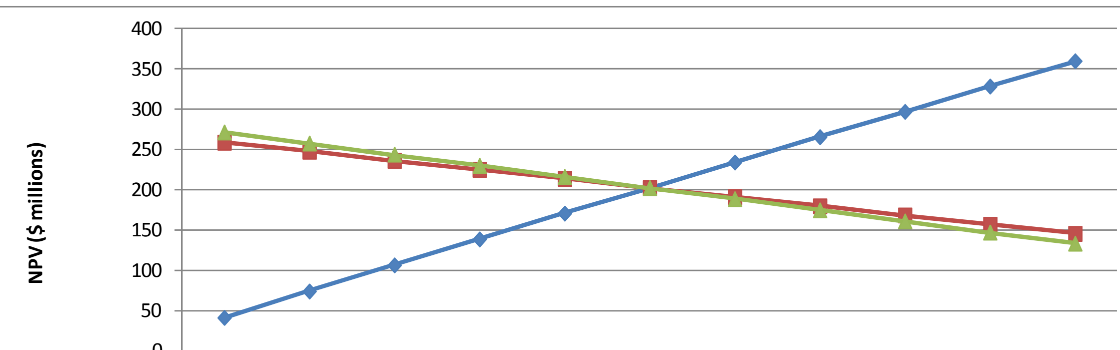
$ million	LOM Total	Base Case Discounted at 8%/y	Discounted at 10%/y	Discounted at 12%/y	IRR (%)
Gross Sales	1,604	971	867	777	
less Royalties	-	-	-	-	
Net Sales Revenue	**1,604**	**971**	**867**	**777**	
Mining Costs	287	183	166	150	
Processing Costs	217	131	117	105	
G&A costs	50	31	27	24	
Total Cash Operating Costs	**555**	**345**	**310**	**280**	
Net Cash Operating Margin	1,050	626	557	497	
Initial capital	264	251	249	246	
Sustaining capital	35	30	29	28	
Changes in Working Capital	-	5	6	6	
Net Cash Flow Before Tax	**751**	**344**	**279**	**224**	**26**
Taxation Payable	267	142	123	107	
Net Cash Flow After Tax	**484**	**202**	**156**	**117**	**21**

22.5 SENSITIVITY STUDY

22.5.1 Sensitivity to Revenue Factors, Operating and Capital Costs

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 25% above and below base case values. Figure 22.10 shows the results of this analysis.

Figure 22.10
Sensitivity to Capital, Operating Costs and Revenue



	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Revenue drivers	41	73	106	138	170	202	233	265	296	328	359
Operating costs	258	247	236	224	213	202	190	179	168	156	145
Capital costs	271	257	243	229	216	202	188	174	160	147	133

The results show that the project is most sensitive to revenue factors, with an adverse change of 25% reducing NPV_8 to $41 million. The project is less sensitive to operating costs, with an adverse change of 25% required to reduce NPV_8 to $145 million. Capital costs have the least impact on returns, with a 25% increase in cost reducing NPV_8 to $133 million.

In further analysis, Micon notes that simultaneously applying an increase of more than 40% to both capital and operating costs simultaneously will be required in order to reduce NPV_8 to zero.

Micon concludes that project returns are sufficiently robust to withstand adverse changes in the principal value drivers of the project within the limits of accuracy of the estimate.

22.5.2 Gold Price and Discount Rate

The sensitivity of project returns to changes in the gold price was determined, while maintaining a 60:1 ratio between the gold and silver price, and holding lead and zinc prices constant at base case values. The results are as shown in Table 22.4. Trailing averages are calculated as of 31 March, 2012.

Table 22.4
Sensitivity to Gold Price and Discount Rate

Gold Price $/oz	NPV after tax Discounted at 8%/y ($ million)	After-tax IRR (%)	Notes
1,320	202	21	36-month trailing average
1,469	264	24	24-month trailing average
1,645	337	28	12-month trailing average
1,674	349	29	1 month trailing average

All else being equal, economic break-even for the project (i.e., an NPV_8 of zero) occurs with a gold price of $850/oz and $14.17/oz silver.

22.6 CONCLUSION

Micon concludes that this study demonstrates the economic potential of the project as proposed, and that further development is warranted.

23.0 ADJACENT PROPERTIES

The J&L Property is situated in a well mineralized area of British Columbia, surrounded by several different types of mineralized showings, all within 10 km of Main Zone portals.

The following information has been obtained from internet sources. The mineralization described is not necessarily indicative of mineralization on the J&L Property and the author has not independently verified the information.

The Mastodon deposits are located 5 km to the south of the Main Zone. The Mastodon is a group of deposits and showings which include the Mastodon (082M 005), Mastodon North (082M 195), Lead King (082M 094), Little Slide (082M 006) and Little Slide No. 3 (082M 196). The area is a series of polymetallic (Zn, Pb, Cd, Ag, Au, Cu) breccia, replacement-type bodies that are tabular (Mastodon - 90 by 60 by 3 m) in Badshot Limestone which may be structurally controlled. Teck-Cominco had the property up until 1992. It has many of the same characteristics as the J&L Main Zone and could be a parallel mineralized structure. Their programs failed to discover sufficient surface indications of mineralization. The entire Mastodon group has had several geochemical surveys completed with several lead/zinc anomalies having been outlined to-date. Surface drilling of the anomalies, though, has been discouraging.

The Copper Queen showing (082M 004) is 7 km to the southwest of the J&L. This polymetallic (Cu, Zn, Ag) showing is considered a Kuroko massive sulphide-type deposit (G06). Little work has been done on this deposit to define its overall dimensions.

The Locojo showing (082M 264) is 5 km to the east of the Main Zone. It is a new discovery that has recently been exposed from under a glacier. Weymin Mining Corporation was the original group to stake this showing. The showing is considered a Besshi-type massive sulphide (Cu-Zn-Pb) deposit (G04). It has had very little exploration due to its remote location.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information that should be included in this Technical Report in order to make it not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

This PEA was completed by Micon based on a resource estimate by P&E. It was prepared to analyze the potential economic viability of developing Huakan's J&L Main Zone deposit as underground mining operation with an on-site mineral processing facility.

The PEA is based on the proposed mining and processing of the combined measured, indicated, and inferred mineral resources as defined in Section 14. The portion of the resources included in the mine plan is based on an NSR cut-off value of $110/t, and is presented in Table 25.1.

Table 25.1
Portion of the Mineral Resources included in the Mine Plan

Resource Class	Resource (000 t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
Measured	1,091	6.32	65.91	2.28	4.16
Indicated	1,046	7.10	66.08	2.04	3.94
Inferred	2,759	6.20	72.43	2.22	3.24

(1) Numbers are rounded to reflect the precision of a resource estimate.
(2) Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
(3) The NSR cut-off of $110/t was derived from $75/t mining, $25/t processing and $10/t G&A.

The results of the PEA study comprise the following:

- Target nominal mill feed rate of 1,500 tonnes per year;
- Total mill feed of 4.9 million tonnes;
- Life-of-mine waste development of 1.2 million tonnes;
- Life of mine operations just under 10 years, with mill running through Year 11;
- Longhole stoping with backfilling; dense media separation, milling, flotation, pressure oxidation and carbon-in-leach processing to produce gold doré, a lead concentrate and by-product zinc hydroxide;
- Estimated LOM feed head grades of 6.4 g/t Au, 69.8 g/t Au, 2.2% Pb, and 3.6% Zn;
- Estimated LOM gold recovery of 90.6% to doré and Pb concentrate;
- Estimated LOM silver recovery of 80.4% to doré and Pb concentrate;

- Estimated LOM lead recovery of 74.2% to Pb concentrate;
- Estimated LOM zinc recovery of 74.5% to Zn hydroxide;
- Estimated LOM payable gold production of approximately 915,000 oz and average annual production of approximately 101,000 oz;
- Estimated LOM payable silver production of approximately 8,831,000 oz and average annual production of approximately 973,800 oz;
- Estimated LOM lead production of approximately 176,198,000 lb and average annual production of approximately 19,431,000 lb;
- The project will employ approximately 200 workers;
- Access to the site will be via the existing Forest Service road, upgraded as required;
- Electrical power will be provided by the local utility supplier; on-site generation will be for emergency purposes only;
- Mine closure costs are estimated to be approximately $6.5 million.

The results of the PEA study are summarized in Table 25.2. All costs are in Canadian dollars. It should be noted that this PEA is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that will enable them to be categorized as mineral reserves, and there is no certainty that the conclusions of the PEA will be realized.

Table 25.2
Summary of the J&L Property PEA Study Base Case Results

Item	Unit	LOM Value	Average Annual Value
Total mill feed production	t (000s)	4,896	540.0
Total waste production (Yrs 1-6 only)	t (000s)	1,240	206.7
Average gold grade	g/t Au	6.4	
Average gold process recovery	%	90.6	
Average silver process recovery	%	80.4	
Average lead process recovery	%	74.2	
Average zinc process recovery	%	74.5	
Total gold production	oz (000s)	915	101
Total silver production	oz (000s)	8,831	974
Total lead production	lb (000s)	176,198	19,431
Total zinc production (in hydroxide)	lb (000s)	131,336	14,484
Mine Life (at steady state)	Years	9.1	
Pre-production capital cost	$ millions	263.7	
Sustaining capital	$ millions	35.1	
LOM operating cost	$ millions	554.7	

Item	Unit	LOM Value	Average Annual Value
LOM cash operating cost	$/t mill feed	113	
Average base case gold price	$/oz	1,320	
Average base case silver price	$/oz	22	
Average base case lead price	$/lb	0.98	
Average base case zinc price	$/lb	0.94	
Discount for zinc in hydroxide	%	50	
		Pre-tax	**After Tax**
LOM net revenue	$ millions	1,604	n/a
LOM Project cash flow	$ millions	751	484
NPV @ 8.0% discount rate	$ millions	344	202
NPV@ 10.0 % discount rate	$ millions	279	156
NPV@ 12.0 % discount rate	$ millions	224	117
Project IRR	%	26	21

26.0 RECOMMENDATIONS

Following the completion of this positive PEA study, it is recommended that Huakan advance the development of the J&L Property to the pre-feasibility stage and that baseline data collection continue in order to support potential permitting.

26.1 BUDGET

Huakan has carried out Phase I of its work on the J&L deposit. It is estimated that to complete a pre-feasibility study on the J&L deposit, including drilling, metallurgical testing, and engineering design will cost in the range of $24 million, spread over a further two phases. The next phase, Phase II, of the work program will cost $4.4 million. A budget for Phase II has been compiled by Huakan and is shown in Table 26.1. Micon believes that the estimate is reasonable.

Table 26.1
J&L Phase II Development and Drilling Tracking Budget

Item	Cost ($)
Environmental Studies	40,000
First Nations Consultation	30,000
PAG Waste Pad Construction	54,000
Portal Shotcreting	55,300
Capital Expenditures	261,000
General Parts and Inventory	50,000
Rehabilitation/Slashing Underground Workings	406,800
Underground Development	935,200
Equipment Rental	20,000
Underground Diamond Drilling Program	1,072,000
Updating of Resource	15,000
Subtotal	**2,939,300**
Camp Costs	
Generator Fuel and Maintenance	560,700
Management/Supervision	105,000
Camp costs and water treatment	277,000
Safety Equipment and Training	26,000
Snow Removal and Routine Road Maintenance	92,000
Subtotal	**1,060,700**
Contingency	400,000
Total	**4,400,000**

The Phase II program comprises the following:

- Extend the length of the main track drift 250 m, for a total of 452 m of drift with dimensions 2.2 m by 2.5 m;
- Drive seven new drill bays;

- Drill 40 holes totaling 6,700 m;

The selection of drillhole locations is based on 60 m centres, with the objective of increasing mineral resources in the indicated category.

On completion of work in Phase II, a third phase is planned. Work in Phase III is expected to cost $19.6 million. Although no detailed budget has been completed for Phase III, this cost estimate is derived from industry experience with similar scale projects. Micon believes this estimate is reasonable.

Phase III drilling includes:

- Drive Ramp 1 decline approximately 2,180 m;

- Drive five cross cuts totaling 976 m at elevations ranging from approximately 730 m to 660 m;

- Drill 47 holes totaling 7,800 m.

27.0 DATE AND SIGNATURE PAGE

The effective date of the resource estimate used in this Preliminary Economic Assessment is 16 May, 2011. The effective date of the Preliminary Economic Assessment is 24 April, 2012.

"Tracy J. Armstrong" *{signed and sealed}*

Tracy J. Armstrong, P.Geo.
P&E Mining Consultants Inc.
06 June, 2012

"Fred H. Brown" *{signed and sealed}*

Fred H. Brown, CPG, Pr.Sci.Nat.
P&E Mining Consultants Inc.
06 June, 2012

Catherine A. Dreesbach *{signed and sealed}*

Catherine A. Dreesbach, P.E
Micon International Limited
06 June, 2012

Bogdan Damjanovic *{signed and sealed}*

Bogdan Damjanovic, P.Eng.
Micon International Limited
06 June, 2012

Christopher Jacobs *{signed and sealed}*

Christopher Jacobs, CEng, MIMMM
Micon International Limited
06 June, 2012

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29.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, CPG, Pr.Sci.Nat.

I, Fred H. Brown, residing at Suite B-10, 1610 Grover St., Lynden WA, 98264 USA, do hereby certify that:

1. I am an independent geological consultant and have worked as a geologist continuously since my graduation from university in 1987.

2. This certificate applies to the technical report titled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit, Revelstoke B.C., Canada" (the "Technical Report"),with an effective date of April 24, 2012.

3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987 and obtained a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005. I am registered with the South African Council for Natural Scientific Professions as a Professional Geological Scientist (registration number 400008/04), the American Institute of Professional Geologists as a Certified Professional Geologist (certificate number 11015) and the Society for Mining, Metallurgy and Exploration as a Registered Member (#4152172).

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

- Underground Mine Geologist, Freegold Mine, AAC 1987-1995
- Mineral Resource Manager, Vaal Reefs Mine, Anglogold.. 1995-1997
- Resident Geologist, Venetia Mine, De Beers 1997-2000
- Chief Geologist, De Beers Consolidated Mines 2000-2004
- Consulting Geologist 2004-2012

4. I have visited the Property that is the subject of this Technical Report on December 17, 2010.

5. I am responsible for authoring Sections 14 and 23 of this Technical Report and co-authoring sections 4-12.

6. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.

7. I have had prior involvement with the Project as a co-author of the technical report titled "Technical Report and Resource Estimate J&L Property, Revelstoke, British Columbia Canada", with an effective date of May 16, 2011.

8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.

9. 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 the Technical Report not misleading.

Effective Date: April 24, 2012
Signed Date: June 06, 2012
{SIGNED AND SEALED}
[Fred H. Brown]

____
Fred H. Brown CPG, Pr.Sci.Nat.

CERTIFICATE OF QUALIFIED PERSON

TRACY J. ARMSTRONG, P.GEO.

I, Tracy J. Armstrong, P.Geo., residing at 2007 Chemin Georgeville, res. 22, Magog, QC J1X 0M8, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the technical report titled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit, Revelstoke, BC, Canada" (the "Technical Report") with an effective date of April 24, 2012.
3. I am a graduate of Queen's University at Kingston, Ontario with a B.Sc. (HONS) in Geological Sciences (1982). I have worked as a geologist for a total of 25 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Order of Geologists of Québec (License No. 566), the Association of Professional Geoscientists of Ontario (License No. 1204) and the Association of Professional Engineers and Geoscientists of British Columbia (License 34027);

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

Underground production geologist, Agnico-Eagle Laronde Mine	1988-1993
Exploration geologist, Laronde Mine	1993-1995
Exploration coordinator, Placer Dome	1995-1997
Senior Exploration Geologist, Barrick Exploration	1997-1998
Exploration Manager, McWatters Mining	1998-2003
Chief Geologist Sigma Mine	2003
Consulting Geologist	2003-present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 9 through 12 of the Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Property, as co-author on a previous Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. 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 the Technical Report not misleading.

Effective date: April 24, 2012
Signing Date: June 6, 2012
{SIGNED AND SEALED}
[Tracy Armstrong]

____
Tracy J. Armstrong, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Christopher Jacobs

As co-author of this report entitled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit Revelstoke B.C., Canada", with an effective date of 24 April, 2012 (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 precious and base metal deposits;
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 visited the J&L Property on October 27, 2011;
7. I am responsible for the preparation of Sections 20, 21 (except subsections 21.1.1, 21.1.2, 21.2.1 and 21.2.2), 22, and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
8. I am independent of Huakan International Mining Inc., as defined in Section 1.5 of NI 43-101;
9. I have had no previous involvement with the property;
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 06th day of June, 2012

"Christopher Jacobs" {signed and sealed}

Christopher Jacobs, CEng, MIMMM

CERTIFICATE OF QUALIFIED PERSON

Catherine A. Dreesbach

As co-author of this report entitled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit Revelstoke B.C., Canada", with an effective date of 24 April, 2012 (the "Technical Report"), I, Catherine A. Dreesbach, do hereby certify that:

1. I am employed as a Senior Mining Engineer by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135 email: cdreesbach@micon-international.com
2. I hold the following academic qualifications:
   B.S. Physics, University of California, Davis, 1992;
   M.S. Mining Engineering, Montana Tech of the University of Montana, 1998
   M.S. Environmental Engineering, Montana Tech of the University of Montana, 1998
3. I am a Licensed Professional Engineer, registered with the State of Montana Board of Professional Engineers and Land Surveyors
   (License Number PE 15922);

   Also, I am a professional member in good standing of The Society for Mining, Metallurgy, and Exploration (Member, Division Officer, Committee Member, Associate Editor);
4. I have worked in the minerals industry for 14 years; my work experience includes over 8 years as a mining engineer and regulatory inspector in various hard rock and coal mines, 5 years as a mining engineer in underground gold and trona mines, and the remainder as an independent consultant on various precious and polymetallic deposits.
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 visited the J&L Property on October 27, 2011;
7. I am responsible for the preparation of Sections 2, 3, 15, 16, 18, 19, 21.1.1, 21.2.1, 24, and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
8. I am independent of Huakan International Mining Inc., as defined in Section 1.5 of NI 43-101;
9. I have had no previous involvement with the property;
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 06th day of June, 2012

"Catherine A. Dreesbach" {signed and sealed}

Catherine A. Dreesbach, P.E.

CERTIFICATE OF QUALIFIED PERSON

Bogdan Damjanović

As co-author of this report entitled "A Preliminary Economic Assessment of the Main Zone, J&L Deposit Revelstoke B.C., Canada", with an effective date of 24 April, 2012 (the "Technical Report"), I, Bogdan Damjanović, do hereby certify that:

1. I am employed as a metallurgist by, and carried out this assignment for:
   Micon International Limited, Suite 900 – 390 Bay Street, Toronto, ON, M5H 2Y2
   tel. (416) 362-5135 email: bdamjanovic@micon-international.com
2. I hold the following academic qualifications:
   B.A.Sc., Geological and Mineral Engineering, University of Toronto, 1992
3. I am a Professional Engineer registered with the Professional Engineers of Ontario. (registration number 90420456);

   Also, I am a professional member in good standing of: The Canadian Institute of Mining, Metallurgy and Petroleum (Member);
4. I have worked in the minerals industry for 20 years; my work experience includes 8 years as a metallurgist on gold, copper/nickel and lead/zinc/gold deposits; and the remainder as an independent consultant when I have worked on a variety of precious and base metal deposits;
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 visited the J&L Property on October 27, 2011;
7. I am responsible for the preparation of Section 13, 17, 21.1.2, 21.2.2 and the portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
8. I am independent of Huakan International Mining Inc., as defined in Section 1.5 of NI 43-101;
9. I have had no previous involvement with the property;
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 06th day of June, 2012

"Bogdan Damjanović" {signed and sealed}

Bogdan Damjanović, P.Eng.