TECHNICAL REPORT ON THE
2014 MINERAL RESOURCE UPDATE

MOSS MINE GOLD-SILVER PROJECT

MOHAVE COUNTY, ARIZONA, USA

Latitude 35º 6’ 00” N
Longitude 114º 26’ 52” W

for Northern Vertex Corporation

Qualified Persons:

David M.R. Stone, P. Eng.
David G. Thomas, P. Geo.
Daniel Kilby, P. Eng.
Douglas Brownlee, P. Geo.

Report Date: December 30, 2014
Effective Date: October 31, 2014

MINEFILL SERVICES, INC.
P.O. Box 725, Bothell, Washington, USA 98041
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DATE AND SIGNATURE PAGE

The Effective Date of this technical report, entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' is October 31, 2014.

CERTIFICATE – Dr. DAVID STONE, P. Eng.

I, David Stone, P. Eng., of PO Box 725, Bothell, Washington, USA, as the principal author of this report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' with an Effective Date of October 31, 2014 and which was prepared for Northern Vertex Mining Corporation (the "Issuer"), do hereby certify that:

1. I am currently employed as President of MineFill Services, Inc., that is a Washington, USA, domiciled Corporation.
2. I am a graduate of the University of British Columbia with a B.Ap.Sc in Geological Engineering, a Ph.D. in Civil Engineering from Queen's University at Kingston, Ontario, Canada, and an MBA from Queen's University at Kingston, Ontario, Canada.
3. This certificate applies to the technical report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' with an Effective Date of October 31, 2014 and a Report Date of December 28, 2014 (the "Technical Report") that was prepared for the Issuer.
4. I have practiced my profession for over 30 years and have considerable experience in the preparation of engineering and financial studies for base metal and precious metal projects, including Preliminary Economic Assessments, Preliminary Feasibility Studies and Feasibility Studies.
5. I am a licensed Professional Engineer in Ontario (PEO #90549718) and I am licensed as a Professional Engineer in a number of other Canadian and US jurisdictions.
6. I have read the definition of 'Qualified Person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
7. I visited the subject property during November 15 and November 16, 2014.
8. I am responsible for Sections 1, 18 and 19, with the exception of those portions related to geology and mineralization, deposit type, exploration, drilling, sample preparation analysis and security, quality assurance/quality control, data verification and the Mineral Resource estimate that is the subject of the Technical Report, Sections 2 through 6, Section 7.3 in conjunction with Mr. Daniel Kilby, P. Eng., Section 13 and Sections 15 through 17.
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and NI 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (October 31, 2014), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

// Signed and Sealed //

David Stone, P. Eng.

DATED at Bothell, Washington, USA, this 30th day of December, 2014.

CERTIFICATE – Mr. DAVID G. THOMAS, P. Geo.

I, David G. Thomas, P. Geo., of 1051 Homer Street, Vancouver, British Columbia, Canada, as a co-author of this report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA', with an Effective Date of October 31, 2014 and which was prepared for Northern Vertex Mining Corporation (the "Issuer"), do hereby certify that:

1. I am an Associate Geologist with the geological consulting firm of Fladgate Exploration Consulting Corporation of Thunder Bay, Ontario, Canada.
2. I am a graduate of Durham University, in the United Kingdom, with a Bachelor of Science degree in Geology, and I am a graduate of Imperial College, University of London, in the United Kingdom, with a Master of Science degree in Mineral Exploration.
3. This certificate applies to the technical report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' with an Effective Date of October 31, 2014 and a Report Date of December 28, 2014 (the "Technical Report") that was prepared for the Issuer.
4. I have practiced my profession for over 19 years. In that time I have been directly involved in the review of exploration programs, geological models, exploration data, sampling, sample preparation, quality assurance-quality control, databases, and Mineral Resource estimates for a variety of mineral deposits, including epithermal gold deposits (Mexico, Argentina, Bulgaria and Serbia).
5. I am a member in good standing of the Association of Professional Geoscientists of British Columbia (APEGBC NRL # 149114). I am also a member of the Australasian Institute of Mining and Metallurgy (MAusIMM # 225250).
6. I have read the definition of 'Qualified Person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
7. I visited the subject property from September 15 to September 18, 2014.
8. I am responsible for those portions of Sections 1, 18 and 19 related to the Mineral Resource estimate that is the subject of the Technical Report, Sub-Section 10.3.4 and Section 14.
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and NI 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (October 31, 2014), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

// Signed and Sealed //

David G. Thomas, P. Geo.

DATED at Vancouver, British Columbia, Canada, this 30th day of December, 2014.

CERTIFICATE – Mr. DANIEL KILBY, P. Eng.

I, Daniel Kilby, P. Eng., of 726 12th Street East, North Vancouver, British Columbia, Canada, as a co-author of this report entitled ‘Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA’, with an Effective Date of October 31, 2014 and which was prepared for Northern Vertex Mining Corporation (the “Issuer”), do hereby certify that:

1. I am the General Manager – Exploration for Northern Vertex Mining Corporation.
2. I am a graduate of the University of British Columbia, with a Bachelor of Applied Science degrees in Geological Engineering.
3. This certificate applies to the technical report entitled ‘Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA’ with an Effective Date of October 31, 2014 and a Report Date of December 28, 2014 (the “Technical Report”) that was prepared for the Issuer.
4. I have practiced my profession for over 35 years. In that time I have been directly involved in review of exploration programs, geological models, exploration data, sampling, sample preparation, quality assurance-quality control, databases, and mineral resource estimates for a variety of mineral deposits, including an epithermal gold deposits (Nevada, USA).
5. I am a member in good standing of the Association of Professional Engineers of British Columbia (APEGBC License #14283).
6. I have read the definition of ‘Qualified Person’ set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a ‘Qualified Person’ for the purposes of NI 43-101.
7. I most recently visited the subject property from September 15 through September 17, 2014.
8. I am responsible for those portions of Sections 1, 18 and 19 relating to geology and mineralization, deposit type, exploration and drilling, Section 7 (including Section 7.3 in conjunction with Dr. David Stone, P. Eng.) and Sections 8 through 10 (the latter with the exceptions of Sub-Sections 10.1.2, 10.2.2 and 10.3.4).
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and NI 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (October 31, 2014), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

// Signed and Sealed //

Daniel Kilby, P. Eng.

DATED at Vancouver, British Columbia, Canada, this 30th day of December, 2014.

CERTIFICATE – Mr. DOUGLAS BROWNLEE, P. Geo.

I, Douglas Brownlee, P. Geo., of 4794 Quesnel-Hydraulic Road, Quesnel, British Columbia, Canada, as a co-author of this report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA', with an Effective Date of October 31, 2014 and which was prepared for Northern Vertex Mining Corporation (the "Issuer"), do hereby certify that:

1. I am a self-employed consulting Geologist.
2. I am a graduate of the University of Alberta in Canada, with a Batchelor of Science degree in Geology Specialization.
3. This certificate applies to the technical report entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' with an Effective Date of October 31, 2014 and a Report Date of December 28, 2014 (the "Technical Report") that was prepared for the Issuer.
4. I have practiced my profession for over 30 years. In that time I have been directly involved with exploration programs for a variety of mineral deposits (Canada, United States, Mexico, Venezuela and Ghana) building and reviewing geological models, designing and setting up exploration data, sampling, sample preparation, quality assurance-quality control programs and databases.
5. I am a member in good standing of the Association of Professional Geoscientists of Alberta (APEGA # 47377).
6. I have read the definition of 'Qualified Person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'Qualified Person' for the purposes of NI 43-101.
7. I have not visited the subject property.
8. I am responsible for those portions of Sections 1, 18 and 19 relating to sample preparation analysis and security, quality assurance/quality control and data verification, Sub-Sections 10.1.2 and 10.2.2, Sections 11 and 12, and Appendix A.
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and NI 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (October 31, 2014), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

// Signed and Sealed //

Douglas Brownlee, P. Geo.

DATED at Quesnel, British Columbia, Canada, this 30th day of December, 2014.

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NOMENCLATURE AND ABBREVIATIONS

Abbreviation	Unit or Description
AA	Atomic Absorption
AAC	Arizona Administrative Code
AAS	atomic adsorption spectrophotometry
ADEQ	Arizona Department of Environmental Quality
Ag	silver
APP	aquifer protection program
ASLD	Arizona State Land Department
Au	gold
AWQS	aquifer water quality standards
BADCT	best available demonstrated control technology
B.C.	British Columbia, Canada
BDV	block dispersion variance
BHL	Hartmut W. Baitis, Robert B. Hawkins & Larry L. Lackey
BLM	Bureau of Land Management
cm	centimetre
CSRM	certified standard reference materials
Cu	copper
CV	coefficient of variation
EqAu	equivalent gold (ounces or grade)
FAAS	flame atomic absorption spectrophotometric
FLPMA	Federal Land Policy and Management Act of 1976
ft	feet
g	gram
g/t	grams per tonne
ha	hectare
Hg	mercury
ICAP-OES	inductively coupled argon plasma – optical emission spectrophotometer
ICP-AES	inductively coupled plasma atomic emission spectrometer
ISGC	Idaho State Gold Company, LLC
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
L	litre
m	metre
M	million
m^2	metre squared
MCF	mine call factor
M+I	Measured plus Indicated (categories of Mineral Resource)
ml	milli-litre
mm	millimetre
MRE	Mineral Resource estimate
MRM	Mineral Resource model
MSGP	multi-sector general permit
Mt	million tonnes
NaCN	sodium cyanide
oz	troy ounce (31.10346 g)
oz/t	troy ounce per short ton
P_{80} (or any other subscript number)	% of material (indicated by the number) passing a specified mesh size
PEA	Preliminary Economic Assessment
RSE	relative standard error of a kriged estimate
SA:V	surface area to volume ratio
SD	standard deviation (statistical function)
SMU	selective mining unit
SWPPP	stormwater pollution prevention plan
t	metric ton (or tonne)

Unless otherwise stated, all dollar figures are in United States dollars (US$). The metric system is employed; for the sake of clarity equivalent US Customary units are sometimes stated in parentheses.

1 SUMMARY

1.1 Issuer

This technical report has been prepared at the request of the issuer, Northern Vertex Mining Corporation (the "Company") that is incorporated in British Columbia, Canada ("B.C."). The Company has its offices at Vancouver, B.C., and it is listed on the TSX-V (trading symbol: NEE) and on the OTCQX (trading symbol: NHVCF). The Company's focus is on the reactivation of the Moss Mine Gold-Silver Project in Mohave County, northwest Arizona, USA (the "Moss Mine Project"), which is the only project or property that the Company has an interest in. The Company has the right to earn-in a 70% property interest in that portion of the Moss Mine Project that is subject to a joint venture agreement with Patriot Gold Corporation, a Nevada, USA, domiciled corporation (see Sub-Section 4.4.2).

1.2 Moss Mine Project

The Moss Mine Project area is located some 22 km by road to the east of Bullhead City, in the historically significant San Francisco (Oatman) Mining District of Mohave County, Arizona. It comprises a total area of approximately 4,030.8 hectares, centred on Latitude 35º 6' 00" North, Longitude 114º 26' 52" West, which was the approximate location of an historical headframe associated with (limited) historical underground mine workings that exploited the Moss Vein. The Company's activities have thus far mainly focused on the exploitation of the Moss Vein, West Extension and their associated stockworks that contain the gold-silver mineralization of interest. The target mineralization outlined is contained within a central area of 15 patented lode claims (102.8 hectares).

After signing the joint venture agreement with Patriot Gold in March 2011, the Company undertook a three-phase exploration drilling program that was completed in 2013. During 2013 the Company's main focus was on its Phase I Pilot Plant activities ("Phase I") that comprised openpit mining, on-site heap leaching and processing of a bulk sample of Moss Vein mineralized material, with off-site carbon stripping and doré production. All Phase I activities were completed during Q4 2014.

A Preliminary Economic Assessment ("PEA") was compiled in 2013. The results are reported in the 2013 Technical Report. The parameters assumed within the scope of the PEA are no longer applicable so the results of the PEA are no longer relevant.

Phase II Commercial Operations ("Phase II") comprise future activities that will include openpit mining, heap leaching and on-site processing to produce doré. The Phase II operations are the subject of an on-going feasibility study of which the 2014 Mineral Resource update, that is the subject of this technical report, is a part. Completion of the feasibility study fulfills the terms of the earn-in agreement with Patriot Gold.

1.3 This Technical Report

This technical report has been prepared with the purpose of providing an updated, National Instrument ("NI") 43-101 disclosure of the Company's 2014 Mineral Resource update for the Moss Mine Project. Details of Moss Mine Project exploration drilling and data verification to October 31, 2014 are provided, along with details of the 2014 Mineral Resource update and completed metallurgical testwork, the latter inclusive of results' interpretation.

1.4 2014 Mineral Resource Update

The Mineral Resources that are the subject of this technical report (Table 1.1) were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, by application of a cut-off grade that incorporated mining and metallurgical recovery parameters. The estimated Mineral Resources are constrained to a pit shell based on commodity prices, metallurgical recoveries and operating costs. Long-term metal prices of US$1,250/oz Au and US$20.0/oz Ag were applied along with metallurgical recovery rates of 82% for gold and 65% for silver. The stated Mineral Resources have an effective date of October 31, 2014.

Table 1.1: Moss Mine Project Mineral Resource Estimate by David Thomas, P. Geo.
(undiluted, pit constrained, 100% in-pit recovery, effective date October 31, 2014)

Category (0.25 g/t Au Cut-Off)	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq (g/t)**	**AuEq (oz)**
Measured	4,860,000	0.97	10.4	152,000	1,630,000	1.10	172,000
Indicated	10,620,000	0.66	8.7	225,000	2,980,000	0.77	263,000
Measured + Indicated	*15,480,000*	*0.76*	*9.3*	*377,000*	*4,610,000*	*0.87*	*435,000*
Inferred	2,180,000	0.55	5.6	38,000	390,000	0.62	43,000

Footnotes to Mineral Resource statement:

- *The Qualified Person ("QP") reviewed the Company's QA/QC programs on the Mineral Resources data. After removing samples with data quality issues, the QP concludes that the collar, survey, assay, and lithology data are adequate to support Mineral Resources estimation.*
- *Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were modelled based on quartz veining and gold grades.*
- *Raw drillhole assays were composited to 1.52 m lengths broken at domain boundaries.*
- *Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.*
- *Block grades for gold and silver were estimated from the composites using ordinary kriging interpolation into 3 m x 3 m x 3 m blocks coded by domain.*
- *A dry bulk density of 2.51 g/cm^3 was used for material with a depth less than 12 m from surface. A dry bulk density of 2.58 g/cm^3 was used for all other material. The dry bulk densities are based on 506 specific gravity measurements.*
- *Blocks were classified as Measured, Indicated and Inferred in accordance with CIM Definition Standards 2014. Inferred resources are classified on the basis of blocks falling within the mineralised domain wireframes (i.e. reasonable assumption of grade/geological continuity) with a maximum distance of 100 m to the closest composite. Indicated resources are classified based on a drillhole spacing of 50 m. Measured resources are classified based on a 25 m x 12.5 m drillhole spacing.*
- *The Mineral Resource estimate is constrained within an optimized pit with a maximum slope angle of 65º.*
- *Metal prices of $1,250/oz and $20.0/oz were used for gold and silver, respectively.*
- *Metallurgical recoveries of 82% for gold and 65% for silver were applied.*
- *A 0.25 g/t gold cut-off was estimated based on a total process and G&A operating cost of $6.97/t of mineralized material mined.*
- *The contained gold and silver figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding.*
- *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
- *The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.*

The gold equivalent ("AuEq") grades and ounces stated on Table 1.1 were determined by applying the following formulae:

Factor A (gold) = 1 / 31.10346 x metallurgical recovery (82%) x smelter recovery (99%) x refinery recovery (99%) x unit Au price (US$1,250/oz)

Factor B (silver) = 1 / 31.10346 x metallurgical recovery (65%) x smelter recovery (98%) x refinery recovery (99%) x unit Ag price (US$20.0/oz)

AuEq = Au grade + (Ag grade x [Factor B / Factor A])

The applied metallurgical recovery rates were those determined through analysis of the results of the metallurgical testwork programs completed up to the Effective Date of this technical report of October 31, 2014 (Section 13). With the exception of the first testwork program that encompassed gravity separation of native gold and electrum (which option was subsequently not pursued), the seven subsequent testwork programs focused on cyanidation/heap leaching (the option of conventional flotation was not tested). The results show that recovery rates of 82% for gold and 65% for silver may be applied for purposes of Mineral Resource estimation and Moss Mine Project planning, as long as:

- the heap leach feed comprises P_{95} 6.35 mm (1/4") mineralized material of the type used during the Phase I heap leaching operation;
- the crushed and screen mineralized material is agglomerated using cement; and
- a Merrill-Crowe type recovery system for silver is employed.

An overall gold recovery rate of 82% for gold was achieved from the Phase I heap leach. Silver recovery was, however, lower than 65% because a Merrill-Crowe type recovery system for silver was not employed.

The applied grade cut-off (0.25 g/t Au) was estimated using a unit cost for mining mineralized material of US$6.36/t, including waste, a unit process cost (heap leach) of US$4.42/t and a unit on-site G&A cost of US$2.55/t. It was based on the generally accepted practice that a decision is made at a pit rim if mined material above the marginal cut-off grade will lose less money if it is sent to the mill rather than if it is sent to the waste dump. It is considered for further processing if it contains a value that is greater than the cost to process it.

1.5 Reconciliation of the 2014 and 2013 Mineral Resource Estimates

The outcomes of the step-wise reconciliation analyses (see Section 14.13), from base case normalization through to model alignment, are summarized on Table 1.2. The results show that no material difference exists between the normalized 2014 Mineral Resource estimate ("MRE") and fully adjusted 2013 MRE, in terms of AuEq ounces defined using the 2013 MRE equivalent metal price ratio (Au 1 : Ag 50). This may be expected because largely the same database was used for both estimates (300 additional assays were included in the 2014 Mineral Resource model ["MRM"]). It was instead mainly the differences of approach when compiling the two MRMs and subsequent MREs that led to the difference in AuEq ounces apparent in the base case models (e.g. the use of wireframes and domains within the scope of the 2014 MRM, which were not included within the scope of the 2013 MRM). If the AuEq differences between the base case models are examined in a logical, step-wise manner the MRE outcomes are nearly identical in terms of AuEq ounces.

Table 1.2: A Summary of Outcomes, 2013 to 2014 Reconciliation Analysis

Modelled Case	2013 Estimate				2014 Estimate			
	Step-Wise Difference (oz AuEq)		Adjusted Model		Normalized Model (per Table 16.1)		Difference to Step-Wise Adjusted 2013 MRE (oz AuEq)	
	M+I	Inferred	M+I (oz AuEq)	Inferred (oz AuEq)	M+I (oz AuEq)	Inferred (oz AuEq)	M+I	Inferred
Normalized Models	-	-	654,000	82,000	472,000	50,000	- 182,000	- 32,000
Re-Block 2013 MRM to 3 m x 3 m x 3 m	- 13,000	-	641,000	82,000	472,000	50,000	- 169,000	- 32,000
2014 Wireframe Constraint (FW only)	- 106,000	- 10,000	535,000	72,000	472,000	50,000	- 63,000	- 22,000
2014 Wireframe Constraint (HW only)	- 115,000	- 51,000	420,000	21,000	472,000	50,000	+ 52,000	+29,000
Mineralization within 2014 MRM Wireframe	+ 51,000	+29,000	471,000	50,000	472,000	50,000	+ 1,000	0

Note: To conform with the 2013 Mineral Resource Estimate, an equivalent metal price ratio of Au 1 : Ag 50 was used

1.6 Conclusions and Recommendations

In MineFill's opinion, the 2014 Mineral Resource update provides a robust estimate that has been compiled using best industry practices and which conforms to the requirements of CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves:

- the Company's exploration drilling program, drillhole surveys, sampling, security, sample preparation and assaying procedures have been carried out in accordance with CIM Best Practice Guidelines and are suitable to support Mineral Resource estimation;
- the Company's exploration and drilling programs supply sufficient information for Mineral Resource estimation and classification;
- the Company's sampling and assaying includes adequate quality assurance procedures;
- the 2014 MRE is well constrained by three-dimensional wireframes representing geologically realistic volumes of mineralization (exploratory data analysis conducted on assays and composites shows that the wireframes are suitable domains for Mineral Resource estimation);
- the Mineral Resources were classified using confidence intervals scaled to volumes of production relevant to the Moss Mine Project; and
- the Mineral Resources are reported using economic and technical criteria such that the Mineral Resources have reasonable prospects of economic extraction.

Additional bottle roll tests on West Extension mineralized material are, however, recommended to establish whether its metallurgical response is similar to that of Moss Vein mineralized material. Other recommendations concerning the paragenetic model presented in Section 7.2.4 and the determination of the grain sizes and deportment of native gold, electrum and acanthite are also made. The details of the recommended program are presented in Section 18 and they are not repeated here.

2 INTRODUCTION

2.1 Issuer

This report has been prepared at the request of the issuer, Northern Vertex Mining Corporation (the "Company") that is incorporated in British Columbia, Canada ("B.C."). The Company's head office is at Suite 1820, 1055 West Hastings Street, Vancouver, B.C., V6E 2E9 and its registered office is at Suite 1500, 1055 West Georgia Street, Vancouver, B.C., V6E 4N7. It is listed on the Venture Exchange of the Toronto Stock Exchange (trading symbol: NEE) and on the OTCQX (trading symbol: NHVCF). The Company was originally called Northern Vertex Capital, Inc. It changed its name to Northern Vertex Mining Corporation in February 2012 (see the Company's news release dated February 14, 2012).

The Company is focused on the reactivation of the Moss Mine Gold-Silver Project in Mohave County, northwest Arizona, USA (the "Moss Mine Project"), which is the only project or property that the Company has an interest in. The Company has the right to earn-in a 70% property interest in that portion of the Moss Mine Project that is subject to a joint venture agreement with Patriot Gold Corporation ("Patriot Gold", Sub-Section 4.4.2) of Suite D165, 3651 Lindell Road, Las Vegas, Nevada 89103, USA (OTC trading symbol: PGOL). The Company is the joint venture operator; all Project site activities are wholly managed through its USA subsidiary – Golden Vertex Corporation of Suite 101, 2440 Adobe Road, Bullhead City, Arizona 86442 ("Golden Vertex").

2.2 Moss Mine Project

The Moss Mine Project area is located some 22 km by road to the east of Bullhead City, in the historically significant San Francisco (Oatman) Mining District of Mohave County, Arizona. It comprises approximately 102.8 hectares in 15 patented lode claims plus 468 surrounding unpatented lode claims and one Arizona State exploration permit with a total area of approximately 3,928.0 hectares (total overall area = 4,030.8 hectares). The claims are centred on Latitude 35º 6' 00" North, Longitude 114º 26' 52" West, which was the approximate location of an historical headframe associated with historical underground mine workings. As part of the Company's community relations plan, the headframe was in 2013 moved to the Recreation Park at Bullhead City, Arizona.

The historical underground workings, that are limited in extent, exploited the Moss Vein. The Company's activities have thus far mainly focused on exploitation of the Moss Vein, West Extension and associated stockworks that contain the gold-silver mineralization of interest. The target mineralization outlined is contained within the area of the 15 patented lode claims.

The Company has adopted a phased approach to the development of the Moss Mine Project. Exploration drilling was carried out during 2011 to 2013, following which Phase I Pilot Plant activities were undertaken in 2013 and 2014 ("Phase I"). They comprised openpit mining of a bulk sample of mineralized material from the Moss Vein and its associated stockwork, which material was heap leached at site. The pregnant solution was processed through carbon columns located next to the heap leach area. Carbon stripping and doré production was carried out off-site. All Phase I activities were completed during Q4 2014. Phase II Commercial Operations ("Phase II") comprise future activities that will include openpit mining, heap leaching and on-site processing to produce doré. The Phase II operations are the subject of an on-going feasibility study of which the 2014 Mineral Resource update, that is the subject of this Technical Report, is a part. Completion of the feasibility study fulfills the terms of the earn-in agreement with Patriot Gold.

In 2014 the Company also started exploration work on the larger Moss Mine project area, as defined by the 468 unpatented lode claims and an Arizona State exploration permit. The results of this exploration work are described in Section 9.

2.3 This Technical Report

This report is entitled 'Technical Report on the 2014 Mineral Resource Update, Moss Mine Gold-Silver Project, Mohave County, Arizona, USA' (this "Technical Report"), the data cut-off date for which is October 31, 2014. It has been prepared with the purpose of providing an updated, National Instrument ("NI") 43-101 disclosure of the Company's 2014 Mineral Resource update for the Moss Mine Project. Details of the Moss Mine Project are presented, along with:

- details of Moss Mine Project exploration drilling and data verification to October 31, 2014;
- details of the 2014 Mineral Resource update for the Moss Mine Project; and
- details and an interpretation of completed metallurgical testwork.

Although a Preliminary Economic Assessment ("PEA") has been compiled and the Phase I operations are complete, the Moss Mine Project does not qualify as an Advanced Property, as defined in Section 1.1 of National Instrument ("NI") 43-101 'Standards of Disclosure for Mineral Projects' ('*an 'advanced property' means a property that has (a) mineral reserves, or (b) mineral resources the potential economic viability of which is supported by a preliminary economic assessment, a pre-feasibility study or a feasibility study*'). To the best of MineFill's knowledge, no Mineral Reserves have been estimated for the Moss Mine Property. As a result of the updated Mineral Resource estimate, which is the subject of this Technical Report, the parameters assumed within the scope of the PEA are no longer applicable so the results of the PEA are no longer relevant. 'Additional Requirements for Advanced Property Technical Reports' (Item 15 through Item 22 of Form 43-101F1 – Technical Reports) are not, therefore, included in this Technical Report.

Completion of the on-going Feasibility Study will change the status of the Moss Mine Property, as defined by Section 1.1 of National Instrument ("NI") 43-101 'Standards of Disclosure for Mineral Projects'. The results of the Feasibility Study will be reported on its completion, in a separate Technical Report.

2.4 Sources of Information

The information contained in this Technical Report was compiled from various published and internal Company documents and reports by contributing consultants and the Qualified Persons (authors) of this Technical Report, as well as documents sourced by means of web searches and observations made during the Qualified Persons' site visits. The various reports, documents and files are cited, where appropriate. A full list of the cited reports, documents and files is provided in Section 19 of this Technical Report. The key documents referenced herein include:

- copies of the various legal agreements relating to the Moss Mine Project, between various individuals groups and companies;
- various news releases by the Company, sourced from its website (www.northernvertex.com);
- United States Bureau of Land Management status reports for the patented and unpatented lode claims that comprise the Moss Mine project area;

- a memorandum and appended schedule to the Company by Brian Munson of CDM Smith entitled 'Proposal for Moss Mine Phase II Permit Analysis for Feasibility Study' and dated September 02, 2014.
- consultancy reports to the Company by Douglas Brownlee, P. Geo., entitled 'Report on Geological Model, Moss Project, Arizona, USA' and dated August 23, 2014, and 'Verification of Golden Vertex Corp., Moss Mine Drill Hole Database' dated December 31, 2013;
- consultancy reports to the Company by David Thomas, P. Geo., entitled 'Moss Mine Project, 2014 Mineral Resource Update' and dated October 24, 2014 and '2014 Model Reconciliation to 2013 PEA Model' dated October 03, 2014; and
- consultancy reports to the Company by Stephen Godden, Independent Mining Consultant, entitled 'Moss Mine Gold-Silver Project, Mineralogical and Metallurgical Review' and dated November 23, 2014 and 'Moss Mine Gold-Silver Project, 2013 to 2014 Mineral Resource Estimates' Reconciliation (Summary)' dated October 09, 2014.

Certain historical, geographical and local resource data were extracted from a Technical Report by Scott E. Wilson, Jack McPortland, Stewart Redwood, et al entitled 'Amended Technical Report and Preliminary Economic Assessment for the Moss Mine Gold-Silver Project' and dated June 18, 2013 (the "2013 Technical Report", which was reviewed by the principal author of this Technical Report [Dr. David Stone, P. Eng.], as part of a larger due diligence process). Reference was also made to a Technical Report by Scott E. Wilson Consulting, Inc. of Highlands Ranch, Colorado, entitled 'NI 43-101 Technical Report, Northern Vertex Capital Inc., Moss Mine, Mohave County, Arizona, USA' and dated October 20, 2011 (the "2011 Technical Report"). Both the 2013 Technical Report and the 2011 Technical Report are listed on www.sedar.com.

MineFill Services, Inc. ("MineFill") has relied almost entirely on information derived from work completed by the authors of published data sources, Company staff members and Company consultants. Although MineFill has reviewed much of the available data and the principal author of this Technical Report has visited the Project area, these tasks only validate a portion of the entire dataset. MineFill has made judgements about the general reliability of the underlying data that is assumed to be both accurate and valid, based on the professional status of the reports' authors and the nature of their reports.

Much of the background information on the Moss Mine Project, such as the history, past exploration, exploration drilling, sampling and assaying, has been reported by others. This past information has been updated only when it was relevant to do so and/or when it was clear that additional information was required.

2.5 Qualified Persons

The Qualified Persons (authors) of this Technical Report are:

Dr. David Stone, P. Eng. - Mining Consultant and President of MineFill Services, Inc. of Bothell, Washington State, USA. Dr. Stone is the principal author of this Technical Report. He is responsible for Sections 2 through 6, Section 7.3 in conjunction with Mr. D. Kilby, P. Eng., Section 13, Sections 15 through 17 and, in conjunction with the other Qualified Persons, Sections 1, 18 and 19. He has reviewed earlier Technical Reports relating to the Moss Mine Project, as well as project-related documents and news releases. He visited the Project area during November 15 and

16, 2014, during which time he toured the project site, visited the underground working and inspected drillcore at the Project's core shack located at Bullhead City, Arizona.

Mr. David Thomas, P. Geo. – Geological Consultant and President of DKT Geosolutions, Inc. of Vancouver, B.C. Mr. Thomas is a co-author of this Technical Report. He is responsible for Sub-Section 10.3.4, Section 14 and, in conjunction with the other Qualified Persons, Sections 1, 18 and 19. He has reviewed earlier Technical Reports relating to the Moss Mine Project, as well as project-related documents and news releases. He visited the Project area between September 15 and September 18, 2014, during which time he toured the Project site, visited the underground workings and inspected drillcore at the Project's core shack located at Bullhead City, Arizona.

Mr. Daniel Kilby, P. Eng. – Geologist and General Manager Exploration for the Company. Mr. Kilby is a co-author of this Technical Report. He is responsible for Sections 7 through 10 (except for Sub-Sections 10.1.2, 10.2.2 and 10.3.4), Section 7.3 in conjunction with Dr. D. Stone, P. Eng., and, in conjunction with the other Qualified Persons, Sections 1, 18 and 19. He has visited the Project Area on numerous occasions, the last time being September 15 through 17, 2014. He has toured the Project site, visited the underground workings and inspected drillcore at the Project's core shack located at Bullhead City, Arizona. He has also reviewed the geological database.

Mr. Douglas Brownlee, P. Geo. – Consulting Geologist of Quesnel, B.C. Mr. Brownlee is a co-author of this Technical Report. He is responsible for Sub-Sections 10.1.2 and 10.2.2, Sections 11 and 12, Sections 1, 18 and 19 in conjunction with the other Qualified Persons, and Appendix A. He has reviewed the data and reports relating to the Company's drillhole database and carried out a verification of the Company's drillhole database. He has reviewed the Company's QA/QC procedures. In accordance with Section 6.2 (3) of Companion Policy 43-101CP to National Instrument 43-101 'Standards of Disclosure for Mineral Projects', Mr. Brownlee is not required to make a site visit and Mr. Brownlee has not made a site visit (because the scope of his work was limited to a review and verification of the Company's digital drillhole database and a review of the Company's QA/QC procedures, and he did not review any data stored at site that was not also stored in the digital drillhole database).

Meetings have been held at various times between the authors of this Technical Report and Company staff members, either in the Company's Vancouver Offices or at Golden Vertex's Bullhead City offices. The purpose was to discuss a broad range of project-related issues and/or to collect and collate Company information about the Moss Mine Project.

3 RELIANCE ON OTHER EXPERTS

The Moss Mine Project claims and exploration permit information presented in this Technical Report is based on information supplied by the Company. The claims information was cross-checked by reference to United States Bureau of Land Management claim documents. MineFill has made no attempt to verify legal ownership of, or title to, the various claims and the Arizona state exploration permit that comprise the Moss Mine Project area.

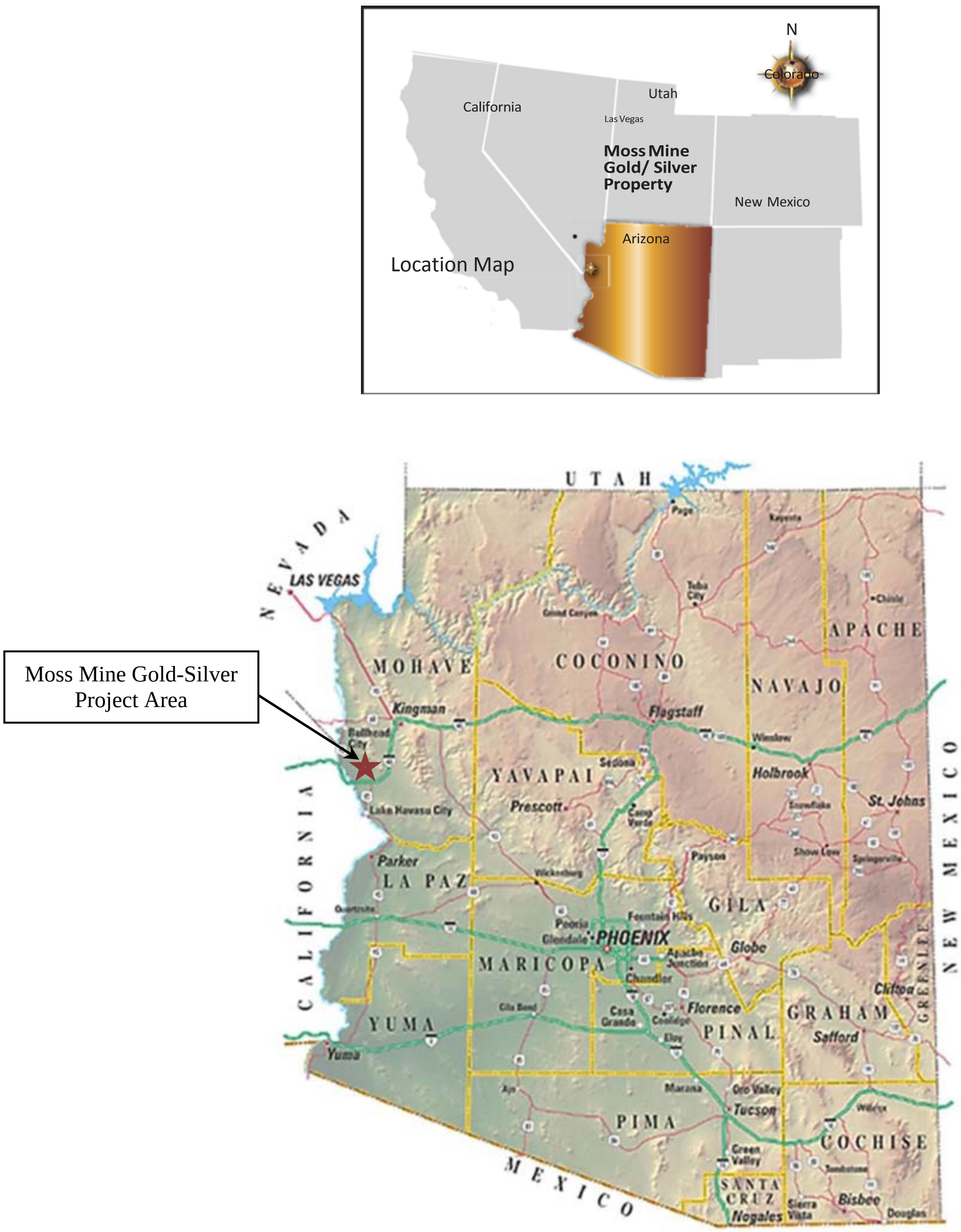
Signed copies of the various agreements pertaining to the Moss Mine Project were seen by MineFill. They were used to compile the information contained in parts of Section 4 of this Technical Report. MineFill is not, however, qualified to assess the validity of the agreements or the information contained therein, including the Company’s buy-in option concerning the Moss Mine Project or Patriot Gold’s Moss Mine Project claims.

MineFill is not qualified to assess environmental issues in the United States and has made no attempt to verify or assess environmental issues or liabilities on the Project area. MineFill can report on observations made during its site visits only, as well as issues that MineFill is made aware of by the Company, but this should not be considered a comprehensive overview of environmental issues.

4 PROPERTY DESCRIPTION AND LOCATION

Northern Vertex Corporation (the "Company") is focused on the reactivation of the Moss Mine Gold-Silver Project in Mohave County, northwest Arizona, USA (the "Moss Mine Project", Figure 4.1), where the Company has the right to earn-in a 70% property interest through joint venture with Patriot Gold Corporation ("Patriot Gold"). As earlier outlined (Section 2.2) and later described (Section 5.6), the Company has been actively working on the Project area as joint venture operator, through its wholly owned US subsidiary Golden Vertex Corporation ("Golden Vertex"). The joint venture agreement with Patriot Gold is described in Sub-Section 4.4.2.

Figure 4.1: A General Location Plan of the Moss Mine Gold-Silver Project
(copied from the Company's website: www.northernvertex.com)



4.1 Property Location

The Moss Mine Project area (the “Project area”) is centred on Latitude 35º 6’ 00” North, Longitude 114º 26’ 52” West (the “Property centre”), which was the approximate location of the historical headframe associated with historical underground mine workings, at the western end of the Moss Vein outcrop (see Section 8 for details, as earlier outlined [Section 2.2] the headframe was in 2013 relocated to Bullhead City). Bullhead City is approximately 15 km to the west and northwest of the Property centre (Figure 4.2).

4.2 Project Area

The total Project area comprises approximately 4,030.8 hectares (“ha”), including:

- 102.8 ha in 15 patented lode claims;
- approximately 3,827.1 ha in 468 unpatented lode claims to which various agreements and royalties apply; and
- one Arizona State exploration permit covering an area of 259 ha (640 acres or one section); but
- approximately 158.2 ha of overlap for a net area of approximately 4,030.8 ha (see Sub-Section 4.2.6 for details).

The total area of the unpatented lode claims and total area of overlap are estimates only. They should not be considered definitive or absolute values; they are stated for information purposes only. This is emphasized because only the patented lode claim boundaries have been surveyed by a registered land surveyor. The areas of the unpatented claims and overlaps were estimated through scrutiny, by MineFill, of AutoCad® claims files supplied by the Company.

The Company’s Moss Mine Project development plans are centred on the patented lode claims where the Company’s completed Phase I activities and future Phase II operations exploited, or are planned to exploit, the Moss Vein, its West Extension and their associated stockworks. Low-level exploration activity only is currently planned in respect of the Silver Creek claim block. An airborne magnetic survey has been carried out over the greater Moss Mine Project area (Section 9). In 2014 the Company initiated a follow-up geological mapping and sampling program to ‘*identify and prioritize areas for future drilling*’ (see the Company’s news release dated September 04, 2014).

4.2.1 Governing Law

Mining in the United States is governed by the Mining Law of 1872, which declared all valuable mineral deposits in lands belonging to the United States to be free and open to exploration and purchase. This law provides citizens of the United States the opportunity to explore, discover and purchase certain valuable mineral deposits on public domain land. The Federal Land Policy and Management Act of 1976 (“FLPMA”) did not amend the 1872 law, but did affect the recording and maintenance of claims: persons holding existing claims were required to record their claims with United States Bureau of Land Management (“BLM”) by October 1979; and all new claims were required to be recorded with BLM. The purpose of FLPMA was to provide BLM with information on the locations and number of unpatented mining claims, mill sites and tunnel sites, to determine the names and addresses of current owners and to remove the large number of titles on abandoned claims.

Figure 4.2: A Google Earth Image Showing the Location of the Moss Mine Gold-Silver Project, Relative to Bullhead City

4.2.2 Mining Claims and Land Rights

A mining claim is defined as a parcel of land for which the claimant has asserted a right of possession and the right to develop and extract a discovered and valuable mineral deposit. This right does not include exclusive surface rights. There are three basic types of minerals on federally-administered lands: locatable, leasable and saleable. Mining claims are staked on locatable minerals on public domain lands. Locatable minerals include both metallic minerals (gold, silver, lead, etc.) and non-metallic minerals (fluorspar, asbestos, mica, etc.).

There are two types of mining claims: lode; and placer. Placer claims are not relevant in the case of the Moss Mine Project and they are not considered further in this Technical Report. Lode claims (which apply in the case of the Moss Mine Project) are described by metes and bounds surveys providing the length and compass bearing of each boundary line from a central point or monument to each corner post, and then sequentially around the perimeter, including a reference to natural objects or permanent monuments.

A patented mining claim is one for which the Federal Government has passed title to the claim holder, thereby making it private land. The owner may mine and remove minerals from these claims in the same manner as unpatented claims. However, the patent provides the owner with full and exclusive title to the minerals on patent-relevant claims, as well as title to the surface area of these claims. The granting of mineral patents was suspended on October 1, 1994.

Unpatented claims are grants of mineral rights on public land owned by the Federal Government and administered by BLM. An annual maintenance fee is payable by September 01 each year to maintain the claim valid, in lieu of assessment work. The claim is forfeited if the fee is not paid. Numerous State and Federal regulations apply to every aspect of the exploration, development and production of natural resources from public lands. These include BLM, United States Environmental Protection Agency, Arizona Mine Inspector's Office and Arizona Department of Environmental Quality.

4.2.3 Arizona State Exploration Permits and Land Rights

Mining claims apply to Federal Lands regulated by BLM, whereas Arizona state exploration permits apply to Arizona State Trust Land regulated by the Arizona State Land Department ("ASLD"). Individual permits apply to a maximum area of 259 ha (640 acres or one whole section), they are valid for one year and are renewable for up to five years. Apart from a rent payable in respect of exploration permits, to keep a permit in good standing the minimum work expenditure requirements are US$10 per acre per year in Years One and Two, rising to US$20 per acre per year for Year Three through Five. Proof of work expenditures must be submitted to ASLD Minerals Section each year, in the form of invoices and paid receipts. If no work was completed the applicant can pay the equal amount to ASLD. An exploration permit does not confer the right to mine on the land subject to the exploration lease.

An exploration Plan of Operation must be submitted annually and approved by ASLD, prior to the start-up of exploration activities. If any surface disturbance is planned as part of exploration activity, archaeological and biological surveys as well as any other applicable permits must be submitted for ASLD prior review. If a discovery of a valuable mineral deposit is made the permitee must apply for a mineral lease before actual mining activities can begin.

4.2.4 Patented Lode Claims

The 15 patented lode claims are listed on Table 4.1, which was compiled from information supplied by the Company. The claim areas were determined by MineFill from an Autocad® file of the claims block supplied by the Company. The stated areas do not include overlaps, the areas of which were removed, by age precedence, from the nominal areas of the various claims. Key information was checked by cross-referencing:

- claim-related status reports located on the website of the Bureau of Land Management ("BLM", www.blm.gov);
- a condition of title report by Chicago Title Insurance Company of Jacksonville, Florida dated August 13, 2013; and
- Patriot Gold's Form 10-K to the United States Securities and Exchange Commission for the fiscal year ended May 31, 2014 (Commission file number 0-32919).

Each of the 15 patented lode claims is owned by Patriot Gold. Seven of the claims are subject to the terms of the MinQuest Agreement (Section 4.4.1); all 15 of the claims are subject to the joint venture agreement between Patriot Gold and the Company (the "Patriot Gold Agreement", Section 4.4.2). Figure 4.3 details the locations of the individual patented claims. Figure 4.4 identifies the claim block boundary in relation to the local topography and the outcrop positions of the Moss Vein and West Extension. The Mineral Resources that are the subject of this Technical Report (the "2014 Mineral Resource update") are located on the Key No. 1, Key No. 2, California Moss Lot 37 (Greenwood), California Moss Lot 38 (Gintoff) and Keystone Wedge patented lode claims.

The Company advised MineFill that the boundaries of the patented lode claims have legally been surveyed ('*the patented claims were laid out and surveyed by registered land surveyors in accordance with federal laws and approved by the United States Surveyor General*'). MineFill has seen a certified copy of the Record of Survey for the claims by Eric L. Stephan (Registered Land Surveyor #29274) of Cornerstone Land Surveying, Inc., located at Bullhead City, Arizona 86439, which is dated 29 February 2012.

Table 4.1: A Summary of the Patented Lode Claims
(Registered Owner Patriot Gold, Inc.) of the Moss Mine Project Area
(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	Mineral Survey	Township/ Range	Section	Date of Location	Date of Amended Location	Date of Mineral Survey	Claim Area (ha)
Key No. 1	MS4484	20 N / 20 W	19	Unknown	Not Applicable	April 1959	7.79
Key No. 2	MS4484	20 N / 20 W	19	Unknown	Not Applicable	April 1959	8.32
California Moss Lot 37 (Greenwood)	MS182	20 N / 20 W	19, 30	Unknown	Not Applicable	Before October 1888	8.20
California Moss Lot 38 (Gintoff)	MS796	20 N / 20 W	19, 20, 29, 30	Feb. 02, 1882	Not Applicable	Before October 1888	8.25
Moss Millsite	MS4484	20 N / 20 W	19	Unknown	Not Applicable	April 1959	5.51
Divide	MS4484	20 N / 20 W	19	Unknown	Not Applicable	April 1959	1.91
Keystone Wedge	MS4484	20 N / 20 W	19, 30	Unknown	Not Applicable	April 1959	4.05
Ruth Extension	MS4485	20 N / 20 W	29, 30	July 02, 1929	June 27, 1958	April 1959	7.78
Omega	MS4484	20 N / 20 W	19, 30	Unknown	Not Applicable	April 1959	8.29
Ruth	MS2213	20 N / 20 W	30	Oct. 15, 1888	Not Applicable	February 1906	7.33
Rattan Extension	MS4485	20 N / 20 W	30	July 02, 1929	June 27, 1958	April 1959	8.36
Rattan	MS857	20 N / 20 W	30	July 19, 1886	Not Applicable	October 1888	8.38
Partnership	MS4485	20 N / 20 W	30	June 27, 1958	June 27, 1958	April 1959	2.38
Mascot	MS4485	20 N / 20 W	30	June 27, 1958	June 27, 1958	April 1959	8.36
Empire	MS4485	20 N / 20 W	30	June 27, 1958	June 27, 1958	April 1959	7.91
						Total	***102.82***

4.2.5 Unpatented Lode Claims

Figure 4.5 is a general reference, colour-coded location plan for the 468 unpatented lode claims that, with the 15 patented lode claims described above and the Arizona State exploration permit described in Sub-Section 4.2.5.5, comprise the overall Moss Mine Project area. Claim plans covering all of the Moss Mine Project-related unpatented lode claims are provided as part of each following sub-section relating to the various claim blocks. The total of 468 unpatented lode claims includes:

- 104 unpatented claims in the name of MinQuest, Inc. (of Reno, Nevada - "MinQuest", a corporation that carries out geological consulting, contracting and exploration services), which are subject to MinQuest Agreement (Sub-Section 4.4.1) and the Patriot Gold Agreement (Section 4.4.2), the former inclusive of a royalty –
    - 63 of the claims were staked by MinQuest on April 26, 27 and 28, 2004 (Moss 11 to Moss 33, Moss 33F, Moss 34 to Moss 39, Moss 39F, Moss 40 to Moss 47, Moss 47B and Moss 48 to Moss 70),
    - 41 of the claims were staked by MinQuest on October 19, 2009 (Moss 1 to Moss 10 and Moss 118 to Moss 148);
- 170 unpatented lode claims staked by Golden Vertex on April 12 to 17 and May 01 to 04, 2011 (GVC 1 to GVC 31, GVC 33 to GVC 65, GVC 67 to GVC 139, GVC 146 to GVC 150, GVC 162, GVC 164 to GVC 168 and GVC 172 to GVC 193) –
    - not all the claims fall within the area of influence of the Patriot Gold Agreement and MinQuest Agreement, in some cases only portions of some the claims are subject to the terms of those agreements,
    - the total of 170 GVC claims does not include eight claims of the GVC series that were rendered invalid for the reasons described in Sub-Section 4.2.5.2;
- 11 unpatented lode claims (Moss 201 to 211) staked by Golden Vertex on June 27, 2012 and September 05, 2012, to fill-in gaps in the block of patented lode claims and along the southern boundary of the Moss 1 to Moss 148 block of claims –
    - all eleven claims fall within the areas of influence of the MinQuest Agreement and the Patriot Gold Agreement and are subject to the terms of those agreements (Section 4.4); and
- 183 unpatented lode claims (Silver Creek 1 to Silver Creek 22, Silver Creek 31 to Silver Creek 54, Silver Creek 63 to Silver Creek 97 and Silver Creek 108 to Silver Creek 209) staked by La Cuesta International, Inc. (of Kingman, Arizona - "La Cuesta") –
    - the Company has a 100% option agreement over all 183 claims (pursuant to the La Cuesta Agreement, which includes a royalty payment – see Sub-Sections 4.4.3 and 4.5.4), and
    - not all the claims fall within the area of influence of the Patriot Gold Agreement and MinQuest Agreement, in some cases only portions of some the claims are subject to the terms of those agreements.

Figure 4.3: A Location Plan for the 15 Patented Lode Claims, Moss Mine Project

(compiled from AutoCad® claims files supplied by the Company)

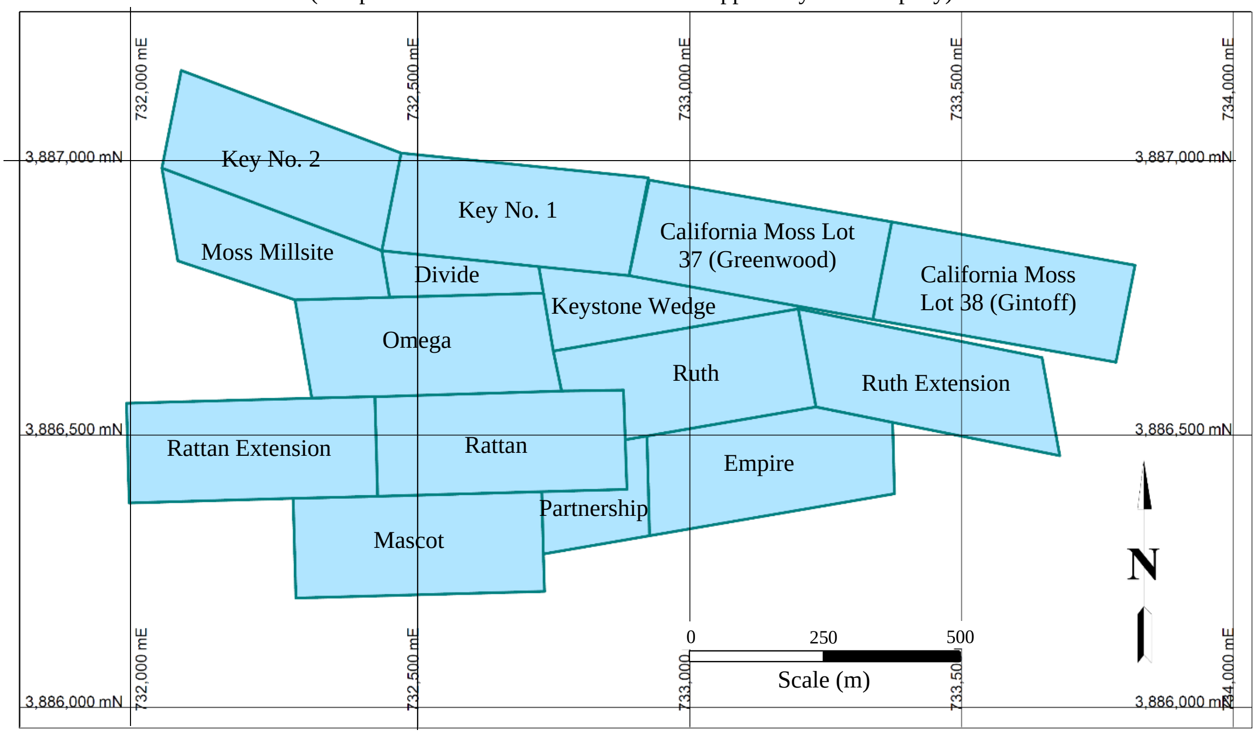


Figure 4.4: A Vulcan® Snapshot of the General Moss Mine Project Area Showing the Boundary of the Patented Claims and the Outcrops of the Moss Vein and West Extension

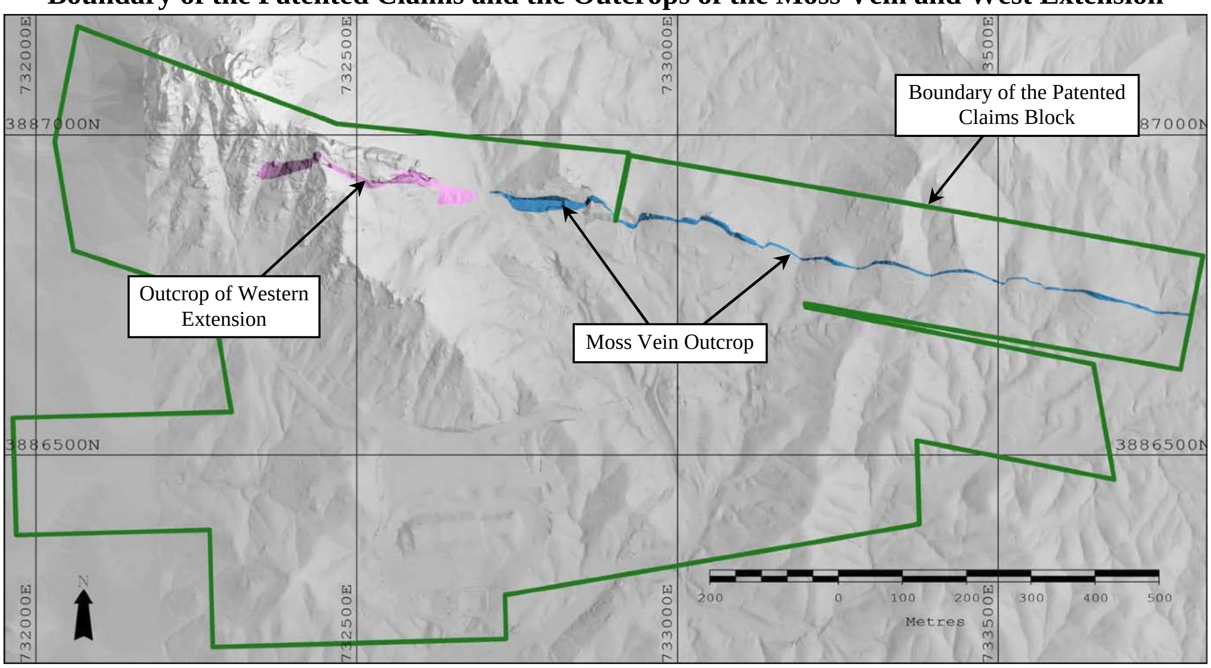
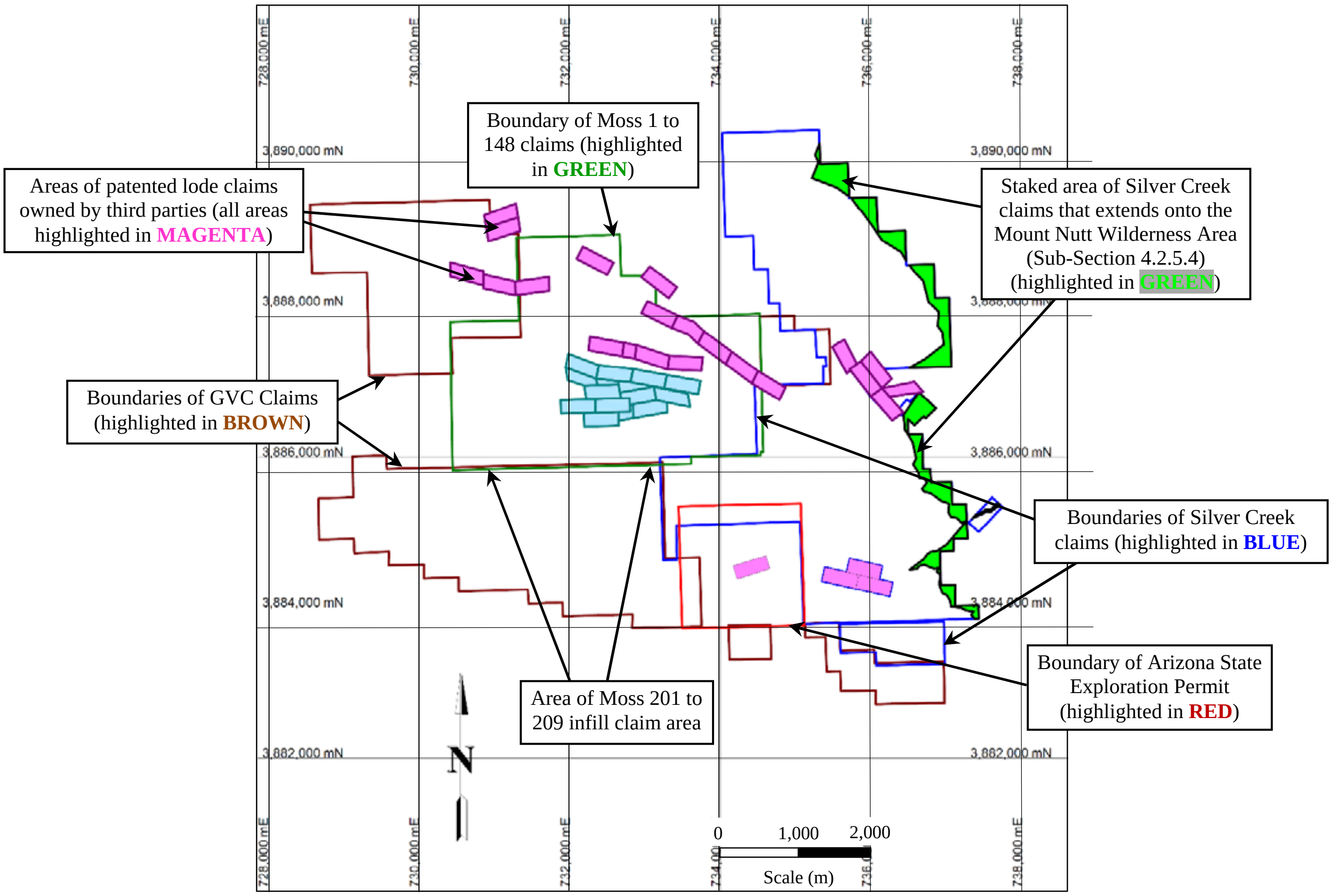


Figure 4.5: A Colour-Coded General Claim Block Reference Plan for the Moss Mine Project Claims and Arizona State Exploration Permit

(compiled from AutoCad® files of the claim areas supplied by the Company)

(refer to the following sub-sections for detailed claim plans)



The Company advised MineFill that the boundaries of the '*unpatented claims were properly laid out and monumented, but they have not been surveyed by a licensed land surveyor*'. According to BLM claims reports secured by MineFill from BLM's website (www.blm.gov), at the time of writing (December 2014) all of the unpatented lode claims listed above are classed as 'active', hence in good standing.

The maximum allowable size of unpatented lode claims in Arizona is 1,500 ft by 600 ft, which dimensions represent a regular unpatented lode claim. The equivalent area of such claims is 9,000 square feet or 8.361 ha. The vast majority of the various unpatented lode claims considered here have areas of 8.361 ha. The areas of individual claims with non-standard dimensions were estimated by MineFill from scrutiny of AutoCad® claims files supplied by the Company.

MineFill used the same AutoCad® files to estimate the portions of individual claims that overlap pre-existing claims and the portions of individual claims that fall within the areas of influence of the MinQuest Agreement and the Patriot Gold Agreement. The results are stated on Tables 4.2 to 4.5, inclusive. It is emphasized that the results are estimates only, that the estimates are stated for information purposes only and they should not be considered as definitive or absolute values.

4.2.5.1 Moss 1 to Moss 148 Series

Table 4.2 (that is in three parts due to its length) and Figure 4.6 summarize the details and locations of the Moss 1 to Moss 148 series of 104 unpatented lode claims that form a single block that surrounds the block of 15 patented lode claims. The total staked area of the Moss 1 to Moss 148 series of claims is estimated by MineFill at 869.54 ha. However, Moss 23 to Moss 28, Moss 33F, Moss 34, Moss 39F, Moss 40, Moss 46, Moss 47, Moss 47B, Moss 55 and Moss 56 overlap the block of patented lode claims described in Section 4.2.4. Patented lode claims take precedence over unpatented lode claims. The active areas of the overlapping Moss claims are stated in Sub-Section 4.2.6 in which the total estimated claim overlap area is defined.

Some of the listed claims occur in two sections (for example Moss 43). Each section of such claims are stated on Table 4.2; some details of individual claims are therefore repeated. The multi-section claims are indicated by the term 'ditto' in the Claim Name, BLM Serial Number and Lead File columns.

Patented lode claims, other than the 15 listed on Table 4.1, exist in the area covered by the Moss 1 to Moss 148 claim series. They are owned by third parties that are independent of the Company and Patriot Gold; their positions are indicated on Figure 4.6. As earlier outlined, patented lode claims have precedence over unpatented lode claims - unless through mutual agreement, activity on unpatented lode claims that overlap patented lode claims cannot take place.

Table 4.2: A Summary of MinQuest's Block of Unpatented Lode Claims (Moss Series), Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
Moss 1	AMC398978	14 0200N 0210W 024 NE, NW, SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 2	AMC398979	14 0200N 0210W 024 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 3	AMC398980	14 0200N 0210W 024 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 4	AMC398981	14 0200N 0210W 024 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 5	AMC398982	14 0200N 0210W 024 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 6	AMC398983	14 0200N 0210W 025 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 7	AMC398984	14 0200N 0210W 025 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 8	AMC398985	14 0200N 0210W 025 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 9	AMC398986	14 0200N 0210W 025 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 10	AMC398987	14 0200N 0210W 025 NE, NW, SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 11	AMC361998	14 0200N 0210W 024 NE, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 12	AMC361999	14 0200N 0210W 024 SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 13	AMC362000	14 0200N 0210W 024 SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 14	AMC362001	14 0200N 0210W 024 SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 15	AMC362002	14 0200N 0210W 024 SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 16	AMC362003	14 0200N 0210W 025 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 17	AMC362004	14 0200N 0210W 025 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 18	AMC362005	14 0200N 0210W 025 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 19	AMC362006	14 0200N 0210W 025 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 20	AMC362007	14 0200N 0210W 025 NE, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 21	AMC362008	14 0200N 0200W 019 NW, SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 22	AMC362009	14 0200N 0200W 019 SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 23	AMC362010	14 0200N 0200W 019 SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 24	AMC362011	14 0200N 0200W 019 SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 25	AMC362012	14 0200N 0200W 019 SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 26	AMC362013	14 0200N 0200W 030 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 27	AMC362014	14 0200N 0200W 030 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 28	AMC362015	14 0200N 0200W 030 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 29	AMC362016	14 0200N 0200W 030 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 30	AMC362017	14 0200N 0200W 030 NW, SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 31	AMC362018	14 0200N 0200W 019 NE, NW, SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 32	AMC362019	14 0200N 0200W 019 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 33	AMC362020	14 0200N 0200W 019 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 33F	AMC362021	14 0200N 0200W 019 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 34	AMC362022	14 0200N 0200W 030 NE, NW	AMC361998	April 26, 2004	8.361	100%	100%

Table 4.2 *continued*: A Summary of the Moss Series of Unpatented Lode Claims, Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
Moss 35	AMC362023	14 0200N 0200W 030 NE, NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 36	AMC362024	14 0200N 0200W 030 NE, NW, SW, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 37	AMC362025	14 0200N 0200W 019 NE, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 38	AMC362026	14 0200N 0200W 019 SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 39	AMC362027	14 0200N 0200W 019 SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 39F	AMC362028	14 0200N 0200W 019 SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 40	AMC362029	14 0200N 0200W 030 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 41	AMC362030	14 0200N 0200W 030 NE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 42	AMC362031	14 0200N 0200W 030 NE, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 43	AMC362032	14 0200N 0200W 019 NE, SE	AMC361998	April 27, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 020 NW, SW	ditto	April 27, 2004	-	100%	100%
Moss 44	AMC362033	14 0200N 0200W 019 SE	AMC361998	April 27, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 020 SW	ditto	April 27, 2004	-	100%	100%
Moss 45	AMC362034	14 0200N 0200W 019 SE	AMC361998	April 27, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 020 SW	ditto	April 27, 2004	-	100%	100%
Moss 46	AMC362035	14 0200N 0200W 019 SE	AMC361998	April 28, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 020 SW	ditto	April 28, 2004	-	100%	100%
Moss 47	AMC362036	14 0200N 0210W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0210W 030 NE	ditto	April 26, 2004	-	100%	100%
Moss 47B	AMC362037	14 0200N 0200W 029 NW	AMC361998	April 28, 2004	8.361	100%	100%
Moss 48	AMC362038	14 0200N 0210W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0210W 030 NE	ditto	April 26, 2004	-	100%	100%
Moss 49	AMC362039	14 0200N 0210W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0210W 030 NE	ditto	April 26, 2004	-	100%	100%
Moss 50	AMC362040	14 0200N 0210W 029 NW, SW	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0210W 030 NE, SE	ditto	April 26, 2004	-	100%	100%
Moss 51	AMC362041	14 0200N 0200W 020 NW, SW	AMC361998	April 27, 2004	8.361	100%	100%
Moss 52	AMC362042	14 0200N 0200W 020 SW	AMC361998	April 27, 2004	8.361	100%	100%
Moss 53	AMC362043	14 0200N 0200W 020 SW	AMC361998	April 27, 2004	8.361	100%	100%
Moss 54	AMC362044	14 0200N 0200W 020 SW	AMC361998	April 27, 2004	8.361	100%	100%
Moss 55	AMC362045	14 0200N 0200W 020 SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 56	AMC362046	14 0200N 0200W 020 SW	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 029 NW	ditto	April 26, 2004	-	100%	100%
Moss 57	AMC362047	14 0200N 0200W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 58	AMC362048	14 0200N 0200W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 59	AMC362049	14 0200N 0200W 029 NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 60	AMC362050	14 0200N 0200W 029 NW, SW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 61	AMC362051	14 0200N 0200W 020 NE, NW, SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 62	AMC362052	14 0200N 0200W 020 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 63	AMC362053	14 0200N 0200W 020 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 64	AMC362054	14 0200N 0200W 020 SW, SE	AMC361998	April 27, 2004	8.361	100%	100%
Moss 65	AMC362055	14 0200N 0200W 020 SW, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 66	AMC362056	14 0200N 0200W 020 SW, SE	AMC361998	April 26, 2004	8.361	100%	100%
ditto	ditto	14 0200N 0200W 029 NE, NW	ditto	April 26, 2004	-	100%	100%
Moss 67	AMC362057	14 0200N 0200W 029 NE, NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 68	AMC362058	14 0200N 0200W 029 NE, NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 69	AMC362059	14 0200N 0200W 029 NE, NW	AMC361998	April 26, 2004	8.361	100%	100%
Moss 70	AMC362060	14 0200N 0200W 029 NE, NW, SW, SE	AMC361998	April 26, 2004	8.361	100%	100%
Moss 118	AMC398988	14 0200N 0210W 024 NW, SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 119	AMC398989	14 0200N 0210W 024 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 120	AMC398990	14 0200N 0210W 024 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 121	AMC398991	14 0200N 0210W 024 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 122	AMC398992	14 0200N 0210W 024 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 123	AMC398993	14 0200N 0210W 025 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 124	AMC398994	14 0200N 0210W 025 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 125	AMC398995	14 0200N 0210W 025 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 126	AMC398996	14 0200N 0210W 025 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 127	AMC398997	14 0200N 0210W 025 NW, SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 128	AMC398998	14 0200N 0210W 013 SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 129	AMC398999	14 0200N 0210W 013 SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 130	AMC399000	14 0200N 0210W 013 SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
ditto	ditto	14 0200N 0210W 024 NE	ditto	Oct. 19, 2009	-	100%	100%
Moss 131	AMC399001	14 0200N 0210W 024 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 132	AMC399002	14 0200N 0210W 024 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 133	AMC399003	14 0200N 0210W 024 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 134	AMC399004	14 0200N 0200W 018 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 135	AMC399005	14 0200N 0200W 018 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 136	AMC399006	14 0200N 0200W 018 SW	AMC398978	Oct. 19, 2009	8.361	100%	100%
ditto	ditto	14 0200N 0200W 019 NW	ditto	Oct. 19, 2009	-	100%	100%
Moss 137	AMC399007	14 0200N 0200W 019 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 138	AMC399008	14 0200N 0200W 019 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 139	AMC399009	14 0200N 0200W 019 NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 140	AMC399010	14 0200N 0200W 018 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%

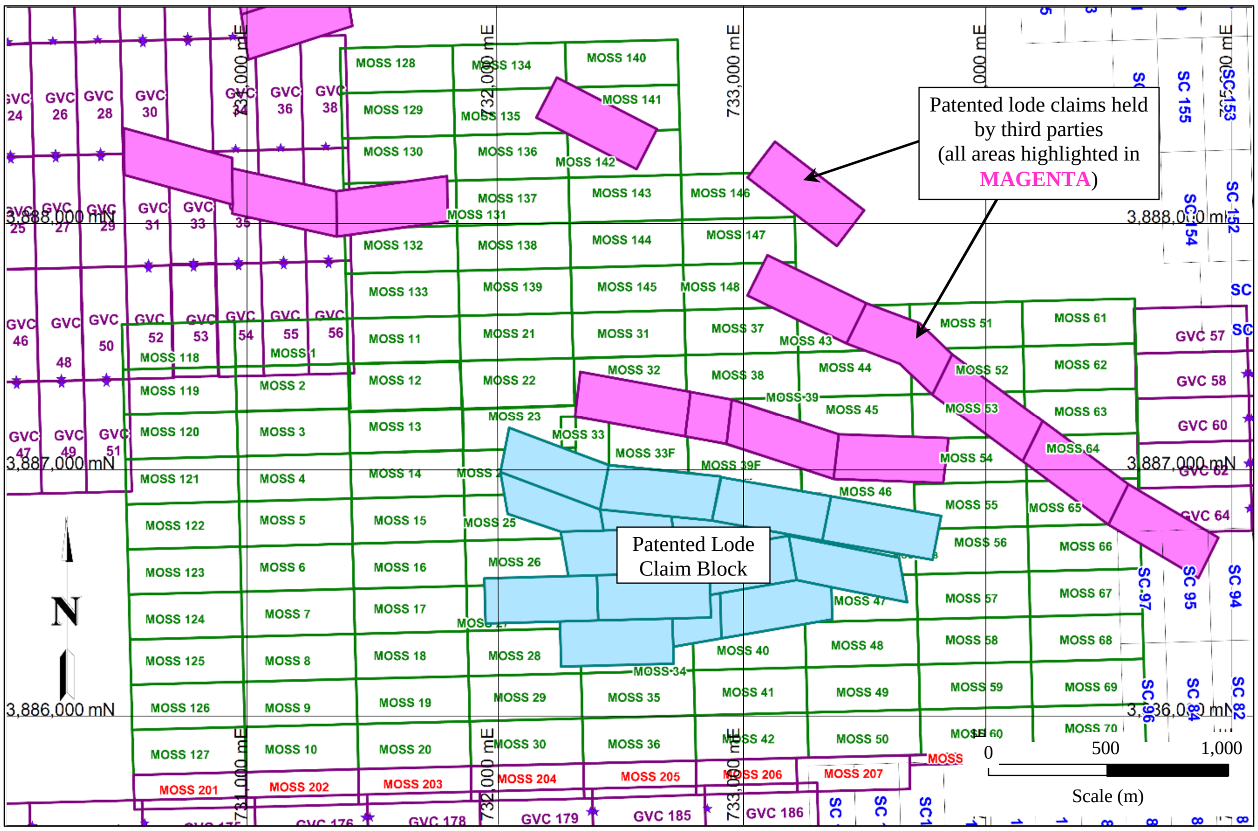
Table 4.2 *continued*: A Summary of the Moss Series of Unpatented Lode Claims, Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
Moss 141	AMC399011	14 0200N 0200W 018 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 142	AMC399012	14 0200N 0200W 018 SW, SE	AMC398978	Oct. 19, 2009	8.361	100%	100%
ditto	ditto	14 0200N 0200W 019 NE, NW	ditto	Oct. 19, 2009	-	100%	100%
Moss 143	AMC399013	14 0200N 0200W 019 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 144	AMC399014	14 0200N 0200W 019 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 145	AMC399015	14 0200N 0200W 019 NE, NW	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 146	AMC399016	14 0200N 0200W 019 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 147	AMC399017	14 0200N 0200W 019 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
Moss 148	AMC399018	14 0200N 0200W 019 NE	AMC398978	Oct. 19, 2009	8.361	100%	100%
				Total Area	***869.544***		

Figure 4.6: A Location Plan for the Moss 1 to Moss 148 (highlighted in GREEN) and Moss 201 to Moss 209 Series (labelled in RED) of Unpatented Lode Claims, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company)



4.2.5.2 GVC Claim Series

Table 4.3 (that is in four parts due to its length) summarizes the details of the GVC series of 170 unpatented lode claims that have an estimated total staked area of 1,421.37 ha. The listed series of staked claims does not include GVC 158 to 161, GVC 163 and GVC 169 to 171 that were found to be invalid: they over-staked an area of already existing, active unpatented lode claims held by a third party. When the third party claims became invalid, the area they covered was staked as part of the Silver Creek series of unpatented lode claims described in Section 4.2.5.4.

Each of the 170 GVC claims has the dimensions hence area of a regular unpatented lode claim (8.361 ha). However, GVC 38, GVC 39 and GVC50 to GVC 56 overlap portions of the Moss 1 to Moss 148 series of claims described in Section 4.2.5.1. The Moss 1 to Moss 148 series of unpatented lode claims takes precedence as they were staked before the GVC series of unpatented claims. The estimated active areas of the overlapping GVC claims are stated in Section 4.2.6 in which the estimated total overlap area is defined.

Some of the listed GVC claims occur in two or even four sections (for example GVC 24 and GVC 26). Each section of such claims is stated on Table 4.3 so some details of individual claims are repeated. The multi-section claims are indicated by the term ‘ditto’ in the Claim Name, BLM Serial Number and Lead File columns.

The percent areas of each claim that are subject to the MinQuest Agreement and to the Patriot Agreement were estimated by MineFill from consideration of the position of the one mile areas-of-interest around the blocks of unpatented lode claims subject to the agreements (see Sub-Sections 4.4.1 and 4.4.2 for details). The positions of the one mile areas-of-interest lines from the Moss claim block boundary were drawn and the areas of each GVC series claim it intersected were estimated using the AutoCad® claims files supplied by the Company. The percentages of each claim were then estimated by dividing the area of any claim located wholly or partially within the one mile line by the total area of the same claim.

To facilitate legibility, the locations of the GVC series of unpatented claims are presented on three plans (Figures 4.7 to 4.9, inclusive). The plans include the blocks of third party patented lode claims that exist on the ground covered by the GVC claims. The position of each illustrated block of GVC series claims, relative to the 15 patented lode claims and the Moss 1 to Moss 148 series of unpatented lode claims, can be determined by reference to Figure 4.5.

Table 4.3: A Summary of the Golden Vertex Block of Unpatented Lode Claims (GVC Series), Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
GVC 1	AMC408939	14 0200N 0210W 014 NW, SW	AMC408939	April 12, 2011	8.361	0.00	0.00
GVC 2	AMC408940	14 0200N 0210W 014 NW, SW	AMC408939	April 12, 2011	8.361	0.00	0.00
GVC 3	AMC408941	14 0200N 0210W 014 NW, SW	AMC408939	April 12, 2011	8.361	0.00	0.00
GVC 4	AMC408942	14 0200N 0210W 014 NW, SW	AMC408939	April 12, 2011	8.361	0.00	4.04
GVC 5	AMC408943	14 0200N 0210W 014 NW, NE, SW, SE	AMC408939	April 12, 2011	8.361	0.00	34.64
GVC 6	AMC408944	14 0200N 0210W 014 NE, SE	AMC408939	April 12, 2011	8.361	0.00	59.23
GVC 7	AMC408945	14 0200N 0210W 014 NE, SE	AMC408939	April 12, 2011	8.361	0.00	94.15
GVC 8	AMC408946	14 0200N 0210W 014 NE, SE	AMC408939	April 13, 2011	8.361	0.00	100.00
GVC 9	AMC408947	14 0200N 0210W 014 NE, SE	AMC408939	April 13, 2011	8.361	2.56	100.00
ditto	ditto	14 0200N 0210W 013 NW, SW	ditto	April 13, 2011	-	2.56	100.00
GVC 10	AMC408948	14 0200N 0210W 013 NW, SW	AMC408939	April 13, 2011	8.361	54.89	100.00
GVC 11	AMC408949	14 0200N 0210W 013 NW, SW	AMC408939	April 13, 2011	8.361	75.18	100.00
GVC 12	AMC408950	14 0200N 0210W 013 NW, SW	AMC408939	April 13, 2011	8.361	86.64	100.00
GVC 13	AMC408951	14 0200N 0210W 013 NW, SW	AMC408939	April 13, 2011	8.361	100.00	100.00
GVC 14	AMC408952	14 0200N 0210W 014 SW	AMC408939	April 12, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0210W 023 NW	ditto	April 12, 2011	-	0.00	0.00
GVC 15	AMC408953	14 0200N 0210W 014 SW	AMC408939	April 12, 2011	8.361	0.00	1.47
ditto	ditto	14 0200N 0210W 023 NW	ditto	April 12, 2011	-	0.00	1.47
GVC 16	AMC408954	14 0200N 0210W 014 SW	AMC408939	April 12, 2011	8.361	0.00	47.22
ditto	ditto	14 0200N 0210W 023 NW	ditto	April 12, 2011	-	0.00	47.22
GVC 17	AMC408955	14 0200N 0210W 014 SW	AMC408939	April 12, 2011	8.361	0.00	92.85
ditto	ditto	14 0200N 0210W 023 NW	ditto	April 12, 2011	-	0.00	92.85
GVC 18	AMC408956	14 0200N 0210W 014 SW, SE	AMC408939	April 13, 2011	8.361	0.00	100.00
ditto	ditto	14 0200N 0210W 023 NW, NE	ditto	April 13, 2011	-	0.00	100.00
GVC 19	AMC408957	14 0200N 0210W 023 NW, NE	AMC408939	April 13, 2011	8.361	0.00	100.00

Table 4.3 *continued*: A Summary of the Golden Vertex Block of Unpatented Lode Claims (GVC Series), Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
GVC 20	AMC408958	14 0200N 0210W 014 SE	AMC408939	April 13, 2011	8.361	0.00	100.00
ditto	ditto	14 0200N 0210W 023 NE	ditto	April 13, 2011	-	0.00	100.00
GVC 21	AMC408959	14 0200N 0210W 023 NE	AMC408939	April 13, 2011	8.361	0.00	100.00
GVC 22	AMC408960	14 0200N 0210W 014 SE	AMC408939	April 13, 2011	8.361	1.54	100.00
ditto	ditto	14 0200N 0210W 023 NE	ditto	April 13, 2011	-	1.54	100.00
GVC 23	AMC408961	14 0200N 0210W 023 NE	AMC408939	April 13, 2011	8.361	64.57	100.00
GVC 24	AMC408962	14 0200N 0210W 014 SE	AMC408939	April 17, 2011	8.361	45.31	100.00
ditto	ditto	14 0200N 0210W 023 NE	ditto	April 17, 2011	-	45.31	100.00
GVC 25	AMC408963	14 0200N 0210W 023 NE	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 26	AMC408964	14 0200N 0210W 013 SE	AMC408939	April 17, 2011	8.361	92.08	100.00
ditto	ditto	14 0200N 0210W 014 SW	ditto	April 17, 2011	-	92.08	100.00
ditto	ditto	14 0200N 0210W 024 NW	ditto	April 17, 2011	-	92.08	100.00
ditto	ditto	14 0200N 0210W 023 SW	ditto	April 17, 2011	-	92.08	100.00
GVC 27	AMC408965	14 0200N 0210W 023 NE	AMC408939	April 17, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 024 NW	ditto	April 17, 2011	-	100.00	100.00
GVC 28	AMC408966	14 0200N 0210W 013 SW	AMC408939	April 17, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 024 NW	ditto	April 17, 2011	-	100.00	100.00
GVC 29	AMC408967	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 30	AMC408968	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 31	AMC408969	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 33	AMC408971	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 34	AMC408972	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 013 SW	ditto	April 17, 2011	-	100.00	100.00
GVC 35	AMC408973	14 0200N 0210W 024 NW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 36	AMC408974	14 0200N 0210W 024 NW, NE	AMC408939	April 17, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 013 SW, SE	ditto	April 17, 2011	-	100.00	100.00
GVC 37	AMC408975	14 0200N 0210W 024 NW, NE	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 38	AMC408976	14 0200N 0210W 013 SE	AMC408939	April 17, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 024 NE	ditto	April 17, 2011	-	100.00	100.00
GVC 39	AMC408977	14 0200N 0210W 024 NE	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 40	AMC408978	14 0200N 0210W 023 NW, NE, SW, SE	AMC408939	April 12, 2011	8.361	0.00	100.00
GVC 41	AMC408979	14 0200N 0210W 023 SW, SE	AMC408939	April 12, 2011	8.361	0.00	100.00
GVC 42	AMC408980	14 0200N 0210W 023 NE, SE	AMC408939	April 12, 2011	8.361	5.72	100.00
GVC 43	AMC408981	14 0200N 0210W 023 SE	AMC408939	April 12, 2011	8.361	9.20	100.00
GVC 44	AMC408982	14 0200N 0210W 023 NE, SE	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 45	AMC408983	14 0200N 0210W 023 SE	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 46	AMC408984	14 0200N 0210W 023 NE, SE	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 47	AMC408985	14 0200N 0210W 023 NE, SE	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 48	AMC408986	14 0200N 0210W 023 NE, SE	AMC408939	April 12, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 024 NW, SW	ditto	April 12, 2011	-	100.00	100.00
GVC 49	AMC408987	14 0200N 0210W 023 SE	AMC408939	April 12, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 024 SW	ditto	April 12, 2011	-	100.00	100.00
GVC 50	AMC408988	14 0200N 0210W 024 NW, SW	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 51	AMC408989	14 0200N 0210W 024 SW	AMC408939	April 12, 2011	8.361	100.00	100.00
GVC 52	AMC408990	14 0200N 0210W 024 NW, SW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 53	AMC408991	14 0200N 0210W 024 NW, SW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 54	AMC408992	14 0200N 0210W 024 NW, SW	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 55	AMC408993	14 0200N 0210W 024 NW, NE, SW, SE	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 56	AMC408994	14 0200N 0210W 024 NE, SE	AMC408939	April 17, 2011	8.361	100.00	100.00
GVC 57	AMC408995	14 0200N 0200W 020 NE, SE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 58	AMC408996	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 59	AMC408997	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 021 SW	ditto	April 16, 2011	-	100.00	100.00
GVC 60	AMC408998	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 61	AMC408999	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 021 SW	ditto	April 16, 2011	-	100.00	100.00
GVC 62	AMC409000	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 63	AMC409001	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 021 SW	ditto	April 16, 2011	-	100.00	100.00
GVC 64	AMC409002	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 029 NE	ditto	April 16, 2011	-	100.00	100.00
GVC 65	AMC409003	14 0200N 0200W 020 SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 021 SW	ditto	April 16, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0200W 028 NW	ditto	April 16, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0200W 029 NE	ditto	April 16, 2011	-	100.00	100.00
GVC 67	AMC409004	14 0200N 0210W 026 SW	AMC408939	April 14, 2011	8.361	0.00	38.67
GVC 68	AMC409005	14 0200N 0210W 026 SW	AMC408939	April 14, 2011	8.361	0.00	20.58
ditto	ditto	14 0200N 0210W 035 NW	ditto	April 14, 2011	-	0.00	20.58
GVC 69	AMC409006	14 0200N 0210W 035 NW	AMC408939	April 14, 2011	8.361	0.00	1.30
GVC 70	AMC409007	14 0200N 0210W 026 SW, SE	AMC408939	April 14, 2011	8.361	0.00	100.00
GVC 71	AMC409008	14 0200N 0210W 026 SW, SE	AMC408939	April 14, 2011	8.361	0.00	100.00
GVC 72	AMC409009	14 0200N 0210W 026 SE	AMC408939	April 14, 2011	8.361	66.94	100.00
GVC 73	AMC409010	14 0200N 0210W 026 SW, SE	AMC408939	April 14, 2011	8.361	0.00	100.00

Table 4.3 *continued*: A Summary of the Golden Vertex Block of Unpatented Lode Claims (GVC Series), Moss Mine Project Area

(compiled from information from various sources, including Company documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
GVC 74	AMC409011	14 0200N 0210W 026 SE	AMC408939	April 14, 2011	8.361	59.28	100.00
GVC 75	AMC409012	14 0200N 0210W 026 SW, SE	AMC408939	April 14, 2011	8.361	0.00	100.00
GVC 76	AMC409013	14 0200N 0210W 026 SE	AMC408939	April 14, 2011	8.361	45.82	100.00
GVC 77	AMC409014	14 0200N 0210W 026 SW, SE	AMC408939	April 14, 2011	8.361	0.00	100.00
ditto	ditto	14 0200N 0210W 035 NW, NE	ditto	April 14, 2011	-	0.00	100.00
GVC 78	AMC409015	14 0200N 0210W 026 SE	AMC408939	April 14, 2011	8.361	27.58	100.00
ditto	ditto	14 0200N 0210W 035 NE	ditto	April 14, 2011	-	27.58	100.00
GVC 79	AMC409016	14 0200N 0210W 035 NW, NE	AMC408939	April 14, 2011	8.361	0.00	94.22
GVC 80	AMC409017	14 0200N 0210W 035 NE	AMC408939	April 14, 2011	8.361	5.71	100.00
GVC 81	AMC409018	14 0200N 0210W 035 NW, NE	AMC408939	April 14, 2011	8.361	0.00	66.15
GVC 82	AMC409019	14 0200N 0210W 035 NE	AMC408939	April 14, 2011	8.361	0.00	100.00
GVC 83	AMC409020	14 0200N 0210W 035 NE	AMC408939	April 14, 2011	8.361	0.00	100.00
GVC 84	AMC409021	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 026 SE	ditto	April 14, 2011	-	100.00	100.00
GVC 85	AMC409022	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 026 SE	ditto	April 14, 2011	-	100.00	100.00
GVC 86	AMC409023	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
GVC 87	AMC409024	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 026 SE	ditto	April 14, 2011	-	100.00	100.00
GVC 88	AMC409025	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
GVC 89	AMC409026	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 026 SE	ditto	April 14, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0210W 035 NE	ditto	April 14, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NW	ditto	April 14, 2011	-	100.00	100.00
GVC 90	AMC409027	14 0200N 0210W 025 SW	AMC408939	April 14, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NW	ditto	April 14, 2011	-	100.00	100.00
GVC 91	AMC409028	14 0200N 0210W 035 NE	AMC408939	April 15, 2011	8.361	98.64	100.00
ditto	ditto	14 0200N 0210W 036 NW	ditto	April 15, 2011	-	98.64	100.00
GVC 92	AMC409029	14 0200N 0210W 036 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 93	AMC409030	14 0200N 0210W 035 NE	AMC408939	April 15, 2011	8.361	70.23	100.00
ditto	ditto	14 0200N 0210W 036 NW	ditto	April 15, 2011	-	70.23	100.00
GVC 94	AMC409031	14 0200N 0210W 036 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 95	AMC409032	14 0200N 0210W 035 NE	AMC408939	April 15, 2011	8.361	23.59	100.00
ditto	ditto	14 0200N 0210W 036 NW	ditto	April 15, 2011	-	23.59	100.00
GVC 96	AMC409033	14 0200N 0210W 036 NW	AMC408939	April 15, 2011	8.361	99.83	100.00
GVC 97	AMC409034	14 0200N 0210W 035 NE, SE	AMC408939	April 15, 2011	8.361	0.00	99.69
ditto	ditto	14 0200N 0210W 036 NW, SW	ditto	April 15, 2011	-	0.00	99.69
GVC 98	AMC409035	14 0200N 0210W 036 NW	AMC408939	April 15, 2011	8.361	51.81	100.00
GVC 99	AMC409036	14 0200N 0210W 036 SW	AMC408939	April 15, 2011	8.361	22.63	22.63
GVC 100	AMC409037	14 0200N 0210W 025 SW, SE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 101	AMC409038	14 0200N 0210W 025 SW, SE	AMC408939	April 15, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NW, NE	ditto	April 15, 2011	-	100.00	100.00
GVC 102	AMC409039	14 0200N 0200W 030 SW	AMC408939	April 15, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 031 NW	ditto	April 15, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0210W 025 SE	ditto	April 15, 2011	-	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NE	ditto	April 15, 2011	-	100.00	100.00
GVC 103	AMC409040	14 0200N 0210W 036 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 104	AMC409041	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NE	ditto	April 15, 2011	-	100.00	100.00
GVC 105	AMC409042	14 0200N 0210W 036 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 106	AMC409043	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NE	ditto	April 15, 2011	-	100.00	100.00
GVC 107	AMC409044	14 0200N 0210W 036 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 108	AMC409045	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NE	ditto	April 15, 2011	-	100.00	100.00
GVC 109	AMC409046	14 0200N 0210W 036 NW, NE, SW, SE	AMC408939	April 16, 2011	8.361	99.62	100.00
GVC 110	AMC409047	14 0200N 0200W 031 NW, SW	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 036 NE, SE	ditto	April 16, 2011	-	100.00	100.00
GVC 111	AMC409048	14 0200N 0210W 036 SW, SE	AMC408939	April 16, 2011	8.361	15.76	22.63
GVC 112	AMC409049	14 0200N 0200W 031 SW	AMC408939	April 16, 2011	8.361	22.63	22.63
ditto	ditto	14 0200N 0210W 036 SE	ditto	April 16, 2011	-	22.63	22.63
GVC 113	AMC409050	14 0200N 0200W 031 SW	AMC408939	April 16, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0210W 036 SE	ditto	April 16, 2011	-	0.00	0.00
GVC 114	AMC409051	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 115	AMC409052	14 0200N 0200W 031 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 116	AMC409053	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 117	AMC409054	14 0200N 0200W 031 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 118	AMC409055	14 0200N 0200W 031 NW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 119	AMC409056	14 0200N 0200W 031 NW, NE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 120	AMC409057	14 0200N 0200W 031 NW, SW	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 121	AMC409058	14 0200N 0200W 031 NW, NE, SW, SE	AMC408939	April 15, 2011	8.361	100.00	100.00
GVC 122	AMC409059	14 0200N 0200W 031 SW	AMC408939	April 15, 2011	8.361	22.63	22.63
GVC 123	AMC409060	14 0200N 0200W 031 SW, SE	AMC408939	April 15, 2011	8.361	22.63	22.63

Table 4.3 *continued*: A Summary of the Golden Vertex Block of Unpatented Lode Claims (GVC Series), Moss Mine Project Area

(compiled from information from various sources, including Company Documents and BLM Claim Reports)

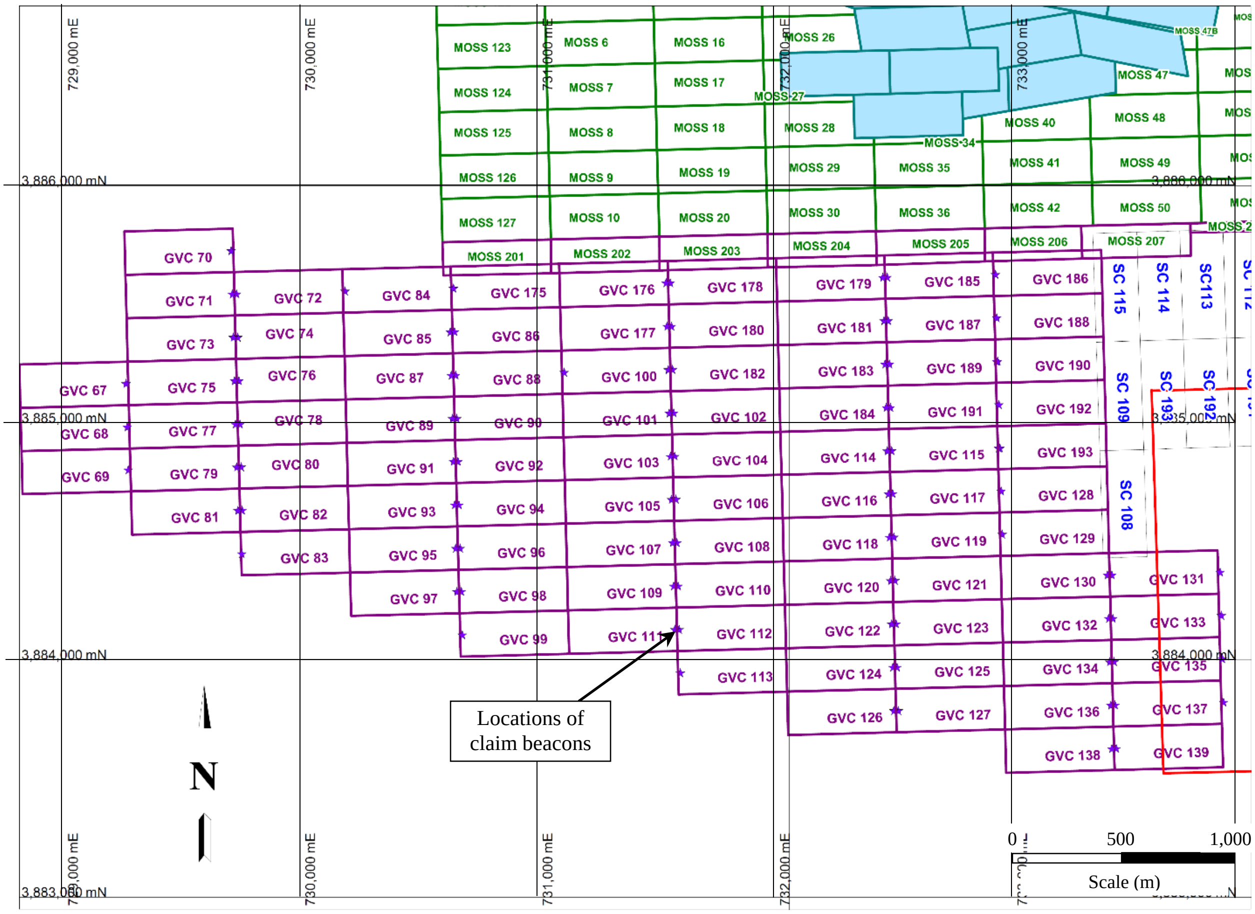
Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
GVC 124	AMC409061	14 0200N 0200W 031 SW	AMC408939	April 15, 2011	8.361	0.00	0.00
GVC 125	AMC409062	14 0200N 0200W 031 SW, SE	AMC408939	April 15, 2011	8.361	0.00	0.00
GVC 126	AMC409063	14 0200N 0200W 031 SW	AMC408939	April 15, 2011	8.361	0.00	0.00
GVC 127	AMC409064	14 0200N 0200W 031 SW, SE	AMC408939	April 15, 2011	8.361	0.00	0.00
GVC 128	AMC409065	14 0200N 0200W 031 NE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 129	AMC409066	14 0200N 0200W 031 NE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 130	AMC409067	14 0200N 0200W 031 NE, SE	AMC408939	April 16, 2011	8.361	100.00	100.00
GVC 131	AMC409068	14 0200N 0200W 031 NE, SE	AMC408939	April 16, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 032 NW, SW	ditto	April 16, 2011	-	100.00	100.00
GVC 132	AMC409069	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	22.63	22.63
GVC 133	AMC409070	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	22.63	22.63
ditto	ditto	14 0200N 0200W 032 SW	ditto	April 16, 2011	-	22.63	22.63
GVC 134	AMC409071	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
GVC 135	AMC409072	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SW	ditto	April 16, 2011	-	0.00	0.00
GVC 136	AMC409073	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
GVC 137	AMC409074	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SW	ditto	April 16, 2011	-	0.00	0.00
GVC 138	AMC409075	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
GVC 139	AMC409076	14 0200N 0200W 031 SE	AMC408939	April 16, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SW	ditto	April 16, 2011	-	0.00	0.00
GVC 146	AMC409082	14 0190N 0200W 005 NW	AMC408939	May 03, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SW	ditto	May 03, 2011	-	0.00	0.00
GVC 147	AMC409083	14 0190N 0200W 005 NW, NE	AMC408939	May 03, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SW, SE	ditto	May 03, 2011	-	0.00	0.00
GVC 148	AMC409084	14 0190N 0200W 005 NE	AMC408939	May 03, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 032 SE	ditto	May 03, 2011	-	0.00	0.00
GVC 149	AMC409085	14 0190N 0200W 004 NW	AMC408939	May 03, 2011	8.361	0.00	0.00
ditto	ditto	14 0200N 0200W 033 SW	ditto	May 03, 2011	-	0.00	0.00
GVC 150	AMC409086	14 0190N 0200W 004 NW	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 162	AMC409091	14 0190N 0200W 004 NW, NE	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 164	AMC409093	14 0190N 0200W 004 NW, NE	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 165	AMC409094	14 0190N 0200W 004 NE	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 166	AMC409095	14 0190N 0200W 004 NW, NE, SW, SE	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 167	AMC409096	14 0190N 0200W 004 NE, SE	AMC408939	May 03, 2011	8.361	0.00	0.00
GVC 168	AMC409097	14 0190N 0200W 004 SE	AMC408939	May 04, 2011	8.361	0.00	0.00
GVC 172	AMC409101	14 0190N 0200W 003 NW	AMC408939	May 04, 2011	8.361	0.00	0.00
ditto	ditto	14 0190N 0200W 004 NE	ditto	May 04, 2011	-	0.00	0.00
GVC 173	AMC409102	14 0190N 0200W 003 SW	AMC408939	May 04, 2011	8.361	0.00	0.00
ditto	ditto	14 0190N 0200W 004 SE	ditto	May 04, 2011	-	0.00	0.00
GVC 174	AMC409103	14 0190N 0200W 003 SW	AMC408939	May 04, 2011	8.361	0.00	0.00
ditto	ditto	14 0190N 0200W 004 SE	ditto	May 04, 2011	-	0.00	0.00
GVC 175	AMC409104	14 0200N 0210W 025 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
GVC 176	AMC409105	14 0200N 0210W 025 SW, SE	AMC408939	May 02, 2011	8.361	100.00	100.00
GVC 177	AMC409106	14 0200N 0210W 025 SW, SE	AMC408939	May 02, 2011	8.361	100.00	100.00
GVC 178	AMC409107	14 0200N 0200W 030 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 025 SE	ditto	May 02, 2011	-	100.00	100.00
GVC 179	AMC409108	14 0200N 0200W 030 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
GVC 180	AMC409109	14 0200N 0200W 030 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 025 SE	ditto	May 02, 2011	-	100.00	100.00
GVC 181	AMC409110	14 0200N 0200W 030 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
GVC 182	AMC409111	14 0200N 0200W 030 SW	AMC408939	May 02, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0210W 025 SE	ditto	May 02, 2011	-	100.00	100.00
GVC 183	AMC409112	14 0200N 0200W 030 SW	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 184	AMC409113	14 0200N 0200W 030 SW	AMC408939	May 01, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 031 NW	ditto	May 01, 2011	-	100.00	100.00
GVC 185	AMC409114	14 0200N 0200W 030 SW, SE	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 186	AMC409115	14 0200N 0200W 030 SW	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 187	AMC409116	14 0200N 0200W 030 SW, SE	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 188	AMC409117	14 0200N 0200W 030 SW	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 189	AMC409118	14 0200N 0200W 030 SW, SE	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 190	AMC409119	14 0200N 0200W 030 SW	AMC408939	May 01, 2011	8.361	100.00	100.00
GVC 191	AMC409120	14 0200N 0200W 030 SW, SE	AMC408939	May 01, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 031 NW, NE	ditto	May 01, 2011	-	100.00	100.00
GVC 192	AMC409121	14 0200N 0200W 030 SE	AMC408939	May 01, 2011	8.361	100.00	100.00
ditto	ditto	14 0200N 0200W 031 NE	ditto	May 01, 2011	-	100.00	100.00
GVC 193	AMC409122	14 0200N 0200W 031 NE	AMC408939	May 01, 2011	8.361	100.00	100.00
				Total Area	***1,421.370***		

Figure 4.7: A Location Plan for the Company's Block of Unpatented Lode Claims (GVC Series, highlighted in PURPLE), Northwest Sector, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company, refer to Figure 4.5 to determine the position of the illustrated claims within the Moss Mine Project Area)

Figure 4.8: A Location Plan for the Company's Block of Unpatented Lode Claims (GVC Series, highlighted in PURPLE), Southwest Sector, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company, refer to Figure 4.5 to determine the position of the illustrated claims within the overall Moss Mine Project Area)



4.2.5.3 Moss 201 to Moss 211 Claim Series

Table 4.4 summarizes the details of the Moss 201 to Moss 211 series of 11 unpatented lode claims. Moss 201 to Moss 209 form a single strip along the southern boundary of the main block of Moss claims, to infill the otherwise open ground. Moss 210 and Moss 211 infill gaps between the surveyed boundaries of the 15 patented lode claims described in Sub-Section 4.2.4.

The claim areas stated on Table 4.4 are the staked areas of each listed claim, estimated by MineFill using the AutoCad® claims files supplied by the Company. However, Moss 201 to Moss 207 overlap one or more claim of the GVC series to the south. The affected GVC claims take precedence over the overlapping Moss claims. The active areas of the overlapping Moss 201 to Moss 207 claims are stated in Section 4.2.6 in which the total overlap area of the claims comprising Moss Mine Project area is defined. The locations of the Moss 201 to Moss 209 claims are detailed on Figure 4.6. The locations of the Moss 210 and Moss 211 claims are detailed on Figure 4.10.

Figure 4.9: A Location Plan for the Company's Unpatented Lode Claims (GVC series, highlighted in PURPLE, and Silver Creek [SC] Series, highlighted in BLUE) and Arizona State Exploration Permit Area (highlighted in RED), Southeast and Central East Sectors, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company, refer to Figure 4.5 to determine the position of the illustrated claims within the overall Moss Mine Project Area)

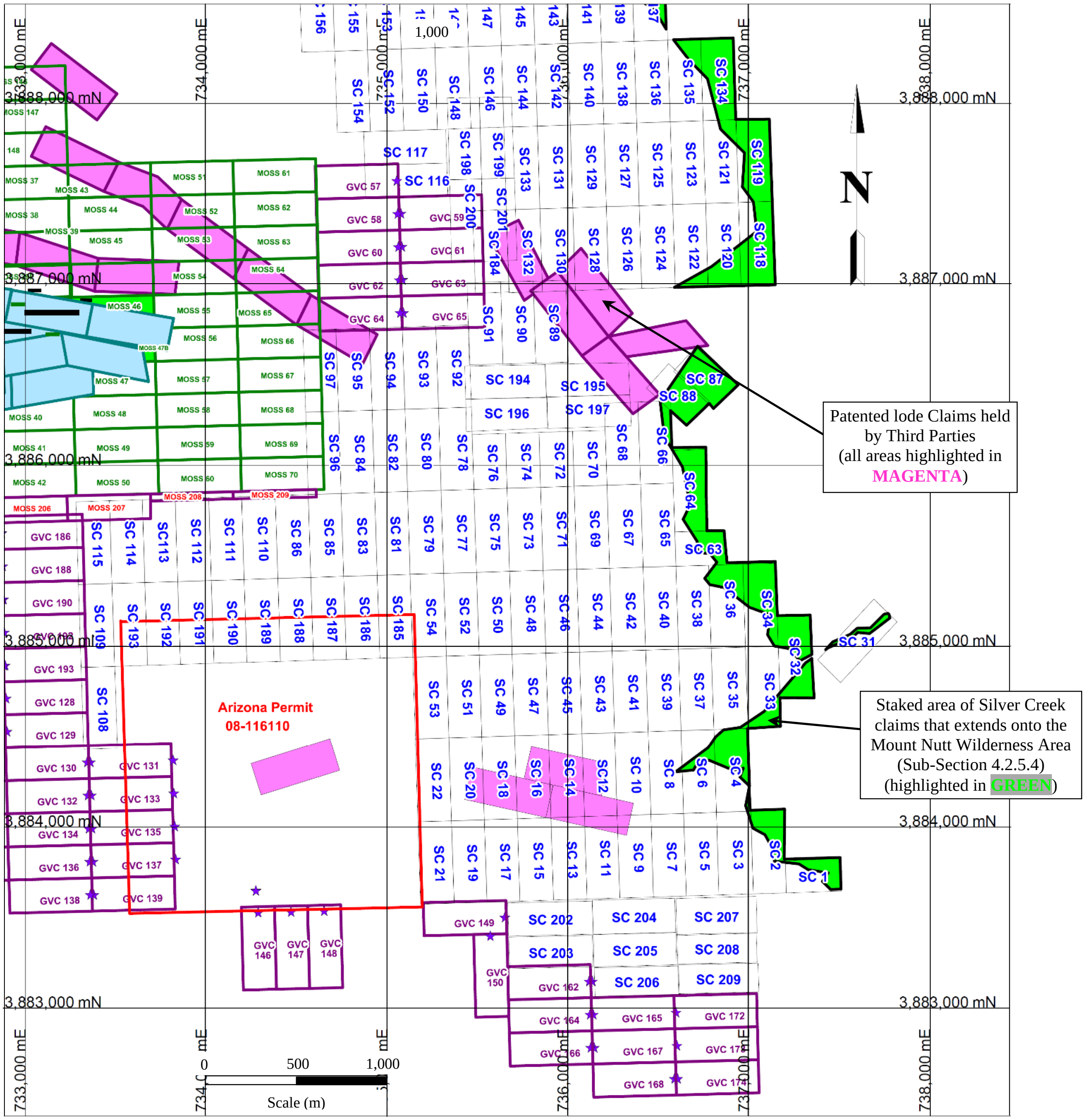


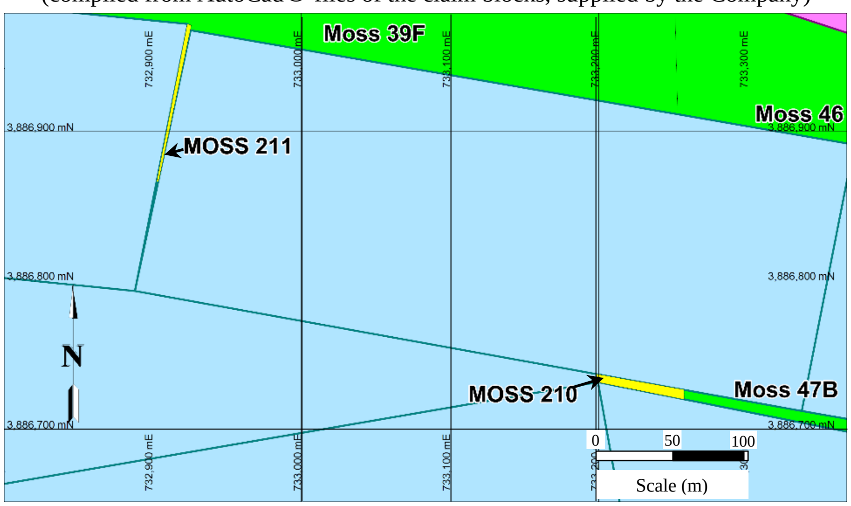
Table 4.4: A Summary of the Company's Unpatented Lode Claims (Moss 201 to Moss 211 Series) of the Moss Mine Project Area

(compiled from information from various sources, including Company Documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Meridian, Township, Range, Sector & Quadrant	Lead File	Date of Location	Staked Area (ha)	% Subject to Agreement	
						MinQuest	Patriot
Moss 201	AMC416914	14 0200N 0210W 025 SW	AMC416914	June 27, 2012	6.45	100%	100%
Moss 202	AMC416915	14 0200N 0210W 025 SW, SE	AMC416914	June 27, 2012	6.45	100%	100%
Moss 203	AMC416916	14 0200N 0210W 025 SE	AMC416914	June 27, 2012	6.45	100%	100%
Moss 204	AMC416917	14 0200N 0200W 030 SW	AMC416914	June 27, 2012	6.45	100%	100%
Moss 205	AMC416918	14 0200N 0200W 030 SW, SE	AMC416914	June 27, 2012	6.45	100%	100%
Moss 206	AMC416919	14 0200N 0200W 030 SE	AMC416914	June 27, 2012	5.67	100%	100%
Moss 207	AMC416920	14 0200N 0200W 029 SW	AMC416914	June 27, 2012	6.45	100%	100%
ditto	ditto	14 0200N 0200W 030 SE	ditto	June 27, 2012	-	100%	100%
Moss 208	AMC416921	14 0200N 0200W 029 SW	AMC416914	June 27, 2012	1.85	100%	100%
Moss 209	AMC416922	14 0200N 0200W 029 SW, SE	AMC416914	June 27, 2012	1.85	100%	100%
Moss 210	AMC420117	14 0200N 0200W 029 NW	AMC420117	Sept. 05, 2012	0.34	100%	100%
ditto	ditto	14 0200N 0200W 030 NE	ditto	Sept. 05, 2012	-	100%	100%
Moss 211	AMC420118	14 0200N 0200W 019 SE	AMC420117	Sept. 05, 2012	0.02	100%	100%
				Total Area	***48.43***		

Figure 4.10: A Location Plan for the Company's Moss 210 and 211 Unpatented Lode Claims, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company)



4.2.5.4 Silver Creek Claims

Table 4.5 (that is in three parts due to its length) summarizes the details of the Silver Creek series of 170 unpatented lode claims (1,487.77 ha). The locations of the claims in the southeast and central east sectors are included on Figure 4.9. Figure 4.11 is a location plan for the Silver Creek claims located in the northeast sector. Each of the plans includes the positions of active patented lode claims that are held by third parties.

Figures 4.9 and 4.11 include the local boundary of the Mount Nutt Wilderness area to the east of the Moss Mine Project Area and highlight the staked areas of the Silver Creek claims that encroach onto the wilderness area. The wilderness area is not open to mineral location and no exploration or related activities are allowed. Pursuant to the La Cuesta Agreement (Sub-Section 4.4.3), the Silver Creek claims listed on Table 4.5 assert rights to only those portions of the claims that are located outside the wilderness preserve.

Table 4.5: A Summary of the Company's Silver Creek Series of Unpatented Lode Claims, Moss Mine Project Area

(compiled from information from various sources, including Company Documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Mohave County Record Number	Section Number	Township	Range	Staked Area (ha)	% Subject to Agreement	
							MinQuest	Patriot
Silver Creek 1	AMC 407863	2011024735	34	20N	20W	8.361	0.00	0.00
Silver Creek 2	AMC 407864	2011024736	34	20N	20W	8.361	0.00	0.00
Silver Creek 3	AMC 407865	2011024737	34	20N	20W	8.361	0.00	0.00
Silver Creek 4	AMC 407866	2011024738	34	20N	20W	8.361	0.00	0.00
Silver Creek 5	AMC 407867	2011024739	34, 33	20N	20W	8.361	0.00	0.00
Silver Creek 6	AMC 407868	2011024740	34, 33	20N	20W	8.361	0.00	0.00
Silver Creek 7	AMC 407869	2011024741	33	20N	20W	8.361	0.00	0.00
Silver Creek 8	AMC 407870	2011024742	33	20N	20W	8.361	0.00	0.00
Silver Creek 9	AMC 407871	2011024743	33	20N	20W	7.763	0.00	0.00
Silver Creek 10	AMC 407872	2011024744	33	20N	20W	7.944	0.00	0.00
Silver Creek 11	AMC 407873	2011024745	33	20N	20W	6.919	0.00	0.00
Silver Creek 12	AMC 407874	2011024746	33	20N	20W	4.069	0.00	0.00
Silver Creek 13	AMC 407875	2011024747	33	20N	20W	8.017	0.00	0.00
Silver Creek 14	AMC 407876	2011024748	33	20N	20W	2.025	0.00	0.00
Silver Creek 15	AMC 407877	2011024749	33	20N	20W	8.333	0.00	0.00
Silver Creek 16	AMC 407878	2011024750	33	20N	20W	2.423	0.00	0.00
Silver Creek 17	AMC 407879	2011024751	33	20N	20W	8.253	0.00	0.00
Silver Creek 18	AMC 407880	2011024752	33	20N	20W	3.732	0.00	0.00
Silver Creek 19	AMC 407881	2011024753	33	20N	20W	8.312	0.00	0.00
Silver Creek 20	AMC 407882	2011024754	33	20N	20W	6.858	8.23	8.23
Silver Creek 21	AMC 407883	2011024755	33	20N	20W	8.361	0.00	0.00
Silver Creek 22	AMC 407884	2011024756	33	20N	20W	8.361	26.70	26.70
Silver Creek 31	AMC 407893	2011024765	34, 27	20N	20W	8.361	0.00	0.00
Silver Creek 32	AMC 407894	2011024766	34	20N	20W	8.361	0.00	0.00
Silver Creek 33	AMC 407895	2011024767	34	20N	20W	8.361	0.00	0.00
Silver Creek 34	AMC 407896	2011024768	34, 27	20N	20W	8.361	0.00	0.00
Silver Creek 35	AMC 407897	2011024769	34	20N	20W	8.361	0.00	0.00
Silver Creek 36	AMC 407898	2011024770	34, 27	20N	20W	8.361	0.00	0.00
Silver Creek 37	AMC 407899	2011024771	33, 34	20N	20W	8.361	0.00	0.00
Silver Creek 38	AMC 407900	2011024772	33, 28, 34, 27	20N	20W	8.361	0.00	0.00
Silver Creek 39	AMC 407901	2011024773	33	20N	20W	8.361	0.00	0.00
Silver Creek 40	AMC 407902	2011024774	33, 28	20N	20W	8.361	0.00	0.00
Silver Creek 41	AMC 407903	2011024775	33	20N	20W	8.361	0.00	0.00
Silver Creek 42	AMC 407904	2011024776	33, 28	20N	20W	8.361	0.00	0.00
Silver Creek 43	AMC 407905	2011024777	33	20N	20W	8.361	0.00	0.00
Silver Creek 44	AMC 407906	2011024778	28, 33	20N	20W	8.361	35.73	35.73
Silver Creek 45	AMC 407907	2011024779	33	20N	20W	8.361	10.28	10.28
Silver Creek 46	AMC 407908	2011024780	33, 28	20N	20W	8.361	97.25	97.25
Silver Creek 47	AMC 407909	2011024781	33	20N	20W	8.361	50.71	50.71
Silver Creek 48	AMC 407910	2011024782	33, 28	20N	20W	8.361	100.00	100.00
Silver Creek 49	AMC 407911	2011024783	33	20N	20W	8.361	85.35	85.35
Silver Creek 50	AMC 407912	2011024784	33, 28	20N	20W	8.361	100.00	100.00
Silver Creek 51	AMC 407913	2011024785	33	20N	20W	8.361	100.00	100.00
Silver Creek 52	AMC 407914	2011024786	33, 28	20N	20W	8.361	100.00	100.00
Silver Creek 53	AMC 407915	2011024787	33	20N	20W	8.361	100.00	100.00
Silver Creek 54	AMC 407916	2011024788	33, 28	20N	20W	8.361	100.00	100.00
Silver Creek 63	AMC 407925	2011024797	28, 27	20N	20W	8.361	0.00	0.00
Silver Creek 64	AMC 407926	2011024798	28, 27	20N	20W	8.361	0.00	0.00
Silver Creek 65	AMC 407927	2011024799	28	20N	20W	8.361	0.00	0.00
Silver Creek 66	AMC 407928	2011024800	28	20N	20W	8.361	0.00	0.00
Silver Creek 67	AMC 407929	2011024801	28	20N	20W	8.361	14.46	14.46
Silver Creek 68	AMC 407930	2011024802	28	20N	20W	8.361	30.12	30.12
Silver Creek 69	AMC 407931	2011024803	28	20N	20W	8.361	97.51	97.51
Silver Creek 70	AMC 407932	2011024804	28	20N	20W	5.485	100.00	100.00
Silver Creek 71	AMC 407933	2011024805	28	20N	20W	8.361	100.00	100.00
Silver Creek 72	AMC 407934	2011024806	28	20N	20W	5.632	100.00	100.00
Silver Creek 73	AMC 407935	2011024807	28	20N	20W	8.361	100.00	100.00
Silver Creek 74	AMC 407936	2011024808	28	20N	20W	5.663	100.00	100.00
Silver Creek 75	AMC 407937	2011024809	28	20N	20W	8.361	100.00	100.00
Silver Creek 76	AMC 407938	2011024810	28	20N	20W	5.563	100.00	100.00
Silver Creek 77	AMC 407939	2011024811	28	20N	20W	8.361	100.00	100.00
Silver Creek 78	AMC 407940	2011024812	28	20N	20W	8.361	100.00	100.00
Silver Creek 79	AMC 407941	2011024813	28	20N	20W	8.361	100.00	100.00
Silver Creek 80	AMC 407942	2011024814	28	20N	20W	8.361	100.00	100.00
Silver Creek 81	AMC 407943	2011024815	28, 29	20N	20W	8.361	100.00	100.00
Silver Creek 82	AMC 407944	2011024816	28, 29	20N	20W	8.361	100.00	100.00
Silver Creek 83	AMC 407945	2011024817	29	20N	20W	8.361	100.00	100.00
Silver Creek 84	AMC 407946	2011024818	29	20N	20W	8.361	100.00	100.00
Silver Creek 85	AMC 407947	2011024819	29	20N	20W	8.361	100.00	100.00
Silver Creek 86	AMC 407948	2011024820	29	20N	20W	8.361	100.00	100.00
Silver Creek 87	AMC 407949	2011024821	28, 27	20N	20W	5.545	0.00	0.00
Silver Creek 88	AMC 407950	2011024822	28, 27	20N	20W	5.569	0.00	0.00
Silver Creek 89	AMC 407951	2011024823	28, 21	20N	20W	8.361	100.00	100.00
Silver Creek 90	AMC 407952	2011024824	28, 21	20N	20W	8.361	100.00	100.00

Table 4.5 *continued*: A Summary of the Company's Silver Creek Series of Unpatented Lode Claims, Moss Mine Project Area

(compiled from information from various sources, including Company Documents and BLM Claim Reports)

Claim Name	BLM Serial Number	Mohave County Record Number	Section Number	Township	Range	Staked Area (ha)	% Subject to Agreement	
							MinQuest	Patriot
Silver Creek 91	AMC 407953	2011024825	28, 21	20N	20W	8.361	100.00	100.00
Silver Creek 92	AMC 407954	2011024826	28	20N	20W	8.361	100.00	100.00
Silver Creek 93	AMC 407955	2011024827	28	20N	20W	8.361	100.00	100.00
Silver Creek 94	AMC 407956	2011024828	29, 28	20N	20W	8.361	100.00	100.00
Silver Creek 95	AMC 407957	2011024829	29	20N	20W	8.464	100.00	100.00
Silver Creek 96	AMC 407958	2011024830	29	20N	20W	8.361	100.00	100.00
Silver Creek 97	AMC 407959	2011024831	29	20N	20W	8.361	100.00	100.00
Silver Creek 108	AMC 407970	2011024842	31	20N	20W	8.361	100.00	100.00
Silver Creek 109	AMC 407971	2011024843	31, 30	20N	20W	8.361	100.00	100.00
Silver Creek 110	AMC 407972	2011024844	29	20N	20W	8.361	100.00	100.00
Silver Creek 111	AMC 407973	2011024845	29	20N	20W	8.361	100.00	100.00
Silver Creek 112	AMC 407974	2011024846	29	20N	20W	8.361	100.00	100.00
Silver Creek 113	AMC 407975	2011024847	29	20N	20W	8.361	100.00	100.00
Silver Creek 114	AMC 407976	2011024848	29, 30	20N	20W	8.361	100.00	100.00
Silver Creek 115	AMC 407977	2011024849	30	20N	20W	8.361	100.00	100.00
Silver Creek 116	AMC 410214	2011044461	21, 20	20N	20W	8.361	100.00	100.00
Silver Creek 117	AMC 410215	2011044462	21, 20	20N	20W	8.361	100.00	100.00
Silver Creek 118	AMC 410216	2011044463	22	20N	20W	8.361	0.00	0.00
Silver Creek 119	AMC 410217	2011044464	22	20N	20W	8.361	0.00	0.00
Silver Creek 120	AMC 410218	2011044465	22	20N	20W	8.361	0.00	0.00
Silver Creek 121	AMC 410219	2011044466	22	20N	20W	8.361	0.00	0.00
Silver Creek 122	AMC 410220	2011044467	21, 22	20N	20W	8.361	0.00	0.00
Silver Creek 123	AMC 410221	2011044468	21, 22	20N	20W	8.361	0.00	0.00
Silver Creek 124	AMC 410222	2011044469	21	20N	20W	8.361	0.00	0.00
Silver Creek 125	AMC 410223	2011044470	21	20N	20W	8.361	0.00	0.00
Silver Creek 126	AMC 410224	2011044471	21	20N	20W	8.361	12.51	12.51
Silver Creek 127	AMC 410225	2011044472	21	20N	20W	8.361	15.37	15.37
Silver Creek 128	AMC 410226	2011044473	21	20N	20W	8.361	100.00	100.00
Silver Creek 129	AMC 410227	2011044474	21	20N	20W	8.361	100.00	100.00
Silver Creek 130	AMC 410228	2011044475	21	20N	20W	8.361	100.00	100.00
Silver Creek 131	AMC 410229	2011044476	21	20N	20W	8.361	100.00	100.00
Silver Creek 132	AMC 410230	2011044477	21	20N	20W	8.361	100.00	100.00
Silver Creek 133	AMC 410231	2011044478	21	20N	20W	8.361	100.00	100.00
Silver Creek 134	AMC 410232	2011044479	22	20N	20W	8.361	0.00	0.00
Silver Creek 135	AMC 410233	2011044480	21, 22	20N	20W	8.361	0.00	0.00
Silver Creek 136	AMC 410234	2011044481	21	20N	20W	8.361	0.00	0.00
Silver Creek 137	AMC 410235	2011044482	21, 16	20N	20W	8.361	0.00	0.00
Silver Creek 138	AMC 410236	2011044483	21	20N	20W	8.361	4.37	4.37
Silver Creek 139	AMC 410237	2011044484	21, 16	20N	20W	8.361	0.00	0.00
Silver Creek 140	AMC 410238	2011044485	21	20N	20W	8.361	87.73	87.73
Silver Creek 141	AMC 410239	2011044486	21, 16	20N	20W	8.361	14.64	14.64
Silver Creek 142	AMC 410240	2011044487	21	20N	20W	8.361	100.00	100.00
Silver Creek 143	AMC 410241	2011044488	21, 16	20N	20W	8.361	25.31	25.31
Silver Creek 144	AMC 410242	2011044489	21	20N	20W	8.361	100.00	100.00
Silver Creek 145	AMC 410243	2011044490	21, 16	20N	20W	8.361	100.00	100.00
Silver Creek 146	AMC 410244	2011044491	21	20N	20W	8.361	100.00	100.00
Silver Creek 147	AMC 410245	2011044492	21, 16	20N	20W	8.361	100.00	100.00
Silver Creek 148	AMC 410246	2011044493	21	20N	20W	8.361	100.00	100.00
Silver Creek 149	AMC 410247	2011044494	21, 16	20N	20W	8.361	100.00	100.00
Silver Creek 150	AMC 410248	2011044495	21	20N	20W	8.361	100.00	100.00
Silver Creek 151	AMC 410249	2011044496	21, 16	20N	20W	8.361	100.00	100.00
Silver Creek 152	AMC 410250	2011044497	20, 21	20N	20W	8.361	100.00	100.00
Silver Creek 153	AMC 410251	2011044498	20, 17, 21, 16	20N	20W	8.361	100.00	100.00
Silver Creek 154	AMC 410252	2011044499	20	20N	20W	8.361	100.00	100.00
Silver Creek 155	AMC 410253	2011044500	20, 17	20N	20W	8.361	100.00	100.00
Silver Creek 156	AMC 410254	2011044501	20, 17	20N	20W	8.361	100.00	100.00
Silver Creek 157	AMC 410255	2011044502	16	20N	20W	8.361	0.00	0.00
Silver Creek 158	AMC 410256	2011044503	16	20N	20W	8.361	0.00	0.00
Silver Creek 159	AMC 410257	2011044504	16	20N	20W	8.361	20.81	20.81
Silver Creek 160	AMC 410258	2011044505	16	20N	20W	8.361	0.00	0.00
Silver Creek 161	AMC 410259	2011044506	16	20N	20W	8.361	53.18	53.18
Silver Creek 162	AMC 410260	2011044507	16	20N	20W	8.361	0.00	0.00
Silver Creek 163	AMC 410261	2011044508	16	20N	20W	8.361	80.82	80.82
Silver Creek 164	AMC 410262	2011044509	16	20N	20W	8.361	0.00	0.00
Silver Creek 165	AMC 410263	2011044510	16	20N	20W	8.361	97.51	97.51
Silver Creek 166	AMC 410264	2011044511	16	20N	20W	8.361	1.16	1.16
Silver Creek 167	AMC 410265	2011044512	17, 16	20N	20W	8.361	100.00	100.00
Silver Creek 168	AMC 410266	2011044513	17, 16	20N	20W	8.361	10.90	10.90
Silver Creek 169	AMC 410267	2011044514	17	20N	20W	8.361	100.00	100.00
Silver Creek 170	AMC 410268	2011044515	17	20N	20W	8.361	17.03	17.68
Silver Creek 171	AMC 410269	2011044516	17	20N	20W	8.361	100.00	100.00
Silver Creek 172	AMC 410270	2011044517	17	20N	20W	8.361	19.75	27.87

Table 4.5 *continued*: A Summary of the Company's Silver Creek Series of Unpatented Lode Claims, Moss Mine Project Area

(compiled from information from various sources, including Company Documents and BLM Claim Reports)

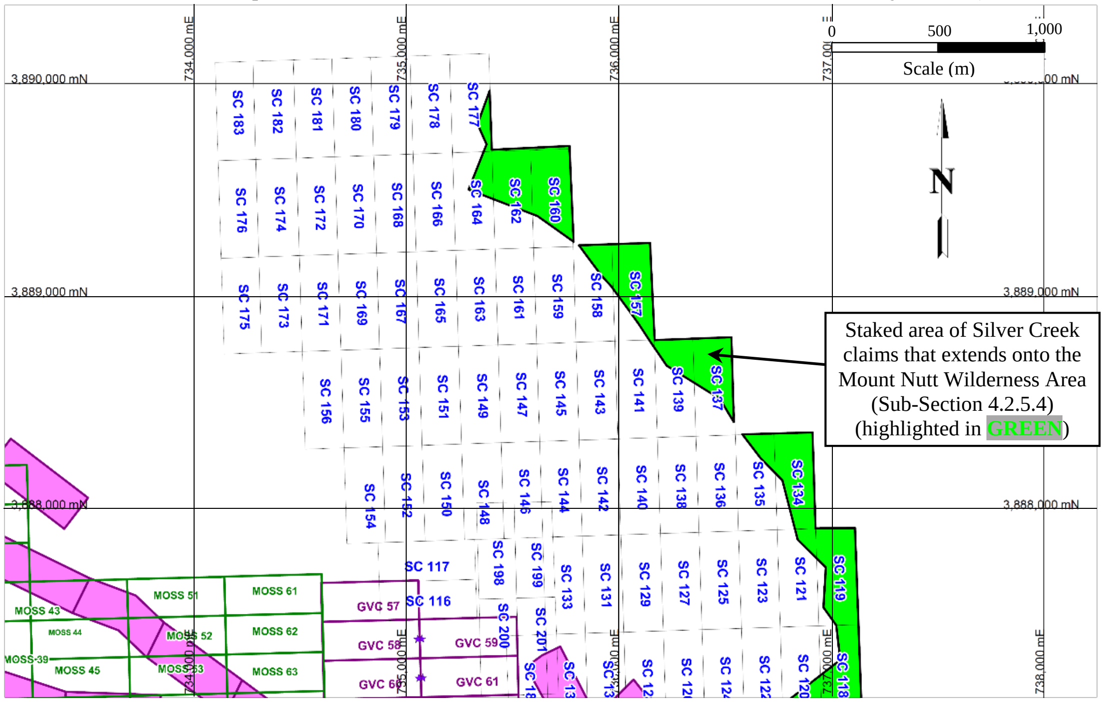
Claim Name	BLM Serial Number	Mohave County Record Number	Section Number	Township	Range	Staked Area (ha)	% Subject to Agreement	
							MinQuest	Patriot
Silver Creek 173	AMC 410271	2011044518	17	20N	20W	8.361	100.00	100.00
Silver Creek 174	AMC 410272	2011044519	17	20N	20W	8.361	20.79	50.84
Silver Creek 175	AMC 410273	2011044520	17	20N	20W	8.361	100.00	100.00
Silver Creek 176	AMC 410274	2011044521	17	20N	20W	8.361	20.03	76.40
Silver Creek 177	AMC 410275	2011044522	16, 9	20N	20W	8.361	0.00	0.00
Silver Creek 178	AMC 410276	2011044523	16, 9	20N	20W	8.361	0.00	0.00
Silver Creek 179	AMC 410277	2011044524	17, 8, 16, 9	20N	20W	8.361	0.00	0.00
Silver Creek 180	AMC 410278	2011044525	17, 8	20N	20W	8.361	0.00	0.00
Silver Creek 181	AMC 410279	2011044526	17, 8	20N	20W	8.361	0.00	0.00
Silver Creek 182	AMC 410280	2011044527	17, 8	20N	20W	8.361	0.00	0.00
Silver Creek 183	AMC 410281	2011044528	17, 8	20N	20W	8.361	0.00	0.00
Silver Creek 184	AMC 410282	2011044529	21	20N	20W	8.361	100.00	100.00
Silver Creek 185	AMC 413137	2012000017	32, 33, 29, 28	20N	20W	8.361	100.00	100.00
Silver Creek 186	AMC 413138	2012000018	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 187	AMC 413139	2012000019	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 188	AMC 413140	2012000020	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 189	AMC 413141	2012000021	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 190	AMC 413142	2012000022	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 191	AMC 413143	2012000023	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 192	AMC 413144	2012000024	32, 29	20N	20W	8.361	100.00	100.00
Silver Creek 193	AMC 413145	2012000025	32, 29, 31, 30	20N	20W	8.361	100.00	100.00
Silver Creek 194	AMC 427718	2014014495	28	20N	20W	8.361	100.00	100.00
Silver Creek 195	AMC 427719	2014014496	28	20N	20W	8.361	77.50	77.50
Silver Creek 196	AMC 427720	2014014497	28	20N	20W	8.361	100.00	100.00
Silver Creek 197	AMC 427721	2014014498	28	20N	20W	8.361	77.50	77.50
Silver Creek 198	AMC 427722	2014014499	21	20N	20W	8.361	100.00	100.00
Silver Creek 199	AMC 427723	2014014500	21	20N	20W	8.361	100.00	100.00
Silver Creek 200	AMC 427724	2014014501	21	20N	20W	8.361	100.00	100.00
Silver Creek 201	AMC 427725	2014014502	21	20N	20W	8.361	100.00	100.00
Silver Creek 202	AMC 428270	2014021863	4	19N	20W	8.361	0.00	0.00
Silver Creek 203	AMC 428271	2014021864	4	19N	20W	8.361	0.00	0.00
Silver Creek 204	AMC 428272	2014021865	4	19N	20W	8.361	0.00	0.00
Silver Creek 205	AMC 428273	2014021866	4	19N	20W	8.361	0.00	0.00
Silver Creek 206	AMC 428274	2014021867	4	19N	20W	8.361	0.00	0.00
Silver Creek 207	AMC 428275	2014021868	4	19N	20W	8.361	0.00	0.00
Silver Creek 208	AMC 428276	2014021869	4	19N	20W	8.361	0.00	0.00
Silver Creek 209	AMC 428277	2014021870	4	19N	20W	8.361	0.00	0.00
					Total Area	***1,487.773***		

4.2.5.5 Arizona State Exploration Permit

The area covered by the Arizona State exploration permit (#08-116110, 259 ha) is identified on Figure 4.9. As can be seen, it overlaps both GVC and Silver Creek series claims. The 'active' area of the exploration permit area is estimated by MineFill (from scrutiny of the AutoCad® claims files provided by the Company) to equal approximately 186.8 ha.

Figure 4.11: A Location Plan for the Company's Optioned Unpatented Lode Claims (Silver Creek [SC] Series, highlighted in BLUE), Northeast Area, Moss Mine Project Area

(compiled from AutoCad® files of the claim blocks, supplied by the Company, refer to Figure 4.5 to determine the position of the illustrated claims within the overall Moss Mine Project Area)



4.2.6 Claim and Permit Overlaps

Table 4.6 summarizes the various overlaps between the various claims and between the Arizona State exploration permit and claims. The active areas of each listed claim were estimated from scrutiny of the AutoCad® claims files supplied by the Company. The total overlap area (estimated by MineFill at 158.16 ha) was deducted from the total estimated area of all the Moss Mine Project patented lode claims, unpatented lode claims and one Arizona State exploration licence (estimated by MineFill and rounded to 4,188.94 ha) to arrive at the estimated total Moss Mine Project area of 4,030.78 ha.

It is emphasized that, for the reasons stated in Section 4.2.5, the areas stated on Table 4.6 are estimates only: none of the unpatented lode claims have been surveyed by a licensed land surveyor; and the stated values are estimates, by MineFill, based on scrutiny of AutoCad® claims files supplied by the Company.

Table 4.6: A Summary of the Estimated Claim and Permit Overlaps, Moss Mine Project
(compiled from scrutiny of the AutoCad® claims files supplied by the Company)

Claim/Permit Name	Area (ha)			Over-Lapping
	Total	Active	Overlap	
Moss 23	8.361	7.48	0.88	Portions of the 15 patented lode claims
Moss 24	8.361	2.94	5.42	
Moss 25	8.361	4.72	3.64	
Moss 26	8.361	6.49	1.87	
Moss 27	8.361	2.27	6.09	
Moss 28	8.361	7.31	1.05	
Moss 33F	8.361	3.61	4.75	
Moss 34	8.361	1.96	6.40	
Moss 39F	8.361	5.45	2.91	
Moss 40	8.361	6.49	1.87	
Moss 46	8.361	4.24	4.12	
Moss 47	8.361	4.09	4.27	
Moss 47B	8.361	0.91	7.45	
Moss 55	8.361	7.84	0.52	
Moss 56	8.361	7.38	0.98	
GVC 39	8.361	7.80	0.56	Moss Claims (portions of the original 104 claims of the Moss 1 to Moss 148 series)
GVC 40	8.361	7.43	0.93	
GVC 50	8.361	7.97	0.39	
GVC 51	8.361	7.32	1.04	
GVC 52	8.361	4.26	4.10	
GVC 53	8.361	4.39	3.97	
GVC 54	8.361	4.39	3.97	
GVC 55	8.361	4.39	3.97	
GVC 56	8.361	3.75	4.61	
Moss 201	6.450	4.81	1.63	Portions of the GVC series of claims
Moss 202	6.450	4.66	1.79	
Moss 203	6.450	4.79	1.66	
Moss 204	6.450	4.83	1.62	
Moss 205	6.450	4.68	1.77	
Moss 206	5.670	4.29	1.38	
Moss 207	6.450	6.03	0.42	
Arizona State Exploration Permit	259.0	186.90	72.10	Portions of the GVC and Silver Creek series of claims
Totals	***504.034***	***345.87***	***158.16***	-

4.3 Taxes, Maintenance Fees and Rent

4.3.1 Patented Lode Claims

Taxes are levied by the State in respect of patented lode claims, for payment to the local county (Mohave County in the case of the Moss Mine Project). The value of a property comprising patented lode claims is assessed by the Property Tax Division of the State's Department of Revenue. The State then applies an assessment ratio to the assessed value to arrive at an assessed full cash value for the patented ground. Primary and secondary tax rates (for 2015, 8.142% and 1.5184%, respectively) are then levied on the assessed full cash value to determine the tax due for the stated patented lode claim or claims. If the tax liability is greater than US$100, 50% of the tax due is payable on or before October 01 of the assessed tax year, with the balance due on or before the first of the following March. If the tax liability is less than US$100, payment is due on or before December 01 of the assessed tax year.

MineFill has seen an original copy of the State's Property Tax Division's assessed property value of the 15 patented lode claims comprising the Moss Mine Property. On this basis MineFill concurs with the Company that the tax liability for 2015 is approximately US$36,000.

4.3.2 Unpatented Lode Claims

To maintain unpatented lode claims as active, hence in good standing, an annual maintenance fee is payable to BLM before September 01 of each year, in respect of the following 12 months. At the time of writing (December 2014) the maintenance fee for 2015 was US$155 per unpatented lode claim (up from US$140 in 2014), plus a filing fee for each block of US$480.

4.3.3 Arizona State Exploration Permit

Rental totalling US$2.00 per acre for the first year of an Arizona State exploration permit is payable to ASLD, which includes Year Two, reducing to US$1.00 per acre through Year Five and the end of the exploration permit. A bond is established based on the proposed exploration activities (typically US$3,000.00 for a single permit). A blanket bond of US$15,000.00 can be paid for five or more permits held by an individual or company.

4.4 Principal Agreements

4.4.1 MinQuest Agreement

The MinQuest Agreement is a mining lease/purchase agreement between MinQuest and Patriot Gold. It was entered into on March 04, 2004. Pursuant to its terms Patriot Gold purchased the Moss Property that is defined in the MinQuest Agreement as:

- seven patented lode claims (Key No. 1, Key No. 2, Moss Millsite, Divide, Keystone Wedge, California Moss Lot 37 [Greenwood] and California Moss Lot 38 [Gintoff]); and
- 63 unpatented lode claims (Moss 11 to Moss 33, Moss 33F, Moss 34 to Moss 39, Moss 39F, Moss 40 to Moss 47, Moss 47F and Moss 48 to Moss 70).

Pursuant to the MinQuest Agreement, a payment of US$50,000 was made by Patriot Gold on signing the MinQuest Agreement, plus reimbursement of filing fees of US$150 per patented and unpatented claim. The agreement is valid for 20 years from the date of signing (March 04, 2004) with automatic extensions '*so long as Patriot Gold holds all or portions of the Property'* (statement taken from the MinQuest Agreement, a certified copy of which was seen by MineFill). Royalties are payable in respect of the MinQuest Agreement, which are detailed in Section 4.5.

4.4.2 Patriot Gold Agreement

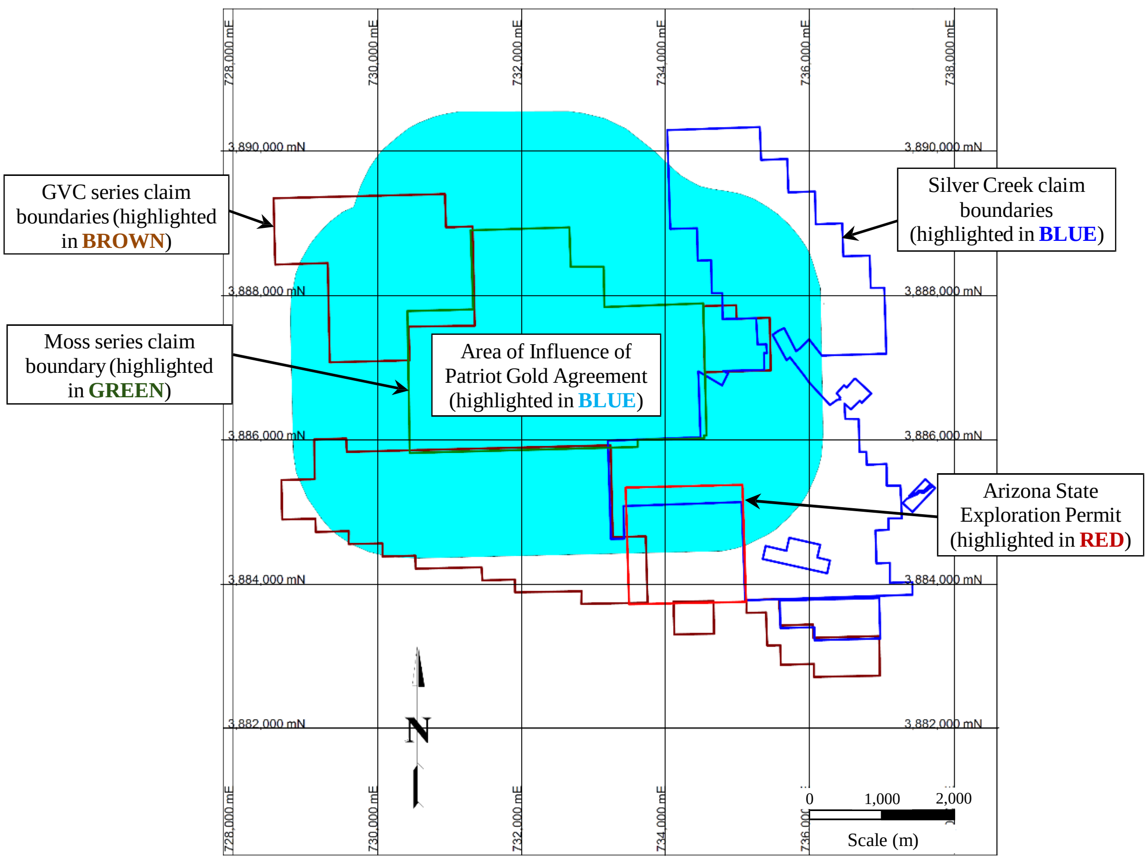
The Patriot Gold Agreement covers all of the 15 patented lode claims listed in Sub-Section 4.2.4 and all of the 104 unpatented lode claims of the Moss 1 to Moss 148 series described in Sub-Section 4.2.5.1. The agreement is an Exploration and Option to Enter Joint Venture Agreement for the Moss Mine Project made between Patriot Gold and Idaho State Gold Company, LLC ("ISGC"), a company registered in Idaho, dated February 28, 2011. The terms of the agreement are for ISGC to earn a 70% interest in the claims by spending US$8.0 million on work on the claims in five years, prepare a bankable feasibility study and make a cash payment of US$0.5 million on signing the agreement.

After signing the Patriot Agreement, ISGC decided not to move forward with the Patriot Gold Agreement and instead assigned it to the Company by means of an Assignment and Assumption Agreement dated March 4, 2011. The Company assumed ISGC's obligations in the Patriot Gold Agreement and the Company made the initial cash payment of US$ 0.5 million to Patriot Gold. ISGC is independent of the Company and, to the best of MineFill's knowledge, ISGC received no payment in respect of the Assignment and Assumption Agreement.

There is a one mile area of influence around the exterior boundary of the claim block detailed in the Patriot Gold Agreement. Pursuant to the agreement, any additional claims staked within this area, either by Patriot Gold or the Company, will the subject to the Patriot Gold Agreement. Figure 4.12 identifies the area of influence defined by the one mile criterion. Tables 4.2 through 4.5 identify the extent to which each unpatented lode claim is subject to the Patriot Agreement (as estimated by MineFill from scrutiny of the AutoCad® claims files supplied by the Company).

Figure 4.12: A Colour-Coded, General Claim Block Reference Plan for the Moss Mine Claims Showing the Extent of the One Mile Zone of Influence Defined in the Patriot Gold Agreement

(compiled using the AutoCad® claims files supplied by the Company, refer to Tables 4.2 through 4.5 for details of the extent to which each unpatented lode claim is subject to the terms of the Patriot Gold Agreement)



At the time of writing (December 2014) the Company had spent a total in excess of US$8.0 million on developing the Moss Mine Project, inclusive of exploration on the Moss Vein and West Extension areas but not including Phase I activities. On completion of a '*bankable feasibility study*' (defined in the Patriot Gold Agreement as meaning '*….an industry accepted report that can be submitted to a bank or other funding group which defines the scope, magnitude, capital costs, rate of return and any and all other items needed to evaluate the viability of a mining operation within the confines of the Property* *and the surrounding Area of Interest'*) the Company will earn its right to form a 70:30 joint venture (70% the Company, 30% Patriot Gold) with pro-rata contribution to all future development costs. In the case of non-contribution, either party will be diluted and if their interest falls below 10% it will convert to a 3.0% NSR royalty.

4.4.3 La Cuesta Agreement

The La Cuesta Agreement covers all of the 183 Silver Creek claims from #1 through #209, as well as the Arizona State exploration permit, that are held in the name of La Cuesta. The agreement is a Mineral Lease and Option Agreement made between the Company and La Cuesta, dated May 07, 2014. Pursuant to the terms of the agreement, full rights to the Silver Creek unpatented lode claims and to the Arizona State exploration permit are transferred to the Company. The primary period of the agreement is 35 years, with extensions allowed up to a maximum of 50 years (although the exploration permit will expire in 2016).

Pursuant to the terms of the agreement, the Company has provided La Cuesta with 100,000 Company shares and has to pay La Cuesta a total of US$85,000 in six month installments over the first 42 months after the date of the agreement, and then US$25,000 every six months thereafter. The payments are credited against future production royalties. Once the production royalty described in Section 4.4.3 starts, no further pre-production payments have to be made.

In addition to the payments outlined, the Company has to spend a minimum of US$15,000 on '*work commitments*' on the leases in Year 1 from the date of the agreement, rising to US$20,000 in Year 2 and US$200,000 in Year 3. No minimum work commitments are required thereafter.

4.5 Royalties and Other Payments

The royalties and other payments summarized in the following sub-sections are payable to Hartmut W. Baitis, Robert B. Hawkins and Larry L. Lackey (collectively "BHL"), MinQuest, to various parties under the Greenwood Agreement and to La Cuesta. To the best of MineFill's knowledge, all of the said corporations and parties are independent of the Company.

Table 4.7 summarizes the royalties payable on each of the patented lode claims and unpatented lode claims, based on MineFill's assessment of the payable royalties and the area of influence of the MinQuest Agreement described in Sub-Section 4.5.1.

Table 4.7: A Summary of Payable Royalties, Moss Mine Project Claims
(compiled from information contained in copies of the legal documents, relating to the various agreements summarized in Sections 4.4 and 4.5, that were provided by the Company)

Patented Claim	Fraction	Royalty	Payable to
California Moss Lot 37 (Greenwood)	-	3.0% NSR 0.5% NSR Sliding Scale*	Various parties in Greenwood Agreement MinQuest, Inc. BHL (finder's fee)
Key No. 1, Key No. 2, California Moss Lot 38 (Gintoff), Moss Millsite, Divide and Keystone Wedge	-	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
Ruth Extension, Omega, Ruth Rattan Extension, Rattan Partnership, Mascot and Empire	-	1.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
Unpatented Claims	**Fraction**	**Royalty**	**Payable To**
All 104 claims of the Moss 1 to Moss 148 series	100%	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
28 GVC Claims	Various (1.5% to 99.9%)	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
88 GVC Claims	100%	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
All 11 claims of the Moss 201 to Moss 2011 series	100%	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
All 183 Silver Creek Claims	-	1.5% NSR	La Cuesta International, Inc.
28 Silver Creek Claims	Various (1.2% to 97.5%)	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)
83 Silver Creek Claims	100%	3.0% NSR Sliding Scale*	MinQuest, Inc. BHL (finder's fee)

Note: * Initially 3% of all exploration and drilling expenditures on the Moss Mine Property until the start of Commercial Production, defined by the Patriot Agreement. On Commencement of commercial production the amount is to be determined by the sliding scale of payments detailed in Sub-Section 4.5.3.

4.5.1 MinQuest, Inc.

Pursuant to the MinQuest Agreement, MinQuest will receive:

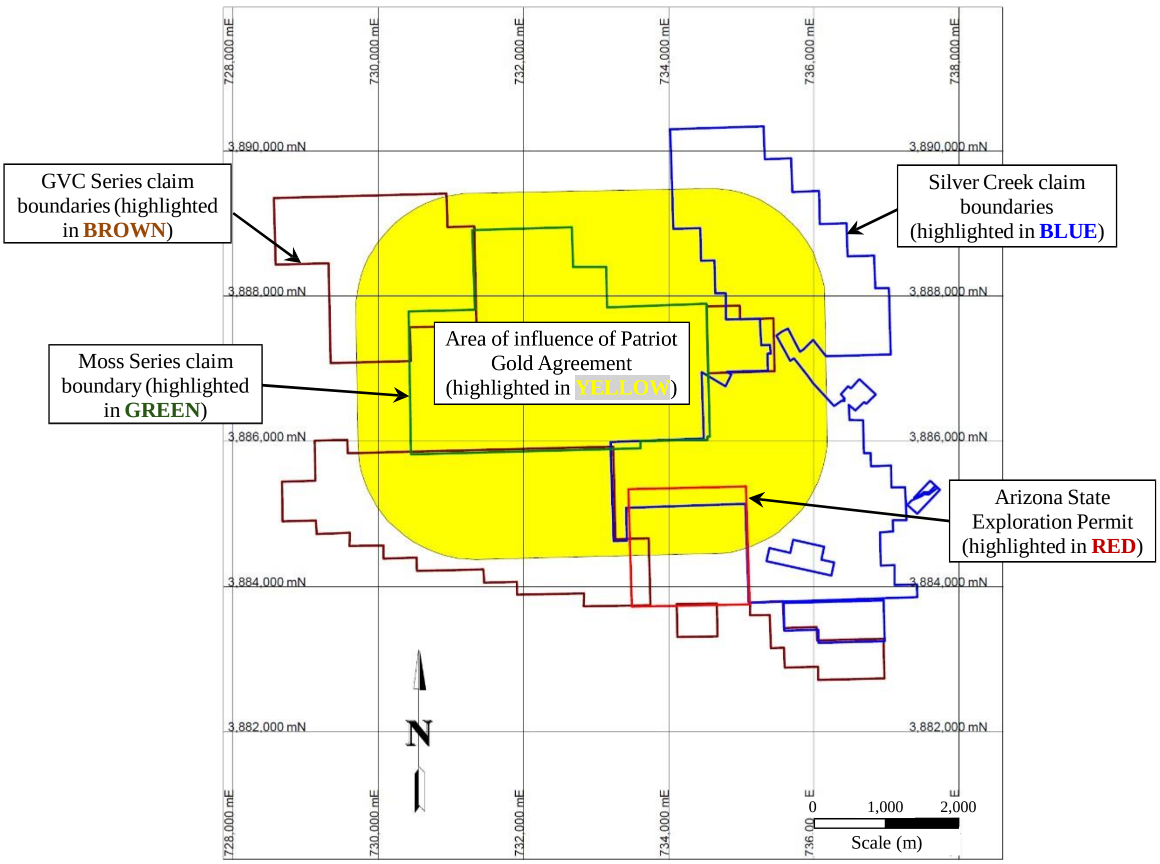
- a 3% net smelter return (NSR) royalty in respect of any and all production from the seven patented lode claims and 63 unpatented lode claims listed in the MinQuest Agreement and on public lands within one mile of the outer perimeter of the said patented and unpatented claims;
- a 1.0% NSR royalty on any and all production from the seven patented lode claims detailed in Sub-Section 4.2.4 and to which no other royalties apply; and
- an over-riding 0.5% NSR royalty on any and all production from those patented lode claims with other royalty interests (limited to the California Moss Lot 37 [Greenwood] lode claim, under the terms of the Greenwood Agreement [Sub-Section 4.5.2]).

The position of the one mile boundary line from the claim block boundary that is the subject of the MinQuest Agreement was drawn and the areas of each claim it intersected were estimated using the AutoCad® claims files supplied by the Company. The percentages of each claim were determined by dividing the estimated area of any claim located wholly or partially within the one mile line by the total estimated area of the same claim.

Figure 4.13 shows the area of influence of MinQuest's one mile boundary line, in respect of the various unpatented lode claim blocks that surround the claim block boundary that is the subject of the MinQuest Agreement (note that the area is smaller than that defined by the Patriot Gold Agreement, per Figure 4.12, because the total block of claims that is subject to the MinQuest Agreement is smaller than the block of claims subject to the Patriot Gold Agreement). Details of the estimated percentages of each unpatented lode claim that is subject to the MinQuest Agreement (hence royalty) are presented on Tables 4.2 through 4.5. The percentages are estimates for the reasons previously outlined: none of the unpatented lode claims have been surveyed by a licensed land surveyor and the fractions of individual claims subject to the MinQuest Agreement were estimated by MineFill from scrutiny of AutoCad® claims files supplied by the Company.

Figure 4.13: A Colour-Coded, General Claim Block Reference Plan for the Moss Mine Claims Showing the Extent of the One Mile Zone of Influence Defined in the MinQuest Agreement

(compiled using the AutoCad® claims files supplied by the Company, refer to Tables 4.2 through 4.6 for details of the extent to which each unpatented lode claim is subject to the terms of the MinQuest Agreement)



4.5.2 Greenwood Agreement

The California Moss Lot 37 (Greenwood) claim is subject to a Purchase Agreement between Patriot Gold and various parties referred to as the Greenwood Agreement that is dated March 2004. The purchase price of US$150,000.00 was paid by Patriot Gold, in addition to which a 3% NSR royalty is payable to the original owners, on gold and silver produced from the claim. In addition and as defined above, a royalty of 0.5% is payable to MinQuest in respect of the California Moss Lot 37 (Greenwood) claim and all other patented claims in which the original vendors have a royalty interest.

4.5.3 Finders Agreement

Pursuant to a Finders Agreement between the Company and BHL, the Company paid a Finder's Fee to BHL in respect of '*certain data, information and consulting services to Northern Vertex concerning the business opportunity and the mineral prospect known as the Moss Mine….*' (a statement forming part of the Finder Agreement, a certified copy of which has been seen by MineFill). An initial payment of US$15,000.00 (equal to 3% of the initial payment under the Patriot Agreement) was made to BHL. Subsequent payments equal to 3% of all Exploration and Drilling Work Expenditures incurred by the Company until the start of Commercial Production, as defined in the Patriot Agreement, have and will be made as quarterly installments, as required by the Finders Agreement.

On commercial production from the Moss Mine, as described in the Patriot Agreement, the Company will pay BHL, on or before 30 days after the end of each calendar quarter, an amount for each troy ounce of gold and silver produced, according to the following schedule:

- for a quarterly average gold price of less than US$700 per troy ounce, US$5.00 per troy ounce of gold produced;
- for a quarterly average gold price equal or greater than US$700 per troy ounce but less than US$1,000 per troy ounce, US$10.00 per troy ounce of gold produced;
- for a quarterly average gold price of greater than US$1,000 per troy ounce, US$15.00 per troy ounce of gold produced;
- for a quarterly average silver price of less than US$15.00 per troy ounce, US$0.10 per troy ounce of silver produced;
- for a quarterly average silver price equal or greater than US$15.00 per troy ounce but less than US$25.00 per troy ounce, US$0.20 per troy ounce of silver produced;
- for a quarterly average silver price of greater than US$25.00 per troy ounce, US$0.35 per troy ounce of silver produced.

The total amount of the payable fee is capped at US$21.00 million and can be purchased by the Company for US$2.40 million, in cash and/or shares, upon mutual agreement and within 90 days of the start of commercial production.

4.5.4 La Cuesta International, Inc.

Pursuant to the terms of the La Cuesta Agreement, the Company will pay La Cuesta a 1.5% NSR royalty on any gold or silver production from the area covered by the Silver Creek claims listed in Sub-Section 4.2.5.4 plus an additional 0.5% NSR royalty on 3rd party claims.

4.6 Environmental Liabilities

The Phase I activities and the planned Phase II operations were and will be limited to the 15 patented lode claims described in Sub-Section 4.2.4. As outlined in Section 4.2.2, the owner of patented lode claims owns the land in law. To the best knowledge of the Qualified Person for this section of this Technical Report (Dr. David Stone, P. Eng.), no environmental liabilities exist as regards the 15 patented lode claims and there is no readily identifiable reason to suppose that any such liabilities exist.

4.7 Permits

The following text is based on an opinion by Mr. Brian Munson of CDM Smith (water, environment, transportation, energy and facilities consultants of Cambridge Massachusetts, USA) regarding permitting requirements for the planned Phase II operations, as stated in a memorandum to Mr. J.R.H. Whittington, President and CEO of the Company entitled ‘Proposal for Moss Mine Phase II Permit Analysis for Feasibility Study’ and dated September 02, 2014.

4.7.1 Required Permits

The permits that were in place to allow the Phase I activities will either have to be renewed, updated or amended for Phase II, the permitting requirements for which should be limited to recognized and conventional permitting programs within the state of Arizona. On this basis, the planned mining and processing operations:

- will require an Aquifer Protection Permit Amendment, a Class 2 Air Quality Permit, a Mined Land Reclamation Permit Amendment and Stormwater Discharge Authorization Update from the State of Arizona;
- will probably not require a Section 404 Dredge and Fill Permit from the US Corps of Engineers in respect of two drainages in which historical mining and other activities have taken place;
- will not require an approval of a Mine Plan of Operations from BLM (the operations will exclusively be located on private [i.e. patented] land;) and
- would probably not require any special access rights across federal lands regulated by BLM if the use of Silver Creek Road (that forms part of the access route to the Moss Mine Property - see Section 5.3) is kept to a practicable minimum.

4.7.2 Aquifer Protection Permit

Aquifer Protection Program (“APP”) permits are issued by the Arizona Department of Environmental Quality (“ADEQ”). APP permits are required to ensure that the groundwater quality in Arizona is maximized, where there is a reasonable probability that pollutants may reach an aquifer. The Arizona Administrative Code (“AAC”) R18-9-A202(A)(5) requires that an application for an APP include a description of the Best Available Demonstrated Control Technology (“BADCT”) to be employed at a specific mining facility. There are five demonstrations required for obtaining an APP permit:

- the facility will be designed, constructed and operated in accordance with BADCT requirements;
- the facility will not cause or contribute to an exceedance of Aquifer Water Quality Standards (“AWQS”) at the point of compliance or, if an AWQS for a pollutant has been exceeded in an aquifer, that no additional degradation will occur (AAC R18-9-A202(A)(8)(a and b));
- the person applying for the APP is technically capable of carrying out the conditions of the permit (AAC R18-9-A202(B));
- the person applying for the APP is financially capable of constructing, operating, closing and assuring proper post-closure care of the facility (AAC R18-9-A203); and

- the facility complies with applicable municipal or county zoning ordinances and regulations (AAC R18-9-A201(A)(2)(c)).

A permittee or applicant is required to propose an applicable point of compliance (i.e. monitoring well) or multiple points of compliance (depending on the operation) to monitor impacts from the operations on groundwater and to ensure that BADCT provisions are effective. Typically, the monitoring is conducted for eight consecutive quarterly observations to establish baseline conditions during the early stages of mine facility development. Alert levels are then established based on this monitoring to signal when impacts may threaten groundwater quality and intervention may be required. Financial assurance is required prior to the issuance of an APP permit.

The APP Permit process typically takes twelve to eighteen months, depending on the complexity of the hydrogeology and mining operations as well as the workload/budget restrictions in place at ADEQ's office at Phoenix, Arizona. However, an APP was in place for the Phase I activities and it is anticipated that an amended APP only will be required for Phase II, for which a lead time of approximately 10 months is anticipated. This lead time includes the statutory thirty day public comment period after ADEQ publishes its decision to approve a permit, prior to its issuance.

4.7.3 Class II Air Quality Permit

The Phase I activities showed that uncontrolled maximum air pollution emissions were less than significant pollutant levels, as defined by AAC. However, based on projected increases in production and expansion of operational equipment during Phase II, it is anticipated that a synthetic minor Class II Air Quality Permit will be required. Obtaining such a permit involves a two-tier approach, with the following major tasks assigned to the first tiered critical path:

- confirming the necessity for an air quality permit;
- establishing the jurisdiction of agencies;
- defining the permit rules and applicable requirements;
- confirming the appropriate air quality permit class;
- securing the permit criteria, obligations, provisions and checklists from the applicable regulatory agency(s);
- preparing the permit application draft outline;
- reviewing air dispersion modeling regulatory requirements;
- defining and selecting approved AERMOD, AERMET, and AERMAP protocols and guidance;
- examining input criteria, availability of site-specific model inputs, data gaps and overall approach to air modeling;
- assembling the air dispersion modeling draft outline; and
- meeting with ADEQ personnel to confirm permit application and modeling approach.

Based on the results of the first tiered effort, the second tiered permit application process would sequentially proceed. The Company anticipates that the lead time to securing a Class II Air Quality Permit will be approximately 17 months, inclusive of ADEQ's administrative completeness review and substantive review.

4.7.4 Mined Land Reclamation Permit

A Mined Land Reclamation Permit in Arizona is issued through the Arizona Mine Inspector's office. An applicant is required, through the application process, to identify 1) the nature of the operations, 2) anticipated impacts and mitigation measures, 3) anticipated post mining land use and 4) reclamation measures required to achieve the post mining land use. Reclamation typically involves those measures necessary to stabilize reclaimed lands (for example, rock armour or re-vegetation) and provide public safety protection (for example, reduce highwalls or fence openpits).

Financial assurance, to ensure that the cost of reclamation will be available if the permittee becomes insolvent, is required as part of the Reclamation Permit application process. The amount of the financial assurance required is adjusted if there is any overlap between the costs of reclamation and the costs for APP closure.

A Mined Land Reclamation Permit was in place for Phase I. The Company anticipates that an amendment only to that permit will be required for Phase II. The Company further anticipates that a lead time of approximately nine months will be required before a permit is secured, inclusive of a review of the permit application that typically takes approximately four months, inclusive of a public comment period.

4.7.5 Stormwater Discharge Authorization

Either an individual National Pollutant Discharge Elimination System or a Multi-Sector General Permit ("MSGP") is required for mining operations in Arizona, depending on the individual operation. The MSGP requires preparation of a Stormwater Pollution Prevention Plan ("SWPPP"), which plan and related authorization were in place for Phase I. The Company anticipates that for Phase II it will be necessary to:

- revise the Phase I SWPPP to include the expanded facilities and provisions for mitigation of storm water run-on and runoff accordingly; and
- file a revision to the existing Notice of Intent with ADEQ to reflect the new changes.

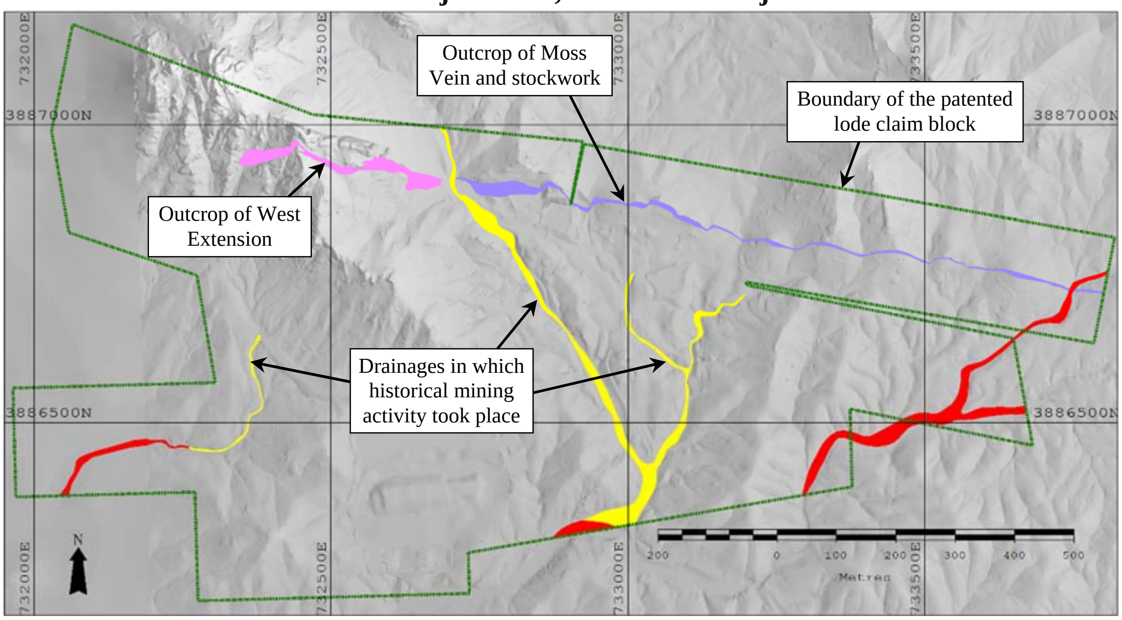
The Company further anticipates that the process leading towards an amended Stormwater Discharge Authorization will take up to approximately three months.

4.7.6 404 Dredge and Fill Permit

A permit is required through the Army Corps of Engineers ("Corps") under Section 404 of the Clean Water Act for the discharge of dredged or fill material into jurisdictional waters. The permit application process involves conducting baseline surveys to define these waters (a Corps determination) and for the presence or absence of any threatened or endangered species or habitats, significant cultural resources or otherwise sensitive lands or habitats that may be impacted within those jurisdictional waters. Any jurisdictional waters (which include ephemeral drainages that are also known as navigable waterways) that are impacted by mining operations must be mitigated under the permit. The permit process can take several years as it invokes provisions set out in the National Environmental Policy Act of 1970, inclusive of an Environmental Assessment or an Environmental Impact Statement, agency consultation and public involvement. A successful permittee is required under the permit to mitigate impacts to jurisdictional waters through rehabilitation or replacement. A bond is also required to cover the costs for mitigation.

MineFill understands that 404 Dredge and Fill Permits are not required for ephemeral drainages in which historical activities (mining or otherwise) have taken place. Four ephemeral drainages exist across the block of 15 patented lode claims detailed in Section 4.2.4 (Figure 4.14). The two drainages highlighted in RED will intentionally be avoided during Phase II; only the two central drainages highlighted in YELLOW will be affected. The need for a 404 Dredge and Fill Permit will be assessed as part of the feasibility study, going forward.

Figure 4.14: A Plan View Vulcan® Snapshot of the Boundary of the 15 Patented Lode Claims Showing the Position of the Drainages That Cut Across the Local Project Area, Moss Mine Project



4.8 Factors and Risks (Qualified Person's Opinion)

Based on its assessment of the standing, access and legal ownership of the land encompassed by the 15 patented lode claims detailed in Section 4.2.4, coupled with the Company's intention to restrict Phase II operations to the patented ground only, MineFill is aware of only two factors that might materially affect its right or ability to perform work on the property: a risk that the Corps might deem the two washes highlighted in YELLOW on Figure 4.14 to be juristictional washes; and BLM might, at some future point, impose restrictions on the use, by Moss Mine traffic, of Silver Creek Road (which extends over federal land and the use of which is required to access the Moss Mine Property – see Section 5.3).

4.8.1 Jurisdictional Washes

If the washes highlighted in YELLOW on Figure 4.14 were deemed to be jurisdictional washes, it would trigger the need for a 404 Dredge and Fill Permit before any Phase II-related activities could impact those washes. Assuming a successful conclusion to the permitting process, a lead time of up to approximately 18 months would be required. Preliminary analysis of Phase II options suggests that the majority of the required lead time could be accommodated within the scope of mine planning and production scheduling.

4.8.2 Property Access

To the best of MineFill's knowledge, under the existing Mining Law and applicable BLM regulations, the Company has the legal right to make reasonable use of Silver Creek Road for legitimate mining-related purposes. To the best of the MineFill's knowledge, no issues concerning the use of Silver Creek Road were raised by BLM during Phase I. However, the continued use of Silver Creek Road might require a Right of Way permit or other land use authorization from BLM.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

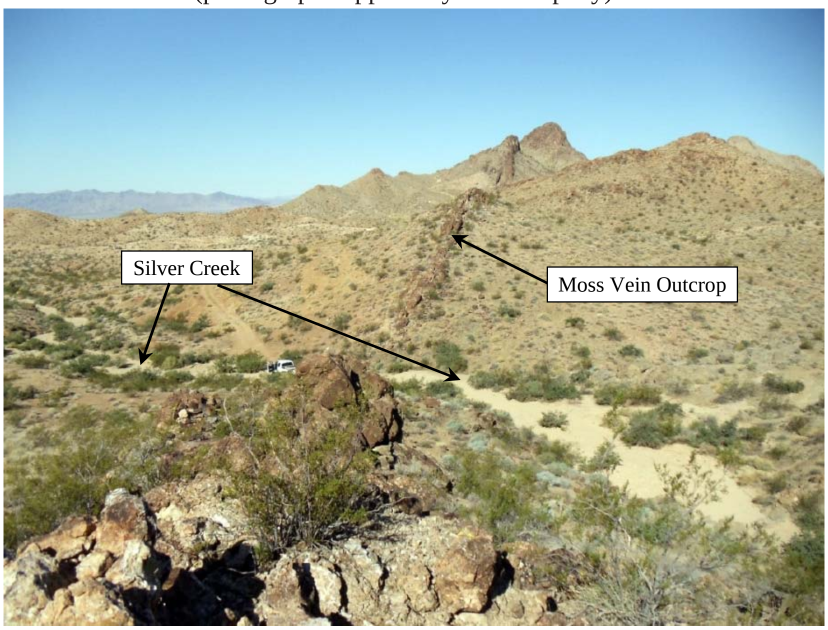
The Moss Mine Project area is located on the Davis Dam 1:100,000 scale topographic map (30 x 60 minute quadrangle) of the United States Geological Survey, BLM's surface management status and desert access guide maps and the Kingman, Arizona 1 x 2 degree, 1:250,000 topographical map (USGS).

5.1 Topography, Elevation and Vegetation

The Moss Mine Project area is located in the Black Mountain Range in the southern part of the basin-and-range topographic province. Elevations in the general area vary from 200 m (at Davis Dam, on the Colorado River) to 1,543 m (the peak of Mount Nutt). Elevations across the Project area vary from an average low of approximately 658 m to a local maximum of approximately 820 m at the western end of the Property (Figure 5.1). The Moss vein forms a prominent east-west ridge across the northern portion of the block of 15 patented lode claims described in Section 4.2.4. It is the Moss Vein and its associated stockworks from which a bulk sample was extracted by openpit mining during Phase I. It is the Moss Vein, its associated stockworks and Western Extension that are the principal targets for exploitation during Phase II.

The local Project area is drained by Silver Creek at the eastern end of the block of 15 patented lode claims (Figure 5.1), which is dry for most of the year and which drains southwest and then west into Colorado River. Vegetation is in general sparse; it comprises bunch grass, sagebrush and cacti. The Fort Mohave Indian Tribe and other private companies have created an agricultural community that covers several square miles in the fertile fields of Mohave Valley and Fort Mohave, to the immediate south of Bullhead City (the nearest city to the Moss Mine Project area – see Section 5.2). The main crops are cotton and alfalfa.

Figure 5.1: A General View of the Moss Mine Project Area (looking approximately west, from the eastern boundary of the block of patented lode claims) with the Local Topographic High in the Background
(photograph supplied by the Company)



5.2 Population Centres and Transport

The nearest major city to the Moss Mine Property is Las Vegas, Nevada, which is approximately 130 km northwest of the Property centre (Figure 5.2). According to the 2010 census, Las Vegas has a population of some 1.95 million people in the metropolitan area, including 0.58 million people in the city proper. Good quality paved roads (Highways 93 and 95 leading to Highways 68 and 163, respectively) link Las Vegas and Bullhead City, which is approximately 22 km by road and to the west of the Property centre (Section 5.3). Interstate Highway 40 is approximately 40 km to the south of the Property centre. There is an international airport at Las Vegas from where chartered flights can be secured to the Laughlin/Bullhead City International Airport located on the Arizona side of Colorado River, which forms the local boundary between the two states. The nearest railway station is at Needles, Nevada, approximately 32 km to the southwest of the Moss Mine Property centre.

The nearest town to the Project area is Oatman, Arizona, which is approximately 10 km to the south-southwest of the Property centre. According to the 2010 census it had a population of 135 people; during the Oatman boom (Section 6.1) it was a mining town with a population estimated at 10,000. The nearest cities to the Moss Mine property are Bullhead City in Arizona and Lauglin in Nevada. According to the 2010 census, Bullhead City has a population of approximately 39,500 people with approximately 100,000 people living in the Bullhead City-Laughlin area, including adjacent communities. Kingman, Arizona, approximately 37 km due east of the Moss Mine Property centre, is the Mohave County seat. Phoenix is the Arizona state capital, which is approximately 290 km to the southeast of the Moss Mine Property centre.

5.3 Means of Access

Road access to Bullhead City from Las Vegas is straightforward: the approximately 155 km journey takes 1.5 hours (Table 5.1). Moss Mine Property is reached by travelling south on US Highway 95 By-Pass (Bullhead City Parkway) to the turning left (east) onto Silver Creek Road, a graded dirt road. After some 9.0 km a turning left (north) is made onto a local dirt road that leads to the Moss Mine property. No restrictions have been identified for the roads that would affect hauling of equipment or supplies needed for the Moss Mine Project. All materials have and will continue to be transported under Federal Department of Transportation standards and other federal regulations.

Table 5.1: A Summary of the Most Direct Route from Downtown Las Vegas to the Moss Mine Property

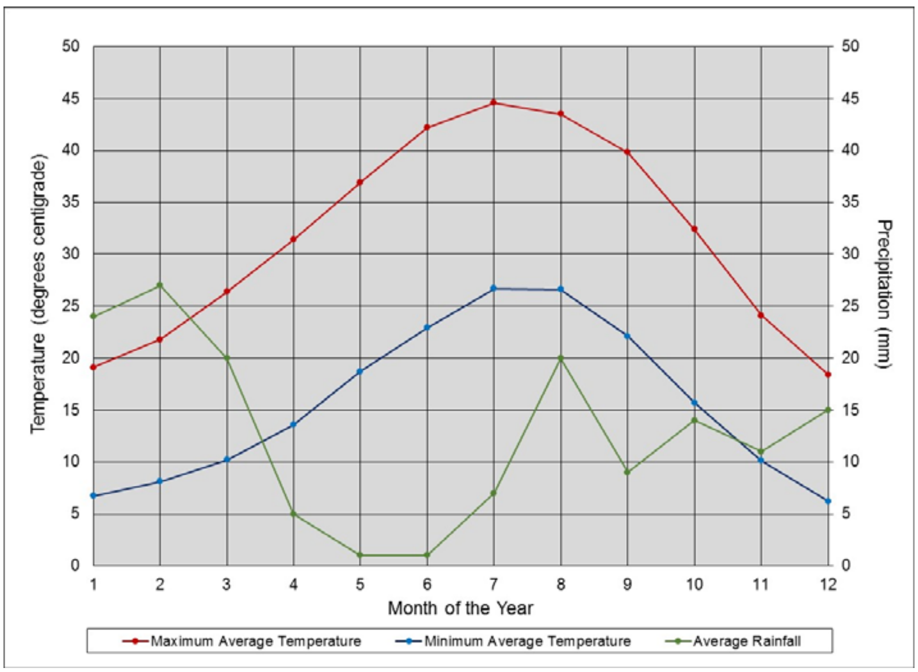
From	To	Road	Distance (km)
Downtown Las Vegas	US Highway 95 turning	Great Basin Highway (US Highways 93/95)	36.0
US Highway 95 turning (right)	Laughlin Highway via Searchlight and Cal-Nev-Ari	US Highway 95	88.5
Laughlin Highway turning (left)	Laughlin	Nevada State Highway 163	31.0
Laughlin	Silver Creek Road via Bullhead City	Arizona State Highway 95 By-pass (Bullhead City Parkway)	8.2
Silver Creek Road (left)	Moss Mine turn-off	Silver Creek Road (graded dirt road)	9.0
Turn north (left)	Moss Mine	Local dirt road	2.5
		Total Distance	***175.2***

Figure 5.2: A Google Earth Image of the Location of the Moss Mine Project Area Showing the Major Roads Linking Bullhead City and Las Vegas

5.4 Climate and Operating Season

The climate in the general Project area is classified as desert (Koppen climate classification BWh). In the Holdridge Life Classification zone it is in a warm temperate latitudinal region, pre-montane to lower montane altitudinal zone and a desert humidity province. There are no climatic constraints on the operating season, although daytime temperatures can exceed 40ºC (104ºF) during June, July and August (Figure 5.3). Heatwaves with temperatures in excess of 50ºC (122 ºF) are not uncommon. The average annual rainfall at Bullhead City is 154 mm (6.06 inches, data ex. www.usclimatedata.com). No rain can fall for months and occasional heavy downpours occur.

Figure 5.3: A Summary of the Monthly Average Temperatures and Rainfall for Bullhead City, Arizona
(compiled from information contained on www.usclimatedata.com)



5.5 Surface Rights, Power, Water and Personnel

5.5.1 Surface Rights

Activities during the Phase I were limited to the 15 patented lode claims described in Sub-Section 4.2.4. Phase II will also be limited to the same 15 patented lode claims. It is established in Sub-Section 4.2.4 that:

- a patented lode claim is one for which the Federal Government has passed title to the claim holder, thereby making it private land; and
- the patent gives the owner full and exclusive title to the surface area of these claims.

5.5.2 Power and Water

Colorado River is approximately 12 km to the west of the Property centre. It flows from north to south and divides the state of Arizona from Nevada and California. Hydroelectric power is generated at Davis Dam on Lake Mohave (approximately 8 km north of Bullhead

City) and at Hoover Dam on Lake Mead (approximately 100 km north-northeast of Bullhead City). A major powerline passes some 6.0 km to the west of the Moss Mine Property centre.

Diesel generated power from rented, portable power generators was used during Phase I. Water for heap leaching and dust suppression was extracted from waterwells located within the boundary of the block of 15 patented lode claims. The Company plans to continue to use diesel generated power and groundwater from the same sources during Phase II.

5.5.2 Personnel

Abundant accommodation, supplies, services and related recreational and light industry facilities are available in the Bullhead City-Laughlin area. The casinos and ancillary services at Laughlin provide much of the local employment, but there is a long history of gold mining in the area from where a potential workforce for the Moss Mine could be found (and was hired for purposes of Phase I). Technical and management roles will continue to be filled by suitable professionals, who would be housed in the Bullhead City-Laughlin area.

5.6 Tailings, Waste, Heap Leach Pad and Plant Areas and Sites

A total of approximately 172,500 t of mineralized material and zero grade waste were mined by openpit during Phase I. Approximately 100,000 t of material was stacked on the heap leach pad (Figure 5.4), with the balance remaining in various stockpiles and waste rock dumps located on patented ground. The metal contained in the pregnant solution from the heap leach pad was adsorbed onto carbon in a conventional carbon column plant (Figure 5.5). The carbon was stripped and doré was produced off-site by third parties. Assaying of the pregnant solution and blastholes was carried out at an on-site laboratory located adjacent to the carbon column plant.

For Phase II, the Company anticipates that the pregnant solution from the heap leach pad will be processed through the existing carbon column plant that might require some upgrading to accommodate an envisioned increase in the production/throughput rate. The Company's project planning encompasses the production of doré on-site in a modular stripping, electro-winning and smelting facility. In MineFill's opinion there is sufficient space for Phase II mining, heap leaching and ancillary facilities on the patented ground.

5.7 Qualified Person's Opinion

In the opinion of the Qualified Person for this section of this Technical Report (Dr. David Stone, P. Eng.), there is no readily identifiable reason to suppose that the Company's on-going Moss Mine Project development plans, through to completion of Phase II, could not successfully and safely be carried out. MineFill understands that the Company will be undertaking a pumping test program and other hydrogeological studies as part of the on-going feasibility study of and for Phase II. MineFill further understands that as part of this work a risk management contingency plan will be developed in the event that sufficient groundwater could not consistently be drawn from the waterwells located on the Moss Mine Property. MineFill agrees with this approach that reflects best industry practice.

Figure 5.4: A General Aerial View (looking approximately north) Of the General Area of the Phase I Heap Leach Pad
(photograph supplied by the Company in December 2013)

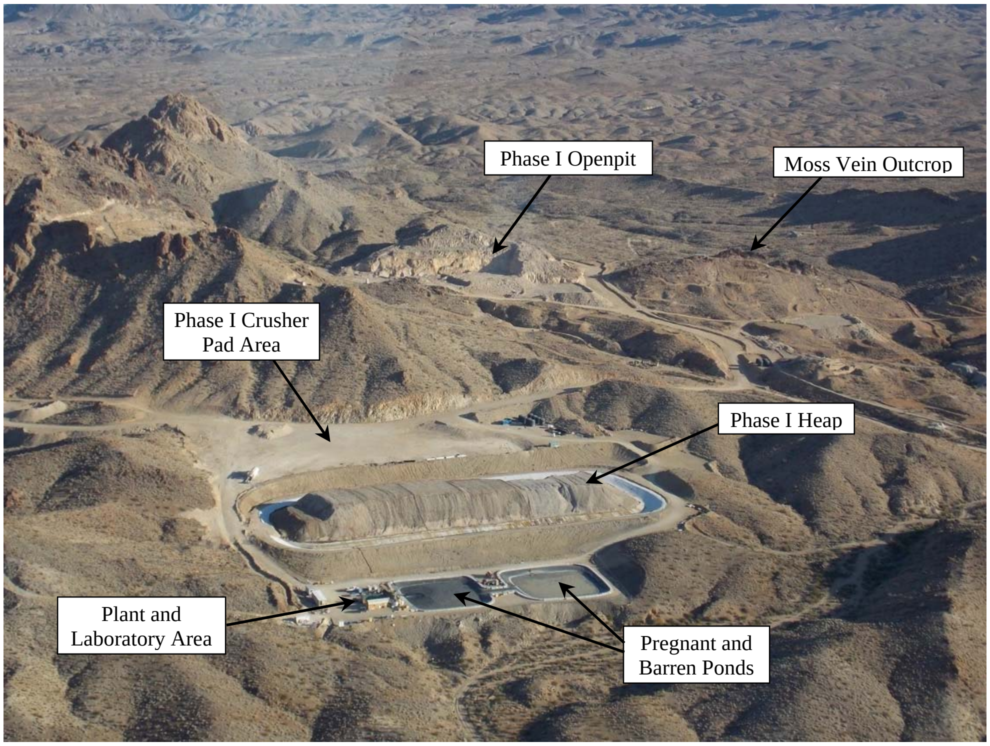


Figure 5.5: A General View (looking approximately southwest) of the Phase I Pilot Plant Area with the Pregnant and Barren Ponds in the Foreground
(photograph supplied by the Company, taken in October 2103)

6 HISTORY

Compilation of the following text relied in part on information contained in the 2013 Technical Report. As part of its due diligence process and to the extent possible, MineFill cross-referenced the information to the source documents. In the opinion of the Qualified Person for this section of the report (Dr. David Stone, P. Eng.) there is no readily identifiable reason to suppose that the information presented in this section does not fairly represent a summary of the history of the Moss Mine Project.

6.1 Discovery and Early Mining (1863 to 1935)

The Moss Mine Project was discovered in 1863 by John Moss (1839-1880). At the time it was reported to be the first major gold discovery in Mohave County. The larger San Francisco Mining District of Mohave County was established in 1864 (Malach, 1977).

The available records show that John Moss was made aware of the Moss Mine area by stories about soldiers from nearby Fort Mojave prospecting for and finding gold. A popular, alternative account of the Moss Vein discovery is that Chief Irataba of the Mojave Tribe led Moss to what became known as the Moss Vein outcrop. Whatever the case, John Moss' name appeared on the first recorded mining claim called the Moss Lode, under the ownership of the San Francisco Gold and Silver Company. It was reported that a '*shoot containing more than $200,000 in gold*' was mined in a 3 m wide and 3 m deep glory hole on the claim, to the east of the later site of Allen Shaft (Figure 6.1).

The available records show that Moss sold the Moss Lode to Dahrean Black and that it was later sold to the Gold Giant Mining and Milling Company of Los Angeles. The area around the glory hole was explored by numerous holes and tunnels, but no other substantial quantities of gold are reported to have been found. The Ruth Vein was subsequently discovered and a 70 m (230 ft) shaft was sunk and '*hundreds of feet of tunnels*' were developed (Malach, 1977). The Moss Mine is reported to have produced approximately 12,000 ounces of gold until it was closed in 1866 due to '*unfriendly Indians*' (Durning & Buchanan, 1984).

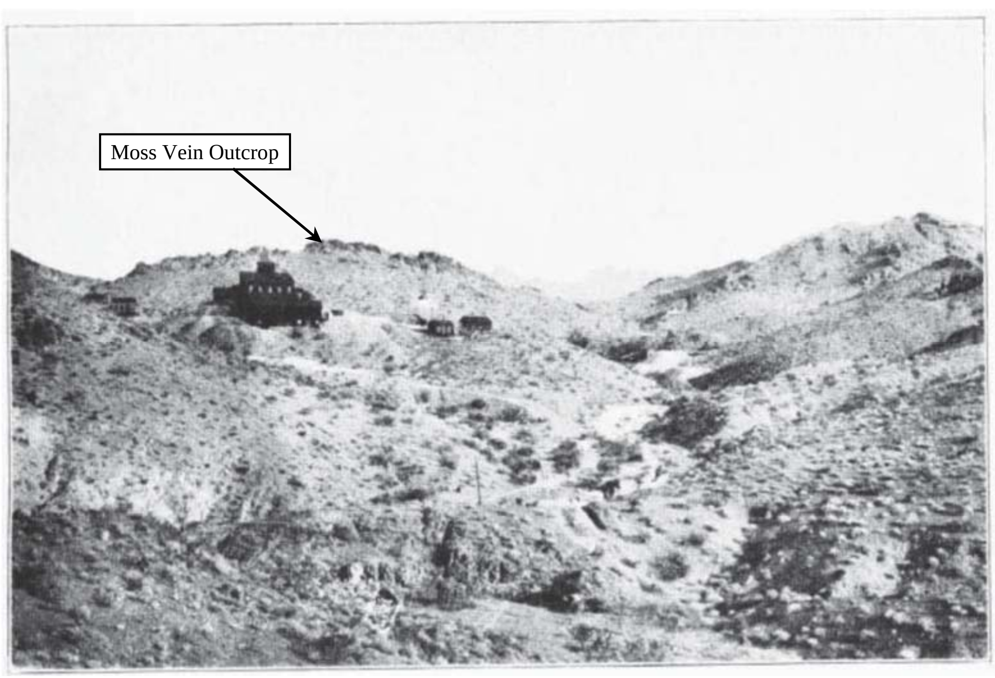
Following its abandonment in 1866, there was little mining activity in the district until the opening of the Tom Reed mine in 1901 and the discovery of the regionally famous Gold Road Vein in 1902. The town of Vivian was founded in that year; its name was changed to Oatman in 1908. In 1906, the Tip Top and Ben Harrison mineralized shoots were discovered. In 1915 and 1916 the Big Jim, Aztec and United Eastern mineralized bodies were discovered on the Tom Reed Vein. Mining activity increased and the population of Oatman grew to a reported 10,000 (today referred to as the Oatman boom, 1915 to 1917). By the mid-1920s the population of Oatman had fallen to a few hundred. In 1933, an increase in the gold price from US$20 to US$35 per ounce resulted in a brief flurry of activity, but all the local mines were closed by 1942 (Ransome, 1923; Sherman & Sherman, 1969; Varney, 1994).

Historical underground mine plans of the Moss Mine in the Company's database are dated May 10, 1915 by Goldroad Mines Co. of Goldroad, Arizona, and September 25, 1920 by the Moss Mines Co. of Gold Road, Arizona. These show the Allen Shaft and levels at 60 ft (18.3 m), 75 feet (22.9 m), 125 feet (38.1 m) and 220 feet (67 m). The plans show that Moss Mine was operating between 1915 and 1920.

The available records show that the Ruth Mine (Figure 6.2) was accessed by a 60º degree incline shaft to drifts on 100-, 200- and 300-ft Levels. Activity appears to have continued through to mid-1935, by which time approximately 183 m (600 ft) of drifting is reported to have been completed (unverified information supplied by the Company). The Company advised MineFill that plans detailing these workings are not available.

Figure 6.1: An Historical Photograph (looking approximately east-northeast) of the Allen Shaft at Moss Mine, 1920 – 1921
(copied from Ransome [1923], Plate IX-B)

Figure 6.2: An Historical Photograph (looking approximately northeast) of the Ruth and Moss Mines, 1920 – 1921
(copied from Ransome [1923], Plate IV-B)



6.2 Previous Exploration and Development (1982 to 2009)

Table 6.1 summarizes the work carried out on the Moss Mine Property by previous owners and operators, up to and including Patriot Gold's last exploration program in 2009. The comments contained in the following sub-sections apply.

Table 6.1: A Summary of Exploration and Development Work Carried Out by Previous Owners and Operators on the Moss Mine Property (the 15 patented lode claims) to 2009
(compiled from information contained in the 2003 Technical Report and in Various Cross-Referenced Documents Supplied by the Company)

Company	Date	Work Completed	Comments
Moss Mine	1860 to 1920	Surface holes and underground mining	12,000 oz of gold reported to have been extracted
Ruth Mine	1900? to 1935	Underground mining	Approx. 24,400 t of mineralized material extracted
BF Minerals	1982	54 rotary airtrack holes, four reverse circulation ("RC") holes for a total of approximately 1,885 m (6,190 ft)	Only assayed Moss Vein material.
Harrison Minerals	1987 to 1988 (exact dates unknown)	Rehabilitated Allen Shaft and deepened it to 91.4 m (300 ft)	Constructed headframe in 1987, reportedly left broken mineralized material in stopes, 3,000 to 5,000 short tons trucked to Tyrol mill.
Billiton Minerals	1990	21 RC holes for a total of 2,190.4 m (6,925 ft)	Preliminary analysis of gold and silver deportment, preliminary metallurgical tests.
Magma Copper Company	1991	21 RC holes for a total of 3,012.5 m (9,890 ft)	Developed local geological maps. Metallurgical testwork carried out by McClelland Laboratories.
Reynolds Metals Explorations,Inc.	1991	11 drillholes for m (4,865 ft), plus two RC holes (152.3 m, 500 ft)	Collar co-ordinates not available.
Golconda Resources	1993	19 RC holes for a total of 931.5 m (3,058 ft)	-
Addwest Minerals International Ltd.	1996 to 1997	30 RC holes for a total of 2,502.8 m (8,217 ft) plus six diamond drillholes for a total of 507.8 m (1,667 ft)	Developed a new geological model.
Patriot Gold Corporation	2004 to 2009	43 RC holes for a total of 3,596.4 m (11,807 ft) plus 12 diamond drillholes for a total of 2,085.3 m (6,846 ft)	Consolidated land position, carried out geological studies and surveys. Contracted Metcon Research to carry out metallurgical testwork.

BF Minerals drilled 54 shallow air track rotary holes and four reverse circulation ("RC") holes for a total of 6,190 feet (1,997 m) at Moss Mine in 1982. BF Minerals leased the mine to Harrison Minerals in 1987 and 1988, during which period Allen Shaft was dewatered and extended to the 91 m (300 ft) level (Figures 6.3 and 6.4). The 300 Level underground workings were reported to have been rehabilitated and mined, with a small amount of mineralized material extracted and hauled to Tyro Mill in Mohave County (Section 6.4). The Moss Vein on 300 Level was reported to be 12.2 m (40 ft) wide and contain to contain 8.6 g/t Au (reported as 0.25 oz/ton Au).

The Moss Mine area was explored by Billiton Minerals (then part of Shell Mining Company of the Royal Dutch Shell group) from 1989 to 1992. They drilled 21 RC holes totaling 2,190.4 m (6,925 ft) in 1990/1991. Gold and silver deportment analyses, grain size determinations, a gravity separation test and preliminary bottle roll tests were also carried out (Baum and Lherbier, 1990).

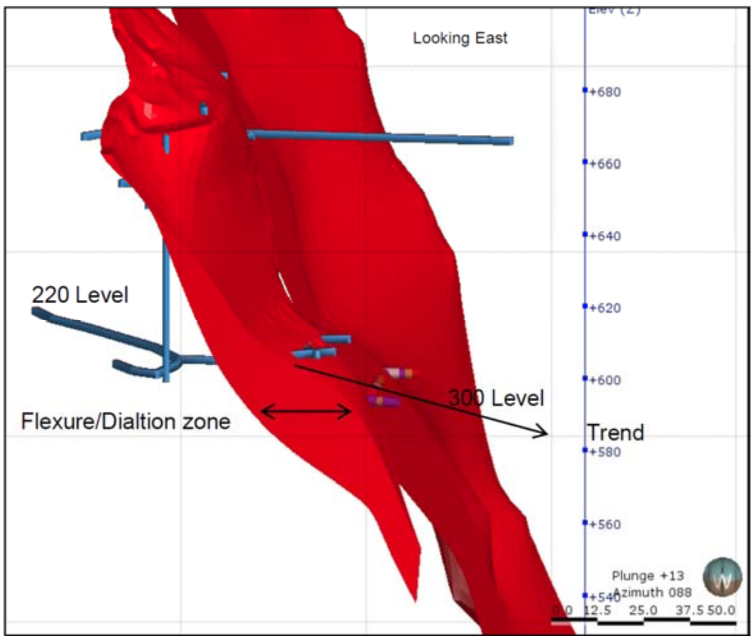
Magma Copper Company ("Magma Copper") sub-leased the property from Billiton in 1990 and 1991, and drilled 21 RC holes totaling 3,012.5 m (9,890 ft). During this period Mintec, Inc. ("Mintec") was contracted to compile an '*indicated in-situ resource estimate*' and an '*estimated pit resource*' (terminology used by Mintec that does not conform with CIM 2014 Definitions Standards for Mineral Resources and Mineral Reserves and cannot, therefore, be relied upon), the outcomes of which are summarized in Section 6.3. McClelland Laboratories, Inc. of Sparks, Nevada ("McClelland Laboratories"), was contracted to carry out preliminary metallurgical testwork, the results of which are presented in Section 13 (Mineral Processing and Metallurgical Testing).

Figure 6.3: A 2012 Photograph (looking approximately northeast) of the Allen Shaft Headframe at the Moss Mine Project Site with Historical Moss Vein Workings Evident in the Background

(photograph supplied by the Company, the headframe was moved to Bullhead City Recreational Park, as part of the Company's community relations program, prior to Phase I openpit mining activity that removed Moss Vein material to the right of the photograph)

Figure 6.4: A Snapshot (looking east) of the Known Historical Mine Workings Leading From Allen Shaft with the Local Moss Vein Identified in RED, Moss Mine Project

(copied from a 2014 consultancy report to the Company by Doug Brownlee)



Golconda Resources optioned the Moss Mine in 1993 and drilled 19 shallow RC holes for a total of 931.5 m (3,058 ft). They terminated the option in 1994. Addwest Minerals International Ltd. (“Addwest”) acquired the property in 1995 and staked additional claims. In 1996, Addwest drilled 30 RC holes totaling 2,502.8 m (8,210 ft) and six diamond drillholes totaling 507.8 m (1,667 ft).

In 2004, MinQuest acquired the seven patented lode claims that are subject to the MinQuest Agreement described in Section 4.4.1 and staked the Moss 1 to Moss 148 series of 104 unpatented lode claims. MinQuest did not, however, carry out any exploration work for its own account.

Patriot Gold acquired the claims from MinQuest in March, 2004 and carried out exploration through 2009, including 43 RC holes totaling 3,596.4 m (11,807 ft) and 12 diamond drillholes totaling 2,085.3 m (6,846 ft). MinQuest carried out the exploration programs for Patriot Gold. Kappes, Cassidy & Associates also undertook various metallurgical test programs for Patriot Gold, summaries of which are presented in Section 13.

In February 2011 the Company entered into an agreement with Patriot Gold to acquire a 70% interest in the 15 patented lode claims and 104 unpatented lode claims of the Moss 1 to Moss 148 series. Details of the Patriot Gold Agreement are presented in Section 4.4.2.

6.3 Historical Mineral Resources and Mineral Reserves

The only known Mineral Resource estimate relating to the Moss Mine Project that was compiled by previous owners or operators of the property is that by Mintec, in 1991, for Magma Copper. The ‘*indicated in-situ resource estimate’* (terminology used by Mintec) totalled 6.71 Mt at 1.302 g/t Au for a 0.686 g/t Au cut-off (reported as 7.4 million short tons at 0.038 oz per ton Au for a 0.02 oz per short ton grade cut-off). The ‘*estimated pit resource*’ (terminology used by Mintec that does not conform with CIM 2014 Definitions Standards for Mineral Resources and Mineral Reserves) totalled 2.63 Mt at 1.51 g/t Au (reported as 2.9 million short tons grading 0.044 oz per ton Au), with a 1.96:1 strip ratio. The criteria used to estimate the resource are not fully known but in MineFill’s opinion, the Mineral Resources outlined were not estimated to CIM standards and cannot therefore be relied upon. The estimates are presented here for information purposes only.

MineFill is not aware of any Mineral Reserve estimate relating to the Moss Mine or Moss Mine Project area.

6.4 Property Production

Production details for the historical Moss mine are limited. As previously outlined (Section 6.1), a total of some 12,000 oz of gold is estimated to have been produced prior to 1920, and that in (probably) 1988, a total of between 3,000 and 5,000 short tons were extracted and hauled to Tyro Mill in Mohave County (unverified information supplied by the Company, the grade of the mineralized material is unknown). In MineFill’s opinion this suggests that the underground workings are limited in extent: by necessity (due to the prevailing gold price) selective high-grading of Moss Vein material only would have locally been carried out.

The available records for Ruth mine suggest that prior to 1907, ‘*several hundred tons*’ of mineralized material had been extracted, for processing at Hardyville. During the Oatman boom the mine was extended and, according to Ross Barkley, mine superintendent in the 1930s, approximately 22,680 t (reported as 25,000 short tons) were mined on 100 Level (unverified information supplied by the Company). Mining ceased when a geological fault was encountered.

In 1933 Ross Barkley and two partners obtained a bond and lease on the Ruth Mine, found mineralized material on the other side of the intersecting geological fault and, during 1933 and 1934, '*shipped US$25,000 worth*' of mineralized material (reported to be worth US$14.70 per short ton, thereby yielding an output of some 1,543 tonnes or 1,700 short tons of mineralized material) to the Tom Reed mill (unverified information supplied by the Company). When the mine changed hands in 1935 shipments totalling 500 short tons at US$9.45/short ton were made in February, along with 900 short tons at US$13.00/short ton in March and 1,200 short tons at US$14.00/short ton in April (unverified information supplied by the Company). For the gold price prevailing at the time (US$35/oz Au), the production records outlined suggest grades of between approximately 9.0 g/t and 14.0 g/t Au for the extracted material, hence selective high-grading along what were known as pay shoots (i.e. high-grade zones of mineralized material).

7 GEOLOGICAL SETTING AND MINERALIZATION

The Moss deposit lies in Oatman Mining District of Mohave County, Arizona. The regional geology is shown on the Geologic Map of Mohave County, Arizona at 1:375,000 scale (Wilson & Moore, 1959). The geology of Oatman Mining District was mapped at 1:48,000 scale by Ransome (1923, Figure 7.1). The western part of Oatman Mining District is covered by geological mapping at 1:24,000 scale for the proposed State Route 95 realignment corridor (Pearthree et al., 2009, Map Sheet 4 of 5). There is neither a published 7½ minute quadrangle geological map for Oatman nor a regional 1 x 2 degree geological map.

The geology of Oatman Mining District has been described in memoirs and papers by Schrader (1909), Ransome (1923), Lausen (1931), Thorson (1971), Clifton et al. (1980), Durning & Buchanan (1984), DeWitt et al. (1991) and Harris (1998). It is also referred to in Korzeb (1988). The bladed quartz after calcite textures found in Oatman Mining District were described and illustrated by Grout (1946). Oatman samples were included in a study of strontium isotopes in veins by Reesman (1968) and samples were analyzed for lead isotopes as part of a regional metallogenic survey of Arizona by Bouse et al. (1999). Marsh & McKeon (1983) used Oatman as a test area for early studies of hyperspectral analysis for alteration mapping.

7.1 Oatman Mining District

Oatman Mining District lies on the southwest flank of the Black Mountains within an eroded volcanic centre of Lower Miocene age (+23 to 18 million years old ["Ma"] - Durning and Buchanan, 1984). It is formed by a thick sequence of andesite, trachyte, latitic dacite, dacite and rhyolite volcanic rocks intruded by monzonite to granite plutons. The centre of the volcanic complex was probably at Oatman, based on the concentration of rhyolite to latite dykes and plugs and two epizonal plutons. The basement is Precambrian gneiss, schist and granite. The volcanic rocks have a 10° to 35° regional dip to the east, which is attributed to rotation along a west-dipping, low-angle detachment fault near the base of the volcanic rocks.

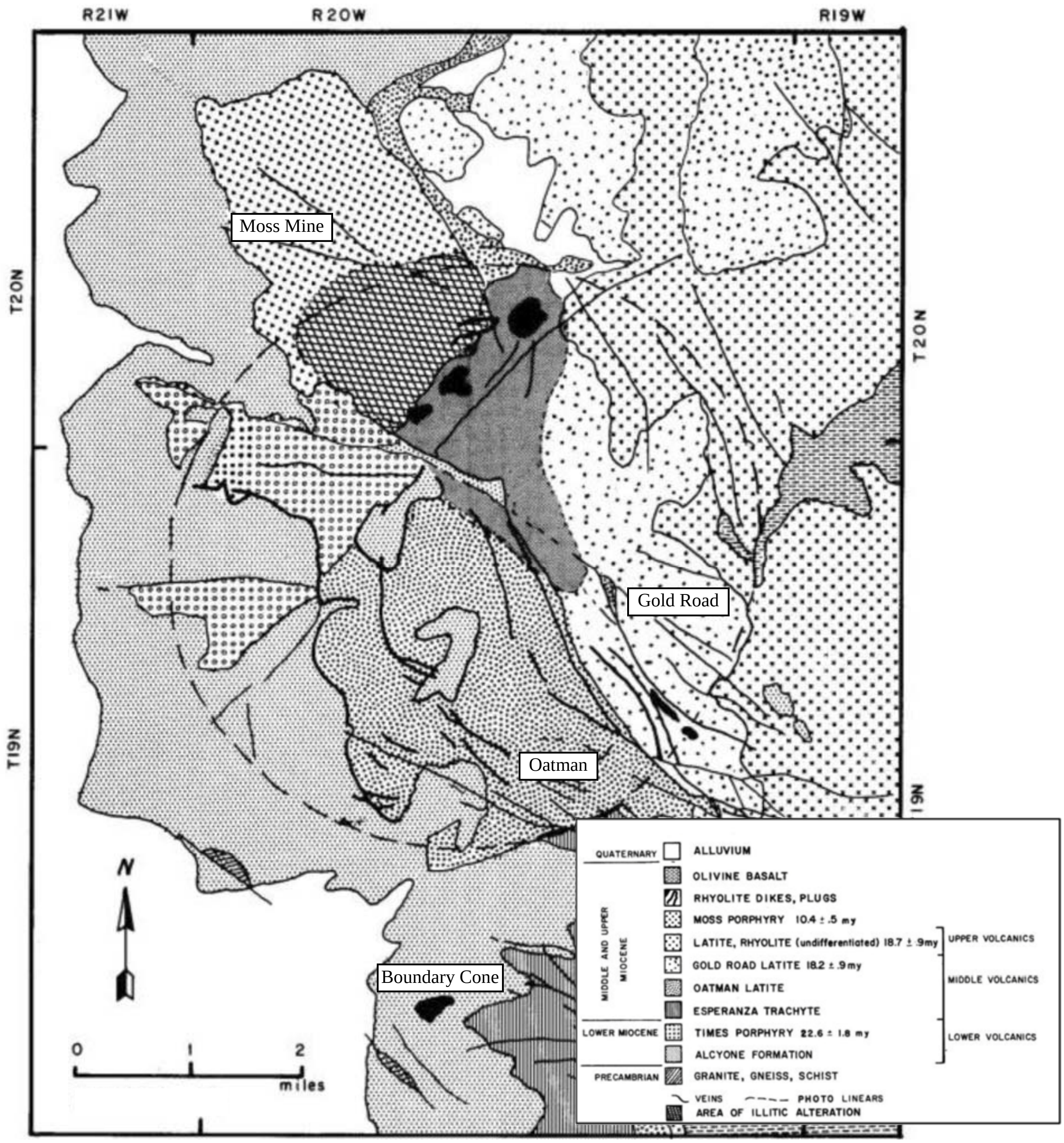
7.1.1 Stratigraphy

The volcanic stratigraphy of Oatman Mining District summarized below is from DeWitt et al. (1991) and from Durning and Buchanan (1984), who based it on Thorson (1971) and their own work. The Precambrian basement is composed of biotite schist, granite gneiss and biotite granite. It is overlain by the Lower Miocene Alcyone Formation or Lower Volcanics, a sequence of welded trachyte tuffs, quartz latite flows, tuff breccias and minor carbonaceous shales and limestones. The Alcyone Formation is called the Peach Spring Tuff on the map of Pearthree et al. (2009).

The Middle Volcanics unconformably overlie the Lower Volcanics. The lowest formation is the Esperanza Formation, a quartz latite lava flow that is approximately 55 m to 305 m thick and which is overlain by the Middle Miocene Oatman and Gold Road Formations. The Oatman formation is host to most of the mineralized veins in Oatman Mining District. It comprises a sequence of non-biotitic, pyroxene latitic andesite flows, tuffs and flow breccias that are up to approximately 305 m thick at Oatman. Gold Road Formation is a sequence of biotitic pyroxene latitic andesite to dacite lava flows and lithic ash flows that are up to approximately 245 m thick. Gold Road Formation has been dated at 18.6 ±0.9 Ma (Thorson [1981], re-estimated by DeWitt et al. [1991]).

Figure 7.1: A Regional Geological Map of Oatman Mining District, Mohave County, Arizona, USA

(copied from Durning and Buchanan [1984], the grid system is based on the Gila and Salt River Township and Range Land Survey System)



Upper Volcanics unconformably overlie Middle Volcanics. The both comprise series of trachyte, quartz latite and rhyolite tuffs and flows. The lowest unit, the Antelope Quartz Latite, has been dated at 19.2 ±0.9 Ma (Thorson [1981] re-estimated by DeWitt et al. [1991]). Two small stocks and a series of dykes and plugs intrude the volcanic rocks in the Oatman area:

- Times Porphyry is a granophyre laccolith, which intrudes Alcyone Formation and has been dated by potassium-argon at 23.1 ±1.8 Ma (Thorson [1981], re-estimated by DeWitt et al. [1991]); and
- Moss Porphyry is a 3.2 km by 6.4 km, concentrically zoned stock with an outer monzonite border, an inner porphyritic tonalite margin and a central tonalite-granodiorite that has been dated at 10.7 ±0.5 Ma by potassium-argon (Thorson [1981] recalculated by DeWitt et al. [1991]), although uranium-lead dating of zircon gives an age of 18.5 ±2.5 Ma (DeWitt et al. [1991]).

The rhyolite and rhyolite porphyry dykes and sills are compositionally similar to Times Porphyry and are localized along northwest trending faults. The geochemistry of the volcanic rocks is alkalic to subalkalic, shoshonitic (highly potassic) calc-alkaline (DeWitt et al., 1991).

7.1.2 Mineralization

Mineralization in Oatman Mining District is hosted by a series of west-northwest to northwest trending, north dipping faults with up to 91 m to 183 m of dip-slip displacement and localized quartz veins, rhyolite dykes and plugs. The veins are sulphide-poor with quartz, calcite, adularia, chlorite and electrum. Quartz pseudomorphs after calcite are common. Quartz textures vary from banded chalcedonic to coarsely crystalline, but most is banded and fine grained. Pale green fluorite is reported in veins near Moss Porphyry and Times Porphyry, as an abundant gangue mineral that often contains included gold. Fluorite is rare elsewhere. Most of the historical gold production came from two veins: the Tom Reed vein and the Gold Road vein. Clifton et al. (1980) observed that gold mineralization is restricted to a maximum vertical interval of 310 m (average 180 m).

The age of mineralization is poorly constrained to the time interval between approximately 22 and 11 Ma (DeWitt et al., 1991). There is a potassium-argon date of 21.2 ±2.1 Ma from an impure mixture of adularia and quartz from the Kokomo vein, which is an approximate age. Veins cut the Moss porphyry which has a zircon date of 18.5 ±2.5 Ma and a potassium-argon cooling date of 10.7 ±0.5 Ma.

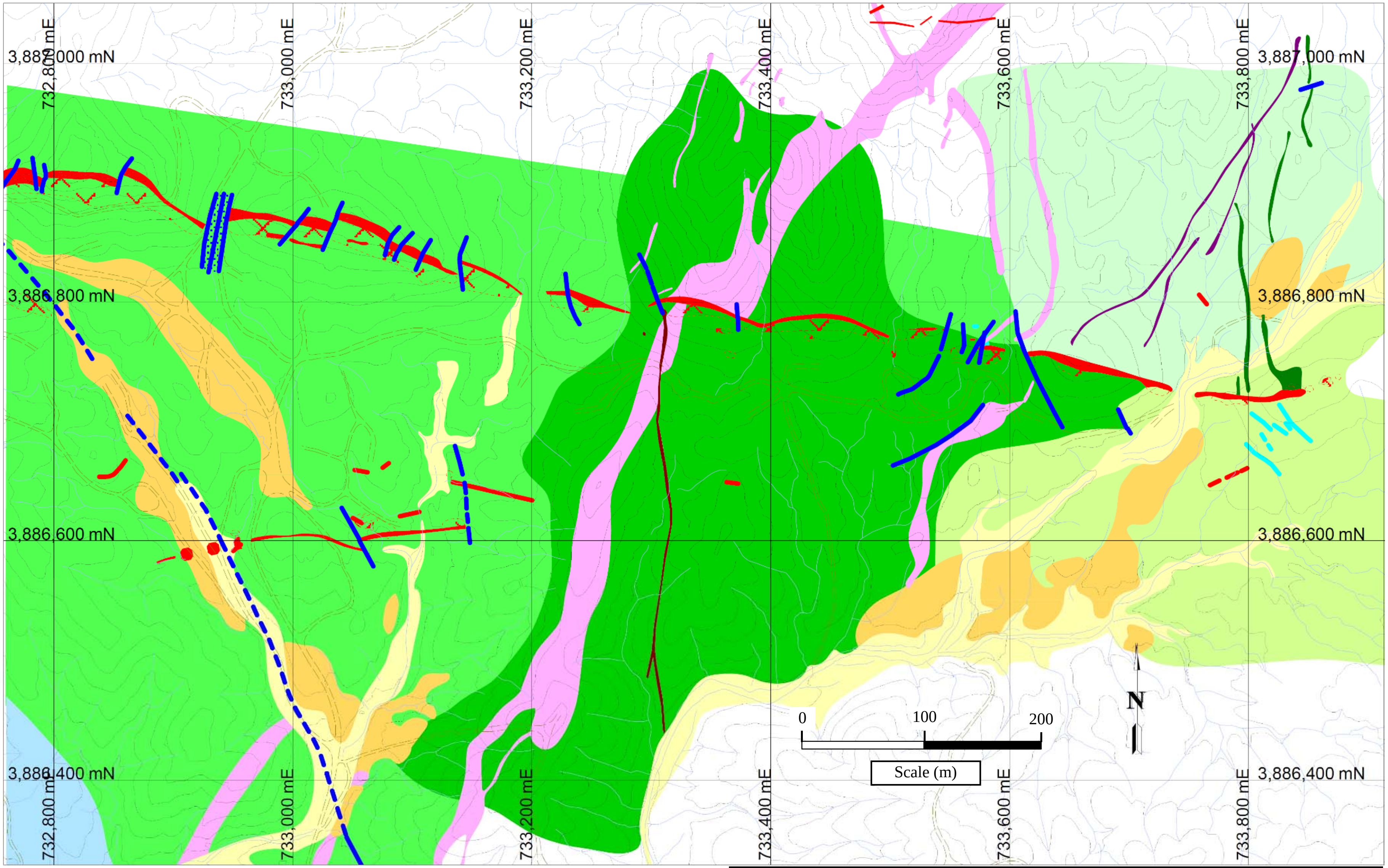
7.2 Moss Mine Property Geology

The geology of the local Moss Mine Project area defined by the 15 patented lode claims (Sub-Section 4.2.4) was mapped by Cuffney (2013). The geology and mineralization of the same area is described in consultancy reports by Baum and Lherbier (1990), Hudson (2011), Cuffney (2013) and Brownlee (2014).

7.2.1 Host Rocks

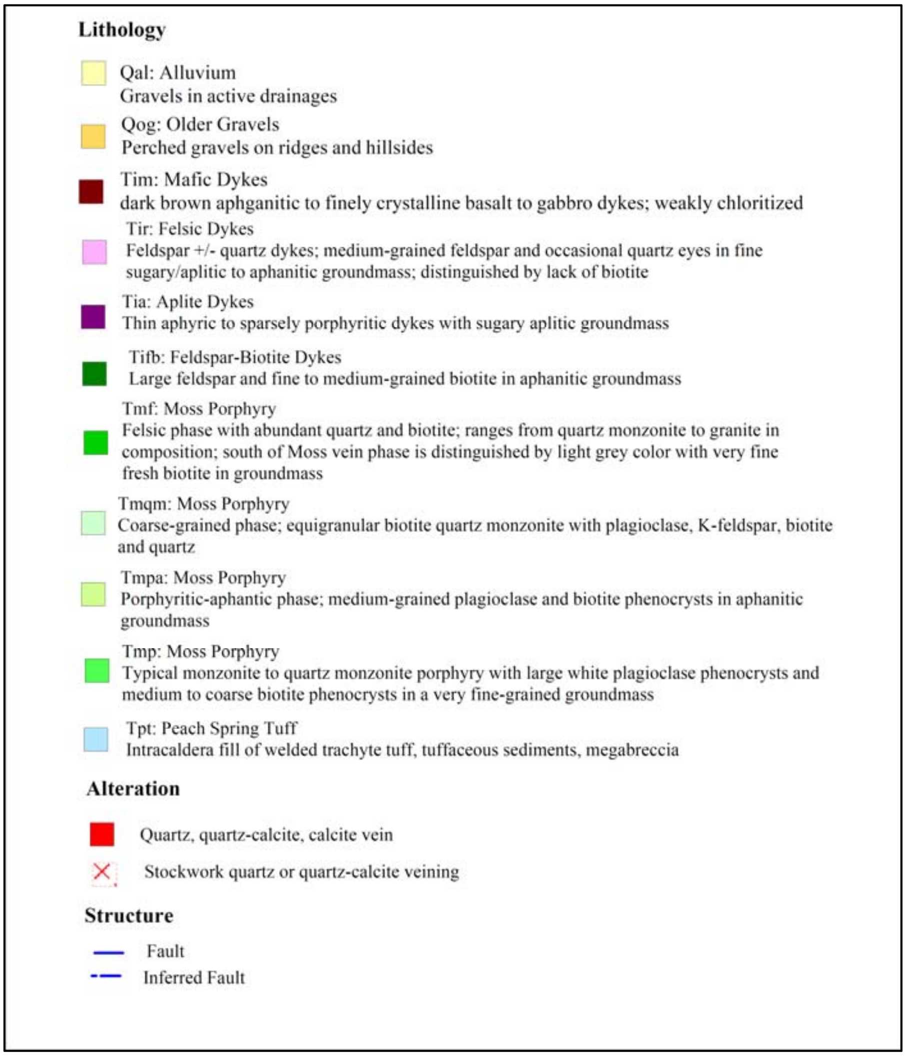
The host rocks of the Moss deposit is the Moss porphyry, a uniform monzonite to quartz monzonite porphyry intrusion. It is coarse grained with 4 mm to 10 mm diameter plagioclase phenocrysts with biotite and lesser hornblende. There is also a fine grained quartz monzonite porphyry, with 1 mm to 2 mm diameter plagioclase phenocrysts with minor biotite and minor magnetite, which is a later phase intrusive that cross-cuts the coarse porphyry and forms an intrusive breccia matrix in places. There is also an equi-granular quartz monzonite with abundant quartz and feldspar, and a quartz latite porphyry (Figure 7.2).

Figure 7.2: A Geology Map of the Moss Mine Project Area, Within and Immediately Around the Block of 15 Patented Lode Claims
(mapped by Cuffney [2013] and supplied by the Company, original in Brownlee [2014])



REFER TO FOLLOWING PAGE FOR THE LEGEND TO THIS PLAN

LEGEND TO FIGURE 7.2 ON THE PRECEDING PAGE



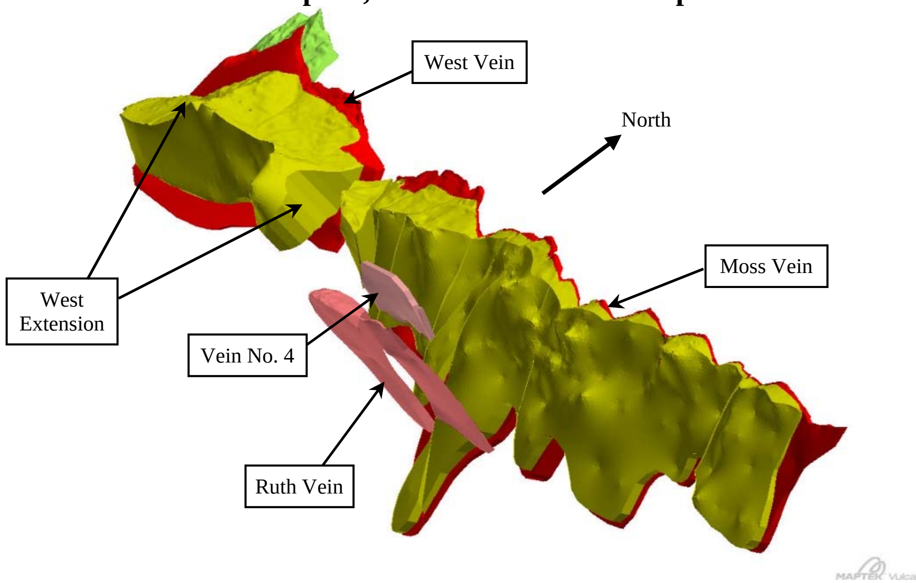
7.2.2 Mineralization

The gold-silver mineralization is contained within three main veins and their associated stockworks: the dominant Moss Vein; a western extension of the Moss Vein (the "West Vein"); and the Ruth Vein to the south of the Moss Vein. Moss Mine Project drillhole logs and assay database indicate a potential for other mineralized veins that are both similar to and sub-parallel to the Ruth Vein. For purposes of geological domaining they have been termed Vein No. 4.

Inferred Mineral Resources have been identified on the Ruth Vein (see Section 14) which, along with Vein No. 4, remains an exploration target. The focus of the Company's near-term, Moss Mine Project development plans is the Moss Vein and West Extension (West Vein and its associated stockworks) that are targeted for exploitation during Phase II. The Ruth Vein is not considered further in this section of this Technical Report.

Figure 7.3 identifies the component parts of the Moss deposit, as defined by the 2014 Mineral Resource model: vein material is identified in RED; hangingwall stockwork in YELLOW, and footwall stockwork in GREEN.

Figure 7.3: An Oblique Vulcan® Snapshot View (looking northwest) of the Component Parts of the Moss Deposit, from Surface to the Deepest Drillhole Intersections



7.2.2.1 Moss Vein

The dominant Moss Vein strikes N276E° (right-hand rule) and dips at approximately 70° to the south. Associated with the Moss Vein are stockwork veins and veinlets that are mainly concentrated on the vein's hangingwall side. The footwall contact is a well-defined shear structure.

The Moss Vein's footwall and hangingwall contacts are consistent along its developed length: the footwall contact is marked by a well-developed and persistent shear/fault; and the hangingwall contact is defined in part by vein content and by grade. In contrast, the position of the hangingwall contact of the hangingwall stockwork is more interpretive (it is defined predominantly by gold grade). Minor stockwork veins and veinlets also exist on the Moss Vein's footwall side, at two locations defined by drilling that may be associated with potential flexure dilation zones.

7.2.2.2 West Vein

The West Vein appears to be an extension of the Moss Vein, to the west of the Canyon fault: local field mapping suggests that there is little apparent displacement across the fault structure; West Vein has the same orientation and dip as the Moss Vein; but West Vein's footwall and hangingwall contacts are not as distinct; and its gold-silver mineralization persistently reports lower grades than the Moss Vein. The stockwork associated with West Vein (the "West Extension stockwork") is more extensive and better developed than that on the hangingwall side of the Moss Vein. The West Extension stockwork is also contiguous to a stockwork developed to the immediate west of the Canyon fault. These characteristics suggest that West Extension might represent a different vein assemblage that has been fault-displaced to its current position that could be a geological coincidence only. This possibility is examined further in Sub-Section 7.2.6.

7.2.3 Vein Mineralogy

Cuffney (2013) describes the Moss vein as '….*not a simple planar fissure-fill vein. The main vein is best described as a "breccia vein" (as opposed to a brecciated vein). The vein ranges from nearly solid white quartz and/or calcite through quartz-calcite with small floating clasts of wallrock, to brecciated wallrock veined and cemented by quartz-calcite* (Figures 7.4 and 7.5). *The hangingwall of the vein contains scattered thin quartz-calcite veins and breccia veins over many ten's* (sic) *of feet. Quartz-calcite veining may occur either as thin planar veins (often quartz veins with calcite cores), irregular veins with sinuous borders, or highly irregular breccia infillings*.'

7.2.4 Gold-Silver Mineralization

7.2.4.1 Mineralogy

The gold-silver mineralization of interest:

- is associated with the quartz-calcite veins and stockworks described above;
- extends from surface to at least 370 m below surface (highest outcrop to lowest drill intersection), within a boiling zone defined by the bladed textures in quartz pseudomorphs after calcite (the upper levels of the paleo-hydrothermal system have been removed by erosion); and
- is predominantly in the form of native gold and silver-rich native gold (or electrum, a naturally occurring alloy of gold and silver with Au:Ag ratios varying between approximately 80:20 and 20:80); although
- very fine grained, minor and grey to black sulphides (probably acanthite, a silver sulphide), may be present in very thin grey bands, known as ginguro banding, in unoxidized or weakly oxidized parts.

Preliminary petrography identified native gold and acanthite in four out of six sections studied (Hudson, 2011), although the identification of acanthite was tentative due to the very small grain size (three to 100 microns, or 0.003 mm to 0.1 mm). In addition, microscopic analysis showed that:

- minor pyrite replacing mafic phenocrysts is developed in the Moss porphyry, which replacement is related to early and weak chlorite-clay-(calcite-pyrite) alteration (see below);
- minor pyrite also occurs in early-stage grey quartz veins that are not related to the gold-silver mineralized Moss Vein and West Vein and their associated stockworks; and
- sparse sulphides only are contained within the Moss Vein and West Vein and their associated stockworks, the minor pyrite fraction of which is developed separately from the gold-silver mineralization of interest and which is typically oxidized to jarosite or goethite pseudomorphs.

Figure 7.4: Quartz Vein Texture of Bladed Quartz, Moss Mine Project Area
(copied from a project report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)

Figure 7.5: Brecciated Quartz Vein with Clasts of Wallrock, Moss Mine Project Area
(copied from a project report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)

7.2.4.2 Paragenetic Sequence

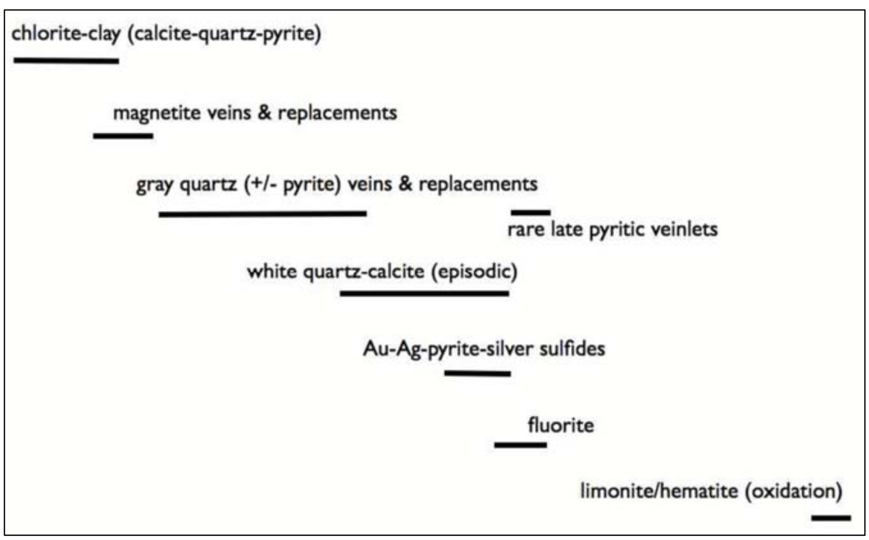
Figure 7.6 summarizes a preliminary paragenetic sequence of alteration and mineralization, based on drillcore logging (Cuffney, 2013). The following details apply:

- the early-phase, weak chlorite-clay-(calcite-pyrite) alteration is a known, district-scale feature;
- magnetite, occurring as aggregates of grains replacing feldspars (and possibly mafics) and rarely as black veinlets, is a product of early-stage alteration –

- unaltered quartz monzonite has fairly strong magnetism due to abundant small grains of disseminated magnetite; however
- magnetite was to a large extent destroyed during early silicification, quartz-calcite veining and ultimately oxidation, thereby yielding zones of low magnetism;

- the early/pre-mineralization silification phase, which is very fine grained, varies between grey quartz veinlets, replacement silicification, microbreccias and breccias;
- the early silicification phase is cut by later stage, coarse grained, light grey to white quartz-calcite veins and veinlets containing the gold-silver mineralization of interest;
- adularia (a feldspar mineral found in low temperature hydrothermal/epithermal deposits) is reported and bands of chalcedony (a cryptocrystalline form of silica, comprising quartz and moganite) may occur, but are rare;
- massive to vuggy, coarse calcite commonly forms a late stage vein infill; and
- late-stage fluorite also occurs, following which the deposit was oxidized, as evidenced by the presence of limonite (an iron oxide, see Sub-Section 7.2.5)

Figure 7.6: A Preliminary Paragenesis for the Moss Deposit
(copied from an internal Company report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)



7.2.4.3 Deportment

Hudson (2011) reports that the native gold, electrum and acanthite are often intergrown, although some gold grains are isolated. Hudson (2011) also reports that the gold-silver mineralization typically occurs as inclusions in calcite grains; only one section were a few micron-sized gold grains seen to be encapsulated in quartz. However, subsequent analysis of quartz-calcite ratios and gold grade shows that the higher grades tend to follow quartz.

Figures 7.6 and 7.7 are photomicrographs of acanthite and gold contained in Moss Vein mineralized material. Pyrite, that is usually partially oxidized, occurs within calcite, interstitial to other grains, as well as in quartz. Traces only of pyrite (or goethite and/or hematite) are present in vein material. Chlorite/illite is also present in vein material, but it does not appear to contain opaque minerals.

Figure 7.7: Photomicrograph of AR-69C (85.50 m to 87.17 m) Showing Grey Acanthite with Traces of Intergrown Gold (right) within Grains of Granular Calcite (low relief) Infilling Quartz (high relief)

(copied from a project report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)

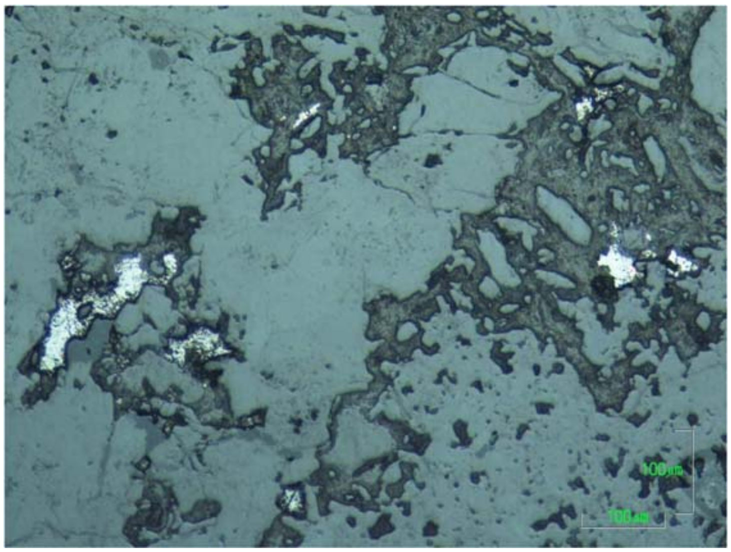
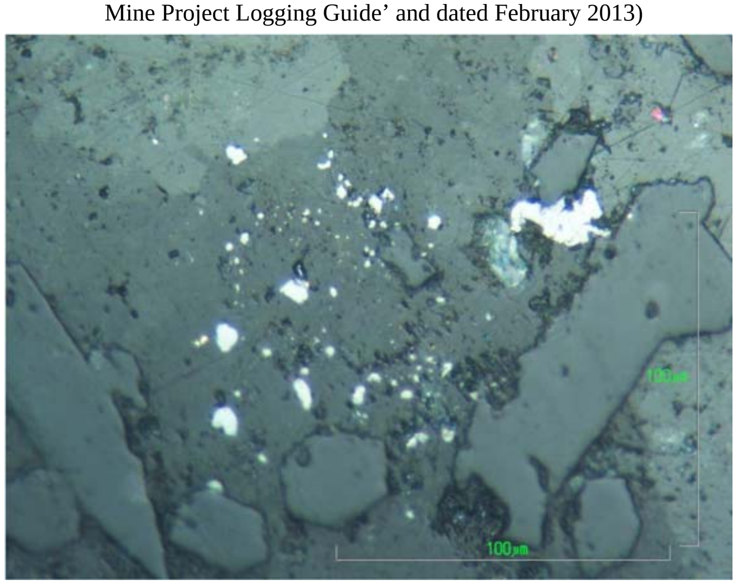


Figure 7.8: Photomicrograph of AR-69C (89.00 to 89.46 m) Showing Many Grains of Grey Acanthite and a Grain of Bright Gold in Calcite (low relief) with Intergrown Quartz Laths from the Recrystallized Acicular Calcite Band

(copied from an internal Company report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)



7.2.4.4 Grain Size

Hudson (2011) notes that grains of native gold (and presumably electrum) vary in the one to 20 micron range (0.001 mm to 0.02 mm). Hudson's finding broadly agrees with that of Baum & Lherbier (1990) who identified, from the results of microscopic gold particle size analysis on Moss Vein samples, the approximate gold/electrum grain diameters summarized on Table 7.1. This data shows that between 60% and 90% of the gold grains are less than 50 microns (or 0.05 mm) in diameter.

Table 7.1: A Summary of Microscopic Gold Particle Size Analysis, Moss Vein Material
(compiled from information contained in Baum & Lherbier (1990)

Grain Size		**Percent of Gold Grains in Sample**	
Microns	**Millimetres**	**444-1-2**	**444-3**
< 5	< 0.005	60%	21%
5 – 20	0.005 – 0.02	21%	15%
20 – 50	0.02 – 0.05	10%	24%
50 – 100	0.05 – 0.1	7%	22%
>100	>0.1	2%	18%
Total	-	*100%*	*100%*

7.2.5 Oxidation

The paragenetic sequence described above establishes that the gold-silver mineralization of interest is late-stage and that it is not associated with any alteration episodes that might otherwise affect the metallurgical response of the target mineralization. The gold-silver mineralization phase does, however, predate the limonite/haematite (oxidation) phase, with the result that its metallurgical response could, in theory at least, be affected by the presence or lack of selective oxidation of the acanthite (silver sulphide) fraction. The minor pyrite fraction of the mineralized veins is not considered relevant for the reasons earlier described: it does not contain gold-silver mineralization that is instead (with the exception of the acanthite fraction) in the form of native gold and electrum that is not susceptible to oxidation effects.

Figures 7.9 and 7.10 are examples of the evidence of selective, late-stage rockmass oxidation found in diamond drillholes from across the Moss deposit: limonite staining may be seen along joint planes, the presence of which suggests the passage of oxygenated (hence oxidizing) groundwater (limonite is an iron-rich oxide).

7.2.5.1 Preliminary Findings

Hudson (2011) states that '*the depth of oxidation can be in excess of 91 m to 152 m (300 to 500 feet)*'. A similar finding is detailed in a mining report by geologist M. C. Godbe III to BF Minerals (April 26, 1982) who states that: '*The Moss Mine was developed over a vertical range from surface to the 300 level. All* (of the mined mineralized material was) *within the oxidized zone. The recently concluded drilling shows oxidation phenomenon well below the present water table (140 feet below the shaft collar), to at least 500 feet below the present surface.*' Hudson (2011) goes on to state that '*BF Minerals deepened the Allen Shaft to the 300 foot level and trucked* (mineralized material) *from the 300 level to the Tyro Mill'*.

Figure 7.9: An Example of Quartz Vein with Black Argentite Cutting Monzonite Porphyry and Showing Typical Limonite Staining (oxidation) along a Joint Plane (drillhole AR-69C, 70.41 m)
(copied from a project report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)

Figure 7.10: An Example Quartz Vein Material with Bladed Texture from Calcite Replacement and Showing Limonite Staining (oxidation) along a Joint Plane (drillhole AR-69C, 80.77 m)
(copied from a project report by Bob Cuffney entitled 'Moss Mine Project Logging Guide' and dated February 2013)

The Company's Moss Mine Project Core Logging Guide (February 2013) states that: '*The REDOX zone at Moss is not a simple boundary and is not related to the present static water table*' and '*It is not uncommon for the vein to be oxidized to depths in excess of 500 ft (152 m), with unoxidized and thin, partially oxidized zones in the hangingwall.'* Cuffney (2013) states that '*The drillholes show that the water level is between 12.2 m and 45.7 m (40 to 150 feet) below surface. There is ample evidence of oxidized rock below the water level in several of the core holes. The fact that oxidation is deeper than the present water table is interpreted to indicate that oxidation is related to a lower water table in the past, and that the water table has risen to its present level after oxidation took place*'.

7.2.5.2 Three-Dimensional Modelling

The efficacy of the preliminary findings outlined was tested by analyzing oxidation data from 151 drillholes along the trend of the Moss deposit: a total of 1,324 intervals of various lengths were logged for limonite content (0% to 5%), which intervals extended to depths of approximately 210 m below the measured watertable. The elevation of the watertable was measured in more than 100 drillholes and wells, by means of piezometers.

Figure 7.11 summarizes the elevation of the measured watertable, clipped to the positions of known (i.e. surface mapped and drillhole interpolated) significant faults. The modelled mineralization is as defined by the 2014 MRM: vein material is identified in RED; hangingwall stockwork in YELLOW, and footwall stockwork in GREEN. The BRIGHT GREEN spheres mark the surface positions of the drillholes in which the surface watertable was measured.

Figure 7.12 shows the portions of the mineralized zones that are above the static watertable only, as well as the collar positions and traces of the 151 drillholes in which oxidation data was recorded. It may be seen that oxidation (as defined by the presence of limonite) extends well below the surface watertable. This is confirmed by Figure 7.13 that shows the mineralized veins below the surface watertable and the distribution of oxidation within and around the same. This clearly demonstrates that:

- oxidation (as evidenced by the presence of limonite) is developed to depths that are significantly greater than the present (surface) watertable;
- oxidation must be related to a lower watertable that existed in the past; and
- the current watertable rose to its current elevation after oxidation took place.

Figure 7.11: An Oblique Vulcan® Snapshot View (looking northwest) of the Moss Deposit, Highlighting the Surface Watertable (clipped to dominant faults) and the Positions of the Drillholes in which the Elevation of the Watertable was Measured

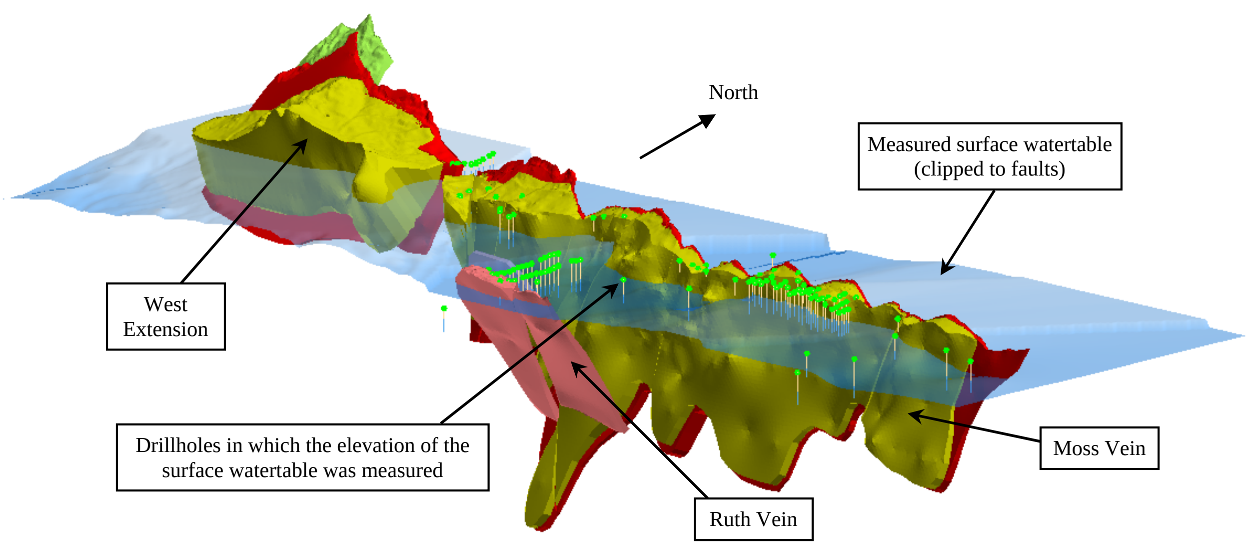


Figure 7.12: An Oblique Vulcan® Snapshot View (looking approximately northwest) of the Moss Deposit Above the Surface Watertable, Highlighting the Position of the Surface Watertable and the Extent of Oxidation (limonite) Below the Surface Watertable

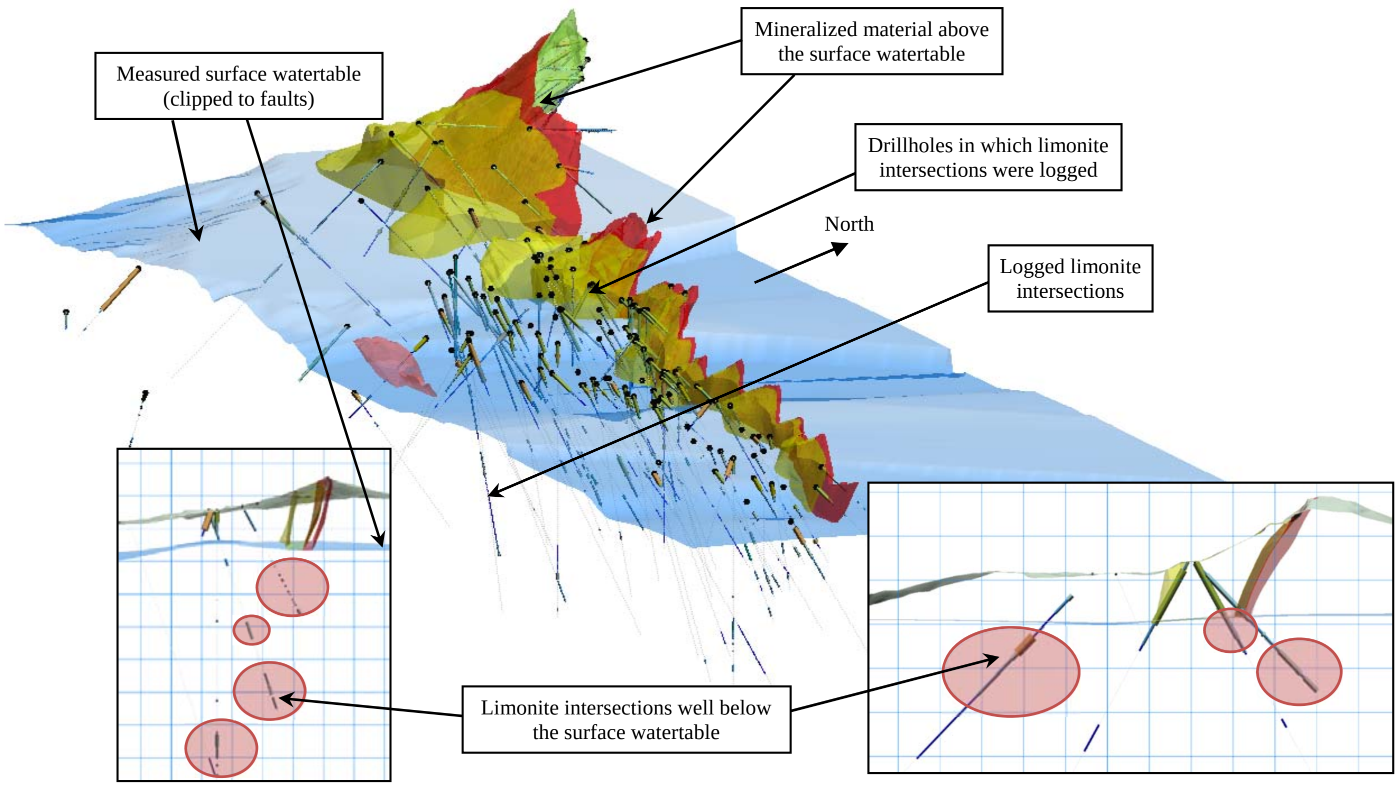
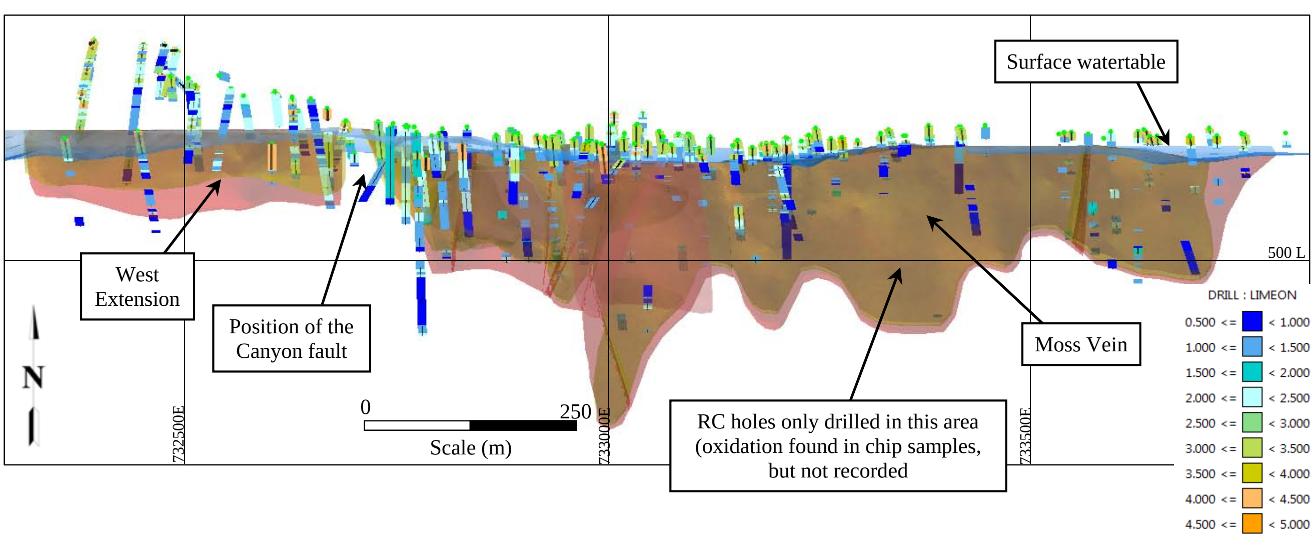


Figure 7.13: A Long-Section Vulcan® Snapshot (looking north) of the Moss Deposit Below the Surface Watertable, Highlighting the Amount and Extent of Logged Oxidation (limonite) Below the Surface Watertable



7.2.6 Faulting

The footwall contact of the Moss Vein is a readily identifiable and persistent shear that dips at an average of 70º to the south. A total of 27 faults that cut across the Moss Vein have been identified by mapping (Figure 7.14), which faults were numbered 1 to 27 from west to east. The faults' strikes and dips were defined by structural mapping. A relative chronology was compiled based on surface topology and their interactions with adjoining intersecting faults. No faults have been identified in the area of West Extension.

The Canyon fault is the most prominent structure that separates the Moss Vein area from West Extension. The Canyon fault appears to displace the Moss Vein and West Extension by a very small amount. However, regional geology plans for the general area show it to be a dominant structure and local drilling suggests that groundwater is not preferentially accumulated within the fault zone. The Canyon fault might, therefore, be a relative compression structure of the strike-slip structural type.

The regional dominance of the Canyon Fault suggests that it might have a large lateral displacement. If this is the case, West Vein and its associated stockworks are likely to be fault-displaced features that are not directly related to the Moss Vein and its associated stockworks. In other words their closely contiguous location, leading to the interpretation that West Extension is an extension of the Moss Vein, might only be a geological coincidence (which possibility is also suggested by the dissimilarity of the mineralized grades – mineralized material from West Extension is consistently lower grade than Moss Vein mineralized material). Whatever the case, the similarity of mineralization and deposit type suggest that the Moss Vein and its associated stockworks are genetically of the same mineralization phase as West Vein and its associated stockworks.

Field data shows that 24 of the mapped faults have dips that are equal to or greater than 80º (the exceptions are Fault 3 that dips at 50º, Fault 12 that dips at 65º and Fault 24 that dips at 40º). All the faults, except the Canyon Fault and the four faults that trend a few degrees east of north/west of south, displace the Moss Vein by small amounts in the left-lateral direction.

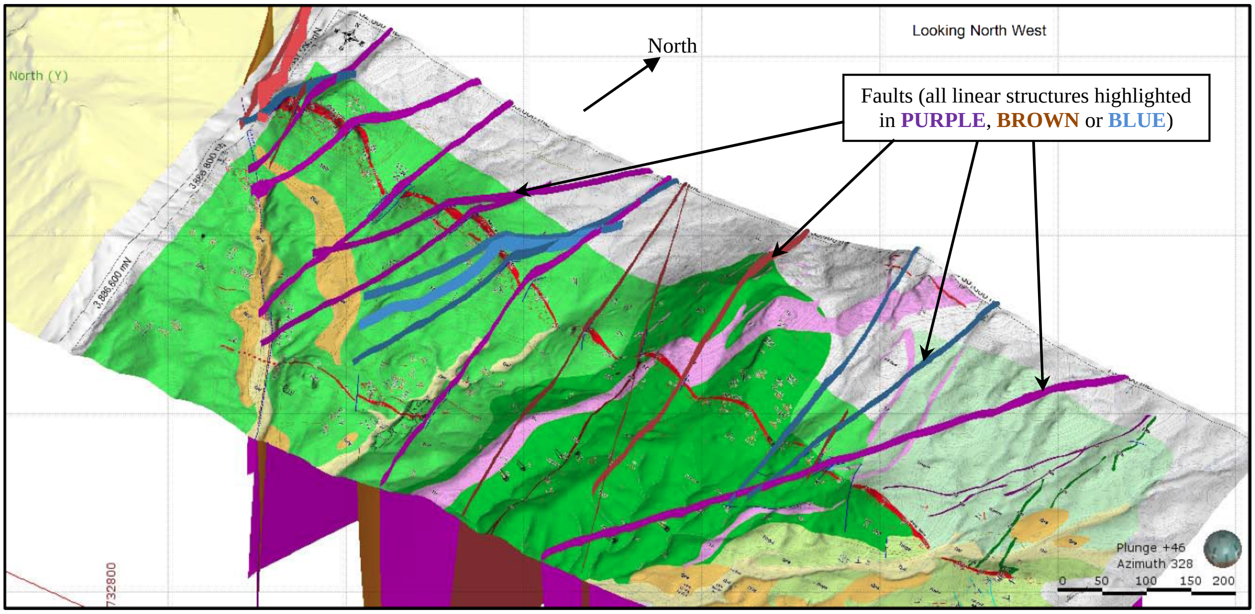
7.2.7 Dykes

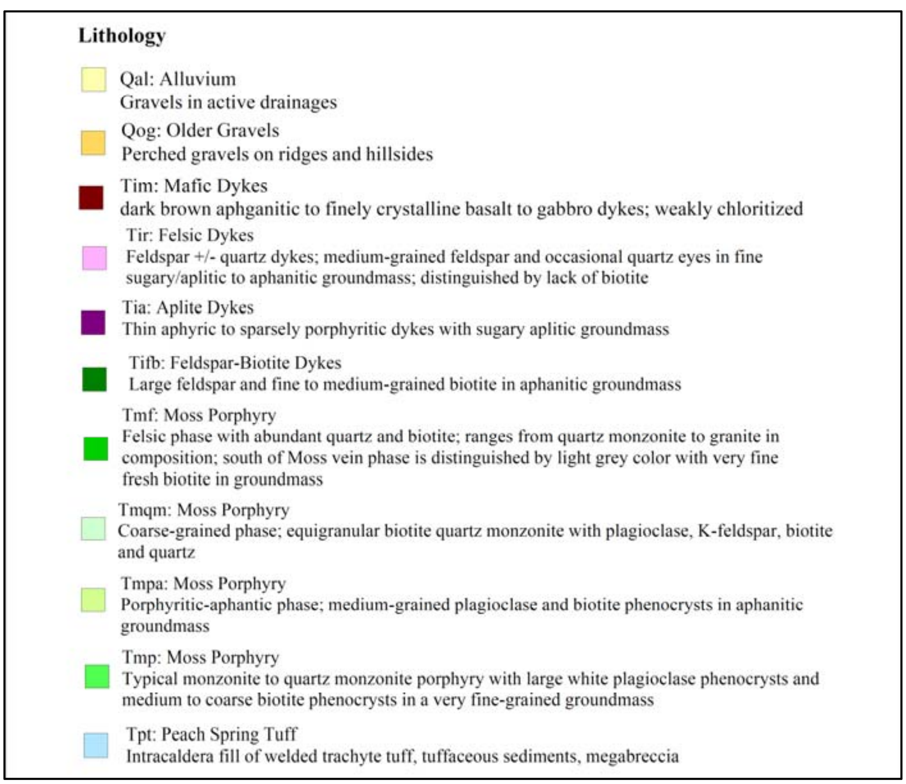
Four different types of dyke have been identified by surface mapping:

- mafic dykes (dark brown, aphganitic to finely crystalline basalt to gabbro that are weakly chloritized);
- feldspar dykes with minor quartz (medium grained feldspar with occasional quartz in a fine grained, sugary/aplitic to aphanitic groundmass);
- aplite dykes (thin aphyric to sparsely porphyritic with a sugary/aplitic groundmass); and
- feldspar-biotite dykes (large feldspar and fine- to medium-grained biotite in an aphanitic groundmass).

Surface mapping shows that the dykes may have been developed along faults or that there has been subsequent offset movement along either the hangingwalls or footwalls of the dykes. They all predate the Moss Vein, as evidenced by surface mapping and the development of Moss Vein-related stockworks within each dyke mass. No dykes have been identified in the area of West Extension because the dykes predate the Moss quartz monzonite that hosts West Extension and the western portion of the Moss Vein.

Figure 7.14: An Oblique Snapshot View (looking approximately northwest) of the Mapped Surface Geology, Draped on the Surface Topography, Highlighting the Positions and Trends of the 27 Mapped Faults
(copied from Brownlee [2014])





7.3 Qualified Persons' Opinion

In the opinion of the Qualified Persons for this section of this Technical Report (Dr. David Stone, P. Eng. and Mr. Daniel Kilby, P. Eng.), the geology of the Moss deposit is straightforward and amenable to exploitation through openpit mining. The deposit itself appears to be a conventional oxide type and it is not necessary to differentiate between mineralized material located above and below the present watertable:

- the economic minerals of interest are native gold and electrum, which are not susceptible to surface weathering effects, as well as acanthite;
- apart from acanthite, the presence of sulphides is limited to minor to very minor pyrite that either pre-dates the gold-silver mineralization phase or is contemporaneous but not intimately associated with the gold-silver mineralization of interest; therefore
- with the exception of the acanthite fraction, gold and silver (from electrum) recovery is not constrained by considerations of encapsulating sulphides or oxides.

The paragenetic model described in Sub-Section 7.2.4 emphasizes the separate and late-phase nature of the target gold-silver mineralization, the characteristics of which suggest that it should be very amenable to cyanide leaching. The deportment characteristics of the target mineralization further suggest that metal recovery through cyanidation is probably proportional to the size of the mineralized material subjected to leaching: the finer the material the greater the amount of gold-silver mineralization that would be released so the greater would be the overall metallurgical recovery. Cyanide solution penetration through micro-cracks and other flaws in the gangue minerals would nevertheless play a contributing role, although the amount and rate of cyanidation would inevitably depend on the surrounding thickness of encapsulating material. With this in mind, it might reasonably be expected that low gold and silver recovery rates would probably be realized from coarse-crushed mineralized material with small amounts of associated fines.

Evidence for the likely amenability of the target gold-silver mineralization to rapid cyanidation is further provided by its measured grain size. The importance of this key physical characteristic is in the surface area to volume ratio ("SA:V") of the grains which, by definition, is very high. Physical chemistry shows that SA:V is inversely proportional to size and that materials with high ratios are much more reactive than their coarser counterparts. In the case of native gold and electrum, this means that they can be expected to be rapidly adsorbed when exposed to a sodium cyanide solution.

8 DEPOSIT TYPE

The Moss deposit may be characterized as a brecciated and steeply dipping (average 70˚) quartz-calcite vein and stockwork system which extends over a strike length of approximately 1,400 m. It is of the low sulphidation (adularia-sericite) epithermal vein type, which is described by Henley & Ellis (1983) and Heald et al. (1987). Epithermal gold-silver deposits form in the near-surface environment from hydrothermal systems typically within 1.5 km of the earth’s surface (Taylor, 2007). They are commonly found associated with centres of magmatism and volcanism, but they can also form in shallow marine settings. Hot spring deposits and both liquid- and vapour-dominated geothermal systems are commonly associated with epithermal deposits.

Epithermal deposits comprise one of three sub-types: high sulphidation; intermediate sulphidation; and low sulphidation. Each sub-type is identified by its characteristic alteration mineral assemblages, occurrences, textures and, in some cases, characteristic suites of associated geochemical elements (for example, mercury, antimony, arsenic and thallium). Copper, lead, zinc and other sulphide minerals may also occur in addition to pyrite, native gold and electrum. In some epithermal deposits, notably those of the intermediate-sulphidation sub-type, base metal sulphides may comprise a significant proportion of the mineralization assemblage.

The quartz vein textures (massive, breccia, vuggy, bladed quartz replacing calcite, colloform banding and ginguro banding), adularia and the very low sulphide content of the Moss deposit are typical of low sulphidation epithermal veins. Gold is native and silver typically occurs as acanthite or combined with gold in electrum. Copper is present, but in very minor quantities (see also Section 13.9).

The platey or bladed calcite characteristics of the Moss deposit is indicative of the boiling zone of the hydrothermal fluid, which calcite is commonly replaced by quartz. Adularia (a low temperature variety of orthoclase) is also indicative of the boiling zone in which gold is deposited out of solution. No paleosurface or shallow features, such as silica sinters, chalcedony or a steam-heated acid leach cap, are preserved in the Moss deposit. This indicates that the top of the hydrothermal system has been eroded, thereby exposing the gold depositional zone.

John (2001) described the Miocene and early Pliocene low sulphidation epithermal gold-silver deposits of northern Nevada as related to a potassium-rich, tholeiite series, bimodal basalt-rhyolite magmatic assemblage formed during continental rifting. These deposits include the Midas (Ken Snyder), Sleeper, DeLamar, Mule Canyon, Buckhorn, National, Hog Ranch, Ivanhoe and Jarbridge districts. Sillitoe (2002) described the association of low sulphidation gold-silver deposits with rifting and bimodal volcanism in northern Nevada, northern Chile, Patagonia and Japan. In contrast, low sulphidation mineralization of the Moss deposit is hosted by an alkalic to sub-alkalic shoshonitic volcanic centre.

9 EXPLORATION

9.1 Previous Owners and Operators (1982 to 2009)

Exploration work by previous owners and operators of and on the Moss Mine property is summarized in Section 6.2. This includes work carried out in 1982 by BF Minerals through programs by Billiton Minerals in 1990, Magma Copper, Golconda Resources, Addwest and finally by Patriot Gold whose last exploration program was in 2009. The nature and disposition of the Moss deposit is such that in each case the main focus of the exploration work was on drilling, underground channel sampling and the development of geology maps for the Moss Mine Property area.

No stream-sampling, soil-sampling or geophysical work appears to have been carried out by previous owners and operators, to demonstrate the possibility of additional mineralization on the Moss Mine Project area. This, in the opinion of the Qualified Person for this section of this Technical Report (Mr. Daniel Kilby, P. Eng.), is unsurprising because until MinQuest staked the Moss series of 104 unpatented lode claims in 2004 and 2009 (Sub-Section 4.2.5), the focus had been solely on the patented ground, hence on the Moss Vein and its associated stockworks. Details of the drilling programs carried out by previous owners and operators are presented in Section 10.1.

9.2 The Company (2011 through 2014)

9.2.1 2011 Exploration Program

The main focus of the Company's 2011 exploration program was the Phase One infill and confirmation drilling program described in Section 10.2. A limited surface sampling program was, however, carried out to test for extensions to the Moss Vein. The results are presented in the Company's news release dated May 10, 2011, but they are not repeated here: in the opinion of the Qualified Person for this section of this Technical Report (Mr. Daniel Kilby, P. Eng.), while surface sampling provides insight into the presence or lack of mineralized material, the grades are not necessarily indicative of the mineralization as a whole and cannot, therefore, be relied upon.

9.2.2 2012 Exploration Program

In 2012, the Company's 2012 exploration effort on the Moss Mine Property was again focused on drilling (the Phase Two program described in Section 10.3). However, the Company also carried out a channel sampling program at 1.52 m (5 ft) intervals across the backs/inverts/crowns of the accessible drifts and crosscuts of the historical underground workings in the vicinity of Allen Shaft (see Section 6.2). A total of 207, 1.52 m (5ft) long samples were taken by hammer and chisel. The sample series is numbered UG2012-01 to UG2012-207.

The channel sample data supplements that compiled by previous owners and operators of the Moss Mine Property, which earlier data totals 109 channel samples in series UG65-1 to UG65-41, UG220-01 to UG220-46, UG300-1 to UG300-3 and UG98-1 to UG98-20. All the listed channel samples were entered in the Moss Mine Project drillhole assay database, as notional short holes for use in Mineral Resource estimation.

Table 9.1 summarizes the significant intersections of the UG2012-1 to UG2012-207 series of channel samples, as reported by the Company in news releases dated June 26, July 19 and August 16, 2012. Figure 9.1 details the locations of the Company's underground channel samples that are identified only by their sample number. The full identification number for each channel sample may be defined by adding UG2012 before the stated number.

Table 9.1: A Summary of Significant Intersections, the Company's 2012 Underground Channel Sampling Program, Moss Mine Project
(compiled from information contained in Company news releases)

Sampling Area	Sample Interval		Length	Assay Grades (g/t)	
	From	To	(m)	Au	Ag
Office Crosscut – 60 level	48.77	89.92	41.15	1.61	8.3
incl.	62.48	85.34	22.86	2.38	11.8
Main Drift West – 60 Level	1.52	92.96	91.44	2.26	14.9
incl.	3.05	12.19	9.14	4.48	23.3
Main Drift East – 60 Level	1.52	7.62	6.10	4.73	35.1
South Crosscut off Main Drift 30' W	1.52	7.62	6.10	1.83	6.4
North Crosscut off Main Drift 40' W	1.52	15.24	13.72	1.64	11.2
incl.	1.52	6.10	4.57	2.28	9.0
incl.	9.14	10.67	1.52	5.29	29.4
North Crosscut off Main Drift Station 60' W	1.52	9.14	7.62	0.98	12.1
North Crosscut off Main Drift Station 150' W	1.52	13.72	12.19	1.49	9.5
incl.	6.10	7.62	1.52	3.69	29.7
North Crosscut off Main Drift Station 200' W	9.14	10.67	1.52	1.88	10.3
Sub-Drift East from Office Crosscut at Station 260' N	1.52	6.10	4.57	2.40	15.5
Sub-Drift East from Office Crosscut at Station 275' N	1.52	4.57	3.05	1.24	6.0
1921 Hill #2 Crosscut	7.62	32.00	24.38	4.42	20.4
incl.	13.72	22.86	9.14	9.72	44.4

9.2.3 Qualified Person's Opinion

In the opinion of the Qualified Person for this section of this Technical Report (Mr. Daniel Kilby, P. Eng.) no factors, which could result in sample bias, may readily be identified in the channel sampling procedure or assay outcomes (the Company's channel samples were assayed at Inspectorate America Corporation's laboratory located at Sparks, Nevada). However, other than a very minor gradient designed to facilitate water egress, the sampled historical underground excavations are horizontal and they are at various different orientations to the Moss Vein. The sample intervals stated on Table 9.1 do not, therefore, reflect in any way the true thickness of the intersected mineralization.

9.2.4 2013/2014 Exploration Program

In addition to the Phase Three drilling program described in Section 10.4, the Company carried out an airborne magnetic survey described in a consultancy report to the Company by Precision GeoSurveys, Inc. of Vancouver, B.C., ("Precision GeoSurveys") that is entitled 'Moss Gold-Silver Survey Block' and dated June 2013. Figure 9.2 provides a summary of the results of the airborne magnetic survey and its interpretation, by Precision GeoSurveys.

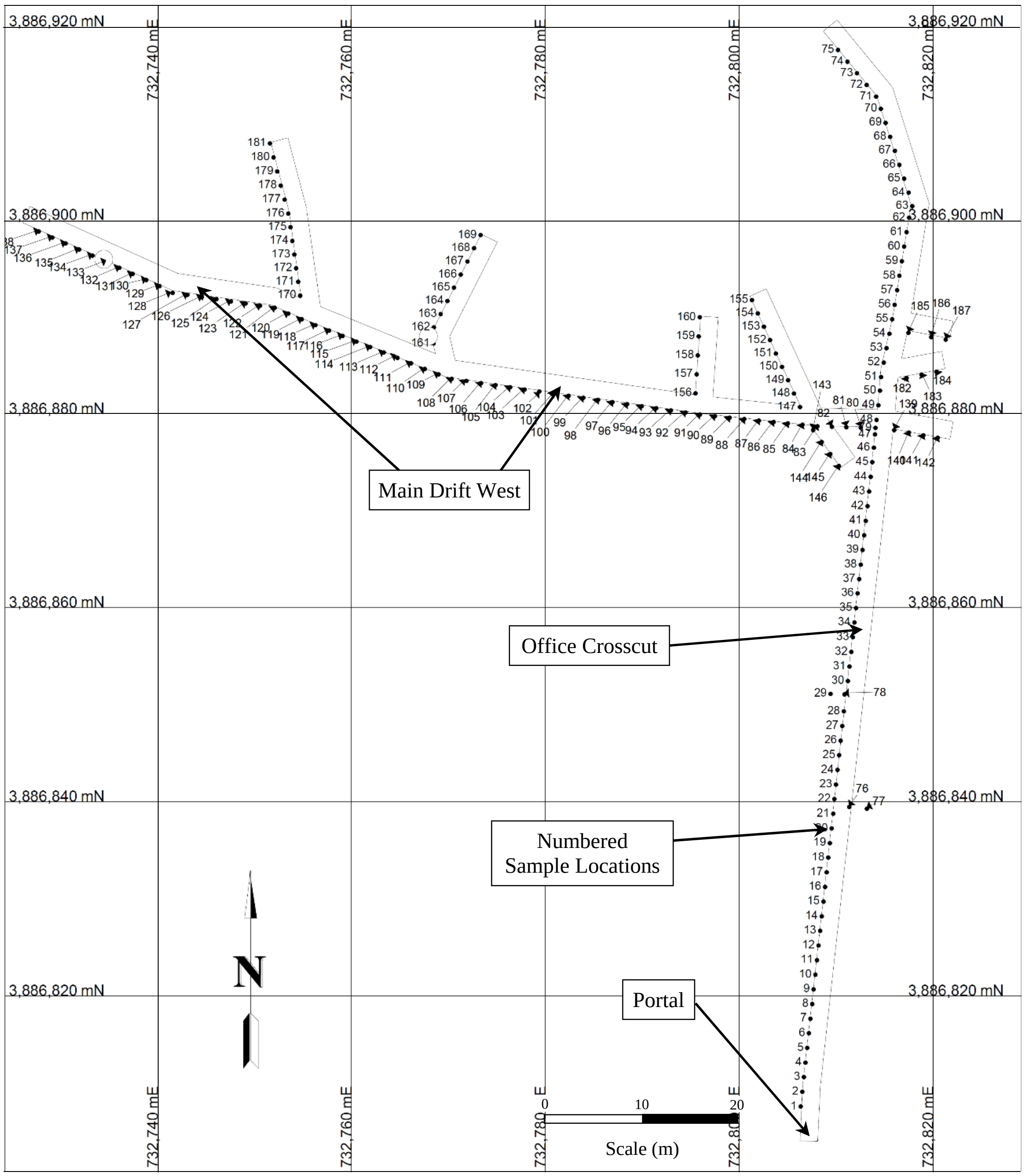
The results show that magnetics are an effective method of identifying potential mineralized structures on the Moss Mine Project area - both magnetic highs and lows correspond with known mineralized structures:

- the Moss deposit lies along a well-defined magnetic high that suggests that is approximately three kilometres of unexplored potential on the one structure;

- including the structure related to the Moss deposit, there are a total of nine linear magnetic anomalies, totalling approximately 21 km of potential strike length, associated with either known mineral occurrences or historic workings (one such structure includes nearly six kilometres of the mapped extension of the structure hosting the regionally famous Gold Road deposit; and
- several other linear magnetic lows and highs occur across the Moss Mine Project area that require ground work to determine if they are mineralized (Figure 9.4).

Figure 9.1: A Location Plan for the Company's 2012 Underground Channel Samples, 60 Level, Historical Mine Workings, Moss Mine Project

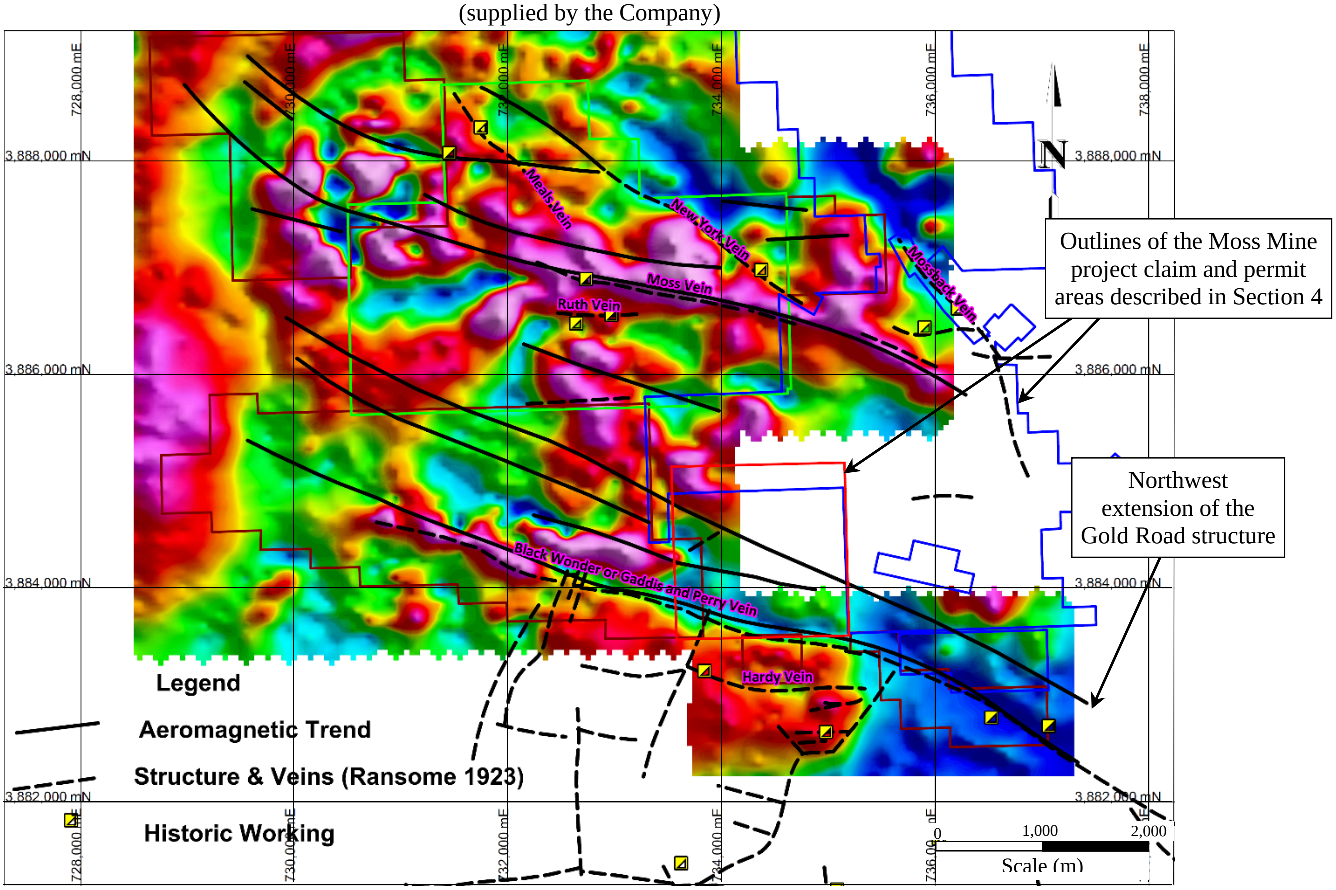
(compiled using the AutoCad® claims files files and channel sample database supplied by the Company)



It was in consequence of the magnetic survey results outlined that the Company subsequently started a geological mapping and sampling program to '*identify and prioritize areas for future drilling where new resources may be discovered*' (see the Company news release dated September 04, 2014). The Company's target areas include:

- 1,500 m of under-explored Moss Vein structure outside the current resource limits;
- nearly six kilometres of unexplored extension of the Gold Road structure that hosts the Gold Road mine (reported by Durning and Buchanan [1984] to have produced 484,000 oz Au);
- numerous historical workings along known but unexplored veins, including Rattan Vein to the south of the Moss Vein where the single RC hole drilled by the Company in 2012 (AR-136R) intersected 1.52 m (5 ft) of mineralized material grading 13.072 g/t Au and 67.0 g/t Ag;
- the intersection of the Gold Road and Eastern United structures (Durning and Buchanan [1984] reported that Gold Road produced 484,000 oz Au at 10 g/t Au and United Eastern 769,000 oz Au at 35.76 g/t Au); and
- five previously identified target areas of the Silver Creek claims.

Figure 9.2: A Plan of the General Moss Mine Project Area Showing the Overall Claim Area, The Locations of Known Historical Workings, Magnetic Intensity and the Related Structures
(supplied by the Company)



10 DRILLING

10.1 Previous Owners and Operators (1982 to 2009)

Table 10.1 summarizes the details of the 221 holes (16,706.75 m) completed by previous owners of the Moss Mine Property. The list identifies only those holes for which the collar co-ordinates are known and have been verified. The LH98-1 to LH98-15 series of holes completed by Addwest in 1998 were drilled as up-holes in the historical underground workings. In each case the holes were drilled to explore the Moss Vein, based on knowledge of its attitude and extent from field mapping and related geological field work.

Table 10.1: A Summary of the Holes Drilled by Previous Owners for which the Collar Positions are Known, Moss Mine Property
(compiled from information supplied by the Company)

Company	Year	Type	Number	Total Metres	Average Depth (m)	Drillhole Series	
						From	To
BF Minerals	1982	Air Trac	54	1,438.66	26.6	M-1-30	M-25-60
		RC	3	356.62	118.9	M-27-68	M-29-60
Billiton Minerals	1990	RC	21	2,110.74	100.5	MM-1	MM-21
Magma Copper	1991	RC	21	3,014.47	143.5	MC-1	MC-21
Golconda Resources	1993	RC	14	822.35	58.7	MR-1	MR-14
		RC	3	143.29	47.8	BX-4	BX-6
Addwest Minerals	1996	RC	30	2,504.54	83.5	M96-1	M96-30
	1996	Core RX	6	508.10	84.7	MC96-1	MC96-6
	1998	Longhole	14	122.53	8.8	LH98-1	LH98-15
Patriot Gold	2004 to 2005	RC	43	3,598.78	83.7	AR-01	AR-44R
	2007, 2009	Diamond Drillholes	12	2,086.66	173.9	AR-45C	AR-56C
		Totals	***221***	***16,706.75***			

10.1.1 Collar Locations

Significant effort was expended by the Company in 2011 to verify the collar positions of all the holes drilled by previous owners of the Moss Mine Property. Individual collars were located in the field and surveyed using a differential GPS instrument. If the collars could not be located the originally reported co-ordinates were used.

As part of the verification process, all the collar coordinates were converted from UTM NAD27 Zone 11 metres to Arizona State Plane West NAD27 (0203) US Feet, using Corpscon Version 6 (Brownlee, 2014). A comparison of collar positions was then made. Good overall correspondence was found: variations of less than 1.52 m and 3.05 m were found for eastings and northings, respectively. Small groups were found where the difference in eastings was 9.14 m and the difference in northings was up to 12.19 m. In the opinion of the Qualified Person for this sub-section of this Technical Report (Mr. Daniel Kilby, P. Eng.) this probably reflects the method used to position the collars (i.e. from topographic maps rather than by survey).

10.1.2 Downhole Surveys

No records of any downhole surveys in holes drilled by previous owners appear to exist. In the opinion of the Qualified Person for this sub-section of this Technical Report (Mr. Douglas Brownlee, P. Geo.), this does not represent a data limiting constraint: most of the holes are short or even very short (Table 10.1) and, based on the analysis of drillhole deviation presented in Sub-Section 10.3.2, any downhole deviation is likely to be limited to a few degrees. Deviation of the holes drilled by previous owners of the Moss Mine Property does not, therefore, impact the veracity of the Company's drillhole database.

10.1.3 Drillhole Data

Table 10.2 (which is in four parts due to its overall length) summarizes the collar locations, hole lengths, azimuths and inclinations of each of the holes completed by previous owners and operators of the Moss Mine Property for which the collar positions are known. Figure 10.1 is a colour-coded collar location plan for the listed holes on which the drillhole traces are indicated.

Table 10.2: A Summary of the Collar Co-ordinates, Hole Lengths, Azimuths and Inclinations of the Drillholes Completed by Previous Owners for which the Collar Co-ordinates are Known, Moss Mine Property

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Hole Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
M-1-60	732,833.81	3,886,868.05	669.65	33.53	349.0	-60º
M-2-30	732,852.64	3,886,870.50	672.08	24.38	346.0	-30º
M-2-60	732,852.64	3,886,870.50	672.08	24.38	346.0	-60º
M-3-30	732,888.53	3,886,873.80	671.17	21.34	334.0	-30º
M-3-60	732,888.53	3,886,873.80	670.56	27.43	334.0	-60º
M-4-30	732,929.48	3,886,843.72	653.49	27.43	344.0	-30º
M-4-60	732,929.48	3,886,843.72	656.84	27.43	344.0	-60º
M-5-30	732,792.93	3,886,877.70	671.78	21.34	332.0	-30º
M-5-60	732,792.93	3,886,877.70	671.78	24.38	332.0	-60º
M-6-30	732,773.99	3,886,878.30	663.85	18.29	4.0	-30º
M-6-60	732,773.99	3,886,878.30	663.85	22.86	4.0	-60º
M-7-70	732,731.86	3,886,888.82	659.89	30.48	4.0	-70º
M-8-30	732,539.16	3,886,904.50	719.33	24.38	350.0	-30º
M-8-45	732,539.16	3,886,904.50	719.33	27.43	350.0	-45º
M-8-60	732,539.16	3,886,904.50	719.33	30.48	350.0	-60º
M-9-30	732,551.71	3,886,912.53	715.06	21.34	343.0	-30º
M-9-45	732,551.71	3,886,912.53	715.06	24.38	343.0	-45º
M-9-60	732,551.71	3,886,912.53	715.06	27.43	343.0	-60º
M-10-30	732,561.99	3,886,915.61	723.29	21.34	0.0	-30º
M-10-45	732,561.99	3,886,915.61	723.29	18.29	0.0	-45º
M-10-60	732,561.99	3,886,915.61	723.29	21.34	0.0	-60º
M-11-30	732,578.48	3,886,914.93	716.28	21.34	355.0	-30º
M-11-45	732,578.48	3,886,914.93	716.28	22.86	355.0	-45º
M-11-60	732,578.48	3,886,914.93	716.28	24.38	355.0	-60º
M-12-30	732,595.36	3,886,911.82	706.22	27.43	358.0	-30º
M-12-45	732,595.36	3,886,911.82	706.22	30.48	358.0	-45º
M-12-60	732,595.36	3,886,911.82	706.22	33.53	358.0	-60º
M-13-30	732,615.15	3,886,913.08	701.04	21.34	16.0	-30º
M-13-45	732,615.15	3,886,913.08	701.04	19.81	16.0	-45º
M-13-60	732,615.15	3,886,913.08	701.04	24.38	16.0	-60º
M-14-30	732,624.31	3,886,894.16	689.46	12.19	25.0	-30º
M-14-45	732,624.31	3,886,894.16	689.46	9.14	25.0	-45º
M-14-60	732,624.31	3,886,894.16	689.46	33.53	25.0	-60º
M-15-30	732,557.14	3,886,895.93	711.10	28.96	342.0	-30º
M-15-45	732,557.14	3,886,895.93	711.10	27.43	342.0	-45º
M-15-60	732,557.14	3,886,895.93	711.10	30.48	342.0	-60º
M-16-30	732,565.58	3,886,898.95	705.61	30.48	349.0	-30º
M-16-45	732,565.58	3,886,898.95	705.61	33.53	349.0	-45º
M-16-60	732,565.58	3,886,898.95	705.61	36.58	349.0	-60º
M-17-45	732,570.00	3,886,903.67	712.62	33.53	351.0	-45º
M-17-60	732,570.00	3,886,903.67	712.62	36.58	351.0	-60º
M-18-45	732,568.76	3,886,885.63	703.78	36.58	350.0	-45º
M-19-45	732,562.19	3,886,881.45	704.09	36.58	356.0	-45º
M-20-45	732,575.63	3,886,890.13	703.48	36.58	350.0	-45º
M-21-30	732,638.27	3,886,896.14	681.53	21.34	16.0	-30º
M-21-45	732,638.27	3,886,896.14	681.53	18.29	16.0	-45º
M-21-60	732,638.27	3,886,896.14	684.28	21.34	16.0	-60º
M-21-60A	732,636.53	3,886,893.34	684.28	30.48	16.0	-60º
M-22-45	732,653.13	3,886,889.61	674.83	21.34	193.0	-45º
M-23-45	732,654.20	3,886,894.22	674.83	36.58	13.0	-45º
M-24-70	732,679.50	3,886,894.74	671.47	36.58	176.0	-70º
M-25-30	732,810.50	3,886,871.87	671.17	27.43	358.0	-30º
M-25-60	732,810.50	3,886,871.87	671.17	30.48	358.0	-60º
M-26-63	732,869.52	3,886,820.71	654.41	91.44	331.0	-63º
M-27-68	732,863.74	3,886,820.22	654.10	121.92	309.0	-68º
M-28-78	732,893.38	3,886,800.14	651.66	124.97	0.0	-78º
M-29-60	732,577.82	3,886,832.24	690.37	109.73	5.0	-60º

Table 10.2 *continued*: A Summary of the Collar Co-ordinates, Hole Lengths, Azimuths and Inclinations of the Drillholes Completed by Previous Owners for which the Collar Co-ordinates are Known, Moss Mine Property

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
MM-1	732,838.44	3,886,819.70	653.19	109.73	0.0	-70º
MM-2	732,838.44	3,886,819.70	653.19	73.15	0.0	-45º
MM-3	732,849.87	3,886,837.70	652.86	91.44	0.0	-90º
MM-4	732,887.72	3,886,805.14	652.27	94.49	15.0	-65º
MM-5	732,729.73	3,886,860.38	658.37	76.20	320.0	-60º
MM-6	732,734.19	3,886,836.12	662.94	64.01	320.0	-50º
MM-7	732,804.00	3,886,837.00	653.19	100.58	12.0	-60º
MM-8	732,778.78	3,886,834.53	652.27	91.44	12.0	-55º
MM-9	732,571.11	3,886,822.87	691.29	137.16	345.0	-60º
MM-10	732,588.23	3,886,840.51	688.24	135.64	10.0	-60º
MM-11	732,634.81	3,886,862.16	678.18	67.06	5.0	-45º
MM-12	732,672.04	3,886,871.01	670.56	60.96	5.0	-65º
MM-13	732,812.27	3,886,780.41	648.61	153.92	0.0	-65º
MM-14	732,812.27	3,886,780.41	648.61	109.73	15.0	-45º
MM-15	733,357.01	3,886,689.28	649.22	117.35	15.0	-60º
MM-16	733,342.51	3,886,677.27	650.14	91.44	160.0	-65º
MM-17	733,418.25	3,886,692.24	656.56	108.20	15.0	-65º
MM-18	733,505.00	3,886,737.00	656.23	103.63	0.0	-60º
MM-19	733,635.53	3,886,698.37	655.32	108.20	10.0	-60º
MM-20	733,776.74	3,886,652.57	651.12	99.06	15.0	-65º
MM-21	734,332.54	3,886,419.12	676.66	117.35	10.0	-65º
MC-1	732,887.77	3,886,775.55	649.22	140.21	12.0	-65º
MC-2	732,809.04	3,886,785.79	649.83	158.50	2.0	-65º
MC-3	733,086.00	3,886,747.00	638.86	152.40	22.0	-65º
MC-4	733,103.25	3,886,742.70	636.12	146.30	12.0	-65º
MC-5	733,050.61	3,886,749.45	644.65	138.68	10.0	-65º
MC-6	732,,979.47	3,886,751.39	644.96	152.40	10.0	-65º
MC-7	732,921.77	3,886,771.47	647.70	143.26	10.0	-65º
MC-8	733,152.82	3,886,731.55	633.68	135.64	12.0	-65º
MC-9	733,227.74	3,886,705.81	634.59	146.30	12.0	-65º
MC-10	733,287.96	3,886,692.52	648.92	170.69	12.0	-65º
MC-11	732,922.69	3,886,808.72	651.66	91.44	12.0	-65º
MC-12	732,981.59	3,886,789.29	648.61	91.44	12.0	-65º
MC-13	733,117.36	3,886,769.93	639.78	85.34	12.0	-65º
MC-14	733,060.52	3,886,791.56	654.71	91.44	12.0	-65º
MC-15	733,,087.51	3,886,777.80	644.35	91.44	12.0	-65º
MC-16	733,167.53	3,886,756.01	637.34	109.73	12.0	-65º
MC-17	732,821.65	3,886,829.52	655.32	54.86	12.0	-65º
MC-18A	732,329.39	3,886,826.57	665.99	231.65	2.0	-46.5
MC-19	732,333.26	3,886,829.75	667.51	213.36	37.0	-37
MC-20	732,335.68	3,886,830.13	667.51	316.99	67.0	-30
MC-21	733,889.00	3,885,610.00	630.94	152.40	0.0	-90
MR-1	732,810.50	3,886,871.87	671.17	33.53	0.0	-60º
MR-2	732,852.65	3,886,870.19	672.08	42.67	346.0	-60º
MR-3	732,848.13	3,886,868.52	671.17	54.86	298.0	-45º
MR-4	732,818.56	3,886,868.16	669.95	42.67	305.0	-45º
MR-5	732,804.05	3,886,873.18	672.39	64.01	287.0	-60º
MR-6	732,792.61	3,886,877.99	671.78	18.29	332.0	-60º
MR-7	732,792.61	3,886,877.99	671.78	56.08	288.0	-60º
MR-8	733,022.35	3,886,867.49	671.17	51.82	214.0	-45º
MR-9	733,038.10	3,886,871.06	671.47	60.96	150.0	-45º
MR-10	733,147.36	3,886,831.92	656.84	59.13	237.0	-45º
MR-11	732,719.74	3,886,904.90	656.84	30.48	150.0	-45º
MR-12	732,778.45	3,886,835.13	652.27	64.01	12.0	-45º
MR-13	733,304.33	3,886,751.93	646.79	54.86	36.0	-45º
MR-14	733,429.78	3,886,749.62	660.50	45.72	52.0	-45º
BX-4	733,025.91	3,886,870.66	671.78	21.34	235.0	-45º
BX-5	732,752.49	3,886,817.50	662.03	60.96	160.0	-45º
BX-6	732,572.99	3,886,821.41	691.29	60.96	137.0	-45º
M96-1	732,757.92	3,886,875.33	655.93	36.58	0.0	-45º
M96-2	732,758.01	3,886,872.59	660.20	76.20	0.0	-90º
M96-3	732,775.09	3,886,863.39	661.11	91.44	0.0	-90º
M96-4	732,760.38	3,886,846.74	654.71	91.44	0.0	-90º
M96-5	732,776.98	3,886,833.55	655.02	103.63	0.0	-90º
M96-6	732,744.42	3,886,887.10	660.50	54.86	0.0	-90º
M96-7	732,743.71	3,886,871.52	658.37	60.96	0.0	-90º
M96-8	732,727.46	3,886,874.04	656.84	50.29	0.0	-60º
M96-9	732,715.11	3,886,878.51	658.37	25.91	0.0	-45º
M96-10	732,756.56	3,886,823.74	660.50	132.59	0.0	-90º
M96-11	732,746.08	3,886,836.21	662.03	91.44	0.0	-75º
M96-12	732,709.53	3,886,815.80	670.56	117.35	10.0	-73º

Table 10.2 *continued*: A Summary of the Collar Co-ordinates, Hole Lengths, Azimuths and Inclinations of the Drillholes Completed by Previous Owners for which the Collar Co-ordinates are Known, Moss Mine Property

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
M96-13	732,699.36	3,886,856.34	665.38	74.68	0.0	-80º
M96-14	732,810.38	3,886,828.85	654.71	80.77	0.0	-45º
M96-15	732,810.77	3,886,826.42	654.41	129.54	0.0	-80º
M96-16	732,819.71	3,886,809.81	653.73	134.11	0.0	-65º
M96-17	732,836.48	3,886,788.73	652.87	170.69	0.0	-68º
M96-18	732,838.60	3,886,842.89	658.06	62.48	0.0	-45º
M96-19	732,900.12	3,886,836.66	657.15	62.48	0.0	-70º
M96-20	732,901.42	3,886,871.48	669.34	30.48	0.0	-90º
M96-21	732,884.63	3,886,871.84	669.34	36.58	0.0	-90º
M96-22	732,868.48	3,886,871.32	669.04	45.72	0.0	-90º
M96-23	732,868.82	3,886,775.27	651.23	160.02	0.0	-64º
M96-24	732,968.04	3,886,774.51	645.87	117.35	0.0	-60º
M96-25	732,963.67	3,886,822.65	655.63	61.57	0.0	-60º
M96-26	733,024.17	3,886,766.32	643.84	105.16	0.0	-60º
M96-27	733,022.04	3,886,799.02	649.51	71.63	0.0	-45º
M96-28	733,205.18	3,886,742.90	636.42	99.06	0.0	-50º
M96-29	733,185.47	3,886,767.27	638.25	60.96	3.0	-45º
M96-30	733,147.51	3,886,774.33	636.87	68.58	0.0	-50º
MC96-1	732,804.00	3,886,834.00	654.41	95.10	0.0	-63.5º
MC96-2	733,230.88	3,886,752.97	641.48	76.20	0.0	-50º
MC96-3	733,200.00	3,886,738.00	635.20	119.79	0.0	-60º
MC96-4	732,856.17	3,886,827.90	655.63	76.81	0.0	-45º
MC96-5	732,834.00	3,886,854.00	668.43	92.66	0.0	-90º
MC96-6	732,772.65	3,886,863.31	659.89	47.55	0.0	-45º
LH98-1	732,871.40	3,886,892.26	677.02	4.27	4.0	10º
LH98-2	732,875.61	3,886,898.94	682.14	3.66	335.0	8º
LH98-3	732,864.80	3,886,902.86	682.75	9.75	13.0	10º
LH98-4	732,863.57	3,886,903.13	682.75	9.75	348.0	16º
LH98-5	732,811.97	3,886,896.95	683.08	9.75	4.0	18º
LH98-6	732,864.00	3,886,901.20	682.14	0.00	0.0	0º
LH98-7	732,805.43	3,886,902.22	685.59	9.75	12.0	19º
LH98-8	732,851.41	3,886,893.92	678.18	10.97	359.0	14º
LH98-9	732,798.84	3,886,902.91	685.44	8.53	12.0	20º
LH98-10	732,843.62	3,886,895.51	678.89	9.75	353.0	14º
LH98-11	732,787.07	3,886,890.14	674.64	9.75	16.0	8º
LH98-12	732,832.26	3,886,881.06	670.56	12.19	346.0	13º
LH98-13	732,778.06	3,886,896.67	675.74	9.75	14.0	14º
LH98-14	732,831.58	3,886,902.08	685.80	8.53	5.0	10º
LH98-15	732,774.86	3,886,901.45	676.96	6.10	355.0	15º
AR-01	732,873.01	3,886,883.96	672.51	30.48	10.0	-45º
AR-02	732,863.38	3,886,834.88	652.62	77.72	10.0	-45º
AR-03	733,349.50	3,886,744.11	649.20	60.96	10.0	-45º
AR-04	733,350.70	3,886,776.65	659.91	30.48	10.0	-45º
AR-05	733,393.60	3,886,767.57	655.57	30.48	10.0	-45º
AR-06	733,389.75	3,886,741.90	652.67	60.96	10.0	-45º
AR-07	733,390.26	3,886,740.14	652.55	91.44	10.0	-70º
AR-08	733,428.49	3,886,774.38	664.78	30.48	10.0	-45º
AR-09	733,427.41	3,886,749.16	658.37	60.96	10.0	-45º
AR-10	733,427.97	3,886,727.61	657.15	91.44	10.0	-55º
AR-11	733,453.61	3,886,764.43	670.26	30.48	10.0	-45º
AR-12	733,449.40	3,886,748.38	658.12	60.96	5.0	-62º
AR-13	733,481.19	3,886,761.47	657.76	30.48	10.0	-45º
AR-14	733,479.61	3,886,730.03	651.47	60.96	8.0	-45º
AR-15	733,524.00	3,886,757.00	658.37	30.48	10.0	-45º
AR-16	733,582.72	3,886,743.10	656.84	30.48	10.0	-45º
AR-17	733,570.14	3,886,716.32	647.84	54.86	10.0	-45º
AR-18	733,516.63	3,886,733.12	648.59	52.43	10.0	-45º
AR-19	733,593.78	3,886,737.15	655.63	30.48	10.0	-45º
AR-20	733,590.21	3,886,707.46	651.05	68.58	10.0	-60º
AR-21	733,632.07	3,886,736.38	660.84	30.48	10.0	-45º
AR-22	733,626.70	3,886,705.81	653.86	60.96	9.0	-45º
AR-23	733,387.73	3,886,727.14	650.75	128.02	10.0	-70º
AR-24	733,571.00	3,886,713.00	650.14	109.73	10.0	-68º
AR-25	733,676.81	3,886,718.93	658.88	30.48	10.0	-45º
AR-26	733,675.66	3,886,693.02	655.32	60.96	11.0	-50º
AR-27	733,730.29	3,886,700.62	647.46	32.01	10.0	-45º
AR-28	733,730.38	3,886,698.90	647.30	67.06	10.0	-76º
AR-29	732,598.52	3,886,908.50	696.81	42.67	12.0	-45º
AR-30	732,592.69	3,886,858.18	683.85	91.44	10.0	-45º
AR-31	733,472.00	3,886,702.00	653.19	106.68	10.0	-42º

Table 10.2 *continued*: A Summary of the Collar Co-ordinates, Hole Lengths, Azimuths and Inclinations of the Drillholes Completed by Previous Owners for which the Collar Co-ordinates are Known, Moss Mine Property

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
AR-32	733,514.40	3,886,705.71	649.88	106.68	10.0	-45º
AR-33	733,450.00	3,886,711.00	654.71	115.82	10.0	-50º
AR-34	733,450.00	3,886,710.00	654.71	144.78	10.0	-65º
AR-35	733,058.00	3,886,692.00	638.56	179.83	10.0	-63º
AR-35BR	733,058.00	3,886,695.00	638.56	213.36	10.0	-63º
AR-36R	732,757.00	3,886,762.00	654.41	155.45	10.0	-64º
AR-37R	733,448.61	3,886,648.67	647.82	190.50	10.0	-55º
AR-38R	733,176.00	3,886,682.00	633.37	179.83	10.0	-55º
AR-39R	733,016.15	3,886,645.79	634.99	121.92	0.0	-90º
AR-40R	732,652.00	3,886,422.00	620.88	121.92	10.0	-45º
AR-42R	732,571.00	3,886,823.00	689.76	182.88	10.0	-50º
AR-44R	733,369.26	3,886,728.68	650.14	109.73	10.0	-60º
AR-45C	732,764.00	3,886,760.00	657.76	304.80	10.0	-65º
AR-46C	732,992.01	3,886,651.19	637.60	346.25	10.0	-77º
AR-47C	733,176.47	3,886,680.89	636.58	249.94	10.0	-70º
AR-48C	732,780.00	3,886,846.00	651.66	71.63	10.0	-45º
AR-49C	732,837.15	3,886,828.73	652.39	74.68	10.0	-45º
AR-50C	733,473.45	3,886,700.85	649.21	146.61	10.0	-60º
AR-51C	732,960.99	3,886,780.41	644.47	153.62	10.0	-65º
AR-52C	733,232.90	3,886,753.14	641.46	90.22	10.0	-45º
AR-53C	733,627.41	3,886,709.55	655.27	99.36	0.0	-90º
AR-54C	733,470.61	3,886,699.82	649.10	153.01	0.0	-90º
AR-55C	733,351.89	3,886,680.21	656.48	166.12	0.0	-90º
AR-56C	733,108.00	3,886,736.00	637.03	230.43	0.0	-90º

10.1.4 Use of Data

To the best of the Qualified Person's knowledge, other than Mintec's 1991 preliminary Mineral resource estimate noted in Section 6.3 (which estimate was not compiled to CIM standards and cannot, therefore, be relied upon), the drillholes listed on Table 10.2 were not used for anything other than general exploration work targeted as assessing the mineralized potential of the Moss Vein.

10.2 The Company (2011 through 2013)

Since entering into the joint venture agreement with Patriot Gold in February 2011, the Company has carried out three drilling programs on the Moss Mine Property. The programs are termed Phase One through Phase Three; Phase Three was completed in 2013 since when no further exploration drilling has been carried out. The Company has instead focused on the Phase I activities described in Sections 2.2 and 5.6, and the on-going feasibility study of the planned Phase II operations.

The Phase One drilling program was supervised by MinQuest; the Phase Two and Phase Three programs were supervised directly by Golden Vertex. Table 10.3 summarizes the type and number of holes drilled during each program phase. Figure 10.2 is a colour-coded collar location plan for the listed holes.

Figure 10.1: A Colour-Coded Plan of the Locations of the Collars of the Drillholes Completed by Previous Owners for which the Collar Co-ordinates are Known, Moss Mine Project

(compiled from data contained in the drillhole database supplied by the Company)

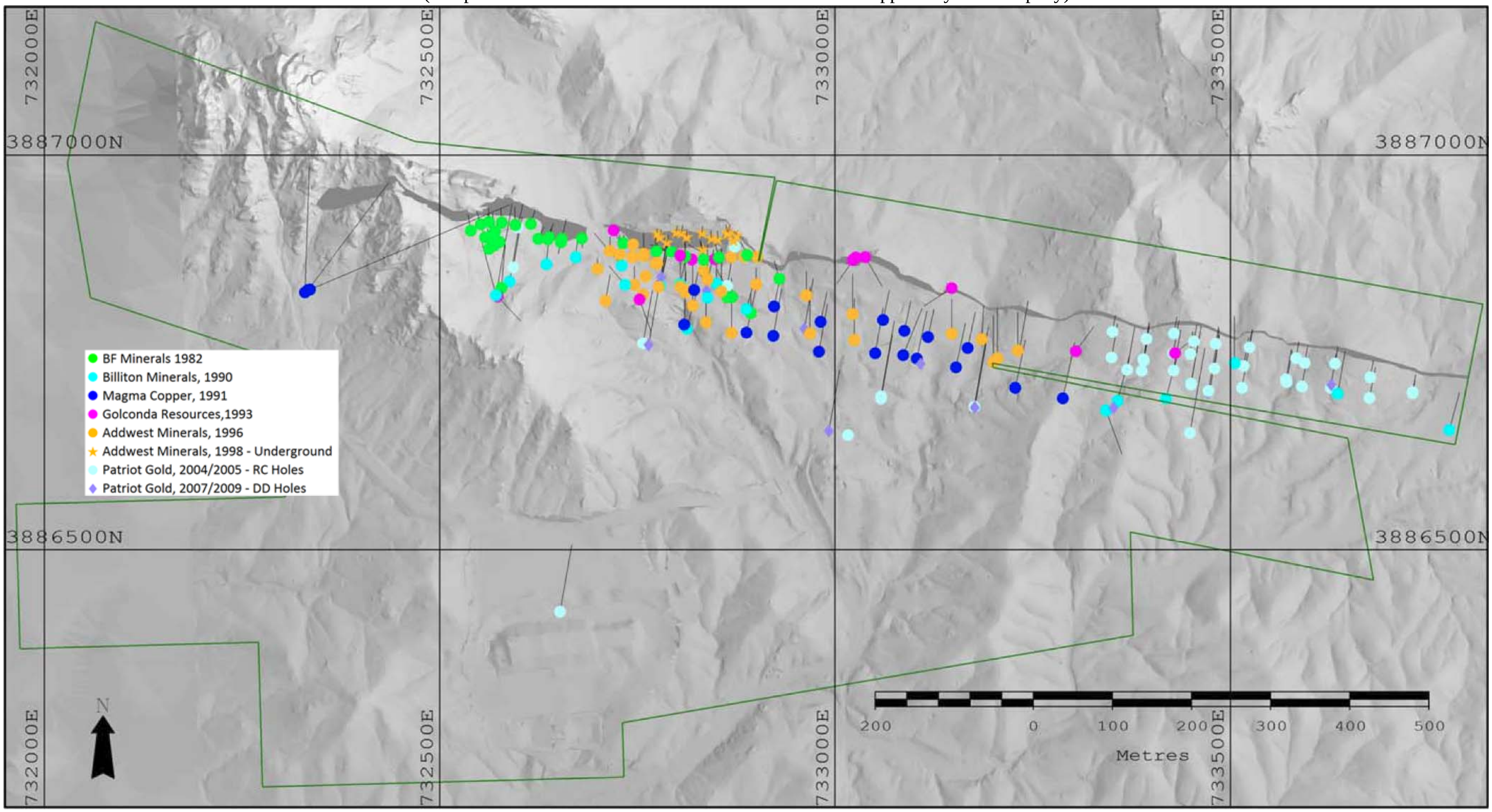


Table 10.3: A Summary of Drillholes Completed by the Company Over Its Three-Phase, 2011 to 2013 Infill and Mineral Resource Expansion Drilling Program, Moss Mine Project

(compiled from information supplied by the Company)

Program Phase	Type	Number of Holes	Total Metres	Drillhole ID Numbers	
				From	To
Phase One	RC	54 (incl. AR-58RD)	6,277.36	AR-57R AR-78R AR101R MW-1R	AR-68R AR-99R AR119R -
	Diamond Drillhole	10	794.31	AR-70C AR-100C	AR-77C -
	Sub-total	*64*	*7,071.67*	-	-
Phase Two	RC	19	2,375.00	AR-120R	AR-138R
	Diamond Drillhole	23	2,720.25	AR-139C	AR-161C
	Sub-total	*42*	*5,105.25*	-	-
Phase Three	Diamond Drillhole	36	3,968.86	AR-162C AR-188C	AR-172C AR-212C
	Orientated Diamond Drillhole	15	1,453.29	AR-173C	AR-187C
	Percussion	323	8,603.28	0+00A ADIT-E-75-1 DIKE-1 RATTAN-CP1 RATTAN-S1 Ruth-1-3 Ruth-2-1 RuthShaft-1 RuthDump-3 MW2012-1 WW-1	21+50G ADIT-W-125-9 DIKE-29B RATTAN-CP3 RATTAN-S2-3 Ruth-1-19 Ruth-2-19 RuthShaft-3 RuthDump-11 MW2012-3 WW-2
	Sub-total	*349*	*10,594.29*	-	-
Totals	*RC* *Diamond Drill* *Percussion*	*73* *84* *323*	*8,652.36* *8,936.71* *8,603.28*	- - -	- - -
Overall Totals		**480**	***26,192.35***	-	-

10.2.1 Collar Locations

The collars of all the Phase One to Phase Three drillholes were surveyed by Company personnel, using a differential GPS. The locations of individual collars are marked by a plastic pipe set in and concreted into the top of the drillhole, with an adjacent metal rod or wooden stake set in concrete and marked with flagging tape (Figure 10.3). Each collar monument is marked with the drillhole number, azimuth and inclination. The collar co-ordinates were verified as part of a larger due diligence program that included the holes drilled by previous owners and operators of the Moss Mine Property.

Figure 10.2: A Colour-Coded Plan of the Locations of the Collars of the Drillholes Completed by the Company During its Three-Phase (2011 to 2013) Drilling Program, Moss Mine Project

(compiled from data contained in the drillhole database supplied by the Company)

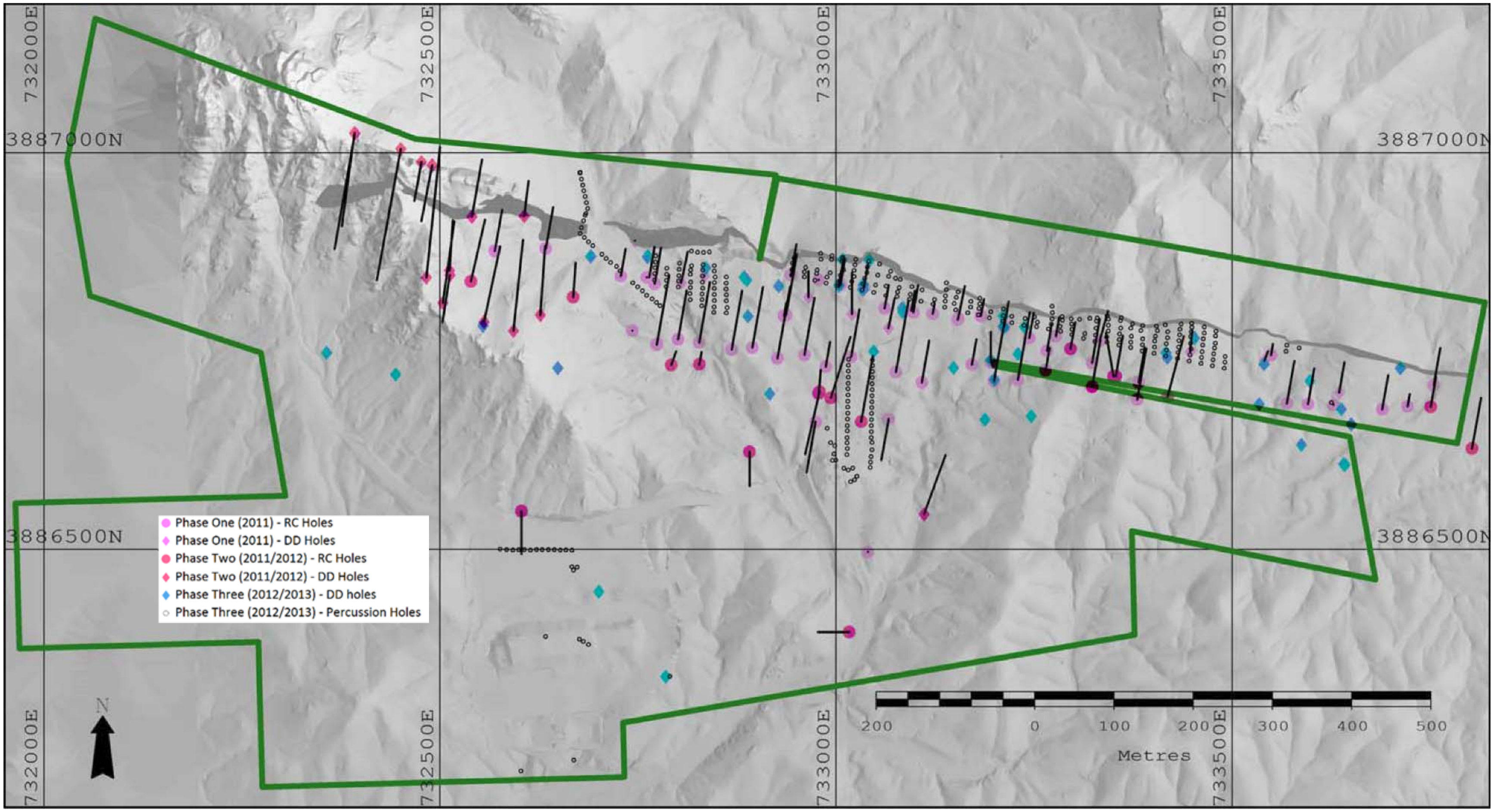


Figure 10.3: An Example of A Drillhole Collar Marked by a Plastic Pipe with an Adjacent Metal Rod Set in Concrete and Marked with Flagging Tape, the Company's Phase Three (2012/2013) Drilling Program, Moss Mine Project
(copied from the 2013 Technical Report that is listed on www.Sedar.com)

10.2.2 Downhole Surveys

Downhole directional surveys were carried out by Company personnel using a Reflex EZ-Trac tool. Downhole surveys were carried out for:

- 21 RC and diamond drillcore holes during the Phase One program;
- four RC holes (i.e. all those greater than 152 m/500 ft long) and 17 core holes (i.e. all but one) during the Phase Two program; and
- all 42 core holes in Phase Three (the short [less than 30 m] and sometimes very short [less than 10 m] and uniformly vertical percussion holes were not downhole surveyed).

The available downhole surveys were reviewed by Douglas Brownlee, P. Geo., by analyzing the change in azimuth and inclination from the collar of individual holes. The objective was to determine whether a standard deviation (azimuth and inclination) could be determined based on type of drilling and drillhole orientation (azimuth). It was found that, with the exception of the 2012 diamond drillholes, it is not possible to determine a standard deviation (azimuth/inclination).

Downhole deviation was calculated by subtracting the initial azimuth or inclination from the reading recorded at each test depth. To reduce the amount of noise caused by the initial collar reading, the first downhole reading was used instead of the collar data. The outcomes result in positive or negative values compared with the initial azimuth or inclination. A positive azimuth value means that the hole is turning to the right, whereas a negative value means that the hole is turning to the left. A positive inclination value means the hole is flattening, whereas a negative value means that the hole is steepening.

For purposes of analysis the diamond drillholes were grouped by year and the holes in general were grouped according to the sector each hole azimuth falls into, as well as its inclination. By virtue of the orientation of the diamond drillholes, only three azimuth quadrants applied to diamond drillholes and only one to the RC drillholes, whereas three inclination groups applied to the diamond drillholes (Groups C, D and E) and all four applied to the RC drillholes (Groups A through D):

- azimuth sectors (clockwise) – A = 350º to 25º (“northeast azimuth sector”) and B = 165º to 200º (“south azimuth sector”) and D = 045º to 145º (drillhole AR-210C only); and
- inclination (negative downwards) – A = -90º to -80º, B = -79º to -70º, C = -69º to -50º and D = -49º to -40º.

The results of the downhole deviation analysis are presented as a series of 18 scatter plots in Appendix A. The following conclusions apply:

- no overall or average trends can be identified as regards either azimuth or inclination deviation in the diamond drillhole dataset, deviations instead tend to be variable and localized -
  - downhole azimuth and inclination deviation in the diamond drillholes is very small to negligible to drillhole depths of at least 50 m, thereafter the deviations are limited to a few degrees only; and
- as might be expected, azimuth and inclination deviation in the RC holes is persistently higher than in the diamond drillholes -
  - azimuth deviation tends to be in the positive direction (i.e. to the right, or to the east in the majority of cases, which may be expected as the drill string turns clockwise), but
  - the same holes tend to steepen with depth (which is also typical of RC holes), and
  - to RC drillholes depths of 100 m, deviation is less than 10º in all but a few cases and less than 5º in the majority of cases.

In all cases it is the combination of azimuth and inclination deviation that determines overall drilling accuracy. If the drillholes are plotted in three-dimensions then in each case the end points of the holes are close to the ideal/perfect traces of the undeviated holes with the same start azimuths and inclinations to the holes’ end points. In the opinion of the Qualified Person for this Sub-Section of this Technical Report (Mr. Douglas Brownlee, P. Geo.), drillhole deviation is not, therefore, considered a limiting issue as regards the veracity of the drillhole database, inclusive of those holes without downhole surveys and the holes drilled by previous owners and operators of the Moss Mine Property.

10.2.3 Phase One Drilling (2011)

The purpose of the Company’s Phase One drilling program was to confirm previous results reported from the holes drilled by previous owners and operators of the Moss Mine Property, to infill previous drilling and to expand the Company’s knowledge of the Moss deposit that ‘*remains open on strike and at depth*’ (see the Company’s news release dated April 14, 2011). The program was carried out in March to June 2011, by Envirotech Drilling LLC of Winnemucca, Nevada. A total of 64 holes were completed for 7,071.67 m, including 10 diamond drillholes (794.31 m) and 54 RC drillholes (6,277.36 m). The

RC drilling program included one waterwell (MW-R1) that was sampled, assayed and included in the drillhole and assay databases for the Moss Mine Project.

In the majority of cases the complete lengths of the drilled holes were sampled and assayed, and the majority of samples were 1.52 metres (5 ft) in downhole length. The majority of the holes were drilled at inclinations that did not intersect either the Moss Vein or West Vein at right angles and the sampling method did not reflect the true thickness of the mineralization. Data from 58 of the Phase One holes, together with the drillhole data compiled by previous owners, eight trenches and 53 channel samples, was used to compile the 2011 Mineral Resource estimate detailed in the 2011 Technical Report.

Table 10.4 summarizes the azimuths, dips and lengths of the Phase One drillholes. Table 10.5 (that is in two parts due to its overall length) summarizes the significant drillhole intercepts, as reported by the Company in news releases dated May 19, June 22, July 21, August 16, 24 and 30 and October 14, 2011.

10.2.4 Phase Two Drilling (2011/2012)

The purpose of the Company's Phase Two drilling program was to '*expedite the path to a production decision on the Moss Gold-Silver Project*' by testing '*the newly discovered western extension of the Moss deposit*' and, by infill drilling, '*to render inferred resources into the indicated resources category*' (see the Company's news release dated December 05, 2011). The program was carried out in two parts. The first part consisted of 19 RC holes (2,375 m) drilled during November and December 2011 by Diversified Drilling LLC of Missoula, Montana. The second part consisted of 23 diamond drillholes (2,730.25 m) drilled in January to March 2012 by Timberline Drilling, Inc. of Hayden Lake, Idaho.

In the majority of cases the complete lengths of the drilled holes were sampled and assayed. The majority of samples were 1.52 metres (5 ft) in downhole length. The majority of the holes were drilled at inclinations that did not intersect either the Moss Vein or West Vein at right angles and the sampling method did not reflect the true thickness of the mineralization. The data, together with the drillhole data compiled by previous owners, the Phase One drilling data and data from the first part of the Phase Three drilling program was used to compile the October 2012 Mineral Resource estimate that was subsequently amended in the 2013 Technical Report.

Table 10.6 summarizes the azimuths, dips and lengths of the Phase One drillholes. Table 10.7 (that is in two parts due to its overall length) summarizes the significant drillhole intercepts, as reported by the Company in news releases dated February 27, March 27, April 02, April 30 and May 16, 2012.

Table 10.4: A Summary of the Collar Co-ordinates, Lengths, Azimuths and Inclinations of the Drillholes Completed by the Company During Its Phase One Drilling Program, Moss Mine Project

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Hole Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
AR-57R	732,960.04	3,886,745.18	642.55	172.21	10.0	-65º
AR-58R	732,936.80	3,886,795.47	645.40	109.73	10.0	-45º
AR-58RD	732,934.17	3,886,794.55	645.42	39.62	10.0	-65º
AR-59R	732,894.28	3,886,754.40	649.40	182.88	10.0	-65º
AR-60R	733,018.90	3,886,742.78	643.18	178.31	10.0	-70º
AR-61R	732,986.96	3,886,730.23	639.20	182.88	10.0	-80º
AR-62R	732,925.98	3,886,740.86	645.06	182.88	10.0	-60º
AR-63R	732,771.00	3,886,835.00	650.75	117.35	10.0	-67º
AR-64R	733,075.39	3,886,723.66	636.56	182.88	10.0	-60º
AR-65R	733,108.34	3,886,710.57	631.90	155.45	10.0	-70º
AR-66R	733,171.41	3,886,733.03	633.09	187.45	10.0	-80º
AR-67R	733,229.46	3,886,713.86	640.58	134.11	10.0	-50º
AR-68R	733,199.29	3,886,712.39	634.83	140.21	10.0	-55º
AR-69C	733,195.36	3,886,735.63	637.94	90.83	0.0	-65º
AR-70C	733,020.05	3,886,796.64	649.36	103.94	0.0	-60º
AR-71C	733,066.00	3,886,779.00	645.26	91.44	12.0	-65º
AR-72C	732,833.61	3,886,844.95	657.43	85.95	0.0	-90º
AR-73C	732,746.00	3,886,889.00	665.07	54.86	0.0	-90º
AR-74C	732,804.44	3,886,836.08	654.95	94.49	12.0	-65º
AR-75C	732,965.01	3,886,817.87	649.09	65.53	0.0	-60º
AR-76C	733,635.00	3,886,698.00	652.27	76.81	10.0	-60º
AR-77C	733,447.00	3,886,749.00	658.37	58.83	5.0	-62º
AR-78R	733,266.23	3,886,750.30	639.39	80.77	10.0	-65º
AR-79R	733,627.94	3,886,682.38	651.37	190.50	10.0	-90º
AR-80R	733,323.00	3,886,735.00	644.65	79.25	10.0	-45º
AR-81R	733,264.38	3,886,725.57	642.93	166.12	10.0	-80º
AR-82R	733,381.89	3,886,710.58	654.07	172.21	10.0	-75º
AR-83R	733,323.22	3,886,705.21	649.82	233.17	10.0	-75º
AR-84R	733,549.00	3,886,745.00	653.80	80.77	10.0	-80º
AR-85R	733,380.06	3,886,689.13	658.71	251.46	10.0	-75º
AR-86R	733,568.90	3,886,684.04	649.19	158.50	10.0	-70º
AR-87R	733,689.38	3,886,676.83	646.17	86.87	10.0	-60º
AR-88R	733,721.25	3,886,680.47	646.20	91.44	10.0	-80º
AR-89R	733,754.35	3,886,707.17	647.73	67.06	10.0	-45º
AR-90R	733,182.94	3,886,794.32	636.63	30.48	10.0	-45º
AR-91R	733,212.12	3,886,780.53	643.64	45.72	10.0	-45º
AR-92R	733,243.83	3,886,766.43	645.38	45.72	10.0	-45º
AR-93R	733,277.31	3,886,768.70	644.78	48.77	10.0	-60º
AR-94R	733,332.55	3,886,763.17	653.87	54.86	16.0	-45º
AR-95R	733,153.68	3,886,789.66	641.35	60.96	12.0	-45º
AR-96R	733,153.94	3,886,790.96	641.65	121.92	12.0	-80º
AR-97R	733,122.00	3,886,797.00	655.00	79.25	10.0	-80º
AR-98R	733,097.48	3,886,798.82	647.48	45.72	10.0	-40º
AR-99R	733,095.00	3,886,801.00	652.27	60.96	10.0	-65º
AR-100C	733,540.73	3,886,738.11	650.45	71.63	20.0	-80º
AR-101R	733,595.66	3,886,683.38	651.60	132.59	10.0	-70º
AR-102R	732,827.17	3,886,760.96	640.89	182.88	10.0	-65º
AR-103R	732,800.47	3,886,764.93	649.37	182.88	10.0	-70º
AR-104R	732,773.67	3,886,758.08	650.20	243.84	10.0	-80º
AR-105R	732,742.74	3,886,775.78	656.22	91.44	0.0	-90º
AR-106R	733,061.20	3,886,803.94	652.11	51.82	10.0	-55º
AR-107R	733,035.18	3,886,830.68	660.28	42.67	10.0	-45º
AR-108R	733,005.50	3,886,833.62	658.97	45.72	10.0	-45º
AR-109R	732,943.00	3,886,846.00	655.93	54.86	10.0	-45º
AR-110R	732,978.00	3,886,840.00	658.37	79.25	0.0	-90º
AR-111R	732,633.50	3,886,879.19	676.04	74.68	10.0	-45º
AR-112R	732,773.85	3,886,759.39	650.59	152.40	10.0	-55º
AR-113R	732,761.00	3,886,843.00	655.32	115.82	0.0	-90º
AR-114R	732,569.04	3,886,876.22	694.07	74.68	10.0	-45º
AR-115R	732,974.00	3,886,661.00	637.64	91.44	190.0	-45º
AR-116R	733,066.18	3,886,664.40	638.11	91.44	190.0	-55º
AR-117R	732,764.00	3,886,843.00	655.32	91.44	10.0	-65º
AR-118R	732,728.30	3,886,844.68	663.11	68.58	10.0	-60º
AR-119R	732,868.23	3,886,752.07	647.26	178.31	10.0	-65º
MW-1R	733,040.24	3,886,496.98	626.27	51.82	0.0	-90º

Table 10.5: A Summary of the Significant Drillhole Intercepts from the Company's Phase One Drilling Program, As Reported in Company News Releases, Moss Mine Project

Drillhole ID	Sample Interval From	Sample Interval To	Length (m)	Assay Grades (g/t) Au	Assay Grades (g/t) Ag	True Thickness (m)
AR-57R	105.16	166.12	60.96	0.41	5.87	Not Known
incl.	152.40	161.54	9.14	1.11	13.30	
AR-58R	54.86	74.68	19.82	1.35	14.08	
incl.	59.44	71.63	12.19	2.06	19.68	
AR-59R	112.78	167.64	54.86	0.53	6.98	
incl.	112.78	123.44	10.66	1.13	14.73	
AR-60R	79.25	158.50	79.25	0.45	3.77	
incl.	140.21	150.88	10.67	1.67	8.80	
AR-61R	115.82	182.88	67.06	0.45	3.94	
incl.	128.02	129.54	1.52	5.86	61.30	
and	163.07	172.21	9.14	0.72	8.16	
AR-62R	100.58	172.21	71.63	0.83	11.16	
incl.	109.74	117.35	7.61	3.42	47.89	
and	109.74	143.26	33.52	1.27	18.28	
AR-63R	7.62	82.30	74.68	0.36	5.27	
incl.	7.62	18.29	10.67	0.82	9.01	
AR-64R	131.06	175.26	44.20	0.71	8.69	
incl.	143.26	173.74	30.48	0.94	10.84	
AR-66R	150.89	184.40	33.51	1.25	30.17	
incl.	163.08	173.74	10.66	2.89	67.89	
AR-67R	80.77	126.49	45.72	1.47	20.0	39.6
incl.	105.16	114.30	9.14	4.43	52.4	7.9
AR-68R	83.82	134.11	50.29	1.24	13.0	41.2
incl.	108.20	126.49	18.29	1.81	18.3	15.0
AR-69R	77.72	90.83	13.11	4.86	45.8	9.3
incl.	85.80	90.83	50.3	10.26	99.5	3.6
AR-70C	38.10	74.68	36.58	0.61	5.5	38.0
incl.	38.10	42.67	4.57	1.12	4.9	3.5
incl.	62.48	68.58	6.10	1.56	18.6	4.7
AR-71C	19.81	73.15	53.34	0.84	10.4	37.7
incl.	59.44	68.58	9.14	2.35	25.9	6.5
AR-72C	51.82	85.95	34.13	1.67	10.3	11.7
incl.	78.03	83.06	5.03	6.30	14.1	1.7
AR-73C	3.05	46.94	43.89	1.98	31.1	15.0
incl.	32.00	41.15	9.15	3.52	67.8	3.1
AR-74C	18.29	89.92	71.63	1.20	17.5	50.7
incl.	70.87	83.82	12.95	3.24	49.8	9.2
AR-75C	45.72	59.44	13.72	2.70	21.7	10.5
incl.	53.34	57.91	4.57	5.69	46.5	3.5
AR-76C	51.82	76.81	24.99	2.11	31.1	19.1
incl.	70.10	76.81	6.71	6.45	77.0	5.1
AR-77C	32.00	54.86	22.86	1.06	19.5	17.0
AR-78R	47.24	80.77	33.53	1.12	22.9	23.7
incl.	64.01	73.15	9.14	2.83	52.0	6.5
AR-79R	111.25	176.78	65.53	0.25	4.7	22.4
AR-80R	60.96	70.10	9.14	0.83	10.8	8.3
AR-81R	96.01	112.78	16.77	0.52	3.2	8.4
AR-82R	149.35	172.21	22.86	0.39	4.9	13.1
AR-83R	204.22	227.08	22.86	0.44	2.7	13.1
AR-86R	111.25	152.40	41.15	0.96	16.2	26.5
incl.	144.78	149.35	4.57	4.74	87.2	2.9
AR-90R	15.24	25.91	10.67	2.87	24.9	9.7
AR-91R	6.10	45.72	39.62	0.82	14.5	35.9
incl.	6.10	27.43	21.33	1.25	22.8	19.3
and	22.86	27.43	4.57	3.46	60.5	4.1
AR-92R	19.81	39.62	19.81	1.19	24.3	18.0
incl.	32.00	39.62	7.62	2.05	42.2	6.9
AR-93R	30.48	33.53	3.05	2.06	41.7	2.8
AR-95R	32.00	60.96	28.96	0.22	7.4	26.2
incl.	57.91	60.96	3.05	0.54	20.9	2.8
AR-96R	54.86	67.06	12.20	0.88	5.4	6.1
AR-97R	28.96	54.86	25.90	0.82	8.9	13.0
incl.	38.10	48.77	10.67	1.44	14.9	5.3
AR-98R	15.24	38.10	22.86	2.55	21.6	21.5
incl.	21.34	33.53	12.19	4.22	32.0	11.5
AR-99R	19.81	47.24	27.43	1.53	15.0	19.4
incl.	28.96	38.10	9.14	3.51	26.7	6.5

Table 10.5 *continued*: A Summary of the Significant Drillhole Intercepts from the Company's Phase One Drilling Program, As Reported in Company News Releases, Moss Mine Project

Drillhole ID	Sample Interval From	Sample Interval To	Length (m)	Assay Grades (g/t) Au	Assay Grades (g/t) Ag	True Thickness (m)
AR-101R	73.15	129.54	56.39	1.27	13.4	36.2
incl.	79.25	85.34	6.09	4.14	34.2	3.9
and	106.68	129.54	22.86	1.70	17.6	14.7
AR-102R	0.00	140.21	140.21	0.23	3.6	99.1
incl.	64.01	68.58	4.57	1.20	47.6	3.2
AR-103R	0.00	166.12	166.12	0.19	2.7	106.8
incl.	67.06	92.96	25.90	.36	4.7	16.6
AR-104R	0.00	190.50	190.50	0.18	1.7	95.3
incl.	163.07	167.64	4.57	1.44	4.0	2.3
AR-105R	0.00	86.87	86.87	0.18	3.1	29.7
incl.	1.52	10.67	9.15	0.52	2.3	3.1
and	64.01	65.53	1.52	0.90	48.2	0.5
AR-106R	21.34	47.24	25.90	0.77	7.2	21.2
incl.	33.53	47.24	13.71	1.22	11.3	11.2
and	38.10	45.72	7.62	1.76	15.5	6.2
AR-107R	15.24	35.05	19.81	1.36	11.6	18.0
AR-108R	15.24	24.38	9.14	2.56	15.0	8.3
AR-109R	6.10	25.91	19.81	0.37	4.3	18.0
and	38.10	39.62	1.52	1.56	32.9	1.4
AR-110R	41.15	77.72	36.57	2.15	19.9	12.5
incl.	53.34	73.15	19.81	3.49	30.4	6.8
AR-111R	0.00	36.58	36.58	0.38	6.3	33.2
incl.	4.57	21.34	16.77	0.54	8.4	15.2
and	60.96	62.48	1.52	2.18	1.0	1.4
AR-112C	15.24	144.78	129.54	0.31	5.0	106.1
incl.	60.96	79.25	18.29	1.08	21.2	15.0
incl.	137.16	140.21	3.05	1.79	2.8	2.5
AR-113R	0.00	42.67	42.67	0.73	7.9	14.6
incl.	0.00	6.10	6.10	2.62	16.7	2.1
AR-114C	0.00	65.53	65.53	0.54	5.9	59.4
incl.	0.00	13.72	13.72	1.55	10.7	12.4
incl.	38.10	42.67	4.57	0.69	16.4	4.1
AR-115R	76.20	79.25	3.05	1.00	0.7	2.8
AR-116R	22.86	24.38	1.52	1.27	1.5	1.2
AR-117R	0.00	19.81	19.81	1.05	12.5	14.0
and	39.62	57.91	18.29	0.53	3.4	12.9
AR-118R	0.00	62.48	62.48	0.22	4.4	47.9
incl.	7.62	12.19	4.57	0.52	9.5	3.5
AR-119R	121.92	166.12	44.20	0.39	7.3	31.3
incl.	143.26	150.88	7.62	0.73	17.5	5.4

Table 10.6: A Summary of the Collar Co-ordinates, Lengths, Azimuths and Inclinations of the Drillholes Completed by the Company During Its Phase Two Drilling Program, Moss Mine Project

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Hole Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
AR-120R	732,979.00	3,886,699.00	639.67	161.54	193.0	-60º
AR-121R	732,978.00	3,886,697.00	639.54	239.27	5.0	-83º
AR-122R	732,993.27	3,886,691.26	636.77	235.31	18.0	-70º
AR-123R	733,031.77	3,886,661.24	637.30	184.40	10.0	-63º
AR-124R	733,295.91	3,886,752.73	640.94	99.06	10.0	-70º
AR-125R	733,323.20	3,886,705.23	649.30	109.73	10.0	-66º
AR-126R	733,750.55	3,886,679.90	649.35	76.20	10.0	-60º
AR-127R	733,418.37	3,886,693.01	655.99	156.97	15.0	-60º
AR-128R	733,350.29	3,886,718.01	646.71	161.54	350.0	-70º
AR-129R	732,827.34	3,886,733.30	640.94	60.96	10.0	-75º
AR-130R	732,791.86	3,886,732.41	646.53	53.34	20.0	-70º
AR-131R	732,668.00	3,886,818.00	679.15	106.68	4.0	-66º
AR-132R	732,539.00	3,886,838.00	696.09	114.30	13.0	-46º
AR-133R	733,352.89	3,886,718.13	646.90	152.40	10.0	-70º
AR-134R	733,802.61	3,886,627.27	649.82	129.54	10.0	-60º
AR-135R	733,911.95	3,886,624.47	652.75	129.54	20.0	-60º
AR-136R	732,602.82	3,886,548.36	637.12	76.20	180.0	-45º
AR-137R	733,016.43	3,886,396.09	617.20	68.58	270.0	-55º
AR-138R	732,890.68	3,886,623.08	635.02	60.96	180.0	-45º
AR-139C	732,592.59	3,886,775.28	705.12	206.35	6.0	-56º
AR-140C	732,555.37	3,886,786.65	701.80	194.77	12.0	-60º
AR-141C	732,626.56	3,886,795.46	692.28	151.79	4.0	-55º
AR-142C	732,511.80	3,886,851.62	706.09	110.64	4.0	-55º
AR-143C	732,512.36	3,886,845.98	706.03	110.49	189.0	-57º
AR-144C	732,482.74	3,886,842.76	717.35	166.12	6.0	-5º
AR-145C	732,482.74	3,886,842.76	717.35	234.70	6.0	-47º
AR-146C	732,606.10	3,886,920.00	698.89	60.96	7.0	-43º
AR-147C	732,606.10	3,886,920.00	698.89	30.48	7.0	-15º
AR-148C	732,503.48	3,886,810.79	714.78	154.84	8.0	-48º
AR-149C	732,540.06	3,886,919.22	718.44	73.15	10.0	-3º
AR-150C	732,392.36	3,887,024.76	757.99	213.36	189.9	-46º
AR-151C	732,392.22	3,887,024.16	758.84	30.48	189.8	-7º
AR-152C	732,392.05	3,887,024.58	758.23	136.55	187.8	-30º
AR-153C	732,450.48	3,887,004.84	755.02	200.56	190.4	-56º
AR-154C	732,450.48	3,887,004.84	755.02	213.36	190.4	-38º
AR-155C	732,476.53	3,886,988.36	752.98	76.20	190.3	-57º
AR-156C	732,476.53	3,886,988.36	752.98	59.44	190.3	-33º
AR-157C	732,476.53	3,886,988.36	752.98	30.48	190.1	3º
AR-158C	732,476.53	3,886,988.36	752.98	16.76	189.8	32º
AR-159C	732,489.95	3,886,984.27	747.35	89.92	190.8	-37º
AR-160C	732,489.95	3,886,984.27	747.35	30.78	191.1	0º
AR-161C	733,111.10	3,886,544.60	623.60	138.07	20.0	-55º

Note: AR-157C, AR-158C and AR-160C were drilled in West Extension, in an area of difficult topography that required horizontal or up-hole drilling.

Table 10.7: A Summary of the Significant Drillhole Intercepts from The Company's Phase Two Drilling Program, As Reported in Company News Releases, Moss Mine Project

Drillhole ID	Sample Interval From	Sample Interval To	Length (m)	Assay Grades (g/t) Au	Assay Grades (g/t) Ag	True Thickness (m)
AR-120R	45.72	47.24	1.52	1.08	0.3	1.2
and	57.91	60.96	3.05	1.31	0.6	2.3
AR-121R	135.64	166.12	30.48	0.77	7.1	13.8
incl.	135.64	143.26	7.62	1.75	15.2	3.5
and	178.31	217.93	39.62	0.34	9.6	18.0
incl.	193.55	201.17	7.62	0.58	18.9	3.5
and	224.03	239.27	15.24	0.38	5.1	6.9
AR-122R	140.21	178.31	38.10	0.29	3.5	24.5
and	184.40	222.50	38.10	0.28	6.4	24.5
incl.	205.74	213.37	7.63	0.43	13.9	4.9
AR-123R	152.40	169.16	16.76	0.45	2.9	12.3
and	178.31	184.40	6.10	0.78	4.4	4.5
AR-124R	38.10	39.62	1.52	0.83	1.0	1.0
and	60.96	85.34	24.38	1.13	12.8	15.7
incl.	67.06	80.77	13.72	1.79	18.9	8.8
and	73.15	77.72	4.57	3.25	22.8	2.9
AR-125R	79.25	105.16	25.91	0.55	10.4	18.0
incl.	91.44	96.01	4.57	0.88	31.0	3.2
AR-126R	53.34	71.63	18.29	0.37	3.7	14.0
AR-127R	89.92	97.54	7.62	0.33	3.9	5.8
and	114.30	141.73	27.43	0.62	7.6	21.0
incl.	131.06	137.16	6.10	1.16	13.4	4.7
AR-128R	94.49	152.40	57.91	0.59	6.4	37.2
incl.	131.06	146.30	15.24	1.24	8.8	9.8
AR-129R	19.81	44.20	24.38	0.35	3.1	14.0
AR-130R	7.62	21.34	13.72	0.23	11.9	8.8
and	25.91	47.24	21.34	0.32	7.5	13.7
AR-131R	6.10	30.48	24.38	0.29	4.3	16.9
and	45.72	65.53	19.81	0.29	4.7	13.8
and	71.63	88.39	16.76	0.20	6.3	11.6
and	91.44	97.54	6.10	0.68	3.6	4.2
AR-132R	0.00	44.20	44.20	0.36	7.0	39.7
and	53.34	77.72	24.38	0.36	8.1	21.9
and	79.25	94.49	15.24	0.22	6.5	13.7
AR-133R	102.11	120.40	18.29	0.30	3.5	11.8
and	126.49	149.35	22.86	0.74	6.8	14.7
incl.	146.30	149.35	3.05	2.36	29.7	2.0
AR-136R	45.72	47.24	1.52	17.71	67.0	Not Known
and	64.01	65.53	1.52	0.87	0.3	Not Known
AR-138R	42.67	44.20	1.52	1.64	42.5	Not Known
and	50.29	57.91	7.62	0.28	5.1	Not Known
AR-139C	16.76	61.78	45.02	0.38	2.7	36.42
and	82.08	88.70	6.61	0.25	5.6	5.35
and	96.01	106.00	10.00	0.41	6.8	8.09
and	114.30	138.70	24.38	0.33	5.8	19.73
and	148.00	160.30	12.34	0.42	6.1	9.99
AR-140C	14.94	16.46	1.52	1.43	0.4	1.17
and	31.85	71.32	39.47	0.30	4.1	30.24
and	83.52	85.04	1.52	0.36	3.3	1.17
and	89.61	139.90	50.29	0.48	16.3	38.53
incl.	93.88	114.00	20.12	0.85	33.0	15.41
and	150.57	161.24	10.67	0.35	3.4	8.17
AR-141C	0.00	42.06	42.06	0.65	2.2	34.5
incl.	2.44	11.58	9.14	1.35	2.9	7.5
and	62.94	115.21	52.27	0.44	5.0	42.8
incl.	87.78	89.31	1.52	2.46	8.3	1.2
AR-142C	0.00	89.00	89.00	0.62	8.2	72.91
incl.	2.90	8.53	5.64	2.84	47.9	4.62
incl.	55.78	64.92	9.14	0.90	4.2	7.49
incl.	75.59	77.11	1.52	3.86	43.9	1.25
AR-143C	5.49	19.20	13.72	0.39	5.3	Not Known
incl.	13.11	14.63	1.52	1.53	29.7	Not Known
and	69.49	90.83	21.34	0.54	14.1	Not Known
incl.	86.26	87.93	1.68	3.10	126.2	Not Known
AR-144C	0.00	17.07	17.07	0.91	4.9	16.5
incl.	2.44	3.81	1.37	4.45	25.3	1.3
and	35.36	78.64	43.28	0.55	5.1	41.8
and	151.18	151.94	0.76	0.66	13.2	0.7

Table 10.7 *continued*: A Summary of the Significant Drillhole Intercepts from The Company's Phase Two Drilling Program, As Reported in Company News Releases, Moss Mine Project

Drillhole ID	Sample Interval From	To	Length (m)	Assay Grades (g/t) Au	Ag	True Thickness (m)
AR-145C	0.00	1.52	1.52	0.29	8.6	1.36
and	4.57	34.14	29.57	0.84	9.5	26.34
incl.	22.74	26.21	3.47	3.05	22.7	3.10
and	44.20	47.24	3.05	0.31	3.5	2.72
and	52.88	55.93	3.05	0.31	2.1	2.72
and	70.10	72.05	1.95	0.59	3.7	1.74
and	77.72	82.30	4.57	0.47	4.2	4.07
and	88.39	90.22	1.83	0.24	5.8	1.63
and	115.80	117.40	1.52	0.62	0.4	1.36
and	153.90	158.50	4.57	0.37	3.5	4.07
AR-146C	7.32	19.51	12.19	0.32	3.0	11.2
and	53.04	54.56	1.52	2.03	0.8	1.4
AR-147C	0.00	20.57	20.57	0.52	4.8	20.50
incl.	5.49	13.11	7.62	0.86	6.7	7.59
and	26.52	28.04	1.52	0.34	0.6	1.52
AR-148C	7.62	8.84	1.22	0.34	1.2	1.08
and	17.62	20.73	3.11	0.37	6.4	2.75
and	28.35	29.87	1.52	1.05	1.2	1.35
and	65.07	83.21	18.14	0.70	4.5	16.01
incl.	65.07	78.64	13.56	0.85	4.9	11.98
and	93.88	109.12	15.24	0.33	8.2	13.46
and	115.52	116.74	1.22	0.25	3.0	1.08
and	119.94	121.55	1.62	0.26	2.9	1.43
and	122.83	127.41	4.57	1.23	13.1	4.04
and	141.28	147.22	5.94	0.31	0.4	5.25
AR-149C	0.00	11.40	11.40	0.75	13.3	10.90
incl.	2.96	4.33	1.37	2.30	28.6	1.31
and	60.05	61.57	1.52	2.28	8.6	1.46
AR-150C	93.12	106.68	13.56	0.23	6.5	5.5
and	126.49	170.38	43.89	0.33	3.8	17.9
and	178.00	213.36	35.36	0.44	5.2	14.4
AR-151C	11.89	27.74	15.85	0.33	4.1	14.12
AR-152C	15.09	38.10	23.01	0.30	3.2	14.79
and	42.37	52.58	10.21	0.27	2.7	6.56
and	58.83	136.55	77.72	0.51	6.1	49.96
incl.	60.35	64.92	4.57	2.19	57.0	2.94
AR-153C	26.21	28.50	2.29	0.29	1.5	0.55
and	43.89	47.24	3.35	0.34	1.3	0.81
and	86.26	87.78	1.52	0.53	3.9	0.37
and	104.55	107.59	3.05	0.41	1.5	0.74
and	149.96	153.01	3.05	0.44	6.4	0.74
and	162.46	163.98	1.52	0.30	5.0	0.37
AR-154C	29.87	44.81	14.94	0.32	1.1	7.91
and	57.30	72.54	15.24	0.38	1.0	8.08
and	83.52	84.73	1.22	0.42	0.4	0.65
and	90.83	167.03	76.20	0.52	3.8	40.38
incl.	160.02	161.85	1.83	5.27	4.0	0.97
and	189.89	191.41	1.52	0.21	7.0	0.81
and	195.99	197.82	1.83	0.28	6.9	0.97
and	203.61	205.13	1.52	0.29	5.1	0.81
and	210.01	211.23	1.22	0.38	0.9	0.65
and	212.45	213.36	0.91	0.27	4.3	0.48
AR-155C	15.85	31.39	15.54	0.29	3.0	3.50
AR-156C	7.92	23.16	15.24	0.30	4.1	9.17
and	40.84	42.06	1.22	0.16	154.8	0.73
and	49.68	51.21	1.52	0.37	0.9	0.92
and	52.73	57.30	4.57	0.29	1.1	2.75
AR-157C	6.10	30.48	24.38	0.30	3.7	23.32
AR-158C	3.05	15.85	12.80	0.57	4.1	12.52
incl.	10.67	11.89	1.22	2.22	0.8	1.19
AR-159C	0.00	9.14	9.14	9.14	23.5	5.0
incl.	0.00	1.52	1.52	1.52	101.9	0.8
and	21.34	51.51	30.18	51.51	4.9	16.4
and	74.37	89.92	15.54	89.92	13.2	8.5
AR-160C	0.00	19.20	19.20	0.28	6.1	18.04
and	28.35	29.87	1.52	0.33	5.2	1.43

10.2.5 Phase Three Drilling (2012/2013)

The Company's Phase Three drilling program was in two parts. The first consisted of 42 HQ diameter, diamond drillholes (5,095.25 m), drilled by Timberline Drilling, Inc. of Hayden Lake, Idaho. The program included 23 oriented holes, drilled for purposes of geotechnical data acquisition, which were sampled and assayed and included in the Company's drillhole database. The overall program also included six PQ diameter diamond drillholes (AR-188C to AR-193C). With the exception of the mineralized intervals that were taken for metallurgical testing (see Section 13.8), each of the PQ diameter holes was sampled and assayed at regular intervals. The results were included in the Company's drillhole database.

The second part of the Company Phase Three drilling program comprised percussion drilling 323 holes, by Arizona Drilling & Blasting of Tempe, Arizona. The percussion hole program was carried out between October 2012 and March 2013 using a tracked, Atlas Copco ECM 590 drill rig with a 75 mm (3") hole diameter (Figure 10.4). All the holes were vertical and the depth was in each case limited to a maximum of 29 m (96 ft). The depth limitation was applied because percussion holes deeper than 30.5 m (100 ft) are in Arizona considered wells and require a well permit from the state. Individual holes were stopped short if they hit old workings, water or heavily fractured ground. All of the percussion holes were sampled, assayed and were included in the Company's drillhole database.

Figure 10.4: An Atlas Copco ECM 590 Percussion Drill Rig and Sample Splitting Crew, the Company's Phase Three Drilling Program, Moss Mine Project
(copied from the 2013 Technical Report that is listed on www.sedar.com)

The overall Phase Three drilling program was focused '*in the starter open pit area*' where there is '*significant vein outcropping at surface*'. The objective was to '*test the extensions of the known mineralization, in mine modelling, to streamline the initial mine design of the starter pit and to add additional resources to the planned Phase 2 Mine Plan*' (see the

Company's news releases dated September 26, 2012 and November 15, 2012). The percussion drilling program was designed with several objectives in mind:

- 187 holes (4,888.38 m) were for resource infill drilling on the Moss Vein, on the south side of a steep slope (called Hill #1) where access for RC and diamond drilling rigs is difficult –
  - the holes were drilled on a grid pattern at 15.2 m (50 ft) spacings in the east-west direction and at 7.6 m (25 ft) spacings in the north-south direction, and
  - the hole identification numbers (0+00A to 21+50G) are named by their grid coordinates;
- 49 holes of the ADIT-E-75-1 to -W-125-9 series (1,355.45 m) to verify mineralization above the 65 Level mine workings on the south side of Hill #1;
- 28 holes of the DIKE-1 to -29B series (729.69 m) to investigate mineralization on the west side of the Canyon fault;
- 47 condemnation holes in the area proposed for waste rock storage (Ruth, Ruth Shaft and Ruth Dump series holes, 1,224.69 m);
- seven condemnation holes of the RATTAN series (80.47 m), drilled in the area of the proposed test heap leach pad and ponds; and
- five other percussion holes were drilled by a truck-mounted rig supplied by Drilltech, two for water monitoring (MW2012-1 to MW2012-3) and two water wells (WW-1 and WW-2).

In the majority of cases the complete lengths of the drilled holes were sampled and assayed, and the majority of samples were 1.52 metres (5 ft) in downhole length. The majority of the holes were drilled at inclinations that did not intersect either the Moss Vein or West Vein at right angles and the sampling method did not reflect the true thickness of the mineralization. The data, together with the drillhole data compiled by previous owners, the Phase One and Phase Two drilling data and data from ten of the Phase Three holes was used to compile the October 2012 Mineral Resource estimate and its amendment detailed in the 2013 Technical Report.

Table 10.8 summarizes the azimuths, dips and lengths of the Phase Three drillholes. Table 10.9 summarizes the significant drillhole intercepts, as reported by the Company in news releases dated July 03, October 13 and October 17, 2013.

Table 10.8: A Summary of the Collar Co-ordinates, Lengths, Azimuths and Inclinations of the Diamond Drillholes Completed by the Company During Its Phase Three Drilling Program

(compiled from data contained in the verified drillhole database supplied by the Company)

Hole ID	UTM NAD 27 Co-ordinates (m)		Elevation (m)	Hole Length (m)	Azimuth (true)	Inclination
	Easting	Northing				
AR-162C	733033.85	3886826.97	659.98	43.8912	0.0	-90
AR-163C	733034.09	3886828.30	660.07	44.86656	11.0	-45
AR-164C	733034.18	3886833.82	660.79	78.6384	0.0	-90
AR-165C	733003.41	3886832.25	658.64	71.3232	0.0	-90
AR-166C	733003.58	3886833.56	658.79	48.1584	10.0	-45
AR-167C	733041.73	3886864.37	673.62	27.1272	0.0	-90
AR-168C	733007.86	3886865.15	673.49	30.1752	0.0	-90
AR-169C	733210.95	3886794.08	645.15	63.7032	0.0	-90
AR-170C	733237.86	3886780.42	648.98	31.6992	11.6	-45
AR-171C	733237.91	3886780.73	649.07	36.2712	15.0	-60
AR-172C	732690.32	3886869.44	664.41	64.9224	10.0	-45
AR-173C	732689.97	3886869.23	664.47	107.8992	340.0	-53
AR-174C	733195.03	3886738.43	637.91	117.0432	25.0	-60
AR-175C	733199.63	3886714.20	634.97	146.304	25.0	-60
AR-176C	733229.61	3886746.32	640.88	86.868	15.0	-55
AR-177C	732883.71	3886843.25	654.41	111.252	0.0	-60
AR-178C	732887.16	3886839.52	654.14	117.0432	180.0	-60
AR-179C	732888.64	3886793.97	648.61	174.9552	180.0	-45
AR-180C	733084.06	3886799.12	649.47	51.5112	0.0	-60
AR-181C	733083.57	3886804.06	649.88	61.8744	0.0	-45
AR-182C	733083.62	3886803.48	649.83	55.1688	180.0	-50
AR-183C	733416.33	3886747.77	658.23	92.6592	0.0	-60
AR-184C	733417.97	3886741.62	656.87	97.2312	180.0	-50
AR-185C	732762.04	3886869.36	659.61	104.5464	270.0	-60
AR-186C	732761.89	3886868.38	659.59	46.6344	210.0	-45
AR-187C	732883.78	3886842.92	654.38	82.296	0.0	-45
AR-188C	732835.55	3886854.78	659.20	105.156	0.0	-90
AR-189C	732927.62	3886831.98	649.15	103.3272	0.0	-90
AR-190C	733211.21	3886780.99	643.47	132.2832	0.0	-90
AR-191C	733452.34	3886766.16	664.51	121.6152	0.0	-90
AR-192C	733598.06	3886712.28	651.91	150.5712	283.0	-85
AR-193C	733212.01	3886780.08	643.55	149.0472	0.0	-90
AR-194C	733637.88	3886676.80	651.29	77.1144	144.0	-60
AR-195C	732916.14	3886696.25	647.57	221.8944	23.0	-65
AR-196C	733047.37	3886749.50	639.85	230.124	0.0	-90
AR-197C	733534.02	3886683.19	648.61	149.9616	10.5	-60
AR-198C	733540.15	3886733.53	650.40	70.104	10.5	-45
AR-199C	733587.41	3886631.84	644.15	174.9552	10.5	-70
AR-200C	733148.23	3886728.80	635.20	136.5504	10.5	-65
AR-201C	733187.79	3886663.56	638.50	238.9632	10.5	-65
AR-202C	732784.85	3886339.88	621.31	15.24	0.0	-90
AR-203C	732700.39	3886446.90	625.80	15.24	0.0	-90
AR-204C	732443.85	3886720.38	652.90	170.0784	10.5	-45
AR-205C	732356.81	3886747.85	644.11	154.8384	10.5	-45
AR-206C	733246.28	3886667.98	655.79	166.4208	10.5	-45
AR-207C	733712.20	3886728.21	656.99	60.0456	10.5	-45
AR-208C	733650.34	3886657.70	647.17	114.9096	10.5	-55
AR-209C	733640.53	3886607.49	646.03	32.6136	345.0	-60
AR-210C	733642.35	3886606.69	646.67	202.9968	35.0	-60
AR-211C	732648.60	3886728.37	690.83	184.0992	10.5	-45
AR-212C	732554.17	3886781.81	704.45	249.936	190.5	-45

Table 10.9: A Summary of the Significant Drillhole Intercepts from the Company's Phase Three Drilling Program, As Reported in Company News Releases, Moss Mine Project

Drillhole ID	Sample Interval From	Sample Interval To	Length (m)	Assay Grades (g/t) Au	Assay Grades (g/t) Ag	True Thickness (m)
AR-180	10.06	39.32	29.26	1.42	12.55	Not reported
AR-195	165.50	174.60	9.10	0.54	0.9	
and	182.30	218.90	36.60	0.91	12.1	
AR-196	196.29	203.91	7.60	0.95	32.36	
AR-197	110.60	113.70	3.10	1.08	10.2	
and	119.80	123.80	4.00	0.72	7.7	
and	125.90	144.20	18.30	1.8	16.4	
AR-198	41.80	61.60	19.80	1.08	5.8	
AR-199	133.80	139.90	6.10	1.81	3.9	
and	149.00	153.60	4.60	2.05	58.9	
AR-200	106.10	118.30	12.20	2.03	47.2	
AR-201	194.50	197.50	3.00	0.54	5.1	
and	203.60	211.20	7.60	0.7	5.6	
and	218.80	233.70	14.90	1.53	31.5	
AR-204	75.60	84.40	8.80	0.37	3.3	
and	114.90	121.00	6.10	1.06	49.3	
and	133.50	136.50	3.00	4.27	37.5	
AR-205	86.30	98.50	12.20	0.49	3.25	
and	106.10	109.10	3.00	0.53	25	
AR-206	128.14	131.37	3.23	1.80	27.77	
and	140.20	152.70	12.50	2.05	24.5	
AR-208	97.23	106.38	9.15	0.95	18.72	
incl.	101.80	106.38	4.57	1.49	26.77	
AR-210	182.58	191.41	9.83	0.75	5.40	
AR-211	32.92	39.01	6.10	1.40	1.83	
and	77.11	84.73	7.62	1.52	2.36	
and	93.90	98.40	4.50	2.01	6.2	
and	104.50	112.20	7.60	0.44	4.1	

10.3 Drilling Type and Sample Comparisons

To the best of the Company's knowledge, none of the samples from the drilling programs carried out by previous owners and operators of the Moss Mine Property are available. The Company was not, therefore, able to carry out any check analyses. The Company instead drilled, over two separate programs, a number of holes twinned with previously drilled holes to establish whether any drilling, sampling or recovery factors impacted the accuracy and reliability of the results. The Qualified Person for the 2014 Mineral Resource update described in Section 14 (Mr. David Thomas, P. Geo.) also compared the three drillhole sample types collected by the Company, with the same objectives in mind. The analyses and results are summarized in the following sub-sections.

10.3.1 Diamond Drillholes versus Previously Drilled RC Holes

During its Phase One (2011) drilling program the Company drilled nine diamond drillholes to twin RC holes drilled by previous owners and operators of the Moss Mine Property. The results were analyzed in a report by Huebert (2011), in which drilling and sampling techniques are compared. The collar positions of six of the twinned holes collars are within five metres, but the collars of three of the twinned pairs are between 10 m and 31 m apart. Huebert excluded the widely spaced twin pairs from his statistical comparison, the results of which are presented in the 2013 Technical Report.

Huebert considered several variables including drilling type, RC sampling method, sample preparation, assay technique and geological. Although the assays match up fairly well on strip logs, Huebert (2011) concluded that:

- the length of comparative intervals between the previously drilled RC holes and the Company's diamond drillholes compare quite well, indicating no significant contamination in RC drilling;

- the gold assay results compared well;
- few high grade intervals had significant increases in gold grade, for example an interval in core hole AR-69C and RC hole MC96-3 has respective gold grades of 8.13 g/t and 3.66 g/t, which biases the overall comparison;
- the overall comparison shows an overall increase of 6.7% at grades between 0.5 g/t to 2.0 g/t Au and 17.3% increase at grades above 2.0 g/t Au;
- the dataset is not large enough to be able to assess whether the increase is due to the drilling method or geological variation; but
- there is scatter on either side of the unity line, apart from the one instance of a core hole returning significantly higher grades, which suggests there is no bias to either dataset; and
- higher silver results were fairly consistently observed in the diamond drillholes (above 5 g/t Ag the sum of differences was over 30% greater in the diamond drillholes), which suggests that RC drilling carried out by previous owners and operators on the Moss Mine Property might be under-estimating the insitu silver grades.

10.3.2 Diamond Drillholes versus Percussion Holes

During the Company’s Phase Three drilling program, six diamond drillholes were twinned with percussion holes to ensure that representative samples were being collected by percussion drilling. The collars of the diamond drillholes were sited as close as practicably possible to the original, twinned percussion hole: the distances the collars of twinned holes varied between 0.70 m and 2.0 m.

The same 1.52 m (5 ft) sample intervals were used for diamond drillcores as were used for percussion holes. Company personnel plotted comparative strip logs of assays for each pair of twinned holes, which are presented in the 2013 Technical Report. The percentage difference of the summation of assays of common intervals of diamond drillhole assays to percussion hole assays varied between -28% and 85%, with the average showing the drillcore returned gold assays that were 21% higher than the percussion holes, over the same sample intervals. Despite the small differences between the collar positions of the twinned holes, the differences were attributed to geological variation between the holes, which is typical of the type of mineralization that characterizes the Moss deposit.

10.3.3 Percussion Holes versus RC Holes Drilled by Previous Owners

The Company also twinned two percussion holes with two RC holes drilled in 1996. The same findings and conclusions as for the diamond drillhole/percussion hole twins outlined above were made.

10.3.4 Overall Data Type Comparisons

A part of a larger due diligence process, the Qualified Person for the 2014 Mineral Resource update described in Section 14 (Mr. David Thomas, P. Geo.) compared the three drillhole sample types collected by the Company to establish whether any drilling, sampling or recovery factors impacted the accuracy and reliability of the results (RC, diamond drillcore and percussion drill hole samples, including RC samples collected by Addwest, Billiton, Magma and Patriot Gold).

Comparisons were limited to those samples falling within the Moss Vein and its hangingwall stockwork. Diamond drillcore data was chosen as the reference data type. Three metre composites were used to estimate nearest-neighbour models for each sample type. A search ellipse radius of 5 m was used (i.e. a maximum distance of 10 m from a sample), which the Qualified Person considered a reasonable distance based upon the first structure of the variogram models described in Section 14. The block grades estimated by each estimation method were then compared. The results are summarized on Tables 10.10 and 10.11. They show that:

- the Company's percussion drillholes are not globally biased with respect to the Company's drillcore samples;
- the drilling methods behave differently in the Moss Vein and its hangingwall stockwork –
    - generally, RC and percussion drilling both return significantly lower average grades than diamond drilling in the Moss Vein; whereas
    - in the hangingwall stockwork, RC and percussion drilling return significantly higher average grades than core drilling.

Table 10.10: A Summary of Sample Type Comparisons, The Company's Drillhole Data, Moss Vein, Moss Mine Project

Comparison	Drillcore			Percussion			RC			% Difference
	Nos.	Mean (Au g/t)	SD	Nos.	Mean (Au g/t)	SD	Nos.	Mean (Au g/t)	SD	Mean (g/t Au)
Core vs Percussion	125	1.03	0.80	125	0.84	0.34	-	-	-	18.4%
Core vs RC	45	1.51	1.49	-	-	-	45	1.65	2.99	-9.3%
RC vs Percussion	-	-	-	100	1.61	4.97	100	1.60	4.37	0.6%
Core vs Billiton RC	125	2.47	4.26	-	-	-	125	1.66	2.74	32.8%
Core vs Magma RC	None	-	-	-	-	-	-	-	-	-
Core vs Patriot RC	22	1.49	0.03	-	-	-	22	1.71	0.13	-14.8%
Core vs Addwest RC	106	1.83	1.28	-	-	-	106	1.13	1.64	38.3%

Note: Nos. = number, SD = Standard Deviation

Table 10.11: A Summary of Sample Type Comparisons, The Company's Drillhole Data, Moss Vein Hangingwall Stockwork, Moss Mine Project

Comparison	Drillcore			Percussion			RC			% Difference
	Nos.	Mean (Au g/t)	SD	Nos.	Mean (Au g/t)	SD	Nos.	Mean (Au g/t)	SD	Mean (g/t Au)
Core vs Percussion	475	0.52	0.45	475	0.60	0.57	-	-	-	-15.4%
Core vs RC	89	0.66	0.37	-	-	-	89	0.95	0.87	-43.9%
RC vs Percussion	-	-	-	361	0.32	0.17	361	0.59	0.32	-45.8%
Core vs Addwest RC	96	0.45	0.10	-	-	-	96	0.65	0.35	-44.4%
Core vs Billiton RC	234	0.53	0.20	-	-	-	234	1.18	2.86	-122.6%
Core vs Magma RC	None	-	-	-	-	-	-	-	-	-
Core vs Patriot RC	94	0.27	0.1	-	-	-	94	0.28	0.19	-3.7%
Core vs Patriot Core	None	-	-	-	-	-	-	-	-	-

Note: Nos. = number, SD = Standard Deviation

10.4 Qualified Person's Opinion

In the opinion of Qualified Person Mr. Daniel Kilby, P. Eng., it is in theory possible that loss of fines (silica dust) from the Moss Vein is significant when using RC or percussion drilling, not least due to the fineness and deportment of the native gold and electrum described in Sub-Section 7.2.4). In the case of percussion drilling this is considered unlikely due to the dust capture systems employed when drilling:

- percussion drilling was carried out by an Atlas Copco ECM 590 drill rig powered by a Cummins 220 horse power (HP) Tier III diesel engine with on board, 250 cubic feet per minute (CFM), 140 pounds per square inch (psi) compressor supplying flushing air;
- the on-board, hydraulically powered dust collector was mounted on the starboard side of the rig, with a 12.7 cm (5 inch) spiral suction hose connected to an adjustable sliding boot below the centralizer on the feed mast, through a drill mast mounted venture;
- the boot could be lowered to form a seal around the collar of the hole being drilled;
- coarse cuttings, accounting for about 80% of the total cuttings, were discharged from the forward venturi; and
- the remaining 20%, comprising the fine fraction, was discharged from the dust collector; and
- polythene sample bags (46 cm x 61 cm) were attached to both the forward venturi discharge and the dust collector discharge to ensure 100% sample collection (Figure 10.5).

Figure 10.5: Detail of the Drill Used For 2013 Percussion Drilling, Showing the Dust Collector (rear) and the Venturi (forward) Used for Sample Collection, Moss Mine Project
(copied from the 2013 Technical Report)

It should also be noted that the majority of the percussion holes were dry: for the reasons described in Section 10.3.5 their lengths were limited to a maximum of 29 m (96 ft), which meant that the majority did not intersect the watertable (and in any event, as earlier described, drilling was stopped if a percussion hole hit water because it caused the cuttings to stick to the drill rods resulting in possible contamination between intervals).

In the case of RC drilling and again for the reasons described in Sub-Section 7.2.4, fine to ultrafine grains of native gold and electrum would inevitably be liberated during the drilling process. Although the drilling method is pneumatic, the majority of the holes were drilled to depths that exceeded the depth to the surface watertable and the samples were recovered wet. The possibility therefore exists that some of the fine to ultrafine gold and silver mineralization would be flushed away by virtue of the presence of groundwater. Whatever the cause, the overall effect is to under-value the insitu gold and silver grades in RC holes in particular, which feature is consistently seen across the three comparative analyses described above.

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Little or no information concerning sample preparation, analysis and security during the drilling programs carried out by previous owners and operators of the Moss Mine Property is available. In the opinion of the Qualified Person for this section of this Technical Report (Mr. Douglas Brownlee, P. Geo.) this may be expected because the work was carried out either before the introduction of NI 43-101 or by USA companies that did not (and do not) report in accordance with NI 43-101. This is not seen as a limitation of the Company's drillhole database because the drilling, logging and sampling was carried out by reputable exploration and mining companies, and there is sufficient documentation to justify the inclusion of the information in the Company's drillhole database. It was for these reasons that the Company focused on verifying drillhole collars and downhole surveys and the veracity of RC and percussion drilling versus diamond drilling (Section 10) and verifying the assay database (Section 12). The following comments and descriptions apply to the Company's sampling procedures and protocols employed during its three-phase drilling program described in Section 10.

11.1 Reverse Circulation Drilling

RC cuttings were sampled and analyzed in 1.52 m (5 ft) increments over the entire lengths of each drillhole. Cuttings were collected via a rotating splitter mounted on the drill.

11.1.1 Phase One Sampling

For the Phase One drilling program, two samples were captured over each sample interval. The first involved adjusting the baffles to recover a 2.3 kg to 4.5 kg (5 lb to 10 lb) split of the total recovered sample, in olefin bags. A second sample was collected in a large bucket, from the rejects of the rotary splitter, and thoroughly mixed by hand before collecting a +4.5 kg (+10 lb) sample, also in olefin bags. The bags were labeled with the drillhole number and sample interval. Both samples were assayed to compare sampling method until one method was determined to be superior to the other.

11.1.2 Phase Two Sampling

For the Phase Two program, cuttings were collected via a rotating splitter mounted on the drill in a three-way split to give Samples A and B of 3.2 kg to 4.5 kg (7lb to 10 lb) each. One sample split reported to an olefin bag, the second to a bucket and the third to a chip sample tray down the center into a screen perched on a small bucket. Each of the samples were labeled with the drillhole number and sample interval. Only Sample A was assayed.

11.1.3 Security

The samples to be assayed were removed from the jobsite each day and placed in secure storage. From there, they were sent via UPS freight carrier directly to the responsible assay laboratory for preparation and analysis (Section 11.4). Before being sent, coarse crushed blank, low- and medium-gold content standards were added to the sample streams.

For the Phase One program, the second (backup) samples were kept at site until assaying was complete. Those samples assaying greater than 0.34 g/t Au, plus two samples above and below the assay interval, were collected and placed in secure storage. The remaining seconds were discarded. For the Phase Two program, Sample B (backup) samples were stored underground in a gated and locked crosscut (the Office Crosscut – see Figure 9.1).

11.2 Diamond Drilling

HQ size core was drilled and collected in 3.0 m (10 ft) lengths, then laid in waxed cardboard or plastic core boxes. Intervals were marked with wooden blocks every 0.6 m to 0.9 m (2 ft to 3 ft).

11.2.1 Phase One Sampling

For the Phase One drilling program, the core was logged at site by a MinQuest contract geologist who marked sample intervals not exceeding 1.52 m (5 ft). The logged and marked core was transported from the Moss Mine Project site by the same contract geologist, to secure storage facility located at Reno, Nevada (rented by MinQuest). After detailed geotechnical logging by an engineer, the core was cut in half using a diamond tipped saw, under the supervision of a Project Geologist. One half of the core was sampled at the intervals marked by the contract geologist, placed in olefin bags and labeled with consecutive sample numbers. Blanks and standards were added to the sample stream. Sample batches were sent by courier to the responsible assay laboratory for preparation and analysis (Section 11.4). The remaining one-half core was kept in secure storage until required for metallurgical testing.

11.2.2 Phases Two and Three Sampling

In the Phases Two and Three programs, the core was transported from the Moss Mine Project site by a Company employee and to the Company's secure core logging and storage warehouse in Bullhead City, Arizona (rented by the Company). The core was logged by a Project Geologist and sample intervals were marked at intervals not exceeding 1.52 m (5 ft). After geotechnical logging, the core was cut in half using a diamond tipped saw. One half of the core was sampled at the intervals marked by the Project Geologist, placed in olefin bags and labeled with a consecutive sample number. Blanks and standards were added to the sample stream. Sample batches were sent by courier to the responsible assay laboratory for preparation and analysis (Section 11.4). The remaining one-half core was kept in secure storage until required for metallurgical testing.

11.2.3 Security

The spilt cores remaining from the Company's Phases Two and Three drilling programs is stored at the Company's secure coreshack in Bullhead City, Arizona. The Company is in the process of recovering samples from the Phase One program stored by MinQuest at Reno, Nevada, for storage at the Company's secure core shack at Bullhead City, Arizona.

11.3 Percussion Drilling

The initial drillrod (T-51 thread = 51 mm diameter) in each drill string was 4.2 m (14 ft) long, allowing for 3.6 m (12 ft) in the hole and 0.6 m (2 ft) from the centralizer clamping device to the hole collar. Subsequent drill rods were 3.6 m (12 ft) long. The sample interval for the 76 mm (3 inch) diameter holes was set at 1.82 m (6 ft), thereby to allow a sample to be collected at the mid-point of the drill rod and when the next rod was added. At the end of each sample interval, the driller would cease drilling, continue to flush with air for 10 seconds with the dust collector on, then shut off the flushing air and the dust collector suction to allow the dust collector and venturi to fully discharge. The quantity of drill cuttings in each sample was approximately 22.7 kg (50 lb).

The coarse and fine sample from each interval was combined and then split on-site using a Jones riffle splitter. The final two splits from each sample interval were kept, with one sent for assay and the second stored on site in the 60 ft Level cross-cut of the historical underground workings. A screened sample was saved in chip trays from each interval. Additional splits from about 5% of the sample intervals were also saved to provide duplicates for assay. Certified Standard Reference Materials ("CSRM") and blanks were also inserted into the samples for QA/QC. The samples were boxed and sent by UPS courier to the assay laboratory for sample preparation and assay.

11.4 Sample Preparation and Analysis

RC and drillcore samples were sent to the assay laboratory as separate orders for each complete drillhole to ensure that each drillhole had an individual assay certificate. The percussion drill samples were sent either as complete or as combined holes.

11.4.1 Phase One Samples

The samples of the Phase One drilling program were sent to the ALS Chemex laboratory in Reno, Nevada (now called ALS Minerals), which is ISO 9001:2008 and ISO 10725 certified and independent of the Company. The core samples were prepared by crushing in a jaw crusher, riffle splitting and pulverizing. All samples were analyzed for gold and silver:

- gold was analyzed by fire assay on a 30 g sample and atomic adsorption spectrophotometry ("AAS") finish (ALS Chemex method AA23);
- samples with assay returns over the maximum detection limit of 10 g/t Au were re-run by fire assay on a 30 g sample with gravimetric finish (ALS Chemex method Au-GRA21);
- silver was analyzed by multi-acid digestion (hydrofluoric acid, nitric acid and perchloric acid, with a hydrochloric acid leach) and AAS finish (ALS Chemex method AA61);
- samples with assay returns over the maximum detection limit of 100 g/t Ag were re-run using the same method as outlined for the primary assay, but with a higher limit of detection (ALS Chemex method Ag-AA62); and
- drillcore samples were also analyzed for 33 additional elements by four acid, near-total digestion and inductively coupled plasma atomic emission spectrometer ("ICP-AES") finish (ALS Chemex method ME-ICP61).

11.3.2 Phases Two and Three Samples

The samples of the Phase Two and Phase Three drilling program were sent to the Inspectorate America Corporation's laboratory in Sparks, Nevada ("Inspectorate"). Inspectorate is a Bureau Veritas Group Company, it is ISO 9001:2008 certified and it is independent of the Company.

RC samples, percussion samples and core samples were prepared by drying for up to 24 hours, crushing in a jaw crusher to $+P_{70}$ 10 mesh, riffle splitting of about 250 g, and pulverizing the split to $+P_{85}$ 200 mesh (Inspectorate method SP-RX-2K). All samples were analyzed for gold and silver:

- gold was analyzed by fire assay on a one assay ton (30 g) sample and atomic adsorption spectrophotometry (AAS) finish, with a detection range of 0.005 to 10.0 ppm (Inspectorate method Au-1AT-AA);

- samples with assay returns above the upper limit of detection were re-assayed by fire assay on a 1 assay ton sample with gravimetric finish, with a detection range of 1 to 1,000 ppm (Inspectorate method Au-1AT-GV);
- silver was analyzed by aqua regia digestion and AAS finish, with a detection range of 0.1 to 200 ppm (Inspectorate method Ag-AR-TR);
- for RC samples silver was also analyzed as part of a 30 element package, by four-acid, near-total digestion and an ICP-AES finish (Inspectorate method 30-4A-TR).

11.5 Quality Assurance/Quality Control

During its Phase One to Phase Three drilling programs, the Company's quality assurance/quality ("QA/QC") protocols included the insertion of standards, blanks and duplicates in the assay sample stream and by carrying out replicate analyses. QA/QC is monitored by a Project Geologist using a spreadsheet in which a written QA/QC log was maintained to indicate problems, actions taken and resolutions. Standards were monitored by standard type and by laboratory certificate or sequence of analysis. If the assay return for a standard was outside three standard deviations of the standard value, the standard and at least one sample either side were re-assayed. Blanks were monitored in a spreadsheet and the intervals were re-assayed if there was an anomalous sample. The QA/QC data is organized by drilling program phase, which is the order followed in the following sub-sections.

11.5.1 Phase One QA/QC

The QA/QC procedures, protocols and results for the Company's Phase One drilling program are detailed in the 2011 Technical Report. The QA/QC procedures used were replicate samples, commercial standards and blanks. Replicate analyses of pulps were carried out for all samples over 0.4 g/t Au. Analysis of the results yielded very good correlations. Replicate silver assays were carried out for most samples over 15 g/t (plus one complete order) and showed a good correlation, although with more scatter than gold.

The Company used three blanks and five standards during Phase One. Two of the standards were internal standards prepared by MinQuest (who managed the Phase One drilling program); and three standards were CSRM. The blank samples returned acceptable results with most below detection limit and all but one within three times the detection limit. The MinQuest standards gave poor replicate assays, some outliers and a high degree of variability in one. It was in consequence of this that it was recommended that the MinQuest standards were not used in future sample streams. By comparison, the CSRM showed acceptable results, with only a few samples falling outside acceptable limits.

It was also recommended in the 2011 Technical Report that:

- the number of QA/QC samples be increased to 11 samples per 100 sample numbers (12% of the total), comprising five CSRM, two blanks, two field duplicates (drillcore or RC cuttings split), as well as two preparation duplicates;
- 5% check assays (new pulp prepared from the coarse reject) and 5% replicate assays (same sample pulp) of samples above a cut off of 0.05 ppm Au should be carried out at a second external laboratory; and
- a project-relevant, written QA/QC protocol be compiled.

11.5.2 Phase Two QA/QC

The QA/QC procedures, protocols and results for the Company's Phase Two drilling program are described in the 2013 Technical Report. Table 11.1 summarizes the QA/QC samples used during the Company's Phase Two drilling and sampling program. The total number of all samples (including blanks, CSRM, duplicates and replicates) was 3,370.

Table 11.1: A Summary of the QA/QC Samples Used During the Company's Phase Two Drilling and Sampling Program, Moss Mine Project
(compiled from information contained in the 2013 Technical Report)

QA/QC Sample	Number	Percent of Total Samples
CSRM	93	2.8%
Blanks	62	1.8%
Duplicates	27	0.8%
Replicates	175	5.2%
Total	***357***	***10.6%***

The blanks and CSRM used for the Phase Two drilling program are summarized on Table 11.2. The three standards are CSRM purchased from CDN Resource Laboratories Ltd. of Vancouver B.C. ("CDN Labs."), the certificates for which are available at www.cdnlabs.com. One of the blanks is a certified fine grained blank from CDN Labs. The second blank is a coarse rock blank collected from the field and assayed ten times for gold and silver. All the assays results returned below detection limits reports.

Table 11.2: A Summary of the Blanks and Standards (CSRM) Used During the Company's Phase Two Drilling and Sampling Program, Moss Mine Project
(compiled from information contained in the 2013 Technical Report)

Material	Au (g/t)	2 Standard Deviations	Ag (g/t)	2 Standard Deviations
Blanks				
CDN-BL-9	<0.01	-	-	-
Coarse Blank	<0.005	-	<0.1	-
Standards (CSRM)				
CDN –GS-2J	2.360	0.200	-	-
CDN-GSP-7E	0.766	0.086	-	-
CDN-ME-15	1.386	0.102	34.0	3.7

11.5.2.1 Certified Standard Reference Materials

The assays returns for the CSRM were analyzed with reference to performance gates of plus or minus two standard deviations ("±2 SD") and ±3 SD:

- values within ±2 SD and a single value between ±2 SD and ±3 SD were deemed acceptable; but
- two or more consecutive values between ±2 SD and ±3 SD and any value greater than ±3 SD were deemed acceptable.

Assay returns for any CSRM deemed to be unacceptable were re-assayed, along with a block of regular samples either side of the re-assayed CSRM. The results are presented in the 2013 Technical Report. They show that overall the blanks and standards returned acceptable results. In two cases the re-assays were similar to the original assay and outside acceptable limits, thereby indicating a problem with the CSRM sample such as a sample switch, contamination or inhomogeneity. In all cases a check plot of the unknown samples showed a good correlation with the repeats, and so the original samples were retained in the database.

11.5.2.2 Blanks

The assays returns for the blanks were analyzed with reference to performance gates of plus or minus three times the recommended or average value for the blank (warning) and five times the recommended or average value (reject). The results are presented in the 2013 Technical Report. They show that values for the coarse blank are uniformly acceptable, with most value below the detection limit. The values for gold for the blank CDN-BL-9 are mostly acceptable – only one sample failed and the following sample returned a value over the warning line, thereby indicating possible contamination. The values for silver for the blank CDN-BL-9 were acceptable for samples assayed by four-acid digestion with an ICP-AES finish. Many of the silver values are above the reference lines for samples assayed by aqua regia and AAS, but the detection limit is much lower and the silver values have the same range as those analyzed by ICP-AES. The results were (and are) therefore deemed to be acceptable.

11.5.2.3 Field Duplicates

The duplicate samples comprised splits from Sample A of the rotary cuttings from RC drilling; no duplicates were taken of the core samples. Scatter plots of gold and silver in duplicate samples are presented in the 2013 Technical Report, as scatter plots of original versus duplicate assay pairs. The plot for gold shows excellent repeatability below 1.0 g/t and is skewed only by two higher grade samples that assay lower in the duplicates (despite which, the r^2 correlation coefficient for the duplicates database is 0.9818). This was (and is) considered to reflect geological variability and an artifact of the small sample population. Silver shows more variability than gold, but the scatter lies close to the unity line and indicates no bias (the r^2 correlation coefficient is 0.8994).

11.5.2.4 Replicate Assays

Replicate analyses of pulp were made at the same laboratory when it was deemed that a CSRM did not return acceptable assay results. A batch of regular samples on either side of the CSRM were also repeated. A total of 105 replicate analyses were carried out for gold (95 excluding CSRM) and 71 for silver (70 excluding CSRM). The results are presented in the 2013 Technical Report. They show a very good correlation with some scatter for gold that was (and is) considered to reflect geological variability and an artifact of the small sample population (despite which the r^2 correlation coefficient for the database is 0.8853). The results for silver are near-ideal - the r^2 correlation coefficient for the database is 0.9974.

11.5.3 Phase Three QA/QC

The QA/QC procedures, protocols and results for the Company's Phase Three drilling program are described in the 2013 Technical Report. Table 11.3 summarizes the QA/QC samples used during the Company's Phase Three drilling and sampling program, the protocol for which required the insertion of 2.5 CSRM, 1.5 blanks and one duplicate per book of 50 sample tags. The total number of all samples (including blanks, CSRM, duplicates and replicates) was 5,530.

Table 11.3: A Summary of the QA/QC Samples Used During the Company's Phase Three Drilling and Sampling Program, Moss Mine Project
(compiled from information contained in the 2013 Technical Report)

QA/QC Sample	Number	Percent of Total Samples
CSRM	278	5.03%
Blanks	130	2.35%
Duplicates	92	1.66%
Replicates	1,241	22.44%
Total	***1,741***	***31.48%***

The blanks and CSRM used for the Phase Three drilling program are summarized on Table 11.4. The CSRM were purchased from CDN Labs. (the certificates are available at www.cdnlabs.com). Two of the blanks are certified fine grained blanks from CDN Labs. The other blanks were uncertified, coarse rocks collected from the field and labeled as Coarse Bullhead Blank, Mint Slate Blank and Loose.

Table 11.4: A Summary of the Blanks and Standards (CSRM) Used During the Company's Phase Three Drilling and Sampling Program, Moss Mine Project
(compiled from information contained in the 2013 Technical Report)

Material	Au (g/t)	2 Standard Deviations	Ag (g/t)	2 Standard Deviations	Samples
Blanks					
CDN-BL-9	<0.01	-	-	-	5
CDN-BL-10	<0.01	-	-	-	116
Coarse Bullhead Blank	Not recorded	-	-	-	3
Mint Slate Blank	Not recorded	-	-	-	3
Loose	Not recorded	-	-	-	5
Not Specified	Not recorded	-	-	-	3
Standards (CSRM)					
CDN-GS-2J	2.36	0.20	-	-	11
CDN-GS-1L	1.16	0.10	-	-	48
CDN-GS-5H	3.88	0.28	50.4	2.70	30
CDN-GS-9A	9.31	0.69	-	-	28
CDN-GS-P7H	0.799	0.050	-	-	54
CDN-GS-1K	0.867	0.098	-	-	31
CDN-GS-2M	2.210	0.244	-	-	41
CDN-GS-1P5E	1.52	0.11	-	-	32
CDN-ME-16	1.48	0.14	30.8	2.2	1
CDN-ME-15	1.386	0.102	34.0	3.7	2

11.5.3.1 Certified Standard Reference Materials

The assays returns for the CSRM were analyzed with reference to the same performance gates as described for the Company's Phase Two drilling and sampling program (±2 SD and ±3 SD). Assay returns for any CSRM deemed to be unacceptable were re-assayed, along with a block of regular samples either side of the re-assayed CSRM. The results are presented in the 2013 Technical Report. They show that the results for gold were (and are) acceptable with only a few outliers. The results for silver are generally lower than the recommended values due to different dissolution methods: the assays used aqua regia, whereas the CSRM was certified by four-acid digestion, which gives a near-total digestion and tends to yield higher results. The scatter plot also indicates that there may be some instrumental drift of the analyses with time as there is a minor trend of increasing values in time.

11.5.3.2 Blanks

The assays returns for the blanks were analyzed with reference to the same performance gates as earlier described for the Company's Phase Two drilling and sampling program. The results are presented in the 2013 Technical Report. They show that the assay returns for gold are acceptable, with most values below the detection limit. The values for silver show scatter with values up to 0.8 g/t and are similar to the Phase Two data for CDN-BL-9.

11.5.3.3 Field Duplicates

The duplicate samples comprised Sample A splits of the rotary cuttings from RC drilling; no duplicates were taken of the core samples. Seventy five duplicate samples were prepared and analyzed at ALS Minerals' laboratory at Reno, Nevada for gold and silver by fire assay on a 30 g sample and gravimetric finish (ALS Minerals method MEGRA21). The lower limits of detection are 0.05 ppm gold and 5 ppm silver.

The results are presented in the 2013 Technical Report, as scatter plots of original versus duplicate assay pairs. The plot for gold shows a very good correlation except for the one highest grade sample which does not match at all. There are also two samples in the 1.5 g/t range where the duplicate returned a higher result. However, the ALS Minerals analysis was gravimetric with a higher limit of detection than the original assay (0.05 ppm versus 0.005 ppm), and 30 of the 75 samples were below the gravimetric detection limit so they could not be compared.

Only 16 of the 75 silver samples were above the gravimetric detection limit (5 ppm) so there is limited data available for comparison. There appears to be significant spread, but this may be due to the difference in analytical method used. In general it is recommended that duplicates should be assayed using the same method as was used to assay the original sample. This was done for 17 of the field duplicates (at Inspectorate). The results are presented in the 2013 Technical Report. They show good correlation, but with a slight high bias in the originals. It should, however, be noted that the data population is too small to make a statistically valid comparison.

11.5.2.4 Replicate Assays

Replicate analyses of pulp were made at the same laboratory when it was deemed that a CSRM did not return acceptable assay results. A batch of regular samples on either side of the CSRM were also repeated. A total of 1,242 replicate analyses were carried out for gold. The results are presented in the 2013 Technical Report. They show a group of moderately high-grade samples with low original assays which ran correct values on replicate analysis. The problem was traced to the laboratory dilutor and the original assays were replaced by the replicate assays in the Company's drillhole database. The remainder of the results show good correlation with the originals.

11.6 Qualified Person's Opinion

Based on the review of the QA/QC programs described above, the Qualified Person for this section of this Technical Report (Mr. Douglas Brownlee, P. Geo.) is of the opinion that:

- the Company's exploration drilling program, drillhole surveys, sampling, security, sample preparation and assaying procedures have been carried out in accordance with CIM Best Practice Guidelines and are suitable to support Mineral Resource estimation;
- the Company's exploration and drilling programs supply sufficient information for Mineral Resource estimation and classification; and
- the Company's sampling and assaying includes adequate quality assurance procedures.

12 DATA VERIFICATION

The Qualified Person for this section of the Technical Report (Mr. Douglas Brownlee, P. Geo.) carried out a comprehensive data verification of the Company's drillhole database, during November and December 2013. All relevant, available data was utilized including reports, certificates, logs and ancilliary data in digital format for all the holes drilled by previous owners and operators of the Moss Mine Property, as described and defined in Section 10.1, and for all the holes drilled by the Company over its three drilling programs described in Section 10.2.

The verification focused on the available data and its format, what data was collected, back-up reference material, data consistency and the accuracy and reliability of the data. The verification focused on the available data and its format, what data was collected, back-up reference material, data consistency and the accuracy and reliability of the data. The Qualified Person was given unlimited access to all data stored on the Company's digital storage site (hosted by Egnyte) and he was not limited as regards data acquisition and analysis. The results are presented in a consultancy report to the Company that is entitled 'Verification of the Golden Vertex Corp. Moss Mine Drillhole Database' and dated December 31, 2013.

Verification of the Moss Mine drillhole database indicates that there are no errors or inconsistencies that would have any material effect on the database. In the opinion of the Qualified Person for this section of the Technical Report (Mr. Douglas Brownlee, P. Geo.) the database is accurate and suitable for use in Mineral Resource estimation. The following is a summary of the Qualified Person's findings and the subsequent actions taken to rectify the issues raised:

- the current master drillhole database is acceptable with only minor errors and problems found during the verification process (the minor errors and problems were itemized and were rectified); however
- details of the drillhole collar and downhole survey verification are presented in Sections 10.1 and 10.2 –
  - all holes without verified collar positions were excluded from the database, and
  - two downhole surveys were found to be referenced but not included in the database (AR-121R and AR-128R) (which omission was rectified);
- excluding standards, duplicates and blanks there are approximately 26,539 assay samples in the master database -
  - 17,458 have electronic certificates and all the results were verified,
  - 1,975 have copies of paper certificates (Jacobs Assay Office), five to ten results per page were verified,
  - 7,106 entries have no certificates (electronic or hardcopy) and were checked against printouts and reports (five to 10 results verified per page) and the Patriot Gold database (all results verified),
  - 2,179 have ICP analytical results have electronic certificates (part of the 17,458 assays with electronic certificates), all of which were verified,
- only one error was found for the samples (a standard) with either electronic or paper certificates (#1047330 in the database had 0.71 gpt Au from a partial certificate, which should have been 0.795 gpt Au from the final certificate, which error has been rectified);

- conversion errors from troy ounces per short ton to grams per tonne in the Billiton Minerals database were rectified; and
- there are 506 specific gravity ("SG") results in the master database that was compiled from the laboratory certificates, no errors were found during the verification process.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Eight metallurgical testwork programs have been carried out on samples of mineralized material from the Moss deposit (Table 13.1). Details and an analysis of the testwork programs and their outcomes are presented in a consultancy report to the Company by Stephen Godden, Independent Mining Consultant entitled ‘‘Moss Mine Gold-Silver Project, Mineralogical and Metallurgical Review,’ and dated November 23, 2014. Summaries of the test programs are presented in the following sections. An interpretation of the results (the Qualified Person’s opinion) is presented in Section 13.10.

A mixture of metric an US Customary units were used in the various original test program reports. The metric system of units is used here although, for the sake of clarity, US Customary units are sometimes stated in parentheses. Tyler mesh sizes are stated, along with sieve apertures in metric units.

Table 13.1: A Summary of Metallurgical Testwork Programs on Samples of Mineralized Material from the Moss Deposit

(compiled from information contained in the various testwork program reports cited in the following Sub-Sections)

Report Date	Laboratory	Test Program		
		Bottle Roll	Column Leach	Other Tests
December 1990	Billiton Minerals	-	-	Gravity separation
May 1991	McClelland Laboratories	15	-	Head & tail analysis (Au only)
January 1992	McClelland Laboratories	2	-	Head & tail analysis (Au and Ag)
June 2008	Metcon Research	4	3	Head & tail screen analysis Particle size vs. recovery analysis
January 2010	Kappes, Cassidy & Associates	2	4	Head & tail analysis Head screen analysis Cyanide shake tests
November 2012	Kappes, Cassidy & Associates	28	4	Head analysis Head & tail screen analysis Cyanide shake tests Variability testing
July 2012	Kappes, Cassidy & Associates	2	-	Head & tail analysis
February and April 2013	McClelland Laboratories	6	3	Head analysis Head & tail screen analysis
Totals	-	***59***	***14***	-

13.1 Billiton Minerals, 1990

Billiton Minerals (“Billiton”) carried out a preliminary metallurgical test program in 1990, the results of which are detailed in Baum and Lherbier (1990). Portions of the report are missing and are not, therefore, available for review.

Bottle roll tests are noted in Baum and Lherbier (1990), but no information is available (the relevant report sections are missing). The report sections relating to gravity testwork are, however, available: tests were carried out on 800 g splits from samples 444-1-2 and 444-3 (locations unknown), by means of heavy liquid separation at a specific gravity of 2.95. Prior to separation the sample splits were crushed to minus 48 mesh (0.30 mm) and deslimed at 400 mesh (0.037 mm). The particle size distribution of the gold grains is summarized on in Sub-Section 7.2.5.4. In the case of sample 444-1-2, it was reported that ‘*almost half the gold reported to the -400 mesh slimes fraction*’, which ‘*represents 30% of the sample weight*’. 28.7% Of the gold occurred in ‘*the* *gravity tailings (float fraction) due to encapsulation in (light) silica gangue*'. The balance of the gold (26.8%) reporting to the gravity concentrate that assayed 38 oz Au per short ton. In contrast, the majority (69.4%) of the somewhat coarser gold contained in the 444-3 sample split was recovered in a high-grade gravity concentrate (71.8 oz Au per short ton), with 18% reporting to the float fraction and 12% reporting to the minus 400 mesh slimes.

The results of the gravity tests may be expected, due to the fineness and deportment of the gold mineralization described in Sub-Section 7.2.5. Gravity separation was not considered further in any of the subsequent metallurgical testwork programs, due to the likely highly variable recovery rate and amount of work (hence cost) required to liberate the gold grains.

13.2 McClelland Laboratories, 1991

The first direct agitation (bottle roll) cyanidation tests on mineralized material from the Moss deposit, for which data is available, were carried out in 1991, by McClelland Laboratories. The report, prepared for Magma Copper Company, is packaged with the Baum and Lherbier (1990) report.

A total of fifteen, 96 hour bottle roll tests were carried out on one bulk ore sample (one test) and on 14 cuttings intervals from RC drillholes located on the Moss Mine Project area. The objective was to determine gold recovery, recovery rates and reagent requirements. Table 13.2 summarizes the cutting intervals by drillhole, sample number and drillhole depth.

Table 13.2: A Summary of the Metallurgical Drillhole Samples, McClelland Laboratories, 1991 Test Program, Moss Mine Project

(compiled from information contained in McClelland Laboratories' 1991 to Magma Copper Corporation)

RC Drillhole	Sample #	From (m)	To (m)	Sample Length (m)
MM-1	30	47.24	48.77	1.53
	64	96.01	97.54	1.53
MM-2	22	35.05	36.58	1.53
	23	36.58	38.10	1.52
	31	45.72	47.24	1.52
	33	48.77	50.29	1.52
	36	53.34	54.86	1.52
	37	54.86	56.39	1.53
MM-4	18	25.91	27.43	1.52
MM-8	30	44.20	45.72	1.52
	49	73.15	74.68	1.53
	56	83.82	85.34	1.52
MM-14	72	108.20	109.73	1.53
MM-18	58	86.87	88.39	1.52

The bulk ore sample was evaluated at P_{100} 25.4 mm (1") feed size. The drillhole cuttings intervals were evaluated at the as-received feed size, for which no particle size data is available. For purposes of analysis and based on observations made of RC drillhole cuttings from other Moss Mine Project holes, the nominal feed size was assumed to be P_{100} 12.7 mm (1/2").

13.2.1 Sample Preparation and Head Assays

The crushed bulk ore sample was '*thoroughly blended and split to obtain a* (3.0 kg) *sample for bottle roll testing*', as well as samples for triplicate direct head assay. The RC drillhole cuttings intervals were '*thoroughly blended and split to obtain* (one kilogram) *samples for bottle roll testing'*. The cuttings intervals from drillhole MM-2 only were split further to obtain a 300 g sample for direct head assay. Head samples were in each case assayed using the fire assay method to determine gold content only.

13.2.2 Test Procedures

The samples for testing were mixed with water to achieve 40% by weight solids. Natural pulp pH values were measured and lime was added to adjust the pH of the pulps to 11.0, before adding sodium cyanide solution at a concentration of 1.0 kg/t of solution.

Leaching was carried out by rolling the pulps in bottles on laboratory rolls for 96 hours; rolling was suspended after 2, 6, 24, 48 and 72 hours to allow the pulps to settle and to enable samples of the pregnant solution to be taken for analysis by Atomic Absorption ("AA") methods. At each planned rolling break, pregnant solution volumes were measured, cyanide concentrations and pH levels were determined. Make-up water, equivalent to that withdrawn for sampling, was then added to the pulps, cyanide concentrations were restored to initial levels and lime was added, as necessary, to maintain the leaching pH at between 10.8 and 11.2. Rolling was then resumed.

After 96 hours rolling ceased and the pulps were filtered to separate liquids and solids. Final pregnant solution volumes were measured and sampled for analysis; final pH and cyanide concentrations were determined. The leached residues were washed, dried weighed and assayed (in triplicate) to determine residual gold contents.

13.2.3 Results

Table 13.3 summarizes the bottle roll test results, from which it may be seen that:

- modest to good gold recoveries were achieved (after 96 hours they ranged from 51.9% to 78.1%, with an average of 62.4%); but
- gold recovery was fairly rapid (extraction was substantially complete in six to 24 hours, after which additional gold was recovered, but at a much diminished rate);
- cyanide consumptions were low to moderate (0.05 kg/t to 0.65 kg/t) - most consumption occurred early in the leaching cycles; and
- at 1.7 kg/t to 2.95 kg/t, lime consumption was low to moderate (an average of 82.5% of the lime was added at the start of the leaching cycles).

Figure 13.1 summarizes the gold metallurgical recovery curves for the fourteen P_{100} 12.7 mm RC cuttings samples tested. Figure 13.2 is a snapshot view of the Moss Vein, looking north, on which the locations of the bottle roll tested samples are identified. It may be seen that, with the exception of samples MM-1-64 and MM-8-56 that yielded higher overall recovery rates, very good repeatability between the tests was achieved (final recoveries varied between 51.9% and 66.7%, with an average of 60.0%). Although the distribution of the metallurgical samples was limited (Figure 13.2), the results provide the first indication of the likely repeatability of the metallurgical response of Moss Vein mineralization to cyanidation. A test program directed at assessing metallurgical variability by Metcon in 2008 and by Kappes, Cassidy & Associates ("KCA") in 2011/2012 establish in more detail the repeatability of metallurgical response (Sections 13.4 and 13.6).

No ready explanation for the comparatively higher recovery rates from samples MM-1-64 and MM-8-56 can be found. However, it might be due to a comparatively more favourable (i.e. finer) overall particle size. There is no direct evidence to support this assumption, but for the reasons later identified, there is a direct and clear relationship between particle size and metallurgical recovery of both gold and silver from mineralized material from the Moss Vein.

Table 13.3: A Summary of Bottle Roll Test Results, McClelland Laboratories, 1991 Test Program, Moss Mine Project

(compiled from data contained in McClelland Laboratories' 1991 report to Magma Copper Corporation)

Parameter	Sample							
	Bulk	MM-1-30	MM-1-64	MM-2-22	MM-2-23	MM-2-31	MM-2-33	MM-2-36
% Au extracted in 2 hours	10.2	20.0	13.3	22.6	28.2	19.4	38.4	29.5
…. 6 hours	18.4	38.0	33.9	39.3	42.5	33.1	52.0	41.2
….24 hours	29.0	49.7	63.3	47.4	54.3	45.0	59.0	50.2
….48 hours	34.7	54.9	71.7	49.3	58.9	50.0	63.0	54.4
….72 hours	38.6	57.4	74.7	51.1	61.8	52.7	63.9	55.8
….96 hours	42.1	60.0	75.0	51.9	64.3	53.8	64.6	58.1
Sample Data								
Feed Size	P_{100} 25.4 mm	P_{100} 12.7 mm						
Extracted Au (g/t)	3.086	0.720	0.926	0.480	0.617	0.960	1.817	0.857
Tail Grade (g/t)*	8.948	0.480	0.309	0.446	0.343	0.823	0.994	0.617
Calculated Head (g/t)	7.337	1.200	1.234	0.926	0.960	1.783	2.811	1.474
Head Assay (g/t)	-	-	-	0.960	0.891	1.577	3.189	1.303
Predicted Head (g/t)	8.948	0.994	1.577	1.783	1.097	1.886	2.434	1.303
Chemistry								
NaCN Consumption (kg/t)	0.075	0.12	0.05	0.07	0.135	0.645	0.43	0.23
Lime Added (kg/t)	1.70	2.15	2.95	2.95	2.30	2.55	2.30	2.10
Final pH	11.0	11.1	11.1	10.9	10.8	10.9	10.9	10.9
Natural pH (40% Solids)	7.1	7.8	7.8	7.8	7.6	7.7	7.7	7.7
Parameter	**Sample**							
	MM-2-37	MM-4-18	MM-8-30	MM-8-49	MM-8-56	MM-14-72	MM-18-58	-
% Au extracted in 2 hours	27.1	20.3	21.0	23.3	36.9	14.8	22.2	-
…. 6 hours	38.1	33.3	39.5	38.7	57.8	29.4	40.4	
….24 hours	47.9	44.1	52.9	54.3	70.3	45.4	55.7	
….48 hours	52.6	48.8	59.5	61.0	72.5	51.5	61.2	
….72 hours	54.5	51.4	63.8	64.5	74.7	54.6	64.4	
….96 hours	57.1	53.4	66.7	66.5	78.1	57.5	66.4	
Sample Data								
Feed Size	P_{100} 12.7 mm							
Extracted Au (g/t)	0.823	1.063	0.480	4.286	0.857	1.577	3.051	-
Tail Grade (g/t)*	0.617	0.926	0.240	2.160	0.240	1.166	1.543	
Calculated Head (g/t)	1.440	1.989	0.720	6.446	1.097	2.743	4.594	
Head Assay (g/t)	1.303	-	-	-	-	-	-	
Predicted Head (g/t)	1.337	1.920	0.549	6.343	0.960	2.640	4.354	
Chemistry								
NaCN Consumption (kg/t)	0.16	0.235	0.14	0.365	0.375	0.305	0.22	-
Lime Added (kg/t)	1.95	1.90	2.15	1.80	2.50	2.50	1.95	
Final pH	10.8	10.7	10.8	10.8	10.9	11.1	10.9	
Natural pH (40% Solids)	7.7	8.0	8.2	8.1	8.3	8.3	8.4	

Note: * - average of three assays

13.3 McClelland Laboratories, 1992

Late in 1991 McClelland Laboratories carried out bottle roll tests on two cuttings intervals from RC drillholes from the Moss Mine Project: drillhole MC-5, Sample 56 (83.82 m to 85.34 m); and drillhole MC-14, Sample 28 (45.15 m to 42.67 m). The objective was to determine precious metal recovery, recovery rates and reagent requirements. McClelland Laboratories' report, prepared for Magma Copper Company, is dated January 29, 1992 and is packaged with the Baum and Lherbier (1990) report.

Figure 13.1: Gold Metallurgical Recovery Curves for Fourteen RC Cuttings Samples, McClelland Laboratories, 1991 Test Program, Moss Mine Project
(compiled from data contained in McClelland Laboratories' 1991 report to Magma Copper Corporation)

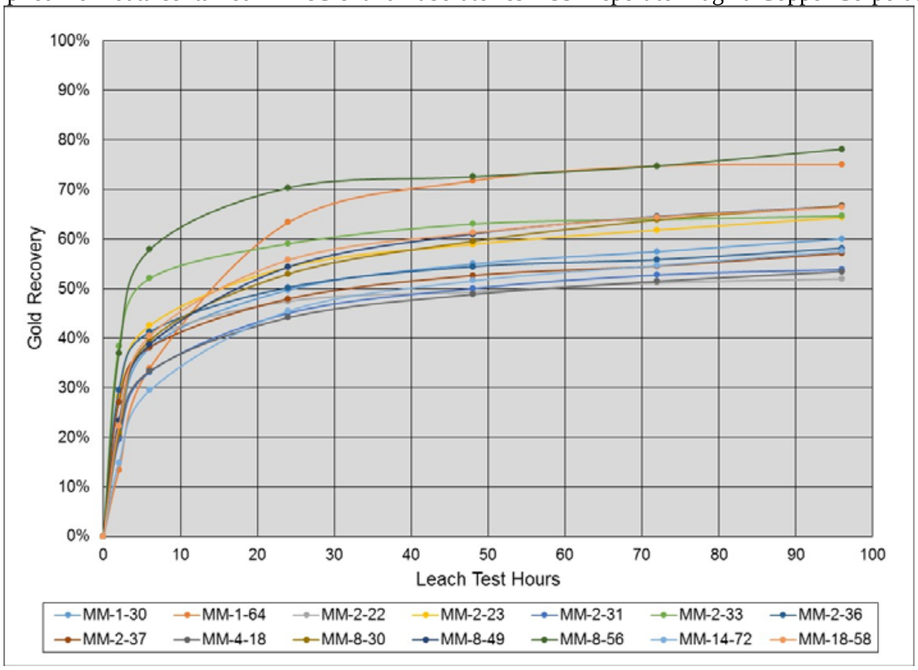
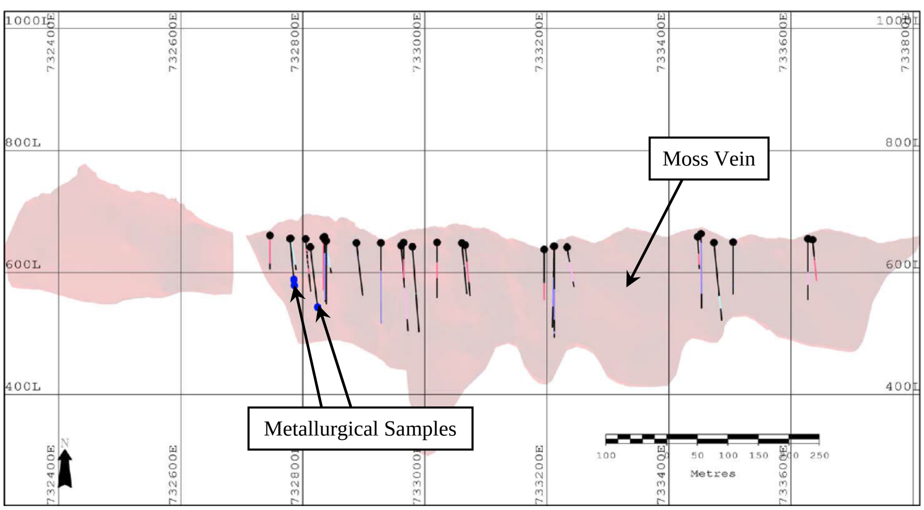


Figure 13.2: A Vulcan® Snapshot View of the Moss Vein (looking north) Showing the Positions of the Drillhole Samples used for Bottle Roll Testing (highlighted in BLUE), McClelland Laboratories 1991 Test Program, Moss Mine Project



A total of two bottle roll tests, each of 96 hour duration at 40% solids, were carried out on the as-received feed size, stated as nominal 10 mesh (1.70 mm). Both intervals were '*thoroughly blended and split to obtain* (one kilogram) *samples for bottle roll testing*' and a sample for direct head assay using the fire assay technique. In all respects the bottle roll test procedures were identical to those described above for McClelland Laboratories' 1991 test program. The results are summarized on Table 13.4, from which it may be seen that:

- very good gold recoveries were achieved (after 96 hours they were 78.7% and 87.9%, with silver recoveries of 59.4% and 70.0%, respectively), which results were significantly better than those achieved in 1991 for the coarser, P_{100} 12.7 mm RC cutting samples; and
- gold recovery was fairly rapid (Figure 13.3 – the majority of the gold was in both cases extracted in less than 10 hours and extraction was substantially complete within 24 hours, after which additional gold was recovered, but at a much reduced rate);
- cyanide consumption was low (it averaged 0.26 kg/t, with most consumption occurring early in the leaching cycles); and
- at an average of 1.65 kg/t, lime consumption was low (an average of 80.0% of the lime was added at the start of the leaching cycles, with the balance added during the leaching cycles).

Table 13.4: A Summary of Bottle Roll Test Results, McClelland Laboratories, 1992 Test Program, Moss Mine Project
(compiled from information contained in McClelland Laboratories' 1992 report to Magma Copper Company)

Parameter	Sample			
	MC-5 (56)		MC-14 (28)	
	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted in 2 hours	41.2	35.0	31.7	15.2
…. in 6 hours	78.2	50.0	59.6	26.8
…. in 24 hours	83.0	62.0	73.4	43.6
…. in 48 hours	85.5	67.0	76.8	51.3
…. in 72 hours	87.0	69.0	78.1	55.8
…. in 96 hours	87.9	70.0	78.7	59.4
Base Data				
Feed Size	P_{80} 10 mesh (1.7 mm)			
Extracted Au (g/t)	0.994	2.40	1.269	14.06
Tail Grade (g/t)*	0.137	1.03	0.343	9.60
Calculated Head (g/t)	1.131	3.43	1.611	23.66
Head Assay (g/t)	1.166	4.46	1.749	19.20
Chemistry				
Cyanide Consumption (kg/t)	0.29		0.23	
Lime Added (kg/t)	1.40		1.85	
Final pH	11.1		11.0	
Natural pH (40% Solids)	8.4		8.4	

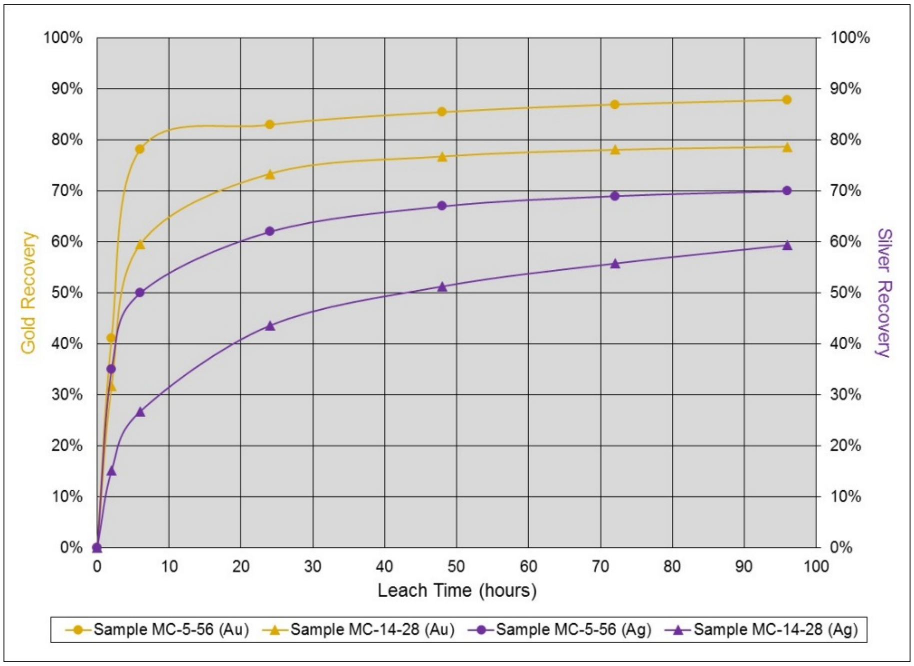
Note: * - average of three assays

13.4 Metcon Research, 2008

In 2008 Metcon Research ("Metcon", of Tuscon, Arizona) completed a metallurgical test program, on behalf of Patriot Gold, on two samples identified as oxide and sulphide composites. The test program is detailed in Metcon's report dated June 2008.

The primary objective of the test program was to generate gold and silver extraction and cyanide and lime consumption data at two different crush sizes. The leach tests were carried out utilizing bottle roll and column leach techniques in locked cycle. The oxide composite comprised material from two drillhole intervals - AR-48C (9.14 m to 62.5 m) and AR-49C (13.72 m to 64.01 m). The sulphide composite comprised material from drillhole AR-50C (102.1 m to 125.9 m). The average grades of the composite samples were 1.147 g/t Au and 13.3 g/t Ag ('oxide' sample) and 1.264 g/t Au, 20.2 g/t Ag for the 'sulphide' sample.

Figure 13.3: Metallurgical Recovery Curves for Bottle Roll Tested, P_{80} 10 Mesh Samples, McClelland Laboratories, 1992 Test Program, Moss Mine Project
(compiled from data contained in McClelland Laboratories' 1992 report to Magma Copper Company)



13.4.1 Bottle Roll Tests

Bottle roll tests were carried out on P_{100} 10 mesh (1.70 mm) samples and on P_{100} 150 mesh (0.105 mm) samples. The bottles were agitated on laboratory rolls for 96 hours, at a pulp density of 33.3% solids, using a leach solution containing 1.0 g/L of sodium cyanide; the pH was maintained at between 11.0 and 11.5, using hydrated lime. Rolling was temporarily suspended after 6, 24 and 48 hours to allow the pulps to settle and to enable samples of the pregnant solution to be taken for analysis using AA methods. At each planned rolling break, pregnant solution volumes were measured, cyanide concentrations and pH levels were determined. Make-up water, equivalent to that withdrawn for sampling, was then added to the pulps, cyanide concentrations were restored to initial levels and lime was added, as necessary, to maintain the leaching pH at between 10.8 and 11.2. Rolling was then resumed. Table 13.5 summarizes the results, from which it may be concluded that:

- there little difference in the recovery rates for the samples identified as 'oxide' and 'sulphide' (which outcome may be expected, for the reasons discussed in Sub-Section 7.2.6);
- moderate to good recovery rates were achieved for the P_{100} 1.70 mm samples (after 96 hours, gold recoveries of 63.8% and 67.2% were achieved along with silver recoveries of 37.4% and 56.5%, which were significantly less than those achieved for the same nominal particle size in McClelland Laboratories' 1992 test series);
- exceptional results were achieved for the 150 mesh samples (after 96 hours they were 97.07% and 92.20% for gold and 79.43% and 83.06% for silver); and

- gold recovery from the P_{100} 150 mesh samples in particular was rapid (the majority of the contained metal was in each case extracted to pregnant solution in approximately 10 hours, at 24 hours extraction was substantially complete, although additional metal was recovered, but at a much slower rate – see Figure 13.4).

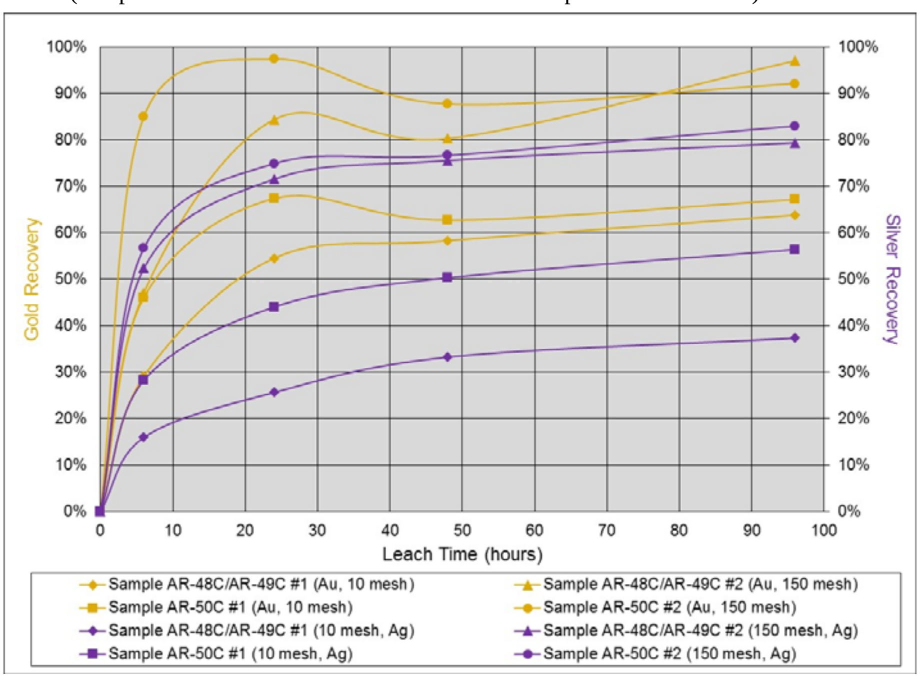
Table 13.5: A Summary of Bottle Roll Test Results,
Metcon Research, 2008 Test Program, Moss Mine Project
(compiled from data contained in Metcon's 2008 to Patriot Gold)

Parameter	Sample							
	AR-48C/AR -49C #1		AR-48C/AR -49C #2		AR-50C #1		AR-50C #2	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted in 6 hours	29.14	15.99	47.17	52.42	46.16	28.29	85.10	56.76
…. in 24 hours	54.58	25.65	84.43	71.58	67.44	44.07	97.46	74.93
…. in 48 hours	58.34	33.27	80.36	75.58	62.78	50.33	87.79	76.70
…. in 96 hours	63.87	37.37	97.07	79.43	67.22	56.46	92.20	83.06
Base Data								
Feed Size	P_{100} 10 mesh (1.70 mm)		P_{100} 150 mesh (0.15 mm)		P_{100} 10 mesh (1.70 mm)		P_{100} 150 mesh (0.15 mm)	
Tail Grade (g/t)*	0.45	10.50	0.04	3.40	0.27	6.90	0.07	3.00
Calculated Head (g/t)	1.24	16.60	1.36	16.40	0.82	15.80	0.89	17.60
Head Assay (g/t)	1.27	15.10	1.27	15.10	0.72	13.00	0.72	13.00
Chemistry								
Cyanide Consumption (kg/t)**	-0.05		-0.04		0.02		-0.05	
Lime Added (kg/t)	0.82		1.03		0.65		0.70	
Final pH	11.29		11.21		11.36		11.43	

Note: * - average of three assays

** - Metcon attributed the negative cyanide consumptions to '*analytical technique precision*'

Figure 13.4: Metallurgical Recovery Curves for Bottle Roll Tested, P_{100} 10 Mesh and P_{100} 150 Mesh Samples, Metcon Research, 2008 Test Program, Moss Mine Project
(compiled from data contained in Metcon's 2008 report to Patriot Gold)



The significant difference in the metallurgical recovery rates for P_{80} 1.70 mm material tested by McClelland Laboratories in 1992 (average 65.5% Au and 46.9% Ag versus 83.3% Au and 64.7% Ag for the McClelland Laboratories' tests) is probably due to the very low cyanide concentrations in Metcon's tests (<0.02 kg/t versus an average of 0.26kg/t for McClelland Laboratories' tests). The very low cyanide consumptions reported for the bottle roll tests on 150 mesh material does not, however, appear to have affected the results (exceptional recoveries were achieved). This tends to suggest gold and silver mineralization that is highly amenable to cyanidation.

13.4.2 Column Leach Tests

The column leach tests were carried out on the samples identified as 'oxide', at crush sizes of P_{80} 25.4 mm (1"), P_{80} 12.7 mm (½") and P_{80} 6.35 mm (¼"). Single test charges from oxide composite sample were reconstituted and loaded into 10 cm (4 inch) diameter PVC columns to a height of approximately 3.5 m. Prior to loading the columns, the sample was agglomerated with water and lime.

After agglomerating and loading the test charges, the columns were allowed to rest for six days, after which the loaded columns were subjected to locked cycle leaching using a feed solution containing 0.5 g/L of sodium cyanide at a pH of approximately 11.5. Throughout the leach cycle the pH of the feed solution was adjusted, using hydrated lime, to maintain an effluent pH of between 11.0 and 11.5.

Lime was blended into test charge at a dosage of 0.8 kg/t (100% addition of the preliminary bottle roll consumption). The cyanide concentration was 0.5 g/L and the leach solution application flow rate was 12 L/hr/m^2. Sixty days of continuous leaching was followed by intermittent leaching (one week on/one week off) for a leach cycle total of 109 days.

The leached residues from the column tests were screened using the same sieve sizes as used to prepare the head screen assay analyses. Sample pulps from each leach residue screen fraction were submitted for gold and silver assays and the assays results from the head and leach residue screen analyses were utilized to calculate extraction by screen fraction.

Table 13.6 summarizes the results of Metcon's locked cycle column leach tests, from which it may be seen that:

- at 38.7% to 66.3%, gold extraction rates were modest, as were the silver recovery rates (14.1% to 42.1%); but
- there are step-wise changes in gold and silver recovery from the P_{80} 25.4 mm material through the P_{80} 12.7 mm material to the P_{80} 6.35 mm material;
- gold recovery from the P_{80} 12.7 mm sample is at the bottom end of the range of results for the majority of the P_{100} 12.7 mm samples tested during McClelland Laboratories' 1991 bottle roll test program (excluding the two outliers reflected in the results for samples MM-1-64 and MM-8-56, gold recoveries of between 51.9% and 66.7% were achieved); and
- cyanide consumption in particular was very low, with slight reductions in both lime and cyanide consumption from the P_{80} 25.4 mm test to the P_{80} 6.35 mm test.

The generally moderate recovery rates outlined may be attributed to the very low cyanide consumptions that typically are much higher in column leach tests than in bottle roll tests. Metcon's test results do not reflect this fundamental difference in the test types which, in some respects, renders the results non-representative in terms of metal recovery potential.

Table 13.6: A Summary of Column Leach Test Results, Metcon Research, 2008 Test Program, Moss Mine Project
(compiled from data contained in Metcon's 2008 report to Patriot Gold')

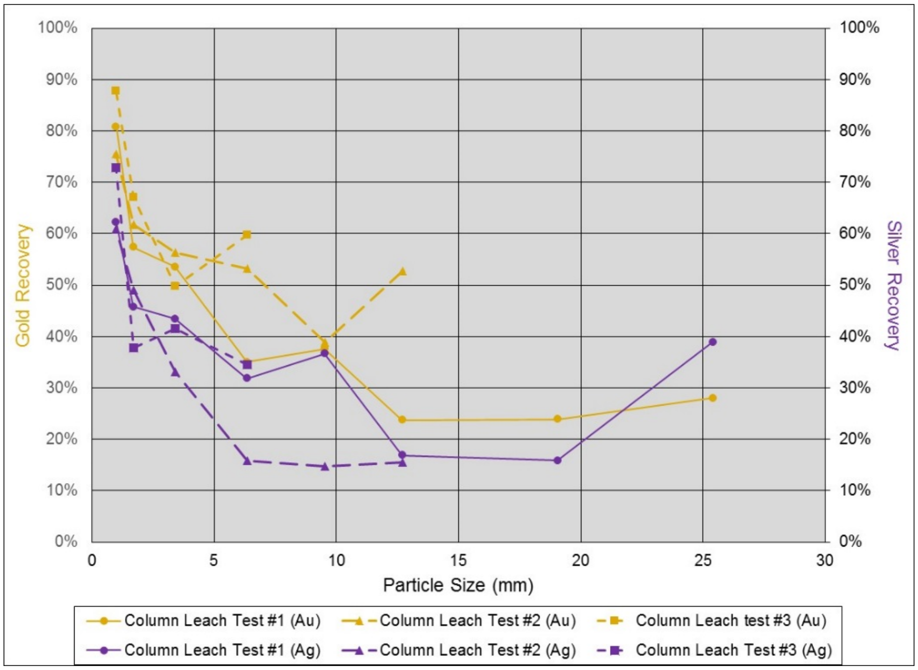
Test	Nominal Crush Size	Metallurgical Products	Volume/Weight (L/kg)	Assays (g/t)		% Extraction		Reagent Consumption (kg/t)	
				Au	Ag	Au	Ag	NaCN	CaO
#1	P_{80} 25.4 mm (1")	Feed Solution	209.31	0.02	0.15	38.66	14.12	0.09	0.96
		Pregnant Solution	210.48	0.13	0.51				
		Leach Residue	39.69	0.89	11.85				
		Calculated Head	-	1.44	13.68				
		Assay Head	40.04	1.25	13.58				
#2	P_{80} 12.7 mm (½")	Feed Solution	208.43	0.02	0.06	51.95	24.16	0.07	0.87
		Pregnant Solution	209.03	0.14	0.75				
		Leach Residue	39.69	0.59	11.28				
		Calculated Head	-	1.21	14.75				
		Assay Head	40.01	1.22	14.88				
#3	P_{80} 6.35 mm (¼")	Feed Solution	208.32	0.03	0.38	66.31	42.11	0.06	0.85
		Pregnant Solution	209.42	0.19	1.47				
		Leach Residue	39.42	0.44	7.96				
		Calculated Head	-	1.30	13.54				
		Assay Head	40.03	1.27	15.14				

Table 13.7 summarizes the results of the particle analysis carried out on the three nominal crush sizes and compares this with the distribution of sieve sample weights, gold and silver assays, and recoveries. It may again be seen that gold and silver extraction is sensitive to particle size, with the progressively finer fractions from the nominal crush size reporting ever-higher recovery rates. Figure 13.5 emphasizes the relationship between particle size and recovery.

Table 13.7: A Summary of Sieve Fraction Analysis Results, Metcon Research, 2008 Test Program, Moss Mine Project
(compiled from data contained in Metcon's 2008 report to Patriot Gold)

Screen Fraction (mm)	Sample Weight		Gold				Silver			
	(kg)	Distrib. (%)	Head (g/t)	Content (g)	Distrib. (%)	Recovery (%)	Head (g/t)	Content (g)	Distrib. (%)	Recovery (%)
Column Leach Test #1 (P_{80} 25.4 mm)										
25.40	8.01	20.00	1.07	0.0086	17.13	28.04	10.8	0.0865	15.91	3.89
19.05	19.15	47.83	1.24	0.0237	47.47	23.86	13.8	0.2643	48.61	15.85
12.70	6.01	15.01	1.43	0.0086	17.18	23.74	14.7	0.0883	16.25	16.89
9.53	1.76	4.40	1.46	0.0026	5.14	37.56	16.8	0.0296	5.44	36.69
6.35	1.77	4.42	1.43	0.0025	5.06	34.98	15.8	0.0280	5.14	31.80
3.40	1.38	3.45	1.12	0.0015	3.09	53.57	15.9	0.0219	4.04	43.40
1.70	0.71	1.77	1.19	0.0008	1.69	57.39	15.9	0.0113	2.08	45.79
<1.70	1.25	3.12	1.30	0.0016	3.25	80.80	11.0	0.0138	2.53	62.18
Total & Averages	*40.04*	*100.00*	*1.25*	*0.0500*	*100.00*	*29.16*	*13.58*	*0.5436*	*100.00*	*13.49*
Column Leach Test #2 (P_{80} 12.7 mm)										
12.70	8.03	20.07	0.95	0.0076	15.62	52.78	12.8	0.1028	17.28	15.44
9.53	10.44	26.09	1.33	0.0139	28.43	38.91	15.8	0.1650	27.71	14.69
6.35	8.15	20.37	1.35	0.0110	22.53	53.14	16.7	0.1361	22.86	15.86
3.40	5.70	14.25	1.33	0.0076	15.52	56.39	16.0	0.0912	15.32	33.13
1.70	2.94	7.35	1.44	0.0042	8.67	61.81	14.9	0.0438	7.35	48.99
<1.70	4.75	11.87	0.95	0.0045	9.24	75.43	11.9	0.0565	9.49	60.77
Total & Averages	*40.01*	*100.00*	*1.22*	*0.0488*	*100.00*	*52.35*	*14.88*	*0.5954*	*100.00*	*24.81*
Column Leach Test #3 (P_{80} 6.35 mm)										
6.35	8.05	20.11	1.20	0.0097	19.00	59.78	14.8	0.1191	19.66	34.44
3.40	13.48	33.67	1.23	0.0166	32.60	49.79	17.0	0.2292	37.82	41.50
1.70	6.49	16.21	1.72	0.0112	21.95	67.14	13.8	0.0896	14.78	37.84
<1.70	12.01	30.00	1.12	0.0135	26.45	87.72	14.0	0.1681	27.75	72.75
Total & Averages	*40.03*	*100.00*	*1.27*	0.0509	*100.00*	*65.53*	*15.1*	*0.6060*	*100.00*	*48.24*

Figure 13.5: A Scatter Plot of Particle Size vs. Gold and Silver Recovery, Metcon Research, 2008 Test Program, Moss Mine Project
(compiled from data contained in Metcon's 2008 report to Patriot Gold)



13.5 Kappes, Cassidy & Associates, 2010

In 2010 Kappes, Cassidy & Associates ("KCA") carried out a metallurgical test program on mineralized samples from the Moss deposit. The program comprised two bottle roll tests and four column leach tests. The program and its results are presented in KCA's March 2011 report to Patriot Gold.

13.5.1 Sample Preparation

KCA was supplied with 124 drillcore samples, from which two composite samples were created utilizing 88 of the received drillcore samples: Composite 1 comprised mineralized material from drillhole AR-51C; and Composite 2 comprised mineralized material from drillholes AR-52C and AR-53C (Table 13.8).

Table 13.8: A Summary of KCA's #1 Composite, by Drillhole Interval And Grade, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)

Drillhole	Drillhole Interval (m)			Interval Weight		Assays (g/t)	
	From	To	Length	kilograms	Distrib. (%)	Au	Ag
AR-51C	51.82 77.72	59.44 134.11	7.61 56.24	20.22 151.57	11.77 88.24	0.511 0.807	4.5 8.5
#1 Composite Totals & Averages				*171.79*	*100%*	*0.772*	*8.0*
AR-52C AR-53C	35.05 54.86	80.77 56.39	45.61 21.28	119.24 57.57	100 100	0.857 0.505	10.9 10.1
#2 Composite Totals & Averages				*176.81*	*100%*	*0.742*	*10.7*

The two composites were separately but identically prepared:

- the samples comprising the composites were blended and screened at 50.8 mm (2”), with any oversize material crushed to minus 50.8 mm and then mixed with the screened undersize fraction;
- each sample was then screened at 50.8 mm, 44.45 mm, 38.10 mm, 31.75 mm, 25.4 mm and 12.7 mm (2”, 1.75”, 1.5”, 1.25”, 1” and 0.5”), and size adjusted; and
- following size adjustment, portions from each separate size fraction were recombined and utilized as follows –
    - the 100% passing 50.8 mm sample material was labeled ‘A’ (approximately 172.4 kg or 380 lb), which sample was coned three times and then quartered,
    - opposite quarters were combined and utilized for column and bottle roll leach tests (approximately 86.2 kg or 190 lb),
    - from one quarter (approximately 43.1 kg or 95 lb) a 20.4 kg (45 lb) split was taken and utilized for a head screen analysis,
    - the second quarter was combined with the reject material from the previous quarter and stage-crushed to 100% passing 12.70 mm and labeled ‘B’ (totalling approximately 65.8 kg or 145 lb), and
    - Sample B was then screened at 12.70 mm, 9.53 mm, 7.92 mm, 6.35 mm, 3.35 mm, 1.68 mm, 0.59 mm and 0.21 mm (1/2”, 3/8”, 5/16”, 1/4”, 1/8”, 10 mesh, 28 mesh and 65 mesh), and size adjusted.

13.5.2 Head Screen Analysis

The weights for each size fraction of the screened samples (to 65 mesh, 0.21 mm) were recorded and from each size fraction two equal portions were split and pulverized to P_{80} 200 mesh (0.074 mm). The pulverized sample splits were then assayed for gold, using the fire assay method, and for silver using wet chemistry methods. Table 13.9 summarizes the weighted average results of the head screen analyses. The calculated P_{80} crush sizes for the prepared samples were: Sample 1A - 35.6 mm (1.4”); Sample 1B - 10.2 mm (0.4”), Sample 2A - 30.5 mm (1.2”); and Sample 2B - 10.2 mm (0.4”).

The results show an approximately even spread of gold and silver grades across the size fractions, which result may be expected by virtue of the style of gold-silver mineralization and its deportment, as described in Sub-Section 7.2.4.

13.5.3 Cyanide Shake Tests

KCA carried out a total of four preliminary cyanide shake tests on splits of Composite #1 and #2 (two samples run in duplicate) that were pulverized to a target size of P_{80} 200 mesh (0.074 mm). The tests included the following elements:

- a 15.0 g portion of the sample material was placed into a 100 mL centrifuge tube;
- 30 mL of a sodium cyanide solution were added to the tube (a strong solution of 5.0/L of water); and

- the centrifuge tube was agitated for 24 hours, following which the slurry was checked for pH, Au, Ag and Cu.

The results of the tests are summarized on Table 13.10. It may be seen that consistently very good gold and silver recoveries were achieved (88% to 90% for gold and 82% to 93% for silver). The results were only slightly inferior to those achieved by Metcon in 2008 by bottle rolling 150 mesh samples (approximately 92% to 97% for gold and approximately 79% to 83% for silver).

Table 13.9: A Summary of Head Screen Analysis Results, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)

Sample #	Passing (mm)	Retained (mm)	Sample Weight (kg)	Distrib. (%)	Cumulative Weight (%)		Gold		Silver	
					Retained	Passing	g/t	Weight %	g/t	Weight %
1A P_{80} 35.6 mm (1.4")	-	44.45	0.00	0.0	-	-	-	-	-	-
	44.45	31.75	3.08	14.4	14.4	100.0	0.343	6.9	6.51	10.26
	38.10	25.40	5.78	27.0	41.4	85.6	0.583	21.7	7.20	21.31
	25.40	19.05	5.11	23.9	65.3	58.6	0.720	24.4	8.23	21.05
	19.05	9.53	1.41	6.6	71.9	34.7	0.960	9.0	14.74	10.68
	15.88	7.94	2.54	11.9	83.8	28.1	1.063	17.4	13.37	17.05
	9.53	6.35	1.21	5.7	89.5	16.2	0.891	7.1	12.34	7.51
	6.35	1.68	1.34	6.3	95.7	10.5	0.823	7.3	10.97	7.36
	1.68	0.59	0.41	1.9	97.7	4.3	0.994	2.7	9.94	2.06
	0.59	0.21	0.23	1.1	98.7	2.3	0.754	1.1	9.26	1.08
	0.21	Pan	0.27	1.3	100.0	1.3	1.406	2.5	12.00	1.65
Totals and Averages			***21.38***	***100.0***	***-***	***-***	***0.789***	***100.0***	***10.19***	***100.00***
1B P_{80} 10.2 mm (0.4")	-	19.05	0.00	0.0	-	-	-	-	-	-
	19.05	12.70	0.76	5.2	5.2	100.0	0.651	4.7	8.23	4.50
	12.70	9.53	2.69	18.4	23.6	94.8	0.720	18.5	12.34	23.76
	9.53	7.94	2.58	17.6	41.2	76.4	0.651	16.3	9.60	17.41
	7.92	6.35	2.56	17.5	58.7	58.8	0.583	14.6	9.60	17.19
	6.35	3.18	2.97	20.3	79.0	41.3	0.583	16.9	8.57	17.63
	3.35	1.68	1.18	8.0	87.0	21.0	0.823	9.5	8.91	7.32
	1.68	0.59	0.89	6.1	93.1	13.0	1.029	8.7	8.23	5.16
	0.59	0.21	0.55	3.7	96.8	6.9	0.754	4.0	8.57	3.32
	0.21	-	0.47	3.2	100.0	3.2	1.509	6.8	11.31	3.71
Totals and Averages			***14.64***	***100.0***	***-***	***-***	***0.779***	***100.0***	***10.70***	***100.00***
2A P_{80} 30.5 mm (1.2")	-	44.45	0.00	0.0	-	-	-	-	-	-
	44.45	31.75	1.14	5.2	5.2	100.0	0.137	1.3	3.09	1.79
	38.10	25.40	5.67	25.8	31.0	94.8	0.309	13.1	6.86	18.86
	25.40	19.05	7.13	32.5	63.5	69.0	0.686	35.9	9.26	31.62
	19.05	9.53	1.73	7.9	71.3	36.5	0.754	9.9	10.97	9.01
	15.88	7.94	3.03	13.8	85.1	28.7	1.029	22.8	14.74	21.47
	9.53	6.35	1.27	5.8	90.9	14.9	0.583	5.6	9.94	6.00
	6.35	1.68	1.20	5.5	96.4	9.1	0.720	6.4	11.66	6.77
	1.68	0.59	0.35	1.6	98.0	3.6	0.789	2.0	10.97	1.84
	0.59	0.21	0.20	0.9	98.9	2.0	0.720	1.1	9.60	0.94
	0.21	Pan	0.25	1.1	100.0	1.1	1.063	2.0	14.06	1.71
Totals and Averages			***21.97***	***100.0***	***-***	***-***	***0.678***	***100.0***	***10.43***	***100.00***
2B P_{80} 10.2 mm (0.4")	-	19.05	0.00	0.0	-	-	-	-	-	-
	19.05	12.70	0.81	4.7	4.7	100.0	1.509	8.4	22.29	8.61
	12.70	9.53	2.91	16.9	21.6	95.3	0.960	19.2	13.71	18.84
	9.53	7.94	2.99	17.4	39.0	78.4	0.720	14.9	11.66	16.51
	7.92	6.35	2.91	16.9	55.9	61.0	0.720	14.5	10.97	15.37
	6.35	3.18	3.45	20.0	75.9	44.1	0.823	19.7	11.66	19.21
	3.35	1.68	1.54	9.0	84.8	24.1	0.651	7.0	11.66	8.52
	1.68	0.59	1.23	7.2	92.0	15.2	0.857	7.3	9.94	5.87
	0.59	0.21	0.73	4.2	96.2	8.0	0.686	3.5	9.94	3.51
	0.21	-	0.65	3.8	100.0	3.8	1.234	5.5	11.66	3.56
Totals and Averages			***17.23***	***100.0***	***-***	***-***	***0.925***	***100.0***	***13.43***	***100.00***

Table 13.10: A Summary of Cyanide Shake Test Results, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)

Parameter	Sample							
	Composite 1, Split A		Composite 1, Split B		Composite 2, Split A		Composite 2, Split B	
% Metal extracted in 24 hours	Au	Ag	Au	Ag	Au	Ag	Au	Ag
	88	82	90	81	88	86	88	93
Base Data								
Feed Size	P_{80} 200 mesh		P_{80} 200 mesh		P_{80} 200 mesh		P_{80} 200 mesh	
Extracted Grade (g/t)	0.603	8.43	0.617	8.33	0.634	9.73	0.634	10.52
Tail Grade (g/t)	0.082	1.85	0.069	1.95	0.086	1.58	0.086	0.79
Head Assay (g/t)	0.686	10.29	0.686	10.29	0.720	11.31	0.720	11.31
Leach Parameters								
Cyanide Concentration (g/L)	5.0		5.0		5.0		5.0	
Final pH	10.1		10.0		10.0		10.0	

13.5.4 Bottle Roll Tests

KCA carried out 96 hour bottle roll cyanide leach tests on splits of Composites #1 and #2 that were pulverized to a target size of P_{80} 200 mesh (0.074 mm). The tests were carried out separately but identically:

- a 1,000 g portion of pulverized sample material was placed into a 2.5 L bottle and slurried with the addition of 1,500 mL of water;
- the slurry was mixed thoroughly and the pH checked and adjusted as necessary with hydrated lime to achieve a pH of 11.0;
- sodium cyanide was added to the slurry at a concentration of 1.0 g/L of added water;
- the bottles were placed onto a set of laboratory rolls and the slurries were checked at 2, 4, 8, 24, 48, 72 and 96 hours for pH, dissolved oxygen, cyanide concentration, gold, silver and copper; and
- following completion of the 96 hour leach period, the slurries were individually filtered, washed, dried and weighed; and
- two equal, 500 g portions of each dried slurries were split out, pulverized individually to a target size of P_{80} 200 mesh (0.074 mm) and then assayed for gold and silver.

The results are summarized on Table 13.11, from which it may be concluded that:

- exceptional extraction rates were achieved (90% and 93% for gold, and 86% and 95% for silver) –
  - the average gold extraction rate (91.5%) is very similar to Metcon's average gold recovery from the bottle roll tests on 150 mesh pulps (94.7%), whereas
  - the average silver recovery rate (89.5%) was somewhat higher than Metcon's average silver recovery from the same bottle roll tests (81.3%);
- the average gold extraction rate after 24 hours is nearly identical to the average extraction rate achieved by the cyanide shake tests (87.5% versus 88.5%);
- the average silver extraction rate after 24 hours (69.0%) was lower than that achieved by the cyanide shake tests (85.5%);

- gold recovery was rapid (the majority of the contained metal was in both cases extracted to pregnant solution in less than 10 hours, and at 24 hours extraction was substantially complete – see Figure 13.6);
- silver recovery was by comparison moderately fast (the majority of the contained metal was extracted to pregnant solution after approximately 12 hours to 15 hours, after which extraction continued to the end of the tests at 96 hours, at which point and in theory at least, the trends of the recovery curves suggest that additional silver could have been recovered); and
- at 0.065 kg/t and 0.5 kg/t, respectively, consumptions of cyanide and lime were low.

Table 13.11: A Summary of Bottle Roll Test Results, KCA, 2010 Test Program, Moss Mine Project

(compiled from data contained in KCA's 2011 report to Patriot Gold)

Parameter	Sample			
	Sample 1A		Sample 1B	
	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted in 2 hours	19	29	30	34
…. in 4 hours	40	38	53	43
…. in 8 hours	66	50	74	55
…. in 24 hours	88	67	87	71
…. in 48 hours	87	75	90	79
…. in 72 hours	90	76	91	80
…. in 96 hours	91	77	92	80
filtrate & wash	90	93	93	86
Base Data				
Feed Size	P_{80} 200 mesh		P_{80} 200 mesh	
Tail Grade (g/t)	0.069	0.69	0.069	1.71
Extracted Grade (g/t)	0.651	9.94	0.686	10.29
Calculated Head (g/t)	0.720	10.63	0.754	12.00
Head Assay (g/t)	0.686	10.29	0.720	11.31
Chemistry				
Cyanide Consumption (kg/t)	0.065		0.065	
Lime Consumption (kg/t)	0.5		0.5	
Final pH	10.8		10.8	

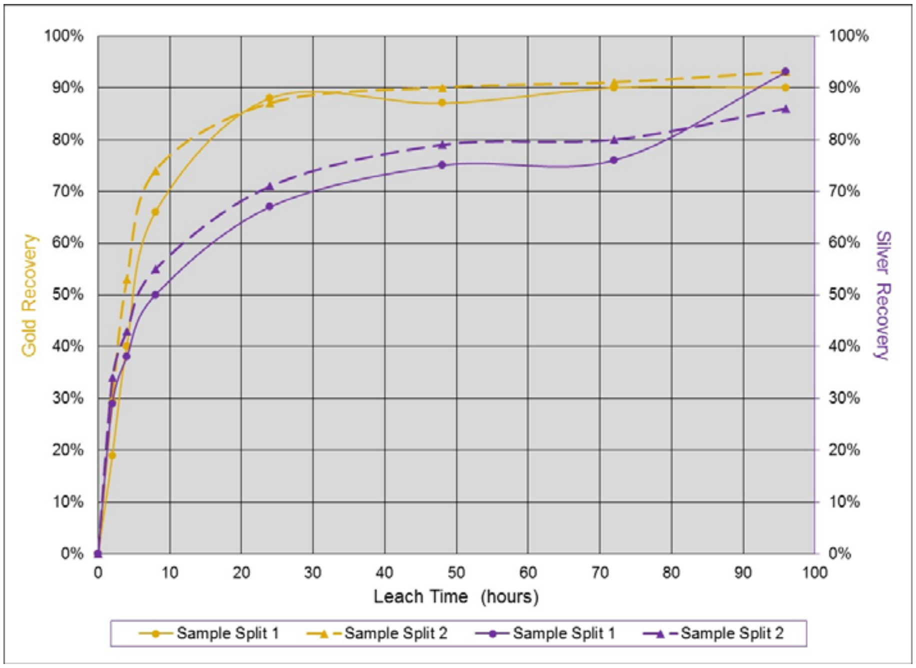
13.5.5 Column Leach Tests

As part of its 2010 program, KCA completed four column tests on Moss Vein material, which tests lasted between 207 and 208 days. The earlier described samples were used:

- Sample 1A with a crush size of P_{100} 50.8 mm and a calculated size of P_{80} 35.6 mm;
- Sample 1B with a crush size of P_{100} 12.7 mm and a calculated size of P_{80} 10.2 mm;
- Sample 2A with a crush size of P_{100} 50.8 mm and a calculated size of P_{80} 30.5 mm; and
- Sample 2B with a crush size of P_{100} 12.7 mm and a calculated size of P_{80} 10.2 mm.

The column tests were run as a continuously drained drip leach tests. The material to be leached was placed into a Plexiglas column and alkaline cyanide solution was continuously distributed onto the material through a header of Tygon tubing with glass capillary drip tubes. The solution flow rate dripping onto the material was controlled with a peristaltic pump, at a rate of approximately 0.16 to 0.20 L/minute/m^2 of column surface area.

Figure 13.6: Bottle Roll Test Metallurgical Recovery Curves for Pulverized Material, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)



The solution exiting the leach column was collected in the bottom tank where it was checked during each cycle for pH, sodium cyanide, gold and silver. Copper content was also checked periodically. The solution was then passed through a bottle of activated carbon over a period of 24 hours to recover the gold and silver in solution.

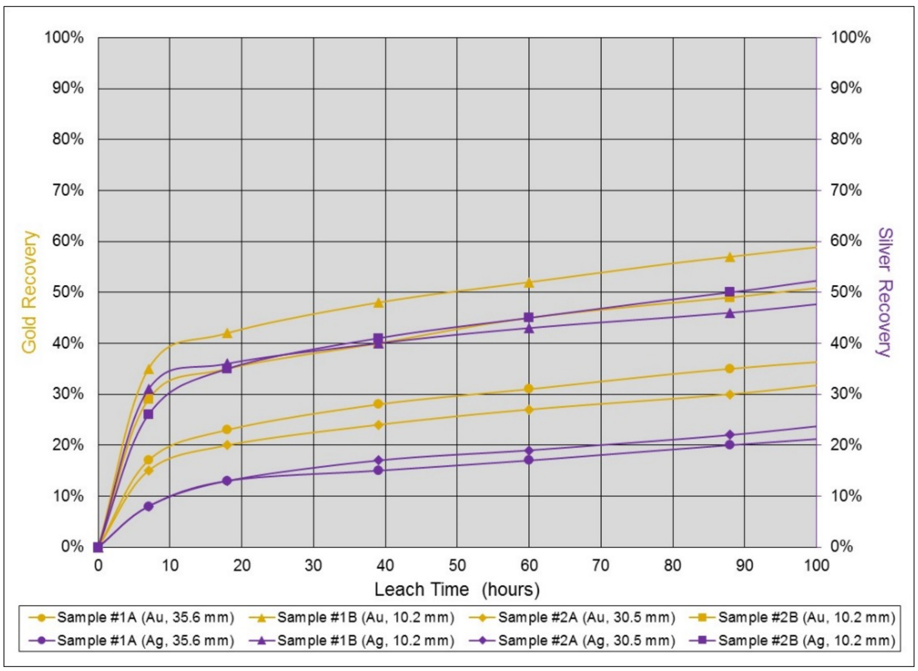
After passing through the bottle of activated carbon, the solution was re-assayed for pH, sodium cyanide, gold and silver. Sodium cyanide was then added, when necessary, to maintain the solution at target levels of approximately 0.5 g/L. The leach solution was then recycled to the material for another 24 hour leach period. Two batches of leach solution were used: while one batch was applied to the column the other was run through the carbon.

Table 13.12 and Figure 13.7 summarize KCA's column leach test results, from which it may be seen that there are marked improvements in gold and silver recovery from the B samples, that were crushed to P_{100} 12.7 mm, compared with the A samples that were crushed to P_{100} 50.8 mm. This repeats the trend first seen in Metcon's 2008 column leach test results. The gold recoveries reported by KCA (2010) and Metcon (2008) compare favourably, but significant variance exists as regards silver recovery.

Table 13.12: A Summary of Column Leach Test Results, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)

Parameter	Sample							
	#1A		#1B		#2A		#2B	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted, days 0 to 7	17	8	35	31	15	8	29	26
…. days 8 to 18	23	13	42	36	20	13	35	35
…. days 19 to 39	28	15	48	40	24	17	40	41
…. days 40 to 60	31	17	52	43	27	19	45	45
…. days 61 to 88	35	20	57	46	30	22	49	50
…. days 89 to 120	38	23	61	50	34	26	53	55
…. days 121 to 208	44	30	66	57	39	32	57	61
Base Data								
Feed Size	P_{80} 35.6 mm (1.4")		P_{80} 10.2 mm (0.4")		P_{80} 30.5 mm (1.2")		P_{80} 10.2 mm (0.4")	
Tail Grade (g/t)	0.480	5.83	0.274	4.46	0.446	6.86	0.343	3.77
Extracted Grade (g/t)	0.377	2.40	0.514	5.83	0.274	3.09	0.446	6.17
Calculated Head (g/t)	0.823	8.57	0.789	10.29	0.754	10.29	0.789	9.94
Chemistry								
Cyanide Consumption (kg/t)	1.15		2.30		1.51		2.40	
Lime Consumption (kg/t)	2.0		2.0		2.0		2.0	

Figure 13.7: Column Leach Test Recovery Curves, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)



13.6 Kappes, Cassidy & Associates, 2011/2012

In late 2011 KCA started a series of bottle roll tests on pulverized composite samples and on crushed samples of diamond drillcore material from across the Moss Vein. A total of four column leach tests were also carried out. All of the tests were completed in 2012, the main objectives of which were to test the metallurgical response of Moss Vein material at different crush sizes, and to test for metallurgical variability across the Moss Vein and with depth. The results are detailed in a KCA report to Patriot Gold dated November 2012.

13.6.1 Sample Preparation

KCA was supplied with samples from nine diamond drillholes from across the Moss Vein area. Fourteen composites were compiled from the supplied material, once each supplied interval had been stage crushed to P_{100} 12.7 mm (1/2") and size adjusted to a target size of P_{80} 6.35 mm (1/4"):

- ten regional composites were generated based on region and hole ID (5 kg splits were used for head analysis and coarse bottle roll test work); and
- four grade composites were compiled based on geographic location and assay grade (high or low), which composites were used for head analyses, pulverized and crushed material bottle roll tests.

Splits of the ten regional composites were used to generate two generalized zone composites that were used for head analyses, head screen analyses with assays by size fraction, bottle roll testing and column leach testing. Portions of the reject material from each regional and grade composite were also used to generate a generalized Moss mine composite for head screen analyses, assays by size fraction and column leach testing. Summary details of the various composites are presented on Table 13.13. Figure 13.8 identifies the locations of the various composites.

The high- and low-grade composites were prepared separately but identically by:

- splitting out 1.5 kg portions of the mixed drillcore samples and then crushing the material to a nominal 10 mesh (1.70 mm);
- splitting out two final, 200 g portions that were ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm), which final splits were used for head analysis; and
- ring and puck pulverizing the remaining material to a target size of P_{80} 200 mesh (0.074 mm), from which 1,000 g portion were split out and used for bottle roll testing.

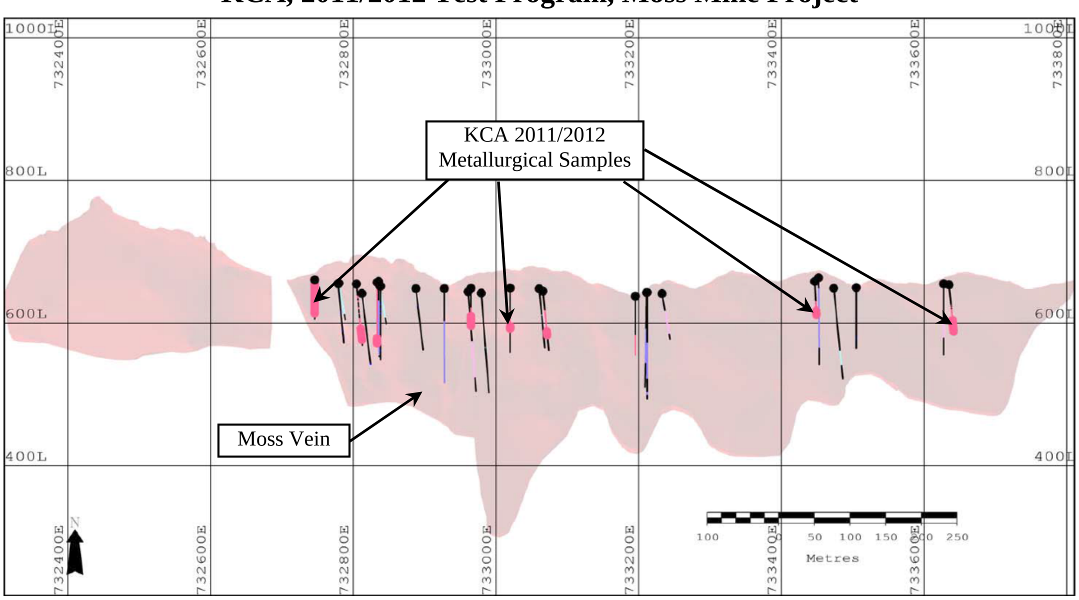
The generalized Upper- and Lower-Zone composites were separately and identically prepared by:

- splitting out and separately storing 14 kg portions of the material comprising the overall composites;
- stage crushing the remaining material to P_{100} 12.7 mm (1/2") and then size adjusting the material to a target size of P_{80} 6.35 mm (1/4");
- from the P_{100} 12.7 mm crushed material fractions –
  - 40 kg portions were split and used for column leach testing,
  - 20 kg portions were split, weighed, oven dried and used for head screen analysis with assays by fraction size,
  - the dried material was screened at 12.7 mm, 9.5 mm, 6.35 mm, 3.35 mm, 2.36 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and 65 mesh (0.212 mm), and

Table 13.13: A Summary of Composites, KCA, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's Npovember 2012 report to Patriot Gold)

Composite	Drillholes	Overall Weight (kg)	Head Analyses Au (g/t)	Head Analyses Ag (g/t)	Head Screen Analyses Crush Size (mm)	Head Screen Analyses Calc. P_{80} Size (mm)	Head Screen Analyses Au (g/t)	Head Screen Analyses Ag (g/t)
Regional Composites								
Lower West #A	AR-72C	27.63	0.653	5.11	-	-	-	-
Lower West #B	AR-74C Split	11.37	1.622	30.60	-	-	-	-
Lower Central #A	AR-75C	11.78	1.258	9.70	-	-	-	-
Lower Central #B	AR-69C	11.72	1.546	21.31	-	-	-	-
Lower East	AR-76C	12.18	0.988	19.30	-	-	-	-
Upper West #A	AR-73C	16.22	1.541	28.82	-	-	-	-
Upper West #B	AR-74C Split	6.45	1.160	15.26	-	-	-	-
Upper Central #A	AR-70C	13.15	0.642	8.71	-	-	-	-
Upper Central #B	AR-71C	15.34	0.834	12.70	-	-	-	-
Upper East	AR-77C	7.16	0.945	20.05	-	-	-	-
Grade Composites								
Upper High Grade	AR-70C, AR-71C, AR-73C, AR-74C, AR-77C	12.62	2.706	45.40	-	-	-	-
Lower High Grade	AR-69C, AR-72C, AR-74C, AR-75C, AR-76C	12.12	5.584	46.10	-	-	-	-
Upper Low Grade	AR-70C, AR-71C, AR-73C, AR-74C, AR-77C	9.73	0.290	6.79	-	-	-	-
Lower Low Grade	AR-69C, AR-72C, AR-74C, AR-75C, AR-76C	10.34	0.600	8.01	-	-	-	-
Generalized Zone Composites								
Upper Zone	Upper West #A Upper West #B Upper Central #A Upper Central #B Upper East *Overall*	5.94 7.00 9.14 1.06 0.36 *23.50*	1.981 0.648 1.034 1.104 1.138 *0.999*	17.69	12.7	6.28	1.075	17.73
Lower Zone	Lower West #A Lower West #B Lower Central #A Lower Central #B Lower East *Overall*	21.42 5.16 1.64 5.58 2.02 *35.82*	0.878 1.871 1.232 2.225 1.418 *1.119*	12.31	12.7	6.29	1.171	12.09
Moss Mine Composite								
Composite A Composite B	All Regional and Grade Composites	106.82	- -	- -	9.5 6.3	7.32 4.78	1.087 1.138	14.22 14.39

Figure 13.8: A Vulcan® Snapshot View (looking north) of the Moss Vein, Showing the Positions of the Grade Composites and Regional Composites (highlighted in RED), KCA, 2011/2012 Test Program, Moss Mine Project



  - each sieved size fraction was weighed and then crushed to a nominal top size of 10 mesh (1.70 mm), as required, and two 200 g portions were split, ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm) and then assayed for gold and silver,
  - a further 10 kg portion was split and crushed to a nominal size of 1.70 mm, from which two final, 500 g portions were split, ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm) and then used for head analysis; and
- final 6 kg portions were split from which –
  - 5 kg splits were taken and utilized for duplicate bottle roll tests, and
  - the remaining 1 kg portions were ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm) and then used for bottle roll testing.

The Moss Mine composite was prepared by stage crushing the relevant material to P_{100} 9.5 mm. The crushed material was labelled Sample #A, from which:

- a 30 kg portion was split for column leach testing;
- a 10 kg portion was split, oven dried and then used for head screen analysis with assays by size fraction –
  - the dry material was screened at 9.53 mm, 6.35 mm, 3.35 mm, 2.36 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and 65 mesh (0.212 mm), and
  - each sieved size fraction was weighed and then crushed to a nominal top size of 1.70 mm (as required) and two 200 g portions were split, ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm) and then assayed for gold and silver,
- a 45 kg portion was split and stage crushed to P_{100} 6.35 mm (1/2”) and then labelled Sample #B, from which –
  - a 30 kg portion was split for column leach testing; and
  - 10 kg portions were split, weighed, oven dried and used for head screen analysis with assays by fraction size,
  - the dried material was screened at 6.35 mm, 3.35 mm, 2.36 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and 65 mesh (0.212 mm), and
  - each sieved size fraction was weighed and then crushed to a nominal top size of 10 mesh (1.70 mm), as required, and two 200 g portions were split, ring and puck pulverized to a target size of P_{80} 200 mesh (0.074 mm) and then assayed for gold and silver.

13.6.2 Head Analysis

Head analyses for gold and silver were carried out on each of the regional, grade and zone composites. A portion of the head material was crushed to nominal 10 mesh (1.70 mm), from which duplicate 500 g splits were individually ring and puck pulverized to P_{100} 150 mesh (0.105 mm). The pulverized splits were then assayed using standard fire assay methods with a flame atomic absorption spectrophotometric (“FAAS”) finish for gold and four-acid digestion with a FAAS finish for silver. Table 13.14 summarizes the results; the average grades are also detailed on Table 13.13.

Table 13.14: A Summary of Head Analysis Results for Gold and Silver, KCA, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Composite	Head Analysis					
	Assay 1 Au (g/t)	Assay 2 Au (g/t)	Average Au (g/t)	Assay 1 Ag (g/t)	Assay 2 Ag (g/t)	Average Ag (g/t)
Regional Composites						
Lower West #A	0.590	0.717	0.653	5.21	5.01	5.11
Lower West #B	1.642	1.601	1.622	30.21	30.99	30.60
Lower Central #A	1.186	1.330	1.258	9.39	10.01	9.70
Lower Central #B	1.567	1.526	1.546	21.81	20.81	21.31
Lower East	0.941	1.035	0.988	19.61	18.99	19.30
Upper West #A	1.464	1.618	1.541	28.22	29.42	28.82
Upper West #B	1.101	1.219	1.160	15.50	15.02	15.26
Upper Central #A	0.597	0.687	0.642	9.02	8.40	8.71
Upper Central #B	0.801	0.867	0.834	12.99	12.41	12.70
Upper East	0.970	0.919	0.945	19.90	20.19	20.05
Grade Composites						
Upper High Grade	2.645	2.767	2.705	45.00	45.81	45.40
Lower High Grade	4.646	6.523	5.584	45.60	46.59	46.10
Upper Low Grade	0.271	0.309	0.290	6.79	6.79	6.79
Lower Low Grade	0.535	0.665	0.600	8.19	7.82	8.01
Zone Composites						
Lower Zone	1.066	1.173	1.119	12.62	12.00	12.31
Upper Zone	0.991	1.008	0.999	17.79	17.59	17.69

13.6.3 Head Screen Analysis

Portions from the two zone composite and portions from each crush size of the Moss mine composites were utilized for head screen analyses with assays by size fraction:

- the Upper and Lower Zone composites (P_{100} 12.7 mm [1/2"]) were screened at 12.7 mm, 9.5 mm, 6.35 mm, 3.35 mm, 2.36 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and (65 mesh (0.212 mm);
- the P_{100} 9.53 mm (3/8") Moss Mine composite was screened at 9.5 mm, 6.3 mm, 4.75 mm, 3.35 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and (65 mesh (0.212 mm); and
- the P_{100} 6.35 mm (1/4") Moss Mine composite was screened at 6.35,mm 4.75 mm, 3.35 mm, 10 mesh (1.70 mm), 28 mesh (0.600 mm) and (65 mesh (0.212 mm);

Each sieved fraction was weighed and crushed to a nominal size of 1.70 mm, as necessary. Two 200 g portions were then split out and pulverized to a target size of P_{80} 200 mesh (0.074 mm). Each pulverized portion was individually assayed for gold and silver using fire assay with an AA finish for gold and four-acid digestion with an AA finish for silver. Table 13.15 summarizes the overall weighted average results. The calculated P_{80} crush sizes for the prepared samples were: Upper Zone – 6.28 mm, Lower Zone – 6.29 mm, Moss Mine Composite A – 7.32 mm; and Moss Mine Composite B – 4.78 mm.

13.6.4 Cyanide Shake Tests

Preliminary cyanide shake tests were carried out on 15 g portions of the pulverized composites, placed into a 50 mL centrifuge tube with a screw cap. A volume equivalent to 30 mL of cyanide solution (5g/L concentration) at ambient air temperature was then added and the pulp and cyanide solution mixed by shaking. The slurry was then agitated on a table action shaker for a period of 24 hours at room temperature, the slurry was then centrifuged and the resulting clear solution was analyzed for pH, gold and silver using FAAS methods. If the measured pH was less than 9.0 the test was re-run with the addition of 0.1 g of hydrated lime. Table 13.16 summarizes the results.

Table 13.15: A Summary of KCA's Head Screen Results, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Sample #	Passing (mm)	Retained (mm)	Sample Weight (kg)	Distrib. (%)	Cumulative Weight (%)		Gold		Silver	
					Retained	Passing	g/t	Weight (%)	g/t	Weight (%)
Upper Zone Composite	-	12.70	0.00	-	-	-	-	-	-	-
	12.70	6.35	3.93	19.75	19.7	100.0	0.979	18.0	15.75	17.6
	6.35	3.36	6.66	33.47	53.2	80.3	1.068	33.3	18.26	34.5
	3.36	2.38	2.09	10.50	63.7	46.8	1.051	10.3	18.99	11.3
	2.38	1.70	1.20	6.03	69.7	36.3	1.011	5.7	17.81	6.1
	1.70	0.600	2.79	14.02	83.8	30.3	1.011	13.2	17.81	14.1
	0.595	0.212	1.25	6.28	90.1	16.2	1.298	7.6	18.10	6.4
	0.210	Pan	1.98	9.95	100.0	9.9	1.298	12.0	18.10	10.2
Totals and Averages			***19.90***	***100.00***	-	-	***1.075***	***100.0***	***17.73***	***100.0***
Lower Zone Composite	-	12.70	0.00	-	-	-	-	-	-	-
	12.70	6.35	3.98	20.05	20.1	100.0	1.275	21.8	10.59	17.6
	6.35	3.36	6.45	32.49	52.5	79.9	1.113	30.9	12.00	32.3
	3.36	2.38	1.79	9.02	61.6	47.5	1.085	8.4	12.31	9.2
	2.38	1.70	1.25	6.30	67.9	38.4	1.176	6.3	12.81	6.7
	1.70	0.600	2.90	14.61	82.5	32.1	1.176	14.7	12.81	15.5
	0.595	0.212	1.43	7.20	89.7	17.5	1.198	7.4	12.99	7.7
	0.210	Pan	2.05	10.33	100.0	10.3	1.198	10.6	12.99	11.1
Totals and Averages			***19.85***	***100.00***	-	-	***1.171***	***100.0***	***12.09***	***100.0***
Mine Composite A (P_{100} 9.5 mm)	-	9.50	0.00	-	-	-	-	-	-	-
	9.50	6.35	2.86	28.83	28.8	100.0	1.108	29.4	14.50	29.4
	6.35	4.76	1.87	18.85	47.7	71.2	1.042	18.1	13.51	17.9
	4.76	3.36	1.30	13.10	60.8	52.3	1.104	13.3	15.91	14.7
	3.36	1.70	1.54	15.52	76.3	39.2	1.128	16.1	14.40	15.7
	1.70	0.600	1.08	10.89	87.2	23.7	1.012	10.1	12.36	9.5
	0.595	0.212	0.54	5.44	92.6	12.8	0.821	4.1	12.70	4.9
	0.210	Pan	0.73	7.36	100.0	7.4	1.310	8.9	15.41	8.0
Totals and Averages			***9.92***	***100.00***	-	-	***1.093***	***100.0***	***14.22***	***100.0***
Mine Composite A (P_{100} 9.5 mm)	-	6.35	0.00	-	-	-	-	-	-	-
	6.35	4.76	2.06	20.72	20.7	100.0	1.398	24.5	14.80	21.3
	4.76	3.36	1.75	17.61	38.3	79.3	0.900	13.4	15.70	19.2
	3.36	1.70	2.27	22.84	61.1	61.7	1.226	23.6	14.71	23.3
	1.70	0.600	1.80	18.11	79.3	38.9	1.113	17.0	12.50	15.7
	0.595	0.212	0.89	8.95	88.2	20.7	1.080	8.2	12.51	7.8
	0.210	Pan	1.17	11.77	100.0	11.8	1.335	13.3	15.39	12.6
Totals and Averages			***9.94***	***100.00***	-	-	***1.183***	***100.0***	***14.39***	***100.0***

Table 13.16: A Summary of Cyanide Shake Test Results, KCA, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Composite	Average Head Grade		Final pH	Pregnant Solution		Extraction			
	Au (g/t)	Ag (g/t)		Au (mg/L)	Ag (mg/L)	Au (g/t)	Au (%)	Ag (g/t)	Ag (%)
Upper Zone	0.999	11.70	10.1	0.49	7.55	0.980	98%	15.10	85%
			10.1	0.51	7.45	1.020	<100%	14.90	84%
Lower Zone	1.119	12.30	10.1	0.58	5.45	1.160	<100%	10.90	89%
			10.2	0.62	5.30	1.240	<100%	10.60	86%

13.6.5 Bottle Roll Tests – Pulverized Material

Cyanide bottle roll tests were completed on pulverized portions of the Upper- and Lower-High Grade composites and Upper- and Lower-Zone composites (total of four tests). Each test was run for a total of 96 hours using industry standard procedures, with solution sampling performed at 2, 4, 8, 24, 48, 72 and 96 hours. The tests utilized 1.0 kg of pulverized material (P_{80} 200 mesh, 0.074 mm) slurried with 1,500 mL of tap water. Sodium cyanide was added and maintained at 1.0 g/L of solution and the pH of the solution was maintained at 11.0, with the addition of hydrated lime. Table 13.17 and Figure 13.9 summarize the results, from which it may be concluded that that:

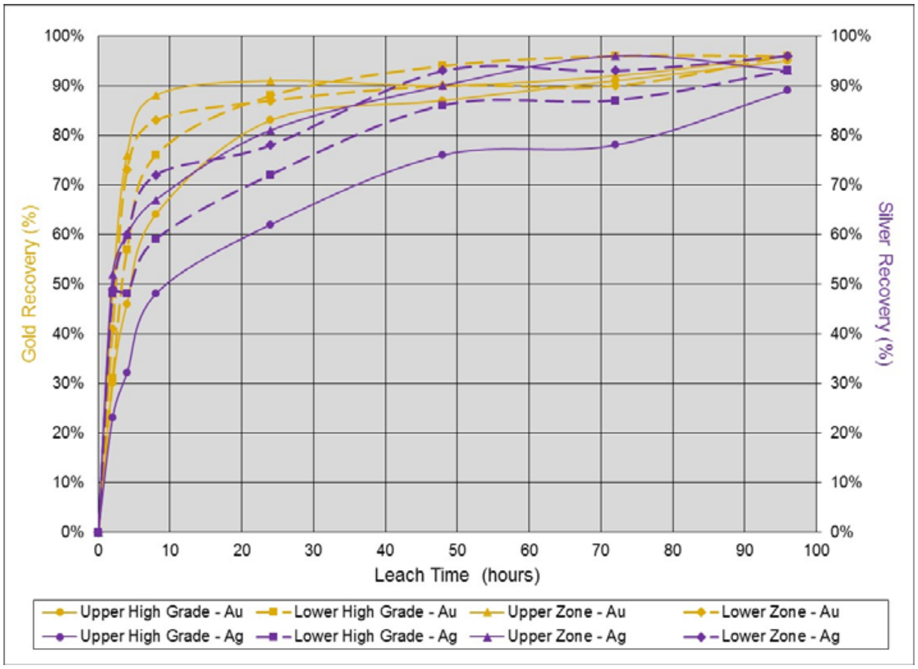
- exceptional extraction rates were achieved (95% to 96% Au, 89% and 96% Ag) –
  - the average gold extraction rate (95.8%) is nearly identical to Metcon's average gold recovery from the bottle roll tests on 150 mesh pulps (94.7%), whereas
  - the average silver recovery rate (92.8%) is very similar to the average silver recovery rate from the KCA's 2010 bottle roll tests program (89.5%);
- in common with KCA's 2010 bottle roll test program –
  - gold recovery was rapid (the majority of the contained metal was in each case extracted to pregnant solution in less than 10 hours and at 24 hours, extraction was substantially complete);
  - silver recovery was by comparison moderately fast (the majority of the contained metal was extracted to pregnant solution after approximately 12 hours to 15 hours, after which extraction continued to the end of the tests at 96 hours, at which point and in theory at least, the trends of the recovery curves suggest that additional silver might have been recovered);
- at 0.14 kg/t ton and 0.63 kg/t, respectively, consumptions of cyanide and lime were low, but slightly higher than for KCA's 2010 bottle roll test program.

Table 13.17: A Summary of Bottle Roll Test Results on Pulverized Material, KCA, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Parameter	Sample							
	Upper High Grade		Lower High Grade		Upper Zone		Lower Zone	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted in 2 hours	30	23	31	48	48	52	41	49
…. in 4 hours	46	32	57	48	76	60	73	60
…. in 8 hours	64	48	76	59	88	67	83	72
…. in 24 hours	83	62	88	72	91	81	87	78
…. in 48 hours	87	76	94	86	90	90	90	93
…. in 72 hours	91	78	96	87	92	96	90	93
…. in 96 hours	91	79	97	88	91	95	90	92
filtrate & wash	95	89	96	93	96	93	96	96
Base Data								
Feed Size	P_{80} 200 mesh		P_{80} 200 mesh		P_{80} 200 mesh		P_{80} 200 mesh	
Tail Grade (g/t)	0.120	5.11	0.171	3.60	0.048	1.20	0.051	0.62
Extracted Grade (g/t)	2.423	42.77	3.846	43.50	0.995	14.06	1.143	13.13
Calculated Head (g/t)	2.538	48.03	4.019	47.00	1.041	15.16	1.195	13.74
Chemistry								
Cyanide Consumption (kg/t)	0.13		0.12		0.09		0.22	
Lime Consumption (kg/t)	0.50		0.50		0.50		1.00	
Final pH	10.6		10.4		10.5		10.5	

Figure 13.9: Bottle Roll Test Metallurgical Recovery Curves for Pulverized Material, KCA, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)



13.6.6 Bottle Roll Tests – Coarse Material

A total of 18 bottle roll tests were carried out on coarse samples (those crushed to P_{100} 12.7 mm [1/2"]) of the regional, grade and zone composites (the latter as duplicates):

- 5 kg portions of head material were slurried with 5,000 mL of tap water, in a 20 L carboy;
- the slurries were mixed thoroughly and the pH checked and adjusted with hydrated lime, as required, to 10.5 to 11.0;
- sodium cyanide was added to the slurry to achieve a target concentration of 1.0 g/L; and
- the bottles were rolled on laboratory rolls, solution sampling was carried out at 2, 4, 8, 24, 48, 72, 96, 120, 144, 168, 192, 216, 240, 264, 288, 312, 336, 360, 384 and 408 hours;
- additional hydrated lime and cyanide were added after each sample period, as required and to adjust the slurry to the target levels;
- on completion of the leach period, individual slurries were filtered, washed and dried; and
- duplicate portions of the tailings were split out, individually ring and puck pulverized to P_{80} 200 mesh (0.074 mm) and assayed for residual gold and silver.

A distinct difference between these and the other bottle roll tests reported herein was the limitation of physical rolling to one minute in every hour, to '*avoid particle size reduction* *in time*' (as defined by KCA's internal standard) and thereby render the results '*more reliable with respect to determining the effect of crush size on precious metal recovery*'. The effect of this is evident from consideration of the gold and silver recovery results summarized on Tables 13.18 through 13.20: significantly and consistently low metal recoveries were achieved.

Table 13.18: A Summary of KCA's Bottle Roll Test Results on Coarse Material, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Parameter	Sample							
	Upper High Grade		Lower High Grade		Upper Low Grade		Lower Low Grade	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal Extracted	35	32	42	26	39	57	51	78
Base Data								
Feed Size	P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")	
Tail Grade (g/t)	2.052	28.51	2.410	38.11	0.199	1.10	0.237	0.79
Extracted Grade (g/t)	1.121	13.20	1.740	13.54	0.124	1.43	0.246	2.86
Calculated Head (g/t)	3.173	41.71	4.150	51.65	0.323	2.52	0.483	3.65
Chemistry								
NaCN Consumption (kg/t)	0.22		0.36		0.20		0.41	
Lime Consumption (kg/t)	0.60		0.60		0.60		0.60	
Final pH	10.4		10.4		10.3		10.2	

Table 13.19: A Summary of KCA's Bottle Roll Test Results on Coarse Material, Regional Composites, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

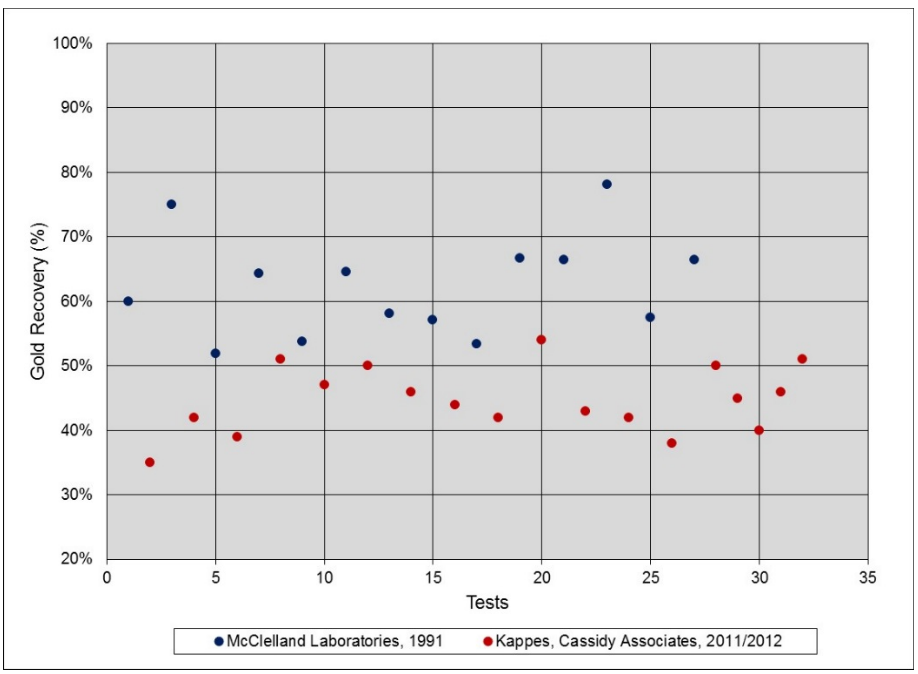
Parameter	Sample									
	Upper West #A		Upper West #B		Upper Central #A		Upper Central #B		Upper East	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal Extracted	47	28	50	30	46	68	44	52	42	37
Base Data										
Feed Size	P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")	
Tail Grade (g/t)	0.891	20.71	0.504	7.51	0.315	1.10	0.435	2.76	0.645	12.10
Extracted Grade (g/t)	0.785	7.88	0.498	3.25	0.266	2.36	0.335	2.97	0.476	7.01
Calculated Head (g/t)	1.675	28.58	1.002	10.76	0.581	3.46	0.770	5.73	1.120	19.11
Chemistry										
NaCN Consumption (kg/t)	0.17		0.12		0.15		0.15		0.12	
Lime Consumption (kg/t)	0.60		0.60		0.60		0.60		0.60	
Final pH	10.4		10.3		10.4		10.5		10.4	
Parameter	**Sample**									
	Lower West #A		Lower West #B		Lower Central #A		Lower Central #B		Lower East	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted	54	62	43	21	42	46	38	35	50	45
Base Data										
Feed Size	P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")	
Tail Grade (g/t)	0.242	0.89	1.037	24.81	0.603	3.06	1.085	10.70	0.494	8.81
Extracted Grade (g/t)	0.279	1.47	0.774	6.72	0.434	2.55	0.668	5.67	0.492	7.33
Calculated Head (g/t)	0.521	2.36	1.811	31.53	1.037	5.61	1.753	16.37	0.986	16.14
Chemistry										
NaCN Consumption (kg/t)	0.19		0.15		0.19		0.19		0.24	
Lime Consumption (kg/t)	0.60		0.60		0.60		0.60		0.60	
Final pH	10.4		10.5		10.4		10.3		10.3	

Table 13.20: A Summary of KCA's Bottle Roll Test Results on Coarse Material, Zone Composites, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Parameter	Sample							
	Upper Zone #A		Upper Zone #B		Lower Zone #A		Lower Zone #B	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal Extracted	45	31	40	31	46	30	51	33
Base Data								
Feed Size	P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")		P_{100} 12.7 mm (1/2")	
Tail Grade (g/t)	0.608	10.85	0.734	10.70	0.660	10.49	0.566	9.70
Extracted Grade (g/t)	0.502	4.77	0.485	4.84	0.554	4.46	0.599	4.82
Calculated Head (g/t)	1.109	15.62	1.218	15.54	1.214	14.95	1.165	14.53
Chemistry								
Cyanide Consumption (kg/t)	0.39		0.37		0.33		0.30	
Lime Consumption (kg/t)	0.60		0.60		0.60		0.60	
Final pH	10.4		10.4		10.3		10.3	

In the case of gold, the recovery rates were consistently lower than those achieved by McClelland Laboratories in 1991 on similar mineralized material of the same nominal size that was subjected to standard, 96 hour bottle roll tests (albeit that for want of detailed information, a nominal size of P_{100} 12.7 mm had to be assumed for the RC chip samples used in McClelland Laboratories' 1991 bottle roll test program). Figure 13.10 emphasizes this, from which it may be seen that KCA's recoveries were up to half those achieved by McClelland Laboratories in 1991. KCA's average gold recovery for the test series was 44.7% compared with an average of 62.4% for McClelland Laboratories' 1991 test series.

Figure 13.10: A Comparison of Gold Recoveries from Bottle Roll Tests on 12.7 mm Material, McClelland Laboratories (1991) vs. KCA (2011/2012), Moss Mine Project
(compiled from data contained in McClelland Laboratories 2011 report to Billiton Minerals and in KCA's November 2102 report to Patriot Gold)



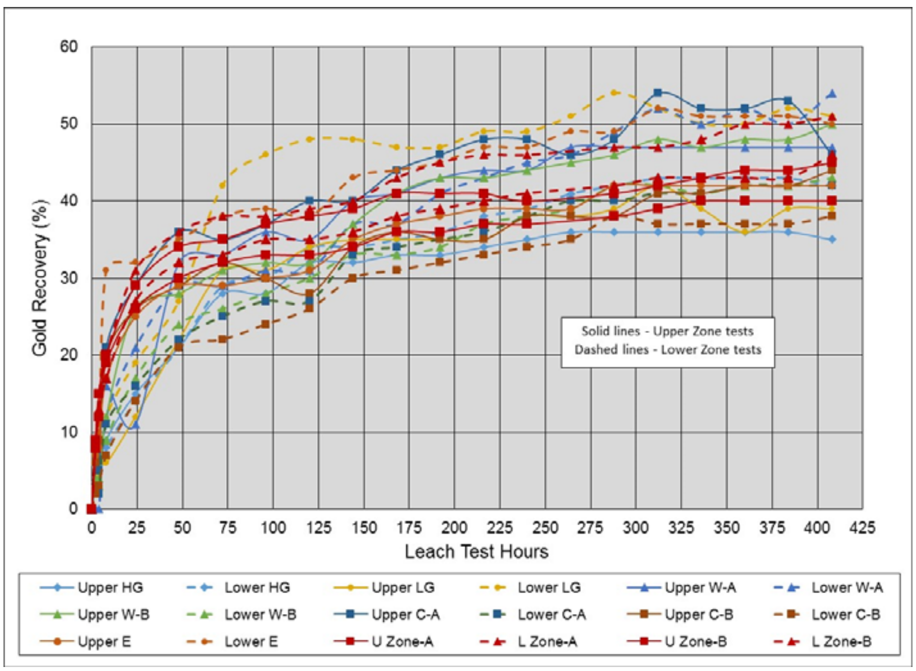
The cumulative gold and silver recovery rates achieved during KCA's 2011/2012 bottle roll tests on coarse material are summarized on Figures 13.11 and 13.12, from which it may be seen that:

- there is good repeatability between the gold recovery curves, but the silver recoveries vary widely;
- there is no clustering of results for either gold or silver by grade (high- and low-grade composites), by geographic location (west, central or east composites) or by elevation (upper or lower composites); and
- the overall Moss Mine composites report average gold recovery curves for the data, but recovery curves that are low-end skewed in the case of silver.

The gold recovery results suggest that there is no selectivity in metallurgical response by grade, depth or geographic position. This finding is supported by the findings of the oxidation analysis presented in Sub-Section 7.2.5, as well as by the repeatability of McClelland Laboratories' 1991 gold recovery results.

Figure 13.11: Metallurgical Recovery Curves for Gold, KCA's Bottle Roll Tests On Coarse Material, 2011/2012 Test Program, Moss Mine Project

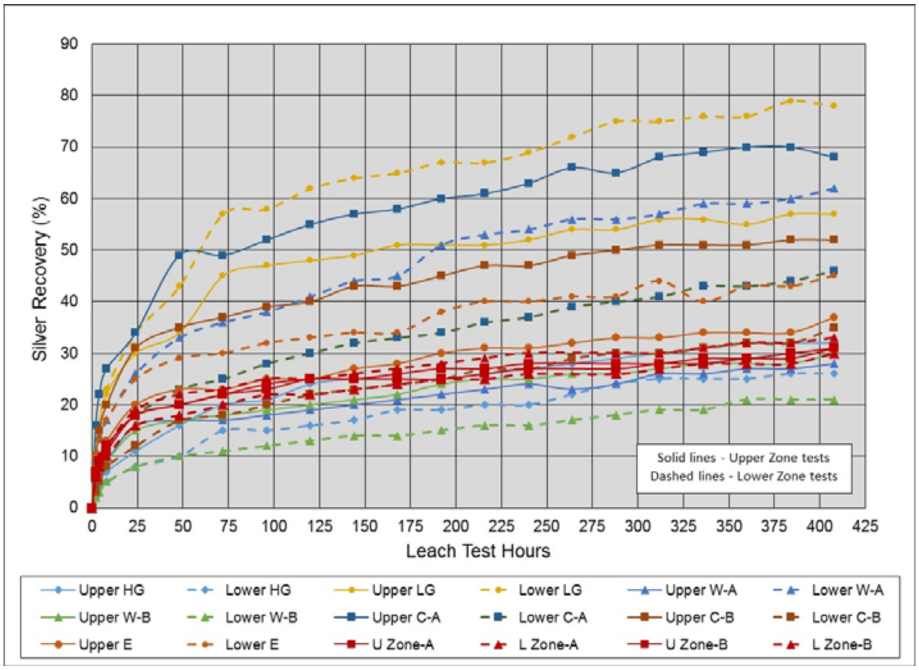
(compiled from data contained in KCA's November 2012 report to Patriot Gold)



13.6.7 Extended Bottle Roll Tests

Cyanide bottle roll tests lasting 2,616 hours (109 days) were carried out on crushed portions of the Upper West, Lower Central and Lower East regional composites. Each portion was tested at two crush sizes: P_{100} 9.53 mm (3/8") and P_{100} 6.35 mm (1/4") and each test utilized 2.0 kg of crushed material that was slurried with 2,000 mL of tap water. Sodium cyanide was added and maintained at 1.0 g/L of solution and the pH was maintained at 11.0, with the addition of hydrated lime.

Figure 13.12: Metallurgical Recovery Curves for Silver, KCA's Bottle Roll Tests On Coarse Material, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)



For the first seven days of the tests, the bottles were rolled for one minute per hour. Thereafter each test bottle was agitated twice per day by hand. After completion of the leach period the slurries were individually filtered, washed and dried. Duplicate portions were then split out and individually ring and puck pulverized to P_{80} 200 mesh (0.074 mm). The pulverized portions were then assayed for residual gold and silver content.

Table 13.21 and Figure 13.13 summarize the results. It may be concluded that:

- there is very good repeatability between the results for gold and good repeatability between the results for silver (which again suggests metallurgical uniformity across the Moss deposit, especially as regards gold);
- the average recovery rates are slightly higher for the P_{100} 6.35 mm (1/4") material compared with the P_{100} 9.53 m (3/8"), which result repeats the size-related recoveries seen in earlier test programs;
- at 0.54 kg/t to 0.69 kg/t cyanide consumption was low, as was lime consumption that varied between 1.0 kg/t and 1.25 kg/t; and
- although the recovery rates were very slow, due to the periodic nature of bottle rolling/agitation, the overall results match closely those achieved from Metcon's 2008 column leach tests on P_{80} 6.35 mm (1/4") material (66.3% Au, 42.1% Ag).

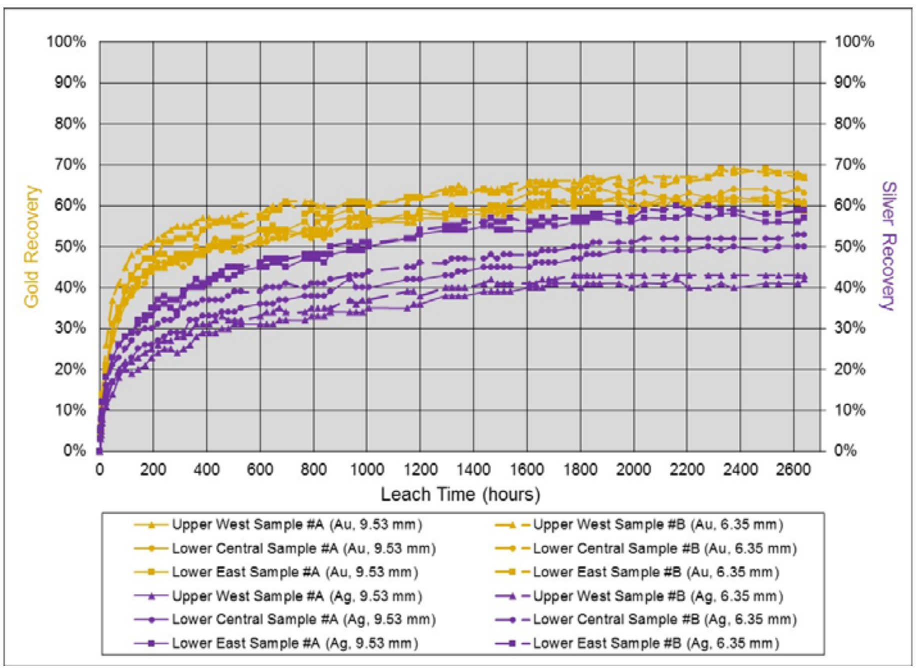
Table 13.21: A Summary of KCA's Extended Bottle Roll Test Results, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Parameter	Sample											
	Upper West #A		Upper West #B		Lower East #A		Lower East #B		Lower Central #A		Lower Central #B	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Extracted metal: 109 days	59	42	67	43	60	57	67	59	63	50	61	53
Base Data												
Feed Size	P_{100} 9.53 mm (3/8")		P_{100} 6.35 mm (1/4")		P_{100} 9.53 mm (3/8")		P_{100} 6.35 mm (1/4")		P_{100} 9.53 mm (3/8")		P_{100} 6.35 mm (1/4")	
Tail Grade (g/t)	0.662	16.51	0.514	17.61	0.408	7.90	0.333	7.82	0.349	3.96	0.387	3.81
Extracted Grade (g/t)	0.962	11.78	1.051	13.14	0.605	10.33	0.685	11.26	0.601	3.95	0.618	4.23
Calculated Head (g/t)	1.623	28.29	1.565	30.74	1.013	18.23	1.018	19.08	0.949	7.91	1.006	8.04
Chemistry												
Cyanide Consumption (kg/t)	0.54		0.59		0.56		0.69		0.57		0.64	
Lime Consumption (kg/t)	1.25		1.25		1.25		1.00		1.00		1.00	
Final pH	10.7		10.7		10.8		10.5		10.3		10.3	

Figure 13.13: Metallurgical Recovery Curves for Gold and Silver, KCA's Extended Bottle Roll Tests, 2011/2012 Test Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)



13.6.8 Column Leach Tests

Column leach tests were carried out on Upper Zone, Lower Zone and Moss Mine composites. The Upper and Lower Zone material was leached for 203 days, the Moss Mine composite material was leached for 198 days. The tests were run as continuously drained drip leach tests: in their program report KCA states that '*this type of test most accurately reflects actual heap leach conditions and is normally run when the material contains enough fines to prevent channeling of solution down individual rock faces*'.

The crushed material split-out for column test work was blended with lime or agglomerated with cement, as necessary, and then loaded into a plastic column. Alkaline cyanide solution was continuously distributed onto the material through Tygon tubing. The flow rate of solution dripping onto the material was controlled with a peristaltic pump to 10 L/hr to 12 L/hr/m^2 of column surface area.

The solution exiting each leach column was collected in a bottom tank. Leach solution was checked each cycle for pH, cyanide, gold and silver; copper content was periodically checked. The solution was then passed through a bottle of granular, activated carbon over a period of 24 hours to extract the gold and silver in solution. After passing through the activated carbon, the solution was re-assayed for pH, cyanide, gold and silver. Sodium cyanide was then added, if necessary, to maintain the solution at target level of 0.5 g/L, and the leach solution was recycled to the material for another 24 hour leach period. Two batches of leach solution were used: while one batch was applied to each column the other was run through carbon. Extraction rates were calculated from consideration of the calculated head grades and assayed tail grades only.

Table 13.22 summarizes KCA's 2011/2012 column leach test results, from which it may be seen that:

- while gold recovery remains approximately the same over the tested material sizes, there is a marked improvement in silver recovery between the coarser (12.7 mm, ½") samples at 39% and 40% and the finer (9.5 mm [3/8"] and 6.35 mm [1/4"]) samples at 58% and 59%, respectively;
- the results for the P$_{100}$ 6.35 mm (1/4") material are essentially the same as for KCA's standard bottle roll tests on the same material type;
- at 1.38 kg/t to 1.76 kg/t, cyanide consumption was much higher than for Metcon's 2008 column leach test program; and
- at 2.01 kg/t, lime consumption for the column leach tests on 12.7 mm material was approximately double the consumption reported for Metcon's 2008 column leach test program.

Table 13.22: A Summary of the Column Leach Test Results, KCA's 2011/2012 Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Parameter	Sample							
	Upper Zone		Lower Zone		Moss Mine #A		Moss Mine #B	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted, days 0 to 7	38	18	39	18	39	25	39	25
…. days 8 to 20	44	23	47	23	47	32	46	33
…. days 21 to 44	50	28	53	28	53	39	52	39
…. days 45 to 203	65	40	68	39	70	58	67	59
Base Data								
Feed Size	P$_{100}$ 12.7 mm (1/2")		P$_{100}$ 12.7 mm (1/2")		P$_{100}$ 9.5 mm (3/8")		P$_{100}$ 6.35 mm (1/4")	
Tail Grade (g/t)	0.430	11.08	0.373	10.34	0.334	5.30	0.374	5.20
Extracted Grade (g/t)	0.785	7.50	0.784	6.74	0.771	7.31	0.775	7.57
Calculated Head (g/t)	1.215	18.58	1.157	17.08	1.105	12.61	1.149	12.77
Chemistry								
Cyanide Consumption (kg/t)	1.38		1.76		1.60		1.75	
Lime Consumption (kg/t)	2.01		2.01		1.01		1.01	

Comparison of KCA's 2011/2012 column leach test results with earlier test programs (bottle roll and column leach) shows again the repeatability of outcomes as regards gold, but the generally more variable nature of the overall recoveries reported for silver. For example:

- the gold recovery results for the P_{100} 12.7 mm (1/2") material (65% and 68%) fall within the mid-range of the gold recovery results for McClelland Laboratories' 1991 bottle roll tests on P_{100} 12.7 mm material (51.9% to 75%); and
- gold recovery is essentially the same as that realized through column leach testing by Metcon is 2008, but silver recovery is higher (59% versus Metcon's 42.1%).

13.6.9 Tail Screen Analysis

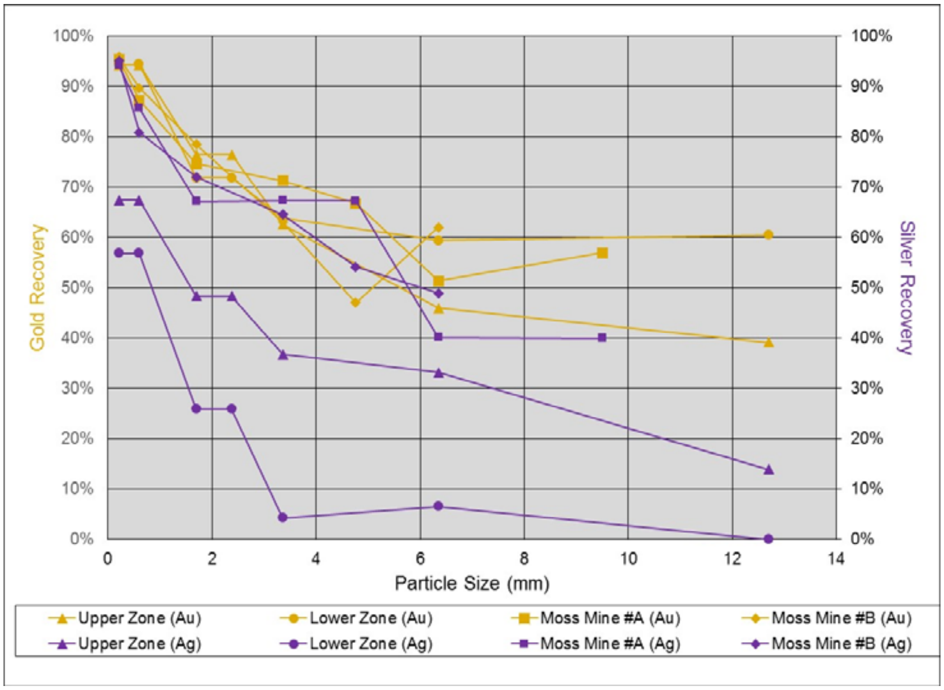
The tails from the column leach tests were analyzed and gold and silver recoveries were determined by size fraction. Table 13.23 details the results that repeat the finding of Metcon's 2008 program: both gold and silver recovery is sensitive to particle size. Figure 13.14 emphasizes this.

Table 13.23: A Summary of the Head, Recovered and Tail Assays by Size Fraction, KCA's 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Composite	Passing (mm)	Head Screen Analysis		Extracted Grade		Tail Screen Assays		Extraction by Fraction	
		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
Upper Zone	12.70	0.979	15.75	0.383	2.19	0.596	13.56	39.12	13.90
	6.35	1.068	18.26	0.491	6.06	0.577	12.20	45.97	33.19
	3.36	1.051	18.99	0.658	6.99	0.393	12.00	62.61	36.81
	2.38	1.011	17.81	0.773	8.60	0.238	9.21	76.46	48.29
	1.70	1.011	17.81	0.773	8.60	0.238	9.21	76.46	48.29
	0.595	1.298	18.10	1.224	12.20	0.074	5.90	94.30	67.40
	0.210	1.298	18.10	1.224	12.20	0.074	5.90	94.30	67.40
Lower Zone	12.70	1.275	10.59	0.771	-1.92	0.504	12.51	60.47	-18.13
	6.35	1.113	12.00	0.661	0.79	0.452	11.21	59.39	6.58
	3.36	1.085	12.31	0.692	0.52	0.393	11.79	63.78	4.22
	2.38	1.176	12.81	0.845	3.31	0.331	9.50	71.85	25.84
	1.70	1.176	12.81	0.845	3.31	0.331	9.50	71.85	25.84
	0.595	1.198	12.99	1.131	7.38	0.067	5.61	94.41	56.81
	0.210	1.198	12.99	1.131	7.38	0.067	5.61	94.41	56.81
Moss Mine #A	9.50	1.108	14.50	0.631	5.79	0.477	8.71	56.95	39.93
	6.35	1.042	13.51	0.535	5.42	0.507	8.09	51.34	40.12
	4.76	1.104	15.91	0.737	10.70	0.367	5.21	66.76	67.25
	3.36	1.128	14.40	0.804	9.70	0.324	4.70	71.28	67.36
	1.70	1.012	12.36	0.755	8.30	0.257	4.06	74.60	67.15
	0.595	0.821	12.70	0.716	10.88	0.105	1.82	87.21	85.67
	0.210	1.310	15.41	1.250	14.52	0.060	0.89	95.42	94.22
Moss Mine #B	6.35	1.398	14.80	0.867	7.24	0.531	7.56	62.02	48.92
	4.76	0.900	15.70	0.423	8.50	0.477	7.20	47.00	54.14
	3.36	1.226	14.71	0.773	9.50	0.453	5.21	63.05	64.58
	1.70	1.113	12.50	0.873	9.00	0.240	3.50	78.44	72.00
	0.595	1.080	12.51	0.969	10.11	0.111	2.40	89.72	80.82
	0.210	1.335	15.39	1.279	14.64	0.056	0.75	95.81	95.13

Figure 13.14: A Scatter Plot of Particle Size vs. Gold and Silver Recovery, KCA's 2011/2012 Program, Moss Mine Project

(compiled from data contained in KCA's November 2012 report to Patriot Gold)



13.7 Kappes, Cassidy & Associates, 2012

In mid-2012 KCA completed a series of 96 hour bottle roll tests on crushed samples of four, minus 200 mm rock samples from the Moss deposit. The results are detailed in a KCA report to Golden Vertex dated July 30, 2012 and entitled 'Moss Mine, Report on Metallurgical Testwork'.

13.7.1 Sample Preparation

The four rock samples were combined to form a single composite which was crushed to a nominal top size of 19.05 mm (3/4") and then split in half using a Jones Riffle Splitter. One half of the material was split again; one split sample quarter was stage crushed to P_{100} 10 mesh (1.70 mm) and then split into 500 g portions.

13.7.2 Bottle Roll Tests

Two 500 g portions of the head material were pulverized in a laboratory rod mill, one portion to a target size of P_{80} 100 mesh (0.150 mm) and the second to a target size of P_{80} 150 mesh (0.105 mm). The calculated P_{80} sizes for the pulverized portions were P_{80} 0.102 mm and P_{80} 0.085 mm, respectively. The material was utilized for cyanide bottle roll leach testing, as follows:

- the prepared samples were placed in separate 2.5 L test bottles and slurried with 750 mL of tap water;
- the slurry was mixed thoroughly, the pH checked and then adjusted, through the addition of hydrated lime, to between 10.5 and 11.0;

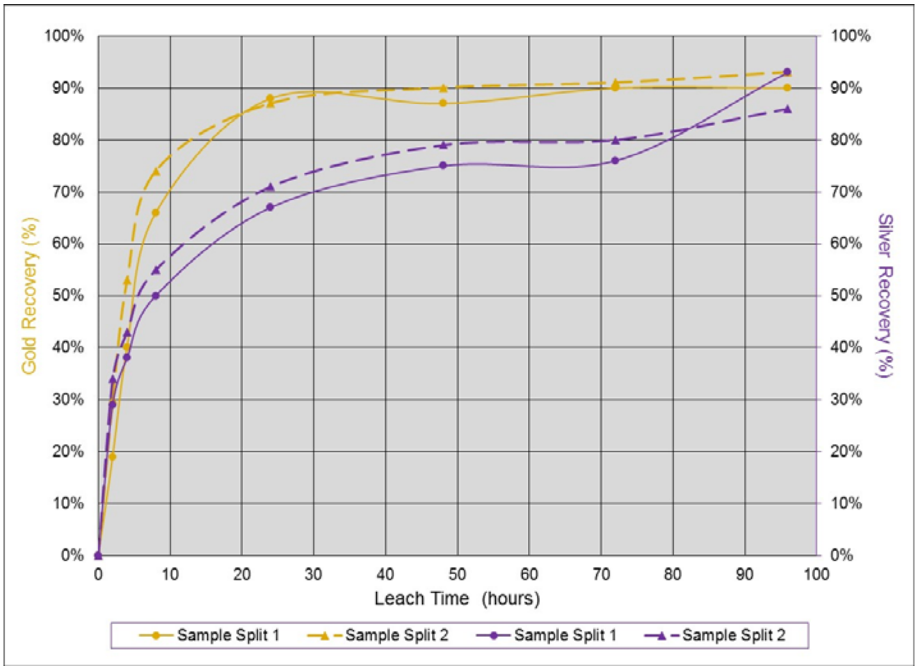
- sodium cyanide was added to the slurry to a target amount of 1.0 g/L;
- the bottle was placed onto a set of laboratory rolls, rolling continued throughout the duration of the tests;
- the slurries were checked at 2, 4, 8, 24, 48, 72 and 96 hours for pH, cyanide concentration, gold and silver;
- on completion of the tests the slurry was filtered, washed and screened at 200 mesh(0.074 mm) –
  - the oversize was dried and screened at 65 mesh (0.212 mm), 100 mesh (0.150 mm) and 150 mesh (0.105 mm) to determine particle size,
  - the size fractions were recombined, duplicate portions were split out and then ring and puck pulverized to P_{80} 200 mesh (0.074 mm), and
  - the pulverized portions were then assayed for residual gold and silver content standard fire assay methods, with a FAAS finish, for gold and four-acid digestion with a FAAS finish for silver.

Table 13.24 summarizes the results of the bottle roll tests, from which it may be seen that excellent recoveries were achieved for both gold (96% and 98%) and silver (89% and 90%) and that cyanide and lime consumptions were low. The results, for the most part, match closely the results of the 2008 Metcon bottle roll tests on 150 mesh material and KCA's 2011 and 2011/2012 bottle roll tests on P_{80} 200 mesh material. Again in common with the 2008 Metcon tests and KCA's 2011 and 2011/2012 tests, gold recovery was rapid - the majority of the gold was extracted in less than 10 hours and largely complete at approximately 24 hours (Figure 13.15).

Table 13.24: A Summary of the Bottle Roll Tests, KCA's 2012 Program, Moss Mine Project
(compiled from information contained in KCA's July 2012 report to Golden Vertex)

Parameter	Sample			
	Sample #A		Sample #B	
	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted in 2 hours	15	18	23	18
…. in 4 hours	31	26	33	26
…. in 8 hours	55	37	69	41
…. in 24 hours	87	62	90	63
…. in 48 hours	90	77	96	77
…. in 72 hours	93	84	97	83
…. in 96 hours	94	88	98	88
filtrate & wash	96	89	98	90
Base Data				
Feed Size	P_{80} 100 mesh		P_{80} 150 mesh	
Tail Grade (g/t)	0.120	1.92	0.065	1.71
Extracted Grade (g/t)	2.647	15.57	2.595	16.25
Calculated Head (g/t)	2.767	17.45	2.661	17.97
Chemistry				
Cyanide Consumption (kg/t)	0.465		0.96	
Lime Consumption (kg/t)	1.00		1.00	
Final pH	11.3		11.4	

Figure 13.15: Bottle Roll Test Recovery Curves for Pulverized Material, KCA's 2012 Test Program, Moss Mine Project
(compiled from information contained in KCA's July 2012 report to Golden Vertex)



13.8 McClelland Laboratories, 2013

Early in 2013 McClelland laboratories received the following Moss Mine Project PQ diameter drillcore sections for metallurgical testing: drillhole AR-188C – 27.13 m to 104.97 m; drillhole AR-189C – 46.63 m to 131.83 m; drillhole AR-190C – 51.66 m to 99.97 m; drillhole AR-191C – 15.24 m to 98.91 m; and drillhole AR-193C – 71.32 m to 121.92 m with 140.21 m to 144.78 m. The objectives of the tests were to test the metallurgical responses of various crush sizes to cyanide leaching, and to establish whether metallurgical recoveries from low- and high-grade samples would differ. The results of the test program are detailed in two reports to Northern Vertex Mining Corporation. The first is dated February 11, 2013 and is entitled 'Heap Leach Amenability Evaluation - Various Crusher Product Ore Samples from the Moss Project'. The second is dated April 26, 2013 and is entitled 'Heap Leach Amenability Evaluation – Lower Grade Moss Composite, 2 x Thru Rolls #2'.

13.8.1 Sample Preparation

The first samples received by McClelland Laboratories comprised samples designated as:

- conventionally (cone) crushed minus 25.4 mm (1") material (termed CC, -1" #1);
- material crushed in one pass through a modified rolls crusher (1 x Thru Rolls #1); and
- material crushed in two passes through a modified rolls crusher (2 x Thru Rolls #1).

Head screen analysis of sample CC, -1" #1 indicated a lack of fines (only 0.5% by weight at P_{100} 100 mesh), which suggested that crushed fines were not included in the sample split taken at site. A second, conventionally (cone) crushed sample of minus 25.4 mm (1") material was therefore obtained, which was designated sample CC, -1" #2.

A 40 kg split of sample CC, -1” #2 was conventionally crushed using a jaw and rolls crusher to P_{95} 6.35 mm (1/4”). A lower grade sample than the 1 x Thru Rolls and 2 x Thru Rolls #1 samples was also received from site, and then crushed using a modified rolls crusher and designated 2 x Thru Rolls #2.

13.8.2 Head Screen Analysis

Head screen analyses were carried out on each of the received samples, at the as-received crush sizes, to determine head grades and value distributions. Each approximately 20 kg sample was wet screened to obtain top size to 200 mesh (0.074 mm) size fractions. Each sieved size fraction was dried, weighed, crushed (if coarser than 10 mesh), blended and split to obtain samples for gold and silver assay. The results are summarized on Table 13.25.

13.8.3 Bottle Roll Tests

Direct agitated cyanidation tests of 96 hour duration were carried out on splits of samples CC, -1” #2, 1 x Thru Rolls, 2 x Thru Rolls #1 and 2 x Thru Rolls #2, at the as-received crush sizes per Table 13.25, to determine precious metal recovery, recovery rates, reagent requirements and amenability to heap leaching. Bottle roll tests were also carried out on pulverized splits of samples 1 x Thru Rolls and 2 x Thru Rolls #1 to determine maximum achievable precious metal recovery, recovery rates and reagent requirements. These latter splits were pulverized in a laboratory stainless steel ball mill to P_{80} 200 mesh (0.064 mm).

All the bottle roll tests were identically carried out:

- 2 kg charges of prepared material were slurried to achieve 40% solids pulp densities;
- the pH of each slurry was measured and hydrated lime was added to adjust the measured pH to between 10.8 and 11.0;
- sodium cyanide was added to the alkaline pulps to achieve a cyanide concentration equivalent to 1.0 g/L;
- rolling was temporarily stopped at 2, 6, 24, 48, 72 and 96 hours to take samples of pregnant solution to test for pH and cyanide concentration, and to assay for gold and silver (pH and cyanide concentrations were adjusted, as appropriate);
- after 96 hours the slurries were filtered, washed, dried, weighed and assayed in triplicate for gold and silver.

The results of the bottle roll tests are summarized on Table 13.26 and Figure 13.16. The following comments apply:

- the metal recoveries from the minus 25.4 mm (1”) sample (30.6% Au, 17.9% Ag) are broadly similar to the recoveries achieved by Metcon in 2008, for similarly sized but column leached material (38.66% Au, 14.12% Ag);
- the metal recoveries from the P_{95} 6.35 mm (1/4”) samples (59.0% and 67.6% Au, 33.3% and 44.6% Ag) are similar to the recoveries, for similarly sized but column leached material, achieved by Metcon in 2008 (66.31% Au, 42.11% Ag) and by KCA’s 2011/2012 bottle roll and column leach tests (61% to 67% Au, 43% to 59% Ag); and

- the metal recoveries from the P_{80} 200 mesh samples (96.5% and 97.3% Au, 80.0% and 84.2% Ag) are very similar to the results for bottle roll tests on 150 mesh material achieved by Metcon in 2008 (92.20% and 97.07% Au, 79.43% and 83.06% Ag), by KCA on 200 mesh material in 2010 (90% and 93% Au, 93% and 86% Ag) and by KCA on 200 mesh material in 2011/2012 (95% to 96% Au, 89% to 96% Ag).

Table 13.25: A Summary of Head Screen Analysis Results, McClelland Laboratories, 2013 Test Program, Moss Mine Project

(compiled from data contained in McClelland Laboratories' 2013 report to Northern Vertex)

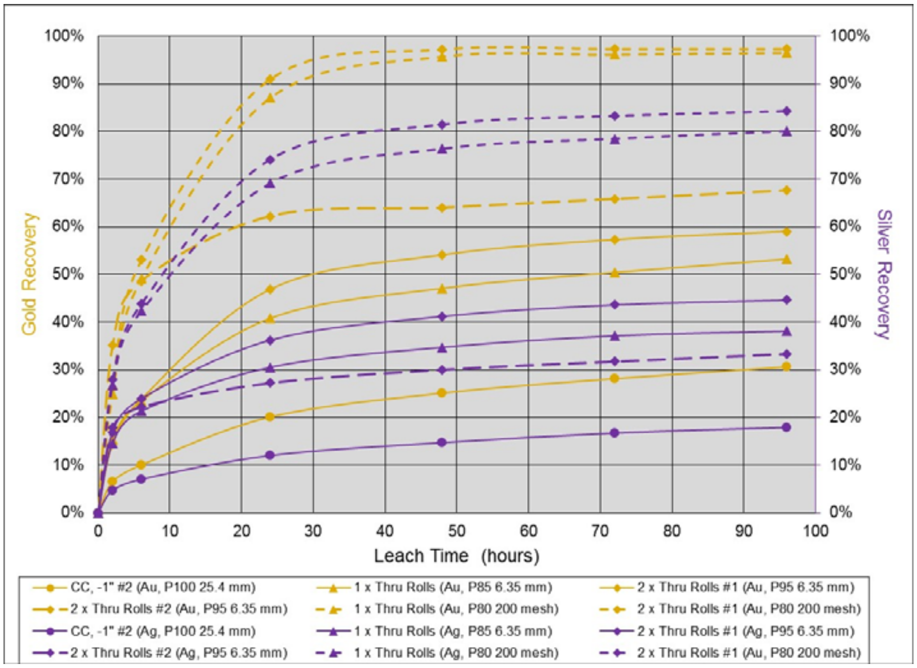
Sample #	Passing (mm)	Retained (mm)	Distrib. (%)	Cumulative Weight (%)		Gold		Silver	
				Retained	Passing	g/t	Weight %	g/t	Weight %
CC, -1" #1	-	19.05	8.7	8.7	100.0	2.859	9.0	24.00	12.9
	19.05	12.70	33.3	42.0	91.3	2.811	33.7	16.01	33.0
	12.70	6.35	31.2	73.2	58.0	2.921	32.8	18.00	34.7
	6.35	1.70	19.9	93.1	26.8	2.571	18.4	12.00	14.7
	1.70	0.85	3.5	96.6	6.9	2.530	3.2	12.99	2.8
	0.85	0.425	1.7	98.3	3.4	2.431	1.5	6.00	0.6
	0.425	0.21	0.9	99.2	1.7	1.471	0.5	3.02	0.2
	0.21	0.15	0.3	99.5	0.8	2.119	0.2	4.01	0.1
	0.15	0.074	0.2	99.7	0.5	1.851	0.1	12.00	0.2
	0.074	Pan	0.3	100.0	0.3	5.431	0.6	44.98	0.8
Totals and Averages			***100.0***	***-***	***-***	***2.777***	***100.0***	***16.18***	***100.0***
CC, -1" #2	-	19.05	10.0	10.0	100.0	2.599	9.3	24.00	11.1
	19.05	12.70	22.0	32.0	90.0	3.171	24.9	19.99	20.4
	12.70	6.35	25.8	57.8	68.0	3.141	28.9	24.00	28.7
	6.35	1.70	24.9	82.7	42.2	2.681	23.8	22.01	25.4
	1.70	0.85	8.7	91.4	17.3	2.239	7.0	20.98	8.5
	0.85	0.425	4.0	95.4	8.6	1.800	2.6	13.99	2.6
	0.425	0.21	2.4	97.8	4.6	1.999	1.7	11.01	1.2
	0.21	0.15	1.2	99.0	2.2	1.690	0.7	13.99	0.8
	0.15	0.074	0.4	99.4	1.0	3.329	0.5	23.01	0.4
	0.074	Pan	0.4	100.0	0.6	3.021	0.6	32.98	0.9
Totals and Averages			***100.0***	***-***	***-***	***2.811***	***100.0***	***21.57***	***100.0***
1 x Thru Rolls (P_{85} ¼")	-	6.35	14.8	14.8	100.0	2.808	16.0	24.58	13.7
	6.35	1.70	48.8	63.6	85.2	3.021	56.5	30.99	56.8
	1.70	0.85	12.1	75.7	36.4	2.270	10.5	20.98	9.5
	0.85	0.425	7.6	83.3	24.3	2.541	7.4	18.00	5.1
	0.425	0.21	5.0	88.3	16.7	1.190	2.3	12.99	2.4
	0.21	0.15	2.1	90.4	11.7	1.639	1.3	12.00	1.0
	0.15	0.074	3.2	93.6	9.6	1.550	1.9	12.99	1.6
	0.074	Pan	6.4	100.0	6.4	1.649	4.1	41.01	9.9
Totals and Averages			***100.0***	***-***	***-***	***2.606***	***100.0***	***26.61***	***100.0***
2 x Thru Rolls #1 (P_{95} ¼")	-	6.35	4.6	4.6	100.0	4.550	8.2	30.17	5.4
	6.35	1.70	40.2	44.8	95.4	2.719	43.0	29.01	45.4
	1.70	0.85	15.4	60.2	55.2	1.389	8.4	22.01	13.2
	0.85	0.425	11.3	71.5	39.8	3.051	13.6	22.01	9.7
	0.425	0.21	8.3	79.8	28.5	1.971	6.4	16.01	5.2
	0.21	0.15	3.0	82.8	20.2	1.371	1.6	13.99	1.6
	0.15	0.074	5.2	88.0	17.2	1.320	2.7	8.98	1.8
	0.074	Pan	12.0	100.0	12.0	3.401	16.1	37.99	17.7
Totals and Averages			***100.0***	***-***	***-***	***2.541***	***100.0***	***25.68***	***100.0***
2 x Thru Rolls #2 (P_{95} ¼")	-	6.35	4.5	4.5	100.0	0.466	6.9	6.99	5.2
	6.35	1.70	30.8	35.3	95.5	0.405	40.9	7.41	37.7
	1.70	0.85	15.2	50.5	64.7	0.209	10.5	5.31	13.4
	0.85	0.425	12.1	62.6	49.5	0.161	6.4	5.79	11.5
	0.425	0.21	8.2	70.8	37.4	0.178	4.8	5.01	6.8
	0.21	0.15	4.9	75.7	29.2	0.154	2.5	5.21	4.2
	0.15	0.074	6.9	82.6	24.3	0.315	7.1	5.69	6.5
	0.074	Pan	17.4	100.0	17.4	0.367	20.9	5.11	14.7
Totals and Averages			***100.0***	***-***	***-***	***0.305***	***100.0***	***6.17***	***100.0***

Table 13.26: A Summary of Bottle Roll Test Results,
McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data contained in McClelland Laboratories' 2012 report to the Company)

Parameter	Sample											
	CC, -1" #2		1 x Thru Rolls		2 x Thru Rolls #1		2 x Thru Rolls #2		1 x Thru Rolls		2 x Thru Rolls #1	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Extracted Metal in 2 hours	6.5	4.7	15.4	14.5	14.4	16.9	35.1	17.8	24.9	26.8	26.0	27.9
…. 6 hours	10.0	7.0	23.4	21.3	23.8	23.9	48.6	22.2	49.1	42.3	53.0	43.8
…. 24 hours	20.1	12.0	40.8	30.5	46.9	36.2	62.2	27.2	87.1	69.1	91.0	74.1
…. 48 hours	25.1	14.7	47.1	34.7	54.1	41.2	64.0	30.0	95.7	76.3	97.2	81.4
…. 72 hours	28.1	16.7	50.4	37.1	57.3	43.6	65.8	31.7	96.1	78.4	97.3	83.2
…. 96 hours	30.6	17.9	53.2	38.1	59.0	44.6	67.6	33.3	96.5	80.0	97.3	84.2
Base Data												
Feed Size	-25.4 mm (1")		P_{85} 6.35 mm (1/4")		P_{95} 6.35 mm (1/4")		P_{95} 6.35 mm (1/4")		P_{80} 200 mesh		P_{80} 200 mesh	
Tail Grade (g/t)*	1.920	21.94	1.131	13.03	1.200	14.06	0.137	2.06	0.103	5.14	0.069	4.11
Extracted Grade (g/t)	0.857	4.80	1.303	8.23	1.714	11.31	0.274	4.11	2.537	20.57	2.743	21.94
Calculated Head (g/t)	2.743	26.74	2.434	21.26	2.914	25.37	0.377	6.17	2.640	25.71	2.811	26.06
Head Assay (g/t)	2.811	21.60	2.606	26.74	2.537	25.71	0.309	6.17	2.606	26.74	2.537	25.71
Chemistry												
Cyanide Consumption (kg/t)	<0.05 lb/st		0.19		0.07		0.21		0.15		0.29	
Lime Consumption (kg/t)	1.7		1.6		1.8		1.9		1.8		1.8	
Final pH	11.1		10.8		10.7		11.0		10.8		10.6	

Note: * - average of three assays

Figure 13.16: Bottle Roll Test Metallurgical Recovery Curves for Different Sized Material, McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data contained in McClelland Laboratories' 2013 report to the Company)



13.8.4 Column Leach Tests

Column percolation leach tests were carried out on samples 1 x Thru Rolls (P_{85} 6.35 mm, ¼”), 2 x Thru Rolls #1 (P_{95} 6.35 mm) and 2 x Thru Rolls #1 (P_{95} 6.35 mm), to determine precious metal recovery, recovery rates, reagent requirements and amenability to agglomeration heap leaching. Each of the tests followed the same procedure:

- agglomerate each approximately 40 kg charge using 2.0 kg/t of material, approximately 9% moisture and a 72 hour cure;
- load agglomerates into 0.1 m x 0.3 m PVC leach columns in a manner to minimize compaction;
- apply a leach solution of 1.0 g/L over the charges at a rate of approximately 0.20 L/minute/m^2 of column cross-section area;
- measure daily pregnant solution volumes by weighing and sample (30 mL) for gold, silver, pH and cyanide;
- pump daily pregnant solutions through a three-stage carbon circuit for adsorption of the dissolved metals;
- measure daily barren solution volumes by weighing and sample (30 mL) for gold, silver, pH and cyanide;
- add make-up water and cyanide and recycle barren solutions daily;
- advance carbons when value breakthrough to barren occurs, assay all loaded carbons for gold and silver when leaching and rinsing is complete;
- continue the above daily procedure until Au values in pregnant solution approach AA detection limits, at which time initiate a 1 week test/1 week leach intermittent cycle;
- when cyanide leaching is terminated, initiate a water rinse cycle to meet Washoe County regulatory requirement (no detectable cyanide, gold or silver);
- after rinsing, let column charges free drain then remove residues, air dry and assay for gold and silver.

The results are summarized on Table 13.27 and Figure 13.17. It may be seen that excellent gold recovery rates were achieved and that minor improvements might have been realized if the tests had had run for a longer period. Silver recoveries were more varied and this, in conjunction with increasing recovery within increasing fineness of the head feed repeats the same outcomes reported for earlier test programs. In common with KCA’s 2011/2012 column leach tests, cyanide consumptions were high (as might be expected for the test type).

13.8.5 Tail Screen Analysis

Table 13.28 summarizes the tail screen analyses for the column leach feed. It may be seen that, although not as marked as for the analyses completed by Metcon in 2008 and KCA in 2011/2012, there is a relationship between particle size and metal recovery. Figure 13.18 emphasizes this.

Table 13.27: A Summary of Column Leach Test Results, McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data in McClelland Laboratories' two 2013 reports to the Company)

Parameter	Sample					
	1 x Thru Rolls		2 x Thru Rolls #1		2 x Thru Rolls #2	
	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
% Metal extracted, day 2	4.1	2.1	0.9	0.0	4.8	0.6
…. day 5	29.7	17.3	42.6	26.4	56.7	20.2
…. day 10	41.0	28.1	56.7	43.4	70.2	26.4
…. day 15	46.0	33.9	62.2	51.9	74.0	29.2
…. day 20	49.2	37.3	65.3	55.9	76.0	30.9
…. day 30	53.8	41.9	69.9	61.4	77.9	32.0
…. day 40	57.7	45.5	73.1	65.2	79.8*	33.7*
…. day 60	64.4	51.6	77.9	70.2	82.7	35.4
…. day 78	69.5	55.8	80.8	73.4	-	-
…. day 92	72.7	58.4	82.7	75.3	-	-
…. day 106 (end of leach)	75.2	60.5	84.1	76.6	82.7**	36.0
…. day 112 (end of rinse)	75.3	61.3	84.6	76.6	82.7***	36.0
Base Data						
Feed Size	P_{85} 6.35 mm		P_{95} 6.35 mm		P_{95} 6.35 mm	
Tail Grade (g/t)****	0.617	8.23	0.411	5.14	0.062	3.91
Extracted Grade (g/t)	1.886	13.03	2.263	16.80	0.295	2.19
Calculated Head (g/t)	2.503	21.26	2.674	21.94	0.357	6.10
Head Assay (g/t)	2.537	23.66	2.709	24.69	0.329	5.69
Chemistry						
Cyanide Consumption	1.48		1.57		0.82	
(kg/t)	2.0		2.0		2.0	
Lime Consumption (kg/t)	10.6		10.7		11.0	
Final pH						

Notes: * - recovery on Leach Day 44, ** - recovery on Leach Day 63,
*** - recovery on Leach Day 73, **** - average of three assays

Figure 13.17: Column Leach Test Metallurgical Recovery Curves, McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data in McClelland Laboratories' two 2013 reports to the Company)

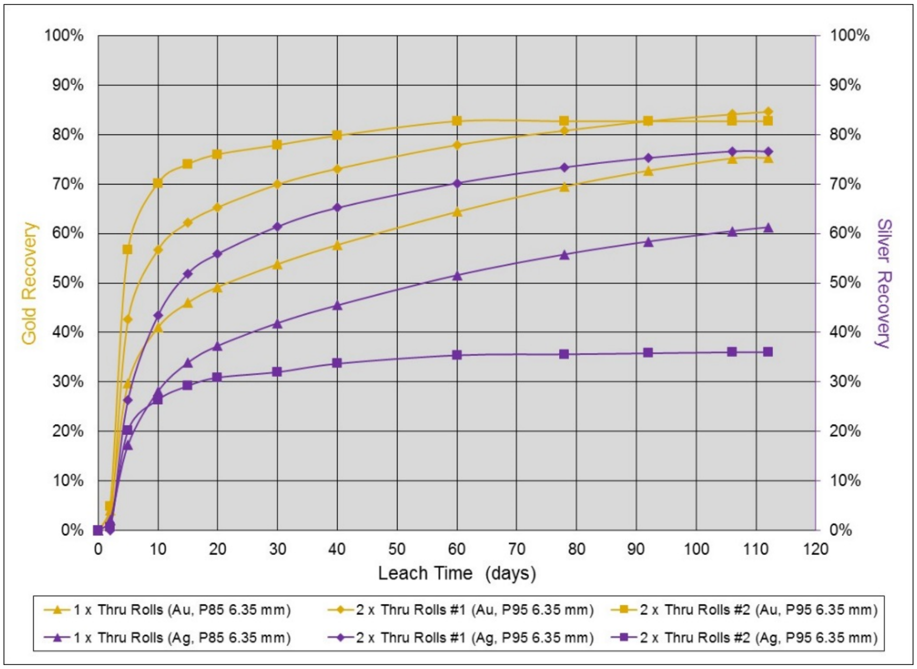
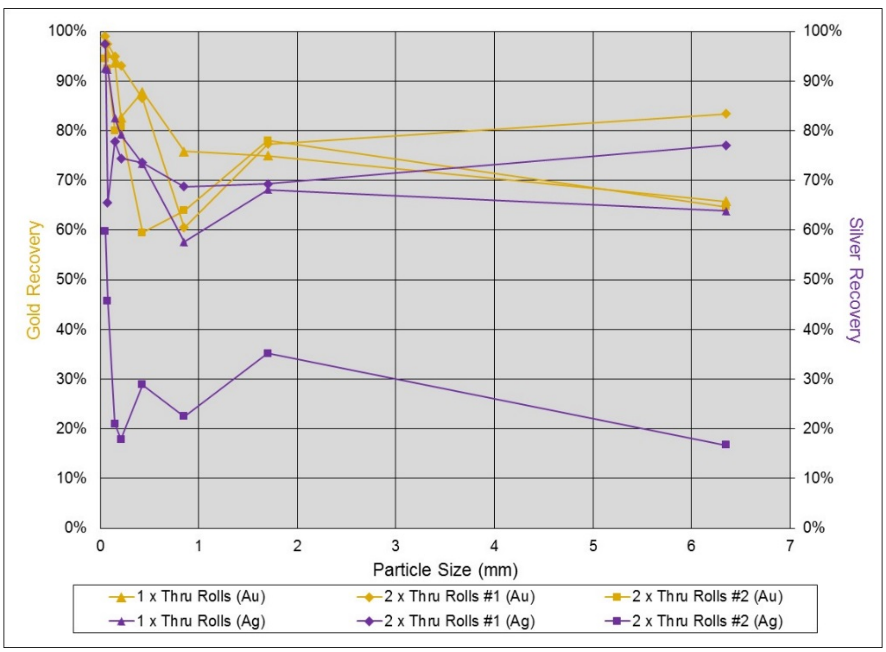


Table 13.28: A Summary of the Head, Recovered and Tail Assays by Size Fraction, McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data in McClelland Laboratories' two 2013 reports to the Company)

Composite	Screen Fraction (mm)	Head Screen Assays		Tail Screen Assays		Extraction by Fraction	
		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
1 x Thru Rolls (P_{85} 6.35 mm)	6.35	2.808	24.58	0.960	8.9	65.8	63.8
	1.70	3.021	30.99	0.754	9.9	75.0	68.1
	0.85	2.270	20.98	0.549	8.9	75.8	57.6
	0.425	2.541	18.00	0.309	4.8	87.8	73.3
	0.21	1.190	12.99	0.206	2.7	82.7	79.2
	0.15	1.639	12.00	0.103	2.1	93.7	82.5
	0.074	1.550	12.99	0.069	1.0	95.5	92.3
	pan	1.649	41.01	0.034	3.1	97.9	92.4
2 x Thru Rolls #1 (P_{95} 6.35 mm)	6.35	4.550	30.17	0.754	6.9	83.4	77.1
	1.70	2.719	29.01	0.617	8.9	77.3	69.3
	0.85	1.389	22.01	0.549	6.9	60.5	68.7
	0.425	3.051	22.01	0.411	5.8	86.5	73.6
	0.21	1.971	16.01	0.137	4.1	93.0	74.4
	0.15	1.371	13.99	0.069	3.1	95.0	77.8
	0.074	1.320	8.98	0.034	3.1	97.4	65.5
	pan	3.401	37.99	0.034	1.0	99.0	97.4
2 x Thru Rolls #2 (P_{95} 6.35 mm)	6.35	0.466	6.99	0.165	5.8	64.7	16.7
	1.70	0.405	7.41	0.089	4.8	78.0	35.2
	0.85	0.209	5.31	0.075	4.1	63.9	22.5
	0.425	0.161	5.79	0.065	4.1	59.5	28.9
	0.21	0.178	5.01	0.034	4.1	80.7	17.9
	0.15	0.154	5.21	0.031	4.1	80.0	21.0
	0.074	0.315	5.69	0.024	3.1	92.4	45.8
	pan	0.367	5.11	0.021	2.1	94.4	59.7

Figure 13.18: A Scatter Plot of Particle Size vs. Gold and Silver Recovery, McClelland Laboratories, 2013 Test Program, Moss Mine Project
(compiled from data in McClelland Laboratories' two 2013 reports to the Company)



13.9 Deleterious Elements

The Moss deposit is described in Section 8 as of the low sulphidation, epithermal type. Mercury, antimony, arsenic and thallium are identified as potential associated elements, as are base metals such as copper, lead and zinc. The associated elements can be environmentally problematic; base metals (copper in particular) can reduce the efficiency of cyanidation resulting is sometimes significant reductions in gold and silver recovery.

13.9.1 Head Analysis – KCA 2010 Test Program

As part of its 2010 metallurgical testwork program, KCA carried out head analyses on portions of the Moss Mine composites described in Section 13.5, including copper assays and determination of total sulphur. The following procedures were employed:

- head assays were completed utilizing standard lead collection fire assaying procedures -
  - total copper was determined utilizing ICAP–OES (inductively coupled argon plasma – optical emission spectrophotometer) and cyanide-soluble copper by FAAS methods;
- total sulphur (sulphur speciation) was determined by means of a LECO CS 400 sulphur determinator with induction furnace –
  - each sample set included two quality control samples, a blank and a standard check,
  - analyses of the material samples determined total sulphur,
  - sulphate determinations were made by pre-treating the samples in an electric kiln, and
  - sulphide sulphur was calculated by subtracting the sulphate sulphur from the total sulphur.

The results are summarized on Table 13.29. It can be seen that very minor copper is present in the assayed samples, along with very minor sulphur. The small amount of sulphide aligns with the mineralogical analysis presented in Sub-Section 7.2.4 (apart from acanthite, only minor pyrite is present in the mineralized mass).

Table 13.29: A Summary of Average Head Analysis Results, KCA, 2010 Test Program, Moss Mine Project
(compiled from data contained in KCA's 2011 report to Patriot Gold)

Composite	Copper (ppm)		Total Sulphur (%)	Sulphide Sulphur (%)	Sulphate Sulphur (%)
	Total	Soluble			
#1	13	5	0.24	0.14	0.10
#2	7	7	0.40	0.31	0.09

13.9.2 Head Analysis – KCA 2011/2012 Program

The same outcomes as described for KCA's 2010 analysis were realized from KCA's analysis of samples of the zone composites described in Section 13.6, as part of its

2011/2012 testwork program, using a LECO CS 400 unit. Table 13.30 summarizes the results determined using ICAP-OES and FAAS methods for copper. The analysis also included mercury, very minor amounts of which were found using cold vapor/atomic absorption methods.

Table 13.30: A Summary of Average Head Analysis Results, Zone Composites, KCA, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's November 2012 report to Patriot Gold)

Composite	Total Sulphur (%)	Sulphide Sulphur (%)	Sulphate Sulphur (%)	Copper (ppm)		Mercury (ppm)
				Total	Soluble	
Upper Zone	0.24	0.15	0.09	15	5.33	0.08
Lower Zone	0.20	0.13	0.07	17	5.41	0.06

13.9.3 Multi-Element Analysis

Three separate multi-element analyses of samples have been carried out: by KCA in 2010 on two composite samples by means of ICAP-OES; by KCA, as part of its 2011/2012 test program for the Moss Mine Project, on the two zone composites described in Section 13.6 and by means of ICAP-OES; and in 2013 by ALS Chemex on behalf of McClelland Laboratories, using four-acid digestion and ICP-AES and ICP-MS instruments. The results are summarized on Table 13.31, from which it may be seen that:

- there is good results repeatability between samples and laboratories;
- there is no analyzed element that stands out as exceptional in terms of its assay grade;
- very minor to negligible amounts of copper, lead and zinc are present in the tested samples (which for copper, replicates KCA's head analysis results summarized above and which conforms with the deposit type characterization described in Section 8);
- very minor to negligible amounts of antimony, arsenic, mercury and thallium are present in the tested samples (which for mercury, replicates KCA's 2011/2012 head analysis results summarized above); and
- overall, mineralized material from the Moss deposit may be described as 'clean', insofar as the amounts of potentially deleterious elements are minor to negligible.

13.10 Qualified Person's Opinion

The Qualified Person for this section of this Technical Report is Dr. David Stone, P. Eng. The following interpretation of the Moss Mine Project metallurgical testwork programs represents the opinion of the Qualified Person as regards the overall scope and applicability of the overall database of metallurgical testwork results and the amenability to cyanidation of mineralized material from the Moss deposit.

To fill the only data gap identified as a result of the review, it is recommended that up to six standard bottle roll tests are carried out on P_{95} 6.35 mm (1/4") crushed drillcore samples of West Extension mineralized material (vein and stockwork). Bottle roll tests only are required due to the very good repeatability between bottle roll and column leach tests across the seven metallurgical test programs that included cyanidation testing data (which in part attests to the ready amenability to cyanidation of the tested material).

Table 13.31: A Summary of the Results of the Three Multi-Element Analyses Carried Out on Samples of Moss Deposit Mineralized Material, Moss Mine Project

(compiled from data contained in KCA's 2010 and 2011/2012 reports and McClelland Labs. 2013 report cited in earlier sub-sections)

Constituent	Unit	KCA Sample # (2010)		KCA Sample # (2011/2012)		McClelland Labs. (2013)
		44181B	44182B	Upper Zone	Lower Zone	2 x Thru Rolls #2
Aluminum (Al)	%	4.63	4.84	3.68	4.49	5.43
Arsenic (As)	%	0.0043	0.0045	0.0019	0.0022	0.0015
Barium (Ba)	ppm	649	714	528	615	900
Beryllium (Be)	ppm	-	-	-	-	6.46
Bismuth (Bi)	ppm	11	8	13	13	19
Carbon (total)	%	1.77	1.08	1.70	1.08	-
Calcium (Ca)	%	5.64	3.21	5.33	3.37	1.61
Cadmium (Cd)	ppm	6	6	1	1	0.25
Cerium (Ce)	ppm	-	-	-	-	102.0
Cobalt (Co)	ppm	4	6	4	4	6
Chromium (Cr)	ppm	50	60	60	55	17
Caesium (Cs)	ppm	-	-	-	-	1.51
Copper (Cu)	%	0.0013	0.0019	0.0015	0.0017	0.0025
Cu (cyanide soluble)	ppm	5.06	7.18	5.33	5.41	-
Iron (Fe)	%	1.57	1.47	1.33	1.47	2.27
Gallium (Ga)	ppm	-	-	-	-	14.55
Germanium (Ge)	ppm	-	-	-	-	0.12
Hafnium (Hf)	ppm	-	-	-	-	0.7
Mercury (Hg)	ppm	0.06	0.07	0.08	0.06	0.04
Indium (In)	ppm	-	-	-	-	0.044
Potassium (K)	%	3.33	3.74	2.82	3.35	4.00
Lanthanum (La)	ppm	-	-	-	-	523
Lithium (Li)	ppm	-	-	-	-	48.9
Magnesium (Mg)	%	0.67	0.63	0.54	0.61	0.63
Manganese (Mn)	ppm	513	402	539	422	369
Molybdenum (Mo)	ppm	1	6	2	3	3.38
Sodium (Na)	%	1.31	1.35	0.88	1.04	1.45
Niobium (Nb)	ppm	-	-	-	-	14.9
Nickel (Ni)	%	0.0014	0.0014	0.0011	0.0011	0.0012
Phophorous (P)	ppm	-	-	-	-	740
Lead (Pb)	%	0.0030	0.0063	0.0013	0.0023	0.0027
Rubidium (Rb)	ppm	-	-	-	-	160.0
Rhenium (Re)	ppm	-	-	-	-	<0.002
Sulphur (total)	%	0.24	0.40	0.24	0.20	0.01
Sulphur (sulphide)	%	0.14	0.31	0.15	0.13	-
Sulphur (sulphate)	%	0.10	0.09	0.09	0.07	-
Antimony (Sb)	ppm	<2	<2	<2	<2	3.99
Scandium (Sc)	ppm	-	-	-	-	5.6
Selenium (Se)	ppm	<5	<5	<5	<5	<1
Tin (Sn)	ppm	-	-	-	-	1.5
Strontium (Sr)	ppm	318	234	279	228	232
Tantalum (Ta)	ppm	-	-	-	-	0.91
Tellurium (Te)	ppm	9	7	4	3	0.11
Thorium (Th)	ppm	-	-	-	-	13.8
Titanium (Ti)	%	0.19	0.22	0.16	0.18	0.29
Thallium (Tl)	ppm	-	-	-	-	1.71
Uranium (U)	ppm	-	-	-	-	1.50
Vanadium (V)	ppm	29	25	24	27	41
Tungsten (W)	ppm	<10	<10	30	28	1.8
Yttrium (Y)	ppm	-	-	-	-	14.6
Zinc (Zn)	%	0.0048	0.0057	0.0037	0.0043	0.0050
Zirconium (Zr)	ppm	-	-	-	-	18.9

The data gap does not affect the principal outcomes of the metallurgical review, hence the opinions stated here. Robust assessments of heap leach metallurgical recovery rates may instead be made and recovery rates of 82% for gold and 65% for silver may be applied for purposes of Mineral Resource estimation and Moss Mine Project planning, as long as:

- the heap leach feed comprises P_{95} 6.35 mm (1/4”) mineralized material of the type used during the Phase I heap leaching operation;
- the crushed and screen mineralized material is agglomerated using cement; and
- a Merrill-Crowe type recovery system for silver is employed.

An overall gold recovery rate of 82% for gold was achieved from the heap leach during the Phase I. Silver recovery was, however, lower than the recommended 65% because a Merrill-Crowe type recovery system for silver was not employed.

13.10.1 Deposit Characterization

The geological and mineralogical characteristics of the Moss deposit described in Section 7.2 show it is a conventional oxide deposit type and as such it does not display variable metallurgical responses and does not, therefore, require different metallurgical processes to optimize metal recoveries. It may instead be considered as a single metallurgical entity:

- analysis of the oxidation characteristics presented in Section 7.2.5 shows that oxidation (as evidenced by the presence of limonite) extends to depths of at least 210 m below the surface watertable;
- as earlier described (Section 7.3) the economic minerals of interest are native gold, electrum (a naturally occurring alloy of gold and silver) and acanthite (a silver sulphide) encapsulated in quartz and, to a lesser extent, calcite;
- native gold, electrum and gangue minerals such as quartz are not susceptible to surface weathering effects/oxidation; and
- apart from acanthite, the presence of sulphides is limited to minor to very minor pyrite –
    - sample analyses by KCA in 2010 and 2011/2012 show that the sulphur content in the form of sulphide is limited to less than 0.31% (average 0.18%),
    - the paragenetic sequence for the deposit (Sub-Section 7.2.4.2) shows that two minor pyrite phases exist - the first pre-dates the gold-silver mineralization phase of interest and the second is contemporaneous but not associated with the gold-silver mineralization of interest, therefore
    - gold and silver recovery from native gold and electrum would not be constrained by considerations of encapsulating sulphides or oxides, however
    - silver recovery from acanthite is a special case that is discussed in Sub-Section 13.10.3.

13.10.2 Amenability to Cyanidation

It is established in Sub-Section 7.2.4 that the principal economic minerals of interest are in the form of fine to ultrafine grains, that the importance of this key physical characteristic is in the surface area to volume ratio (or SA:V) of the grains which, by definition, is very high and that the minerals of economic significance can, therefore, be expected to rapidly adsorb into solution when exposed to sodium cyanide. The results of the metallurgical tests bears this finding out:

- maximum liberation of the target minerals would be achieved in pulverized samples, such as those with P_{80} to P_{100} 100 mesh (0.15 mm) to 200 mesh (0.074 mm) sizes;
- cyanide shake and/or bottle roll tests, on pulverized samples by Metcon in 2008, by KCA in 2010, in 2011/2012 and in 2012, and by McClelland Laboratories in 2013 consistently achieved recoveries in excess of 88% and up to 99% for gold and in excess of 80% and up to 95% for silver;
- up to 60% gold recovery was typically realized within five or six hours of the introduction of cyanide solution; and
- the recoveries and recovery rates outlined were achieved irrespective of the reported cyanide consumption rate (that was very low for the Metcon tests and very high for both sets of cyanide shake tests, the latter as may be expected for the test type).

In other words, the Moss deposit is very amenable to cyanidation, especially as regards the recovery of gold that is consistently rapid and comprehensive in fine grained and pulverized head feeds. However, the same metallurgical test results consistently show that variability in silver recovery exists when the nominal size of the crushed material exceeds approximately P_{80} 6.35 mm. The potential for a reduced overall average silver recovery rate is included in the recommended heap leach recovery rate for silver: analysis shows that for a heap leach feed comprising P_{95} 6.35 mm (1/4”) mineralized material of the type used during the Phase I heap leaching operation, silver recovery of approximately 73% could in theory be achieved (see Sub-Section 13.10.5).

13.10.3 Silver Recovery

Test work by McClelland Laboratories in 1991, by Metcon in 2008 and by KCA in 2011/2012 demonstrates well the uniformity of the Moss Vein as regards gold recovery on cyanidation: very good results’ repeatability between tests was realized, irrespective of sample grade, geographical location or depth. The KCA 2011/2012 test series shows that this is not the case with silver: variability in the silver recovery rate by grade was found, as Figure 13.19 suggests. Variability in silver recovery was also found in the other test programs, with the exception of McClelland Laboratories’ 1991 program that did not consider silver. There is no quantitative evidence to suggest why this might be the case:

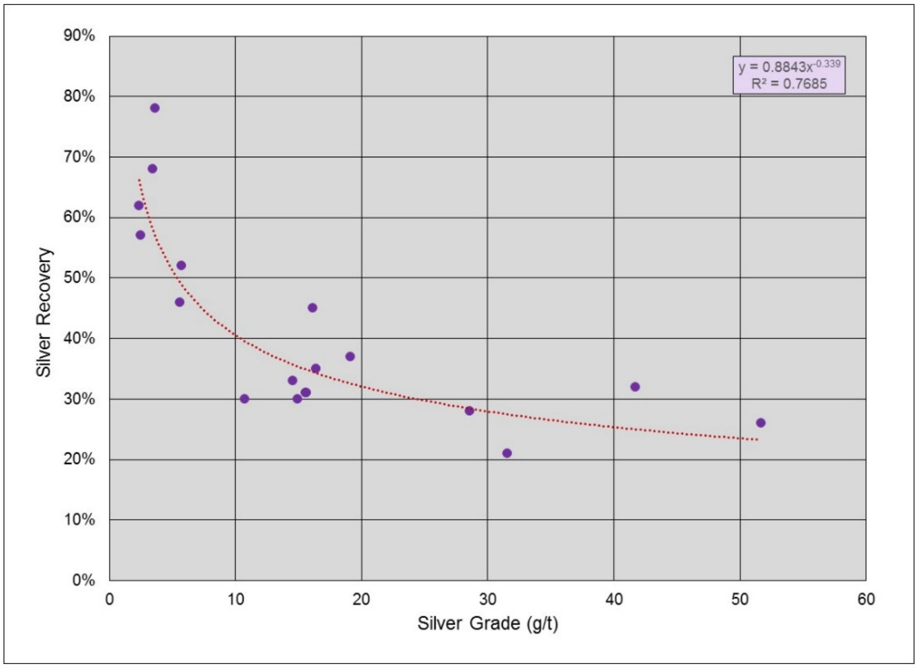
- total silver grades would inevitably be captured during laboratory head and tail assays using either the fire assay or four-acid digestion method with an AA finish (i.e. the methods used within the scope of the metallurgical test programs described above);
- the method employed of determining final recovery rates (subtracting the assayed tail grades from the calculated head, dividing the result by the calculated head and multiplying the outcome by 100) is robust and does not introduce error;
- the behaviour of electrum on cyanidation will be the same as for native gold; but
- the maximum silver content in electrum typically approximates to 80% by weight; however
- the assay database for the Moss Vein shows that, above a cut-off of 0.25 g./t Au, –
    - the gold grades vary up to 29.74 g/t Au (average 1.17 g/t Au), and
    - the silver grades vary up to 305.5 g/t Ag (average 11.84 g/t Ag); therefore

- either above-average amounts of acanthite or above average amounts of silver-rich electrum along with little or no native gold must be present where significant silver grades exist.

The deportment of silver in high and very high silver grade material needs to be confirmed by thin section analysis. If the dominant mineral is acanthite, the strong correlation between silver recovery and silver grade for KCA's 2011/2012 series of coarse bottle roll tests (Figure 13.19) may be explained:

- at a nominal feed size of 12.7 mm, liberation of the acanthite would likely be variable but moderate at best (a significant fraction would probably have remained encapsulated in the gangue minerals; therefore
- the cyanide solution would have been unable to effectively dissolve what might be coarse grains of acanthite; therefore
- the overall silver recoveries reduced were low or very low, despite 408 hours of cyanidation over the term of the bottle roll tests; and
- the limited amount of physical rolling (one minute in every hour – see Section 4.5) would only serve to exacerbate recovery potential.

Figure 13.19: A Scatter Plot of Calculated Silver Head Grades and Recoveries for Coarse Material, KCA, 2011/2012 Test Program, Moss Mine Project
(compiled from data contained in KCA's report to Patriot Gold entitled 'Moss Mine Project, Report on Metallurgical Testwork, November 2012')



13.10.4 Results' Repeatability

Table 13.32 summarizes the recovery rates achieved over the eight metallurgical test programs described above, by test type and feed size. The results of the 18 bottle roll tests on the P_{100} 12.7 mm (1/2") regional, grade and zone composites of KCA's 2011/2012 test program are not included for the reasons discussed in Sub-Section 13.6.6: intermittent rolling resulted in gold recovery rates that were up to 50% lower (approximately 30% lower on average) than the recoveries reported for similarly sized material in other test programs. This renders the results unsuitable for consideration in test repeatability analysis (but valid for purposes of metallurgical variability analysis). Figures 13.20 and 13.21 are scatter plots of the same data for gold (Figure 13.20) and silver (Figure 13.21). All the data points are for P_{80} material, except those with black borders that are for P_{85} to P_{100} material, as detailed on Table 13.32.

It may be seen that while there is results variability for each head feed particle size (which reflects variable test conditions and feed characteristics), the overall database of test results reflects a robust repeatability between test types: no test type consistently reports higher or lower results than any other test type. The results for each head feed particle size are instead mixed. In the opinion of the Qualified Person, this confirms the straightforward nature of the metallurgical response of the economic minerals of interest to cyanidation and it identifies that column leach tests are not ideally required to test the metallurgical response of mineralized material from the Moss Vein. Standard bottle roll tests may instead be used.

**Figure 13.20: A Scatter Plot of Gold Recoveries by Test Type,
Moss Mine Project Metallurgical Programs**

(compiled from data contained in the metallurgical test program reports cited above)

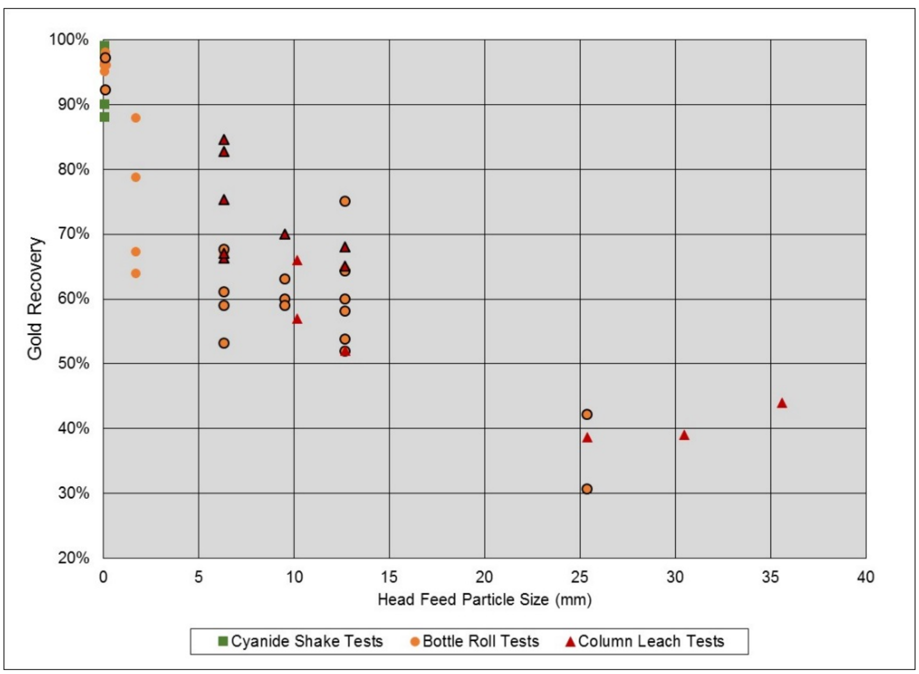


Table 13.32: A Summary of Metal Recovery Rates by Test Type and Head Feed Particle Size, Moss Mine Project

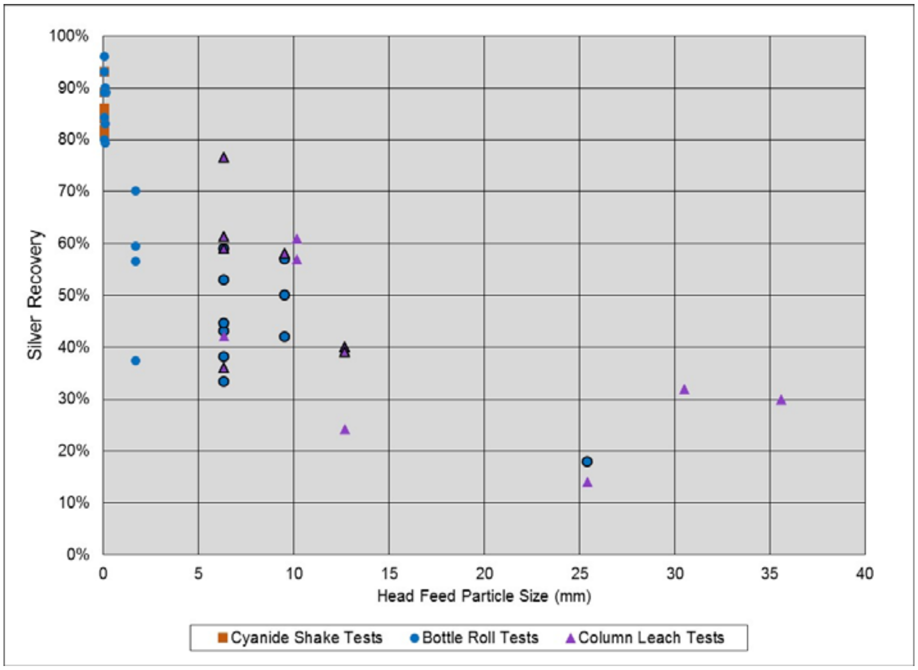
(compiled from data contained in the metallurgical test program reports cited above)

Source	Test Type	Sample Size (P_{80} unless otherwise stated)																					
		35.56 mm (1.4”)		30.48 mm (1.2”)		25.4 mm (1”)		12.7 mm (1/2”)		10.16 mm (2/5”)		9.53 mm (3/8”)		6.35 mm (1/4”)		1.7 mm (10 mesh)		0.15 mm (100 mesh)		0.105 mm (150 mesh)		0.09 mm (200 mesh)	
		Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)
McClelland Labs., 1991	BT	-	-	-	-	*42.1*	-	*60.0* *75.0* *51.9* *64.3* *53.8* *64.6* *58.1*	- - - - - - -	-	-	-	-	-	-	-	-	-	-	-	-	-	-
McClelland Labs., 1992	BT	-	-	-	-	-	-	-	-	-	-	-	-	-	-	87.9 78.7	70.0 59.4	-	-	-	-	-	-
Metcon Research, 2008	BT	-	-	-	-	-	-	-	-	-	-	-	-	-	-	*63.9* *67.2*	*37.4* *56.5*	-	-	*97.1* *92.2*	*79.4* *83.1*	-	-
	CT	-	-	-	-	38.7	14.1	52.0	24.2	-	-	-	-	66.3	42.1	-	-	-	-	-	-	-	-
KCA, 2010	ST	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	88 90 88 88	82 81 86 93
	BT	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	90 93	93 86
	CT	44	30	39	32	-	-	-	-	66 57	57 61	-	-	-	-	-	-	-	-	-	-	-	-
KCA, 2011/2012	ST	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98 99 99 99	85 84 89 86
	BT	-	-	-	-	-	-	-	-	-	-	*63* *60* *59*	*50* *57* *42*	*61* *67* *67*	*53* *59* *43*	-	-	-	-	-	-	95 96 96 96	89 93 93 96
	CT	-	-	-	-	-	-	*65* *68*	*40* *39*	-	-	*70*	*58*	*67*	*59*	-	-	-	-	-	-	-	-
KCA, 2012	BT	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	96	89	98	90	-	-
McClelland Labs., 2013	BT	-	-	-	-	*30.6*	*17.9*	-	-	-	-	-	-	*53.2* *59.0* *67.6*	*38.1* *44.6* *33.3*							96.5 97.3	80.0 84.2
	CT	-	-	-	-	-	-	-	-	-	-	-	-	*75.3* *84.6* *82.7*	*61.3* *76.6* *36.0*	-	-	-	-	-	-	-	-

Notes: ST = cyanide shake text, BT = bottle roll test, CT = column leach test
All samples P_{80}, except those highlighted in *GREEN* (P_{85}), in *RED* (P_{95}) or *PURPLE* (P_{100}).
The abnormally low Metcon results, highlighted in *ORANGE*, are attributed to the very low cyanide consumption realized during the tests.
Excluded from the summary are KCA’s 2011/2012 bottle roll test results on coarse material - the metal recoveries are up to 50% lower (approximately 30% lower on average) than the recoveries reported for similarly sized material, for the reasons discussed in Sub-Section 4.6.6.

Figure 13.21: A Scatter Plot of Silver Recoveries by Test Type, Moss Mine Project Metallurgical Programs

(compiled from data contained in the metallurgical test program reports cited above)



13.10.5 Correlations and Trends

13.10.5.1 Cyanide Consumption

Scrutiny of the datasets for each of the test programs described above shows that there is no correlation between cyanide consumption and either head grade of the tested samples or metal recovery from those samples:

- cyanide consumptions for KCA's cyanide shake tests were uniformly high (5.0 kg/t), as can be expected for the test type;
- with the exception of KCA's 2012 bottle roll tests on P_{80} 100 mesh samples for which abnormally high cyanide consumption rates were reported (0.47 kg/t and 0.96 kg/t), cyanide consumptions for the various bottle roll tests on pulverized material (P_{100} 150 mesh or finer) varied between approximately less than 0.1 kg/t and 0.3 kg/t;
- with the exception of KCA's 2011/2012 bottle roll test series for which very low cyanide consumptions were reported, cyanide consumptions for the various bottle roll tests on non-pulverized material varied between approximately 0.1 kg/t and 0.7 kg/t, with the majority varying around 0.3 kg/t; and
- at 0.8 kg/t to 2.40 kg/t (with only one test reporting a value lower than 1.0 kg/t and only one test reporting a value higher than 2.0 kg/t) cyanide consumption rates for the column leach tests were uniformly higher than those for the bottle roll tests, which is typical of test type.

13.10.5.2 Recovery and Particle Size

Figures 13.20 and 13.21 establish that there is a strong correlation between metallurgical recovery and the nominal size (diameter) of the mineralized material subjected to cyanide leaching. A similar relationship is evident from the results of the head screen analyses by Metcon in 2008, by KCA in 2011/2012 and by McClelland Laboratories in 2013.

Figures 13.22 and 13.23 are scatter plots that summarize the head screen analysis results and compare them with the recovery versus head feed particle size data summarized on Figures 13.20 and 13.31. It may be seen that while broadly similar correlations are apparent for both gold (Figure 13.22) and silver (Figure 13.23) and with the exception of the 100 mesh to 200 mesh pulps, the majority of the head screen analysis results report lower recovery rates compared with the results for bottle roll- and column leach-tests at the same nominal particle sizes. This result may be expected: the grain sizes of the economic minerals of interest are fine to ultrafine so any liberated grains would only be captured in either the fine to very fine sieved fractions or the pan. The effect will be to skew downwards the reported recoveries for all coarser sieve size fractions, except the finest and near finest. As much can be seen in the results summarized on Figures 13.22 and 13.23. In this regard it should be emphasized that the recovery rates by size fraction are calculated by subtracting the tail grade from the head grade for size fraction, dividing by the head grade and then multiplying by 100 to derive a percent recovery value.

Figure 13.22: A Scatter Plot of Gold Recoveries by Test Type, Head Screen Fraction and Particle Size, Moss Mine Project Metallurgical Programs
(compiled from data contained in the metallurgical test program reports cited above)

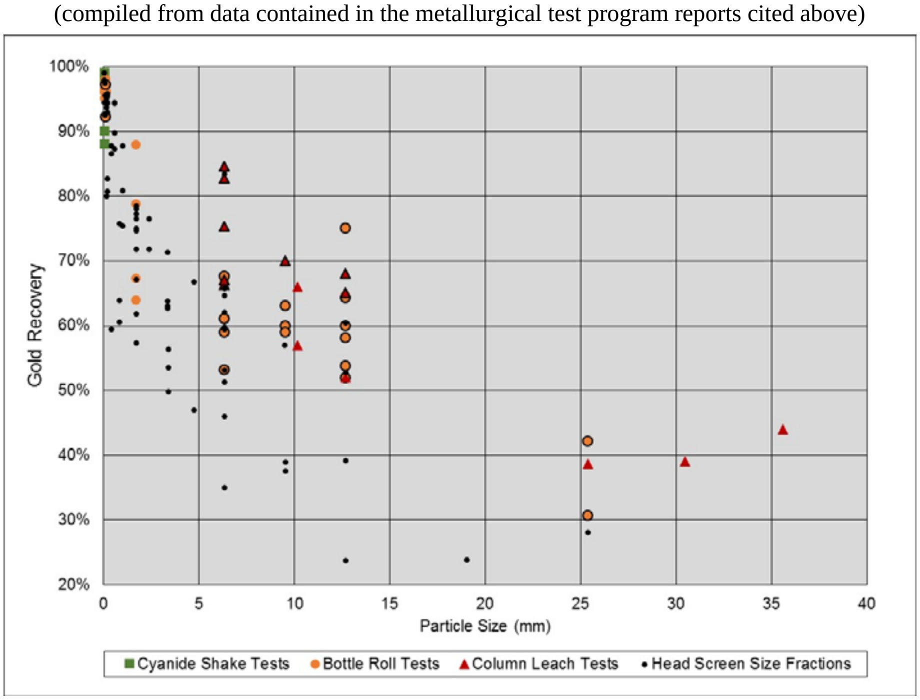
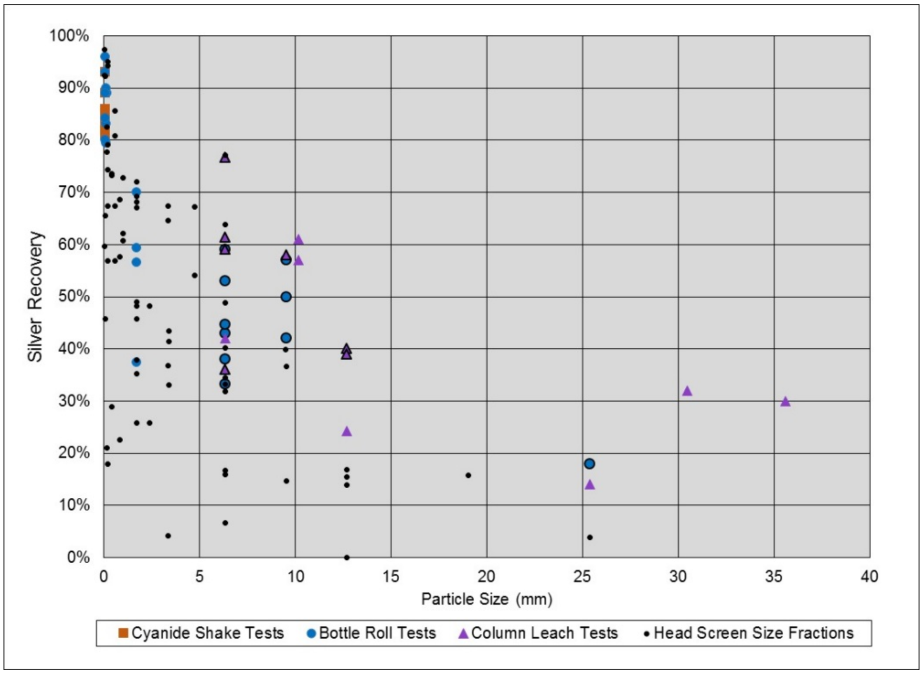


Figure 13.23: A Scatter Plot of Silver Recoveries by Test Type, Head Screen Fraction and Particle Size, Moss Mine Project Metallurgical Programs
(compiled from data contained in the metallurgical test program reports cited above)



It is for the reasons outlined that only those datasets summarized on Figures 13.20 and 13.21 were used for the basis of analysis to determine best case, average and worst case recovery curves for gold and silver. Figures 13.24 and 13.25 are scatter plots of the utilized database of results for gold (Figure 13.24) and silver (Figure 13.25) on which the outlier results excluded in analysis are highlighted and the best fit recovery curves and their mathematical expressions are identified. The following comments and conclusions apply:

- it is the nominal crush size of the material tested that for the most part determines the best, average and worst case recveory rates (the P_{80} to P_{100} value is of secondary importance – as long as the generated fines are present in the mix);
- the best case recovery curves were in both cases limited to the reported best case recoveries for column leach testing only (for example, in the opinion of the QP, a higher value for +P_{80} 12.70 mm material was achieved in a bottle roll test, but its use when defining the best case gold recovery curve would yield an unrealistic result);
- a very good correlation coefficient (r^2 = 0.8753, as defined by the average recovery curve) applies to the overall gold recovery dataset, exclusive of identified outliers, with exceptional correlation coefficients (r^2 = 0.992 and 0.9933) for the best case and worst case recovery curves, respectively;

- a good correlation coefficient (r^2 = 0.7829, as defined by the average recovery curve) applies to the overall silver recovery dataset (which reflects the greater scatter of data compared with the gold results), exclusive of identified outliers, with exceptional correlation coefficients (r^2 = 0.9702 and 0.9283) for the best case and worst case recovery curves, respectively;
- the best case gold recovery curve for +P_{80} 6.35 mm material predicts a recovery of 83.0% for gold (which is slightly higher than the 82.0% achieved from the Phase I heap leach, which suggests that the particle size distribution of the agglomerated, Phase I heap leach material was near ideal); and
- the best case silver recovery curve for +P_{80} 6.35 mm material predicts a recovery of 72.7% for silver, if a Merrill-Crowe type recovery process is employed.

Figures 13.24 and 13.25 confirm the earlier interpretation of the mineralogy and deportment of the economic minerals of interest:

- a strong relationship between particle size and recovery must exist:
- the amount of recovery is primarily related to the amount of native gold, electrum and acanthite that is liberated and thereby exposed to direct cyanidation;
- the quantity of the economic minerals of interest that is liberated must be directly proportional to the amount of work (i.e. crushing and/or grinding) carried out on the mineralized material prior to cyanidation (i.e. the finer the head feed the greater the recovery of both gold and silver); although
- at least partial cyanidation of gold and silver mineral encapsulated in especially calcite (and to a lesser extent quartz) would occur as cyanide solution penetrated the encapsulating gangue minerals along cracks and other micro-flaws; and
- the amount of such cyanidation would be proportional to the thickness of the surrounding gangue minerals, hence the extent to which a cyanide solution could penetrate the material (i.e. the thicker the surrounding gangue minerals the larger the particle size hence the less the amount of metal extracted by cyanidation).

Figure 13.24: A Scatter Plot of Gold Recoveries by Test Type and Nominal Head Feed Size, with the Interpreted Best Fit Best, Average and Worst Case Recovery Curves, Moss Mine Project
(compiled and interpreted from data contained in the metallurgical test program reports cited above)

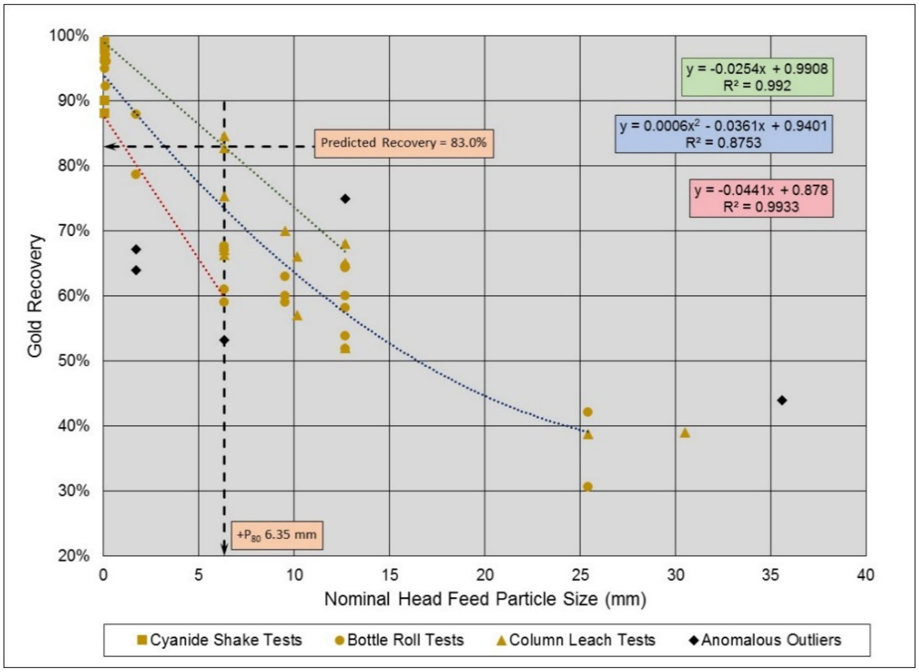
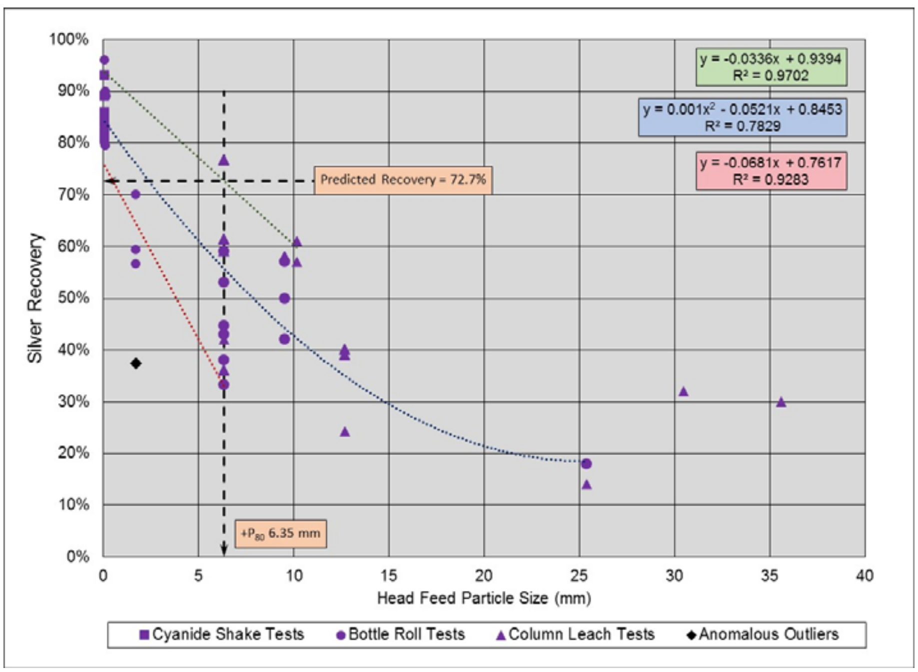


Figure 13.25: A Scatter Plot of Silver Recoveries by Test Type and Nominal Head Feed Size, with the Interpreted Best Fit Best, Average and Worst Case Recovery Curves, Moss Mine Project
(compiled and interpreted from data contained in the metallurgical test program reports cited above)



13.10.6 Metallurgical Test Coverage

13.10.6.1 By Grade

Figures 13.26 and 13.27 summarize the ranges of calculated head grades for gold (Figure 13.26) and silver (Figure 13.27) by test type for each of the cyanide shake-, bottle roll- and column leach-tests carried out over the seven test programs for which data is available. It may be seen that overall, the test series comprehensively covered the range of gold and silver grades available across the Moss deposit (it is earlier established that above a cut-off of 0.25 g/t Au, the assay database for the Moss deposit shows that the gold grades average 1.17 g/t Au and vary up to 29.74 g/t Au, whereas the silver grades average 11.84 g/t Ag and vary up to 305.5 g/t Ag).

Figure 13.26: A Scatter Plot of the Calculated Gold Head Grades of the Samples and Composites Used for Metallurgical Testing, by Test Type, Moss Mine Project
(compiled and interpreted from data contained in the metallurgical test program reports cited above)

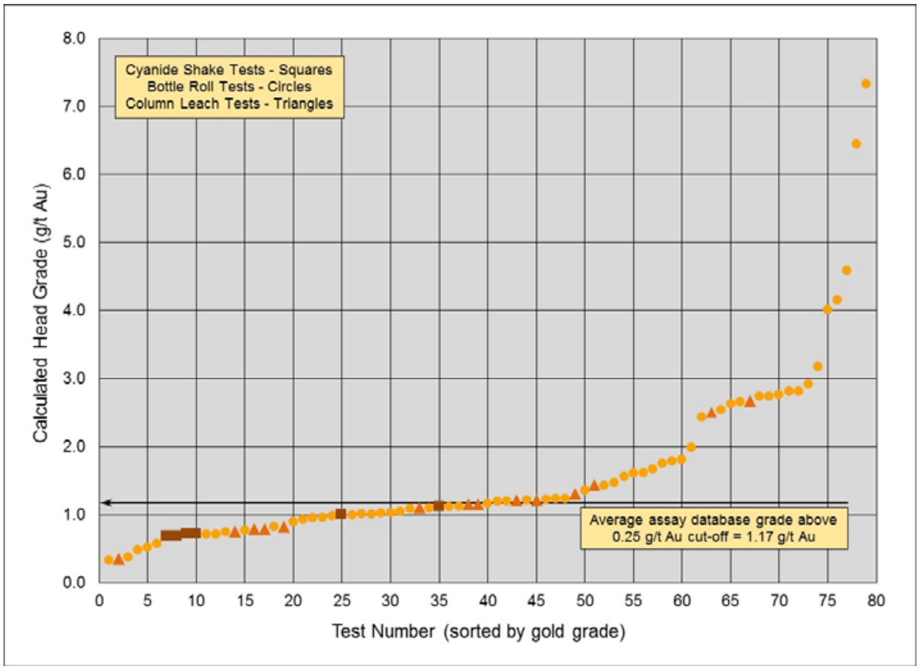
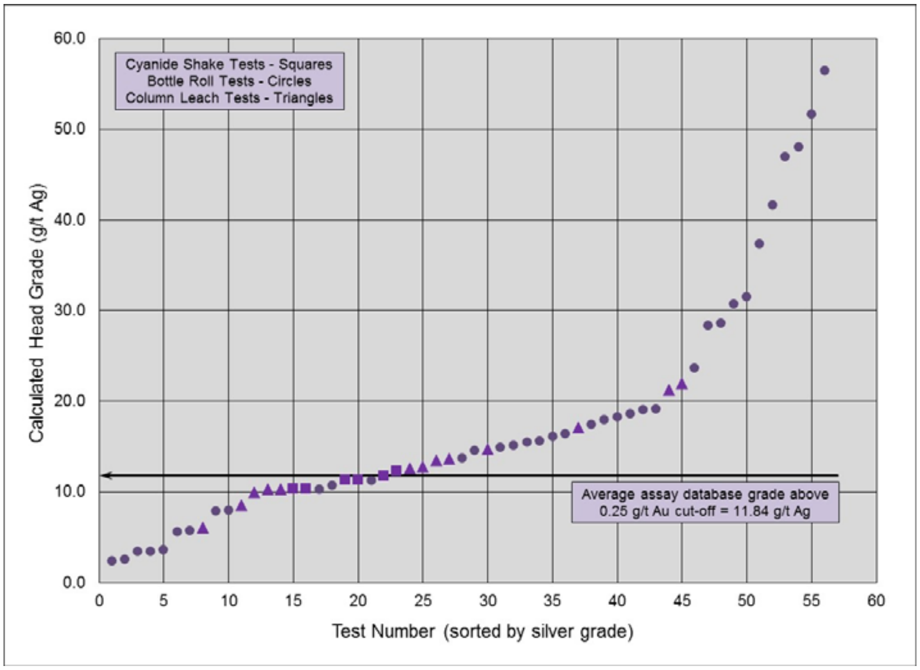


Figure 13.27: A Scatter Plot of the Calculated Silver Head Grades of the Samples and Composites Used for Metallurgical Testing, by Test Type, Moss Mine Project
(compiled and interpreted from data contained in the metallurgical test program reports cited above)



13.10.6.2 By Location and Depth

Figure 13.28 is a long-section, looking north, of the Moss Vein and West Vein on which are highlighted the sample intervals used over the seven metallurgical test programs that are described in Section 4 and that included cyanidation test results. Table 13.33 summarizes the 22 intersecting metallurgical drillhole samples that total 377.50 m in length:

- a very good distribution of samples is evident across the Moss Vein, within the estimated Phase II pit area (additional tests to cover the possibility of metallurgical variability along the strike length of the Moss Vein are not required); but
- samples for metallurgical testwork are conspicuous by their absence in the Western Extension (a targeted bottle roll metallurgical testwork program for mineralized material from the Western Extension is recommended).

The same general conclusions apply as regards the hangingwall and footwall stockworks. Figure 13.29 is a snapshot view of the Moss Vein's hangingwall stockwork (as defined by the 2014 MRM, looking north) on which are highlighted the 30 intersecting, metallurgical drillhole samples that total 452.10 m in length (Table 13.34). Figure 13.30 is a snapshot view of the two, minor Moss Vein footwall stockworks (as defined by the 2014 MRM, looking approximately north on which are highlighted the seven, intersecting metallurgical drillhole samples that total 26.68 m in length (Table 13.35).

Figure 13.28: A Long-Section Vulcan® Snapshot View (looking north) of the Moss Vein and West Vein Showing the Distribution of Metallurgical Test Samples (that are colour-coded by test program)

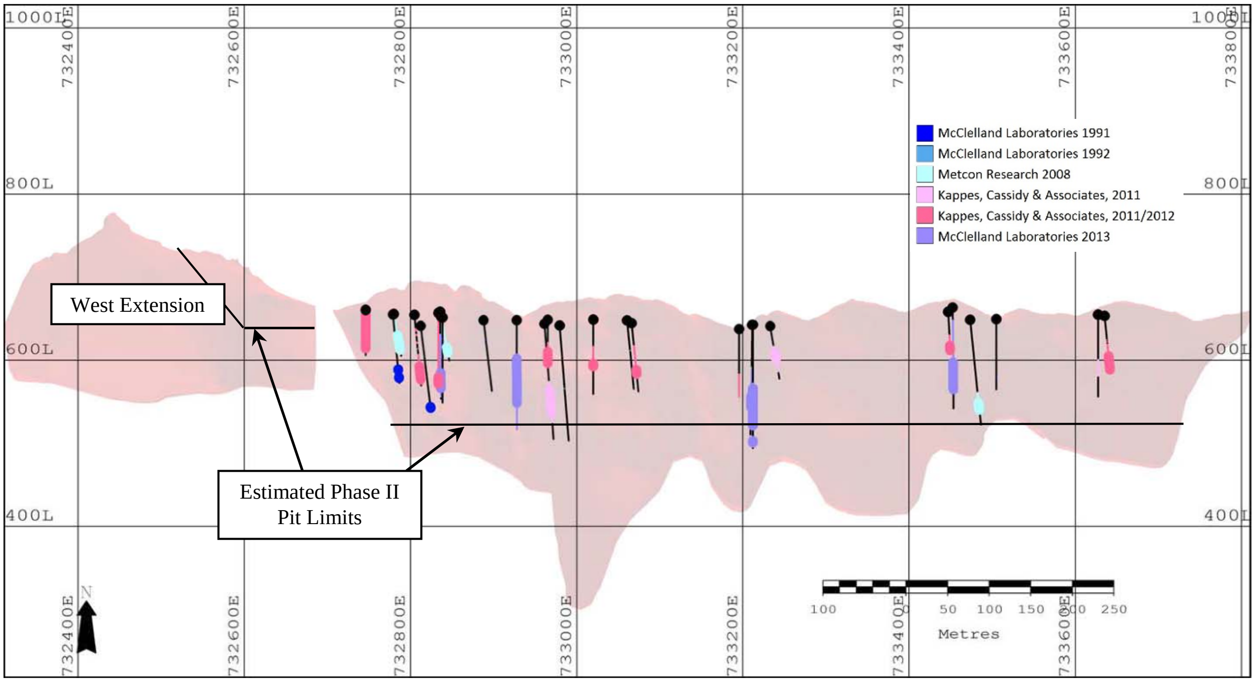


Table 13.33: A Summary of the Metallurgical Drillhole Samples that Intersect the Moss Vein, Moss Mine Project

(compiled from data contained in the metallurgical test program reports cited above)

Drillhole	Sample Interval From (m)	Sample Interval To (m)	Sample Length (m)	Test Program
MM-8	73.15	74.68	1.53	McClelland Laboratories, 1991
MM-8	83.82	85.14	1.32	
MM-14	108.20	61.89	1.53	
AR-48C	36.26	61.89	25.63	Metcon Research, 2008
AR-49C	51.60	61.75	10.15	
AR-50C	116.26	125.90	9.64	
AR-51C	88.61	118.74	30.13	Kappes, Cassidy & Associates, 2011
AR-52C	44.95	56.52	11.57	
AR-70C	61.57	65.96	4.39	Kappes, Cassidy & Associates, 2011/2012
AR-71C	62.26	68.58	6.32	
AR-72C	78.43	85.95	7.52	
AR-73C	3.05	46.94	43.89	
AR-74C	68.58	86.56	17.98	
AR-75C	44.70	60.95	16.24	
AR-76C	56.62	75.83	19.21	
AR-77C	46.39	53.34	6.95	
AR-188C	73.83	92.20	18.37	McClelland Laboratories, 2013
AR-189C	46.85	100.40	53.55	
AR-190C	86.58	99.97	13.39	
AR-191C	66.23	98.91	32.68	
AR-193C	77.19	121.92	44.73	
AR-193C	140.21	142.53	2.32	
		Total	***377.50***	

Figure 13.29: A Long-Section Vulcan® Snapshot View (looking north) of the Hangingwall Stockworks of the Moss Vein and West Vein Showing the Distribution of Metallurgical Test Samples (that are colour-coded by test program)

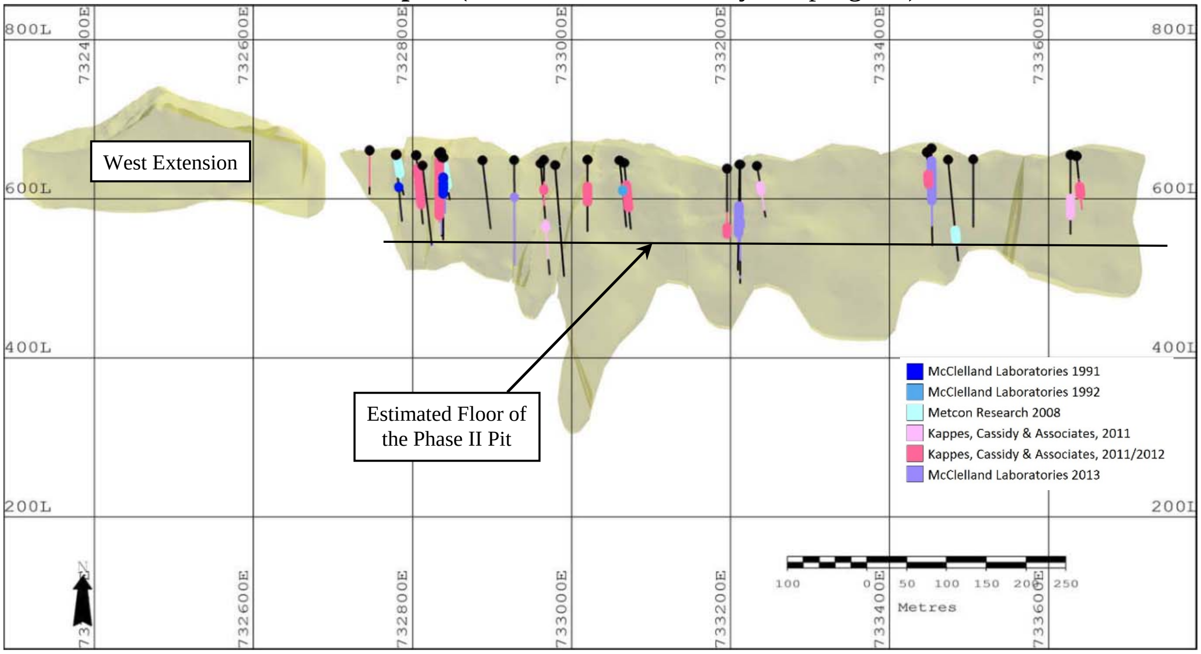


Table 13.34: A Summary of the Metallurgical Drillhole Samples that Intersect the Hangingwall Stockwork of the Moss Vein, Moss Mine Project
(compiled from data contained in the metallurgical test program reports cited above)

Drillhole	Sample Interval From (m)	Sample Interval To (m)	Sample Length (m)	Test Program
MM-1	47.24	48.77	1.53	McClelland Laboratories, 1991
MM-2	35.05	38.10	3.05	
MM-2	45.72	47.24	1.52	
MM-2	48.77	50.29	1.52	
MM-2	53.34	56.39	3.05	
MM-8	44.20	45.72	1.52	
MM-14	41.45	42.67	1.22	McClelland Laboratories, 1992
AR-48C	9.14	34.26	25.12	Metcon Research, 2008
AR-49C	13.87	50.69	36.82	
AR-50C	102.11	116.19	14.08	
AR-51C	85.34	88.61	3.27	Kappes, Cassidy & Associates, 2011
AR-52C	35.05	44.81	9.76	
AR-53C	54.86	76.20	21.34	
AR-69C	80.77	90.83	10.06	Kappes, Cassidy & Associates, 2011/2012
AR-70C	38.86	61.57	22.71	
AR-71C	30.48	62.26	31.78	
AR-72C	9.14	78.43	69.29	
AR-74C	18.29	22.86	4.57	
AR-74C	25.91	28.96	3.05	
AR-74C	36.58	39.62	3.04	
AR-74C	45.72	47.24	1.52	
AR-74C	53.34	68.58	15.24	
AR-75C	42.67	44.70	2.03	
AR-76C	44.26	56.62	12.35	
AR-77C	32.00	45.87	13.87	
AR-188C	27.13	73.83	46.70	McClelland Laboratories, 2013
AR-189C	46.63	46.85	0.22	
AR-190C	51.66	86.58	34.92	
AR-191C	15.24	66.01	50.77	
AR-193C	71.32	77.19	5.87	
		Total	***452.10***	

Figure 13.30: A Long-Section Vulcan® Snapshot View (looking north) of the Footwall Stockworks of the Moss Vein and West Vein Showing the Distribution of Metallurgical Test Samples (that are colour-coded by test program)

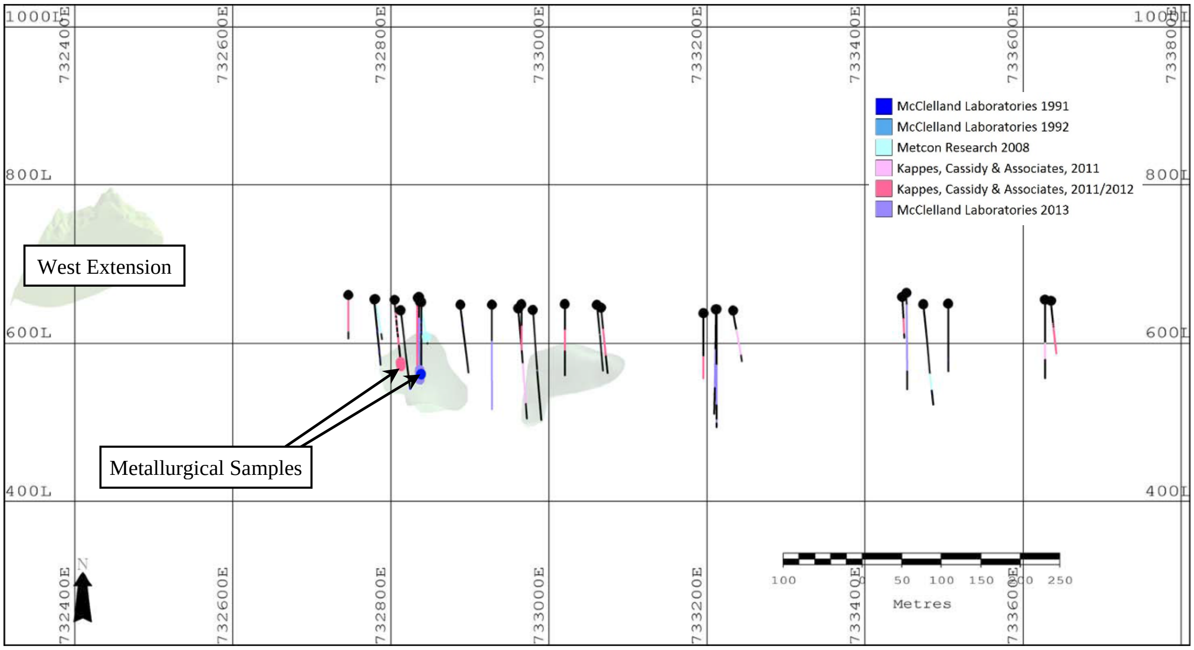


Table 13.35: A Summary of the Metallurgical Drillhole Samples that Intersect the Footwall Stockworks of the Moss Vein, Moss Mine Project
(compiled from data contained in the metallurgical test program reports cited above)

Drillhole	Sample Interval		Sample Length (m)	Test Program
	From (m)	To (m)		
MM-1	96.01	97.54	1.53	McClelland Laboratories, 1991
AR-49C	61.75	64.01	2.26	Metcon Research, 2008
AR-51C	118.87	124.97	6.10	Kappes, Cassidy & Associates, 2011
AR-70C AR-74C	65.96 86.56	68.58 92.96	2.62 6.40	Kappes, Cassidy & Associates, 2011/2012
AR-188C AR-188C	92.20 103.20	100.20 104.97	8.00 1.77	McClelland Laboratories, 2013
		Total	***28.68***	

14 MINERAL RESOURCE ESTIMATION

The Qualified Person for the 2014 Mineral Resource estimate (that is the subject of this Technical Report) is Mr. David G. Thomas, P. Geo. With the exception of Sub-Section 14.11 that has different source references (cited in Sub-Section 14.11) the following text and its supporting tables and figures are those presented in his consultancy report to the Company entitled 'Moss Mine Project, 2014 Mineral Resource Update' and dated October 24, 2014. A reconciliation of the 2014 Mineral Resource estimate to the 2013 Mineral Resource estimate (that is detailed in the 2013 Technical Report) is presented in Section 14.11.

14.1 Mineral Resource Statement

Mineral Resources for the Moss Mine Project (Table 14.1) were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, by application of a cut-off grade that incorporated mining and metallurgical recovery parameters. The estimated Mineral Resources are constrained to a pit shell based on commodity prices, metallurgical recoveries and operating costs. Long-term metal prices of US$1,250/oz Au and US$20.0/oz Ag were applied in analysis along with metallurgical recovery rates of 82% for gold and 65% for silver (per the recommended values defined in Section 13). The stated Mineral Resources have an effective date of October 31, 2014. The gold equivalent ("AuEq") grades and ounces stated on Table 14.1 were determined by applying the following formulae:

Factor A (gold) = 1 / 31.10346 x metallurgical recovery (82%) x smelter recovery (99%) x refinery recovery (99%) x unit Au price (US$1,250/oz)

Factor B (silver) = 1 / 31.10346 x metallurgical recovery (65%) x smelter recovery (98%) x refinery recovery (99%) x unit Ag price (US$20.0/oz)

AuEq = Au grade + (Ag grade x [Factor B / Factor A])

Table 14.1: Moss Mine Project Mineral Resource Estimate by David Thomas, P. Geo.
(undiluted, pit constrained, 100% in-pit recovery, effective date October 31, 2014)

Category (0.25 g/t Au Cut-Off)	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)	AuEq (g/t)	AuEq (oz)
Measured	4,860,000	0.97	10.4	152,000	1,630,000	1.10	172,000
Indicated	10,620,000	0.66	8.7	225,000	2,980,000	0.77	263,000
Measured + Indicated	*15,480,000*	*0.76*	*9.3*	*377,000*	*4,610,000*	*0.87*	*435,000*
Inferred	2,180,000	0.55	5.6	38,000	390,000	0.62	43,000

Footnotes to Mineral Resource statement:

- *The Qualified Person ("QP") reviewed the Company's QA/QC programs on the Mineral Resources data. After removing samples with data quality issues, the QP concludes that the collar, survey, assay, and lithology data are adequate to support Mineral Resources estimation.*
- *Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were modelled based on quartz veining and gold grades.*
- *Raw drillhole assays were composited to 1.52 m lengths broken at domain boundaries.*
- *Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.*
- *Block grades for gold and silver were estimated from the composites using ordinary kriging interpolation into 3 m x 3 m x 3 m blocks coded by domain.*
- *A dry bulk density of 2.51 g/cm^3 was used for material with a depth less than 12 m from surface. A dry bulk density of 2.58 g/cm^3 was used for all other material. The dry bulk densities are based on 506 specific gravity measurements.*
- *Blocks were classified as Measured, Indicated and Inferred in accordance with CIM Definition Standards 2014. Inferred resources are classified on the basis of blocks falling within the mineralised domain wireframes (i.e. reasonable assumption of grade/geological continuity) with a maximum distance of 100 m to the closest composite. Indicated resources are classified based on a drillhole spacing of 50 m. Measured resources are classified based on a 25 m x 12.5 m drillhole spacing.*
- *The Mineral Resource estimate is constrained within an optimized pit with a maximum slope angle of 65º.*
- *Metal prices of $1,250/oz and $20.0/oz were used for gold and silver, respectively.*

- *Metallurgical recoveries of 82% for gold and 65% for silver were applied.*
- *A 0.25 g/t gold cut-off was estimated based on a total process and G&A operating cost of $6.97/t of mineralized material mined.*
- *The contained gold and silver figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding.*
- *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
- *The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.*

14.2 Key Assumptions/Basis of Estimate

The Qualified Person reviewed the database used to estimate the Mineral Resources and several drillholes and underground channel samples were removed for the reasons outlined on Table 14.2. The Qualified Person confirmed that the remaining collar, downhole survey, assay and lithology data are adequate to support Mineral Resource estimation

Table 14.2: A Summary of Moss Mine Project Data Not Used in Mineral Resource Estimation

Hole ID/Series	Reason Not Used
M-26-63	Suspected assay mix-up
MC-18, LH98-6, RAL-01, RAL-02, WO Series, BX-01, 3 and 7	No Collar Coordinates
23+00G	Samples Lost
AR-58RD	No assays
Ruth Dump Series	Waste dump condemnation drilling
UG220 Series	Unverified data from 1915/1920 underground channel sample maps
WW Series	Waterwell holes. Sampling method not representative.
UG300 Series	From underground channels, no supporting data

There are a total of 986 drillholes, channel samples and trenches for a total of approximately 42,800.5 m within the Moss Mine Project database used to support Mineral Resource estimation (Table 14.3). Drillholes have intercepted mineralization at depths of up to 370 m below surface.

Table 14.3: Moss Mine Project Data Types Used to Support Mineral Resource Estimation

Data Type	Number	Total Length (m)
Air Trac	54	1,438.7
Channel Samples	270	459.1
Drillcore	117	13,912.2
Longhole	14	122.5
Percussion Holes	336	8,732.7
RC Drillholes	187	18,046.9
Trench	8	88.4
All Data	***986***	***42,800.5***

The drillhole database, provided by the Company, is in MS Excel® files. The database cut-off date for purposes of Mineral Resource estimation purposes was 17 August, 2014. The collar, downhole survey, lithology and assay data was imported into MineSight®, a commercial mining software program. The Qualified Person exported the data and checked that the imported data was the same as the original data. No significant differences were found.

Topographic contour lines were based on a surface supplied by the Company, with two metre contour intervals. The topography is based on a Lidar survey. The Qualified Person compared the drillhole collars with the topographic surface and found only minor differences (less than one metre) in elevation between the drillhole collars and the surveyed topography.

14.3 Wireframe Models and Mineralization

The Moss Vein forms a prominent ridge trending approximately 110° with an average dip of approximately 70° to the south. There is a fault along the entire length of the footwall contact of the Moss Vein, which is not a simple fissure vein. The thickest accumulations of quartz and calcite are typically in the hangingwall next to the fault. However, the width is variable along strike and it reduces to zones of breccia veins and stockwork veining in some areas. The hangingwall above massive accumulations of quartz and calcite is brecciated and stockworked by quartz and calcite veins of similar character with the amount and thickness of veining decreasing away from the footwall fault. Higher grade mineralization usually correlates with the intensity of quartz and calcite introduced along the entire length of the mineral deposit. The Moss Vein and its hangingwall stockwork zone are continuous along a strike length of approximately 1,500 m and to a vertical depth of approximately 250 m.

Brecciation and quartz stockwork veining is locally present in the footwall of the Moss Vein. Three zones with limited extents of 100 m along strike and 100 m down dip have been partially delimited by drilling.

Two veins (the Ruth Vein and Vein No. 4) trend in an east-west direction with a dip of approximately 60° to the north. These veins have a more limited strike length of between 100 m and 150 m and a vertical extent of 150 m to 200 m.

A northwest trending, steeply-dipping post-mineral fault, termed the Canyon fault, divides the deposit into eastern and western segments. A zone of breccia-hosted mineralization follows the trend of the Canyon fault. Two sets of structures have been recognized: a northwest trending set; and a northeast trending set. The structures locally offset the mineralization of interest by only a few metres horizontally.

The Company provided wireframe models of the mineralized zones, which were compiled by Douglas Brownlee, P. Geo., a co-author of this Technical Report, using Leapfrog geological modeling software. The Qualified Person for the 2014 Mineral Resource estimate reviewed the wireframe models and found the boundaries to be correctly snapped to the drillhole intercepts. The Qualified Person also inspected drillholes displaying gold and silver grades and found that no significant zones of mineralization fall outside the wireframes. Each mineralized zone was coded separately, as summarized on Table 14.4.

Table 14.4: A Summary of the Moss Mine Project Domain Codes

Domain	Code
Canyon Stockwork	1
MossVein	2
Ruth Vein	3
Footwall Stockwork	4
Footwall Stockwork 2	5
Hangingwall Stockwork	6
Vein No. 4	8
Footwall Stockwork West	9
Hangingwall Stockwork West	10
West Vein	11

Figure 14.1 (which repeats Figure 7.3) shows the wireframe models used to constrain the Mineral Resource estimation: vein material is identified in RED; hangingwall stockwork in YELLOW, and footwall stockwork in GREEN. The volume constrained by the wireframes was compared with the volume of the blocks coded to each domain. Table 14.5 summarizes the results; minor differences only were found to exist.

Figure 14.1: An Oblique Vulcan® Snapshot View (looking northwest) of the Component Parts of the Moss Deposit, from Surface to the Deepest Drillhole Intersections

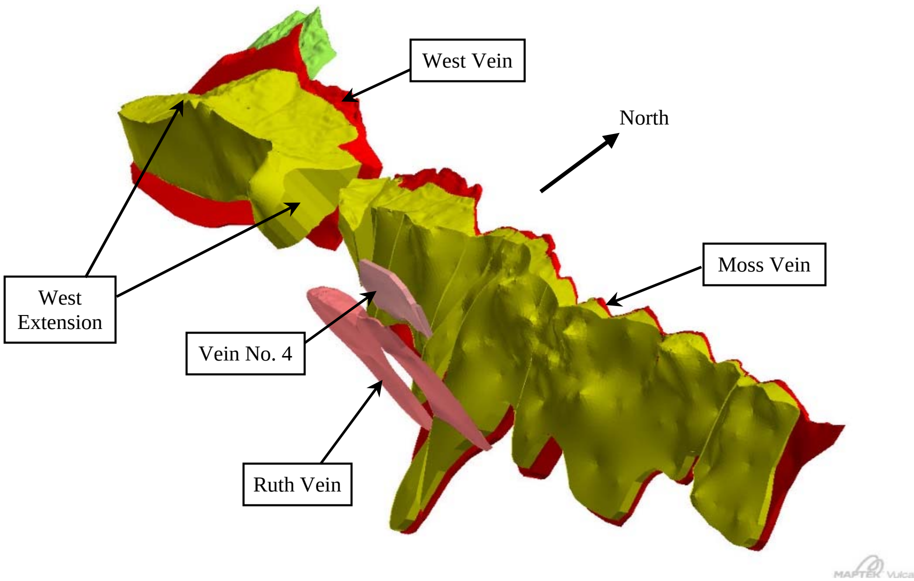


Table 14.5: A Summary of the Comparison of Block Volumes with Wireframe

Domain	Code	Number	Volume (m^3)	% Difference
Canyon Stockwork	1	9,149	247,023	-0.4%
MossVein	2	75,372	2,035,044	0.7%
Ruth Vein	3	19,666	530,982	-0.3%
Footwall Stockwork	4	4,689	126,603	-0.7%
Footwall Stockwork 2	5	2,837	76,599	-1.1%
Hangingwall Stockwork	6	95,990	2,591,730	0.6%
Vein No. 4	8	1,761	47,547	-1.1%
Footwall Stockwork West	9	14,592	393,984	-0.2%
Hangingwall Stockwork West	10	93,233	2,517,291	0.0%
West Vein	11	39,115	1,056,105	-0.2%

14.4 Exploratory Data Analysis (EDA)

Exploratory data analysis comprised basic statistical evaluation of the assays and composites for gold, silver and sample length.

14.4.1 Assays

14.4.1.1 Histograms and Probability Plots

Log-scaled histograms and probability plots for gold and silver within each domain show limited evidence for mixed populations. The log-scaled histogram for the hangingwall stockwork zone shows the presence of an included low-grade population, comprising 10% of the samples. The Qualified Person concluded that this amount of included, low-grade material does not warrant further domaining. Figures 14.2 and 14.3 are histograms and probability plots for the Moss Vein and Hangingwall Stockwork zone.

14.4.1.2 Grade Capping/Outlier Restrictions

The length weighted, normal-scaled and log-scaled histograms and probability plots of the assays were evaluated to define grade outliers for gold and silver within each of the domains separately. Tables 14.6 and 14.7 summarize the capping grade thresholds and the amount of metal removed within the domains. Capping was completed on the assays prior to compositing.

Figure 4.2: Moss Vein Histograms and Probability Plots, Assays

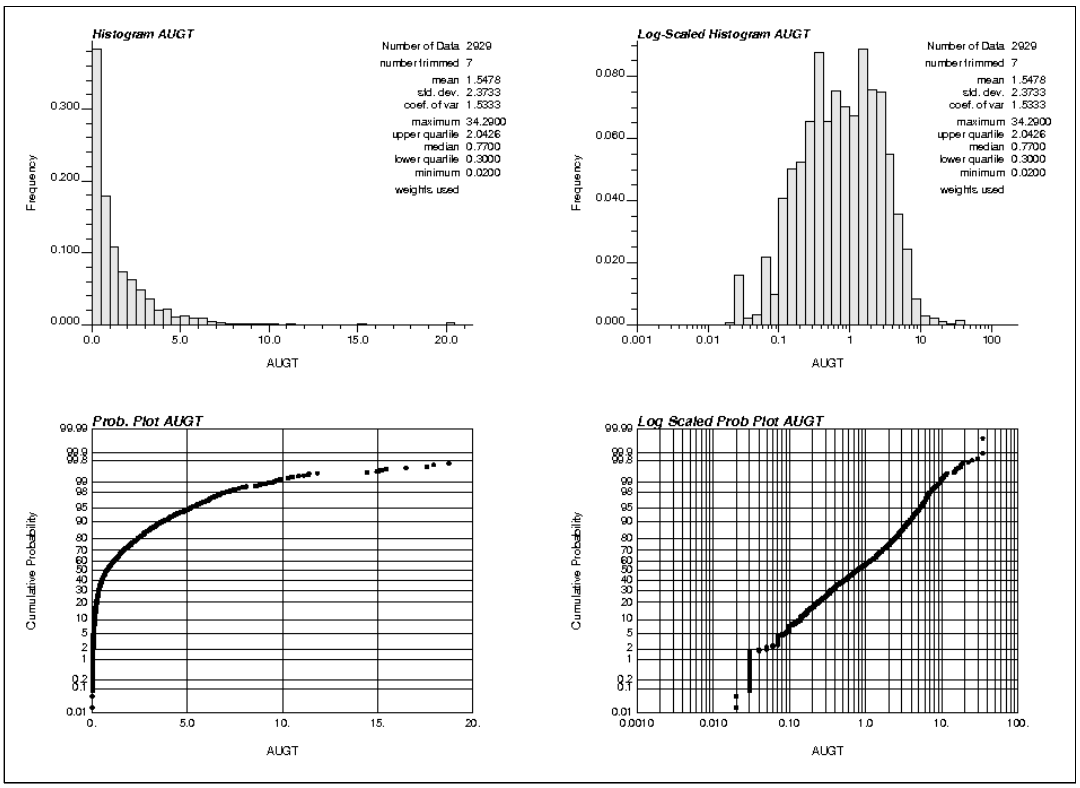


Figure 14.3: Hangingwall Stockwork Histograms and Probability Plots, Assays

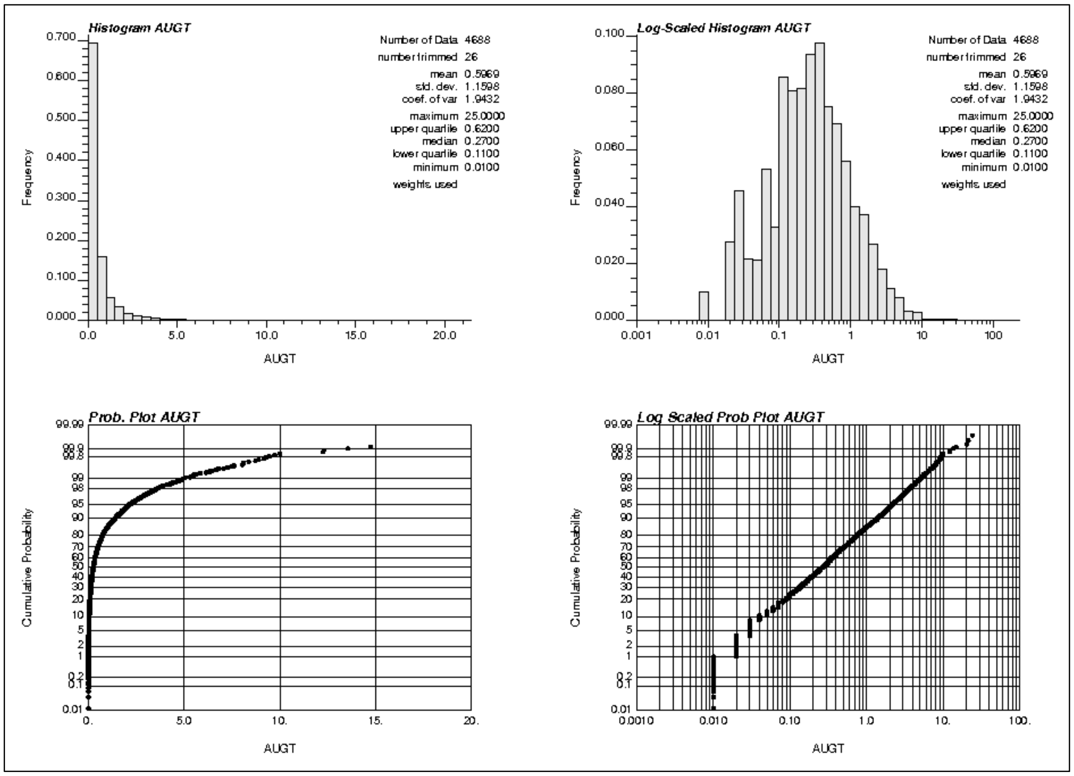


Table 14.6: A Summary of Length-Weighted Assay Statistics for Gold Within Each Domain

Domain	Code	Number	Min. (g/t)	Max. (g/t)	Mean (g/t)	CV	Capping Threshold (g/t)	Capped Mean (g/t)	Capped CV	% Metal
Canyon Stockwork	1	580	0.03	4.53	0.35	1.36	3.0	0.35	1.28	-1.5%
MossVein	2	2,929	0.02	34.29	1.55	1.53	20.0	1.52	1.36	-1.7%
Ruth Vein	3	309	0.01	8.55	0.39	1.77	3.0	0.37	1.42	-5.3%
Footwall Stockwork	4	119	0.03	5.83	1.01	1.06	4.0	0.97	0.95	-3.7%
Footwall Stockwork 2	5	69	0.01	2.78	0.50	1.18	2.0	0.48	1.10	-3.7%
Hangingwall Stockwork	6	4,688	0.01	25.00	0.60	1.94	2.0	0.59	1.70	-2.0%
Vein No. 4	8	182	0.01	5.22	0.32	2.01	2.0	0.28	1.51	-12.4%
Footwall Stockwork West	9	332	0.01	2.85	0.28	0.97	1.5	0.27	0.76	-3.1%
Hangingwall Stockwork West	10	1,165	0.01	8.23	0.43	1.38	5.2	0.43	1.31	-0.8%
West Vein	11	781	0.03	13.71	0.58	1.49	5.2	0.56	1.16	-3.8%

Table 14.7: A Summary of Length-Weighted Assay Statistics for Silver Within Each Domain

Domain	Code	Number	Min. (g/t)	Max. (g/t)	Mean (g/t)	CV	Capping Threshold (g/t)	Capped Mean (g/t)	Capped CV	% Metal
Canyon Stockwork	1	580	0.3	52.2	3.9	1.2	25.0	3.79	1.03	-2.9%
MossVein	2	2,736	0.0	202.3	16.1	1.3	140.0	16.00	1.30	-0.7%
Ruth Vein	3	309	0.0	56.0	3.0	1.9	20.0	2.70	1.45	-8.9%
Footwall Stockwork	4	110	0.7	71.0	10.2	1.1	40.0	9.90	0.97	-3.4%
Footwall Stockwork 2	5	69	0.5	66.9	8.0	1.4	40.0	7.58	1.20	-4.8%
Hangingwall Stockwork	6	4,588	0.0	257.0	6.1	1.6	115.0	6.11	1.53	-0.5%
Vein No. 4	8	182	0.0	99.4	4.4	2.7	40.0	3.76	2.10	-14.1%
Footwall Stockwork West	9	333	0.0	154.8	4.1	2.7	22.0	3.17	1.17	-22.4%
Hangingwall Stockwork West	10	941	0.0	71.0	4.9	1.4	60.0	4.90	1.35	-0.3%
West Vein	11	427	0.3	70.9	5.8	1.3	45.0	5.68	1.16	-1.5%

The coefficient of variation ("CV") values of the capped assays within each domain are generally below 1.5. The Qualified Person concluded that the domains comprise single mineralized populations and that no further domaining was necessary. The amounts of metal removed from each domain are consistent with the amount of drilling: less metal was removed from well-drilled domains and more metal is removed from less well-drilled domains.

14.4.2 Composites

To normalize the weight of influence of each sample, the assay intervals were regularized by compositing the drillhole data into 1.5 m lengths using the mineralization zone domain boundaries to break the composites. The 1.5 m composites were back-tagged using the mineralization zone solids and the assay intervals were regularized into 3.0 m lengths using the same methodology. The 3.0 m composites were then used to estimate nearest-neighbour models for model validation purposes. Table 14.8 and 14.9 summarize the 1.5 m composite statistics. Table 14.10 summarizes a comparison between length-weighted 3.0 m composites and assays.

Table 14.8: A Summary of Length-Weighted 1.5 m Composite Statistics, Gold

Domain	Code	Number	Min. (g/t)	Max. (g/t)	Mean (g/t)	CV	Capped Mean (g/t)	Capped CV	% Metal Removed
Canyon Stockwork	1	642	0.03	4.24	0.35	1.29	0.35	1.20	-1.5%
MossVein	2	3,083	0.02	34.29	1.55	1.46	1.52	1.29	-1.7%
Ruth Vein	3	344	0.00	6.90	0.39	1.61	0.37	1.35	-5.2%
Footwall Stockwork	4	119	0.03	5.56	1.00	1.00	0.97	0.90	-3.7%
Footwall Stockwork 2	5	71	0.01	2.78	0.50	1.14	0.48	1.07	-3.7%
Hangingwall Stockwork	6	5,198	0.00	25.00	0.60	1.75	0.58	1.57	-1.9%
Vein No. 4	8	190	0.01	5.13	0.32	1.92	0.28	1.40	-12.2%
Footwall Stockwork West	9	328	0.00	2.53	0.28	0.91	0.27	0.71	-3.1%
Hangingwall Stockwork West	10	1,153	0.00	7.80	0.43	1.23	0.43	1.17	-0.8%
West Vein	11	792	0.03	11.11	0.58	1.41	0.56	1.12	-3.8%

Table 14.9: A Summary of Length-Weighted 1.5 m Composite Statistics, Silver

Domain	Code	Number	Min. (g/t)	Max. (g/t)	Mean (g/t)	CV	Capped Mean (g/t)	Capped CV	% Metal Removed
Canyon Stockwork	1	642	0.3	50.0	3.9	1.2	3.79	0.98	-2.9%
MossVein	2	2,879	0.0	188.7	16.1	1.3	15.96	1.24	-0.7%
Ruth Vein	3	343	0.0	51.8	3.0	1.8	2.69	1.35	-8.9%
Footwall Stockwork	4	109	0.7	61.8	10.2	1.0	9.90	0.92	-3.4%
Footwall Stockwork 2	5	71	0.5	66.9	8.0	1.3	7.59	1.17	-5.0%
Hangingwall Stockwork	6	5,101	0.0	144.1	6.1	1.5	6.10	1.43	-0.5%
Vein No. 4	8	190	0.0	79.4	4.4	2.5	3.78	2.00	-14.3%
Footwall Stockwork West	9	328	0.1	124.0	4.1	2.5	3.17	1.11	-22.6%
Hangingwall Stockwork West	10	928	0.0	65.3	4.9	1.3	4.86	1.24	-0.3%
West Vein	11	423	0.3	68.5	5.8	1.2	5.68	1.10	-1.5%

Table 14.10: A Summary of Length-Weighted 3.0 m Composite Comparison with Assays, Gold

Domain	Code	Length-Weighted Assays		3.0 m Length Weighted Composites		% Difference	
		Uncapped Mean (g/t)	Length Sum (m)	Uncapped Mean (g/t)	Length Sum (m)	Uncapped Mean (g/t)	Length
Canyon Stockwork	1	0.35	942.9	0.35	959.8	-0.2%	1.8%
MossVein	2	1.55	4,601.1	1.55	4,668.2	-0.2%	1.5%
Ruth Vein	3	0.39	504.5	0.39	514.4	-0.3%	2.0%
Footwall Stockwork	4	1.01	177.3	1.00	180.2	-0.1%	1.6%
Footwall Stockwork 2	5	0.50	106.5	0.50	108.2	-0.3%	1.6%
Hangingwall Stockwork	6	0.60	7,669.3	0.60	7,799.1	-0.3%	1.7%
Vein No. 4	8	0.32	279.2	0.32	283.6	0.2%	1.6%
Footwall Stockwork West	9	0.28	480.5	0.28	489.2	-0.5%	1.8%
Hangingwall Stockwork West	10	0.43	1,704.6	0.43	1,732.3	-0.5%	1.6%
West Vein	11	0.58	1,162.4	0.58	1,180.2	-0.2%	1.5%

It should be noted that the length-weighted mean grades of both 1.5 m and 3.0 m length composites are very similar to those of the assays. The Qualified Person is, therefore, confident that the compositing process worked as intended:

- within the domains, the capped CV values of the composites are low to moderate (less than 1.5); and
- the CV values indicate that no further domaining is warranted.

Histograms and probability plots for the Moss Vein and Hangingwall Stockwork are presented as Figures 14.4 and 14.5.

14.4.2.1 Scatter Plots and Regression Analysis

The Qualified Person examined an assay scatterplot between gold and silver for all of the mineralization domains (Figure 14.6). The scatter plot shows a moderate correlation (correlation coefficient of 0.65) between gold and silver and the presence multiple gold-silver correlations. These possibly represent multiple mineralizing events. The scatterplot also shows the presence of zeros within the silver assays.

Figure 14.4: Moss Vein Histograms and Probability Plots, Composites

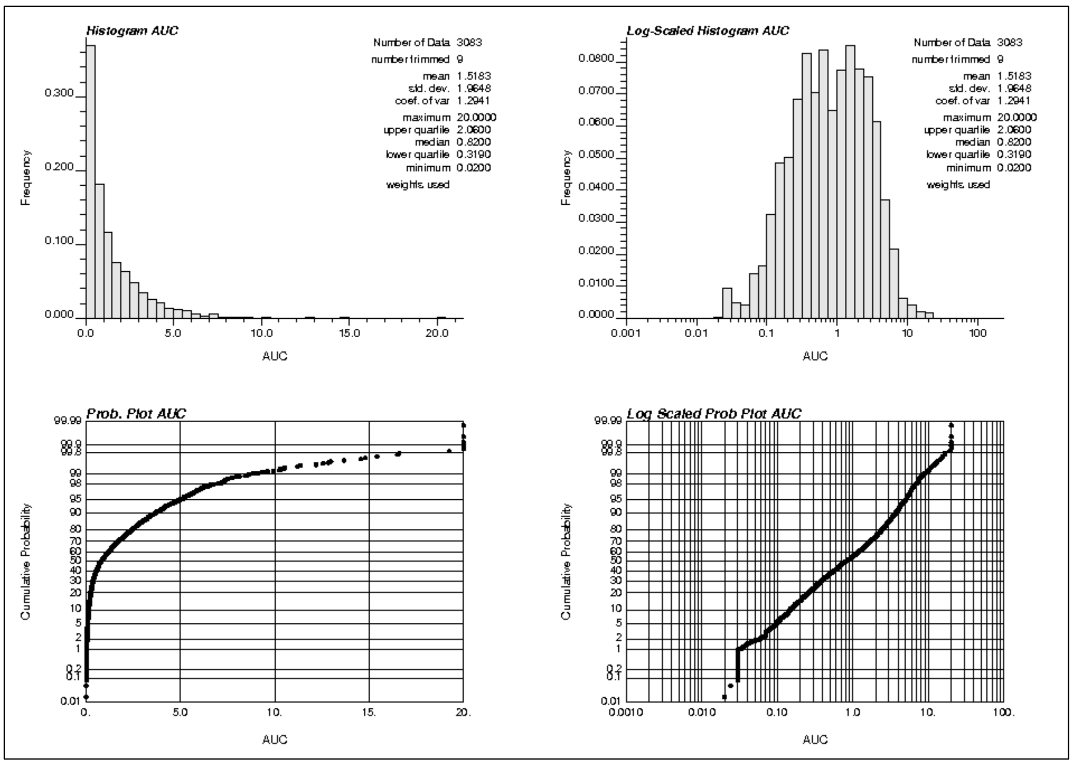


Figure 14.5: Hangingwall Stockwork Histograms and Probability Plots, Composites

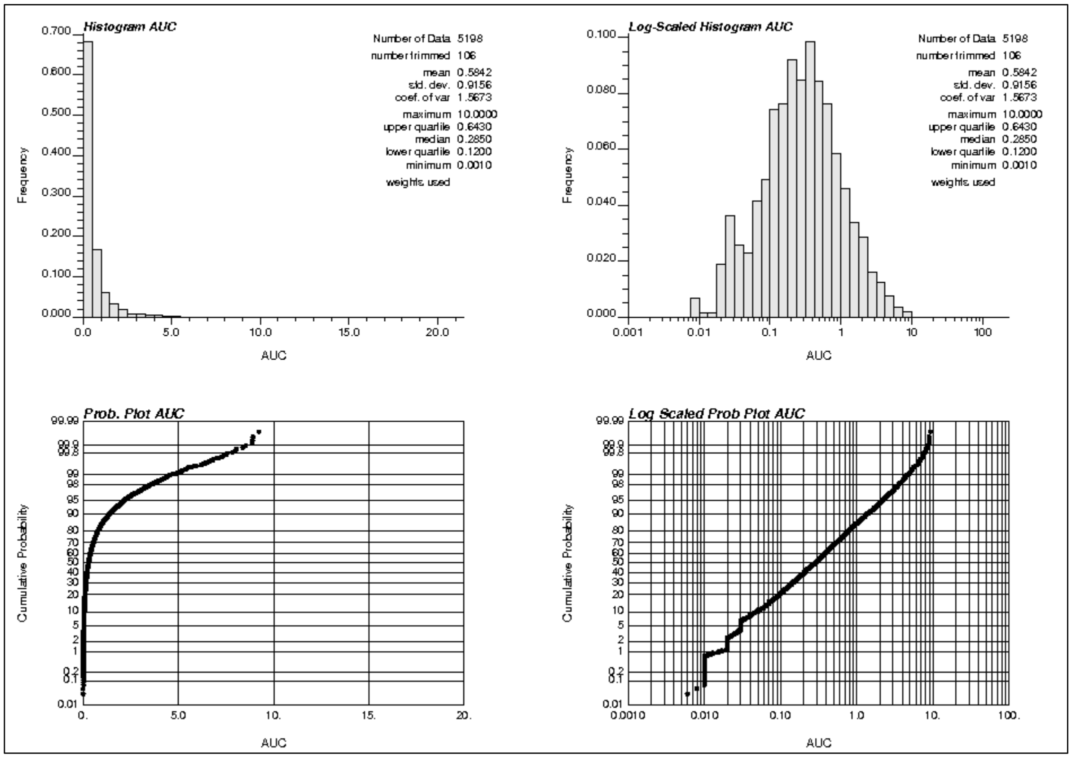
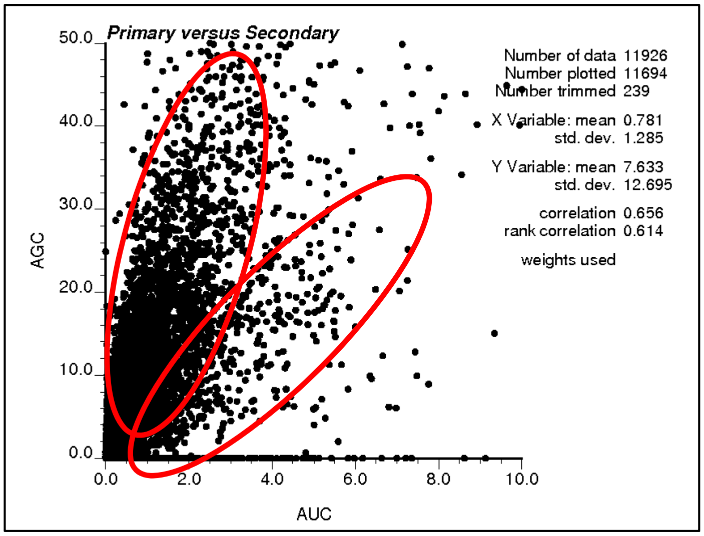


Figure 14.6: A Scatter Plot of Silver versus Gold, All Domains



Note: Populations with different gold to silver ratios are outlined in red

There are a total of 904 samples from 57 drillholes (BF Minerals Air Trac holes) and eight trenches with missing silver assays. Linear regression analyses of silver versus gold was completed for groups of domains and silver grades were assigned to missing silver assays. The regression formulas for each group of domains for both 1.5 m and 3.0 m composites are presented on Tables 14.11 and 14.12.

Table 14.11: 1.5 m Composite Linear Regression Formulas, Silver versus Gold

Domain	Coefficient (m)	Constant (c)	Correlation Coefficient (r^2)
Vein Domains	7.756	4.743	0.496
Hangingwall Stockworks	7.169	1.959	0.561
Footwall Stockworks	8.708	2.524	0.533

Note: Regressions are shown using the formula y = mx + c

Table 14.12: 3.0 m Composite Linear Regression Formulas, Silver versus Gold

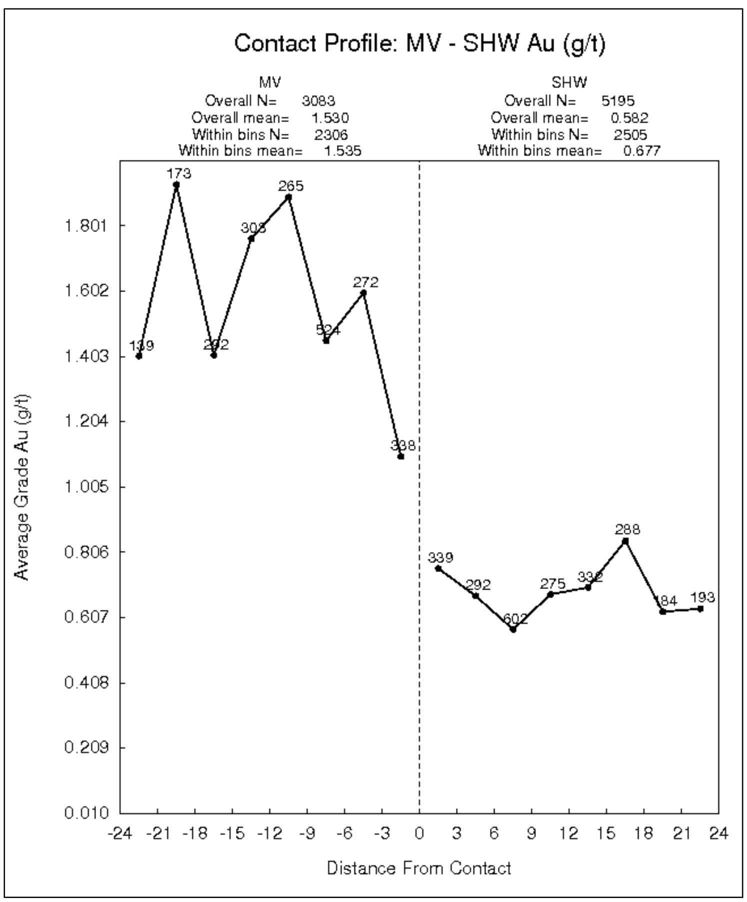
Domain	Coefficient (m)	Constant (c)	Correlation Coefficient (r^2)
Vein Domains	7.356	4.993	0.496
Hangingwall Stockworks	7.237	1.925	0.589
Footwall Stockworks	8.942	2.206	0.476

Note: Regressions are shown using the formula y = mx + c

14.4.2.2 Contact Profiles

Contact plots displaying average grades of gold in distance classes on either side of the contact between the Moss Vein and Hangingwall Stockwork were compiled (Figure 14.7). The contact profiles show that there is a sharp change in grade across the contact, in consequence of which the contact was used as a hard boundary during grade estimation.

Figure 14.7: Contact Profile, Moss Vein and Hangingwall Stockwork

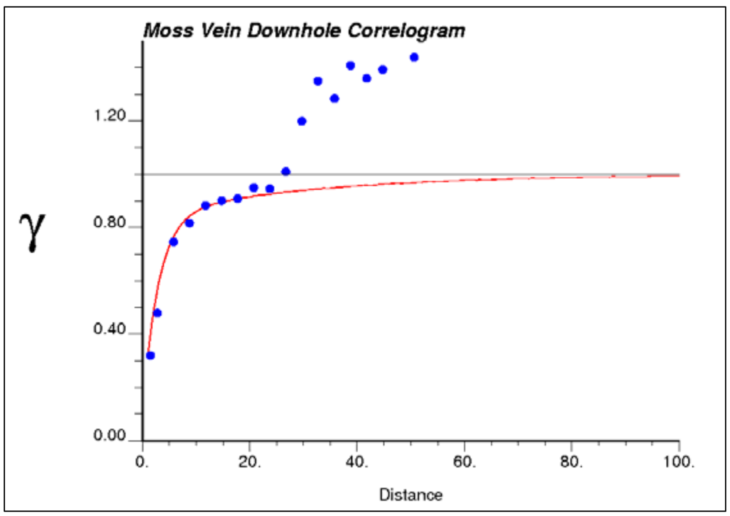


14.4.2.3 Variography

Experimental down-the-hole variograms were calculated from the 1.5 m capped composites and fitted models of the down-the-hole correlograms using GSLIB software. Down-the-hole correlogram models were used to select the nugget effect used in subsequent modelling of directional correlograms. Directional experimental correlograms were calculated using the same set of capped 1.5 m composites and fitted models of the directional correlograms using SAGE® software. The directions of anisotropy were selected to coincide with the trend directions of the mineralization. No obvious plunge direction was observed within the plane of the mineralization.

Down-hole and directional correlograms were constructed for gold and silver using three groups of domains (vein, hangingwall stockwork and footwall stockwork). The variograms show very low nugget effects of 10% of the total variance. The ranges of correlation generally vary between 20 m and 30 m. The downhole variogram for the Moss Vein and West Vein is presented as Figure 14.8

Figure 14.8: A Downhole Variogram, Moss Vein/West Vein



A nugget effect, single spherical model and either a nested spherical or exponential model were used to fit the experimental correlograms. Tables 14.13 and 14.14 summarize the correlogram models. Figure 14.9 is an example of the fitted model in the three main directions. The correlogram models were re-oriented to follow changes in the dip and strike of the mineralized domains.

Table 14.13: A Summary of the Variogram Models and Rotation Angles, Gold

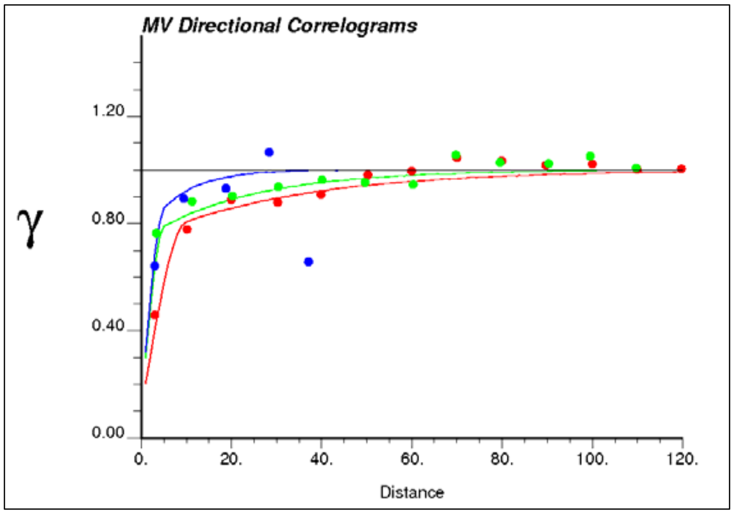
Domain	Code	Nugget Effect	Sills 1st Structure	Sills 2nd Structure	1st Structure Type	2nd Structure Type	Range 1st Structure X	Range 1st Structure Y	Range 1st Structure Z	Range 2nd Structure X	Range 2nd Structure Y	Range 2nd Structure Z	Rotation Angles (GSLIB LRR Convention) Z-Axis	Rotation Angles (GSLIB LRR Convention) X-Axis	Rotation Angles (GSLIB LRR Convention) Y-Axis
Canyon Stockwork	1	0.100	0.709	0.191	Spherical	Exponential	5	8	5	65	90	45	50	-85	0
MossVein	2	0.100	0.642	0.258	Spherical	Exponential	5	10	5	70	100	25	10	70	0
Ruth Vein	3	0.100	0.713	0.187	Spherical	Spherical	9	15	11	65	30	15	0	-55	0
Footwall Stockwork	4	0.100	0.382	0.518	Spherical	Exponential	5	5	5	60	21	19	10	70	0
Footwall Stockwork 2	5	0.100	0.382	0.518	Spherical	Exponential	5	5	5	60	21	19	10	70	0
Hangingwall Stockwork	6	0.100	0.709	0.191	Spherical	Exponential	5	8	5	65	90	45	10	70	0
Vein No. 4	8	0.100	0.713	0.187	Spherical	Spherical	9	15	11	65	30	15	0	-65	0
Footwall Stockwork West	9	0.100	0.382	0.518	Spherical	Exponential	5	5	5	60	21	19	10	70	0
Hangingwall Stockwork West	10	0.100	0.709	0.191	Spherical	Exponential	5	8	5	65	90	45	10	70	0
West Vein	11	0.100	0.642	0.258	Spherical	Exponential	5	10	5	70	100	25	10	60	0
Canyon Stockwork	11	0.100	0.642	0.258	Spherical	Exponential	5	10	5	70	100	25	10	50	0

Table 14.14: A Summary of the Variogram Models and Rotation Angles, Silver

Domain	Code	Nugget Effect	Sills 1st Structure	Sills 2nd Structure	1st Structure Type	2nd Structure Type	Range 1st Structure X	Range 1st Structure Y	Range 1st Structure Z	Range 2nd Structure X	Range 2nd Structure Y	Range 2nd Structure Z	Rotation Angles (GSLIB LRR Convention) Z-Axis	Rotation Angles (GSLIB LRR Convention) X-Axis	Rotation Angles (GSLIB LRR Convention) Y-Axis
Canyon Stockwork	1	0.100	0.658	0.242	Spherical	Spherical	5	8	5	60	75	35	50	-85	0
MossVein	2	0.100	0.680	0.220	Spherical	Spherical	5	5	5	86	40	23	10	70	0
Ruth Vein	3	0.100	0.781	0.119	Spherical	Spherical	9	7	10	25	50	15	0	-55	0
Footwall Stockwork	4	0.100	0.340	0.560	Spherical	Spherical	5	5	5	20	30	40	10	70	0
Footwall Stockwork 2	5	0.100	0.340	0.560	Spherical	Spherical	5	5	5	20	30	40	10	70	0
Hangingwall Stockwork	6	0.100	0.658	0.242	Spherical	Spherical	5	8	5	60	75	35	10	70	0
Vein No. 4	8	0.100	0.781	0.119	Spherical	Spherical	9	7	10	25	50	15	0	-65	0
Footwall Stockwork West	9	0.100	0.340	0.560	Spherical	Spherical	5	5	5	20	30	40	10	70	0
Hangingwall Stockwork West	10	0.100	0.658	0.242	Spherical	Spherical	5	8	5	60	75	35	10	70	0
West Vein	11	0.100	0.680	0.220	Spherical	Spherical	5	5	5	86	40	23	10	60	0
Canyon Stockwork	11	0.100	0.680	0.220	Spherical	Spherical	5	5	5	86	40	23	10	50	0

Figure 14.9: A Directional Variogram, Moss Vein/West Vein

(major (Y) axis direction is shown in **RED**, minor axis (X) is shown in **GREEN**, vertical (Z) axis is shown in **BLUE**)



14.4.3 Estimation/Interpolation Methods

The block model consists of regular blocks (3 m along strike x 3 m across strike x 3 m vertically). The block size was chosen such that geological contacts are reasonably well reflected and to support selective openpit mining. The ordinary kriging grade interpolation method was used in three passes with increasing search distances.

The composite selection parameters for grade estimation in each domain (minimum, maximum, maximum number of composites per hole and quadrant search) were adjusted so as to minimize bias (as measured against a nearest-neighbour model) and to produce grade estimates with a variance approximating those predicted from the variograms models and using a selective mining unit ("SMU") of 6 m x 6 m x 6 m. A quadrant search restriction was implemented to increase the de-clustering of the data during estimation. The data used for estimation includes clustered underground channel samples. Tables 14.14 through 14.17 summarize the search distances and search ellipse orientations for the estimation domains.

Grade estimation used a composite and block matching scheme based on the domain codes. For example, composites coded to the Moss Vein were only used to estimate blocks falling within the Moss Vein. The orientation of the search ellipse was adjusted where the wireframe showed a significant change in orientation. The same grade estimation plan was used for both gold and silver.

14.5 Density Assignment

A total of 506 SG determinations have been performed on drillcore samples collected from material within the mineralized zones. The determinations were performed by ALS Chemex laboratory using unsealed immersion technique to measure the weight of each sample in air and in water (ALS Chemex standard OA-GRA08). The measurements were plotted against downhole depth; the plot shows a decrease in SG within 12 m of surface. An SG of 2.51 was, therefore, assigned to blocks within 12 m of surface and an SG of 2.58 was assigned to blocks more than 12 m in depth. The SG values have been used directly as the dry bulk density to report the tonnage estimates of the Mineral Resource.

Table 14.15: A Summary of the Grade Model Interpolation Plan, Pass 1

Domain	Code	Search Ellipse Dimensions - Pass 1			Composite Restrictions				Rotation Angles (GSLIB LRR Convention)			Number of Holes	
		X-Axis	Y-Axis	Z-Axis	Min.	Max.	Maximum Per Hole	Max. Per Quadrant	Z-Axis	X-Axis	Y-Axis	Minimum	Maximum
Canyon Stockwork	1	25	25	20	4	12	3	6	50	-85	0	2	4
MossVein	2	25	25	20	4	18	3	6	10	70	0	2	4
Ruth Vein	3	25	25	20	3	12	2	2	0	-55	0	2	4
Footwall Stockwork	4	25	25	20	4	18	3	None	10	70	0	2	6
Footwall Stockwork 2	5	25	25	20	3	12	2	2	10	70	0	2	4
Hangingwall Stockwork	6	25	25	20	4	18	3	6	10	70	0	2	4
Vein No. 4	8	25	25	20	3	12	2	2	0	-65	0	2	4
Footwall Stockwork West	9	25	25	20	4	18	3	6	10	55	0	2	4
Hangingwall Stockwork West	10	25	25	20	4	18	3	6	10	70	0	2	4
West Vein East	11	25	25	20	4	18	3	6	10	60	0	2	4
West Vein West	11	25	25	20	4	18	3	6	10	50	0	2	4

Note: Search ellipse orientations are given using the LRR rotation convention as used in GSLIB

Table 14.16: A Summary of the Grade Model Interpolation Plan, Pass 2

Domain	Code	Search Ellipse Dimensions - Pass 1			Composite Restrictions				Rotation Angles (GSLIB LRR Convention)			Number of Holes	
		X-Axis	Y-Axis	Z-Axis	Min.	Max.	Maximum Per Hole	Max. Per Quadrant	Z-Axis	X-Axis	Y-Axis	Minimum	Maximum
Canyon Stockwork	1	50	50	20	3	12	2	2	50	-85	0	2	4
MossVein	2	50	50	20	4	18	3	6	10	70	0	2	4
Ruth Vein	3	50	50	20	3	12	2	2	0	-55	0	2	4
Footwall Stockwork	4	50	50	20	4	18	3	None	10	70	0	2	6
Footwall Stockwork 2	5	50	50	20	3	12	2	2	10	70	0	2	4
Hangingwall Stockwork	6	50	50	20	4	18	3	6	10	70	0	2	4
Vein No. 4	8	50	50	20	3	12	2	2	0	-65	0	2	4
Footwall Stockwork West	9	50	50	20	4	18	3	6	10	55	0	2	4
Hangingwall Stockwork West	10	50	50	20	4	18	3	6	10	70	0	2	4
West Vein East	11	50	50	20	4	18	3	6	10	60	0	2	4
West Vein West	11	50	50	20	4	18	3	6	10	50	0	2	4

Note: Search ellipse orientations are given using the LRR rotation convention as used in GSLIB

Table 14.17: A Summary of the Grade Model Interpolation Plan, Pass 3

Domain	Code	Search Ellipse Dimensions - Pass 1			Composite Restrictions				Rotation Angles (GSLIB LRR Convention)			Number of Holes	
		X-Axis	Y-Axis	Z-Axis	Min.	Max.	Maximum Per Hole	Max. Per Quadrant	Z-Axis	X-Axis	Y-Axis	Minimum	Maximum
Canyon Stockwork	1	100	100	40	2	12	2	4	50	-85	0	1	6
MossVein	2	100	100	40	2	18	2	6	10	70	0	1	9
Ruth Vein	3	100	100	40	2	12	2	2	0	-55	0	1	4
Footwall Stockwork	4	100	100	40	2	18	2	None	10	70	0	1	9
Footwall Stockwork 2	5	100	100	40	2	12	2	None	10	70	0	1	6
Hangingwall Stockwork	6	100	100	40	2	18	2	6	10	70	0	1	9
Vein No. 4	8	100	100	40	2	12	2	2	0	-65	0	1	4
Footwall Stockwork West	9	100	100	40	2	18	2	6	10	55	0	1	9
Hangingwall Stockwork West	10	100	100	40	2	18	2	6	10	70	0	1	9
West Vein East	11	100	100	40	2	18	2	6	10	60	0	1	9
West Vein West	11	100	100	40	2	18	2	6	10	50	0	1	9

Note: Search ellipse orientations are given using the LRR rotation convention as used in GSL

14.6 Block Model Validation

The Moss Mine Project block model was validated to ensure appropriate honouring of the input data. Nearest-neighbour grade models were created from 3.0 m composites to validate the ordinary kriging grade models.

14.6.1 Visual Inspection

A visual inspection of block grade versus composited data was carried out in section and plan view. Block grade versus composited data showed a good reproduction of the data by the model.

14.6.2 Metal Removed By Capping

The impact of capping was evaluated by estimating uncapped and capped grade models. Generally the amounts of metal removed by capping in the models are consistent with the amounts calculated during the grade capping study on the composite.

14.6.3 Global Bias Checks

A comparison between the ordinary kriging and nearest-neighbour estimates was completed on all classified blocks to check for global bias in the grade estimates. Differences were generally within acceptable levels (less than 10%). The domains with larger differences between the nearest-neighbour model and ordinary kriging model either have a low number of composites or are those with drilling oblique to the trend of the mineralization. The Qualified Person concluded that the nearest-neighbour model does not provide a robust reference for validation. The summary statistics are presented on Tables 14.18 and 14.19.

A comparison between the ordinary kriging and nearest-neighbour estimates was completed on Measured and Indicated blocks to check for global bias in the grade estimates. Differences were within acceptable levels (less than 5%). The summary statistics are presented on Table 14.20 and 14.21.

Table 14.18: A Summary of the 3.0 m Composite, Nearest-Neighbour (NN) and Ordinary Kriging Model Statistics Comparison, Gold

Domain	Code	3.0 m Capped Composites		NN Blocks Capped		Kriged Blocks Capped		% Differences	
		Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean (Composites - NN)	Mean (NN - Kriged)
Canyon Stockwork	1	0.35	322	0.34	9,149	0.33	9,149	-1.5%	-3.0%
MossVein	2	1.52	1,677	1.11	75,372	1.15	75,372	-26.7%	2.9%
Ruth Vein	3	0.37	172	0.43	19,666	0.44	19,666	16.0%	3.7%
Footwall Stockwork	4	0.97	67	0.94	4,689	0.97	4,689	-2.9%	2.9%
Footwall Stockwork 2	5	0.48	38	0.46	2,837	0.44	2,837	-4.8%	-4.7%
Hangingwall Stockwork	6	0.58	2,706	0.50	95,990	0.51	95,990	-14.5%	2.3%
Vein No. 4	8	0.28	95	0.35	1,761	0.34	1,761	23.9%	-0.7%
Footwall Stockwork West	9	0.27	164	0.32	14,592	0.29	14,592	18.8%	-9.6%
Hangingwall Stockwork West	10	0.43	632	0.39	93,233	0.41	93,233	-8.0%	4.2%
West Vein	11	0.56	425	0.51	39,115	0.52	39,115	-8.8%	1.6%
All Domains		*0.78*	*6,298*	*0.59*	*356,404*	*0.61*	*356,404*	*-24.3%*	*2.5%*

Table 14.19: A Summary of the 3.0 m Composite, Nearest-Neighbour (NN) and Ordinary Kriging Model Statistics Comparison, Silver

Domain	Code	3.0 m Capped Composites		NN Blocks Capped		Kriged Blocks Capped		% Differences	
		Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean (Composites - NN)	Mean (NN - Kriged)
Canyon Stockwork	1	3.78	322	3.97	9,149	3.72	19,666	5.1%	-6.5%
MossVein	2	15.95	1,677	13.58	75,372	13.97	4,689	-14.8%	2.9%
Ruth Vein	3	2.68	172	2.94	19,605	3.10	2,837	9.6%	5.4%
Footwall Stockwork	4	9.90	67	9.39	4,689	9.63	95,990	-5.1%	2.5%
Footwall Stockwork 2	5	7.59	38	7.63	2,837	6.78	1,761	0.6%	-11.2%
Hangingwall Stockwork	6	6.10	2,706	6.21	95,990	6.30	14,592	1.8%	1.5%
Vein No. 4	8	3.78	95	5.31	1,761	5.20	93,233	40.3%	-2.0%
Footwall Stockwork West	9	3.17	164	3.29	14,592	2.79	39,115	3.8%	-15.2%
Hangingwall Stockwork West	10	4.86	632	4.78	93,233	5.35	356,404	-1.7%	11.8%
West Vein	11	5.68	425	6.48	38,973	6.60	9,149	14.1%	1.9%
All Domains		*8.25*	*6,298*	*7.12*	*356,201*	*7.36*	*637,436*	*-13.8%*	*3.4%*

Table 14.20: A Summary of the Measured and Indicated Nearest-Neighbour (NN) and Ordinary Kriging Model Statistics Comparison, Gold

Domain	Code	Kriged Blocks Capped		NN Blocks Capped		% Differences
		Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean (NN - Kriged)
Canyon Stockwork	1	0.33	9,086	0.34	9,086	3.6%
MossVein	2	1.15	75,372	1.11	75,372	-2.9%
Ruth Vein	3	0.33	4,074	0.32	4,074	-3.4%
Footwall Stockwork	4	0.95	4,538	0.94	4,538	-0.6%
Footwall Stockwork 2	5	0.43	2,837	0.46	2,837	5.6%
Hangingwall Stockwork	6	0.53	88,904	0.52	88,904	-2.3%
Vein No. 4	8	0.34	1,761	0.35	1,761	0.7%
Footwall Stockwork West	9	0.31	1,510	0.34	1,510	8.4%
Hangingwall Stockwork West	10	0.40	53,976	0.39	53,976	-2.1%
West Vein	11	0.45	21,929	0.46	21,929	1.9%
All Domains	*N/A*	*0.66*	*264,222*	*0.65*	*264,222*	*-1.9%*

Table 14.21: A Summary of the Measured and Indicated Nearest-Neighbour (NN) and Ordinary Kriging Model Statistics Comparison, Silver

Domain	Code	Kriged Blocks Capped		NN Blocks Capped		% Differences
		Mean Au (g/t)	Number	Mean Au (g/t)	Number	Mean (NN - Kriged)
Canyon Stockwork	1	3.72	9,149	3.97	9,149	7.0%
MossVein	2	14.24	73,451	13.84	73,451	-2.8%
Ruth Vein	3	3.09	17,501	2.92	17,501	-5.4%
Footwall Stockwork	4	9.63	4,689	9.39	4,689	-2.4%
Footwall Stockwork 2	5	6.78	2,837	7.63	2,837	12.6%
Hangingwall Stockwork	6	6.37	94,649	6.28	94,649	-1.4%
Vein No. 4	8	5.20	1,761	5.31	1,761	2.1%
Footwall Stockwork West	9	2.79	14,512	3.27	14,512	17.5%
Hangingwall Stockwork West	10	5.25	76,840	4.67	76,840	-11.2%
West Vein	11	6.41	35,346	6.44	35,346	0.6%
All Domains	*N/A*	*7.50*	*330,735*	*7.28*	*330,735*	*-3.0%*

14.6.4 Local Bias Checks

A check for local bias was carried out by plotting the average gold and silver grades of composites, nearest-neighbour and ordinary kriging models in swaths oriented along the model northings, eastings and elevations. The swath plots were reviewed; only minor discrepancies between the neareast neighbour and ordinary kriging model grades were found. In areas where there is significant extrapolation beyond the drillholes, the swath plots indicate less agreement for all variables. Figures 14.10 and 14.11 are the gold swath plots for the Moss Vein and Hangingwall Stockwork, respectively.

Figure 14.10: Gold Swath Plots by Easting, Northing and Elevation for the Moss Vein

(the upper Swath plots show the grades, lower swath plots show number of blocks or composites)
(the **GREEN** lines represent the ordinary kriging mdel, the **MAGENTA** lines represent the nearest-neighbour model and the BLACK line represents composites)

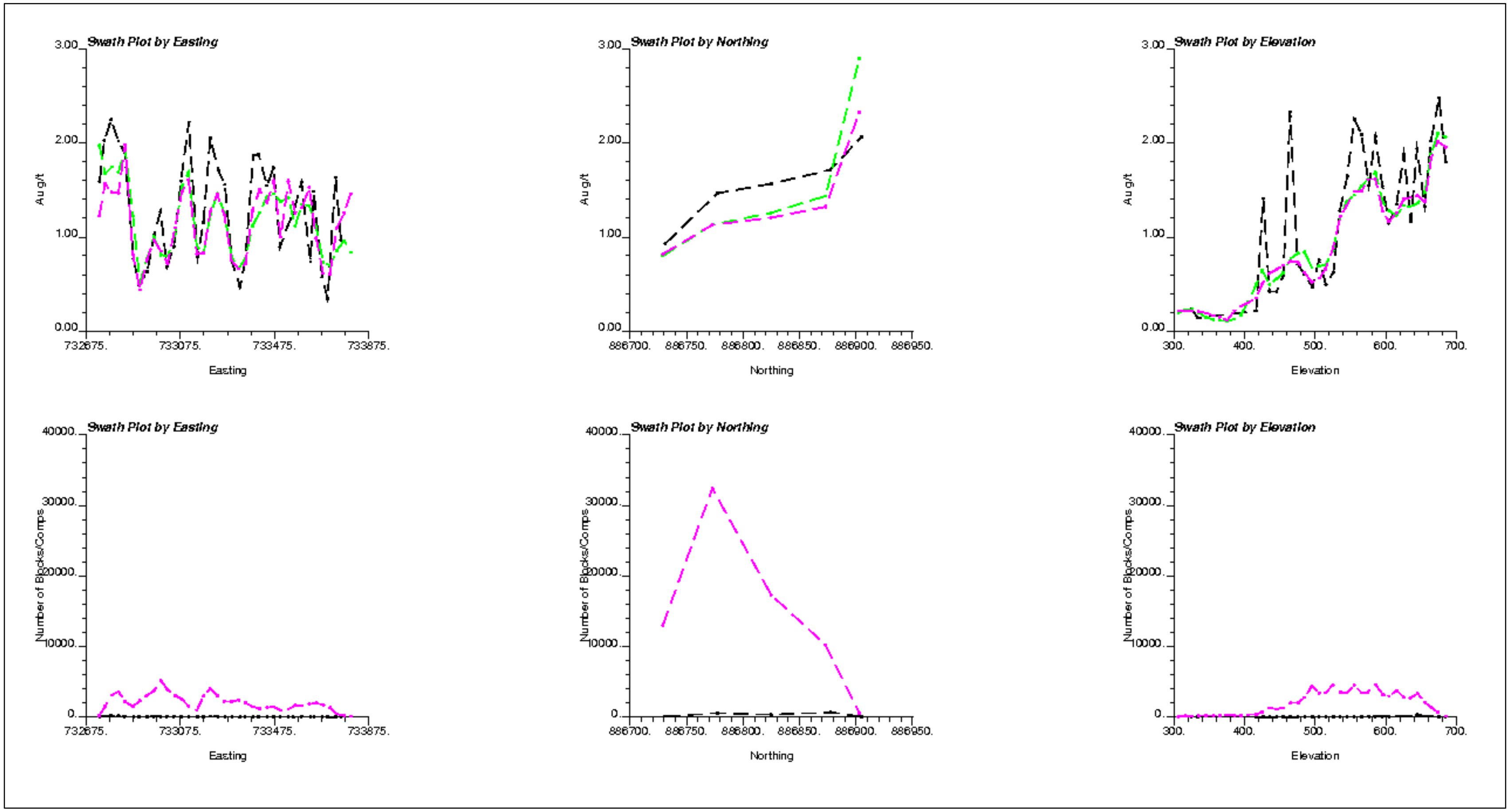
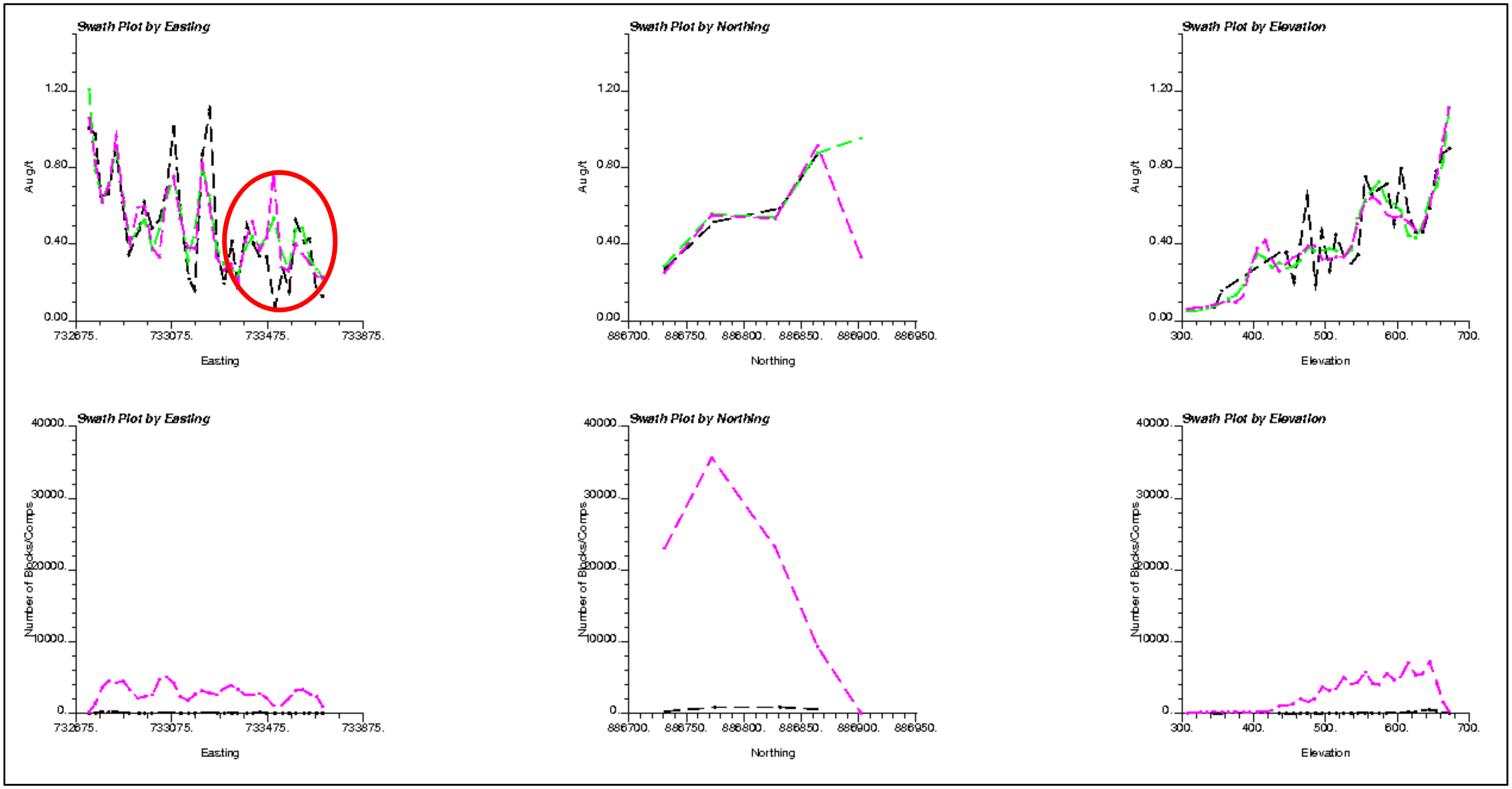


Figure 14.11: Gold Swath Plots by Easting, Northing and Elevation for the Hangingwall Stockwork

(the upper Swath plots show the grades, lower swath plots show number of blocks or composites)
(the GREEN lines represent the ordinary kriging mdel, the MAGENTA lines represent the nearest-neighbour model and the BLACK line represents composites)
(the area with a minor discrepancy is highlighted by the RED circle)



14.6.5 Grade Smoothing

A 6 m x 6 m x 6 m block size is considered suitable, by the Qualified Person, to represent a selective mining unit ("SMU") for an openpit mining operation with production rates between 4,000 and 5,000 tonnes per day.

Block model variance impacts predicted tonnes and grade (model selectivity) above any given cut-off grade. Usually a higher model variance will result in less predicted tonnes and higher predicted grade above a given cut-off grade. In other words, a higher model variance results in a higher model selectivity. Model selectivity is typically measured by comparing model grade-tonnage curves with calculated target model grade-tonnage curves. Target model grade-tonnage curves are calculated by correcting for change of support from a reference distribution (usually the de-clustered sample grade distribution [i.e. a nearest-neighbour model]) to the target distribution (in this case, a 3 m x 3 m x 3 m block grade distribution. Target grade-tonnage curves are dependent on the target model variance ("TMV"). TMV is given by:

TMV = Reference Distribution Variance x Block Dispersion Variance ("BDV")

Block dispersion variance is obtained from a unit sill variogram model. The Qualified Person conducted the change of support selectivity check on Measured and Indicated ordinary kriging block gold estimates within each domain, using Measured and Indicated blocks from a nearest-neighbour model reference distribution. The variance correction factors used in the Discrete Gaussian Model corrected grade-tonnage curves were calculated using the grade correlogram models based on 1.5 m composites. A correction factor was applied for the change of support from 1.5 m composites to 3.0 m composites, using the following formula:

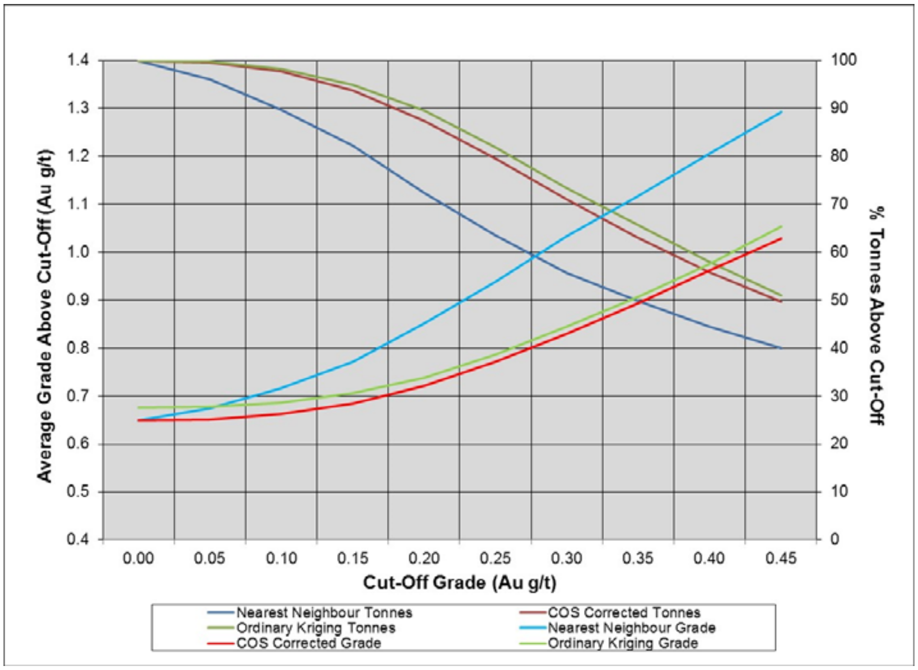
$$BDV_{adj} = BDV_{1.5m} \times (CV_{1.5m})^2 / (CV_{3m})^2$$

The grade-tonnage curves were combined within each variogram domain to give the grade-tonnage curve for the entire deposit. The grade-tonnage curves for the chosen 6 m x 6 m x 6 m SMU were then compared. The results are shown in Table 14.222. They show that the grade-tonnage curve of the ordinary kriging model closely matches the Discrete Gaussian Model corrected grade-tonnage curve, assuming a 6 m x 6 m x 6 m SMU. At cut-off grades between 0.1 g/t and 0.4 g/t Au, and assuming a 6 m x 6 m x 6 m SMU, the ordinary kriging model reports a difference of between +0.6% and +4.0% in the tonnes compared to the Discrete Gaussian Model corrected model. Based upon the modelled variograms and in the opinion of the Qualified Person, the internal grade dilution within the kriged gold grade model is appropriate for the SMU. The grade-tonnage curve is shown on Figure 14.12.

Table 14.22: A Summary of Grade-Tonnage Curve Data

Cut-Off (g/t Au)	Nearest-Neighbour			Herco			Kriged			% Differences		
	Tonnes	Grade (g/t Au)	Metal	Tonnes	Grade (g/t Au)	Metal	Tonnes	Grade	Metal	Tonnes	Grade	Metal
0.00	100	0.649	100	100	0.649	100	100	0.676	100	0.0%	4.2%	0.0%
0.05	96.026	0.675	99.81	99.581	0.652	99.977	99.69	0.678	99.983	0.1%	4.0%	0.0%
0.10	89.815	0.716	99.094	97.732	0.663	99.752	98.289	0.687	99.82	0.6%	3.6%	0.1%
0.15	82.188	0.771	97.622	93.753	0.685	98.970	94.972	0.706	99.192	1.3%	3.1%	0.2%
0.20	72.383	0.852	94.990	87.43	0.722	97.252	89.552	0.738	97.772	2.4%	2.2%	0.5%
0.25	63.592	0.939	91.961	79.499	0.772	94.497	81.978	0.786	95.244	3.1%	1.8%	0.8%
0.30	55.726	1.033	88.637	71.108	0.83	90.943	73.391	0.845	91.754	3.2%	1.8%	0.9%
0.35	49.808	1.117	85.684	63.105	0.894	86.941	65.645	0.907	88.042	4.0%	1.5%	1.3%
0.40	44.459	1.206	82.601	55.925	0.961	82.800	58.099	0.976	83.859	3.9%	1.6%	1.3%
0.45	40.064	1.292	79.724	49.702	1.028	78.732	51.002	1.053	79.409	2.6%	2.4%	0.9%

Figure 14.12: Nearest-Neighbour, COS Corrected and Ordinary Kriging Gold Grade-Tonnage Curves, All Domains



14.7 Drillhole Spacing Study

For Mineral Resource classification, the Qualified Person used the criteria that grade, tonnage and metal estimates should have a 90% confidence interval of ±15%. Measured Mineral Resources consider a quarterly production increment while Indicated Mineral Resources consider an annual production increment. The drillhole spacing study for the Moss Mine Mineral Resource model used the kriged estimation of the tonnage of mineralized material within a monthly production panel with indicator variograms (which, for purposes of this early stage analysis was modeled above a 0.20 g/t cut-off grade) and the kriged estimation of grades with grade variograms.

An expected relative standard error of the kriged estimate (“RSE”) can be calculated, even when grades are unknown, provided that the data location and the variogram parameters are known. A RSE is obtained by multiplying the normalized ordinary kriging standard deviation by the composite CV.

The relative accuracy at a 90% confidence limit on monthly production grades is obtained by multiplying the RSE by 1.645 (obtained from the standard normal distribution). The results are then scaled to quarterly and annual production. Assuming independence between the monthly panels the equations become Quarterly 90% = (1.645 x RSE) / $\sqrt{3}$ and Annual 90% = (1.645 x RSE) / $\sqrt{12}$.

The kriged estimation of metal within a monthly panel of production (with dimensions of 100 m east-west, 50 m north-south and 12 m in height) was simulated using idealized vertical or inclined drillhole grids with hole spacings varying from 12.5 m (easting) x 12.5 m (northing) up to 100 m (easting) x 100 m (northing). The capped composite CV was calculated for the Moss Vein and the Hangingwall Stockwork domains. The kriged estimation of tonnage was simulated by using an indicator variogram above a 0.2 g/t Au cut-off. The results are shown graphically on Figures 14.15 and 14.16.

The results based on estimation of gold grades suggest that a drill grid with a spacing of 25 m (east-west) x 12.5 m (north-south) would be sufficient to classify Measured Mineral Resources and a drill spacing of 50 m x 50 m would be sufficient to classify Indicated Mineral Resources.

It is recommended that the Company carefully evaluate and identify areas of the deposit with higher risk (e.g. areas with significantly higher grades than the average grade of the deposit areas with more discontinuous grades or areas which rely heavily on historic data types) and consider strategically located holes in those areas to mitigate the risks. Additional drilling would mitigate the risk by increasing local confidence in the estimated tonnage and grade above cut-off.

Figure 14.13: Drillhole Spacing Study Results, Moss Vein Gold Grades

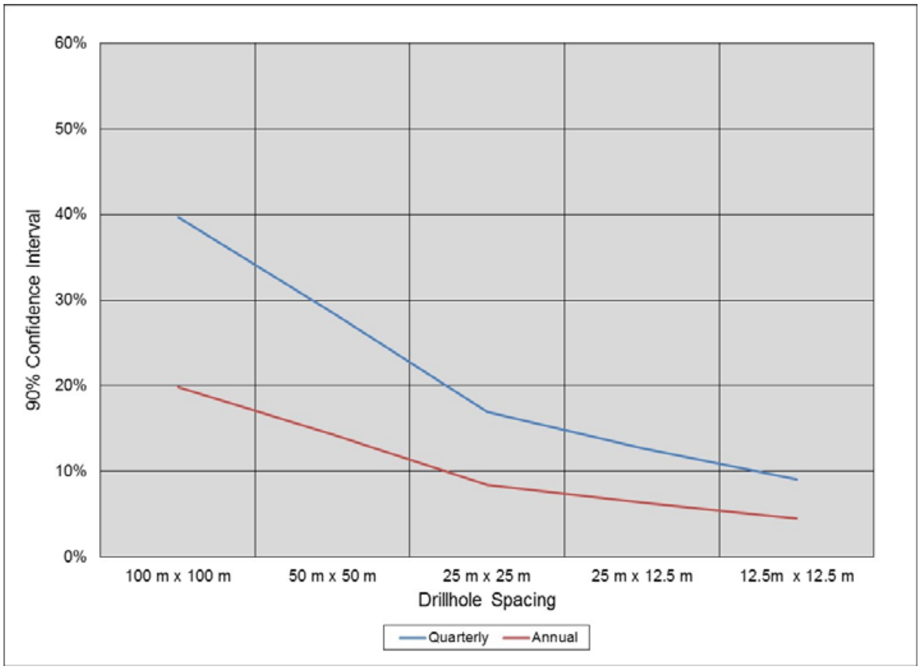


Figure 14.14: Drillhole Spacing Study Results, Hangingwall Stockwork Gold Grades

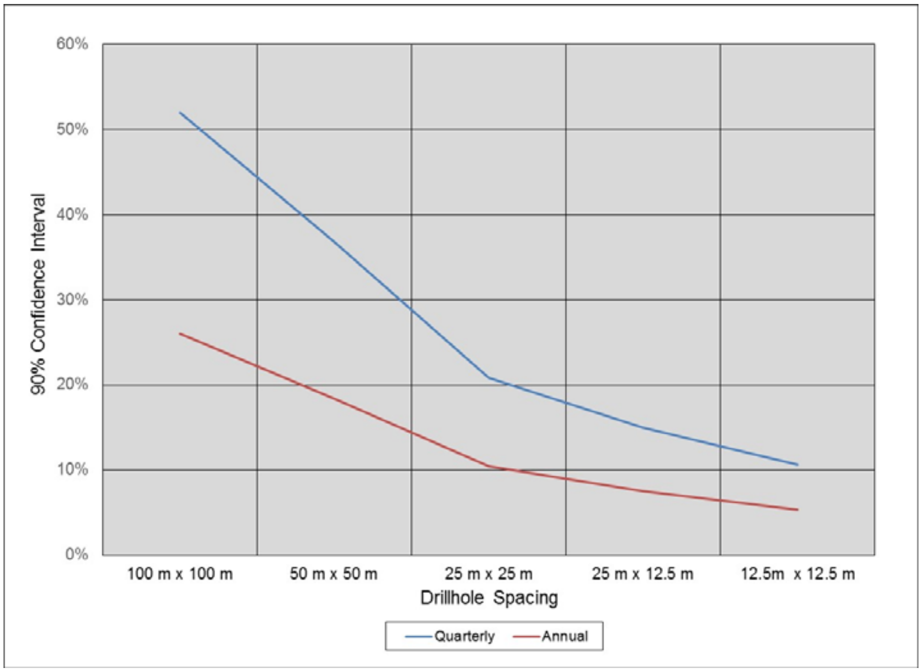


Figure 14.15: Drillhole Spacing Study Results, Moss Vein Tonnage (Indicator Variogram)

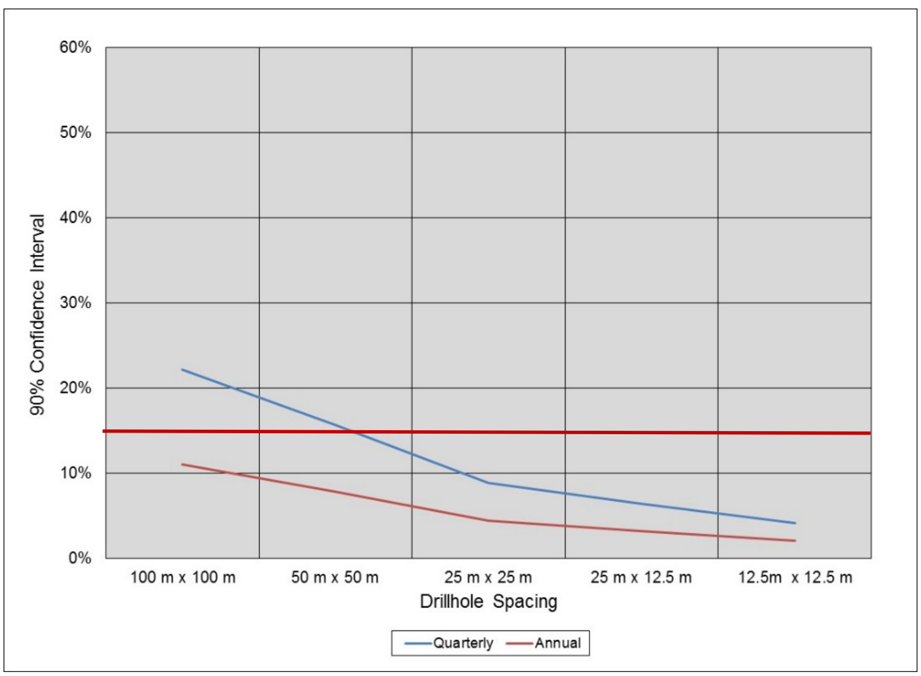
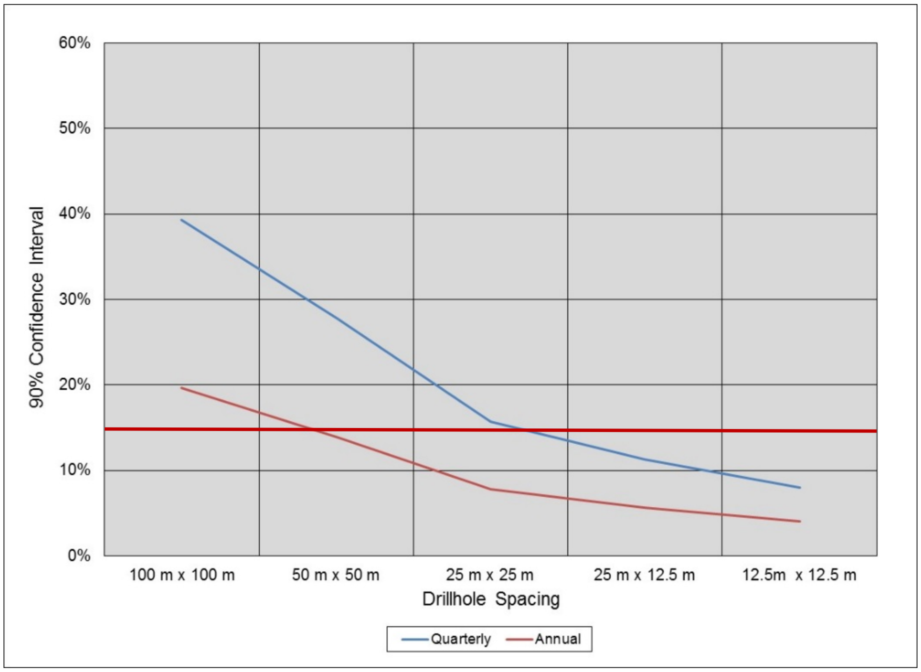


Figure 14.16: Drillhole Spacing Study Results, Hangingwall Stockwork Tonnage (Indicator Variogram)



14.8 Classification of Mineral Resources

The blocks were classified with an average distance from two holes of less than 38.5 m (i.e. with a 50 m x 50 m spacing) to the Indicated category, and with an average distance from three holes of less than 15.4 m (i.e. approximately a 25 m x 12.5 m spacing) to the Measured category. An annealing-based smoothing algorithm was used to remove either isolated blocks of Measured category material within areas of mostly Indicated category material, or isolated blocks of Indicated category material within areas of mostly Measured category blocks.

The geological model, data quality, geological continuity and metallurgical characteristics were reviewed for classification of Measured and Indicated mineral resources. The Ruth Vein, Vein No. 4 and West Footwall Stockwork domains are supported by drilling at orientations which are oblique to the dip of the domains. Significant difficulties were encountered with model validation in these domains. Measured or Indicated category Mineral Resources were, therefore, downgraded to the Inferred category. Blocks within the mineralization wireframe were classified into the Inferred Mineral Resource category where samples fell within 100 m of the block centroid. The mineralization solids represent the limit at which grade continuity can reasonably be assumed. A maximum distance of 100 m permits a reasonable local estimate of grades (as demonstrated by model validation).

14.9 Reasonable Prospects for Economic Extraction

The classified blocks were assessed for reasonable prospects of economic extraction by applying preliminary economics for potential openpit mining methods. Metallurgical testwork has been completed for the mineralization (see Section 13). Process and operating costs, metal prices, metallurgical recovery and a 65° slope angle were used to optimize a pit shell using a Lerchs-Grossman algorithm. The assessment does not represent an economic analysis of the deposit, but was used to determine reasonable assumptions for the purpose of determining the Mineral Resource. The assumed long-term metal prices were US$1,250/oz for gold and US$20.0/oz for silver. The assumed metallurgical recoveries were 82% for gold and 65% for silver, per the recommended recovery rates defined in Section 13.

14.10 Marginal Cut-Off Grade Estimation

A marginal gold cut-off value was estimated using a unit cost for mining mineralized material of US$6.36/t, including waste, a unit process cost (heap leach) of US$4.42/t and a unit on-site G&A cost of US$2.55/t. The marginal cut-off is based on the generally accepted practice that a decision is made at a pit rim if mined material above the marginal cut-off grade will lose less money if it is sent to the mill rather than if it is sent to the waste dump. It is considered for further processing if it contains a value that is greater than the cost to process it. On this basis and using the same metal prices and metallurgical recovery rates stated above, a gold cut-off grade of 0.25 g/t was selected for reporting Mineral Resources potentially amenable to an openpit mining method.

14.11 Mineral Resources Split East and West of the Canyon Fault

Tables 14.23 and 14.24 summarize the estimated Mineral Resources to the east (comprising the Moss and Ruth Veins) and to the west of the Canyon fault (the latter comprising West Extension – see Section 7). The tabulations show that there is a significantly higher Measured and combined Measured plus Indicated tonnage to the east of the Canyon fault. The gold equivalence formulae applied for purposes of compiling Tables 14.23 and 14.24 are same as those stated in Section 14.1.

Table 14.23: Mineral Resources East of the Canyon Fault (0.25 g/t Au cut-off)
(undiluted, pit constrained, 100% in-pit recovery, effective date October 31, 2014)

Category	Tonnes	Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)	AuEq (g/t)	AuEq (oz)
Measured	4,265,000	1.03	10.9	141,000	1,490,000	1.17	160,000
Indicated	4,910,000	0.87	11.8	137,000	1,860,000	1.02	161,000
Measured + Indicated	*9,175,000*	*0.94*	*11.4*	*278,000*	*3,350,000*	*1.09*	*321,000*
Inferred	805,000	0.60	4.5	16,000	120,000	0.66	17,000

Note: see Section 14.1 for the statement of the gold equivalence formulae

Table 14.24: Mineral Resources West of the Canyon Fault (West Extension, 0.25 g/t Au cut-off)
(undiluted, pit constrained, 100% in-pit recovery, effective date October 31, 2014)

Category	Tonnes	Au (g/t)	Ag (g/t)	Au (ozs)	Ag (ozs)	AuEq (g/t)	AuEq (oz)
Measured	595,000	0.54	7.3	10,000	140,000	0.63	12,000
Indicated	5,710,000	0.48	6.1	88,000	1,110,000	0.55	102,000
Measured + Indicated	*6,305,000*	*0.48*	*6.2*	*98,000*	*1,250,000*	*0.56*	*114,000*
Inferred	1,375,000	0.52	6.3	23,000	280,000	0.59	26,000

Note: see Section 14.1 for the statement of the gold equivalence formulae

14.12 Sensitivity of the Mineral Resources

The sensitivity of the Mineral Resources to changes in gold and silver prices was assessed by reporting the estimated Mineral Resources for several lower and higher cut-off grades (Table 14.25). The results show that the Mineral Resources are not highly sensitive to increasing cut-off grades (a proxy for decreasing metal prices). It is, therefore, concluded that the Mineral Resources are robust with respect to the choice of long-term metal price used for reporting.

Table 14.25: Moss Mine Project Mineral Resource Sensitivity

Cut-Off Grade (Au g/t)	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Measured					
Cut-Off Grade (Au g/t)	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**
0.10	5,565,671	0.87	9.5	155,696	1,699,849
0.15	5,395,128	0.89	9.7	155,002	1,686,496
0.20	5,163,451	0.93	10.0	153,691	1,664,940
0.25	**4,862,368**	**0.97**	**10.4**	**151,514**	**1,633,515**
0.30	4,510,637	1.02	11.0	148,400	1,590,573
Indicated					
Cut-Off Grade (Au g/t)	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**
0.10	12,425,993	0.59	7.9	236,667	3,169,880
0.15	12,180,571	0.60	8.0	235,635	3,150,624
0.20	11,597,637	0.62	8.3	232,300	3,093,732
0.25	**10,615,449**	**0.66**	**8.7**	**225,153**	**2,978,417**
0.30	9,390,849	0.71	9.3	214,336	2,817,128
Measured and Indicated					
Cut-Off Grade (Au g/t)	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**
0.10	*17,991,664*	*0.68*	*8.4*	*392,364*	*4,869,729*
0.15	*17,575,699*	*0.69*	*8.6*	*390,637*	*4,837,120*
0.20	*16,761,088*	*0.72*	*8.8*	*385,992*	*4,758,672*
0.25	***15,477,817***	***0.76***	***9.3***	***376,667***	***4,611,932***
0.30	*13,901,486*	*0.81*	*9.9*	*362,735*	*4,407,700*
Inferred					
Cut-Off Grade (Au g/t)	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**
0.10	2,736,733	0.47	4.9	41,786	433,465
0.15	2,651,846	0.49	5.0	41,444	430,226
0.20	2,444,003	0.51	5.3	40,255	416,716
0.25	**2,181,132**	**0.55**	**5.6**	**38,373**	**394,405**
0.30	1,911,644	0.59	6.0	35,985	367,930

14.13 Reconciliation of the 2014 and 2013 Mineral Resource Estimates

The results of a reconciliation analysis of the 2014 Mineral Resource model (“MRM”) and Mineral Resource estimate (“MRE”) with the 2013 MRM and declared 2013 MRE are presented in the following sub-sections. It should be emphasized that the base model for the reconciliation was the 2014 MRE at a grade cut-off of 0.20 g/t Au. This does not materially influence the reconciliation process or its outcomes because the 2014 and 2013 MRMs were adjusted using a step-wise process to achieve normalized models that allowed direct comparisons between the models to be made.

The following text and its supporting tables and figure were taken from a consultancy report to the Company by David Thomas, P. Geo., entitled ‘2014 Model Reconciliation to 2013 PEA Model’ dated October 03, 2014 and a consultancy report by Stephen Godden, Independent Mining Consultant, entitled ‘Moss Mine Gold-Silver Project, 2013 to 2014 Mineral Resource Estimates’ Reconciliation (Summary)’ and dated October 09, 2014.

14.13.1 Statement of Reconciliation

Table 14.26 summarizes the outcomes of the step-wise reconciliation analyses described below. The results show that no material difference exists between the normalized 2014 MRE and fully adjusted 2013 MRE, in terms of AuEq ounces defined using the 2013 MRE equivalent metal price ratio (Au 1 : Ag 50). This may be expected because largely the same database was used for both estimates (300 additional assays were included in the 2014 MRM). It was instead the differences of approach when compiling the two MRMs and subsequent MREs that led to the difference in AuEq ounces apparent in the base case models. If the AuEq differences between the base case models are examined in a logical, step-wise manner the MRE outcomes are nearly identical in terms of AuEq ounces.

Table 14.26: A Summary of Outcomes, 2013 to 2014 Reconciliation Analysis

Modelled Case	2013 Estimate				2014 Estimate			
	Step-Wise Difference (oz AuEq)		Adjusted Model		Normalized Model (per Table 16.1)		Difference to Step-Wise Adjusted 2013 MRE (oz AuEq)	
	M+I	Inferred	M+I (oz AuEq)	Inferred (oz AuEq)	M+I (oz AuEq)	Inferred (oz AuEq)	M+I	Inferred
Normalized Models	-	-	654,000	82,000	472,000	50,000	- 182,000	- 32,000
Re-Block 2013 MRM to 3 m x 3 m x 3 m	- 13,000	-	641,000	82,000	472,000	50,000	- 169,000	- 32,000
2014 Wireframe Constraint (FW only)	- 106,000	- 10,000	535,000	72,000	472,000	50,000	- 63,000	- 22,000
2014 Wireframe Constraint (HW only)	- 115,000	- 51,000	420,000	21,000	472,000	50,000	+ 52,000	+29,000
Mineralization within 2014 MRM Wireframe	+ 51,000	+29,000	471,000	50,000	472,000	50,000	+ 1,000	0

Note: To conform with the 2013 Mineral Resource Estimate, an equivalent metal price ratio of Au 1 : Ag 50 was used

14.13.2 Base Case Mineral Resource Estimates

Table 14.27 summarizes the 2013 (0.30 g/t Au grade cut-off) and the 2014 MRE at a grade cut-off of 0.20 g/t Au. The former is the estimate detailed in the 2013 Technical Report, the latter represents the first-pass outcome of the Mineral Resource estimation process described above. The 2014 MRE at a grade cut-off of 0.20 g/t Au forms part of the sensitivity analysis described in Section 14.10. For purposes of the 2013 MRE a 1 Au : 50 Ag metal price equivalence was employed. The AuEq grades and ounces stated on Table 14.27 for the 2014 MRE at a grade cut-off of 0.20 g/t Au were determined by applying the following formulae:

Factor A (gold) = 1 / 31.10346 x metallurgical recovery (82%) x smelter recovery (99%) x refinery recovery (99%) x unit Au price (US$1,450/oz)

Factor B (silver) = 1 / 31.10346 x metallurgical recovery (65%) x smelter recovery (98%) x refinery recovery (99%) x unit Ag price (US$22.50/oz)

AuEq = Au grade + (Ag grade x [Factor B / Factor A])

Table 14.27: A Summary of the 2013 Mineral Resource Estimate and the 2014 Mineral Resource Estimate at a Grade Cut-Off of 0.20 g/t Au

2013 Mineral Resource Estimate (0.30 g/t Au grade cut-off) *(undiluted, 100% recovery, unconstrained, 1 Au : 50 Ag metal price equivalence)*							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	12,611,000	0.85	9.10	345,000	3,690,000	1.03	419,000
Indicated	9,978,000	0.60	6.68	192,000	2,143,000	0.73	235,000
Measured + Indicated	*22,589,000*	*0.74*	*8.03*	*537,000*	*5,833,000*	*0.90*	*654,000*
Inferred	3,957,000	0.52	6.30	66,000	801,000	0.65	82,000
2014 Mineral Resource Estimate at a Grade Cut-Off of 0.20 g/t Au *(undiluted, 100% recovery, LG pit constrained, AuEq as stated above)*							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	5,235,000	0.92	9.9	154,000	1,670,000	1.04	175,000
Indicated	11,890,000	0.62	8.3	237,000	3,160,000	0.72	275,000
Measured + Indicated	*17,125,000*	*0.71*	*8.8*	*391,000*	*4,840,000*	*0.82*	*450,000*
Inferred	2,690,000	0.50	5.1	43,000	440,000	0.56	49,000

14.13.3 Normalized Mineral Resource Estimates

The step-wise process used to align the base case 2014 MRE and the previously declared 2013 MRE removed the principal differences between the two estimates:

- the 2013 MRE was unconstrained, whereas the 2014 MRE reflects the mineralization above grade cut-off within the boundaries of an optimized Lerchs-Grossman pit;
- the 2013 MRE used a 0.30 g/t Au grade cut-off, whereas the base case 2014 MRE was for a 0.20 g/t Au grade cut-off;
- for purposes of the 2013 MRE, a ratio of Au 1 : Ag 50 of the assumed metal prices was used to estimate gold equivalence, whereas metal prices and recoveries were used to estimate gold equivalence for purposes of the 2014 MRE (as defined by the formulae stated above); and
- for purposes of the 2013 MRE, tonnes and ounces were rounded to the nearest thousand whereas, for the 2014 MRE, rounding to the nearest 5,000 t, 1,000 oz Au and AuEq, and 10,000 oz Ag was employed.

The consequence of each step-wise adjustment to the base case 2014 MRE is summarized on Table 14.28 (values highlighted in GREEN are added to the gold equivalent ounces estimated for the base case 2014 MRE, RED values are subtracted). The process yielded a normalized 2014 MRE comprising 472,000 oz AuEq in the Measured and Indicated categories of Mineral Resource, plus 50,000 oz AuEq in the Inferred category of Mineral Resource, at a grade cut-off of 0.3 g/t Au.

Table 14.28: A Summary of Outcomes, 2014 MRE Alignment to 2013 MRE Base Parameters, Moss Mine Project

Modelled Case	2014 Estimate			
	Step-Wise Difference (oz AuEq)		Adjusted Model	
	M+I	Inferred	M+I (oz AuEq)	Inferred (oz AuEq)
Estimate at a 0.20 g/t Au grade cut-off (per Table 14.27)	-	-	450,000	49,000
Removal of LG Pit Constraint	+18,000	+6,000	468,000	55,000
Alignment of Grade Cut-Offs (to 0.30 g/t Au)	- 33,000	-8,000	435,000	47,000
AuEq Alignment (to 2013 MRE Au 1 : Ag 50 AuEq model)	+37,000	+3,000	472,000	50,000
Normalized 2014 MRE	**-**	**-**	***472,000***	***50,000***

The lower portion of Table 14.29 summarizes the results of the adjustments to the base case 2014 MRE, as summarized above and in terms of Mineral Resource category. The upper part of Table 14.29 summarizes the base case (declared) 2013 MRE, per Table 14.27. Comparison of the MREs summarized on Table 14.29 shows that an additional 182,000 oz AuEq in the Measured plus Indicated ("M+I") categories were estimated within the scope of the base case 2013 MRE, compared to the normalized 2014 MRE, with an additional 32,000 oz AuEq in the Inferred category.

Table 14.29: A Summary of the Base Case 2013 MRE and Normalized 2014 MRE
(undiluted, 100% recovery, unconstrained, 0.30 g/t Au grade cut-off, AuEq = 1 Au : 50 Ag)

2013 Mineral Resource Estimate (per Table 14.27)							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	12,611,000	0.85	9.10	345,000	3,690,000	1.03	419,000
Indicated	9,978,000	0.60	6.68	192,000	2,143,000	0.73	235,000
Measured + Indicated	22,589,000	*0.74*	*8.03*	*537,000*	*5,833,000*	*0.90*	*654,000*
Inferred	3,957,000	0.52	6.30	66,000	801,000	0.65	82,000
Normalized 2014 Mineral Resource Estimate (adjusted, as summarized above)							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	4,536,000	1.02	10.9	149,000	1,594,000	1.24	181,000
Indicated	10,301,000	0.69	9.2	230,000	3,041,000	0.88	291,000
Measured + Indicated	14,837,000	0.79	9.7	379,000	4,635,000	0.99	472,000
Inferred	2,258,000	0.58	5.7	42,000	415,000	0.69	50,000

14.13.4 Differences (Mineral Resource Models)

Table 14.30 summarizes the principal differences between the MRMs on which the base case 2013 MRE and the normalized 2014 MRE depend. Some of the differences outlined are technical (geostatistical) in nature and were found to have little or no impact in terms of reconciling the declared 2013 MRE with the normalized 2014 MRE. The 182,000 oz AuEq (M+I) plus 32,000 oz AuEq difference between the two MREs, identified on Table 14.29, may instead be attributed to the differences between the block sizes and the lack of wireframes used within the scope of the 2013 MRM. The impact of normalizing these differences is described in the following sub-sections.

Table 14.30: A Summary of Key Differences between the 2013 MRM and 2014 MRM

2013 Mineral Resource Model	**2014 Mineral Resource Model**
Mineral Resource Model	
Mineralization constrained on footwall side only (no hard boundary on hangingwall side)	Wireframes employed to constrain tonnes & grade estimates (footwall & hangingwall sides)
Footwall constraint inconsistent with surface geology	Wireframes fully consistent with geology (at and below surface)
Single domain used	10 different domains considered, as defined by 2014 MRM wireframes
Block Model	
5 m x 5 m x 5 m blocks, plus sub-blocks on footwall contact	3 m x 3 m x 3 m blocks
5 m composites used	1.5 m composites
Single bulk density value used	Two average bulk mean densities used (above and below 12 m bs)
Tonnes & Grade Estimation	
No geostatistical analysis carried out	Full geostatistical analysis carried out
Single search ellipse used; orientated at oblique angle to the Moss Vein	Moss Vein search ellipse N100ºE/70º, search ellipses for other domains' orientated to wireframes
ID^3 estimation method used with single pass	Ordinary kriging used with multiple passes
All composites used above constraining footwall surface	Composite restrictions applied to ensure appropriate amount of internal dilution

14.13.5 Re-Blocking the 2013 Mineral Resource Model

Table 14.31 provides a summary of the base case 2013 MRE (per Table 14.29) and the results for the same model when it has been re-blocked to 3 m x 3 m x 3 m blocks (i.e. to conform with the 2014 MRM). It may be seen that re-blocking results in 13,000 fewer AuEq ounces in the M+I category but the same estimated amount of AuEq ounces in the Inferred category. This reduces the difference in the amount of AuEq ounces (M+I) between the adjusted base case 2013 MRE and normalized 2014 MRE to 169,000 oz AuEq.

Table 14.31: A Summary of Results, Base Case and Re-Blocked 2013 MRM
(undiluted, 100% recovery, unconstrained, 0.30 g/t Au grade cut-off, 1 Au : 50 Ag AuEq)

Base Case 2013 Mineral Resource Estimate (per Table 14.29)							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	12,611,000	0.85	9.10	345,000	3,690,000	1.03	419,000
Indicated	9,978,000	0.60	6.68	192,000	2,143,000	0.73	235,000
Measured + Indicated	*22,589,000*	*0.74*	*8.03*	*537,000*	*5,833,000*	*0.90*	*654,000*
Inferred	3,957,000	0.52	6.30	66,000	801,000	0.65	82,000
Re-blocked 2013 Mineral Resource Estimate							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	12,820,000	0.81	8.79	335,000	3,621,000	0.99	408,000
Indicated	10,196,000	0.58	6.54	190,000	2,142,000	0.71	232,000
Measured + Indicated	*23,016,000*	*0.71*	*7.79*	*525,000*	*5,763,000*	*0.87*	*641,000*
Inferred	4,087,000	0.50	6.15	66,000	808,000	0.62	82,000

14.13.6 Footwall Mineralization

If the wireframes used for the 2014 MRM are applied to the re-blocked 2013 MRM, 3.95 Mt of above cut-off mineralized material may be identified in the footwall of the Moss Vein, which material probably doesn’t exist. Table 14.32 summarizes this:

- the top portion of summarizes the re-blocked 2013 MRE, per Table 14.31;
- the middle portion summarizes the “additional” footwall material that probably doesn’t exist; and
- the bottom portion summarizes the re-blocked and adjusted 2013 MRE (adjusted to exclude the “additional” footwall material).

Table 14.32: A Summary of Results, Re-Blocked and Adjusted 2013 MRM
(adjusted to exclude footwall mineralization that probably doesn’t exist)
(undiluted, 100% recovery, unconstrained, 0.30 g/t Au grade cut-off, 1 Au:50 Ag equivalence)

Re-blocked 2013 Mineral Resource Estimate (per Table 14.31)							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	12,820,000	0.81	8.79	335,000	3,621,000	0.99	408,000
Indicated	10,196,000	0.58	6.54	190,000	2,142,000	0.71	232,000
Measured + Indicated	*23,016,000*	*0.71*	*7.79*	*525,000*	*5,763,000*	*0.87*	*641,000*
Inferred	4,087,000	0.50	6.15	66,000	808,000	0.62	82,000
“Additional” Footwall Mineralization							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	1,797,000	0.87	9.44	50,000	545,000	1.06	61,000
Indicated	1,686,000	0.68	7.49	37,000	406,000	0.83	45,000
Measured + Indicated	*3,482,000*	*0.78*	*8.50*	*87,000*	*951,000*	*0.95*	*106,000*
Inferred	466,000	0.54	8.00	8,000	120,000	0.70	10,000
Re-Blocked and Adjusted 2013 Mineral Resource Estimate							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	11,023,000	0.80	8.68	285,000	3,076,000	0.97	347,000
Indicated	8,510,000	0.56	6.35	153,000	1,736,000	0.69	187,000
Measured + Indicated	*19,533,000*	*0.70*	*7.66*	*438,000*	*4,812,000*	*0.85*	*535,000*
Inferred	3,621,000	0.49	5.91	58,000	688,000	0.61	72,000

The results summarized on Table 14.32 show that the re-blocked 2013 MRE per Table 14.31, compared with the re-blocked and adjusted 2013 MRM, over-estimates the amount of AuEq ounces by some 106,000 oz AuEq in the M+I category and by some 10,000 oz AuEq in the Inferred category. This may be attributed to the lack of wireframes to constrain estimation in the 2013 MRM (grade smearing into the footwall can and did occur, by virtue of the geostatistical/probabilistic method used to estimate block grades). The effect is to:

- further reduce the difference in the amount of AuEq ounces (M+I) between the adjusted base case 2013 MRE and normalized 2014 MRE to 63,000 oz AuEq; and
- reduce the difference in the amount of AuEq ounces (Inferred category) between the adjusted base case 2013 MRE and normalized 2014 MRE to 22,000 oz AuEq.

14.13.7 Hangingwall Mineralization

An additional 8.49 Mt of above cut-off mineralized material may also be identified in the hangingwall of the Moss Vein, if the wireframes used for the 2014 MRM are applied to the re-blocked 2013 MRM. In common with the “additional” footwall material considered above, this material probably doesn’t exist. Table 14.33 summarizes the outcome if this material is excluded:

- the top portion summarizes the re-blocked and adjusted base case 2013 MRE, per Table 14.32;
- the middle portion summarizes the “additional” hangingwall material, which probably doesn’t exist; and
- the bottom portion summarizes the re-blocked and further adjusted base case 2013 MRE.

Table 14.33: A Summary of Results, Re-Blocked and Further Adjusted 2013 MRM
(adjusted to exclude hangingwall mineralization that probably doesn’t exist)
(undiluted, 100% recovery, unconstrained, 0.30 g/t Au grade cut-off, 1 Au:50 Ag equivalence)

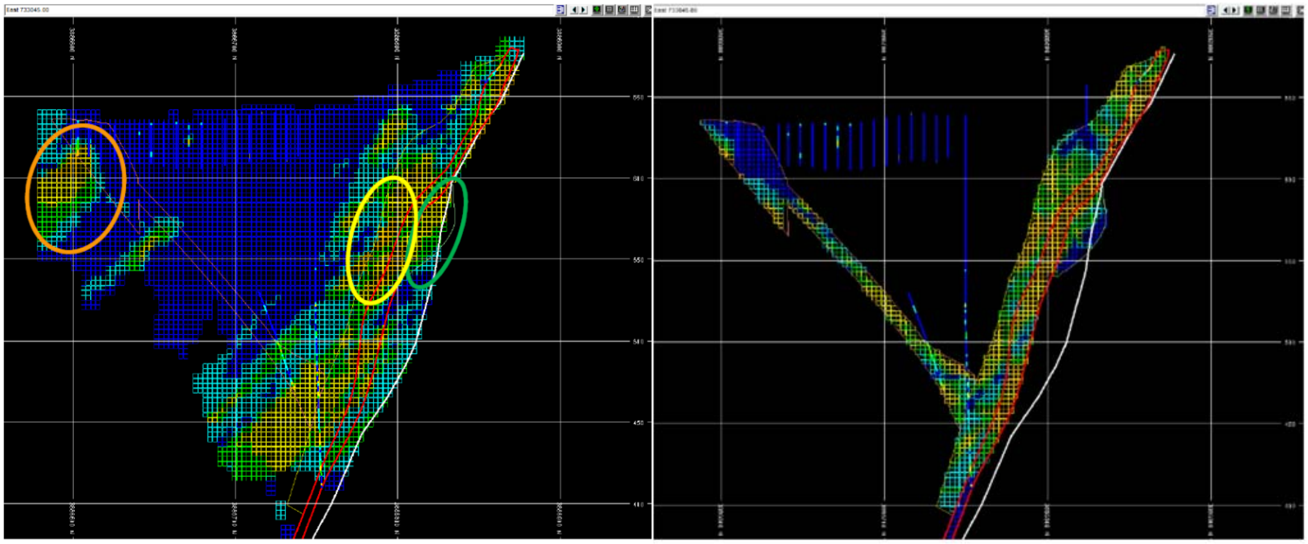
Re-Blocked and Adjusted 2013 Mineral Resource Estimate (per Table 14.32)							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	11,023,000	0.80	8.68	285,000	3,076,000	0.97	347,000
Indicated	8,510,000	0.56	6.35	153,000	1,736,000	0.69	187,000
Measured + Indicated	*19,533,000*	*0.70*	*7.66*	*438,000*	*4,812,000*	*0.85*	*535,000*
Inferred	3,621,000	0.49	5.91	58,000	688,000	0.61	72,000
“Additional“ Hangingwall Mineralization							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	2,391,000	0.51	5.44	40,000	418,000	0.62	48,000
Indicated	3,538,000	0.49	5.05	55,000	574,000	0.59	67,000
Measured + Indicated	*5,929,000*	*0.50*	*5.21*	*95,000*	*992,000*	*0.60*	*115,000*
Inferred	2,561,000	0.50	5.93	41,000	488,000	0.62	51,000
Re-Blocked and Further Adjusted 2013 Mineral Resource Estimate							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	8,632,000	0.88	9.58	245,000	2,658,000	1.07	299,000
Indicated	4,972,000	0.61	7.28	98,000	1,162,000	0.76	120,000
Measured + Indicated	*13,604,000*	*0.78*	*8.73*	*343,000*	*3,820,000*	*0.96*	*420,000*
Inferred	1,060,000	0.48	5.87	17,000	200,000	0.60	21,000

The results summarized on Table 14.33 show that the re-blocked and adjusted 2013 MRE (per Table 14.32) over-estimates the amount of AuEq ounces by some 115,000 oz AuEq in the M+I category and by some 51,000 oz AuEq in the Inferred category. This may again be attributed to the lack of wireframes to constrain estimation in the 2013 MRM (grade smearing into the hangingwall occurred by virtue of the geostatistical/probabilistic method used to estimate block grades). The effect is to:

- *reverse* the difference in AuEq ounces (M+I), between the re-blocked and further adjusted 2013 MRE and normalized 2014 MRE, to 52,000 oz AuEq in favour of the latter; and
- *reverse* the difference in AuEq ounces (Inferred category), between the re-blocked and further adjusted 2013 MRE and normalized 2014 MRE, to 29,000 oz AuEq in favour of the latter.

Figure 14.17 provides an example of the effect of grade smearing caused by a lack of wireframes within the scope of the 2013 MRM (the snapshot on the left), and compares this with the wireframe-constrained 2014 MRM. The orientation of the mineralized blocks to the left of the 2013 MRM snapshot (which constitutes the Ruth Vein) is a function of the orientation of the search ellipse that remained constant across the 2013 MRM. Domain-specific search ellipses were used within the scope of the 2014 MRM.

Figure 14.17: Section 733,045 East, Looking West, 2013 MRM (Left) 2014 MRM (Right)
(the 2014 MRM Moss Vein is shown in **RED** outline, the 2013 MRM footwall structure is shown in WHITE outline)
(areas with footwall mineralization are highlighted in**GREEN**, hangingwall mineralization is highlighted in **YELLOW**)
(Ruth vein mineralization with an inappropriate search orientation highlighted in **ORANGE**)



14.13.8 Constrained Mineralization

Constraining the mineralization in the re-blocked 2013 MRM with the 2014 MRM wireframes further shows that at a grade cut-off of 0.30 g/t Au, the 2014 MRM contains 2.419 Mt of additional mineralized material compared with the 2013 MRM. Table 14.33 summarizes this:

- the top portion summarizes the mineralization contained within the wireframes in the 2014 MRM;
- the middle portion summarizes the mineralization contained within 2014 MRM wireframes when they are applied to the 2013 MRM; and
- the bottom portion summarizes the difference between the two.

Table 14.34: A Summary of Results, Mineralization within the 2014 MRM Wireframes
(2014 MRM versus re-blocked and further adjusted 2013 MRM, per Table 14.33)
(undiluted, 100% recovery, unconstrained, 0.30 g/t Au grade cut-off, 1 Au:50 Ag equivalence)

Above Grade Cut-Off Mineralization In 2014 MRM							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	4,535,000	1.02	10.93	149,000	1,594,000	1.24	181,000
Indicated	10,295,000	0.69	9.19	230,000	3,041,000	0.88	291,000
Measured + Indicated	*14,830,000*	*0.79*	*9.72*	*379,000*	*4,635,000*	0.99	471,000
Inferred	2,254,000	0.58	5.73	42,000	415,000	0.69	50,000
Above Grade Cut-Off Mineralization In Re-Blocked 2013 MRM, Using 2014 MRM Wireframes							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	8,632,000	0.88	9.57	246,000	2,657,000	1.07	299,000
Indicated	4,973,000	0.61	7.27	98,000	1,163,000	0.76	121,000
Measured + Indicated	*13,605,000*	*0.78*	*8.73*	*343,000*	*3,820,000*	0.95	420,000
Inferred	1,060,000	0.5	5.86	17,000	200,000	0.62	21,000
Difference Between The Wireframe Constrained 2104 and Re-Blocked 2013 MRMs							
Category	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Au (oz)**	**Ag (oz)**	**AuEq g/t**	**AuEq (oz)**
Measured	-4,097,000	0.14	1.36	-97,000	-1,063,000	0.17	-118,000
Indicated	5,322,000	0.08	1.91	132,000	1,878,000	0.12	170,000
Measured + Indicated	*1,225,000*	*0.01*	*0.99*	*36,000*	*815,000*	0.03	51,000
Inferred	1,194,000	0.08	-0.12	25,000	215,000	0.08	29,000

To make the base case 2013 MRE directly comparable with the normalized 2014 MRE, the additional tonnage contained within the 2014 MRM wireframes, as described, should be added to the re-blocked and further adjusted 2013 MRE, per Table 14.33. This may be justified by consideration of the grade estimation plan, the search ellipse orientation selected for the 2013 MRE and the replacement of missing silver grades by linear regression against gold in the 2014 MRM: the 2013 MRM search ellipse was oriented at an oblique angle to the Moss Vein, therefore areas within the wireframe appear as waste within the 2013 MRM. The ID^3 grade interpolation method used in the 2013 MRM (also) results in a rapid decrease in tonnes above increasing cut-offs.

Put another way, the mineralization that would otherwise have been defined within the 2014 wireframes was, in the 2013 MRM, smeared to the hangingwall and footwall of the Moss Vein in particular. If the smeared mineralization is deducted from the 2013 MRE then, to compensate for this, the additional mineralization within the 2014 MRM wireframes should be added to the 2013 MRE. The effect is to:

- increase the amount of M+I AuEq ounces attributable to a fully adjusted 2013 MRE by 51,000 oz AuEq; and
- increase the amount of Inferred AuEq ounces attributable to a fully adjusted 2013 MRE by 29,000 oz AuEq; hence

- compared with a fully adjusted 2013 MRE, yield a normalized 2014 MRE containing one thousand more ounces M+I AuEq but the same amount of Inferred AuEq ounces.

14.14 Factors That May Affect The Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- the applied, long-term commodity price and exchange rate assumptions;
- the operating cost assumptions;
- the applied metallurgical recovery rates and any changes that might result from additional metallurgical testwork;
- changes to the tonnage and grade estimates as a result of new assay and bulk density information;
- future tonnage and grade estimates may vary significantly as more drilling is completed;
- permitting of mining operations on land which is not registered as a patented lode claim; and
- any changes to the slope angle of the pit walls as a result of geotechnical information would affect the pit shell used to constrain the Mineral Resources.

14.15 Qualified Person's Opinion

The Qualified Person (Mr. David Thomas, P. Geo.) is of the opinion that the Mineral Resources for the Moss Mine Project have been performed to best industry practices and conform to the requirements of CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. The Mineral Resource estimate is well-constrained by three-dimensional wireframes representing geologically realistic volumes of mineralization.

Exploratory data analysis conducted on assays and composites shows that the wireframes are suitable domains for Mineral Resource estimation. Grade estimation has been performed using an interpolation plan designed to minimize bias in the average grade and to provide grade estimates with a variance approximating those predicted from the variograms models and using an SMU of 6 m x 6 m x 6 m.

It is concluded as a result of validation of the Mineral Resource block model that:

- visual inspection of block grade versus composited data shows a good reproduction of the data by the model;
- checks for global bias in the grade estimates show differences generally within acceptable levels (less than 10%). Domains with larger differences between the nearest-neighbour and ordinary kriging models either have a low number of composites or are those with drilling oblique to the trend of the mineralization (the nearest-neighbour model therefore does not provide a robust reference for validation);
- checks for global bias in the grade estimates on Measured and Indicated blocks show differences within acceptable levels (less than 5%);
- checks for local bias (swath plots) indicate good agreement for all variables, except in areas where there is significant extrapolation beyond the drillholes;

- a check on grade smoothing (model selectivity) for potential openpit mining using a global change-of-support correction shows that the amount of smoothing is acceptable around the cut-off grades of interest and are generally less than 5%;
- the impact of capping as assessed by estimating uncapped and capped grade models - generally the amounts of metal removed by capping in the models are consistent with the amounts calculated during the grade capping study on the composites;
- the Mineral Resources were classified using confidence intervals scaled to volumes of production relevant to the Moss Mine Project;
- the Mineral Resources are constrained and reported using economic and technical criteria such that the Mineral Resources have reasonable prospects of economic extraction; and
- the Mineral Resources are not highly sensitive to changes in cut-off grade and is therefore not sensitive to small to moderate changes (increases or decreases) in the gold price.

15 ADJACENT PROPERTIES

There are no adjacent properties that are of relevance to the Moss Mine Project.

16 OTHER RELEVANT DATA AND INFORMATION

There is no additional information that is pertinent to the 2014 Mineral Resource update that is the subject of this Technical Report.

17 INTERPRETATION AND CONCLUSIONS

17.1 2014 Mineral Resource Update

The 2014 Mineral Resource estimate ("MRE"), which is the subject of this Technical Report, reflects a significant reduction in the quantity of equivalent gold ounces compared with the 2013 MRE stated in the 2013 Technical Report (450,000 oz AuEq versus 645,000 oz AuEq in the Measured plus Indicated categories). A reconciliation analysis is provided in Section 14.13. This shows that if the two Mineral Resource models ("MRM") and subsequent MREs are normalized and adjusted to thereby allow them to be directly compared, no material difference exists. This may be expected because largely the same database was used for both estimates (300 additional assays were included in the 2014 MRM). It was instead mainly the differences of approach when compiling the two MRMs and subsequent MREs that led to the difference in AuEq ounces apparent in the base case models (i.e. the MREs stated in this Technical Report and the 2013 Technical Report). Of particular importance are the use of wireframes and geological domains within the scope of the 2014 MRM: the lack of wireframes in the 2013 MRM and the use of a single geological domain resulted in grade smearing to the hangingwalls and footwalls of both the Moss Vein and Ruth Vein. The effect was to create mineralization (i.e. to populate MRM blocks with grades), the presence of which is not supported by local drillhole data.

17.2 Risks and Uncertainties

The 2014 MRE forms part of an on-going feasibility study of Phase II (Commercial Operations) of the Company's Moss Mine project development plan. During Phase II mineralized material from the Moss Vein, its Western Extension and associated stockworks, hence a portion of the 2014 MRE, will be exploited.

No significant technical risks or uncertainties are identified as regards the target mineralization:

- the geology of the Moss deposit is straightforward and amenable to exploitation through openpit mining;
- the deposit appears to be a conventional oxide type (it is not necessary to differentiate between mineralized material located above and below the present watertable);
- the amounts of potentially deleterious elements are minor to negligible; and
- analysis of metallurgical test results and the outcomes of Phase I show that Moss Vein mineralized material is very amenable to cyanidation.

Additional bottle roll tests on West Extension mineralized material are, however, recommended to fill a data gap and thereby establish whether its metallurgical response is similar to that of Moss Vein mineralized material (see Section 18).

A heap leach cyanidation process, of the type used during Phase I, may reasonably be anticipated to be suitable for extracting gold and silver from Western Extension mineralized material. However, if lower gold and/or silver metallurgical recovery rates were found to apply, this might affect the Mineral Resources if a higher grade cut-off was required (for Western Extension mineralized material only) to compensate for the reduction in gold and/or silver recovery. In MineFill's opinion, this would not materially affect the Moss Mine Project:

- while a reduction in Mineral Resource tonnes for the West Extension would inevitably result from the selective adoption of a higher grade cut-off, the analysis provided in Section 14.12 shows that the Moss deposit is not especially sensitive to changes in cut-off grade (i.e. any reduction is likely to be small); and, further to which,
- the analysis provided in Section 14.11 shows that –
  - the AuEq grade of Moss Vein mineralized material (to the east of the Canyon fault) is nearly twice that of the West Extension (1.09 g/t AuEq versus 0.56 g/t AuEq), and
  - there are nearly three time as many AuEq ounces within the Mineral Resources to the east of the Canyon fault (321,000 oz AuEq versus 114,000 oz AuEq [estimated using the AuEq formulae stated in Section 14]).

18 RECOMMENDATIONS

To fill the data gap identified as a result of the metallurgical review presented in Section 13, it is recommended that up to six standard bottle roll tests are carried out on P_{95} 6.35 mm (1/4”) crushed drillcore samples of West Extension mineralized material (vein and stockwork). Bottle roll tests only are required due to the very good repeatability between bottle roll and column leach tests across the seven metallurgical test programs that included cyanidation testing. A quote for six bottle roll tests and related head screen analyses was in December 2014 secured from McClelland Laboratories. The estimated total program cost is US$11,500, inclusive of samples’ identification, packaging and transport. The Company plans to have the tests undertaken in January or February 2015.

It is recommended that the preliminary paragenetic model presented in Sub-Section 7.2.4 is finalized, in part by carrying out additional mineral petroscopy on polished sections. Apart from mineralogical determinations, it is recommended that the program includes analyses of the deportment and grain sizes of native gold, electrum and acanthite. The program should include material from both the Moss Vein and West Vein (West Extension) and their associated stockworks. A quote for such a program was received from Robert Cuffney, Consulting Geologist. The estimated total program cost is US$12,000, inclusive of samples’ identification, packaging and transport. The Company initiated the program in December 2014.

In Section 14.7 it is recommended that the Company carefully evaluate and identify areas of the deposit with higher risk as regards drillhole spacing and the classification of Mineral Resources (e.g. areas with significantly higher grades than the average grade of the deposit areas with more discontinuous grades or areas which rely heavily on historic data types). The Company plans to undertake such evaluations and carry to out additional drilling, if required.

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APPENDIX A

SCATTER PLOTS

- DRILLHOLE DEVIATION ANALYSIS -

Figure A.1: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 2012 Diamond Drillholes, NE Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

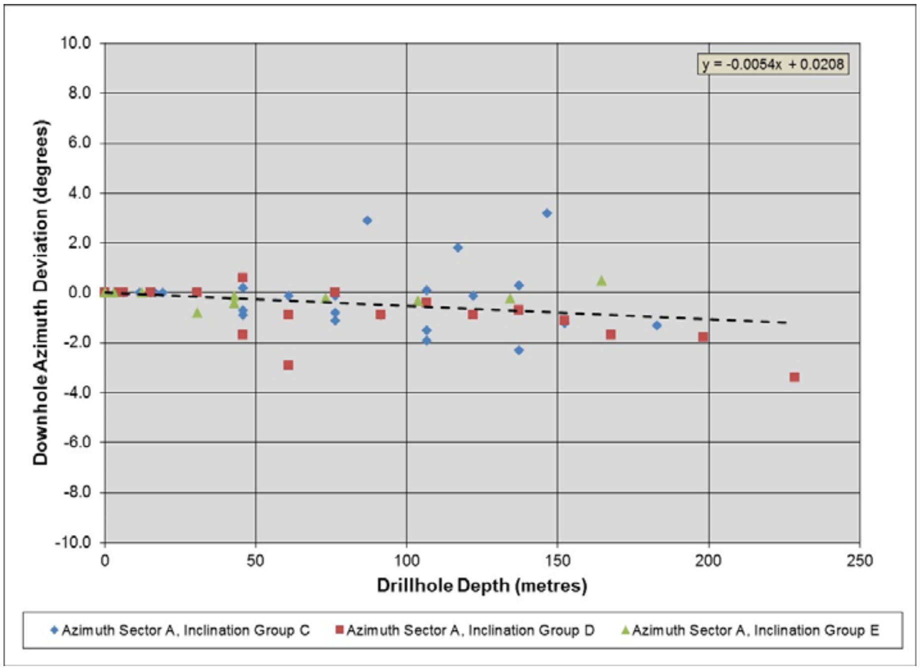

Figure A.2: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 2012 Diamond Drillholes, South Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

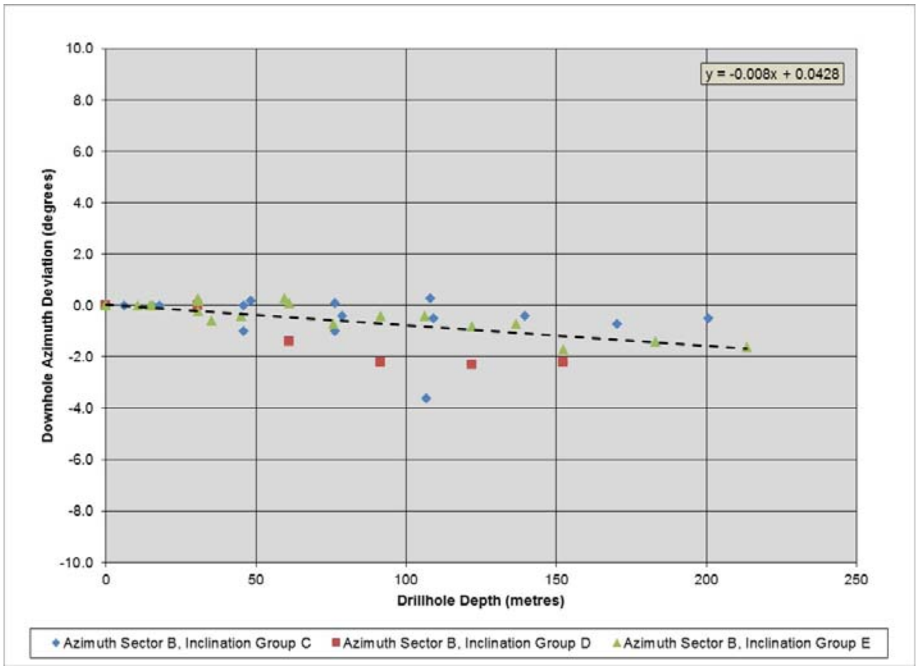

Figure A.3: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 2012 Diamond Drillholes, NE Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

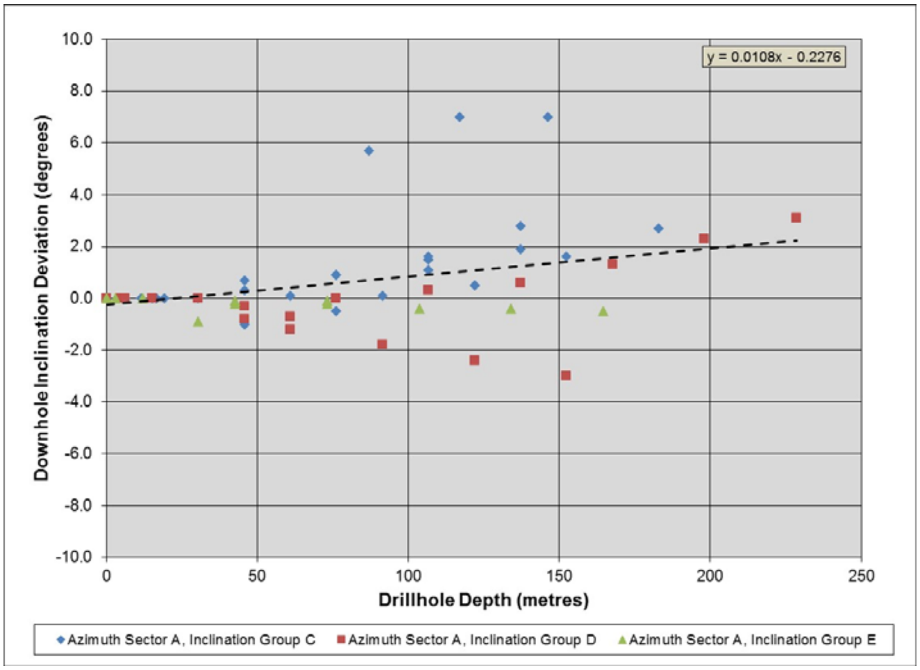

Figure A.4: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 2012 Diamond Drillholes, South Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

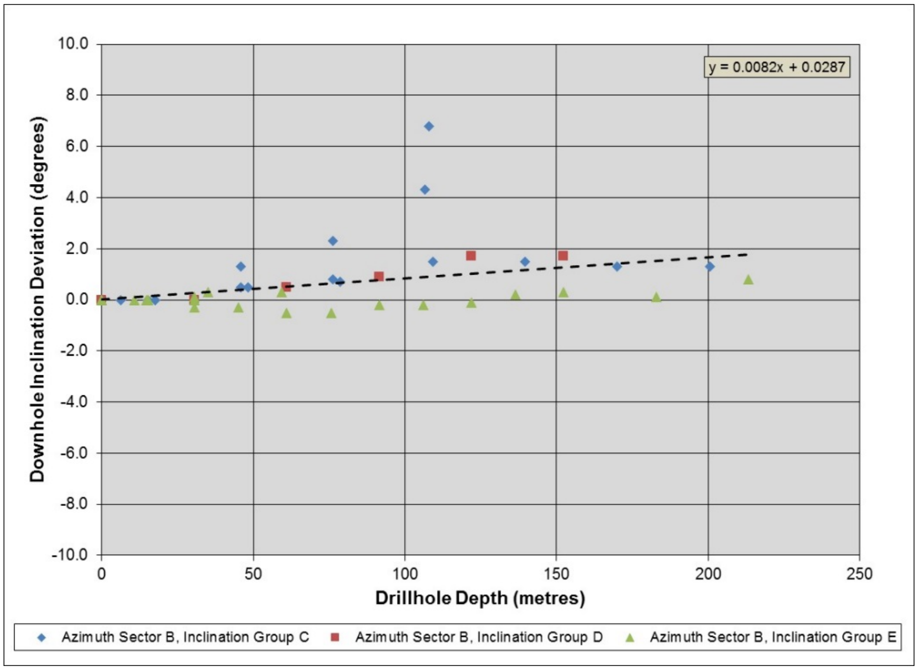

Figure A.5: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 2013 Diamond Drillholes, NE Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

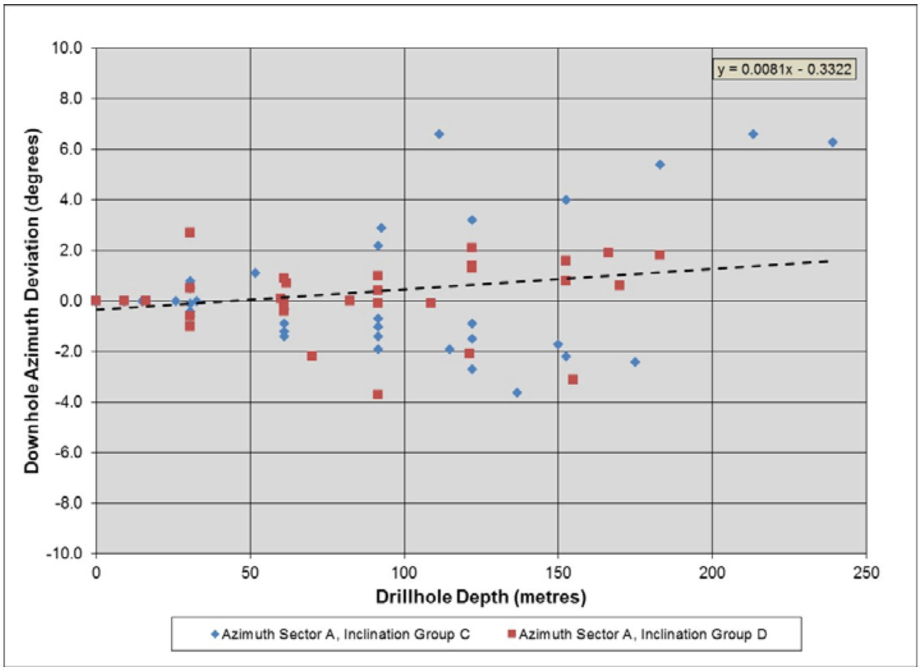

Figure A.6: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 2013 Diamond Drillholes, South Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

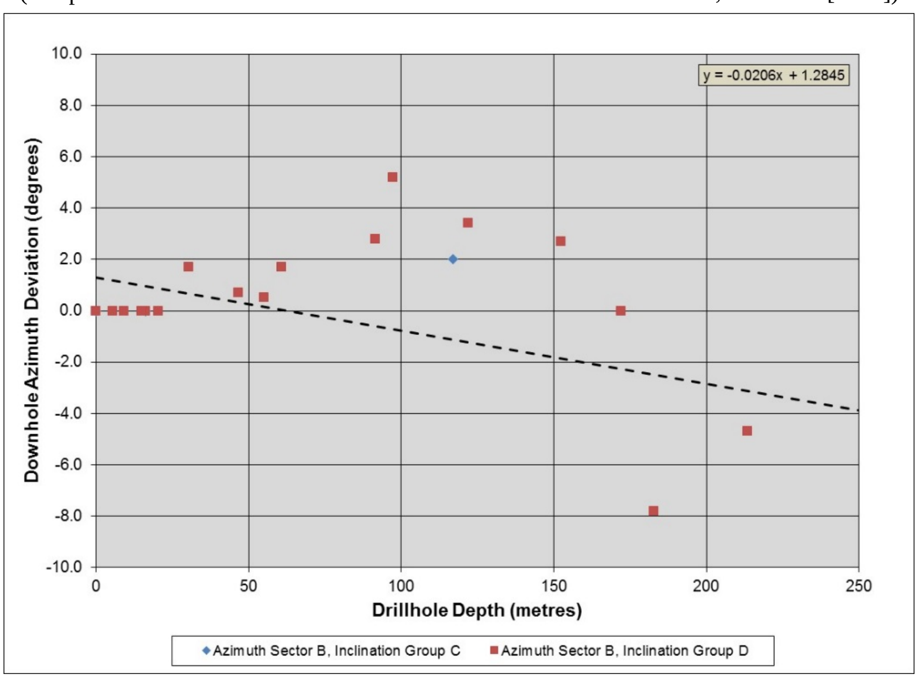

Figure A.7: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 2013 Diamond Drillholes, NE Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

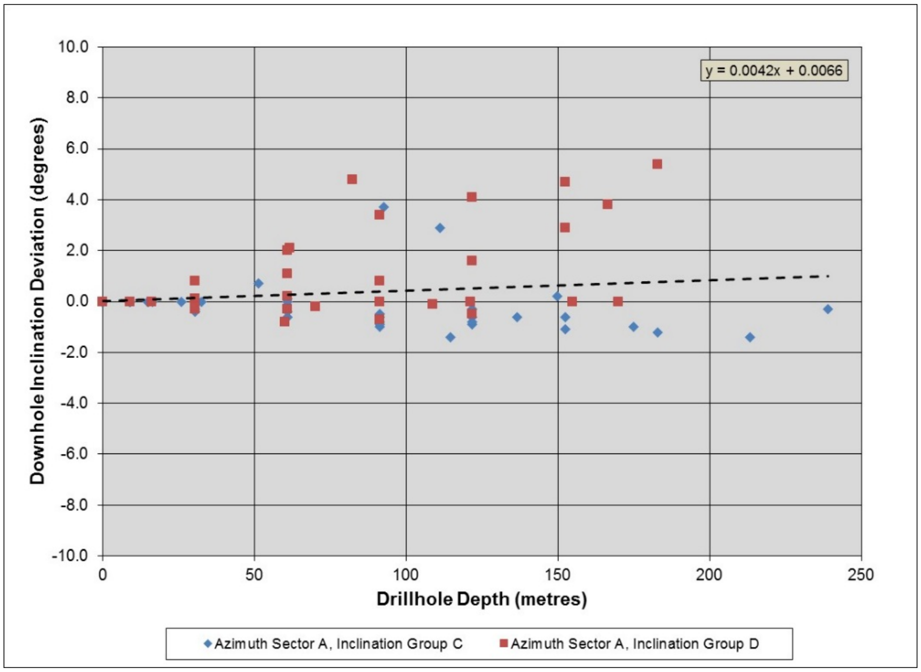

Figure A.8: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 2013 Diamond Drillholes, South Azimuth Sector, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

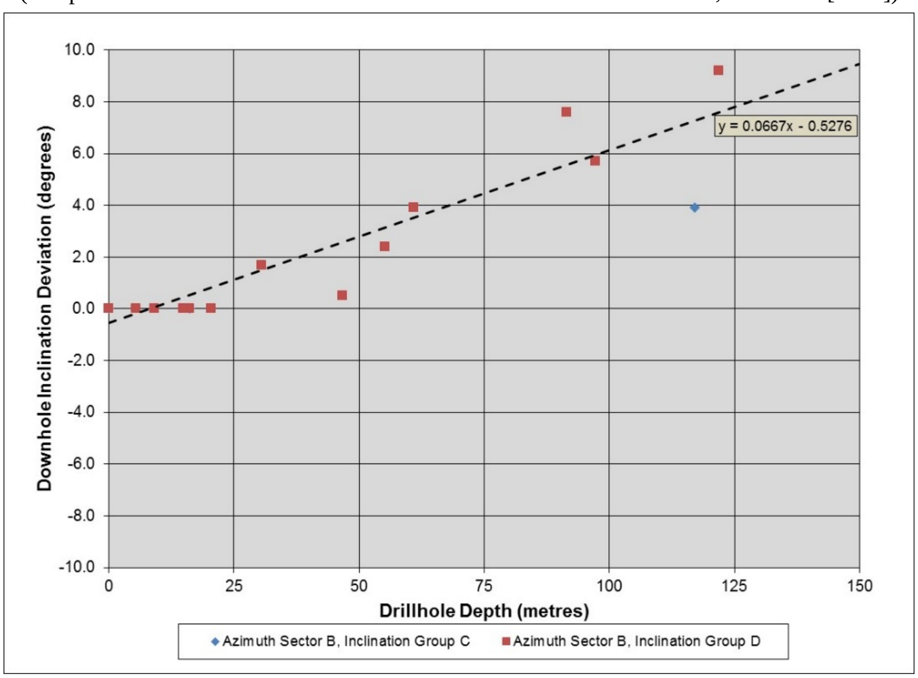

Figure A.9: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 5.25" RC Drillholes (2012), NE Azimuths, Group C Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

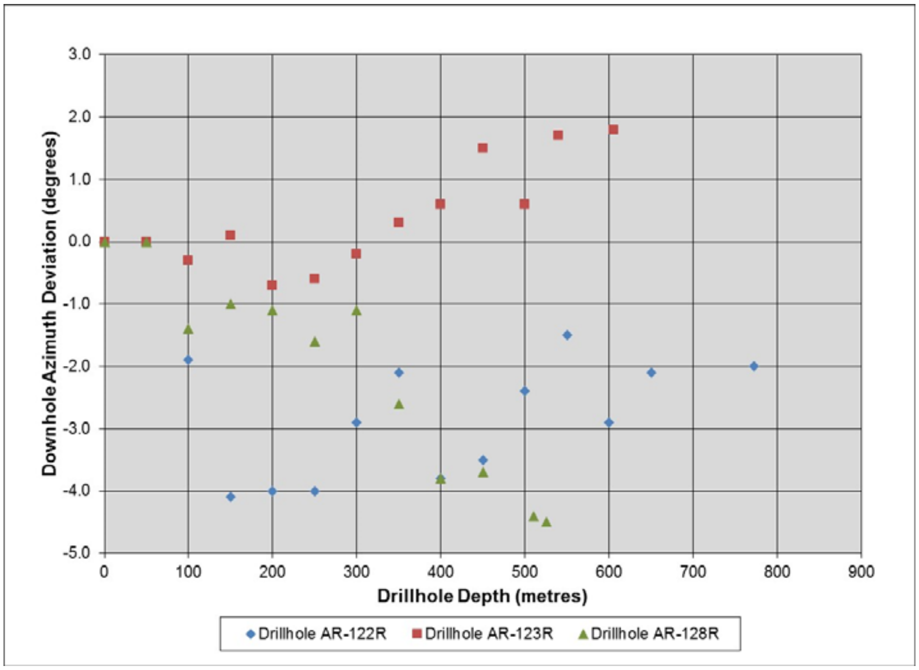

Figure A.10: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 6" RC Drillholes, NE Azimuth Sector, Group A Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

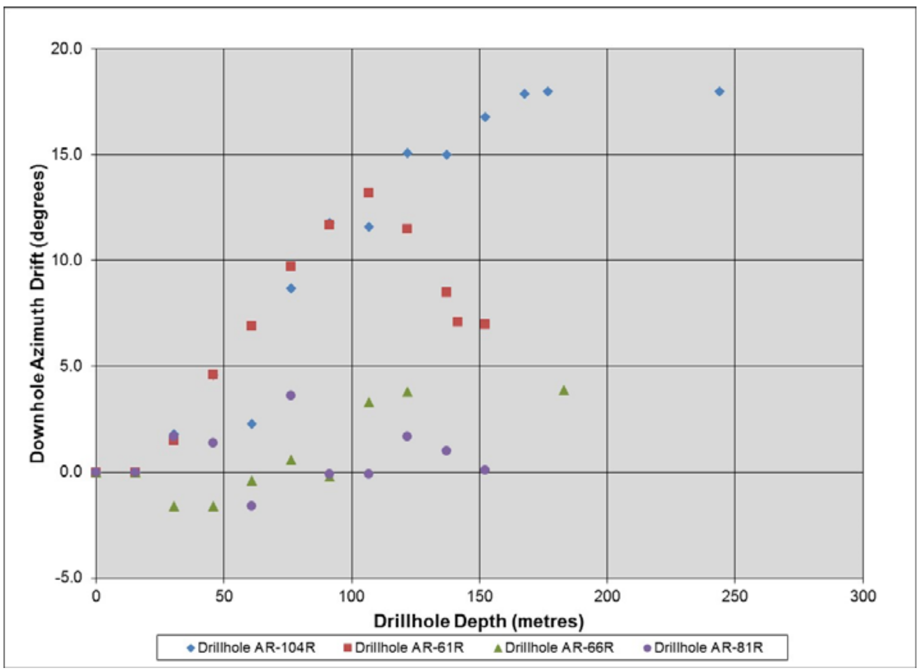

Figure A.11: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 6" RC Drillholes, NE Azimuth Sector, Group B Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

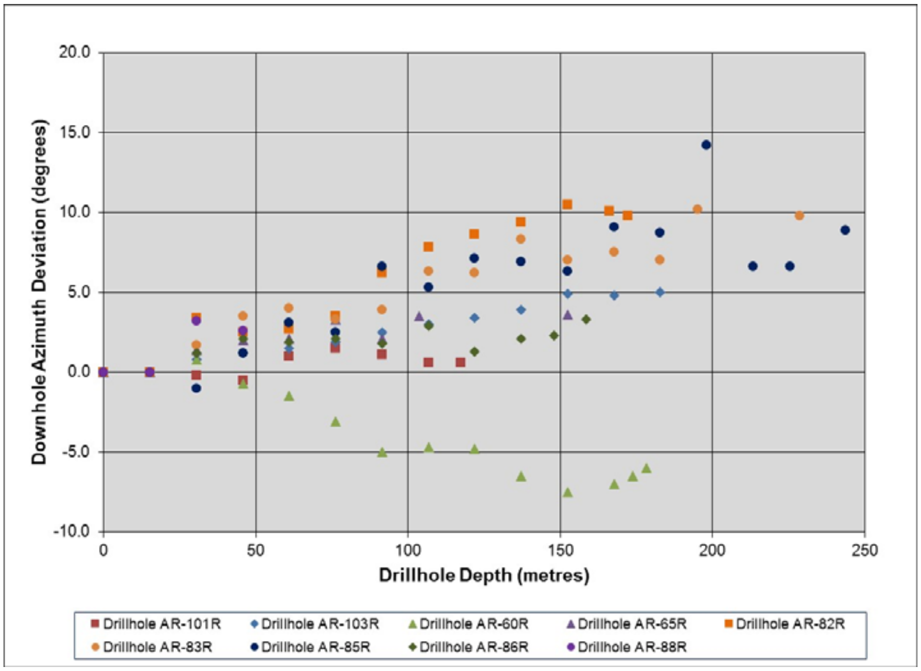

Figure A.12: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 6" RC Drillholes, NE Azimuth Sector, Group C Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

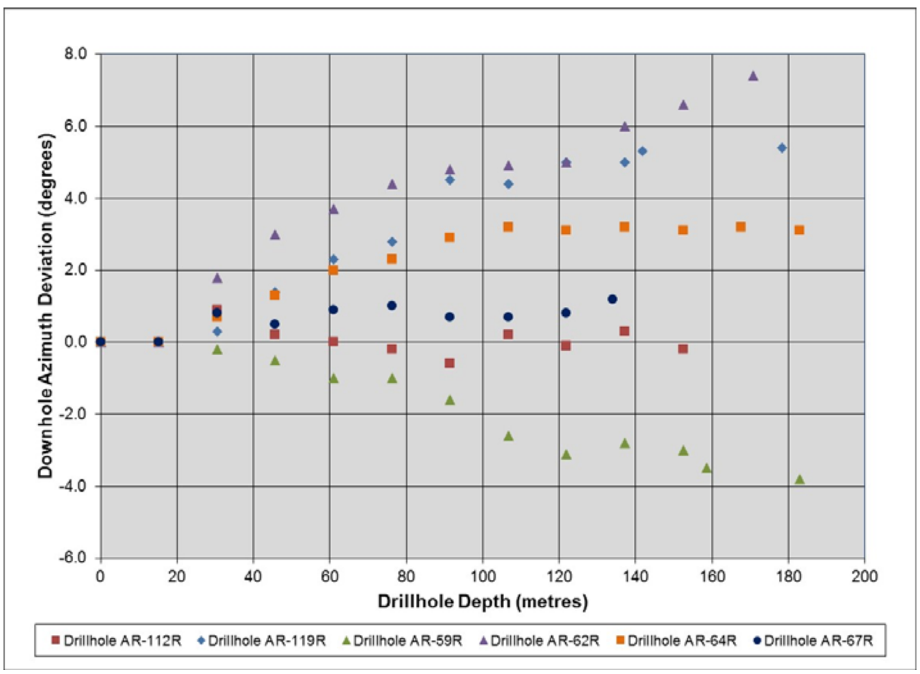

Figure A.13: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Azimuth Deviation, 6" RC Drillholes, NE Azimuth Sector, Group D Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

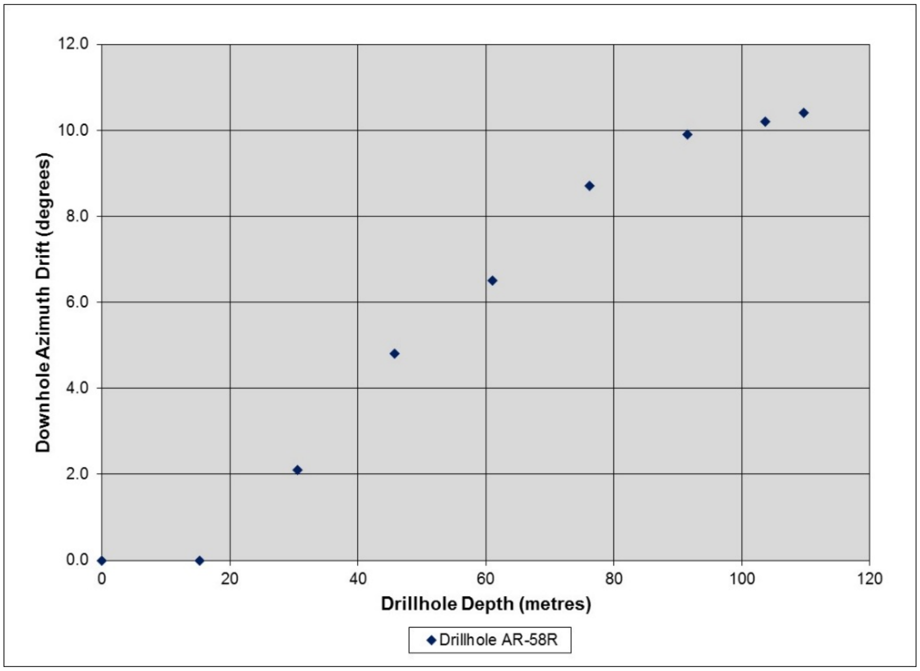

Figure A.14: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 5.25" RC Drillholes (2012), NE Azimuths, Group C Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

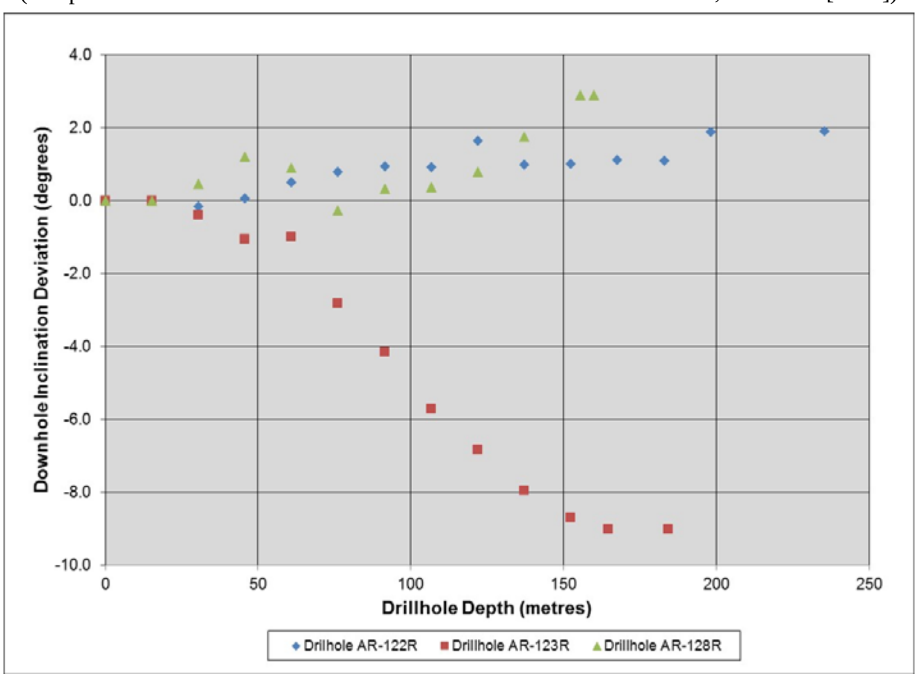

Figure A.15: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 6" RC Drillholes, NE Azimuth Sector, Group A Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

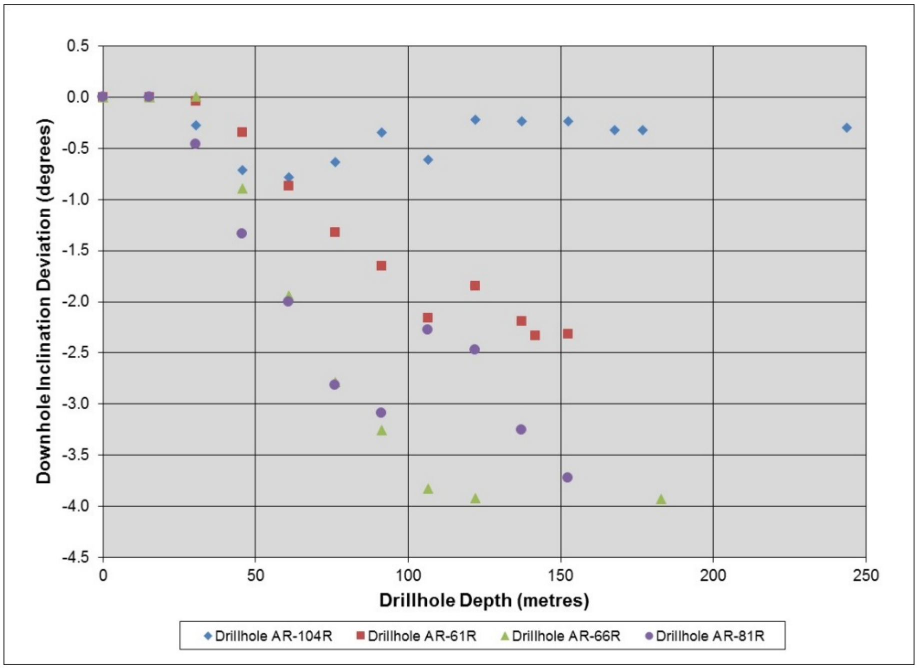

Figure A.16: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 6" RC Drillholes, NE Azimuth Sector, Group B Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

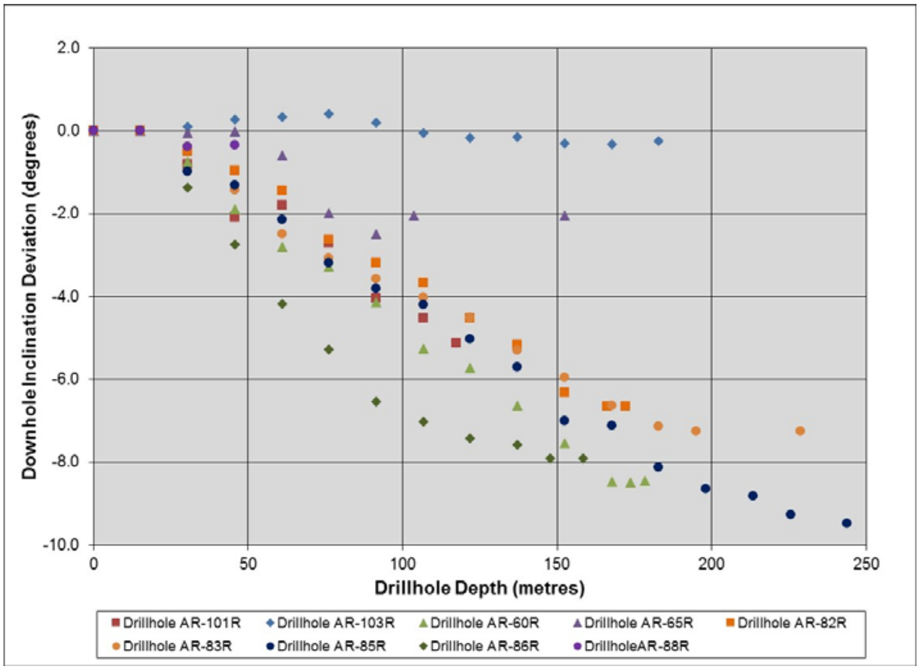

Figure A.17: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 6" RC Drillholes, NE Azimuth Sector, Group C Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])

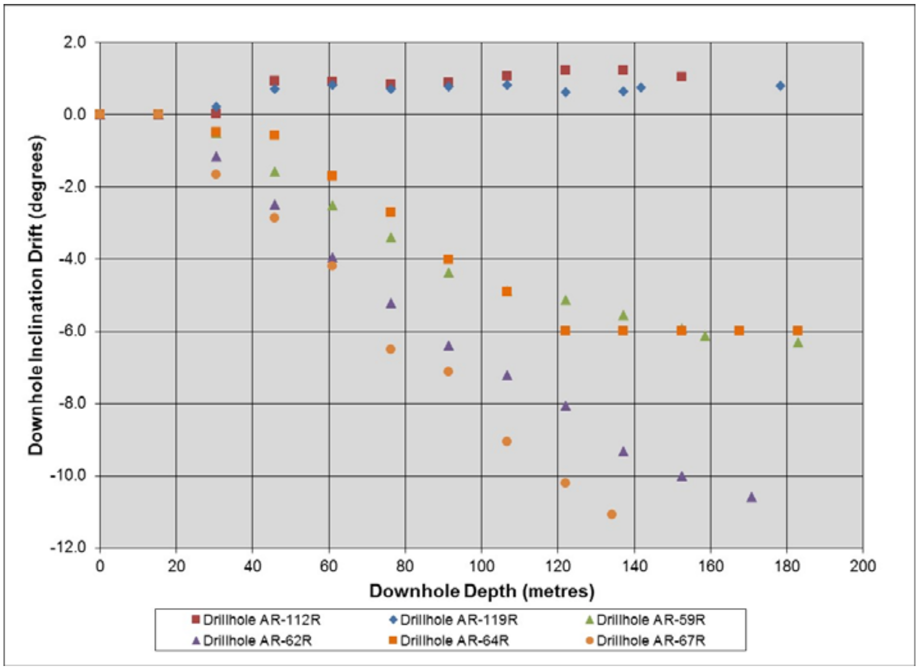

Figure A.18: A Scatter Plot and Trendline Showing the Magnitudes of Downhole Inclination Deviation, 6" RC Drillholes, NE Azimuth Sector, Group D Inclinations, Moss Mine Project
(compiled from data contained in the drillhole database verification files, Brownlee [2014])