Amended Preliminary Economic Assessment Technical Report

on the

Willa Property

Slocan Mining Division
British Columbia
Canada

BCGS Maps 082F083, 084, 094
NTS Map 082F014W

Latitude 49°53' N, Longitude 117°22' W

For
Discovery Ventures Inc.
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By

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Amended January 19, 2016, from May 26, 2014 original report

Effective date: November 30, 2015

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1.0 Summary

1.1 Introduction

This Amended Preliminary Economic Assessment Technical Report on the Willa Property was prepared for Discovery Ventures Inc at the request of Mr. Akash Patel, president of Discovery Ventures. The purpose of the Report is to disclose an updated Mineral Resource estimate and to report the results of a Preliminary Economic Assessment based on a preliminary Mine Plan. This Report is amended from the Preliminary Economic Assessment & Technical Report, Willa MAX Project, for Discovery Ventures Inc, dated May 26, 2014, and authored by W. M. Ash, PEng.

The Report is based on a review of previous reports filed by a number of operators who have conducted mineral exploration programs within and around the current boundaries of the Property. Some of the reports have been filed for assessment credit with the British Columbia Ministry of Energy and Mines ("BCMEM"). Significant data were also obtained from private reports, logs and maps from previous operators.

1.2 Reliance on Other Experts

Details of the status of mineral title ownership on the Property were obtained from the BC Mineral Tenures Online ("MTO") database system managed by the BCMEM.

Copies of the various Option Agreements and Amendment Agreements were provided to the co-author Gilmour. Although the author has no reason to believe this information is inaccurate, a detailed audit of the option agreements among the Vendors and various parties has not been completed and the author Gilmour is relying solely on the information that has been provided by the various parties.

FortyTwo Metals, which is 100% owned by Discovery Ventures, has a non-capital loss and an undepreciated capital cost (UCC), totalling about $C 47 million. These amounts have been confirmed by the accounting firm of MNP, in Vancouver. MNP states that upon amalgamation of FortyTwo and Discovery Ventures, it does not foresee any limitation on the use of the tax pool by the amalgamated entity. Discovery Ventures has confirmed that approval of the amalgamation has been granted by the TSX Venture Exchange, with just internal reorganization of the two companies to complete.

1.3 Property Description and Location

The 24 mineral titles comprising the Property total 5,620 hectares in area. The titles are in good standing to at least, February 18, 2016, with the title on which the Mineral Resource is located being good until January 19, 2020. The titles are registered on MTO in the name Discovery Ventures Inc. The title has been acquired from Billingsley, Brickner, Kress and Richards, the Vendors of the Property. Discovery Ventures has purchased the Property from the Vendors, subject to monthly advanced royalty payments.

Discovery Ventures has acquired FortyTwo Metals Inc and thereby acquired assets that include an underground molybdenum mine, crushing, milling and concentrating facilities (the "MAX mill"), tailings storage facilities, mineral titles and mineral leases, including Mineral Lease 577449 on which the MAX mill is located. In this Report is it proposed that mine product from the Willa be processed at the MAX Mill.

The Willa Property, containing the Willa gold-copper-silver deposit, is located in the steeply incised Aylwin Creek Valley of southeastern British Columbia, south of Silverton and east of Slocan Lake. The Property is mainly located in the northwest portion of BC Geographic System map 082F084.

The Property is located approximately 7.5 km south of Silverton and 12 km south of New Denver, along Highway 6. The nearest airport is at Castlegar, 86 km to the south via Highways 6 and 3A. Nelson is 88 km to the south via Highway 6. Vernon, in the Okanagan Valley, is 254 km west via Highway 6 and the Needles ferry. Revelstoke, on Highway 1, is 162 km north via Highways 6 and 23 and the Galena Bay ferry. The MAX mill is 142 km north of the Property via road.

Discovery Ventures has a Mines Act Permit for mineral exploration on the Property, valid to December 31, 2017. Approval is granted for the rehabilitation of the 1025 and 1100 levels and portals, sampling and geological mapping, and removal of a 2 t sample from underground for metallurgical testing. Discovery Ventures also has an exploration permit, valid until August 2020, which includes an underground 10,000 t bulk sample, core drilling and rock sampling. Discovery Ventures does not have a Mine Permit on the Property.

1.4 Accessibility, Physiography, Climate, Local Resources and Infrastructure

The Property is located approximately 7.5 km south of Silverton and 12 km south of New Denver, along Highway 6. The nearest airport is at Castlegar, 86 km to the south via Highways 6 and 3A. Access to Vernon, in the Okanagan Valley, is 254 km west via Highway 6 and the Needles ferry. The Property is in forested, rugged terrain with the main portal at 1025 metres above sea level. An access road to the Property is located approximately 8 km south of Silverton near the south junction of Highway 6 and the Red Mountain Road. Both these roads are maintained year round. From Red Mountain Road a 4-wheel-drive road for 1.2 km accesses the 1025 Level portal area.

The Property is in forested, rugged terrain with elevations ranging from 550 m at Slocan Lake to 2,415 m along the ridge separating Aylwin and Congo Creeks. Slopes are frequently greater than 35°.

The climate is typical of the Central Kootenay Region, with cool winters and warm summers. Although annual snow fall at the portal approximates 3 m, underground exploration, development and mining could be carried out year round.

The closest single-phase power line runs along Red Mountain Road. Three-phase power would likely require a new 7.5-km line from Silverton along Highway 6.

1.5 History

The Property has been explored on and off since 1893. However, the majority of the work to develop the Property has occurred only more recently. In the mid-1960s Cominco, AMAX Exploration and Western Mining Company identified copper and molybdenum mineralization. During the early 1980s the Willa deposit was explored on surface by a joint venture between Rio Algom and BP. In April 1985, Northair Mines joined the joint venture and completed the present underground workings. In 2004, Bethlehem Resources (1996) Corporation completed a 39-hole underground core drill program.

As well in the 1980s, 2,455 metres of underground working were completed. In 1988, a 494 tonne underground bulk sample was collect and treated. The reported grade from belt sampling during processing was 5.55 grams/tonne gold and 0.69% copper.

1.6 Geological Setting and Mineralization

The Lower Jurassic Elise Formation of the Rossland Group, the oldest rocks on the Property, comprises about 75% of a roof pendant within the Nelson Batholith. The rocks range from volcanic siltstone and tuff to coarse breccias and flows. The metavolcanic roof pendant is cut by felsic porphyritic intrusions. A heterolithic breccia intrudes the volcanic and intrusive rocks. Lamprophyre dykes and faults crosscut and sometimes displace the older lithologies.

The significant mineralization at the Willa deposit is gold-copper-silver, predominantly within the heterolithic breccia and feldspar porphyry intrusions, and to a lesser extent, the Elsie Formation. The mineralization is controlled by extensive silica-rich micro-fractures, faults, shears and breccias, predominantly within the heterolithic breccia unit that formed a zone of weakness for the emplacement of the mineralization.

The predominant minerals are chalcopyrite, with varying amounts of pyrite and pyrrhotite. The sulphides tend to be encapsulated within sheets of silica. Propylitic alteration occurred during mineralization, as did zones of intense silicification and minor pyritization.

1.7 Deposit Types

The Willa deposit is a sub-volcanic breccia-hosted-type deposit. This type of deposit represents a transition from porphyry copper to epithermal conditions with a blending of porphyry and epithermal characteristics. The porphyry units have intruded into the volcanic rocks and a pyrite, silica-rich mineralized stockwork breccia and closely-spaced sheeted veins with local massive to disseminated replacement zones have been imprinted on the sequence. The mineralization in these types of deposits is usually polymetallic and in the case of the Willa deposit contains significant gold, copper and silver.

1.8 Exploration

Since acquiring the Property in 2012, Discovery Ventures has carried out metallurgical studies, minor surface exploration and minor underground rehabilitation, but no drilling.

1.9 Drilling

Drilling at Willa comprises 593 holes totalling 56,312 m completed between 1965 and 2004. Note that all drill results are from previous owners and Discovery Ventures has not conducted any of its own drilling to date. The vast majority of the drilling was collared from the underground workings and was drilled on behalf of Northair between 1986 and 1988. The location of the core from prior to 2003 is unknown, although it has been reported that, due to improper core storage, there is no way to identify the core.

The majority of the drilling was conducted within a block measuring approximately 250 m by 50 m, and over a vertical range of 200 m. Almost 20,000 samples were collected in the various drilling programs. Areas of significant mineralization were originally drilled at approximately 25-m line spacing and infill-drilling across the majority of the mineralized zones reduced the spacing to 12.5 m

1.10 Sample Preparation, Analyses and Security

Quality Control / Quality Assurance programs, as defined by CIM Exploration Best Practices, were not then implemented by the exploration operators during the 1965 to 1988 drill programs, although the exploration was carried out by major and intermediate companies.

The 2004 drill program was carried out under CIM Exploration Best Practices.

1.11 Data Verification

As part of preparing the Mineral Resource disclosed in this report, some of the drillhole results in the database were selected and gold, copper and silver assay results were cross checked against primary sources by Mr. Waldegger. Holes completed prior to 2004 did not have original assay certificates from the lab to verify therefore database results were cross checked against results transcribed on to the paper geological logs. No significant discrepancies were noted.

During a site visit in November 2015, Mr. Waldegger and Mr. Gilmour visited the main portal for the 1025 level.

Mr. Waldegger is of the opinion that the data in the drillhole database fairly reflects the data recorded in the paper records and can be relied upon and used for Mineral Resource Estimation.

1.12 Mineral Processing and Metallurgical Testing

Various metallurgical studies have been carried out, including included gravity separation using high-gravity concentration, cyanidation, copper flotation, pyrite flotation, thickening and copper concentrate thickening. The results of these studies have been incorporated in the economic analysis.

It is recommended that continued metallurgical test work be conducted to include, but not be restricted to:

- Optimization of reagent dosages for the Willa mineral
- Confirmatory test work for cycloning requirements at the Max mill
- Final design of milling flow sheet

A 10,000 tonne bulk sample should be taken for confirmatory pilot plant testwork.

1.13 Mineral Resource Estimates

MFW Geoscience Inc estimated that the Property includes 198,000 tonnes of Measured Mineral Resources at average grades of 5.36 g/t gold, 0.83% copper, and 8.3 g/t silver, and 627,000 tonnes of Indicated Mineral Resources at average grades of 4.97 g/t gold, 0.86% copper, and 9.5 g/t silver, and 151,000 tonnes of Inferred Mineral Resources at average grades of 4.21 g/t gold, 0.71% copper, and 9.8 g/t silver. Mineral Resources were estimated by Mike Waldegger, PGeo, at a 3.0 g/t gold equivalent cut-off grade and is based on drillhole samples collected between 1980 and 2004. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

A drill program is recommended to validate the geology logging, assess analytical bias with respect to the historical sampling, to significantly increase the confidence in the bulk density model and to significantly increase confidence in the relationship between grade and bulk density. A drill program is also recommended to determine if the resource can be expanded in areas of limited drilling or where the mineralization is not completely cut off.

1.14 Mineral Reserve Estimates

There are no mineral reserve estimates for the Property.

1.15 Mining Methods

Due to the wide widths and relatively low grade of the mineral economy dictates that low-cost underground mining methods must be employed. The mining method chosen for recovery of the mineral is long-hole open-stope mining, with sub-levels developed at 15-m vertical intervals.

1.16 Recovery Methods

The Willa mineral responds well to the typical grinding and flotation conditions. The testwork data are considered adequate for process design purposes provided the data are interpreted conservatively and flexibility is incorporated into the flotation circuit to permit easy circuit changes. Three concentrates will be produced from the Willa deposit: gravity (coarse gold), chalcopyrite (copper/gold) and pyrite (gold).

With the present required grind of 80% <50 μm, the MAX mill has a production capacity for Willa mineral, of between 300 and 350 tonnes per day. With the completion of the installation of the larger ball mill, the MAX mill will be capable of processing in excess of 500 tpd.

1.17 Project Infrastructure

The project infrastructure includes both the Willa mine and MAX mill. They are connected by 142 km of road. The Willa deposit has been developed over several decades, with 2,385 m of underground workings. Power for the Willa mine will be serviced by diesel generation near the 1025 L portal.

The present infrastructure at the MAX property includes the MAX mine, mill and tailings pond.

1.18 Market Studies and Contracts

A market study was conducted, based on 2012 data obtained from Wood Mackenzie, an International-based research and consulting firm dealing in the fields of Energy, Mining and Metals. There are no existing metal contracts regarding production from the Willa.

1.19 Environmental Studies, Permitting, and Social of Community Impact

The Property is permitted for exploration activities such as road building, and underground rehabilitation, drilling, bulk sampling. Certain environment studies, as described in this Report, will be needed to meet permit requirements. There is no mine permit for the Property.

It is recommended that continued environmental testwork be conducted.

With regards to socioeconomic considerations, it is recommended that Discovery Ventures proceed with public information meetings be held at Silverton, New Denver, Nakusp, Trout Lake, plus consultative discussions with First Nations that may be affected by the project.

1.20 Capital and Operating Costs

The CAPEX of the entire project is less than twenty-two million dollars ($C), which includes not only the mine development but also for the acquisition and renovations of the modern milling facilities of the MAX mine. The total operating costs are estimated at about $C 98/t.

1.21 Economic Analysis

An economic analysis was prepared and is based on Measured and Indicated mineral resources from eight planned stopes in the West Zone of the Willa Mineral Resources. It does not include Inferred Mineral Resources nor does it include the Mineral Resources of the North and East Zones.

The tonnage and average grade for the diluted, recoverable mineral is estimated at 617,460 tonnes grading 4.51 g Au/t, 0.81 % Cu, and 8.39 g Ag/t, with an in-situ value of $C 217.58/t.

Based on a mine and mill production rate of 146,000 tonnes per year, the capital cost for the project is estimated at approximately $C 21.3 million, of which approximately $C 3.0 million has been spent to date. The revenue is estimated at $C 140.3 million, the operating costs at $C 62.5 million, cumulative cash flow at $C 56.1 million, the after-tax Net Present Value (based on a 10% discount rate) at $C 38.7 million, and the after-tax Internal Rate of Return at 83%. The mine has an estimated lifespan of 4.2 years.

The projected recoverable gold by a combination of gravity and flotation presently constitutes approximately 80% of the revenue of the operation as compared to 18% of the revenue from the copper and 2% for silver.

Due to the accumulated tax pools of about $C 47 million at the MAX operation (which was included as part of the purchase price of the MAX property), it is anticipated that there will be no income tax payable over the (present) 4.25-year life of the Willa mine.

Metals prices were assumed to be $US 1200/oz. for gold, $US 2.75/lb. for copper, $US 16.50/oz. for silver, and a US dollar at $C 1.30. Milling recoveries were assumed to be 83.9% for gold and 90% for copper and 82% for Silver.

The PEA is preliminary in nature and there is no certainty that it will be realized.

2.0 Introduction and Terms of Reference

This Amended Preliminary Economic Assessment Technical Report (the "Report") on the Willa Property (the "Property") was prepared for Discovery Ventures Inc ("Discovery Ventures") at the request of Mr. Akash Patel, president of Discovery Ventures. The purpose of the Report is to disclose an updated Mineral Resource estimate and to report the results of a Preliminary Economic Assessment based on a preliminary Mine Plan. This Report is amended from the Preliminary Economic Assessment & Technical Report, Willa MAX Project, for Discovery Ventures Inc, dated May 26, 2014, and authored by W. M. Ash, PEng.

The Report is based on a review of previous reports filed by a number of operators who have conducted mineral exploration programs within and around the current boundaries of the Property and have filed this work for assessment credit with the British Columbia Ministry of Energy and Mines ("BCMEM"). These assessment reports are available as free downloadable Adobe Portable Document Format (PDF) files from the BCMEM Assessment Report Indexing System ("ARIS"). Other significant data are from the BP Minerals ("BP") – Rio Algom Joint Venture and a later BP – Rio Algom – Northair Mines Ltd ("Northair") Joint Venture. [Much of the reporting refers to Riocanex, a subsidiary of Rio Algom]. They include geological drill logs with analytical results, including at times copies of signed laboratory reports, from underground drill programs, maps showing analytical results of sampling of underground workings, and various reports. A list of the materials cited is contained in the References section of the Report. The authors are satisfied that the information contained in publicly available assessment reports and internal company reports was collected and processed in a professional manner following industry best practices applicable at the time, and that the historical data give an accurate indication of the nature, style and possible economic value of mineral occurrences on the Property. Exploration carried out in 2004 was in accordance with CIM Exploration Best Practices.

There are three co-authors: Wayne Ash ("Ash"), PEng; William Gilmour ("Gilmour"), PGeo; and Michael Waldegger ("Waldegger"), PGeo. Ash visited the Property on October 29, and the MAX mill site on October 30, 2015. Gilmour and Waldegger visited the Property and the MAX mill site on November 2, 2015.
Ash is solely responsibility for the following sections:

- 1.4-1.5, 1.12, 1.14-1.21, 5, 6, 13. 15-22, 25.1. 25.3-25.10, 26.1, 26.3-26.4

Gilmour is solely responsible for the following sections:

- 1.1-1.3, 1.6-1.10, 2-4, 7-11, 23-24, 27

Waldegger is solely responsible for the following sections:

- 1.11, 1.13, 12, 14, 25.2, 26.2

Units of measure in this report are metric; monetary amounts referred to are in Canadian dollars ($C) or United States dollars ($US), as noted.

3.0 Reliance on Other Experts

Details of the status of mineral title ownership on the Property were obtained from the BC Mineral Tenures Online ("MTO") database system managed by the BCMEM. This system is based on mineral tenures acquired electronically on-line using a grid cell selection system. Mineral title boundaries are based on lines of latitude and longitude.

Copies of the various Option Agreements and Amendment Agreements were provided to the co-author Gilmour. Although the author has no reason to believe this information is inaccurate, a detailed audit of the option agreements among the Vendors and various parties has not been completed and the author Gilmour is relying solely on the information that has been provided by the various parties.

FortyTwo Metals, which is 100% owned by Discovery Ventures, has a non-capital loss of about $C 15 million and an undepreciated capital cost (UCC) of about $C 32 million, for a total of about $C 47 million. These amounts have been confirmed by the accounting firm of MNP, in Vancouver. MNP states that upon amalgamation of FortyTwo and Discovery Ventures, it does not foresee any limitation on the use of the tax pool by the amalgamated entity. Discovery Ventures has confirmed that approval of the amalgamation has been granted by the TSX Venture Exchange, with just internal reorganization of the two companies to complete. Therefore, it is reasonable to assume that the amalgamation will be completed before there is any mining on the Willa and that the tax pool could be used.

4.0 Property Description and Location

The Willa Property, containing the Willa gold-copper-silver deposit, is located in the steeply incised Aylwin Creek Valley of southeastern British Columbia, south of Silverton and east of Slocan Lake (Figures 1 and 2). The Property is mainly located in the northwest portion of BC Geographic System map 082F084, with some of the mineral titles in the northeast of 082F083 and the southwest of 082F094. This area corresponds to NTS map sheet 082F14W.

The portal to the 1025 Level underground workings of the Willa deposit is at 5526270 north and 473400 east, NAD 83 UTM. From the portal, Slocan Lake is about 2 kilometres ("km") west and Highway 6 about 700 m west.

The Property is located approximately 7.5 km south of Silverton and 12 km south of New Denver, along Highway 6. The nearest airport is at Castlegar, 86 km to the south via Highways 6 and 3A.

Nelson is 88 km to the south via Highway 6. Access to Vernon, in the Okanagan Valley, is 254 km west via Highway 6 and the Needles ferry. Revelstoke, on Highway 1, is 162 km north via Highways 6 and 23 and the Galena Bay ferry. The MAX mill is 142 km north of the Property via road.

4.1 Mineral Titles

The 24 mineral titles (claims) comprising the Property, shown in Table 1, total 5,620 hectares ("ha") in area. The titles are in good standing to at least February 18, 2016, with the title on which the Mineral Resource is located good until January 19, 2020. The good-to dates in Table 1 are dependent on the acceptance by the BCMEM of Assessment Report 35168 and the related Statements of Work 5521512 and 5544546. The title information was obtained using the Mineral Tenure Online ("MTO") search engine available on the British Columbia Geological Survey Branch website. All titles listed in the table are in the Slocan Mining Division within the NTS map sheet 82F/14W and BC Map Sheets 82F.083, 82F.084 and 82F.094. The title map shown in Figure 2 was generated from GIS spatial data downloaded from the Government of BC, Integrated Land Management Branch (ILMB), Land and Resources Data Warehouse (LRDW) data discovery and retrieval system (http://archive.ilmb.gov.bc.ca/lrdw/). These spatial layers are generated by the MTO electronic staking system that is used to locate and record mineral titles in British Columbia.

Most of the Property is underlain by Crown land; that is, the surface rights are owned by the Province of British Columbia. Two mineral titles are underlain by four private district lots, as shown on Figure 2. This area is west of the Willa deposit and the private surface rights will not interfere with work on the Willa deposit.

A mineral title (542206) owned by a third party abuts the eastern portion of the East Zone (Figure 7.2)

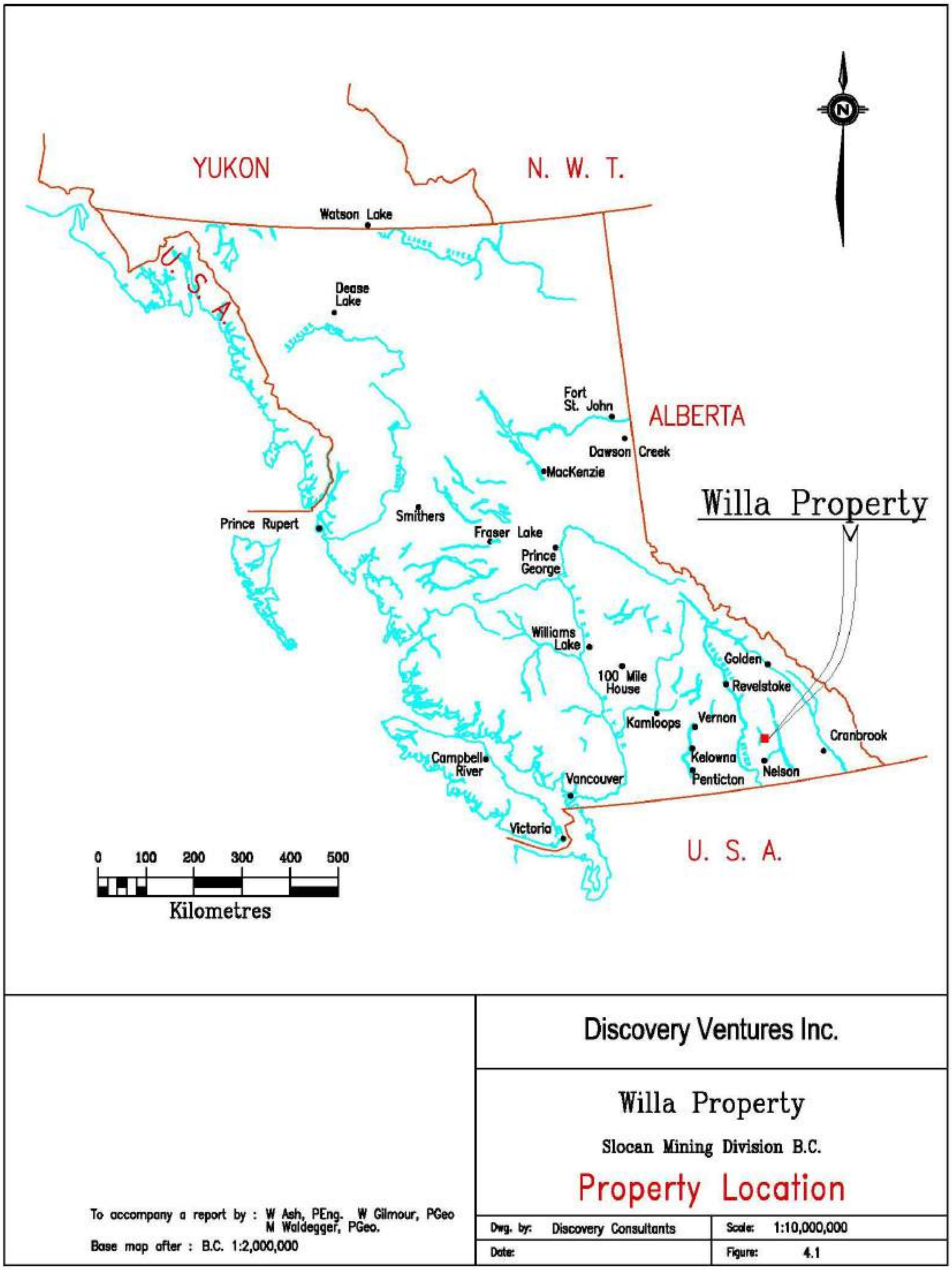
Portions of mineral titles 849697 and 708162 overlap onto pre-existing Crown-granted mineral claims, and hence the Property has no mineral title in this area (Figure 4.2). The vast majority of the mineral exploration has been carried out on title 849697, on which the Willa deposit is located.

A very small portion of titles 8396665 and 839666 is reduced in areal extent due to the Kokanee Glacier Provincial Park. This area is about 8 km from the Willa deposit and is in a separate drainage basin, and therefore is not significant.

Table 4.1: List of Mineral Titles, Willa Property, BC

Mineral Title	Area (ha)	Issue Date	Good To Date
699945	187.5	2010/jan/15	2017/jun/19
700003	166.6	2010/jan/15	2017/jun/19
708162	270.6	2010/feb/26	2017/jun/19
839663	520.5	2010/dec/03	2017/jun/19
839664	520.5	2010/dec/03	2017/jun/19
839665	515.9	2010/dec/03	2017/jun/19
839666	503.6	2010/dec/03	2017/jun/19
839668	499.4	2010/dec/03	2017/jun/19
839669	270.5	2010/dec/03	2017/jun/19
839670	187.4	2010/dec/03	2017/jun/19
839671	20.8	2010/dec/03	2016/apr/04
844403	229.1	2011/jan/25	2017/jun/19
845349	124.9	2011/feb/03	2016/apr/04
846135	520.8	2011/feb/11	2017/jun/19
849697	437.2	2011/mar/24	2020/jan/19
865017	62.4	2011/jul/07	2016/feb/18
904361	62.4	2011/oct/02	2017/jun/19
904409	166.5	2011/oct/03	2017/jun/19
904889	41.6	2011/oct/05	2017/jun/19
905610	20.8	2011/oct/06	2017/jun/19
1010951	20.8	2012/jul/10	2017/jun/19
1029377	187.3	2014/jul/03	2017/jun/19
1029379	62.4	2014/jul/03	2017/jun/19
1033093	20.8	2015/jan/03	2018/jan/03
Total	5620.3		

The good-to dates of these titles are as of the report amendment date, January 19, 2016.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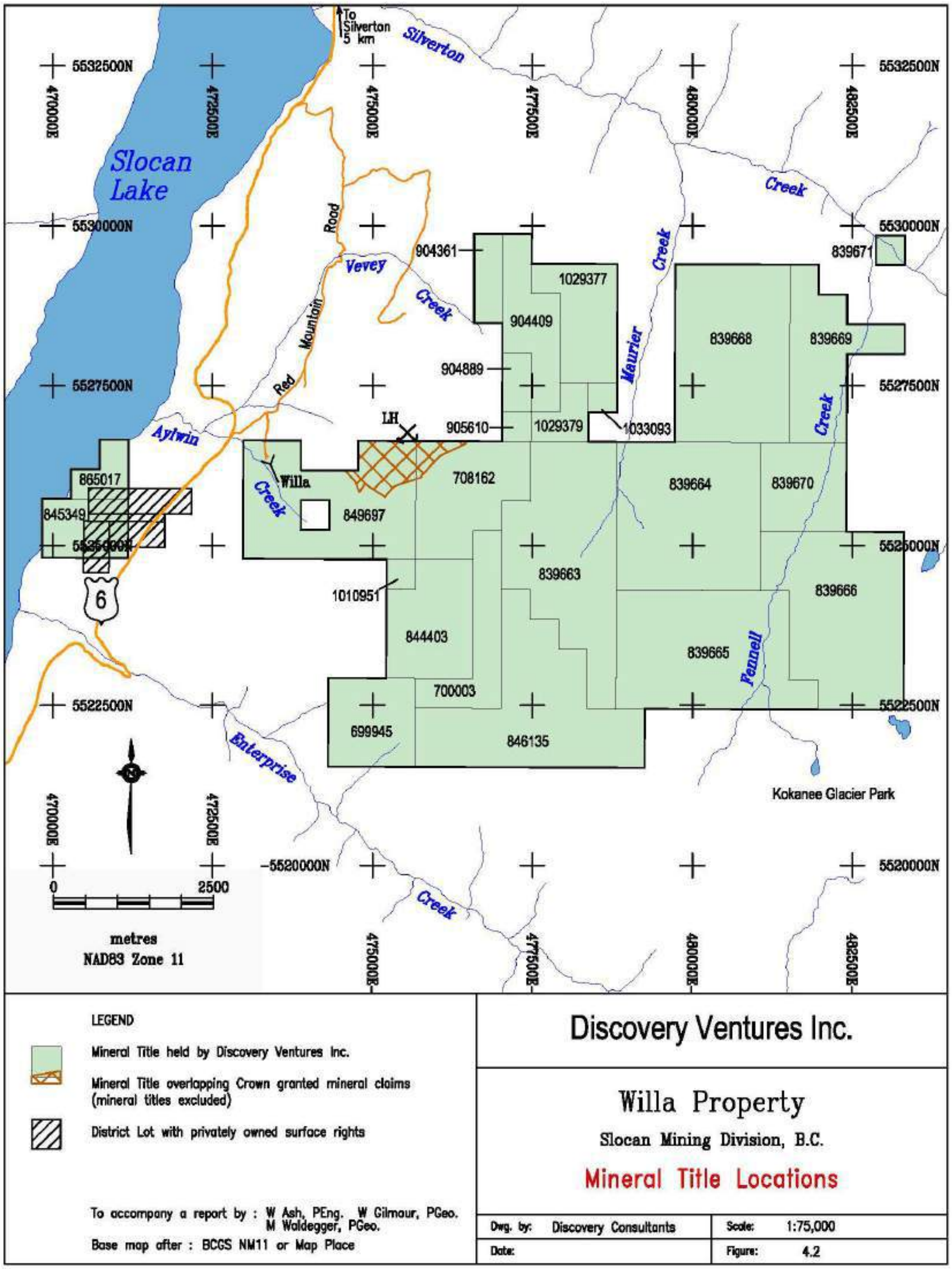
N. W. T.
U. S. A.
Watson Lake
Dease Lake
Fort St. John
ALBERTA
Dawson Creek
MacKenzie
Willa Property
Prince Rupert
Smithers
Fraser Lake
Prince George
Williams Lake
100 Mile House
Golden
Revelstoke
Kamloops
Vernon
Cranbrook
Kelowna
Nelson
Penticton
Campbell River
Vancouver
Victoria
U. S. A.
0 100 200 300 400 500
Kilometres
Discovery Ventures Inc.
Willa Property
Slocan Mining Division B.C.
Property Location
To accompany a report by : W Ash, PEng. W Gilmour, PGeo M Waldegger, PGeo.
Base map after : B.C. 1:2,000,000
Dwg. by: Discovery Consultants
Scale: 1:10,000,000
Date:
Figure: 4.1


To Silverton 5 km
Silverton
Creek
Slocan Lake
Vevey
Creek
Red Mountain Road
Maurier Creek
Creek
Aylwin
Willa
Creek
Fennell
Enterprise
Creek
LH
904361
1029377
904409
904889
905610
1029379
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846135
5532500N
5530000N
5527500N
5522500N
5520000N
470000E
472500E
475000E
477500E
480000E
482500E
Kokanee Glacier Park
0
2500
metres
NAD83 Zone 11
LEGEND
Mineral Title held by Discovery Ventures Inc.
Mineral Title overlapping Crown granted mineral claims (mineral titles excluded)
District Lot with privately owned surface rights
To accompany a report by : W Ash, PEng. W Gilmour, PGeo. M Waldegger, PGeo.
Base map after : BCGS NM11 or Map Place
Discovery Ventures Inc.
Willa Property
Slocan Mining Division, B.C.
Mineral Title Locations
Dwg. by: Discovery Consultants
Scale: 1:75,000
Date:
Figure: 4.2

4.2 Title Ownership

The Property comprises 24 mineral titles, totalling 5,620 ha. The titles are registered on MTO as to Discovery Ventures.

The Vendors of the Property were R.J. Billingsley, R. Bickner, D.E. Kress and S.C. Richards. They formed a BC incorporated company, 0951719 B.C. Ltd, after acquiring the Property and on November 16, 2012 optioned the Property ("Option Agreement") to Discovery Ventures. Under the terms, Discovery Ventures had the right to earn 100% in the Property, subject to a 2.5% net smelter return ("NSR") in favour of the Vendors. Subsequent to the original agreement, the Vendors acquired four additional mineral titles, which were included in the agreement with Discovery Ventures. Discovery Ventures could earn 80% interest by:

- Paying $C 50,000 to the Vendors upon execution of the Option Agreement
- Paying $C 150,000 and issuing 1,000,000 shares to the Vendors upon TSX Venture Exchange approval ("Approval") on the transaction
- By the first anniversary of the Approval paying $C 350,000 and issuing 4,000,000 shares to the Vendors, and incurring a further $C 500,000 of work on the Property
- By the second anniversary of the Approval paying $C 1,000,000 and issuing 6,000,000 shares to the Vendors, and incurring another $C 500,000 of work on the Property
- By the third anniversary of the Approval paying $C 1,450,000 and issuing 7,000,000 shares to the Vendors, and incurring a final $C 500,000 of work on the Property
- Upon exercise of the option, Discovery Ventures could purchase the remaining 20% interest at fair market value.

After various amended agreements and new agreements, on September 24, 2013, January 13, 2014, February 27, 2014, April 15, 2014, June 16, 2014, September 26, 2014 and November 26, 2014, an Amendment to Assignment, Assumption and Framework Agreement (Willa Property) was signed on September 14, 2015. It states that the Vendors grant to Discovery Ventures sole and exclusive right and option (the "Option") to acquire a 100% interest in the Property, subject to a 2.5% NSR, free and clear of all liens, claims and rights or interests of any other person. Discovery Ventures may exercise the Option by paying the Vendors $C 130,000 [paid].

Furthermore, if Commercial Production has not commenced by September 15, 2015 and subsequent years, an additional $C 144,000 ("Advanced Royalty") will be paid to the Vendors in each respective anniversary year in monthly instalments.

As well, if Discovery Ventures has exercised the Option but either (i) Commercial Production has not commenced before September 15, 2020, or (ii) the Vendors have provided Discovery Ventures with written notice of default of the Advance Royalty and Discovery Ventures has not cured such default within three months of receiving such notice, Discovery Ventures will transfer back to the Vendors 100% interest in the Property, which will be left in gold standing for a period of three years or more.

On November 4, 2013, Discovery Ventures entered into an agreement with Roca Mines Inc ("Roca") and FortyTwo Metals Inc ("FortyTwo"), a wholly-owned subsidiary of Roca, whereby Roca granted an exclusive option to Discovery Ventures to acquire all the issued and outstanding common shares of FortyTwo for $C 950,000 by January 8, 2014. After subsequent amended agreements (January 5, 2014, March 6, 2014 and October 5, 2015) and the issuance of 3,000,000 shares and 3,000,000 share purchase warrants of Discovery Ventures, Discovery Ventures finalized the acquisition of 100% of the shares of FortyTwo. FortyTwo holds assets that include an underground molybdenum mine, crushing, milling and concentrating facilities (the "MAX mill"), tailings storage facilities, approximately $C 47 million in tax losses, mineral titles and mineral leases, including Mineral Lease 577449 on which the MAX mill is located.

4.3 Title Acquisition and Work Requirements

In British Columbia, an individual or company holds the available subsurface mineral rights as defined in Section 1 of the Mineral Tenure Act by acquiring title to a mineral title; this is now done by electronic staking. In addition to mineral rights, a mineral title conveys the right to use, enter and occupy the surface of the claim or lease for the exploration and development or production of minerals, including the treatment of ore and concentrates, and all operations related to the business of mining providing the necessary permits have been obtained.

In order to maintain a mineral title in good standing, exploration work (assessment work) or cash in lieu to the value required must be submitted prior to the expiry date. The amount required is specified by Section 8.4 of the British Columbia Mineral Tenure Act Regulation. In 2012, when the regulations were changed, the anniversary years for all claims in BC were reset to year 1. This act states that the value of exploration and development work required to maintain a mineral title for one year is at least:

- $C 5 per hectare during each of the first and second anniversary years, and
- $C 10 per hectare during the third and fourth anniversary years, and
- $C 15 per hectare during the fifth and sixth anniversary years, and
- $C 20 per hectare for subsequent anniversary years.

Assessment value can be allocated to titles contiguous to those on which the exploration was carried out. Cash in lieu fees are double those of exploration requirements. Up to 10 years of work or cash in lieu can be applied on a mineral title. A change in anniversary date can be initiated at any time and for any period of time up to 10 years. In order to obtain credit for the work done on the Property, Discovery Ventures must file a Statement of Work and submit an Assessment Report documenting the results of the work done on the Property. This report must also include an itemized statement of costs.

A Mining Lease is the production tenure for mining. Both subsurface mineral rights and surface rights are held by the lease holder. The assessment work system of maintaining title does not apply for mineral leases; an annual tax of $C 20/ha is applied. This Report does not include an estimate as to the timing of obtaining a lease or of the cost. Once a Mining Lease is granted, the value of exploration, development or mining on the lease cannot be allocated to contiguous titles for the purpose of maintaining their title.

4.4 Permits Required to Conduct Exploration

Prior to initiating any physical work such as drilling, trenching, bulk sampling, camp construction and access upgrading or construction, a Notice of Work ("NoW") permit application must be filed with, and approved by, the BCMEM. The permit authorizing this work must be granted prior to commencement of the work and the permit will likely require the posting of a reclamation bond. The filing of the NoW initiates engagement and consultation with other stakeholders including First Nations. Information supplied by the Mineral Titles Branch of the BCMEM lists the following First Nations that may have aboriginal interests in the area of the Property:

- Ktunaxa Nation Council – Akisqnuk First Nation
- Ktunaxa Nation Council – Lower Kootenay Band
- Ktunaxa Nation Council – St. Mary's Indian Band
- Ktunaxa Nation Council – Tobacco Plains Indian Band
- Okanagan Nation Alliance – Penticton Indian Band
- Okanagan Nation Alliance –Lower Similkameen Indian Band
- Okanagan Indian Band
- Shuswap Indian Band

The Mineral Titles Branch also states that there are no Wildlife Management areas on the Property.

Discovery Ventures has a Mines Act Permit (MX-5-786) for mineral exploration on the Property, valid from November 17, 2014 to December 31, 2015. Approval is granted for the rehabilitation of the 1025 and 1100 levels and portals, sampling and geological mapping, and removal of a 2 t sample from underground for metallurgical testing. Water samples shall be taken from all drainages from the underground workings and analyzed for mineral content and pH values. Discovery Ventures received written notice on Sept 18, 2015 from the Inspector of Mines (Permitting), that the expiry date of the permit was extended to December 31, 2017. Discovery Ventures has posted a $C 10,000 reclamation bond with the BCMEM. The bond will be returned to Discovery Ventures upon BCMEM inspection and approval of reclamation.

On July 3, 2015 Discovery Ventures applied with the BCMEM for an exploration permit, which includes an underground 10,000 t bulk sample, core drilling and rock sampling. An amended permit (MX-5-463) was issued on November 2, 2015, and is valid until August 16, 2020. The existing reclamation bond for this permit is $C 13,000.

The existing road to the Property from the Red Mountain Road is not suitable as a haulage road. On August 20, 2015, Discovery Ventures was granted a permit (S25934) for road construction and maintenance, by the BC Ministry of Lands, Forests and Natural Resource Operations ("MLFNRO"). Also, on November 2, the BCMEM granted a Free Use Permit, which enables Discovery Ventures to cut up to 50 m^3 of timber.

Discovery Ventures does not have a Mine Permit on the Property and no application has been made this time. Under the BCMEM Mines Fee Regulation, there is a fee for the review of an application. This fee must be paid before the processing of the application can begin. This Report does not include an estimate as to the timing of obtaining a mining permit or of the cost. An additional reclamation bond is likely for this permit.

The MAX mill has BCMEM permits in good standing, but which may need amending and/or extending if material is going to be shipped from the Willa and if new tailings are to be produced.

Work on the Property needs to follow the Mines Act, the Health, Safety and Reclamation Code for Mines in British Columbia, and the Handbook for Mineral & Coal Exploration in British Columbia.

4.5 Environmental Liabilities

In 1990, Robert L. Hallam Environmental Management Ltd (Hallam, 1990b) reported that: "On the basis of site sampling and geological evaluation it has been determined that the Willa Project waste rock stockpiles have a near neutral acid-base accounting and are unlikely to result in a significant acid generating or environment concern".

However, the BC Environmental Assessment Office stated in 2003 (Connolly 2003a and 2003b) that the BCMEM had identified the potential for the project to create acid rock drainage and metal leaching, and therefore an Environmental Assessment Certificate would be needed.

The underground work in the 1980s produced waste rock, which is now a waste dump about 100 m northwest of the 1025 Level portal.

Discovery Ventures is required to post reclamation bonds for any permitted exploration.

4.6 Other Liabilities

Aside from the above (Sections 4.1 to 4.5), the co-author Gilmour is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5.0 Accessibility, Physiography, Climate, Local Resources and Infrastructure

Access to the Property is as follows: from the bridge crossing Silverton Creek, in Silverton, south on Highway 6 for 7.7 km; then left onto Red Mountain Road South for 0.3 km, then right. Roads to this point are paved and maintained year round. Leaving the paved road on a 4-wheel-drive road there is a locked and posted gate at 0.8 km and the Willa mine dump at 1.2 km. From there it is 0.3 km on foot to the main portal (1025 Level) (Figure 4.2). The upper portal (1100 Level) is accessed by switchbacks (on a quad) for another 0.75 km.

The nearest airport is at Castlegar, 86 km to the south via Highways 6 and 3A. Nelson is 88 km to the south via Highway 6. Access to Vernon, in the Okanagan Valley, is 254 km west via Highway 6 and the Needles ferry. Revelstoke, on Highway 1, is 162 km north via Highways 6 and 23 and the Galena Bay ferry.

The nearest scheduled flight access to the area is either the Castlegar airport, with daily flights from Vancouver and Calgary, and the Trail airport with daily flights from Vancouver. Castlegar airport is noted for its difficult flight conditions that often cause cancellations during cloudy winter/spring days. The Property can also be reached by vehicle using Highway 3 or the Coquihalla/TransCanada Highway from Vancouver in approximately 9 to 10 hours.

The access route from the Property to the proposed mill site on the MAX property (a distance of 142 km), via Highways 6, 23 and 31, is shown on Figure 3. In the village of Trout Lake one turns south on Westside Road for 2.3 km; then right on a posted and gated road for an additional 4 km to the mill site.

The Property is in forested, rugged terrain with elevations ranging from 550 metres at Slocan Lake to 2,415 metres ("m") along the ridge separating Aylwin and Congo Creeks. Slopes are frequently greater than 35°. If a production decision is made, it is proposed that the mineralized material having economic value be shipped off-site to the MAX mill for processing.

At low elevations the undergrowth can be quite thick with devil's club, stinging nettles and slide alder. The predominant deciduous trees are poplar and birch, commonly occurring at lower elevations. The main portion of the Property is treed by western hemlock, western red cedar, Douglas fir and larch. At higher elevations, the forest grades into Engelmann spruce and alpine fir, and then into alpine meadows and windswept ridges (Heather, 1985).

The climate is typical of the Central Kootenay Region, with cool winters and warm summers. A maximum temperature range of between −34° and +40° Celsius ("C") (Heather, 1985) has been recorded but in many years the range is −5° and +30°. The nearest weather station is at New Denver, approximately 14 km to the north, at an elevation of 560 m above mean sea level. The average January temperature in New Denver is -0.3° C. The average August temperature is 28° C. Precipitation throughout the year is relatively constant at 60 to 75 millimetres ("mm") per month and averages 880 mm per annum. Note that the elevation of the 1025 Level portal is approximately 500 m higher than at the weather station, so precipitation can be expected to be somewhat higher than at New Denver, with annual snowfall of three to four metres at the portal. Underground exploration, development and mining could be carried out year round.

The nearest population centre is the village of Silverton (population 195, 2011 census). The Village of New Denver (population 504) is 5 km north of Silverton. The Village of Slocan (population 296) is 17 km south of the Willa Property. The major communities of Nelson (population 10,230), Castlegar (population 7,816), Trail (population 7,681), and Revelstoke (population 7,139), are within a 2 to 3 hour drive from the Property.

The closest single-phase power line runs along Red Mountain Road. Three-phase power would likely require a new 7.5-km line from Silverton along Highway 6. Buried telephone lines run along Red

Mountain Road. There is fibre-optic cable running all the way up the Slocan Valley. Cellular phones have limited coverage in this area. Potable water is not available on the Property.

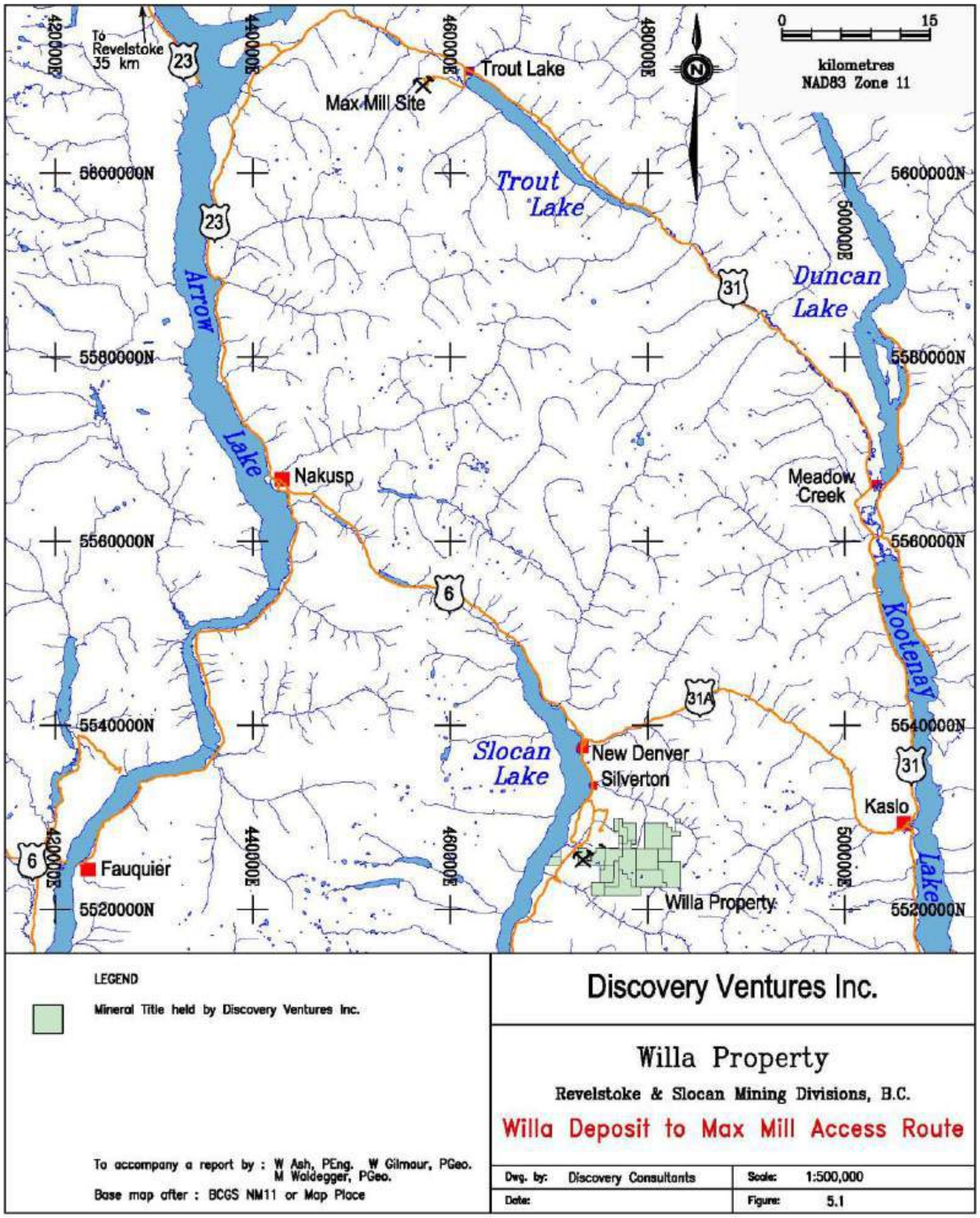


Figure 5.2 Willa Deposit to MAX Mill Access Route 3D View

6.0 History

The Property has been explored on and off since 1893. However, the majority of the work to develop the Property has occurred only more recently. In the mid-1960s Cominco, AMAX Exploration ("Amex") and Western Mining Company ("Western Mining") identified copper and molybdenum mineralization. During the early 1980s the Willa deposit was explored on surface by a joint venture between Rio Algom and BP. In April 1985, Northair Mines ("Northair") joined the joint venture and completed the present underground workings. In 2004, Bethlehem Resources (1996) Corporation ("Bethlehem") completed a 39-hole underground core drill program. An underground survey by R.C. Powers and Associates was completed to tie in underground data to UTM NAD 83 coordinates.

Exploration work carried out by the Vendors and by Discovery Ventures is described in **9.0 Exploration**.

A brief summary of the exploration history follows:

1893 Three adits driven by prospectors: Willa No.1, 80 m; Willa No.2, 10 m; Rockland, 70 m

1899 Crown-granted claim known as the Rockland owned by Willow Gold Mining Co produced 300 tons.

1900 Rockland claim was then owned by Spinks, Graves and Watson. They completed 91 m combined of drifting in the Willa No. 1, Willa No. 2 and Rockland adits.

1901 Granby Consolidated inspected the property and chip sampled the Willa No. 1, Willa No. 2 and Rockland adits.

1904 BC Minister of Mines Annual Report (p 173) describes the Willa showing as a mineralized zone about 60 feet wide carrying copper sulphides with gold values.

1912 Granby Consolidated again inspected the property and again turned it down (low grade).

1930s Claims were transferred to W. J. Nicholls.

1936 Optioned to Slocan Lake Gold Mining Company. They completed another 32 m of drifting and 3.6 m of raising.

1955 Egil Lorntzsen sampled the Rockland adit and found only trace gold.

1964 The Rockland and adjoining claims (17 Crown-granted claims) were purchased by Northlode Exploration from D.H. Hawkings.

1964-1965 Cominco optioned the property and completed four diamond drillholes. Minor copper and gold mineralization was found.

1964-1967 Hawkings property was optioned to Rockland Mining. They carried out a small surface diamond drilling program near the mineralized occurrences.

1967-1969 AMAX completed trenching and geochemical sampling of the surface. Gold-silver-copper and copper-molybdenum anomalies were discovered (1968) leading to a surface diamond drill program in 1969. Good mineralization was intersected.

1970 Western Mining and AMAX completed a joint venture on the claims. An underground diamond drilling program (457 m) for molybdenum was completed using some of the old workings as setups.

1970 Rockland Mining Ltd reported started drilling at the Little Daisy portal.

1971-1975 Crown-granted claims lapse.

1978-1979 Pete Leontowicz and Bill Wingert acquired the property.

1979 Rio Tinto Canadian Exploration ("Riocanex") optioned the property and staked more claims. BP staked the adjoining properties. Note that some reports refer to Rio Algom, the parent company of Riocanex. Lithogeochemistry survey started.

1980-1984 The Aylwin Joint Venture (Riocanex 50%, BP 50%) explored the property with geochemical surveys and drilling (6,580 m), including some deep (>300 m) holes, which discovered the main mineralized zones.

1984 BP had acquired 72% interest in the property by funding the majority of the exploration.

1985 Northair optioned the property for 50% ownership (BP 36%, Riocanex 14%).

1986-1987 Northair drove the 1025 Level drift and the associated crosscuts. Bench-scale testwork was undertaken by Lakefield Research.

1987 Northair completed the declines and the intermediate 1013 Drift.

The 1025 Level drifts and crosscuts were extended.

The 1100 Level drift and the 1100 Raise were completed.

A rib, face and muck sampling program was completed.

Environmental baseline studies started.

1988 Northair completed an underground diamond drill program, bringing the total of drillholes on the property to 556. Northair retained Rescan Environmental, which completed and filed a Stage One Environmental Assessment Study, and calculated a mineral resource. A 494 tonne ("t") bulk sample of material was collect and processed at a flotation mill (Petersen, 1988b). A jig concentrate was also produced. Northair ceased all exploration and underground development, by which time Northair had earned a 75% joint venture interest in the Willa JV thereby leaving BP Canada and Rio Algom with 25% between them. An historical mineral resource, estimated by D.B. Petersen, PEng, of Northair, included "Proven" and "Probable" categories at a 3.4 grams/tonne ("g/t") gold ("Au") cut-off grade. The "Mining and Geological Ore Reserves", were estimated by Petersen at 495,176 t grading 5.90 g/t Au and 0.868% copper ("Cu") (Petersen, 1988a). A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves and Discovery Ventures is not treating the historical estimate as current mineral resources or mineral reserves.

1990-1991 Treminco Mining Ltd optioned the property and carried out minor sampling and mapping on portions of the 1100 and 1025 Level drifts, but did no development work or diamond drilling. Two economic studies on grade and tonnage, and on how to best mine and process the deposit were completed.

2000-2002 All title held by Treminco and the joint venture partners was dropped and the claims reverted free and clear to Leontowicz and Wingert.

2002 Orphan Boy Resources ("Orphan Boy"), by way of an Option to Purchase Agreement, acquired the property from Leontowicz, Wingert and Hudock.

2003 Orphan Boy converted the Willa data (i.e., drillhole, topography, underground workings) into digital format. An NI 43-101 Technical Report disclosing Mineral Resources was completed (Makepeace, 2003). The report disclosed an estimated Measured Mineral Resource (at a 3.5 g/t Au cut-off grade) of 487,989 t grading 6.77 g/t Au, 0.97% Cu and 11.59 g/t Ag, an Indicated Mineral Resource of 292,457 t grading 5.31 g/t Au, 0.65% Cu and 11.94 g/t Ag, and an Inferred Mineral Resource of 216,177 t grading 6.55 g/t Au, 0.57% Cu and 7.32 g/t Ag. The basis of the estimate was drillhole data from holes completed between 1965 and 1988. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources and Discovery Ventures is not treating the historical estimate as current mineral resources. Orphan Boy transferred title to Bethlehem, a wholly-owned subsidiary of Orphan Boy, with the intent of shipping the mineral to the Bethlehem mill. The Bethlehem mill (Noranda's Goldstream Mill) was located about halfway between Revelstoke and Mica Creek (approximately 230 km from the Property).

2004 Bethlehem completed, with supervision from Discovery Consultants, a 39-hole underground core drill program (Gilmour, 2004). Northair had completed 2,455 m of underground workings in the 1980s and these were refurbished in 2004 to facilitate the drill program. An underground survey by R.C. Power and Associates was completed to tie in underground data to UTM NAD 83 coordinates. The Mine Manager was Stephen Phillips. The drilling is summarized in section **10.0 Drilling**. Minor surface exploration was carried out on the Property, with no discovery of any Willa-style mineralization (Gilmour, 2005).

Scott Geophysics conducted an induced polarization (IP) orientation survey to evaluate its usefulness as an exploration tool on the Property. In total, 3 line-km were run over and off the deposit. The survey successfully detected the Willa mineralized breccia pipe.

2005 Bethlehem filed an updated NI 43-101 Technical Report (Chapman and Makepeace, 2005) disclosing an update Mineral Resource. The report disclosed an estimated Measured Mineral Resource of 495,784 t grading 7.81 g/t Au, 0.94% Cu, and 12.16 g/t Ag, and an Indicated Mineral Resource of 262,415 t grading 5.71 g/t Au, 0.67% Cu and 13.26 g/t Ag. The basis of the estimate was drillhole data from holes completed between 1965 and 2004. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources and Discovery Ventures is not treating the historical estimate as current mineral resources.

Hatch Consultants and others completed a preliminary mine design and mine schedule based on the 2005 Mineral Resource estimate. However, with the prevailing metals prices of that time ($US 436/troy ounce ("oz") Au, $US 1.73/pound ("lb") Cu and $US 7.26/oz Ag) the estimated operating costs did not yield the 15% internal rate of return ("IRR") that was needed to proceed with production. Bethlehem ceased work on the project but retained ownership of the property in anticipation of an up-swing in metals prices.

2006 The key mineral title (390104) inadvertently lapsed and the Willa deposit was staked by the Vendors in September, 2006. No technical work has been conducted on the property since that time and the adit portals were closed under the direction of the BCMEM.

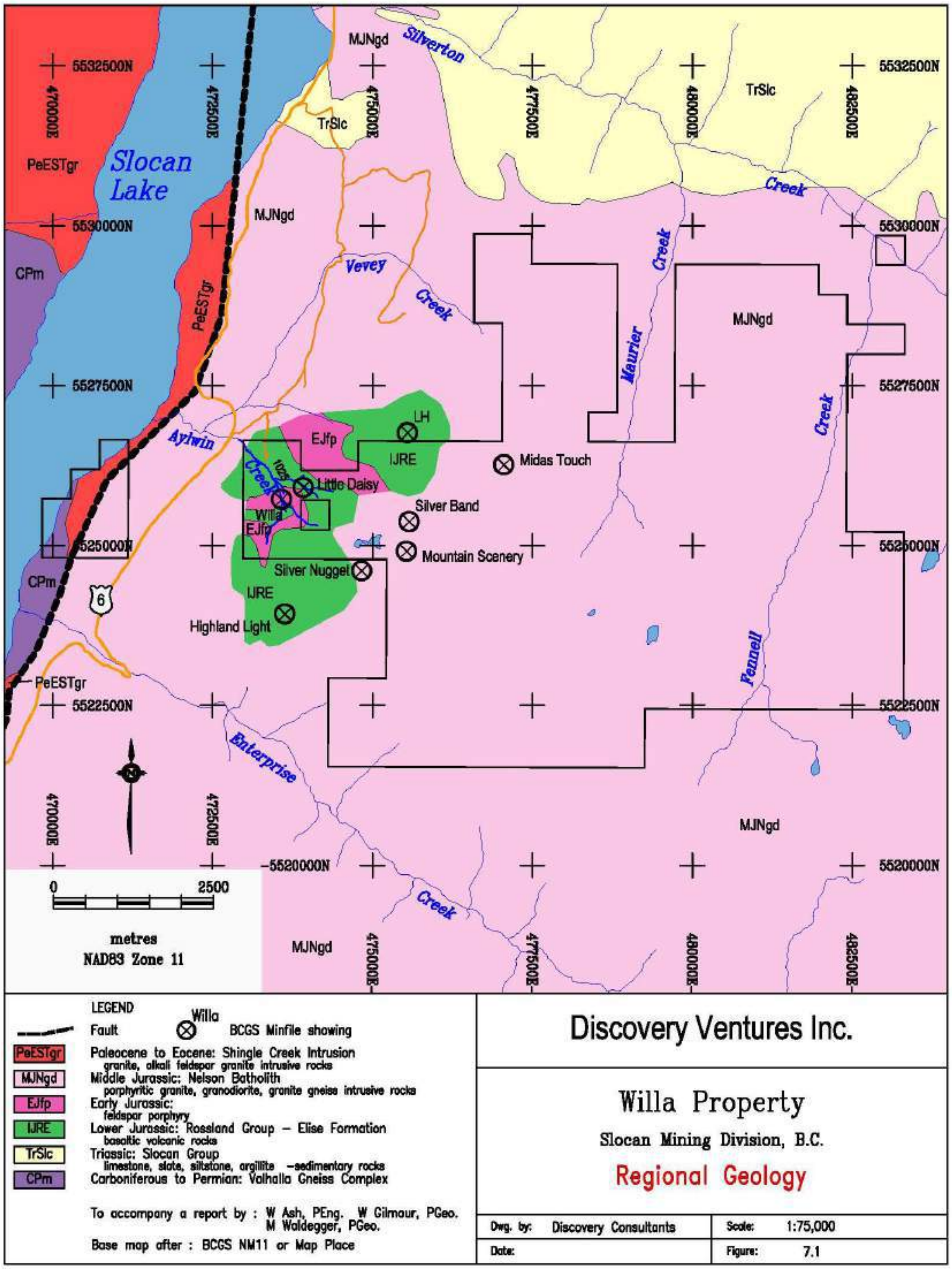
7.0 Geological Setting and Mineralization

7.1 Regional Geology

The Willa deposit is situated in the southern portion of the Selkirk Mountains within the Omineca Crystalline Belt. The deposit is within a roof pendant, composed of basic volcanic and volcaniclastic rocks, which were intruded by felsic dykes (Figure 4). The volcanic rocks have been correlated to the Lower Jurassic Elise Formation of the Rossland Group (map BCGS MN11). The pendant is enclosed and intruded by the Middle Jurassic Nelson Batholith, which is predominantly composed of coarse-grained granodiorite to quartz monzonite. North of the Property and north of the Nelson Batholith is the Upper Triassic Slocan Group metasedimentary sequence of rocks that contains numerous silver-lead-zinc deposits. Structurally adjacent to the west of the Rossland Group is the Precambrian Valhalla Gneiss Complex, which is a highly metamorphosed terrane of remobilized granites and granitic gneisses.

The Slocan Lake Fault is the major structure in the area. This north-trending fault has a dip of 35° to 40° to the east and is of Eocene age.

Generally, the roof pendant rocks have been weakly metamorphosed to lower greenschist facies.


Slocan Lake
Silverton Creek
Vevey Creek
Maurier Creek
Aylwin Creek
Fennell Creek
Enterprise Creek
MJNgd
TrSlc
PeESTgr
CPm
EJfp
IJRE
LH
Little Daisy
Willa
Silver Band
Midas Touch
Mountain Scenery
Silver Nugget
Highland Light
6
5532500N
5530000N
5527500N
5525000N
5522500N
5520000N
470000E
472500E
475000E
477500E
480000E
482500E
0
2500
metres
NAD83 Zone 11
LEGEND
Fault
Willa
BCGS Minfile showing
Paleocene to Eocene: Shingle Creek Intrusion
granite, alkali feldspar granite intrusive rocks
Middle Jurassic: Nelson Batholith
porphyritic granite, granodiorite, granite gneiss intrusive rocks
Early Jurassic:
feldspar porphyry
Lower Jurassic: Rossland Group – Elise Formation
basaltic volcanic rocks
Triassic: Slocan Group
limestone, slate, siltstone, argillite –sedimentary rocks
Carboniferous to Permian: Valhalla Gneiss Complex
To accompany a report by : W Ash, PEng. W Gilmour, PGeo. M Waldegger, PGeo.
Base map after : BCGS NM11 or Map Place
Discovery Ventures Inc.
Willa Property
Slocan Mining Division, B.C.
Regional Geology
Dwg. by: Discovery Consultants
Scale: 1:75,000
Date:
Figure: 7.1

7.2 Property Geology

The Property geology has been described in great detail by Heather (1985) and Wong and Spence (1995). The lithology is summarized in the table below (Table 2) with corresponding rock codes. The geology is illustrated in Figure 4.

The Elise Formation ("Elise") of the Rossland Group, the oldest rocks on the Property, comprises about 75% of a roof pendant within the Nelson Batholith. The rocks range from volcanic siltstone and tuff to coarse breccias and flows. The metavolcanic roof pendant is cut by two felsic porphyritic intrusions. The first intrusion is a quartz latite porphyry ring-dyke and two radial dykes that have a central core of feldspar porphyry and a crosscutting breccia pipe. The second intrusion consists of two sub-parallel igneous bodies consisting of white feldspar porphyry and hornblende feldspar porphyry. A heterolithic breccia intrudes the volcanic and intrusive rocks. Lamprophyre dykes and faults crosscut and sometimes displace the older lithologies.

Table 7.1: Willa Rock Classification

Age	Description		Code
	Lamprophyre Dykes (L)		8
Middle Jurassic	Nelson Batholith		7
Lower Jurassic	Heterolithic Breccia (Hbx)		6
	Feldspar Porphyry (Fp)	White Feldspar Porphyry	5
		Feldspar Porphyry	4
		Hornblende Feldspar Porphyry	3
	Quartz Latite Porphyry (Qlp)		2
	Rossland Group, Elise Formation (V)	Pyroclastic Rocks	1
		Augite Porphyry	
		Volcanic Siltstone	
		Biotite Schists	

Source: Heather, 1985

7.2.1 Rossland Group

Fragmental pyroclastic rocks make up roughly 70% of the Elise. They range from volcanic agglomerates and conglomerates to fine-grained crystal and lithic tuffs (Heather, 1985). Augite porphyry sills or flows are present around the known Willa mineralization. The unit ranges from dark green to black-green and yellow-green and is usually altered (iron-stained or bleached). The augite and plagioclase phenocrysts are euhedral to subhedral and range in size from 0.5 to 4.0 mm. The matrix is composed of augite, feldspars and biotite.

Volcanic siltstone (commonly hornfelsed) is usually interbedded with the augite porphyry and makes up only a small portion of the Elise. The siltstone varies from green (actinolite-quartz-plagioclase-orthoclase) to grey to pink (biotite-plagioclase-orthoclase). Biotite schist is found predominantly to the south and southeast of the heterolithic breccia. This black schist has been found in core and in outcrops and is believed to be related to the augite porphyry unit.

7.2.2 Quartz Latite Porphyry

The quartz latite porphyry unit forms a ring and radial dyke complex within the Elise. Its composition ranges from quartz monzonite to granodiorite with large phenocrysts of plagioclase. The ring dyke structure is elliptical in shape trending 050° with a 5-km by 1-km size. The radial dyke radiates both inward and outward from the ring structure. There is up to 7 % pyrite in the quartz latite porphyry and when it is exposed on surface, the blocky, fractured outcrop has a limonitic stain.

7.2.3 Feldspar Porphyries

The feldspar porphyry intrusive stock is centred within the quartz latite porphyry. It has an elliptical shape that trends 000°. This unit has phenocrysts of plagioclase and quartz with minor pyrite, apatite, titanite and magnetite. Outcrops are oxidized with skins of limonite and manganese oxide.

The white feldspar porphyry intrusive has been identified in two elongated bodies 1 km north of the quartz latite porphyry ring dyke. These highly altered units have large plagioclase and small quartz phenocrysts with minor pyrite and hornblende.

The hornblende feldspar porphyry forms small intrusive bodies and dykes within the quartz latite porphyry and the feldspar porphyry. The large plagioclase and small hornblende phenocrysts are within a groundmass of orthoclase and quartz.

7.2.4 Heterolithic Breccia

The heterolithic breccia lies within the core of the quartz latite porphyry ring dyke and is roughly cylindrical in shape. The cross-section of this pipe is 350 m (north-south) by 200 m. The outer portions of the breccia pipe form a crackle breccia texture. All the above lithologies other than the white feldspar porphyry have been identified as angular to rounded fragments of the pipe. The fragments normally show propylitic or potassic alteration. The matrix of the pipe is an altered iron-rich rock flour composed of plagioclase, quartz and orthoclase with minor actinolite and biotite.

7.2.5 Nelson Batholith

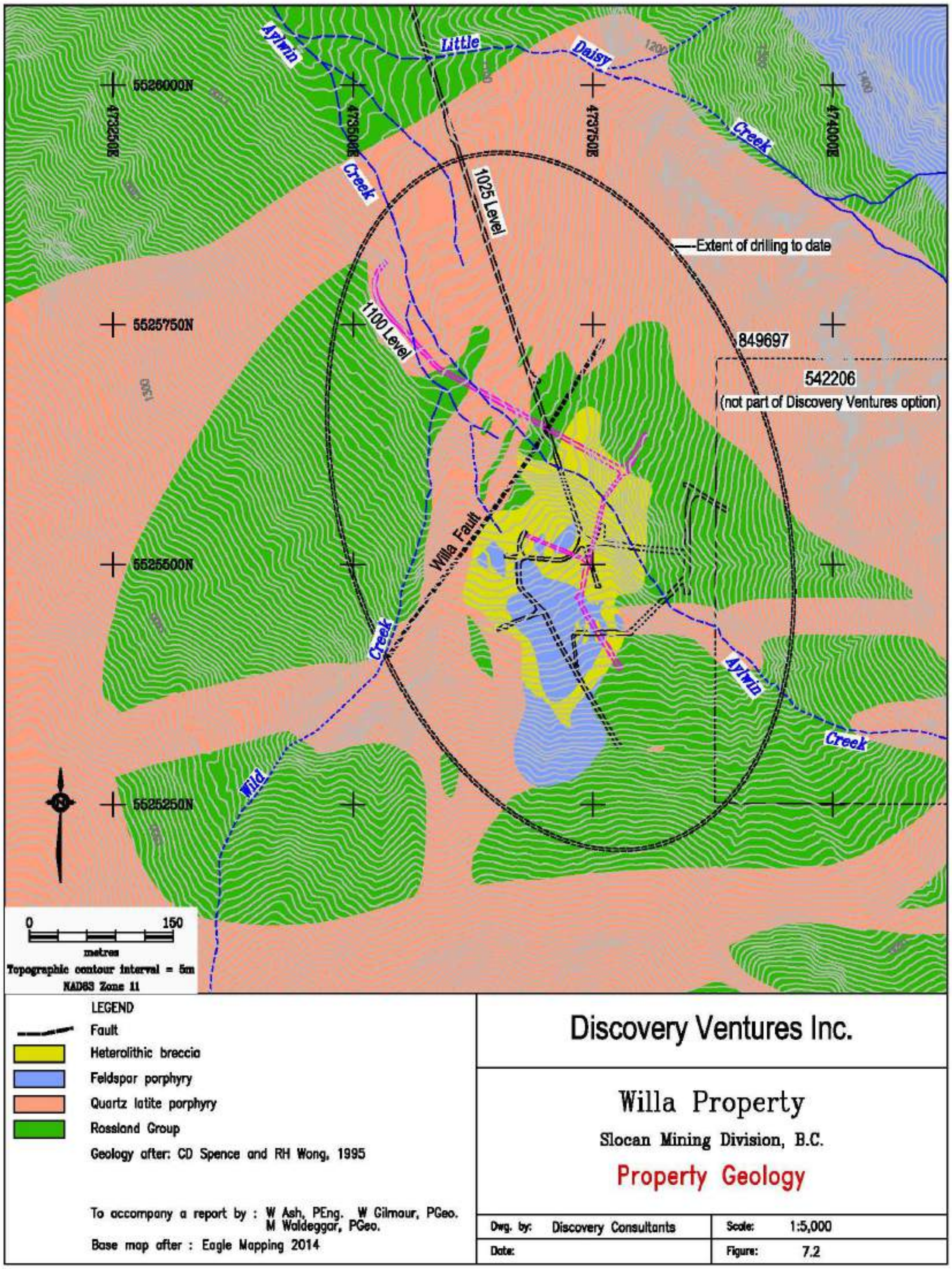
The Nelson Batholith is composed of a variety of granitic rocks ranging from porphyritic granite, quartz monzonite, syenite to granodiorite. The batholith encircles the volcanic roof pendant and does not outcrop near the Willa deposit. Granitic pegmatite dykes have been intersected in some of the deep drillholes but the main Nelson Batholith has not been intersected.

7.2.6 Lamprophyre Dykes

Mafic intrusive rocks, lamprophyre dykes, are intersected in many drillholes. They normally trend north-south, are steeply dipping and are usually less than 1 m thick. These dark green to black dykes are composed of plagioclase and/or pyroxene and biotite and contain varying amounts of orthoclase, quartz, amphibole, chlorite and olivine with minor apatite, titanite, zircon and magnetite.

7.3 Structure

There are several types of faults, which have been thought to localize the gold-copper-silver mineralization (Petersen, 1988). ‘Paleo-faults’ are north striking, vertically dipping faults that have been active throughout the mineral emplacement. They are thought to control the lamprophyre dykes. ‘Flat faults’ strike easterly and dip 15° to the north and may have reacted with the ‘Paleo-faults’ to create vertical conduits for the mineral emplacement. ‘Dislocation faults’ strike northeast and dip from 45° southeast to vertical and offset the ‘Paleo-faults’. The Willa Fault, which has a strike of 040° and dips vertically, has no apparent offset. Contact faults follow the contact of the heterolithic breccia and probably serve as conduits for gold-copper-silver mineralization, but are narrow.


Aylwin
Little
Daisy
Creek
1025 Level
1100 Level
Extent of drilling to date
849697
542206
(not part of Discovery Ventures option)
Willa Fault
Wild
5526000N
5525750N
5525500N
5525250N
473250E
473500E
473750E
474000E
0
150
metres
Topographic contour interval = 5m
NAD83 Zone 11
LEGEND
Fault
Heterolithic breccia
Feldspar porphyry
Quartz latite porphyry
Rossland Group
Geology after: CD Spence and RH Wong, 1995
To accompany a report by : W Ash, PEng. W Gilmour, PGeo. M Waldeggar, PGeo.
Base map after : Eagle Mapping 2014
Discovery Ventures Inc.
Willa Property
Slocan Mining Division, B.C.
Property Geology
Dwg. by: Discovery Consultants
Scale: 1:5,000
Date:
Figure: 7.2

7.4 Mineralization

The significant mineralization at the Willa deposit is gold-copper-silver (MINFILE 082FNW071). Minor molybdenite-bearing quartz veins occur in the quartz latite porphyry.

The gold-copper-silver mineralization is predominantly within the heterolithic breccia unit, although mineralization has been identified in the feldspar porphyry, the quartz latite porphyry and the Elsie volcanic rocks. The mineralization is controlled by extensive silica-rich micro-fractures, faults, shears and breccias, predominantly within the heterolithic breccia unit that formed a zone of weakness for the emplacement of the mineralization.

The predominant minerals are chalcopyrite ($CuFeS_2$), with varying amounts of pyrite (FeS_2) and pyrrhotite ($Fe_{1-x}S$). The sulphides tend to be encapsulated within sheets of silica. Propylitic alteration occurred during mineralization, as did zones of intense silicification and minor pyritization.

The quartz-molybdenite stockwork is weak but extensive in the quartz latite porphyry and volcanic rocks north and west of the heterolithic breccia. Molybdenite occurs within the quartz and along the quartz vein boundaries. Due to the low grade of the molybdenum and the relatively small size, no potentially economic mineralization is known.

The Au-Cu-Ag mineralization does not appear to be bound to any particular lithology; however, most of the mineralization has been observed within the breccia pipe, mostly in the heterolithic breccia but also in the feldspar porphyry. A minor amount of mineralization has been observed crossing over the pipe boundaries into the volcanic rocks. The mineralization has been observed to be somewhat bound laterally by intersecting lamprophyre dykes which may or may not be useful as an exploration tool.

Three zones of Au-Cu-Ag mineralization have been observed at the Willa deposit: The West Zone, North Zone (called the Main Zone in some historical reports), and the East Zone.

Figure 7.2 illustrates the extent of drilling and observed mineralization.

7.4.1 West Zone

The West Zone is almost entirely contained within the feldspar porphyry and heterolithic breccia units at the southwest margin within the breccia pipe. The West Zone is the largest of the three mineralized zones and most extensively explored. The zone strikes to the northwest, with the northwestern most tip trending north, following the boundary of the breccia pipe. The overall strike length is 200 m and mostly occurs within a vertical extent of 100 m. The zone is approximately 100 m wide and dips to the northeast.

The West Zone has been exposed on the 1025 Level (1000 Crosscut, 950 South Drift, 950 North Drift, 950 Crosscut and the 1025 Raise) that runs parallel to and within the mineralization at approximately midway through the vertical extent of the zone. It has been documented that the mineralized zone is cut off to the north by the Willa Fault and has been displaced to the south by crosscutting faults.

Drillhole data seem to indicate that mineralization continues beyond these faults and into other lithologies.

7.4.2 North Zone

The North Zone lies higher and to the north of the other zones. It appears to have been offset at several locations along its crude north-south strike. This zone is exposed on surface by Aylwin Creek. The Willa No. 1 and No. 2 adits were collared in the North Zone on either side of the creek. The exposed portion of the zone is stained with limonite and chlorite. The zone is continuous and has a strike length of approximately 175 m, a width of 50 m and a depth of 125 m. The zone is partially composed of heterolithic breccia as well as feldspar porphyry, quartz latite porphyry and volcanic rocks of the Elise. The offsets of the North Zone can be subdivided into North Zone and Upper North Zone.

7.4.3 East Zone

The East Zone is presently the smallest of the three zones. This may be due simply to the lack of drilling in this area. The majority of the zone is below the 1013 Decline, the 1013 Level and the 993 Decline. The East Zone is discontinuous and appears to be a series of sub-parallel veins striking west and northwest up to 80 m with widths up to 5 m. Mineralization extends 175 m downdip and is mostly located east of the contact with the breccia pipe and within the Elise. Further exploration is required to better define this zone. A mineral title (542206) owned by a third party abuts the eastern portion of the East Zone (Figure 7.2)

7.4.4 Other Showings

The Little Daisy (MINFILE 082FNW070) is a gold-silver polymetallic vein in granitic rocks. On the Property and/or within the Elise there are several polymetallic silver-lead-zinc±gold veins (Figure 3). The Silver Nugget (082FNW072) and Highland Light (082FNW075) are hosted in the Elise. The Mountain Scenery (082FNW074), Silver Band (082FNW073) and Midas Touch (082FNW269) are hosted in the Nelson Batholith. At the LH (082FNW212), gold, along with pyrite, pyrrhotite and arsenopyrite, occurs in silicified shear zones within the Elise.

7.5 Alteration

Alteration around the Property has been described in detail by R.H. Wong and C.D. Spence in "Copper-gold mineralization in the Willa breccia pipe, southeastern British Columbia", (1995):

Hydrothermal alteration evident in the area of the Willa Deposit is a consequence of three discrete, but successive intrusive events and two major episodes of mineralization. Overlap and over-printing of the various alteration assemblages has resulted in a complex zonation. The two earliest alteration assemblages, associated with molybdenum mineralization, consist of biotite-pyrite and quartz-pyrite-molybdenite. These are spatially associated with quartz latite porphyry. Following this and associated with intrusion of feldspar porphyry is a potassic assemblage of K-feldspar/biotite accompanied by up to 5% disseminated pyrite. Emplacement of heterolithic breccia was closely followed by pervasive calcium metasomatism resulting in a prograde calc-silicate alteration assemblage. It is believed that most, if not all, of the gold-copper-silver mineralization accompanied this alteration. Retrograde alteration of this calc-silicate assemblage has resulted in the formation of and over-printing by minerals such as epidote, actinolite, gypsum, quartz, calcite, and zeolites. Emplacement of the Nelson Batholith produced only minor propylitic effects in the Willa area.

7.5.1 Biotite-Pyrite Assemblage

Fine- to medium-grained black biotite accompanied by 2% to 5% disseminated and fracture-filling pyrite is locally preserved within mafic volcanic rocks adjacent to quartz latite porphyry. While more probably a product of contact metasomatism related to intrusion rather than a true hydrothermal alteration, it remains a recognizable assemblage where it is not over-printed by later hydrothermal events. The biotite also occurs as felted masses predominantly pseudomorphic after augite and may comprise up to 50% of the rock. Ubiquitous pyrite associated with this assemblage contributes to the large pyritic halo, which encloses the overall intrusive complex.

7.5.2 Quartz-Pyrite-Molybdenite Assemblage

Quartz-pyrite-molybdenite stockwork veins, sheeted veins, and pervasive flooded zones occur mainly in quartz latite porphyry but are also found within volcanic rocks adjacent to the quartz latite porphyry contact. Alteration around individual veins is best developed in the quartz latite porphyry where quartz-sericite-pyrite envelopes pass outward into zones of albitized plagioclase. Where the veins cut biotite-pyrite altered volcanic rocks, wall rock alteration is minimal, probably due to the dense, impermeable nature of theses rocks.

7.5.3 K-feldspar - Biotite Assemblage

A K-feldspar/biotite assemblage is spatially associated with the feldspar porphyry plug and locally is superimposed upon the first two alteration assemblages. Heather (1985) recognized a horizontal zonation within this assemblage with K-feldspar dominant in the centre of the plug and biotite more prominent on the periphery. K-feldspar dominant alteration is marked by partial to total replacement of groundmass and plagioclase phenocrysts to yield a hard, whitish-coloured rock with the original textures obscured. Biotite-dominant alteration is marked by the development of disseminated, fine-grained, purple biotite laths, which impart a distinct pinkish colouration to the rock. Biotite development is most intense near the margins of the feldspar porphyry in zones of monolithic breccia. In these zones, fine-grained biotite, albitized plagioclase, and pyrite form the matrix of the brecciated rock.

7.5.4 Prograde Calc-Silicate Assemblage

Following development of the heterolithic breccia pipe, prograde calc-silicate alteration resulting from calcium metasomatism associated with the gold-copper-silver mineralization event resulted in replacement of the breccia matrix, and formation of veins and fracture-fillings in crackle-fractured clasts and peripheral crackle zones. The alteration assemblage consists of various combinations of the following materials listed in order of decreasing abundance: pyroxene, amphibole, epidote, garnet, plagioclase, K-feldspar, quartz, anhydrite, sphene, and calcite.

7.5.5 Retrograde Assemblage

Retrograde alteration of earlier calc-silicate alteration minerals is widespread as crosscutting veinlets and replacements. Epidote is particularly prominent as an alteration product within the garnet-anhydrite and pyroxene zones. Veinlets of amphibole, pyroxene, calcite, and quartz are seen to cut all of the prograde assemblages. Fibrous clusters of zeolite locally replacing garnet and sparry gypsum replacing anhydrite are also considered to represent retrograde alteration as the hydrothermal system collapsed.

7.4.6 Late-stage Veinlets

Veinlets consisting of varying proportions of calcite, chlorite, quartz, and gypsum are seen to crosscut all rock types, including Nelson Batholith. They are especially common adjacent to late shear zones and may contain minor pyrite, hematite, or magnetite.

8.0 Deposit Types

The Willa deposit is a sub-volcanic breccia-hosted-type deposit (L01 – BC Mineral Deposit Profiles). This type of deposit represents a transition from porphyry copper to epithermal conditions with a blending of porphyry and epithermal characteristics. The porphyry units have intruded into the volcanic rocks and a pyrite, silica-rich mineralized stockwork breccia and closely-spaced sheeted veins with local massive to disseminated replacement zones have been imprinted on the sequence. The mineralization in these types of deposits is usually polymetallic and in the case of the Willa deposit contains significant gold, copper and silver.

Other deposit types on the Property are small in size compared to the Willa deposit and some may have been superimposed on the known mineralization. The primary focus of the historic exploration and development work on the Willa deposit is based on the sub-volcanic breccia-hosted mineralization. Future exploration will continue to focus on this type of deposit.

9.0 Exploration

Mineral exploration and development carried out before the acquisition of the Property by the Vendors is summarized in section **6.0 History**.

2012 A structural analysis of mineralized zones on Title 708162 was completed (Sookochoff, 2012) for the Vendors of the Property.

2012 Discovery Ventures commissioned a Technical Report on the Willa Deposit and the historical Mineral Resources prepared in 2005 were disclosed again (Ash and Makepeace, 2012).

2013 VLF-EM surveys were carried out on Title 843349 (Sookochoff, 2013) for the Vendors and Discovery Ventures.

2014 Met-Solve Laboratories ("Met-Solve") completed a gravity recovery gold test of mineralized core (from 2004 drilling), including SG measurements (see Section 13.4 for details). Also, Eagle Mapping Ltd completed an aerial LiDAR survey over the Property.

2014 In August and September, Discovery Ventures carried out a geological and geochemical study on mineral title 849697. The purpose of this program was to use scatterplots to plot various elements to assist the mapping of geology and alteration.

2014 From November 24 to December 3, 2014, Discovery Ventures retained the services of Genex Mining Company Ltd ("Genex"). Genex started physical work, including re-opening the portals, preparing equipment for underground track installation, and rehabilitating the underground. At the 1100 Level portal a 4-foot culvert was installed to establish ventilation to the underground workings. At the 1025 Level portal, 120 m^3 of overburden were removed and stored on the waste dump.

2015 An underground visit on Level 1025 by Ash on October 29, encountered a collapse blocking further access at 300 m. On the 1100 Level a similar collapse was at 100 m from the portal.

10.0 Drilling

Drilling at Willa comprises 593 holes totalling 56,312 m completed between 1965 and 2004. Note that all drill results are from previous owners and Discovery Ventures has not conducted any of its own drilling to date. The vast majority of the drilling was collared from the underground workings and was drilled on behalf of Northair between 1986 and 1988. The location of the core from prior to 2003 is unknown, although it has been reported that, due to improper core storage, there is no way to identify the core.

The majority of the drilling was conducted within a block measuring approximately 250 m by 50 m, and over a vertical range of 200 m. Almost 20,000 samples were collected in the various drilling programs. Areas of significant mineralization were originally drilled at approximately 25-m line spacing and infill-drilling across the majority of the mineralized zones reduced the spacing to 12.5 m

10.1 1965 to 1970 Drilling

1965 Cominco completed 297 m from 4 surface holes. Holes ranged in length from 14 to 113 m. The drill core size was AX. Work was directed by A.B. Mawer. No significant mineralization was intersected in any of the drilling. The name of the drilling company is unknown to the authors. It is not known is any downhole deviation tests were completed or if the drillhole locations were surveyed. No drill core remains to be inspected.

1969 AMAX completed 57 m from 5 surface holes, under the supervision of R.W. Phendler. Holes ranged in length from 102 to 168 m. Core recovery averaged 93%. Significant mineralization was intersected in one hole in what is now referred to as the North Zone. There is one significant sample from a 32 m sample interval; the sample collection details are unknown to the Report authors. The size of the drill core and the name of the drilling company are unknown to the authors. It is not known if any downhole deviation tests were completed or if the drillhole locations were surveyed. No drill core remains to be inspected.

1970 Western Mining and AMAX, under the supervision of R.W. Phendler, completed 1,241 m from 6 underground holes from underground workings that were close to surface. Holes ranged in length from 153 to 248 m. Core recovery averaged about 80%. No significant mineralization was intersected. The size of the drill core and the name of the drilling company are unknown to the authors. It is not known if any downhole deviation tests were completed or if the drillhole locations were surveyed. No drill core remains to be inspected.

10.2 1980 to 1984 Drilling

1980 Riocanex and BP completed 2,626 m from 8 surface holes, under the supervision of D.C. Durgin. Riocanex was the program operator. Holes ranged in length from 107 to 813 m. Drilling was conducted by Cameron McCutcheon using a Longyear 56A for long holes and a modified Longyear 38 for shorter holes. Drill core size was NQ except BQ in one hole beyond 594 m depth. Holes 80-1 and 80-2 were surveyed by using a Sperry Sun gyroscopic instrument; however, the database does not indicate a change in azimuth along the length of the boreholes. The other four holes were surveyed for dip only by acid bottle test. Dip tests were completed on each hole at intervals ranging from 50 m in the shortest hole to 250 m in the longest hole. In general 3 borehole dip tests were completed on each hole at intervals ranging from 50 m in the shortest hole to 250 m in the longest hole. Old and new drillhole locations were surveyed by Ray Johnston and Associates ("Johnston") of Nelson. No drill core remains to be inspected.

1981 Riocanex and BP completed 3,130 m from 13 surface holes. Holes ranged in length from 109 to 375 m. Riocanex was the program operator. Drilling was conducted by Cameron McCutcheon using Longyear 38 and 56A drills and recovering NQ core. Drillhole dip tests were determined by acid tests, at approximately every 50 m. All 1981 drill locations and all 1980 survey data were surveyed to a benchmark by Johnston. No drill core remains to be inspected.

1982 Riocanex and BP completed 1,300 m from 3 surface holes. Holes ranged in length from 208 to 566 m. Riocanex was the program operator. Drilling was conducted by D.W. Coates Enterprises and supervised by L. Haynes. Drill core was NQ, except in one hole was where it was reduced at depth to BQ. Hole DH82-24 was surveyed using a Sperry Sun gyroscopic instrument, with tests every 30 m, and indicating a change in dip and azimuth with depth. The other holes were surveyed for change in dip by acid tests, completed every 50 to 150 m. Drillhole locations were surveyed by Johnston. No drill core remains to be inspected.

1983 BP assumed management of the joint venture and completed 1,748 m from 6 surface holes. Holes ranged in length from 222 to 362 m. BP was the program operator. Drilling was conducted by Canadian Mine Services using a Super Drill. Drill core was NQ. All drillholes were surveyed with a Sperry Sun single-shot instrument used in conjunction with a magnetic susceptibility meter. Tests were completed every 20 to 60 m. Drillhole 83-30 was also surveyed with a Sperry Sun multi-shot gyroscopic instrument. Drillhole locations were surveyed by Johnston. No drill core remains to be inspected.

1984 Riocanex and BP completed 5,497 m from 17 surface holes. Holes ranged in length from 109 to 375 m. BP was the program operator. Drilling was conducted by F. Boisvenu Drilling using a 56A drill. Drill core size was HQ and NQ. All boreholes were surveyed with a Sperry Sun single-shot compass-type instrument used in conjunction with a magnetic susceptibility meter, with tests completed every 25 to 240 m, and indicating changes in azimuth and dip. Hole locations were surveyed by Johnston.

10.3 1985 to 1988 Drilling

1985 Northair became a partner in the joint venture and completed 1,274 m from 6 underground holes. The purpose of the drilling was to collect geotechnical information and no assay samples were collected. Drilling was carried out from 1025 level and holes ranged in length from 31 to 588 m. Drilling was conducted by F. Boisvenu Drilling using JKS a 300-U electric-hydraulic drill and a Boye BBU-s air drill. Drill core was BQ. It is not known if any downhole deviation tests were completed. The drillhole locations were surveyed. No drill core remains to be inspected.

1986 Northair completed 6,869 m from 56 underground holes. The purpose of the drilling was to explore the limits of mineralization in the West Zone and explore for mineralization at the south and eastern margins of the intrusive complex. Drilling was carried out from the 1025 level and holes ranged in length from 18 to 223 m. Drilling was conducted by Boisvenu Drilling using a JKS 300-U electric-hydraulic drill. Drill core was BQ. It is not known if any downhole deviation tests were completed. The drillhole locations were surveyed. Drilling was successful in establishing the limits of the West Zone and discovering mineralization in the East Zone. No drill core remains to be inspected.

1987 Northair completed 9,730 m from 189 underground holes in 1987 from the 1025 and 1013 levels and holes ranged in length from 11 to 192 m. The purpose of the underground drilling was to delineate the West and East Zones. The program was supervised by F.G. Hewett. The size of the drill core was both NQ and BQ. A high-grade gold intersection in 1983 (DDH 83-25) was confirmed by DDH 87-173 and DDH 87-174. The name of the drilling company is unknown to the authors. It is not known if any downhole deviation tests were completed. Although there are no supporting data, it is assumed that the drillhole locations were surveyed. No drill core remains to be inspected.

Northair also completed 1,698 m from 17 surface holes in 1987. Holes ranged in length from 58 to 135 m. The purpose of the surface drilling was to delineate the North Zone. The size of the drill core and the name of the drilling company are unknown to the authors. Borehole dip tests were generally completed twice per hole, ranging from 10 m to 130 m, and the database does not indicate a change in azimuth along the length of these holes. Several holes did not have any dip tests recorded. No drill core remains to be inspected.

1988 Northair completed 14,965 m from 212 underground holes from the 1100, 1025, 1000 and 913 levels and holes ranged in length from 14 to 253 m. Supervision was provided R.J. Beckett. Drilling focussed on the West and East Zones with few holes focussing on the North Zone. The size of the drill core was both NQ and BQ. The name of the drilling company is unknown to the authors. It is not known if any downhole deviation tests were completed. Although there are not supporting data, it is assumed that the drillhole locations were surveyed. No drill core remains to be inspected.

10.4 2004 Drilling

Several unexplored areas around the deposit were identified in a 3D digital model (Makepeace, 2003). This led to an underground drill program, carried out in 2004 (Gilmour, 2004), on six areas adjoining known areas of Au-Cu-Ag mineralization that had either not been drilled or were under-drilled. Bethlehem completed 5,284 m from 39 underground holes from the 1025 and 1013 levels and ranged in length from 32 to 321 m. Both areas were accessed via the 1025 Level portal along a tracked passage using an electric tram locomotive pulling a flat deck utility car. Rubber tired hand carts were used to move the core and equipment in areas where trackless mining had been conducted. An electric tugger was used to assist the movement of core and equipment on the decline.

Drilling was conducted by Advanced Drilling of Surrey, BC and supervised by William Gilmour of Discovery Consultants. Personnel from Discovery Consultants spotted the collar locations, conducted core sampling and geological and geotechnical logging. Holes were drilled using a B-10 drill for the shorter holes and a Mini Myte drill for longer holes. Both drills were electric-hydraulic powered and designed for underground operation. Drill core size was BTW.

Hole locations were surveyed using standard survey techniques (theodolite) by a survey crew under the supervision of R.C. Power, B.C.L.S. of Vernon, BC. The hole collars were surveyed into UTM NAD83 coordinates which had been established earlier in the year by Power and Associates from legal survey bench marks and carried into the Property and underground. Drillhole set-ups were surveyed for location, dip and azimuth. Foresights and back sights were marked for the drillers. The drillers used a taught string between them to correctly align the drill. The holes were marked with labelled wooden plugs after the drill moved to the next set-up. The markers were driven into the hole collars to make identification easier for the surveyors and to distinguish the new holes from the numerous old holes from earlier programs. After drilling was completed the locations, dips and azimuths of all holes were resurveyed.

A Sperry-Sun Magnetic Directional Single-Shot instrument was used to determine dips and azimuths at the end of most drillholes.

Photo 1: 2004 Drilling

This drilling in the peripheral areas of the Willa deposit was not successful in locating significant new zones of potentially economic Au-Cu-Ag mineralization. Drilling along the western edge of the West Zone better contributed to the delineation of the boundaries of the zone.

The drill core is stored in cross-stacked piles at the MAX mill in a partially enclosed storage shelter. Evidence that the boxes were stored outside at some point is obvious and numerous boxes are in poor condition. Aluminum core box labels remain legible as do sample interval markings on the core trays. Most of the remaining half-core mineralized intervals were sampled completely in 2014 for metallurgical studies.

11.0 Sample Preparation, Analyses and Security

11.1 1965 to 1988 Drill Programs

The following is a summary of procedures during this period. Quality Control / Quality Assurance (“QC/QA”) programs, as defined by CIM Exploration Best Practices, were not then implemented by the exploration operators, although the exploration was carried out by major and intermediate companies. The analytical laboratories would have had their own internal QC/QA programs, but there is no evidence that results were reported to the operators.

1965 Samples were collected as 1.52 m (5 feet) lengths and sampling was not continuous throughout the drillholes. The assay database largely consists of repetitive assay grades for gold and copper, no results for silver are available.

1969 Only results for gold and copper were available. The assay database has a mix of variable length sample intervals.

1970 Only three of the six holes were partially sampled for molybdenum and copper only.

1980 The upper portions of holes DH80-1 and DH80-2, and all core from the remaining holes, were split and sampled at 2 m intervals. Copper, gold and silver were assayed at Chemex, with gold by fire assay / gravimetric methods.

1981 All core was sampled at 2 m intervals. Copper, gold and silver were analysed at Chemex, with gold by fire assay / atomic absorption methods on one assay ton (29.2 g) sub-samples. Copper and silver were analysed by atomic absorption on 5 g subsamples.

1982 Core with visible mineralization was sampled at 2 m intervals. Copper, gold and silver were analysed at Chemex, with gold by fire assay / gravimetric methods.

1983 Mineralized sections of core were split and sampled at 2 m intervals. Copper, gold and silver were assayed at Vangeochem, North Vancouver, with gold by fire assay / atomic absorption methods on 20 to 30 g sub-samples.

1984 Mineralized sections of core were split and analysed at 2 m intervals. Well mineralized samples were sent to Vangeochem for gold, silver and copper assaying. Gold and silver were assayed by fire assay / gravimetric methods on 20 to 30 g subsamples. Vangeochem reportedly ran duplicate analysis on about 10% of the assays, and that unusually high gold values were checked. Many of the sections of core that were not well mineralized were geochemically analysed by ICP methods after aqua regia digestion of a 0.5 g subsample. This analysis was generally done on 2 m samples, at a maximum interval of 6 m.

1986 Mineralized sections of core were split and analysed at 1 m or 2 m intervals. Samples were sent to Vangeochem or Chemex for gold, silver and copper assaying. It is reported that for two holes the whole core was sent to the laboratory. For two other holes one half of the core was sent to Vangeochem and the other to Chemex. Gold and silver were assayed by fire assay / gravimetric methods on 20 to 30 g subsamples. Some duplicate gold and silver were reportedly completed by Vangeochem.

1987 Mineralized sections of core were split and analysed, with the vast majority of the samples being at 1-m intervals. Samples were sent to Vangeochem for gold and copper assaying. Gold was assayed by fire assay / gravimetric methods on one assay ton sub-samples. Assay sheets from Vangeochem show that duplicate gold and copper analysis was done every fifth sample. The nature of the duplicate samples is not known, but is likely to be duplicate pulp assays.

1988 Only Northair drill logs are available. The logs show gold, silver and copper assay results. Although in previous drill programs only visibly mineralized core was sampled and analysed. It appears that in this program all of the core was sampled and analysed.

11.2 2004 Drill Program

This section is a summary of the 2004 drill program, which followed certain procedures to ensure the data collected were compliant with CIM Exploration Best Practices. A sampling and analytical protocol for the underground drill program was developed by J.A. Chapman, PEng, of Bethlehem. Discovery Consultants of Vernon BC was responsible for ensuring the protocol was carried out. Acme Analytical Laboratories Ltd ("Acme") of Vancouver, BC, was the main laboratory. ALS Chemex ("Chemex") of North Vancouver, BC, handled duplicate core samples. These labs were independent of Orphan Boy and Bethlehem. Discovery Consultants was responsible for the spotting of drilholes, the geological and the geotechnical logging of the core, and the quality control aspects of the drill program.

The 2004 drill core was placed in marked boxes and transported off the Property to a core logging facility on Red Mountain Road. All aspects of quality control and quality assurance were monitored. The collection and security of core samples, the core recovery, the sample preparation procedures, the precision of the analytical results, the reproducibility of check samples, and the reproducibility of standards were all satisfactory. The precision of results is excellent. This is most likely due to the generally fine grained nature of the gold and demonstrates the reliability of drill core in determining grade. Section 13.4.3.1 of the Report mentions that coarse gold particles (>100 μm) were recovered during metallurgical tests.

The co-author Gilmour is of the opinion that the sample collection and preparation, sample analysis and sample security for the 2004 drill program is adequate and that the results can be relied upon.

11.2.1 Sample Collection and Preparation

The core was marked at 2.0-m intervals and tagged for splitting and sampling. The two-metre length was standard, except at the start or the end of each hole where occasionally sample length varied to accommodate casing.

A field blank sample was inserted into the sample sequence at sample numbers ending in 49 or 99 (about 100-metre intervals). The field blank material was obtained from an aplitic quartz-feldspar vein, exposed on the right rib of the 1025 Level a short distance in from the portal. The blank material generally contained no visible sulphides. The aplitic material was broken into core-sized pieces, placed into labelled plastic bags, and tagged similarly to the regular core samples.

Rock saws were set up outside the core facility to longitudinally split the core for sampling. A mechanical splitter was used in some zones if it became impractical to cut the harder lithologies using the rock saws. Half the core was shipped to Acme for analysis while the other half was placed back in the core boxes.

Duplicate core samples were taken at 25-sample intervals (about 50-metre) from samples with numbers ending in 00, 25, 50, and 75. Here the remaining half core was longitudinally split to form ¼ cores. One portion of the quartered core was sent to Chemex for check analysis and the remaining portion was kept in the core box. The split core was re-photographed and stored for future reference.

At Acme the samples were dried and all the material in the samples was crushed to >70% minus 2.0 mm. After thorough mixing the sample was riffle split to produce a 250 g sub-sample. The remaining 2.0 mm reject was placed in sealed plastic bags and stored at Acme for possible future analysis and for metallurgical testing. The 250 sub-sample was pulverized to >95% -100 Tyler mesh. After thoroughly mixing, 29.2 g (one assay ton) was split off for gold and silver analysis and a 1.0 g sub-sample was taken for analysis of base metals and other elements. The remaining pulp sample was placed in sealed plastic bags and stored at Acme for possible future analysis.

At Chemex a similar process of sample preparation and analysis was carried out on duplicate core samples.

11.2.2 Sample Analysis

Gold and silver values were determined by classical lead-collection fire assay on a 30 g subsample, followed by ICP emission spectrometry. Base metals and other elements were determined by aqua regia digestion on 1.0 g sample, followed by ICP emission spectrometry. Note that digestion is only partial for some minerals, especially silicates. However, digestion is excellent for base-metal sulphides.

11.2.3 Sample Security

The drillers transported the core to the portal at the end of each ten-hour shift using a combination of handcarts, tugger, and the electric locomotive. The core was received by personnel from Discovery and transported by pickup truck to a rented core logging facility located 4.3 km from the mine site.

The core logging facility was located within a gated yard, owned and occupied by a local resident. Three very vocal dogs guarded the yard and associated buildings. A large greenhouse with an industrial garage door was utilized as the core logging facility. Core logging benches were built and water hoses were obtained to adapt the greenhouse for core processing.

Sawn or mechanically split core samples were placed in plastic bags along with pre-numbered sample tags. The corresponding sample number was written on both sides of the sample bag. The sample bags were closed with plastic cable ties and packed into poly woven rice bags fitted with a security tag and a cable tie fastener. Partially filled rice bags were locked in a large aluminum box when unattended until sufficient samples were processed to completely fill and secure a shipping bag.

Regular sample shipments contained a total of 150 samples. Most of the samples were driven to Nakusp by Discovery Consultants personnel and were transferred to bonded shippers who trucked the samples to Acme in Vancouver.

The duplicate field samples were also similarly placed in plastic bags and locked in the security box until sufficient samples accumulated to fill a shipping bag. Duplicate samples were packed in poly woven rice bags fitted with security tags and shipped separately to Chemex via bonded shippers.

The split core was stored in temporary, roofed core racks beside the greenhouse. The core was subsequently removed from the temporary racks, carefully cross-stacked and shipped to a storage facility at Bethlehem's Goldstream mill site for permanent storage. The drill core has since been shipped and is stored in cross-stacked piles at the MAX mill in a partially enclosed storage shelter. Evidence that the boxes were stored outside at some point is obvious and numerous boxes are in poor condition. Aluminum core box labels remain legible as do sample interval markings on the core trays. Most of the remaining half-core mineralized intervals were sampled completely in 2014 for metallurgical studies.

11.2.4 Quality Control and Quality Assurance Program

11.2.4.1. Contamination

For the 2004 underground drill program, Discovery Consultants inserted a coarse blank sample every 50 samples, totalling 50. The blank material was obtained from an aplitic quartz-feldspar vein, was not drill core, and therefore was not blind to the laboratory. The purpose of these blanks was to check for contamination within the preparation (crushing, pulverizing) process.

Acme inserted a silica sand sample at the start of every sample batch, totalling 18. The purpose of these blanks was to check for any preparation contamination from the preceding batch. The results were monitored on a regular basis.

The results demonstrate no significant contamination during the sample preparation process.

11.2.4.2 Precision

Duplicate samples are prepared and analysed to measure precision. Precision is defined as the percent relative variation at the two standard deviation (95%) confidence level. In other words, a result should be within two standard deviations of the mean, 19 times out of 20. The higher the precision number the less precise the results. Precision varies with concentration – usually the lower the concentration the higher the precision number. The precision values are determined from Thompson-Howarth plots. The duplicate sample results pair the original Acme results with a Chemex check core sample, another sub-sample from the rejects, or another sub-sample from the pulps.

Precision is a measure of the error in the analytical results from a variety of sources: (1) core sampling; (2) sample preparation and sub sampling; and (3) analysis. The core duplicates measure the error in all three of these parameters; the reject duplicates, the error in preparation, sub sampling and analysis; and the pulp duplicates, mainly the error in analysis. The duplicates are inserted into the sample stream after the original sample. The homogeneity of the samples should increase with the process, from core sampling through to the analysis of sample pulps. The results of the duplicate samples were monitored on a regular basis.

The precision for the sample collection, preparation and analysis is excellent. This is most likely due to the fine grained nature of the gold. Wong and Spence (1995) reported that the average gold grain size is 10 μm (micrometres or microns).

Core duplicates

In total, 107 duplicate pairs of core samples were analysed. The pairs comprised ½-core samples analysed by Acme and ¼-core samples analysed by Chemex. This procedure will only give an estimate of the sampling precision, as a different sized sample was analysed by a different laboratory.

At the 95% confidence level the precision values indicate about a ± 2% error for 2.0 g/t Au values and about a ± 2.5% error for 0.3% Cu values. This is the total error for sampling, preparation and analysis. This error is probably higher than if two ½-core samples were analysed consecutively within the same laboratory.

Reject Duplicates

Acme systematically produced, about every 30 to 40 samples (89 pairs in total), another pulp sample from the saved reject (crushed) material.

At the 95% confidence level the precision values indicate about a ± 0.6% error for 2.0 g/t Au values and about a ± 0.7% error for 0.3% Cu values. This is the total error for preparation and analysis.

Pulp Duplicates

The laboratory systematically analysed, about every 30 to 40 samples (89 pairs in total), another pulp sample.

At the 95% confidence level the precision values indicate about a ± 0.9% error for 2.0 g/t Au values and about a ± 0.3% error for 0.3% Cu values. This is the error for analysis.

11.2.4.3 Accuracy

Acme inserted two standards into the sample stream about every 35 samples. One was a gold assay standard and other was the multi-element standard. The results were monitored on a regular basis.

From the laboratory standards, almost all of the Au values were within acceptable limits. The Cu standard results were consistent, with almost all ranging between 0.55 and 0.57%. The results of the standards demonstrated no significant problems with the accuracy of Au and Cu values.

11.2.5 Geotechnical Measurements

These measurements were taken to assist in any future resource calculations or mining plans. The core was logged and photographed using natural light in the greenhouse.

11.2.5.1 Rock Quality Designation (RQD) and Core Recovery

Before splitting, the RQD (rock quality designation) values – the percent of core pieces within an interval that are greater than 10 cm in length – were determined. Percent recovery values were also measured. The good recovery indicates that the analytical values are representative of the in situ rock.

11.2.5.2 Specify Gravity

Specific gravity values were determined in the field on stored ¼ core samples (every 25th sample). Determinations were done by weighing pieces of quartered core of convenient length first in air and then submerged in water.

Acme measured the specific gravity of reject material from core samples (also every 25th sample). The field values are consistent with the laboratory values, although about 1.5 to 2.8 % higher on average. A difference is expected as the laboratory values are for a split of crushed rock from a 2.0 m interval while the field determination is from a piece of core from the same interval.

Bulk density determinations were conducted on several samples by sealing the core in a thin layer of wax to prevent water from entering cracks, vugs or pores in the rock. The sealed core was weighed as was the field specific gravity test. No significant difference was detected between the bulk density and the specific gravity indicating that the core was solid in nature with low internal porosity.

12.0 Data Verification

The following section describes the verification procedures used by the Qualified Persons to verify the data used in this technical report for the purpose of Mineral Resource estimation.

In 2015, as part of preparing the Mineral Resource disclosed in this report, Mr. Waldegger selected 5% of the drillholes from each drilling campaign and cross checked all gold, copper and silver assay results against primary sources. Holes completed prior to 2004 did not have original assay certificates from the lab to verify therefore Mr. Waldegger cross checked the database against results transcribed on to the paper geological logs. The logs were legible and neatly organized into bound reports grouped by drill campaign. Holes completed in 2004 had assay certificates that were stored in a binder at Discovery Consultants office in Vernon, BC. Mr. Waldegger checked the database against these certificates. Only one discrepancy was observed (less than 0.01% error rate), where a gold value was recorded in the database at a higher grade (3.02 g/t vs 2.61 g/t) than in the drill log. Mr. Waldegger also checked density values against written reports, in some cases lab reports, and checked underground face and wall sample data against original assay lab certificates.

During a site visit in November 2015, Mr. Waldegger and Mr. Gilmour visited the main portal for the 1025 level. A GPS location could not be established using a hand held Garmin GPS 60 unit due to the steep terrain and satellite positions. The location of the portal is as described in previous documentation. They also inspected the 2004 drill core located at the MAX mill, 142 km north of the project. The drill core was observed to be stored in cross-stacked piles under a three-walled shelter. The aluminum labels on the boxes were still legible. The mineralized intervals from two holes were laid out to inspect but no mineralized core remained in the boxes. Numerous boxes were rotting and covered in mould. No drill core from previous drilling was available to inspect, and from personal communication with Gilmour and Ash, none of the core exists. Therefore no independent samples were collected for verification of tenor of mineralization.

In 2004, Discovery Consultants verified the 2004 data by checking the database against certificates received from the lab. Several channel samples were completed by Mr. Makepeace, PGeo, in the underground workings to confirm the tenor of mineralization from the West Zone.

Mr. Waldegger is of the opinion that the results recorded in the drillhole database fairly reflect the data recorded in the paper records and can be relied upon and used for Mineral Resource Estimation.

13.0 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Testwork Conducted to Date

Six programs of metallurgical tests have been conducted on the mineralogy of the Willa deposit to date:

- Lakefield Research ("Lakefield") of Lakefield, Ontario, in 1985 to 1987 on behalf of Northair: bench-scale testwork on West Zone high-grade
- G. Hawthorn in 1988 on behalf of Northair: bench-scale and pilot plant testwork on West Zone high-grade
- UBC 4th Year students in 1996, supervised by Dr. George Poling & Dr Rimas Pakalnis: bench-scale tests on lower grade material from the lower dump
- H. Winckers & Associates at PRA Labs ("PRA") of Richmond, BC, in 2003-2004 on behalf of Bethlehem: Bench-scale testwork on high-grade drift-wall chip samples from West Zone
- G&T Metallurgical Services ("G&T"), subsidiary of ALS Chemex, of Kamloops, BC, in 2014 on behalf of Discovery Ventures: testwork on lower grade material from lower dump
- Met-Solve of Langley, BC: Bench-scale gravity concentration testwork conducted on lower dump material in January 2014, and gravity and flotation tests conducted on 2004 drill core (intermediate-grade) in 2015

13.2 Early Metallurgical Testwork (1985 to 2004)

Metallurgical investigations on five Willa mineral composites were completed by Lakefield between the autumn of 1985 and the end of 1987. Pilot plant trials were conducted in June and July of 1988 under the direction of G. Hawthorn, who also performed a small number of bench scale tests on a Willa composite. The pilot plant was operated at less than optimal conditions. A small scale test program on two Willa 1025 Level mineral composites was conducted in 2003, to confirm earlier test conditions and results.

The metallurgical response of the Willa composites tested by Lakefield fell within a fairly narrow range and tended to improve the confidence level of the projections. However, the sample head grades were also very similar and tended to be taken from a high-grade section of the West Zone. From the results of the locked cycle tests (repetitive batch flotation tests), gold recoveries averaging 81% and copper recoveries averaging 93% to a concentrate grading 24% Cu were projected for mineral grading in the order of 7.0 g/t Au and 0.9% Cu. The 2003 PRA testwork indicated the potential for an incremental two percent increase in gold recovery in an auriferous pyrite concentrate, recovered from a rougher scavenger concentrate. Preliminary indications were that projections for gold recovery below head grades of 7.0 g/t Au should be calculated using a constant tail of 1.1 g/t Au (note that recent testwork suggests gold in tailings grade to average 0.79 g/t). The head grades of the composites are shown in the following table.

Table 13-1: Pre-2003 Metallurgical Composite Head Assays

Test Campaign	Composite	Head Assay				
		Au (g/t	Ag (g/t)	Cu (%)	S (%)	Fe (%)
Lakefield P.R. 1	Composite 2-85	6.1	14.0	0.97	6.82	8.30
Lakefield P.R. 2	Composite 2-85	6.1	14.0	0.97	6.82	8.30
Lakefield P.R. 3	Willa Bulk	4.1	13.2	0.76	5.48	7.51
	Willa Bulk 1	3.6	10.6	0.74	5.31	7.60
	Composite 3	7.0	12.9	0.91	6.00	7.78
Lakefield P.R. 4	Willa Bulk 1	3.6	10.6	0.74	5.31	7.60
	Willa High-Grade	5.9	8.3	0.70	6.10	7.50
	Composite 2-87	7.0	13.1	0.93	6.81	7.38
Hawthorn P.R. 1	Composite W-1	7.4	11.3	1.04	-	-
Pilot Plant	Lot 1	2.4	-	0.44	-	-
	Lot 2	3.1	-	0.50	-	-
	Lot 3	6.0	-	0.69	-	-
	Lot 4	9.1	-	1.02	-	-

Three mineral zones have been identified in the Willa gold-copper deposit: the West Zone, the North Zone, and the East Zone. The focus of previous and current testwork is the West Zone. Mineral from Willa West Zone is amenable to treatment by conventional comminution (particle size reduction), flotation, and dewatering processes, as indicated by the results of metallurgical testwork performed. The ratios between gold, copper and silver are generally similar between the three zones. No significant difference in the metallurgical characteristics is therefore expected. If a production decision is made, the West Zone will likely be the first zone to be mined since its development is most advanced and the grade tends to be higher than the average. Therefore during the first year of production sufficient time would be available for confirmatory metallurgical testwork to be conducted on the two other zones.

Project development plans include the use of the MAX mill, 135 km north of the Property. This modern, 500 tonnes per day ("tpd") flotation plant was operated as late as 2011 for ore of the MAX molybdenum mine. Minor improvements and alterations are required to facilitate processing of the Willa mineral.

In February 2003, a new metallurgical test program was initiated to confirm results of the previous work and to further optimize the mineral processing parameters. This work, conducted by PRA Labs ("PRA") of Richmond, BC, on two of five chip sample composites obtained from the walls of the 1025 Level drift and crosscut, was conducted at PRA. The analysis of the PRA samples is shown in the following table.

Table 13-2: PRA Metallurgical Composite Head Assays

Composite	Head Assay				
	Au (g/t)	Ag (g/t)	Cu (%)	S (%)	Fe (%)
A	11.6	14.6	1.09	-	8.8
B	3.3	10.1	0.77	-	13.4

The composite head grades are at the top and the bottom of the expected grade range.

The Willa gold-copper mineralization is hosted in a breccia pipe that occurs at the centre of a hypabyssal complex of quartz and feldspar porphyritic intrusions. Auriferous and argentiferous chalcopyrite, pyrite, and magnetite mineralization comprise three zones within and peripheral to the breccia pipe. All zones are associated with pervasive calc-silicate alteration, which has overprinted the earlier phyllic and potassic events.

Pyrite and chalcopyrite are the principal sulphides present in the West Zone and occur in varying proportions. Pyrrhotite and sphalerite generally grade less than 1%.

Gold occurs in native form as inclusions and micro-veinlets in pyrite and as grains along contacts between pyrite and either chalcopyrite or silicates. The average gold grain size is 10 μm. Silver values are associated with sphalerite, which commonly occurs as inclusions in chalcopyrite and pyrite (Wong and Spence, 1995).

The sulphides in the mineralization of the West Zone appear to be simple and clean with regard to potential penalty elements reporting to the concentrate.

The gold mineralization is very fine grained. Metallic gold assays of the Willa Bulk 1 and Willa High-Grade Composites showed that the +100M ("M", as in Tyler mesh size) fractions of these samples contained 16.5% and 87.6% of the gold in 4.3% and 7.8% of the weight, respectively. This may be an indication that the gold mineralogy in the West Zone is variable.

During the various campaigns in the 1980s, approximately 36 bench gravity and flotation tests were conducted on Willa composites. A few cyanidation leach tests were done on gravity and flotation tailings. Most of these tests included a single-gravity gold pre-concentration step at a coarse grind followed by further grinding and rougher and cleaner flotation. Most of this work was conducted at Lakefield, with the exception of a pilot trial on bulk composites conducted, in 1988, in the Silver Ridge Resources "Standard" plant near Slocan, BC. Control of the processing conditions during this test was poor and the results are therefore not considered indicative.

Gravity pre-concentration was included in many of the Lakefield tests. The process involved tabling the feed at a relatively coarse grind, on a 1/8 size Wilfley table, followed by concentrate cleaning on a Mozley separator. The test results are summarized below.

Table 13-3: Lakefield Pre-Concentration Gravity Test Results

Comp. No.	Test No.	Grind (% -200M)	Head Au (g/t)	Gravity Concentrate (% Wt.)	Au (g/t)	Au Rec. (%)
2-85	1	37	7.23	0.30	942	35.2
2-85	7	62	6.28	0.30	692	33.0
2.85	10	37	7.23	0.18	1291	32.0
3	18	55	7.02	0.16	1467	34.1
HG	26	55	6.35	0.05	3436	26.1
2-87	26A	55	8.37	0.01	12786	11.7
Flotation Tailings						
3	16	91	1.40	0.01	3821	33.6
2-87	27	92	1.49	0.30	9	2.0

Tests 16 and 27 were conducted on the flotation tailings of Tests 15 and 21. The results indicate that a significant percentage of free gold was recovered in the first test, whereas the gravity gold recovery in the second test was negligible. A likely explanation for the difference in results is that Test 15 did not include gravity pre-concentration.

It should be noted that it is current practice to use centrifugal gravity concentrators, which are more effective for fine gold recovery. A proper appraisal of gravity gold recovery potential would require testwork using this equipment.

The results of the best tests on each composite using Test 4 conditions are shown in the following table.

Table 13-4: Lakefield Flotation Test Results

Comp. No.	Test No.	Type	Head Assay		Gr. Rec.	Cu Conc.		Recovery G + F		Roc Tail
			Au (g/t)	Cu (%)	Au (%)	Au (g/t)	Cu (%)	Au (%)	Cu (%)	Au (g/t)
Comp 2-85	4	B	7.23	0.89	35.2	81	19.5	83	92.3	0.89
Willa Bulk	14	B	4.67	0.72	-	135	23.9	80	93.9	0.63
Comp 3	18	LCT	7.02	0.87	34.1	80	20.8	83	93.5	1.13
High-Grade	21A	B	5.55	0.71	-	146	23.4	73	92.3	1.20
Comp 2-87	26A	B	8.37	0.95	11.1	157	21.0	79.7	93.8	1.45
PP Comp 4	GFL-2	B	6.24	0.66	22.4	83	14.3	67.2	73.7	2.11
Comp W-1	GF-4	B	6.13	0.93	15.6	127	26.5	78.8	86.6	1.19

*B=batch test; LCT=locked cycle test

During the course of testwork two locked cycle tests were performed, one on Composite 3 and one on Willa High-Grade. The results of the locked cycle tests (LCT) and batch tests (B) on the same samples are compared in the following table. Unfortunately, the batch tests did not include gravity pre-concentration.

Table 13-5: Lakefield Locked Cycle and Corresponding Batch Results

Comp. No.	Test No.	Grind (%) (-200M)	Gravity Conc.		Conc. Assays			Recovery			Tails
			Au (g/t)	Rec. (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)
C-3	18-LCT	76.7	1467	34	80	240	20.8	83.0	85.0	94	1.13
C-3	15-B	90.5	-	-	135	323	26.6	66.0	74.4	94	1.10
HG	21C-LCT	95.7	3436	26	125	231	27.4	78.5	82.5	95	1.32
HG	21A-B	97.7	-	-	153	n.a.	25.4	64.3	n.a.	84	1.20

The gold recoveries shown are the overall recoveries of gravity plus flotation where applicable. The locked cycle tests produced higher gold recoveries than the batch tests. The tailings gold assays were similar for the batch and locked cycle tests, indicating that the cleaner recoveries in the locked cycle tests were high. High copper recoveries were obtained in all tests. The results of Tests 15 and 18 suggest that the flotation conditions of Test 18 were not fully optimized.

A cyanidation test was done on the scavenger tailings of Test 21CL containing 1.4 g/t Au. The flotation test included gravity pre-concentration. A gold extraction of 70% was obtained; the leach residue assayed 0.35 g/t Au. The flotation tails included both pyrite and gangue minerals. This indicates that the gold lost in the flotation tailings of this test is most likely in the form of coated or middling particles, although no distinction was made between the gold content of the pyrite versus the gangue minerals.

Pilot plant tests were run on bulk samples of Willa mineral in the summer of 1988. In total, 494 t of material from four lots were processed. The grade of the first two lots was less than 4 g/t Au. Lots 3 and 4 assayed 6.0 and 9.1 g/t Au, respectively. The results of the pilot plant work are very difficult to assess because:

- The pilot plant was fairly archaic
- There was an absence of a gravity circuit
- Relatively inexperienced operators
- Timely assays were not available
- Inadequate process control
- Very few pilot plant runs had complete copper/gold metal balances

The data generated in the pilot plant trials were considered by Winckers as being of questionable value, and the results were generally inferior to those obtained in bench scale tests.

The Rod Mill Work Index and Abrasion Index were determined at Lakefield on a 50/50 composite of Composites A and E. The results are shown in the table below:

Table 13-6: Grindability Tests – 1025 Level Samples

Composite	BM WI (kWh/t)	RM WI (kWh/t)	Grind P80 (micron)	Abrasion Index
Comp S-A	14.1		62.5	
Comp S-E	14.5		62.8	
Waste R-4	14.0		59.9	
Waste R-6	14.2		60.5	
Comp A+E		14.1		0.7231

The mineral and potential waste dilution composites have about the same hardness and can be classified as medium range.

The abrasion index ("AI") is a measure of how abrasive the mineral is, resulting in ball and liner wear in the mill. Most ores fall into the range of 0.0 to 1.0. AIs above 1.0 are normally associated with chert. The AI for the Willa mineral, at 0.7231, falls into a high-medium range.

The locked cycle test of rougher scavenger and cleaner scavenger tails samples were submitted to Vizon SciTec, of Vancouver, BC, for acid-base accounting. The rougher scavenger tailings had a small positive net neutralization potential, while this parameter was highly negative for the other samples.

Results and review of the past testwork indicate the following:

- Head grades of the composites tested are within range of the estimated mineable mineral grade (based on a cut-off grade of 3.5 g/t Au).
- Willa mineral composites tested responded well to conventional flotation conditions, yielding consistently high copper recoveries and somewhat lower and more variable gold recoveries.
- The composites tested may not represent the full range of the gold-copper mineralization within the deposit.
- Willa mineral is grind sensitive. Composites tested at Lakefield showed stronger grind sensitivity than those tested at PRA. The best gold and copper recoveries are obtained at grinds finer than 90% -200M. The optimum grind is estimated to be 80% passing 55 µm.

- The Bond Ball Mill Work Index ("WI") data of a small number of composites tested indicate that the mineral and waste rock dilution materials are relatively soft. This was confirmed by the results of the comparative WI study of five composites tested at Lakefield. A WI of 14.2 kWh/t can be used future studies.
- Gravity pre-concentration was used in a large percentage of the batch tests and in all locked cycle tests. Winckers postulated that the net benefit of gravity pre-concentration on the gold recovery cannot be determined from the types of tests performed and the results obtained to that time.
- Rougher flotation mass recoveries in the order of 15% are required to optimize gold recovery. Incremental gold recovery in an auriferous pyrite concentrate produced from a rougher scavenger concentrate may be possible.
- The Willa mineral exhibits a relatively strong concentrate grade recovery relationship, with both gold and copper recoveries dropping off steeply above concentrate grades of 25% Cu. Further process optimization using locked cycle tests may improve this condition.
- Gold and copper recoveries obtained in the testwork ranged between 80 to 84% and 90 to 94%, respectively, at a concentrate grade of 20 to 24% Cu.
- The Willa High-Grade Composite and Composite 2-87 appear to be less amenable to flotation than the other composites.
- A multi-element analysis of the locked cycle test concentrate produced from Composite A+E indicates that the concentrate is clean with low levels of deleterious elements.

13.3 Summary of University of British Columbia Testwork (1996)

Bench-scale metallurgical testwork was conducted in 1996 by the 4th year Metallurgical Engineering students (supervised by Dr. George Poling, Head of the Metallurgical Department) on a sample recovered from the 1025 Level dump, grading 4.0 g/t Au and 0.74% Cu. The testwork included gravity separation using jigs, cyanidation, copper flotation, pyrite flotation, thickening and copper concentrate thickening.

Gravity Separation: The work, using a 1/8-size Wilfley concentrating table, was conducted on -48M +200M product and recovered less than 15% of the gold, indicating that the majority of the gold is finer than 200M. However, it is the coarse gold (>200M particles) that is important since the method of sample assaying by smelters neglects highs and therefore little if any coarse gold content is paid for from the smelter.

Cyanidation: Whole-ore cyanidation testwork by bottle rolls recovered 82% of the gold on a 90% <200M feed.

Copper Flotation: Lime consumption was 0.3 kg/t to maintain the required 10.5 pH for flotation. A projection of 85 to 90% recovery estimated for a 20% Cu content product.

Pyrite Flotation: A pyrite concentrate of 95 to 98% recovery of pyrite were projected, leaving no sulphides in the tailings.

Thickening: Thickening tests were done on the final tails (after recovery of the copper and pyrite concentrate), indicating the requirement for a thickener surface area of 1.03 m^2/t/day. As there are no plans to incorporate thickening to the final tails prior to discharge, this data are not applicable.

Filtering: Due to the lack of a plate-and-frame filter press, a vacuum filter was used. As this does not produce the required low moisture content of 8%, the filtration data are not applicable.

13.4 Recent Metallurgical Testwork (2014 to 2015)

Since late 2013, two batteries of metallurgical testwork have been undertaken on behalf of Discovery Ventures. The first was conducted on dump samples taken in November 2013 with testwork carried out by G&T Labs (a subsidiary of ALS) and by Met-Solve Labs of Langley, on behalf of Discovery Ventures in early 2014.

The second battery, which is on-going, incorporated 2004 diamond drill core, recovered in December, 2014.

First Battery of Metallurgical Testwork:

As the Willa mine was inaccessible in 2013, the samples for the first battery of metallurgical tests on Willa mineral were personally taken by co-author Ash in mid-November 2013 from the Lower Mineral dump. These consisted of development muck recovered from tunnelling on the West Zone mineralization (circa 1986/87). Having been subjected to the environment for at least 25 years, these showed some signs of oxidation. The sampling consisted of 14 sample bags of "closed-eye" mineralized muck totalling 181 kg, taken at approximately equal intervals along the face of the dump. On completion of sampling, Ash personally transported them back to Vancouver and delivered them to ALS Chemex in North Vancouver the following day. The North Vancouver facility of ALS Chemex then sent the samples to the ALS metallurgical test lab in Kamloops where they were dried, weighed, crushed to <3.35 mm (1/4") and combined to form a bulk composite upon which metallurgical scoping tests would be performed.

G&T Labs (Kamloops) was chosen to conduct the flotation and ABA estimation. The purposes of G&T test-work were to assess:

- The grade and elements contained in the head sample
- The percentage recovery of sulphide minerals from the bulk composite sample
- Assess the final tails for Acid Base Accounting (ABA) and potentially-deleterious substances
- The projected potential for using high-gravity process for recovery of a high-grade gold concentrate.

Met-Solve Laboratories of Langley, BC was chosen for the gravity testwork because Met-Solve had:

- A wide background of experience with high-gravity separation
- A lab-size model of the Falcon concentrator expected to be used in production
- Experience in prediction of the potential gravity recovery using the Falcon in the plant grinding circuit, through calculations using a mathematical model.

Therefore, co-author Ash had G&T ship approximately 70 kg of the crushed composite to Met-Solve in late December, 2013, for gravity recovery tests.

ALS Chemex Testwork:
The samples were composited and stage-crushed to <6M and rotary split into 2-kg charges where they were stored under nitrogen at -10°C until needed in the test program.

The head grade assayed 3.77 g Au/t, 0.38% Cu, 7 g Ag/t, 5.4% Fe and 4.99% S. A 2-kg sample of this was ground to a nominal size of 80% <75um for flotation. The scoping test resulted in flotation recoveries of 77.7% for gold, 93.4% for copper, 86% for silver, and 92.6% for sulphur.

The potentially-deleterious elements in the rougher tails (equivalent to the final tails) included:

Table 13.7: Analysis of Final Tails

Element	ppm	Element	ppm
Ag	0.94	Mo	62.6
As	4.3	Pb	8.1
Au	0.84	S	0.84
Cd	0.18	Sb	1.7
Cu	607	Te	0.95
Hg	0.02	U	0.7
		Zn	59

The Net Neutralization Potential = -11 t $CaCo_3$/kt

For the flotation testwork, the mineral was ground to a nominal 75 µm, and resulted in recoveries of 77.7% for Au, 93.4% for Cu, and 92.6% for S. Note that no pre-concentration by gravity methods was employed.

Met-Solve Testwork (January, 2014):
Earlier gravity tests (1985-88, 2005), conducted by Lakefield and PRA employed standard gravity processes (concentrating tables), which resulted in recoveries of 30 to 32% of the gold from a head grade of 7.0 g Au/t, prior to flotation. No coarse gold was noted by either company but both recommended further research on gravity separation. Therefore, as Met-Solve employed more efficient, high-gravity laboratory equipment (Falcon L40 Concentrator) for gold recovery, co-author Ash had G&T ship 25 kg of crushed sample to Met-Solve for gravity testwork (the second test set).

The purpose of the first Met-Solve test (January, 2014) was to determine the efficiency of gravity-recoverable gold (GRG) present using high-gravity methods (Falcon Concentrator), and to predict the potential gravity recovery in the plant grinding circuit through calculations using a mathematical model.

Met-Solve received the 70 kg of crushed bulk sample composite from G&T in late December, 2013. Ten-kg sub-samples were extracted from the bulk composite by homogenization of the received sample.

A 3-stage, 10-kg GRG test was conducted using a laboratory-scale Falcon L40 centrifugal concentrator, at a final grind of 80%<67 μm. Met-Solve employed more efficient, high-gravity laboratory equipment (Falcon Concentrator) for gold recovery. Multi-element ICP scans were conducted on a sub-sample of the head sample and gravity tails. Gravity circuit mathematical modeling based on the GRG results was used to predict the potential gravity recovery in the plant grinding circuit.

The calculated head was 3.71 g Au/t and 0.35% Cu. Note that while the calculated head grade was 3.71 g/t Au, the assayed head grade was 5.46 g/t Au, suggesting that while the great majority of the gold particles were very fine (<20 μm, some coarse gold (>73 μm or +200M) existed in the sample. In the test, the 10-kg sub-samples were screened at 1,295 μm (12M) and tested as follows:

- The rougher concentrate of the 1st pass through the Falcon contained 12.5% of the head Au. Hand Panning reduced this to 9.3%. The pan tails, containing 3.2% of the head Au, was combined with the 1st pass Falcon tails, re-ground to 80% passing 352 μm or 40M, and charged to the Falcon for a 2nd pass.
- The rougher concentrate of the 2nd pass through the Falcon yielded 13.1% of the head grade. Hand panning yielded 8.3% of the head gold grade.
- The 2nd-pass tails, containing 4.8% of the head Au, were combined with the 2nd pass Falcon tails, re-ground to 87 % <67 μm (-225M), and put through the Falcon for the 3rd pass.
- The rougher concentrate of the 3rd pass through the Falcon yielded 27% of the head grade. Hand panning of this yielded 13.5% of the head grade.

The total combined rougher Falcon Concentrate therefore contained 2.3% of the gold in 3.88% of the head weight, grading 51.92 g Au/t. The combined panned (cleaned) Falcon concentrate was 31.9% of the gold in 0.46% of the head weight, and graded 251 g Au/t. From the above, Met-Solve projected an in-plant recovery of 39% of the gold in a 200% recirculating load in the grinding circuit, or 45.5% in a 375% recirculating load. The projection was based on the extensive research of the fundamental understanding of gravity gold recovery within grinding circuits, much of which was conducted by the late Dr. Andre Laplant of McGill University, plus expected throughput data provided by Rob Robson, (metallurgical advisor for Discovery Ventures)

2nd Battery of Metallurgical Testwork:

Between the 1980s and 2005, the metallurgical testwork was conducted mainly on relatively high grade samples taken from tunnelling in a restricted area of the West Zone. Subsequent to the publication of the 2014 PEA, there was some discussion by metallurgical consultants that the mineral grade of the sample upon which the historical (1985 to 2005) testwork was conducted was higher than the average grade. Since in general, lower head grades result in lower recoveries, it was considered advisable to conduct metallurgical tests on material grading similarly to the low and average grades within the mineral resource estimate. The company contracted to do the present and on-going metallurgical testwork is Met-Solve.

In the historical testwork, only one test was conducted (by PRA) to recover a pyrite concentrate. The rest left sulphides in the tailings. It is expected that overall, the mineral of the Willa mine will average somewhere between four and ten percent pyrite.

As it was suspected that the pyrite might contain significant (potentially-recoverable) gold, as part of the current testwork, recovery of a pyrite concentrate was conducted, as well as recovery of a gravity gold concentrate and a copper/gold concentrate.

For the current metallurgical test program, co-author Ash personally recovered, bagged and labelled 93 selected samples of split core from the existing 2004 core boxes (located at the MAX mill) in mid-December, 2014, and personally delivered them immediately to Met-Solve in Langley. The selected samples were designed to cover a wider range of the West Zone than the samples tested between 1985 and 2004, and cover a wider range of values (1.5 to over 35 g/t Au). The samples are therefore expected to be more representative of the mineral encountered in mining.

At Met-Solve each of the 93 samples was dried, weighed (average >3 kg each), and initially crushed to <6600 µm. Each was then homogenized, from which 200 g was removed, pulverized and tested for specific gravity, using kerosene as the emersant. The remaining portions of the samples were then broken into three categories; Sub-Grade ($C 50 to $114/t), Intermediate-Grade ($115 to $174/t), High-Grade ($175/t or greater). The dollar values for the samples were based on the following assumptions:

- World price silver: $US 20/troy oz.
- World price gold: $US 1,200/toyr oz.
- World price copper: $US 3.00/lb.
- $US in $C: $1.10
- Milling recovery gold: 82%
- Milling recovery copper: 90%
- Milling recovery silver: 82%
- Concentrate shipment to Vancouver: $C 82/dry t
- Copper grade shipped: 24%
- Transport to Asia & smelting cost: $C 220/dry t
- Smelter deduction on gold: $US 10/troy oz
- Smelter deduction on copper: 1 metric unit
- Copper refining charge: $US 0.22/lb Cu
- Smelter deduction on silver: $US 1.00/troy oz
- Smelter payment gold: 97.5% of world price
- Smelter payment on silver: 90% of world price
- NSR to royalty holder: 2.5%

The in-situ prices used for each of the three metals were therefore assumed at:

- Gold: $C 38.81/gram
- Copper: $C 45.14/percent

- Silver: $C 0.48/gram

The grades, dollar values and specific gravities for each grade category are as follows:

Table 13.8: Summary of Sample Head Grades

Test Series	Value Category	Sample Quantity	Total Wt (kg)	Average Au g/t	Average Cu %	Average Ag g/t	Average C$ Value	Average SG
OX300	Sub Grade	28	89	1.98	0.39	6.3	$ 98	2.72
OX200	Inermediate Grad	29	92	3.40	0.60	8.6	$ 164	2.78
OX100	High Grade	36	114	9.69	1.27	27.9	$ 448	2.82

Assays based on 2004 assaying conducted by Acme

Purposes of the Tests were to determine:

- A practical bulk specific gravity calculation of the Mineral Resources
- The practical optimum grind for liberation of gold and copper
- The recovery of gold in a high-gravity circuit
- The overall recovery of gold and copper by flotation, with various reagents in the various categories of mineral
- The recovery of a pyrite concentrate, determine whether it is auriferous and if so, whether it contains sufficient value to warrant further treatment for gold recovery
- The approximate Acid Base Accounting (ABA) of the final tails, assess possible metallic contaminants,
- The toxicity of the final effluent on water-borne insects.

To date, testwork has been limited to the Intermediate-Grade (the OX200 series). This category was chosen as this is believed to best represent the average grade in the mine plan.

13.4.1 Bulk Mineral Resource Specific Gravity

Based on the results of the specific gravity ("SG") testwork and assuming that 40% of the mineral mined would be high-grade, 45% intermediate-grade and 15% sub-grade (dilution), a more appropriate bulk SG of 2.78 to 2.80 has been estimated.

13.4.2 Optimum Grind

The optimum liberation particle size for gold and copper were conducted, including 80% <75, 68, 53, and 45 µm, and confirmed tests conducted by Lakefield and PRA that the optimum liberation particle size is 50 to 55 µm. This was further confirmed by Process Mineralogical Consulting, which conducted analysis on replicate 30 mm polished block sections by the Tescan Integrated Mineral Analyser (TIMA).

13.4.3 Gold Recovery by High-Gravity Methods

Assay methods conducted by smelters on concentrates neglect coarse gold assays so coarse gold content is not paid for by smelters. Recovery of coarse gold in the mill allows an overall increase in gold recovery. The percentage increase in overall recovery due to the recovery of the coarse gold is generally dependent upon the head grade but cannot be practically defined.

Met-Solve used a lab-scale (Falcon L-40) concentrator for gravity concentration of gold. The unit recovered 51.9, 52.3, 54.6% and 66.4% of the gold in the four tests. Note that for calculations, the 66.4% recovery in Test OX204 was omitted due to a coarse gold particle(s) occurring in the panned concentrate, resulting in too great a variance between the assay head and calculated head.

Significant coarse gold particles (>100 μm) were noted in the gravity concentrate, thus confirming that:

- Coarse gold does exist,
- It is advisable to incorporate a Falcon Concentrator in the circuit for coarse gold recovery
- It is advisable to install an on-site smelter for recovery of coarse gold in the form of bullion.

The high-gravity unit chosen for production is the Falcon SB 750. However, in order to maintain a favourable water balance, Met-Solve recommends treating only 50% of the cyclone underflow in production, which will reduce the gravity recovery to approximately 37%, with a grade greater than 500 g/t Au. The coarsest gold (>100 μm) would be recovered by tabling and then smelted on site while the table tails (containing only fine gold) would be combined with the copper concentrate with a grade still expected to exceed 500 g/t Au.

The coarse gold recovery cannot be specifically defined. Therefore, in order to err on the conservative side, while the gravity and refining circuits are included in the capital cost estimate, for accounting purposes it is assumed that the coarse gold as well as the fine, reports to the copper concentrate.

13.4.4 Metallurgical Recoveries

Four tests have been conducted on the Intermediate-Grade composite sample. Test No. OX204 is the culmination of the test results and includes estimates of the reagents required and dosages. A summary of the current testwork conducted to date by Met-Solve is shown in the following tables:

Table 13.9: Head, Concentrate and Tails Assays

Test No.	Calc Head		Panned Gravity Conc	Copper Conc		Pyrite Conc		Tailings	
	Au (g/t)	Cu (%)	Au (g/t)	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)
OX201	4.49	0.59	483	13.8	6.16	3.47	0.56	0.99	0.05
OX202	4.45	0.59	783	10.9	4.1	2.29	0.33	0.78	0.04
OX203	3.95	0.56	447	30.6	20.8	4.75	0.56	0.6	0.06
OX204	6.79	0.57	571	27.4	12.5	3.22	0.3	0.71	0.03

Table 13.10: Percent Recoveries

Test #	Gravity Conc Au (g/t)	Copper Flot Conc Au (g/t)	Copper Flot Conc Cu (%)	Pyrte Flot Conc Au (g/t)	Pyrte Flot Conc Cu (%)	Tailings Au (g/t)	Tailings Cu (%)	Overall Au (%)
OX201	51.9	26	88.1	3.4	4.2	18.7	7.1	81.3
OX202	52.3	31.6	92.6	1.9	2.1	14.2	5.2	85.8
OX203	54.6	21.9	84.3	10.9	9.6	12.6	6.1	87.1
OX204	70.2	16.5	66.8	4.5	1.4	8.7	2.6	91.3

Note that for simplification, silver has been excluded as it is a minor by-product, contributing no more than 3 % of the value of the recovered minerals.

In Test OX203, the rougher copper concentrate assayed 20% Cu and the clearer concentrate assayed 33% Cu, but resulted in high copper reporting to the pyrite concentrate (9.6%). While cleaning may recover most of the copper from the pyrite, there is no certainty that the same results will occur in production. As the copper in the pyrite concentrate may be unrecoverable, the reagent addition was optimized in Test OX204 to produce a 12.5% rougher concentrate and 25.6% cleaner concentrate, thereby reducing the copper reporting to the pyrite to just 1.4%. This confirms earlier work by Winckers indicating that copper concentrates grading greater than 25.5% result in an inordinate loss of copper. Therefore, omitting the copper recovery results of Test OX203, and assuming that all the copper in the pyrite concentrate is unrecoverable, the average overall recovery of copper is 90.7%.

A gold recovery of 84% was achieved. Gold reporting to final tails varied from 0.6 to 1.0 g/t. Recovery of silver was restricted to Test No. OX204, with overall recovery of 92.4%, based on a calculated head of 11.2 g Ag/t. The silver associated with the pyrite concentrate is 7.1% and may or may not be recoverable.

Based on the first three tests, an average overall gold recovery of 84.7% was achieved. However, as a significant portion of the gold was found to report to the pyrite concentrate, it was necessary to perform cyanide bottle rolls on the pyrite concentrate to determine the percentage of gold which could not be recovered from the pyrite concentrate. The bottle-roll test was conducted on a composite of all the pyrite concentrates recovered in the four tests, and this resulted in a gold recovery of 80.2% of the gold from the pyrite in eight hours. Thus, if recovery by cyanidation were to be used for recovery of the gold from the pyrite concentrate, the overall gold recovery would be reduced by 0.8%, bringing the average overall potential gold recovery down from 84.7% down to 83.9%. Note that while the eight-hour bottle rolls recovery was 80.2%, cyanidation for 24 hours recovered 83.8% of the gold and at 48 hours, 84.3%. However, increasing the cyanide gestation time to 24 or 48 hours would result in increased capital and operating cost which is not warranted, considering the marginal recovery increase.

Gold reporting to final tails varied from 0.60 to 0.99 g/t. Recovery of silver was restricted to Test No. OX204, with overall recovery at 92.9% based upon a calculated head grade of 11.2 g/t. However, due to the lack of confirming evidence in the other three tests (in which silver recovery was not assessed), and the possibility that the silver associated with the pyrite concentrate (7.1%) may not all be economically recoverable, the silver recovery for the Preliminary Economic Assessment is based on locked cycle tests conducted 1988 by Lakefield, at 82%.

13.4.5 Pyrite Concentrate

The metallurgical testwork has shown the pyrite concentrate to be auriferous, with grades averaging 3.4 g/t Au, and ranging from 2.25 to 4.75 g/t Au. Several alternatives exist for treatment of the auriferous pyrite concentrate. It may be shipped for further treatment abroad, or may be treated to recover the gold in Canada, leaving the barren pyrite residue to be encapsulated and therefore, unavailable for oxidation or production of acid. For the short term, it will be stored as a concentrate in a separate lined storage pond while investigations are being conducted on how to recover the best value on the product. Further research on each of these treatment alternatives will be conducted during the production phase.

13.4.6 Environmental Assessment

The quality of the flotation tailings (equivalent to final tails) was evaluated on the bases of Acid Rock Drainage (ABA) and contents of the deleterious substances.

Table 13.11: Analysis of Supernatant

Item	OX201	OX202	OX203	OX204	Avg
Total Sulfur (%)	0.39	0.25	0.28	0.37	**0.32**
S as sulfate (%)	0.14	0.15	0.12	0.2	**0.15**
S as Sulfide (%)	0.25	0.1	0.16	0.17	**0.17**
Acid Potential (kg $CaCO_3$/t)	8	3.2	5.1	5.4	**5.44**
Total Carbon (%)	0.19	0.19	0.19	0.19	**0.19**
Neutralization Potetial (kg $CaCO_3$	15.2	15.2	15.2	15.2	**15.2**
Net Neut. Pot. (kg $CaCO_3$/t)	7.2	12	10.1	9.8	**9.76**
NP/AP	**1.9**	**4.75**	**2.97**	**2.79**	**3.1**

The ideal NP/AP ratio for tailings (ratio of neutralizing potential to acid-producing potential) is 4.0 or higher. The average NP/AP of the four flotation tails samples is 3.1, which is considered close to ideal since few metals mines produce tailings with as low sulphide content. Furthermore, the tailings will be stored under water in the tailings pond and finally, capping the tailings at the end of production with an impervious veneer cover such as clay remains a potential. Therefore, the tailings can be considered benign.

The tailings pond supernatant will be discharged into a trout-bearing creek. The toxicity is determined by a 96-hour LC50 for trout and *daphnia magna*. The *daphnia magna* (water lice), which are a major food source for trout fingerlings, are far more susceptible to toxicity than trout. The latest test was conducted on the solution from the OX200 series, which showed a *daphnia magna* pass at an effluent concentration of 78%. On-going testing will be conducted on other collectors. Settling tests will also be conducted with various flocculants and coagulants to ensure that the tailings will settle out in the tailings pond, in an effort to ensure that the final effluent reporting to the environment is non-toxic to aquatic life.

13.5 Milling Reagents

Table 13.12: Projections for Milling Reagent Usage

Reagent	Use	g/t	$C/t
SPRI-1004	Oily-Cu/Au Collector	106	$0.69
Flex 51 PM	Pyrite/Au Collector	125	1.02
Sodium Metabisulphite($Na_2S_2O_5$)	ORP Control (Pyrite Depression)	230	0.25
Copper Sulphate ($CuSO_4$)	Pyrite Activation	209	0.82
Diethylenetriamine (DETA)	Pyrite Depressant	40	0.31
MIBC (Aerofroth 70)	Frother (Alcohol)	20	0.08
Lime (CaO)	pH Control (Pyrite Depressant)	1040	0.47
Sulphuric Acid (H_2S0_4)	pH Control (Pyrite Activator, tails)	520	0.37
		TOTAL	**$4.01**

13.6 Summary of Recent Test Results

The results of the testwork conducted to date on the Intermediate grade mineral indicate the following:

- The arithmetic average overall recovery of the gold in the three tests (OX201, OX202, OX203) for the intermediate-grade category was 85.1%, approximately 3% higher than the average recovery achieved in the historic (1984 to 2005) testwork conducted on higher-grade mineral. Note that in test OX204, the calculated head was anomalously-high (7.07 g Au/t) (undoubtedly due to the nugget effect of coarse gold) and this test was consequently omitted from the calculation.
- The higher overall gold recovery than that attained in historic testwork is expected to be due to the recovery of the auriferous pyrite concentrate and lower content of gold in the final tailings.
- The gold grade of the pyrite concentrate produced from the four tests varied from 2.29 to 4.75 g Au/t and averaged 3.4 g Au/t. Further testwork is warranted to determine if it is possible to increase the gold grade in the pyrite concentrate further by re-cleaning, without adversely affecting the overall recovery or the quality of the final tailings.
- For the cash flow analysis, the gold recovery used is 84%.
- The gold recovered by the Falcon consistently exceeded 50% for the intermediate-grade mineral (as compared to 30 to 33% found historically in standard-gravity concentration testwork). However, in recovering the coarse gold fraction by tabling, following high-gravity concentration, the greatest percentage of the gold will be too fine to recover by tabling so the resulting table tails will be combined with the copper concentrate.
- The arithmetic average of the copper recovery in the four tests is 94.1%, which compares well with the best results of the historic testwork. However, excluding anomalous test OX203 (in which 9.6 % of the copper reported to the pyrite concentrate), an average of 3.6% of the copper may be expected to report to the pyrite concentrate. Additional cleaning of the pyrite may or

may not recover additional copper but whether or not the additional cleaning warrants the cost of additional flotation remains to be determined.

- While the single test showed a 92.4% for silver, inadequate testwork has been conducted on silver and therefore the recovery of silver will be based on the historical estimate of 82%.
- The cyanidation test was conducted on the combination pyrite concentrates from the four tests. The cyanidation recovery from the pyrite concentrate, with an average grade of 3.22 g Au/t, resulted in a recovery of 77.4% in 4 hours, 80.2% in 8 hours and 84.4% in 48 hours.
- Due to the high gold content (>20 g/t Au in copper concentrate) the ore buyer will accept concentrates grading as low as 20% Cu, without penalty. The recent test results indicate that re-cleaner copper concentrate particles are essentially free of contaminants and grades as high as 33% Cu may be achieved. However, an increase in copper content above 25.5% (Winckers' report and confirmed in Test OX203) will result in higher copper reporting to the pyrite concentrate, which may not be recoverable. In consequence, a grade of 24% Cu is used for the cash flow analysis.

The recoveries stated above may not necessarily be achieved for the entire deposit. Furthermore, locked cycle tests were not conducted, and while these normally result in marginally higher recoveries, there exists a potential for lower overall recoveries.

14.0 Mineral Resource Estimates

14.1 Key Assumptions/Basis of Estimate

Mineral Resources on the Willa deposit were estimated from technical data collected up to the 27 of July, 2004. The estimate was completed by Mr. Michael Waldegger, PGeo, of MFW Geoscience Inc ("MFW").

Gold, copper, and silver grades from drillhole samples formed the basis of the Mineral Resource estimate. All drillholes were completed by historic operators and drillholes completed prior to 1980 were not considered for this estimate.

MFW assumed Willa would be mined using conventional underground methods only.

The geological model and domain constraints used in the Mineral Resource estimate were prepared using Leapfrog Geo TM Version 2.2.1 (Leapfrog) supplied by Aranz Geo, and the interpolation of block grades was carried out using GEOVIA GEMS^{TM} Version 6.7.1 ("GEMS") supplied by Dassault Systems. Spatial analysis of drill data was carried out using Sage2001 software provided by Issaks & Co.

14.2 Drillhole Database

MFW received drillhole data from Discovery Consultants. The data were provided in MS Excel worksheets, and included collar locations in NAD 83, borehole deviation surveys, sample assays for gold, copper and silver, lithology logs, geotechnical measurements, and specific gravity determinations.

MFW imported the data into Leapfrog and upon 3D inspection of the 593 holes, which comprise the drillhole database, 26 holes were excluded from the database. Errors in the database were identified and corrected (overlapping and zero length intervals, negative and non-numeric data, and duplicate samples). The final database was imported into GEMS.

Of the 567 holes used for modelling, 528 were drilled prior to NI 43-101 coming into effect and the remaining 39 were completed in 2004. All samples were of half diamond drill core at 2 m sample lengths and some at 1 m lengths. Samples were analysed for copper, gold, and silver; however, the database has numerous zero values which were assumed to mean not assayed or less than detection limit. Samples with zero grade for all three elements were deleted from the database, and where a zero grade for one to two elements was recorded while the third element was greater than zero, the zeros were deleted. All missing intervals or blanks were replaced in the software's compositing routines with very low values (half the detection limit).

Out of a total of 19,362 samples from drillholes used in the estimate, 82% had a value for gold, 99% for copper, and only 68% for silver. Throughout the mineralized intersections, however, gold and copper were sampled with very few missing intervals, and silver results for 80% of the drillhole intersections. Outside of the mineralized zones it was common for either every 5th sample interval to be analysed or not at all.

14.3 Exploratory Data Analysis

Statistical analysis of the Naïve database was completed to characterize the grade and density data used in the Mineral Resource estimate. All analyses were completed on the final accepted database and excluded the holes that were removed.

14.3.1 Sample Grade

Histograms, probability plots, box and whisker plots, and scatter plots, along with summary statistics, were used to analyse the raw assay data. These tools were useful in characterising grade distributions per rock type.

Table 14.1 presents a comparison of drilled length per logged lithology and metal contained in each lithological category. Each assay interval was coded from the geology log and the gold equivalent ("Au Eq") grade was multiplied by the assay length and summed by lithological category. The table demonstrates that 98% of the metal is contained in lithologies coded as heterolithic breccia, feldspar porphyry, volcanic rocks, and quartz latite porphyry.

Table 14.1 Lithological Comparison in Metal Content

Rock Type	Intersected Length (m)	% of total length	% of total metal
Heterolithic Breccia	18,936.0	37.1%	49.7%
Volcanic Rocks	14,904.8	29.2%	13.7%
Feldspar Porphyry	9,909.1	19.4%	30.8%
Quartz Latite Porphyry	5,058.4	9.9%	3.6%
Overburden	856.6	1.7%	0.0%
Lamprophyre Dyke	825.4	1.6%	0.3%
Shear Fault Gouge	208.7	0.4%	0.4%
Nelson Granite	131.8	0.3%	0.0%
-	131.5	0.3%	0.3%
(blank)	55.5	0.1%	1.1%
White Feldspar Porphyry	38.6	0.1%	0.0%
Quartz Vein	10.5	0.0%	0.0%
Pegmatite	7.4	0.0%	0.0%

Table 14.2 Summary Statistics of Assay Samples

	Length (m)	Au (g/t)	Cu (%)	Ag (g/t)
Valid Cases	19362	15932	19170	13238
Mean	1.87	1.54	0.273	4.2
Variance	0.13	20.94	0.280	389.2
Std. Deviation	0.36	4.58	0.53	19.73
Variation Coefficient	0.2	3.0	1.9	4.7
Minimum	0.1	0.001	0.002	0.1
25th Percentile	2	0.17	0.05	0.8
Median	2	0.48	0.12	2
75th Percentile	2	1.58	0.33	4.4
Maximum	4	346.77	47	1876.5

Scatter plots and Pearson Correlation tables (Table 14.3) were prepared and a positive correlation was observed between each element; however, the correlation coefficients were below 0.5, which means there is quite a bit of scatter. Although the scatter still remains, better correlation is demonstrated when comparing capped values.

Table 14.3 Pearson Correlation Table of Assay Samples (Entire Dataset)

	Au	Cu	Ag
Au	1.00	0.41	0.18
Cu	0.41	1.00	0.28
Ag	0.18	0.28	1.00

	CAPAu	CAPCu	CAPAg
CAPAu	1.00	0.65	0.57
CAPCu	0.65	1.00	0.77
CAPAg	0.57	0.77	1.00

14.3.2 Density

In total, 220 sample intervals were tested for bulk density. It was observed that the density of significantly mineralized samples was higher than those that were not mineralized therefore a background density was established per rock type based on test results of samples under 1 g/t Au Eq (Table 14.4).

Table 14.4 Mean Density per Rock Type Above and Below 1 g/t Au Eq

Rock Type	Mean Density (< 1 g/t)	Mean Density (>1 g/t)
Fp	2.67 (n=1)	2.82 (n=17)
Hbx	2.71 (n=36)	2.80 (n=97)
L	2.84 (n=1)	(n=0)
Qlp	2.63 (n=22)	2.73 (n=1)
V	2.81 (n=39)	2.9 (n=4)
Entire sample	2.73 (n=99)	2.81 (n=119)

Within the heterolithic breccia, a trend of increasing density with increasing sample grade was observed (Figure 14.1). A similar trend was observed for samples of feldspar porphyry.

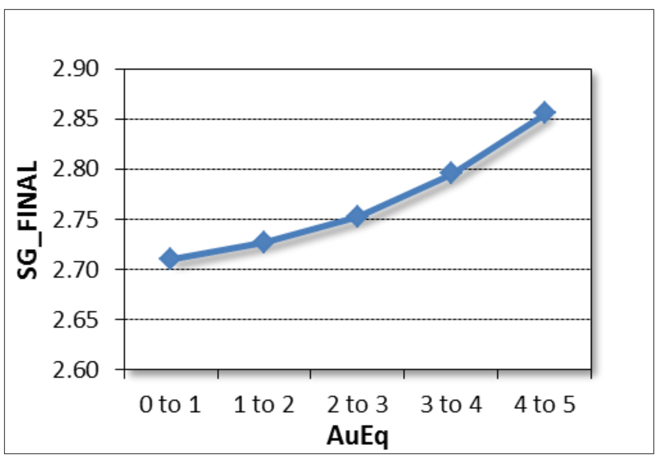


Figure 14.1 Trend of Density with Sample Grade

The mean density was determined by Au Eq grade bins on samples of heterolithic breccia and feldspar porphyry only and is presented in Table 14.5. These values were used to code the block model as described later in this section of the Report.

Table 14.5 Mean Density per Grade Bin for Hbx and Fp

Bin of Au Eq	Hbx Mean Density	Fp Mean Density
0-1	2.71 (n=36)	2.67 (n=1)
1-2	2.73 (n=11)	2.62 (n=1)
2-3	2.75 (n=20)	2.69 (n=2)
3-4	2.80 (n=19)	2.75 (n=3)
4-5	2.86 (n=12)	(n=0)
5-6	2.78 (n=4)	2.91 (n=3)
6-7	2.75 (n=5)	2.91 (n=3)
7-8	2.86 (n=26)	2.85 (n=5)

During the 2004 drilling campaign a second method to determine sample density was used. Discrete samples of drill core were tested at site and systematically returned higher density results than those from the analysis conducted by Acme on assay reject material from the same sample intervals. It is unclear as to why a systematic high bias was observed and since each of the three other methods tested assay sample reject material, MFW rejected the site results from the density modelling.

14.4 Geological Model

MFW completed wireframe modelling of geological units based on drillhole logs using Leapfrog. The 3D model covers most of the extent of the drilling data, excluding only the deep information from a very small number of holes. The 3D model was guided by the description of the Willa geological model presented in **Section 8** of this report.

Using the intrusion modelling tools in Leapfrog, a package of volcanic rocks (V and Qlp) was modeled as the oldest units with the 3D model extents. These units were intruded by group of rocks defined by feldspar porphyry and heterolithic breccia. This intrusion was subsequently refined to model the feldspar porphyry intrusion as a separate unit from the heterolithic breccia. Finally, using the vein modelling tools in Leapfrog, a suite of cross-cutting lamprophyre dykes and the Willa fault were modelled. Figure 14.2 presents a plan view section of the geological model at 1025 m above sea level.

The above described model was then clipped using the erosional surface modelling tool to model the bottom of overburden. In areas of poor drillhole coverage and in mountainous terrain, overburden surface modelling can produce undesired results, with the surface crossing over the topographic surface or producing unrealistically thick overburden volumes. To prevent this in the Willa 3D model, the overburden surface was produced from an offset topographic surface by 1 m and the hard drillhole data. The topographic surface was modelled based on points from DEM grids prepared from a Lidar survey.

Two underground hand drawn maps were used as verification. The maps were drawn in 1987 and are considered valid, however 40% of the drilling was completed after that time in 1988 and 2004. In general the 3D model agrees with the underground map; however, the maps did not attempt to depict the lamprophyre dykes.

In general there are few holes that present conflicting geological logs when compared to their neighbouring holes. There appears however to be numerous conflicts in logging between the rock types Hbx and Fp in the southern portion of the west zone. Some effort in re-coding the intersections based on the descriptions in the drillhole logs could improve the confidence in the geological model in this area.

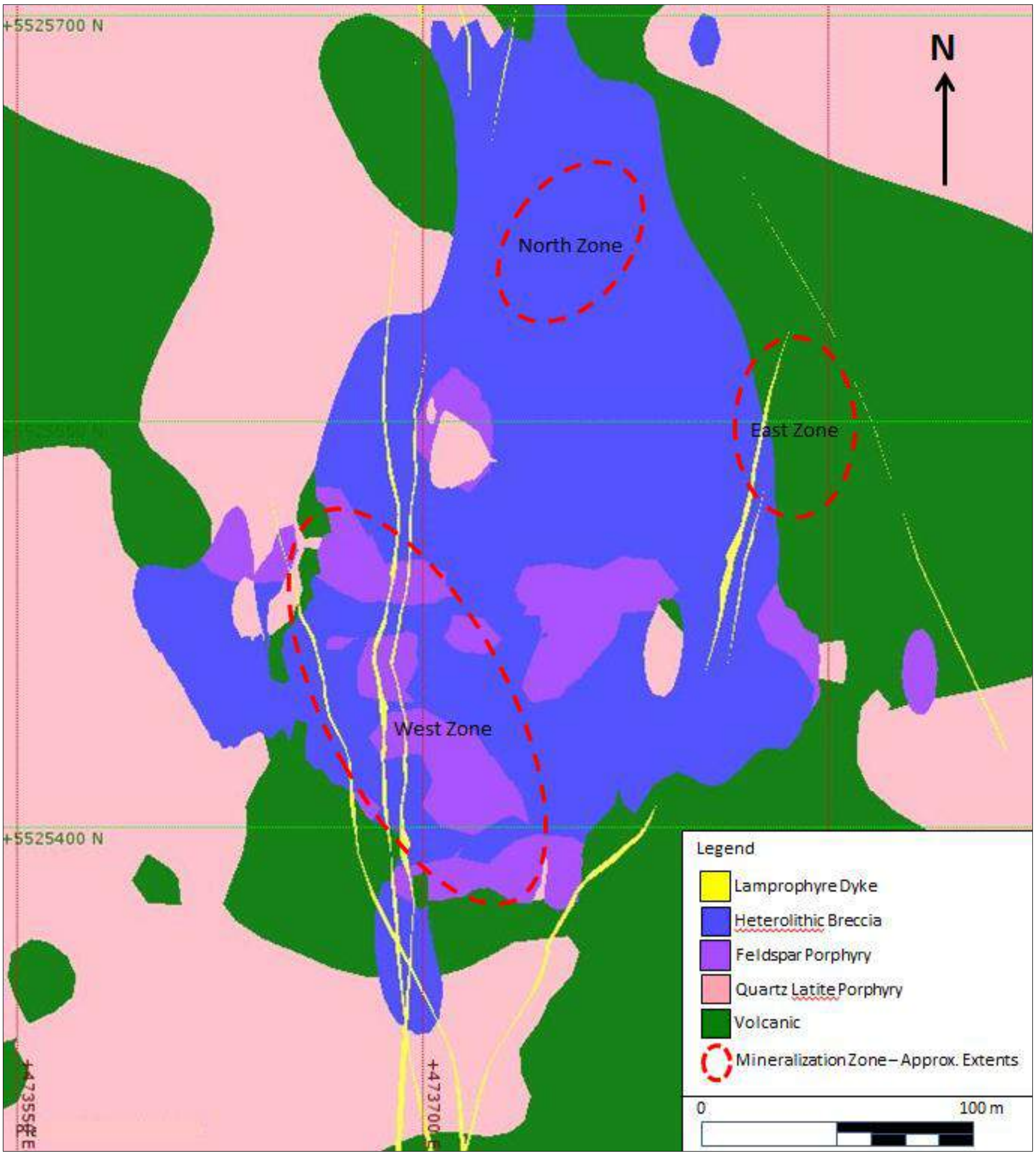


Figure 14.2 Plan View Section of Geological Model at 1025 masl

Source: Leapfrog Geological Model, Drawn by M. Waldegger, Date: October 2015

14.5 Metal Equivalences

MFW used a recoverable metal value per tonne ("RMV") to model the mineralized zones used to constrain the grade estimate. A gold equivalent grade was also calculated and used as the basis of a cut-off grade to report the mineral resources. Metal prices and metallurgical recoveries are presented in Table 14.6. The RMV used for wireframe modelling was guided by the three year average metal prices and the Au Eq used to report the Mineral Resources were based on the one year average metal prices. The metallurgical recoveries were based on testwork completed by Met-Solve, under the supervision of a metallurgist.

Table 14.6 Metal Prices and Recoveries

Commodity	Recovery	Three Year Average Metal Price ($US)	One Year Average Metal Price ($US)
Gold	82%	$1400 per oz.	$1200 per oz.
Copper	90%	$3.15 per lb.	$2.75 per lb.
Silver	82%	$20.00 per oz.	$16.50 per oz.

The following simplified formula was used to calculate the recoverable metal value:

$$RMV\ (\frac{\$}{t}) = (Au\ Grade\ x\ 36.9091) + (Cu\ Grade\ x\ 62.5010) + (Ag\ Grade\ x\ 0.5273)$$

The following simplified formula was used to calculate the equivalent grade:

$$Au\ Eq\ (\frac{g}{t}) = (Au\ Grade\ x\ 0.82) + (Cu\ Grade\ x\ 1.0024) + (Ag\ Grade\ x\ 0.0113)$$

14.6 Mineralization Model

MFW completed wireframe modelling of the mineralized zones based on drillhole assay data using Leapfrog to separate out potentially economic material from the surrounding rock mass.

Based on communications with Mr. Ash (who indicated that the cost per tonne to mine at Willa was about $C 115/t) MFW used a $US 100/t RMV cut-off grade when modelling the mineralization using Leapfrog's intrusion modelling method. Drillhole samples are first coded as either mineralized or not, based on the cut-off grade, and then a surface is modelled to enclose the segments coded as mineralized. Fine adjustments were made to the surface by applying local "structural trends" that guided the resulting surface shapes as opposed to using one fixed trend or no trend (isotropic). A final modification to the drillhole database was a 2 m extension at zero grade in 11 holes that ended in significant mineralization. This prevented the modelling algorithms from over estimating the volume, which can be a significant concern if a hole which terminates in significant mineralization occurs in an area of limited drilling.

Three regional zones of mineralization were observed: the West Zone, the North Zone, and the East Zone.

The West Zone is the largest of the three mineralized zones and most extensively explored. The modelled zone is almost entirely contained within the feldspar porphyry and heterolithic breccia units at the southwest margin of the intrusion. The zone was observed to trend to the northwest, with the northwestern most tip trending north. The overall strike length is 200 m and mostly occurs within a vertical extent of 100 m. The zone dips to the northeast.

The North Zone is located in the northern portion of the intrusive body and is proximal to the surface. The North Zone is modelled as a plunging semi-cylindrical shaped mineralized body stretching 120 m downdip with a diameter of approximately 20 m. Some near surface mineralization was also interpreted to form a tabular body striking 50 m to the north east, extending 60 m steeply downdip to the southeast with a width of 5 m. MFW compared the underground face and wall sampling completed in 1987 to the modelled mineralized zone and found good agreement.

The East Zone is modelled as a series of sub-parallel veins striking west or northwest up to 80 m with widths up to 5 m. Mineralization spans 175 m downdip and mostly is located east of the contact with the intrusive rocks and within the volcanic rocks. The mineralization is cut off by a claim that is not owned by the company.

The mineralized zones were reviewed in 3D (Figure 14.3) and in plan and cross-section to visually ensure the wireframes captured the mineralized samples. The model captured most of the assays above the cut-off grade; however, some samples assayed above the cut-off grade were excluded from the model on the basis of lacking continuity of mineralization. The mineralized shell captures assays below the cut-off grade in areas where the mineralization exists but at lower grades. Internal waste zones were modelled out where multiple holes demonstrated continuity.

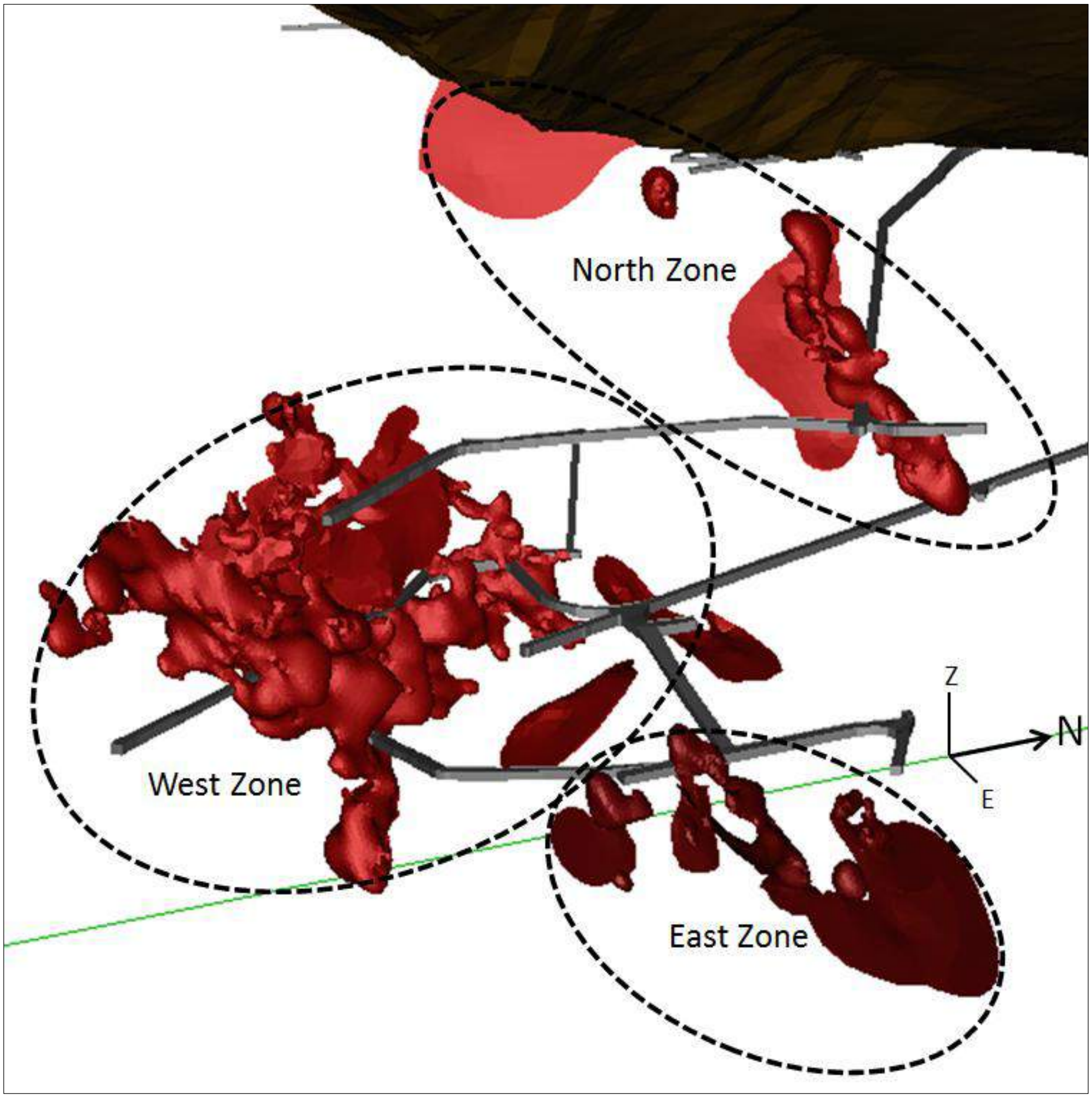


Figure 14.3 3D View of Mineralized Zones from the Southeast

14.7 Analysis of Grouped Data

Statistical analysis of the sample data within and outside of the modelled mineralized zones used for resource estimation was completed to characterize the data used in the resource model.

14.7.1 Sample Grade

MFW used a variety of graphical plots including histograms, probability plots, box and whisker plots, scatter plots, and contact plots, along with summary statistics, to analyse the raw assay data within and outside of the mineralized domains. These tools were useful in characterising grade distributions per mineralized zone, evaluating the nature of the domain boundaries with respect to change in grade, and in identifying high-grade outliers.

A total of 3,216 sample intervals are contained within the wireframes representing the mineralized zones and 16,146 sample intervals are outside of the zones. Table 14.7 and Table 14.8 present the summary statistics inside and outside of the domains. Only 80% of the samples within the wireframes were assayed for silver while nearly all of the samples were assayed for gold and copper.

Table 14-7 Summary Statistics for Samples within Mineralized Zones

	Au (g/t)	Cu (%)	Ag (g/t)	Sample Length (m)
Valid Cases	3212	3213	2241	3216
Mean	5.37	0.89	13.2	1.7
Variance	82.87	1.09	2123.9	0.25
Std. Deviation	9.1	1.04	46.1	0.5
Variation Coefficient	1.7	1.2	3.5	0.3
Minimum	0.05	0.01	0.1	0.1
25th Percentile	2.47	0.49	5.5	1
Median	3.57	0.70	8.9	2
75th Percentile	5.69	1.04	14.7	2
Maximum	346.77	47	1876.5	3.3

Table 14.8 Summary Statistics for Samples outside of Mineralized Zones

	Au (g/t)	Cu (%)	Ag (g/t)	Sample Length (m)
Valid Cases	12720	15957	10997	16146
Mean	0.57	0.15	2.4	1.9
Variance	0.66	0.03	15.8	0.1
Std. Deviation	0.82	0.16	4.0	0.3
Variation Coefficient	1.4	1.1	1.7	0.2
Minimum	0.001	0.002	0.1	0.2
25th Percentile	0.12	0.04	0.7	2
Median	0.29	0.09	1.4	2
75th Percentile	0.82	0.21	3.0	2
Maximum	26.64	2.6	252.3	4

To assess bias between the major drilling campaigns of 1986 to 1988 (90% of data), MFW compared the grade distributions of the sample data on probability plots by drilling campaign for the West Zone. The assay data for each of the three drilling campaigns exhibited similar grade distributions for gold demonstrating that no bias between drilling campaigns was observed. There was not sufficient numbers or representative spatial coverage from the other campaigns to complete a proper comparison, and since the other campaigns only represent 10% of the sample data, MFW did not attempt to establish if a bias was present in any one of those campaigns. MFW therefore incorporated all of the drilling data into the database for use in resource estimation.

MFW also observed grade distributions to be very similar between each lithological unit within the West Zone demonstrating that the mineralization was not restricted to any one lithology within the zone based on geological logs. Since the sampling was not broken by geological contacts, MFW was not able to demonstrate statistically that the lamprophyre dykes were barren of mineralization; however, those samples with significant contributions from the dykes were observed to be lower in grade than neighbouring samples. Therefore it is assumed that the dykes are barren.

Between the three mineralized zones, the gold grade distributions were very similar; however, the North Zone presents a slightly lower copper grade compared to the West and East Zones. The mean copper grade for the North Zone is 0.7% while for the West and East Zones the mean copper grade is 0.9%. Mean silver grades are less than 15 g/t in all Zones.

Contact evaluation plots were prepared for gold, copper and silver to analyse the nature of the change in grade across the West Zone domain boundary. In each case the contact was interpreted to be sharp and these observations support the use of hard boundaries during grade estimation.

14.7.2 Core Recovery

Core recovery data from the 2004 drilling were available for review. Previous reports on earlier crilling commented on the good quality of drill core and high recoveries but Discovery does not have any of the raw core recovery data from these drilling campaigns.

The average core recovery for the 2004 drilling was 99.3% and only 10 intervals had recoveries less than 50%. Given that there were very few drill runs with poor recovery, and only recovery information for a small proportion of the drill database, MFW did not treat samples differently according to core recovery during the estimate of grade.

14.8 Treatment of High-Grade Outliers

MFW analysed histograms and probability plots to assess the distributions for high-grade outliers. All distributions were right screwed with long high-grade tails. MFW also analysed metal content per decile and centile (Parrish analysis) to determine how much metal was being contributed by each bin. When too much of the metal in the deposit is attributed to just a few high-grade samples, capping can be warranted.

MFW capped raw assays as tabled below, prior to compositing. Sample data were analysed per mineralized zone and accordingly only the West Zone was determined to require capping. The effect of capping was low with less than 3.5% of the metal being cut as a result of capping 0.5% of the West Zone samples. The impact of capping on the resource model is discussed in Section 14.12.

Table 14.9 Capping Levels for West Zone

	Au(g/t)	Cu(%)	Ag(g/t)
Capping Level	50	7	100
Number of Samples Capped	11	2	4
Percentage of Metal Capped	3.4	1	11.7

14.9 Sample Compositing

The capped drillhole assays were composited using GEMS compositing routines to 2 m lengths down the drillholes and broken by domain boundary. Where intervals shorter than 1 m in length remained at the end of a domain interval, all composite lengths were adjusted slightly so that each composite was the same length. If the remaining interval was less than 2 m but greater than 1 m, the composite was created at that length.

Missing intervals or intervals defined in the database with no assay information were replaced with very low grades (half detection limit) during the compositing procedure. The greatest impact of replacing missing values was during compositing silver grades as non-assayed intervals comprised 20% and 27% of the West and North zone drilling. Silver was analysed for 96% of East zone drilling. Summary statistics for the composites are presented in Table 14.10.

Table 14.10 Summary Statistics for 2 m Composites

	Length (m)	Au (g/t)	Au Capped (g/t)	Cu (%)	Cu Capped (%)	Ag (g/t)	Ag Capped (g/t)
Valid cases	2727	2727	2727	2727	2727	2727	2727
Mean	1.98	5.19	5.03	0.871	0.863	10.8	9.7
Variance	0.007	46.33	23.93	0.553	0.349	1767.8	117.4
Std. Deviation	0.08	6.81	4.89	0.744	0.591	42.0	10.8
Variation Coefficient	0.04	1.3	1.0	0.9	0.7	3.9	1.1
Minimum	1	0.001	0.001	0.001	0.001	0.1	0.1
25th Percentile	2	2.47	2.47	0.495	0.495	2.7	2.7
Median	2	3.5	3.5	0.7	0.7	7.2	7.2
75th Percentile	2	5.62	5.62	1.03	1.03	13	13
Maximum	2.78	186.9	50	24.495	6.11	1876.5	100

14.10 Spatial Analysis

Using commercially-available Sage2001 software, experimental correlograms for gold, copper and silver were computed from the composites for each mineral zone. Downhole experimental correlograms were fitted to determine the nugget effect.

For each grade element, experimental correlograms in thirty-seven directions were used as the input data to prepare a correlogram model with a nugget and two nested spherical structures for gold and copper, and one for silver. In general, the correlogram models are robust and reflect the orientation of the mineralization. Effectively, the maximum ranges were quite short at about 10 m; however, some models have a small component of a long range contribution reflecting the strike length of the West Zone. Gold displayed a moderate nugget effect of approximately 43% of the sill and while the orientation of the maximum range was within the general plane of mineralization, the model is not highly anisotropic, with a ratio of only about 1.4 to 1.

Correlogram models were not able to be fitted with confidence for either the North Zone or the East Zone as these zones have far fewer samples.

The variogram models presented in Table 14-11 were used during grade interpolation using the Ordinary Kriging (OK) estimator.

Table 14.11 Grade Correlogram Models

Zone	Grade	Nugget	C1	1st Structure Ranges			1st Structure Rotation Angles			C2	2nd Structure Ranges			2nd Structure Rotation Angles		
				X	Y	Z	Z	Y’	Z’		X	Y	Z	Z	Y’	Z’
West Zone	Au	0.427	0.529	10.2	7.3	6.3	-39	61	35	0.044	18.9	225.8	29.0	23	52	9
	Cu	0.206	0.626	6.9	8.5	7.6	-67	62	0	0.168	29.6	82.5	162.9	22	11	40
	Ag	0.279	0.621	16.4	3.7	9.2	2	17	1	-	-	-	-	-	-	-

Note: Correlogram models are given in the real world coordinate system using the RRR rotation convention as used in GSLIB

14.11 Estimation/Interpolation Methods

14.11.1 Block Model Parameters

Resources were estimated using 3D block modelling techniques. GEMS was the software with which the model was prepared. The block size was set to 4 m cubes, and the model was not rotated. The block size is approximately 1/3rd the drillhole spacing and carries a percent model to reflect the proportion of the block that is contained within the modelled mineralized zones.

14.11.2 Block Grades

Block grade estimates were based on the composited assay data from drillholes. Only composites from each modelled zone were used to estimate block grades for the corresponding zone. A single-pass estimation strategy was used to estimate block grades for capped and uncapped gold, copper, and silver.

The grades were computed using Ordinary Kriging. A minimum of two composites from one drillhole within 30 m was required to estimate the grade of the block. A maximum of two composites from a single hole, and only the nearest 12 samples were used in the estimate of each block grade. Each block carries additional data including the number of holes and the number of composites used to estimate the grade, the distance to the nearest composite, the average distance to the composites used, and the variance of the composites used.

Ordinary kriging requires a distance weighting formula which was presented in a previous section. Interpolation of grade in the East and North Zones used the same correlogram model as for the West Zone.

Grades were also interpolated using Inverse Distance (ID^2) to the second power, and by Nearest Neighbour (NN) methods for validation.

Blocks outside of the mineralized domains were also estimated with grades. Block grades were interpolated using Ordinary Kriging, using the correlogram model from the West Zone, for the purpose of estimating dilution grade in the economic analysis which is discussed in **Section 16**. A similar search strategy was used however a high-grade search restriction was utilized to prevent isolated high-grade composites from being used to estimate grade in blocks farther than 10 m from a high-grade composite. The high-grade thresholds were set to 3 g/t Au, 0.7% Cu, and 15 g/t Ag.

4.11.3 Bulk Density

Bulk density was assigned to each block as a volume weighted average based on the lithology wireframes the block contained (Table 14.12). To account for the observation that an increase in density is linked to an increase in sample grade, the density model for the heterolithic breccia and feldspar porphyry rock types were adjusted based on Table 14.13. Not enough density sampling was available within the volcanic rocks or quartz latite porphyry to establish a trend with any confidence; however, it is MFW's opinion that the bulk density of these rock types would also be positively affected by an increase in grade.

Table 14.12 Bulk Density by Lithology Wireframe

Lithology	Bulk Density
Heterolithic Breccia	2.71
Feldspar Porphyry	2.65
Lamprophyre Dyke	2.84
Quartz Latite Porphyry	2.63
Volcanic Rocks	2.81

Table 14.13 Bulk Density by Au Eq Grade Bin

Bin of Au Eq (g/t)	Heterolithic Breccia Mean Density	Feldspar Porphyry Mean Density
0-1	2.71	2.65
1-2	2.73	2.65
2-3	2.75	2.69
3-4	2.80	2.75
4-5	2.86	2.83
>5	2.86	2.91

14.12 Block Model Validation

Five validation exercises were completed on the Willa resource model.

Visual comparison of block and composite grades on cross-sections and plans were conducted. An expected level of smoothing was observed. No obvious discrepancies between block and composite grades were observed. No large unsupported areas of high-grade blocks were observed.

A global comparison of contained Au metal between the capped OK, ID^2 and NN models was completed. The ID^2 model reported 4% fewer gold ounces above a 3.0 g/t ID Au cut-off grade than the OK model. The NN model reported 30% fewer gold ounces above a 3.0 g/t NN Au cut-off grade than the OK model. ID and NN models were only estimated for gold for the purpose of validation.

A global comparison of contained metal between the capped and uncapped models was also completed. Capping grades removed less than 5% Au metal above a 3.0 g/t Au Eq cut-off grade, 2% of the Cu metal, and 15% of the Ag metal.

A local comparison of ID^2 block grade to NN block grade using swath plots was completed. The ID^2 blocks generally honour the grade distribution of the NN blocks, indicating no local bias was observed in the model. Any deviations noted corresponded to areas where there were only a small number blocks.

A wireframe representing the volume of rock collected for the underground bulk sample collected in 1988 was modelled and used as a constraint to report tonnage and grade from the block model. The wireframe reports 480 tonnes at an average grade of 5.39 g/t Au, 0.61% Cu and 6.2 g/t Ag. The reported grade from belt sampling during processing of the 494 tonne bulk sample was 5.55 g/t Au and 0.69% Cu. The current model is 3% lower in Au grade and 12% lower in Cu grade than the reported belt sampling; however, these results are considered to be within the accuracy of the estimate.

14.13 Classification of Mineral Resources

Resources were classified as Measured, Indicated and Inferred in compliance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), guided by confidence levels of the drilling and sampling methods, geological understanding and interpretation, spatial distribution of samples, and quality of the grade estimate. Classification was not downgraded because of lack of access to inspect mineralization in core or in underground workings, but because of the belief that the technical work conducted on the Property was completed professionally and consistent to standard practices for the time period.

Within the mineralized zones, the spacing of drillhole intersections was sufficient to reasonably assume continuity of mineralization and assign Indicated to most of the blocks within the core of the zones. Blocks with grades interpolated using 12 composites, from at least 6 drillholes, which have an average distance of 15 m to the centre of the block, were classified as Indicated Resources.

Generally, the Inferred category was assigned to the outer fringe of the mineralized zones, where grade was extrapolated. Blocks with grades interpolated using less than 12 composites were classified as Inferred Resources. Also the blocks interpolated using 12 composites, from at least 6 drillholes, which have an average distance of greater than 15 m to the centre of the block, were classified as Inferred Resources.

Where underground development intersected the West Zone, blocks within 10 m of an underground drillhole collar were classified as Measured Resources.

14.14 Reasonable Prospects of Economic Extraction

MFW considered all blocks above a 3.0 g/t Au Eq cut-off grade that were within the domains modelled to have a reasonable likelihood of eventual economic extraction by underground mining methods. At a $US 1200/oz gold price, 3.0 g/t Au is equivalent to approximately $US 115/t. The block model was exported to a mining engineer who prepared a preliminary mine design and an economic analysis (**Section 16**).

14.15 Mineral Resource Statement

The Willa Mineral Resource Estimate has an effective date of November 30, 2015. The estimate is summarised in Table 14.14, above a 3.0 Au Eq cut-off grade.

Table 14.14 Willa Mineral Resource Tabulation above 3.0 g/t Au Eq Cut-off Grade

Zone	Resource Category	Density (t/m3)	Tonnage (t)	Au (g/t)	Cu (%)	Ag (g/t)	Au (oz)	Cu (lbs)	Ag (oz)
WEST ZONE	Measured	2.824	198,000	5.36	0.83	8.3	34,000	3,623,000	53,000
	Indicated	2.815	537,000	4.96	0.89	9.7	86,000	10,537,000	167,000
	Measured + Indicated	2.817	735,000	5.07	0.87	9.3	120,000	14,097,000	220,000
	Inferred	2.773	68,000	4.19	0.70	11.5	9,000	1,049,000	25,000
NORTH ZONE	Indicated	2.809	45,000	4.92	0.59	5.9	7,000	585,000	9,000
	Inferred	2.749	68,000	4.24	0.76	8.8	9,000	1,139,000	19,000
EAST ZONE	Indicated	2.806	45,000	5.17	0.81	10.2	7,000	804,000	15,000
	Inferred	2.795	15,000	4.15	0.56	6.9	2,000	185,000	3,000
TOTAL	Measured	2.824	198,000	5.36	0.83	8.3	34,000	3,623,000	53,000
	Indicated	2.814	627,000	4.97	0.86	9.5	100,000	11,888,000	192,000
	Meas+Ind	2.816	825,000	5.07	0.85	9.2	134,000	15,460,000	244,000
	Inferred	2.764	151,000	4.21	0.71	9.8	20,000	2,364,000	48,000

1. Mineral Resources do not include external dilution.
2. Tonnages are rounded to the nearest 1,000 tonnes, and grades are rounded to one or two decimal places as shown.
3. Material quantities and grades are expressed in metric units and contained metal in imperial units.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.16 Factors That May Affect the Mineral Resource Estimate

MFW is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other factors that could materially affect the mineral resource estimate.

14.17 Comparison to Previous Mineral Resource Estimate

The previous estimate of Mineral Resources was prepared in 2012 for Discovery Ventures. The Measured and Indicated Resources were unchanged from those disclosed in a 2005 Technical Report for Bethlehem Resources (1996) Corporation; however, in 2012 there was the addition of Inferred Resources. The only material technical information collected, that is pertinent to the mineral resource estimate, since the 2012 estimate was 96 samples of mineralized drill core tested for dry bulk density.

The estimate presented in this report has 21% more volume at a 3% lower density for 17% more tonnes. The reported tonnage is 23% lower in Au and Ag grades and 4% higher in Cu grade than the previous estimate. The amended 2016 mineral resource contains, based on the updated tonnages and grades, 10% fewer ounces of gold, 21% more pounds of copper, and 8% fewer ounces of silver.

A detailed review of the previous model was not completed by MFW; however, the factors contributing to the largest changes are likely due to the following changes in methodology:

- The updated estimate used estimation constraints based on a grade shell model and applied hard boundaries during grade estimation. The previous estimate did not use estimation constraints other than the search radius extent.

- The updated estimate of grade used the Ordinary Kriging estimator. The previous estimate used Inverse Distance to the 3rd power.

- The updated estimate was reported above a 3 g/t gold equivalent cut-off grade, which considered recoverable contributions from gold, copper and silver. The previous resource was reported above a 3.5 g/t gold only cut-off grade.

- The updated estimate introduced a variable density model based on tests on samples of drill core and taking into account the observation of an increase in density with an increase in sample grade. The previous estimate applied a constant density of 2.9 t/m^3.

- The current estimate applying different capping levels to raw assays.

- The current estimate was classified differently, with far less tonnage reporting to the measured category.

15.0 Mineral Reserve Estimates

There are no mineral reserve estimates for the Property.

16.0 Mining Methods

16.1 Mine Layout

Access to the mineral zones of the Willa deposit is via two existing adits. The main haulage adit (at 1,025 m above mean sea level) is termed the 1025 L, and "Upper" adit is at 1,100 masl (1100 L). The 1025 L is equipped with 60 pound rail on 30-inch (750 mm) track gauge for 1,000 m into the adit but a ground collapse at approximately 300 m into the adit prevented inspection by co-author Ash beyond that point. Air and water lines have been removed from both adits. Where viewed by co-author Ash, the 1025 L access tunnel averaged about 3.5 m square while the 1100 L averaged about 3.5 m high by 3.6 m wide and appears to have been driven by trackless methods. The mine has been developed by almost 2,400 m of development, as shown in the following table:

Table 16.1: Total Present Mine Development

Section	From	to	Length (m)
1100 Level			660 m
1025 Level			1,250 m
1013 Level			160 m
Decline	1025 L	1013 L	130 m
Decline	1013 L	993 L	150 m
Vent Esc Rse	1025 L	1100 L	106 m
Vent Raise	993 L	1025 L	45 m
TOTAL			2,385 m

16.2 General Pre-Production Development

In addition to rehabilitation of the present drifts and raises, items which will be installed in the pre-production phase will include underground explosives and detonator magazines, a refuge chamber, rail siding and an underground transformer station.

Three main mineral zones have been explored by diamond drilling, including the North zone, the East Zone and the West Zone. For the cash flow analysis, only the West Zone is considered as it has been explored in more detail, and appears to be the highest grade. The other two zones warrant additional exploration to confirm economic viability.

The west zone occurs between 1000 m and 1200 m into the mine from the 1025 L portal. The West Zone has been traced along strike for over 200 m, and vertically over 130 m, with mineralization open to depth. Eight stopes have been defined to date in the West Zone, ranging in Measured and Indicated tonnage potentials from 2,500 to over 180,000 tonnes. The mineral occurs over wide widths, at times exceeding 30 metres, with the mineral extending in some instances over vertical distances of 75 m and other instances over 100 m in length. The majority of the stopes in the lower section of the West Zone (those with bases below the 1025 L) will generally be collared at an elevation of 990 m elevation. The cost of extending the decline down from the 990 L to an elevation of 970 m (or deeper) has not been incorporated in the assessment of the development required. Although the West Zone is open at least to an elevation of 940, additional diamond drilling is required to better direct the route that the decline should take.

As in the past, track drifts will continue to have cross-sectional dimensions of 3.5 m x 3.5 m. Declines and drifts serviced by ST-7 Scooptrams are assumed to have minimum dimensions of 4.3 m wide and 3.66 m high. Access, escape-way and ventilation raises are assumed to have minimum dimensions of 1.5m wide and 1.5 m from sill to back (height). An Alimak-driven ore-pass required between the 1025L and 1100 L is assumed to have minimum dimensions of 2.5 m x 3.0 m. Slot raises in stopes, measuring 2.2 m x 2.2 m will be drilled by long-hole drills and blasted by the vertical retreat method. These on-mineral raises have not been incorporated as development costs because blasting them by retreat methods is included as part of the stoping costs. However, 2.2 m x 2.2 m raises driven in waste using the long-hole drill drilling and retreat-blasting method have been included where they occur in waste (that is, the knuckle-back muck pass between the 1025 L and 1100 L).

The total diluted and recoverable tonnage, based on Measured and Indicated Resources, in the eight stopes is estimated at 617,460 tonnes grading 4.51 g Au/t, 0.81% Cu, and 8.4 g Ag/t. This applies to the West Zone only, and between elevations of 970 and 1100 m.

The diluted and recoverable mineral resource for the eight stopes is as follows:

Table 16.2 Diluted and Recoverable Mineral Resource Summary

Stope #	Tonnes	Au (g/t)	Cu (%)	Ag (g/t)	$ Value/t
1	183,387	4.25	1.03	13.23	$220.35
2	41,131	5.67	0.88	7.64	$265.66
3	180,035	4.82	0.68	5.17	$221.96
4	11,010	3.22	0.47	4.46	$149.49
5	114,446	4.45	0.72	6.12	$209.97
6	13,601	4.6	0.64	7.26	$212.44
7	71,187	4.04	0.77	8.9	$197.73
8	2,663	3.81	0.91	10.29	$196.19
Total	**617,460**	**4.51**	**0.81**	**8.39**	**$217.58**
Internal dilution, overbreak dilution and mining recovery included in the above					

The dollar values in Table 16.2 are based upon Table 22.1 calculations:

- Gold = $38.77 / gram
- Copper = $48.10 / percent
- Silver = $0.47 / gram

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Preliminary Economic Assessment is preliminary in nature, and although it does not include Inferred Mineral Resources, there is no certainty that the Preliminary Economic Assessment will be realized.

The above recoverable Mineral Resource assumes 5% over-break dilution and 95% stope recovery. The recoverable tonnage and grade may be considerably lower, depending upon potential for increased dilution due to un-intended ground collapses, inaccurate projections of stope boundaries, price decreases in commodities, or other events.

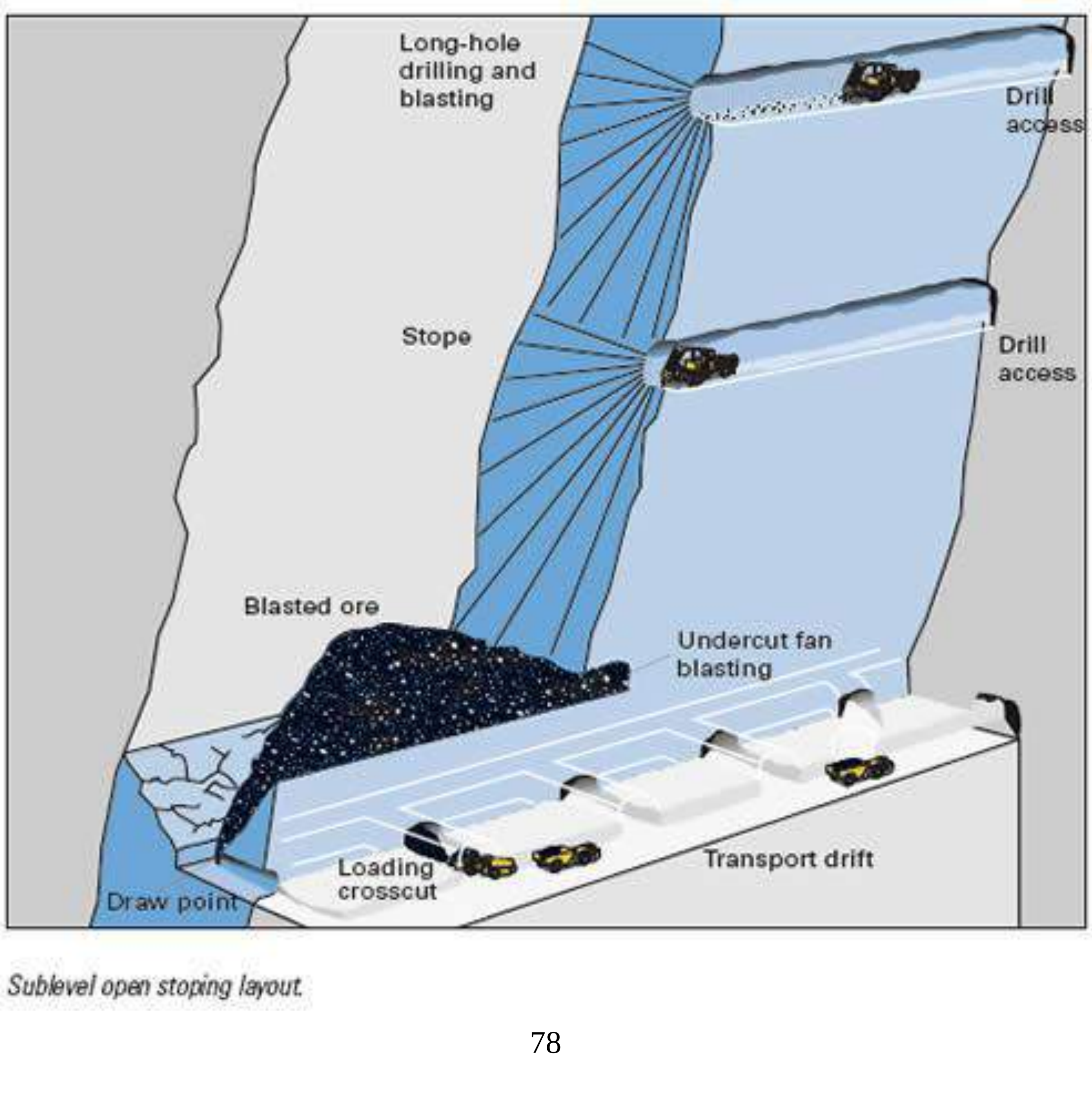
16.3 Stoping Methods

Due to the wide widths and relatively low grade of the mineral economy dictates that low-cost underground mining methods must be employed. The mining method chosen for recovery of the mineral is long-hole open-stope mining, with sub-levels developed at 15-m vertical intervals. The long-hole stopes are designed for:

- Minimum widths of 3 metres, maximum widths at times exceeding 30 m,
- Minimum lengths of 20 m and at times, exceeding 100 m.
- Vertical dimensions varying from 20 m to 75 m.

The 15-m interval for the sub-levels will allow for confirmatory percussion-drill testing for stope boundaries, and where possible, for maintaining blast-hole lengths at 20 m or less.

Fig. 16.1 Typical long-hole mining method



Sublevel open stoping layout.

16.4 General and Stope Development (drifting and raising)

The total development required for recovery of the above tonnage is approximately 1,450 m in waste and 1,700 m in mineral, as tabulated in Table 16.3 below.

Table 16.3 Mine Development Summation

DEVELOPMENT FOR LOWER STOPES				
	Waste	Waste	Mineral	Mineral
LEVEL	(m)	(t)	(m)	(t)
990	190	9,298	113	4,898
1005	20	123	66	2,248
1010	19	638	323	11,000
1025	150	5,035	182	6,198
Muck Raises	25	428	0	-
1025 Misc.	30	756	0	-
TOTAL 990 to 1025 L	434	16,278	684	24,344
DEVELOPMENT FOR UPPER STOPES				
1025	135	4,531	64	2,180
1040	152	5,102	367	12,498
1055	103	3,457	227	7,730
1065	8	269	30	1,022
1070	67	2,249	166	5,653
1085	119	3,994	135	4,597
1100	208	6,982	21	715
Raise to 1055	40	658		
Alimak Access Rse	76	1,562		
Muck Rse to 1100	78	1,034		
Misc LH Muck Rses	25	428		
Total Upper Mine	**1011**	**30,266**	**1010**	**34,395**
Grand Total Development	**1,445**	**46,544**	**1,694**	**58,739**

Internal dilution (sub-grade mineral and waste occurring within the stope boundaries) is incorporated into each stope base-tonnage and grade. In addition, the assumptions incorporated in each stope include:

- Mineable recovery assumed to be 95%
- External dilution (over-break) of 5%
- Each mineral block used in mineral resource calculation is cubic with 4 m per side (total 64 m^3 per block)
- Mineral density (t/m^3) was incorporated into each individual mineral block, based upon gold content, and ranging from 2.72 to 2.86 t/m^3
- Mining method is long-hole open stoping with preference for (but not limited to) down-hole blast-holes.
- Blast-holes are assumed to be 64 to 75 mm diameter, based upon Boart-Longyear Stopemaster (or equivalent).

- Broken mineral in the stopes is to be recovered from drawpoints or from the stope bases by remote-controlled ST7-Scooptram (load-haul-dump (LHD)) units.
- Mineral (and waste) recovered from below the 1025 L will be double-handled by intermediate storage in either the muck bay immediately adjacent to the rail-loading facility on the 1025 L, or on the 1013 Sub-level.
- The muck will be transferred to 4.5 m^3 rail cars on the 1025 L, by ST-3.5 Scooptram (or equivalent), for rail haulage to the tipple outside the 1025 L portal.
- For mineral and waste recovered from the stopes with bases on or above the 1025 L, the muck will be recovered from the draw points and loaded directly into the rail cars, thereby eliminating the up-grade haul and the double handling associated with extraction of mineral from below the 1025 L.

Pre-production development, development during the production phase, stoping and underground haulage are assumed to be conducted by mining contractors. Production includes stope drilling, blasting and muck haulage to the surface tipple. Transport of the mineral from the tipple to the MAX mill for the life of the mine is assumed to be done by separate contractor and is not considered part of the underground mining cost.

Percussion testhole drilling will be conducted from the on-mineral stope development headings in order to confirm stope boundaries. As these holes will duplicate as blast-holes, the cost of the testholing is considered part of the blast-hole drilling cost.

Due to the wide stopes, short-term pillars may be required (depending upon geotechnical conditions determined during final development and stoping). Recovery of short-term pillars may induce caving within some stopes and lead to "pulling to dilution" of the pillar muck.

Waste-rock produced by development during the pre-production phase and part of the first year of production will be temporarily stored on the 1025 L dump. A portion of it is expected to have potential acid generating (PAG) qualities. Therefore, when the first stope below the 1025 L is mined out, the PAG muck on the 1025 L dump will be returned underground and dumped into the open stope. And thereafter, all waste with apparent PAG potential will be dumped directly into the stope. Waste rock showing low PAG will remain on the dump and on closure, will be encapsulated to prevent or mitigate the potential for ARD.

Upon closure at the end of the mine life, the lower mine area will be flooded, covering the PAG waste rock, and essentially eliminating oxidizing conditions and thus, any acid rock drainage (ARD).

16.5 Mining Sequence

Due to the reduced development cost required for pre-production development for stopes below the 1025 L, production will commence on the stopes with bases on the 990 L. However, production of these stopes will not be complete before production commences from some stopes with bases on the

1025 L or above. Therefore, for simplification of the cash flow analysis, the overall average recoverable grade is assumed for the mine life.

Fig. 16.2 Present Development: Plan View

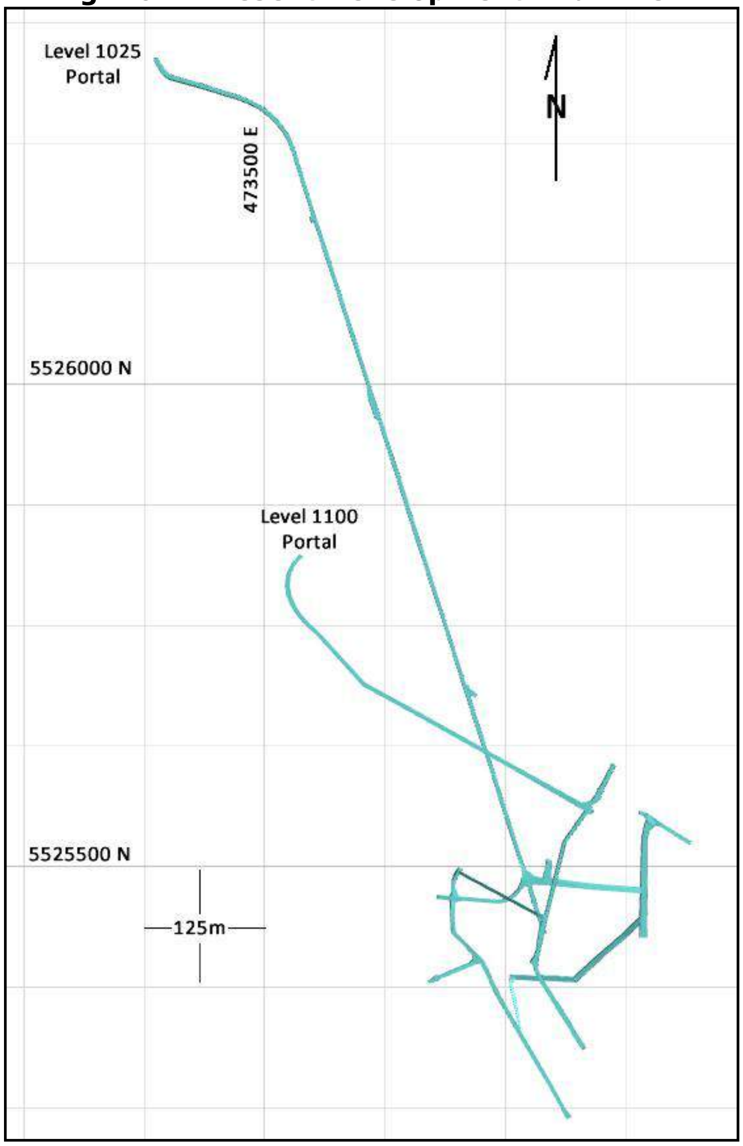


Fig. 16.3 Present Development: Isometric View Looking NW

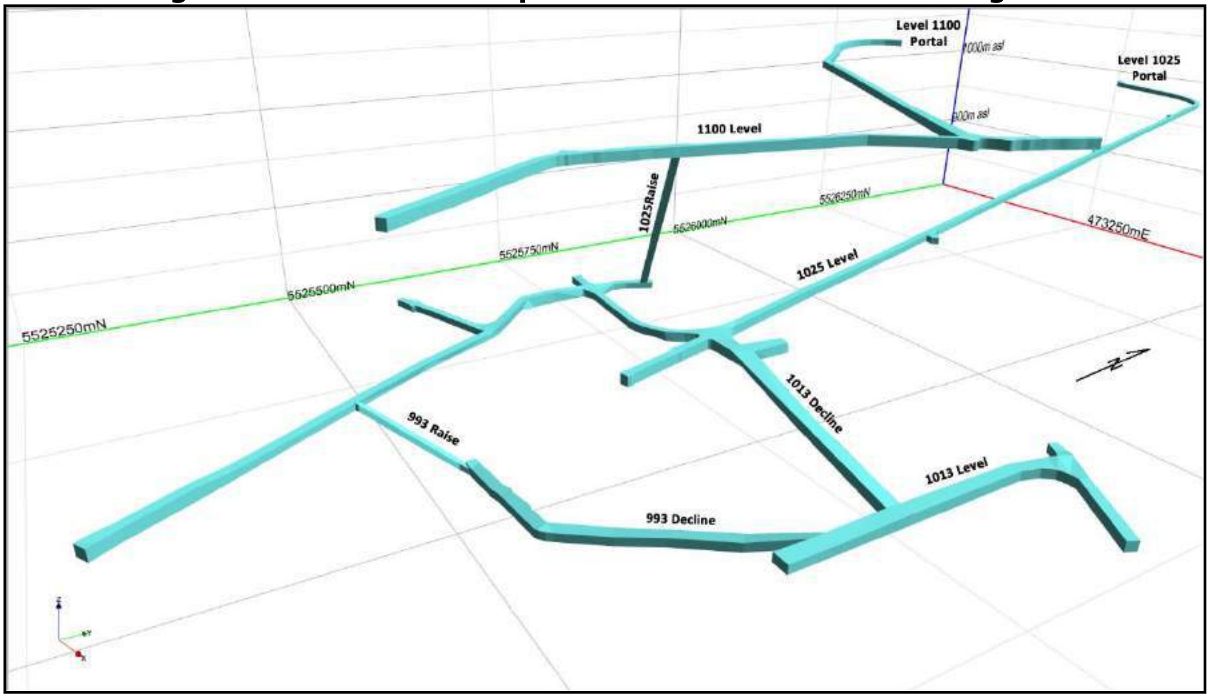


Fig. 16.4 Present Development and Proposed Development

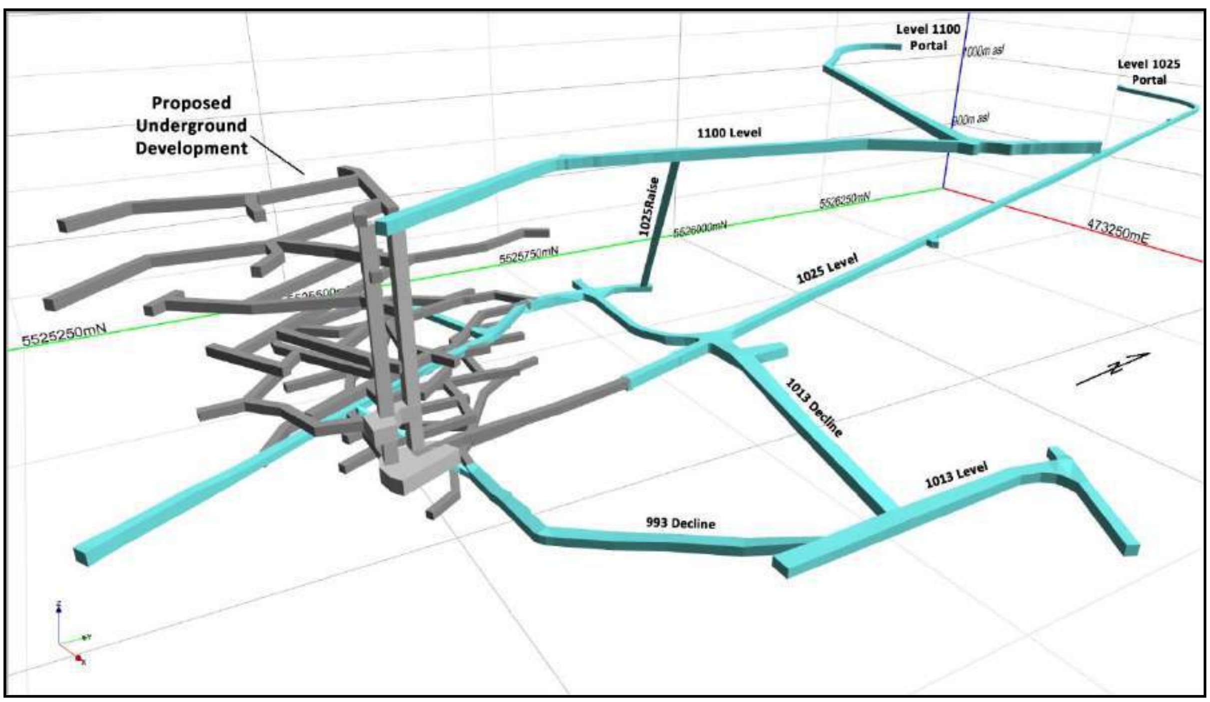


Fig. 16.5 2015 Resource with Present Development

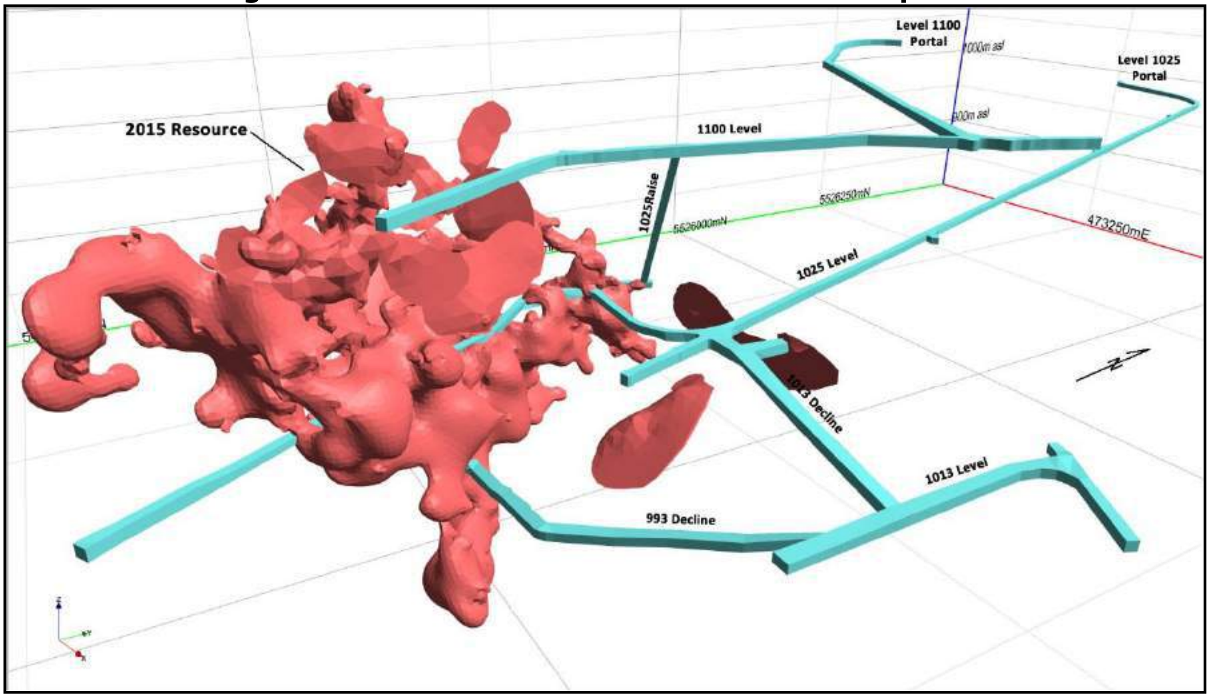


Fig. 16.6 Proposed Development Stope No. 1

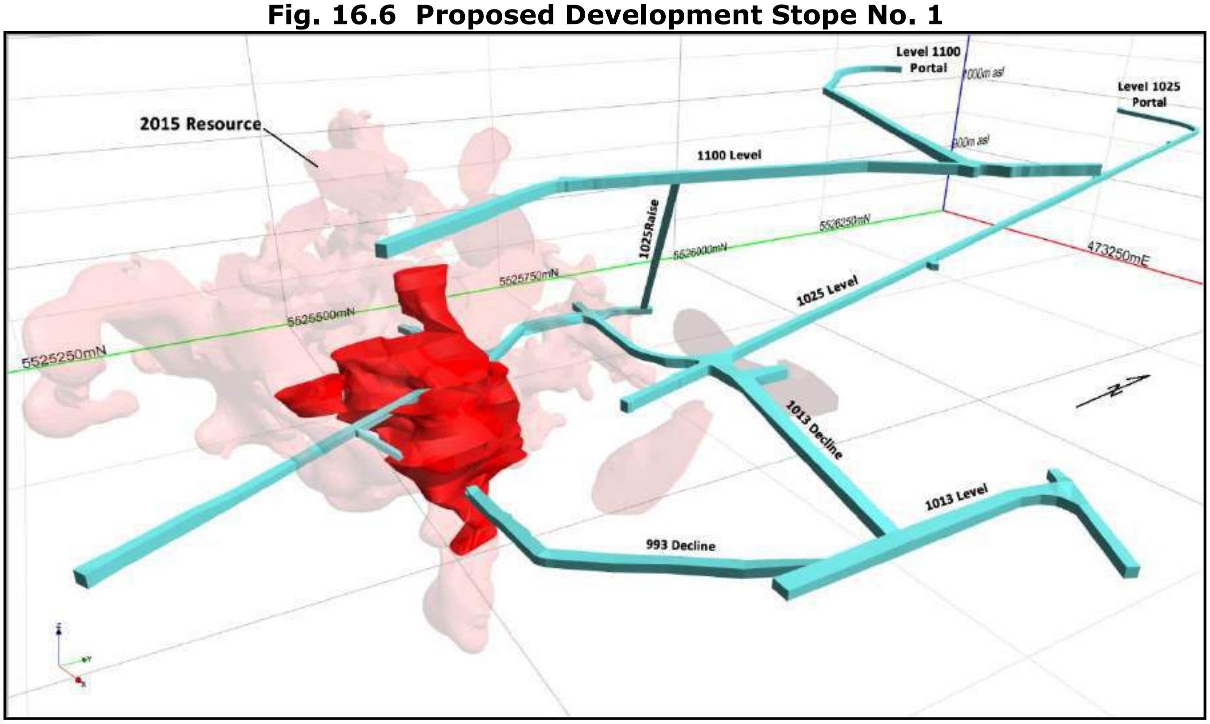


Fig. 16.7 Proposed Development Stope No. 2

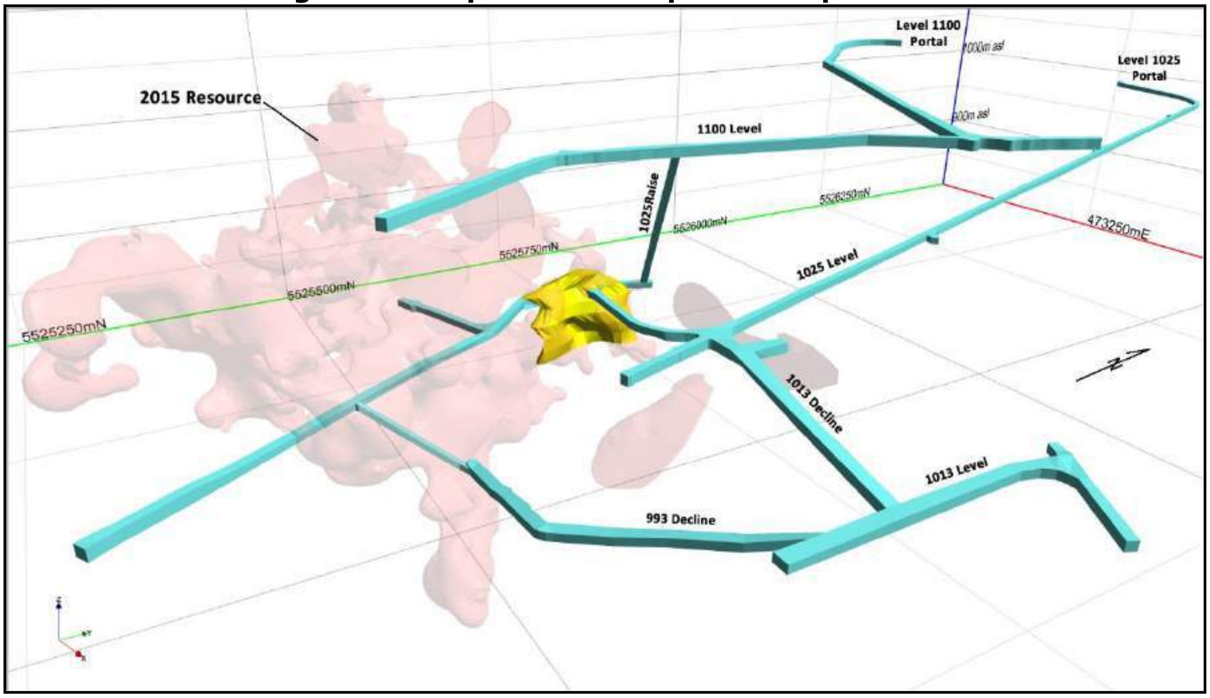


Fig. 16.8 Proposed Development Stope No. 3

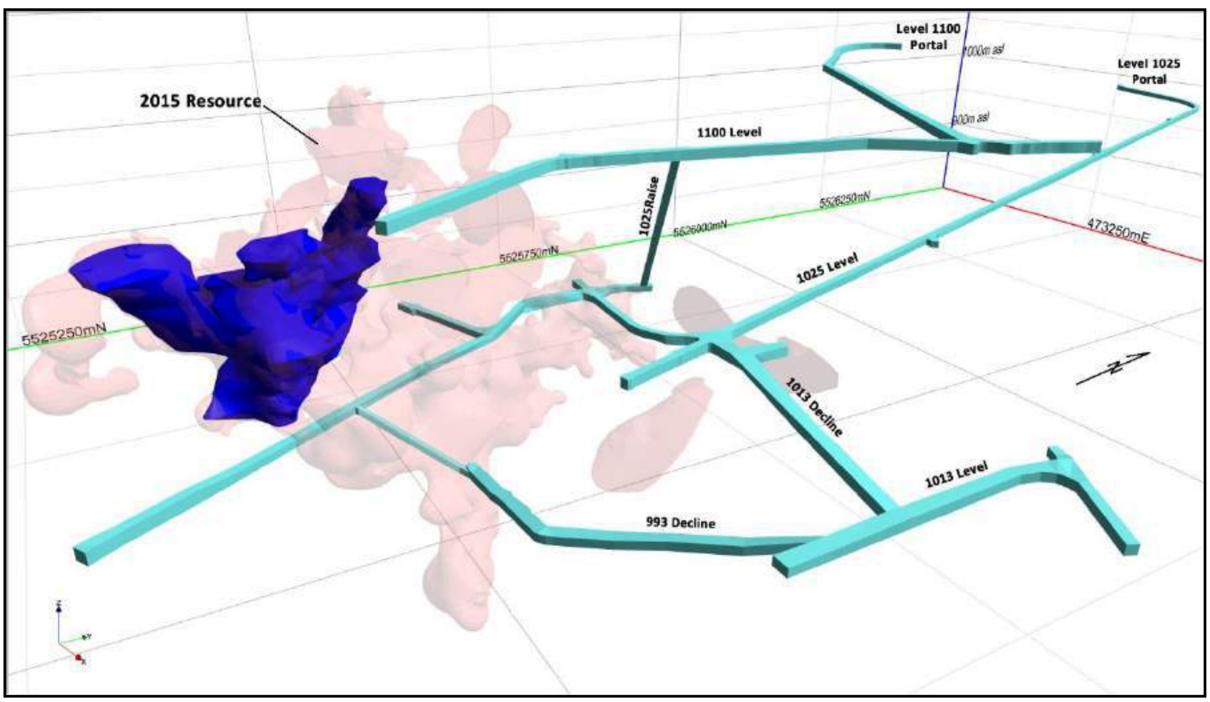


Fig. 16.9 Proposed Development Stope No. 4

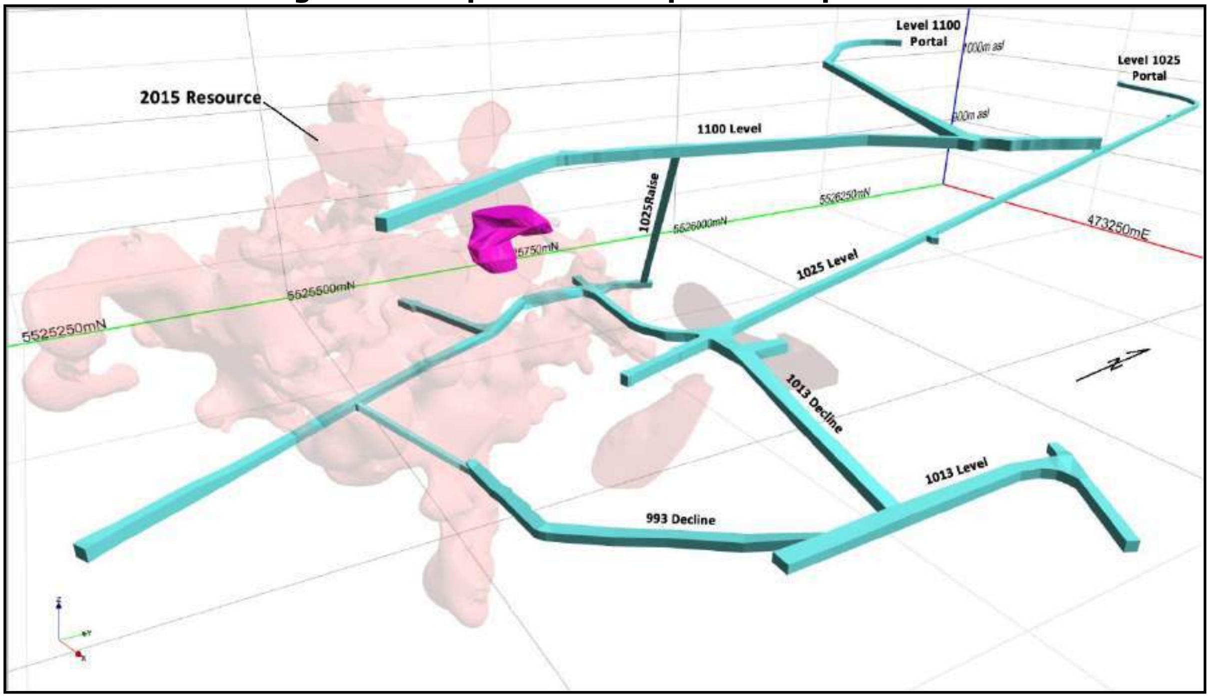


Fig. 16.10 Proposed Development Stope No. 5

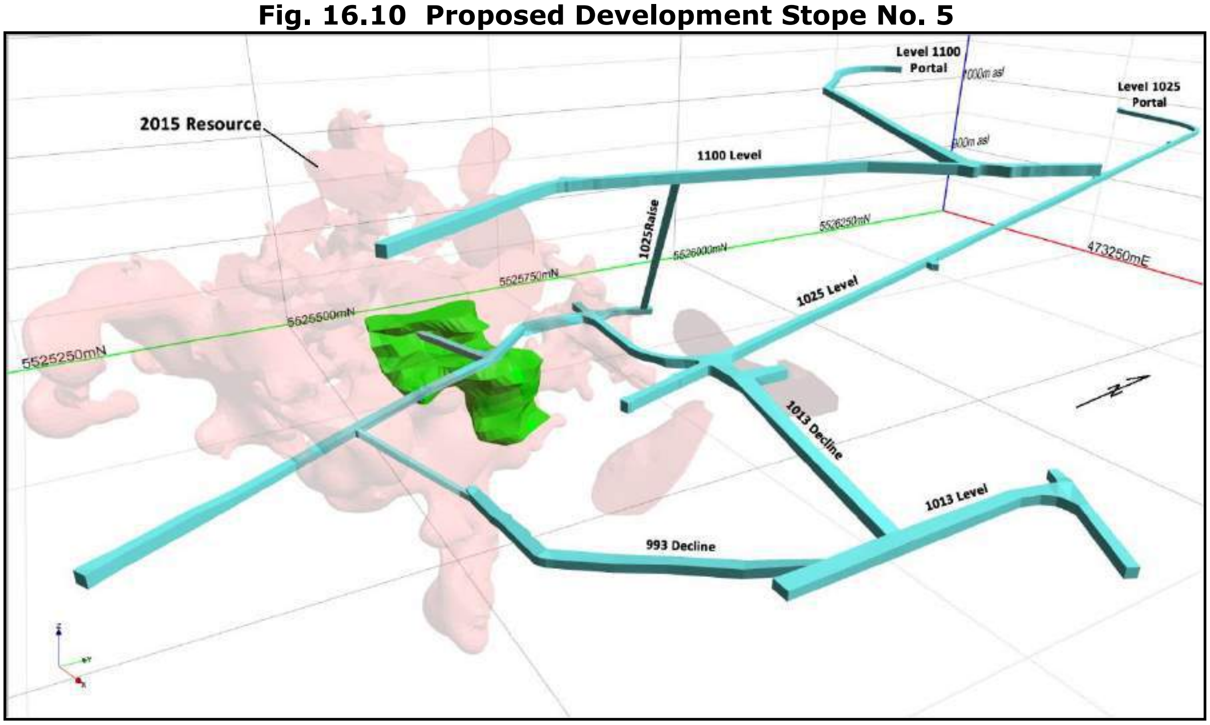


Fig. 16.11 Proposed Development Stope No. 6

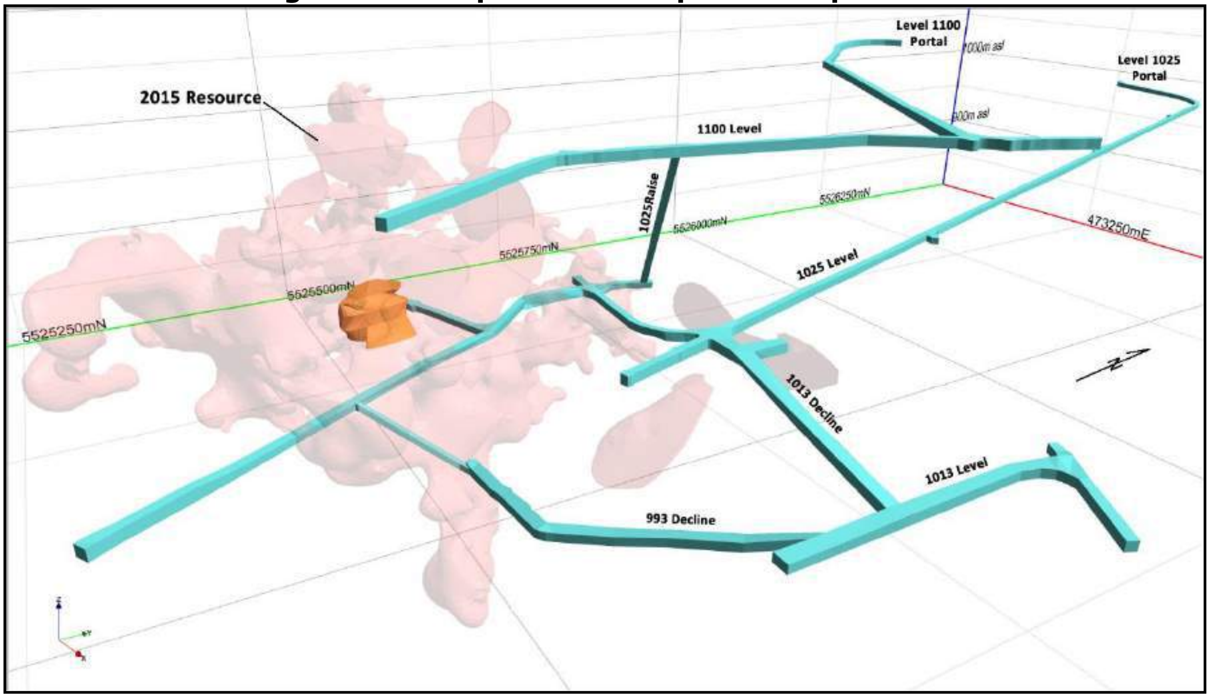


Fig. 16.12 Proposed Development Stope No. 7

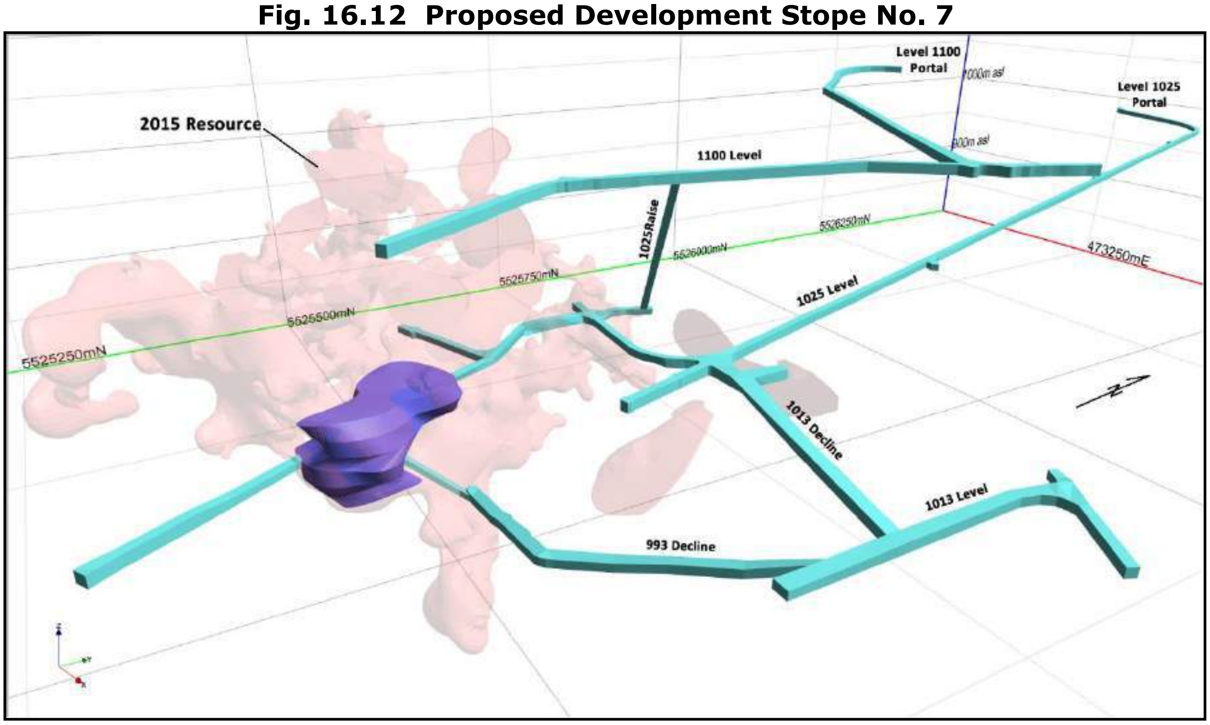


Fig. 16.13 Proposed Development Stope No. 8

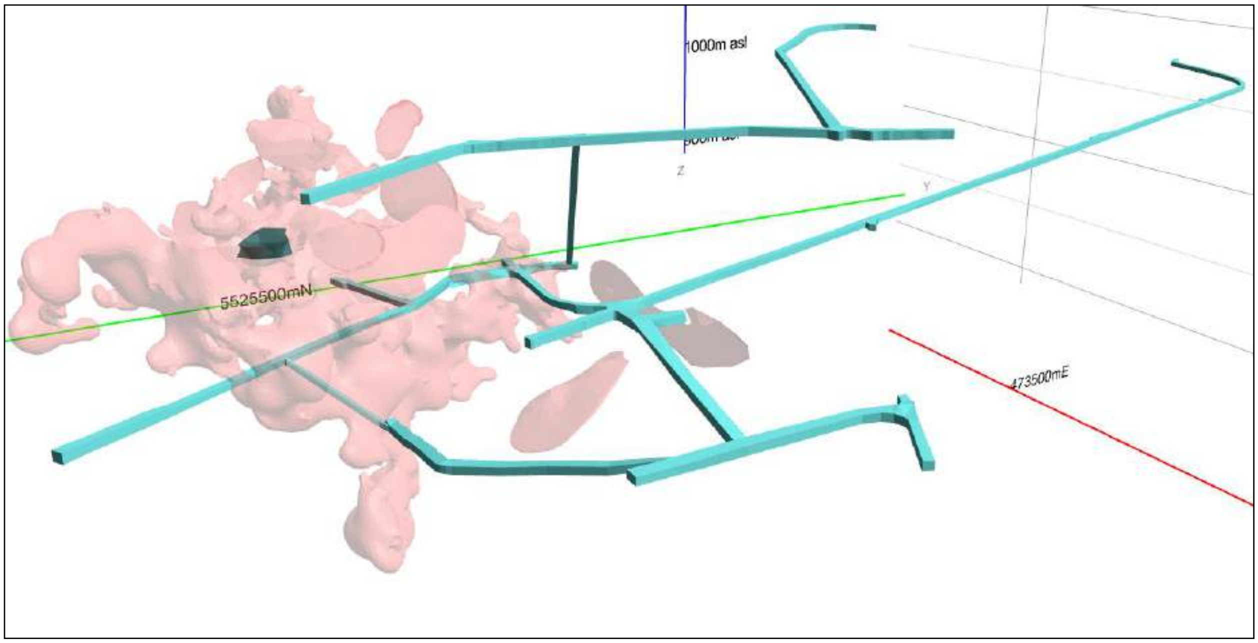


Fig. 16.14 Proposed Development – All Stopes

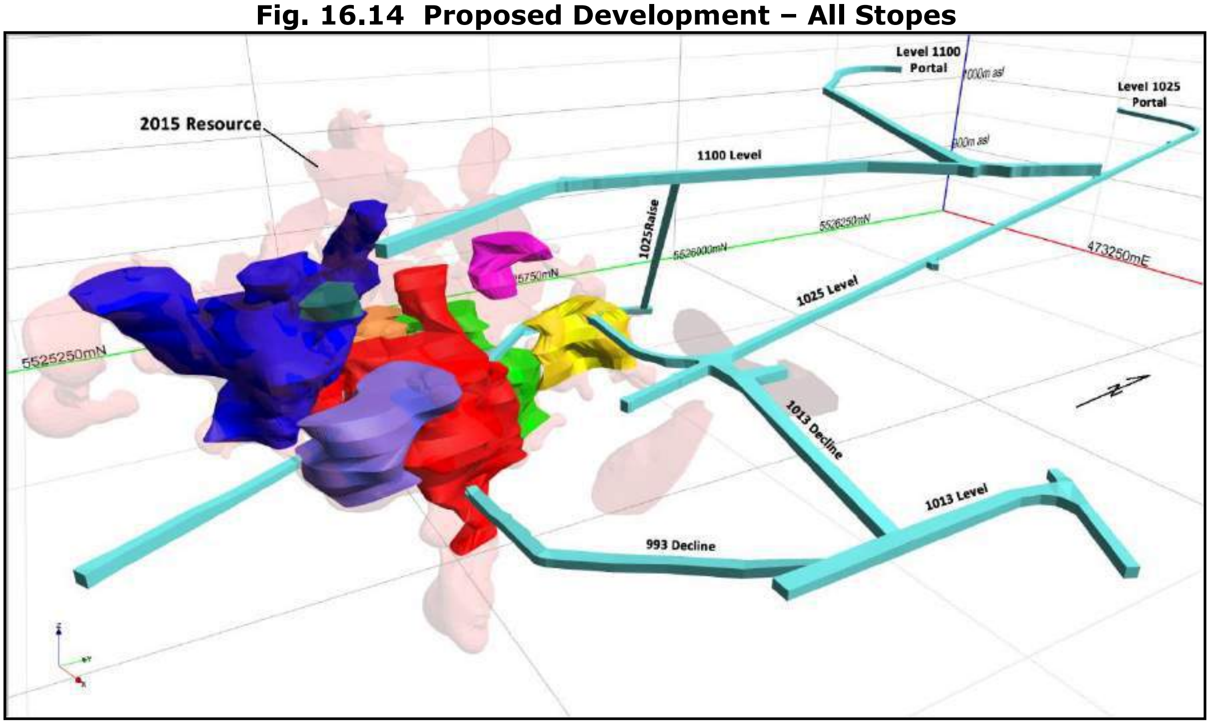


Fig. 16.15 Proposed Development – All Stopes, Looking NE

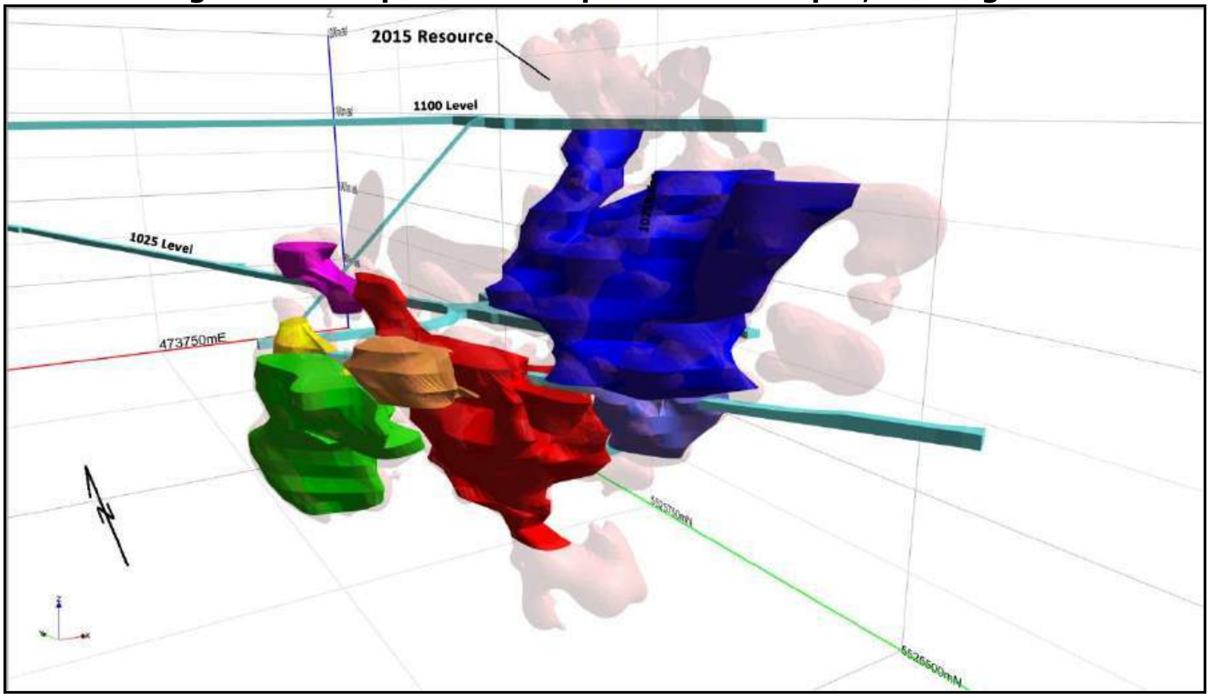


Fig. 16.16 Proposed Development – All stopes with proposed underground development (Looking NE)

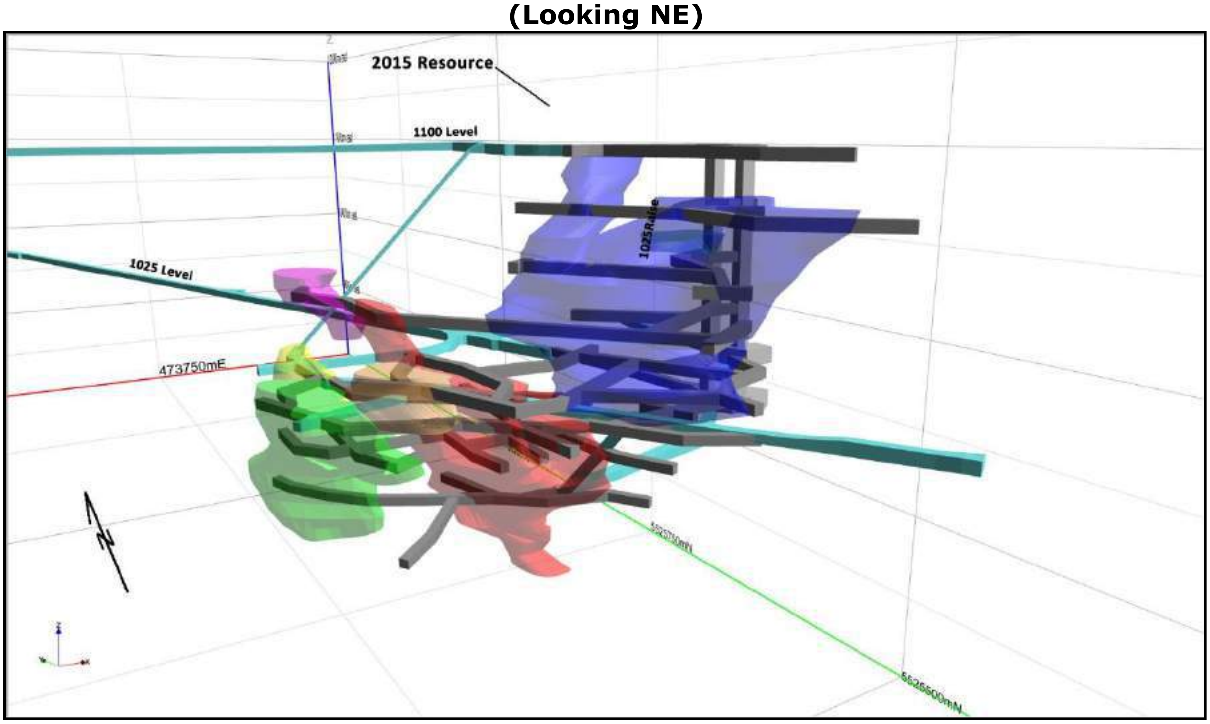
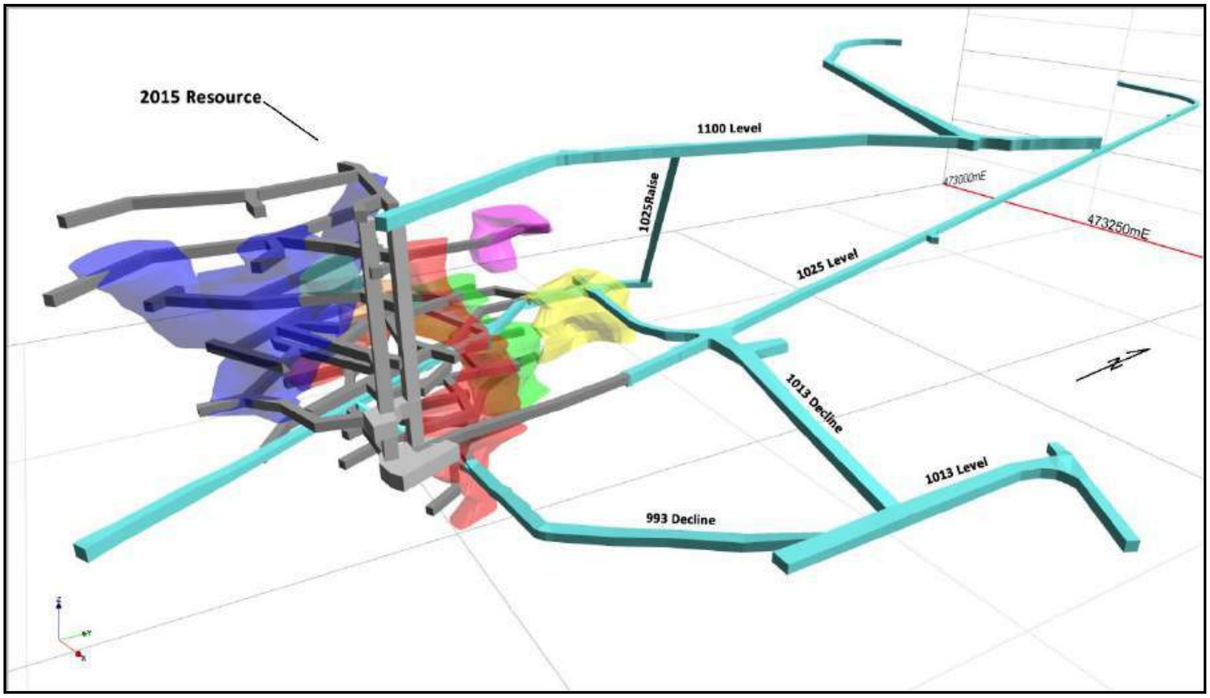


Fig. 16.17 Proposed Development – All Stopes with proposed underground development (Looking NW)



16.6 Underground Haulage Methods

Based upon best economics and other important factors such as ventilation, the haulage methods proposed for the underground operation includes a combination of load-haul-dump (LHD) equipment (such as Scooptrams), and rail haulage to surface.

Broken muck extracted from development or stopes below the main haulage level (1025 L) will be retrieved by remote-controlled diesel ST-7 Scooptrams and hauled up the decline, either to a storage bay located immediately adjacent to the 1025 L main track drift, or to a dead-end drift on the 1013 sub-level. A separate 3 m^3 Scooptram will load 6 only, six to eight-tonne rail cars for track haulage to the tipple outside the 1025 L portal. The locomotive may be either a battery or diesel-powered, 8 to 12-tonne locomotive. Muck from the proposed stopes above the 1025 L will be extracted from the 1025 L draw-points by the Scooptrams and loaded directly into rail cars for haulage to the surface tipple. For the cash flow analysis, the underground haulage costs are estimated at $C 9.44/t, based on the haulage from stopes to the lower dump.

16.7 Surface Mineral Haulage to the South Red Mountain Road

The present road between the 1025 L portal and the South Red Mountain Road is 1.2 km but averages over 15% grade and is too steep for production haulage. For production haulage the road will be re-routed, widened to 9 m (complete with ditch and safety berm) and will have a length of 1.8 to 2.0 km, with a downgrade of 8%. As the haulage road will be designed for one way traffic (but used for both transporting mineral down-hill and supplies up-hill), lay-bys and run-away lanes will be required. All vehicles using the road will be equipped with radios for on-road communication.

16.8 Mineral Transport to Mill

The trucking contractor may elect to haul the mineral directly from the 1025 L tipple to the MAX mill or to dump the muck on a pad immediately adjacent to the South Red Mountain road for trans-loading highway-permitted vehicles for transport to the MAX mill. The unit underground haulage cost includes transport from the tipple to the lower dump.

The MAX mill is located 142 miles km north of the Willa mine, at Trout Lake, all but 6 km of the route is on paved road, most of which is on Provincial highway 6. A general quotation from a local trucking firm of $C 28/t has been used in the cash flow model.

16.8 Projected Mine Life

The projected mine life will depend on the average daily production rate. Theoretically, the Willa mine has a production capability of 500 tonnes per day, as does the MAX mill. Under that condition, the mine, with a present recoverable mineral resource of 617,000 tonnes, would have a life-span of 3.4 years.

However, with respect to the mill:

- The 500 tonnes-per-day (tpd) was based upon a rougher flotation feed at 80% <110 µm. However, the rougher flotation feed required for the Willa mineral is 80% <55 µm. This may reduce the productivity of the present ball mills to between 300 and 335 tpd. By bringing the 9' x 15' Hardinge ball mill on stream, the mill throughput can be increased back to 500 tpd, but only with the addition of extra capital cost.
- Theoretical throughput of the mill does not include scheduled and unscheduled downtime. Under favourable operating conditions, a mill is off-line for 5% of the time, which reduces the operating daily throughput to 475 tpd.
- Mills rarely meet optimum mill throughput during the first year of operation since:
  i) On-going, on-site metallurgical testwork is required to meet and exceed recovery projections.
  ii) The crews require time to gain the necessary experience in refining each step of the process.
  iii) Further additional adjustments are required to each circuit and reagent additions.
  iv) In general for the first months of operation, particularly in gold mining, low grade mineral is charged to the mill in order to tighten the mills and prevent metallic gold particles from lodging in behind mill liners, pipes, and other locations where it may otherwise collect.

With Respect to the Willa mine:

- Highway transport through Silverton, New Denver, and Nakusp may be limited to restricted hours, which might increase the number of haulage trucks required (at additional transport unit cost).

- In addition to development and mine production, preparation for final stope design is dependent upon percussion drilling of intermediate holes for confirmation of stope boundaries in each stope prior to production in each stope, resulting in additional manpower requirement and interference between the production and exploration crews.
- A high mine-production rate would reduce the mine life, resulting in a shorter time-limit for on-going underground exploration, thus resulting in the loss of mineral resources which otherwise might be encountered and developed.

With respect to mineral haulage to the mill:

- Road weight-limit restrictions to 50% loads are normally in affect for 6 weeks during spring break-up. This could effectively double the transportation cost to over $C 50 per tonne delivered to the mill during the spring breakup period and reduce profits. Theoretically, the extra mineral could be hauled during the winter months and stockpiled at the MAX mill for milling operation during the break-up period. However, this impractical for two reasons. First, the snowfall is heavy at the MAX mill and would accumulate on the stockpile, causing havoc in the crushing circuit. Second, stockpiling during the winter months would result in having to exceed the practical in-mine-haulage capabilities at the Willa and probably, the highway transport by the contractor. Therefore, the winter stockpiling alternative is not a practical solution.

The best alternative would be to cease haulage during the spring breakup period, but this would reduce the overall daily tonnage hauled (on a yearly basis) to 440 tpd.

For this reason, the projected mine life for the cash flow analysis is based upon a production rate of 400 tonnes per day.

Based on present Measured and Indicated mineral resources, the 400-tpd option would increase the mine life-span to 4.2 years. While this option would increase the CAPEX pay-back time and lower the NPV and IRR, it has the advantages of allowing additional time during the production phase for:

- In-fill drilling to better-outline stoping blocks in the North and East Zones,
- Expansion drilling to upgrade Inferred Resources to Measured and Indicated Resources.
- Conduct exploratory and follow-up drilling, which has, to date, encountered significant mineral intersections outside the general boundaries of the three presently-known mineral zones.

Note that the West Zone is open to depth below 940 m elevation, although the present mineral resource is only included to 970 m elevation. Further decline development is required for exploratory drilling or production access below the 990 L but has not been included in the footage or tonnage of development of Table 16.3.

17.0 Recovery Methods

The Willa mineral responds well to the typical grinding and flotation conditions. The testwork data are considered adequate for process design purposes provided the data are interpreted conservatively and flexibility is incorporated into the flotation circuit to permit easy circuit changes. Gravity concentration is also required in order to recover the coarse gold content.

The copper concentrate samples from the PRA metallurgical testwork were assayed for Pb, As, Sb, Hg, Se, Te to determine whether sufficient deleterious metals or substances occurred in the concentrate which could result in significant penalties by a smelter.

MAX Mill Assessment

The MAX mine and mill operation complex was purchased by Discovery Ventures for processing of the Willa mineral. The MAX mine and mill were operated from 2008 to 2011 at a rate often exceeding 500 tpd over periods of several months. However, due to the finer grind required for liberation of the copper compound from the Willa mineral, the present grinding capacity is only capable of handing from 300 to 325 tpd. On the other hand, a larger ball mill (2.75 m diameter x 4.57 m long) is installed (but requires some additional adjuncts, including liners), which would allow the mill to process over 500 tpd.

Amenability of Willa Mineral to the Present MAX Mill Equipment

The MAX mill is a modern mill, constructed in 2007/08, designed to treat the molybdenum ore of the MAX mine and to produce a molybdenite concentrate. The units in operation between 2008 and 2011 included six only Denver Sub-A Rougher cells #25, (each, 1.4 m^3) cells, four only Denver Sub-A cleaner #24 cells (each, 0.68 m^3), and two only Denver Sub-A #15 cells (each 0.34 m^3) for cleaning. However, additional flotation cells installed but never commissioned, and include ten only Agitair #48 cells (1.13 m^3/cell), and twelve only Wemco #66 cells (each, 1.87 m^3), in various states of repair.

Three concentrates will be produced from the Willa deposit: gravity (coarse gold), chalcopyrite (copper/gold) and pyrite (gold). The gravity (gold) concentrate will make up less than 1% of the total concentrate produced.

Although the tonnage of copper/gold flotation concentrate is expected to approximate 2.25 to 2.5 times the amount of molybdenite concentrate tonnage, there is more than adequate rougher flotation capacity for the Willa copper concentrate (although, additional cleaner flotation cells and adjuncts (pipe, pumps, etc.) are required).

The 2015 metallurgical testwork has confirmed that a pyrite concentrate contains potentially-economically significant gold values. There are more than adequate additional cells for rougher (and cleaning, if advisable) of pyrite concentrate.

Photo 17.1 MAX Mill Interior

Installed Equipment in the Mill

Crusher Section

The dimensions of the MAX Crusher building measure 15 m long, 12 m wide, and 15 m high, with concrete foundation, steel I-beam frame, and wood and tin siding. The present installed crushing equipment is ideally suited for the Willa mineral. When the MAX mine was in production, the crushing plant operated between four and five hours per day to produce the 500 tpd required for treatment. Therefore, the crushing plant is fully adequate for crushing the Willa mineral. However, the coarse ore bin is open to the environment and will have to be covered in order to prevent precipitation from mixing with the coarse mineral. In addition, the coarse ore bin should be equipped with a grizzly and rock-breaker.

A summary of the equipment presently in the crushing facility includes:

- Coarse Ore Bin
- Apron Feeder
- Jaw Crusher: Traylor (36” x 42”) (91 cm x 107 cm) c/w coolant fan and coolant tank heater
- 3 conveyors
- Permanent Magnet
- Screen Deck (5’ x 10’) (1.5 m x 3 m)
- Cone Crusher: Symons 5.5 ft (1.67 m) short head
- Miscellaneous: electric heater, air compressor, lighting and overhead crane.

Dimensions of the main MAX mill building are 49 m (160’) long, 24.4 m (80’) wide and 12 to 15 m high. The mill is built on 3 levels, concrete foundation, I-beam frame with wood and tin siding and roofing. No building expansion is necessary for treatment of the Willa mineral. There is ample room for the installation of the additional required flotation cells, gravity circuit, lime and sulphuric acid circuits and on-site smelter for production of gold bullion. In addition, a building adjunct measuring 15 m x 10.7 m has been installed and houses the 9’ x 15’ ball mill.

Grinding Section

The equipment presently installed in the main mill building (and adjunct) include:

- Fine Ore Bin
- Feed Conveyor
- Belt Scale
- 2 only 7' x 7' (2.13 m x 2.13 m) Marcy grate-discharge ball mills in series
- Discharge boxes for each ball mill
- A 9' x 15' (2.75m diameter x 4.57 m) Hardinge ball mill (installed but requiring liners and adjunts (conveyor, pump boxes, pumps, cyclones, etc.)
- 5" x 4" (12.7 cm x 10 cm) SRL pump on #1 ball mill discharge box
- Feed box for the #2 ball mill
- Glandwater pumps: 2 only (1 standby)
- 2 only 5" x 4" x 14" (12.7 cm x 10cm x 36 cm) SRL pumps on #2 ball mill discharge box
- 2 only 10" (25 cm) Krebb cyclones

Rougher Concentrate Re-Grinding

- 1 only 4' x 7' (1.2 m x 2.1 m) re-grind ball mill
- 2 only 3" x 3" x 10" (7.6 cm x 7.6 cm x 25.4 cm) SRL pumps (1 on standby)
- 3 only 2: x 2" x 10" (5 cm x 5 cm x 25.4 cm) SRL pumps (1 on standby)
- 1 only 4" (10 cm) Krebbs Cyclone

Rougher Flotation

- Conditioner tank
- Rougher flotation cells: 6 only Gardner Denver Sub-A #24 cells (each, 1.4 m^3)
- 2 only 2" x 2" x 10" (5 cm x 5 cm x 25.4 cm) SRL Pumps (1 on standby)

Cleaner Flotation

- 4 only Gardner Denver Sub-A #18 cleaner cells (each 0.68 m^3)
- 2 only Gardner Denver Sub-A #15 cleaner cells (each 0.34 m^3)

Concentrate Thickening

- 1 only 18' diameter x 8' deep (5.5 m x 2.4 m) thickener
- 1 only 30' diameter x 10' deep (9 m x 3 m) thickener (not used)

Concentrate Thickening

- 1 only Thickened concentrate holding tank (4.7 m^3 capacity)
- 2 only Sperry automatic plate-and-frame concentrate filters (each with 35 m^2 of filter area)

Concentrate Drying

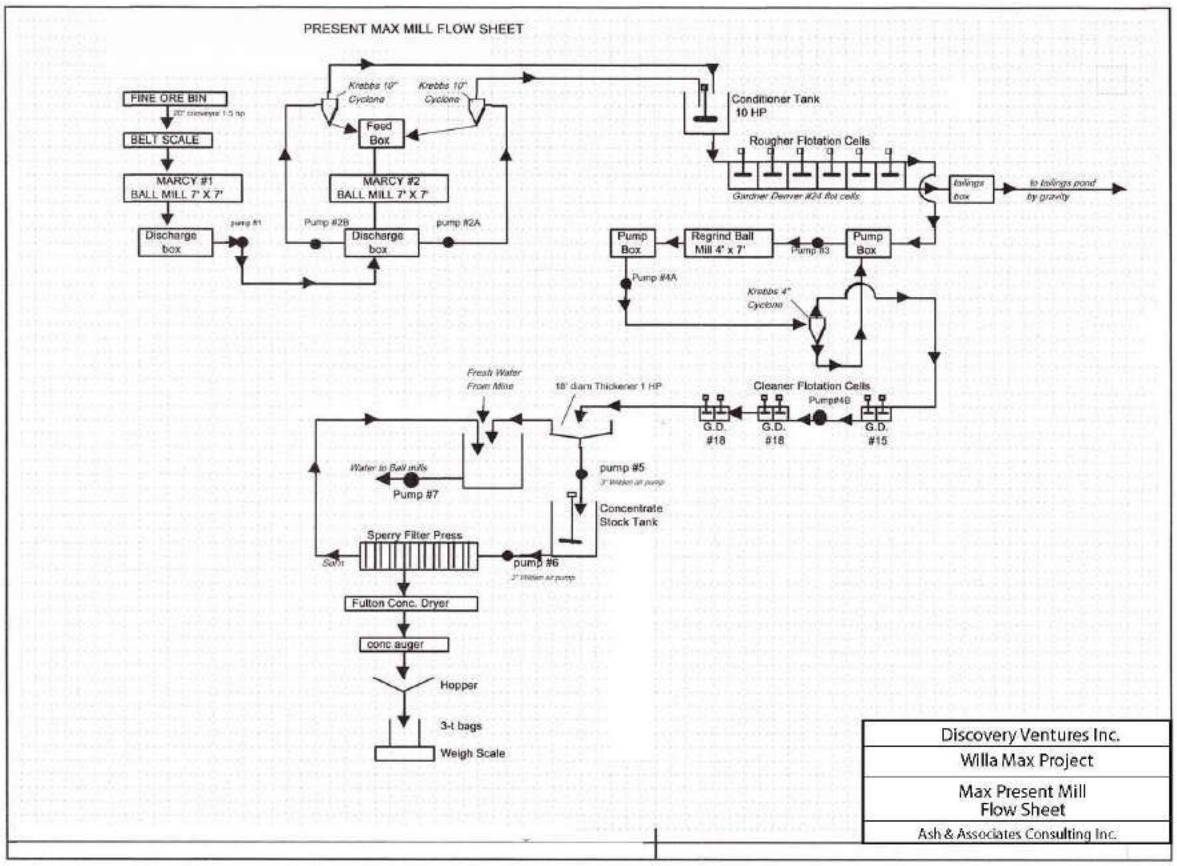
- Fulton Dryer (not used)

Other (Miscellaneous)

- Process Water Tank c/w agitator

- 3 only 2.5” (6.4 cm) Gallagher sump pumps
- Sand piper Pump (air operated)
- Overhead Crane
- 3 only Compressors for mill air (2 on line, one on standby)
- Stock tank
- 2 only Berkeley pumps for process water (1 on standby)
- Concentrate Stock Tank
- Wildon 3” (7.5 cm) Air Pump
- Concentrate Screw Conveyor (9” x 19’)(23 cm x 5.8 m)
- Concentrate Hopper & Scale
- 10 only Gallagher Agitair #48 flotation cells (each 1.13 m^3 capacity) complete with agitators and motors (Installed but never commissioned)
- 12 only Wemco #66 flotation cells (each 1.87 m^3 capacity)(require repair)

Figure 17.1 MAX Mill Present Flow Sheet



Additional Equipment Required

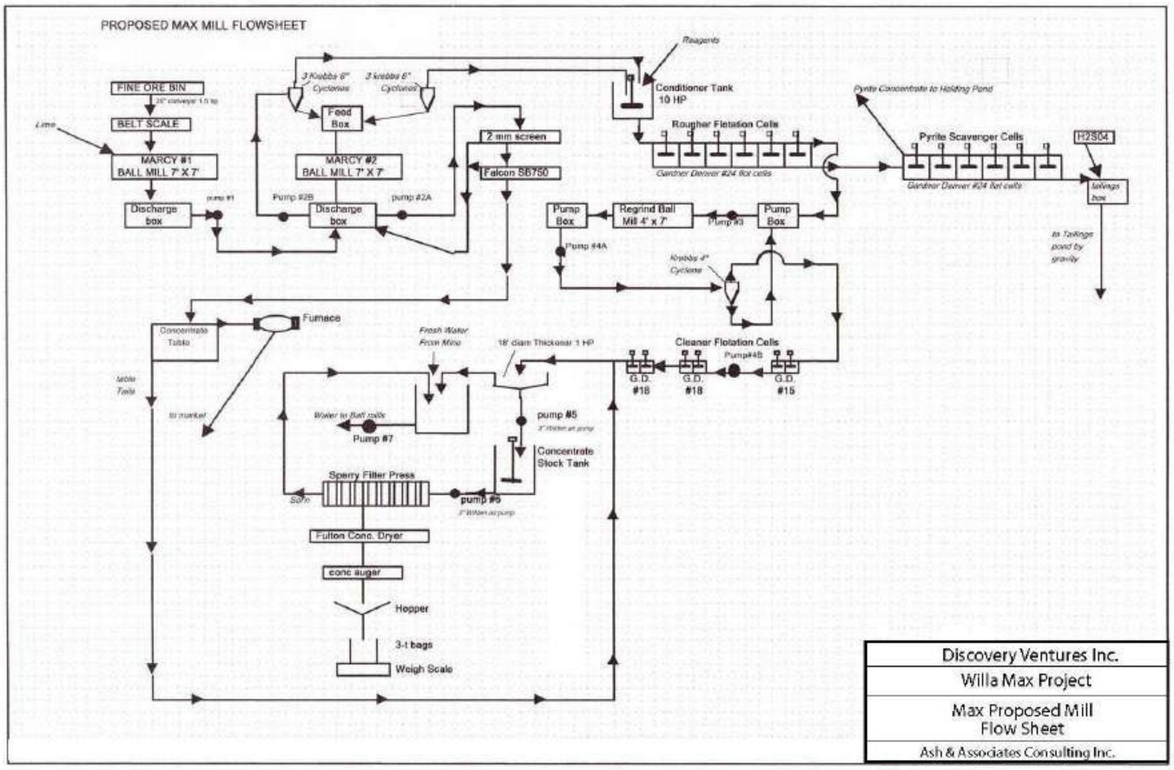
The main addition required for the mill is a gravity concentrator with high centrifugal speed (Falcon concentrator) for maximizing the recovery of the coarse gold. The methods of sampling and assaying by the smelters result in payment of the fine gold (<73 µm) but no payment for coarse gold particles in the concentrate (> 73 µm). It is therefore advisable to table off the very coarse gold from the Falcon concentrate for on-site smelting, while melding the table tails (estimated in the range of 0.75 tpd) with the copper concentrate. While the recovery of gold by the use of the Falcon concentrator is expected to be over 40%, the vast percentage of the gold in the Falcon concentrator will be extremely fine and may be combined with the copper concentrate, to be paid for by the smelter. The coarse gold content is expected to constitute no more than a few percent of the total gold content and therefore, the on-site smelter need be of very limited capacity. The table tails are expected to average approximately 0.75 tpd and are expected to reduce the grade of the copper concentrate only marginally. Assuming a flotation copper concentrate of 24.5%, the combined copper/gravity concentrate is estimated to have a dry grade of 23% Cu.

As previously mentioned with respect to the flotation circuit, Discovery Ventures also plans to produce a rougher pyrite concentrate that will serve two purposes, effectively de-sulphurization of the tailing product while offering the potential for retaining a significant percentage of gold (for later disposition). The mill is presently equipped with far more rougher flotation capacity than is required for the concentration of the copper concentrate and pyrite concentrate combined. However, additional copper cleaner cells are required.

Additions required for the processing of the Willa mineral will include:

- A Falcon SB750 high-gravity gold concentrator c/w screen, autopac, concentrate and underflow return pump, concentrate holding tank with agitator and/or air injection system, half-size Deister concentrating table, pump for pumping table tails to a table tails holding tank with agitator and/or air injection, a pump to pump the table tails to the copper concentrate thickener, a Wabi Furnace for smelting doré complete with adjuncts.
- A lime circuit for maintaining the required flotation pH in the range of 11.0 to 11.5 for collection of a copper concentrate.
- A sulphuric acid dosing system to reduce the pH to 7.5 to 8.0 for recovery of the pyrite concentrate, and to bring the final effluent to within the allowable pH discharge limit.
- Additional cleaner flotation cells for producing the copper concentrate. Note that the concentrate tonnage is expected to be approximately double that of the past MoS_2 concentrate.
- Six only 6” (15 cm) Krebb cyclones (smaller cyclones for fine overflow product are preferred to the present larger ones but more are required).
- Additional rougher flotation cells for producing a potential auriferous pyrite concentrate (presently on site).
- A high-density polyethylene-lined short-term pyrite concentrate holding pond.

Figure 17.2 MAX Mill Proposed Flow Sheet



Mill Recovery Projection

PRA (2004) results projected mill recoveries as follows:

Gold: 81.4%

Copper: 93.2%

Silver: 82%

The gold recovery projected by Winckers (metallurgical consultants, 2004) was based upon flotation plus a standard single-gravity process.

Recovery projections estimated from the 2015 metallurgical testing by Met-Solve, employing a Falcon Lab concentrator and projections on the production model (SG 750) are:

Gold: 83.9%

Copper: 90.7%

Silver: 82%.

Note that inadequate testwork has been conducted on the silver recovery by Met-Solve and in consequence, the PRA projection of 82% is used. However, for the Cash Flow Analysis, the recoveries used are:

Gold: 83.9%

Copper: 90%

Silver: 82%

The metallurgical test samples taken on behalf of Northair (1985-1988) and Bethlehem (2004) were restricted to those taken from the West Zone only. The reason was that due to the higher grade nature of the West Zone, it was actively being drilled and/or developed at that particular time. However, there is no reason to suspect that recoveries will be any different in the East Zone or from above the 1100 L.

The majority of the metallurgical testwork conducted to date was focused mainly on recovery of gold and copper since those two metals represented in the range of 97% of the recoverable $C values/t.

Process tests which were not covered in past studies by Lakefield or PRA include thickening, filtering, and environmental testwork. Based on a production rate of 400 tpd, the mill is expected to produce a tonnage of chalcopyrite concentrate of approximately 8 tonnes per day.

Thickening: The MAX mill contains two thickeners; 5.5 m (18 ft) diameter and 9.1 m (30 ft) diameter. To date, the 5.5 m diameter thickener has been used for thickening the molybdenum concentrate while the 9.1 m diameter thickener was installed but has not been operated. Testwork on thickening or filtering of the copper concentrate was not conducted. However, Winckers Associates (metallurgical consultants for Bethlehem Corp. in 2004) assessed design parameters based upon operational data from comparable applications. For thickening, they assumed 0.5 m^2/t/d. The 5.5 m diameter thickener has a surface are of over 23m^2 (a capacity equivalent 46 t/d), which allows a safety factor of 400 to 500% and is therefore of sufficient holding capacity.

Concentrate filtering: The MAX mill contains two automatic plate-and-frame filter presses each with a surface area of 34 m^2. In the past, only one has been used on a part-time basis for filtering the molybdenum concentrate. Filtration testwork was not conducted by PRA in 2004. However, based upon operating data from comparable operations, Winckers conservatively estimated a required filter unit area of 400 kg/m^2/day. The two plate-and-frame filter presses on site have a total filter area of 68 m^2 so are capable of handling 28 t/day of concentrate. Since the projected daily copper concentrate production is expect to be in the range of 10 to 14 t/day, the plate-and-frame filter-presses together appear to have a safety factor of approximately 300 % and are therefore considered to have sufficient capacity.

Environmental Testwork: Flotation of chalcopyrite and depression of the pyrite requires a pH in the range of 11.0 to 11.5, enhanced by the addition of DETA and sodium metabisulphite (compared to a near-neutral pH required for molybdenite). Then, the pH is reduced to 7.5 to 8.0 by the addition of sulphuric acid for collection of the pyrite, prior to discharging the tailings to the TDF.

Finally, testwork will be conducted on determining the effect of quick lime on the sulphate content of the MAX mine discharge water since the combination of the two may precipitate out in the form of gypsum and may lead to a reduction in the dissolved sulphate content in the mine discharge. This testwork will be conducted during the production phase.

Pyrite Disposition: Bench-scale scavenger flotation testwork was conducted by UBC students (supervised by Dr. George Poling, Head of the Metallurgical Department) in 1996 to recover the pyrite prior to discharge of the final tailings to the tailings pond.

The pyrite concentrate assayed 95 to 98% pyrite, with less than 1% sulphur remaining in the final tailings discharge. At Met-Solve (2015), the final tails consistently assayed less than 0.5% sulphur, of which approximately half was in the form of sulphate.

While the testwork conducted by UBC had a pyrite content of 10%, the average may vary from this. The pyrite content of the Intermediate metallurgical sample tested by Met-Solve had a pyrite content of approximately 7%.

Confirmatory testwork will be conducted prior to the production phase. The study will include:

a) acid-base-accounting of the final tails after removal of the pyrite,
b) assessment of the gold content of the pyrite concentrate,
c) various methods of gleaning further values from the pyrite concentrate,
d) alternatives for final disposition of the pyrite concentrate.

Water Requirements: The drainage from the MAX mine will continue to supply mill needs. The mine drainage far exceeds the MAX mill requirements.

18.0 Project Infrastructure

18.1 Willa Area

The project infrastructure includes both the Willa mine and MAX mill. They are connected by 142 km of road. Access to the Willa property is via the South Red Mountain Road, which intersects Highway 6 approximately 8 km south of the village of Silverton.

The main entrance to the mine access road is approximately 400 m from the intersection of the South Red Mountain road with the highway. The mine access road distance from the South Red Mountain Road to the main production level (the 1025 L) is about 1.2 km but is too steep for production haulage. For production haulage the road will be re-routed, widened to 9 m (complete with ditch and safety berm) and will have a length of 1.8 to 2.0 km and an average downgrade of 8%. The haulage road will be designed for one way traffic but used for both transporting mineral down-hill and supplies up-hill. Therefore lay-bys and run-away lanes will be required. All vehicles using the road will be equipped with radios for on-road communication.

Dwellings along the South Red Mountain Road are serviced by single phase power lines and telephone lines. However, communication by cell phone is limited to one small area on the 1025 L dump. Power for the Willa mine will be serviced by diesel generation near the 1025 L portal. Potable water is not available on the property and no toilet facilities have been installed.

The nearest population centres include Silverton (pop. 250, 8 km N), New Denver (pop. 650, 17 km N). The villages of Silverton and New Denver tend to be mining oriented, with a labour supply of miners and mining contractors. Larger communities in the area include Nakusp (pop. 1,600, 64 km N), Revelstoke (pop. 7,200, 162 km N) Nelson (pop. 10,000, 83 km S), Castlegar (pop. 7,800, 78 km S), and Trail (pop. 7,700, 104 km SW). Revelstoke, Nelson, Castlegar and Trail are serviced by rail transport.

18.2 Willa Mine

The total present development in the Willa mine consists of:

Section	From	to	Length (m)
1100 Level			660 m
1025 Level			1,250 m
1013 Level			160 m
Decline	1025 L	1013 L	130 m
Decline	1013 L	993 L	150 m
Vent Esc Rse	1025 L	1100 L	106 m
Vent Raise	993 L	1025 L	45 m
TOTAL			**2,385 m**

Co-author Ash was unable to inspect the 1025 L beyond a ground collapse approximately 300 m in from the portal, or the 1100 L beyond 150 m in from that portal because of another ground collapse.

The 1025 L is tracked at least to the ground collapse and the 2005 Technical Report indicated the track runs at least 1,000 m in from the portal. Where viewed by co-author Ash, the 1025 L access tunnel averaged about 3.5 m square in cross-section, while the 1100 L averaged about 3.5 m high by 3.7 m wide and appears to have been driven by trackless methods. In 2004/05, the Willa Mine was equipped with air and water lines, a surface repair shop, battery locomotive charging station, compressor house and dry/office. However, these were removed on mine closure circa 2005/06.

In 2005, both portals were well timbered and it was reported that the ground conditions inside the mine were excellent except for timbering in one short section of the 1025 L. Between then and 2014, a contractor hired by the Ministry of Mines blocked the entrances with muck, but in doing so, destroyed the portal timber and damaged the mine drainage culvert at the portal, resulting in collapse of track outside the portal. In addition to the required repair of the access to the portals, each portal will require considerable remedial work, including removal of overhanging trees and overburden from above the adits, supporting the portals with rock bolts, screen, straps and timber, plus installation of doors to prevent casual access.

In 2014, Discovery Ventures hired a mining contractor to gate the access road, re-open the portals and inspect the mine. The portals were re-opened and secured with screening in order to prevent casual mine entrance but inspection beyond the cave-in about 300 m in from the portal was not possible. The mine inspection by co-author Ash on October 29, 2015 confirmed the collapse in the

1025 L and also found a ground collapse in the 1100 L, located about 150 m in from the portal. The recent inspection showed significant deterioration of the backs (ceilings) of both access drifts. Prior to further drill testing or bulk sampling, scaling, re-rock-bolting, screening and strapping are is necessary at least in the access drifts inspected and likely throughout the mine.

No Geotechnical Study has been undertaken.

Although no water issues from the 1100 L adit, according to geologists on site between in 2003 and 2005, the 1025 L discharges approximately 25 l of water/sec and is reported to increase by as much as 50% during the spring freshet). The water discharging from the portal is channelled through a 45 cm culvert and discharges about 200 m from the portal and 125 m from Aylwin Creek. Due to the interconnection of the two main levels, a chimney effect is established and except for certain outside temperatures in the range of 6° C, strong natural ventilation occurs through the mine.

The 1025 L mine dump makes the largest footprint on the property, measuring about 0.5 hectares in area and containing approximately 60,000 tonnes of waste rock. Other than small, scattered pieces and pockets of high-sulphide-content mineral left from earlier excavations, little if any staining, (suggestive of significant oxidation) has occurred. The dump is stable and is showing signs of natural re-growth.

Access to the 1100 L is via a timbered bridge (in good condition) over upper portion of Aylwin Creek. The portal is entered though sandy glacial debris. Access was re-established again in 2014 by installation of a temporary 10-m-long x 1.2-m diameter culvert with a screen secured to the entrance to prevent casual access. Continued sloughing of the loose glacial debris above the portal is beginning to re-block the entrance. For permanent access, an excavator is required to terrace the glacial debris and the trees overhanging the slough above the portal should be removed for safety reasons. The culvert should be removed and the timber overhanging the slough used for ground support at the portal.

18.3 MAX Mine/Mill

The present infrastructure at the MAX property includes the MAX mine, mill and tailings pond.

Access to the MAX mine is closed. Rail, air pipe and cable may be recovered from the mine for use at the Willa Mine but Discovery Ventures will require a permit to re-open the mine to do this. No mining of mineral from the MAX mine will be undertaken during the life of the Willa Mine.

The MAX property presently has active permits for mining, mineral processing, tailings discharge and environmental discharge, for operations up to 1,000 tpd for the mineral of the MAX mine. Custom milling and new environmental permits will be required due to the differing chemistry of the Willa mineral and tailings, and associated reagents used in recovery of the values.

18.4 Considerations for the Mill Production

A milling rate of 400 tonnes per day is assumed for the model, for the following reasons:

- Under optimum conditions, the Willa Mine is capable of tramming out 500 tonnes per day (tpd) of muck but that includes both mineral and waste rock. Overall, the waste development is expected to constitute approximately 10% on top of the mineral production, which would effectively reduce the mineral production rate down to approximately 450 tpd.
- Transportation of muck to the mill during the spring breakup period is normally about six weeks, in which the Ministry of Highways restricts loads to 50% of the normal capacity. This could effectively double the transportation cost to over $C 50 per tonne delivered to the mill during the spring breakup period and reduce profits.
- Theoretically, the extra mineral could be hauled during the winter months and stockpiled at the MAX mill for milling operation during the break-up period. However, this is impractical for two reasons. First, the snowfall is heavy at the MAX mill and would accumulate on the stockpile, causing havoc in the crushing circuit. Second, stockpiling during the winter months would require exceeding the practical in-mine-haulage capabilities of the Willa. Therefore, the stockpiling alternative is not feasible.
- In addition to mining and development production, preparation for final stope design is dependent upon percussion drilling of intermediate holes for confirmation of stope boundaries in each stope prior to production in each stope, resulting in additional manpower requirement and interference between the production and exploration crews.
- A high mine-production rate would reduce the mine life, resulting in a short time limit for continued underground exploration.

For these reasons a production rate of 400 tpd is considered as optimal for the cash flow analyses.

18.5 Tailings Disposal Facility (TDF)

Based upon the design engineered by Klohn Crippen Berger, TDF is designed for a capacity of 1.8 million tonnes once the dams are raised to their maximum height. Based on the present heights of the dams, the TDF has a capacity to hold 600,000 tonnes of tailings, of which 480,000 tonnes of mine tailings have been deposited by Roca Mines, leaving a remaining capacity of approximately 120,000 tonnes. However, the effluent of any new tailings deposited will require time to settle, with is not included in the present remaining capacity. Therefore, for the present mineral resource of the West Zone of the Willa mine, the dams will be required to be increased in two stages over a three-year period. Due to the heavy winter snowfall, the dams must be raised during the period when the ground is devoid of snow and therefore, the first dam-raising must be done prior to full mine and mill production.

The additional infrastructure required at the MAX mill complex is restricted to the necessary equipment for the alteration from the processing of molybdenum ore to gold/copper and pyrite concentrates, and tailings facility enlargement, as specified in the Capital Cost estimate and cash flow analysis.

Metallurgical testwork conducted between 1983 and May, 2015, shows that:

- Large samples of sub-grade, low-grade and high-grade mineralization have all shown low concentrations of potentially-deleterious elements, well within government guidelines.
- An acceptable-grade gold/copper concentrate can be gleaned (24%). The copper concentrate is of particular interest by concentrate buyers due to the high content of gold (>20 g Au/t).
- The recovery of an auriferous pyrite concentrate is warranted. The recovery of the auriferous pyrite will result in very low sulphides (<0.4%) reporting to final tails. The final tailings from the metallurgical testwork indicate just slight acid-generation-potential, but less than the acid generation potential of the MAX tailings presently in the TDF. On conclusion of mineral processing, the tailings in the TDF may be encapsulated by the introduction of a bentonite clay covering.
- In LC50 tests conducted on *daphnia magna*, the tailings effluent has passed the government requirement for quality at a concentration of 78% mill effluent. Further testwork is required to ensure consistent passes for both *daphnia magna* and rainbow trout at 100 % concentration.

18.6 Pyrite Holding Facility

Several alternatives for the treatment and disposal of the gold-enriched pyrite might include (but are not necessarily limited to) any of the following possible options:

- Shipment (through an intermediate buyer) to smelters in the USA,
- Shipment to a gold mining operation in the USA (or Canada) for recovery of the gold, with disposal of the final tailings at their facility,
- Further processing on site for gold recovery, with the final tailings pumped underground at the MAX mine for disposal in the vacant stopes
- Processing on site with the final tailings encapsulated for disposal on site.

For the first year to year-and-a half of production, the auriferous pyrite concentrate will be stored in an on-site HDPE-lined, water-covered, short-term holding pond. These and other potential options will be investigated during that period to determine the most economically viable alternative.

19.0 Market Studies and Contracts

A market study was conducted by co-author Ash, based on 2012 data obtained from Wood Mackenzie (WM), an International-based research and consulting firm dealing in the fields of Energy, Mining and Metals.

19.1 Copper Concentrate Analysis

From the metallurgical testwork conducted by G. Hawthorn, Process Research Associates Ltd. and Lakefield Research, the copper concentrate is expected to have the following approximate analysis as shown in Table 19.1:

Table 19.1 Copper Concentrate Analysis

Element	*Grade*	*Element*	*Grade*
Cu	20 to 26 %	P_2O_5	0.31 %
Au	120 to 180 g/dmt	SiO_2	3.21 %
Ag	200 to 300 g/dmt	TiO_2	0.04 %
Al_2O_3	0.8 %	Bi	< 2 ppm
S	34 %	Cd	0.4 ppm
Fe	20-25 %	Hg	< 3 ppm
CaO	0.4 %	Sb	< 5 ppm
MgO	0.13 %	Se	132 ppm
Zn	0.33 %	Ni	62 ppm
BaO	0.01 %	Zr	10 ppm
Pb	0.005 %	As	41 ppm
K_2O	0.03 %	Cl	9 ppm
MnO	0.01 %	F	297 ppm
Na_2O	0.49 %		

The final tailings (after recovery of the copper and pyrite concentrates from the bulk sample), which were taken by Ash from the Lower Mineral Dump at the Willa mine in December 2013) returned the following elements of potential concern:

Table 19.2: Final Tailings Content from 2014 G&T Metallurgical Test

Element	Grade	Element	Grade
Ag	0.94 ppm	Mo	63 ppm
As	4.3 ppm	Pb	8.1 ppm
Bi	0.31 ppm	S	0.80%
Cd	0.18 ppm	Sb	1.7 ppm
Cu	607 ppm	U	0.7 ppm
Hg	0.02 ppm	Zn	59 ppm

The metallurgical testwork conducted indicates that the recovery of both copper and gold tend to drop off above a concentrate grade of 25%. In the cash flow analysis, it is assumed the concentrate will contain 23 to 24% copper (with the inclusion of the table tails of the gravity process), with a gold content in the copper concentrate grading approximately 100 g/t.

The gold content of copper concentrate ranks it in the 97 percentile range for the 190 mines surveyed worldwide. In general, with most smelter schedules, a gold content of less than 1 g/t in the concentrate is not paid for. Thirty-five percent of the 190 mines listed by WM graded less than 1 g Au/t. At a gold price of $US 1200/oz, one g Au/t is worth in the range of $US 38.50 ($C 50/oz). Thus, while the copper content is below the world average, the high gold content makes it an ideal concentrate for blending with low-grade gold values from other mines.

19.2 Smelting Penalty Charges

In addition to the treatment and refining charges (TC/RC) many smelters impose penalty charges for certain deleterious elements contained in the concentrate. Globally, typical penalty thresholds for various elements are shown in Table 19.2:

Table 19.3 Penalty Elements (in $US)

Element	"Free" Limit	Penalty Cost Range
As	2000 ppm	$1.5-$2.5 /%
Al2O3	3%	$1 to $2 /%
Sb	500 ppm	$1.00 to $2.00 /100 ppm
Bi	200 ppm	$1.50 to $3.00 / 100 ppm
Cd	300 ppm	$1.00 to $5.00 / 100 ppm
Cl	300 ppm	$1.00 to $3.00 /100 ppm
F	300 ppm	$1.00 to $2.00 / 100 ppm
Pb	1.00%	$1.00 to $5.00 / %
Hg	5 ppm	$0.10 to $5.00 / %
Ni	0.50%	$1.00 / 0.1%
Se	300 ppm	$1.50 / 100 ppm
SiO2	10.00%	$1.00 / %
Zn	3.00%	$1.00 to $5.00 / %

On this basis, the Willa concentrate will not be penalized for deleterious metals.

19.3 Treatment (smelting) and Refining Charges

According to the WM data, approximately 44% of copper concentrate production was consumed by integrated companies (in-house) in 2011 while the remaining 56% of all copper-in-concentrate products were sold as custom concentrates to third parties (traders) on the spot market.

Major companies with large open pit mines and mine life-spans measured in decades generally negotiate long-term contracts directly with smelters. However, it is common for custom concentrate sellers to commit no more than 80-90% of their annual output under long-term contract. In this manner, unforeseen production disruptions are unlikely to affect deliveries under these arrangements. Any residual output is typically sold in the spot market, on a tender basis. The bulk of the bidding is done by traders.

Companies producing low annual tonnages of concentrate, such as would be the case of Discovery Ventures, almost invariably sell their concentrates through traders. The largest traders are shown below:

Table 19.4 Largest Traders

Company	kt/annum
Trafigura	2,300
Glencore	1,700
L. Dreyfus	850
CWT (MR)	800
JP Morgan	800
Transamine	350
BMAG	350
Ocean Partners	300
Gerald	200
Traxys	150

Trafigura, the world's largest trader of copper concentrate, has a warehouse in Richmond, BC where the concentrates from different Western Canadian mines are blended prior to shipping the blended concentrate to smelters in Mexico or Asia. Small sellers, such as those which would include Discovery Ventures, normally ship their concentrate in two-tonne bags (Cost per bag $C 31 each). Assuming moisture content of 8%, this equates to 1.85 dry tonnes (dt) per bag (an additional cost in the range of two cents per contained lb of Cu). Negotiations with a local trucking firm have resulted in a budget cost estimate of $C 85 per dt of concentrate for shipment to Vancouver (including tote bags).

Trafigura has issued Discovery a general smelter schedule for the purchase of the gold-bearing copper concentrate. Under normal circumstances, copper concentrates are blended with copper concentrates from other Canadian mines at the depot prior to shipment of the concentrates in bulk form to destinations in Asia or Mexico. However, due to the desirability of the high expected gold content in the Willa copper concentrate, the concentrate delivered in any one-month period will be shipped in the 2-t bags directly to Korean and/or Japanese smelters and refiners.

In early 2014, when the world price of copper was $US 3.30 /lb, co-author Ash received a general smelting and refining schedule issued by Trafigura to Discovery, as follows:

1. **Shipment to Vancouver:** Discovery pays for the shipping of the concentrate in bags and insurance from the MAX mill to the Richmond depot, either in rail cars or trucks.

2. **Payments:** For copper concentrates grading above 20% Cu, the payment will be 96.5% of the copper content at the world price (subject to a minimum deduction of one unit off the percent of copper in each dt). For the gold content, the payment is 97.5% of world price, with a deduction of $US 10/g Au. For the silver content, the payment is 90% of world price subject to a minimum deduction of 1.0 oz per dt.

3. **TC/RC (Treatment Charge and Refining Charge):** The treatment (smelting) charges generally rise and fall with the price of copper. In January, 2014, when the price of copper was at $US3.31, Trafigura quoted to Discovery a general TC of $US 220 per dry tonne, plus a refining charge (RC) of

$US 0.22 per payable pound of copper. This included insurance, handling and shipment from the Richmond depot to the Asian smelters and refiners. By October 2015, when the world price of copper had dropped to $US 2.32/lb, the Trafigura agent quoted a TC of $US 200 /dry tonne and an RC of $US 0.20/lb.

4. **Initial Payment:** Trafigura has offered payment of 90% of the estimated value (based on the supplier's assays) for all concentrate received in Vancouver during one calendar month, with payment within three working days after month end. The final 10% is paid within two months after delivery to Vancouver and is based upon the smelter's final assays and taking into account any penalties and NSR royalties.

Note that all TC/RC are in $US and while transport to Vancouver is in $C. The US dollar is presently approximately 30% higher than the Canadian dollar.

In order to establish an overall, realistic value of the precious and base metals in the stopes, a calculation was made based on a one-year average metals prices of $US 1200 /oz Au, $US 2.75 /lb Cu, and $US 16.50 /oz Ag, and a $US at $C 1.30. The calculation took into account:

- Milling losses
- Projected tonnage and grade of concentrate
- Transport of the concentrate to the Trafigura out-loading facility in Richmond, BC, smelting and refining costs, deductions and payments
- NSR royalty to Willa property Vendors

The Canadian dollar value for each metal in the minerals shipped to the mill was determined as:

- Gold: $C 38.77 /gram
- Copper: $C 48.10 per percent
- Silver: $C 0.47 /gram

The projected recoverable gold by a combination of gravity and flotation presently constitutes approximately 80% of the revenue of the operation as compared to 18% of the revenue from the copper and 2% for silver.

Contracts

General quotes have been given by various contractors including:

- Trafigura (Concentrate TC/RC)
- Mining Contractor (unit cost for driving decline)
- Quadra Chemicals (milling reagents)
- Nakusp Sand and Gravel (Mineral transport from Willa to MAX)
- Jade Line Holdings (copper concentrate shipment MAX to Richmond)
- Orica Canada (Explosives)

No firm contracts have been negotiated

20.0 Environmental Studies, Permitting, and Social or Community Impact

20.1 Solution Characteristics

Due to the dual-property nature of the WILLAMAX Project, three separate types of effluent are considered:

At the Willa mine

- ARD Potential due to percolation of precipitation through the dumps,
- Mine seepage water with low suspended solids content issuing from drillholes and excavations in the mine,
- Mine seepage water with high suspended solids content (due to mining activities).

At the MAX mine

- Mine seepage with insignificant suspended solids (since mining activity will not be conducted during the operating life of the Willa mine).
- Mill effluent discharging to the environment.

20.2 Willa Waste Dumps & Mineral

Assessment of ARD Potential of Mine Waste Rock

It is expected that the great majority of the waste rock on the 1025 L and 1100 L dumps will have low potential for acid generation (PAG), but that the ARD potential may increase In order to determine this during the pre-production phase, samples of waste rock be taken at 20-m intervals throughout the present mine workings (total 125 samples), with sample locations marked and categorized as to the six main rock types, for future reference.

Each of the samples will be analysed for carbon and sulphur by a commercial lab, using a Leco carbon/sulphur analyser. The assay results of each will be compared with the proximity to the West Zone in order to assess whether there is a definitive increase in ARD potential with respect to the proximity of the mineral zone. Sub-samples of the ground samples of each rock type will then be combined. The homogenized samples will then be re-analyzed for carbon, total sulphur and sulphate in order to establish a better estimate of the PAG of the existing mine dumps.

Mitigation of Waste Dump ARD

Due to the expected low ARD potential from the great majority of the waste rock at the Willa mine, all waste rock development muck produced during the pre-production phase and the first year of production which shows significant PAG will be stored on the present waste dump. However, it will be returned underground and dumped in the empty stopes below the 1025 L once those stopes have been mined out. All waste muck produced after the completion of mining of those stopes showing PAG will be dumped directly into the open stopes rather than storing it temporarily on surface. Note that on mine closure, the mine seepage will fill all waste-filled stopes to the 1025 L, thereby permanently encapsulating it all in a subaqueous state.

The present mine dump at the 1025 L portal is required for the tipple and haulage truck loading facilities. On mine closure, all surface structures and waste dumps will be reclaimed. The dumps will be sloped to shed water, covered by HDPE sheeting and topped by a minimum of 0.5 m of tamped boulder till, topsoil and seeded. Thus mitigating or eliminating potential ARD by encapsulating the waste dumps.

20.3 Underground Control of Potential ARD in Mine Seepage

In addition to the planned stopes below the 1025 L, open stopes are also being planned for the zones between the 1025 L and 1100 L. Depending upon seepage water sources and courses, these seepage channels might eventually lead to the significant production of ARD from the stope walls and minor broken muck remaining in the stope. This situation will be monitored during the development and production stages. Should it be found that these seepages significantly adversely affect the long-term quality of seepage water issuing from the 1025 L adit, a bond will be put up prior to mine closure for the installation of a concrete bulkhead in the 1025 L adit. Once installed, the water will continue rising behind the bulkhead until it rises to the 1100 m elevation, at which time it will issue from the 1100 portal instead, thus submerging the stopes above the 1025 L. Due to the reduced area from which the mine drainage water originates, the volume of water discharging from the 1100 level is expected to be radically reduced in volume and contaminant load. All diamond drilling conducted from surface was collared well above the 1100 m elevation so no issuance from abandoned drillholes is possible. No firm plans have yet been made to develop stopes above the 1100 m elevation. However, should that be case, and should the long term mine discharge indicate a potential significant ARD hazard, the 1100 adit will likewise be bulkheaded, with a drain pipe extending in the mine to the highest level mined.

20.4 Willa Mine Drainage Volume

According to the 2005 Technical Report, the discharge from the Willa mine apparently varies from 25 litres/second ("l/sec") to 40 l/sec with the surge occurring during the spring freshet. Since the mid-1980s, the mine effluent has been discharged onto a side hill approximately 100 m from Aylwin Creek, eventually reaching the creek about 300 metres downstream. Note that the slope of Aylwin Creek is too steep for the habitation of fish.

a) During production, two classes of mine seepage will exist:A significant percentage of the mine drainage apparently issues from the ventilation raise connecting the 1025 L and 1100 L, plus from several drillholes. The analysis during periods when actual development is not being conducted, show it to be almost non-existent in suspended solids and to have acceptably low deleterious substances to allow for discharge directly into Aylwin Creek. The company will set up systems to capture this water and discharge it separately via a pipeline to the relatively flat area adjacent to the Red Mountain Road where it will be distributed over a wide area for percolation to ground.

b) During the production phase the seepage water which cannot be captured at its source, is expected to contain significant suspended solids due to mining activities. This will be captured at the portal and piped downhill to the relatively flat area near the Red Mountain Road where it will be discharged to a settling pond. The pond-overflow will be re-captured, piped further downhill and will likewise be distributed to ground through a perforated horizontal poly line.

c) The distribution lines in both cases will be drilled at regular intervals with 2 to 5-mm diameter holes in order to spread the discharge over a wide area, allowing it to drain through the soil before entering the creek. Should the settling pond continue to contain unacceptably high suspended solids, flocculent will be added to the flow immediately prior to entering the pond.

Nitrate and ammonium are expected to increase in the mine (as with most mines), due to the use of ammonium nitrate explosives. In order to mitigate this situation, utmost care will be taken to maintain good "house-keeping practices" to ensure a minimum wastage of ANFO-based explosives.

20.5 Mill Tailings Disposal Facility (TDF)

MAX Mine Area Environmental Considerations

Permits for the MAX Mill are presently in place for discharge of effluent based on mill production of both 500 tpd and 1,000 tpd.

Annual Water Quality and Biological Monitoring programs have been filed with provincial and federal regulators and the site has been in compliance during and post operations.

Water quality requirements specified for the MAX Effluent Discharge Permit (PE-18167) for Site I and Site TP is as follows:

Parameter	Maximum	30-day Average
Total Suspended Solids (TSS)	30 mg/L	<15 mg/L
Molybdenum (dissolved)	3 mg/L	
Iron (dissolved)	1 mg/L	
Ph	6.5 – 9	
Toxicity	100%1	

[1] *Effluent must pass 96-hr LC50 rainbow trout and 48-hr LC_{50} Daphnia magna toxicity tests.*

Site I represents the sampling site for effluent from the mine portals. Discharge from Site I enters Wilkie Inlet D downstream of Site B. Monitoring of this site has been on-going since 2007. The permit allows for an average mine discharge of 90,000 m^3/month (23 l/sec, with a maximum of 180,000 m^3/month (46 l/sec.)

Site TP represents the sampling site for discharge from the present tailings facility, located at the head of Shrub Creek. Water from the tailings facility was discharged to the environment in January, March, April, November and December of 2010. The permit allows for a monthly discharge of 98,000 m^3/mo, with discharge permitted 8 months of the year.

During the operation of the MAX mine and Mill, acute toxicity testing was carried out on samples for Site I and Site TP, with all samples passing. At 100% effluent concentration, all tests had a survival rate of >90% at their respective end points; 96 hours for rainbow trout and 48 hours for *D. Magna*. Between 2007 and December 2010, 30 samples were submitted for toxicity testing, all of which passed, indicating that the effluent is not directly toxic to either test species.

During the productive years (between 2007 and 2011), the MAX mill used a portion of the MAX mine effluent discharge for mill water. The rest of the mine discharge was piped to a settling pond before discharging into the same subsidiary stream of Wilke Inlet.

The ideal NP/AP ratio for tailings (ratio of neutralizing potential to acid-producing potential) is 4.0 or higher. The average NP/AP of the four Willa flotation tails samples assessed in the Met-Solve 2015 testwork is 3.1. This is considered close to ideal since few metals mines produce tailings with as low sulphide content. Furthermore, the tailings will be stored under water in the tailings pond and finally, at the end of production, the tailings may be capped with an impervious veneer cover (such as clay). Therefore, the tailings can be considered benign.

LC 50 Test of *Daphnia Magna*

The tailings pond supernatant resulting from processing the Willa mineral, will be discharged into a trout-bearing creek. The toxicity is determined by a 96-hour LC50 for trout and 48 hours for *daphnia magna*. The *daphnia magna* (water lice), which are a major food source for trout fingerlings, are far more susceptible to toxicity than trout. The latest test was conducted on the solution from the OX200 series. The test passed in an effluent concentration of 78%. In an effort to ensure that the final effluent reporting to the environment is non-toxic to aquatic life at 100% concentration. On-going testing will be conducted on other collectors and settling tests will also be conducted with various flocculants and coagulants to ensure that the tailings will settle out in the TDF.

Figure 20.1 General Plant Layout

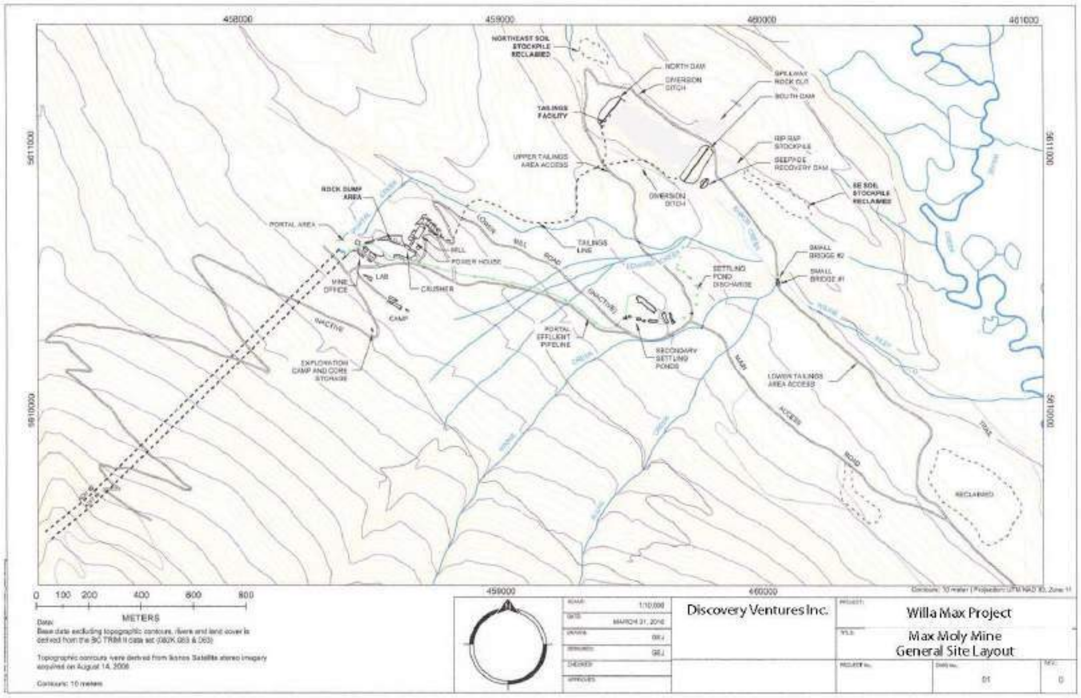


Figure 20.2 Tailings Pond Plan View

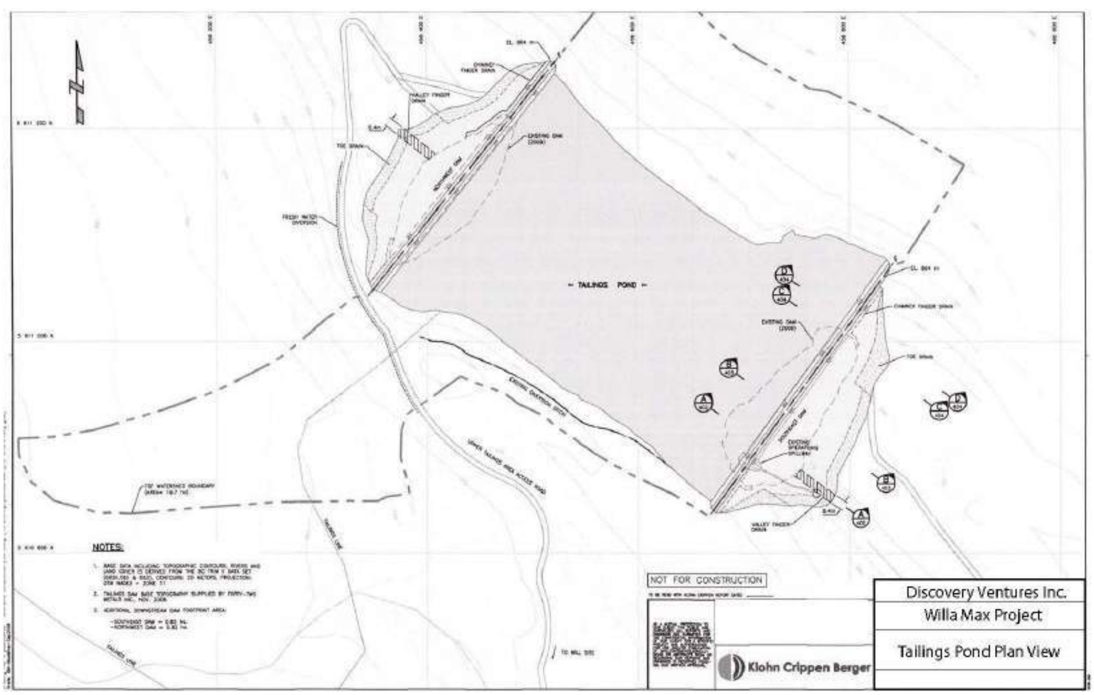
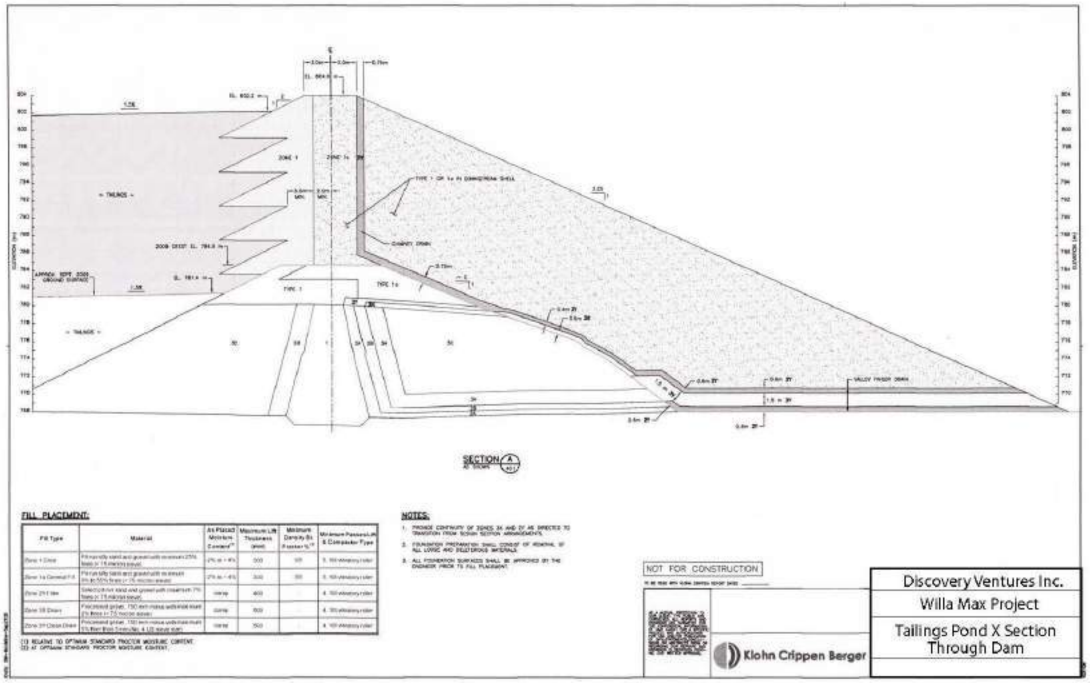


Figure 20.3 Tailings Dam X-Section



The British Columbia Ministry of Energy & Mines (BCMEM) approved the initial design of the TDF for a holding capacity of 600,000 tonnes of MAX Mine tailings sub-aqueously. It also approved the 2009 Klohn Crippen engineered design for increasing the dam height in stages to accommodate 1.8 million tonnes of tailings.

To date, the MAX mine has produced 482,000 tonnes, leaving a present, unused capacity in the TDF of roughly 120,000 tonnes of tailings (approximately 10 months at full production of 400 tpd of Willa mineral). Deposition of the Willa tailings will require increasing the dam height on a yearly basis, as per design, of which the first stage must be done commencing during the summer months of the pre-production phase.

Discovery Ventures has opted to scavenge the pyrite from the flotation tails prior to discharge of the tails to the TDF, thus reducing the sulphur (as sulphide) to less than 0.4%.

20.6 Mitigation of ARD Affects Due to Sulphide Content of Willa Mineral

While theoretically, government approval should be secured for Discovery Ventures to discharge all Willa tailings in the MAX tailings pond, the company has opted (as noted above) to float off the remaining pyrite as a concentrate after recovery of the copper concentrate, thus, mitigating (or eliminating) potential for ARD of the tailings in the TDF.

Several alternatives are available for disposal of the pyrite product, depending upon the content of gold associated with the pyrite concentrate. The final disposition will only be determined after production commences and therefore, the pyrite concentrate will be deposited sub-acqueously in a separate, short-term, poly-lined holding pond. The installation of the pyrite holding pond has been included in the project capital cost.

The pyrite holding pond will be constructed so that, at the end of the life of the mine, the solution segment will be allowed to drain into an intermediate well. The effluent will be extracted, treated to remove contaminants, and discharged to the main TDF. Once the interstitial moisture in the solids portion has dropped to a low level, the pyrite holding pond will be filled with till, contoured to a gentle out-slope and covered with an HDPE cover. The HDPE will then be covered by a minimum of 0.5 m of till and topsoil, and seeded, thereby encapsulating the pyrite.

20.7 pH Control

For the flotation of molybdenite, a pH in the range of 7.5 to 8.5 is optimal, resulting in an effluent which, in the past, met the environmental discharge guidelines of a 6.5 to 8.5 pH range. The pH required for flotation of chalcopyrite (and depression of the pyrite) must be brought up to the range of 11.0 to 11.5 by the addition of calcium hydroxide. In order to recover the pyrite, the pH must be reduced back to the range of 7.0 to 7.5 by the introduction of sulphuric acid. The majority of the sulphuric acid is expected to combine with the calcium hydroxide to form gypsum (which will tend to precipitate out in the TDF).

20.8 Permitting

Permits though the BC Ministry of Energy and Mines have been issued for the following:

For Willa Deposit:

- An amendment to the Exploration Permit was issued to Discovery Ventures on September 18, 2015 by the Ministry of Energy and Mines, extending the permit expiry date to December 31, 2017.
- A 10,000 tonne Bulk Sample permit was issued to Discovery Ventures on November 2, 2015 by the Ministry of Energy and Mines, with an expiry date of August 16, 2020.

Additional permits will be applied for, including:

- Small Mine Permit (allowing for up to 75,000 tpy of production)
- Mine permit (for production of 500 tpd).

For the MAX Mine:

The Production Permit for 1,000 tpd is in place but an amendment will be required to allow for the processing of off-site mineral (specifically, the Willa mineral).

Work on the Property needs to follow the Mines Act, the Health, Safety and Reclamation Code for Mines in British Columbia, and the Handbook for Mineral & Coal Exploration in British Columbia.

20.9 Socioeconomic Considerations

Community Considerations including that of First Nations were studied by Jo Harris and Associates (2003) relative to mining the Willa deposit and processing at Bethlehem's Goldstream plant. The study is well documented and extensively referenced. The following conclusions are paraphrased from the report:

"The Goldstream Willa Gold Project does not have the capacity to negatively impact First Nations. Accordingly, no additional ethnographic or archeological research is required or anticipated in relation to the Project. Conversely, the Project represents a potential opportunity for First Nations' members to participate in employment and contracting opportunities."

"Overall, the socioeconomic impact of the project will be extremely positive. The most significant impact will be the positive effects associated with employment creation. The in-migration of educated and experienced employees in the area will enhance the work-skills base in the region. The required housing, emergency, health and education services are in place. Property taxes associated with the project will assist in maintaining these services, and in turn will act to offset population drift. The project is clearly pivotal, as no other economic initiatives are anticipated at this time."

Open house meetings were held in Silverton and Revelstoke in 2003 and again in 2004 for presentation of the proposed mining at the Willa and the re-start of Bethlehem's Goldstream mill north of Revelstoke. According to the NI 43-101 Technical Report of 2005, the presentations received a very positive response from the public.

However, as the processing site will now be at Trout Lake, open house meetings will be held during the pre-production stage, in Silverton, New Denver, Nakusp and Trout Lake. The Silverton/New Denver area has a long history of mining and will benefit directly from the Willa operation. The re-start of the process plant at the MAX mine will directly benefit Trout Lake. Communications will also be arranged with all First Nations bands which may be affected by the project.

20.10 Willa Mine Closure

The present development muck dump from the 1100 L is estimated at 20,000 t. It has been placed adjacent to Aylwin Creek. Acid Base Accounting (ABA) testwork conducted on the various waste rock types indicate an average Net Neutralization Potential in the range of -57 kg/t. It is therefore considered-mildly acid producing. On closure, the company will slope the surface of the dump where practical, cover it with a HDPE cover and a tamped 0.5-m thick layer of boulder till to prevent downward percolation, followed by a veneer of topsoil, plus and re-seeding to government specifications.

The tonnage on the present mine dump at the 1025 L portal is estimated at 60,000 tonnes. The dump is required throughout the pre-production and production phases for a tipple base, truck reloading system and surface facilities. On mine closure, the same re-contouring and dump-encapsulating system will be conducted on this as on the 1100 L dump. Finally, the access road will be re-contoured, ripped and planted with government approved seeding.

As mining will also be conducted above the 1025 L, there is a possibility that with time, water percolating through the open stopes above this level could result in ARD/ML. Should this occur, DVN will deposit a bond for the construction of a concrete bulkhead in the 1025 L adit, allowing the water to build up behind it and discharge through the 1100 L portal.

20.11 MAX Mine Closure

The MAX mine presently has an approved Mine Closure Plan. The MAX Mine presently embraces a mineral resource exceeding 40 million tonnes. When the Willa mine is mined out, the mill may be maintained in an operable state, for potential for continued mining of the MAX mine when the price of molybdenum improves, or for the exploitation of other local deposits.

21.0 Capital and Operating Costs

21.1 Capital Costs

The capital cost includes:

- Acquisition of the Willa property
- Acquisition of the MAX property
- Administration and exploration costs to date
- Administration from the present time to commencement of production (assumed to be twelve months).
- Capital cost at the Willa mine, including (but not limited to) rehabilitation of the Willa mine, installation of surface and underground facilities, the extraction of the 10,000 tonne bulk sample, and stope development required to allow mine production to commence.
- Since the 10,000 bulk sample will be milled during the pre-production period, the costs include transportation of the mineral to the mill, milling it, and the NSR due to the Property Vendors. However, the shipment of the copper concentrate, smelting and refining are included in the in-situ value of the mineral.
- The stope development will result in approximately 23,000 additional tonnes of broken mineral being produced. It has been included as being shipped to the mill but is assumed to be stockpiled there (but not processed).
- The capital cost at the MAX project includes the purchase cost of the MAX mine and mill, the outstanding financial obligations with respect to the purchase (such as debts owed by FortyTwo Metals to creditors). It also includes repairs to the present milling and auxiliary complexes, acquisition and installation of the additional equipment required, the two phases of dam raising of the tailings disposal facility (TDF) with holding capacity for the present projected Willa mine life, and construction of the pyrite holding pond.

Mine development costs incurred during the production phase are considered part of the operating costs rather than capital costs.

The capital costs for the project are tabulated in Appendix 1.

Caution: There are no certainties that:

- A permit for production will be issued to Discovery Ventures for production (other than for the 10,000 tonne bulk sample, issued November 2, 2015).
- A permit will be issued to allow the Willa mineral to be processed at the MAX mill.
- Other permits will be issued.
- Sufficient funding can be arranged to allow for the Willa mine to be brought into production.
- The mine and milling complexes can be production-ready within the specified twelve-month time frame.

The Capital cost for the Willa mine is projected at $C 13,962,000 (**see Appendix 1, Table 2**). The Pre-production head office administration expense (12 months) (**see Appendix 1, Table 1**), is spread over the pre-production period and is charged to the Willa capital cost rather than distributing it over both the Willa and MAX capital cost.

Phase 1 for the Willa mine includes:

- Administration (6 months)(**one half of Appendix 1 Table 1**)
- Mine rehabilitation
- Underground development and Installations
- Surface construction
- Short-term access road construction
- Buildings & mobile Equipment
- Recovery of a 10,000 tonne bulk sample
- Shipment of the bulk sample to the MAX mill
- Processing the bulk sample at the MAX Mill

Phase 2 for the Willa mine will include preparation for full production and will include:

- Administration (6 months)(**one half of Appendix 1, Table 1**)
- Additional Equipment installation
- Additional underground development in preparation of the stopes below the 1025 L for production.

The capital cost for the MAX operation (**Appendix 1, Table 3**) is estimated at $C 7,319,000 and includes:

- Acquisition of the MAX property
- Outstanding MAX financial obligations
- Milling, laboratory, and bullion refining purchases
- Building Repairs
- Equipment Repair
- New equipment installations
- Dam raising of the present TDF (including the first lift during the pre-production phase and the second lift during the production phase)
- Installation of the auriferous pyrite holding pond.

21.2 Operating Costs

A general cost of $C 2,000 per m is assigned for all mining development, which was received from local mining contractor, based upon driving of a decline for LHD haulage. This price is assumed for all development including track-drift, sub-level drift, and Alimak raising.

A general contract of $C 28.00 per tonne was quoted by Nakusp Sand & Gravel for transportation of mineral from the Lower Dump of the Willa Mine (to be located adjacent to the South Red Mountain Road) to the MAX Mill. The price was given in early 2014 and although the inflation to 2016 is projected to be approximately 5%, the price of diesel fuel has decreased from $C 1.20/l down to $C 1.00 or less per litre. Therefore, the price of $C 28 is assumed to remain valid.

The following unit costs are based on a mining and milling production of 400 tonnes per day (146,000 tonnes per year):

Table 21.1 Summation of Unit Costs During Production Phase

Based on production of 400 tpd or 146,000 tonnes per year

Category	Unit $C/t	Total $C/t
General Administration	**5.89**	**$5.89**
Willa Unit costs		
Supervision	4.49	
Stope Drilling and Blasting	10.48	
Muck Haulage to 1025 Portal	7.36	
Muck Haulage to SRM Road	2.08	
Total Unit Mining Cost		**$24.41**
Haulage SRM Rd to Max Mill		**$28.00**
Max Unit Costs		
Salaried Personnel	4.48	
Hourly Labour	15.04	
Consumables	7.42	
Power	12.53	
Total Unit Milling Costs		**$39.47**
TOTAL OP COST PER TONNE		**$97.77**

22.0 Economic Analysis

The economic analysis applies to the potentially-economically-viable Measured and Indicated mineral resources in the eight designated West Zone stopes only, between 970 and 1100 m elevations.

It does not include Inferred resources.

It does not include the mineral resources of the North and East Zones, which together consist of a base tonnage and grade of:

- Indicated category 90,000 t @ 5.04 g Au/t, 0.66 % Cu and 8.0 g Ag/t
- Inferred category 83,000 t @ 4.22 g Au/t, 0.72 % Cu and 8.4 g Ag/t

These have been neglected for the present economic assessment because the drilling conducted to date is too wide spread to form a Measured Resource category, and additional diamond drilling is warranted to extend the zones and/or upgrade more of the Inferred category to Indicated or Measured.

The potentially-economically-viable West zone tonnage and grade includes only those portions which are amenable to low-cost underground mining techniques (long-hole retreat mining). For simplicity in the cash flow analysis, the tonnage and grade used for the cash flow analysis is the average of all the eight projected stopes. The average tonnage and grade consists of:

- All blocks of mineral within the stope boundaries, with an in-situ value grading $C 100/tonne or greater, plus
- The internal dilution within the stope boundaries (consisting of those blocks of mineral grading less than $C 100 per tonne), plus
- The external dilution (assuming 5%) and based upon the average grade of the internal dilution, and also assuming:
- A mining recovery of 95% of the average diluted tonnage and grade.
- A one-year average world price for gold at $US 1200 / oz
- A one-year average world price for copper at $US 2.75 / lb
- A one-year average world price for silver at $US 16.50 / oz
- An American dollar at $C 1.30

The tonnage and average grade for the diluted, recoverable mineral from the eight stopes is estimated at 617,460 tonnes grading 4.51 g Au/t, 0.81 % Cu, and 8.39 g Ag/t, with an in-situ value of $C 217.58 **(see Table 16.2).**

The in-situ grade is defined as the value per tonne after taking into account:

- The estimated milling recovery of each mineral
- The shipment of concentrate from the mill site to Vancouver
- The shipment of the concentrate from Vancouver to Asia
- Smelting and refining costs, deductions and payments as per the buyer’s general contract
- NSR royalty payment to the claim holders of 2.5%.

From the resulting in-situ value, the unit operating cost must be subtracted, with the remainder equivalent to a pre-tax profit (after deduction of the capital cost).

The cash flow analysis assumes a daily mine and mill production rate of 400 tonnes per day (146,000 tonnes per year).

The metallurgical testwork conducted by Met-Solve in 2015 indicates a projected average recovery for the Intermediate grade mineral of 83.9% for gold and 91.2% for copper. However, the cash flow analysis assumes milling recoveries of 83.9% for gold and 90% for copper. Insufficient testwork has been conducted by Met-Solve on recovery of silver. Therefore, a recovery of 82% is assumed, based upon metallurgical testwork conducted prior to 2005. Note that silver contributes less than 3% of the total in-situ value of the mineral.

Based on a mine and mill production rate of 146,000 tonnes per year, the capital cost for the project is estimated at approximately $C 21.3 million, of which approximately $C 3.0 million has been spent to date. The revenue is estimated at $C 140.3 million, the operating costs at $C 62.5 million, cumulative cash flow at $C 56.1 million, the after-tax Net Present Value (based on a 10% discount rate) at $C 38.7 million, and the after-tax Internal Rate of Return at 83%. The mine has an estimated lifespan of 4.2 years.

Due to the accumulated tax pools of about $C 47 million at the MAX operation (which was included as part of the purchase price of the MAX property), it is anticipated that there will be no income tax payable over the (present) 4.25-year life of the Willa mine.

Table 22.1 Dollar Values for Minerals			151116		
American dollar in Canadian Funds	$1.30				
GOLD DOLLAR VALUE PER GRAM			**$US**		**$Cdn**
Gold Word Price per oz			$1,200	/oz	$1,560
Grams per oz	31.1035	g/oz			
Gross value of 1 g Au			$38.58	/g	$50.16
Mill recovery assumed	82	%			
Gold recovered from mill			$31.64	/g	$41.13
Smelter deduction			$10.00	/oz	$13.00
value per oz left			$1,190	/oz	$1,547
Value of g left			$31.37	/g	$40.78
Smelter pays of world price	97.5	%			
Smelter return on gold			$30.59	/g	$39.76
NSR to Property Seller	2.5%				
VALUE IN-SITU PER g Au			**$29.82**	**/g**	**$38.77**
COPPER VALUE PER PERCENT			**$US**		**$Cdn**
Copper World price per lb assumed			$2.75	/lb	
Lbs Cu in 1% of 1 dmt	22.046	lbs			
Mill recovery	90	%			
Copper recovered from mill	19.841	lbs			
% Copper in copper concentrate shipped	24.0	%			
Lbs dry copper concentrate	82.673	lbs			
Moisture Content assumed	8.0	%			
Lbs wet copper concentrate	89.861	lbs			
Shipping cost to Vancouver per tonne (Appendix 3 Table 6)					$84.78
Equivalent shipping to Vanc cost for each %					$3.46
Treatment (Smelting) Cost per dmt			$200.00	/dmt	
Equivalent smelting cost for each percent			$7.50		$9.75
Smelter deduction for loss	1	%			
Percent copper padi for	23.0	%			
Lbs of copper paid for	19.015	lbs			
Refining charge per lb paid for			$0.22	/ lb	
Total refining charge			$4.18		$5.44
Total cost for 1% Cu in situ					$18.64
Gross Smelter payment			$52.29	/%	$67.98
NSR on Copper before payment to seller			$37.95		$49.33
NSR to Property Seller	2.5%				
VALUE IN-SITU PER Cu%			**$37.00**	**/g**	**$48.10**
SILVER DOLLAR VALUE PER GRAM			**$US**		**$Cdn**
Silver World Price per oz			$16.50	/oz	$21.45
Grams per oz	31.1035	g/oz			
Gross value of 1 g Ag			$0.530	/g	$0.69
Mill recovery assumed	82	%			
Silver recovered from mill			$0.435	/g	$0.57
Smelter deduction			$1.00	/oz	$1.30
value per oz left			$15.50	/oz	$20.15
Value of g left			$0.409	/g	$0.53
Smelter pays of world price	90.0	%			
Smelter return per gram silver			$0.368	/g	$0.48
NSR to Property Seller	2.5%				
VALUE IN-SITU PER g Ag			**$0.36**	**/g**	**$0.47**
IN SITU VALUE SUMMATION	**$Cdn**				
Gold	**$38.77**	**/g**			
Copper	**$48.10**	**/%**			
Silver	**$0.47**	**/g**			

Table 22.2 CASH FLOW ANALYSIS																							
Annual Mine and Mill production	**146,000**	**tonnes per year**										**Assumed**											
$US DOLLAR IN $CDN FUNDS	**$1.30**											**Smelter**											
							World	**Smelter**	**Mined**		**Mill Recov**	**Paym't**											
							Price	**Pays for**	**Grade**	**Units**	**(%as Dec)**	**%(Dec)**											
					$US-->	gold /oz	$1,200	$1,190	4.51	g/t	0.839	0.975											
					$US-->	silver/oz	$16.50	$15.50	8.39	g/t	0.82	0.90											
					$US-->	copper/lb	$2.75		0.81	%	0.90	0.965											
					Pre-Production		**Production -->**																
Quarter Ending Assumptions	**Base**	**Units**	**TOTAL**	**Mar-16**	**Sep-16**	**Dec-16**	**Mar-17**	**Jun-17**	**Sep-17**	**Dec-17**	**Mar-18**	**Jun-18**	**Sep-18**	**Dec-18**	**Mar-19**	**Jun-19**	**Sep-19**	**Dec-19**	**Mar-20**	**Jun-20**	**Sep-20**	**Dec-20**	**Mar-21**
Off ore development (m)		m	**1,413**	200	202	0	200	200	106	200	200	105											
On ore devopment (m)		m	**1,694**	100	292	292	150	150	131		150	150	150	129									
On Ore Development (tonnes)		mt	**59,290**	3500	10220	10220	5250	5250	4585	0	5250	5250	5250	4515	0	0	0	0					
Stope Drill Blast		mt	**558,210**				30,750	30,750	31,415	36,000	30,750	30,750	30,750	31,485	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	17,560
On-Ore Development & Stope Muck Trammed		mt	**617,500**	3500	10220	10220	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	17,560
Mineral left untrammed end of period		mt	**617,500**	614,000	603,780	593,560	557,560	521,560	485,560	449,560	413,560	377,560	341,560	305,560	269,560	233,560	197,560	161,560	125,560	89,560	53,560	17,560	0
Tonnes shipped to mill		mt	**617,500**	3500	10220	10,220	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	17,560
Tonnage milled		mt	**617,500**	3500	10220	10220	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	36,000	17,560
Total oz Au produced		oz	**75,122**	426	1,243	1,243	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	4,380	2,136
Total oz Ag produced		oz	**136,585**	774	2,261	2,261	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	7,963	3,884
Total tonnes Cu produced		mt	**5,002**	28	83	83	292	292	292	292	292	292	292	292	292	292	292	292	292	292	292	292	142
Assumed concentrate grade	24.00	%Cu																					
Concentrate produced		dmt	**20,841**	118	345	345	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	1,215	593
Moisture in Wet Concentrate	8.00	%																					
Tonnes wet Concentrate shipped to Vanc.		wmt	**22,653**	128	375	375	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	644
REVENUE (in $C thousands)																							
Gold payment by smelter			**113,308**	642	1,875	1,875	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	6,606	3,222
Silver payment by smelter			**2,477**	14	41	41	144	144	144	144	144	144	144	144	144	144	144	144	144	144	144	144	70
Copper payment by smelter			**33,565**	192	559	559	1,971	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	1,955	954
Treatment Charges (in US$)	-200.00	<--$US	**-5,419**	-31	-90	-90	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-316	-154
NSR less Treatment charge			**143,932**	817	2,386	2,386	8,405	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	8,390	4,092
NSR Royalty	2.50	%	**-3,598**	-20	-60	-60	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-210	-102
Revenue Paid by Period ($thousands)			**140,334**	797	2,326	2,326	8,195	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	8,180	3,990
COSTS (in $C thousands)																							
Capital Cost Willa			**13,962**	7,182	3,390	3,390	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Capital Cost Max			**6,319**	6,319	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Max Raise Tailings Dam			**1,000**	0	500	0	0	0	500	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Equipment purchases			**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
General, Administration & Corp Costs	5.89	per tonne	**3,496**	0	0	0	212	212	212	212	212	212	212	212	212	212	212	212	212	212	212	212	103
Cu Conc Ship to Vancouver	85.00	per t Conc	**1,771**	10	29	29	103	103	103	103	103	103	103	103	103	103	103	103	103	103	103	103	50
Development :	2,000.00	/m	**4,042**	0	0	0	700	700	474	400	700	510	300	258	0	0	0	0	0	0	0	0	0
Willa Supervision	4.49	per tonne	**2,992**	327	0	0	162	162	162	162	162	162	162	162	162	162	162	162	162	162	162	162	79
Stope Drill & Blast	10.48	per tonne	**5,850**	0	0	0	322	322	329	377	322	322	322	330	377	377	377	377	377	377	377	377	184
Stope, muck, tram, haul to lower dump & Gen	7.98	per tonne	**4,737**	0	0	0	287	287	287	287	287	287	287	287	287	287	287	287	287	287	287	287	140
Ship to Mill	28.00	per tonne	**16,620**	0	0	0	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	1,008	492
Milling	39.48	per tonne	**23,434**	0	0	0	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	1,421	693
Total CAPEX			**21,677**	13,838	3,919	3,419	0	0	500	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total OPEX			**62,546**	0	0	0	4,216	4,216	3,997	3,971	4,216	4,026	3,816	3,781	3,571	3,571	3,571	3,571	3,571	3,571	3,571	3,571	1,742
Total Costs			**84,223**	13,838	3,919	3,419	4,216	4,216	4,497	3,971	4,216	4,026	3,816	3,781	3,571	3,571	3,571	3,571	3,571	3,571	3,571	3,571	1,742
Cashflow (in $C thousands)																							
Cash Flow this Quarter			**56,111**	-13,041	-1,593	-1,093	3,979	3,964	3,683	4,209	3,964	4,154	4,364	4,398	4,609	4,609	4,609	4,609	4,609	4,609	4,609	4,609	2,248
Cash Flow this Year			**56,111**			-15,727				15,836				16,881				18,437				18,437	2,248
Cumulative Cash Flow				-13,041	-14,634	-15,727	-11,748	-7,784	-4,100	109	4,073	8,227	12,591	16,990	21,599	26,208	30,817	35,426	40,035	44,645	49,254	53,863	56,111
EBITDA			**77,788**	797	2,326	2,326	3,979	3,964	4,183	4,209	3,964	4,154	4,364	4,398	4,609	4,609	4,609	4,609	4,609	4,609	4,609	4,609	2,248
Depreciation Available			**210,138**	13,838	17,029	19,502	18,355	17,208	16,561	15,378	14,195	13,012	11,829	10,646	9,463	8,280	7,097	5,915	4,732	3,549	2,366	1,183	0
Depreciation Used			**21,677**	728	946	1,147	1,147	1,147	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	0
Earnings before Interest & Taxes			**56,111**	68	1,380	1,179	2,832	2,817	3,000	3,026	2,781	2,971	3,181	3,216	3,426	3,426	3,426	3,426	3,426	3,426	3,426	3,426	2,248
Tax payable (no Tax Loss)	50000	0.26	**14,589**	18	359	307	736	732	780	787	723	773	827	836	891	891	891	891	891	891	891	891	585
Tax payable			**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Earnings After Taxes			**56,111**	68	1,380	1,179	2,832	2,817	3,000	3,026	2,781	2,971	3,181	3,216	3,426	3,426	3,426	3,426	3,426	3,426	3,426	3,426	2,248
-less CAPEX			**-21,677**	-13,838	-3,919	-3,419	0	0	-500	0	0	0	0	0	0	0	0	0	0	0	0	0	0
-plus depreciation			**21,677**	728	946	1,147	1,147	1,147	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	1,183	0
NPV after tax	10%		**38,672**																				
IRR after tax	10%		**83%**																				

23.0 Adjacent Properties

The most significant adjacent property is the LH (Figure 3). Historical work on the LH includes 518 m of underground workings. Gold, along with pyrite, pyrrhotite and arsenopyrite, occurs in silicified shear zones within the Elise Formation. It is not known if there is a genetic relationship between the LH and the Willa mineralization. Exploration work, comprising drilling, has been carried out on the property this year (2015).

24.0 Other Relevant Data and Information

The authors have reviewed the sources of information cited under References. The authors are not aware of any additional sources of information that might significantly change the conclusions presented in this technical report.

25.0 Interpretations and Conclusions

25.1 Mineral Processing and Metallurgical Testing

Various metallurgical studies have been carried out, including included gravity separation using high-gravity concentration, cyanidation, copper flotation, pyrite flotation, thickening and copper concentrate thickening. The results of these studies have been incorporated in the economic analysis.

All samples with a known origin upon which the metallurgical testwork has been were taken from the West zone only. No metallurgical testwork was carried out for the North or East zones. However, there is no direct or indirect evidence to suggest that significant differences may be encountered in metallurgical characteristics between the West Zone and the other two main zones.

25.2 Mineral Resource Estimate

MFW Geoscience Inc. estimated that the Willa Property includes 198,000 tonnes of Measured Mineral Resources at average grades of 5.36 g/t gold, 0.83% copper, and 8.3 g/t silver, and 627,000 tonnes of Indicated Mineral Resources at average grades of 4.97 g/t gold, 0.86% copper, and 9.5 g/t silver, and 151,000 tonnes of Inferred Mineral Resources at average grades of 4.21 g/t gold, 0.71% copper, and 9.8 g/t silver. Mineral Resources were estimated by Mike Waldegger, P.Geo., at a 3.0 g/t gold equivalent cut-off grade and is based on drillhole samples collected between 1980 and 2004. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

25.3 Mineral Reserve Estimates

There are no mineral reserve estimates for the Property.

25.4 Mining Methods

As of 2005, ground conditions in the Willa mine were considered by on-site geologists as being excellent, (other than one timbered section on the 1025 L). However, in late October, 2015, when the accessible underground access ways were inspected by co-author Ash, it was determined that the ground conditions of the underground access ways have deteriorated since 2005. Access to most of the mine is presently barred by a major ground collapse about 300 m into the 1025 L adit, and about

150 m into the 1100 L adit. Based upon general back (ceiling) conditions noted in the short, accessible portions of the adits it appears highly probable that extensive re-support of access ways throughout the mine will be required. This must include re-support with rock bolts, screen and straps, and will require timbering and blocking where known (and as-yet-uninspected) ground collapses have occurred.

There is a risk that geotechnical conditions and stability of the rock may preclude that temporary or permanent pillars be left over what is planned in this assessment.

25.5 Recovery Methods

The Willa mineral responds well to the typical grinding and flotation conditions. The testwork data are considered adequate for process design purposes provided the data are interpreted conservatively and flexibility is incorporated into the flotation circuit to permit easy circuit changes. Gravity concentration is also required in order to recover the coarse gold content.

With the present required grind of 80% <50 µm, the MAX mill has a production capacity for Willa mineral, of between 300 and 350 tonnes per day. With the completion of the installation of the larger ball mill, the MAX mill will be capable of processing in excess of 500 tpd.

25.6 Project Infrastructure

The project infrastructure includes both the Willa mine and MAX mill. They are connected by 142 km of road. The Willa deposit has been developed over several decades, with 2,385 m of underground workings.

25.7 Market Studies and Contracts

A market study was conducted, based on 2012 data obtained from Wood Mackenzie, an International-based research and consulting firm dealing in the fields of Energy, Mining and Metals. There are no existing metal contracts regarding production from the Willa.

25.8 Environmental Studies, Permitting, and Social or Community Impact

The Property is permitted for road building, and underground rehabilitation, drilling, bulk sampling. Certain environment studies, as described in this Report, will be needed to meet permit requirements. The is no mine permit for the Property.

25.9 Capital and Operating Costs

The CAPEX of the entire project is less than twenty-two million dollars ($C), which includes not only the mine development but also for the acquisition and renovations of the modern milling facilities of the MAX mine.

25.10 Economic Analysis

An economic analysis was prepared and is based on Measured and Indicated mineral resources from eight planned stopes in the West Zone of the Willa Mineral Resources. It does not include Inferred Mineral Resources nor does it include the Mineral Resources of the North and East Zones.

The tonnage and average grade for the diluted, recoverable mineral is estimated at 617,460 tonnes grading 4.51 g Au/t, 0.81 % Cu, and 8.39 g Ag/t, with an in-situ value of $C 217.58/t.

Based on a mine and mill production rate of 146,000 tonnes per year, the capital cost for the project is estimated at approximately $C 21.3 million, of which approximately $C 3.0 million has been spent to date. The revenue is estimated at $C 140.3 million, the operating costs at $C $62.5 million, cumulative cash flow at $C 56.1 million, the Net Present Value at $C 38.7 million, and the Internal Rate of Return at 83%. The mine has an estimated lifespan of 4.2 years.

The projected recoverable gold by a combination of gravity and flotation presently constitutes approximately 80% of the revenue of the operation as compared to 18% of the revenue from the copper and 2% for silver.

Due to the accumulated tax pools of about $C 47 million at the MAX operation (which was included as part of the purchase price of the MAX property), there will be no tax associated with the (present) 4.25-year life of the Willa mine.

Metals prices were assumed to be $US 1200/oz. for gold, $US 2.75/lb. for copper, $US 16.50/oz. for silver, and a US dollar at $C 1.30. Milling recoveries were assumed to be 83.9% for gold and 90% for copper and 82% for Silver.

The PEA is preliminary in nature and there is no certainty that it will be realized.

26.0 Recommendations

26.1 Mineral Processing and Metallurgical Testing

It is recommended that continued metallurgical test work be conducted to include, but not be restricted to:

- Optimization of reagent dosages for the Willa mineral
- Confirmatory test work for cycloning requirements at the Max mill
- Final design of milling flow sheet

A 10,000 tonne bulk sample should be taken for confirmatory pilot plant testwork.

26.2 Mineral Resource Estimates

To validate the geology logging, assess analytical bias with respect to the historical sampling, and to significantly increase the confidence in the bulk density model, it is recommended that 10 drillholes

(1,000 m) be completed from existing underground workings. Holes should be equally spaced across the West Zone.

To determine if the resource can be expanded in areas of limited drilling or where the mineralization is not completely cut off, it is recommended that several additional holes (1,000 m) be drilled.

To significantly increase confidence in the relationship between grade and bulk density the following procedures are recommended:

- Collect one sample for bulk density every 2 m.
- Bulk density tests should be on samples weighting approximately 100 g (½ core samples) and should include both the buoyancy method on solid rock and the pychometer method on pulp material to validate previous testing results.
- The samples should be assayed for gold, copper and silver.
- Continue with 2 m sample interval, except break samples at lithological boundaries.
- Include an ICP multi-element suite to develop alteration model.
- QC/QA sampling revised to include ½ core duplicates sent to the same lab as the next sample in the sequence.
- A CRM (analytical standard) should be included in the sample stream.

26.3 Mining Methods

For the Willa deposit, due to the wide widths and relatively low grade of the mineral zones, the appropriate mining method should be low-cost long-hole mining.

26.4 Environmental Studies, Permitting, and Social of Community Impact

It is recommended that continued environmental testwork be conducted to include, but not be restricted to:

- Assessing the underground drainage sources and flows to determine which can be captured and disposed of directly to Aylwin Creek without treatment
- Testing of the settling of mine water with and without flocculent and/or coagulants for final design of mine water settling pond
- Testing of flocculent and coagulant for settling in the Max tailings disposal facility
- On-going LC50 testing on *daphnia magna* and rainbow trout at the Max tailings pond facility
- During the bulk sampling and mining, all walls of main and auxiliary access drifts should be sampled for assessment of the general ARD potential of dump material.

With respect to the auriferous pyrite holding pond, it is recommended that the location, size and design be conducted and permitting for use be applied for.

With regards to socioeconomic considerations, it is recommended that Discovery Ventures proceed with public information meetings be held at Silverton, New Denver, Nakusp, Trout Lake, plus consultative discussions with First Nations that may be affected by the project.

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Date and Signatures

Effective Date: November 30, 2015

Amended Date: January 19, 2016

“signed and sealed”

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Wayne M. Ash, PEng

Date and Signatures

Effective Date: November 30, 2015

Amended Date: January 19, 2016

“signed and sealed”

____

William R. Gilmour, PGeo

Date and Signatures

Effective Date: November 30, 2015

Amended Date: January 19, 2016

“signed and sealed”

Michael F. Waldegger, PGeo

Certificate of Qualified Person

I, Wayne M. Ash, P. Eng. do hereby certify that:

1. I am a consulting Mining Engineer for Ash & Associates Consulting Ltd., 502-935 Marine Drive, West Vancouver, BC, V7T 1A7.

2. I am a graduate of the Provincial Institute of Mining, Haileybury, Ontario with certification as a Mining Technologist (1965) and from Michigan Technological University, Houghton, Michigan with a B.S. degree in Mining Engineering, (1969).

3. I am the co-author of a Report on the Property entitled "AMENDED PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT on the WILLA PROPERTY, SLOCAN MINING DIVISION, BRITISH COLUMBIA", for Discovery Ventures Inc, with an Amended Date of January 19, 2016 and an Effective Date of November 30, 2015. I am responsible for Sections, 1.4-1.5, 1.12, 1.14-1.21, 5, 6, 13, 15-22, 25.1, 25.3-25.10, 26.1, 26.3-26.4.

4. I have been practicing my profession since graduation. I have over 50 years of experience in the minerals industry, ranging from grass-roots-roots exploration through mining, mineral beneficiation and sale of concentrates.

5. I am a Professional Engineer with the Association of Professional Engineers and Geoscientists of British Columbia (Membership #104313, Licence #7940).

6. This report is based upon knowledge of the Property gained from the assessment of available documentation on historical exploration, and property visits between 2013 and 2015, with the most recent visit being October 29, 2015.

7. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as deemed in NI 43-101) and past work experience, I fulfill the requirements to be a "qualified person" (QP) for the purposes of NI-43-101, with past experience in the commodity being explored.

8. I am independent of both the Vendor and Issuer of all the tests in Section 1.5 of NI 43-101.

9. As of the date of this Certificate, to be best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

10. I have read NI 43-101, 43-101CP, and form 43-1-1F1, and the Report has been prepared in compliance with that instrument and form.

Effective Date: November 30, 2015
Amended Date: January 19, 2016

"signed and sealed"

____
Wayne M. Ash, P. Eng.

Certificate of Qualified Person

William R. Gilmour, BSc, PGeo

I, William R. Gilmour, BSc, PGeo, do hereby certify that:

1. I am a consulting geologist in mineral exploration with Discovery Consultants, 201, 2928 29th Street, Vernon, BC, V1T 5A6.
2. I am a 1970 graduate of the University of British Columbia with a Bachelor of Science degree in geology.
3. I am the author of a Report on the Property entitled "AMENDED PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT on the WILLA PROPERTY, SLOCAN MINING DIVISION, BRITISH COLUMBIA," for Discovery Ventures Inc, with an Amended Date of January 19, 2016 and an Effective Date of November 30, 2015. I am responsible for sections 1.1-1.3, 1.6-1.10, 2-4, 7-11, 23-24, and 27.
4. I have been practicing my profession since graduation. I have over 40 years experience in mineral exploration on for a variety of base and precious metals, uranium and diamonds. My working experience includes grassroots & reconnaissance exploration, project evaluation, geological mapping, planning and execution of drilling programs, and project reporting and project management.
5. I am a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia (membership #109681).
6. This report is based upon knowledge of the Property gained from the management of an exploration program carried out on the Property in 2004, the study of available documentation on historical exploration, and a property visit carried out on November 2, 2015.
7. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as deemed in NI 43-101) and past work experience, I fulfill the requirements to be a "qualified person" (QP) for the purposes of NI 43-101 with past experience in the commodity being explored.
8. I am independent of both the Vendor and the Issuer applying all of the tests in section 1.5 of NI 43-101.
9. As of the date of this Certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
10. I have read NI 43-101, 43-101CP, and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.

Effective Date: November 30, 2015
Amended Date: January 19, 2016

"signed and sealed"

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William R. Gilmour, PGeo

Certificate of Qualified Person

Michael F Waldegger, B.Sc., P.Geo.

I, Michael F Waldegger, B.Sc., P.Geo., do hereby certify that:

1. I am a consulting Resource Geologist with MFW Geoscience Inc, with a business address of 7082 Jasper Dr., Vernon, BC, V1H 1P2.
2. I am a graduate of the University of Ottawa with a Bachelor of Science degree in geology (B.Sc. Hons., 1998).
3. I am a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia (membership #33582).
4. I have been practicing my profession continuously since graduation.
5. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as deemed in NI 43-101) and past work experience, I fulfill the requirements to be a "qualified person" (QP) for the purposes of NI 43-101.
6. My relevant experience includes 18 years working as a geologist in the mining sector. Most relevant to the subject of this report are the 9 years estimating mineral resources on numerous projects around the world in base metals and precious metals deposits. I have also been involved in numerous drill programs in a management capacity, on site logging, sample chain of custody, and managing drill hole databases.
7. I visited the property on November 2, 2015.
8. I am the co-author of the Report entitled "AMENDED PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT on the WILLA PROPERTY, SLOCAN MINING DIVISION, BRITISH COLUMBIA," for Discovery Ventures Inc, with an Amended Date of January 19, 2016 and an Effective Date of November 30, 2015. I am responsible for sections1.11, 1.13, 12, 14, 25.2 and 26.2. I have no prior involvement with the property that is the subject of this Technical Report.
9. As of the date of this Certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
10. I am independent of both the Vendor and the Issuer applying all of the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101, 43-101CP, and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.

Effective Date: November 30, 2015
Amended Date: January 19, 2016

"signed and sealed"

____
Michael F. Waldegger, PGeo

Appendix I

	151118 Appendix 1 Table 1				
	PRE-PRODUCTION ADMINISTRATION EXPENSE (12 mo duration)				
		qty	**units**	**$/unit**	**Total**
ADMINISTRATION (TIME 12 Mo)					
	Head Office exp	12	mo	20,000	240,000
	New bond for land disturbanc	1	only	20,000	20,000
	Planning	12	mo	15,000	180,000
	Environmental monitor, report	12	mo	10,000	120,000
	Community Meetings	4	only	4,000	16,000
	Govt meetings	2	only	3,000	6,000
	Metallurgical Test work	6	mo	8,000	48,000
	Total Administration				**$ 630,000**

All dollars in Canadian funds

Appendix 1 Table 2

	WILLA CAPEX				
		qty	units	$/unit	Total
ACCRUED COSTS TO DATE					
	Purchase Costs Willa to date				3,083,000
	Expenses to date (met, consult, admin)				781,000
	Advance NSR Payments to date				479,500
	Advance NSR Payments for Pre-Production (1 yr)				144,000
	TOTAL ACCRUED COSTS TO DATE				**4,487,500**
ADMINISTRATION					
	Administration Phase 1 (6 months)				**315,000**
	PHASE 1: REHAB & BULK SAMPLE				
	Phase 1 Assumed 6 months				
		qty	units	$/unit	Total
UNDERGROUND DEVELOPMENT, INSTALLATIONS					
	Rehab 1025 & 1100 L	2000	only	200	400,000
	Mine drain line clean water (8" poly)	1,200	m	10.00	12,000
	Install Air Line	1200	m	10.00	12,000
	Water line 2" poly 130 psi from 1100L	1600	m	6.25	10,000
	PAG wall Sampling and Assaying	66	smple	60.00	4,000
	Diamond drilling West Zone (incl assay)	450	m	150	68,000
	TOTAL				**506,000**
SURFACE CONSTRUCTION ON SITE					
	Supervision (6 mo)				385,000
	Repair portals	2	only	15,000	30,000
	Repair portal slough	1	only	5,000	30,000
	Install drain line to settling pond	500	m	4.00	2,000
	Install power line to 1100 Portal	500	m	4.00	2,000
	Concrete blocks for bulk sample tipple	100	only	160.00	16,000
	Tipple Ties	3	mbf	1,000.00	3,000
	Tipple rail removal from Max mine	200	m	30.00	6,000
	Remove power cable from Max Mine	1,800	m	4.50	8,100
	Remove air line & vics from Max Mine	1,800	m	6.00	10,800
	Haulage rail, pipe, cable	2	trucks	800.00	1,600
	install tipple concrete blocks	1	only	6,000	6,000
	Install rail to tipple	100	m	40.00	4,000
	Mine drain water dirty water (4" poly)	500	m	5.00	2,500
	Install drain line to settling pond	500	m	4.00	2,000
	Total Surface Construction Phase 1				**124,000**
ACCESS ROAD CONSTRUCTION (short term)					-
	Run-away lanes	4	only	2,500	10,000
	Level present road				20,000
	Total short term road				**30,000**
CAPITAL EXPENDITURES					
	BUILDINGS				
	Lunch room/lamp room, check-in	1	only	6,000	6,000
	Mine office/First Aid	1	only	6,000	6,000
	Mechanic Shop	1	only	6,000	6,000
	Surface Storage	1	only	5,000	5,000
	Gen Set lean-to construction	1	only	4,000	4,000
		qty	units	$/unit	Total
	Compressor lean to construction	1	only	4,000	4,000
	Electrical supplies, set up	1	only	10,000	10,000
	Toilet facility (pump out)	1	only	5,000	5,000
	Total Bldgs Phase 1				**46,000**
COMPANY MOBILE EQUIPMENT					
	4-tonne Battery locomotive c/w charger	1	only	40,000	40,000
	8-tonne diesl locomotive	1	only	50,000	50,000
	6-tonne ore cars	8	only	10,000	80,000
	flat cars	3	only	4,000	12,000
	Main Fan (20 hp) from Max hookup	1	only	2,000	2,000
	Flat car c/w roof for rock-bolting	1	only	10,000	10,000
	Rock Breaker UG on wheels diesel (ug)	1	only	30,000	30,000
	total mobile equipment phase 1				**224,000**
	Sub-total Equipt, development, Installations				**930,000**
	Contingencies (20 %)				**186,000**
10,000 TONNE BULK SAMPLE					
	Bulk Sample drift: 100 m	100	m	3,000	300,000
	Bulk Sample Slash & haul to lower dump	6,500	t	50	325,000
	Bulk sample haul to mill	10,000	t	28	280,000
	Bulk sample Processing at Max	10,000	t	43	430,000
	Ship & Smelt Conc (accounted for elsewhere)				-
	SUB TOTAL				**1,335,000**
	GRAND TOTAL PHASE 1				**7,253,500**

Appendix 1 Table 2 cont'd

	PHASE 2: SURFACE AND UNDERGROUND MINE DEVELOPMENT				
Phase 2 Assumed 6 months					
	PREP FOR MINE PRODUCTION	**qty**	**units**	**$/unit**	**Total**
ADMINISTRATION					
	Head Office Administration				315,000
	Advance NSR Payments for Pre-Production (1 yr)				144,000
	Willa Supervision (6 mo)				385,000
	TOTAL ADMINISTRATION & SUPERVISION				**844,000**
EQUIPMENT AND INSTALLATION					
	Concrete tipple blocks for production	250	only	160	40,000
	Install tipple blocks, install bldgs	1	only	10,000	10,000
	Timber cutting pond (0.5 hectares)	0.5	hect	4,000	2,000
	Settling Pond construction	8	days	1,200	9,600
	Timber cutting lower dump	1	hect	4,000	4,000
	Mine Rescue Equipment	6	units	25,000	150,000
	New gates	2	only	1,000	2,000
	New road right of way timber	4	hect	4,000	14,400
	New access road	1,800	m	20	36,000
	Diesel locomotive (10 tonne)	1	only	50,000	40,000
	Backhoe	1	only	30,000	30,000
	Rock breaker surf on backhoe	1	only	30,000	30,000
		qty	**units**	**$/unit**	**Total**
	flat cars	3	only	4,000	12,000
	Total Equipment & installation				**380,000**
UNDERGROUND DEVELOPMENT					
	Off-mineral Stope Development	404	m	3,000	1,212,000
	On-Mineral stope development	684	m	3,000	2,052,000
	Misc off-mineral development	50	m	3,000	150,000
	total UG development	**1,138**	**m**		**3,414,000**
	Transport Mineral to Max Mill fr Ph2 develo	**34,000**	**t**	28	**952,000**
	SUB-TOTAL PHASE 2				**5,590,000**
	Contingencies (20%)				**1,118,000**
	TOTAL PHASE 2				**6,708,000**
	GRAND TOTAL PHASE 1 & 2				**$ 13,961,500**

All dollars in Canadian funds

Appendix 1 Table 3 MAX CAPEX

Annual Production		*146,000*	*tonnes*		
	ITEM	**qty**	**units**	**C$/unit**	**total C$**
COSTS ACCRUED TO DATE					
	Max Acquisition				2,560,000
	Outstanding financial Obligations re Max				2,600,000
Total Costs & Obligations Accrued to date					**5,160,000**
Milling Equipment Purchases					
	9 x 15 ball mill liners	1	set	35,000	
	Feeder Conveyor	1	only	15,000	
	Cyclones	6	only	24,000	
	Falcon SB750	1	only	75,000	
	Falcon Autopac	1	only	12,000	
	Falcon screen	1	only	20,000	
	Falcon mezanine	1	only	6,000	
	GRG holing tank c/w Agitator	1	only	12,000	
	Diester Table	1	only	15,000	
	Table Tails Pump Box	1	only	3,000	
	Table Tails Pump	1	only	3,000	
	Table Tails holding tank	1	only	5,000	
	TT air for tank	1	only	1,000	
	TT Agitator	1	only	4,000	
	TT pump	1	only	3,000	
	Lime tank c/w agitator	1	only	6,000	
	Lime Tank pump (Moyno)	1	only	3,000	
	Lime circulating piping	1	only	500	
	Sulfuric pump	1	only	4,000	
	Sulfuric Vent duct & fan	1	only	300	
	Sulfuric piping	1	only	500	
	Total Purchases				**247,300**
Lab Equipment Purchases					
	Leco Carbon/Sulfur Analyser	1	only	30,000	
	lab ball mill	1	only	6,000	
	Flotation Machine	1	only	10,000	
	pan filter	1	only	3,000	
	Vacuum pump	1	only	500	
	centrifuge & Misc Filters	1	only	1,000	
	Fire Assay Shipping Crate (20x 8)	1	only	5,000	
	Fire Assay Hood & Ducting	1	only	5,000	
	Fire Assay Fan	1	only	2,000	
	Fire Assay bag house	1	only	3,000	
	Fire Assay Furnaces low grade	1	only	6,000	
	FireAssay cupel furnace low grade	1	only	6,000	
	Fire Assay Furnaces high grade	1	only	2,500	
	Fire Assay tools	1	only	2,000	
	micro bead balance	1	only	13,000	
	Total Lab Equipment Purchases				**95,000**
Bullion Smelting Purchases					
	Furnace (incl 2 cruscibles)	1	only	8,000	
	Hood (sheet metal made on site)	1	only	1,000	
	Implements (molds, tongs etc)	1	set	3,000	
	Propane piping (SS)	1	only	1,000	
	Bullion Balance	1	only	500	
	Safe	1	only	2,000	
	Total BullionSmelt Purchases				**15,500**

Appendix 1 Table 3 cont'd MAX CAPEX

Building Repairs					
	Generator Bldg	1	only	2,000	
	Mill Roof	1	only	20,000	
	Secondary Electrical Room roof	1	only	2,000	
	Conveyor Gallery with snow shed	1	only	5,000	
	Lower Camp Repair	1	only	6,000	
	Mine Camp Repair	1	only	4,000	
	Cover over Coarse Ore Bin	1	only	15,000	
	Total Bldg Repairs				**54,000**
Equipment Repair					
	Apron Feeder	1	only	2,000	
	Jaw Crusher	1	only	5,000	
	Cone Crusher	1	only	5,000	
	Screen Deck	1	only	2,000	
	Crusher transfer points	3	only	4,000	
	Chutes	3	only	4,000	
	Grizzly over Coarse Ore Bin	1	only	15,000	
	Bearings on 7 x 7 ball mills	2	only	15,000	
	Wemco Flotation Cells	1	only	10,000	
	Coverall Building	1	only	2,000	
	Generators	1	only	10,000	
	Repair Wemco Flotation Cells	1	only	10,000	
	Total Equipment Repair				**84,000**
Equipment Installation					
	9 x 15 ball mill liners			5,000	
	Feeder Conveyor			5,000	
	Cyclones & piping	6	only	3,000	
	Falcon SB750	1	only	1,000	
	Falcon Autopac	1	only	1,000	
	Falcon screen	1	only	1,000	
	Falcon mezanine	1	only	2,000	
	GRG holing tank	1	only	200	
	GRG holding Tank Agitator, & Pipir	1	only	1,000	
	Diester Table	1	only	500	
	Diester Table cage	1	only	10,000	
	Table Tails Pump Box & Piping	1	only	400	
	Table Tails Pump	1	only	200	
	Table Tails holding tank	1	only	400	
	TT air piping for tank	1	only	400	
	TT Agitator	1	only	300	
	TT pump	1	only	500	
	Lime tank c/w agitator	1	only	2,000	
	Lime Tank pump (Moyno)	1	only	200	
	Lime circulating piping	1	only	500	
	Sulfuric pump (moyno SS)	1	only	200	
	Sulfuric Ventilation	1	only	200	
	Sulfuric piping	1	only	2,000	
	Furnace (incl 2 cruscibles)	1	only	8,000	
	Hood (sheet metal made on site)	1	only	1,000	
	Propane piping (Cert. plumber)	1	only	5,000	
	Safe	1	only	2,000	
	Total Equipment Installation				**53,000**
Other Capital Costs					
	Installation of a pyrite pond				**250,000**
	Raising of the dams for mine lifespan				**1,000,000**
	Contingencies on Equipment, Installation (20%)				**359,760**
	TOTAL MAX CAPEX				**$ 7,318,560**

All dollars in Canadian funds