TECHNICAL REPORT
AND RESOURCE ESTIMATE
J&L PROPERTY
REVELSTOKE, BRITISH COLUMBIA
CANADA

PREPARED FOR

850-580 Hornby Street
Vancouver, BC, V6C 3B6
Canada

By

P&E Mining Consultants Inc.

NI-43-101 & 43-101F1
Technical Report No. 216

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Signing Date: June 23, 2011
This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Huakan International Mining Inc. ("Huakan") by P&E Mining Consultants Inc. ("P&E"). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in P&E's services and is based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended to be used by Huakan subject to the terms and conditions of its contract with P&E. This contract permits Huakan to file this report as a Technical Report with the Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, "Standards of Disclosure for Mineral Projects". Any other use of this report by a third party is at that party's sole risk.
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</table>
EXECUTIVE SUMMARY

The J&L property represents one of the largest undeveloped polymetallic deposits in British Columbia. The Property is 35 kilometres north of Revelstoke, British Columbia, Canada. The Property consists of 20 mineral tenure claims and 10 crown granted claims for a total of 3,051.73 hectares. The 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, giving the Company 100% unencumbered interest in the J&L property. In December 2010, the Company changed its name to Huakan International Mining Inc. (Huakan).

The Property lies strategically within a geologically attractive lithologic package that hosts numerous mineral deposits. The Property itself has two known and significant precious and polymetallic mineral deposits. The Main Zone is a shear hosted replacement deposit overprinting a pre-existing silver-lead-zinc deposit (the Yellowjacket). The sheeted sulphide vein system is composed of banded massive and stringer arsenopyrite-pyrite-sphalerite-galena mineralization with appreciable content of gold and silver. The Main Zone has been traced on surface by prospecting, trenching and soil sampling for a strike length of over 3 kilometres and traced by drilling for 1.5 kilometres strike length by 0.8 kilometres down dip. The Main Zone generally dips about 60 degrees to the northeast with an average true thickness of 2.5 metres but can reach 15 metres true thickness. The Main Zone is the focus of recent drilling by Huakan and the resulting NI 43-101 resource estimate contained in this report.

The Yellowjacket deposit is a very siliceous sphalerite-galena (Zn-Pb-Ag) stratabound carbonate replacement deposit that sub parallels and is in the immediate hanging wall of the Main Zone. The Yellowjacket per se is believed to be the remains of its former self, the majority of which was cut and remobilized by a major shear zone, which ultimately became occupied by the Main Zone sulphide vein system. The Yellowjacket is not currently the focus of exploration activity and remains with an historic resource estimate dating back to 1991. The Yellowjacket occurs in a series of lenticular bodies each up to 8 metres thick.

Several major mining companies have explored and advanced the J&L deposits. There are a total of 266 drill holes that have been completed on the Property from 1983 to present. This translates to 31,186 metres of drilling. The two most significant assets on the Property are the 1.9 kilometres long 830 drift that has exposed the Main Zone for approximately 0.8 kilometres in length as well as the 550 metre long 832 trackless drift that provides year round underground access to the 830 drift. Several raises and cross cuts have aided in the extraction of several bulk samples and drill stations for defining the deposits. The bulk samples have been used to conduct metallurgical testwork on the Main Zone mineralization. The Main Zone is a complex polymetallic deposit high in arsenic which creates a challenge in the production of saleable zinc and lead concentrates and the economic recovery of gold. Extensive metallurgical testing between the mid 1980’s and 2006 have considered various options and have produced numerous effective options for acceptable recoveries of gold, silver, zinc and lead by making 3 separate concentrates, including using heavy media separation. Limited metallurgical testwork has been performed on the Yellowjacket Zone which appears to have a simpler metallurgy than the Main Zone.

In late 2010, the J&L property underwent renewed exploration activity by Merit/Huakan with the completion of a 7,897 metre, 60 hole, underground drill program focused on the Main Zone with the objective of verifying historic drilling and sampling and infilling a 800 metre strike by
200 metre dip face of the Main Zone with 30 metre centers to support a NI 43-101 compliant resource.

It is apparent that both the Main Zone and the Yellowjacket have potential to expand their current dimensions as defined by the current drill pattern. The Main Zone, in particular, with its tabular predictable geometry and grade, has already a laterally extensive size defined by drilling and remains open in a number of directions. Its surface strike length has been established to be in excess of 3 kilometres.

This report contains details of a Mineral Resource Estimate prepared for the J&L property by P&E (2011). The mineral resource estimate as presented in this report was prepared in accordance with the Canadian Securities Administrators’ National Instrument 43-101 and has been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines.

Based on the economic parameters listed in the Table below, a 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:

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

### TABLE-I
**J&L Economic Parameters**

<table>
<thead>
<tr>
<th>Element</th>
<th>Metal Price $US/lb or oz</th>
<th>Concentrate Recovery %</th>
<th>Smelter Payable %</th>
<th>Refining Chg. $US/lb or oz</th>
<th>Refining Chg. $C/lb or oz</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>$0.99</td>
<td>80%</td>
<td>95%</td>
<td>$0.00</td>
<td>$0.00</td>
</tr>
<tr>
<td>Zn</td>
<td>$0.95</td>
<td>72%</td>
<td>85%</td>
<td>$0.00</td>
<td>$0.00</td>
</tr>
<tr>
<td>Ag</td>
<td>$21.01</td>
<td>88%</td>
<td>91%</td>
<td>$0.50</td>
<td>$0.53</td>
</tr>
<tr>
<td>Au</td>
<td>$1.183</td>
<td>92%</td>
<td>96%</td>
<td>$15.00</td>
<td>$15.79</td>
</tr>
</tbody>
</table>

| $C/$US | $0.950 |

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

<table>
<thead>
<tr>
<th>Element</th>
<th>Payable Metal $C/tonne/g or %</th>
</tr>
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<tbody>
<tr>
<td>Pb</td>
<td>$17.46</td>
</tr>
<tr>
<td>Zn</td>
<td>$13.49</td>
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<tr>
<td>Ag</td>
<td>$0.56</td>
</tr>
<tr>
<td>Au</td>
<td>$34.91</td>
</tr>
<tr>
<td>TOTAL</td>
<td>$66.42</td>
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</tbody>
</table>

| 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 |
The mineral resource estimate for the J&L deposit is reported at a NSR cut-off grade of CDN$110.00 as depicted in the following Table.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Tonnes</th>
<th>Au (g/t)</th>
<th>Au (ozs)</th>
<th>Ag (g/t)</th>
<th>Ag (ozs)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Main Zone</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Measured</td>
<td>1,202,000</td>
<td>6.71</td>
<td>259,200</td>
<td>69</td>
<td>2,664,600</td>
<td>2.4</td>
<td>4.46</td>
</tr>
<tr>
<td>Indicated</td>
<td>1,165,700</td>
<td>6.92</td>
<td>259,200</td>
<td>64.9</td>
<td>2,432,100</td>
<td>2.01</td>
<td>3.86</td>
</tr>
<tr>
<td>Measured &amp; Indicated</td>
<td>2,367,700</td>
<td>6.81</td>
<td>518,400</td>
<td>66.95</td>
<td>5,096,700</td>
<td>2.21</td>
<td>4.16</td>
</tr>
<tr>
<td>Inferred</td>
<td>4,538,100</td>
<td>5.19</td>
<td>757,500</td>
<td>67.8</td>
<td>9,887,800</td>
<td>2.16</td>
<td>2.99</td>
</tr>
<tr>
<td><strong>Footwall Zone</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Inferred</td>
<td>292,800</td>
<td>4.54</td>
<td>42,700</td>
<td>49</td>
<td>461,900</td>
<td>0.91</td>
<td>0.73</td>
</tr>
</tbody>
</table>

(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 $1,183/oz, Ag $21/oz, Pb $0.99/lb, Zn $0.95/lb
   - Exchange rate of US$0.95US = $1.00CDN
   - 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 US$185/t
   - Mass pull of 5% and 8% concentrate moisture content.

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

Additional exploration to expand the resource on the Main Zone is recommended. The program would involve extending underground workings either by extending the 830 decline or extending the 830 track drift cross-cuts in order to establish drill bays. The down dip and strike towards the southeast on the Main Zone hold the best potential to build new resources. There is a 600 metre strike length of the Main Zone where the Main Zone could double its dip length. In addition, although the strike of the Main Zone towards the northwest is open, the southeast direction shows more promise. The Main Zone is traceable by drilling from the 830 drift up dip to surface with Inferred Resources. There is potential to upgrade these Inferred Resources to Indicated Resources with infill drilling to 60 metre centers. This is probably a secondary priority to the down dip and strike length target areas due to the generally thinner nature of the Main Zone in the up dip area.

If silver, lead and zinc prices continue to rise and remain high, the Yellowjacket deposit could show potential economic merits and may be considered for further exploration and definition.

P&E Mining Consultants Inc.
J&L Property Technical Report No. 216
1.0 INTRODUCTION AND TERMS OF REFERENCE

1.1 TERMS OF REFERENCE

P&E Mining Consultants Inc. (P&E) was engaged by Huakan International Mining Inc. (“Huakan” or the “Company”) to provide a National Instrument 43-101 compliant resource estimate for the Main Zone on the J&L property (the “Property”) located 35 kilometres north of Revelstoke, BC. Resource estimation work was undertaken in compliance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserve definitions that are referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects. This Technical Report has been prepared in compliance with the requirements of Form 43-101.F1.

Mr. Fred Brown, CPG, Pr.Sci.Nat, Dr. Wayne Ewert, P.Geo., and Ms. Tracy Armstrong P.Geo., of P&E, served as the Qualified Persons responsible for preparing the Technical Report. The resource estimation work was conducted by Fred Brown, CPG, Pr.Sci.Nat.

Paul Cowley, P.Geo. a consultant and VP Exploartion for Huakan supervised the 2010-2011 drilling and sampling activities for the J&L Main Zone exploration program and was an invaluable “Source of Information” for P&E.

1.2 SOURCES OF INFORMATION

In preparing this report, P&E obtained information from geological reports, maps and miscellaneous technical papers listed in the References (Section 19) of this report, as well as from public information, and Huakan’s (Merit’s) experience in British Columbia.

The site was visited on December 17, 2010 by Mr. Fred Brown, CPG, Pr.Sci.Nat., during which time the 2010 underground drilling program was reviewed and many of the drill collars were substantiated. P&E collected a number of representative drill core sample intervals both from historic drill holes as well as from the current 2010 drill program for independent confirmation analysis. These samples were sent to ALS Chemex Laboratories in NorthVancouver, BC and the results are included in Section 13 of this report.

1.3 UNITS AND CURRENCY

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 ton (“oz /T”) are specifically stated. Base metal assay values are given in percent (“%”) or in parts per million (“ppm”). The CDN$ is used throughout this report. For the purposes of this report 1 CDN$ = 1 US$.

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

1.4 GLOSSARY AND ABBREVIATION OF TERMS

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 below.

“$” and “CDN$” means the currency of Canada.
“AA” is an acronym for Atomic Absorption, a technique used to measure metal content subsequent to fire assay

“asl” means above sea level

“Au” means gold

“Azi” means azimuth

“Company” means Huakan International Mining Inc. previously Merit Mining Corp.)

“CIM” means the “Canadian Institute of Mining, Metallurgy and Petroleum”

“CSA” means the Canadian Securities Administrators

“DDH” means diamond drillhole

“E” means east

“el” means elevation level

“g/t” means grams per tonne

“g/t Au” means grams of gold per tonne of rock

“ha” means Hectare

“IP” means Induced Polarization

“kg” means kilogram

“km” means kilometre equal to 1,000 metres or approx. 0.62 statute miles

“m” means metric distance measurement equivalent to approximately 3.27 feet

“M” means million

“Ma” means millions of years

“Mt” means millions of tonnes

“N” means north

“NE” means northeast

“NI 43-101” means Canadian Securities Administrators National Instrument 43-101

“NTS” means National Topographic System

“NW” means northwest.

“NSR” is an acronym for “Net Smelter Return”, which means the amount actually paid to the mine or mill owner from the sale of ore, minerals and other materials or concentrates mined and removed from mineral properties, after deducting certain expenditures as defined in the underlying smelting agreements

“oz/T” means ounces per ton

“P&E” means P&E Mining Consultants Inc.

“PEA” means a Preliminary Economic Assessment study

“ppm” means parts per million

“Property” means the J&L property in the Revelstoke area, B.C., Canada

“S” means south

“SE” means southeast

“SEDAR” means the System for Electronic Document Analysis and Retrieval

“SW” means southwest

“t” means metric tonne equivalent to 1,000 kilograms or approximately 2,204.62 pounds

“T” means Short Ton (standard measurement), equivalent to 2,000 pounds

“US$” means the currency of the United States

“UTM” means Universal Transverse Mercator grid coordinate system as used for this project (NAD 83- Zone 11).

“W” means west
2.0 RELIANCE ON OTHER EXPERTS

P&E has assumed that all the information and technical documents listed in the Sources of Information section of this report are accurate and complete in all material aspects. While we carefully reviewed all the available information presented to us, we cannot guarantee its accuracy and completeness. We reserve the right, but will not be obligated to revise our report and conclusions if additional information becomes known to us subsequent to the date of this report.

Although copies of the licenses and work contract were reviewed, P&E has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties. Information relating to tenure was reviewed by means of the public information available through the Mineral Titles Branch of the BC Ministry of Energy, Mines, and Petroleum Resources MTO land tenure database. P&E has relied upon this public information, as well as tenure information from Haukan and has not undertaken an independent detailed legal verification of title and ownership of the J&L Property claims.

A legal land survey of the claims has not been undertaken.

The authors have drawn heavily on the documents listed in the Sources of Information as well as upon information gained during the site visit, however, the conclusions and recommendations are exclusively the authors. The results and opinions outlined in this report are dependent on the aforementioned information being current, accurate and complete as of the date of this report and it has been assumed that no information has been withheld which would impact the conclusions or recommendations made herein.

A draft copy of the report has been reviewed for factual errors by Huakan. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.
3.0 PROPERTY DESCRIPTION AND LOCATION

Material relevant to this section is contained in the following Technical Report but summarized below:


The J&L property is located in southeastern British Columbia, approximately 32 air kilometres 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 3.1).

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 hectares if there was 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 3.1, the crown grant claims in Table 3.2 and both are illustrated in Figure 3.2.

<table>
<thead>
<tr>
<th>Tenure Number</th>
<th>Claim Name</th>
<th>Expiration Date</th>
<th>Mining Division</th>
</tr>
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<tr>
<td>398402</td>
<td>J1</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
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<tr>
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<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398406</td>
<td>J5</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398407</td>
<td>J6</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398408</td>
<td>J7</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398409</td>
<td>J8</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398410</td>
<td>J9</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398411</td>
<td>J10</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398412</td>
<td>J11</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>398413</td>
<td>J12</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>399179</td>
<td>Sage</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>399180</td>
<td>J13</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>399181</td>
<td>J14</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>399182</td>
<td>J15</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>401774</td>
<td>Brush</td>
<td>15/11/2017</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>606405</td>
<td>Yellow Jacket</td>
<td>31/06/2011</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>620103</td>
<td>Hardpan</td>
<td>31/06/2011</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>805402</td>
<td>A &amp; E - W</td>
<td>30/06/2011</td>
<td>Revelstoke</td>
</tr>
</tbody>
</table>

*Note: Claim status as of April 15, 2011*
TABLE 3.2
J & L CROWN GRANTED CLAIMS

<table>
<thead>
<tr>
<th>Claim Number</th>
<th>Claim Name</th>
<th>Mining Division</th>
</tr>
</thead>
<tbody>
<tr>
<td>L 14821</td>
<td>Goat Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14822</td>
<td>Goat No. 2 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14823</td>
<td>Goat No. 3 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14824</td>
<td>Goat No. 4 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14825</td>
<td>Goat No. 5 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14826</td>
<td>Goat No. 6 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14827</td>
<td>View Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14828</td>
<td>View No.2 Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 14829</td>
<td>Creek Fraction</td>
<td>Revelstoke</td>
</tr>
<tr>
<td>L 7408</td>
<td>Aberdeen</td>
<td>Revelstoke</td>
</tr>
</tbody>
</table>

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 NSR royalties owing to any parties.

Figure 3.1  Regional Location Map
Figure 3.2  Claim Map, J&L Property
4.0 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 ACCESSIBILITY

Vehicle access to the area is via Provincial Highway 23, 32 kilometres 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 kilometres 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 plowing has become wholly undertaken 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.

4.2 CLIMATE

The summer weather is considered moderate with average temperatures between 16° to 30° celsius, with long stretches of sun and rain. The rain at times can be very heavy. The average precipitation is 65 cm/year. Winters are long and are characterized by heavy snowfalls (1 to 4 metres) 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.

4.3 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, which include forestry, tourism, transportation (mainly CP Rail) and public services.

4.4 INFRASTRUCTURE

The Property has several adits and numerous trenches. Only two (2) 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 832 portal. Electric power is produced by on-site diesel generators and a satellite phone and internet system is in place. Selkirk Helicopters sometimes use the Property as a re-fueling station for their operations. A skid mounted fuel tank is located about 200 metres from the 832 portal on the Forest Service Road.

The nearest population center is the Town of Revelstoke (population approx. 8,500), which is located approximately 32 kilometres 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 is present on the eastern end of the town (owned by Huakan). 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 2 kilometres north of Revelstoke and produces power for a large portion of British Columbia. There are no powerlines running along Highway 23 although there is an underground telephone line.

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

4.5 PHYSIOGRAPHY

The topography is characteristic of the Selkirk Mountains. The elevation ranges from 700 to 3,050 metres, mean sea level. The topographic relief is a result of recent alpine glaciation. Incised creeks such as McKinnon Creek created narrow valley floors while major creeks (i.e. 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. Numerous avalanche chutes occur in the area. All of these conditions make traversing the Property hazardous and time consuming. An avalanche chute occurs beside the 830 portal which prompted the driving of the 832 trackless drift. The 832 trackless drift allows safe year round access to the underground network. Flat ground is limited on the Property, however, there appears to be enough for a millsite and waste rock piles should the project advance to production. There is a tributary valley 3km upstream on Carnes Creek that might be servicable as a tailings facilities, but would require study to prove up.

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 that can change its flow volume 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 mineral 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 at approximately 1,980 metres elevation. The Carnes and Tumbledown Glaciers are immediately east of the Property boundary.
5.0 HISTORY

5.1 EARLY REGIONAL 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 kilometres 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. A summary of property activities is presented in Table 5.1. A depiction of the underground workings is presented in Figure 5.1.

<table>
<thead>
<tr>
<th>Year</th>
<th>Company</th>
<th>Exploration</th>
</tr>
</thead>
<tbody>
<tr>
<td>1912</td>
<td>E. McBean</td>
<td>Collaring of the 986 level portal (91 metres long) and 2 shallow shafts (each 46 metres deep).</td>
</tr>
<tr>
<td>1924</td>
<td>E. McBean</td>
<td>Metallurgical tests were attempting to resolve problems due to the high arsenic content of the ore.</td>
</tr>
<tr>
<td>1924-1927</td>
<td>Porcupine Goldfields Development and Finance Company</td>
<td>43 metres of underground drifting on two levels. In 1925, Mr. 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.</td>
</tr>
<tr>
<td>1928</td>
<td>Geological Survey of Canada</td>
<td>J&amp;L area mapped under the direction of Dr. H. Gunning.</td>
</tr>
<tr>
<td>1934</td>
<td>Mr. T. Arnold</td>
<td>Crown granted claims acquired.</td>
</tr>
<tr>
<td>1935</td>
<td>Raindor Gold Mines</td>
<td>Property optioned, 986 Level Adit extended a further 152 m on the Main Zone.</td>
</tr>
<tr>
<td>1946</td>
<td>Raindor Gold Mines</td>
<td>2 shafts deepened, collectively to 117 metres.</td>
</tr>
<tr>
<td>1952</td>
<td>Asarco</td>
<td>Several trenches were completed on the Main Zone.</td>
</tr>
<tr>
<td>1965</td>
<td>Westair Mines Ltd.</td>
<td>A new portal (830 level adit) was collared to explore the Main Zone. Total length was 297 metres. It is one of the major underground assets on the Property. A road (12.4 Km) was built into the property from the Big Bend road (now Hwy 23).</td>
</tr>
<tr>
<td>1980</td>
<td>Pan American Minerals</td>
<td>Property leased from T. Arnold.</td>
</tr>
<tr>
<td>1981-1985</td>
<td>BP Minerals, Selco Division</td>
<td>Surface exploration program. 830 Level (Tracked) Adit extended an additional 1,333 metres of drift and cross cuts. 64 underground drill holes advanced for 2,640 m.</td>
</tr>
<tr>
<td>1988</td>
<td>Pan American Minerals</td>
<td>830 Level (Tracked) Adit extended an additional 250 metres of drift and cross cuts and completed 4 raises totalling 120 metres.</td>
</tr>
</tbody>
</table>
### TABLE 5.1
**SUMMARY OF HISTORICAL EXPLORATION**

<table>
<thead>
<tr>
<th>Year</th>
<th>Company</th>
<th>Exploration</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>sample from 3 “Take-Down-Backs” was collected for metallurgical studies.</td>
</tr>
<tr>
<td>1991</td>
<td>Cheni Gold Mines</td>
<td>Cheni 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 metres was collared.</td>
</tr>
<tr>
<td>1996</td>
<td>Weymin Mining Corporation</td>
<td>3 surface boreholes were advanced for 503 metres of drilling. A 120 tonne underground bulk sample was retrieved from the 830 level for metallurgical studies from six sample locations.</td>
</tr>
<tr>
<td>1998</td>
<td>H.A. Simons</td>
<td>Completed ‘McKinnon Creek Property Scoping Study.’</td>
</tr>
<tr>
<td>2004</td>
<td>BacTech Mining Corporation</td>
<td>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.</td>
</tr>
<tr>
<td>2007</td>
<td>Merit Mining Corp.</td>
<td>100% interest in the J&amp;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 1363 metres of drilling.</td>
</tr>
<tr>
<td>2008</td>
<td>Merit Mining Corp.</td>
<td>Rehabilitation of 832 drift was extended a further 550 metres with the 5 metre by 5 metre profile (to allow for 30 ton trucks) and connected to the 830 track drift, approximately 310 metres in from the 830 portal. Tunnelling completed by September 2008 at which time the program was suspended due to financial constraints and a downturn in world metal prices.</td>
</tr>
<tr>
<td>2010</td>
<td>Merit Mining Corp.</td>
<td>Mining activities resume to generate a NI-43-101 compliant resource.</td>
</tr>
</tbody>
</table>
5.2 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 5.2.

<table>
<thead>
<tr>
<th>Year</th>
<th>Drillholes</th>
<th>Total Metres</th>
<th>Company</th>
</tr>
</thead>
<tbody>
<tr>
<td>1962 to-1967</td>
<td>U/G</td>
<td>183</td>
<td>Westairs Mines Ltd.</td>
</tr>
<tr>
<td>1983 to 1984</td>
<td>65 UG drill holes</td>
<td>2640</td>
<td>BP Selco Ltd.</td>
</tr>
<tr>
<td>1987 to 1988</td>
<td>20 UG drill holes</td>
<td>1914</td>
<td>Pan American Minerals</td>
</tr>
<tr>
<td>1988 to 1989</td>
<td>32 UG drill holes</td>
<td>2985</td>
<td>Equinox Resources Ltd.</td>
</tr>
<tr>
<td>1990 to 1991</td>
<td>50 UG drill holes</td>
<td>13889</td>
<td>Equinox Resources Ltd/Cheni Gold Mines Ltd.*</td>
</tr>
<tr>
<td>1997</td>
<td>3 UG drill holes</td>
<td>503</td>
<td>Weymin Mining Corp.</td>
</tr>
<tr>
<td>2006</td>
<td>2-3 UG holes</td>
<td>undisclosed</td>
<td>BachTech Mining Corp.</td>
</tr>
<tr>
<td>2007</td>
<td>9 SFC drill holes</td>
<td>1363</td>
<td>MMerit Mining Corp.</td>
</tr>
<tr>
<td>2010-2011</td>
<td>60 UG drill holes</td>
<td>7897</td>
<td>Merit/Huakan International Mining Inc</td>
</tr>
</tbody>
</table>

5.3 HISTORIC RESOURCE ESTIMATES

In 1991, Equinox Resources Limited completed a historical resource estimate for the J&L deposit (Table 5.3).
TABLE 5.3
SUMMARY OF THE HISTORICAL RESOURCE ESTIMATES FOR THE J&L DEPOSIT

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

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

In 2007, David Makepeace 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 (Figure 5.2). Makepeace used a standard circular polygonal calculation, weighted average method for both zones.

**Figure 5.2** Historical Mineral Estimate Map – J&L Property
The 1998 Scoping study by H.A. Simons provided an analyses of 6 cases, exclusively on the Main Zone (The Yellowjacket Zone was not analyzed). The two favoured cases were:

- Base case @ 1,000 tpd with all processing at McKinnon Creek;
- Base case @ 1,500 tpd, grind, float and pressure oxidize at Goldstream, required a 4 million tonne deposit. (Note: the Goldstream facility was sold by International Bethlehem Mining Corp. To Barkerville Gold Mines in fall 2010 and the mill is planned to be relocated to Wells, BC in the summer of 2011).

The Capital costs for the above scenarios above were $81.7 million and $115 million, respectively. At metal prices of $US 350 per oz Au (currently > $1500), $US 6 per oz Ag (currently > $38.00/oz), $US 0.55/lb Zn (currently $0.98/lb) and $US 0.30/lb Pb (currently $1.12/lb) and a $Can to $US Exchange Rate of 0.70 (currently $1.05) 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 EQ and $180/oz EQ, respectively.
6.0 GEOLOGICAL SETTING

6.1 REGIONAL GEOLOGY

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., 1996, 7 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 map area. Recent mapping by provincial government geologists has outlined the regional geology of the area.

6.2 PROPERTY 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 Lardeo 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.

The following is a brief description of the main geological units present on the J&L property as provided by P. Cowley (pers. communication). A stratigraphic column displaying the age relationships of units is presented below and detailed in Figure 6.1.

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)

6.2.1 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.
6.2.2 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 original J&L claims. 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).

6.2.3 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 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.

6.2.4 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. 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 600 metres northwest of the North Zone Pit (approximately 500 metres 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.

6.2.5 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).

6.2.6 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 6.1  J&L Geology Map

LEGEND

- X: Sulfide Mineralization
- 6: A&E (Pb-Zn-Ag-Au)
- 7: Roseberry (Au)
- I: Undivided Intrusive
- Iu: Upper Index Fm.
- II: Lower Index Fm.
- B: Badshot Fm.
- M: Mohican Fm.
- L: Limestone
- H: Hamill Group

J&L PROJECT GEOLOGY MAP

Huakan International Mining

Scale: 1:50,000  Map Sheet: N.T.S 82M/RE
Date: May 17, 2011  Drafted By: T. Song, GIT
6.3 DEPOSIT GEOLOGY

Proximal to the Main Zone, the lithological assemblage consists of phyllite and schist (87%), limestone (8%), quartzite (5%), and rare dykes as defined 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 amount, 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 metres, 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 metres. 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 centimeter 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.

6.3.1 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 transposed 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 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., 1996, 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, sulfide 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 stike-length.

For much of the Main Zone exposed along strike in the underground, 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 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.
7.0 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, VMS (Gold stream) and 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 7.1.

Figure 7.1 A generic SEDEX-type deposit model (Wilkinson 2007)

7.1 YELLOWJACKET ZONE

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

Irish type deposits are characterized by:

- Active tectonics during sedimentation and some of the mineralization;
- Deposits are hosted by Carboniferous carbonates, basal section of the Waulsortian mud mound complex and Navan beds;
- 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 (Figure 7.2);
• 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.

Figure 7.2  Diagrammatic cross section showing mineral zoning in sedimentary exhalative Zn-Pb deposits (from Briskey, 1986)

Typically, the Yellowjacket deposit occurs about 30 metres into the hangingwall (northeast) from and sub parallels the Main Zone, however, locally there is Yellowjacket type mineralization in the immediate hangingwall 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.
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 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 kilometres. The drilling to date demonstrates that the Main Zone continues for a striklength of at least 1.2 kilometres and in the down dip direction for at least 800 metres. 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 metres thick of sheeted sulphide veining but the sheeting can reach up to 15 metres 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 Exhalitite 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.
8.0 MINERALIZATION

8.1 MAIN ZONE

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 metres but the sheeted sulphide veining can reach 15 metres in true thickness. The continuity of the zone is broken by its absence in a few places within narrow sections as shown in Figure 8.1.

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

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 10’s of metres but appear to resume in an adjacent band. Individual massive sulphide bands frequently range from 5 centimetres to 1 metre 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. Where the mineralization narrows, it is almost completely composed of arsenopyrite. Mineralization widens and 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 sulfide 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 sulfides 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 metres, it appears to have limited extent both along the strike and vertically.

Gangue minerals to the Main Zone 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 metres wide. Marblization occurs immediately at the contact between limestone and the margins of the mineralized zone, varying in width from 0.1 to 1 metre.

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 hanginwall zones of similar Type I, II and III mineralization occur proximal to the Main Zone. The HM1 zone lies approximately 5 metres into the hangingwall of the Main Zone. The HM2 zone lies approximately 8-10 metres into the hangingwall of the Main Zone.
The FM1 zone lies approximately 2.5-10 metres into the footwall of the Main Zone and has a degree of continuity so that a resource can be calculated. The FM2 zone lies approximately 15-20 metres into the hangingwall of the Main Zone.

8.2 YELLOWJACKET ZONE

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 metres into the hangingwall rock 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 metres but remains open laterally in both directions and at depth. The deposit appears to pitch to the southeast at 30°.

Verification drilling and sampling would be required in order to justify a NI 43-101 compliant resource on the Yellowjacket deposit. It is apparent that the Yellowjacket Zone has potential to expand beyond the limits of the current drill pattern. The deposit was only discovered in 1991, late in the exploration history of the evaluation of the Main Zone.

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 occurring 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.

8.3 OTHER SHOWINGS

The Roseberry showing lies on the J&L property (082M 091) 4.5 kilometres to the northwest of the Main Zone (see Figure 3.2). 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 metres.

The A & E showing lies on the J&L property (082M 099) 5 kilometres north of the Main Zone (see Figure 3.2) and 2 kilometres 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 4 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 metres along strike and 160 metres 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 metres to 150 metres 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 metres horizontally and 90 metres 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 metre. This zone could be tested further by diamond drilling.
9.0 EXPLORATION

9.1 PREVIOUS EXPLORATION

Details of historical exploration on the Property can be found in Section 5.0 History

9.1.1 Geological / Prospecting

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 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. 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 property 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 118º15’N with a total of 699 kilometres flown covering an area of 232 square kilometres. 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 zone 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 of 1982, covering a portion of the area covered by the earlier Questor survey. The Dighem II survey was carried out along 396 kilometres of flight lines covering an area of 396 square kilometres. 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, one kilometre south of the McKinnon Creek valley.

During July of 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 kilometres of coverage testing 700 metres 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 metres. 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 metres 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 1982 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 kilometres 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.

The authors are 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 except underground drifting and drilling.
10.0 DRILLING

10.1 HUAKAN DIAMOND DRILLING 2007-2010

The Company at the time known as Merit Mining, completed a 1,363 metre, 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 zinc-lead-silver 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 metre underground drill program with the objective of verifying historic drilling and to broaden the known resource in the Main Zone. This drilling was intended to provide more data for a National Instrument 43-101 compliant resource estimate. The program was expanded to 7,897 metres 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 metres down the hole, usually including a near to collar test as well as a near to bottom hole test. All azimuth readings taken during the downhole surveys had a declination factor of 17º added to them to give true azimuth readings for the pertinent region of British Columbia. Other data collected were dip angles recorded at the various downhole reading sites as well as magnetic susceptibility. Any erroneous results caused by strong magnetism put a small number of the azimuth readings in question, and the readings were therefore eliminated.

10.1.3 Core Recovery and Storage

Core recoveries throughout the 2010/2011 underground J&L drill program were normally >90% and often >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 of Aldergrove, BC was the drilling contractor for the 2010/2011 program. Drilling took place between November 11, 2010 to January 31, 2011. Drilling was carried out on
two ten hour 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 SAMPLING METHOD AND APPROACH

A total of 956 split core samples from the Huakan 2010/2011 diamond drill program were collected and analyzed by Huakan. Sampling was carried out where visual sulphide concentrations became apparent beyond non-mineralized host-rocks. Sample intervals were generally less than 0.5 metres in stronger sulphide concentrations, with lesser mineralization sampled in intervals ranging from between 0.5 to 1.0 metre and rarely to 1.5 metres. Occasional narrower sample intervals ranged from between 0.25 to 0.5 metres on massive vein sections.

A total of 427 bulk density measurements were taken on site on the 2010/2011 core by competent Company geological staff. The bulk density measurements utilized the wet immersion technique.

P & E has not observed any adverse drilling or sampling factors that would affect the accuracy and reliability of the core samples. All core is considered to be representative of the mineralization that was drilled.
12.0 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 metres long but rarely greater than 1.5 metres long. A minor number of samples taken were less than 0.5 metres in length, but not less than 0.25 metres 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 (ie. duplicate sample #18 would be 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 would place 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 Kamloops, B.C. Huakan has archived all of the original assay certificates for both the 2007 and 2010/2011 drill programs.

Eco Tech Laboratory Ltd. 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 250g) are catalogued and logged into the sample-tracking database. During the logging in process at the lab, 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 gram 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 gram sample size is fire assayed along with certified reference materials using appropriate fluxes. The flux used is pre-mixed, purchased from Anachemia 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 dore 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 gram sample is digested with a 3:1:2 (HCl:HNO3:H2O) for 90 minutes in a water bath at 95°C. The sample is then diluted to 10ml 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 lab. 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 >30g/t, Cu, Pb, Zn >1.0% are run as an assay.

When samples returned over limits for lead, zinc, silver and arsenic, they were run for 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 RO 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.
13.0 DATA VERIFICATION

13.1 SITE VISIT AND INDEPENDENT SAMPLING

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) 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:


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 13.1 to Figure 13.4.
Figure 13.1  P & E Verification Sample Results for Gold

Figure 13.2  P & E Verification Sample Results for Silver
13.2 HUAKAN QC PROGRAM

13.2.1 Reference Material

For the 2010/2011 drill programs Huakan geologists inserted certified reference materials and blanks, which were obtained from CDN Resource Laboratories 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.

13.2.2 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.

13.2.3 Duplicate Sampling Program

The 2010/2011 J&L drill program implemented a duplicate core sampling procedure which was designed 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 would provide the 50% split for the 18th or duplicate sample.

In addition, P&E examined the lab 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.

13.2.4 Sampling and QC Recommendations

The QC program as implemented and monitored by Huakan has ensured a robust, accurate and precise database. 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.
14.0 ADJACENT PROPERTIES

The J&L property is situated in a well mineralized area of the province, surrounded by several different types of mineralized showings, all within 10 kilometres of Main Zone portals.

The Mastodon Mine is five kilometres 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 x 60 x 3 metres) 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 seven kilometres 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 five kilometres 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 on it due to its remote location.
15.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Huakan has not conducted any mineral processing or metallurgical testing on the J&L property. The following information in this section describes mineral processing and metallurgical testing performed by previous companies and have been compiled by Huakan from earlier study reports by H.A. Simons and AMEC. P&E have not independently reviewed or assessed the testwork.

15.1 SUMMARY

The majority of the work to date has focused on producing saleable lead and zinc concentrates by various flotation separation techniques. Fewer 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.

J&L is a fine grained massive polymetallic orebody with very complex mineralogy. Major minerals present include pyrite, arsenopyrite, galena, bournonite, freibergite, sphalerite and chalcopyrite. Preliminary test work indicates the ore is amenable to heavy media separation whereby up to half of the feed weight can be rejected with less than 15% of the metal values. The liberation size and specific gravity for heavy media separation have not yet been optimized. Ores require very fine primary grinding (P80=44µm) followed by regrinding to achieve only partial liberation.

Gold appears in two modes, the majority of the gold (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 gold liberates on grinding and 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. Current lead and zinc concentrates have high levels of arsenic, antimony and mercury. 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 extractions from arsenopyrite concentrates treated by pressure oxidation and cyanidation are approximately 90% and 50%, respectively. Extractions for arsenopyrite concentrates treated by biooxidation and cyanidation are approximately 90% and 80%, respectively.

Numerous mineralogical and metallurgical studies and reports have been carried out for the J&L Main Zone deposit, with all known studies/reports segregated and listed in the References section of this report. The metallurgy studies and assessments date from 1982 through to 2005 with particularly 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. and Weymin Mining Corporation. The Weymin bulk sample work was substantial and was incorporated into the H.A. Simon’s scoping study. The last metallurgical study for J&L was carried out in 2005 on behalf of BACTECH Mining Corporation by Process Research Associates Ltd. (PRA).
15.2 METALLURGICAL TESTWORK

15.2.1 Flotation

The following mineralogical observations are lengthy but important, as they are considerate of all the testwork carried out to date and they explain the difficulties encountered with conventional techniques.

In 1982, Lakefield assessed the mineralogy of samples provided by B P Resources (BPR). Lakefield described the major mineralization consisting of pyrite and arsenopyrite, with which chalcopyrite, galena, and sphalerite were associated. Sulphide grain size ranged from 1,200 µm to less than 10 µm. Bedded and unliberated sulphides were finer grained from a maximum of 210 µm to less than 10 µm.

In 1985, the B P Research Center (BPRC) in the United Kingdom carried out an extensive mineralogical study on six ore types ground to -53 µm + 38 µm. The following major minerals were encountered: sphalerite (ZnS); arsenopyrite (FeAsS); pyrite (FeS2); galena (PbS); antimonian galena ((Pb, Sb)S) and lead sulfosalts bournonite (CuPbSbS3); and meneghinite (Pb15Cu Sb7S24).

In most samples, sphalerite was either locked or contaminated by inclusions of lead sulphides. The disseminated lockings can be antimonian. Lead sulphides occurred mainly as disseminations in sphalerite and were invariably pure galena. Grains over 10 µm are occasionally antimonian. Arsenopyrite is mostly liberated in all samples although it rarely contains disseminated inclusions of lead sulphides. Pyrite was similar to the arsenopyrite. A very small number of auriferous grains (3 of electrum and 1 of native gold) were located. The grains were all in the 2 µm to 20 µm size range. Argentiferous grains all occurred as freibergite (Cu, Ag)10(Fe, Zn)2Sb4S13 in the 2 µm to 100 µm size range. Roughly 27% of the freibergite was associated with the pyrite, 23% with the galena, 17% with the sphalerite and 6% with the arsenopyrite. The remainder of the freibergite was associated with chalcopyrite, antimonian galena and gangue minerals.

BPRC proceeded to produce lead, zinc and arsenopyrite flotation concentrates. Clean lead concentrates were impossible to achieve due to the antimonian nature of the galena. In general, for every 10% lead, 1% antimony was present. Lead concentrates also contained 2% to 3% arsenic and 10% to 20% zinc due to the fine grained nature of the galena-sphalerite-arsenopyrite lockings. A lead concentrate averaging 50% lead is typical and contained 60% to 65% of the silver and, more significantly, 30% to 35% of the gold.

In contrast to lead, high grade zinc concentrates could be produced with lower levels of antimony and arsenic. Arsenic levels in the concentrate ranged from 0.5% to 1%. Lead in the zinc concentrate ranged from 1% to 5%, depending on the amount of lead inclusions in the sphalerite. Gold and silver reporting to the zinc concentrates are associated with the lead and arsenic inclusions.

Arsenopyrite-pyrite concentrates were easily produced with 80% of the arsenic recovered to a concentrate containing 10%-27% arsenic. This percentage was dependent on the arsenic grade in the ore. Gold grade mirrored arsenic content, being approximately 1 ppm per 1% arsenic in the concentrate.
Recoveries of the lead to the lead circuit averaged approximately 70%, with losses of the lead to the zinc and arsenopyrite circuits being high generally due to 10 µm to 20 µm locked galena. In some ore types, lead is exsolved in sphalerite making it unrecoverable to the lead concentrate. Sphalerite, being coarser grained than galena, showed high recoveries to a zinc concentrate of 70% to 80%. Higher zinc recoveries are limited because of the losses to the lead circuit. Arsenic and pyrite were generally coarse grained and liberated with regrinding so that high recovery is possible. Some fine grained intergrowths of arsenopyrite and pyrite with sphalerite and especially galena occurred leading to arsenic in those concentrates. The bulk of antimony (75% to 80%) reports to the lead concentrate because it occurs in solid solution with galena or as freibergite. Silver, like antimony, mostly reported to the lead concentrate (60%-65%) since it also occurs in solid solution with the galena as freibergite or as electrum, which floated with the lead. Silver reported to the arsenopyrite concentrate again as freibergite. BPRC considered there were two modes of gold occurrence: fracture filling/veinlets in arsenopyrite, or coarse grains locked to gangue and other sulphides. This gold can liberate on grinding and float with the lead or stay locked with the galena-arsenopyrite-pyrite middlings reporting to the lead concentrate. This observation was supported by gold extraction by cyanidation in the order of 10%.

In 1985, Lakefield carried out more mineralogy to assist in explaining poor metallurgical response in the lead zinc separation, when high zinc deportment to the lead and low lead concentrate grade was observed. Greater than 60% of the sphalerite contained inclusions of fine grained galena, lead sulphosalts plus arsenopyrite. Bournonite was more abundant than galena. In purest form, bournonite is 42.6% lead and galena is 86% lead. The galena grain size ranged from 800 µm to less than 5 µm. Sixty percent of the pyrite was identified as fine grained inclusions to coarse grained attachments to arsenopyrite plus zinc and lead minerals. Arsenopyrite replaced some pyrite and was found to have common and abundant inclusions in sphalerite. Fractured arsenopyrite grains were commonly cemented by bournonite and galena. Electrum (Au Ag) was observed in over 50 occurrences, all less than 5 µm and associated with arsenopyrite, bournonite and galena.

In 1987, Noranda Research evaluated four composites from the 830 Adit. Noranda produced a bulk lead zinc concentrate with 50% combined lead plus zinc. Metal recoveries were 60% to 65% lead, 80% to 85% zinc, 20% to 25% gold and 60% silver. The bulk concentrate contained up to 1% copper, 1% antimony, 100 ppm mercury and 2% to 5% arsenic. Noranda identified one of the four composites that did not respond well. They went on to succeed in recovering an arsenopyrite concentrate for pressure oxidation, but attempts to separate arsenopyrite from pyrite in order to upgrade the arsenopyrite concentrate were unsuccessful.

Later in 1987, Met Engineers carried out a program commissioned by Panamerican Minerals (PAM) that produced results similar to those achieved by Lakefield. A fine primary grind to 80% passing 80 µm with regrinding to 80% passing 20 µm was identified.

In November 1987, Bacon Donaldson carried out several batch flotation tests aimed at producing saleable lead, zinc and arsenopyrite concentrates. Locked cycle testing was terminated after four cycles due to high circulating loads of zinc and silver. Arsenopyrite and lead concentrates were pressure leached using the Arseno process (nitric acid leach). Pressure leaching the lead concentrate produced a residue containing 50.9% lead and 1.34% arsenic.

In March 1989, Lakefield carried out a program commissioned by ERL. Roughly 37 composites of 50 drill holes and a bulk sample were evaluated. The bulk sample (generally less than 2 inches) responded well to heavy media separation at a specific gravity (SG) of 3.0. Drill hole
composites crushed to 10 mesh showed varied response, but the SG and liberation size analysis needed to be optimized. Direct cyanidation of the ore resulted in approximately 20% gold extraction and 30% silver recovery. Cyanidation of the lead concentrate recovered 76.9% of the gold and 5.6% of the silver in the concentrate. The flotation response of each of the drill hole composites was highly variable. After three cleaning stages, lead concentrates ranged from 25% to just over 50% lead with arsenic concentrations of 1.3% to 12%. Some tests resulted in 50% lead and 2.5% arsenic. Very good zinc grades were achieved in five of the samples with grades in the 55% zinc range. After three cleaning stages, zinc concentrates ranged from 13% to 55% zinc. Preliminary acid base accounting tests on the +1/4” float and arsenopyrite tailings indicate that they are not acid generating. Recoveries of 90% of the gold and 57% of the silver were obtained after pressure leaching and cyanidation of the arsenopyrite concentrate.

ERL bulk sampled the ore at three specific underground locations, all of which indicated different metallurgical properties. The ore zone was drilled and blasted, and then mucked into 45 gallon metal barrels. A fourth bulk sample for gravity test work was taken from a 500 tonne stockpile mined out during PAM’s raise program. The bulk samples were then transported to Lakefield Laboratories in Lakefield, Ontario.

In December 1989, Lakefield carried out six pilot plant tests on Composite 2 and two tests on Composite 3. The tests were commissioned by ERL and funded by Placer. HMS tests on bulk composites showed that a 25% weight reduction could be obtained with less than a 5% loss of values. Milling of the ore showed significant variance in grindability. Composite 1, low arsenic (1.01%), was evaluated in the bench tests only. ERL projected metallurgical results based on a 50/40/10 mix of Composites of 1, 2 and 3.

In March 1991, Lakefield carried out pyrite and arsenopyrite separation tests for CGM along with wet high intensity magnetic separation (WHIMS), magstream separation tests, gravity separation and cyanidation of bulk flotation concentrates. WHIMS and magstream did not provide a satisfactory separation between pyrite and arsenopyrite. Cyanidation of a bulk sulphide concentrate ground to less than 20 µm resulted in less than 10% gold and 60% silver extraction. Gravity recovery from Composites 2 at 100%-200 mesh was approximately 6% for gold and 0.4% for silver. Cyanidation of the gravity tailings resulted in an additional 26% gold and 16% silver extraction.

In May 1991, Bacon Donaldson carried out work commissioned by CGM that focused on separating 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. They were able 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 -1/2” and screened at 4, 8 and 14 mesh. Heavy media separations were carried out on SG 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. Head samples showed mercury levels of 37 ppm to 96 ppm.
In December 1991, Bacon Donaldson carried out variability testing commissioned by CGM to determine if samples with varying pyrite and arsenopyrite content would still be responsive to the separation technique. Only partial lead cleaning tests, no zinc flotation was undertaken. Tests were directed solely at obtaining a low arsenic bearing pyrite concentrate. A better understanding of the parameters was obtained and regrinding with pyrite cleaning appeared beneficial.

In May 1993, BRGM (CGM’s parent company) carried out a review of work to date, completed some of their own testwork and commented on work conducted during the EEC project by Imperial College. They conclude that the preferred process is an adaptation if the Bacon Donaldson flowsheet, which consisted of pyrite flotation from the lead rougher tailing at an elevated temperature, and pH reduction with sulphuric acid instead of SO2. Similar if not better, results were obtained without the acid addition. Tests carried out to find a new reagent for selective flotation of pyrite at Imperial College were promising, but the reagent had an adverse effect on zinc metallurgy and was therefore unsuitable for J&L.

In 1997-1999, Weymin Mining Corporation focused on metallurgical test work that produced numerous effective options for acceptable recoveries of gold, zinc and lead by making 3 separate concentrates and including use of heavy media separation. They extracted six bulk samples were from the 830 Tracked Level. They were blended into an average composite and tested.

The gravity concentration test using a Knelson Concentrator indicated that at a concentration ratio of 760, the gold recovery was 2.5% and the gravity concentrate assayed 227 g/t Au.

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

At a primary grind size of 60% passing 325 mesh, the Pb rougher concentrate typically recovered about 25 % of the Au, 80 % of the Ag, 80 % of the Pb, 30 % of the Zn and 10 % of the As. A locked cycle test indicated that Pb recovery was 80.4 % at a concentrate grade of 50.0 % Pb. Corresponding Au and Ag recoveries in the Pb concentrate were 17.5 % and 78.5%, respectively.

Two other locked cycle tests were conducted. One test showed that Pb recovery was 73.5 % at a concentrate grade of 54.6% Pb and the corresponding Au and Ag recoveries in the Pb concentrate were 14.0 % and 74.4 %, respectively. The other test showed that Pb recovery was 64.9 % at a grade of 57.3 % Pb and the corresponding Au and Ag recoveries were 9.4 % and 59.8 %, respectively.

Variability samples had a higher Pb recovery. In batch tests, Pb recovery was 64.3 % at a grade of 56.2 % Pb, and 69.0 % at a grade of 51.2 % Pb.

The pyrite concentrate removed 10.8 % of the mass and took away 6.3 % of the Au, 6.3 % of the Ag, 3.3 % of the Pb and 3.1 % of the Zn. Direct cyanidation of the pyrite concentrate could only extract 17.3 % of the Au at a sodium cyanide consumption of 8.09 kg/t.

Zn and As were readily floatable after activation with copper sulphate. Although they were floated together into the bulk rougher concentrate, Zn was found to be more floatable than As and was floated first. The final Zn concentrate consisted of 7.0 % of the mass, assayed 51.8 % Zn.
and recovered 72% of the Zn. The Zn concentrate also contained 4.2% of the Au and 5.4% of the Ag.

The As concentrate was produced by separating the Zn/As bulk concentrate. It consisted of 31.2% of the mass, assayed 22.0% As and recovered 90.3% of the As and 69.7% of the Au. The concentrate also contained 8.3% of the Ag, 7.6% of the Pb and 7.2% of the Zn.

The As concentrate was pressure leached to oxidize the arsenopyrite and to release the associated Au. Nearly complete arsenopyrite oxidation was achieved when leached at 8% solids for 3 hours at 190° C with 100 psi O2 over-steam pressure, and with the addition of 5 g/L H2SO4, 1 g/L Fe2+ and 1 g/L Fe3+. The sulphide sulphur content in the As concentrate was reduced from 18.3% to 0.21% and 92.4% of the As and 97.4% of the Fe were fixed in the leach residue. Cyanidation of the residue extracted 96% of the Au at a sodium cyanide consumption of 3.77 kg per tonne of As 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.

From all of the historic test work H.A. Simon’s used recoveries of 80.4% for the Pb, 72.0% for the Zn, 91.8% for the Au and 88.1% for the Ag in their 1998 scoping study.

In 2005 BacTech Mining Corporation continued metallurgical testing on the same six samples extracted by Weymin Mining Corporation.

The samples were tested for bulk density and Bond Work Index. Flotation tests were initially done using the individual samples. The flotation procedure used was the four-product procedure developed in the 1998 metallurgical tests. The four flotation concentrates were a gold-bearing arsenopyrite concentrate, a lead concentrate, a zinc concentrate and a pyrite concentrate.

The flotation tests produced results generally in keeping with those obtained in the previous study. Overall recoveries for lead, zinc, gold and silver were very high, although the selective separation of lead and zinc for the final lead concentrate remained problematic.

In order to obtain more consistent results, a blended composite sample was used for the next phase of the flotation test work. This minimized the variability of the results from different individual samples. Also, it was felt by management that the practical operability of the four-product flotation procedure was deemed to be difficult since this involved the heating up of the pulp to 50 to 65° C and using sulphur dioxide gas as a reagent for the pyrite flotation stage. A simplified three-product flotation procedure, generating a gold-bearing arsenopyrite/pyrite concentrate, a lead concentrate and a zinc concentrate was proposed. This three-product procedure produced results equivalent to the four-product procedure.

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 Zn 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/cm³, 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 would 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.

Issues yet to be resolved with this approach include previous marketing studies showed penalties in smelter contracts will affect economics or made concentrates unsaleable:

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

**15.2.2 Chemical Approaches and Refractory Gold Process Testing**

Testing was completed on J&L products using refractory gold technology to free locked precious metals. Extensive laboratory studies were completed on the Cashman (chloride) and Redox (nitrate) process technologies.

The Cashman process is a chloride-based total chemical oxidation process. Its objective would to make the lead and zinc contents of a bulk flotation concentrate soluble as chlorides. Most of the silver and part of the gold would also report to the filtrate. The remaining gold and silver would be recovered from the leach residue by cyanidation.

The Cashman work on J&L Main Zone was supervised by Lyall Lichty in the Mountain States laboratory, in 1985. A bulk concentrate was successfully produced. Processing of this concentrate dissolved over 95% of the gold and over 90% of the silver. Metal separation and recovery was not demonstrated.

The Redox Process is another chemical approach which uses a nitrate/nitrite system to oxidize the sulphide minerals and dissolve the metallic species. Testing was completed at Bacon Donaldson Inc., the inventors of the process. The testing focused mainly on gold recovery, with less attention paid to base metals. The gold remains in the residue, for recovery by conventional cyanidation.

R.C. Smith in February 1988 provided a good overview of the metallurgical test work that previously had been conducted. Roasting achieved, at best, a gold recovery of 75%. Silver recovery was better than the other oxidation process at 74% but the economics are substantially
controlled by gold rather than silver. Pressure oxidation gave gold extractions at 170 °C and a 3 hour residence time up to 93%, with NaCN consumptions at 0.72 kg/t and CaO consumptions of 1 kg/t (Lakefield Tests). The low cyanide and lime consumptions conflict with several other test results; the residue pretreatment may have been responsible. A true reagent consumption would best be obtained on a residue that is neutralized and cyanided without washing or drying beyond what is done commercially. Assuming that washing is conducted (e.g., CCD thickening), a reagent consumption must be included to neutralize the washings.

For bioleaching, Smith focused on the concentrate results obtained by International Bioleaching, who treated both a zinc flotation tails products and an arsenopyrite concentrate: 87% Au extracted with an NaCN consumption of 2.8 kg/t and a CaO consumption of 3.4 kg/t. For the tails, Au extraction was 95.9% with complete oxidation assumed after 11 days. Reagent consumptions were 16.3 kg/t NaCN and 7 kg/t CaO. Thiourea leaching for gold was examined as a way to reduce neutralization costs, but available information suggested it might be too expensive.

In a batch bioleach amenability test, Coastech determined an extraction of 89% Au in 25 days, with NaCN consumption of 13 kg/t and CaO consumption of 53 kg/t.

The completed tests demonstrated amenability to pressure oxidation and biooxidation.
16.0 MINERAL RESOURCE ESTIMATES

16.1 INTRODUCTION

The mineral resource estimate presented herein is reported in accordance with the Canadian Securities Administrators’ National Instrument 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 FH Brown, MSc (Eng) CPG Pr.Sci.Nat., an independent Qualified Persons in terms of NI43-101, from information and data supplied by Huakan International Mining Inc. The effective date of this estimate is May 16, 2011. A draft copy of this report was reviewed by Huakan for factual errors.

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

16.2 PREVIOUS RESOURCE ESTIMATES

An historic mineral resource estimate was prepared by Equinox Resources Ltd. in 1991 for the Main Zone, listing “Proven and Probable Ore Reserves” of 1.7 million tonnes grading 7.38 g/t gold, 75.9 g/t silver, 2.64 % lead and 4.43 % zinc, and “Possible Ore Reserves” of 1.9 million tonnes grading 7.12 g/t gold, 85.5 g/t silver, 3.32 % lead and 3.48 % zinc.¹

An historic mineral resource estimate was completed by Equinox Resources Ltd. in 1991 for the Yellow Jacket Zone, listing “Probable Ore Reserves” of 693,000 tonnes grading 52.3 g/t silver, 2.45 % lead and 7.06 % zinc, and “Possible Ore Reserves” of 337,000 tonnes grading 53.1 g/t silver, 2.50 % lead and 7.15 % zinc.²

P&E has not independently verified these historic mineral resource estimates, and makes no assurances as to their validity or economic viability, in whole or in part. These historical estimates have not been shown to be in accordance with the mineral resources or mineral reserves classifications contained in the CIM Definition Standards on Mineral Resources and Mineral Reserves, as required by National Instrument 43-101. Accordingly, P&E is not treating these historical estimates as current mineral resources or mineral reserves as defined in NI43-101 and such historical estimates should not be relied upon. It should be further noted that these historic resource estimates have been superseded by the NI 43-101 compliant mineral resource estimate that is the subject of this report. 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.

² Ibid.
16.3 DATA SUPPLIED

All sampling data were compiled by Huakan, who supplied a Microsoft AccessTM 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 (Table 16.1). 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>
<thead>
<tr>
<th>Type</th>
<th>Record Count</th>
<th>Total Meters</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling</td>
<td>253</td>
<td>31,410.41</td>
</tr>
<tr>
<td>Underground Chip Sampling</td>
<td>223</td>
<td>529.15</td>
</tr>
<tr>
<td>Surface Trench Sampling</td>
<td>32</td>
<td>85.57</td>
</tr>
<tr>
<td>Not used</td>
<td>29</td>
<td>140.41</td>
</tr>
</tbody>
</table>

16.4 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.

16.5 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 16.2).

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

Based on the economic parameters listed in Table 16.3 a 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:

\[ \text{NSR} = (\text{Pb} \% \times \$17.46) + (\text{Zn} \% \times \$13.49) + (\text{Ag} \, \text{g/t} \times \$0.56) + (\text{Au} \, \text{g/t} \times \$34.91) - \$18.25 \]

TABLE 16.3
J&L ECONOMIC PARAMETERS

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>$0.99</td>
<td>80%</td>
<td>95%</td>
<td>$0.00</td>
</tr>
<tr>
<td>Zn</td>
<td>$0.95</td>
<td>72%</td>
<td>85%</td>
<td>$0.00</td>
</tr>
<tr>
<td>Ag</td>
<td>$21.01</td>
<td>88%</td>
<td>91%</td>
<td>$0.50</td>
</tr>
<tr>
<td>Au</td>
<td>$1.183</td>
<td>92%</td>
<td>96%</td>
<td>$15.00</td>
</tr>
</tbody>
</table>

| $/CUS   | $0.950      |

<table>
<thead>
<tr>
<th>Concentration Ratio</th>
<th>20</th>
<th>Pb/Zn Blended</th>
</tr>
</thead>
<tbody>
<tr>
<td>Smelter Treatment Charge $US/dmt</td>
<td>$185</td>
<td>Pb/Zn Blended</td>
</tr>
<tr>
<td>Concentrate Shipping Charge $C/tonne</td>
<td>$65</td>
<td></td>
</tr>
<tr>
<td>Moisture Content</td>
<td>8%</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Element</th>
<th>Payable Metal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>$17.46</td>
</tr>
<tr>
<td>Zn</td>
<td>$13.49</td>
</tr>
<tr>
<td>Ag</td>
<td>$0.56</td>
</tr>
<tr>
<td>Au</td>
<td>$34.91</td>
</tr>
<tr>
<td>TOTAL</td>
<td>$66.42</td>
</tr>
</tbody>
</table>

| 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 |

TABLE 16.2
SPECIFIC GRAVITY VALUES

<table>
<thead>
<tr>
<th></th>
<th>Marginal</th>
<th>Main Zone</th>
<th>P&amp;E</th>
</tr>
</thead>
<tbody>
<tr>
<td>Count</td>
<td>396</td>
<td>256</td>
<td>18</td>
</tr>
<tr>
<td>Minimum</td>
<td>2.61</td>
<td>2.66</td>
<td>2.76</td>
</tr>
<tr>
<td>Maximum</td>
<td>5.03</td>
<td>5</td>
<td>4.18</td>
</tr>
<tr>
<td>Average</td>
<td>3.31</td>
<td>3.52</td>
<td>3.36</td>
</tr>
</tbody>
</table>
16.7 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 meters 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/tonne 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 (Figure 16.1).

Figure 16.1 Isometric projection of mineral resource domains.

16.8 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 grade of 0.001 was used to populate a small number of un-sampled intervals. Composites that were less than 0.25m 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.

### 16.9 EXPLORATORY DATA ANALYSIS

P&E generated summary statistics for the Main Zone composite data (Table 16.4) and the Footwall Zone composite data (Table 16.5). A total of 2,474 composites were generated for the Main Zone, and 79 for the Footwall Zone.

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 (Table 16.6).

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 16.2).

| TABLE 16.4 |
|---|---|---|---|---|
| **MAIN ZONE COMPOSITE SUMMARY STATISTICS** | **Length (m)** | **Au (g/t)** | **Ag (g/t)** | **Pb %** |
| Mean | 0.49 | 5.05 | 51.64 | 1.81 |
| Length-Weighted Mean | 4.57 | 46.75 | 1.64 | 3.01 |
| CV | 0.08 | 1.48 | 1.34 | 1.46 |
| Median | 0.50 | 2.50 | 22.96 | 0.64 |
| Mode | 0.50 | 0.10 | 0.30 | 0.01 |
| Standard Deviation | 0.04 | 7.48 | 69.08 | 2.64 |
| Sample Variance | 0.00 | 55.94 | 4772.59 | 6.98 |
| Kurtosis | 17.88 | 88.94 | 7.54 | 6.54 |
| Skewness | -4.27 | 6.15 | 2.37 | 2.37 |
| Range | 0.25 | 157.19 | 600.98 | 18.56 |
| Minimum | 0.25 | 0.00 | 0.00 | 0.00 |
| Maximum | 0.50 | 157.19 | 600.98 | 18.56 |
| Count | 2,474 | 2,474 | 2,474 | 2,474 |
### Table 16.5
**Footwall Zone Composite Summary Statistics**

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

### Table 16.6
**Main Zone Composite Correlation Matrix**

<table>
<thead>
<tr>
<th></th>
<th>Ag</th>
<th>Au</th>
<th>Pb</th>
<th>Zn</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ag</td>
<td>1.00</td>
<td>0.43</td>
<td>0.85</td>
<td>0.60</td>
</tr>
<tr>
<td>Au</td>
<td>0.43</td>
<td>1.00</td>
<td>0.36</td>
<td>0.31</td>
</tr>
<tr>
<td>Pb</td>
<td>0.85</td>
<td>0.36</td>
<td>1.00</td>
<td>0.69</td>
</tr>
<tr>
<td>Zn</td>
<td>0.60</td>
<td>0.31</td>
<td>0.69</td>
<td>1.00</td>
</tr>
</tbody>
</table>

**Figure 16.2** Main Zone QQ plot for drillhole vs. chip composites

**16.10 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 16.3). Minimal capping levels were therefore selected. Composite grades were capped to the selected threshold values prior to estimation (Table 16.7).

Figure 16.3  Decile analysis results

| Table 16.7  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%              |

16.11 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.
16.12 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 16.8). 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>
<thead>
<tr>
<th>Dimension</th>
<th>Minimum</th>
<th>Maximum</th>
<th>Number</th>
<th>Size (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>X</td>
<td>421,800.00</td>
<td>423,300.00</td>
<td>150</td>
<td>10</td>
</tr>
<tr>
<td>Y</td>
<td>5,680,100.00</td>
<td>5,683,100.00</td>
<td>300</td>
<td>10</td>
</tr>
<tr>
<td>Z</td>
<td>2,100.00</td>
<td>4,000.00</td>
<td>190</td>
<td>10</td>
</tr>
<tr>
<td>Rotation</td>
<td></td>
<td>-45.00</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

16.13 ESTIMATION & CLASSIFICATION

Block density values were calculated using a single pass. Anisotropic inverse distance squared ("ID2") 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 considers that there is sufficient drilling and sampling information, and that this information is of a sufficient quality, to support a Measured, Indicated and Inferred classification for the J&L 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 16.4).
Figure 16.4 Isometric projection of Main Zone block classification

Scale Bar = 500m.
View looking north-east.
Red = Measured.
Green = Indicated.
Blue = Inferred.

16.14 MINERAL RESOURCE ESTIMATE

The mineral resource estimate for the J&L deposit is reported at a NSR cut-off grade of CDN$110.00/tonne in Table 16.9, with an effective date of May 16, 2011.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Tonnes</th>
<th>Au (g/t)</th>
<th>Au (ozs)</th>
<th>Ag (g/t)</th>
<th>Ag (ozs)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured</td>
<td>1,202,000</td>
<td>6.71</td>
<td>259,200</td>
<td>69</td>
<td>2,664,600</td>
<td>2.4</td>
<td>4.46</td>
</tr>
<tr>
<td>Indicated</td>
<td>1,165,700</td>
<td>6.92</td>
<td>259,200</td>
<td>64.9</td>
<td>2,432,100</td>
<td>2.01</td>
<td>3.86</td>
</tr>
<tr>
<td>Measured &amp; Indicated</td>
<td>2,367,700</td>
<td>6.81</td>
<td>518,400</td>
<td>66.95</td>
<td>5,096,700</td>
<td>2.21</td>
<td>4.16</td>
</tr>
<tr>
<td>Inferred</td>
<td>4,538,100</td>
<td>5.19</td>
<td>757,500</td>
<td>67.8</td>
<td>9,887,800</td>
<td>2.16</td>
<td>2.99</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Classification</th>
<th>Tonnes</th>
<th>Au (g/t)</th>
<th>Au (ozs)</th>
<th>Ag (g/t)</th>
<th>Ag (ozs)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred</td>
<td>292,800</td>
<td>4.54</td>
<td>42,700</td>
<td>49</td>
<td>461,900</td>
<td>0.91</td>
<td>0.73</td>
</tr>
</tbody>
</table>

(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 $1,183/oz, Ag $21/oz, Pb $0.99/lb, Zn $0.95/lb
- Exchange rate of US$0.95US = $1.00CDN
- 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 CDN$110 per tonne was derived from $75/t mining, $25/t processing and $10/t G&A.

16.15 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. No significant bias between the block model and the input data was noted.

<table>
<thead>
<tr>
<th>Table 16.10</th>
<th>MAIN ZONE VALIDATION STATISTICS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Element</td>
<td>Model Mean</td>
</tr>
<tr>
<td>Ag g/t</td>
<td>56.61</td>
</tr>
<tr>
<td>Au g/t</td>
<td>4.41</td>
</tr>
<tr>
<td>Pb %</td>
<td>1.78</td>
</tr>
<tr>
<td>Zn %</td>
<td>2.52</td>
</tr>
</tbody>
</table>

In addition, local trends were evaluated by comparing the ID2 block estimates to a nearest neighbour estimate (NN) at zero cut-off along the strike of the Main Zone (Figure 16.5). In general the ID2 block estimates are in good agreement with the NN estimates, and demonstrate no evidence of systematic bias in the model.
Figure 16.5  Main Zone swath plots

![Graphs of Ag, Pb, Au, and Zn concentrations with comparison to NN concentrations.](image-url)
17.0 OTHER RELEVANT DATA AND INFORMATION

P&E is unaware of any other relevant material data and information that would result in misleading statements.
18.0 CONCLUSIONS AND RECOMMENDATIONS

18.1 CONCLUSIONS

The J&L property, in which Huakan holds a 100% unencumbered interest, is located 35 kilometres north of Revelstoke, British Columbia, Canada.

The property has two known precious and polymetallic mineral deposits. The Main Zone is a shear hosted replacement deposit overprinting a pre-existing silver-lead-zinc deposit (the Yellowjacket). The sheeted sulphide vein Main Zone system is composed of banded massive and stringer arsenopyrite-pyrite-sphalerite-galena mineralization with appreciable content of gold and silver. The Main Zone has been traced on surface for a strike length of over three kilometres and traced by drilling for 1.5 kilometres along strike by 0.8 kilometres downdip. The Main Zone generally dips about 60 degrees to the northeast with an average true thickness of 2.5 metres but can reach 15 metres true thickness. It has been the focus of recent drilling by Huakan and the resulting NI 43-101 resource estimate contained in this report.

The Yellowjacket deposit is a very siliceous sphalerite-galena (Zn-Pb-Ag) stratabound carbonate replacement deposit that sub parallels and is in the immediate hanging wall of the Main Zone. The Yellowjacket per se is believed to be the remains of its former self, the majority of which was cut and remobilized by a major shear zone, which ultimately became occupied by the Main Zone sulphide vein system. The Yellowjacket is currently not the focus of recent activity and remains with a historic resource estimate dating back to 1991. The Yellowjacket occurs in a series of lenticular bodies each up to eight metres thick.

The J&L property has been explored by a number of mining companies by trenching, drifting and drilling. There are a total of 266 drill holes that have been completed on the property from 1983 to present, which translates to 31,186 metres of drilling.

The 830 drift and related cross-cuts total 1.9 kilometres exposing the Main Zone for approximately 0.8 kilometres. The 550 metre long 832 trackless drift provides year round underground access to the 830 drift.

Bulk samples have been taken to conduct metallurgical testwork on the Main Zone mineralization, which is a complex polymetallic deposit high in arsenic. The arsenic content creates a challenge in the production of saleable zinc and lead concentrates and the economic recovery of gold. Extensive metallurgical testing between the mid 1980’s and 2006 considered various options and produced numerous effective options for acceptable recoveries of gold, silver, zinc and lead by making three separate concentrates, including using heavy media separation. Limited metallurgical testwork has been performed on the Yellowjacket Zone which appears to have a simpler metallurgy than the Main Zone.

In late 2010, the J&L property underwent renewed exploration activity by Huakan with the completion of a 60 hole 7,897 metre underground drill program focused on the Main Zone with the objective of verifying historic drilling and sampling and infilling a 800 metre strike by 200 metre dip face of the Main Zone with 30 metre centers to support a NI 43-101 compliant resource.
The J&L Main Zone is at an advanced stage of exploration, having been the subject of exploration programs including surface and underground drilling programs carried out under the supervision of Qualified Persons.

P&E is satisfied that the drill sample database and geological interpretations are sufficient to enable the estimation of Mineral Resources. Accepted estimation methods have been used in the generation of a 3D block model of Au, Ag, Pb and Zn grades and assigned densities.

The estimates have been classified with respect to CIM Standards as Measured, Indicated and Inferred, according to the geological confidence and sample spacings that currently define the deposit.

Should Huakan want to upgrade any of the Inferred Resources, infill drilling will be required.

18.2 RECOMMENDATIONS

P&E makes the following recommendations for continuing work at J&L. The work should be completed in two phases, which are not contingent upon one another:

Phase I at an approximate cost of CDN $2.4M, is designed to expand Main Zone Resources 300 to 400 m laterally and up and down dip, (see Table 18.1 and Figure 18.1).

- Extend 830 level track drift approximately 300 to 400 m in order to provide diamond drill bays to test the Main Zone an additional 300 to 400 m along strike and 200 m down dip;
- Drive a 150 m long footwall cross-cut in order to provide diamond drill bays to test the up dip extent of the Main Zone where thickness and grade are favourable to potentially rapidly increase resources;
- 2,500 m underground drilling from these two drifts

**TABLE 18.1**

**PHASE I PROPOSED BUDGET**

<table>
<thead>
<tr>
<th>Start-End Point</th>
<th>Plan Distance</th>
<th>Track Drill Length</th>
<th>Turnaround Increase (15%)</th>
<th>Gradient</th>
<th>Number of Drill Holes</th>
<th>Length of Drill holes</th>
<th>COST</th>
</tr>
</thead>
<tbody>
<tr>
<td>830 Level Drift</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>D1-D2</td>
<td>15.00</td>
<td>15.00</td>
<td>17.25</td>
<td>0.3</td>
<td></td>
<td></td>
<td>65,500</td>
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<tr>
<td>C1-C2</td>
<td>9.73</td>
<td>9.73</td>
<td>11.19</td>
<td>0.3</td>
<td></td>
<td></td>
<td>42,500</td>
</tr>
<tr>
<td>C2-C3</td>
<td>143.05</td>
<td>143.05</td>
<td>164.51</td>
<td>0.3</td>
<td></td>
<td></td>
<td>625,100</td>
</tr>
<tr>
<td>E1-E2</td>
<td>200.00</td>
<td>200.00</td>
<td>230.00</td>
<td>0.3</td>
<td></td>
<td></td>
<td>874,000</td>
</tr>
<tr>
<td><strong>Sub Total</strong></td>
<td><strong>367.78</strong></td>
<td><strong>367.78</strong></td>
<td><strong>422.95</strong></td>
<td></td>
<td></td>
<td></td>
<td><strong>1,407,100</strong></td>
</tr>
<tr>
<td>830 Drill Hole</td>
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<td></td>
<td></td>
<td></td>
<td></td>
<td>498,200</td>
</tr>
<tr>
<td><strong>Contingency (15%)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>315,800</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>367.78</strong></td>
<td><strong>367.78</strong></td>
<td><strong>422.95</strong></td>
<td></td>
<td></td>
<td></td>
<td><strong>2,421,100</strong></td>
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Figure 18.1  Phase I Proposed Layout for Drifts and Diamond Drill Bays

<table>
<thead>
<tr>
<th>Drilling Phase</th>
<th>Start-End Point</th>
<th>Plan Distance</th>
<th>Track Drift Length</th>
<th>Turnaround Increase(%)</th>
<th>Gradient</th>
<th>Number of Drill Holes</th>
<th>Length of Drill Holes</th>
<th>COST</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>D1-D2</td>
<td>15.00</td>
<td>15.00</td>
<td>17.25</td>
<td>0.3</td>
<td></td>
<td></td>
<td>65,500</td>
</tr>
<tr>
<td></td>
<td>C1-C2</td>
<td>9.75</td>
<td>9.75</td>
<td>11.19</td>
<td>0.3</td>
<td></td>
<td></td>
<td>42,500</td>
</tr>
<tr>
<td></td>
<td>C2-C3</td>
<td>143.05</td>
<td>143.00</td>
<td>164.51</td>
<td>0.3</td>
<td></td>
<td>1600</td>
<td>625,100</td>
</tr>
<tr>
<td></td>
<td>E1-E2</td>
<td>200.00</td>
<td>200.00</td>
<td>230.00</td>
<td>0.3</td>
<td></td>
<td>2300</td>
<td>874,000</td>
</tr>
<tr>
<td>Sub Total</td>
<td></td>
<td>397.76</td>
<td>367.76</td>
<td>422.95</td>
<td></td>
<td>25</td>
<td>2.49%</td>
<td>1,057,100</td>
</tr>
<tr>
<td>832 Drift Hole</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>25</td>
<td>2.49%</td>
<td>466,200</td>
</tr>
<tr>
<td>Contingency</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>315,000</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>397.76</td>
<td>367.76</td>
<td>422.95</td>
<td></td>
<td>25</td>
<td>2.49%</td>
<td>2,431,500</td>
</tr>
</tbody>
</table>
Phase II at an approximate cost of CDN $15M, is designed to increase Measured and Indicated Resources, (see Table 18.2 and Figure 18.2).

- Extend 832 decline by 1.7 km to reach the 700 m level;
- Drive a hanging wall crosscut and drift totalling approximately 2,300 metres in order to provide diamond drill bays to test a 1,000 m lateral by 200 m vertical area of the Main Zone;
- 6,200 m of diamond drilling at 60 m centres.

If silver, lead and zinc prices continue to rise and remain high, the Yellowjacket deposit could show potential economic merits and may be considered for further exploration and definition.

### Table 18.2

**Phase II Proposed Budget**

<table>
<thead>
<tr>
<th>Phase II - 700 Level Ramp, Drift &amp; Drill</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drifting Price</td>
</tr>
<tr>
<td>Drill Hole Cost</td>
</tr>
<tr>
<td>Start-End Point</td>
</tr>
<tr>
<td>Ramp (809m Ele to 700m Ele)</td>
</tr>
<tr>
<td>A1-A2</td>
</tr>
<tr>
<td>A2-A3</td>
</tr>
<tr>
<td>A3-A4</td>
</tr>
<tr>
<td>A4-A5</td>
</tr>
<tr>
<td>700L Drift</td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td>Sub Total</td>
</tr>
<tr>
<td>700L Drift Hole</td>
</tr>
<tr>
<td>Camp &amp; Infrastructure</td>
</tr>
<tr>
<td>Contingency (15%)</td>
</tr>
<tr>
<td>Total</td>
</tr>
</tbody>
</table>
Figure 18.2  Phase II Proposed Layout for Drifts and Diamond Drill Bays
19.0 REFERENCES

BCMEMPRT, Annual Reports 1905 (148-150), 1912 (144), 1915 (117), 1916 (193), 1922 (215), 1923 (232), 1924 (204), 1925 (258), 1926 (269), 1927 (290), 1946 (174), 1965 (204)


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Pegg, R., (1982-1985). Assessment Reports related to various physical, geological, geophysical and geochemical surveys carried out on J&L property for BP-Selco Inc. (Assessment Reports: 10664, 10939, 12616, 12634 and 14405)


20.0 CERTIFICATES

CERTIFICATE of AUTHOR

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.


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 34720).

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 did not visit the J&L Property.

5. I am responsible for Sections 9 through 15, 17 and jointly responsible for Sections 18, 19.

6. I am independent of the Issuer applying the test in Section 1.4 of NI 43-101.

7. I have not had prior involvement with the Property that is the subject of this 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: May 16, 2011

Signing Date: June 23, 2011

[SIGNED AND SEALED]

(Tracy J. Armstrong)

________________________________
Tracy J. Armstrong, P.Geo.
CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, CPG, PrSciNat

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.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand and 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.

This report is based on my personal review of information provided by Shore Gold Inc. and on discussions with its representatives. My relevant experience for the purpose of the Technical Report is:

- Chief Geologist, De Beers Consolidated Mines ................................................................. 2000-2004
- Consulting Geologist ........................................................................................................... 2004-2008

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 16.0 and co-authoring sections 18.0 and 19.0 of this Technical Report.
6. I am independent of the issuer applying the test in Section 1.4 of NI 43-101.
7. I have not had any prior involvement with the Project that is the subject of this 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: May 16, 2011
Signed Date: June 23, 2011

{SIGNED AND SEALED}
[Fred H. Brown]

Fred H. Brown CPG, PrSciNat
CERTIFICATE OF QUALIFIED PERSON

WAYNE D. EWERT, P.GEO.

I, Wayne D. Ewert, P. Geo., residing at 10 Langford Court, Brampton, Ontario, L6W 4K4, do hereby certify that:

1. I am a principal of P & E Mining Consultants Inc. who has been contracted by Huakan International Mining Inc.


3. I graduated with an Honours Bachelor of Science degree in Geology from the University of Waterloo in 1970 and with a PhD degree in Geology from Carleton University in 1977. I have worked as a geologist for a total of 39 years since obtaining my B.Sc. degree. I am a P. Geo., registered in the Province of Saskatchewan (APEGS No. 16217), British Columbia (APEGBC No. 18965), and the Province of Ontario (APGO No. 0866).

   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:
   - Principal, P&E Mining Consultants Inc. ................................................................. 2004 – Present
   - Regional Manager, Gold Fields Canadian Mining Limited.................................... 1986 – 1987
   - Supervising Project Geologist, Getty Mines Ltd. .................................................. 1982 – 1986
   - Supervising Project Geologist III, Cominco Ltd. .................................................. 1976 – 1982

4. I have not visited the J&L Property.

5. I am responsible for authoring Sections 1.0 through 10.0, 14.0, and 20.0 in their entirety and for co-authoring sections 18.0 and 19.0 of this Technical Report.


7. I have not had prior involvement with the project that is the subject of this 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: May 16, 2011
Signed Date: June 23, 2011

[SIGNED AND SEALED]
[Wayne Ewert]

Dr. Wayne D. Ewert P. Geo.