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Technical Report

Getchell Project NI 43-101 Technical Report

Premier Gold Mines Limited and i-80 Gold Corp

Humboldt County, Nevada, USA

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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1 Summary

1.1 Introduction

This Technical Report (the Report) provides an update of the Mineral Resource estimates and metallurgy of the Mineral Resources identified within the Getchell Project (Property) located in Humboldt County, Nevada, USA. The Report has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada on behalf of Premier Gold Mines Limited / i-80 Gold Corp (i-80 or the Companies).

On 10 August 2020, Premier entered into a definitive purchase agreement with affiliates of Waterton Global Resource Management, Inc. to acquire from Waterton all of the outstanding membership interests of Osgood Mining Company LLC (OMC).

The Property comprises a number of property parcels which collectively encompass 2,545 acres in the Potosi mining district. The four-square miles of land contain all areas of past gold production and the area of the currently estimated Mineral Resource. This area includes the historical Pinson Mine. OMC controls a 100% interest in the private lands that make up approximately 1,280 acres of the Property through outright ownership. Additionally, OMC controls a 100% interest in unpatented federal lode mining claims covering about 797 additional acres either by outright ownership or via mining lease agreement and owns an undivided 41.67% interest in private land and unpatented federal lode mining claims covering about 468 additional acres.

The Report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators (CSA).

All monetary values shown in the Report are in United States dollars ($).

1.2 Location and history

The Property is located in the Potosi mining district, 27 miles north-east (NE) of Winnemucca, within the south-eastern (SE) part of Humboldt County, Nevada. Access to the Property is provided by a combination of paved interstate and state highways, and well maintained, unpaved private roads. The towns of Winnemucca and Battle Mountain are located 35 miles by road to the south-west and 60 miles to the SE of the Property respectively.

The Property has had a protracted history of gold exploration and mining activities. Gold was initially discovered at the Property in the mid to late 1930's. Approximately 10,000 troy ounces (oz) of gold was produced from the Property between 1949 and 1950. A further 987 thousand ounces (koz) was produced from various open pit mining operations between 1980 and 1999.

Most recent mining on the Property was completed by former owner Atna Resources Ltd. (Atna) between 2012 and 2013 via an underground operation at the Property. Approximately 30,148 tons of ore containing 7,915 oz of gold were mined and shipped to off-site processing facilities during the course of operations.

OMC acquired the Property in May 2016 following a Chapter 11 bankruptcy filing by Atna.

In June 2020, the Companies signed a letter of intent with OMC to acquire the Getchell Project (formally the Pinson Project) from OMC.

1.3 Geology and mineralization

The Property is located on the eastern flank of the Osgood Mountains within the Basin and Range tectonic province of northern Nevada. The Pinson Mine occurs within a north-west (NW) trending structural corridor known as the Getchell gold trend. This trend also encompasses a number of gold deposits located outside the Property including the Preble, Getchell, Turquoise Ridge and Twin Creeks. These deposits are hosted in Paleozoic marine sedimentary rocks. Gold mineralization at the Property is described as a Carlin-type, sediment hosted system.

The Property geology comprises a sequence of Cambrian to Ordovician sedimentary rocks which form part of the Osgood Mountain Terrane and the Osgood Mountains. Much of the Property comprises shales, hornfelsed sedimentary rocks, and limestone interbeds of the Preble Formation and an overlying (or juxtaposed), alternating sequence of limestone, shale, dolomite with tuffaceous shale and intraformational conglomerates belonging to the Comus Formation. The Preble and Comus Formations have been folded into a broad north-plunging anticline and have been intruded by large Cretaceous granodiorite stocks, resulting in irregular contact metamorphism.

Gold mineralization at the Property is strongly structurally controlled, occurring at favorable sites within a fault network occurring around the eastern edge of the Osgood granodiorite, and predominantly within Comus Formation host rocks. Mineralization is commonly associated with the decalcification of carbonate rocks, and the introduction of silica, fine grained pyrite, arsenian pyrite, and remobilized carbon. Continuity of mineralization is highly variable ranging from 40 to 4,500 feet (ft) in strike extent, 250 to 1,800 ft in down-dip extent and 5 to 400 ft in thickness. The underground mineralization has a variable thickness between 5 ft to 30 ft.

Oxidation reaches depths of up to 1,800 ft within shear zones. Oxide mineralization includes pervasive limonite, hematite along with other iron and arsenic oxides. Historical production from the open pits was focused on the oxidized material.

Underground mineralization displays pervasive argillization and decalcification of host lithologies along with the formation of dissolution collapse breccias and intense shearing. Where the alteration is strongest, the altered zones consist of punky, spongy decalcified limestone in an argillically altered fine-grained, carbon-rich matrix (Gustavson Associates 2012). Silicification is minor and occurs as a broad overprint on the zone. Historical underground production included both sulphide and oxide material.

1.4 Data verification and quality assurance and quality control

The Property has been historically drilled using a combination of reverse circulation (RC) and diamond drilling. The majority of drilling was completed from surface. More recent drilling was completed as underground diamond core drilling. Sampling protocols adopted by former Property operators were similar and generally followed industry best practices of the time.

RC samples were collected from the drill cyclone in 5 ft intervals. Diamond core was sampled predominantly as 5 ft intervals however locally adjusted based on geological, alteration and oxidation contacts. RC and core recovery were recorded and considered to be excellent.

Samples were prepared and analyzed by a number of accredited laboratories throughout the Project history, including ALS Chemex, Inspectorate American Laboratories (IAL), and American Assay Laboratories (AAL).

Data validation has been completed by various operators throughout the Project's history. This process comprised the checking of original assay certificates and drillhole records against the digital database. This was completed most recently in April 2019 by OMC.

Quality assurance / quality control (QA/QC) samples including Certified Reference Materials (CRMs), coarse blanks and field duplicate samples were included regularly with samples submitted between 2005 and 2008. A limited number of CRMs were included with drilling completed in 2012.

The Qualified Person (QP) has reviewed available QA/QC data and noted a number of minor issues of concern with CRM precision and duplicate bias however does not consider these issues to be a material concern for a global, long-term Mineral Resource estimate. The QP however cannot guarantee that there are no material impacts on the local scale. Overall, the QP considers the assay database to be acceptable for Mineral Resource estimation.

The QP independently verified 4.9% of the assays in the Underground area, and a further 5.7% of assays in the Open Pit area. This verification was completed by randomly selecting assays from within mineralization wireframes for the various operators and laboratories used throughout the Project's history and comparing the results in the sample assay database against the original assay certificate. Where the assay certificate was not available, relevant original assay laboratory files or handwritten assay logs were consulted. A total of 0.1% and 3.3% of samples verified from the underground and Open Pit areas respectively were found to have errors.

In addition to this, collar locations were checked against the provided topography and it was found that many collars were either above or below topography. This in part is due to drilling taking place prior to mining. Collars with large discrepancies were reviewed and 110 collar surveys were updated.

The QP considers the assay database to be acceptable for Mineral Resource estimation.

1.5 Mineral processing and metallurgical testing

The Companies are evaluating the feasibility of processing material from their Pinson gold deposit in Nevada to produce saleable gold products. They are evaluating mining feed from the Mag and CX Open Pit mines and processing the mined material in a heap leach facility to produce gold bullion and mining feed from the Ogee underground mine and selling this material (in a toll treating arrangement) to a nearby autoclave facility to process the refractory gold associated with Ogee material.

Metallurgical testwork programs were conducted between 1999 and 2013 by metallurgical laboratories on behalf of the Homestake Mining Company (Homestake) (1999) and Atna (2005/6 and 2013/4).

The testwork on the Mag Pit and CX Pit open pit samples showed that:

- Many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. The QP believes this is a risk to gold recovery if it not treated correctly.
- Testwork on ground material showed that Mag Pit feed was amenable to carbon-in-leach (CIL) methods.
- Column leach tests on the Mag Pit samples achieved gold recoveries in the range of 19% to 82%. This variability is largely associated with the grade of total organic carbon (TOC) indicating a preg-robbing mechanism.
- Column leach tests on the CX Pit samples achieved gold recoveries of 82%.

The testwork on the Ogee underground samples showed that:

- Autoclave pre-treatment ahead of cyanide leach testwork demonstrated significant increases in gold recovery relative to baseline cyanide leach tests.

Based on available data, the QP considers that for the purpose of this NI 43-101, using heap leaching for the Mag and CX Open Pit material is reasonable, and it is reasonable to assume that gold recoveries between 48% to 82% for the Mag Pit and 82% for the CX Pit are achievable. The QP also considers that using autoclave pre-treatment on the underground Ogee material is reasonable, it is reasonable to assume that gold recoveries between 78% to 95% are achievable. High grade ore extracted from the Ogee deposit between 2012 and 2013 was historically trucked eight miles to Newmont Mining Corporation's Twin Creeks autoclave facility for processing to produce gold bullion. Gold recoveries from the autoclave processing ranged from 69.2% to 92.6%.

These gold recoveries have been used to derive cut-off grades (COG) for Mineral Resource reporting.

1.6 Mineral Resources

The Mineral Resources for the Pinson deposit have been estimated by Ms Dinara Nussipakynova, P.Geo., of AMC. Ms Nussipakynova is a QP under NI 43-101 and takes responsibility for the Mineral Resource estimates.

The estimated Mineral Resource at Pinson is divided into two parts. One part is proximal to the underground mine. It is referred to as the "Underground area". The other resource area, referred to as the "Open Pit area" is beneath the historical open pits. As the style and grade of mineralization are different for these two areas they are treated as separate deposits. Table 1.1 shows the Mineral Resource estimate for the Underground area. Table 1.2 shows the Mineral Resource estimate for the Open Pit area. These tables report the full Mineral Resource on the Property regardless of the Companies ownership percentage. There are no Mineral Reserves stated at present. The Mineral Resources have been depleted for previous mining.

Table 1.1 Summary of the Underground area Mineral Resource as of 23 July 2020

Classification	Tons (ktons)	Au (opt)	Metal Au (koz)
Measured	184	0.289	53
Indicated	436	0.313	136
Measured and Indicated	**620**	**0.306**	**190**
Inferred	1,676	0.347	581

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- The Mineral Resource COG is based on a metal price of $1,550/oz Au. (cost and other assumptions shown in Table 14.13).
- Underground Mineral Resources as stated are constrained within modeled underground stope shapes using a nominal 15' minimum thickness, above a gold cut-off grade of 0.15 opt Au.
- Drilling results up to 31 December 2015.
- Drilling database provided 18 April 2019.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The numbers may not compute exactly due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 1.2 Summary of Open Pit area Mineral Resource as of 23 July 2020

Classification	Tons (ktons)	Au (opt)	Metal Au (koz)
Measured	10,726	0.068	730
Indicated	11,829	0.046	545
Measured and Indicated	**22,554**	**0.057**	**1,275**
Inferred	1,388	0.047	65

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- Mineral Resources are constrained by an optimized pit shell developed at a metal price of $1,550/oz Au (cost and other assumptions shown in Table 14.31).
- Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone COG is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

The Pinson Underground area was estimated using 117 mineralization domains constructed in Datamine. Samples within mineralization domains were composited to 10 ft lengths and reviewed using probability plots for the existence of outliers. As there were no outliers, capping was not applied. Mineralized domains were grouped into 16 sets based on similar orientations. Gold grades were interpolated using inverse distance squared (ID^2) using a three-pass search.

An indicator method was used to model mineralization in the Open Pit area. This process comprised creating a broad mineralization envelope at each pit area. Drillhole samples were composited to 10 ft and indicator values were then defined for gold grades above and below 0.1 gram per metric tonne (g/t) (0.03 opt). Experimental variograms were then calculated and modelled for indicator data. Ordinary kriging (OK) was used to interpolate indicators in all but a single zone, where ID^2 was used. A 0.3 probability was chosen to separate high and low-grade domains. Data within each zone was reviewed for high grade outliers and capped as appropriate. Gold grades within the high and low-grade models were interpolated primarily using OK in three passes. ID^2 was used where variograms could not be calculated due to insufficient data.

Mineral Resources were classified using an assessment of geological and mineralization continuity, data quality and data density. Wireframes were constructed to code Measured, Indicated, and Inferred Resources into the block model.

Mineral Resources are reported at a COG of 0.15 opt Au for the Underground area and two COG of 0.007 opt Au and 0.014 opt for open pit mining methods. The Company provided the initial COG calculations and the QP verified the reasonableness of the assumptions. The COG is based on actual and benchmark cost data for similar scale of operations and assumptions regarding mineral processing metal recoveries and metal prices. The COG includes all mining, processing and General and Administration (G&A) costs and a gold price of $1,550/oz. A gold metallurgical recovery of 90% was used in establishing the underground COG. The open pit block model was coded with areas of low and high recovery zones with recoveries of 40% and 80% used respectively in establishing the open pit COG. Varying royalties are applied at varying trigger points throughout the mine life, but for simplicity a constant 6% royalty was used for the calculation of COG.

1.7 Interpretation and conclusions

Gold mineralization at the Property comprises two main areas; the Underground and Open Pit areas. Both areas are sites of past production. The Mineral Resource estimates described in the report were prepared using Datamine software. They have been estimated by Ms Dinara Nussipakynova, P.Geo., of AMC, who takes responsibility for these estimates.

Using a 0.15 opt gold COG, Measured and Indicated Underground Resources are estimated at 620,000 tons grading 0.306 opt gold; and Inferred Mineral Resources are estimated at 1,676,000 tons grading 0.347 opt gold. The Underground area Mineral Resources are constrained within modeled underground stope shapes.

Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone COG is 0.007 opt Au. Measured and Indicated Open Pit area Resources are estimated at 22,554,000 tons grading 0.057 opt gold; and Inferred Mineral Resources are estimated at 1,388,000 tons grading 0.047 opt gold. The Open Pit area Mineral Resources were pit-constrained.

The metal price used in determining COGs for the Mineral Resources is $1,550/oz Au. A gold metallurgical recovery of 90% was used in establishing the underground COG. A metallurgical recovery of 40% was used in establishing the open pit COG for the low recovery zone and 80% was used for the high recovery zone.

The Property is subject to a number of royalty obligations.

Numerous data validation campaigns have been undertaken on the Property.

Drilling programs completed at the Property between 2005 and 2015 have included QA/QC monitoring programs which have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams. Some concerns have been highlighted, but the QP does not consider these issues to be material to the global, long term Mineral Resource estimate.

The Companies are presently in the process of reviewing potential options to mine material contained within the Mag and CX Open Pit areas and process this material as a heap leach operation; and to mine Underground Mineral Resources at Ogee and process material at a nearby autoclave facility via a toll treatment arrangement. Based on available data, the QP considers these approaches to be reasonable. Some concerns and gaps in the metallurgical information have been identified and recommendations made to address these. Gold recoveries between 48% to 82% for Mag Pit and 82% for the CX Pit are considered achievable using heap leach. Gold recoveries between 78% to 95% are also considered achievable using an autoclave for the refractory gold associated with the Ogee material.

1.8 Risk

1.8.1 Geological risk

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is a degree of uncertainty attributable to the estimation of Mineral Resources. Until resources are actually mined and processed, the quantity of mineralization and grades must be considered as estimates only. Any material change in quantity of Mineral Resources, mineralization, or grade may affect the economic viability of the project.
- The Mineral Resource estimate was not based on oxidation information. Collection and inclusion of oxidation data and other parameters that would support the determination of processing options could materially impact the COGs.
- Data used to inform the block model is historical in nature. Verification of the source of original data is challenging due to incomplete records. The past production on the Property mitigates some of this risk. Continued efforts should be made to verify the historical data.
- QA/QC monitoring programs have only been completed on the Property between 2005 and 2015. Insertion rates were low, CRMs showed poor precision and duplicate samples showed suboptimal performance. Despite the concerns highlighted above, the QP does not consider

these issues to be material to the global, long term Mineral Resource estimate. The QP however cannot guarantee that there are no material impacts on the local scale.

- The number of bulk density measurements used in the block model is limited (153). Additional sampling may result in minor changes to the density and may affect the tonnage.

1.8.2 Metallurgical risk

- Metallurgical samples do not represent the grade variability of the deposit and test work should be undertaken on samples that represent the low- and high-grade variation of the mineralization. The lack of information on metallurgical performance of such samples remains a risk to the project.
- Deleterious elements (arsenic and mercury) are present in some zones at grades high enough to be a risk to the project. Additional test work on the deportment and fate of these elements is required to define the processes necessary to mitigate their impacts.
- Sample representativity should be improved. Metallurgical sampling has been localized to relatively small portions of the Mineral Resource. The metallurgical response of the samples is likely to represent the general behaviour of the zone, but sampling of at least one other area of each zone to confirm the metallurgical response will reduce uncertainty. Confirmatory testwork on targeted drilled samples is recommended to mitigate the risk.
- Many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. The QP believes this is a risk to gold recovery if it is not treated correctly.

1.9 Opportunities

1.9.1 Geological opportunities

The Pinson Mineral Resource presently excludes several zones of relatively continuous mineralization which were solely defined by drillhole assays that could not be supported by original certificates. Verification of assays in this region, or additional drilling to confirm these results may provide sufficient justification to classify Mineral Resources in these areas.

1.9.2 Metallurgical opportunities

- By developing a geometallurgical model of each of the underground and open pit resources, it is possible to optimize the choice of processing / recovery options.
  - Selective diversion of refractory material to stockpile for toll treatment and non-refractory material to conventional leaching.
  - Selective diversion of preg-robbing feed (open pit Mineral Resource) to appropriate processing to improve recovery.
- Examine flotation of underground feed to reduce the mass of material to an autoclave circuit. The flotation concentrates with high sulphur and gold grades should reduce operating costs and increase throughput through the autoclave.
- Trial roasting as an alternative to autoclave pre-treatment (ahead of cyanide leach) as a method of treating refractory gold in Ogee feed. This takes advantage of the proximity of sulphide roaster facilities in the region. Roasters could also be used to treat carbonaceous material so that preg-robbing issues would be prevented.
- Maximize the potential value of the resource by completing a techno-economic trade-off study looking at the roaster and autoclave options. This study should examine the demand for Pinson material from local roasters and autoclave facilities.

1.10 Recommendations for further work

The QPs make the following recommendations:

1.10.1 Overall project recommendation

A selectively assigned delineation core drilling program of 5,000 feet ($500k) is recommended in Indicated & Inferred areas of mineralization to support de-risking of the existing Open Pit Mineral Resources. A Phase 1 exploration drill program of 35,000 feet ($3.5M) utilizing RC-holes and core tails is recommended at Pinson underground to test areas of highest potential and provide a basis for preliminary development planning. A Phase 2 program ($8M) of underground development and delineation drilling, designed to delineate positive results from the Phase 1 program and further confirm existing Measured and Indicated Mineral Resources, would follow thereafter. The scale of Phase 2 is dependent on Phase 1 results.

Additional detailed recommendation by Section is given below. The total cost of the programs below is $0.35M.

1.10.2 Sample preparation, analyses, and security

1.10.2.1 Data validation

- Complete additional clean-up work on the Datashed database.

1.10.2.2 CRMs

- Purchase additional CRMs at the approximate COGs, average grades, and higher grades of the deposits.
- Include CRMs in every batch of samples submitted at a rate of at least 1 in every 20 samples (5%).
- Ensure that CRMs are monitored in real time on a batch by batch basis, and that remedial action is taken immediately as issues are identified.
- Ensure CRM warnings, failures and remedial action is documented.
- If pulps are available in areas relevant to the current Mineral Resource, the QP recommends that an investigation into analytical precision be completed. This would comprise selecting a number of mineralized intervals associated with poor performing CRMs and completing reanalysis of two separate sub-samples from each pulp using an umpire laboratory. CRMs should be included in this submission. Differences between the grades of the new pulp assays will allow assessment of subsampling variance and geological variance. Differences to the original samples may provide insight into the precision of the original laboratory.

1.10.2.3 Blanks

- The QP recommends that both coarse and pulp blanks are included in future exploration programs. Blank material should be analyzed prior to inclusion in QA/QC programs to ensure the material is below the appropriate analytical detection.
- The QP recommends that fine and coarse blank material be included in each batch. The weight of individual blank samples included in the sample stream should be consistent. Blank samples should comprise 5% of the total sample stream. Blank material should be included after recognized high grade samples.

1.10.2.4 Duplicates

- Field Duplicates, coarse duplicates and pulp duplicates should be regularly inserted into the sample stream.
- The QP recommends that further investigative work be completed to assess duplicate performance and sample bias.

1.10.2.5 Umpire samples / duplicates

- The QP recommends that if historical pulps are available in the areas of the current Mineral Resource, that umpire sampling be completed. Umpire samples should comprise 5% of total samples originally submitted.

1.10.3 Data verification

- Drillhole collars be re-surveyed if they can still be located on the ground.
- Missing original assay certificates, downhole survey logs, original geology, and alteration logs, as well as additional records on the density, should be located if possible.

1.10.4 Mineral processing

- Future testwork programs should be completed on a number of samples that represent the deposit's spatial variability of weathering profile, lithology, and gold grade, and that represent run-of-mine feed from progressive stages of the project.
- Conduct quantitative mineralogy (e.g., QEMScan) on selected samples that represent run-of-mine feed from progressive stages of the project.
- Complete additional autoclave pre-treatment testwork on Ogee samples.
- Conduct comminution testwork on both underground and open pit samples.
- Conduct roaster pre-treatment testwork on Ogee samples, given the proximity of sulphide roaster facilities in the region. The roasting testwork could be trialed as an alternative to autoclave pre-treatment and can be used to treat carbonaceous material.
- Complete flotation testwork ahead of autoclave pre-treatment testwork to produce flotation concentrates with high sulphur and gold grades.
- Test the deportment of arsenic and mercury in the processing of the feed material. This program should cover the CIL, heap leach, and pre-oxidation processes tested during the past test work program.
- Conduct additional column leach testwork on open pit samples. This testwork should be completed at varying crush sizes to determine the optimum crush size.
- Complete additional CIL testwork on open pit material.
- Test alternative options for dealing with the carbonaceous preg-robbing material:
  - Completing resin-in-leach testwork as an alternative to activated carbon.
  - Completing testwork where blinding agents such as kerosene are added to the bottle roll tests.
- Develop a geometallurgical block model for the Pinson material. This model should also include a financial model that determines the most economically viable process route for all blocks in the block model.

1.10.4.1 Geometallurgy

A geometallurgical block model should be developed for the Pinson material. This model should incorporate both Open Pit and Underground areas and include key inputs such as chemical assays (including gold, sulphur speciation, and carbon speciation), mineralogy and testwork parameters. This model would develop relationships between key parameters such as gold grade, sulphide grade, carbon grade and gold recovery. This model should also include a financial model that determines the most economically viable process route for all blocks in the block model. This financial model should include inputs such as gold price, gold grade, tested gold recovery, operating costs, and expected revenue from toll treatment. The model should also account for the capacity of the various process units (heap leach and autoclave) to avoid creating process bottlenecks.

1.10.5 Mineral Resource estimates

- Drillholes should be re-evaluated / re-logged for oxidation to allow for the criteria to be coded into future block model estimations.
- Additional bulk density samples be taken in future drilling campaigns every 30 ft.
- Future updates of the block model include oxidation and other parameters that would support the determination of processing options. This will allow the Mineral Resources to be more accurately reported out with different COGs.

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Abbreviations and acronyms

Abbreviation & acronyms	Description
$	United States dollar
%	Percentage
°	Degree
°C	Degrees Celsius
°F	Degrees Fahrenheit
µm	Micrometre
3D	Three-dimensional
AA	Atomic absorption
AAL	American Assay Laboratories
AAS	Atomic absorption spectroscopy
ACQ	AcQuire
ALS	ALS Laboratories
AMC	AMC Mining Consultants (Canada) Ltd.
amsl	Above mean sea level
As	Arsenic
Atna	Atna Resources Ltd.
Au	Gold
Barrick	Barrick Gold Corporation
BLM	Bureau Land Management
C	Carbon
CIL	Carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CO_2	Carbon dioxide
Coeff. of var.	Coefficient of variation
COG	Cut-off grade
Cordex	Cordex Syndicate
CPI	Consumer Price Index
CPm	Cambrian Middle Preble Formation
CPu	Cambrian Upper Preble Formation
CRM	Certified Reference Material
CSA	Canadian Securities Administrators
CSAMT	Controlled Source Audio-frequency Magneto Tellurics
Cu	Copper
E	East
EM	Electromagnetics
ENE	East-north-east
ESE	East-south-east
EW	East-west
Fm	Formation
ft	Feet
ft^3	Cubic feet
g	Gram
g/L	Gram per litre
g/t	Gram per metric tonne
G&A	General and Administration
gpm/ft^2	Gallons per minute / squared feet
H_2SO_4	Sulphuric acid
Hg	Mercury

Abbreviation & acronyms	Description
HNO_3	Nitric acid
Homestake	Homestake Mining Company
i-80	i-80 Gold Corp.
IAL	Inspectorate American Laboratories
ICPAES	Induced Coupled Plasma Atomic Emission Spectroscopy
ID^2	Inverse distance squared
IP	Induced Polarization
KGD	Cretaceous Granodiorite
km	Kilometre
koz	Thousand ounces
ktons	Kilotonnes
L	Litre
lab	Laboratory
lb	Pound
M	Million
m	Metre
m^3	Cubic metre
Ma	Million years / mega annums
Maxwell	Maxwell Resources
Mg/L	Milligram per litre
mm	Millimetre
MSHA	Mine Safety and Health Administration
MT	Magneto Tellurics
N	North
NaCN	Sodium cyanide
NaOH	Sodium hydroxide
NE	North-east
NGM	Nevada Gold Mines LLC
NI 43-101	National Instrument 43-101
NNE	North-north-east
NNW	North-north-west
Nsamples	Number of samples
NSR	Net Smelter Return
NW	North-west
OCL	Ordovician Lower Comus Formation
OCU	Ordovician Upper Comus Formation
OK	Ordinary kriging
OMC	Osgood Mining Company LLC
opt	Troy ounce per short ton
Ov	Ordovician Valmy
oz	Troy ounce
oz/ton	Troy ounces per short ton
P_{80}	80% Passing
pH	pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution
PMC	Pinson Mining Company
ppb	Parts per billion
ppm	Parts per million
Property	Pinson Property
psi	Pound per square inch

Abbreviation & acronyms	Description
QA/QC	Quality assurance / quality control
QAL	Quaternary alluvium
QP	Qualified Person as defined by NI 43-101
RC	Reverse circulation drilling
Report	Technical Report
RFF	Range Front Fault
RFZ	Range Front Zone
RPD	Relative paired difference
RQD	Rock quality determination
SE	South-east
$sh.t/ft^3$	Short ton per cubic foot
SMD	Small Mine Development
SSE	South-south-east
Standdev	Standard deviation
SW	South-west
T	Tonne
t	Short ton
t/m^3	Tonne per cubic metre
TOC	Total organic carbon
ton	Short ton
UG	Underground
USA	United States of America
w/w	Weight for weight
WNW	West-northwest

2 Introduction

2.1 Purpose

This Technical Report (Report) provides an update of the Mineral Resource estimate and metallurgy of the Mineral Resources identified within the Getchell Property (Property or Project) located in Humboldt County, Nevada, USA. The Report has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada on behalf of Premier Gold Mines Limited / i-80 Gold Corp (i-80 or the Companies). Through the report this abbreviation will be used when referring to the issuer. On 10 August 2020, the Companies entered into a definitive purchase agreement with affiliates of Waterton Global Resource Management, Inc. (Waterton) to acquire from Waterton all of the outstanding membership interests of Osgood Mining Company LLC (OMC).

The Property comprises a number of property parcels which collectively encompass 2,545 acres of land covering the historic Pinson Project. This land package includes a controlled 100% interest in approximately 1,280 acres of private land. Additionally, OMC controls a 100% interest in unpatented federal lode mining claims covering about 797 additional acres either by outright ownership or via mining lease agreement and owns an undivided 41.67% interest in private land and unpatented federal lode mining claims covering about 468 additional acres. The Property is subject to a number of royalty obligations.

This report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilizes the definitions and categories of Mineral Resources and Mineral Reserves as set out in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines (2014) (CIM Standards).

2.2 Terms of reference

In 2020, the Companies commissioned AMC to prepare an updated Technical Report on the Property. This report includes a review of mineral processing and metallurgical testing and an independent estimate of the Mineral Resources of the Property. The Mineral Resource estimate is the basis for this report. The Mineral Resource estimate was prepared by D. Nussipakynova, P.Geo. (BC and ON).

The estimated Mineral Resource on the Property is divided into two parts. One part, referred to as the "Open Pit Area" is beneath the historical open pits. The other resource area is proximal to the Underground mine. It is referred to the "Underground Area". As the style and grade of mineralization are different for these two areas they are treated as separate deposits.

Projected risks and opportunities associated with the Project were compiled together with a list of recommendations for further Project development activities, including ongoing data verification of the historical drilling, logging of oxidation and other parameters that would allow for a refined block model and additional metallurgical test work.

The Mineral Resources were estimated in the local mine grid. Conversions are listed in Section 24.

A list of abbreviation and acronyms is provided after the table of contents.

2.3 Sources of information

This Report has been prepared by AMC for the Companies. The information, conclusions, opinions, and estimates contained herein, for which the named Qualified Persons (QPs) take responsibility, are based on:

- Information available at the time of preparation of this report.
- Assumptions, conditions, and qualifications as set forth in this report.
- Data, reports, and other information supplied by the Companies, OMC and from other sources.

Key sources of information include the diamond drillhole database and metallurgical test work reports. A full reference list is included at the end of the Report. The most recent report often referred to is: Report on the Pinson Project Preliminary Feasibility Study in Humboldt County, Nevada" dated 17 October 2014 (Golder 2014).

2.4 Qualified Persons

A listing of the authors of the Report, together with the sections for which they are responsible, is shown in Table 2.1.

Table 2.1 Persons who prepared or contributed to this Technical Report

<table>
<tr><th colspan="7">Qualified Persons responsible for the preparation of this Technical Report</th></tr>
<tr><th>Qualified Person</th><th>Position</th><th>Employer</th><th>Independent of OMC</th><th>Date of last site visit</th><th>Professional designation</th><th>Sections of Report</th></tr>
<tr><td>Ms D Nussipakynova</td><td>Principal Geologist</td><td>AMC Mining Consultants (Canada) Ltd.</td><td>Yes</td><td>19-21 March 2019</td><td>P.Geo. (BC, ON)</td><td>1 (part), 2 - 12, 14 - 16, 18 - 24, 25 (part), 26 (part), 27 (part)</td></tr>
<tr><td>Dr P Greenhill</td><td>Principal Consultant</td><td>AMC Consultants Pty Ltd</td><td>Yes</td><td>No</td><td>FAusIMM (CP)</td><td>1 (part), 13, 17, 25 (part), 26 (part), 27 (part)</td></tr>
<tr><th colspan="7">Other Experts who assisted the Qualified Persons in the preparation of this Technical Report</th></tr>
<tr><th>Expert</th><th>Position</th><th colspan="2">Employer</th><th>Independent of OMC</th><th>Visited site</th><th>Sections of Report</th></tr>
<tr><td>Dr A Ross</td><td>Geology Manager / Principal Geologist</td><td colspan="2">AMC Mining Consultants (Canada) Ltd.</td><td>Yes</td><td>No visit</td><td>Overall report</td></tr>
<tr><td>Mr S Robinson</td><td>Senior Geologist</td><td colspan="2">AMC Mining Consultants (Canada) Ltd.</td><td>Yes</td><td>No visit</td><td>Parts of Section 11</td></tr>
<tr><td>Mr W Schleiss</td><td>Technical Support Geologist</td><td colspan="2">Elko Mining Group LLC</td><td>No</td><td>Yes</td><td>6, 7, 9, 10 (part), 11 (part)</td></tr>
<tr><td>Mr K Fowlow</td><td>Senior Geologist</td><td colspan="2">Elko Mining Group LLC</td><td>No</td><td>Yes</td><td>10 (part), 11 (part)</td></tr>
<tr><td>Mr W Oakley</td><td>Geology Manager</td><td colspan="2">Elko Mining Group LLC</td><td>No</td><td>Yes</td><td>6 - 11 (part)</td></tr>
<tr><td>Mr B May</td><td>Senior Geologist</td><td colspan="2">Elko Mining Group LLC</td><td>No</td><td>Yes</td><td>8</td></tr>
<tr><td>Mr J Currie</td><td>Manager, Exploration</td><td colspan="2">Waterton Global Resource Management, Inc.</td><td>No</td><td>Yes</td><td>Overall report</td></tr>
</table>

Note: ** QP responsibility for 'part' sections is governed by their respective areas of expertise: Ms D Nussipakynova–Geology and Mineral Resource aspects; Dr P Greenhill–Metallurgical aspects.

NI 43-101 requires at least one Qualified Person (QP) to inspect the Property. As the estimator of the Mineral Resources, Ms Dinara Nussipakynova visited the Property, a site visit by Dr Paul Greenhill was not deemed necessary.

The Companies have been provided with a draft of this Report to review for factual content.

This Report is effective as of 23 July 2020.

2.5 Units of measure and currency

Throughout this Report, measurements are in imperial units and currency is in United States dollars ($) unless otherwise stated.

3 Reliance on other experts

The QPs have relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Report:

- The following disclosure is made in respect of this Expert: Daniel A. Jensen, Shareholder, Parr Brown Gee & Loveless, a Professional Corporation, as advised in a letter of 23 July 2020 to AMC.
- Report, opinion, or statement relied upon information on mineral tenure and status, title issues, royalties, and mining concessions.
- Extent of reliance: full reliance following a review by the QPs.
- Portion of Report to which disclaimer applies: Section 4.2.

4 Property description and location

4.1 Property description and location

The Property is located 27 miles north-east (NE) of Winnemucca, Nevada, in south-eastern (SE) Humboldt County (Figure 4.1). The Project site is 35 miles from Winnemucca by road and is 60 road miles north-west of Battle Mountain, Nevada. The Project area encompasses approximately 2,545 acres in the Potosi mining district, surrounding and including the existing Pinson Mine. The geographic centre of the Property is located at UTM 478,294E and 4,553,515N (NAD27, Zone 11 m).

Figure 4.1 Pinson Project location map

4.2 Ownership, mineral rights, and tenure

4.2.1 Overview

In May 2016, OMC acquired the Property from Atna after Atna filed for Chapter 11 bankruptcy in November of 2015. Atna had acquired its interest in the Property through a series of transactions with Pinson Mining Company (PMC), an affiliate of Barrick Gold Corporation (Barrick), that culminated with Atna negotiating and closing the purchase of all of the interests in the core, four square miles of the Property (Sections 28, 29, 32, and 33, Township 38 North, Range 42 East) in September 2011. The four-square miles of land (roughly 2,545 acres) contain all areas of past gold production and the area of the currently estimated Mineral Resource. OMC now controls the Property.

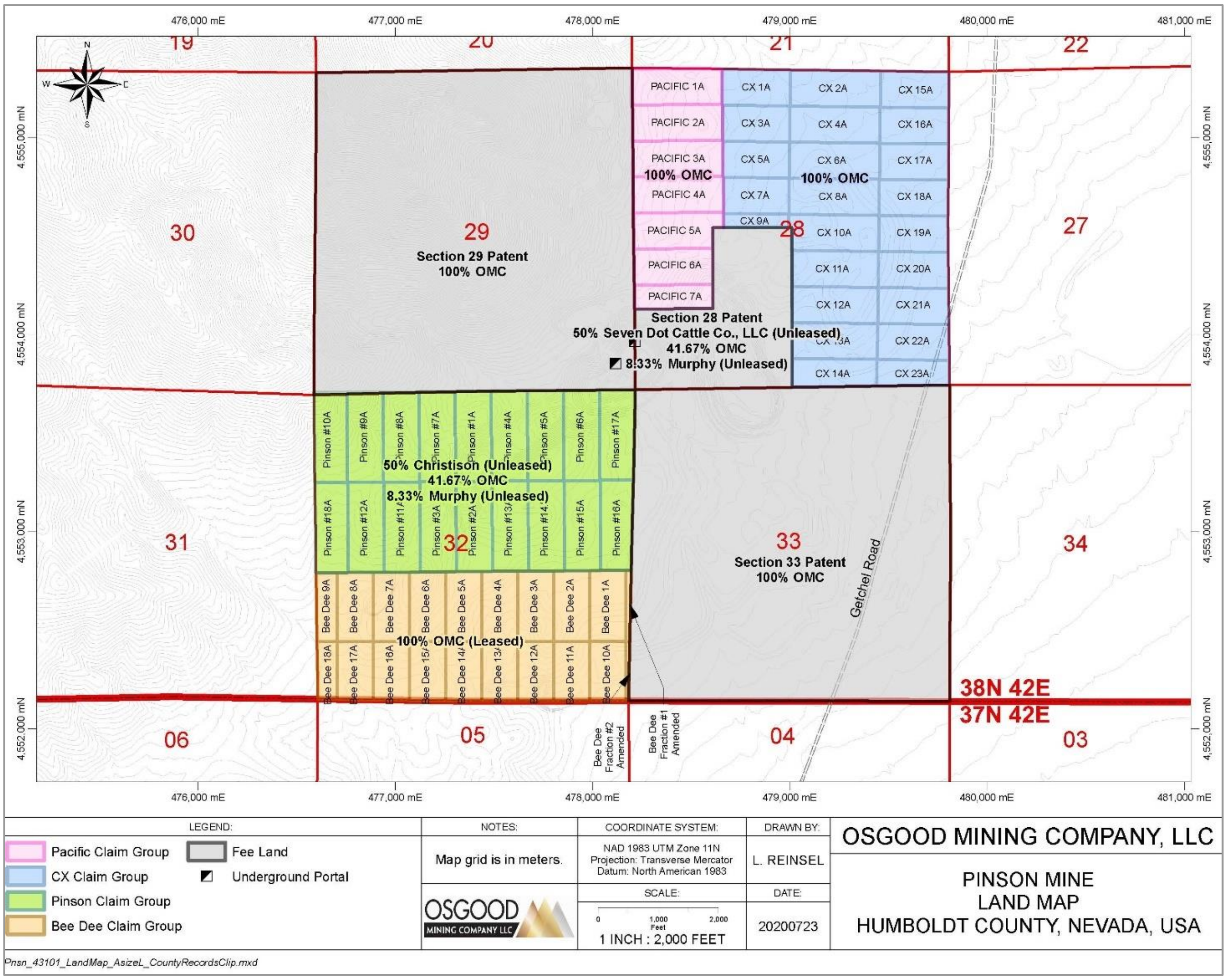
The Property is made up of a number of Property parcels that are either wholly owned by OMC, fractionally owned by OMC (a joint undivided fractional interest), or under lease by OMC.

Approximately 2,545 acres of fee simple (private) land, unpatented federal lode mining claims, and a lease make up the land package controlled by OMC, and the Mineral Resources estimated within this Report. OMC controls a 100% interest in the private lands that make up approximately 1,280 acres of the Property through outright ownership. Additionally, OMC controls a 100% interest in unpatented federal lode mining claims covering about 797 additional acres either by outright ownership or via mining lease agreement and owns an undivided 41.67% interest in private land and unpatented federal lode mining claims covering about 468 additional acres.

4.2.2 Unpatented federal lode mining claims

OMC owns or controls 50 mining claims covering portions of Sections 28 and 32, Township 38 North, Range 42 East. Additionally, OMC owns an undivided 41.67% interest in another 18 mining claims covering part of Section 32, Township 38 North, Range 42 East. Federal holding costs for the unpatented mining claims for 2020 (on a net ownership basis) will be approximately $9,487 in 2020. Land holdings are shown in Figure 4.2.

Figure 4.2 Pinson Property and mining claim map



4.2.2.1 Pacific unpatented federal lode mining claims

OMC owns a 100% interest in the Pacific #1A-7A mining claims located in Section 28, Township 38 North, Range 42 East (see Figure 4.2). These claims were initially staked by the Cordilleran Explorations partnership and are subject to the Royal Gold Royalty, the Cordilleran Royalty, and the PMC Royalty described below.

4.2.2.2 CX unpatented federal lode mining claims

OMC owns a 100% interest in the CX #1A-23A claims located in Section 28, Township 38 North, Range 42 East (see Figure 4.2). These claims were initially staked by PMC and are subject to the Royal Gold Royalty and the PMC Royalty described below.

4.2.2.3 BEE DEE unpatented federal lode mining claims

OMC controls a 100% interest in the BEE DEE group of claims (20 claims) through a Mining Lease Agreement with Franco-Nevada U.S. Corporation (50%) and S&G Pinson, LLC (50%) as the current lessors (the BEE DEE Lease Agreement). These claims are located in Section 32, Township 38 North, Range 42 East (see Figure 4.2). These claims are subject to a leasehold royalty payable to the lessors pursuant to the BEE DEE Lease Agreement, as well as the Royal Gold Royalty and the PMC Royalty described below.

4.2.2.4 Pinson unpatented federal lode mining claims

OMC owns an undivided 41.67% interest in the Pinson #1A-18A mining claims located in Section 32, Township 38 North, Range 42 East (see Figure 4.2). The remaining 58.34% interest in these claims is owned by Diana Sue Christison (16.67%), James Christison (16.67%), Victor Christison (16.67%), and Michael Murphy (8.33%), and is not leased by OMC. The fact that OMC has not leased the unowned 58.34% interest in these claims does not preclude OMC from mining the claims. By law, OMC, as the co-owner of an undivided interest in these claims, has the right to mine the claims without permission or approval from (and even over any objections by) the other co-owners, subject, however, to an obligation on the part of OMC to account to the other co-owners for their proportionate shares of mining revenues less their proportionate shares of mining expenses. These claims are subject to the Royal Gold Royalty and the PMC Royalty described below and are also subject to a royalty initially held by Kate Murphy et al. as described in Table 4.1.

4.2.3 Fee lands

OMC owns a 100% interest in Sections 29 and 33, Township 38 North, Range 42 East. Section 29 is subject to the Royal Gold Royalty, the Cordilleran Royalty and the PMC Royalty described below. Section 33 is subject to the Royal Gold Royalty, the PMC Royalty, the Goldfield Royalty, and the Conoco Royalty described below.

OMC also owns an undivided 41.67% interest in the 120-acre parcel comprising the east ½ of the south-west (SW) ¼ and SE ¼ of the south-west ¼ of Section 28, Township 38 North, Range 42 East. The remaining interest in this parcel is co-owned by Seven Dot Cattle Co., LLC (50% undivided interest) and Michael Murphy (8.33% undivided interest). This parcel is subject to the Royal Gold Royalty and the PMC Royalty described below, as well as a royalty tied to PMC's purchase of this land as described in Table 4.1.

4.2.4 Underlying agreements – unpatented federal mining claims

OMC controls a 100% interest in the 20 BEE DEE unpatented federal lode mining claims by way of the BEE DEE Lease Agreement. The BEE DEE Lease Agreement provides for monthly minimum advance royalty payments to the lessors (currently Franco-Nevada U.S. Corporation (50%) and S&G Pinson, LLC (50%), which minimum advance royalty payments currently total $35,232.96 per year (subject to increases or decreases in accordance with the Consumer Price Index (CPI)). OMC is also required under the BEE DEE Lease Agreement to maintain the leased claims with the Bureau of Land Management (BLM) and Humboldt County, Nevada. The BEE DEE Lease Agreement expires 9 May 2040.

The BEE DEE Lease Agreement imposes a two percent (2%) net mint or smelter returns (NSR) royalty on the BEE DEE claims in favor of the lessors.

4.2.5 Underlying agreements – fee lands

As explained in Section 4.2.3, OMC owns an undivided 41.67% interest in a 120-acre patented fee land parcel in the south-west quarter of Section 28, Township 38 North, Range 42 East. The remaining undivided 58.33% interest in that parcel is not leased by OMC. As noted above with respect to the Pinson unpatented mining claims (which are only partially owned by OMC), the fact that OMC does not own or lease the outstanding 58.33% interest in this land does not preclude OMC from mining the land. By law, OMC, as the co-owner of an undivided interest in the land, has the right to mine the land without permission or approval from (and even over any objections by) the other co-owners, subject, however, to an obligation on the part of OMC to account to the other co-owners for their proportionate shares of mining revenues less their proportionate shares of mining expenses. OMC's right to mine this parcel is subject to a 5/12 of two percent NSR royalty resulting from a Deed dated 8 September 2001 from Kate M. Murphy as grantor, a Deed dated

17 September 2001 from Barbara P. Noceto as grantor, and a Deed dated 24 December 2002 from Patricia B. Phillips as grantor, all to OMC's predecessor in title.

4.2.6 Underlying agreements – royalty agreements

The Property is subject to several royalties. The following section, as summarized in Table 4.1, describes the royalties present on the various properties.

4.2.6.1 Royal Gold Royalty (Royal Gold, Inc. – current owner)

In a NSR Royalty Agreement dated 30 November 1996, PMC agreed to pay Rayrock Mines, Inc. et al. (now Royal Gold, Inc. and D. M. Duncan, Inc.) an overriding NSR royalty that varies depending on the nature of the particular land holding and any underlying royalties existing on that land at the time of the transaction (the Royal Gold Royalty). The Royal Gold Royalty applies to all lands controlled by OMC and the subject of this Report, but it is not payable until 200,000 troy ounces (oz) of gold have been produced. Currently the Royal Gold Royalty would commence after production of approximately 90,000 additional oz of gold from the Property.

For example, on fee lands now owned by OMC, the Royal Gold Royalty holders receive a 2.5% royalty on parcels not subject to an underlying royalty and a 0.5% royalty on parcels subject to a royalty which increases to a 1% NSR royalty if the average gross value per ton of ore produced is greater than $175/ton.

On fee lands leased by OMC (of which there are none at present) and subject to royalties payable to a third-party, the Royal Gold Royalty varies from a minimum of 0.5% to a maximum of 5% depending upon the underlying royalty. The royalty percentage is determined by the difference between a total royalty load of 6% less the underlying royalty; however, the royalty will never exceed 5% or be reduced to less than 0.5%. For example, if the underlying royalty is 4%, then the Royal Gold Royalty would be 6% less 4%, resulting in a 2% royalty payable to the holders of the Royal Gold Royalty. If the underlying royalty is 0.5%, the Royal Gold Royalty would be 6% less 0.5% equalling 5.5%, which is greater than 5%, thus reducing the applicable royalty rate to 5%. If the underlying royalty is 6% or greater, the Royal Gold Royalty rate is limited to 0.5%.

On unpatented lode mining claims not subject to underlying third-party agreements with retained royalties, the Royal Gold Royalty is 2.5%. If the unpatented mining claims have underlying retained royalties, then the royalty percentage is determined as described above under patented lands leased by OMC and subject to an underlying royalty with a maximum of 5% and a minimum of 0.5% dependent upon the underlying royalty load.

Table 4.1 lists the applicable Royal Gold Royalty rates for the various parts of the Property.

Table 4.1 Summary of royalties related to the Property

Section	Property or agreement name	Royalty owner(s)	From %	To %	Remarks
28	**Fee Land**				
	PMC purchase	Successors of Kate Murphy, et al.	2	2	Current royalty rate is 5/12 of 2% (NSR)
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	4% NSR split between Royal Gold (3.9158%) and Duncan (0.0842%)
	PMC Royalty	NGM	10	10	Net profits
28	**Pacific Mining Claims**				
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	1% NSR split between Royal Gold (0.97895%) and Duncan (0.02105%)
	Cordilleran Royalty	Royal Gold	5	5	NSR
	PMC Royalty	NGM	10	10	Net profits
28	**CX Mining Claims**				
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	2.5% NSR split between Royal Gold (2.447375%) and Duncan (0.0526275%)
	PMC Royalty	NGM	10	10	Net profits
29	**Fee Land**				
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	3% NSR split between Royal Gold (2.93685%) and Duncan (0.06315%)
	Cordilleran Royalty	Royal Gold	3	3	NSR
	PMC Royalty	NGM	10	10	Net profits
32	**BEE DEE Mining Claims**				
	BEE DEE Lease Agreement	Franco-Nevada & S&G Pinson	2	10	Current royalty rate is 2% (NSR), split between Franco-Nevada (1%) and S&G Pinson (1%)
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	2% NSR split between Royal Gold (1.9579%) and Duncan (0.0421%)
	PMC Royalty	NGM	10	10	Net profits
32	**Pinson Mining Claims**				
	Murphy royalty	Successors of Kate Murphy, et al.	5.5	7.5	NSR percentage is a sliding scale based on price per oz of gold. Current rate is 7.5% (for gold price higher than $700/oz).
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	4% NSR split between Royal Gold (3.9158%) and Duncan (0.084204%)
	PMC Royalty	NGM	10	10	Net profits
33	**Fee Land**				
	Royal Gold Royalty	Royal Gold & Duncan	0.5	5	1% NSR split between Royal Gold (0.97895%) and Duncan (0.02105%)
	Goldfield Royalty	Franco-Nevada	2	2	NSR
	Conoco Royalty	OMC	5	5	NSR
	PMC Royalty	NGM	10	10	Net profits

Note: All unpatented claims require annual assessment work to maintain validity.

4.2.6.2 Cordilleran Royalty (Royal Gold Inc. - current owners)

The Cordilleran Explorations partnership, the original developer of the Property, received an overriding royalty on several parcels, including all of the patented Section 29, Township 38 North, Range 42 East, consisting of a 3% NSR. Cordilleran Explorations also received a 5% NSR overriding royalty on the Pacific unpatented lode mining claims located in Section 28, Township 38 North, Range 42 East. Royal Gold, Inc. is the current owner of both royalties.

4.2.6.3 Goldfield Royalty (Franco-Nevada U.S. Corporation - current owner)

In 1981, The Goldfield Corporation, in a Special Warranty Deed, reserved to itself a 2% NSR royalty on the production of minerals from privately owned Section 33 of Township 38 North, Range 42 East (the Goldfield Royalty). Section 33 is now owned by OMC. The Goldfield Royalty is now owned by Franco-Nevada U.S. Corporation.

4.2.6.4 Conoco Royalty (OMC - current owner)

In 1982 PMC acquired three and three-quarter square miles of fee lands from Conoco Inc. (Sections 23, 27, 33, and the west half and NE quarter of Section 25, Township 38 North, Range 42 East). Conoco retained a 5% NSR royalty (the Conoco Royalty) on those parcels. Only the Section 33 parcel (which is owned by OMC) is part of the Pinson Project. OMC now owns the Conoco Royalty as to said Section 33. Consequently, while Section 33 is burdened by the Conoco Royalty, that royalty is payable to OMC.

4.2.6.5 PMC Royalty (NGM - current owner)

All of the Property is subject to a 10% net profits royalty, payable to Nevada Gold Mines LLC (NGM) (which acquired the PMC Royalty from PMC on 1 July 2019), that will be triggered after (but only after) the first 120,000 ounces of gold (and / or the gold-equivalent of other minerals) are produced from the Property (the PMC Royalty). The PMC Royalty was created by a Mineral Production Royalty Agreement dated 31 August 2011, which is the reference date for determining when the 120,000-ounce royalty production threshold has subsequently been reached. Currently the Property has produced ~6,834 ounces since the royalty was created.

4.3 Overall holding costs

A summary of costs since 2017 show that annual holding costs on the Getchell Property are typically about $0.5M. These costs include claim and property maintenance fees, environmental monitoring, wages, permits, and equipment maintenance.

4.4 Environmental liabilities

Environmental liabilities associated with historical mining and processing operations at the site are considered minimal. Current closure and reclamation financial sureties approved by the BLM and the Nevada Department of Environmental Protection total approximately $2.1M and cover all unreclaimed historical mining, exploration, and development operations at the Property.

No material environmental issues resulting from mining, exploration and development operations have been identified at the Property. The site is currently and will continue to be monitored in accordance with the permit requirements. OMC is in good standing with all its regulatory obligations under its existing permits.

4.5 Permits

OMC has all the primary permits in place to conduct underground mining operations at the Property. Specifically, underground exploration and mining activities are permitted under Reclamation Permit #0242 and WPCPs NEV2005102 and NEV2005103. Surface exploration disturbance within the plan boundary is permitted under Reclamation Permit #0047 and Plan of Operations NVN-064101.

Open pit mining and mining disturbance outside of the currently permitted areas will require, as appropriate, new approvals and / or amendments to the existing approvals.

A list of major active permits held by OMC for the Property is shown in Table 4.2.

Table 4.2 Active permits

Permit	Number	Agency
Pinson Mine Class II Air Quality Operating Permit	AQOP AP1041-3086.01	NDEP BAPC
Mercury Air Emissions Control Program Tier-3 Non-Permit De Minimis	AQOP AP1041-3089	NDEP BAPC
Water Pollution Control Permit: Pinson Infiltration Project	NEV2005102	NDEP BMRR
Water Pollution Control Permit: Pinson Mining Project	NEV2005103	NDEP BMRR
Pinson Underground Mine Reclamation Permit	#0242	NDEP BMRR
Pinson Mine (surface mine 1980-1999)	#0047	NDEP BMRR
Pinson Mining Plan of Operations (surface mine 1980-1999)	NVN-064101 (N24-83-004P)	BLM
Mining General Stormwater Permit	NVR300000/MSW-42365	NDEP BWPC
Onsite Sewage Disposal System (OSDS) General Permit	GNEVOSDS09S0177 Project Identification #S0049	NDEP BWPC
US ACOE Nationwide Permit	199400663 Nationwide Permit #26	US ACOE
EPA Hazardous Waste Generator	NV099530966	EPA
EPA Toxic Release Inventory	89414PNSNM22MIL	EPA

Pinson is located in the Kelly Creek drainage area. OMC currently controls sufficient water rights to operate the underground mine. Table 4.3 lists the water rights held by OMC.

Table 4.3 Water rights

Application / cert #	Owner	Diversion rate (cfs)	Duty (AFA)	Use
43130 / 13070	Osgood Mining Company, LLC	0.860	491.8	Mining, Milling, and Domestic
51388 / 14222	Osgood Mining Company, LLC	1.280	287.9	Mining, Milling, and Domestic
51427 / 14224	Osgood Mining Company, LLC	0.70	18.32	Mining, Milling, and Domestic
57885	Osgood Mining Company, LLC	0.90	651.57	Dewatering
57887	Osgood Mining Company, LLC	4.00	1076.00	Dewatering
65629	Osgood Mining Company, LLC	1.22	282.15	Dewatering
65630	Osgood Mining Company, LLC	0.47	114.88	Dewatering
65631	Osgood Mining Company, LLC	0.78	563.75	Dewatering
65632	Osgood Mining Company, LLC	1.8	800.00	Dewatering
68182	Osgood Mining Company, LLC	1.4	508.00	Surface (Granite Creek)
68183	Osgood Mining Company, LLC	1.45	525.00	Surface (Granite Creek)
77459	Osgood Mining Company, LLC	12.61	9129.23	Mining, Milling, and Dewatering
78956	Osgood Mining Company, LLC	1.00	723.97	Mining, Milling, and Dewatering
85178	Osgood Mining Company, LLC	2.25	1628.93	Mining, Milling, and Dewatering
85179	Osgood Mining Company, LLC	0.60	434.385	Mining, Milling, and Dewatering

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility and local resources

The Property is accessed by a combination of paved interstate and state highways, and well-maintained, unpaved private roads. Beginning in Winnemucca, travel east on Interstate 80 for 15 miles (24 kilometre (km)) and turn north at the Golconda exit. Proceed through Golconda to Nevada State Highway 789 and continue 16 miles (26 km) to a fork in the road and the end of the paved surface. The right gravel fork leads to the Ken Snyder Mine and the town of Midas. The Pinson deposit is located 4 miles (6 km) north along the left gravel fork, and the Getchell and Turquoise Ridge Mines are 7 miles (11 km) further up the road. The left fork terminates at the Twin Creeks Mine, 15 miles (24 km) north of the end of the pavement.

Winnemucca is the single urban population centre in Humboldt County, boasting a population of more than 7,300, and is the nearest significant source of mining personnel and resources for operations at the Property. Winnemucca is a historical ranching community, which grew to support regional large-scale mining following the discovery of several substantial gold deposits in the 1980s. A general aviation airport serves the local community, and a variety of logistical support is available from resident businesses. The active, relatively close-proximity Getchell / Turquoise Ridge and Twin Creeks mining complexes may provide an additional source of logistical support and skilled labour.

5.2 Topography, elevation, vegetation, and climate

The Property is situated in the Great Basin region of the Basin and Range Physiographic Province. North-south striking mountain ranges and parallel intermontane basins characterize the area. The entire region is a closed drainage system with all the permanent streams flowing to interior “sinks” such as the Carson and Humboldt sinks, or interior lakes such as Pyramid and Walker. Elevations in the area range from about 4,000 ft above mean sea level (amsl) in the basins, to over 9,000 ft amsl in the surrounding ranges. The local terrain near the Project is generally moderate.

Local vegetation consists of mixed sagebrush, shrubs, and grasses. Sagebrush and shrub species include two varieties of sagebrush (big and low); three of rabbitbrush (rubber, green, and low); bitterbrush; little leaf horsebrush; and desert peach. Grasses include Sandberg bluegrass, cheatgrass, Basin wild rye, wheatgrass, needlegrass, pepperweed, Russian thistle, halogeton, phlox, lupine, balsamroot, and Indian paintbrush (BLM 2001).

The climate in the Project area is semi-arid, with little rainfall, low humidity, and generally clear skies. Based on data provided by the Western Regional Climate Centre for the nearby Rye Patch Dam weather station, local average monthly temperatures range from about 43°F in January to around 94°F in July, and annual extremes range from -28 to 111°F. Average annual precipitation is around 7.82 in, and most precipitation falls as snow during the winter months. Winter and wet weather conditions occasionally limit access to the Project site, but in general, mining operations may be conducted year-round on the Property.

5.3 Infrastructure

Existing infrastructure at the Project includes an office building, dry and warehouse facilities, and a lined stockpile area on the surface. Over 9,000 ft of underground workings have been completed and four deep de-watering wells were drilled and cased, two of which are currently being operated. Electrical infrastructure suitable for mine operations is installed and two re-infiltration basins and associated pipelines have been constructed to re-infiltrate water produced in mine dewatering into the valley aquifer.

The mine is accessed through either of two portals, and dual egress has been established for most areas of the mine. Where dual egress is not possible, rescue chambers have been installed. Equipment is repaired in an underground mine shop. Air doors and a ventilation fan provide required air supply to the workings in compliance with Mine Safety and Health Administration (MSHA) standards.

Landline telephone and DSL service are available at the Project site. Cellular phone service is also available, but is dependent on the strength of receiving antennas, topography, and lines of sight.

6 History

The Property has been explored by a number of individuals and mining / exploration companies since the late 1930s. The original discovery on the Property was made by Clovis Pinson and Charles Ogee in the mid to late-1930s, but production did not occur until after World War II, when ore from the original discovery was shipped to and processed at the Getchell mine mill. In 1949 and 1950, total production from the Pinson mine amounted to approximately 10,000 tons grading approximately 0.14 opt.

6.1 Prior ownership and ownership changes

6.1.1 Cordex I Syndicate

The Property remained functionally dormant from 1950 until 1970, when an exploration group known as the Cordex I Syndicate (John Livermore, Peter Galli, Don Duncan, and Rayrock Resources) leased the Property from the Christison Family (descendants of Mr Pinson and Property owners), on the strength of its similarity to the Getchell Property and structural position along the range-front fault zone bordering the Osgood Mountains. Following a surface mapping and sampling program in 1971, 17 reverse circulation (RC) drillholes were completed in and around the 1940s era Pinson pit, confirming low- grade gold values. An 18th step-out hole encountered a 90-ft intercept of 0.17 opt Au. This intercept was interpreted as a subcropping extension of known mineralization NE of the original pit and was the basis for delineation of what would become the "A" Zone at the Property, a 60-by-1,000-ft shear zone. During the late 1970s, the Cordex I Syndicate reorganized into a Nevada Partnership known as PMC, with Rayrock Resources as the Project operator, and began production at the Property.

Cordex Syndicate (Cordex), and its successor, PMC, explored the Property largely through mapping and geochemical sampling. There are three known mapping programs:

1. A regional mapping program from Preble to Getchell by Pete Chapman in the late 1970s.
2. A 1:6000 scale mapping program of the Property in 1983.
3. A 1:2400 scale mapping program of the Pit areas through the active life of the mine.

6.1.2 Pinson Mining Company

PMC began developing the A Pit in 1980 and produced gold the following year. Production from the B Pit began in 1982. Step-out drilling in 1982 – 1983 to the NE of the A Zone intersected two more discrete zones: The C Zone extending east-north-east (ENE) from the A Zone and the CX Zone extending NE from the C Zone. Step-out drilling NE of the CX Zone in 1984 located an apparently independent fault system (striking north-northwest (NNW), dipping steeply east that became the core of the Mag deposit, which went into production in 1987. PMC produced from the CX, CX-West and Mag Pits into the mid to late 1990s, until a combination of falling gold prices and erratic mill feed forced closure of the oxide mill in early 1998. Continued attempts to expand production of oxide ore failed, and all active mining ceased on 28 January 1999 (McLachlan et al. 2000).

6.1.3 Homestake – Barrick

In the 1990s, Homestake Mining Company (Homestake) and Barrick became "fifty-fifty" partners in PMC through purchase of minority interests (McLachlan et al. 2000). Homestake and Barrick conducted an exploration program from 1996 to 2000 through PMC, expending some $12M on the Project. The joint venture explored the deeper feeder fault zones of the Property, exploring for a large, high-grade gold system that would support a refractory mill complex. This work, while successful in identifying gold mineralization with underground grades, failed to identify a deposit of sufficient size to be of development interest to Homestake or Barrick, and the partners concluded the exploration program. Subsequent to that decision, in 2003, Barrick acquired Homestake and drilled an additional three exploration drillholes.

6.1.4 Atna Resources Ltd. earn-in and PMC back-in

In August 2004, Atna Resources Ltd. (Atna) acquired an option to earn 70% Joint Venture interest in the Property from PMC, a wholly owned subsidiary of Barrick, and commenced additional follow-up exploration and development of the Property. Atna completed its earn-in in 2006 and vested in its 70% interest in the Project after expending the required $12M in exploration and development expenditures. PMC elected to back-in to the Project and re-earn an additional 40% interest (bringing PMC's interest to 70% and Atna's to 30%) on 5 April 2006. PMC spent over $30M on the Project during the next three-year period and completed its "claw-back" in early 2009. Their work included surface and underground diamond core drilling, RC rotary drilling, underground drifting, and surface infrastructure construction (rapid infiltration basins, mineralized material stockpile pad, underground electrical service upgrades, etc.). A new mining joint venture was formed in 2009 reflecting the Project's ownership with PMC owning a 70% interest in the venture and Atna owning a 30% interest. PMC, as the majority interest owner, was the operator of the joint venture.

6.1.5 Atna 2011 – 2013 underground development

In September 2011, Atna negotiated the acquisition of PMC's 70% joint venture interest in the core property position at the Getchell Project (previously Pinson Project). The asset purchase and sale agreement include all right title and interest to the core property described above as well as an evergreen processing agreement with Barrick for the processing of underground refractory ores from Pinson at Barrick's Goldstrike facilities.

Development of the Pinson underground mine commenced in early 2012 and mine ramp-up began in late 2012. In total, 6,011 ft of primary and secondary development were completed during 2012 and 2013. The primary spiral ramp was driven to the 4530 level from the 4650 adit level and both top cut and underhand stoping occurred in three Ogee-zone stope blocks during development. Additional secondary access drifts were in progress when the mine was placed on care and maintenance to access the Range Front and Adams Peak mineral zones but were not completed prior to cessation of underground work. Mining was performed by contract-miners utilizing underground mining equipment owned by the contractor. Approximately 30,000 tons of ore containing 7,900 oz of gold were mined and shipped to off-site processing facilities.

Work on the Project continued until June of 2013 when the mine was placed on care and maintenance. This decision was driven by a number of factors including the steep decline in the gold prices in 2013.

In May 2014, the status of the underground mine was changed to an intermittent production status. Under this status, periodic mining from stoping areas developed in 2013 was conducted to develop and test revised stoping methods for the underground and to prove mining economics at small production rates.

6.1.6 Osgood Mining Company LLC acquisition

Since acquiring the Project in 2016, OMC has completed numerous drillhole database compilation and verification campaigns beginning with migration of the ATNA database to Maxwell Datashed Database software in 2017 and database verification and improvement efforts in 2018. In 2016, OMC, with an external consultant, completed a project scale structural geology study that included surface and underground mapping, historical data review and cross section interpretation that was aimed at defining the main structural architecture at Pinson and develop exploration and resource drilling targets. This work formed the basis of an updated 3D litho-structural model that was used for Mineral Resource estimation. From 2017 – 2018, OMC also completed an extensive drill material inventory and salvage program that secured the available drill core and RC chips on the property.

OMC continued to maintain compliance and keep all environmental permits for the site in good standing. This included performing permit related sampling and reporting, as well as renewing permits. In addition, OMC performed regular inspections of the site. During the ownership period, OMC worked with the State of Nevada to close out a Water Pollution Control Permit for a reclaimed portion of the mine, reducing the overall compliance monitoring and reporting liabilities for the operator. In addition, OMC received approval from the State to remove portions of the reclaimed site from the bond.

In addition to these geology and compliance activities, OMC has continued to maintain and improve site infrastructure including a third party review of hydrology and dewatering requirements that resulted in the replacement of pumps (2019) and the upgrading of two dewatering well process controls. Rapid infiltration basins (RIB’s) have been maintained as needed with water flows being tracked and monitored.

6.1.7 i-80

On 10 August 2020, Premier entered into a definitive purchase agreement with affiliates of Waterton Global Resource Management, Inc. to acquire from Waterton all of the outstanding membership interests of OMC.

6.2 Historical Mineral Reserve and production

No known Mineral Resource estimates have been published prior to Atna’s involvement in the Property’s exploration and development.

A QP has not done sufficient work to classify the historical estimate (“initial reserve”) as a current Mineral Resource or Mineral Reserve and the issuer is not treating the historical estimate as current Mineral Resource or Mineral Reserve. The details of the initial reserve can be found in Gustavson Associates (2012).

Historically, the Getchell Project (previously Pinson Project) with small additions from the nearby Preble and Kramer Hill mines was credited with gold production in excess of 1 million ounces, and less than 100,000 oz of silver (Tingley 1998). PMC independently compiled a record of production and credited the Pinson mine Property with production of 986,000 oz of gold through 1999.

Table 6.1 shows the historical production and initial reserve from the Getchell Project.

Table 6.1 Property production summary

Deposit	Year of discovery	Years in production	Initial reserves			Gold produced (troy oz)		References
			Short tons	Gold grade (opt)	Contained gold (oz)	Mill feed	Leach feed	
Gold deposits of the Pinson Mining Company								
A	1963, 1971	1980 – 1985	2,500,000	0.108	270,000	369,753	83,469	Hill 1971, PMC 1993
B	1971	1982 – 1988	3,400,000	0.050	170,000	Included	Above	As above
C	1982	1988 – 1996	233,000	0.017	3,961	10,773	N/A	PMC 1993, 1999
CX	1982	1990 – 1999	1,684,000	0.070	117,880	83,951	33,884	PMC 1993, 1999
CX-West	1993	1994 – 1999			0	3,962	In CX	PMC 1996, 1999
Mag (mill feed)	1984	1987 – 1999	4,300,000	0.080	344,000	301,255	N/A	PMC 199_, 1999
Mag (leach feed)			2,300,000	0.030	69,000	N/A	59,741	Foster and Kretschmer 1991, PMC 1999
Felix	1972	1989 – 1992	355,000	0.030	10,650	1,133	11,641	PMC 1993, 1999
Blue Bell	1972, 1983	1993 – 1994	228,000	0.072	16,416	17,014	1,085	PMC 1993, 1999
Pacific	1984	1992 – 1993	130,000	0.048	6,240	4,939	2,607	PMC 1993, 1999
Pinson Mine		08/1999 – 12/1999				0	2,141	PMC 1999
Pinson Underground		2012-2013	*30,148	*0.263	*7,915	**6,834		Atna mine records
Pinson Mine combined production					**1,016,062**	**799,614**	**194,568**	**Total Pinson Mine production: 994,182 oz gold**
Prior gold production on PMC properties								
Ogee & Pinson	**1945**	**1949 – 1950**					**~10,000**	**Hill 1971**

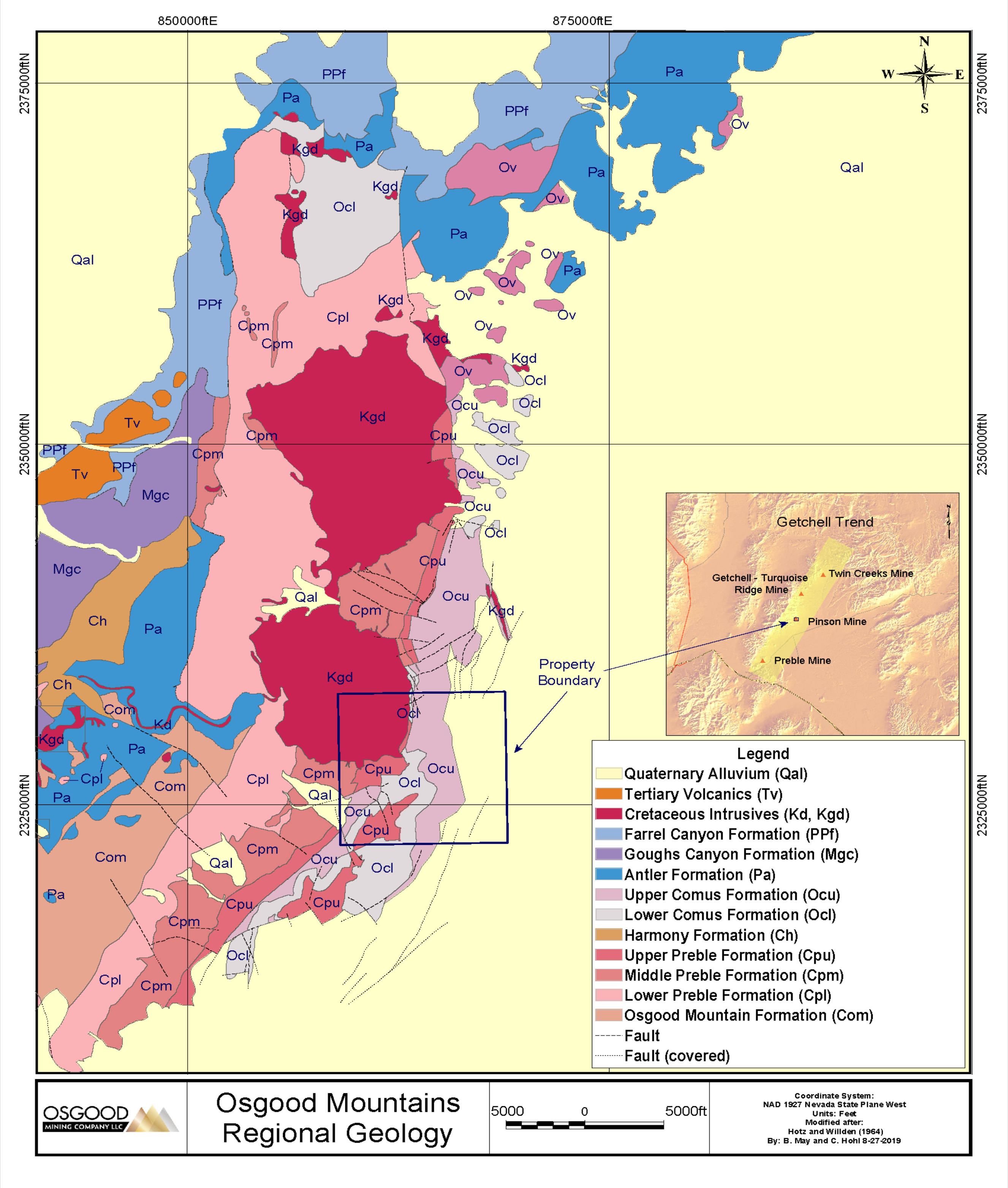
Notes:
*Underground production is tonnage and grade produced and includes minor low-grade development tonnage that was upgraded by screening to a shippable product.
**Underground production reflects ounces recovered at third-party mills from shipped underground ores.

7 Geological setting and mineralization

7.1 Regional geology

The Property is located on the eastern flank of the Osgood Mountains within the Basin and Range tectonic province of northern Nevada. The Pinson mine, together with the Preble, Getchell, Turquoise Ridge and Twin Creeks mines, are on what is referred to as the Getchell gold trend (Getchell trend). The main Getchell trend generally strikes NE-SW and has been cross-cut by secondary north-south and NW-SE-trending structures. The deposits are hosted in Paleozoic marine sedimentary rocks. The rocks are exposed in the Osgood Mountains and have been complexly thrust faulted (Hotz and Willden 1964) and intruded by the Cretaceous-aged (92 Ma) (Silberman et al. 1974) Osgood Mountains granodiorite stock. These units are unconformably overlain by Miocene volcanic rocks. Figure 7.1 is a regional geologic map of the Osgood Range.

Figure 7.1 Regional geological map of a portion of the Osgood Mountains



As mapped by Hotz and Willden (1964) and Jones (1991, cited in McLachlan et al. 2000), the Osgood Mountains are underlain by Cambrian through Permian metamorphic and sedimentary rocks that were deposited on the rifted western margin of the North American Craton. These rocks have been intruded by the Cretaceous-aged Osgood Mountains granodiorite stock which forms the core of the Osgood Mountains, and plunge north (Chevillon et al. 2000, cited in McLachlan et al. 2000). A significant thermal metamorphic aureole surrounds the stock.

At least four Paleozoic terranes, defined by structure, lithology, and age comprise the Osgood Mountains (McLachlan et al. 2000). These include the:

- Osgood Mountain terrane.
- Leviathan allochthon.
- Antler overlap sequence.
- Golconda allochthon.

The Osgood Mountain terrane has been described by Jones (1991, cited in McLachlan et al. 2000) and is comprised of the Cambrian Osgood Mountain Quartzite and the Cambrian Preble Formation. Both units have undergone regional metamorphism and intense, northwest-directed folding (McLachlan et al. 2000). At the Getchell Project, these two units are folded together to form the northwest-verging Pinson anticline.

The Leviathon allochthon described by Stenger et al. (1998) is exposed at the Turquoise Ridge and Twin Creeks mines. The assemblage was mapped as "Ordovician Valmy" by Hotz and Willden (1964), although it differs somewhat from Valmy Formation (Ov) as exposed in other areas of Nevada (McLachlan et al. 2000). The Leviathon allochthon is composed of a thick (>980 ft) sequence of mid-ocean ridge basalts and intercalated pelagic sediments which have been thrust over the Twin Creeks member of the Comus Formation (Stenger et al. 1998). This sequence has not been identified at the Pinson mine.

The Antler overlap sequence in the Osgood Mountains consists of the Pennsylvanian Battle Formation, and Pennsylvanian-Permian Etchart and Adam Peak formations (McLachlan et al. 2000). The Battle conglomerate consists of cobbles and pebbles of quartzite. The Etchart lies conformably on the Battle and consists of calcareous sandstone underlying fossiliferous limestone. South of the Getchell Project, the Battle and Etchart lie unconformably on the Preble Formation and Osgood Mountain Quartzite (McLachlan et al. 2000).

The Golconda allochthon comprised of the Mississippian Goughs Canyon and the Pennsylvanian-Permian Farrel Canyon formations is present along the north-west flank of the Osgood Mountains. The thrust strikes N to NE from the central part of the range to the Dry Hills in the north (McLachlan et al. 2000). These units are not present at the Getchell Project.

7.2 Local geology

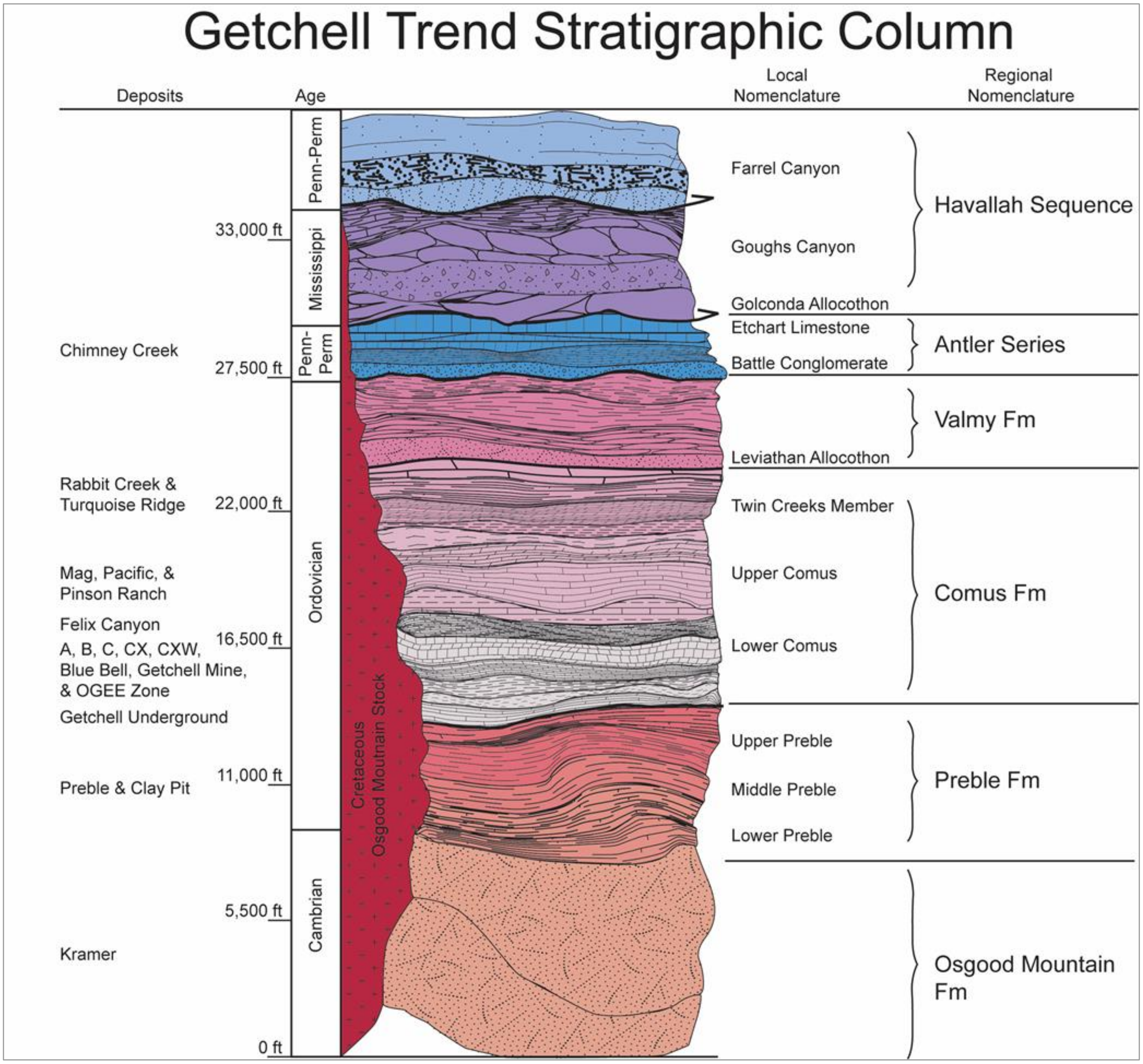
The geology at the Property is typified by folded Cambrian to Ordovician sedimentary rocks that have been intruded by Cretaceous stocks which have been cross-cut by later high-angle structural deformation. Hotz and Willden (1964) suggest the high angle faulting is related to the Basin and Range extension. The older rocks are overlain by Miocene andesitic basalt and the surrounding fault-bounded basins filled with quaternary alluvial (QAL) gravel. The Osgood Mountains have a general NE trend although in the vicinity of the Pinson mine the east flank of the range trends north. Gold mineralization is primarily hosted by fine-grained marine sedimentary rocks that overlie a large stock of Cretaceous granodiorite. The Property is considered to be part of the Osgood Mountain terrane.

At the Property, Cambrian to Ordovician siliciclastic and carbonate rocks have been intruded by the Cretaceous Osgood Mountain granodiorite resulting in the formation of large metamorphosed aureoles with development of several tungsten-bearing skarns. The lowest stratigraphic units recognized at the Property are the Cambrian phyllitic shales, limestone interbeds and various hornfelsed sedimentary rocks of the Preble Formation which are juxtaposed against the granodioritic intrusive. The Preble is overlain by Ordovician sedimentary rocks of the Comus Formation both of which have been folded into a broad, north-plunging anticline. The west flank of the anticline has been over-thrust by the Ordovician Valmy Formation which consists of deep-water siliceous shales and cherts. The core of the anticline and scattered localities along the east side of the Osgood Mountains are unconformably overlain or in fault contact with sandstones and conglomerates of the Battle Formation and limestones of the Etchart Formation.

A second structural trend evidenced by the presence of the Golconda and Humboldt thrusts displaced Mississippian volcanics and Pennsylvania shales eastward along the north-west and southern flanks of the Osgood Mountains. Extension during the Tertiary resulted in outflows of rhyolitic tuffs, Miocene basalt, andesite flows, and younger basalt flows.

Gold mineralization at the Property is primarily hosted in the Comus Formation as shown in Figure 7.2.

Figure 7.2 Stratigraphy of the Getchell trend



Notes: The Mag, Pacific, Felix Canyon, A, B, C, CX, CXW, Ogee Zone and Blue Bell deposits are on the Property.
Source: Osgood Mining Company LLC (modified from Sim 2005).

The stratigraphy of the Osgood Mountains from youngest to oldest is:

- Quaternary:
  - Qal / Qb – Alluvium and basalt.
- Tertiary:
  - Tba – Olivine basalt.
  - Tba – Andesite and basalt flows. Dark green to black aphanitic and weakly porphyritic flows, flow breccia.
  - Tr – Rhyolitic tuffs. Pumice, welded, reworked, tan to white.
  - Tcg – Chert, shale, rhyolite clasts in a sandy matrix.

- Dacite dikes.
- Andesite dikes.

- Cretaceous:
  - Kgd – Granodiorite, quartz diorite. Equigranular, medium grain intergrowths of feldspar, quartz, biotite, and hornblende.
- Permian / Pennsylvanian:
  - PPfc – Farrell Canyon Formation. Interbedded sandstone, chert, shale, siltstone with minor volcanic flows and pyroclastics. Chert, interbedded with sandstone composes up to 50% of the unit.
  - PPa – Adam Peak Formation. Chiefly shale and siltstone with 40-60% dolomitic sandstone and chert. Shales and siltstones are dolomitic. Unit contains 2-3% phosphate.
  - PPe – Etchart Limestone. Limestone, sandy limestone, dolomite. Lower portion is sandy limestone with local pebble conglomerate. Upper portion is pure limestone with interbedded dolomite and sandy dolomite. Minor calcareous shale.
- Pennsylvanian:
  - Pb – Battle Formation. Poorly bedded, poorly sorted boulder and pebble conglomerate with coarse-grained sandstone and minor limestone clasts composed of Osgood Quartzite and chert in a shaley to sandy matrix.
- Ordovician:
  - Ov – Valmy Chert. Chert, shale, quartzite, volcanics (greenstone). Interbedded chert and shale with quartzite greenstone bed on east side of the Osgood Mountains. Interbedded with shale and minor limestone. Quartzite dominant in lower portion and chert, shale in upper portion.
  - Oc – Comus Formation – Alternating sequence of limestone, shale, and dolomite. Distinctive tuffaceous shales and tabular intraformational conglomerate. In the Twin Creeks mine area, numerous mafic igneous sills and dikes exist within the sequence. Igneous rocks are not present in the Pinson portion of the Comus Formation.
- Cambrian:
  - Ch – Harmony Formation. Feldspathic sandstone with pebble conglomerate and interbedded shale. Light olive brown, red, reddish green in color. Paradise Valley cherts are separate but mapped with the Harmony Formation.
  - Cp – Preble Formation – Dominantly phyllitic shale. Light olive to brown. Upper part contains thin interbeds of limestone rhythmically bedded with shale.
  - Com – Osgood Mountain Quartzite. White, gray, light brown, purple brown to green gray, medium to thick-bedded quartzite. Contains Twin Creek Member. Impure quartzite, silty sandstone, phyllitic shale.

7.3 Property geology

The Property is located on the eastern flank of a large Cretaceous granodiorite stock that forms the southern core of the Osgood Mountains. Rocks adjacent to the eastern side of the stock have a general east dip and strike sub-parallel to the trend of the Osgood Mountains. The oldest units exposed against the granodiorite are Cambrian Preble phyllitic shales, interbedded limestones, and various hornfelsed sediments. Overlying the Preble is a thick package of Ordovician Comus sediments. The lowest portion of the Comus is composed of medium to massively bedded, micritic to silty limestone. The middle portion consists of interbedded limestone and shale layers with local interbedded debris flows. The Upper Comus is comprised of mildly to non-calcareous shales with minor shaly limestone interbeds.

The depositional relationship between the Preble Formation and the overlying Comus Formation is not clearly understood. In the Getchell Property the two formations are in fault contact, however, Kretschmer (1984) suggests they are conformable. McLachlan et al. (2000) state that at the Getchell Project, the two formations are in fault contact with each other and subparallel to the Range Front Fault (RFF) that juxtaposes Comus Formation in the hangingwall against the Preble Formation in the footwall.

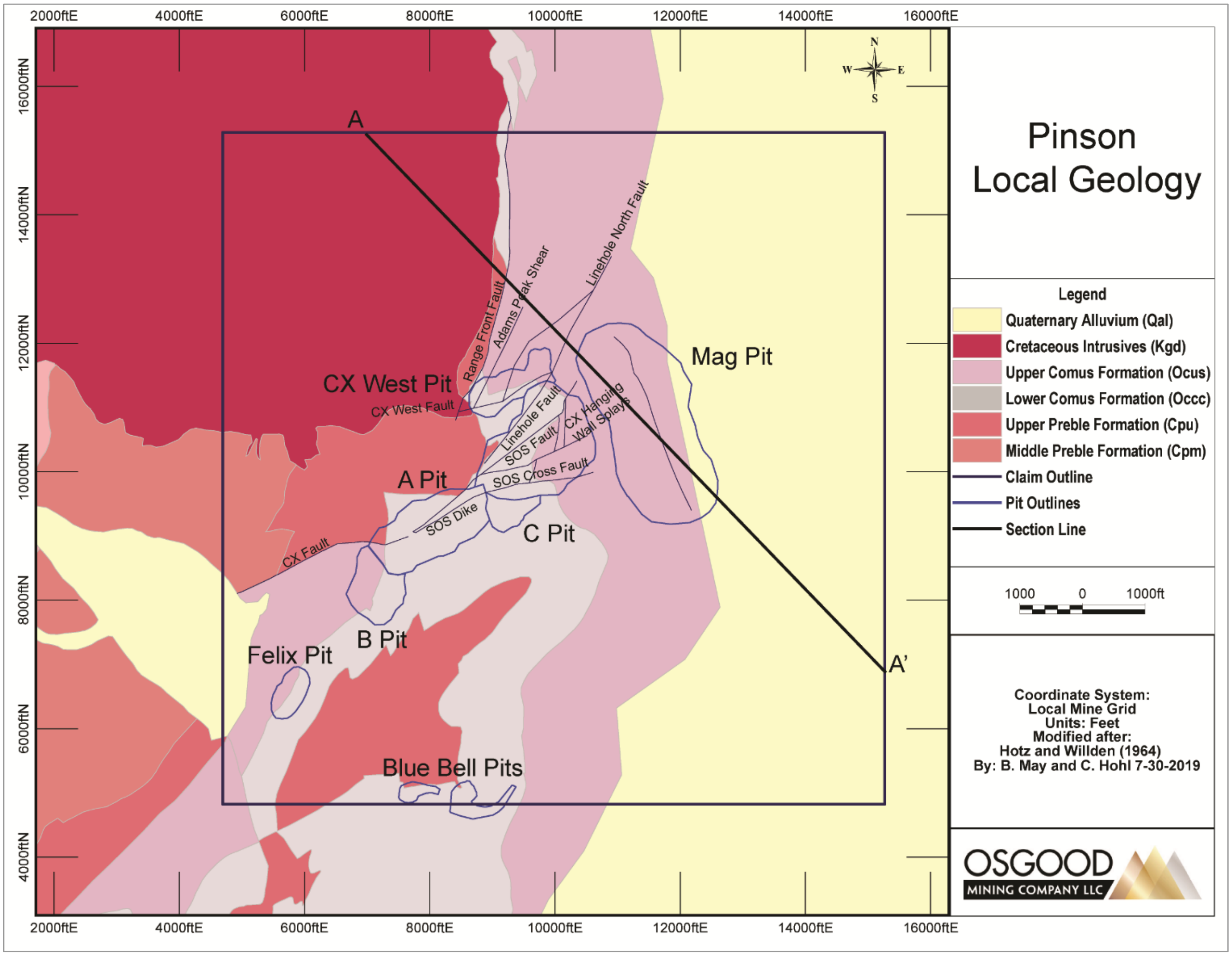
A Cretaceous aged (90 – 92 Ma) (Silberman et al. 1974) granodiorite stock intrudes the Paleozoic section in the southern half of the Osgood Mountains. Emplacement of the stock resulted in the formation of an irregular contact metamorphic aureole which extends as much as 10,000 ft from the intrusive contact. The metamorphic event resulted in the formation of maroon-colored, biotite-cordierite hornfels in the Upper Preble Formation and chiastolite hornfels in the Upper Comus Formation within much of the Property area (McLachlan et al. 2000). In addition, carbonate rocks were metamorphosed to marble and calc-silicates (wollastonite, garnet, diopside, and vesuvianite). Several tungsten-bearing skarn deposits were also formed along the margins of the stock (Silberman et al. 1974). Two tungsten skarns are located on the Property.

Outcrop mapping and historic drilling has revealed the presence of extensive folding of the Paleozoic section in the Osgood Mountains. The most prominent of these folds is the Pinson Anticline. The fold is NE-plunging and NW-verging and extends for a distance of approximately three miles SW from the Pinson mine (McLachlan et al. 2000). Numerous parasitic folds have also been noted along the limbs of the anticline. Where exposed, the Pinson Anticline is cored by the Cambrian Preble Formation and flanked on the north-west and SE by sediments of the Ordovician Comus Formation.

Mineralization on the Property exhibits strong structural control. A wide variety of mineralized structural orientations have been documented. The most important structural feature on the Property is the network of faults that border the escarpment marking the southern and eastern edge of the Osgood granodiorite (Sim 2005). This fault system has been variably interpreted as a single master fault (RFF) (McLachlan et al. 2000) that curves around the stock, or more likely, a network of shorter, straighter segments that collectively accommodate several thousand feet of displacement while making a 50° bend around the SE corner of the stock (Sim 2005). The fault system can be divided into three structural and stratigraphically mineralized zones, with each mineralized zone defined by one or more major structural elements. These are referred to as the Range Front, CX, and Mag Zones. Sedimentary rocks in the vicinity of this system generally dip steeply (easterly) away from the contacts of the granodiorite (Sim 2005).

Figure 7.3 shows the structural and geology map of the Property with the mined-out pits shown for reference.

Figure 7.3 Geology and structural map



Notes: Modified from Hotz and Willden 1964.

The relationship of the Range Front, CX, and Mag Zones to the Property stratigraphy are described below and shown in Figure 7.5.

Figure 7.5 also shows a representative cross-section through the Property.

In addition to this large-scale fault system, there are numerous NW and a few east-west (EW) structures that have been identified by past mapping and drilling (McLachlan et al. 2000). In general, these appear to be mostly older than, and truncated by, the main system. Some of these faults have been re-activated and disrupt the continuity of the main Pinson system (Sim 2005).

7.4 Structural framework

7.4.1 Structural overview

In 2016, OMC contracted Mr Robert Leonardson to generate a comprehensive geological model for the Getchell Project. This work included geological interpretation, structural review, and the identification of exploration targets on the Project. The following structural overview is summarized from his report Pinson Narrative Nov 2016 submitted to OMC.

Structure at the Getchell Project is highly complex and involves varying stress environments hinged upon the inflection of the RFF around the Osgood Mountain complex. Principal stress north of the inflection point is compressional based on bedding relationships with faulting and transpressional south of the inflection due to strike-slip relationships of the many NE striking CX-type faults (Leonardson 2016). A principal thrust orientation of WNW-ESE (285°-105°) is recognized at the Property based on the dominant bedding attitudes of north-north-east (NNE) (015°), and the steep easterly dips of the strata. Three main structural elements have been recognized: a frontal ramp striking north-south that is west-verging and extends north to the Getchell Mine along the range front; a northwest-verging oblique ramp extending to the SW along a second orientation of the range front; an inflection point where the two ramps intersect. These three elements determine the style and orientation of the subsequent development of two thrust fault suites.

The first thrust suite is the W-verging range front on a frontal ramp that rises to the west over the Osgood stock and extends southward from the Getchell Mine area to the inflection point. This imbricated thrust stack sources off a N-S ramp and flat structure approximately 1,500 ft east of the Mag Pit east high wall. South of the inflection point rocks along the SW-oriented ramp produce a second thrust suite that dips SE and strikes SW. Leonardson (2016) terms this suite of thrusts CX type. A series of vertical ENE striking faults, centred on and extending north and south of the inflection point relieved stress between the two ramp structures allowing the rocks on the south to move to the west-southwest forming the second thrust suite.

South-dipping thrusts of the CX type include three, and potentially four, main faults named Adam Peak, CX, Disturbed, and South Mountain; the CX fault is the most significant. The source of these faults is to the east from the same north-south ramp structure that sources the RFF suite. A significant cross over thrust in this suite, the Delaney thrust, appears to connect with the Disturbed and South Mountain thrusts.

The following subsections give details on significant structural features observed across the Property. Pit structural mapping by Chadwick (2002) collected orientation data and cross-cutting relationships.

7.4.2 Faults and faulting

7.4.2.1 Range Front Zone

The Range Front Zone (RFZ) is comprised of a series of NE-trending subordinate faults that form a broad persistent zone of shearing and brecciation along the Range Front Fault (RFF) that bounds the eastern margin of the Osgood Mountains. The RFZ involves the entire stratigraphic sequence at the Property including the Cambrian Preble, Ordovician Comus and Cretaceous granodiorite.

7.4.2.2 Range Front Fault

The RFF is a prominent 020° to 030° striking normal fault, that defines the eastern front of the Osgood Mountains. For much of its length, the fault juxtaposes the Comus Formation in the hangingwall against the Preble Formation in the footwall (McLachlan et al. 2000). The paucity of distinct marker units prevents determination of the amount of offset on the fault, but significant strike-slip displacement is suspected. The contact of the fault exhibits brittle and ductile deformation characteristics. The fault dips to the SE at 55° to 65°.

7.4.2.3 CX West Fault

The CX-West Fault strikes 070° from the southern portion of the exposed granodiorite and has down-dropped stratigraphy significantly to the north. The CX-West along with several other NE trending structures may represent conjugate accommodation structures that link the main northerly trending faults along which major dip-slip faulting occurred (Leonardson 2016). These structural intersections are key controls to mineralization and may often host high-grade pods (McLachlan et al. 2000).

7.4.2.4 Ogee (Linehole) Fault

The Ogee Fault is a steeply dipping, north trending fault that cuts both the Upper and Lower Comus formations. It is also called the Linehole Fault. Drilling indicates that the Ogee Fault is intersected and cut by the CX-West Fault and is slightly offset to the east on the north side of the CX West Fault (Leonardson 2016). The intersection of these two structures is referred to as the Ogee zone and contains significant mineralization.

7.4.2.5 Adams Peak Shear

The Adams Peak Shear, as identified from underground workings, occurs primarily in the Upper and Lower Comus formations. The shear consists of multiple vertical structures originating from the RFF that form a brittle broken area of rock (Gustavson Associates 2012). The structure has not been identified south of the CX West fault and its north-easterly extent is unknown.

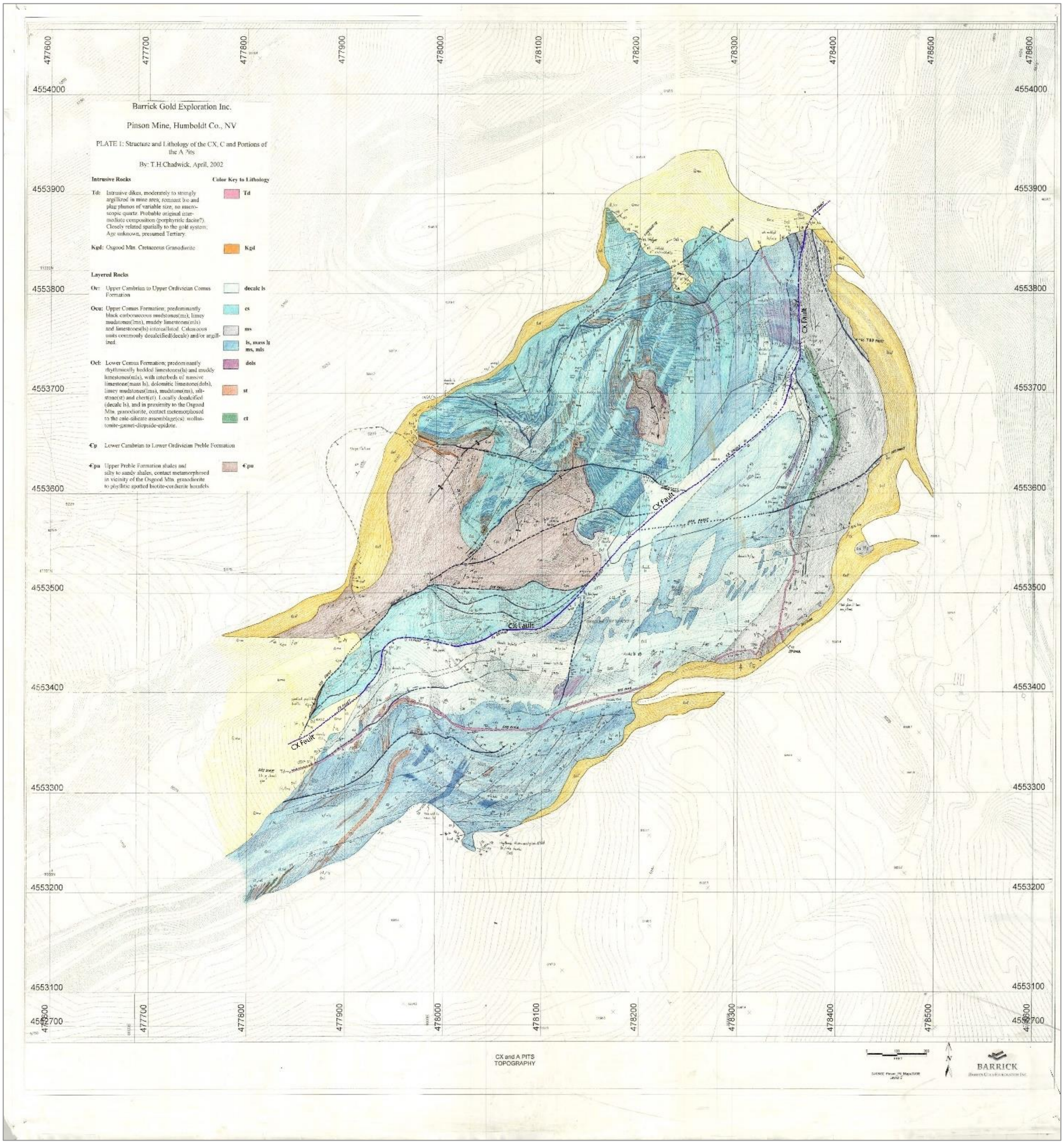
7.4.2.6 CX Zone

The CX Zone was discovered between 1982 and 1983 by PMC during a step-out drilling program along the NE Projection of the A Zone (McLachlan et al. 2000). The zone is comprised of the dominant CX Fault along with a series of subordinate NE trending faults and fault splays. The CX Fault was the primary mineralized structure developed by PMC in the CX open pit.

7.4.2.7 CX Fault

The CX Fault is a complex zone of brittle fracturing that juxtaposes Upper Comus shales against limy beds of the Lower Comus. The fault strikes approximately 035° to 045° and dips 55° to 65° SE Chadwick (2002) as shown in Figure 7.4. Relative movement on the fault is described as dip slip with the SE side downthrown although the amount of displacement is unknown. The fault was an important hydrothermal conduit that focused gold mineralization within the CX Pit.

Figure 7.4 Structure and lithology of the CX, C, and portions of the A Pits



Source: Chadwick 2002.

7.4.2.8 SOS Fault

The SOS Fault is a steeply south-south-east (SSE) dipping structure (McLachlan et al. 2000). The fault has a strike length of 1,400 ft, has an average thickness of 10 ft and intersects the CX fault down dip (Gustavson Associates 2012).

7.4.2.9 CX Fault hangingwall splays

Two near-vertical, NNE-striking, east-south-east (ESE) dipping fault splays have been mapped in the CX Pit (Gustavson Associates 2012; McLachlan et al. 2000). These faults are considered hangingwall splays of the CX Fault and extend southward to the SOS Fault. These faults hosted the majority of ore mined from the CX Pit. Most of the mineralization contained within these structures appears to rake to the NNE and has been intersected in drilling to depths of 1,800 ft (McLachlan et al. 2000).

7.4.2.10 SOS Dike

The SOS Dike strikes approximately 070°, has a near vertical dip, and is interpreted to be Tertiary in age (Gustavson Associates 2012). The dike is exposed within the CX Pit and appears to follow a zone of structural weakness that parallels the SOS Fault.

7.4.2.11 SOS Cross Fault

The SOS Cross Fault is a narrow structure that extends from the footwall of the SOS Fault to the SOS Dike on the south (Gustavson Associates 2012). The fault is near vertical and strikes 020°.

7.4.2.12 Mag Zone

The Mag Zone is defined by a suite of major NW-striking, NE-dipping faults termed the Mag Fault that forms a broad zone of alteration ~1,200 ft in width. The Mag fault suite is cut by moderate to low angle CX-hangingwall fault splays (Leonardson 2016).

7.5 Mineralization

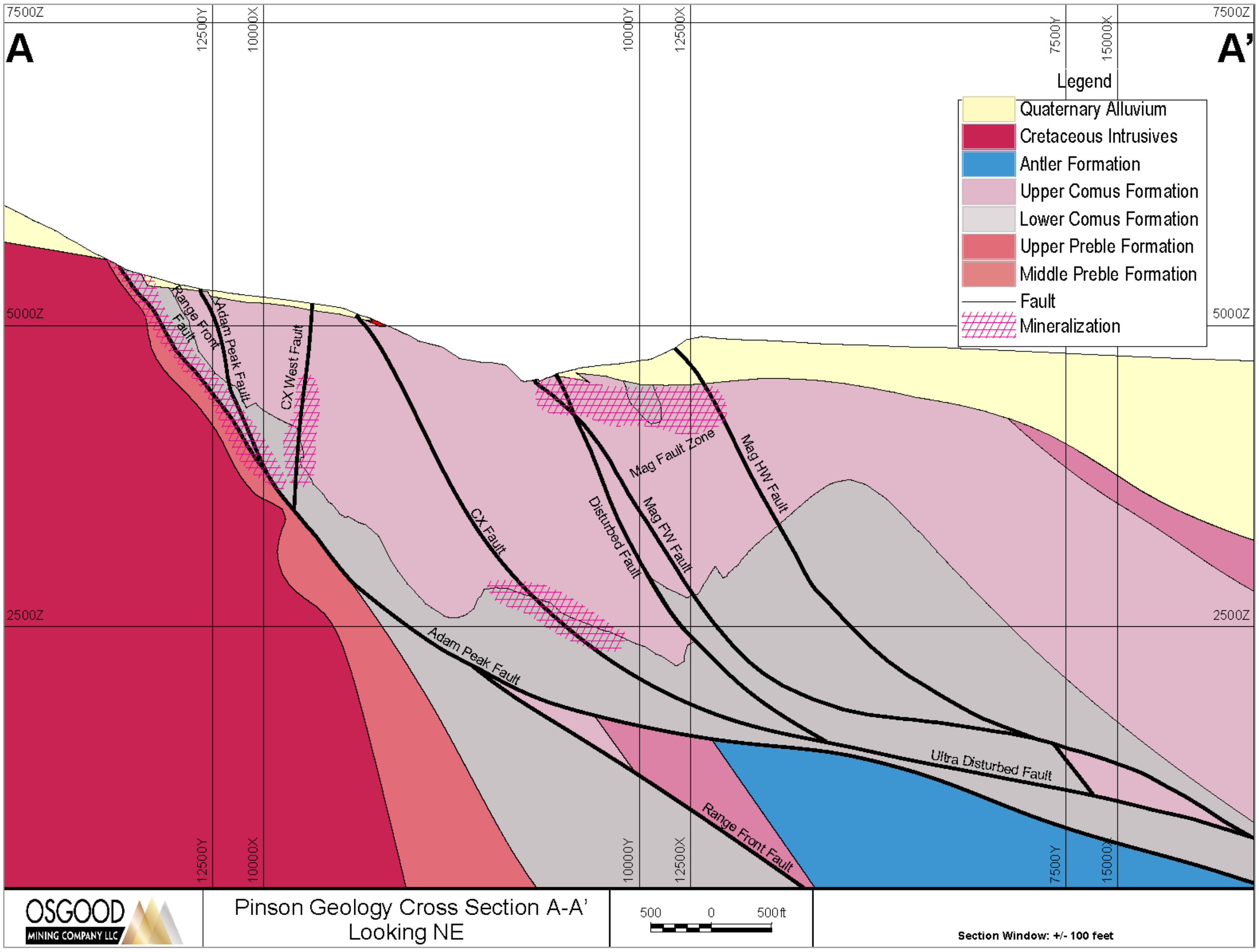
Underground mineralization associated with the CX and RFF typically strikes NE to NNE with moderate to subvertical dip and thickness varying between 5 ft to 30 ft. High-grade gold mineralized zones are moderately discontinuous and occur within near-vertical pipe-like bodies at fault intersections and along fault parallel structural corridors. Gold mineralization is characterized by pervasive sulphide that consists of two stages of pyrite development, an early barren pyrite stage and a gold-bearing arsenian pyrite stage (Ridgley et al. 2005). Megascopically, the gold-bearing pyrite is typically dull brassy to black in color and very fine-grained. Pyrite may also be associated with remobilized carbon imparting a “sooty” appearance to the pyrite. Gold is primarily contained in pyrite as microscopic inclusions or found as rims around fine pyrite grains (Wallace and Wittkop 1983; Foster 1994 cited in Ridgley et al. 2005). Gold mineralization can be found in multiple styles, including fine sulphide associated with quartz veining and brecciation. Foster and Kretschmer (1991) suggest, based on detailed geochemical analyses, that there is a positive Au-Hg correlation, as well as a negative Au-Ba association.

Gold mineralization at the Property is primarily hosted by the Upper and Lower Comus Formations which consist of interbedded shale, siltstone and limestone. The Upper Comus is the primary host lithology in the Mag Zone and currently is host to the majority of surface resources at Pinson deposit (Gustavson Associates 2012). The Upper Comus is also locally mineralized within the A, B, C, CX, CX-West, and portions of the RFZ. The Lower Comus hosts the majority of the higher-grade underground resources.

The Preble rocks are a poor host for gold mineralization but does contain localized gold concentrations where have been brecciated and adjacent to major hydrothermal conduits.

Figure 7.5 is a representative cross-section through the Property. The section shows features of stratigraphy and structure that are factors in the localization of gold mineralization. Prominent features include high-angle fault zones and the primary host lithology (Upper and Lower Comus Formation).

Figure 7.5 Cross-section A-A' looking NE



Oxide mineralization includes pervasive limonite, hematite along with other iron and arsenic oxides. Oxidation is extensive in the CX Fault system, occurring along the entire length of the zone and penetrating to a depth of 1,500 ft. Within the RFF system, oxidation is more variable than within the CX Fault system. In some fault and shear zones, oxidation may be present to depths of 1,800 ft, whereas in others it may only reach to depths of < 500 ft (Ridgley et al. 2005).

7.5.1 Mag Pit mineralization

Gold mineralization within the Mag Pit is hosted by interbedded carbonate and shale of the Upper Comus Formation. The mineralized zone has a N-NW orientation, sub-parallel to the Mag Fault, dips to the ENE and plunges to the SSE (McLachlan et al. 2000). The deposit is tabular, has a strike length of approximately 2,500 ft, varies from 200 to 400 ft in width, and ranges in depth from 250 to 300 ft (Kretschmer 1985; Foster and Kretschmer 1991). Higher grade zones are localized along high-angle NW or NE-trending faults (Foster and Kretschmer 1991 cited in McLachlan et al. 2000). Mineralization within the Mag deposit is more disseminated and lower grade than the Range Front, CX, and Ogee zones (Gustavson Associates 2012).

Gold mineralization is spatially associated with decalcification, kaolinization, white kaolinite fracture-filling, silicification, and quartz veinlets (McLachlan et al. 2000). Foster and Kretschmer (1991, cited in McLachlon et al. 2000) report that with the exception of massive limestones, the original carbonate content of the host lithologies was removed during decalcification leaving a porous silty textured rock.

7.5.2 Underground mineralized Zones

Two areas of high-grade gold mineralization at the Pinson deposit are amenable to underground mining methods as shown by previous operators. These include the Range Front-Ogee Zone and the CX Zone. The Range Front-Ogee Zone is located along and adjacent to the range-bounding fault zone and the CX Zone is within the CX open pit that was mined historically.

7.5.2.1 Range Front-Ogee Zone

RFZ mineralization consists of discontinuous occurrences of pervasive argillization and decalcification within host rock lithologies. Silicification is minor with carbonate alteration (calcite) occurring along the borders of fault zones. Karst and dissolution breccias which occur along bedding and structural intersections within the Lower Comus Formation are particularly receptive to mineralization. The Ogee Zone, which is a near vertical, pipe-like shoot occurs at the intersection of the CX-west and Ogee faults. The upper Ogee Zone is characterized by strong iron oxide staining whereas the lower portion of the zone which is hosted by Lower Comus Formation consists of decalcified limestone-siltstone dissolution breccia (Gustavson Associates 2012). Below the 4,650-ft elevation within the Ogee Zone, sulphide mineralization become prevalent. The zone has a strike length of approximately 350 ft, a vertical extent of 600 ft, and averages 30 ft in width.

7.5.2.2 Range Front Fault

The RFF bounds the eastern front of the Osgood Mountains. Mineralization hosted within the RFF has a strike length of 4,000 ft, a down dip extent of 3,000 ft and averages 100 ft in thickness (Gustavson Associates 2012). Higher grade gold mineralization within the zone is discontinuous with strike lengths between 40 to 200 ft and thicknesses varying from 10 to over 60 ft.

7.5.2.3 CX-West Fault Zone

Gold mineralization along the CX-West fault Zone strikes approximately NE, dips steeply to the NNW and has a strike length of approximately 3,000 ft (Gustavson Associates 2012; McLachlan et al. 2000). The mineralized zone averaged approximately 100 ft in width (Gustavson Associates 2012) and occurred primarily along the fault contact between the Upper and Lower Comus formations (McLachlan et al. 2000).

7.5.2.4 Ogee (Linehole) Fault

The Linehole Fault zone consists of two fault strands, the Linehole North Fault and the Linehole Fault (Gustavson Associates 2012). The Linehole North Fault is the extension of the Linehole Fault north of the intersection with the CX-West Fault and the Linehole Fault the extension to the south of the intersection with the CX-West Fault (Gustavson Associates 2012; McLachlan et al. 2000). The Linehole mineralization strikes to the NE, has a strike length of approximately 4,500 and a downdip extent of 1,800 ft (McLachlan et al. 2000). Mineralization averages approximately 15 ft in width.

7.5.2.5 Adams Peak Shear

The Adams Peak Shear is a broad structural zone that strikes to the NE and dips to the north-west (McLachlan et al. 2000). Mineralization within the shear is highly variable consisting of multiple strands within the structural zone. The mineralization has a strike length of approximately 1,500 ft and continues down dip to the intersection with the RFF (Gustavson Associates 2012). The average width of mineralization is approximately 125 ft.

7.5.2.6 Otto Stope Fault

The Otto Stope Fault is located between the CX-West and Linehole faults. The mineralization has a strike length of approximately 2,000 ft and an average thickness of 10 ft (Gustavson Associates 2012).

7.5.3 CX Zone

The CX Zone mineralization can be described as a series of discontinuous occurrences of pervasive argillization and decalcification within karst and dissolution breccias along bedding and structural intersections within the Lower Comus Formation (Gustavson Associates 2012). Silicification is minor and carbonate alteration (calcite) is common along fault zones. Dissolution breccias formed in the CX Zone are structurally controlled and reflect the geometry of the individual faults. The description of the individual structures that occur within the CX Zone are summarized from Gustavson Associates (2012).

7.5.3.1 CX Fault

The CX Fault is a zone of continuous mineralization with a strike length of approximately 4,500 ft and a width ranging between 10 to 100 ft. Mineralization has a down-dip extent of 1,300 ft as defined by exploration drilling.

The following faults either cut or control the orientation of the mineralization in the CX Zone.

7.5.3.2 SOS Fault

The SOS Fault has an average width of 10 ft and a strike length of 1,400 ft and extends down-dip to its intersection with the CX Fault.

7.5.3.3 CX Fault hangingwall splays

The CX Fault hangingwall splays extend between the CX and SOS faults for approximately 500 ft and have an average thickness of 15 ft. They extend down-dip to their intersection with the CX Fault.

7.5.3.4 CX Fault footwall splay

The CX Fault footwall splay has a strike length of approximately 500 ft, averages 20 ft in width and extends down-dip for 750 ft.

7.5.3.5 SOS Dike

The SOS dike has an average thickness of 15 ft, a strike length of approximately 2,700 ft and extends down-dip to its intersection with the CX Fault.

7.5.3.6 SOS Cross Fault

The SOS Cross Fault strikes between the SOS Fault and the SOS Dike for approximately 700 ft, extends down dip to its intersection with the CX Fault, and has an average width of 5 ft.

7.6 Alteration

Alteration assemblages observed at the Pinson deposit include silicification, decalcification, pyrite, and remobilization of carbon. Alteration mapping by Chadwick outlined the distribution of these assemblages within the pits.

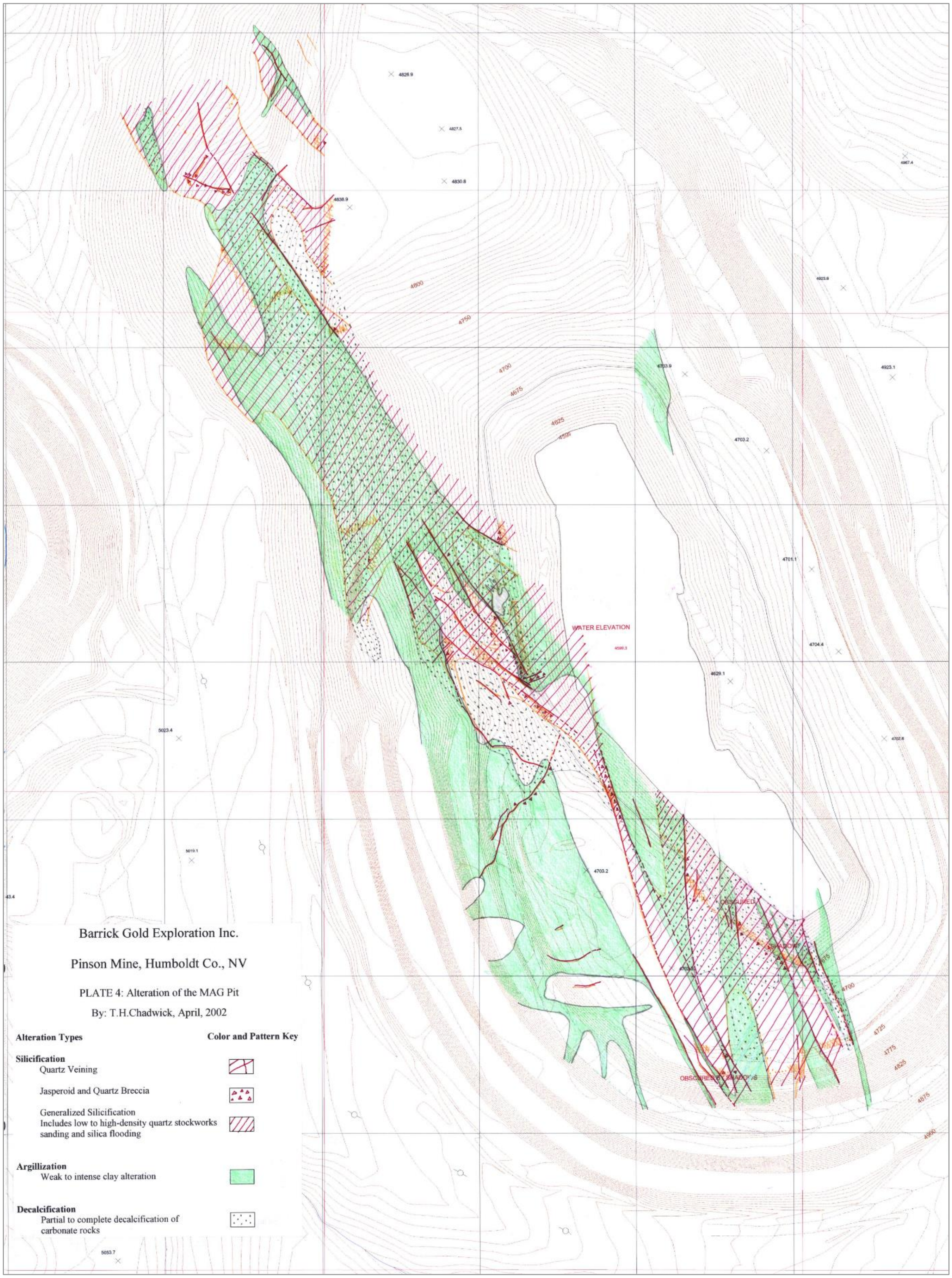
In the CX Zone which follows the strike of the CX Fault and includes the A, B, C, CX, and CX-West pits, McLachlan et al. (2000) documented gradational changes in the style and intensity of observed alteration. In the south-west, within the B Pit, gold mineralization occurs in strongly fractured shale and silty carbonate that has been weakly silicified and clay altered. In the nearby A Pit, alteration consists of intense silicification of carbonate lithologies and formation of gold-rich jasperoid along structures. Gold grains within the jasperoid are typically <5 microns in size and are found as inclusions in arsenian pyrite (McLachlan et al. 2000). Within the C Pit, located NE of the A Pit, higher grades are hosted in decalcified carbonates which have been crosscut by small faults.

Within the CX Pit, mineralization consisted of silica and pyrite replacing carbonate along narrow structures resulting in the formation of intermittent jasperoid and locally silicified wallrock. A large volume of the adjacent hangingwall carbonate-bearing siltstone is decalcified, but barren. Within the CX-West Pit mineralization is hosted in strongly calc-silicate carbonates which exhibit strong argillic alteration.

Mineralization in the Mag Pit is associated with decalcification, kaolinization, white kaolinite fracture-filling, silicification and quartz veining (McLachlan et al., 2000). Except for some massive limestone units, the original carbonate content of the calcareous host lithologies was removed during decalcification resulting in a porous silty textured rock. Silicification occurs as replacement of the decalcified lithologies and healing fault gouge and breccia. Quartz veining and drusy open space coatings are common throughout the deposit. White kaolinite is commonly formed along fractures within the central portion of the deposit and elsewhere occurs as an argillic replacement of the host lithologies (McLachlan et al. 2000). Lithlogy and alteration relationships can be observed in Chadwick’s 2002 pit maps. Chadwick’s alteration map of the Mag Pit is shown in Figure 7.6.

The RFF Zone displays pervasive argillization and decalcification of host lithologies along with the formation of dissolution collapse breccias and intense shearing. Where the alteration is strongest, the altered zones consist of punky, spongy decalcified limestone in an argillically altered fine-grained, carbon-rich matrix (Gustavson Associates 2012 Silicification is minor and occurs as a broad overprint on the zone. Calcite veining is also prevalent along the margins of the RFF.

Figure 7.6 Alteration of the Mag Pit



Source: Chadwick 2002.

8 Deposit types

The structural setting, alteration mineralogy, and mineralization characteristics of the Pinson deposit is consistent with Carlin-type deposits as defined in Radtke (1985) and Hofstra and Cline (2000).

Carlin-type deposits formed in the mid-Tertiary after the onset of extension in an EW trending, subduction-related magmatic belt. The deposits are located along long-lived, deep crustal structures inherited from Late Proterozoic rifting and the formation of a passive margin within Paleozoic carbonate sequences composed of silty limestone to calcareous siltstone. The carbonate sequences are overlain by either structurally controlled siliciclastic sequences controlled by the Early Mississippian-aged Roberts Mountain allochthon or by stratigraphically controlled siliciclastic sequences. The siliciclastic rocks are less permeable than the underlying carbonate rocks which traps fluids along major structures causing them to flow laterally into the permeable and reactive carbonate sequences.

Alteration of host carbonate sequences consists of decalcification, argillization, and selective silicification forming jasperoid and causing carbon flooding. Gangue minerals in Carlin-type deposits consist of calcite, siderite, and ferroan dolomites that can occur as geochemical fronts beyond the mineralized zones.

Gold deposition occurs in arsenian pyrite, is hosted within carbonaceous sequences near major high-angle structural zones and is concentrated in structural traps and / or replacement horizons of reactive and permeable sedimentary beds.

The Carlin-type deposits typically show enrichment in antimony, arsenic, mercury, thallium, and barium, caused by hydrothermal fluids with temperatures up to 300°C. The source of fluids is under debate with two current hypotheses: either regional Eocene magmatism or widespread circulation through Basin-Range extension. Tertiary dikes associated with mineralization and radiometric age dates between 39-42 million years (Ma) provides evidence towards both of the above hypotheses.

Structural pathways, reactive rocks, and sources of heat, gold, sulfur, and iron are required for Carlin-type deposits to form. Large regional structures transecting reactive rocks create contacts, faults, and shears. These secondary structures create pathways and traps for hydrothermal and metalliferous fluids.

9 Exploration

9.1 Introduction

No exploration work has been conducted by the Companies. This section discuss exploration undertaken by previous owners.

Exploration techniques employed on the Property to define additional gold resources have consisted primarily of mapping, geochemical sampling, and drilling. Utilization of these methods has resulted in the discovery of approximately 1 million ounces of gold in several open pit deposits. Several geophysical techniques have also been utilized to aid in the delineation of gold resources, albeit with limited success. The geophysical programs have mostly been applied to exploration programs along strike of known mineralization and as grass-roots applications to locate additional mineralized zones.

Atna became involved in Project planning in July 2004 and began drilling the Property in August 2004 after execution of the earn-in agreement with PMC on 12 August 2004. Atna continued work through April 2006. Atna vested a 70% interest by completing $12M in exploration and development expenditures and completing an NI 43-101 Technical Report of the Project's resources.

9.2 Geologic mapping and geochemical sampling

Cordex, and its successor, PMC, explored the Property through geologic mapping and geochemical sampling. There are three known mapping programs:

- A regional mapping program from the Preble to the Getchell mines conducted in the late 1970's.
- A 1:6000-scale mapping program of the Property in 1983.
- A 1:2400-scale mapping program of the Pinson pit area through the active life of the mine.

Bench mapping in the pits occurred during mining and was followed up by detailed 1:1200-scale mapping of the A, B, C, CX, MAG, CXW, and Blue Bell pits by Tom Chadwick starting in 2000, after mining ceased. These maps were completed under the Homestake / Barrick partnership agreement.

Several geochemical programs were also completed by Cordex and PMC during the active mine life of the Pinson Mine, and by Homestake. These included programs:

- Cordex took rock chip samples in conjunction with mapping programs. A total of 737 rock chip samples were collected. Samples were assayed for gold, silver, arsenic, antimony, and mercury. Select samples were also analyzed for lead, zinc, copper, and manganese. The combined mapping / sampling programs were responsible for the discoveries of the Blue Bell and Felix Canyon deposits (Thompson et al. 2000, cited in Sim 2005).
- PMC completed six float chip geochemical grids consisting of 8,756 samples. These grids covered the MAG deposit, and along strike south of the A and B Pits.
- A biogeochemical sagebrush sampling program was conducted in the 1990's with inconclusive results.
- Under the Homestake / Barrick JV, an additional 312 rock samples and 273 soil samples were collected. These programs were completed on strike south of the existing pit areas, and west of the A, B, C, and CX Pits.

9.2.1 Osgood Mining geologic / structural mapping

In 2016, OMC contracted Mr Robert Leonardson to complete a geological study on the Property that focused on advancing OMC's understanding of the structural framework and to provide guidance on exploration targeting. This work included structural and geologic mapping of the open pits and underground exposures, construction of Property wide cross sections and report writing that included the identification of exploration targets on the Project.

He concluded that potential targets to discover additional gold mineralization are at intersections of the E-dipping N – S faults (Range Front / Mag) with the SE-dipping CX-type faults. Other areas include the intersection of the sub-vertical NW-striking faults with the CX-type faults. Examples of the first type are the CX hanging wall splays where they intersect the Mag Fault in the north half of the Mag Pit. The second example is exemplified by the intersection of the Bluebird Fault Zone with the Delaney thrust in the Blue Bell East pit, and the intersection of the Bluebell 2 Fault with the CX thrust in the CX B Pit. Zones of limestone decarbonization such as seen in the CX Pit are also potential hosts for gold mineralization. These zones indicate strong fluid / vapor flow through the rock mass. Specific areas for exploration include:

- The intersection of the SOS and JP dikes on the south wall of the CX Pit. This area contains the largest block of decarbonization on the Property, and the hydrothermal alteration may represent an "exhaust plume" emanating from depth.
- The Ogee pipe extension located between 1,500 ft to 1,800 ft below the CX-C Pit. A historical hole, HPC-070A intersected a 760-ft interval of low to moderate gold grades above 3,160 ft and high-grade mineralization from 3,160 ft to 3,130 ft near, and just south of the proposed Ogee high-grade down-dip extension.
- The northern continuation of the fault propagated anticline in the western portion of the Mag Pit between the Mag Fault and CX Fault, and to the north of the Mag Pit. The anticline steepens to the south and the best chance to intersect high-grade mineralization would be at the intersection with the Disturbed Fault.
- The intersection of the Adam Peak Fault and the Mag Fault suite north of the Mag Pit.
- The CX-B Pit decarbonatization zone at the intersection of the CX and Bluebell 2 faults on the west limb of the Pinson anticline.
- The Mag Pit decarbonatization on the west wall along a section of the Mag Fault intersection with the CX and HW faults and the Disturbed Fault.
- The Mag Pit decarbonatization on the west wall along a portion of the Mag Fault intersections with the Disturbed Fault.
- Bluebell east pit decarbonatization at the intersection of the Bluebell and Delaney faults.
- Traps and fault intersections along the NNW-trending Mag Fault suite and the NE-trending CX type faults.
- Flat to ramp traps down dip extension of fluids that mineralized the Bluebell, CX(?) between South Mountain Fault, and the southern Mag suite of faults. Flat to ramp traps along the Adam Peak detachment and subsequent faults (CX, Disturbed, and South Mountain).

9.3 Geophysical surveys

Numerous geophysical surveys have been conducted on the Property. These include both regional and detailed surveys. The regional surveys included gravity and aeromagnetics. Detailed surveys involved mostly electromagnetic techniques and included Induced Polarization (IP), Electromagnetics (EM), Magneto Tellurics (MT), and Controlled Source Audio-frequency Magneto Tellurics (CSAMT) surveys. A summary of these techniques includes:

- Airborne EM and magnetics by the USGS at quarter-mile line spacing throughout much of the Getchell Trend.
- Ground-based magnetics over the CX Zone completed in 1970 by Cordex.
- Regional gravity surveys, both public and private, compiled by Homestake in 1997.
- Ground-based magnetic survey at the north edge of the Mag Pit completed in 1998 by Homestake.

- Several generations of AMT (EM, IP, CSAMT) completed by PMC.
- Several CSAMT lines were completed by Homestake between 1998 and 2000.
- Several EM lines were completed by Homestake in 2000.
- A detailed gravity survey over the Property was conducted by Magee Geophysical Services, LLC of Reno, Nevada in October 2006. A total of 2,587 gravity readings were acquired using a 100 m station spacing. The results, which were interpreted by Barrick in 2007 were used to target exploration drilling in 2007 and 2008.

9.4 Underground drifting / evaluation

A small exploration drifting program was conducted on the upper "B" zone by Cordex in the 1970's to conduct bulk testing. Results from this program are unavailable.

In May of 2005, Small Mine Development (SMD) of Boise, Idaho was contracted by Atna to drive exploration drifts, crosscuts, and develop drill stations to complete Atna's evaluation of the Range Front resource area. Both the Range Front and CX resource areas were of interest in Atna's program.

The underground development work completed 1,988 ft of 14-ft by 16-ft adit, 378 ft of decline, and six diamond drill stations (Gustavson Associates 2012). A small minability test was also carried out on the newly defined Ogee Zone to evaluate the potential conditions for future stoping. Approximately 400 tons of material were extracted during this test. The results indicated the possibility of drift and fill as a potential mining method.

During 2008, approximately 693 ft of development drifting was completed, and significant geological data recorded in the RFZ. However, no data on ground conditions was acquired. This data was not collected since it was anticipated that ground conditions would be similar to those encountered at the Getchell Mine, and mineralization would be exploitable by underhand drift and fill stoping methods (Gustavson Associates 2012).

9.5 Trenching and sampling

Atna channel sampled 14 ribs in the Ogee Zone and sent 74 rib and face samples out for assay. Salient results are summarized in Table 9.1. Assays from the samples indicated that no ore-grade mineralization was encountered except where the main drift intersected the Ogee Zone on the 4770 elevation (Gustavson Associates 2012).

Table 9.1 Salient results of the Ogee Zone channel sample assays

Sample No.	From (ft)	To (ft)	Length (ft)	Gold opt (grams/tonne)
North rib				
RFUG-055	76	81	5	0.144 (4.94)
RFUG-056	81	85	4	0.445 (15.26)
RFUG-059	85	88	3	0.274 (9.39)
RFUG-061	88	93	5	1.448 (49.65)
RFUG-063	93	97	4	0.176 (6.03)
RFUG-064	97	101	4	0.739 (25.34
RFUG-067	101	110	9	0.996 (34.15)
Weighted average			**34**	**0.682 (23.38)**
South rib				
RFUG-081	77	80	3	0.106 (3.63)
RFUG-082	80	83	3	0.065 (2.23)
RFUG-083	83	93	10	1.082 (37.10)
RFUG-084	93	96	3	0.894 (30.65)
RFUG-086	96	99	3	0.355 (12.17)
RFUG-087	99	107	8	0.028 (0.96)
RFUG-088	107	112	5	0.228 (7.82)
Weighted average			**35**	**0.470 (16.11)**

Source: Edmondo et al. 2007.

10 Drilling

10.1 Drilling campaigns

10.1.1 Overview

Numerous holes have been drilled in and around the Property prior to 1970. Unfortunately, this drillhole data is no longer available. Since 1970, a total of 2,083 drillholes totaling 955,747.9 ft have been drilled within the Property area. Figure 10.1 shows the drilling by each operator and significant time period. PMC, and its predecessors, Rayrock Mines and the Cordex Syndicate, account for most of these holes; 1,434 holes totaling 554,435 ft. Homestake drilled 165 holes totaling 160,207.7 ft and Barrick drilled 166 holes totaling 122,031.1 ft. Both companies acted as operators for PMC. Atna, the last company to operate at the Pinson mine, drilled 318 holes totaling 119,074.1 ft. Table 10.1 presents the summary of drilling at the Property.

Table 10.1 Summary of drilling on the Property since 1970

Company	Surface RC		Surface core		UG RC		UG core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)	# holes	Footage (ft)	# holes	Footage (ft)		
PMC	1,426	546,313.0	8	8,122.0					1,434	554,435.0
PMC (Homestake)	136	108,335.0	29	51,872.7					165	160,207.7
PMC (Barrick)	39	35,645.0	67	65,700.1	4	930.0	56	19,756.0	166	122,031.1
Atna	29	18,672.0	65	52,847.6	176	32,068.0	48	15,486.5	318	119,074.1
Total	**1,630**	**708,965.0**	**169**	**178,542.4**	**180**	**32,998.0**	**104**	**35,242.5**	**2,083**	**955,747.9**

Note: RC=reverse circulation, UG=underground.
Source: Osgood Mining Company LLC.

Each period of drilling is described in further detail in Sections 10.1.2 to 10.1.10.

10.1.2 PMC drilling 1970 to 1996

Many holes drilled by PMC during this time period were development holes drilled in and adjacent to exiting pits. Over 1,400 holes were drilled within the A, B, C, CX, Mag, CX West, Felix, and Blue Bell pit areas. Many of these holes were drilled vertically and all but eight were either conventional rotary or RC. The eight core holes that were drilled (8,122 ft) were in the B, C, CX, and Mag Pit areas to test stratigraphy, metallurgy, or deep mineralized structures (Golder 2014). Table 10.2 summarizes the drilling PMC conducted through 1996.

Table 10.2 PMC drilling through 1996

Company	Surface RC		Surface core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
PMC	1,426	546,313.0	8	8,122.0	1,434	554,435.0
Total	**1,426**	**546,313.0**	**8**	**8,122.0**	**1,434**	**554,435.0**

Source: Osgood Mining Company LLC.

10.1.3 PMC – Homestake drilling 1997 to 2000

Between 1997 and 2000, 165 holes were drilled by Homestake, as the operator for PMC as shown in Table 10.3. Of the 165 holes drilled, 136 (108,335 ft) were directed into the CX and RFF system.

Table 10.3 Homestake drilling

Company	Surface RC		Surface core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
PMC (Homestake)	136	108,335.0	29	51,872.7	165	160,207.7

Source: Osgood Mining Company LLC.

10.1.4 PMC – Barrick drilling 2003

Four exploration holes were drilled by Barrick, operator at the time for PMC, to test extensions of the CX Fault Zone near its projected intersection with the Mag Pit fault system. The drilling did not identify significant mineralized zones and no additional work was conducted by Barrick (Golder 2014). Table 10.4 shows a summary of the Barrick drilling.

Table 10.4 Barrick drilling 2003

Company	Surface RC		Surface core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
PMC (Barrick)	3	3,340.0	1	3,003.3	4	6,343.3

Source: Golder Associates 2014.

10.1.5 Atna drilling 2004

The drilling by Atna in 2004 followed up on mineralized zones previously identified by PMC and Homestake. Thirty-one holes, totaling 29,739.5 ft were drilled. These holes were comprised of four RC holes (2,217 ft) and 27 core holes totaling 27,522.5 ft (Table 10.5). This drilling program had five objectives:

- Improve the grade and thickness of mineralized zones, especially in areas where drilling consisted of only RC drilling.
- Infill drilling, especially where previous drill spacing was greater than 400 ft.
- Expand mineralized zones both laterally and down-dip.
- Obtain rock quality data on hangingwall, footwall, and mineralized zones.
- To evaluate previously identified targets.

Table 10.5 shows a summary of the Atna drilling.

Table 10.5 Atna drilling 2004

Company	Surface RC		Surface core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
Atna	4	2,217.0	27	27,522.5	31	29,739.5

Source: Osgood Mining Company LLC.

Of the 31 holes drilled, 13 holes (13,000 ft) were drilled into the CX Fault Zone and 18 holes (16,739.5 ft) were drilled into the RFF Zone (Golder 2014).

10.1.6 Atna drilling 2005 – 2006

The objective of the 2005 / 2006 drilling program was to define and delineate Measured and Indicated gold Mineral Resources in the upper portions of the RFF Zone where Atna had outlined a 1,000-ft long by 200- to 500-ft thick mineralized zone during its 2004 drilling program. The drilling program was designed to test the upper RFZ between the 5,000- and 4,400-ft amsl elevations

(Golder 2014). The program used both surface and underground drilling to delineate the zone. A total of 107 drillholes (55,180.1 ft) were drilled between 2005 and 2006 (Table 10.6).

Table 10.6 Atna drilling 2005-2006

Company	Surface RC		Surface core		UG core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)	# holes	Footage (ft)		
Atna	25	16,455.0	34	23,238.6	48	15,486.5	107	55,180.1
Total	**25**	**16,455.0**	**34**	**23,238.6**	**48**	**15,486.5**	**107**	**55,180.1**

Source: Osgood Mining Company LLC.

Surface drilling began in May of 2005. The majority of these holes were core holes which were pre-collared via RC drilling and completed with core drilling. Fifty-nine (59) drillholes, totaling 39,693.6 ft of drilling, were completed from surface.

Underground drilling began in September of 2005 after drifting was completed and underground drill rigs became available. In total, 48 holes aggregating 15,486.5 ft of underground drilling were completed in the Ogee, CX West, and Range Front targets.

10.1.7 PMC (Barrick) drilling 2007

In August of 2007, surface exploration and development drilling began using an Eklund RC drill rig and a Major Drilling core rig. Targets tested included portions of the CX and RFF, Ogee Zone, and the HPR104 area. The HPR104 area is north of the Pinson mine.

Twenty-three (23) surface holes (18,916.2 ft) were completed during the latter part of 2007 as shown in Table 10.7. The results of the drilling were disappointing in that only thin, sub-economic zones of underground mining gold grades were intersected.

Table 10.7 PMC - Barrick drilling 2007

Company	Surface RC		Surface core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
PMC (Barrick)	7	4,935.0	16	13,981.2	23	18,916.2

Source: Osgood Mining Company LLC.

10.1.8 PMC (Barrick) 2008 drilling

Surface drilling began in January of 2008 with three core drills and one RC drill testing areas north of the CX West pit. The core drilling was focused on completing holes pre-collared by RC drilling in 2007 and testing the deep potential of the Getchell Fault system north of the Pinson Mine which had associated gravity and MT anomalies (Golder 2014). RC drilling was primarily focused on pre-collaring holes for follow up core drilling north of the CX / CX West pits. Surface core drilling was completed in April of 2008. RC drilling continued throughout 2008 with the focus on drilling pilot holes for potential dewatering well locations.

Underground exploration began in April 2008 as discussed in Section 9.4. SMD was contracted to rehabilitate existing underground workings, and drive exploration headings into the Ogee and CX zones. SMD supplied an underground RC drill for closely spaced definition drilling, and Connors Drilling was contracted to conduct underground core drilling. The SMD contract was terminated in May of 2008. Connors Drilling remained onsite and brought in a second underground core rig in mid-July. Both core rigs continued operation through mid-December testing the Ogee Zone and conducting widely spaced drilling within the RFZ.

In August 2008, a second surface drilling program was initiated to twin RC holes in key areas of the resource suspected of having downhole contamination. Two core rigs and one RC rig (to pre-collar holes) were utilized. A third surface core rig was also brought in to complete one deep hole to test the Mag fault-Delaney fault intersection south of the resource area. The drilling program was completed in mid-December and all drilling equipment removed from site.

During 2008, total surface drilling included 29 RC holes totaling 27,370 ft and 50 core holes totaling 48,715.6 ft. Underground drilling included 4 RC holes for 930 ft and 56 core holes totaling 19,756 ft (Table 10.8).

Table 10.8 PMC – Barrick drilling 2008

Company	Surface RC		Surface core		UG RC		UG core		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)	# holes	Footage (ft)	# holes	Footage (ft)		
PMC (Barrick)	29	27,370.0	50	48,715.6	4	930.0	56	19,756.0	139	96,771.6

Source: Osgood Mining Company LLC.

10.1.8.1 HPR104 area

During the 2008 drilling program, eight holes were drilled north of the Pinson deposit resource area. These holes were designed to twin earlier PMC drilling which were drilled to test the intersection of the Range Front and Linehole Faults. The results of the initial drilling could not reproduce the thick low-grade intercept identified in an earlier hole, hole HPR104. This was considered to constitute downhole contamination in hole HPR104 and the hole was removed from the database. A second round of core drilling did intersect thin, higher-grade mineralization. Hole BPIN-008 intercepted 21.5 ft grading 0.620 opt at a depth of 1,378 ft (Golder 2014). This mineralization appeared to be structurally controlled by the intersection of the Linehole Fault and the Upper / Lower Comus contact 900 ft NE of the main portal.

10.1.8.2 Deep exploration targets

Two deep drillholes, BPIN-010C and BPIN-011A, were drilled in 2008. Hole BPIN-010C was drilled to a depth of 2,845.5 ft and was designed to test the Lower Comus Formation adjacent to structures identified from a 2006 gravity survey (Golder 2014). The hole bottomed in Upper Preble Formation and assay results proved negative. Hole BPIN-011A was drilled to a depth of 2,778 ft and ended in argillite and shale of the Upper Comus (Golder 2014). The hole was designed to test the projected intersection of the Mag and Delaney faults. Analyses of chip samples indicated a 60-ft zone of low-grade gold (0.029 opt) at 1,440 hosted in silicified Upper Comus claystone and shale (Golder 2014). Subsequent analyses of core from the entire hole indicated narrow zones of mineralization associated with decalcified and pyritized sediments.

10.1.9 2012 Atna Mag Pit core drilling

In 2012, Atna completed four PQ-size core holes, totaling 2,086.5 ft, to acquire samples for column leach testing from mineralized material within the Mag Pit resource area. The holes were drilled along strike of the known mineralized zone with each hole intersecting potential high-grade material. In addition to the metallurgical holes, and additional 56 underground exploration RC holes totaling 7,495 ft were drilled in the Ogee Zone. Table 10.9 summarizes the drilling conducted by Atna in 2012.

Table 10.9 Atna drilling 2012

Company	Surface core		UG RC		Total holes	Total footage
	# holes	Footage (ft)	# holes	Footage (ft)		
Atna	4	2,086.5	56	7,495.0	60	9,581.5

Source: Osgood Mining Company LLC.

10.1.10 2013 – 2015 Atna Underground development RC drilling

Between 2012 and 2015, Atna completed 120 underground RC holes totaling 24,573 ft (Table 10.10). These holes were designed to confirm continuity of mineralization and to delineate stope configuration within the Ogee Zone for mining.

Table 10.10 Atna drilling 2013 – 2015

Company	UG RC		Total holes	Total footage
	# holes	Footage (ft)		
Atna	120	24,573.0	120	24,573.0

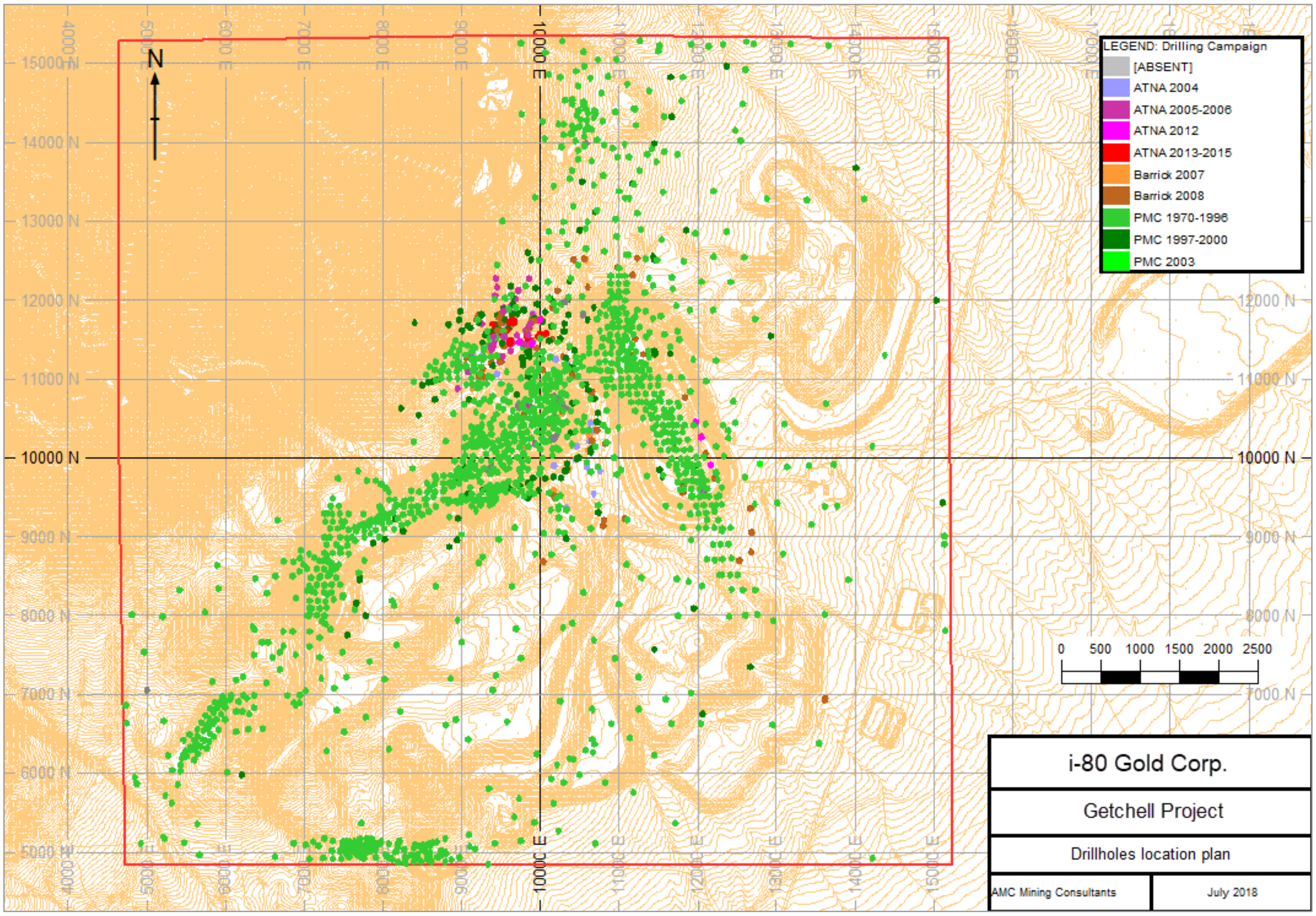
Source: Osgood Mining Company LLC.

10.2 Representative drill sections and plan

Figure 10.1 shows the drill plan of the Property in the area of the current Mineral Resource, shown by a red outline. The drillholes are coded by operator and significant time periods. Figure 10.2 shows a plan view with section lines of the Open Pit area. Figure 10.3 to Figure 10.6 shows representative vertical sections through the four Open Pit areas. Figure 10.7 shows a vertical section through the underground resource area.

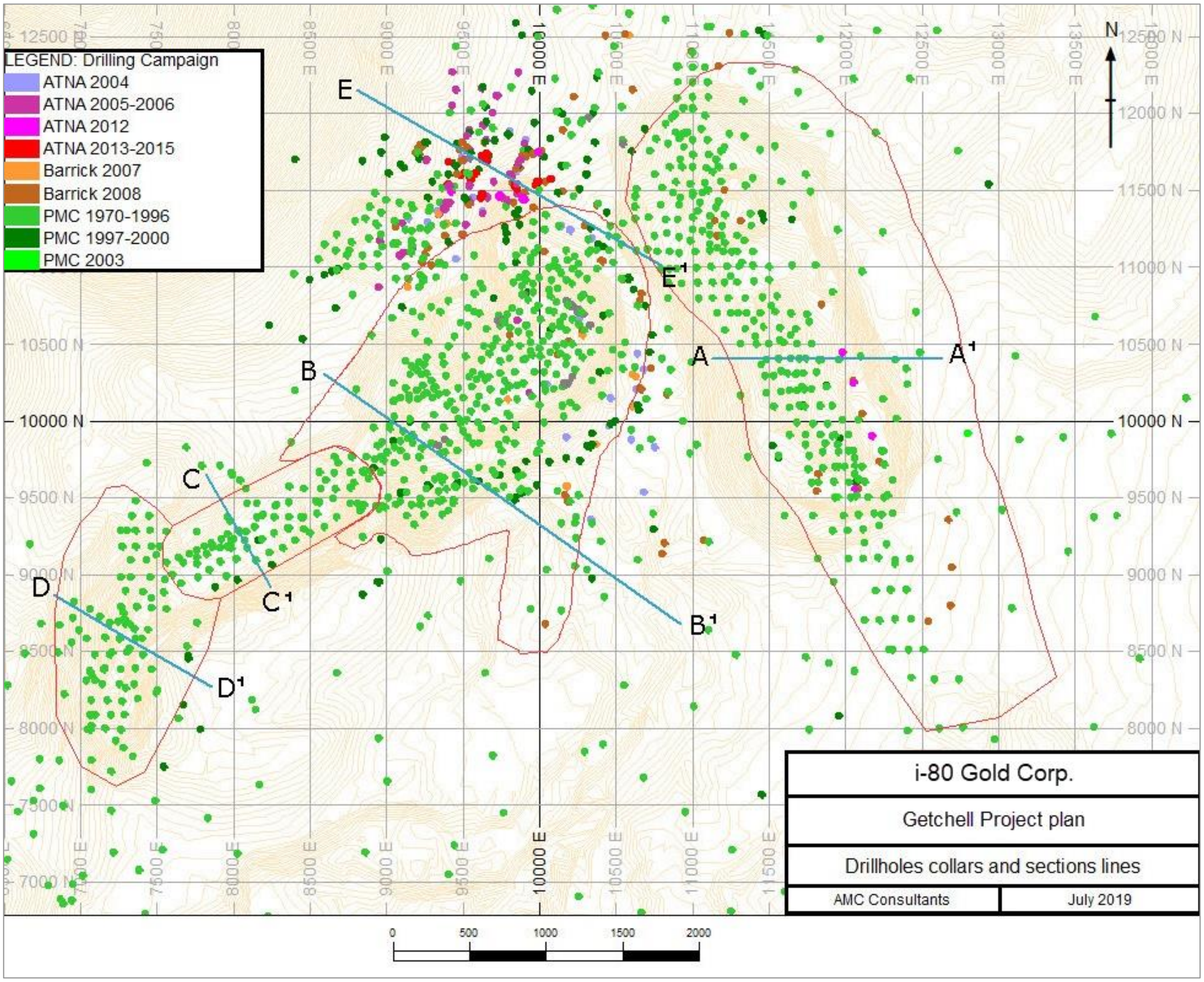
All drill results are from previous operators. the Companies has conducted no drilling on the Property.

Figure 10.1 Drill plan by operator



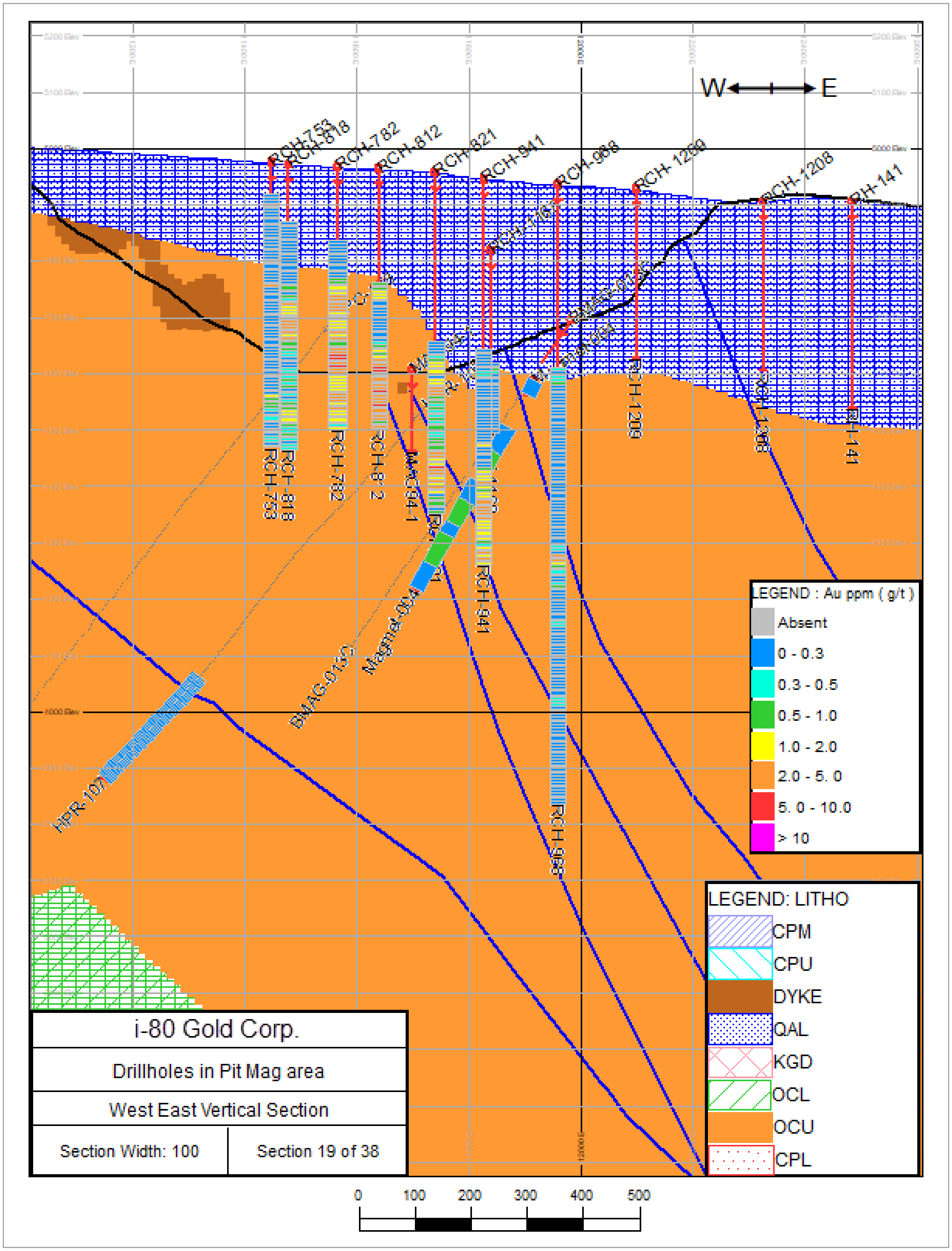
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.2 Plan view sections lines of Getchell Project



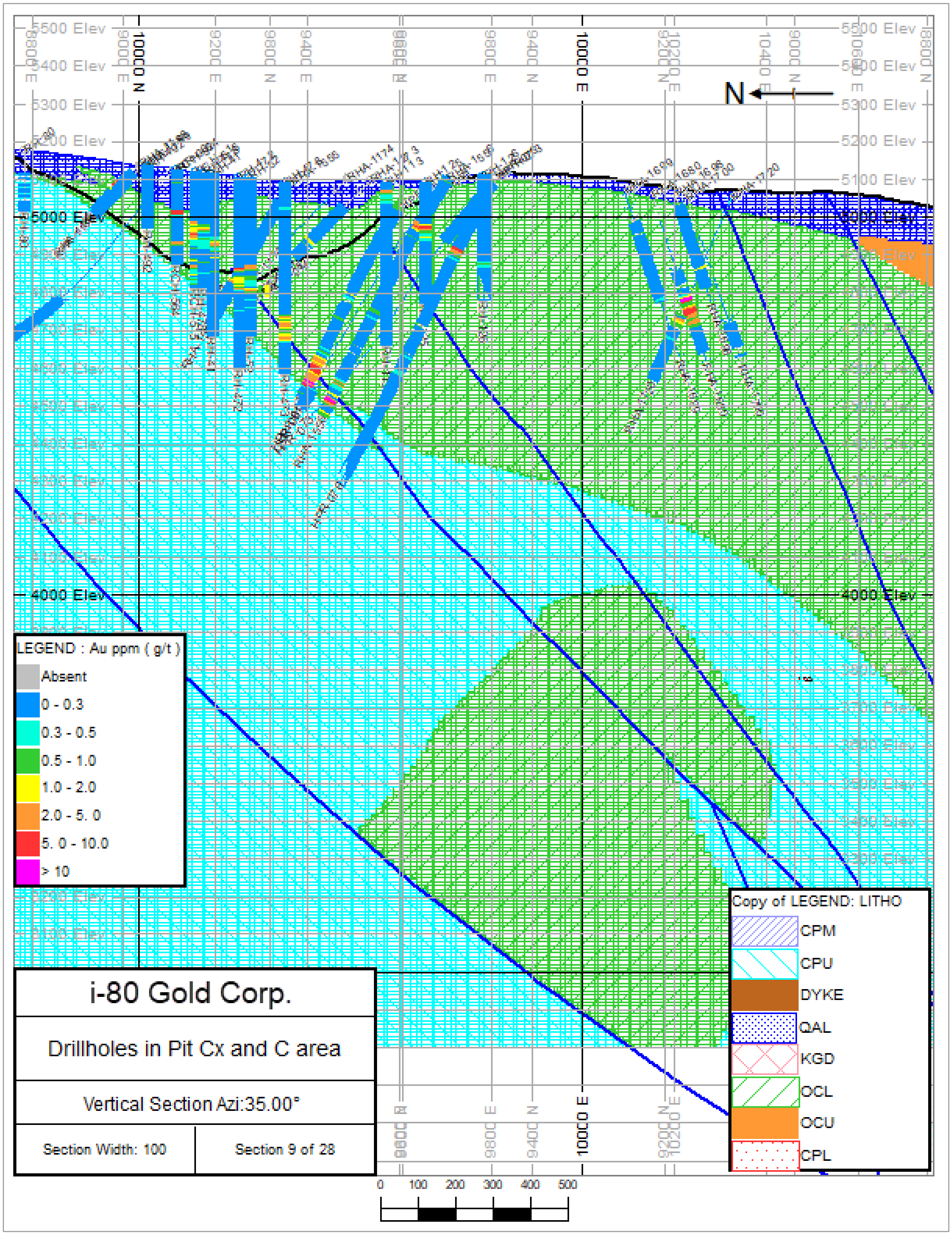
Note: Red outlines show the outline of the open pits.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.3 Vertical section A-A[1] of the Mag Pit area



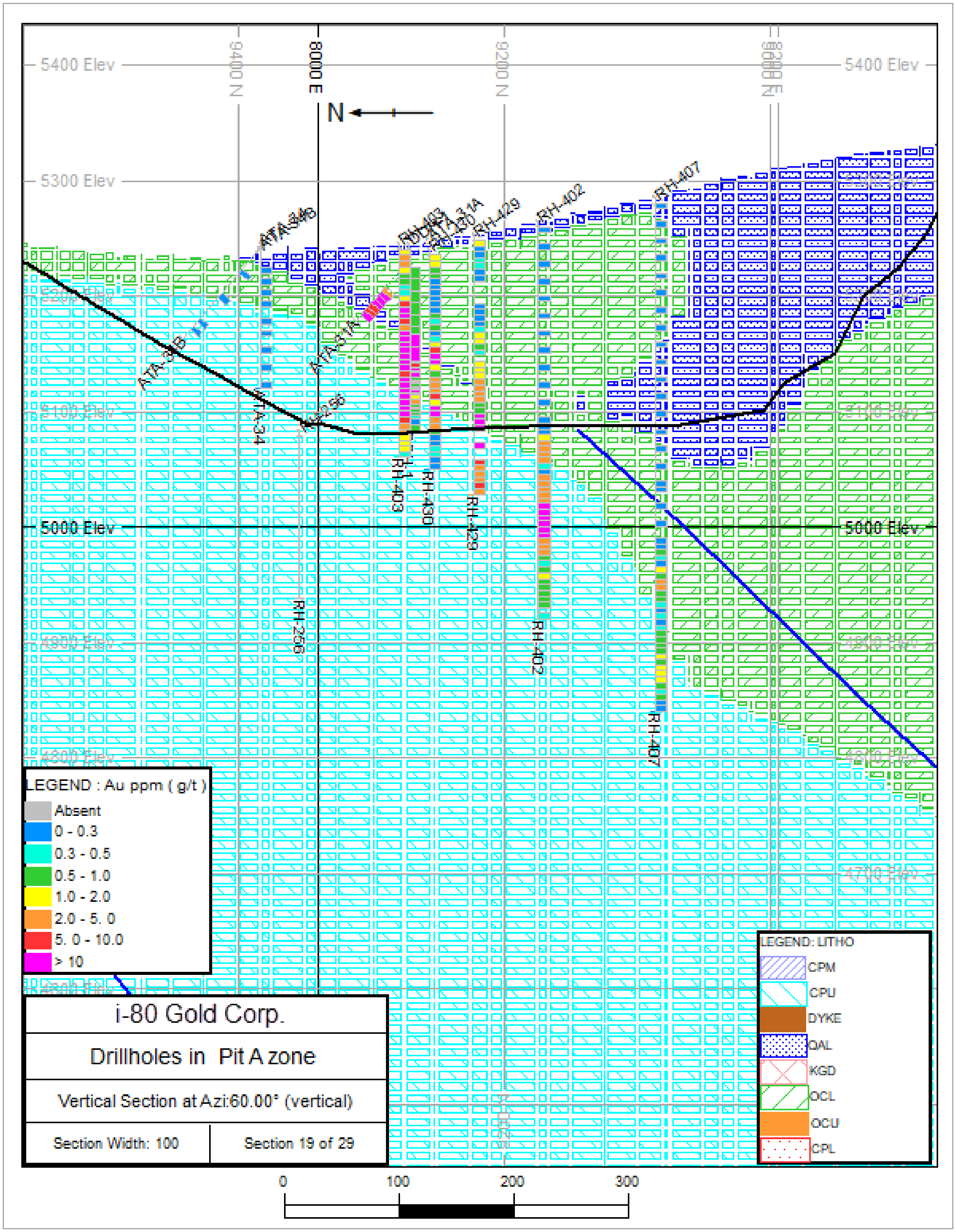
Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.4 Vertical section B-B[1] of the Pit CX and C area



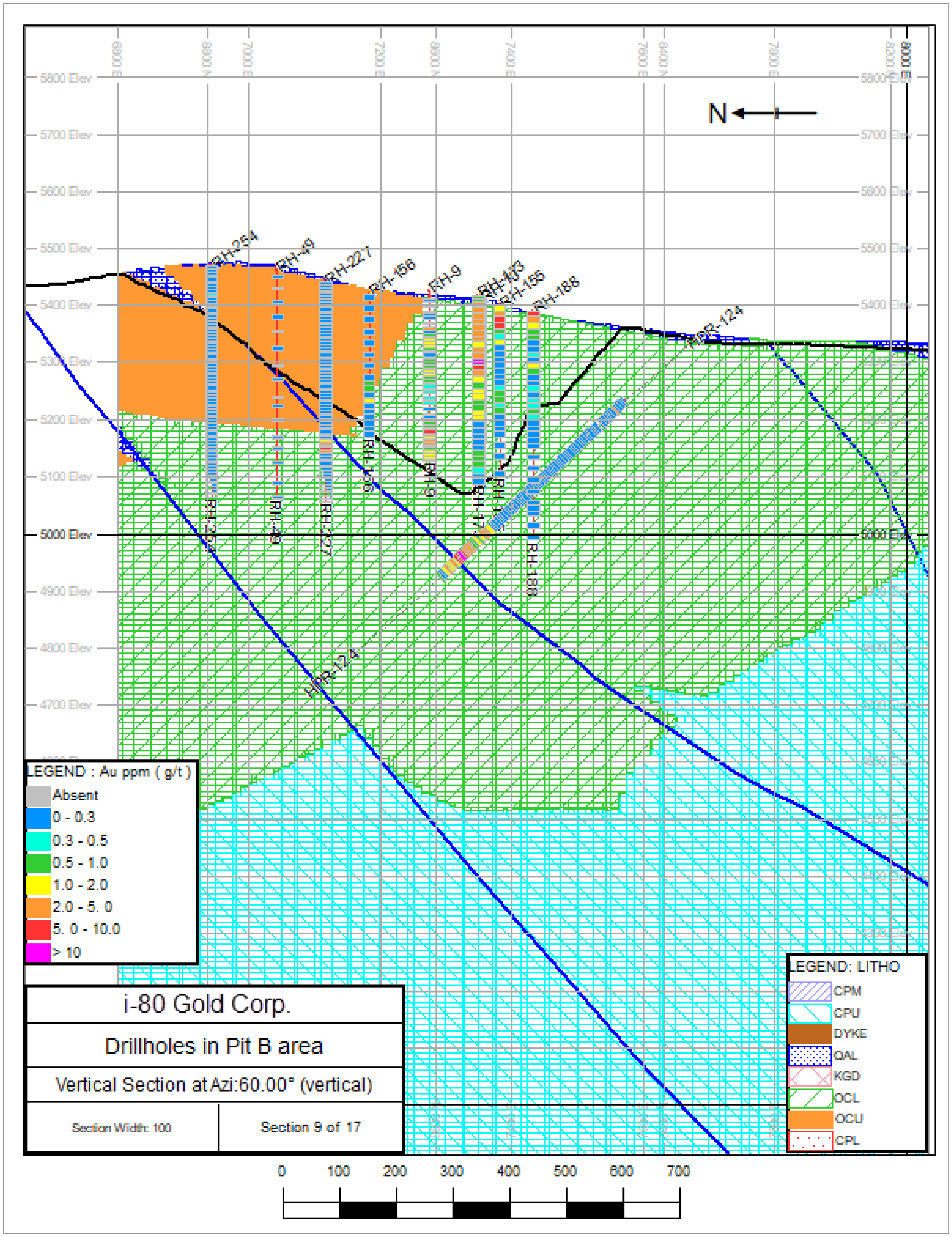
Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.5 Vertical section C-C[1] of the Pit A area



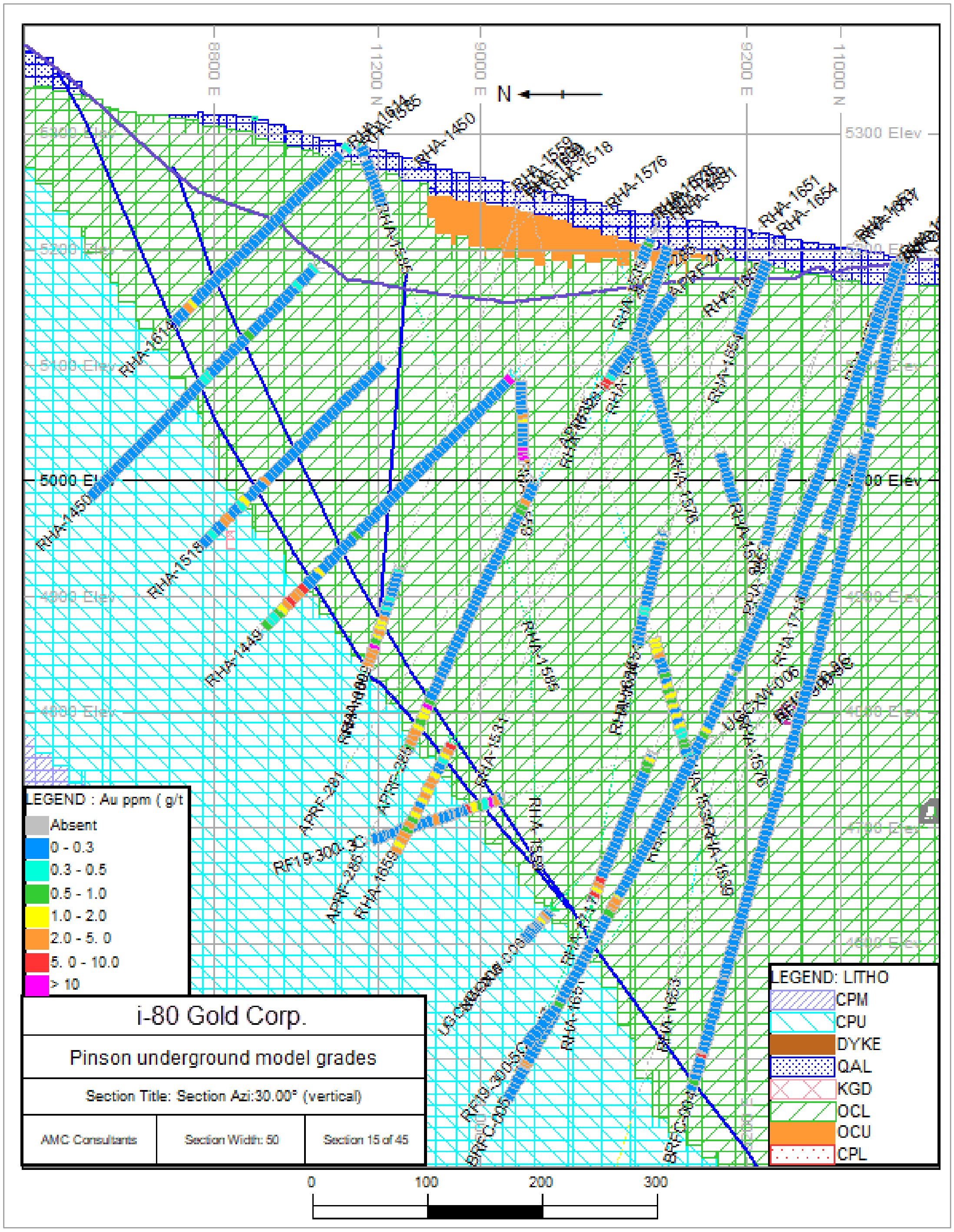
Notes: Blue line is a fault. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.6 Vertical section D-D[1] of the Pit B area



Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

Figure 10.7 Vertical section E-E[1] of the Underground resource area



Notes: Blue lines are faults. Top blue line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd. 2019.

10.3 Drilling, sampling, and recovery factors

There are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results. Drilling has been discussed in this section and sampling and recovery factors are discussed in Section 11.

11 Sample preparation, analyses, and security

11.1 Sampling methods and approach

Drilling at the Property used both surface RC and core drilling along with underground core drilling. The RC drilling was used primarily to pre-collar holes to bedrock followed by core drilling. This was done to minimize costs by not core drilling through unmineralized material overlying the mineralized fault zones. Core drilling provides a higher confidence in sample quality versus RC drilling along with providing additional data for engineering studies and detailed geologic definition of structurally controlled high-grade mineralized zones.

The primary objective of the drilling programs was to collect clean, uncontaminated representative samples that are correctly labeled when drilled and logged, and that can be accurately tracked from the drill rig to the assay laboratory. Both Atna and PMC (Barrick) Exploration used similar sampling and analytical protocols.

11.2 Reverse circulation drilling

11.2.1 Sampling methods

In this drilling method, cuttings produced by the bit are sent up the drill pipe into a cyclone at surface where the sample is homogenized prior to collection. From the cyclone, the sample is processed through a rotary splitter that takes a representative split of the sample (usually a quarter split), sending a split portion to the sample port with the remainder to the reject port. Samples are placed into 10-by-17-inch sample bags that have been clearly labeled with the drillhole number and a unique numbering sequence prepared beforehand using a spreadsheet. This spreadsheet helps in tracking bag numbers, footages drilled, and quality control samples. A representative sample of each interval drilled is also preserved in chip trays that are clearly labeled with the hole number and drill interval for future reference.

11.2.2 Recovery

Sample recovery for RC drilling is measured by weight of material collected which is usually eight to ten pounds of material from the quarter split in a typical six-inch diameter hole. Historical RC sample recovery was excellent. Full five to ten-pound bags of sample were collected from every interval. The only exception were 15 samples out of 6,100 that were collected by Atna. The missing samples occurred in an isolated zone of badly broken ground.

11.2.3 Sample intervals

Typical truck-mounted RC drill rigs use 20-ft drill rods with samples collected in five-ft intervals. Both Atna and PMC utilized this sampling procedure in their drilling programs.

For each RC hole drilled, the drill crew was provided with a sequentially numbered set of sample bags. The outsides of the bags were marked with the drillhole number and a sample number.

To ensure that blanks and standards were inserted into the sample stream correctly (every tenth sample), several steps were taken. First, the sampler was provided with chip trays that were labelled with both the true footage and the corresponding bag number. Second, he was provided with an incompletely labeled set of sample bags which did not include bags for the standards or blanks. Third, since the total depth of the hole was not known prior to drilling, bags for duplicate samples (collected every 100 ft) were labeled with the letters "A", "B", "C", etc. and flagged with a tear-off paper tag.

Samples were allowed to drain / dry at the sample site which was routinely visited by the geologist in charge of the drill program to ensure accurate numbering of the sample suite. Once drained and / or dried, the samples were re-located from the drill site to the shipment staging area where personnel relabeled the bags containing the duplicate samples by assigning the correct sequential number. This ensured that they were "blind" to the laboratory personnel. The samples were then loaded into 4 x 4 x 3 ft wooden crates in preparation for pickup by the lab.

11.2.4 Logging

Representative rock chips for each five feet run were collected in clearly labeled 20 compartment plastic chip trays. These trays were taken to the logging trailer where the geologist logged the chips with the aid of a binocular microscope. The geologist recorded lithology, mineralization, alteration, and other pertinent features on a paper drill log. A schematic graphic log was also produced to aid in interpretation of the stratigraphic sequence.

11.3 Diamond drilling

11.3.1 Sampling methods

At the drill site, the drill crew was responsible for obtaining a complete and representative sample of the cored interval. This interval is usually five feet in length but may be shorter dependent on how difficult the ground conditions are. Core is recovered from the core barrel via a wire line core tube which may be outfitted with an inner "triple-tube".

For core obtained using a triple-tube system, the core was placed on a rack and the drill crew recorded rock quality determination (RQD) values on a worksheet and photographed the core. For holes drilled with conventional core barrels, RQD values were recorded later by a geologist from the core in the box.

At the drill site, once the RQD values were recorded and the core photographed, the drill crew placed the core in waxed cardboard boxes that were labeled with the company name, Property, hole ID, box number, and from-to footage. Core boxes were partitioned in five, two-foot long sections totaling 10 ft in length. As core is drilled, it was placed in the core boxes in sequential order from top of the run to bottom of the run. A wooden block was inserted at the end of each run, and at the driller's discretion, to indicate problems with drilling, such as caving, voids, or core tube mismatches. The last block of each run was marked with the ending footage on the thin edge of the block, and two numbers on the larger surface.

If the core was not photographed for RQD purposes, the drillers marked the breaks they made to fit the core into the core boxes with the letter "M" on each side of the break, so it was not counted in the RQD analysis. After boxing, each core box was securely closed with elastic banding and loaded into the driller's vehicle for transport to the logging area, at which point it was unloaded and logged.

11.3.2 Recovery

Core recovery is measured by the ratio of the length of drill core recovered versus the length of the drilled run and is expressed in percent. Core recovery was excellent with greater than or equal to 99% core recovered (Golder 2014). Where core loss was recorded, it amounted to less than two feet in zones where voids were present in the stratigraphy.

11.3.3 Sample intervals

Once the core was logged, the geologist determined the sample intervals to be sent to the laboratory. The geologist adhered to a set of guidelines to better define boundaries between mineralized material and barren samples. Original core blocks, inserted by the driller to mark the end of a drill run, served as the primary sample boundary, subject to the rules below; where a conflict existed between the inserted core blocks and the guidelines, the guidelines prevailed, and extra blocks were inserted by the geologist to compensate:

- A sample must not cross a geologic contact.
- A sample must not cross an obvious alteration boundary, including oxidation.
- A sample must not exceed seven feet long, and only be that long if it occurred in barren material, with five feet samples being the optimum.
- Any core blocks that do not mark a sample boundary, for whatever reason (such as "cave", "loss", "void", etc.) must be labelled in black marker for photographic visibility.

Each block that marked a sample boundary was outlined or highlighted in red marker, and the interval boundaries entered into a sample sequence log. Sample intervals generally ranged from 1 to 6 ft in length and averaged 4.6 ft.

During the core sampling process, the sampler was provided with the geologic core log and the sample sequence to allow the sampler to have a better understanding of why and how the sample boundaries were picked, and to act as a check on the geologist's accuracy.

The condition of the rock and whether it was mineralized or not dictated the splitting method of the core. Unmineralized rock was split with a hydraulic splitter. Mineralized and silicified intervals were sawn with a water-cooled diamond-bladed rock saw. Mineralized un-silicified was also typically sawn, but in some instances split with the hydraulic splitter. Broken mineralized core was separated and divided into two equal portions.

To avoid sampling bias, whenever possible, the core was sawn or split perpendicular to the trace of visible bedding. The portion of the core to be saved was placed in the core box in its original position with the core blocks in place, and the box rubber banded for additional security. The sampled half of the split core was bagged, and the bags placed in 4 x 4 x 3 ft wooden crates for shipment to the laboratory. The remaining core was palletized, covered with tarps, and moved to industrial shelving on an outdoor cement pad for storage and reference. It is unknown of this storage facility was secure.

11.3.4 Logging

Once the core was received at the logging facility, it was arranged sequentially from top of the hole to bottom of the hole.

Data captured on paper drill logs included footage of the core runs, lithology, alteration, major structural features, bedding dips, and fractures. A horizontal line was drawn across the log indicating footage where core blocks were present within the drilled core. Footage of core cut, and recovery were also recorded. Intervals with no recovery were indicated on the drill log by horizontal lines crossing the entire page with a blanked-out zone of "no information" making it readily apparent where information was missing.

Any discrepancies in the footage shown on the core blocks or in core recovery were noted by the logging geologist on the log. Where there was missing core, additional core blocks were inserted by the geologist reflecting the missing interval and a cursory explanation written on the core block stating why the interval was missing.

Graphic logs of the lithology were also produced to reflect the major rock types using conventional or agreed upon symbols. Major structural features including contact relationships, dips and fractures, bedding, and veins were plotted on the log and described as angle from core axis. Alteration and mineralization styles were also recorded along with a description of the lithology.

11.4 Sample security

Methods for securing samples by companies conducting work at the Property prior to the formation of PMC are unknown. Between 1970 and 1996, during which time PMC was actively mining at the Property, samples were sent to the mine laboratory for analyses. It is not known what provisions PMC employed for sample security.

When Homestake operated PMC, samples were picked up and transported to the laboratory by ALS Chemex as part of the chain of custody. In 2003, and 2007 to 2008, Barrick as operator of PMC conducted drilling programs. It is uncertain what protocols were employed by Barrick to ensure sample security.

Atna conducted exploration and development drilling between 2004 – 2006, and from 2012 – 2015. Once a set of samples was ready for shipment to the laboratory (lab), the lab was contacted for a job number and a pickup time by the lab scheduler. It is unknown if samples were stored onsite or whether the sample storage area was secured. Both RC chips and core samples were placed in numbered bags and the bags placed in 4 x 4 x 3 ft wooden crates for shipping along with a transmittal sheet indicating whether the samples were core or RC cuttings, the range of sample numbers, and the total number of samples. In some instances, an Atna geologist travelling to Reno delivered samples to the lab.

11.5 Sample preparation and analysis

11.5.1 PMC 1970 – 1996

Sample preparation procedures for the Pinson Mine were not recorded.

PMC’s standard assaying practice was to run assays using atomic absorption (AA) methods. For all assays this was generally done on a cyanide leach to aide in identifying leachable feed (Sim 2005). At some unknown point, PMC changed this to only run fire assay with AA finish on samples over 0.01 opt. Check assays were performed on high-grade zone samples at third-party laboratories. Detection limits for the PMC samples varied from <0.003 to <0.001 opt, dependent on the age of the assay.

11.5.2 PMC - Homestake 1997 – 2000

When Homestake operated PMC, assays were analyzed by ALS Chemex in Reno, NV. Samples were prepared at the ALS lab as follows:

- Primary crush and mill to 80% passing -10 mesh.
- 300-gram split of material for pulverization to 90% passing -150 mesh.
- 30-gram split for digestion and assay.

Samples were assayed utilizing the Au-AA23 fire assay method with AA finish. Analyses were reported in parts per billion (ppb). Samples reporting Au values > 10,000 ppb were re-assayed by fire assay with a gravimetric finish.

Detection limits for gold analyses performed by ALS Chemex were 5 ppb and 0.0005 opt. For statistical purposes, most of the Homestake holes that reported "detection limit" gold were converted to 2.5 ppb and 0.0003 opt. (These values were subsequently converted back to -5 ppb and -0.0005 opt in the current database).

11.5.3 PMC Barrick 2000 – 2008

American Assay Laboratories (AAL) located in Sparks, Nevada was utilized by PMC (Barrick) to prepare and analyze samples generated from its drilling programs.

Samples were dried, weighed and crushed using either a roll or jaw crusher. A split of crushed material was pulverized for further analytical work. Samples were analyzed for gold using a one-assay ton (29.116 g) fire assay with AA finish (Fire AA). Samples with a fire assay greater than 0.005 opt were subject to a cyanide soluble leach assay by AA spectroscopy to determine gold recovery and carbon and sulfur analysis for metallurgical evaluation. Samples returning an initial gold assay >5 parts per million (ppm) were subject to fire assay with a gravimetric finish.

In addition to gold, PMC (Barrick) also had the samples analyzed for an additional 69 elements using an aqua regia digestion with an Induced Coupled Plasma Atomic Emission Spectroscopy (ICPAES finish). PMC (Barrick) employed its own internal quality assurance / quality control (QA/QC) protocols. Once the assay results were received via email, the exploration database manager loaded the assay data into AcQuire database management software (ACQ). The ACQ software evaluated the gold values of the standards and flagged any standards that performed outside of acceptable limits. Failed standards were documented and reviewed by the geologist in charge of the project. Dependent on the rate of failure, a selection of samples, or possibly the entire batch was rejected, and another round of analyses requested by the geologist.

When samples needed re-assaying, the lab was notified of the failures and a list of samples to be re-assayed were sent to the lab. Upon receipt of the results of the re-assayed samples by the database supervisor, they were loaded into ACQ and XY scatter plots generated for the geologist to review for approval or rejection. Should the second round of analyses be rejected, a third round would ensue until acceptable results were achieved. Check samples were also collected and sent to a second lab to evaluate potential laboratory bias. It is unknown which laboratories were used to analyze the check samples.

11.5.4 Atna 2004 – 2013

Atna utilized Inspectorate American Laboratories (IAL), an ISO 9002-accredited facility located in Reno, Nevada as their primary analytical lab for the Getchell Project. Sample preparation procedures utilized by IAL follow.

The samples were dried and weighed prior to crushing. Crushing utilized a two-stage process. Once the sample was dried it was passed through a jaw crusher to reduce it to a uniform size. It then passed through a roll mill to reduce the sample to >80% passing -10 mesh. A 300-gram split of this material was obtained using a Jones riffle splitter. The split material was further reduced to >90% passing -150 mesh using a ring and puck pulverizer.

After pulverization, a 30-gram sample of pulp was taken and digested and analyzed for gold using standard fire assay with AA finish. Samples returning gold values greater than 3 g/t were subjected to gravimetric analyses.

11.5.5 Atna Underground 2011 – 2016

The new mine lab which was constructed adjacent to the administration building in 2012, was in operation from 2012 to 2016.

Underground samples were transported to the on-site laboratory by Atna personnel. Samples were logged in and checked against sample transmittal sheets. Samples were then dried and weighed before being passed through a small jaw crusher to minus 3/8 inch passing. Crushed material is then passed through a "Jones" splitter, multiple times if necessary, to produce a 200 grams (g) to 300 g sample split for pulverization. The pulp split is then transferred to the ring and puck pulverizer for grinding to 80% passing 150 mesh. Pulverized material was weighed out to a 30 g fire assay sample charge (Pinson Mine Internal 2015).

11.6 Data validation

11.6.1 Summary

The Property database has been subjected to three major campaigns of data validation by Atna, Barrick and most recently OMC. The details of data validation completed by Atna and Barrick are described in detail in previous Technical Reports Atna (2006), Atna (2007), Gustavson (2012), and Golder (2014). A summary of this work is described herein.

11.6.2 Atna review of prior data

Atna completed a detailed review of historic data as part of due diligence studies, and upon acquiring the Property. This process involved comparing data stored within a historic Microsoft Access database with digital files, databases, Vulcan files, and records stored onsite. Errors were corrected based on a "well maintained filing system containing most, if not all, drill logs, downhole surveys and Homestake assays" (Atna 2007). Validation errors such as overlapping samples, length discrepancies (i.e., surveys beyond hole depth) were investigated and corrected as appropriate.

Atna was unable to verify PMC analytical results as much of the historical analysis had been completed using the mine laboratory and original certificates were not available. To assess historic analytical results, Atna reanalyzed 652 drill sample pulps from mineralized intercepts within the CX and Range Front target area. The pulps were sourced from the onsite pulp library maintained by PMC. Check pulp samples were submitted with Certified Reference Materials (CRMs). Atna concluded that re-assay results confirmed the accuracy of original Homestake and PMC assay results.

Atna subsequently completed two separate database audits. The first audit involved the selection of 20% of the 370 holes within the database, extracting assays greater than 0.08 opt and checking assays. Out of 216 errors, 16 errors were noted and corrected. A second audit was completed by checking 15% of the drillholes completed by Atna in the Phase 2 program of 2006. Out of 1,653 assays a total of 12 errors were identified.

11.6.3 Barrick review of prior data

On exercising their earn-back option with Atna, Barrick conducted a detailed verification review of the historical drillhole database. This included reviewing the use of standards, blanks, and duplicates along with a second round of checks on the data entry and database maintenance. The results of the verification program are documented in an internal Barrick report which concluded that, "…10% of the database was checked, and it was considered adequate for use in a Scoping Level study…" (Golder 2014).

Barrick broadened the scope of their investigation of potential Mineral Resources at the Getchell Project to include open pit potential and initiated a check of the accuracy of the historical database within an area of interest which included checks on drillhole collars for 2,014 holes.

Barrick contracted Geostrata LLC of Bluffdale, Utah to complete data verification checks on historic data. Collar coordinates, downhole surveys, from and to intervals and assay values were reviewed. Six errors were identified out of 208 collars checked. Errors comprised transcription errors where the collar coordinates or hole length was incorrect, and field errors where data had been entered into the incorrect field. Out of a total of 18,013 assays a total of 184 errors were identified (1%). Errors comprised:

- Data in the database but not in the drill log and vice versa.
- Incorrect numbers in the database according to the drill log.
- Discrepancy transcribing nil, trace, no sample, or detection limit values.
- Sample type is recorded in the drill log but not in the database.
- No assay data is available via certificate or drill log but there was data in the database.

Table 11.1 provides a summary of the errors.

Table 11.1 Summary of errors within the database

Company	Total assays reviewed	Missing data	Incorrect numbers	Discrepancies nil, trace, no sample, detection limit	Sample type errors	No certificate or drill log	Total number of errors
Atna	1867	3 (0.16%)	0	0	0	0	3 (0.1%)
PMC (Barrick)							
Cordex	179	0	0	7 (3.9%)	0	0	7 (3.9%)
Cordilleran	435	4 (0.9%)	0	2 (0.4%)	0	1 (0.2%)	7 (1.6%)
PMC (Homestake)	3319	5 (0.15%)	3 (0.09%)	11 (0.3%)	11 (0.3%)	2 (0.06%)	32 (0.9%)
Pinson Mine Co.	12,392	71 (0.57%)	47 (0.3%)	16 (0.1%)	0	1 (0.008%)	135 (1.0%)
Total	**18,013**	**83 (0.46%)**	**50 (0.27%)**	**36 (0.19%)**	**11 (0.06%)**	**4 (0.02%)**	**184 (1.0%)**

11.6.4 OMC data compilation and validation

11.6.4.1 Database compilation

In January of 2017, OMC contracted Maxwell Resources (Maxwell) to perform data migration of the drillhole database into their proprietary DataShed database software. Maxwell was supplied with collar, downhole survey, lithology, and original assay files.

While in operation, both mine labs utilized a digital assay file management system to keep track of assay and other data generated from drilling programs. Only raw digital assay files were located for assays generated by the new mine lab. The new mine lab utilized an Excel file with multiple tabs to record assay data throughout the assaying process. Only the tab marked as "final assay" was used by Maxwell and OMC for data uploads into DataShed. Assay data from the old mine lab was only available as paper copies with hand-written assays on the form. These paper copies were used to validate assay data in the DataShed database.

Maxwell supplied OMC with an SQL database in February of 2017. During the process of migrating the database into the new software, Maxwell noted that assay files were in various formats and that there were multiple errors in collar information.

All gold assays, including Cyanide Au and calculated values, were uploaded into one Au field. There were also a significant number of generic methods that had unknown ("UN_UN") listed for the analytical method. The new data uploaded from the various labs added more analytical methods. After reviewing the database, it was determined that additional Au fields were needed to separate out the various analytical methods i.e., Cyanide Au (Au_CN field) along with a field for calculating ounces per ton (opt) (AU_CALC field). It is important to be able to specify the analytical method used for Au analyses since DataShed automatically ranks the methods from most reliable method to least reliable method.

11.6.4.2 Database corrections

In 2018, OMC corrected the errors found by Maxwell during their data migration process. Errors that were corrected included duplicate holes, core recovery issues, and interval data that went past total hole depth. In addition, assays batches that were not uploaded correctly were flagged with a "NOCERT" or "assay method unknown" identifier.

In April of 2019, OMC contracted AMC and CSA Global to perform separate Mineral Resource updates on the Pinson underground mineralized zone. Upon detailed review of the drillhole database AMC and CSA Global separately expressed concern with the number of "NO CERT" and "assay method unknown" assays. An area of interest surrounding the underground mineralized zone was subsequently defined and original assay certificates were sourced and reloaded where possible. Analytical methods associated with assay data was updated during this process. Standards and blanks were also compiled and uploaded.

Details of assays reloaded are presented in Table 11.2 and Table 11.3.

Table 11.2 Initial data set and 18 April 2019 data subset

	Starting database	**18 April 2019 database**
Samples	77,475	77,660
Number of samples with "NOCERT"	58,740	48,498
Percentage of database with "NOCERT"	75.80%	62.40%

Table 11.3 Assay certificates and samples uploaded by laboratory

Laboratory	**Number of batches**	**Number of samples**
American assay lab	66	9,098
Inspectorate	164	13,626
Pinson Mine	132	2,921
Total	**362**	**25,645**

Notes: Numbers are from the defined area of interest.

Certificate headers contain the certificate identification, analyte, laboratory method, and assay unit. The raw assay headers from all the labs had to be re-formatted to facilitate direct import to DataShed. All certificates, regardless of the lab of origin, had the identifier "_2019" added to the end of the certificate number to aid in separating assays from the same certificate but which had different loading parameters. The 18 April 2019 database described in this section was used in the Mineral Resource estimate.

Significant work has been completed on the transfer of the old database into the new DataShed database and additional clean-up work still needs to be performed on the DataShed database to ensure its completeness and increase the confidence in the data.

11.7 Quality assurance / quality control overview

Quality assurance / quality control (QA/QC) data has been compiled from available databases for all drilling activities completed since 2005. No QA/QC data is available for work occurring prior to this time.

Drilling programs completed at the Property between 2005 and 2015 included QA/QC monitoring programs which comprised the insertion of CRMs, blanks, and duplicates into the sample streams on a batch by batch basis. Table 11.4 provides a summary of QA/QC samples included during this period.

Table 11.4 QA/QC 2005 – 2015

Year	Company	Drill samples	CRM's	Blanks	Field duplicates
2005	Atna	7,330	267	289	23
2006		4,859	265	263	39
2007	Barrick	3,644	123	107	2
2008		17,661	403	265	197
2012		1,515	0	0	0
2013	Atna	3,360	0	0	0
2015		1,320	23	0	0
Total		**39,689**	**1,081**	**924**	**261**

Notes:
- Counts of individual samples. Multiple analyses types per sample (i.e., fire assay and gravimetric).
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Table 11.5 shows the insertion rates of QA/QC samples between 2005-2015.

Table 11.5 QA/QC 2005 – 2015 insertion rates

Year	Company	CRM's	Blanks	Field duplicates	QA/QC [1]
2005	Atna	3.6%	3.9%	0.3%	7.9%
2006		5.5%	5.4%	0.8%	11.7%
2007	Barrick	3.4%	2.9%	0.1%	6.4%
2008		2.3%	1.5%	1.1%	4.9%
2012		0.0%	0.0%	0.0%	0.0%
2013	Atna	0.0%	0.0%	0.0%	0.0%
2015		1.7%	0.0%	0.0%	1.7%
Total		**2.7%**	**2.3%**	**0.7%**	**5.7%**

Notes:
- Counts of individual samples. Multiple analyses types per sample (i.e., fire assay and gravimetric).
- [1] Insertion rate for CRM, Blanks and Field Duplicates combined.
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

11.8 Certified Reference Materials

11.8.1 Description

A total of 37 different CRMs was used at the Property between 2005 and 2015. CRMs were supplied by Rocklabs of New Zealand.

CRMs comprised on average 2.7% (and up to 5.5%) of samples submitted to the laboratory. CRMs insertion formed part of the QA/QC program consistently in the period between 2005 and 2008. CRMs, during this time, were generally included systematically at a rate of 1 in 20 to 1 in 25 samples. CRMs do not appear to have been consistently used since 2008.

CRMs used in the 2005 and 2006 programs are discussed in the 2007 NI 43-101 Technical Report titled "Technical Report Update Pinson Gold Property, Humbolt County, Nevada, USA" effective 1 June 2007 (2007 Technical Report). There is no documentation available regarding CRM procedures for programs after 2006.

Rocklabs CRMs were stored in bulk in plastic bin in the logging trailer. Individual CRMs were created by measuring 100 g of the appropriate CRM into kraft envelopes. Packaged CRMs were then stored in separate labelled bins and inserted regularly into the sample stream.

Table 11.6 and Table 11.7 summarize CRMs by year and company.

Table 11.6 CRMs used in each year

Period	Company	# CRMs	CRMs used
2005	Atna	16	OxA45, OxE21, OxH29, OxK18, OXL25, OxN33, OxP32, SF12, SG31, SI15, SJ10, SK11, SN16, SP17, SQ18, UNKNOWN
2006		16	OxA45, OxE21, OxH29, OxI54, OxJ36, OxK18, OXL25, OxN33, OxP32, SF12, SI15, SJ10, SK11, SN16, SP17, SQ18
2007	Barrick	15	OxA59, OxC58, OxD57, OxF53, OxG60, OxH52, OxI54, OxK48, OxN49, OxP50, SF23, SG31, SJ32, SK33, SN26
2008		18	OxA59, OxC58, OxD57, OxF53, OxG60, OxH52, OxI54, OxJ36, OxK48, OxN49, OxP50, SF23, SG31, SI25, SJ32, SK33, SN26, UNKNOWN
2012	Atna	0	
2013		0	
2015		6	OxK119, OxN117, OxP91, SK78, SN75, SP73

Table 11.7 CRMs used by year and company (2005 – 2015)

CRM ID	Expected Au value (ppm)	Stand dev	Number of CRMs used [1]					
			Atna		Barrick		Atna	Total
			2005	2006	2007	2008	2015	
OxA45	0.081	0.0069	2	13				15
OxA59	0.082	0.0052			3	37		40
OxC58	0.201	0.007			7	30		37
OxD57	0.413	0.012			13	42		55
OxE21	0.651	0.026	30	26				56
OxF53	0.810	0.029			4	30		34
SF12	0.819	0.028	36	18				54
SF23	0.831	0.027			7	38		45
SG31	0.996	0.028	1		4	36		41
OxG60	1.025	0.028			10	27		37
OxH52	1.291	0.025			18	28		46
OxH29	1.298	0.033	24	21				45
SI25	1.801	0.044				22		22
SI15	1.805	0.067	1	4				5
OxI54	1.868	0.066		1	6	33		40
OxJ36	2.398	0.073		3		1		4
SJ10	2.643	0.06	2	16				18
SJ32	2.645	0.068			5	30		35
OxK18	3.463	0.132	21	2				23
OxK48	3.557	0.042			10	22		32
OxK119	3.604	0.105					3	3
SK33	4.041	0.103			9	15		24
SK78	4.134	0.138					4	4
SK11	4.823	0.11	21	26				47
OXL25	5.852	0.105	29	25				54
OxN33	7.378	0.208	33	28				61
OxN49	7.635	0.189			19	6		25
OxN117	7.679	0.207					2	2
SN16	8.367	0.217	17	21				38
SN26	8.543	0.175			2	3		5
SN75	8.671	0.199					4	4
OxP91	14.820	0.3					3	3
OxP50	14.890	0.493			6	3		9
OxP32	14.990	0.44	3	15				18
SP17	18.125	0.434	25	32				57
SP73	18.170	0.42					7	7
SQ18	30.490	0.88	22	14				36

Notes:
- [1] Counts of individual samples. Multiple analyses types per sample (i.e., fire assay and gravimetric).
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

11.8.2 Discussion on CRMs

CRMs are inserted to check the analytical accuracy of the laboratory. An insertion rate of at least 5% of the total samples assayed is advocated. CRMs should be monitored on a batch by batch basis and remedial action taken immediately if required. For each economic mineral, there should be at least three CRMs with values:

1 At around the cut-off change of the deposit.
2 At the expected grade of the deposit.
3 At a higher grade.

The average grade for the Open Pit area Mineral Resource is approximately 1.7 ppm Au at a 0.010 opt Au (0.34 ppm Au) cut-off grade (COG). The average grade of the Underground area Mineral Resource is 10.3 ppm Au at a 0.016 opt Au (0.55 ppm Au) COG. CRMs OxC58, OxD57, and OxE21 cover the approximate COGs of both Mineral Resource areas. CRMs SI25, SI15, and OxI54 cover the average grade of the Open Pit area. The average grade of the Underground area is not covered by a single CRM. Higher grades for the Open Pit and Underground areas are covered by various CRMs.

The QP advocates re-assaying assay batches where two consecutive CRMs occur outside two standard deviations, or one CRM occurs outside three standard deviations of the expected value described on the CRM certificate. Results for CRMs used in the QA/QC program are presented in Table 11.9.

Control charts are used to monitor the analytical performance of an individual CRM over time. Control lines are also plotted on the chart for the expected value of the CRM, two standard deviations above and below the expected value, and three standard deviations above and below the expected value. CRM assay results are plotted in order of analysis. These charts often clearly show analytical drift and bias should they occur.

The QP considers the number of different CRMs historically used on the Property to be excessive. It is preferable to limit the number of different CRMs used on a Project to ensure that each CRM has enough results to enable meaningful analysis. In the QP’s experience between three and five different CRMs are usually adequate to monitor laboratory performance.

Control charts at various grades for the two main campaigns of work are presented for select CRMs (outlined in Table 11.8) are shown in Figure 11.1 to Figure 11.8. Control charts were not plotted for 2015 drilling due to limited data. Control charts show the most relevant method of analysis.

Table 11.8 CRMs selected for control charts

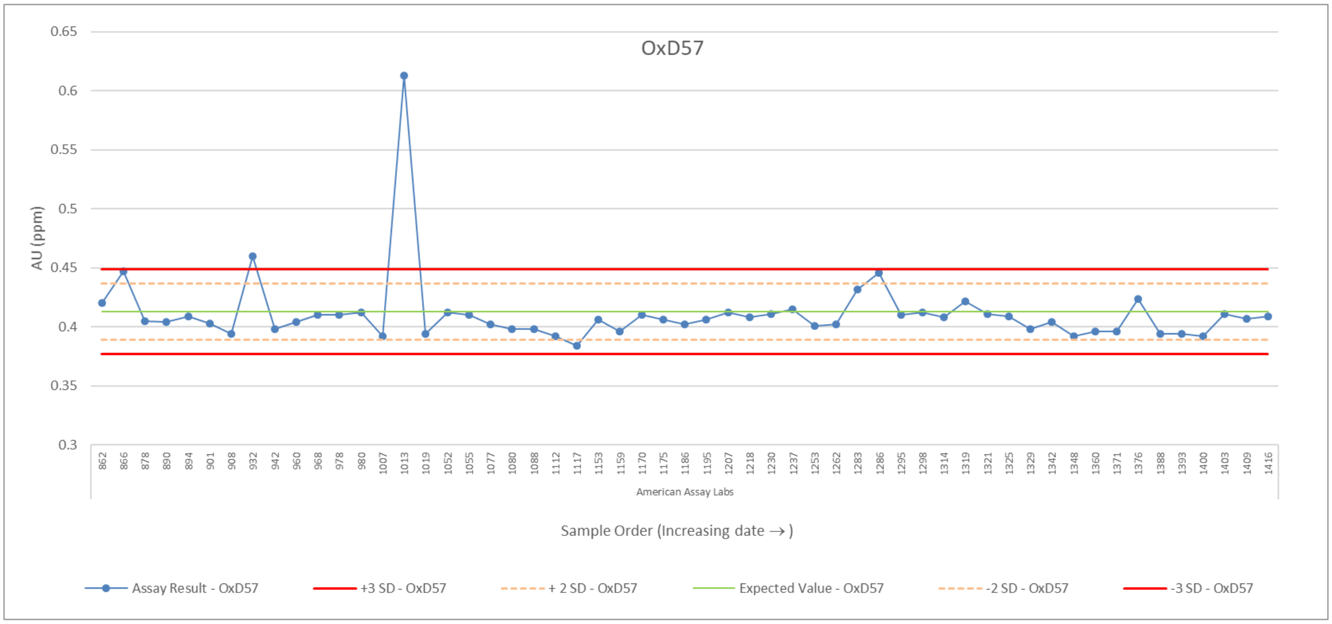
CRM	Au value (ppm)	No. CRMs	Campaign	Notes
OxD57	0.413	55	2007 – 2008	Approximate open pit COG
OxE21	0.651	56	2005 – 2006	Approximate underground COG
OxH52	1.291	46	2007 – 2008	Approximate average grade of open pit
OxH29	1.298	45	2005 – 2006	Approximate average grade of open pit
OxN33	7.378	61	2005 – 2006	Higher grade
OxN49	7.635	25	2007 – 2008	Higher grade
SN16	8.367	38	2005 – 2006	Higher grade
SP17	18.125	57	2005 – 2006	Higher grade

Table 11.9 CRM results (2005 – 2015)

CRM	Expected Au value (ppm)	Expected standdev	Years used	Number of assays	Warning (> 2 standdev)	Fail (> 3 standdev)
OxA45	0.081	0.0069	2005, 2006	15	0	4
OxA59	0.082	0.0052	2007, 2008	40	0	1
OxC58	0.201	0.007	2007, 2008	37	0	1
OxD57	0.413	0.012	2007, 2008	55	3	2
OxE21	0.651	0.026	2005, 2006	56	6	19
OxF53	0.810	0.029	2007, 2008	34	0	3
SF12	0.819	0.028	2005, 2006	54	12	15
SF23	0.831	0.027	2007, 2008	45	1	4
SG31	0.996	0.028	2005, 2007, 2008	41	3	1
OxG60	1.025	0.028	2007, 2008	37	3	1
OxH52	1.291	0.025	2007, 2008	46	7	11
OxH29	1.298	0.033	2005, 2006	45	9	15
SI25	1.801	0.044	2008	22	1	0
SI15	1.805	0.067	2005, 2006	5	0	2
OxI54	1.868	0.066	2006, 2007, 2008	40	4	3
OxJ36	2.398	0.073	2006, 2008	4	0	1
SJ10	2.643	0.06	2005, 2006	18	4	7
SJ32	2.645	0.068	2007, 2008	35	4	7
OxK18	3.463	0.132	2005, 2006	46	9	2
OxK48	3.557	0.042	2007, 2008	33	7	15
OxK119	3.604	0.105	2015	3	0	0
SK33	4.041	0.103	2007, 2008	25	4	4
SK78	4.134	0.138	2015	4	0	1
SK11	4.823	0.11	2005, 2006	94	21	32
OXL25	5.852	0.105	2005, 2006	108	20	33
OxN33	7.378	0.208	2005, 2006	122	23	21
OxN49	7.635	0.189	2007, 2008	44	6	13
OxN117	7.679	0.207	2015	2	0	1
SN16	8.367	0.217	2005, 2006	76	12	8
SN26	8.543	0.175	2007, 2008	9	0	0
SN75	8.671	0.199	2015	4	1	0
OxP91	14.820	0.3	2015	3	1	0
OxP50	14.890	0.493	2007, 2008	17	0	0
OxP32	14.990	0.44	2005, 2006	36	3	6
SP17	18.125	0.434	2005, 2006	114	11	14
SP73	18.170	0.42	2015	7	0	1
SQ18	30.490	0.88	2005, 2006	72	4	5
Total				**1448**	**179**	**253**

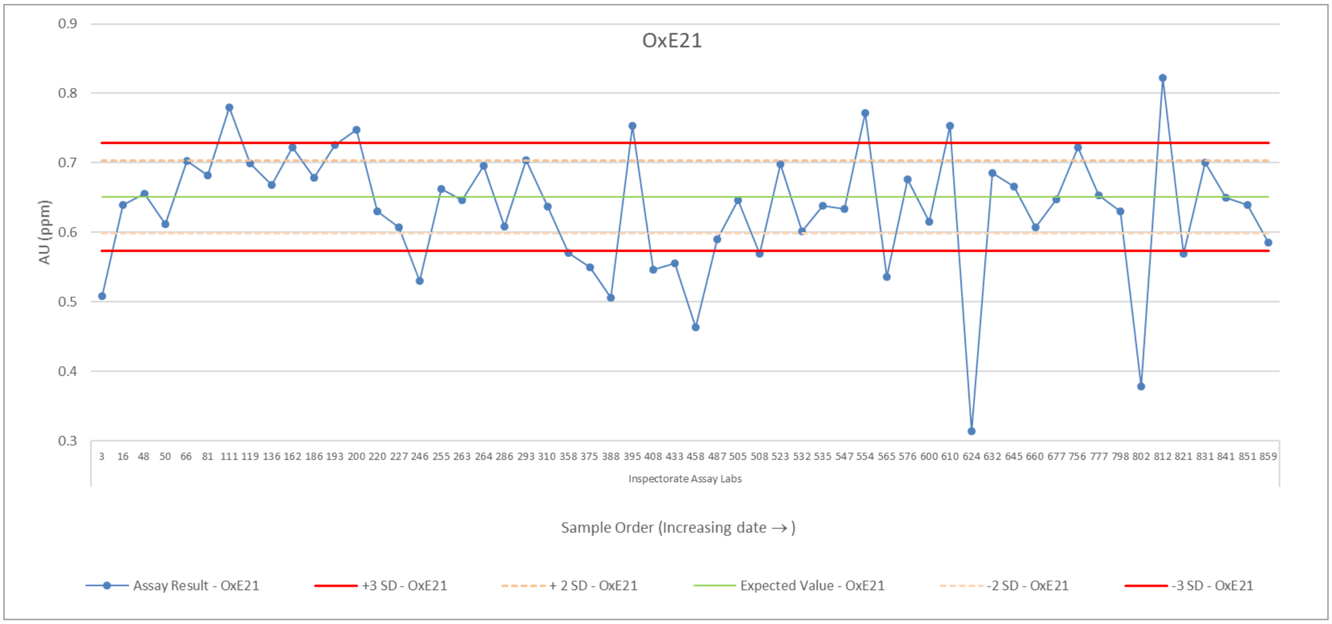
Note: Sorted by CRM expected value.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.1 CRM OxD57 (2007 – 2008) FA-ICP-ES



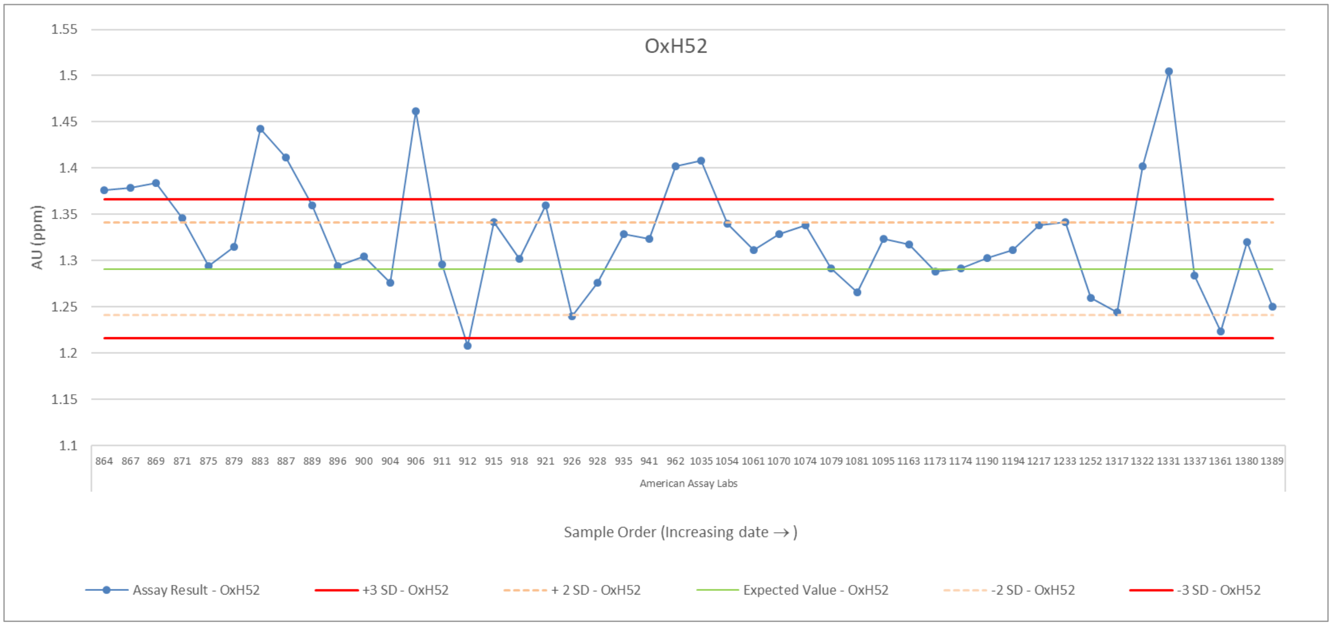
Note: All CRMs analyzed by fire assay with ICP-ES at American Assay Labs.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.2 CRM OxE21 (2007 – 2008) FA-AAS



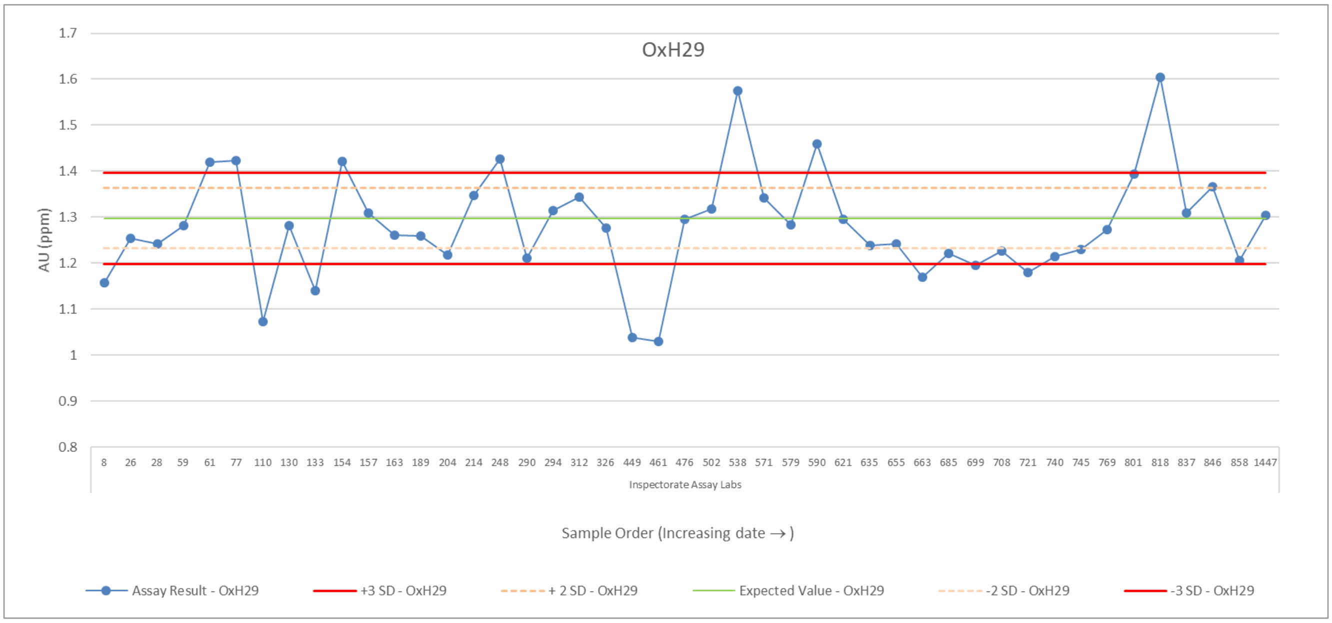
Note: All CRMs analyzed by fire assay with AAS at Inspectorate Assay Labs.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.3 CRM OxH52 (2007 – 2008) FA-ICP-ES



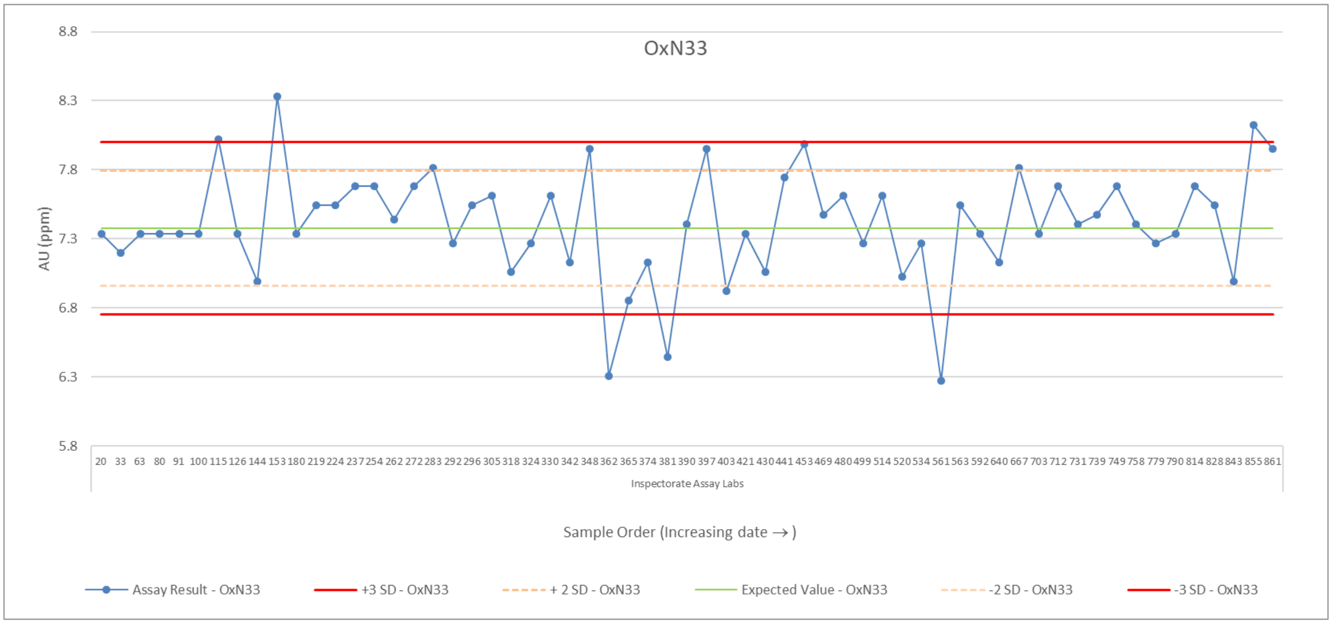
Note: All CRMs analyzed by fire assay with ICP-ES at American Assay Labs.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.4 CRM OxH29 (2005 – 2006) FA-AAS



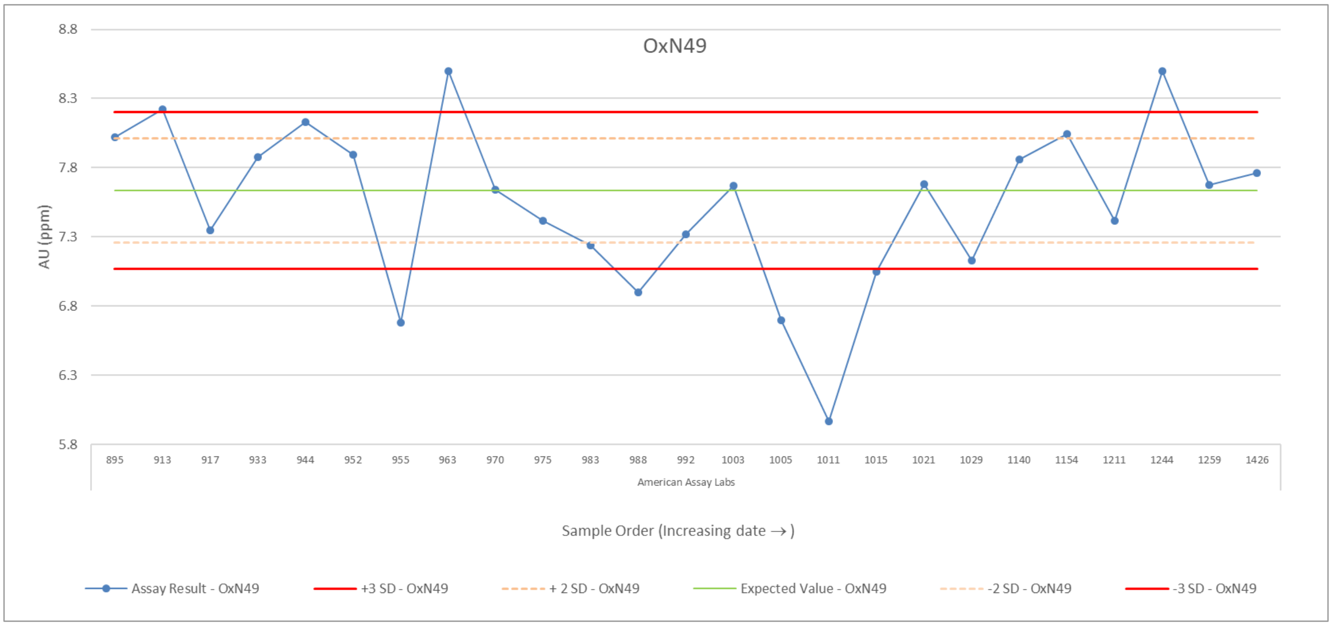
Note: All CRMs analyzed by fire assay with AAS at Inspectorate Assay Labs.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.5 CRM OxN33 (2005 – 2006) FA-GRAV



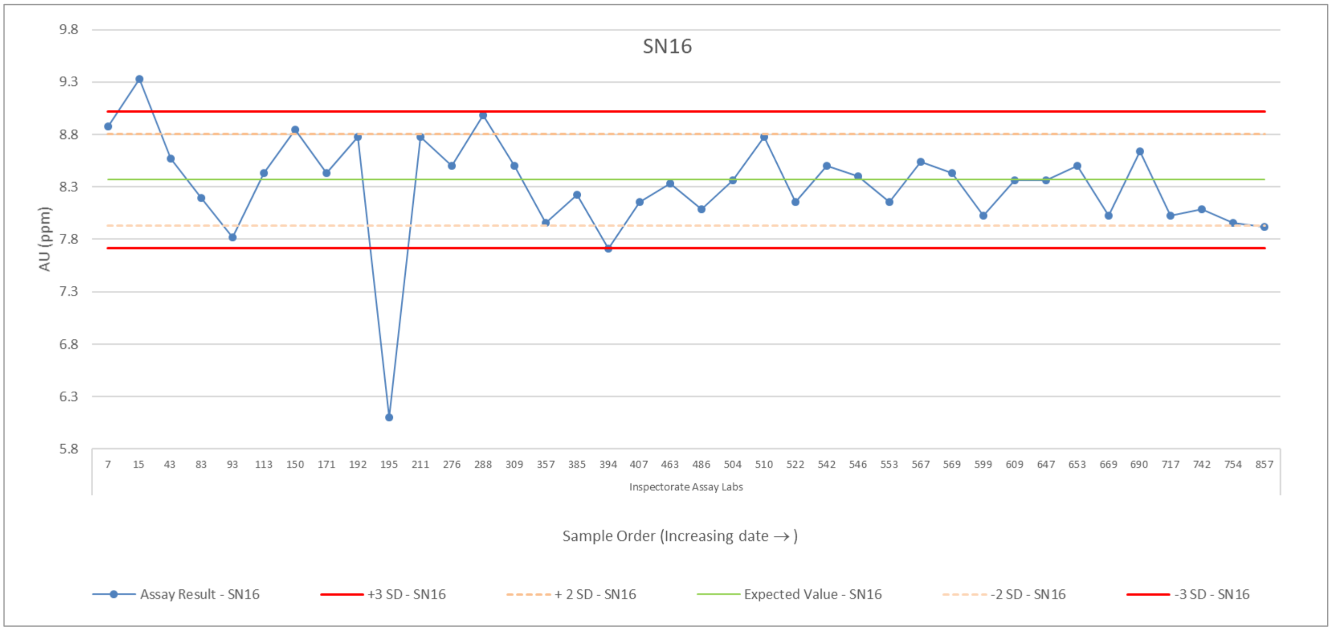
Note: All CRMs analyzed by fire assay with gravimetric analysis at Inspectorate Assay Labs.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.6 CRM OxN49 (2007 – 2008) FA-ICP-ES



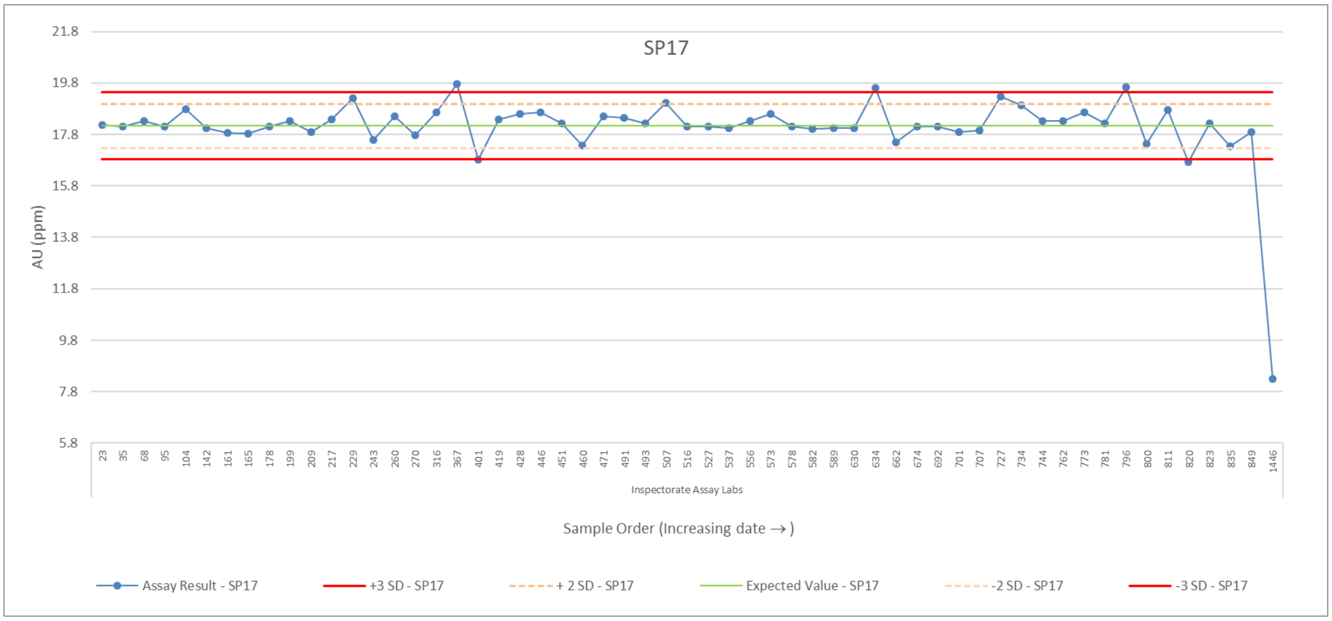
Note: All CRMs analyzed by fire Assay with ICP-ES analysis at American Assay Labs. Assays by FA-GRAV not shown.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.7 CRM SN16 (2005 – 2006) FA-GRAV



Note: All CRMs analyzed by fire assay with gravimetric analysis at Inspectorate Assay Labs. Assays by FA-AAS not shown.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.8 CRM SP17 (2005 – 2006) FA-GRAV



Note: All CRMs analyzed by fire assay with gravimetric analysis at Inspectorate Assay Labs. Assays by FA-AAS not shown.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

The CRM insertion rate of 2.7% is significantly less that the preferred rate of 5%. Programs between 2005 and 2009 included between 1.7% and 5.5% CRMs submitted regularly into the sample stream. There is no record of CRMs being consistently inserted in 2012, 2013, and 2015 drilling which falls short of common industry practice.

CRMs used at the Property between 2005 and 2008 exhibit a consistent, high number of > 2 standard deviation warnings and > 3 standard deviation failures on a number of different CRMs from two different laboratories, over a number of years.

Laboratory assay results have similar means to the expected CRM values, but have significantly larger standard deviations (on average 2.5 times larger) than that specified by Rocklabs (Table 11.10). CRM control charts show results scattered relatively evenly above and below the expected value, suggesting no significant bias.

In general, CRMs show reasonable analytical accuracy, but relatively poor precision, when compared against the certified standard deviation. This poor precision occurs in a number of CRMs from two laboratories, over a period of four years. The QP was unable to definitely determine the cause of CRM high failure rate.

The 2007 Report (Atna 2007) briefly discusses issues with poor CRM performance of 2005 – 2006 samples. CRM performance was investigated, and after discussion with Rocklabs, a moving average method was used to set control limits rather than the certified standard deviation.

While the poor precision of CRMs should be investigated AMC does not consider this to be of a material concern for a global, long-term Mineral Resource estimate.

11.8.3 Recommendations on CRMs

The QP recommends the following for any future programs:

- Purchase additional CRMs at the approximate COGs, average grades and higher grades of the deposits.
- Include CRMs in every batch of samples submitted at a rate of at least 1 in every 20 samples (5%).
- Ensure that CRMs are monitored in real time on a batch by batch basis, and that remedial action is taken immediately as issues are identified.
- Ensure CRM warnings, failures and remedial action is documented.
- If pulps are available in areas relevant to the current Mineral Resource, the QP recommends that an investigation into analytical precision be completed. This would comprise selecting a number of mineralized intervals associated with poor performing CRMs and completing reanalysis of two separate sub-samples from each pulp using an umpire laboratory. CRMs should be included in this submission. Differences between the grades of the new pulp assays will allow assessment of subsampling variance and geological variance. Differences to the original samples may provide insight into the precision of the original laboratory.

11.9 Blank samples

11.9.1 Description

Coarse blank samples were inserted into the sample stream of drill programs completed between 2005 and 2008. Data available suggest that blanks were not included in subsequent programs.

The 2005-2006 programs utilized commercial decorative stone purchased in 50 pound (lb) bags as the source of blank material. The source of blank material used in 2007 – 2008 is unknown.

A total of 924 blanks were included between 2005 and 2008 representing between 1.5 and 5.4% of total samples. In 2005 and 2006 blanks were inserted regularly approximately every 20th sample between CRMs. This was insertion rate was decreased to approximately every 40th or every 80th sample in 2007 and 2008, a rate of 2.9% and 1.5%. There is no record of blanks being included in subsequent drill programs.

Table 11.10 Comparison between CRM values and analytical results

CRM			Analytical results			Comparison	
CRM ID	Expected Au value (ppm)	Standdev	Number of assays	Mean	Standdev	Mean vs expected	Standdev of results vs expected
OxA45	0.0811	0.0069	15	0.079	0.018	98%	257%
OxA59	0.0817	0.0052	40	0.083	0.004	101%	78%
OxC58	0.201	0.007	37	0.202	0.008	101%	111%
OxD57	0.413	0.012	55	0.411	0.031	100%	258%
OxE21	0.651	0.026	56	0.634	0.092	97%	356%
OxF53	0.81	0.029	34	0.821	0.039	101%	136%
OxG60	1.025	0.028	37	1.019	0.034	99%	120%
OxH29	1.298	0.033	45	1.282	0.117	99%	355%
OxH52	1.291	0.025	46	1.326	0.061	103%	246%
OxI54	1.868	0.066	40	1.852	0.103	99%	156%
OxJ36	2.398	0.073	4	2.377	0.160	99%	220%
OxK119	3.604	0.105	3	3.600	0.034	100%	32%
OxK18	3.463	0.132	46	3.580	0.210	103%	159%
OxK48	3.557	0.042	33	3.637	0.256	102%	609%
OXL25	5.852	0.105	108	5.937	0.338	101%	322%
OxN117	7.679	0.207	2	6.994	0.824	91%	398%
OxN33	7.378	0.208	122	7.465	0.464	101%	223%
OxN49	7.635	0.189	44	7.592	0.571	99%	302%
OxP32	14.99	0.44	36	15.080	3.060	101%	695%
OxP50	14.89	0.493	17	14.841	0.414	100%	84%
OxP91	14.82	0.3	3	14.571	0.480	98%	160%
SF12	0.819	0.028	54	0.792	0.081	97%	288%
SF23	0.831	0.027	45	0.835	0.045	101%	167%
SG31	0.996	0.028	41	0.996	0.034	100%	120%
SI15	1.805	0.067	5	1.493	0.476	83%	711%
SI25	1.801	0.044	22	1.819	0.048	101%	108%
SJ10	2.643	0.06	18	2.597	0.205	98%	342%
SJ32	2.645	0.068	35	2.583	0.128	98%	188%
SK11	4.823	0.11	94	4.934	0.381	102%	346%
SK33	4.041	0.103	25	4.117	0.223	102%	216%
SK78	4.134	0.138	4	3.908	0.288	95%	209%
SN16	8.367	0.217	76	8.311	0.476	99%	220%
SN26	8.543	0.175	9	8.533	0.074	100%	42%
SN75	8.671	0.199	4	8.503	0.316	98%	159%
SP17	18.125	0.434	114	18.122	1.264	100%	291%
SP73	18.17	0.42	7	18.279	0.893	101%	213%
SQ18	30.49	0.88	72	30.734	1.161	101%	132%

Note: All data included; Standdev=standard deviation.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

11.9.2 Discussion on blanks

Coarse blanks test for contamination during both the sample preparation and assay process. Blanks should be inserted in each batch sent to the laboratory. In the QP's opinion, 80% of coarse blanks should be less than three times the detection limit.

Table 11.11 shows the assay results from blank materials for drilling completed between 2005 and 2008 and the results of AMC's pass / fail parameters. AAL and IAL are reviewed separately due to the differences in detection limits.

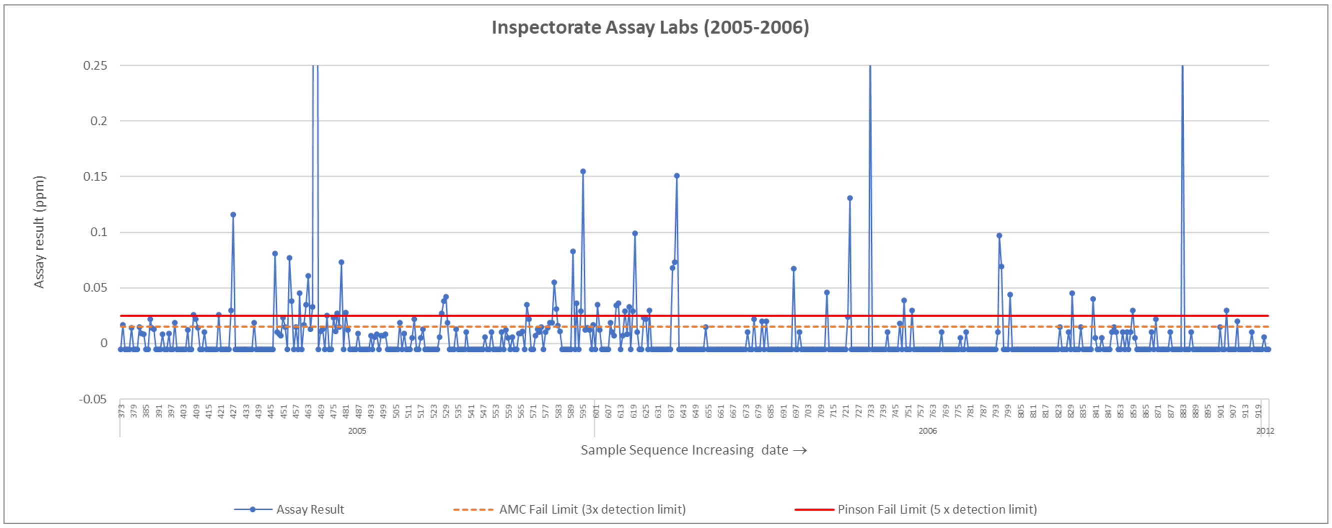
Table 11.11 Blanks

Laboratory	Year	Detection limit (ppm)	Number of samples	# AMC fail (> 3 x DL)
IAL	2005	0.005	289	50
	2006		263	28
	Subtotal		**552**	**78**
AAL	2007	0.003	107	8
	2008		265	14
	Subtotal		**372**	**22**
Total			**924**	**100**

Note: Year=refers to drill year.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

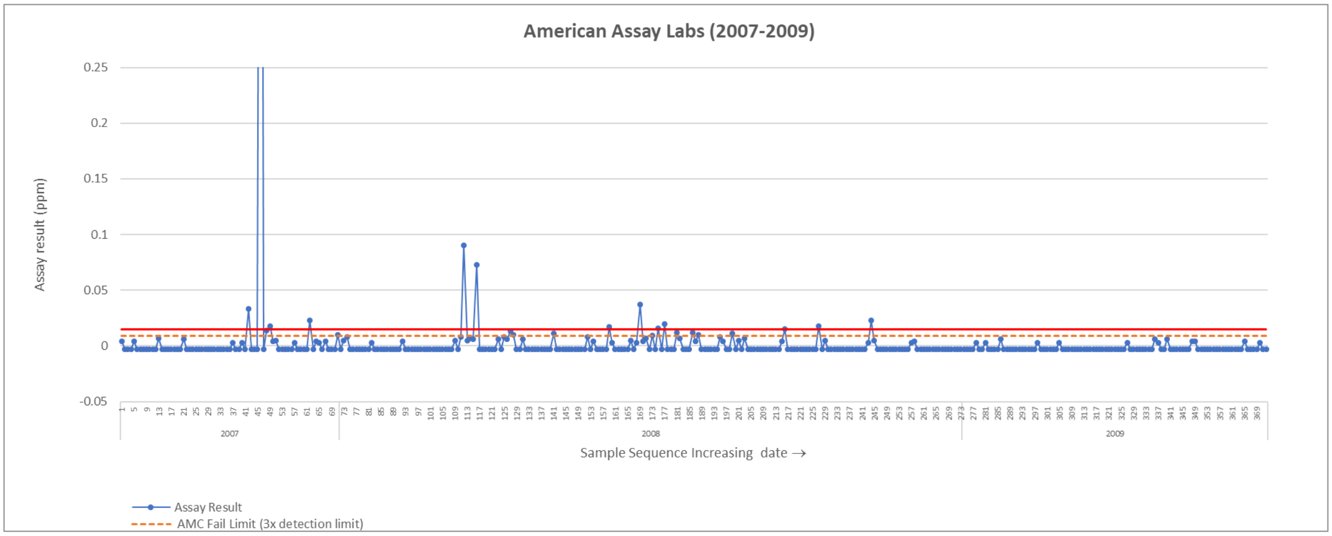
A total of 86% of IAL blanks reported less than three times the detection limit of 0.005 ppm Au. A total of 89% of AAL blanks reported less than the three times the detection limit of 0.003 ppm Au. OMC reviewed blank failures and noted a number of instances where failures appear associated with contamination from preceding samples of high-grade material. Despite some possible level of contamination, the QP does not consider the relatively few numbers of blank failures to be a material concern for a global, long-term Mineral Resource estimate.

Figure 11.9 Inspectorate Assay Labs (2005 – 2006) blank performance chart



Note: Chart related to drillholes drilled between 2005 and 2006.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

Figure 11.10 American Assay Labs (2007 – 2009) blank performance chart



Note: Chart related to drillholes drilled between 2007 and 2008. Assaying continued into 2009.
Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC.

11.9.3 Recommendations on blanks

The QP recommends that both coarse and pulp blanks are included in future exploration programs. Blank material should be analyzed prior to inclusion in QA/QC programs to ensure the material is below the appropriate analytical detection.

The QP recommends that fine and coarse blank material be included in each batch. The weight of individual blank samples included in the sample stream should be consistent. Blank samples should comprise 5% of the total sample stream. Blank material should be included after recognized high grade samples.

11.10 Duplicates

11.10.1 Description

OMC submitted 261 field duplicates as part of the QA/QC program between 2005 and 2008. Duplicate samples comprised between 0.3 and 1.1% of total submitted samples.

Duplicate samples collected in 2005 – 2006 were obtained by taking a 50-50 split of the sample material from the RC drill cyclone. No documentation on the nature of duplicate samples used in subsequent programs was available.

11.10.2 Discussion on duplicates

Coarse, uncrushed duplicate samples monitor sampling variance (including that arising from crushing), analytical variance, and geological variance.

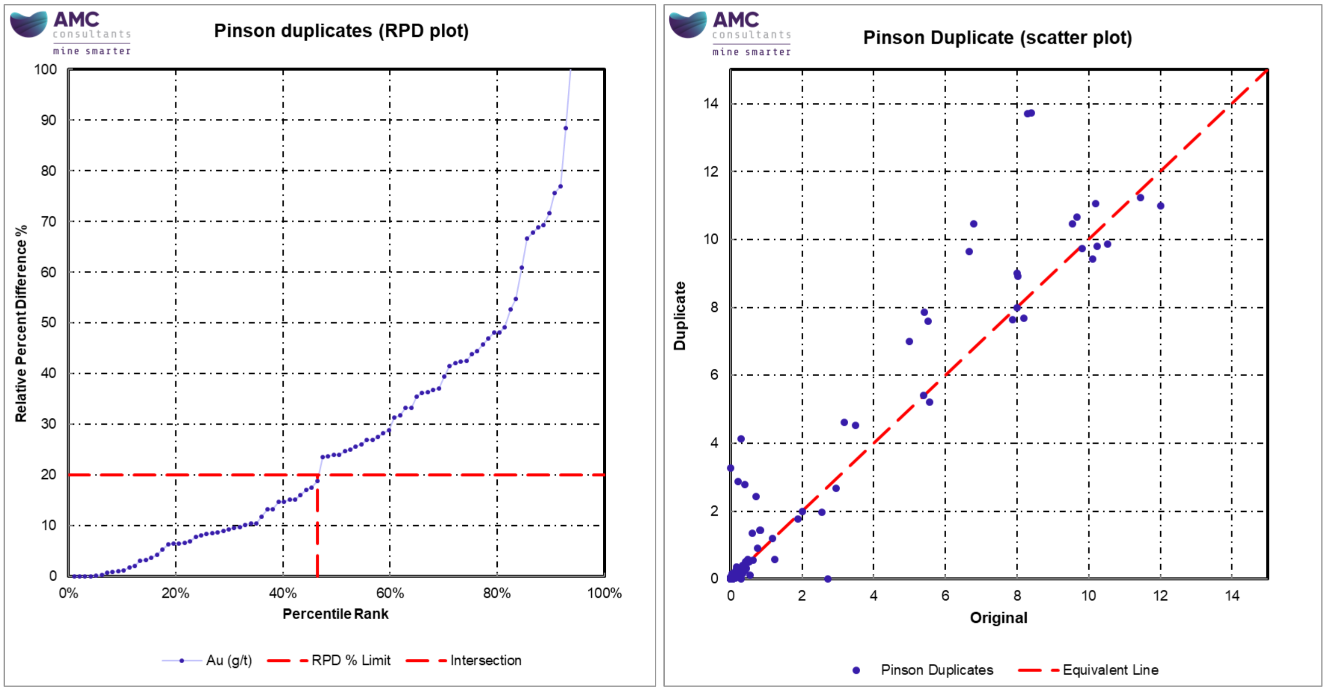
The QP recommends that duplicate samples be selected over the entire range of grades seen on the Project to ensure that the geological heterogeneity is understood. However, the majority of duplicate samples should be selected from zones of mineralization. Unmineralized or very low-grade samples should not form a significant proportion of duplicate sample programs as analytical results approaching the stated limit of lower of detection are commonly inaccurate, and do not provide meaningful assessment of variance.

Duplicate data can be assessed using a variety of approaches. AMC typically assesses duplicate data using both scatterplots and relative paired difference (RPD) plots. These plots measure the absolute difference between a sample and its duplicate. For coarse duplicates, it is desirable to achieve 80 to 85% of the pairs having less than 20% RPD between the original assay and check assay (Stoker 2006). In this analysis, samples less than 15 times the lower limit of analytical detection are excluded.

An RPD plot and scatter plot of duplicate data is presented in Figure 11.11. These plots show that only 46% of samples are within 20% RPD and that duplicate samples are positively biased, and on average have a 16% higher grade than original samples. The QP was unable to determine the cause of sample bias based on historical data.

Whilst the proportion of duplicate samples with assay values within 20% RPD is less than desirable, this is possibly due to the combination of the heterogenous nature of mineralization, uncrushed nature of samples, and sampling variance. The bias seen between original and duplicate samples is concerning. As original samples are biased low, relative to duplicates, this may introduce a level of conservatism into Mineral Resource estimates.

Figure 11.11 Duplicate RPD and scatter plot



11.10.3 Recommendations on duplicates

The QP recommends the following:

- Further investigative work be completed to assess duplicate performance and sample bias.
- Field duplicates, coarse duplicates and pulp duplicates should be regularly inserted into the sample stream.

11.11 Umpire samples

11.11.1 Discussion on umpire samples

Umpire laboratory duplicates are pulp samples sent to a separate laboratory to assess the accuracy of the primary laboratory (assuming the accuracy of the umpire laboratory). Umpire duplicates measure analytical variance and pulp sub-sampling variance.

The 2007 Report suggests that limited umpire sampling was completed at ALS Laboratories (ALS) however AMC was not able to locate any umpire sample data within the drillhole database.

Based on the 2007 Report, umpire samples were selected by identifying mineralized zones with initial assay results greater than 0.1 opt Au. Lower grade material, occurring within these zones was also included. A list of samples was compiled and send to the Inspectorate Laboratories, where original pulps were collected and subsequently dispatched to ALS. Samples were analyzed at ALS using a 30-gram sub-sample of the original pulp using standard fire assay with gravimetric finish.

11.11.2 Recommendations on umpire samples

The QP recommends that if historical pulps are available in the areas of the current Mineral Resource, that umpire sampling be completed. Umpire samples should comprise 5% of total samples originally submitted.

11.12 Conclusions

Drilling programs completed at the Property between 2005 and 2015 have included QA/QC monitoring programs which have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams. The QP has compiled and reviewed available QA/QC data.

In general, the QA/QC sample insertion rates used fall below general accepted industry standards.

CRM samples shows a reasonable level of accuracy, but poor to moderate precision when using standard deviations provided by the CRM supplier.

Blank sample results are considered acceptable and suggest no systematic contamination has occurred throughout the analytical process.

Duplicate sample results show suboptimal performance which may be a result of the heterogenous nature of mineralization, uncrushed samples and sampling variance. Overall duplicate samples appear to be positively biased, with duplicate results returning higher grade than original samples.

Previous reporting suggests that umpire sampling has been completed at the Property. The results of this sampling were not available in the drillhole database and therefore the QP was not able to assess accuracy of the primary laboratory.

Despite the concerns highlighted above, the QP does not consider these issues to be material to the global, long term Mineral Resource estimate. The QP however cannot guarantee that there are no material impacts on the local scale. Overall, the QP considers the assay database to be acceptable for Mineral Resource estimation.

12 Data verification

12.1 Site visit

From the 19 to 21 March 2019, AMC Principal Geologist Ms Dinara Nussipakynova, P.Geo., visited the Property to undertake the following verification steps:

- Discussions with site geologists regarding:
  - Sample collection
  - Sample storage
  - Geological interpretation
  - Data validation procedures
  - Survey procedures
  - Exploration strategy
- An inspection of the core shed and drill core intersections.

12.2 Assay data verification

In 2019, under supervision of Ms Nussipakynova, Marissa Ealey of AMC undertook random cross-checks of assay results in the database with original assay results on the assay certificates, or data source for the Open Pit and Underground Mineral Resource areas.

For the Underground area, the data was requested for 3,340 of the 48,179 assays (6.9%) contained within mineralized wireframes. Assay certificates were chosen from the following laboratories: AAL, Inspectorate, ALS, and the Pinson Laboratory. Details of the data received for the Underground area are shown in Table 12.1.

Table 12.1 Data verification certificate status (Underground area)

Description	Number of samples	Percentage of total
Total number of samples selected for verification	3,340	N/A
Original assay certificates	1,507	45%
Excel file from lab	155	5%
Excel file not from lab	313	9%
Handwritten on assay log	401	12%
Missing files	964	29%

Source: AMC Mining Consultants (Canada) Ltd.

Table 12.2 shows the results of the verification for the data received.

Table 12.2 Data verification results (Underground area)

Data	Total	Selected for validation	% samples verified	# errors noted	% errors noted
Samples	48,179	2,376	4.9%	2	0.1%

Source: AMC Mining Consultants (Canada) Ltd.

For the Open Pit areas data was requested for 5,029 of the 77,321, (6.5%) assays used in the estimation. Assay certificates were chosen from the following laboratories: AAL, Inspectorate, BSi Inspectorate, and ALS Chemex. Details of the Open Pit area data verification are shown in Table 12.3. Assay certificates were available for the majority of the request and this table also lists the number of samples selected for validation that had missing assay certificates in the Open Pit area.

Table 12.3 Data verification results and certificate status (Open Pit area)

Data	Total	Selected for validation	# missing certificates	% missing certs	# samples with certificates	% samples verified	# errors noted	% errors noted
Samples	77,321	5,029	651	0.8	4,378	5.7	143	3.3%

Source: AMC Mining Consultants (Canada) Ltd.

12.3 Collar data verification

In addition to the assay verification, the QP checked collar locations against the provided topography and found that many collars were either above or below topography.

Specifically, the QP noted that drillhole collars were located above topography by a maximum of 65 ft and below topography by a maximum of 145 ft.

The QP notes that in some instances these offsets were due to changes to topography since the time of drilling. Examples follow:

- Drillholes were drilled from surface prior to mining.
- Stockpiles were built at a later date modifying the topography.
- Some drillholes were drilled from benches that have now been mined-out.

OMC validated the collars that the QP highlighted and updated 110 collar locations.

12.4 Data validation

Data validation was carried out using the normal routines in Datamine where the database was checked for collar, survey, and assay inconsistencies, overlaps, and gaps.

12.5 Recommendations

The QP recommends that OMC implement the following:

- Drillhole collars be re-surveyed if they can still be located on the ground.
- Missing original assay certificates, downhole survey logs, original geology, and alteration logs as well as additional records on the density should be located if possible.

12.6 Conclusions

The QP does not consider these issues to have a material impact on Mineral Resource estimates. The QP considers the assay database to be acceptable for Mineral Resource estimation. As discussed in Section 14, Mineral Resource classification takes into the account the presence or absence of original assay certificates.

13 Mineral processing and metallurgical testing

13.1 Introduction

Metallurgical testwork programs were conducted between 1999 and 2013 by metallurgical laboratories on behalf of Homestake (1999) and Atna (2005 / 2006 and 2013 / 2004). The metallurgical testwork programs were completed on samples from the Mag and CX open pit deposits, and the Ogee underground deposit.

The Pinson mine was an operating open pit mine, processing oxide ores through both a conventional mill and heap leach processing facility from 1980 to 1999. Historical production (Section 6.2, Table 6.1 indicated very high recoveries, some in excess of 100%. The QP considers this data to be unreliable and not suitable for estimation of potential gold recoveries from oxide ores. High grade ore extracted from the Ogee deposit between 2012 and 2013 was historically trucked eight miles to Newmont Mining Corporation's Twin Creeks autoclave facility for processing to produce gold bullion. Gold recoveries from the autoclave processing route ranged from 69.2% to 92.6%.

The QP has reviewed the historical metallurgical testwork programs on Pinson feed material including:

- Report on Heap Leach, Direct and CIL Cyanidation, and "Preg-Robbing" Tests – Various Mag Pit Samples and Composites, and CX Pit Bulk Ore, MLI Job No. 2532, Addendum, and Change Orders #1, #2, and #3, March 1999.
- Summary Report on Ore Variability Testing – Mag Pit Pinson Drill Core Composites, MLI Job. No. 3746, 7 February 2013.
- Summary Report on Heap Leach Cyanidation Testing – Mag Pit Pinson Drill Core Composites, MLI Job No. 3746, 26 December 2013.
- Summary Report on Heap Leach Cyanidation Testing – Mag Pit Pinson Drill Core Composites, MLI Job No. 3746, 16 January 2014.
- Pinson Underground Autoclave-Cyanide Leach Tests April 2006.
- Results of Sample Preparation and Head Analysis on Ogee Samples April 2006.
- Wilmot Metallurgical Consulting Met Testwork Results Atna-Pinson Project.
- Dawson Metallurgical Report (14 April 2006).
- Dawson autoclave leach report.

Background on the laboratories used is shown in Table 13.1.

Table 13.1 Laboratories used for testwork

Year	Company	Laboratory	Samples
1999	Homestake	McClelland Laboratories	Mag and CX pits
2005-2006	Atna Resources Ltd	Dawson Metallurgical Laboratories	Ogee underground
2013-2014	Atna Resources Ltd	McClelland Laboratories	Mag pit

Notes:
- Dawson are now part of FL Smidt who are an ISO certified organisation.
- McClelland is accredited under the International Accreditation Service (IAL and also the ILAC-MRA).
- Both are independent of Premier.

Based on available data, the QP considers that for the purpose of this Report, using heap leaching for the Mag Pit and CX Pit material is reasonable, and it is reasonable to assume that gold recoveries between 48% to 82% for Mag Pit and 82% for the CX Pit are achievable by this process. The QP also considers that using autoclave pre-treatment on the underground Ogee material is reasonable, it is reasonable to assume that gold recoveries between 78% to 95% are achievable.

Therefore, these gold recoveries form the basis for the Report.

Tables are from various reports and the units have been left as direct quotes.

13.2 Homestake

In 1999, McClelland Laboratories completed a testwork program on samples sourced from the Mag Pit and CX Pit on behalf of Homestake.

The following scope of work was completed as part of this program:

- Head assays including gold, sulphur speciation and total organic carbon (TOC).
- Preg-robbing tests.
- Cyanide leach bottle roll tests including direct cyanide leach and carbon-in-leach (CIL tests).
- Column leach tests.

This testwork program was completed on the following samples:

- Six Mag Pit bulk ore samples that were sampled from the pit. These samples were labelled as "Mag Pit I" through to "Mag Pit VI".
- One bulk ore sample from the CX Pit labelled "C2".
- Six Mag Pit composites that were made up of drilled cuttings material. These were labelled as "Mag Pit Cuttings Composite 1" through to "Mag Pit Cuttings Composite 6".
- Five Mag Pit composites that were made up of drill core material. These were labelled as "Mag Pit Drillcore Composite 1" through to "Mag Pit Drillcore Composite 5".
- A Mag Pit master composite which was made up of the Mag Pit drill core composites.

The drillhole IDs and intervals used to make up the Mag Pit cuttings and drill core composites are shown in Table 13.2.

Table 13.2 Mag Pit composites for 1999 testwork program

Sample	Drillhole ID	Interval (ft)	
		From	To
Cuttings			
Mag Pit cuttings composite 1	HPC-129	505	555
		600	615
Mag Pit cuttings composite 2	HPC-109	155	165
		190	200
	HPC-129	560	570
		580	585
Mag Pit cuttings composite 3	HPC-129	455	465
		490	495
		570	580
		585	590
Mag Pit cuttings composite 4	HPC-109	255	275
		280	290
		310	320
Mag Pit cuttings composite 5	HPC-129	210	220
		255	270
		275	285
		470	485
	HPC-109	200	205
		215	235
Mag Pit cuttings composite 6	HPC-109	275	280
		290	295
		300	305
		320	340
		345	350
		360	370
Drill core			
Mag Pit drill core composite 1	HPC-142	65	105
Mag Pit drill core composite 2	HPC-142	105	180
Mag Pit drill core composite 3	HPC-142	180	230
Mag Pit drill core composite 4	HPC-143	30	85
		105	135
Mag Pit drill core composite 5	HPC-143	135	235

The proportions of the drill core composites used to make the Mag Pit master drill core composite are shown in Table 13.3. The proportions were selected on a footage weighted basis from the five individual composites.

Table 13.3 Mag Pit master drill core composite proportions from 1999 testwork program

Sample	% of composite
Mag Pit drill core composite 1	11.6%
Mag Pit drill core composite 2	21.7%
Mag Pit drill core composite 3	14.5%
Mag Pit drill core composite 4	24.6%
Mag Pit drill core composite 5	27.6%

Preg-robbing tests were completed on some of the samples to determine the preg-robbing characteristics of the Mag Pit and CX Pit samples. In these tests, barren solutions were "spiked" with a diluted gold solution with a concentration of approximately 1 milligram per litre (mg/L) gold and added to the test slurries. A standard cyanide leach bottle roll test was completed on the slurry with the spiked solution. The pregnant leach solution was then assayed for gold at regular intervals. The percentage of gold that was preg-robbed was determined by the following formula:

$$\frac{\text{Original gold solution concentration} - \text{Final gold solution concentration}}{\text{Original gold solution concentration (\%)}}$$

Preg-robbing testwork completed in this manner is of limited usefulness as the effect of further dissolution of gold during the test is not isolated from the result. More useful preg-robbing test methodologies complete two tests in parallel. The first test is completed without the spiked solution and the other test, using the same sample, is completed with the spiked solution. The difference in gold recovery between the non-spiked test and the spiked test gives an indication of gold recovery lost due to preg-robbing.

Table 13.4 shows the percentages of preg-robbed gold. Negative values indicate where the final gold concentration was higher than the original gold concentration.

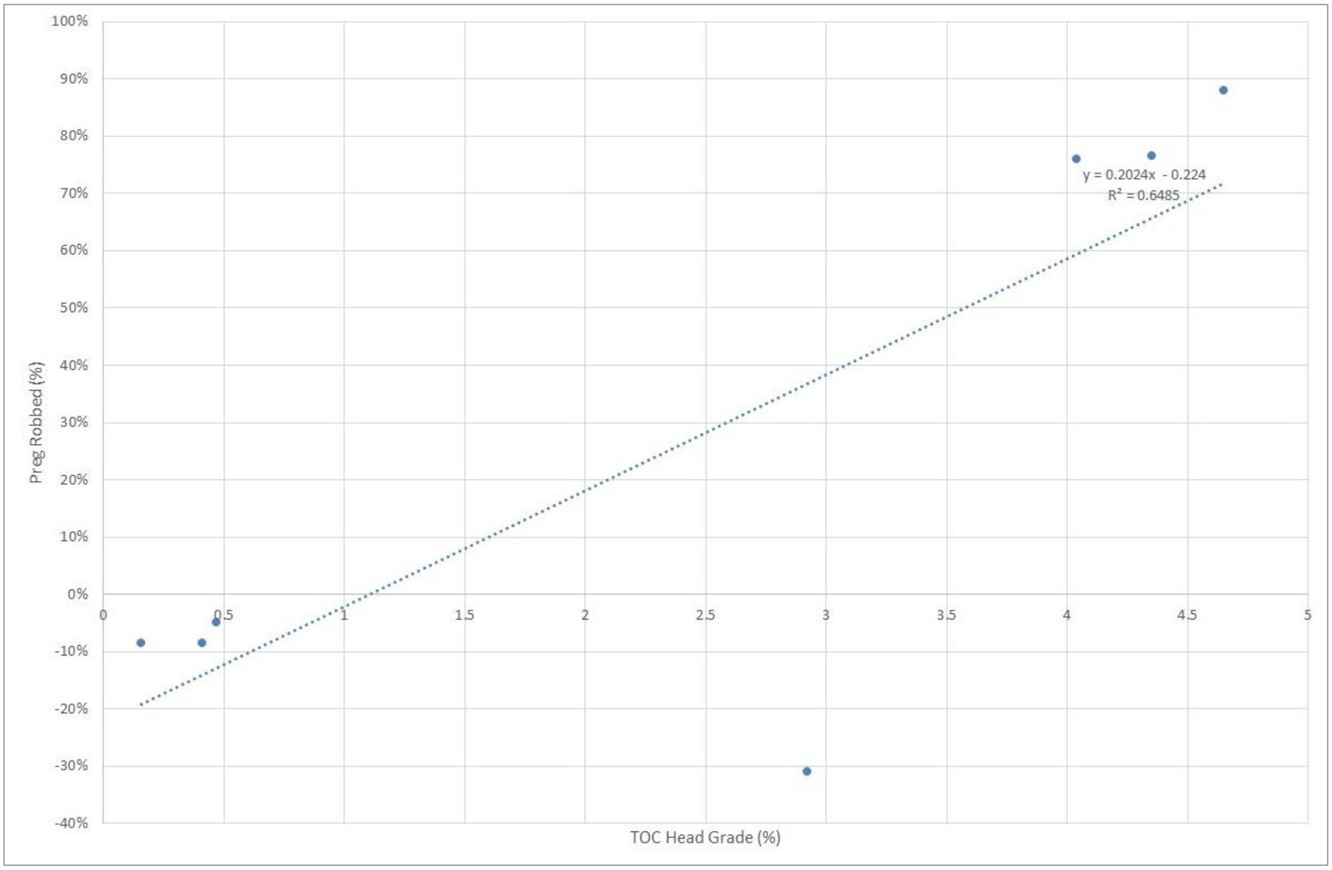
Table 13.4 Preg-robbing test results from the 1999 testwork program

Sample	Sample type	Feed size	Preg-robbed gold (%)
Mag Pit I	Bulk ore	P_{80} 3/4" (19 mm)	87.9%
Mag Pit II	Bulk ore	P_{80} 3/4" (19 mm)	76.6%
Mag Pit III	Bulk ore	P_{80} 3/4" (19 mm)	-8.6%
Mag Pit IV	Bulk ore	P_{80} 3/4" (19 mm)	76.0%
Mag Pit V	Bulk ore	P_{80} 3/4" (19 mm)	-8.6%
Mag Pit VI	Bulk ore	P_{80} 3/4" (19 mm)	-5.0%
CX-2	Bulk ore	P_{80} 3/4" (19 mm)	-31.0%
Mag Pit cuttings composite 1	Drill core	10 Mesh (1.7 mm)	14.1%
Mag Pit cuttings composite 2	Drill core	10 Mesh (1.7 mm)	14.9%
Mag Pit cuttings composite 3	Drill core	10 Mesh (1.7 mm)	15.5%
Mag Pit cuttings composite 4	Drill core	10 Mesh (1.7 mm)	19.1%
Mag Pit cuttings composite 5	Drill core	10 Mesh (1.7 mm)	60.5%
Mag Pit cuttings composite 6	Drill core	10 Mesh (1.7 mm)	47.4%

Table 13.4 shows that many of the samples had relatively high preg-robbing values (greater than 50%), demonstrating that preg-robbing is a potential issue when processing Pinson material.

Figure 13.1 shows the preg-robbed percentage vs. the TOC head grade of the feed material.

Figure 13.1 Preg-robbed percentage vs. TOC head grade from 1999 testwork program



Source: AMC Consultants Pty Ltd based on McClelland Laboratories, Inc. 1999.

Figure 13.1 shows a positive linear trend between the preg-robbed percentage and the TOC head grade. There is one outlier where the TOC grade was 2.9% and the preg-robbed value is -31%. It was unclear in the report as to why this may have occurred.

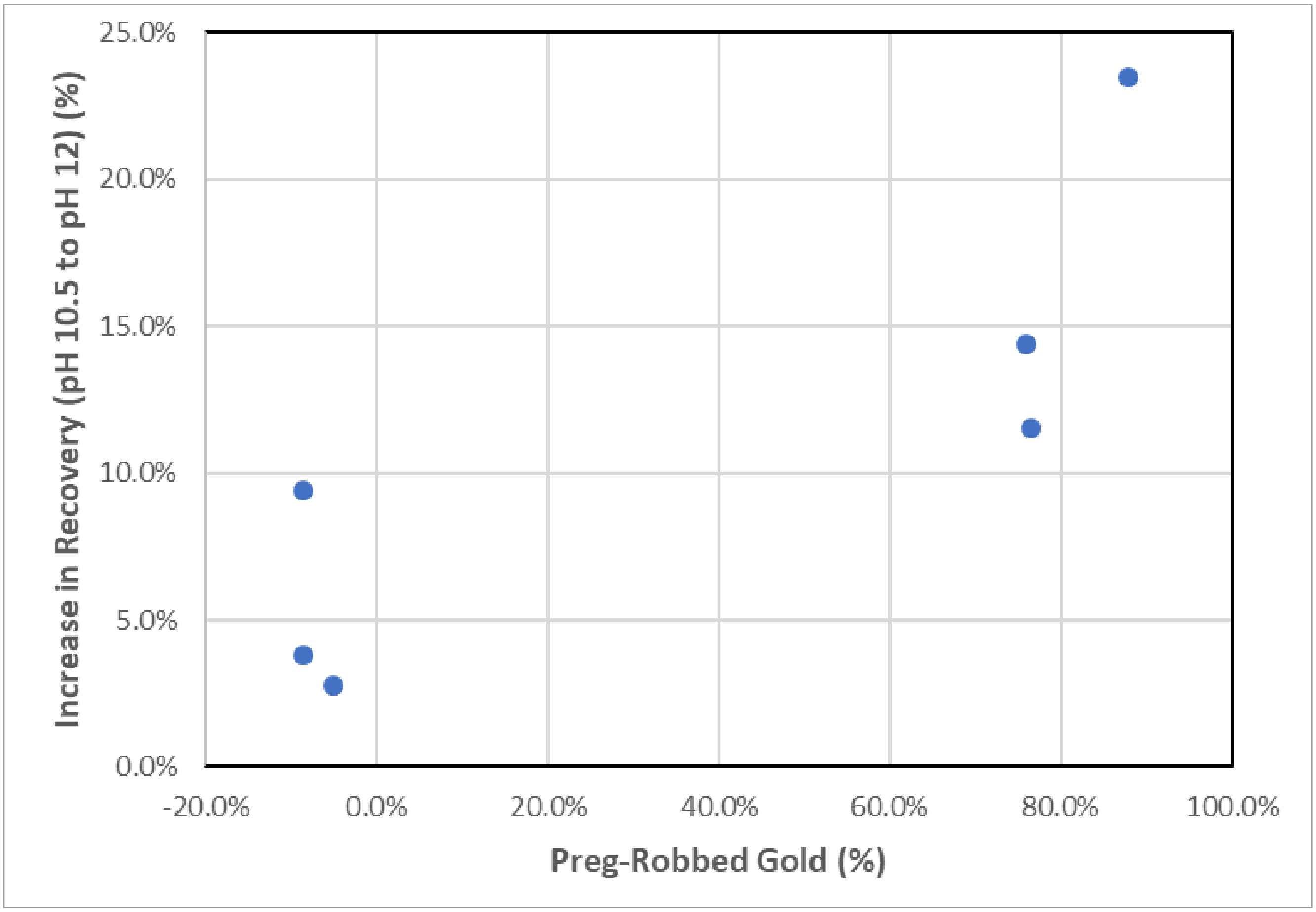
Cyanide leach bottle roll tests were completed on the Mag Pit bulk ore samples using caustic soda (NaOH) to adjust pH, rather than hydrated lime. The testwork report postulated that NaOH passivates the preg-robbing (carbonaceous) surfaces by occupying active carbon sites with OH ions so the $Au(CN)_2$- ions do not absorb onto the active carbon sides. Reducing the amount of $Au(CN)_2$- ions which absorb onto the carbon sites would improve gold recovery. For each sample, two tests were conducted at pH 10.5 and pH 12 (using NaOH to adjust pH). The results of these tests are shown in Table 13.5.

Table 13.5 NaOH bottle roll tests from 1999 testwork program

Sample	Preg-robbing factor (%)	NaOH tests				
		pH 10.5 tests		pH 12.0 tests		Difference in gold recovery between pH 12.0 tests and pH 10.5 tests
		Gold recovery (%)	Cyanide consumption (lb/short tonne)	Gold recovery (%)	Cyanide consumption (lb/short tonne)	
Mag Pit I	87.9	8.5	1.1	32.0	1.0	23.5%
Mag Pit II	76.6	13.2	3.0	24.7	1.5	11.5%
Mag Pit III	-8.6	74.2	1.4	83.6	0.7	9.4%
Mag Pit IV	76.0	26.4	1.6	40.8	0.5	14.4%
Mag Pit V	-8.6	50.0	1.8	53.8	0.4	3.8%
Mag Pit VI	-5.0	62.2	1.1	65.0	0.3	2.8%

The test data shows that higher pH tests (where there was a higher NaOH addition) showed an increase in gold recovery (Figure 13.2) and an associated reduction in cyanide consumption.

Figure 13.2 Effect of pH on gold recovery



Source: McClelland Laboratories, Inc. 1999.

Generally, the largest recovery increases between the pH 12.0 tests and the pH 10.5 tests, were associated with samples which showed the highest preg-robbed gold.

There were no baseline tests using lime on these samples, so a proper comparison between the use of lime and NaOH cannot be completed.

CIL tests were completed on the CX-2 bulk ore and Mag Pit cuttings samples. The objective of these tests was to test the applicability of CIL processes (such as a CIL agitated tank circuit) to Pinson open pit material. The conditions of these tests were:

- Tests were conducted in agitated bench-scale beakers.
- Samples were ground to a P_{80} of 200 mesh (75 µm).
- 72 hours residence time.
- Kinetic samples taken at 6 hours, 12 hours, 24 hours, 36 hours, and 48 hours.
- Hydrated lime was added to raise the pH to 10.5.
- A sodium cyanide (NaCN) concentration of 1 gram per litre (g/L).
- Pulp density of 40% solids weight for weight (w/w).
- Activated carbon was added to absorb the gold in solution onto the carbon.

The results of these tests are shown in Table 13.6.

Table 13.6 CIL tests from 1999 testwork program

Sample	Gold recovery (%)	Cyanide consumption (lb/short tonne)
CX-2 Bulk ore	88.2	2.4
Mag Pit cuttings composite 1	94.0	3.3
Mag Pit cuttings composite 2	75.3	2.3
Mag Pit cuttings composite 3	59.7	3.0
Mag Pit cuttings composite 4	82.9	4.8
Mag Pit cuttings composite 5	55.0	3.0
Mag Pit cuttings composite 6	87.5	3.9

These tests generally achieved high gold recoveries (greater than 75%) (with the exception of the Mag Pit 3 and Mag Pit 5 samples); which demonstrates that the Pinson open pit material is generally amenable to CIL processes.

Column leach tests were conducted on some of the samples from the 1999 program. The conditions of these tests were:

- Residence time of between 50 and 90 days. If the kinetic leach curve demonstrated that a test was approaching terminal gold recovery, the test was stopped.
- Varying crushed sizes.
- Hydrated lime was added for agglomerate the material in the column.
- Lime was added to most tests to raise the pH to 10.5.
- NaOH was added to the Mag Pit I and Mag Pit II samples, given the successful NaOH bottle roll tests on these samples. The pH was initially 10.5 but was increased to 12.0 later in the test to ascertain the impact on leaching.
- NaCN was added at an initial concentration of 1 g/L and was pumped into the columns at a rate of 0.005 gpm/ft^2 of cross-sectional area.
- Three tests with varying particle sizes were conducted on the CX Pit sample to ascertain the impact of crush size on gold recovery.

Three tests were conducted on the Mag Pit master composite where pH and alkali were varied:

- Test 1: pH 10.5 (lime).
- Test 2: pH 11.8 (lime).
- Test 3: pH 11.8 (NaOH).

Table 13.7 shows the results from the from the 1999 column leach tests.

Table 13.7 Column leach tests from 1999 testwork program

Sample	Sample type	Feed size (inches)	Gold recovery (%)	Cyanide consumption (lb/short tonne)
Mag Pit I	Bulk ore	-4″	18.8	9.9
Mag Pit II	Bulk ore	-4″	35.3	9.0
Mag Pit III	Bulk ore	-4″	93.1	4.6
Mag Pit IV	Bulk ore	-4″	49.5	5.3
Mag Pit V	Bulk ore	-4″	51.7	3.9
Mag Pit VI	Bulk ore	-4″	60.7	3.7
Mag Pit 2	Drill core	-1″	69.0	4.0
Mag Pit 3	Drill core	-1″	62.0	1.6
Mag Pit 4	Drill core	-1″	47.9	1.5
Mag Pit 5	Drill core	-1″	61.7	2.1
Mag Pit master (pH 10.5, Lime)	Drill core	-1″	65.0	6.3
Mag Pit master (pH 11.8, Lime)	Drill core	-1″	70.7	4.2
Mag Pit master (pH 11.8, NaOH)	Drill core	-1″	69.0	3.5
CX Pit, CX-2	Bulk ore	-6″	77.7	5.1
CX Pit, CX-2	Bulk ore	P_{80} 3″	81.7	4.8
CX Pit, CX-2	Bulk ore	P_{80} 3/4″	82.2	5.4

This testwork program had the following findings:

- There was a wide range of gold recoveries, varying from 19% to 93%.
- The tests on the Mag Pit drill core samples (with a sizing of -1″) generally had higher recoveries than the tests on the Mag Pit bulk samples (with a feed sizing of -4″).
- There was only marginal improvement in gold recovery by crushing finer in the gold recovery-by-size tests on the CX Pit.

The tests on the Mag Pit master composite sample had the following conclusions:

- Increasing pH demonstrated an increase in gold recovery.
- The NaOH had a slightly lower gold recovery than the lime test (pH 11.8).

13.3 Atna Resources Ltd.

Atna commissioned a number of metallurgical testwork programs to be completed on Pinson samples.

13.3.1 McClelland Laboratories

McClelland Laboratories completed a metallurgical testwork program on Mag Pit samples on behalf of Atna in 2013 / 2014.

The following scope of work was completed as part of this program:

- Head assays including gold, sulphur speciation, TOC and a preg-rob assay using 0.1 troy ounces per short ton (oz/ton) spiked bottle roll test.
- Cyanide leach bottle roll tests.
- Column leach tests.

A summary of the drillholes and intervals used to make up the samples for this program are shown in Table 13.8.

Table 13.8 Sample composite list from 2013 testwork program

Drillhole	Sample	Interval	
		To (ft)	From (ft)
Magmet-001	Magmet-001-01	0.0	27.5
	Magmet-001-02	27.5	99.5
	Magmet-001-03	99.5	157.0
	Magmet-001-04	170.5	228.5
	Magmet-001-05	228.5	251.5
	Magmet-001-06	251.5	302.5
	Magmet-001-07	302.5	364.5
	Magmet-001-08	364.5	415.5
Magmet-002	Magmet-002-01	211.5	254.5
	Magmet-002-02	254.5	292.0
	Magmet-002-03	292.0	337.0
	Magmet-002-04	337.0	397.0
	Magmet-002-05	397.0	446.0
	Magmet-002-06	450.0	497.0
	Magmet-002-07	497.0	567.5
	Magmet-002-08	567.5	599.8
Magmet-003	Magmet-003-01	179.0	225.0
	Magmet-003-02	225.0	283.0
	Magmet-003-03	283.0	304.5
	Magmet-003-04	304.5	361.5
	Magmet-003-05	361.5	409.0
	Magmet-003-06	409.0	459.5
	Magmet-003-07	459.5	514.0
Magmet-004	Magmet-004-01	125.0	148.0
	Magmet-004-02	148.0	220.5
	Magmet-004-03	220.5	270.0
	Magmet-004-04	270.0	330.0
	Magmet-004-05	330.0	372.0
	Magmet-004-06	372.0	415.0
	Magmet-004-07	415.0	439.0
	Magmet-004-08	439.0	496.0
	Magmet-004-09	496.0	551.5

Cyanide leach bottle roll tests were conducted on Mag Pit samples. The objective of these tests was to identify the impact on particle size on gold recovery. The conditions for the bottle roll tests were:

- Two bottle roll tests were completed on each sample type at two sizes - a P_{80} of ¼" (6.35 mm) and a nominal 150 Mesh (105 microns).
- A pulp density of 40% solids w/w.
- Hydrated lime was added to raise the pH to 10.8 to 11.2.
- NaCN was added at a concentration of 1 g/L.
- The residence time was 48 hours.
- Kinetic samples were taken at 6 hours, 12 hours, 24 hours, and 36 hours.

Preg-rob factors were measured for each sample. It was unclear from McClelland's report what method was used to measure the preg-rob factor.

A summary of the results from the bottle roll tests is shown in Table 13.9.

Table 13.9 Bottle roll tests results from 2013 testwork program

Sample	Au head grade (oz/ton)	Preg-rob factor	Gold recovery (%)		
			P_{80} 1/4"	150 Mesh	Difference between 150 mesh test and P_{80} 1/4" test
Magmet-001-01	0.031	0	77.4%	81.3%	3.9%
Magmet-001-02	0.089	93	50.0%	47.8%	-2.2%
Magmet-001-03	0.032	0	72.0%	82.1%	10.1%
Magmet-001-04	0.030	36	57.1%	69.2%	12.1%
Magmet-001-05	0.057	97	17.4%	6.0%	-11.4%
Magmet-001-06	0.050	92	24.4%	27.8%	3.4%
Magmet-001-07	0.058	100	4.3%	5.8%	1.5%
Magmet-001-08	0.018	87	20.0%	25.0%	5.0%
Magmet-002-02	0.030	92	8.3%	9.5%	1.2%
Magmet-002-03	0.005	84	50.0%	40.0%	-10.0%
Magmet-002-04	0.128	76	78.4%	86.4%	8.0%
Magmet-002-05	0.114	78	50.0%	70.7%	20.7%
Magmet-002-06	0.043	95	15.4%	17.1%	1.7%
Magmet-003-04	0.014	0	53.8%	71.4%	17.6%
Magmet-003-05	0.059	10	76.8%	87.0%	10.2%
Magmet-003-06	0.022	11	75.0%	80.0%	5.0%
Magmet-003-07	0.044	48	48.7%	81.0%	32.3%
Magmet-004-04	0.018	42	42.9%	55.6%	12.7%
Magmet-004-05	0.006	15	50.0%	71.4%	21.4%
Magmet-004-06	0.018	19	47.1%	75.0%	27.9%
Magmet-004-08	0.023	10	42.9%	54.5%	11.6%

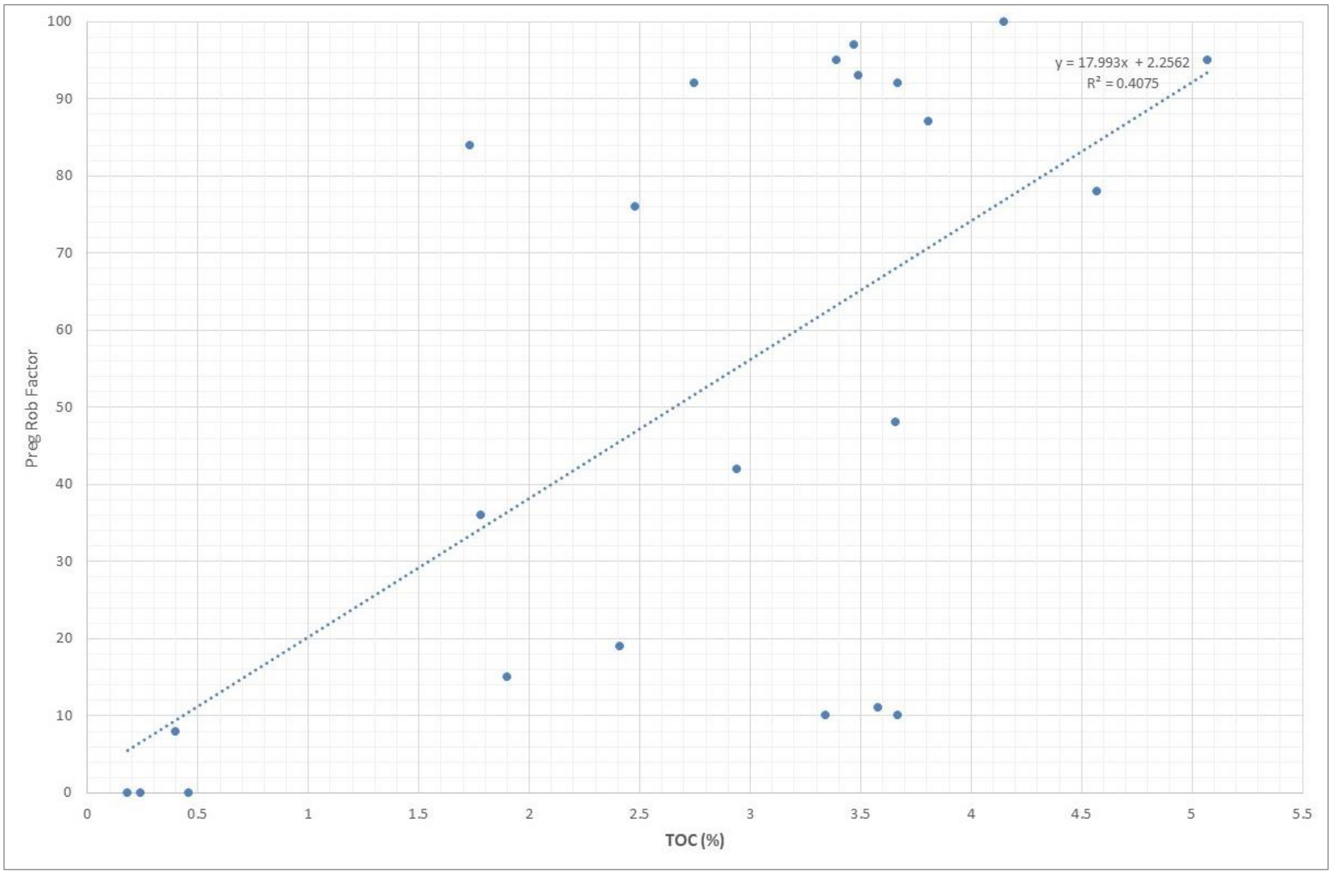
The following findings from these tests were identified:

- There was a range of gold recoveries.
- In general, reducing the feed sizing increased gold recovery.
- Many of the samples had high preg-robbing factors (>50).

McClelland postulated in the report that samples with low recoveries were most likely due to refractory gold (in sulphide minerals) or preg-robbing.

Figure 13.3 shows the preg-robbing factor vs. TOC from samples from the 2013 testwork program.

Figure 13.3 Preg-rob factor vs. TOC head grade from 2013 testwork program



Source: AMC Consultants Pty Ltd based on based on McClelland Laboratories, Inc. February 2013.

Figure 13.3 shows that there is a positive linear trend between the TOC head grade and the preg-robbing factor.

Cyanide leach bottle roll tests were completed on some of the Mag Pit samples with the objective of using NaOH rather than lime for treating preg-robbing. For each sample type, a test was completed at pH 10.5, and another at a pH of 12.0. The conditions of these tests were:

- NaOH was added to raise the pH.
- Samples were crushed to a P_{80} of ¼”.
- Residence time of 48 hours.

Table 13.10 Results from bottle roll tests using NaOH from 2013 testwork program

Sample	Au head grade (oz/ton)	Gold recovery (%)		
		pH 10.5 test	pH 12.0 test	Difference between pH 12.0 test and pH 10.5 test
Magmet-001-02	0.089	53.2	60.8	7.6
Magmet-001-05	0.057	14.0	43.1	29.1
Magmet-002-02	0.030	12.5	20.8	8.3
Magmet-002-06	0.043	20.5	35.1	14.6
Magmet-003-05	0.059	75.0	78.2	3.2

Table 13.10 shows that increasing the pH using NaOH demonstrated an increase in gold recovery for all tests.

Column leach tests were conducted on composites of the Mag Pit samples. The samples used to make up these composites are shown in Table 13.11. It was not clear from the report as to the rationale behind the compositing.

Table 13.11 Sample composition for column leach tests from 2013 testwork program

Sample	% of composite
Mag Column 1	
Magmet-001-01	14.3
Magmet-001-03	15.5
Magmet-001-04	33.2
Magmet-003-05	17.1
Magmet-004-04	19.9
Mag Column 2	
Magmet-002-04	18.5
Magmet-002-05	8.8
Magmet-003-04	28.4
Magmet-003-06	14.5
Magmet-003-07	13.8
Magmet-004-06	13.2
Magmet-004-08	12.8
Mag Column 3	
Magmet-001-02	26.2
Magmet-001-05	6.4
Magmet-001-06	10.1
Magmet-001-07	17.7
Magmet-001-08	13.9
Magmet-002-02	12.2
Magmet-002-06	13.5
Mag Column 4	
Mag Column 1	13.8
Mag Column 2	43.1
Mag Column 3	43.1

Prior to the column leach tests, bottle rolls tests were conducted on each composite to determine gold recovery kinetics and reagent addition rates. The conditions of the bottle roll tests were:

- 48 hours residence time.
- P_{80} of 1/4”.
- Hydrated lime was added to raise the pH to 12.0.
- The conditions of the column leach tests were:
  - Residence time varied between 72 and 76 days.

Generally, the tests were conducted on samples that had been crushed to -2”. The Mag Column 3 sample had an additional test on a sample crushed to ½” to ascertain the impact of size on gold recovery.

Lime was added to agglomerate the ½” column only. The lime was cured in the column for 72 hours prior to applying NaCN. Note, agglomeration is typically only required for particle feed sizes that are 1” or finer.

Lime additions were based on the bottle roll test lime requirements.

1 g/L cyanide was added to the top of the columns at a rate of 0.2 litres per minute/m^2 of column cross sectional area.

The results of the McClelland column tests are shown in Table 13.12.

Table 13.12 Bottle roll and column test results from 2013 testwork program

Sample	Au grade (oz/short ton)	Test type	Feed size (inches)	Leach time (days)	Gold recovery (%)	Cyanide consumption (lb/short ton)
Mag Column 1	0.030	Bottle roll	1/4”	2	52.4	0.6
		Column	2”	73	61.8	2.5
Mag Column 2	0.045	Bottle roll	1/4”	2	71.7	1.1
		Column	2”	76	82.4	3.3
		Column	1/2”	72	82.0	3.8
Mag Column 3	0.038	Bottle roll	1/4”	2	32.5	0.7
		Column	2”	72	50.9	2.6
Mag Column 4	0.039	Bottle roll	1/4”	2	51.2	0.8
		Column	2”	76	65.3	3.0

Table 13.12 shows that the gold recoveries in the column tests varied from 51% to 82%. The Mag Column 2 tests showed there was no benefit to gold recovery by crushing finer.

13.3.2 Dawson metallurgical program

Dawson Metallurgical Laboratories completed metallurgical testwork programs on samples from underground deposits (Ogee) on behalf of Atna. These programs were completed in August 2005 (report date, 30 August 2005) and April 2006 (report date, 14 April 2006).

This testwork program was completed on the following samples:

- A composite form the Ogee underground deposit labelled "Right Rib and Left Rib".
- Composites from the RFZ labelled as:
  - RF_Met-1 (33941)
  - RF_Met-2 (33942)
  - RF_Met-4 (34259)
- Composites from the CX Zone labelled as:
  - APCX-204
  - APCX-211
  - APCX-219
  - APCX-226
- Undefined samples:
  - AMW-002
  - Met1 and Met 2 from the 2005 program
  - Met1 and Met 2 from the 2006 program

The objective of these programs was to ascertain whether autoclave pre-treatment of the feed samples could release refractory gold from the sulphide minerals and improve gold recoveries (relative to baseline tests).

The scope of these testwork programs included:

- Head assays including gold, sulphur speciation, and carbon speciation.
- Baseline cyanide leach shake-out tests on ground feed samples.

Pressure oxidation testwork:

- Grinding of samples to either a P_{80} of 75 µm or 45 µm.

Acid leach stage, where sulphuric acid (H_2SO_4) was added to achieve a pH of 1.8 to 2.0 and processed for one hour. The purpose of this stage was to digest carbonate minerals ahead of the autoclave stage.

The acid leach residue was then processed in an autoclave with the following conditions:

- Temperature of 225°C.
- Residence time of one hour.
- Pulp density of 35% solids w/w.
- Oxygen overpressure of 460 pounds per square inch (psi).

Lime was added to the autoclave residue to raise the pH to a range of 10.0 to 10.5. The autoclave residue then underwent a cyanide leaching shake-out test to determine gold recovery. The shake-out test consists of a shortened (two hours) nitric acid (HNO_3) digestion of the sample, followed by recovery of the solids and shaking of the solids with a cyanide leach solution (0.25% NaCN, 0.10% NaOH) for two hours.

The cyanide leach gold recoveries from the baseline tests and the autoclave tests are shown in Table 13.13. The MET1 and MET2 samples did not undergo baseline cyanide leach tests. It was not clear in the report as to why this was the case.

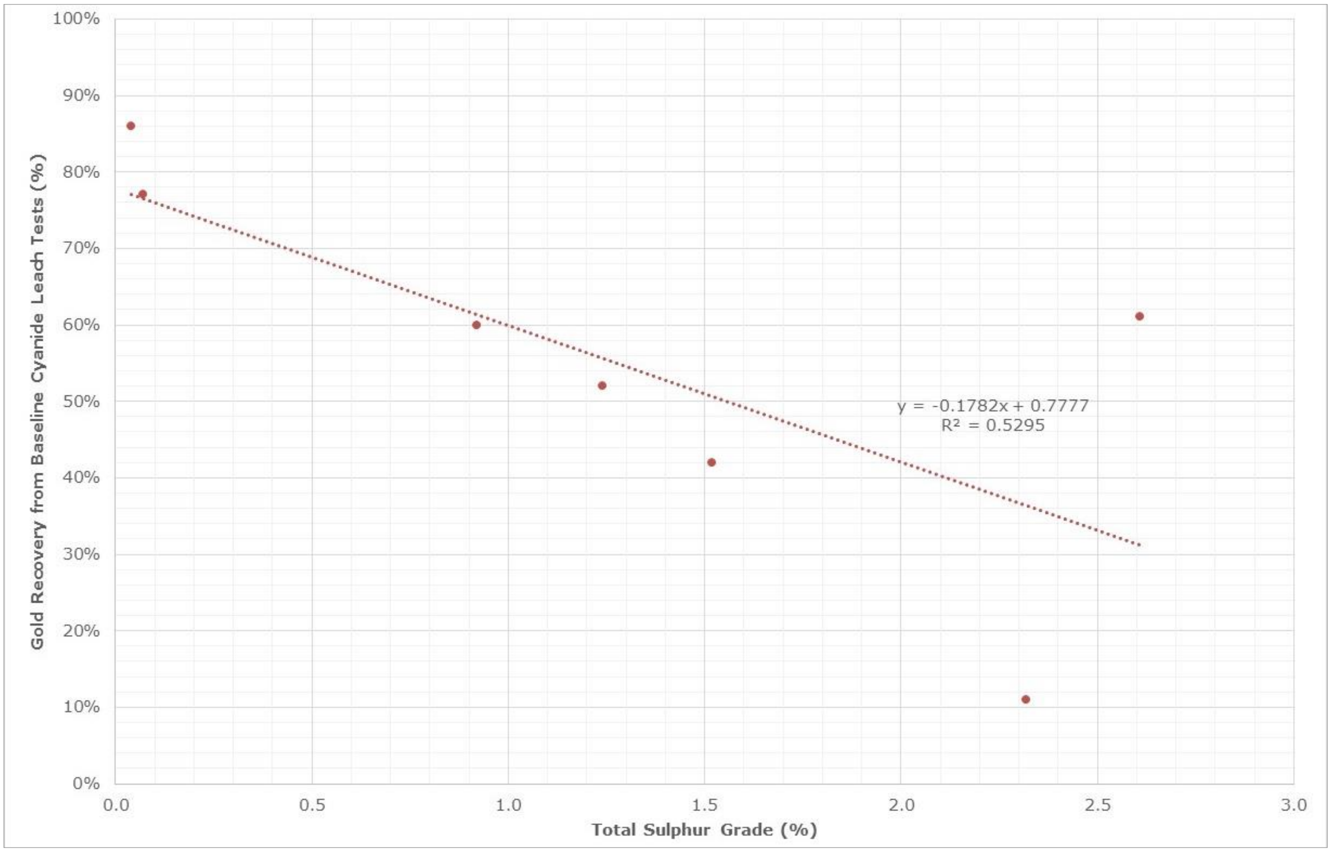
Table 13.13 Autoclave pre-treatment tests from Dawson testwork program

Sample	Year of testwork program	Grind P_{80} (µm)	Gold head assay (oz/ton)	Total sulphur head assay (%)	Total carbon (CO_2) head assay	Cyanide leach gold recovery (%)	
						Baseline tests	Tests on autoclave residue
Ogee samples							
Ogee (Right Rib + Left Rib)	2005	75	0.40	0.04	0.82	86	93
RF Zone samples							
RF_Met-1 (33941)	2005	75	0.24	1.24	2.27	52	93
RF_Met-2 (33942)	2005	75	0.43	2.61	1.76	61	95
RF_Met-4 (34259)	2005	75	0.43	2.32	2.43	11	89
CX Zone samples							
APCX-204	2006	75	0.27	0.00	5.34	94	N/A
APCX-211	2006	75	0.33	0.00	4.29	85	N/A
APCX-219	2006	75	0.33	0.92	0.77	60	91
APCX-226	2006	45	0.56	1.52	2.70	42	94
Undefined samples							
AMW-002	2006	75	0.33	0.07	0.35	77	N/A
MET 1	2005	75	0.51	1.21	2.27	N/A	93
MET 2	2005	75	0.32	2.61	1.76	N/A	95

Table 13.13 shows that pre-treating the material in an autoclave had an increase in gold recovery for all samples. The RF_Met-4 sample recorded the highest increase of 78%.

Figure 13.4 shows the baseline test gold recoveries vs. the total sulphur grade from Dawson testwork program.

Figure 13.4 Baseline gold recovery vs. total sulphur grade program

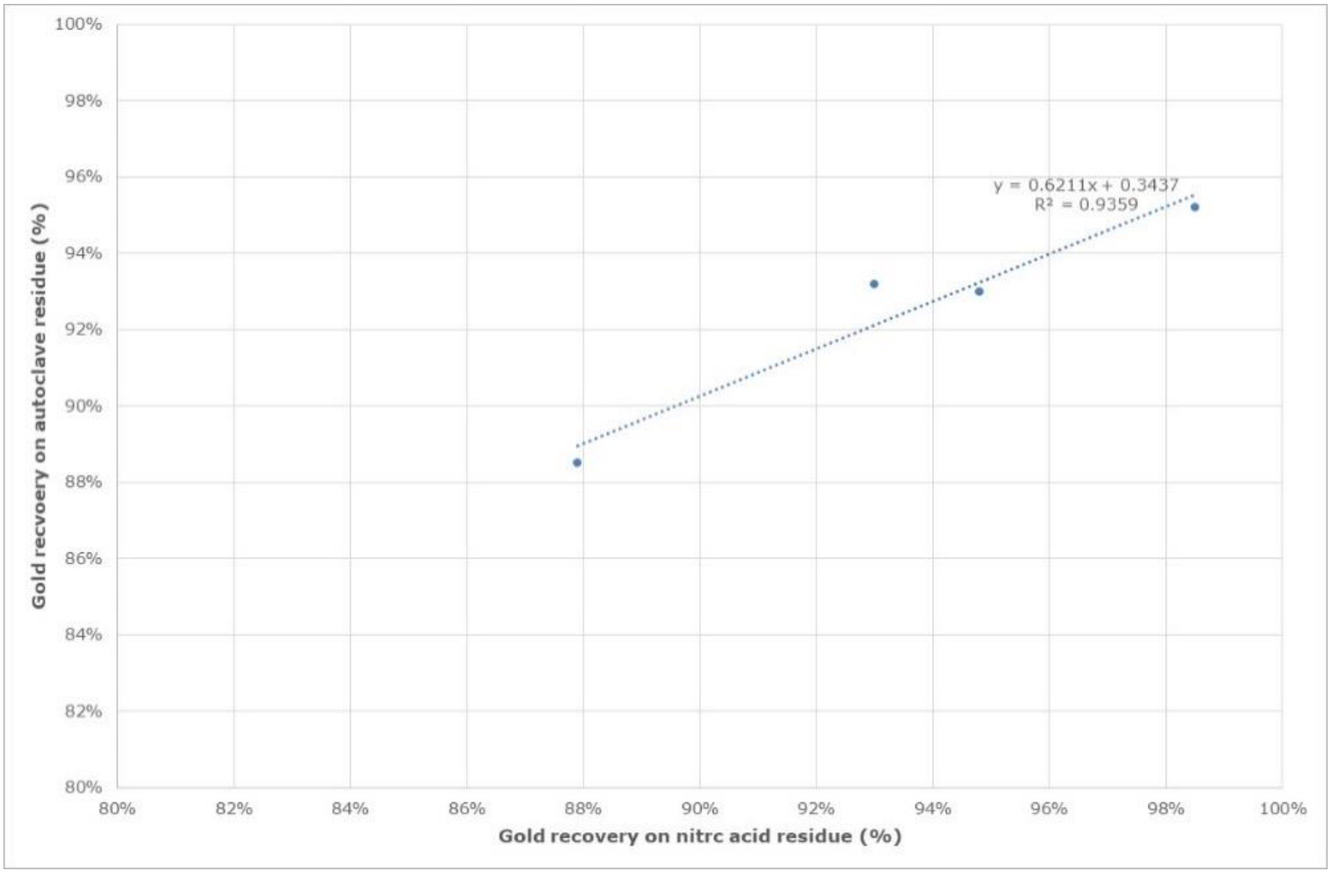


Source: AMC Consultants Pty Ltd based on Dawson 2005.

Figure 13.4 shows a negative linear relationship between the baseline gold recovery and the sulphur head grade. With the exception of the 2.6% S sample, the relationship suggests that a higher sulphur head grade will have more refractory gold that will detrimentally impact gold recovery.

A diagnostic HNO_3 procedure was developed with the objective of providing a proxy test to the autoclave tests. In these tests, a pulverized (to P_{80} of 75 µm) feed sample was digested with a 10% w/w nitric acid solution at 60°C for 1 hour. The nitric leach residue was then processed in a cyanide leach shake-out test with a 0.25% NaCN / 0.10% NaOH solution for two hours. Figure 13.5 shows the gold recovery from the autoclave residue vs. the gold recovery on the nitric acid diagnostic test residue from Dawson testwork program.

Figure 13.5 Autoclave residue gold recovery vs. nitric acid residue gold recovery program



Source: AMC Consultants Pty Ltd based on Dawson 2005.

Figure 13.5 shows there is a very good correlation coefficient (R2) of 0.93, which suggests that the nitric acid diagnostic test is a good predictor of the gold recovery on autoclave bench test residues.

It was noted that there were samples labelled MET 1 and MET 2 in each of the 2005 and 2006 programs.

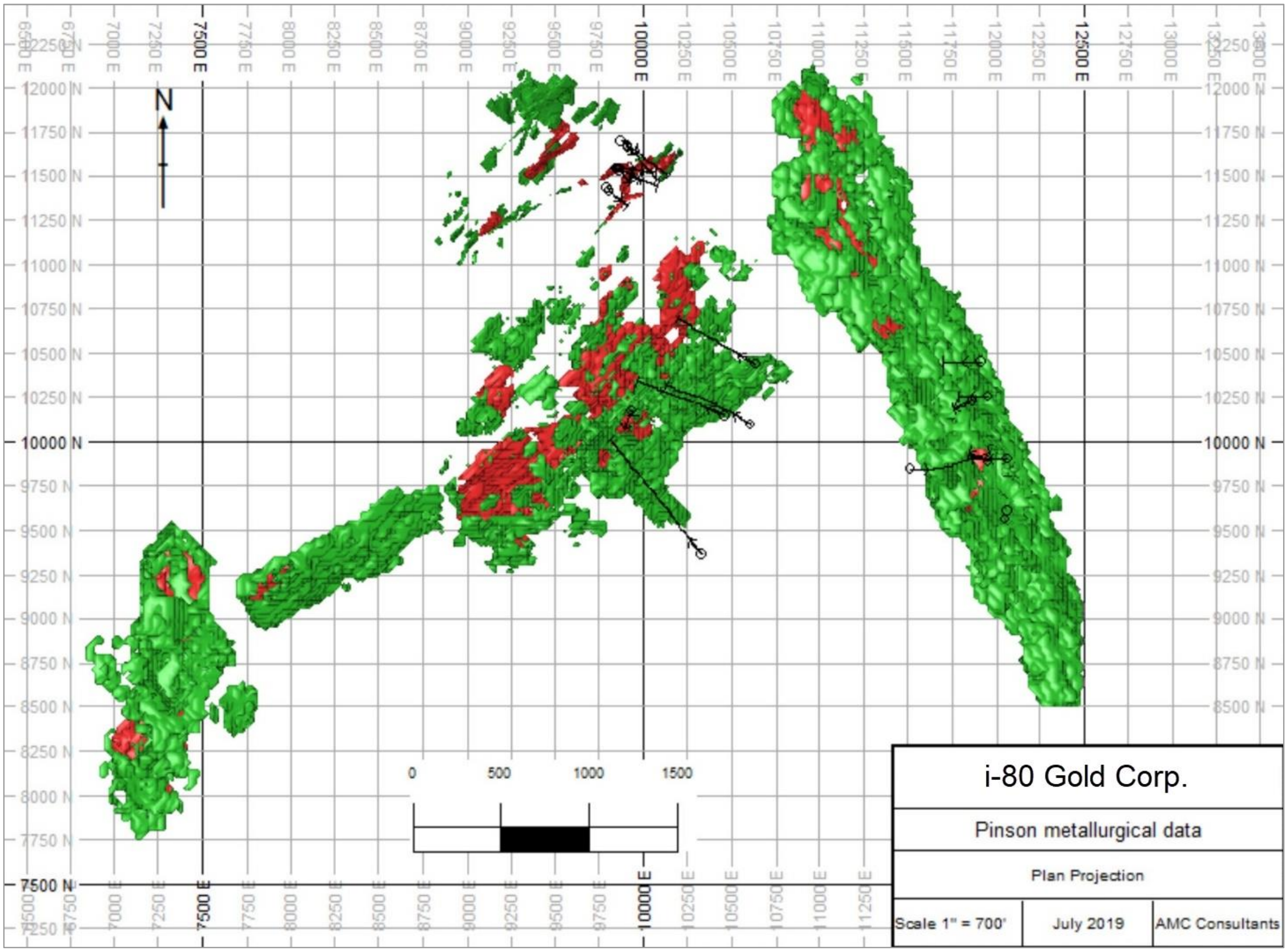
13.4 Sample representivity

13.4.1 Overview

Samples used for metallurgical test work have been sourced from the open pits (Mag Pit and CX Pit) and from drilling of each of the main Mineral Resource area (Mag Pit, CX Pit, and underground). Each of the main zones has been included in test work and key metallurgical characteristics have been defined.

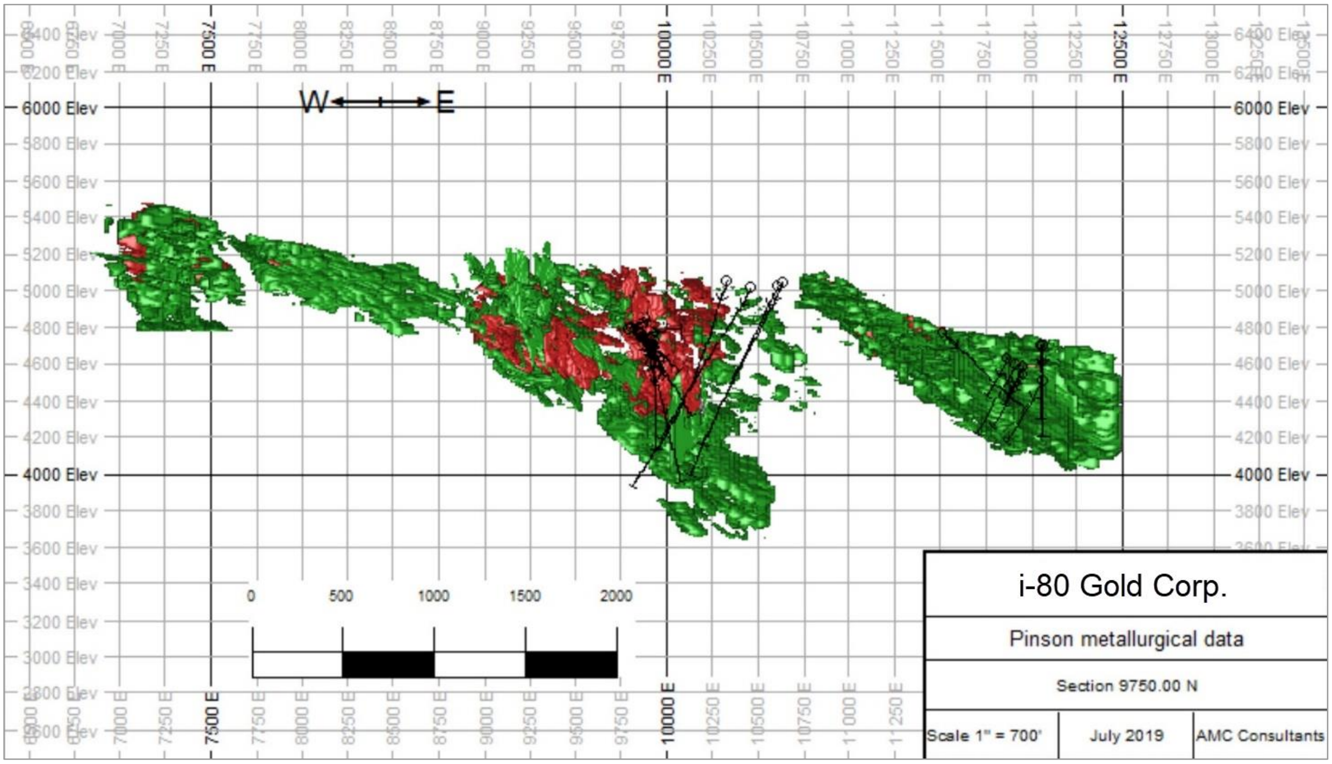
Figure 13.6 and Figure 13.7 show the relationship of the metallurgical samples to estimated blocks.

Figure 13.6 Plan view showing metallurgical sample locations



Note: The green shaded area has been classified as Indicated and red as Measured in the model that has not been constrained by an open pit shell. Thus, these are not regarded as current Mineral Resources. Drillholes used for metallurgical sampling are shown in black.
Source: AMC Mining Consultants (Canada) Ltd.

Figure 13.7 Lateral view showing metallurgical sample locations



Note: See notes for Figure 13.6.
Source: AMC Mining Consultants (Canada) Ltd.

Within each zone, drilling has been localized to relatively small portions of the deposit as seen in Figure 13.8 to Figure 13.10. The metallurgical response of the samples is likely to represent the general behavior of the zone, but sampling of at least one other area of each zone to confirm the metallurgical response will reduce uncertainty. Confirmatory testwork on targeted drilled samples is recommended to mitigate the risk.

13.4.2 Bulk samples

Bulk samples were sourced from the Mag Pit and CX Pit. Six bulk samples of approximately 1,000 lb each were sourced from the Mag Pit and designated Mag Pit I - VI. One bulk sample of approximately 4,300 lb was sourced from the CX Pit. The locations of the samples were not reported so it is not possible to assess whether they are representative of the eventual Mineral Resource volume.

13.4.3 Drillhole samples

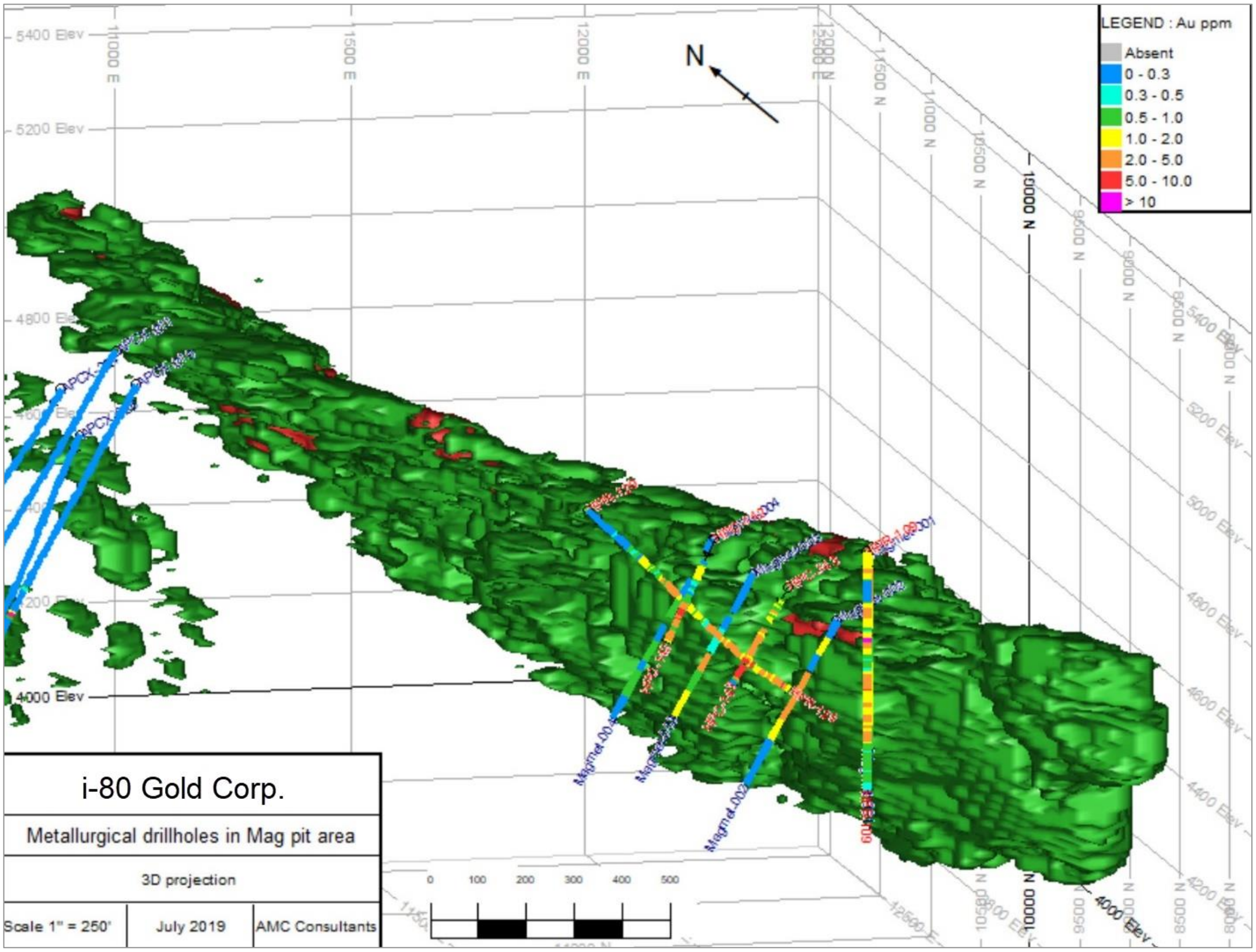
The samples selected from drilling on the Project over its life are listed in Table 13.14. The drillholes used for the samples are plotted on the mineralized domains in Figure 13.8 to Figure 13.10.

Table 13.14 Drillhole sample selection and testing matrix

Sample ID	Location	Testing
HPR109	Mag Pit	Preg-robbing, bottle roll, column percolation leach tests
HPR129	Mag Pit	As above
HPC142	Mag Pit	As above
HPC143	Mag Pit	As above
Magmet-001	Mag Pit	Bottle roll, column percolation leach tests
Magmet-002	Mag Pit	As above
Magmet-003	Mag Pit	As above
Magmet-004	Mag Pit	As above
APCX-204	CX Zone	Bottle roll, column percolation leach tests
APCX-211	CX Zone	As above
APCX-219	CX Zone	As above
APCX-226	CX Zone	As above
AMW-002	CX Zone	Bottle roll, column percolation leach tests
UGOG-004	Underground resource	Head assays and CN soluble Au only.
UGOG-010	Underground	As above
UGOG-013	Underground	As above
UGOG-015	Underground	As above
UGOG-017	Underground	As above
UGOG-018	Underground	As above
UGOG-019	Underground	As above
UGOG-021	Underground	As above
UGOG-022	Underground	As above

Mag Pit drillholes intersect only one end of the mineralized domain. It may be necessary to do further work in the other extent of the zone if it is likely to be included in a future mining plan. Lease boundaries and other factors are to be reviewed. An expanded view of the Mag Pit resource and drillhole is shown in Figure 13.8.

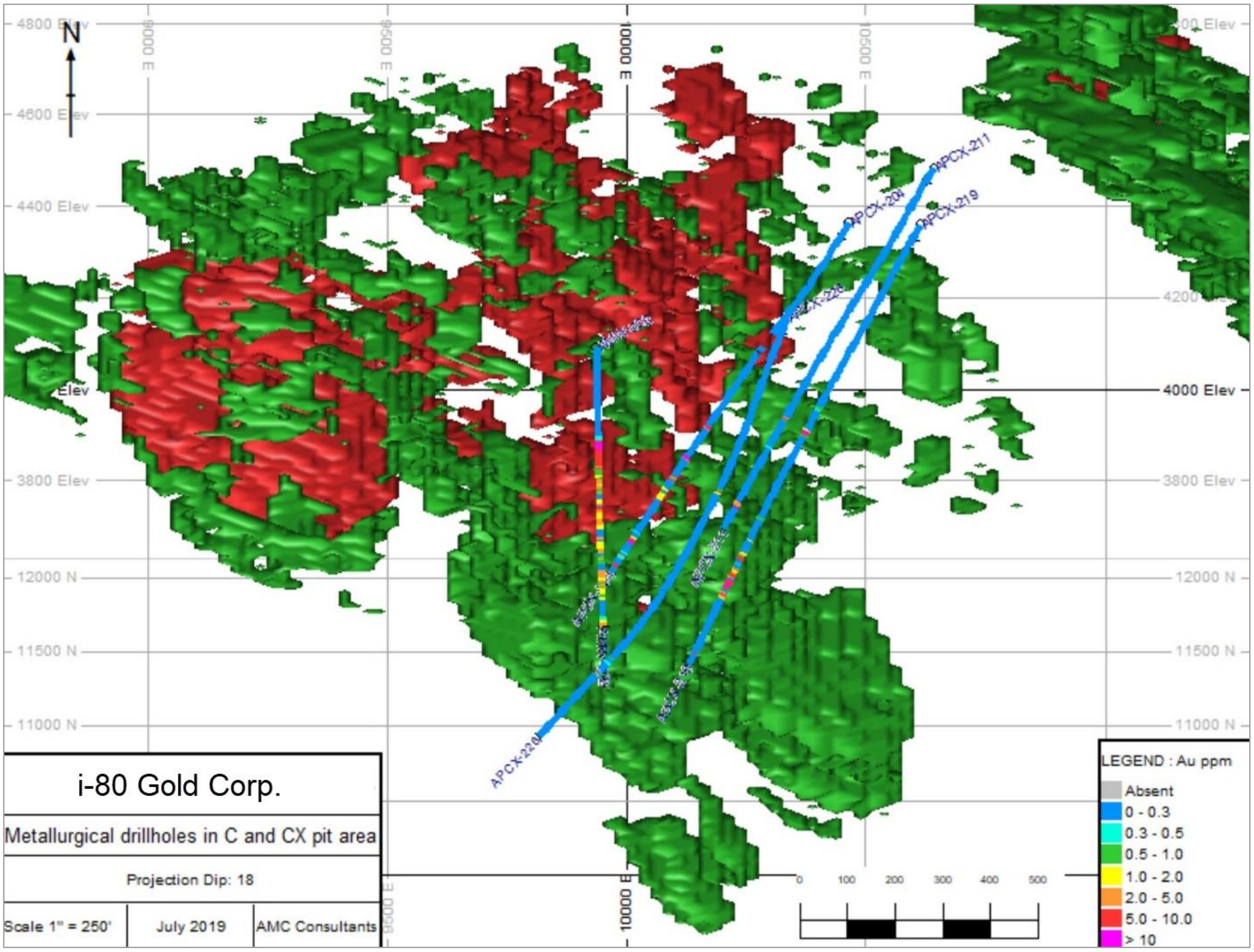
Figure 13.8 Location of drillholes relative to Mag Pit mineralized domain



Note: See notes for Figure 13.6.
Source: AMC Mining Consultants (Canada) Ltd.

The sampling of the CX Pit is heavily clustered, and much of the mineralized domain has not been assessed metallurgically. A view of the CX sampling is shown in Figure 13.9.

Figure 13.9 Location of drillholes relative to CX Pit mineral domains

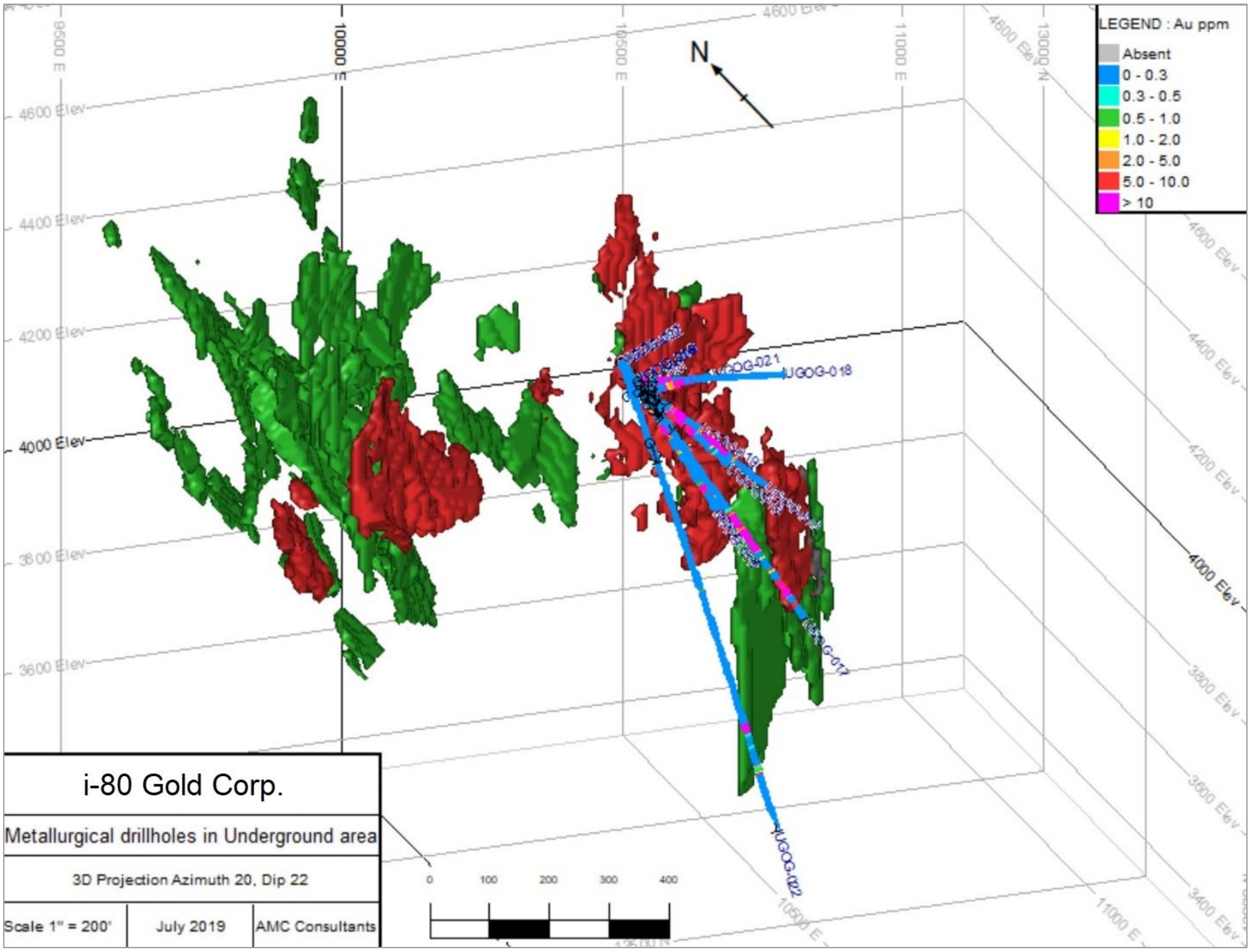


Note: See notes for Figure 13.6.
Source: AMC Mining Consultants (Canada) Ltd.

Ogee (from the old underground developments) metallurgical test drilling intersects a restricted portion of the mineralized domain. The lens parallel to the existing workings is not intersected by any drilling. Dependent upon grade, this material may be included in a future mine plan and so may require additional metallurgical testing.

Location of drillholes relative to the underground (Ogee) mineralized domains is shown in Figure 13.10.

Figure 13.10 Location of drillholes relative to underground mineralized domains



Note: The green shaded area has been classified as Indicated and red as Measured in the model that has not been constrained by stopes. Thus, these are not regarded as current Mineral Resources. Drillholes used for metallurgical sampling are shown in black.
Source: AMC Mining Consultants (Canada) Ltd.

Generally drilling intersects only limited areas of the mineralized domains and testing of additional areas is recommended. The selection of further drilling and sampling for metallurgical testing should be guided by a future mine plan.

The lack of metallurgical testing that spatially represents all zones is a risk to the project.

13.4.4 Metallurgical composite assembly

For all test work conducted, composites for metallurgical test work were prepared by combining drillhole intervals using instructions given by the owner at the time. In general, it is not possible to describe the objective with respect to variability or representivity testing from the information given. That is, it is not apparent that the composites were prepared in such a way that their grade or mineralogy represented the variability or domaining of the deposit.

The samples provided by Homestake in 1999 were composited in a manner which tended to reduce the variability in the provided samples. Table 13.15 displays the composite assay and the highest and lowest assays of the intervals in the composite.

Table 13.15 Composite assays

Composite	Composite assay oz Au/t	High / low interval oz Au/t
Cuttings comp 1	0.094	0.027 – 0.266
Cuttings comp 2	0.070	0.028 – 0.103
Cuttings comp 3	0.068	0.036 – 0.125
Cuttings comp 4	0.074	0.035 – 0.102
Cuttings comp 5	0.059	0.014 - 0.142
Cuttings comp 6	0.076	0.041 – 0.163
Core comp 1	0.032	0.014 – 0.058
Core comp 2	0.077	0.003 – 0.162
Core comp 3	0.098	0.021 – 0.191
Core comp 4	0.058	0.016 – 0.093
Core comp 5	0.146	0.015 – 0.272

The QP concludes that the samples do not represent the variability of the mineralization and test work should be undertaken on samples that represent the low- and high- grade variation of the mineralization. The lack of information on metallurgical performance of such samples remains a risk to the project.

13.5 Deleterious elements

Both arsenic and mercury are present in the mineralization. Elemental concentrations are higher in underground (Ogee) samples (up to 2% in Sample 34259) associated with high gold values. The Mag Pit zone also contains areas of high TOC which have been associated with poor gold recoveries due to preg-robbing of the gold-cyanide complex.

As both the CX Pit underground mineralization and the Ogee underground mineralization exhibit refractory behaviour, the recovery of gold will require oxidative leaching that will solubilize arsenic. Methods exist to manage arsenic and confine it to the solid residue. The selection of a suitable process requires extensive metallurgical development work. Until this is completed, the fate of arsenic-containing residues is a risk to the project.

13.5.1 Homestake mining

Test work conducted for Homestake did not report the presence or deportment of arsenic.

Mercury was assayed in Mag Pit and CX Pit bulk ore samples. Assays ranged from 2.53 ppm to 43.12 ppm Hg, levels high enough to require consideration of mercury capture during refining.

Carbon and copper were assayed to understand the potential contributors to preg-robbing behaviour. Copper was present in the range of 16 to 54 ppm in both the Mag Pit and CX ore samples.

TOC was present at greater than 4% in Mag Pit samples I, II, IV all of which displayed significant preg-robbing characteristics. TOC was less than 0.4% in the other samples and gold recoveries were high.

Preg-robbing results in poor gold recovery which in the various samples and conditions tested ranged from 8.5% to 94%. Further it can result in high cyanide consumption. The proportion of the mineralization with high TOC is unclear and is a risk to further development of the deposits.

13.5.2 Atna Resources 2005

Drillhole samples from the underground extension of the CX Pit were received for autoclave and cyanide leach testing. These were assayed for arsenic and measured values between 0.054% and 1.65% were reported and are tabulated with gold by fire assay in Table 13.16.

Table 13.16 Gold and arsenic assays CX Pit

	Au (ppm) by fire assay	As (%) by AA
APCX-204	8.16	0.066
APCX-211	8.33	0.130
APCX-219	10.25	0.180
APCX-226	17.5	1.650
AMW-002	10.29	0.054

A further four samples designated R Rib & L Rib, 33941, 33942, and 34259 were also received for autoclave and cyanide leach testing. These tests were completed on three composite samples from the RFZ and one composite from the Ogee Zone mineralization. These samples are understood to be samples collected from the previously mined underground deposit. Assays for As and Hg were not reported and assumed not measured.

Approximately 200 samples from nine drillholes in the Ogee Underground resource area were submitted for sample preparation and assaying in March 2006. The individual samples were composited into 21 samples for further work. Gold assays ranged from 7.1 ppm to 54.7 ppm and arsenic from 0.09% to 0.46%.

There is no report of the deportment of As (or Hg) in the test work. Test work will be required to understand the deportment of these elements and whether products and residues need to be further treated.

13.5.3 Atna Resources 2013

Thirty-two (32) drill core composites from the Mag Pit area were submitted for heap leach amenability testing. A full elemental analysis was done on each sample including As, Hg, Cu, and organic carbon. The range of assays is shown in Table 13.6.

Table 13.17 Mag Pit drill core composite assays

Element	Assay range
As	77 ppm to 671 ppm
Hg	2.1 ppm to 30 ppm
Cu	8.7 ppm to 154 ppm
C	0.14% to 5.1%

There is no report of the deportment of As, Hg in the test work.

Twenty-three (23) of these composites were tested for heap leach amenability but neither As or Hg deportment was measured.

13.6 Conclusions

13.6.1 Sample representivity

Within each zone, drilling has been localized to relatively small portions of the mineralized domains as seen in Figure 13.8 to Figure 13.10. The metallurgical response of the samples is likely to represent the general behavior of the zone, but sampling of at least one other area of each zone to confirm the metallurgical response will reduce uncertainty. The lack of this drilling remains a risk to the project.

It is further concluded that the samples do not represent the grade variability of the deposit and test work should be undertaken on samples that represent the low and high-grade variation of the mineralization. The lack of information on metallurgical performance of such samples remains a risk to the project.

13.6.2 Deleterious elements

Deleterious elements (arsenic and mercury) are present in some zones of the deposits at grades high enough to be a risk to the project. Insufficient test work on the deportment and fate of these elements has been completed.

13.6.3 Testwork on Open Pit samples

Cyanide leach bottle roll tests and column leach tests were completed on samples from both the Mag and CX open pits. These tests were commissioned by both Homestead and Atna.

The testwork demonstrated that many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. The lack of representivity of the Mag Pit samples presents a risk to gold recovery in the Mag Pit due to variable and ill-defined preg-robbing characteristics of the feed material.

Bottle roll tests were conducted on Mag Pit samples using NaOH as an alternative to hydrated lime, as a method of treating material with preg-robbing characteristics. These tests demonstrated that raising the pH improved gold recovery and decreased cyanide consumption.

A column leach test on a Mag Pit sample showed that there was no gold recovery benefit in using NaOH rather than lime (at the equivalent pH).

Testwork on ground feed showed that Mag Pit material was amenable to CIL methods.

Column leach tests on the Mag Pit samples achieved gold recoveries in the range of 19% to 82%.

Column leach tests on the CX Pit samples achieved gold recoveries of 82%.

13.6.4 Testwork on underground samples

Testwork on both underground and open pit material showed that there was a negative linear relationship between gold recovery and total sulphur grade. The relationship suggests that a higher sulphur head grade will have more refractory gold that will detrimentally impact gold recovery.

Autoclave pre-treatment ahead of cyanide leach testwork was completed on the Ogee underground samples to treat refractory gold present in sulphide minerals. This testwork demonstrated significant increases in gold recovery relative to the baseline cyanide leach tests.

13.7 Recommendations

The QP recommends the following actions for developing the Property.

13.7.1 Testwork recommendations

Future testwork programs should be completed on a number of samples that represent the deposit's spatial variability of weathering profile, lithology, and gold grade, and that represent run-of-mine feed from progressive stages of the project.

Conduct quantitative mineralogy (e.g., QEMScan) on selected samples that represent run-of-mine feed from progressive stages of the project.

Additional column leach testwork should be conducted. This testwork should be completed at varying crush sizes to determine the optimum crush size.

Further cyanide leach testwork should be completed to determine the optimum alkali and pH parameters for gold recovery.

Preg-robbing tests should be conducted where a test with a spiked gold solution is run in parallel with a test without the spiked solution.

Additional autoclave pre-treatment testwork should be completed on Ogee samples. This testwork should optimize the test parameters including grind size, pressure, temperature, and residence time.

Diagnostic leach tests should be conducted on Pinson open-cut and underground samples. These tests would confirm the deportment of gold in terms of free milling gold, gold in carbonates and gold in sulphides.

Mineralogy testwork should be conducted on Pinson samples to confirm the gold deportment amongst different mineral species. This work will assist in confirming the refractory nature of the gold.

Comminution testwork should be completed on Pinson samples. This should include crushing testwork to design and size the crushing circuit for the heap leach circuit; and testwork on the underground samples to determine the crushing and grinding requirements for autoclave treatment.

Complete additional CIL testwork (where material is ground prior) on open pit material to ascertain the technical and economic viability of processing the material in a CIL circuit.

Roaster pre-treatment testwork (ahead of cyanide leach) should be conducted on the Ogee (underground)samples, given the proximity of sulphide roaster facilities in the Pinson region. The roasting testwork could be trialed as an alternative to autoclave pre-treatment as a method of treating refractory gold. Roasters could also be used to treat carbonaceous material so that preg-robbing issues would be prevented.

A techno-economic trade-off study be completed looking at the roaster and autoclave options. This study should examine the demand for Pinson material from local roasters and autoclave facilities.

Further testing should be done using the nitric acid diagnostic leach procedure as a predictor of gold recovery on autoclave residues.

Complete flotation testwork ahead of autoclave pre-treatment testwork to produce flotation concentrates with high sulphur and gold grades. The objective of the flotation testwork is to reduce the mass of material to an autoclave circuit, to reduce operating costs and increase throughput through the autoclave.

Test alternative options for dealing with the carbonaceous preg-robbing material from the Open Pit areas including:

- Completing resin-in-leach testwork as an alternative to activated carbon. Resin-in leach technology has been proven to have a better affinity for absorbing solution gold than activated carbon, and this would reduce the amount of solution gold that is being preg-robbed.
- Completing testwork where blinding agents such as kerosene are added to the bottle roll tests. The objective of this testwork is to blind the carbonaceous material so that it prevents solution gold getting absorbed to carbonaceous material. During this testwork, care must be taken to ensure that the blinding agents are not overdosed to avoid blinding the activated carbon.
- Additional samples should be subject to metallurgical testing. These samples should be chosen to extend the spatial extent of coverage of the deposit and to represent the grade variation in the different zones.
- A program of testing of the deportment of arsenic and mercury in the processing of the mineralization is required. This program should cover the CIL, heap leach, and pre-oxidation processes tested during the past test work program.

13.7.2 Geometallurgy recommendations

A geometallurgical block model should be developed for the Pinson material. This model should incorporate both Open Pit and Underground areas and include key inputs such as chemical assays (including gold, sulphur speciation, and carbon speciation), mineralogy and testwork parameters. This model would develop relationships between key parameters such as gold grade, sulphide grade, carbon grade and gold recovery. This model should also include a financial model that determines the most economically viable process route for all blocks in the block model. This financial model should include inputs such as gold price, gold grade, tested gold recovery, operating costs, and expected revenue from toll treatment. The model should also account for the capacity of the various process units (heap leach and autoclave) to avoid creating process bottlenecks.

14 Mineral Resource estimates

14.1 Introduction

The Mineral Resources for the Pinson deposit have been estimated by Ms Dinara Nussipakynova, P.Geo., of AMC, who takes responsibility for these estimates.

The QP is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates. Nevada is a mining friendly territory within a stable jurisdiction.

Both the underground and open pit estimates are dated 23 July 2020 and supersede the previous estimates outlined in the "Technical Report on the Pinson Project Preliminary Feasibility Study in Humboldt County, Nevada" dated 17 October 2014 (Golder 2014). The previous open pit estimate had an effective date of 31 December 2013. The previous underground estimate had an effective date of 1 July 2014.

The data used in the 23 July 2020 estimate includes results of all drilling carried out on the Property to 15 April 2019 and is based on the 18 April 2019 database. The last drilling in the Open Pit area was four metallurgical holes drilled in 2012. The last drilling in the Underground area was RC holes drilled by Atna in 2015. The date of the database is noted as although limited drilling has taken place since the previous estimates, considerable work has been done to verify the database and obtain original assay certificates for the Underground area.

The results of the current underground and open pit estimates as of 23 July 2020 are summarized in Table 1.1 and Table 1.2.

The Mineral Resource statement by area is presented in Table 14.3.

Claim ownership in the Mineral Resource area varies. Mineral Resources tables are reported after mining depletion. Tables of the Mineral Resources by claim is provided in Table 14.15 and Table 14.34.

Note that with the exception of Table 14.3 estimates are not combined into a single Mineral Resource table as they are spatially separate and represent two distinct Mineral Resource areas. There are no Mineral Reserves stated at present. The Mineral Resources have been depleted for previous mining.

Table 14.1 Summary of the Underground area Mineral Resource as of 23 July 2020

Classification	Tons (ktons)	Au (opt)	Metal Au (koz)
Measured	184	0.289	53
Indicated	436	0.313	136
Measured and Indicated	**620**	**0.306**	**190**
Inferred	1,676	0.347	581

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- The Mineral Resource COG is based on a metal price of $1,550/oz Au. (cost and other assumptions shown in Table 14.13).
- Underground Mineral Resources as stated are constrained within modelled underground stope shapes using a nominal 15' minimum thickness, above a gold cut-off grade of 0.15 opt Au.
- Drilling results up to 31 December 2015.
- Drilling database provided 18 April 2019.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The numbers may not compute exactly due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.2 Summary of Open Pit area Mineral Resource as of 23 July 2020

Classification	Tons (ktons)	Au (opt)	Metal Au (koz)
Measured	10,726	0.068	730
Indicated	11,829	0.046	545
Measured and Indicated	**22,554**	**0.057**	**1,275**
Inferred	1,388	0.047	65

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- Mineral Resources are constrained by an optimized pit shell developed at a metal price of $1,550/oz Au (cost and other assumptions shown in Table 14.31).
- Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone COG is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.3 Mineral Resource as of 23 July 2020 by area

Classification	Deposit	Tons (ktons)	Au (opt)	Metal Au (koz)
Measured	Pit A	-	-	-
	Pit B	1,940	0.045	86
	Pit C and CX	3,233	0.098	317
	Pit MAG	5,553	0.059	327
	Underground	184	0.289	53
	Total	**10,910**	**0.072**	**783**
Indicated	Pit A	359	0.096	34
	Pit B	2,189	0.051	111
	Pit C and CX	2,348	0.055	128
	Pit MAG	6,933	0.039	271
	Underground	436	0.313	136
	Total	**12,265**	**0.056**	**681**
Measured and Indicated	Pit A	359	0.096	34
	Pit B	4,129	0.048	197
	Pit C and CX	5,581	0.080	445
	Pit MAG	12,485	0.048	597
	Underground	620	0.306	190
	Total	**23,175**	**0.063**	**1,464**
Inferred	Pit A	110	0.029	3
	Pit B	49	0.069	3
	Pit C and CX	620	0.077	47
	Pit MAG	609	0.018	11
	Underground	1,676	0.347	581
	Total	**3,064**	**0.211**	**646**

Note: See footnotes under Table 1.1 and Table 1.2.

14.2 Resource estimation process

The Pinson deposit, for the purposes of modelling and estimations, is broken into two areas. The first area is the Underground area. The second area is the Open Pit area, which is the location of previous open pit mining.

The evaluation of the Mineral Resources for the Pinson Underground area involved the following procedures:

- Constructing mineralization domain.
- Conditioning of data (compositing and capping) for geostatistical analysis and variography.
- Selecting of estimation strategy and estimation parameters.
- Block modelling and grade interpolation using inverse distance squared (ID^2).
- Validation, classification, and tabulation.
- Constraining the estimate by creating mining shapes which demonstrate continuity and economics.
- Preparation of the Mineral Resource Statement.

The evaluation of the Mineral Resources for the Pinson Open Pit area involved the following procedures:

- Constructing mineralization envelopes for each pit.
- Compositing drillholes for each pit.
- Flagging the composites as being above or below a threshold value.
- Calculating and modelling variograms for the indicator variable.
- Estimating the indicator value using ordinary kriging (OK).
- Selecting a nominal probability limit from the estimated indicator value above which is "high grade model" and below which is the "low-grade model".
- Determining suitable search ellipsoid orientations for both models.
- Estimating gold grades using either OK or ID^2 (depending on sample support and area) for the two models.
- Combining the models and classify.
- Code block model with cyanide leach recovery assumptions.
- Combining with geology model (provided by client) to assign densities based on geology and grades.
- Constraining the estimate by a pit optimization shell which demonstrates economics.
- Depleting the model using the topography provided by the client.
- Preparation of the Mineral Resource Statement.

Datamine™ software was used to construct the mineralization wireframes for the Underground area and to constrain domains by kriging Indicators in the Open Pit areas. The estimation was carried out using Datamine™ software. Interpolation of grades was carried out using OK for all the open pit mineralized domains. The underground domains were estimated using ID^2. Datamine™ software was also used for geostatistical analysis and variography.

Database import, bulk density assignment and geological modelling are common to both estimates and are discussed in following sections. The details of the underground estimations are in Section 14.6.2, and the details of the open pit estimations are in Section 14.12.2.

14.3 Data used

14.3.1 Drillhole database

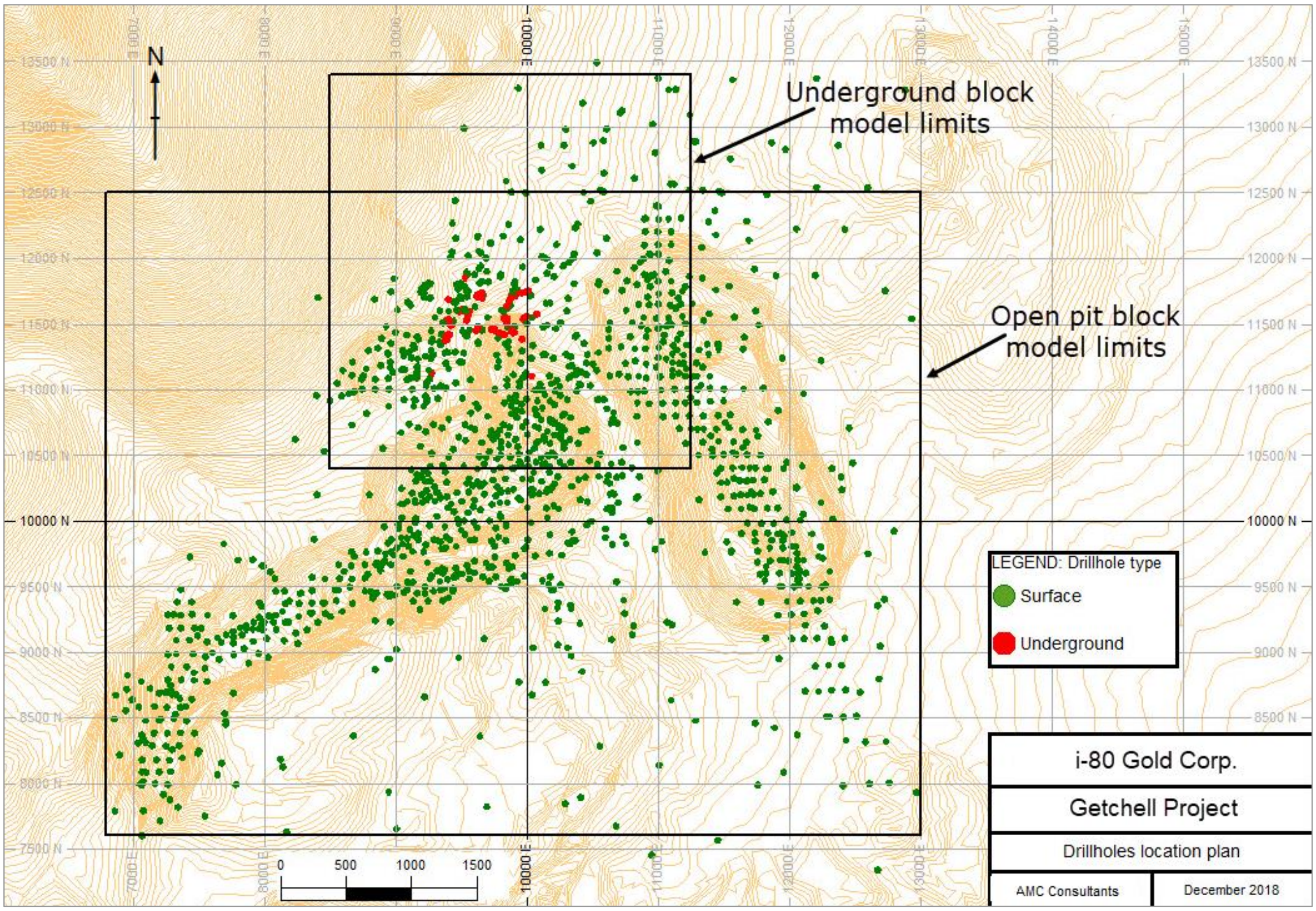
The entire dataset for the Property was provided by OMC on 18 April 2019 as a set of .csv files containing drilling information collar, survey, assays, lithology, alteration, veins, and bulk density. It had a total of 2,855 exploration holes (surface, underground, and trench samples) and assays, collar, and survey data for 695 production holes (surface and underground). Drilling is a mix of RC drilling, diamond drilling, and RC precollar with diamond tails. The exploration assay file contains 212,839 gold assays. The production data assay file contains 1,477 gold assays. There are also 9,321 multi-element assays.

All data was provided in a local (imperial) grid and the modelling was conducted in this local grid.

A subset of the total data was used in the Mineral Resource estimate as shown in Figure 14.1.

For the Underground area this included 48,179 assays contained within mineralized wireframes. For the Open Pit areas this included 77,321 assays used in estimation.

Figure 14.1 Pinson Project drillholes location plan



Note: The box outlines the extents of the data used in the Mineral Resource.
Source: AMC Mining Consultants (Canada) Ltd.

The data used in the underground estimate consists of surface and underground diamond, RC, and combined RC and diamond drillholes. There was a total of 453 drillholes that were used in modelling and informed the Mineral Resource estimate. Production drillholes (not listed below) were also used to guide the interpretation of mineralization domains but were not used in the estimation.

Table 14.4 is a summary of the drillholes used in the Underground area estimation.

Table 14.4 Summary of drillholes used in estimation of the Underground area

Drillhole type	Surface drillholes				Underground drillholes			
	# drillholes	# surveys	# samples	Length (ft)	Drillholes	# surveys	# samples	Length (ft)
RC	128	1,253	14,720	76,825.0	156	738	5,835	29,488
RC and diamond	72	1,642	16,631	79,681.7				
Diamond	11	147	1,582	7,267.1	86	1,930	7,062	30,519.5
Total	**211**	**3,042**	**32,933**	**163,773.8**	**242**	**2,668**	**12,897**	**60,007.5**

Note:
- RC=reverse circulation
- Underground drill data to 31 December 2016.
- Drilling database provided 18 April 2019.

Source: AMC Mining Consultants (Canada) Ltd.

There were a number of negative assays in the database and a description and count is shown in Table 14.4. For the purposes of the Mineral Resource estimate, all negative values were treated as absent data.

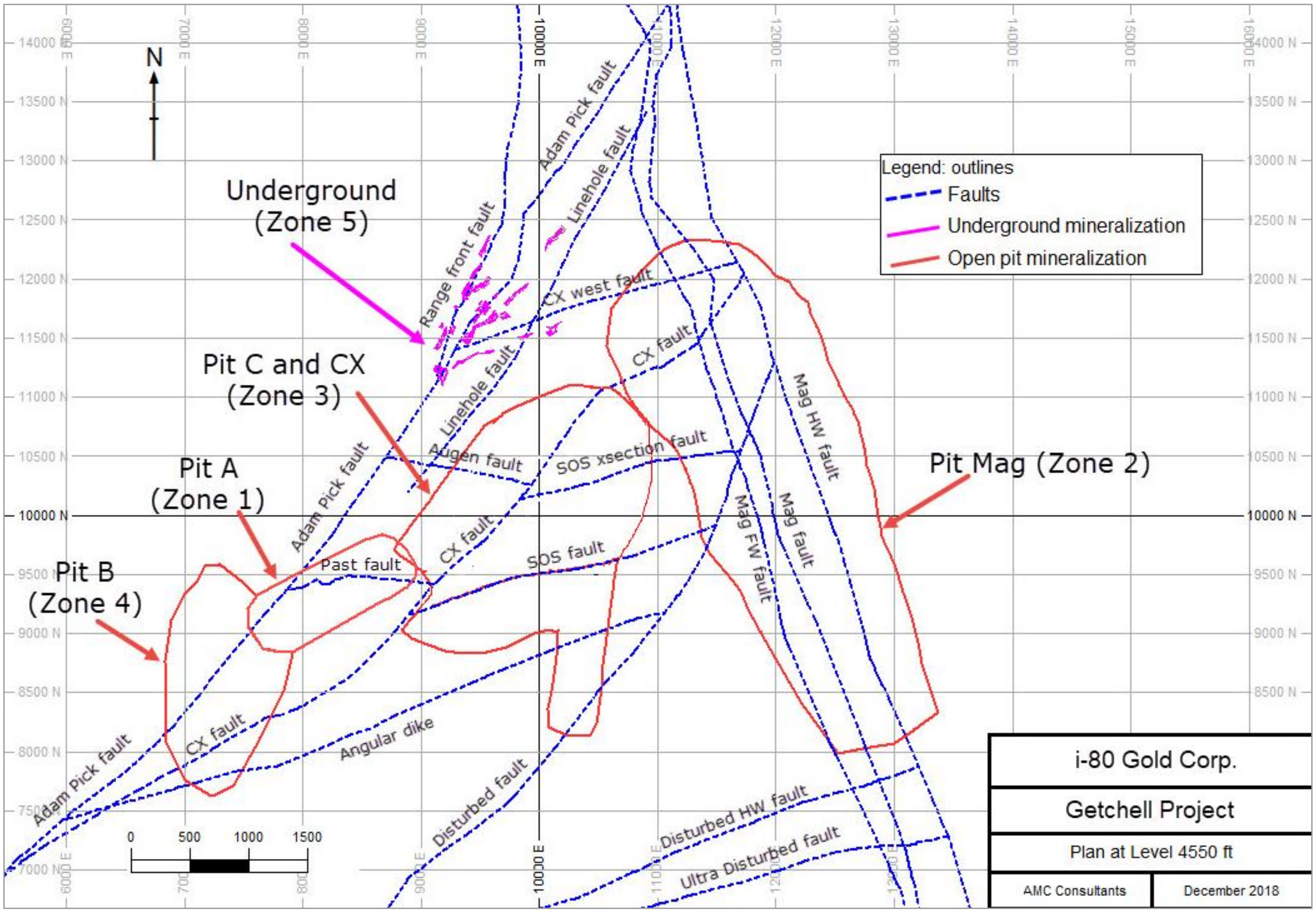
Table 14.5 Replacement values for negative assays in database

Value	Description	Count in export	Replaced value (ppm)
-5556	Sample not received	76	-999
-0.9942853	Below detection limit for -0.029 opt	45	0.5
-0.1714285	Below detection limit for -0.005 opt	49	0.085
0.015	Below detection limit for -0.001 opt	330	0.015
-0.003	Below detection limit for -0.003 ppm	2,302	0.0015
0	Historical assays, not sure the description	26	0
-3394.284	Conversion of -99 opt to ppm, only in Cyanide Export		-99
	Blank		-9
-34.287			-99
-3			-99

Source: Osgood Mining Company LLC.

The Mineral Resource area was divided into five zones; four open pit zones (Zones 1 – 4) and one underground zone (Zone 5). These mineralization zones are shown in Figure 14.2 along with the important structures.

Figure 14.2 Pinson mineralization zones



Source: AMC Mining Consultants (Canada) Ltd. with fault data provided by Osgood Mining Company LLC.

14.3.2 Bulk density

Bulk density was supplied by OMC. The total number of measurements is 153. It is based on a combination of rock type and grade. Bulk density was coded into the block model as show in Table 14.6.

Table 14.6 Bulk density used in block model

Rock code	Au >= 0.008 opt		Au < 0.008 opt	
	No of measurements	Density (t/m^3)	No of measurements	Density (t/m^3)
QAL	Assigned	1.85	Assigned	1.85
KGD	2	2.73	4	2.70
OCu	64	2.50	7	2.70
OCL	29	2.51	30	2.64
CPu	7	2.42	10	2.60
DYKE	Assigned	2.70	Assigned	2.70

Notes: QAL=Quaternary Alluvium, KGD=Cretaceous Granodiorite, OCU=Ordovician Upper Comus Formation, OCL=Ordovician Lower Comus Formation, CPu=Cambrian Preble Formation. See details of rock types in Table 14.7.
Source: Osgood Mining Company LLC.

14.4 Domain modelling

14.4.1 Topography modelling

The topography was supplied by OMC and is dated July 2013 after past mining ceased.

14.4.2 Geological modelling

The geological model was built in Leapfrog software and supplied by OMC. The geological model was reviewed by the QP against drillholes and along sections and accepted. Within the Mineral Resource area, the geology model consists of seven rock types as shown below.

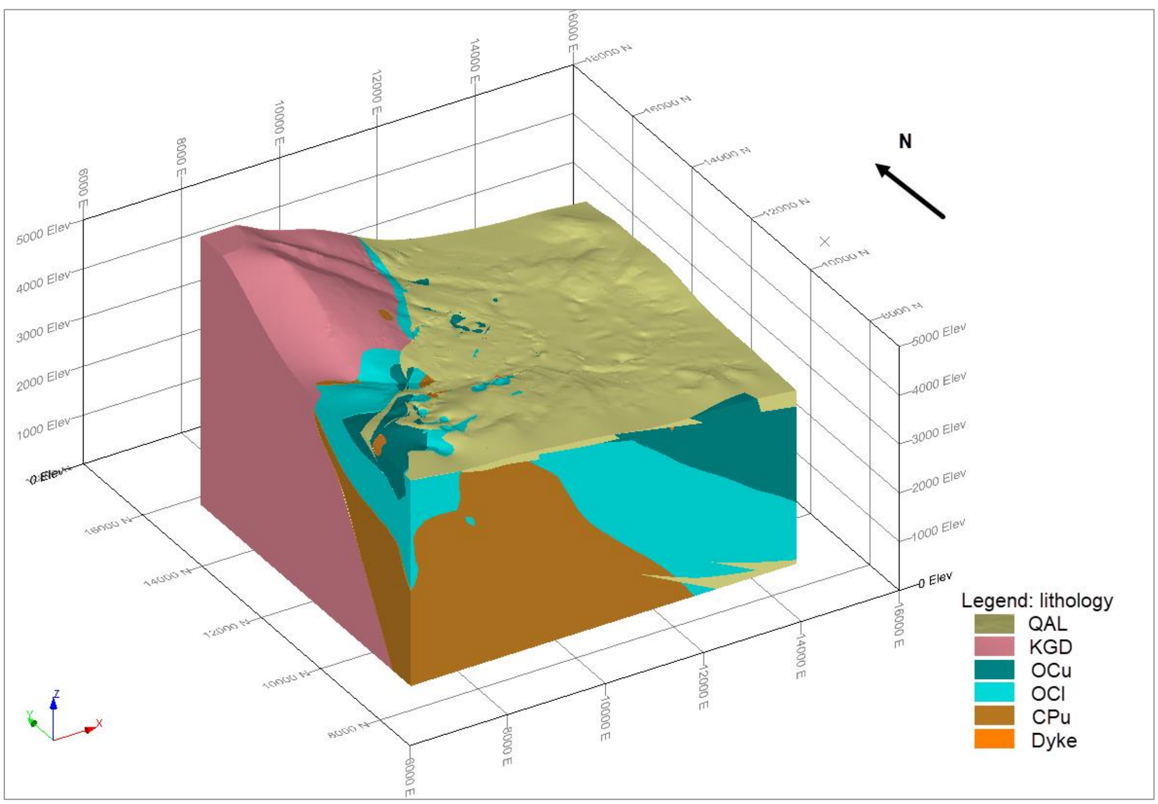
Table 14.7 Rock types in the geological model

Rock code	Rock name	Description
QAL	Quaternary Alluvial	Alluvium gravel
KGD	Cretaceous Granodiorite	Granodiorite and quartz diorite
OCU	Ordovician Upper Comus Formation	Mildly to non-calcareous shales with minor shaly limestone interbeds
OCL	Ordovician Lower Comus Formation	Medium to massive bedded limestone
CPu	Cambrian Upper Preble Formation	Phyllitic shales with limestone interbeds
CPm	Cambrian Middle Preble Formation	Phyllitic shales with limestone interbeds
DYKE	Undefined	Undefined

Source: Osgood Mining Company LLC.

A three-dimensional (3D) view of the Leapfrog geology model is shown below in Figure 14.3.

Figure 14.3 3D view of geology Leapfrog modelling



Notes: QAL=Quaternary Alluvial, KDG=Cretaceous Granodiorite, OCU=Ordovician Upper Comus Formation, OCL=Ordovician Lower Comus Formation, CPu=Cambrian Upper Preble Formation, Dyke= undefined dyke.
Source: AMC Mining Consultants (Canada) Ltd. based on data provided by Osgood Mining Company LLC.

14.4.3 Structural modelling

OMC provided 17 fault surfaces as dxf files for the deposit. The QP built one additional fault model assisted by the digitized map of Chadwick (2002). The structures are shown in Figure 14.2.

Structures were used as a guide for modelling mineralization.

14.4.4 Mineralization modelling

The Pinson deposit, for the purposes of modelling and estimations, is broken into two areas. The first zone is the Open Pit area which is the location of previous open pit mining. The second area is the Underground area. Modelling estimation, classification, validation, and reporting of each of these areas are treated separately in Sections 14.4.5 to 14.7 for the Underground and 14.10 to 14.15 for the Open Pit.

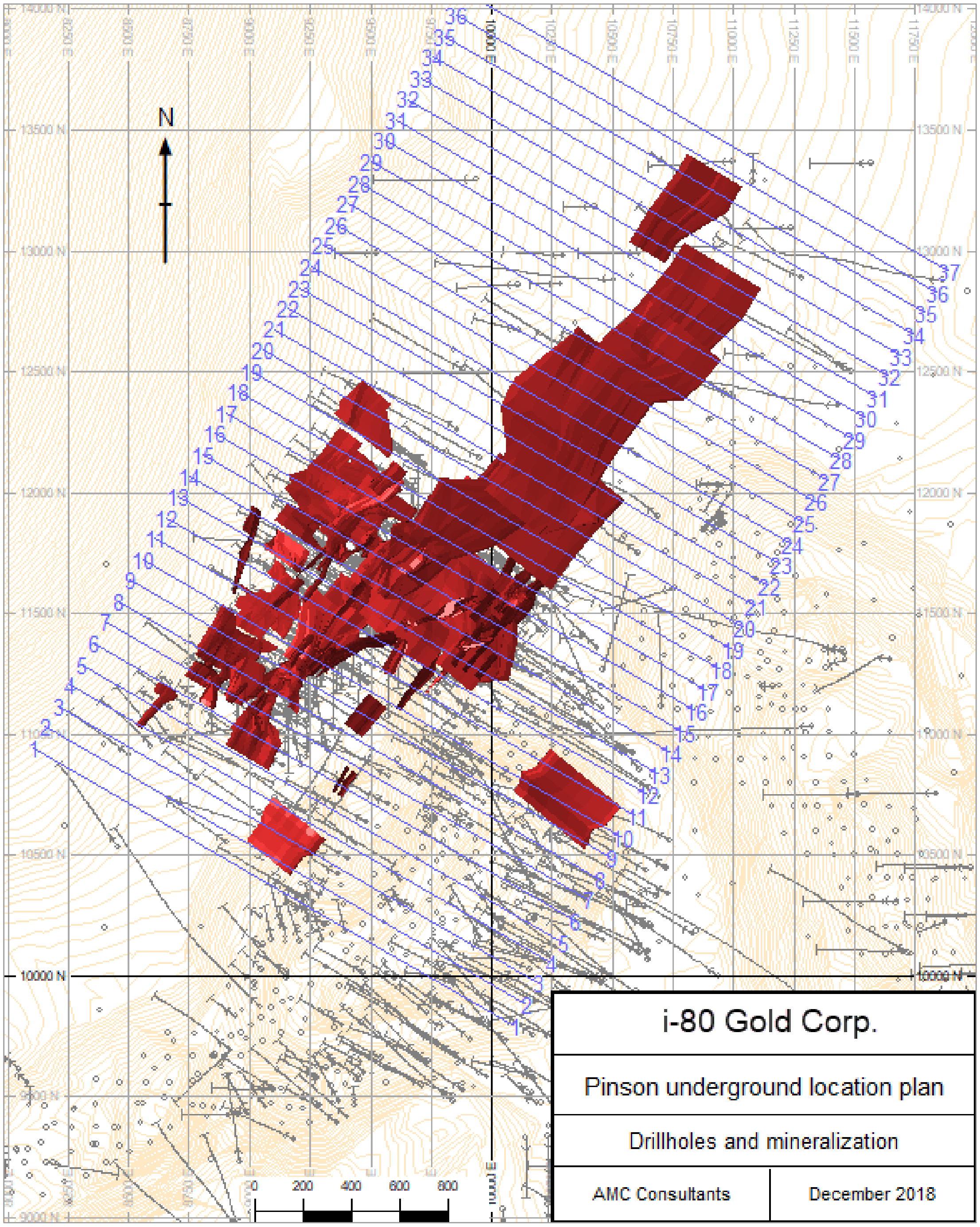
14.4.5 Underground area modelling

As discussed in Section 14.4.3, fault planes were provided by OMC in all but one case.

The QP modelled the mineralization in Datamine. A threshold grade of 1 g/t gold was used to guide the wireframing. All drillhole data was used to guide the shape of the mineralized domains. Some drillholes did not have original assay certificates. Mineral Resource classification accounted for this uncertainty in the data. Domaining included mineralization around single holes for future exploration targets.

Figure 14.4 shows the location of underground mineralization domains in plan view.

Figure 14.4 Underground mineralization location plan

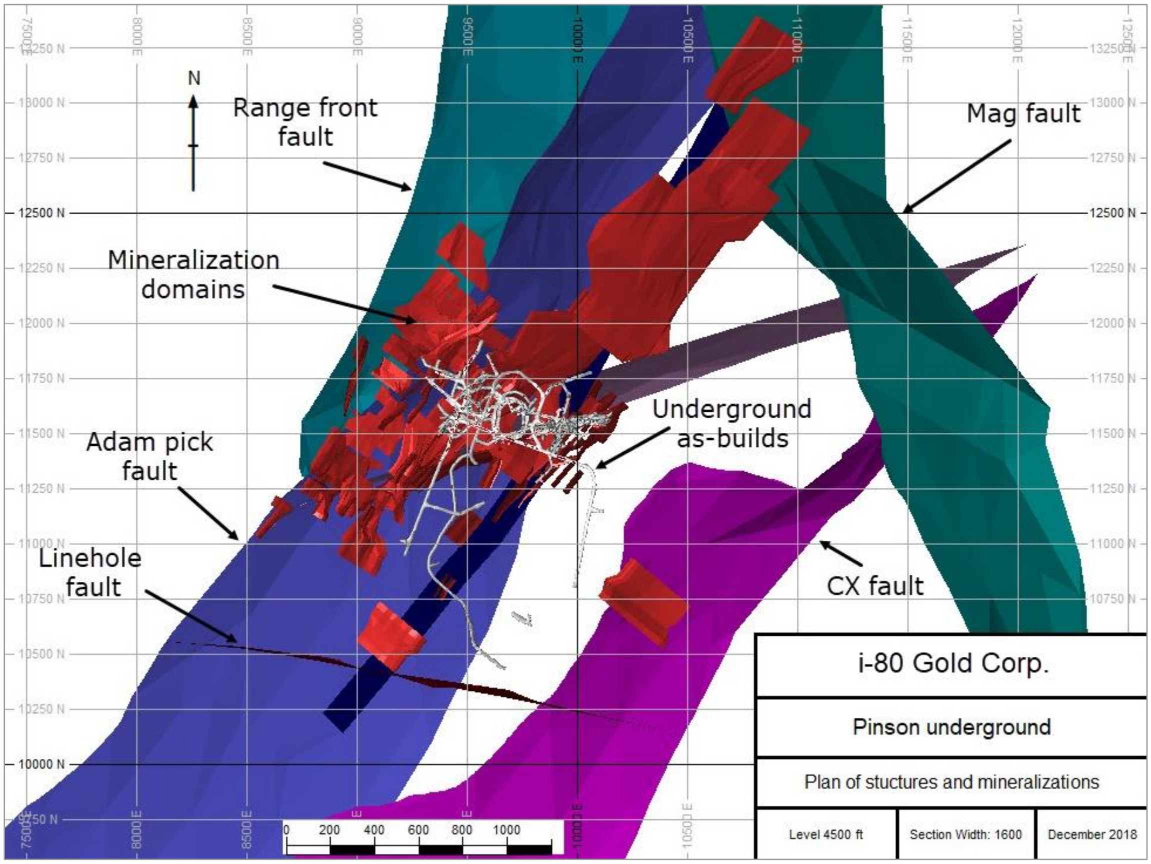


Source: AMC Mining Consultants (Canada) Ltd.

The total number of underground mineralization domains is 117, however 19 of these domains were classified as "potential" because of lack of data and missing certificates. Six domains were classified as a mixture of "potential" and higher resource classification categories.

Figure 14.5 shows a 3D view of the mineralized domains in the Underground area as they relate to the underground workings.

Figure 14.5 3D view of mineralization domains for the Underground area



Note: Pits are excluded in order to highlight structures.
Source: AMC Mining Consultants (Canada) Ltd.

14.5 Underground statistics and compositing

Sample lengths range from 0.04 to 60 ft within the wireframe models. Approximately 50% of the samples were taken at 5 ft intervals. Given this distribution and considering the width of the mineralization, the QP chose to composite on 10 ft lengths. Assays within the wireframe domains were composited starting at the first mineralized wireframe boundary from the collar and resetting at each new wireframe boundary. Samples were composited by domain using Datamine's dynamic compositing tool. This tool composites samples within each zone to 10 ft but adjusts sample length as necessary to avoid sample residuals (i.e., samples lengths less than composite length left over).

Domains were grouped into 16 sets based on similar orientations. The probability plots of each of these sets was viewed. After reviewing the probability plots, The QP decided no capping was necessary.

For ease of illustration, the probability plot of all gold grades within the Underground area domains is shown in Figure 14.6.

Figure 14.6 Probability plot of gold grades of all Underground area samples

Source: AMC Mining Consultants (Canada) Ltd.

The raw and composited assay data for the Underground area is shown in Table 14.8.

Table 14.8 Statistics for raw and composited Underground assay data

Field	Samples	Composites
	Au (ppm)	Au (ppm)
Number of samples	4,892	2,328
Minimum	0	0
Maximum	134.85	103.91
Mean	6.78	6.79
Standard deviation	12.11	10.21
Coefficient of variation	1.79	1.50

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.9 shows the comparison on a domain level of raw and composited data for the top 20 domains (based on number of samples / domain). As no capping was applied, mean grades have not changed.

Table 14.9 Statistics of selected raw and composites data

Domain	Statistic	Raw	Composites	Domain	Statistic	Raw	Composites
109	NSamples	277	107	503	NSamples	165	71
	Minimum	0.00	0.14		Minimum	0.00	0.00
	Maximum	116.57	74.11		Maximum	24.39	18.23
	Mean	18.32	18.32		Mean	3.51	3.51
	Standdev	20.83	17.46		Standdev	4.69	3.87
	Coeff. of var.	1.14	0.95		Coeff. of var.	1.34	1.10
110	NSamples	57	34	507	NSamples	123	58
	Minimum	0.00	0.01		Minimum	0.00	0.32
	Maximum	56.64	26.43		Maximum	36.14	22.28
	Mean	6.10	6.10		Mean	2.71	2.71
	Standdev	8.13	6.22		Standdev	4.92	3.82
	Coeff. of var.	1.33	1.02		Coeff. of var.	1.82	1.41
111	NSamples	115	49	511	NSamples	126	59
	Minimum	0.00	0.01		Minimum	0.01	0.17
	Maximum	114.67	30.17		Maximum	74.12	35.73
	Mean	7.66	7.66		Mean	8.30	8.30
	Standdev	12.03	7.72		Standdev	10.53	7.23
	Coeff. of var.	1.57	1.01		Coeff. of var.	1.27	0.87
112	NSamples	304	150	519	NSamples	74	39
	Minimum	0.00	0.00		Minimum	0.02	0.02
	Maximum	92.91	89.30		Maximum	10.77	5.13
	Mean	8.32	8.32		Mean	1.59	1.59
	Standdev	13.74	11.76		Standdev	1.83	1.30
	Coeff. of var.	1.65	1.41		Coeff. of var.	1.15	0.82
113	NSamples	175	72	902	NSamples	90	40
	Minimum	0.00	0.00		Minimum	0.01	0.05
	Maximum	134.85	103.91		Maximum	24.58	15.99
	Mean	14.16	14.16		Mean	3.59	3.59
	Standdev	23.63	20.34		Standdev	4.73	4.09
	Coeff. of var.	1.67	1.44		Coeff. of var.	1.32	1.14
203	NSamples	107	44	1001	NSamples	92	53
	Minimum	0.00	0.34		Minimum	0.01	0.01
	Maximum	56.98	37.92		Maximum	6.75	6.46
	Mean	10.26	10.26		Mean	0.69	0.69
	Standdev	11.89	8.81		Standdev	1.38	1.29
	Coeff. of var.	1.16	0.86		Coeff. of var.	1.99	1.87
301	NSamples	216	106	1002	NSamples	246	131
	Minimum	0.00	0.00		Minimum	0.00	0.01
	Maximum	37.03	26.53		Maximum	88.80	74.78
	Mean	4.61	4.61		Mean	6.55	6.55
	Standdev	6.60	5.64		Standdev	12.28	10.78
	Coeff. of var.	1.43	1.22		Coeff. of var.	1.88	1.65

Domain	Statistic	Raw	Composites	Domain	Statistic	Raw	Composites
306	NSamples	334	140	1003	NSamples	67	38
	Minimum	0.00	0.00		Minimum	0.00	0.00
	Maximum	93.05	64.35		Maximum	43.89	29.95
	Mean	8.66	8.66		Mean	6.72	6.72
	Standdev	14.99	12.02		Standdev	9.57	7.92
	Coeff. of var.	1.73	1.39		Coeff. of var.	1.42	1.18
404	NSamples	201	113	1004	NSamples	88	53
	Minimum	0.01	0.01		Minimum	0.00	0.00
	Maximum	51.91	41.62		Maximum	54.41	39.10
	Mean	4.76	4.76		Mean	7.17	7.17
	Standdev	8.41	6.83		Standdev	11.87	10.42
	Coeff. of var.	1.77	1.44		Coeff. of var.	1.66	1.45
502	NSamples	85	42	1302	NSamples	91	40
	Minimum	0.00	0.01		Minimum	0.01	0.01
	Maximum	12.96	9.99		Maximum	130.42	66.95
	Mean	2.50	2.50		Mean	8.26	8.26
	Standdev	3.01	2.35		Standdev	14.95	11.70
	Coeff. of var.	1.21	0.94		Coeff. of var.	1.81	1.42

Notes:
- NSamples=number of samples, Standdev=standard deviation, Coeff. of var.=Coefficient of variation. For minimum, maximum, and mean values gold is in g/t.
- Composite grades are not declustered.

Source: AMC Mining Consultants (Canada) Ltd.

14.6 Block model

14.6.1 Block model parameters

The parent block size was 10 ft by 10 ft by 10 ft with sub-blocking employed. Sub-blocking resulted in minimum cell dimensions of 2.5 ft by 2.5 ft by 2.5 ft.

The block model dimensions and are shown in Table 14.10. The block model was not rotated.

Table 14.10 Block model parameters for the Underground area

Parameter	X	Y	Z
Origin	8,500	10,400	2800
Maximum block size (ft)	10	10	10
Minimum block size (ft)	2.5	2.5	2.5
Number of blocks	275	300	330

Source: AMC Mining Consultants (Canada) Ltd.

14.6.2 Grade estimation

Search ellipses were based on the orientation of mineralization and drillhole spacing. Variography was investigated, but due to poor quality variograms, variography was not used to inform the size of the search ellipses.

Interpolation was carried out using three methods: inverse-distance squared, inverse-distance cubed and nearest neighbour. The grades were estimated for each domain individually. OK was not used as an interpolation method due to the small number of samples in many domains.

Mineralization domains were grouped into 16 sets based on similar orientations. The same search parameters were used for each set of wireframes. The search parameters are shown in Table 14.11.

Three passes were used:

- Pass 1: the minimum number of samples allowed to inform a block = 6.
- Pass 2: the minimum number of samples allowed to inform a block = 4.
- Pass 3: the minimum number of samples allowed to inform a block was either 1 or 2.

In all cases, the maximum number of samples allowed to inform a block was 20 and the maximum number of samples allowed from one drillhole was two.

Table 14.11 Search parameters for the Underground area

SREFNUM	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**
SDIST1	100	100	100	100	100	100	100	100	200	100	100	100	100	100	100	100
SDIST2	100	100	100	100	100	100	100	100	200	100	100	100	100	100	100	100
SDIST3	25	25	25	25	25	25	25	40	40	25	25	25	40	30	25	25
SANGLE1	40	40	65	45	30	45	50	45	45	45	20	75	18	45	0	17
SANGLE2	85	-85	-85	60	-65	85	-85	-65	-70	-80	-55	-50	-60	-75	-30	82
SANGLE3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
SAXIS1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
SAXIS2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
SAXIS3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
SVOLFAC2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
SVOLFAC3	4	4	3	4	3	3	3	4	5	3	4	4	5	5	3	3
MINNUM3	1	2	2	2	1	2	2	1	1	2	1	1	1	2	2	2

Notes:

- SREFNUM = Search volume reference number
- SDIST1 = Max search distance in direction 1
- SDIST2 = Max search distance in direction 2
- SDIST3 = Max search distance in direction 3
- SANGLE1 = First rotation angle for search volume
- SANGLE2 = Second rotation angle for search volume
- SANGLE3 = Third rotation angle for search volume
- SAXIS1 = Axis for 1st rotation (1=X,2=Y,3=Z)
- SAXIS2 = Axis for 2nd rotation (1=X,2=Y,3=Z)
- SAXIS3 = Axis for 3rd rotation (1=X,2=Y,3=Z)
- SVOLFAC2 = axis multiplying factor for second dynamic search volume
- SVOLFAC3 = axis multiplying factor for third dynamic search volume
- MINNUM3 = Minimum number of samples for third dynamic search volume

Source: AMC Mining Consultants (Canada) Ltd.

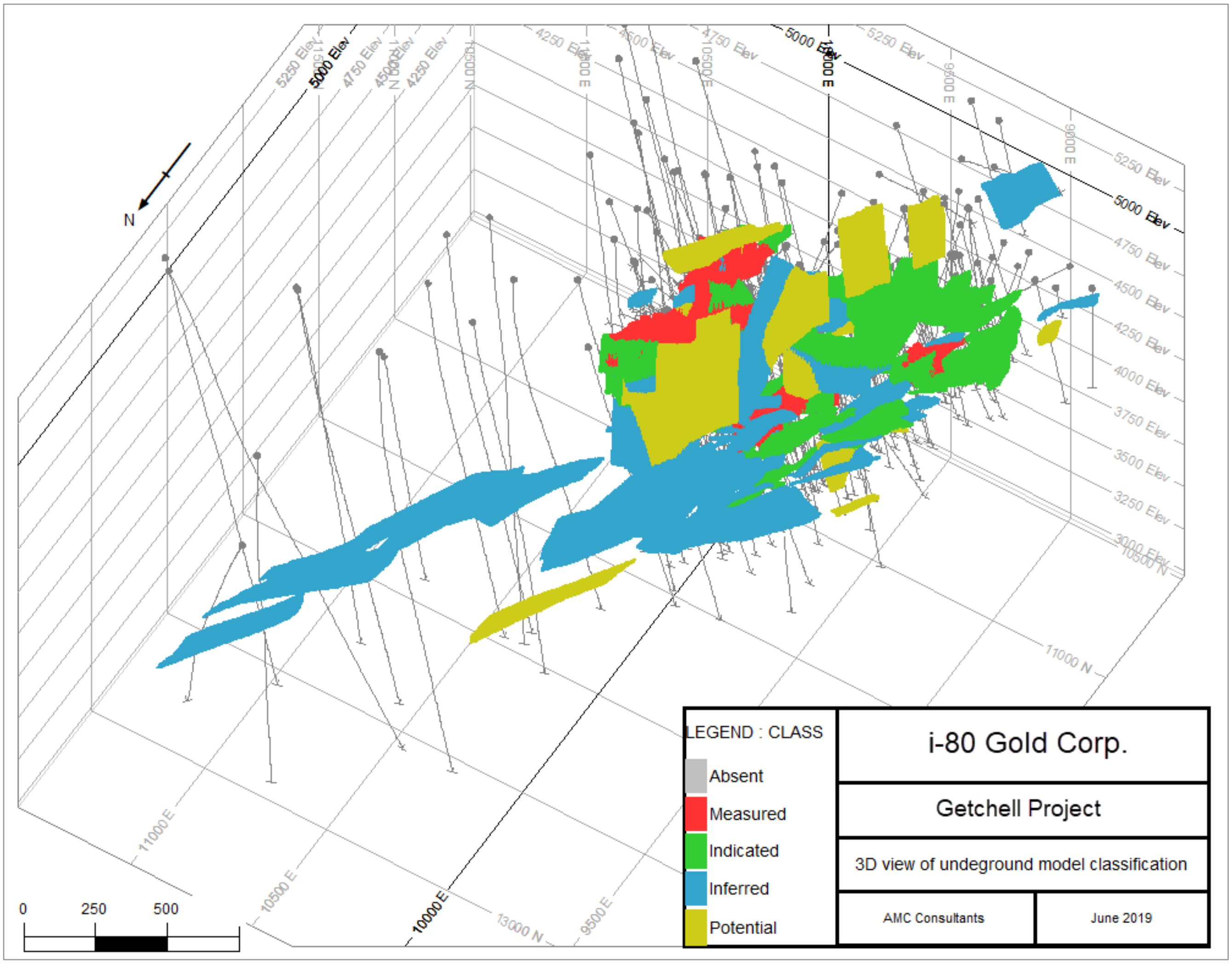
The blocks inside the block model are coded by estimated gold as well as assigned bulk density, by geology and claim status. The entire Underground area block model has been assigned a ZONE code of 5 to distinguish it from the Open Pit area model.

14.6.3 Mineral Resource classification

Mineral Resource classification was completed using an assessment of geological and mineralization continuity, data quality and data density. Data quality included the presence or absence of original assay certificates. Estimation passes were used as an initial guide for classification. Wireframes were generated manually to build coherent areas and the presence or absence of original assay certificates was taken into account.

Figure 14.7 shows a 3D view of the underground Mineral Resource classification. The Mineral Resource presently excludes several zones of relatively continuous mineralization which were solely defined by drillhole assays that could not be supported by original certificates. Verification of assays in this region, or additional drilling to confirm these results may provide sufficient justification to classify Mineral Resources in these areas.

Figure 14.7 3D View of Underground area Mineral Resources classification



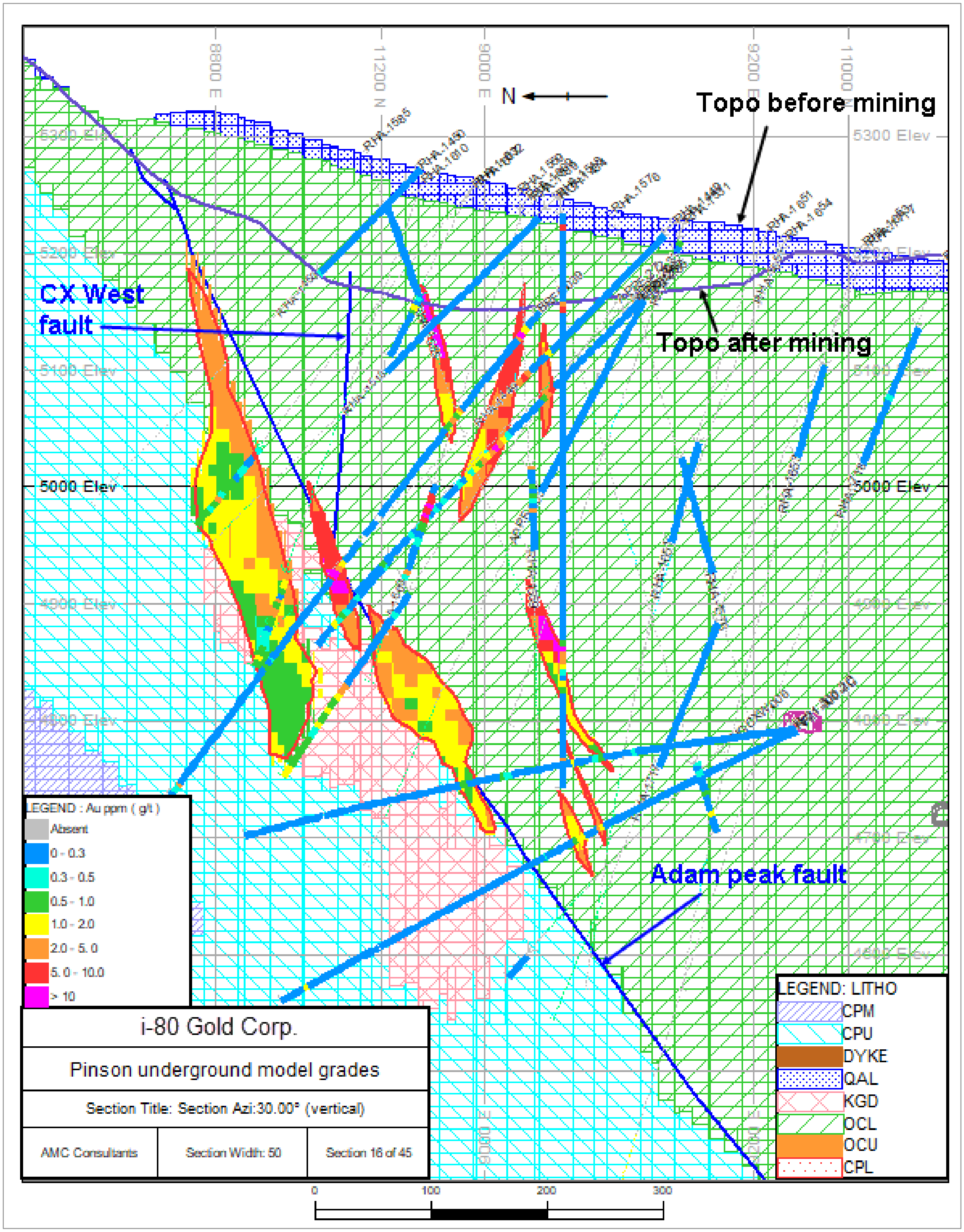
Notes: Waste blocks were assigned the classification=absent but are not shown in the above figure.
Source: AMC Mining Consultants (Canada) Ltd.

14.7 Underground block model validation

The block model was validated in three ways. First visual checks were carried out to ensure that the grades respected the raw assay data and also lay within the constraining wireframes. Secondly the estimate was statistically compared to the final (composited) assay data. Thirdly, swath plots were reviewed. The QP attempted to compare the current block model to historical mining but due to incomplete production records this was not possible.

An example of the drillhole composite gold grades compared to the block model estimated grades is shown in Figure 14.8. The figure shows good agreement between the drillhole composite grades and estimated block model grades.

Figure 14.8 Gold grade cross section of Underground area



Notes: Section Line is Section 16 as shown on Figure 14.4. Mineralization is all classified as Inferred or higher.
Source: AMC Mining Consultants (Canada) Ltd. 2019

Table 14.12 shows the statistical comparison of the composites versus the block model grades for gold for the top 20 domains. Top 20 domains were selected based on the highest number of composites in the domain. The results are satisfactory.

Table 14.12 Comparison of Underground area block model and composites

Domain	Statistic	Composites	Model	Domain	Statistic	Composites	Model
109	NSamples	107	12,706	503	NSamples	71	34,307
	Minimum	0.14	2.22		Minimum	0.00	0.07
	Maximum	74.11	61.34		Maximum	18.23	14.61
	Mean	18.32	18.48		Mean	3.51	3.66
	Standdev	17.46	8.46		Standdev	3.87	2.30
	Coeff. of var.	0.95	0.46		Coeff. of var.	1.10	0.63
110	NSamples	34	2,139	507	NSamples	58	30,023
	Minimum	0.01	0.40		Minimum	0.32	0.36
	Maximum	26.43	14.37		Maximum	22.28	18.32
	Mean	6.10	6.10		Mean	2.71	2.34
	Standdev	6.22	2.49		Standdev	3.82	2.18
	Coeff. of var.	1.02	0.41		Coeff. of var.	1.41	0.93
111	NSamples	49	11,768	511	NSamples	59	74,881
	Minimum	0.01	0.14		Minimum	0.17	0.64
	Maximum	30.17	24.08		Maximum	35.73	33.16
	Mean	7.66	7.92		Mean	8.30	8.45
	Standdev	7.72	4.16		Standdev	7.23	3.73
	Coeff. of var.	1.01	0.52		Coeff. of var.	0.87	0.44
112	NSamples	150	8,125	519	NSamples	39	18,454
	Minimum	0.00	0.50		Minimum	0.02	0.02
	Maximum	89.30	71.15		Maximum	5.13	4.66
	Mean	8.32	7.22		Mean	1.59	1.70
	Standdev	11.76	5.61		Standdev	1.30	0.96
	Coeff. of var.	1.41	0.78		Coeff. of var.	0.82	0.56
113	NSamples	72	3,271	902	NSamples	40	142,011
	Minimum	0.00	2.37		Minimum	0.05	0.17
	Maximum	103.91	59.58		Maximum	15.99	15.08
	Mean	14.16	14.36		Mean	3.59	4.71
	Standdev	20.34	10.46		Standdev	4.09	2.43
	Coeff. of var.	1.44	0.73		Coeff. of var.	1.14	0.52
203	NSamples	44	17,521	1,001	NSamples	53	3,605
	Minimum	0.34	0.84		Minimum	0.01	0.23
	Maximum	37.92	30.38		Maximum	6.46	2.70
	Mean	10.26	9.37		Mean	0.69	0.82
	Standdev	8.81	3.99		Standdev	1.29	0.40
	Coeff. of var.	0.86	0.43		Coeff. of var.	1.87	0.48

Domain	Statistic	Composites	Model	Domain	Statistic	Composites	Model
301	NSamples	106	66,315	1,002	NSamples	131	30,393
	Minimum	0.00	0.06		Minimum	0.01	0.25
	Maximum	26.53	21.89		Maximum	74.78	56.47
	Mean	4.61	4.14		Mean	6.55	7.53
	Standdev	5.64	3.19		Standdev	10.78	4.86
	Coeff. of var.	1.22	0.77		Coeff. of var.	1.65	0.64
306	NSamples	140	28,683	1,003	NSamples	38	7,044
	Minimum	0.00	0.35		Minimum	0.00	0.63
	Maximum	64.35	40.45		Maximum	29.95	21.63
	Mean	8.66	5.83		Mean	6.72	6.87
	Standdev	12.02	3.63		Standdev	7.92	3.48
	Coeff. of var.	1.39	0.62		Coeff. of var.	1.18	0.51
404	NSamples	113	2,042	1,004	NSamples	53	12,457
	Minimum	0.01	0.37		Minimum	0.00	0.07
	Maximum	41.62	23.07		Maximum	39.10	24.04
	Mean	4.76	5.03		Mean	7.17	5.81
	Standdev	6.83	3.40		Standdev	10.42	4.12
	Coeff. of var.	1.44	0.68		Coeff. of var.	1.45	0.71
502	NSamples	42	42,156	1,302	NSamples	40	102,057
	Minimum	0.01	0.09		Minimum	0.01	0.11
	Maximum	9.99	7.84		Maximum	66.95	51.84
	Mean	2.50	2.71		Mean	8.26	8.97
	Standdev	2.35	1.26		Standdev	11.70	6.61
	Coeff. of var.	0.94	0.47		Coeff. of var.	1.42	0.74

Notes:
- NSamples=number of samples, Standdev=standard deviation, Coeff. of var.=Coefficient of variation. For minimum, maximum, and mean values gold is in g/t.
- Composite grades are not declustered.

Source: AMC Mining Consultants (Canada) Ltd.

14.8 Underground Mineral Resource statement

Mineral Resources are reported at a COG of 0.15 opt Au for the Underground area. the Companies provided the initial COG calculations and the QP verified the reasonableness of the assumptions. The COG is based on actual and benchmark cost data for similar scale of operations and assumptions regarding mineral processing metal recoveries and metal prices. Operating costs for an underground mine include mining, General and Administration (G&A), refining and gold transport costs are estimated to be $200/t of feed. Metal price used for gold is $1,550/oz and mineral processing recovery is assumed to be 90%. Varying royalties are applied at varying trigger points throughout the mine life, for simplicity a constant 6% royalty for the calculation of COG was used. Further details of the COG inputs are shown below in Table 14.13.

Table 14.13 Inputs into underground COG calculations

Description	Values	Units
Au Price	1,550	US$/oz
Royalties	6%	%
Costs:		
Mining	100	$/t processed
Haul	30	$/t processed
Process	60	$/t processed
G&A	10	$/t processed
Overall Costs	200	$/t processed
Mill Au recovery	90%	%
Breakeven Au for stopes	0.15	opt

Table 14.14 show a summary of the Underground area Mineral Resource.

Table 14.14 Summary of the Underground area Mineral Resource as of 23 July 2020

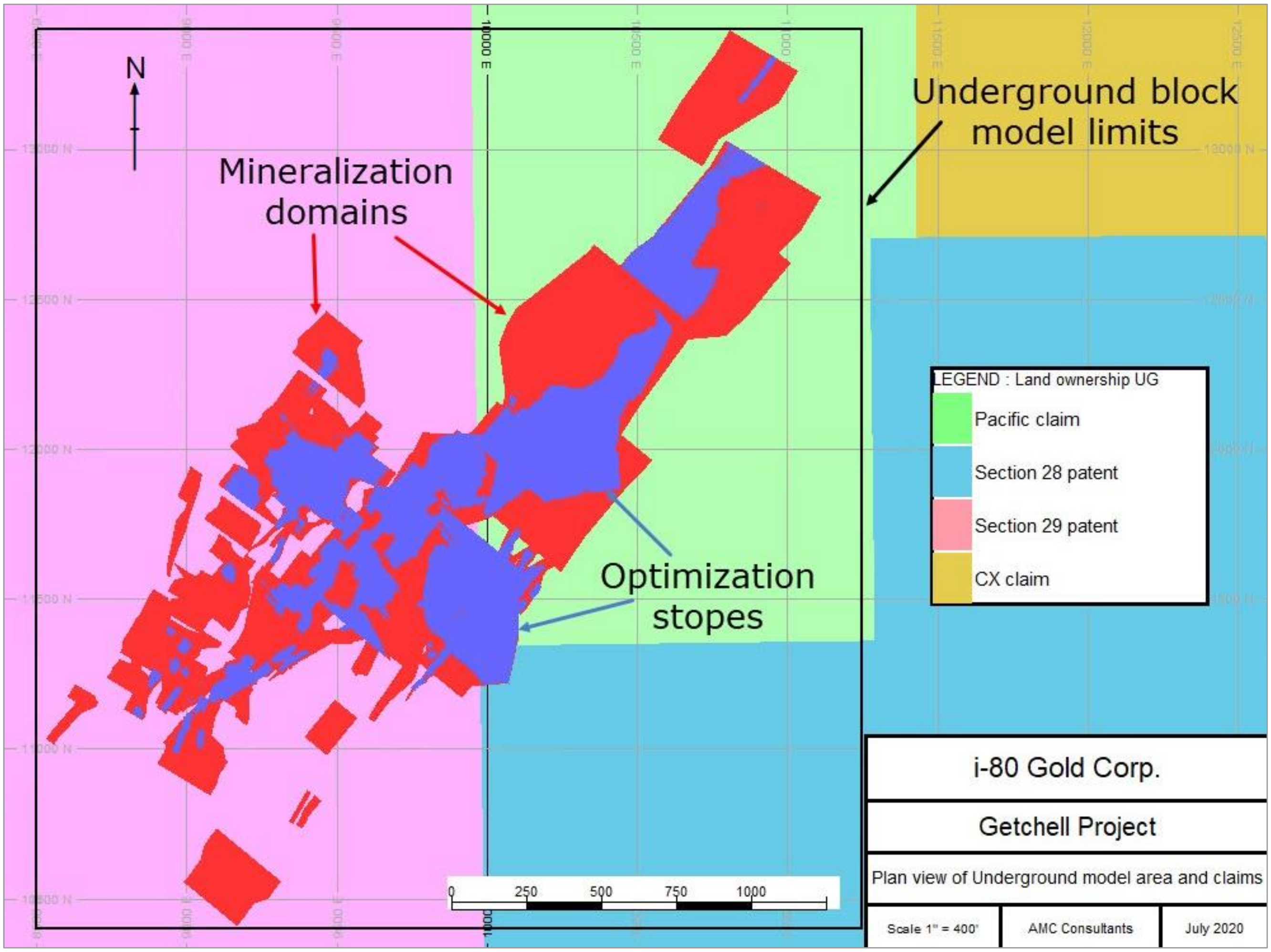
Classification	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Measured	184	0.289	53
Indicated	436	0.313	136
Measured and Indicated	**620**	**0.306**	**190**
Inferred	1,676	0.347	581

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- The Mineral Resource COG is based on a metal price of $1,550/oz Au.
- Underground Mineral Resources as stated are constrained within modelled underground stope shapes using a nominal 15' minimum thickness, above a gold cut-off grade of 0.15 opt Au.
- Drilling results up to 15 April 2019.
- Drilling database provided 18 April 2019.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.9 shows the claim ownership with respect to the Underground area block model. The QP notes that OMC owns 41.67% of Section 28. The small portion of the Underground area block model on this section is Inferred material.

Figure 14.9 Plan view of Underground area block model and claims



Source: AMC Mining Consultants (Canada) Ltd.

A breakdown of the Underground area Mineral Resource by claim is shown in Table 14.15.

Table 14.15 Underground area Mineral Resources 23 July 2020 by claim

Claim	Classification	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Pacific claim	Measured	40	0.388	16
	Indicated	67	0.527	35
	Measured and Indicated	107	0.475	51
	Inferred	1,063	0.403	429
Section 28 patent	Measured	-	-	-
	Indicated	-	-	-
	Measured and Indicated	-	-	-
	Inferred	39	0.271	11
Section 29 patent	Measured	144	0.261	38
	Indicated	369	0.274	101
	Measured and Indicated	513	0.270	139
	Inferred	573	0.247	142
Total	Measured	184	0.289	53
	Indicated	436	0.313	136
	Measured and Indicated	**620**	**0.306**	**190**
	Inferred	1,676	0.347	581

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- The Mineral Resource COG is based on a metal price of $1,550/oz Au.
- Underground Mineral Resources as stated are constrained within modelled underground stope shapes using a nominal 15’ minimum thickness, above a gold cut-off grade of 0.15 opt Au.
- Drilling results up to 31 December 2015.
- Drilling database provided 18 April 2019.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The numbers may not compute exactly due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

14.9 Comparison with previous underground estimate

The previous Mineral Resource estimate on the Property dated 30 June 2014 was published in the Golder Associates 2014 Report for Atna. Changes to the Mineral Resource estimate in this report are due predominantly to:

- New interpretation of mineralized domains.
- Updated structural and geological model.
- New estimation was based on individual domains not large solid panels.
- Updated COG parameters and reporting at different COGs.
- Classification based on additional data verification information.

Table 14.16 shows the comparison of the Underground Mineral Resources with previously published estimate.

Table 14.16 Comparison of Underground Mineral Resources

Mineral Resource	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnage (ktons)	Au (opt)	Metal Au (koz)	Tonnage (ktons)	Au (opt)	Metal Au (koz)	Tonnage (ktons)	Au (opt)	Metal Au (koz)	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Oxide (Atna 2014)	31	0.464	14	245	0.36	88	276	0.372	103	366	0.386	141
Sulphide (Atna 2014)	39	0.605	24	492	0.449	221	531	0.461	244	1,306	0.428	559
Total (Atna 2014)	70	0.543	38	737	0.419	309	807	0.43	347	1,672	0.419	700
AMC 2019	184	0.289	53	436	0.313	136	620	0.306	190	1,676	0.347	581
Difference in %	163	-47	40	-41	-25	-56	-23	-29	-45	0	-17	-17

Notes for the Atna Estimate:
- Atna reported out at COG of 0.22 opt and 0.19 opt for oxide and refractory sulphide Mineral Resources, respectively.
- Source: Golder Associates (2014).

Notes for the AMC estimate:
- See notes on Table 1.1 with respect to the current estimate.
- Source: AMC Mining Consultants (Canada) Ltd.

14.10 Open Pit area modelling

14.10.1 Modelling overview

The use of the indicator method for estimation of mineralization domains is a rapid and satisfactory process for modelling the Open Pit areas at Pinson.

The QP used the following method to prepare the block model:

- Constructing mineralization envelopes for each pit.
- Compositing drillholes for each pit.
- Flagging the composites as being above or below a threshold value.
- Calculating and modelling variograms for the indicator variable.
- Estimating the indicator value using OK.
- Selecting a nominal probability limit from the estimated indicator value above which is “high-grade model” and below which is the “low-grade model”.
- Determining suitable search ellipsoid orientations for both models.
- Estimating gold grades using either OK or ID^2 (depending on sample support) for the two models.
- Combining the models and classify.
- Code block model with cyanide leach recovery assumptions.
- Combining with geology model (provided by client) to assign densities based on geology and grades.
- Depleting the model using the topography provided by the client.

Details of the process follows and there is further discussion of the statistics and variography as it relates to this process in Section 14.9.

14.10.2 Mineralization envelopes for Open Pits

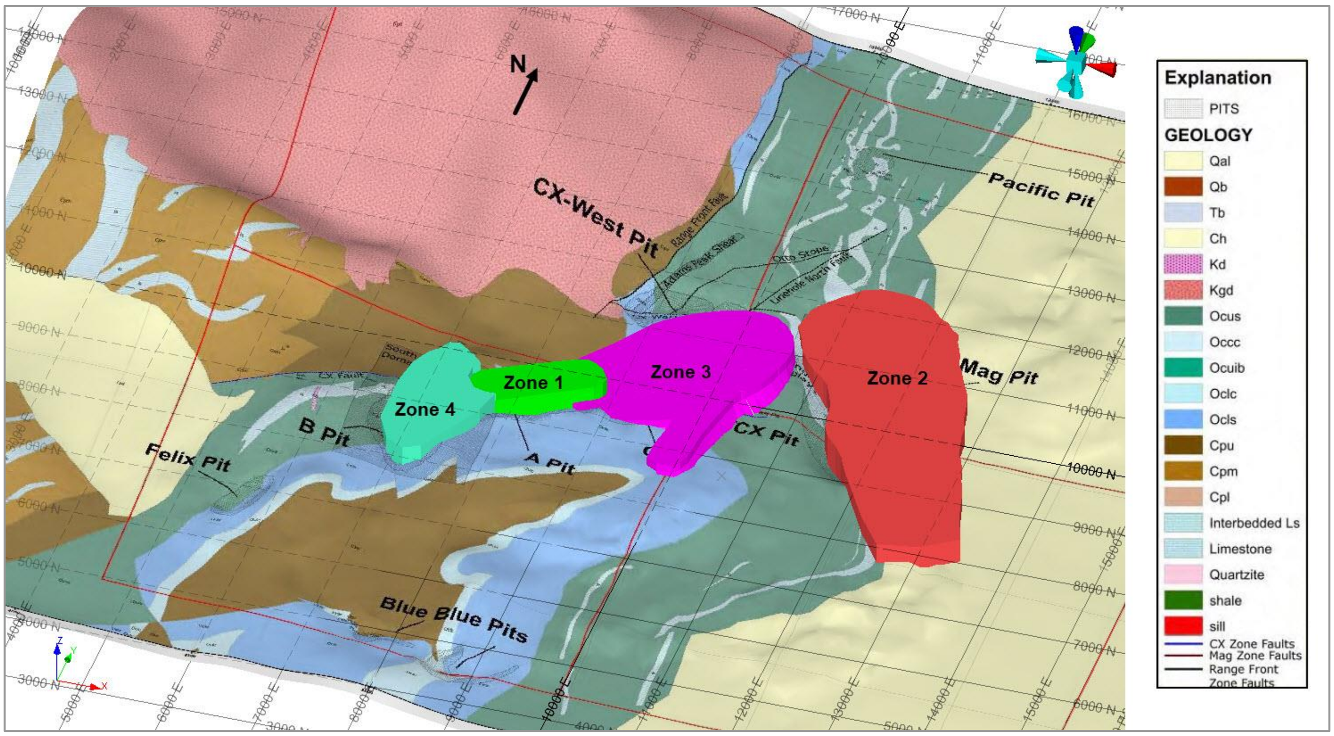
For the purpose of building mineralization envelopes, Pit A, Pit B, and Mag Pit were handled differently than the more complex C and CX Pits.

14.10.2.1 Pit A, Pit B, and Mag Pit

A new mineralization envelope (zone) was constructed for each pit using the indicator estimation method. The new envelopes were based primarily on the grade distribution, but also tied in with the general geology. These envelopes are "soft" interpretations as they are intended to generally capture most of the mineralization in a reasonable manner, while not using strict criteria for unmineralized material incorporated into the shapes.

The purpose of the mineralization envelope is to define the estimation boundaries. The outline of these mineralization envelopes is shown in Figure 14.10.

Figure 14.10 Mineralization envelopes overview



Notes: Zone 1=Pit A, Zone 2=Mag Pit, Zone 3=C and CX Pits, Zone 4=Pit B. At Client's request the Felix Pit and blue pits were not estimated.
Source: Regional map supplied by Osgood Mining Company LLC., Mineralization envelopes (zones) superimposed by AMC Mining Consultants (Canada) Ltd.

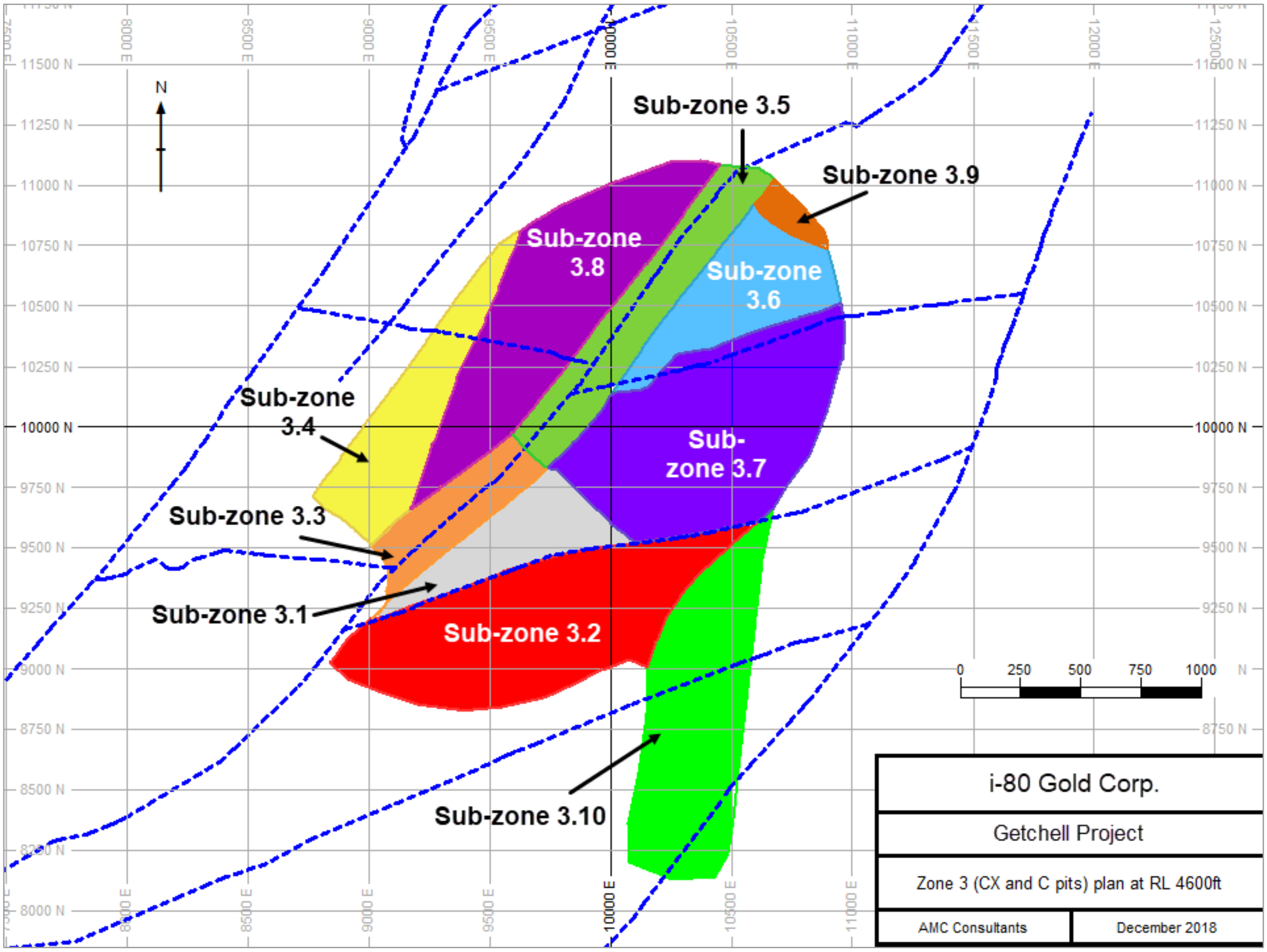
14.10.2.2 Pits C and CX

Multiple new mineralization envelopes were constructed for both pits using the indicator estimation method and to manage issues related to faulting and the subsequent varying orientations of the mineralization. The new envelopes were strongly influence by the pit mapping of Chadwick (2002).

To account for the geological and structural complexity in this area, ten mineralization envelopes (sub-zones) were built inside Zone 3.

Figure 14.11 shows the mineralized envelopes (sub-zones) for the C and CX Pits.

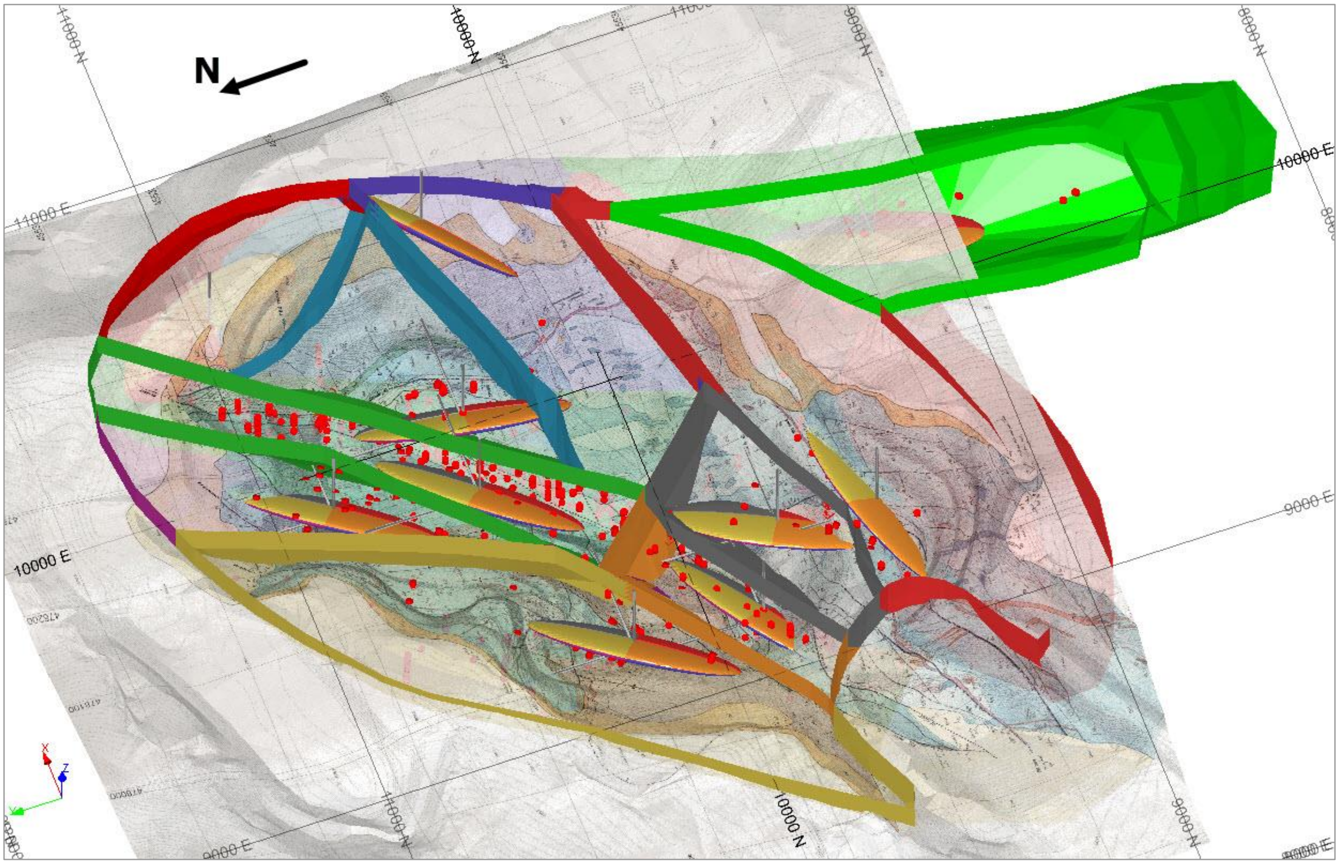
Figure 14.11 Mineralization envelopes for the C and CX Pits



Note: Blue dashed lines are faults.
Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.12 shows the trends of mineralization greater than 1 g/t gold. Mineralization has various orientations due to complex folding and faulting as shown on the underlying (transparent) pit map.

Figure 14.12 3D View of mineralization envelopes for the C and CX Pits



Notes: Sub zones for CX and C Pits (legend as per Figure 14.11). Ellipsoids highlight mineralization trends. Transparent underlay is pit mapping draped onto the 3D surface.
Source: AMC Mining Consultants (Canada) Ltd.; underlay Chadwick 2002.

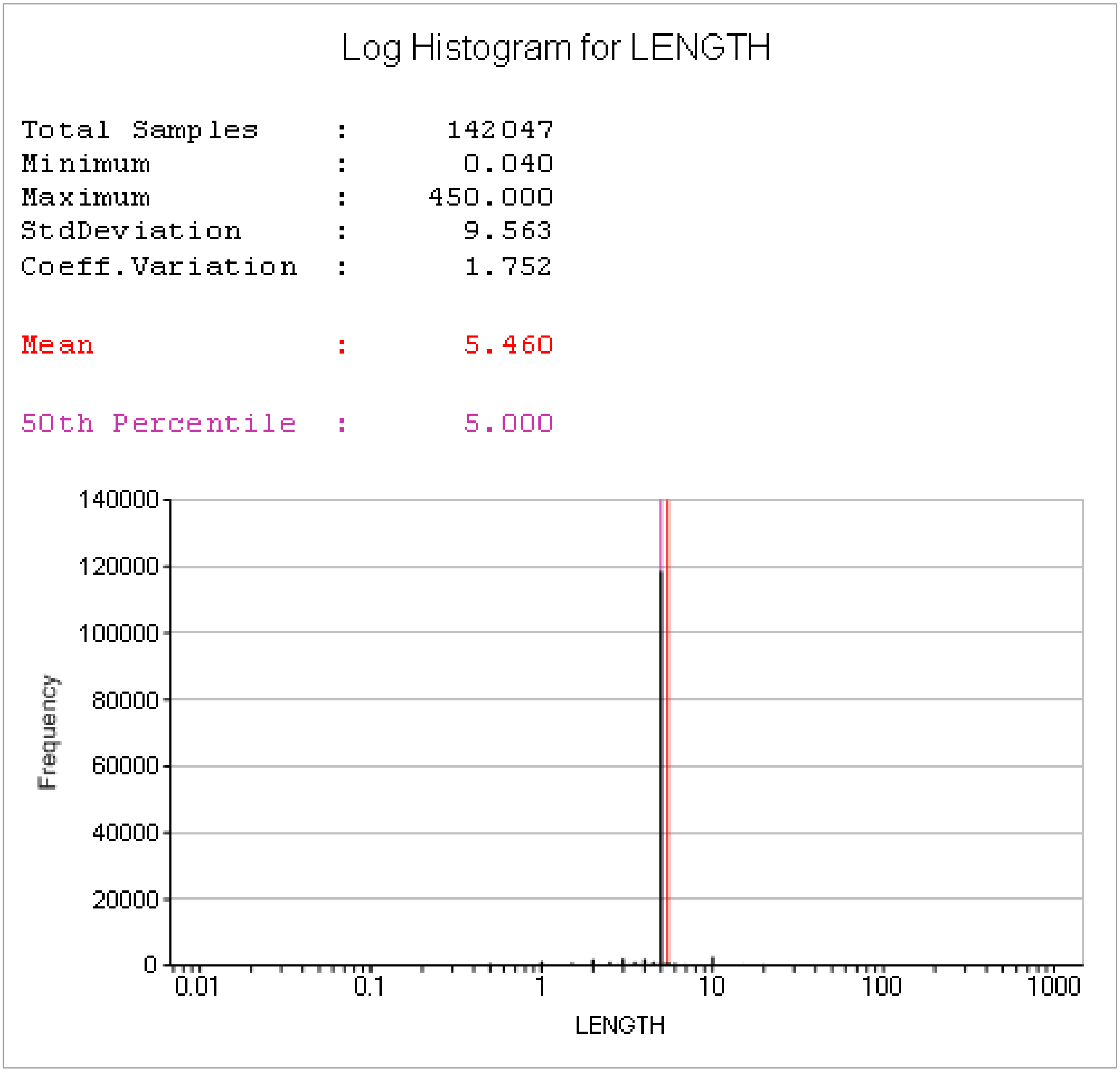
14.11 Open Pit statistics and compositing

14.11.1 Compositing

The drillhole database, coded with interpreted mineralization envelopes via the ZONE / SUBZONE field, was composited to a regular 10 ft downhole composite length as a means of achieving a uniform sample support. Any missing, unassayed intervals were set to zero grade.

The decision to use 10 ft composites considered the common raw sampling intervals in the drillhole data, the amount of data available for the domains, definition of mineralization, and the parent cell sizes used for modelling. Most of the assay data for the mineralized drillhole intersections are from core samples collected for a range of intervals, with 5 ft being the most common downhole lengths (Figure 14.13). A 10 ft composite interval was selected as appropriate as it avoided the decompositing of original sample intervals and was appropriate for the block size.

Figure 14.13 Raw sample interval lengths for drillhole data



Source: AMC Mining Consultants (Canada) Ltd.

14.11.2 Indicator model

The next step to generate a probability or indicator model was to determine an appropriate mineralization threshold grade. An indicator field (AA) was built and everything above 1 g/t gold (0.03 opt) was coded as 1. Anything below 1 g/t was coded as 0.

Variogram models were then created for Pit A, Pit C and Mag Pit where the threshold was greater than 1 g/t gold. Due to the complexity of Pit C and CX variograms were done on the subzones as well.

Variograms were done for Indicator (AA) = 1.

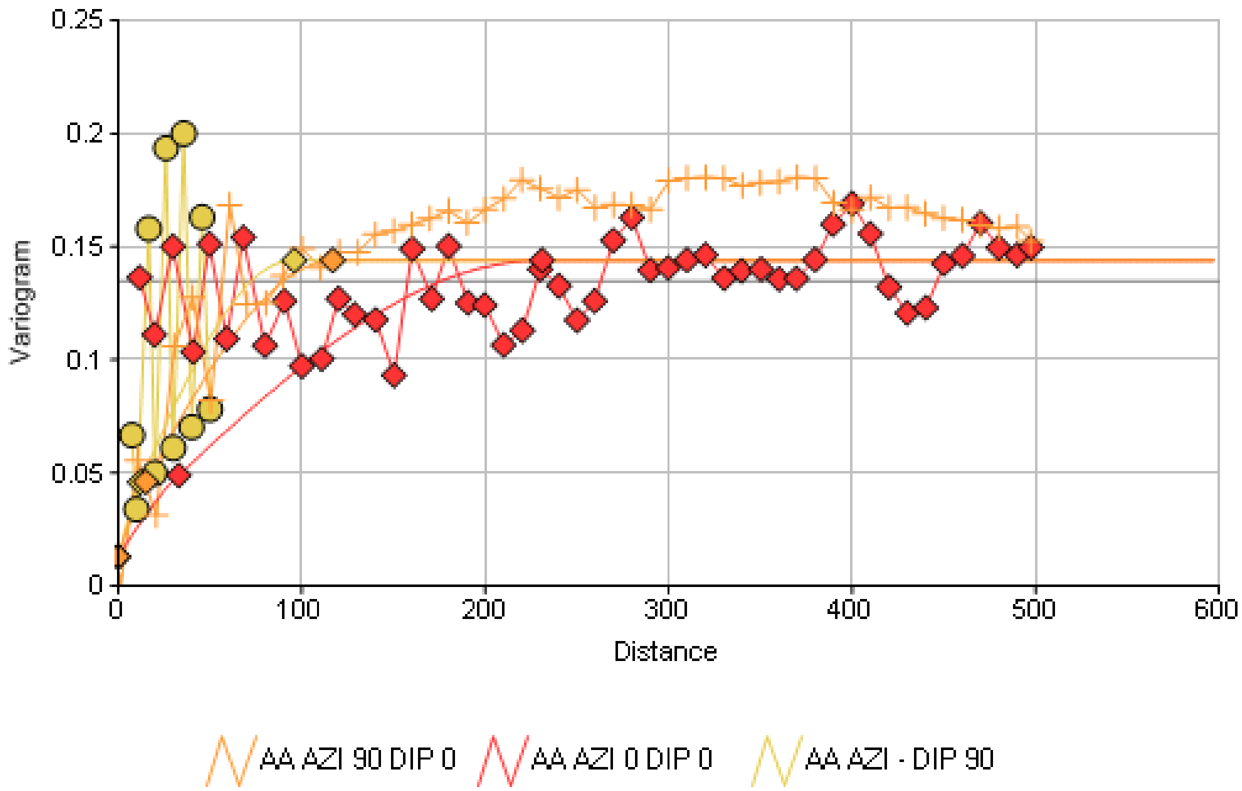
The indicator variogram model for the Mag Pit is shown in Figure 14.14.

Results for the Mag Pit is shown as an example for all sections as it contains the largest number of gold ounces.

Figure 14.14 Variograms for Mag Pit (Zone 2) drillhole data > 1 g/t gold

Variogram of Indicator value AA for Zone 2 (Mag pit)

Type	Variance	90/0	0/0	-/90
Nugget	0.013	-	-	-
Spherical	0.010	14.8	31.6	12.8
Spherical	0.121	117.7	230.0	94.8



Source: AMC Mining Consultants (Canada) Ltd.

Table 14.17 shows the Indicator variograms for the Open Pit areas.

Table 14.17 Indicator variogram models for the Open Pit areas

Pit	A	Mag	C and CX										B
Zone	**1**	**2**	**3**										**4**
Sub-zone	NA	NA	3.1	3.2	3.3	3.4	3.5	3.6	3.7	3.8	3.9	3.10	NA
Z axis rotation angle	57	150	27	75	55	35	40	10	50	35	160	15	15
Y axis rotation angle	-50	50	-40	-50	-50	-50	-50	-50	-60	-50	-60	-35	-35
X axis rotation angle	0	0	0	0	0	0	0	0	0	0	0	0	0
Nugget	0.008	0.013	0.002	0.00	0.01	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.015
Structure 1													
Major range	78.5	14.8	17.5	28.30	25.80	28.30	23.50	24.20	18.80	23.50	18.40	28.90	16.6
Semi-major range	94.2	31.6	21.5	36.3	39.2	36.8	24.9	34.3	24.2	22.2	21.5	31.6	16.6
Minor range	26.2	12.8	12.1	16.80	9.00	13.50	12.80	11.40	10.10	10.10	9.90	13.50	10
Sill component	0.005	0.01	0	0.01	0.03	0.00	0.05	0.00	0.00	0.01	0.01	0.00	0.012
Structure 2													
Major	177.1	117.7	56.5	91.5	141.3	98.7	100.0	84.8	81.4	84.1	55.6	92.8	73.7
Semi-major	227.6	230.0	69.3	118.4	181.6	126.5	117.0	123.8	94.2	105.6	58.7	98.9	101.5
Minor	58.2	94.8	41.0	50.4	44.8	57.4	49.8	44.4	45.1	30.3	35.0	41.0	57.7
Sill component	0.089	0.121	0.017	0.03	0.06	0.04	0.05	0.02	0.01	0.01	0.00	0.01	0.123

Note: Variogram model type is spherical.

To estimate the indicator value, OK was used for each zone. For Pit CX, when there was not enough data to generate a good variogram, ID^2 was used instead.

The parent block size for the Indicator model was 10 ft by 10 ft by 10 ft with block splitting employed. Block splitting resulted in minimum cell dimensions of 2.5 ft by 2.5 ft in the X and Y direction. The smallest Z direction is 1 ft. The block model is not rotated. The Indicator block model extents are shown in Table 14.18.

Table 14.18 Block model parameters for Open Pit area indicator models

Parameter	X	Y	Z
Origin (ft)	6,800	7,600	2,800
Parent block size (ft)	10	10	10
Minimum block size (ft)	2.5	2.5	0.5
No. of blocks	620	580	330

The dimensions of the search radius for the zones / subzones are shown in Table 14.19.

A number of passes were employed, each using different search distances and multiples as follows:

- Pass 1 = 1 x search distance.
- Pass 2 = 2 x search distance.
- Pass 3 = 3 x search distance.

Parameters used for the pass 1 search are summarized in Table 14.19. Note, there was no rotation of the search ellipse along the X axis. For all passes the minimum number of samples is 4 and the maximum number of samples is 12. The minimum number of drillholes to inform a block is two.

Table 14.19 Pass 1 search ellipse parameters for gold

Zone	Sub-zone	Pit	X (ft)	Y (ft)	Z (ft)	Z-Axis rotation (degrees)	Y-axis rotation (degrees)
1	NA	A	175	220	100	57	-50
2	NA	Mag	118	230	95	150	50
3	3.1	C & CX	57	70	40	27	-40
	3.2		92	118	50	75	-50
	3.3		141	181	45	55	-50
	3.4		100	127	57	35	-50
	3.5		100	117	50	40	-50
	3.6		85	124	44	10	-50
	3.7		81	94	45	50	-60
	3.8		84	106	30	35	-50
	3.9		56	59	35	160	-60
	3.10		93	100	41	15	-35
4	NA	B	74	101	58	15	-35

Notes: Z-axis rotation describes the rotation of the ellipse about the Z-axis in a counterclockwise direction when negative and clockwise when positive. Y-axis rotation describes the rotation of the ellipse about the Y-axis in a clockwise direction.
Source: AMC Mining Consultants (Canada) Ltd.

Estimation parameters for AA=1 using OK.

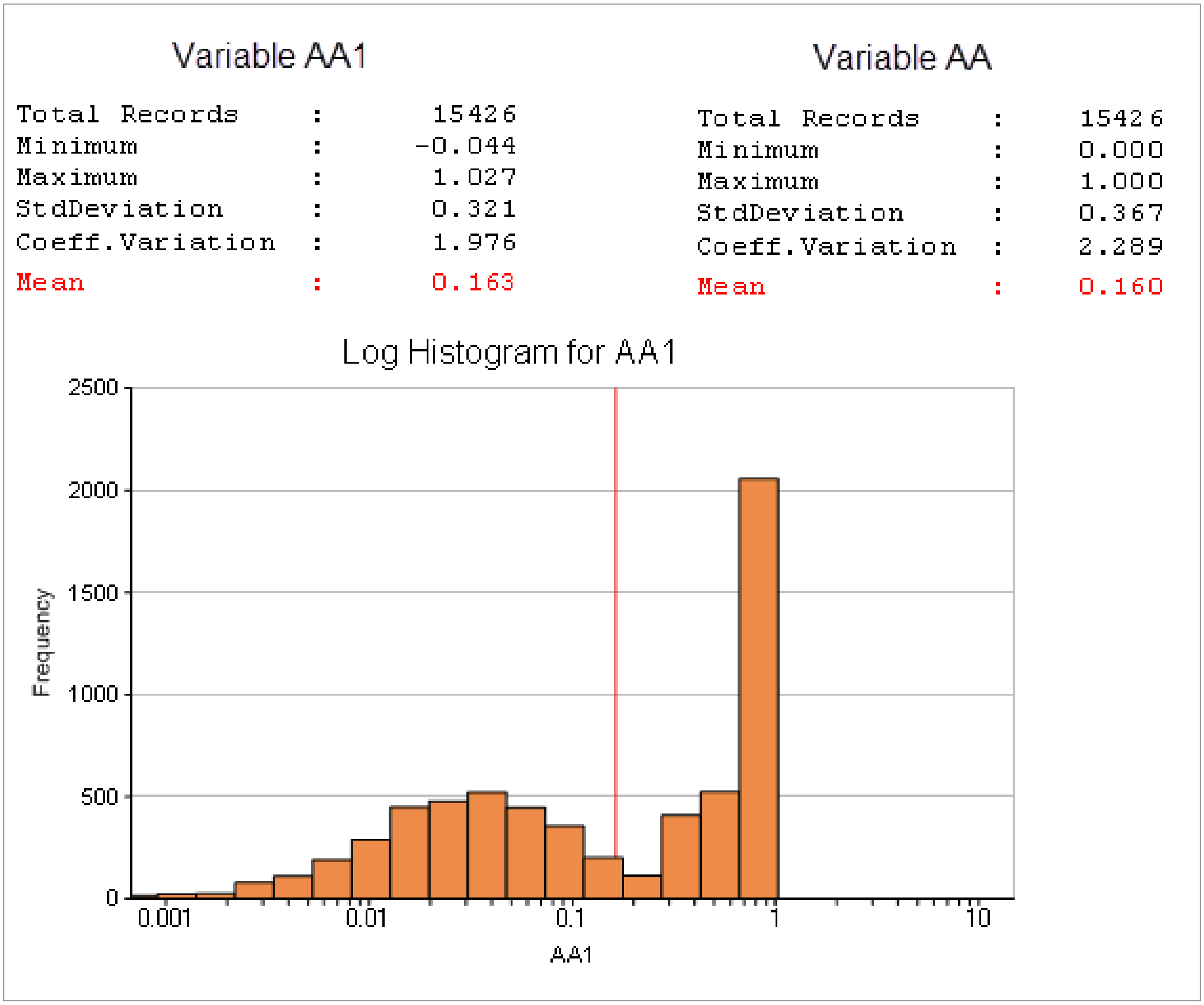
For the CX and C model, there were 10 sub-zones. If there was not enough data for a good variogram the sub-zone was estimated with ID^2 instead of OK.

Field AA1 is a new field that contains the different probabilities.

Figure 14.15 shows there are two populations. Based on the histogram a break between high and low-grades mineralization was established at 0.3 probability.

Based on the results of the Indicator block models an estimated probability of 0.3 was selected based on visual results. This division was coded into the composite file where >0.3 probability is called "high-grade" and less than 0.3 probability is called "low-grade".

Figure 14.15 Log Histogram of the estimated Indicator (AA1)



Note: Input was AA.
Source: AMC Mining Consultants (Canada) Ltd.

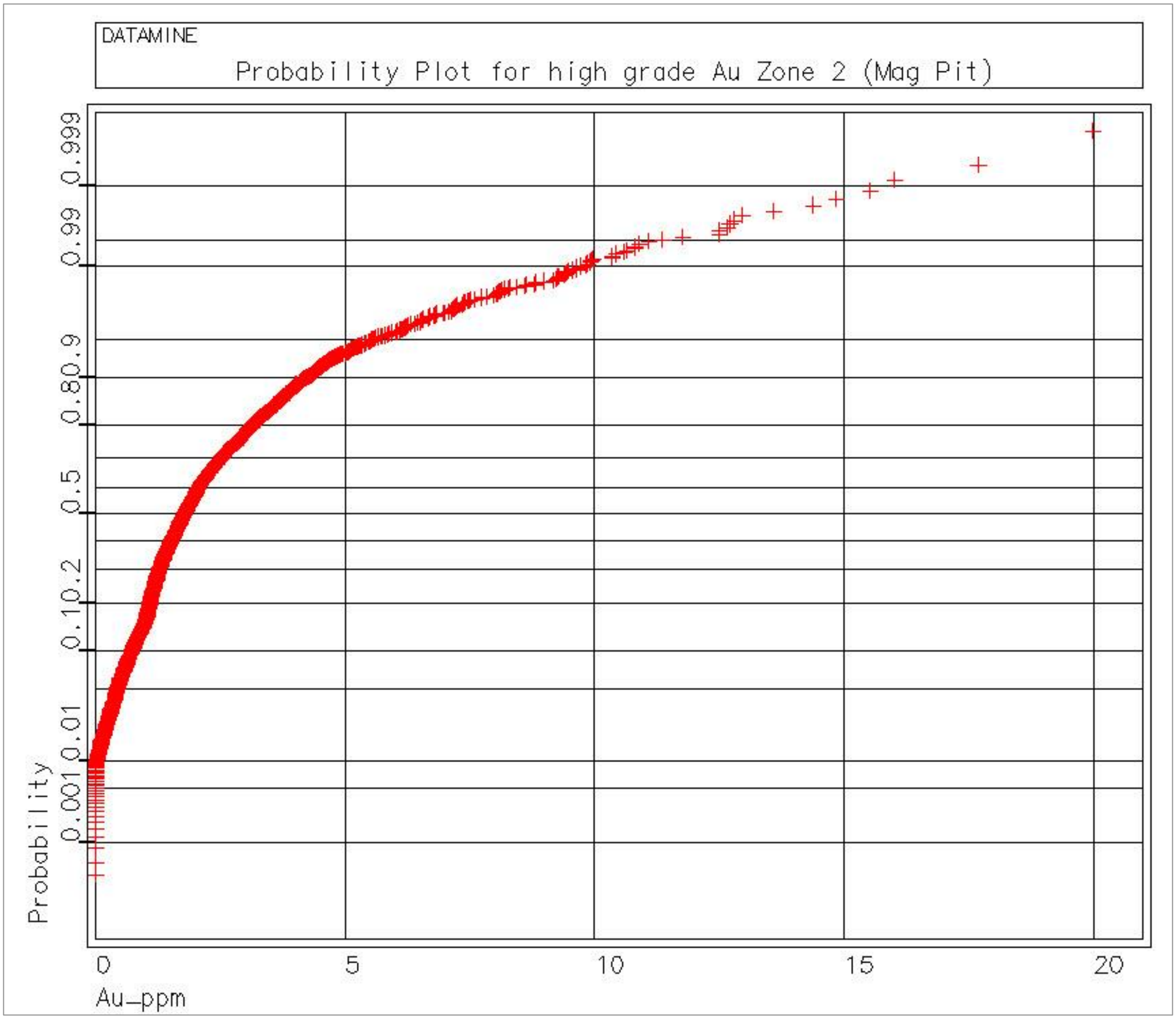
14.11.3 Capping and statistics

Probability plots of the grades for both high-grade and low-grade domains were viewed.

Capping was then assessed. Based on the high-grade probability plots capping was not considered necessary for all zones. The probability plot for the Mag Pit is show in Figure 14.16 below.

Capping is summarized in Table 14.20 and Table 14.21.

Figure 14.16 Probability plot of high grades for Mag Pit (Zone 2)

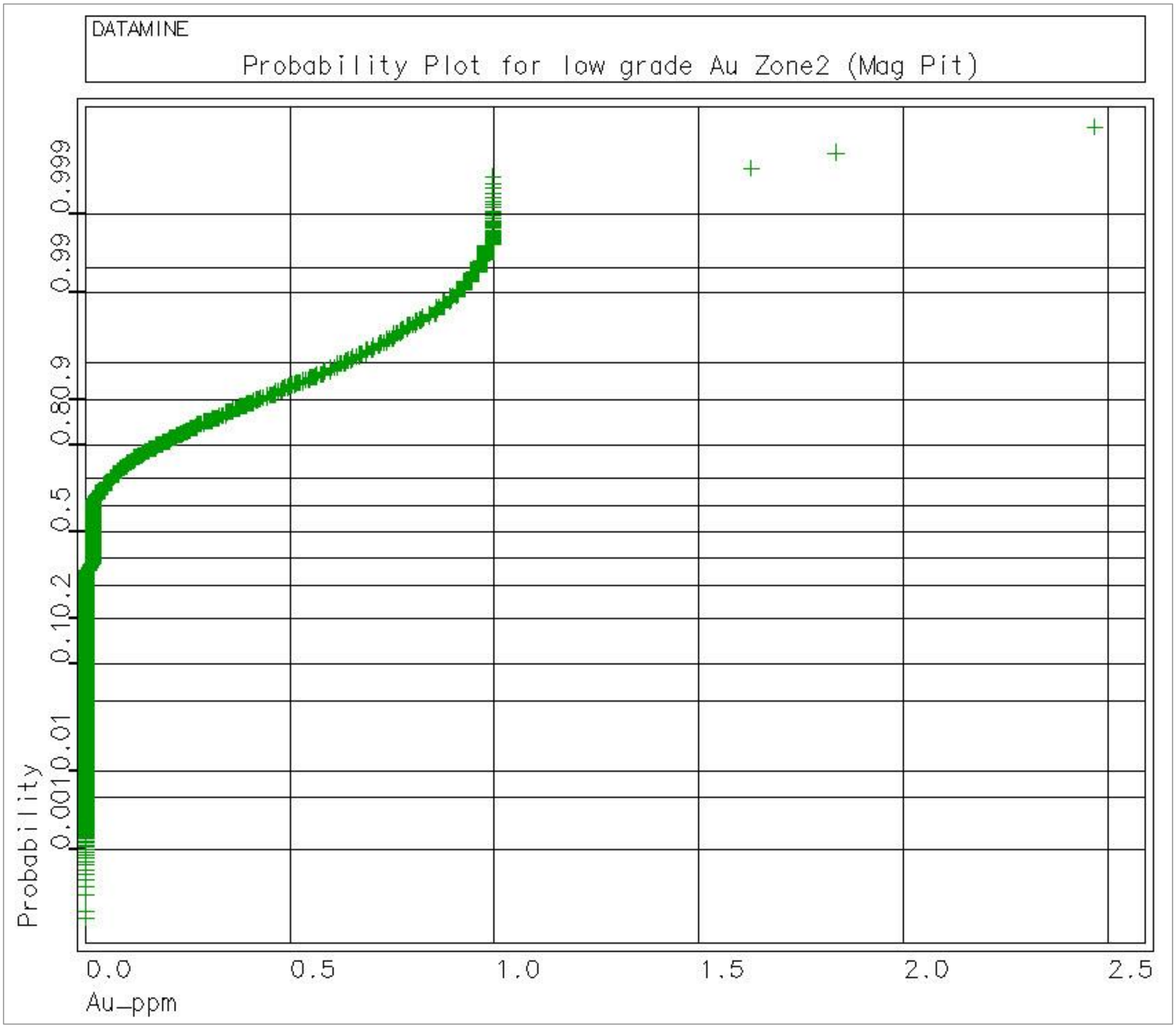


Source: AMC Mining Consultants (Canada) Ltd.

Based on the low-grade probability plot for the Mag Pit capping was applied at 1.22 g/t gold. The probability plot is show in Figure 14.17 below. This cap affects one sample.

Capping is summarized in Table 14.20 and Table 14.21.

Figure 14.17 Probability plot of low grades for Mag Pit (Zone 2)



Source: AMC Mining Consultants (Canada) Ltd.

Table 14.20 shows the statistics before and after compositing for Pit A (Zone 1), Pit B (Zone 4), and Mag Pit (Zone 2). Note that some zones did not require capping.

Table 14.20 Statistics of composites and capped data for Zones 1, 2, and 4

Zone / pit	Data	High-grade		Low-grade	
		Composites	Capped	Composites	Capped
	Field	Au (ppm)	Au (ppm)	Au (ppm)	Au (ppm)
Zone 1 (Pit A)	Nsamples	481	481	1,911	1,911
	Minimum	0.00	0.00	0.00	0.00
	Maximum	99.50	40.00	3.00	1.22
	Mean	4.88	4.65	0.10	0.10
	Standdev	8.46	6.57	0.21	0.19
	Coeff. of var.	1.73	1.41	2.16	1.99
Zone 2 (Mag Pit)	Nsamples	2,938	2,938	12,487	12,488
	Minimum	0.00	0.00	0.00	0.00
	Maximum	19.99	19.99	2.47	1.00
	Mean	2.27	2.27	0.11	0.11
	Standdev	1.85	1.85	0.21	0.21
	Coeff. of var.	0.82	0.82	1.85	1.84
Zone 4 (Pit B)	Nsamples	815	815	2,855	2,855
	Minimum	0.00	0.00	0.00	0.00
	Maximum	29.35	20.00	0.99	0.99
	Mean	2.25	2.23	0.12	0.12
	Standdev	2.25	2.13	0.21	0.21
	Coeff. of var.	1.00	0.95	1.68	1.68

Notes: Nsamples=number of samples, Standdev=standard deviation, Coeff. of var.=coefficient of variation.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.21 shows the statistics before and after compositing for the subzone of Pits C and CX (Subzones 3.1 to 3.10). Note that some subzones zones did not require capping.

Table 14.21 Statistics of composites and capped data for Pits C and CX (Zone 3)

Sub-zone	Data	High-grade		Low-grade	
		Composites	Capped	Composites	Capped
	Field	Au (ppm)	Au (ppm)	Au (ppm)	Au (ppm)
3.1	Nsamples	39	39	1,250	1,250
	Minimum	0.02	0.02	0.00	0.00
	Maximum	7.89	4.00	0.92	0.92
	Mean	1.93	1.63	0.04	0.04
	Standdev	1.88	1.17	0.10	0.10
	Coeff. of var.	0.97	0.72	2.25	2.25
3.2	Nsamples	74	74	1,604	1,604
	Minimum	0.02	0.02	0.00	0.00
	Maximum	32.91	20.00	2.21	1.00
	Mean	5.31	5.02	0.03	0.03
	Standdev	6.29	5.31	0.10	0.09
	Coeff. of var.	1.19	1.06	3.34	2.99
3.3	Nsamples	218	218	1,334	1,334
	Minimum	0.02	0.02	0.00	0.00
	Maximum	24.79	24.79	2.62	1.29
	Mean	3.39	3.39	0.10	0.09
	Standdev	4.26	4.26	0.20	0.18
	Coeff. of var.	1.26	1.26	2.05	1.92
3.4	Nsamples	101	101	1,612	1,612
	Minimum	0.03	0.03	0.00	0.00
	Maximum	22.11	10.00	0.98	0.98
	Mean	2.54	2.33	0.06	0.06
	Standdev	3.08	2.05	0.13	0.13
	Coeff. of var.	1.22	0.88	2.21	2.21
3.5	Nsamples	662	662	4,099	4,099
	Minimum	0.00	0.00	0.00	0.00
	Maximum	48.48	35.00	2.30	1.00
	Mean	4.01	3.99	0.09	0.08
	Standdev	5.06	4.91	0.17	0.16
	Coeff. of var.	1.26	1.23	1.97	1.94
3.6	Nsamples	97	97	2,235	2,235
	Minimum	0.00	0.00	0.00	0.00
	Maximum	19.87	19.87	0.99	0.99
	Mean	3.65	3.65	0.03	0.03
	Standdev	4.00	4.00	0.08	0.08
	Coeff. of var.	1.10	1.10	2.67	2.67
3.7	Nsamples	81	81	4,013	4,013
	Minimum	0.02	0.02	0.00	0.00
	Maximum	15.46	15.46	0.99	0.99
	Mean	2.38	2.38	0.02	0.02
	Standdev	2.83	2.83	0.07	0.07
	Coeff. of var.	1.19	1.19	2.70	2.70

Sub-zone	Data	High-grade		Low-grade	
		Composites	Capped	Composites	Capped
	Field	Au (ppm)	Au (ppm)	Au (ppm)	Au (ppm)
3.8	Nsamples	65	65	1,737	1,737
	Minimum	0.02	0.02	0.00	0.00
	Maximum	29.89	15.00	3.91	1.00
	Mean	4.01	3.78	0.04	0.04
	Standdev	4.74	3.71	0.15	0.11
	Coeff. of var.	1.18	0.98	3.53	2.69
3.9	Nsamples	5	5	555	555
	Minimum	0.96	0.96	0.00	0.00
	Maximum	1.38	1.38	0.69	0.69
	Mean	1.24	1.24	0.07	0.07
	Standdev	0.16	0.16	0.13	0.13
	Coeff. of var.	0.13	0.13	1.88	1.88
3.10	Nsamples	25	25	813	813
	Minimum	0.03	0.03	0.00	0.00
	Maximum	15.19	15.19	0.88	0.88
	Mean	3.68	3.68	0.02	0.02
	Standdev	4.17	4.17	0.06	0.06
	Coeff. of var.	1.13	1.13	2.44	2.44

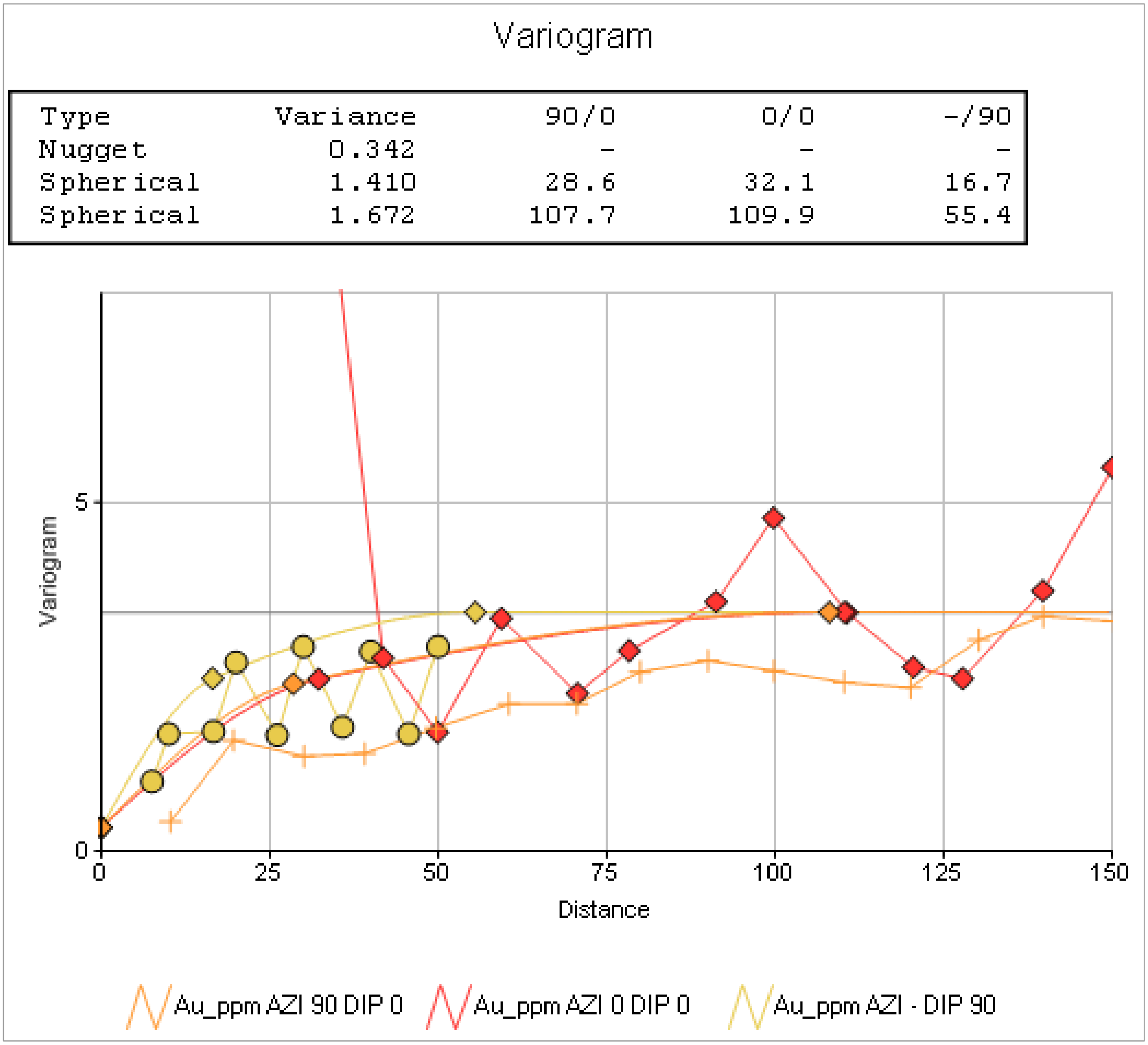
Notes: Nsamples=number of samples, Standdev=standard deviation, Coeff. of var.=coefficient of variation.
Source: AMC Mining Consultants (Canada) Ltd.

14.11.4 Variography

Variography was carried out for each open pit and for the high grade and low grade data sets.

The high-grade variogram model for the Mag Pit is shown in Figure 14.18.

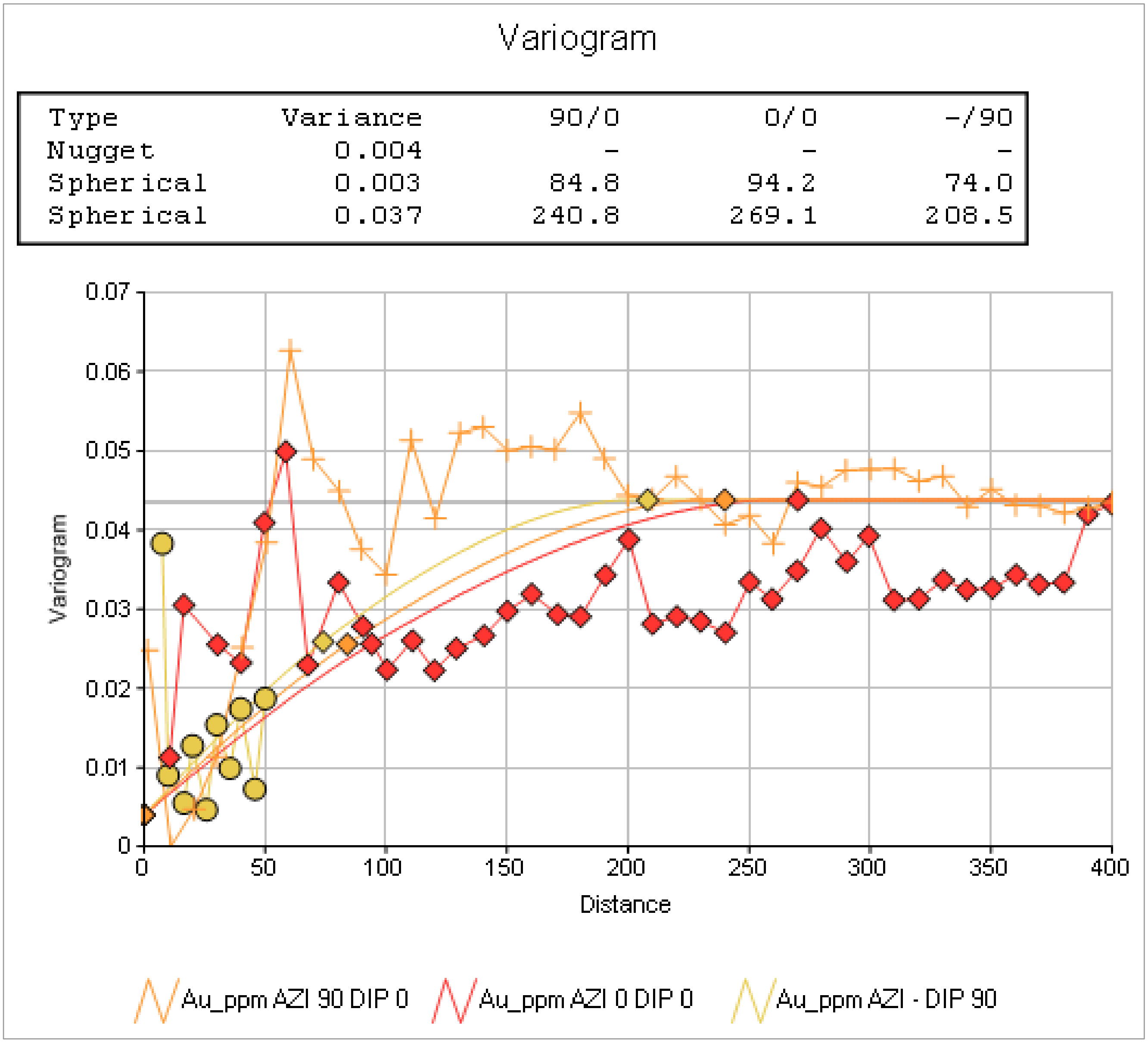
Figure 14.18 High-grade variogram for Mag Pit (Zone 2)



Source: AMC Mining Consultants (Canada) Ltd.

The low-grade variogram model for the Mag Pit is shown in Figure 14.19.

Figure 14.19 Low-grade variogram for Mag Pit (Zone 2)



Source: AMC Mining Consultants (Canada) Ltd.

Table 14.22 and Table 14.23 show the variogram parameters for the high-grade and low-grade gold domains, respectively. In both sets of tables, the first rotation is around the Z-axis and the second rotation is around the Y-axis.

Table 14.22 Variogram parameters – Au high-grade

Zone	1	2	3						4
Sub-zone	NA	NA	3.1	3.2	3.3	3.4	3.5	3.6	NA
Pit	A	MAG	C and CX						B
VANGLE1	57	150	27	75	55	35	40	10	15
VANGLE2	-50	50	-40	-50	-50	-50	-50	-50	-35
NUGGET	4.344	0.342	0.135	2.808	1.828	0.432	2.468	1.636	0.51
ST1PAR1	74.5	28.6	30.8	13.4	31.6	33	27.4	20.2	29
ST1PAR2	91.5	32.1	50.1	26.7	38.9	38.2	36.7	29	47.5
ST1PAR3	10.6	16.7	5.3	8	8.6	11.9	8.7	4	8.6
ST1PAR4	0.317	1.41	0.097	7.156	0.078	0.915	0.648	3.916	0.393
ST2PAR1	175.5	107.7	119.6	85.5	98.9	89.7	90.2	111.6	124.0
ST2PAR2	217.0	109.9	176.7	130.1	127.3	102.4	119.6	137.1	192.5
ST2PAR3	26.6	55.4	27.3	42.8	29.7	36.9	32.1	19.3	45.9
ST2PAR4	38.781	1.672	1.119	18.12	16.377	2.972	21.56	10.805	4.2

Notes: VANGLE1=Rotation angle 1, VANGLE2=Rotation angle 2, NUGGET=Nugget variance, ST1PAR1=Range of first structure (axis 1), ST1PAR2=Range of first structure (axis 2), ST1PAR3=Range of first structure (axis 3) ,ST1PAR4=Variance (first structure), ST2PAR1=Range of second structure (axis 1), ST2PAR2=Range of second structure (axis 2), ST2PAR3=Range of second structure (axis 3), ST2PAR4=Variance (second structure).
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.23 Variogram parameters – Au low-grade

Zone	1	2	3						4
Sub-zone	NA	NA	3.1	3.2	3.3	3.4	3.5	3.6	NA
Pit	A	MAG	C and CX						B
VANGLE1	57	150	27	75	55	35	40	10	15
VANGLE2	-50	50	-40	-50	-50	-50	-50	-50	-35
NUGGET	0.008	0.004	0.001	0.001	0.003	0.002	0.003	0.001	0.004
ST1PAR1	60.1	94.2	13.5	24.5	20.2	31.6	11.4	15	33.6
ST1PAR2	65	94.2	18.8	33.4	27.6	31.6	15.5	19.1	33.6
ST1PAR3	42.6	74	8.1	12.3	11.4	14.8	7.4	11.6	14.8
ST1PAR4	0.006	0	0.003	0.002	0.009	0.006	0.008	0	0.003
ST2PAR1	195.6	240.8	70.0	129.5	100.0	103.6	84.8	115.9	160.1
ST2PAR2	196.2	269.1	93.3	135.0	139.9	129.1	107.0	122.7	213.9
ST2PAR3	111.5	208.5	35.9	61.4	61.2	47.1	53.1	84.5	121.1
ST2PAR4	0.022	0.039	0.006	0.005	0.018	0.008	0.019	0.006	0.036

Notes: VANGLE1=Rotation angle 1, VANGLE2=Rotation angle 2, NUGGET=Nugget variance, ST1PAR1=Range of first structure (axis 1), ST1PAR2=Range of first structure (axis 2), ST1PAR3=Range of first structure (axis 3) ,ST1PAR4=Variance (first structure), ST2PAR1=Range of second structure (axis 1), ST2PAR2=Range of second structure (axis 2), ST2PAR3=Range of second structure (axis 3), ST2PAR4=Variance (second structure).
Source: AMC Mining Consultants (Canada) Ltd.

14.12 Block model

14.12.1 Block model parameters open pit

The parent block size for the models was 25 ft by 25 ft by 10 ft with block splitting employed. Block splitting resulted in minimum cell dimensions of 2.5 ft by 2.5 ft in the X and Y direction.

The smallest Z direction is 1 ft. The block model is not rotated. The block model extents are shown in Table 14.24.

Table 14.24 Block model parameters for open pit high and low-grade models

Parameter	X	Y	Z
Origin (ft)	6,800	7,600	2,800
Parent block size (ft)	25	25	10
Minimum block size (ft)	2.5	2.5	1.0
Number of blocks	248	196	300

Source: AMC Mining Consultants (Canada) Ltd.

14.12.2 Grade estimation of high and low-grade models

To estimate into the high and low-grade models, OK was used in Zones 1, 2, and 4. For Pit C and CX (Zone 3), when there was not enough data to generate a good variogram, ID^2 was used instead.

The dimensions of the search radius for the high-grade zones and subzones are shown in Table 14.25.

A number of passes were employed, each using different search distances and multiples as follows:

- Pass 1 = 1 x search distance
- Pass 2 = 2 x search distance
- Pass 3 = 3 x search distance

The search parameters for high-grade gold are shown in Table 14.25. In all cases, the axis for the first rotation is Z and the second axis of rotation is Y. Note, there was no rotation of the search ellipse along the X axis. For all passes the minimum number of samples is 4 and the maximum number of samples is 12. The minimum number of drillholes to inform a block is two.

Table 14.25 Search parameters for high-grade gold

Zone	Sub-zone	Pit	Max search distance in direction 1	Max search distance in direction 2	Max search distance in direction 3	1st rotation angle for search volume	2nd rotation angle for search volume
1	NA	A	175	217	27	57	-50
2	NA	MAG	110	110	55	150	50
3	3.1	C and CX	120	176	27	27	-40
	3.2		86	130	43	75	-50
	3.3		100	127	30	55	-50
	3.4		90	102	37	35	-50
	3.5		90	120	32	40	-50
	3.6		112	137	20	10	-50
	3.7		80	130	30	50	-60
	3.8		80	130	30	35	-50
	3.9		80	130	30	160	-60
	3.10		80	130	30	15	-35
4		B	124	193	46	15	-35

Notes: The axis for the first rotation is Z and the second axis of rotation is Y. Rotation is in a clockwise direction when positive and counterclockwise when negative.
Source: AMC Mining Consultants (Canada) Ltd.

The search parameters for low-grade gold are shown in Table 14.26. As previously, the axis for the first rotation is Z and the second axis of rotation is Y and there was no rotation of the search ellipse along the X axis.

For all passes the minimum number of samples is 4 and the maximum number of samples is 12. The minimum number of drillholes to inform a block is two.

Table 14.26 Search parameters for low-grade gold

Zone	Sub-zone	Pit	Max search distance in direction 1	Max search distance in direction 2	Max search distance in direction 3	1st rotation angle for search volume	2nd rotation angle for search volume
1	NA	A	195	196	111	57	-50
2	NA	MAG	240	270	200	150	50
3	3.1	C and CX	70	94	36	27	-40
	3.2		130	135	61	75	-50
	3.3		100	140	61	55	-50
	3.4		103	129	47	35	-50
	3.5		85	107	53	40	-50
	3.6		116	123	85	10	-50
	3.7		85	110	50	50	-60
	3.8		85	110	50	35	-50
	3.9		85	110	50	160	-60
	3.10		85	110	50	15	-35
4	NA	B	160	214	121	15	-35

Notes: The axis for the first rotation is Z and the second axis of rotation is Y. Rotation is in a clockwise direction when positive and counterclockwise when negative.
Source: AMC Mining Consultants (Canada) Ltd.

14.12.3 Mineral Resource classification

Classification was carried out using data support as the main criteria. An estimation was run with the search parameters shown in Table 14.27 to roughly outline Measured, Indicated, and Inferred. These results where then used to manually generate contiguous 3D wireframes defining the Measured, Indicated, and Inferred Resource categories. These wireframes were then used to code the block model.

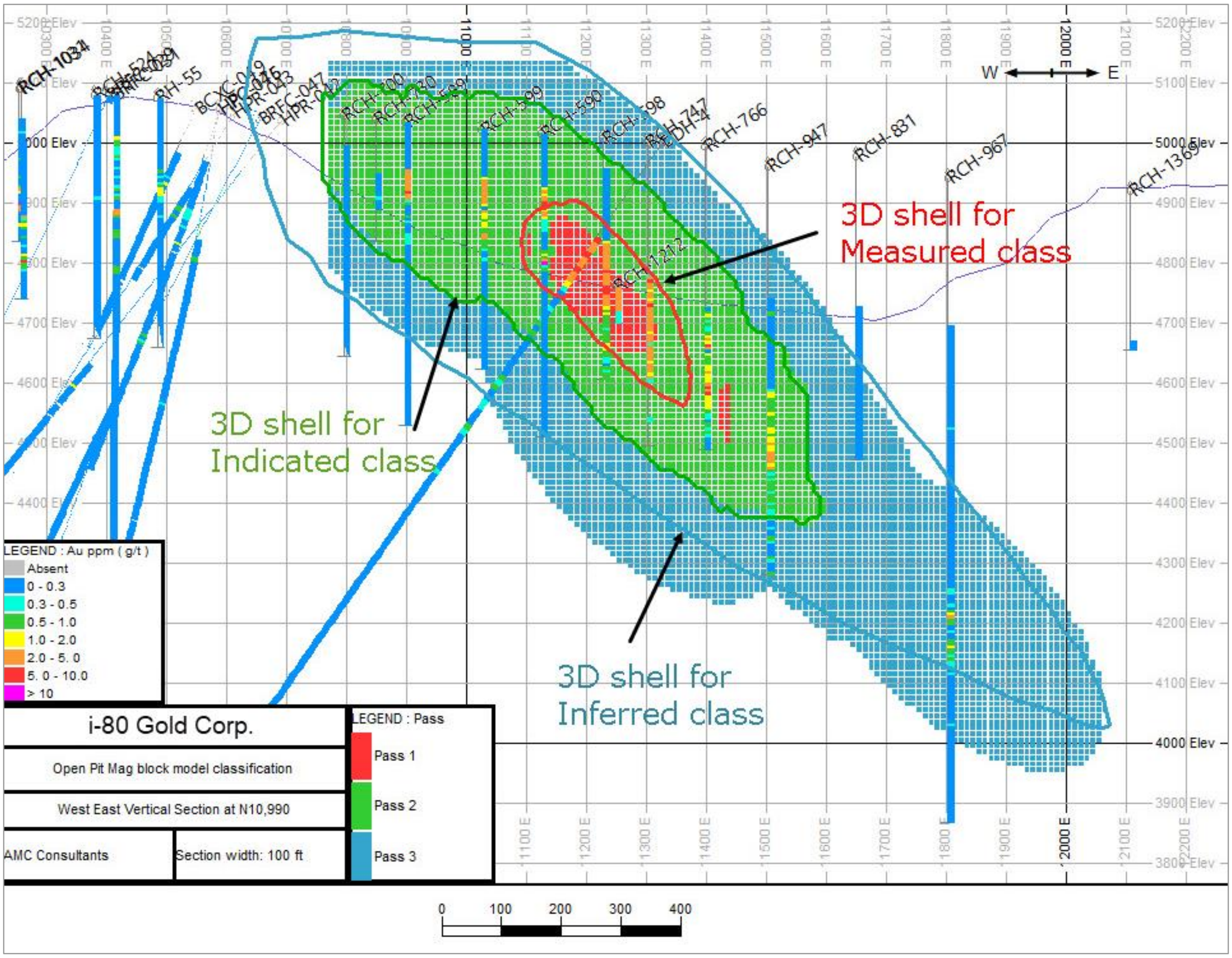
Table 14.27 Search parameters for classification

Pit	Pass	Search distance (ft) on X axis	Search distance (ft) on Y axis	Search distance (ft) on Z axis	Z-Axis rotation (degrees)	Y-Axis rotation (degrees)	Minimum # of composites	Maximum # of composites	Minimum # of drillholes
Pit A	1	90	130	20	57	-53	8	22	4
	2	130	220	100	150	50	6	20	3
	3	20	275	125	150	50	2	16	1
Pit B	1	110	110	50	15	-35	8	30	4
	2	220	220	100	15	-35	6	24	3
	3	275	275	125	15	-35	4	20	2
Pit C and CX	1	110	110	50	35	-50	8	24	4
	2	220	220	100	150	50	6	24	3
	3	275	275	125	150	50	4	20	2
Mag Pit	1	110	110	50	150	50	8	20	4
	2	220	220	100	150	50	6	20	3
	3	330	330	150	150	50	4	20	2

Note: Pass 1 roughly defines Measured, Pass 2 roughly defines Indicated, Pass 3 roughly defines Inferred.
Source: AMC Mining Consultants (Canada) Ltd.

As an example, the Mineral Resource classification for the Mag Pit is presented in Figure 14.20 where red is Measured Mineral Resources, green is Indicated Mineral Resources and blue is Inferred Mineral Resources.

Figure 14.20 Example of Mineral Resource classification of Mag Pit



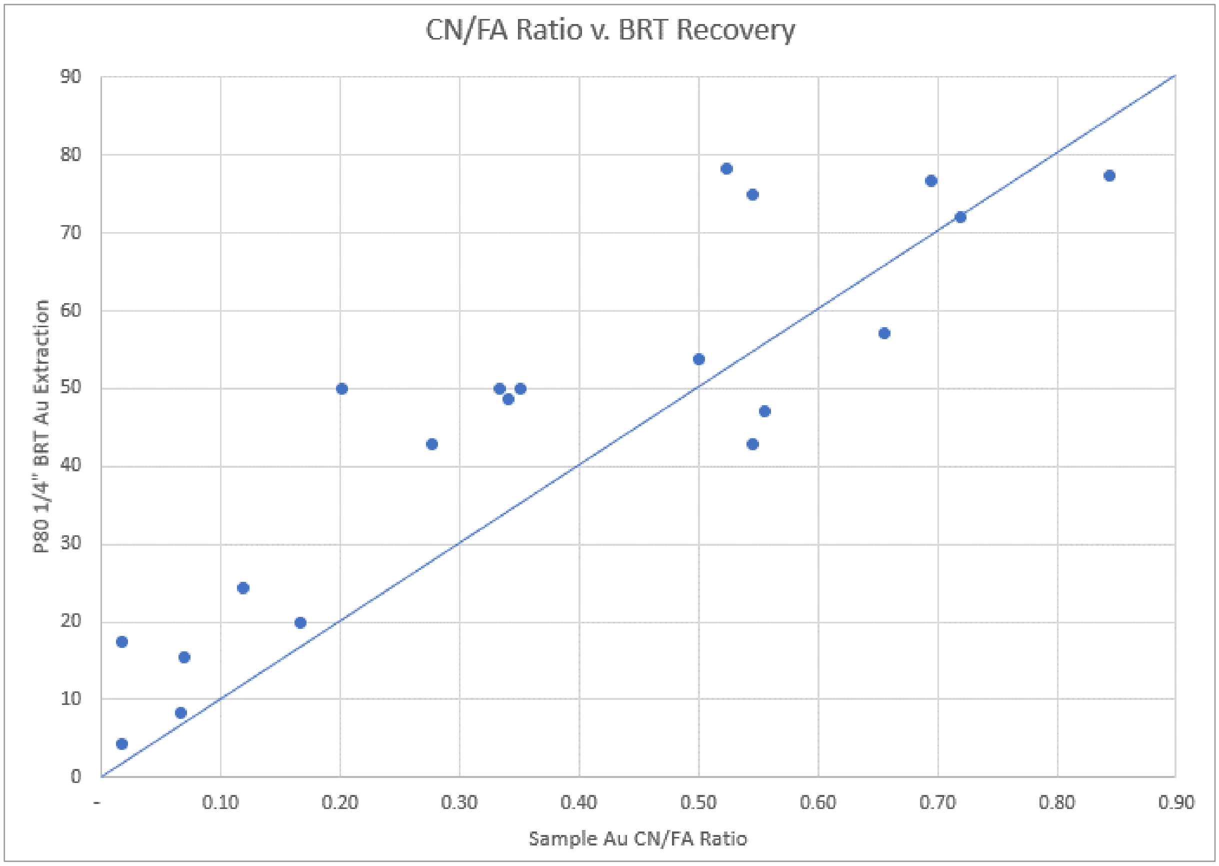
Source: AMC Mining Consultants (Canada) Ltd. 2019

14.12.4 Recovery assumptions

As discussed in Section 13, McClelland Laboratories completed a metallurgical testwork program on Mag Pit samples on behalf of Atna in 2013 / 2014. This included cyanide leach bottle roll tests. A summary of the bottle roll tests is shown in Table 13.8. McClelland postulated in their report that samples with low recoveries were most likely due to refractory gold in sulphide minerals or preg-robbing.

Figure 14.21 shows the relationship between cyanide leach bottle roll and the ratio of the cyanide leach assay to the fire assay (CN/FA).

Figure 14.21 Gold recovery versus CN / FA assay ratios



Source: OMC, July 2020.

To estimate gold recovery, OMC used the available cyanide leach assay / fire assay ratios to build shapes in LeapFrog. The high recovery shapes were based on ratios > 0.6 CN/FA and low recovery shapes were based on ratios of 0.6 to 0.2 ratios. These shapes were reviewed by the QP and found to be acceptable. Based on these shapes a gold recovery of 80% was applied to the high recovery shapes, 40% recovery was applied to the low recovery shapes and a recovery of 0% was applied to areas outside of these shapes.

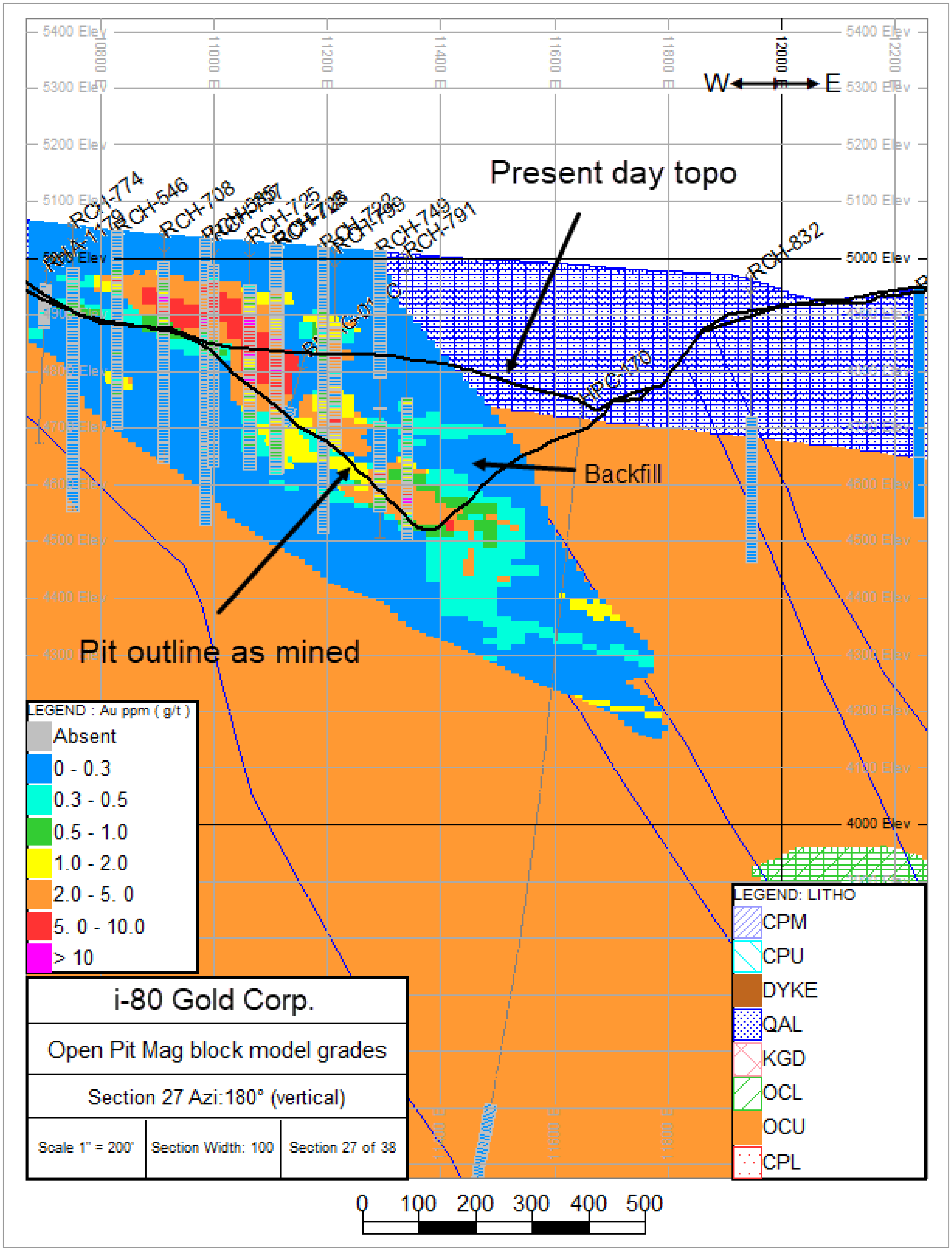
14.13 Validation of Open Pit models

The block models were validated in four ways. First visual checks were carried out to ensure that the grades respected the raw assay data and also lay within the constraining wireframes. Secondly, swath plots were reviewed. Thirdly the estimate was statistically compared to the final (composited) assay data. Fourth, the block model was compared to historical mining.

The block model validation for Mag Pit is shown below as an example.

An example of the drillhole composite gold grades compared to the block model estimated grades is shown in Figure 14.22. The figure shows good agreement between the drillhole composite grades and estimated block model grades.

Figure 14.22 Vertical section showing BM and drillhole grades for Mag Pit (Zone 2)



Source: AMC Mining Consultants (Canada) Ltd. 2019.

Table 14.28 shows the statistical comparison of the composites versus the block model grades for Zone 1 (Pit A), Zone 2 (Mag Pit), and Zone 4 (Pit B). Table 14.29 shows the statistical comparison of the composites versus the block model grades for gold for Zone 3 (CX and C Pits).

Table 14.28 Statistics of composites and block model of Zones 1, 2, and 4

Zone	Data	Data	Composites	Block model	
		Field	Au_ppm	AU_OK	AU_ID^2
Zone 1 (Pit A)	High-grade	Nsamples	481	57,372	57,372
		Minimum	0	0	0.15
		Maximum	40	27.58	30.17
		Mean	4.65	4.14	4.57
		Standdev	6.57	3.42	3.59
		Coeff. of var.	1.41	0.83	0.79
	Low-grade	Nsamples	1,911	830,076	830,076
		Minimum	0	0	0
		Maximum	1.22	0.81	0.92
		Mean	0.1	0.09	0.09
		Standdev	0.19	0.11	0.12
		Coeff. of var.	1.99	1.28	1.31
Zone 2 (Pit Mag)	High-grade	Nsamples	2,938	589,883	589,883
		Minimum	0	0.12	0.12
		Maximum	19.99	9.84	11.28
		Mean	2.27	1.94	2
		Standdev	1.85	0.86	0.91
		Coeff. of var.	0.82	0.44	0.45
	Low-grade	Nsamples	12,488	10,975,581	10,975,581
		Minimum	0	0	0
		Maximum	1	0.94	0.99
		Mean	0.11	0.1	0.1
		Standdev	0.21	0.15	0.15
		Coeff. of var.	1.84	1.44	1.46
Zone 4 (Pit B)	High-grade	Nsamples	815	213,469	213,469
		Minimum	0	0	0.36
		Maximum	20	10.01	10.19
		Mean	2.23	2.16	2.34
		Standdev	2.13	0.91	1.01
		Coeff. of var.	0.95	0.42	0.43
	Low-grade	Nsamples	2,855	1,943,012	1,943,012
		Minimum	0	0	0
		Maximum	0.99	0.89	0.92
		Mean	0.12	0.12	0.12
		Standdev	0.21	0.13	0.12
		Coeff. of var.	1.68	1.07	1.06

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.29 Statistics of composites and block model of Zone 3

Sub-zone	Data	High-grade			Low-grade		
		Composites	Block model		Composites	Block model	
	Field	Au_ppm	AU_OK	AU_ID²	Au_ppm	AU_OK	AU_ID²
3.1	Nsamples	39	6,246	6,246	1,250	420,740	420,740
	Minimum	0.02	0.52	0.64	0	0	0
	Maximum	4	3.61	3.74	0.92	0.58	0.59
	Mean	1.63	1.95	1.95	0.04	0.04	0.04
	Standdev	1.17	0.58	0.56	0.1	0.05	0.05
	Coeff. of var.	0.72	0.3	0.29	2.25	1.27	1.36
3.2	Nsamples	74	11,617	11,617	1,604	1,349,355	1,349,355
	Minimum	0.02	1.04	0.83	0	0	0
	Maximum	20	14.09	13.6	1	0.48	0.62
	Mean	5.02	4.44	4.79	0.03	0.02	0.02
	Standdev	5.31	2.13	2.38	0.09	0.03	0.04
	Coeff. of var.	1.06	0.48	0.5	2.99	1.34	1.53
3.3	Nsamples	218	29,536	29,536	1,334	552,834	552,834
	Minimum	0.02	0.27	0.47	0	0	0
	Maximum	24.79	15.9	16.17	1.29	0.78	1.01
	Mean	3.39	3.49	3.65	0.09	0.08	0.08
	Standdev	4.26	2.06	2.16	0.18	0.09	0.1
	Coeff. of var.	1.26	0.59	0.59	1.92	1.16	1.23
3.4	Nsamples	101	13,957	13,957	1,612	956,201	956,201
	Minimum	0.03	0.63	0.51	0	0	0
	Maximum	10	6.43	6.42	0.98	0.63	0.7
	Mean	2.33	2.27	2.32	0.06	0.04	0.04
	Standdev	2.05	0.78	0.89	0.13	0.06	0.06
	Coeff. of var.	0.88	0.34	0.39	2.21	1.33	1.52
3.5	Nsamples	662	160,230	160,230	4,099	1,734,811	1,734,811
	Minimum	0	0.3	0.43	0	0	0
	Maximum	35	25.17	23.43	1	0.8	0.87
	Mean	3.99	4.22	4.4	0.08	0.09	0.09
	Standdev	4.91	2.72	2.9	0.16	0.11	0.11
	Coeff. of var.	1.23	0.65	0.66	1.94	1.2	1.3
3.6	Nsamples	97	NA	11,915	2,235	NA	918,959
	Minimum	0		0.79	0		0
	Maximum	19.87		13.3	0.99		0.64
	Mean	3.65		3.71	0.03		0.03
	Standdev	4		1.62	0.08		0.05
	Coeff. of var.	1.1		0.44	2.67		1.69
3.7	Nsamples	81	NA	13,907	4,013	NA	1,661,297
	Minimum	0.02		0.39	0		0
	Maximum	15.46		9.13	0.99		0.62
	Mean	2.38		2.56	0.02		0.03
	Standdev	2.83		1.54	0.07		0.04
	Coeff. of var.	1.19		0.6	2.7		1.54

Sub-zone	Data	High-grade			Low-grade		
		Composites	Block model		Composites	Block model	
	Field	Au_ppm	AU_OK	AU_ID²	Au_ppm	AU_OK	AU_ID²
3.8	Nsamples	65	NA	10,702	1,737	NA	1,524,452
	Minimum	0.02		0.89	0		0
	Maximum	15		11.28	1		0.72
	Mean	3.78		5.58	0.04		0.04
	Standdev	3.71		2.4	0.11		0.06
	Coeff. of var.	0.98		0.43	2.69		1.49
3.9	Nsamples	5	NA	2	555	NA	309,634
	Minimum	0.96		1.07	0		0
	Maximum	1.38		1.07	0.69		0.58
	Mean	1.24		1.07	0.07		0.05
	Standdev	0.16		0	0.13		0.07
	Coeff. of var.	0.13		0	1.88		1.52
3.1	Nsamples	25	NA	5,079	813	NA	912,723
	Minimum	0.03		0.76	0		0
	Maximum	15.19		8.8	0.88		0.49
	Mean	3.68		4.14	0.02		0.02
	Standdev	4.17		1.58	0.06		0.03
	Coeff. of var.	1.13		0.38	2.44		1.34

Source: AMC Mining Consultants (Canada) Ltd.

As a final check on the Open Pit area block models, the QP reported out of the block models at a 0.010 opt Au COG in order to compare to historical production data as shown in Table 14.30. The comparison of the total ounces shows a reasonable match between historical production and the estimated AMC model for the mined out volume. Local differences may be due to incomplete record keeping at the time. The past production may also not account for stockpiles.

Table 14.30 Comparison of AMC "mined-out" pits to historical gold production

Area	Golder Associates 2014			AMC 2020			Difference in %
	Gold produced (troy oz)			Estimated mined out blocks at Au 0.010 opt			
Deposit	Mill feed	Leach feed	Total feed	Short tons	Gold grade (opt)	Contained gold (oz)	Estimated vs gold in feed
A	369,753	83,469	453,222	1,782,402	0.142	252,310	
B	Included to above			4,965,964	0.058	259,064	
Subtotal A+B	**369,753**	**83,469**	**453,222**	**6,218,059**	**0.082**	**511,374**	**13%**
C and CX	98,686	33,884	132,570	2,455,773	0.098	240,762	82%
Mag	301,255	59741	360,996	9,736,299	0.064	624,742	73%
Total	**769,694**	**177,094**	**946,788**	**18,410,131**	**0.074789**	**1,376,878**	**45%**

Notes: Source Golder Associates 2014: modified from Table 6-1 of Golder Associates 2014.
Source AMC 2020: AMC Mining Consultants (Canada) Ltd.

14.14 Open Pit Mineral Resource statement

The Companies provided the initial COG calculations and the QP verified the reasonableness of the assumptions. The input parameters are based on actual and benchmark cost data for similar scale of operations and assumptions regarding mineral processing metal recoveries and metal prices. Operating costs for the open pit include mining, processing and G&A. Metal price used for gold is $1,550/oz and mineral processing recovery is assumed to be 80% for the high recovery zone and 40% for the low recovery zone. Varying royalties are applied at varying trigger points throughout the mine life, for simplicity a constant 6% royalty has been used for the calculation of COG. As discussed above, the open pit block model was coded into two recovery zones. The following COG selected to report Mineral Resources are higher than the calculated COG. The low recovery zone COG is 0.014 opt Au. The high recovery zone COG is 0.007 opt Au.

Further details are shown in Table 14.31.

Table 14.31 Inputs into open pit COG calculations

Input parameters	Items	Unit	High recovery zone	Low recovery zone
Feed mining costs	Total feed mining costs	$/t feed mined	2	2
Waste mining costs	Total waste mining costs	$/t mined	2	2
Feed costs (P costs)	Processing cost crusher	$/t feed mined	5	5
	G&A	$/t feed mined	1	1
	Total feed costs	$/t feed mined	6	6
Processing parameters	Gold metallurgical recovery	%	80.0%	40.0%
Net revenue gold	Gold price	$/oz	1,550	1,550
	Selling costs and penalties	$/oz	5	5
	Payable gold	%	99.85%	99.85%
Royalty	Total royalty	%	6.0%	6.0%
Slope angles	40 in Mag North, 45 in Mag South, 45 in CX	2008 Assessment by Golder, assuming pit dewatered and full to partial depressurization		

The Mineral Resource estimation for gold grades was estimated in g/t as this matched the drillhole database. Spatially, data was provided in local grid. The Mineral Resource are reported in in imperial units using the conversion factors outlined in Table 24.2.

Table 14.32 shows a summary of the Open Pit Mineral Resources.

Table 14.32 Summary of Open Pit Mineral Resource as of 23 July 2020

Classification	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Measured	10,726	0.068	730
Indicated	11,829	0.046	545
Measured and Indicated	**22,554**	**0.057**	**1,275**
Inferred	1,388	0.047	65

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- Mineral Resources are constrained by an optimized pit shell developed at a metal price of US$1,550/oz Au (cost and other assumptions shown in Table 14.31).
- Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone OCG is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.
- Mineral Resources shown on a 100% basis.

Source: AMC Mining Consultants (Canada) Ltd.

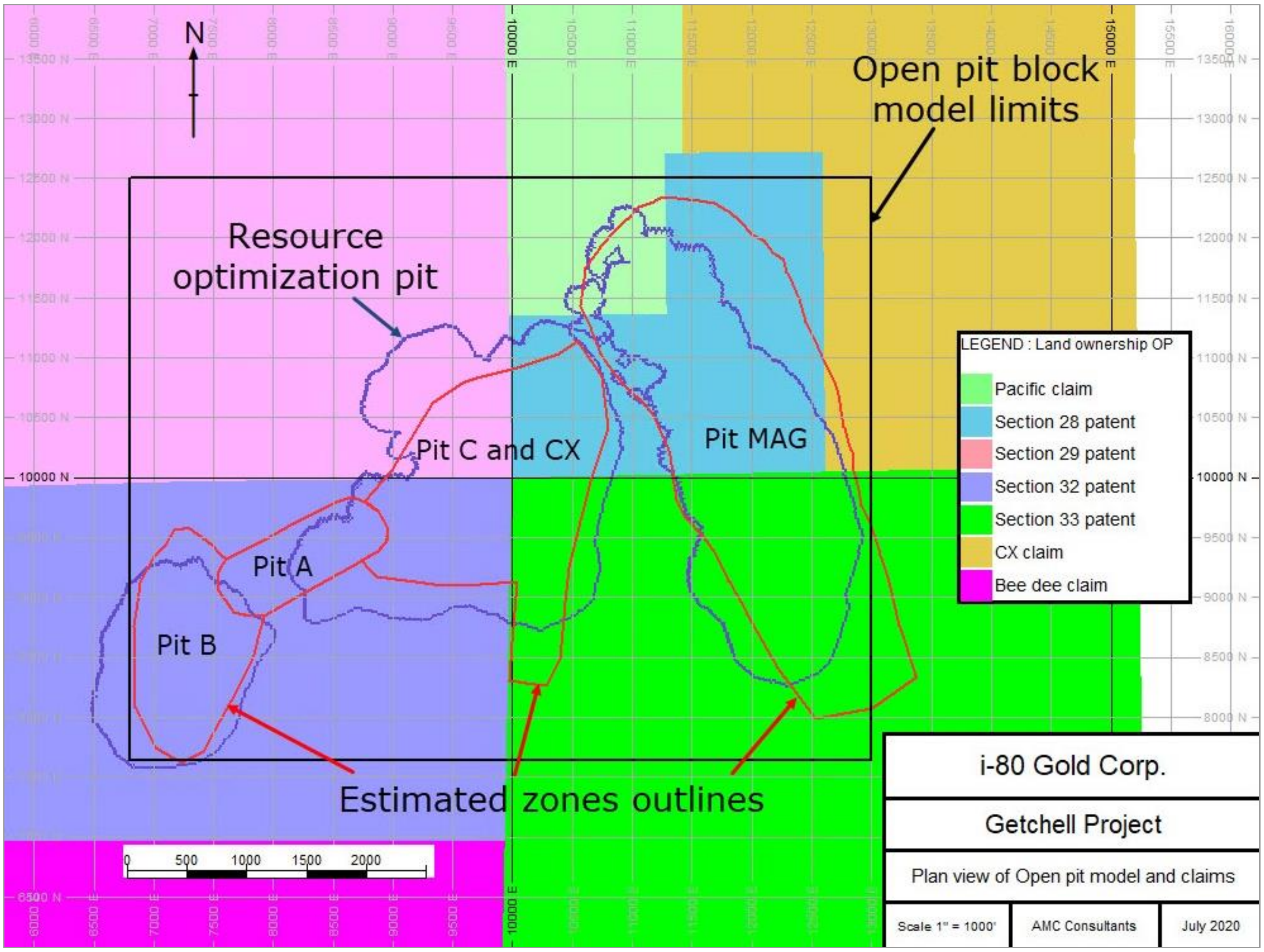
A summary of the Open Pit area Mineral Resource estimates by metallurgical recovery is shown in Table 14.33.

Table 14.33 Summary of Open Pit area Mineral Resource by recovery as of 23 July 2020

Classification	Recovery	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Measured and Indicated	High	13,883	0.059	803
Measured and Indicated	Low	8,671	0.054	472
Measured and Indicated	**Combined**	**22,554**	**0.057**	**1,275**

As the percentage of claim ownership varies, Figure 14.23 shows a plan view of the limits of the Open Pit area block model and the claim boundaries. Outlines of the historical pits are shown for reference.

Figure 14.23 Plan view of Open Pit block model, pit outlines, and claims



Notes: Estimation zones were based around the mined pits.
Source: AMC Mining Consultants (Canada) Ltd.

A summary of the Open Pit Mineral Resource estimates by claim is shown in Table 14.34.

Table 14.34 Pinson Open Pit area Mineral Resources by claim as of 23 July 2020

Claim	Classification	Tonnage (ktons)	Au (opt)	Metal Au (koz)
Pacific claim	Measured	187	0.052	10
	Indicated	336	0.031	10
	Measured and Indicated	**523**	**0.038**	**20**
	Inferred	140	0.012	2
Section 28 patent	Measured	2,188	0.065	142
	Indicated	3,240	0.045	146
	Measured and Indicated	**5,428**	**0.053**	**287**
	Inferred	178	0.043	8
Section 29 patent	Measured	853	0.095	81
	Indicated	879	0.037	32
	Measured and Indicated	**1,732**	**0.066**	**114**
	Inferred	234	0.053	12
Section 32 patent	Measured	3,586	0.074	267
	Indicated	3,402	0.062	211
	Measured and Indicated	**6,988**	**0.068**	**478**
	Inferred	272	0.046	13
Section 33 patent	Measured	3,912	0.059	231
	Indicated	3,971	0.036	145
	Measured and Indicated	**7,883**	**0.048**	**376**
	Inferred	564	0.055	31
Total	Measured	10,726	0.068	730
	Indicated	11,829	0.046	545
	Measured and Indicated	**22,554**	**0.057**	**1,275**
	Inferred	1,388	0.047	65

Notes:
- OMC has 41.67% ownership of Section 28 and Section 32 patents.
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Ms D. Nussipakynova, P.Geo., of AMC takes responsibility for the Mineral Resources.
- Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone OCG is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.35 through Table 14.41 show the Open Pit area Mineral Resource estimates for Zone 1 (Pit A), Zone 4 (Pit B), Zone 3 (Pit C and CX), and Zone 2 (Mag Pit) respectively at a range of COGs. The preferred COG is in bold.

Note there is no low recovery category for Pit A.

Table 14.35 Open Pit area Mineral Resource estimate at a range of cut-offs Pit A – high recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**
0.005	-	-	-	421,681	0.082	34,684	421,681	0.082	34,684	133,018	0.025	3,377
0.007	-	-	-	**358,683**	**0.096**	**34,298**	**358,683**	**0.096**	**34,298**	**110,075**	**0.029**	**3,243**
0.010	-	-	-	315,081	0.108	33,943	315,081	0.108	33,943	81,558	0.037	3,023
0.014	-	-	-	305,009	0.111	33,833	305,009	0.111	33,833	48,629	0.055	2,654
0.020	-	-	-	304,183	0.111	33,800	304,183	0.111	33,800	44,234	0.058	2,581
0.030	-	-	-	302,103	0.112	33,726	302,103	0.112	33,726	43,282	0.059	2,550
0.040	-	-	-	273,948	0.118	32,378	273,948	0.118	32,378	28,444	0.070	2,002
0.050	-	-	-	241,636	0.126	30,428	241,636	0.126	30,428	17,617	0.084	1,475
0.060	-	-	-	210,865	0.134	28,240	210,865	0.134	28,240	14,555	0.088	1,287
0.070	-	-	-	181,010	0.143	25,868	181,010	0.143	25,868	10,001	0.102	1,016
0.080	-	-	-	143,534	0.157	22,558	143,534	0.157	22,558	5,853	0.121	708
0.090	-	-	-	111,293	0.174	19,352	111,293	0.174	19,352	3,935	0.147	580
0.100	-	-	-	94,919	0.185	17,587	94,919	0.185	17,587	3,673	0.152	559

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.36 Open Pit area Mineral Resource estimate at range of cut-off Pit B – high recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**
0.005	1,920,309	0.038	73,401	1,931,409	0.033	63,367	3,851,719	0.036	136,767	1,806	0.046	84
0.007	**1,741,051**	**0.042**	**72,368**	**1,506,174**	**0.041**	**61,026**	**3,247,225**	**0.041**	**133,394**	**1,543**	**0.053**	**82**
0.010	1,559,429	0.045	70,877	1,160,021	0.050	58,328	2,719,449	0.048	129,205	1,385	0.058	81
0.014	1,327,314	0.051	68,141	865,839	0.064	55,113	2,193,153	0.056	123,254	1,385	0.058	81
0.020	1,155,277	0.057	65,412	762,253	0.070	53,528	1,917,530	0.062	118,940	1,385	0.058	81
0.030	1,102,824	0.058	63,978	708,058	0.073	52,021	1,810,882	0.064	115,998	1,385	0.058	81
0.040	802,501	0.065	52,409	543,264	0.084	45,552	1,345,765	0.073	97,961	1,385	0.058	81
0.050	476,464	0.078	36,948	407,773	0.096	39,159	884,237	0.086	76,107	411	0.058	24
0.060	272,587	0.094	25,653	332,392	0.104	34,483	604,980	0.099	60,136	-	-	-
0.070	175,946	0.109	19,248	261,605	0.112	29,184	437,551	0.111	48,431	-	-	-
0.080	139,037	0.119	16,511	208,233	0.119	24,802	347,270	0.119	41,313	-	-	-
0.090	106,834	0.128	13,694	162,101	0.126	20,380	268,935	0.127	34,074	-	-	-
0.100	85,060	0.137	11,616	123,971	0.131	16,211	209,030	0.133	27,827	-	-	-

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.37 Open Pit area Mineral Resource estimate at range of cut-off Pit B – low recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**
0.007	237,239	0.061	14,369	1,113,421	0.049	54,131	1,350,660	0.051	68,500	95,461	0.039	3,682
0.010	212,268	0.067	14,168	831,502	0.062	51,734	1,043,770	0.063	65,902	52,066	0.064	3,321
0.014	**199,042**	**0.070**	**14,004**	**683,055**	**0.073**	**50,021**	**882,097**	**0.073**	**64,025**	**47,357**	**0.069**	**3,270**
0.020	187,101	0.074	13,786	651,138	0.076	49,506	838,239	0.076	63,291	46,893	0.070	3,265
0.030	178,599	0.076	13,546	617,455	0.078	48,434	796,054	0.078	61,980	46,893	0.070	3,265
0.040	153,650	0.082	12,560	547,994	0.084	45,809	701,644	0.083	58,369	41,992	0.073	3,055
0.050	132,926	0.087	11,576	495,347	0.087	43,064	628,273	0.087	54,641	38,110	0.075	2,845
0.060	113,450	0.092	10,409	398,207	0.092	36,796	511,657	0.092	47,205	31,199	0.077	2,415
0.070	88,975	0.097	8,607	292,125	0.097	28,268	381,100	0.097	36,875	18,844	0.082	1,541
0.080	68,101	0.101	6,910	204,123	0.103	21,039	272,224	0.103	27,949	10,899	0.086	937
0.090	50,048	0.107	5,366	140,509	0.106	14,956	190,557	0.107	20,321	5,368	0.091	488
0.100	30,248	0.116	3,519	74,425	0.112	8,324	104,673	0.113	11,844	2,409	0.099	239

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.014 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.38 Open Pit area Mineral Resource estimate at range of cut-offs Pit C – high recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**
0.005	3,851,499	0.082	314,858	3,164,643	0.036	113,471	7,016,142	0.061	428,329	703,458	0.045	31,327
0.007	**3,161,978**	**0.098**	**310,859**	**2,169,882**	**0.050**	**107,621**	**5,331,860**	**0.078**	**418,480**	**487,105**	**0.062**	**29,977**
0.010	2,581,358	0.119	306,053	1,450,891	0.070	101,530	4,032,249	0.101	407,584	355,638	0.081	28,850
0.014	2,317,619	0.131	302,894	1,160,340	0.084	98,030	3,477,959	0.115	400,924	316,768	0.090	28,381
0.020	2,242,928	0.134	301,617	1,112,750	0.087	97,232	3,355,678	0.119	398,849	305,973	0.092	28,207
0.030	2,223,110	0.135	301,046	1,086,825	0.089	96,506	3,309,935	0.120	397,552	304,801	0.092	28,181
0.040	2,142,793	0.139	297,756	1,003,706	0.093	93,547	3,146,499	0.124	391,303	301,532	0.093	28,073
0.050	2,004,307	0.145	290,578	808,254	0.105	84,775	2,812,561	0.133	375,354	257,026	0.102	26,089
0.060	1,812,493	0.154	278,741	648,956	0.117	75,985	2,461,449	0.144	354,726	228,915	0.107	24,540
0.070	1,617,372	0.164	264,763	533,575	0.128	68,523	2,150,947	0.155	333,286	173,824	0.121	21,008
0.080	1,432,518	0.174	249,417	432,046	0.141	60,932	1,864,565	0.166	310,349	132,462	0.134	17,742
0.090	1,293,731	0.183	236,309	380,324	0.148	56,437	1,674,055	0.175	292,746	111,373	0.142	15,869
0.100	1,147,239	0.192	220,622	329,749	0.156	51,439	1,476,987	0.184	272,060	84,080	0.156	13,131

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.39 Open Pit area Mineral Resource estimate at range of cut-offs Pit C – low recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**
0.007	101,861	0.064	6,495	224,473	0.094	21,203	10,635,359	0.039	412,045	245,886	0.075	18,382
0.010	82,526	0.077	6,332	190,228	0.110	20,910	9,205,156	0.043	399,907	152,195	0.116	17,684
0.014	**71,148**	**0.087**	**6,200**	**178,099**	**0.116**	**20,746**	**7,539,954**	**0.050**	**380,615**	**132,727**	**0.132**	**17,457**
0.020	67,154	0.091	6,134	177,816	0.117	20,741	6,347,798	0.057	361,863	129,791	0.134	17,412
0.030	67,065	0.091	6,131	177,130	0.117	20,722	6,080,077	0.058	355,290	129,577	0.134	17,406
0.040	65,466	0.093	6,059	169,716	0.120	20,441	5,024,452	0.063	316,285	127,568	0.136	17,339
0.050	59,405	0.097	5,754	141,952	0.135	19,159	3,377,674	0.071	239,870	118,330	0.143	16,965
0.060	47,436	0.107	5,097	120,026	0.150	18,004	2,033,207	0.081	164,397	113,815	0.147	16,678
0.070	39,546	0.116	4,575	106,064	0.162	17,139	1,292,203	0.089	114,891	110,462	0.149	16,454
0.080	29,982	0.127	3,822	95,913	0.169	16,235	767,098	0.098	74,869	109,073	0.150	16,346
0.090	24,834	0.133	3,309	93,172	0.172	15,992	419,125	0.108	45,095	107,199	0.151	16,193
0.100	21,700	0.137	2,967	87,330	0.176	15,406	228,593	0.117	26,796	103,027	0.153	15,812

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.014 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.40 Open Pit area Mineral Resource estimate at range of cut-offs MAG Pit – high recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**
0.005	2,522,058	0.058	146,035	3,118,298	0.024	74,902	5,640,356	0.039	220,937	687,826	0.010	7,020
0.007	**2,412,865**	**0.060**	**145,372**	**2,532,451**	**0.028**	**71,412**	**4,945,316**	**0.044**	**216,784**	**479,845**	**0.012**	**5,726**
0.010	2,255,670	0.064	144,098	1,841,122	0.036	65,799	4,096,792	0.051	209,897	228,714	0.016	3,662
0.014	2,135,177	0.067	142,731	1,222,506	0.048	58,798	3,357,683	0.060	201,530	83,235	0.024	2,007
0.020	2,055,674	0.069	141,501	929,022	0.058	54,260	2,984,696	0.066	195,761	41,598	0.033	1,355
0.030	2,034,630	0.069	141,004	903,592	0.059	53,640	2,938,222	0.066	194,644	21,735	0.037	812
0.040	1,895,816	0.072	135,772	799,087	0.062	49,793	2,694,903	0.069	185,565	8,348	0.041	344
0.050	1,599,598	0.076	121,878	542,483	0.069	37,412	2,142,081	0.074	159,291	2,143	0.055	118
0.060	1,168,061	0.084	97,657	305,803	0.078	23,842	1,473,863	0.082	121,499	643	0.083	53
0.070	790,494	0.092	72,691	139,480	0.091	12,697	929,974	0.092	85,388	159	0.086	14
0.080	508,084	0.101	51,154	77,128	0.101	7,790	585,212	0.101	58,944	29	0.091	3
0.090	305,752	0.110	33,563	45,025	0.108	4,878	350,777	0.110	38,441	29	0.091	3
0.100	190,508	0.118	22,501	23,471	0.119	2,792	213,979	0.118	25,293	-	-	-

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.007 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.41 Open Pit area Mineral Resource estimate at range of cut-offs MAG Pit – low recovery

Depleted for past mining												
Class	**Measured**			**Indicated**			**Measured and Indicated**			**Inferred**		
COG opt	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**	**Metal Au (koz)**	**Tonnage (ktons)**	**COG opt**	**Tonnage (ktons)**	**Au (opt)**
0.007	3,701,699	0.051	187,123	6,933,660	0.032	224,922	10,635,359	0.039	412,045	386,674	0.021	7,982
0.010	3,470,754	0.053	185,166	5,734,402	0.037	214,740	9,205,156	0.043	399,907	210,824	0.030	6,414
0.014	**3,139,698**	**0.058**	**181,234**	**4,400,257**	**0.045**	**199,381**	**7,539,954**	**0.050**	**380,615**	**129,161**	**0.042**	**5,456**
0.020	2,830,493	0.062	176,347	3,517,305	0.053	185,517	6,347,798	0.057	361,863	105,898	0.048	5,107
0.030	2,731,095	0.064	173,945	3,348,982	0.054	181,345	6,080,077	0.058	355,290	103,904	0.049	5,052
0.040	2,457,461	0.067	163,732	2,566,991	0.059	152,553	5,024,452	0.063	316,285	64,155	0.055	3,508
0.050	1,835,165	0.073	134,760	1,542,509	0.068	105,110	3,377,674	0.071	239,870	23,447	0.067	1,562
0.060	1,260,986	0.081	102,314	772,220	0.080	62,083	2,033,207	0.081	164,397	11,297	0.079	896
0.070	842,955	0.088	74,366	449,248	0.090	40,525	1,292,203	0.089	114,891	8,832	0.084	740
0.080	513,351	0.096	49,295	253,747	0.101	25,574	767,098	0.098	74,869	6,829	0.087	595
0.090	279,812	0.105	29,280	139,313	0.114	15,815	419,125	0.108	45,095	6,312	0.087	549
0.100	154,122	0.113	17,345	74,471	0.127	9,451	228,593	0.117	26,796	4,029	0.091	367

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Mineral Resources are shown at a range of cut-off values. The preferred COG for the Open Pit area Mineral Resources is 0.014 opt Au.
- Drilling results up to 15 April 2019. Mining depletion is based on topography as of July 2013.
- The numbers may not add due to rounding.

Source: AMC Mining Consultants (Canada) Ltd.

14.15 Comparison with previous Open Pit estimate

The Golder 2014 report disclosed a Mineral Resource estimate for the “South Zone” and “Mag Pit”. It was not clear from the report if the “South Zone” was part of the Mag Pit. The report compared results, excluding the South Zone, as shown in Table 14.42.

Changes since the 2014 report include:

- An updated database as a result of verification procedures undertaken by OMC.
- Estimation undertaken using Indicator method rather than constructing grade shell domains.
- Reported 2020 figures are constrained by an open pit shell, Golders were not.

The QP observes that the previous Mineral Resource had significantly higher Measured and Indicated tonnage and the grade was lower presumably by incorporating low grade material outside the current constraining pit shell, and the resulting ounces were lower than the earlier estimate.

It is considered that these changes are due to estimation method and different classification criteria employed by the different QPs, but most significantly due to the constraining pits.

Table 14.42 Mag Pit Mineral Resources comparison at Au 0.010 opt

Resource estimation	Measured			Indicated			Measured + Indicated			Inferred		
	Tons	Au (opt)	Au metal (oz)	Tons	Au (opt)	Au metal (oz)	Tons	Au (opt)	Au metal (oz)	Tons	Au (opt)	Au metal (oz)
Golder 2014	20,631,000	0.034	711,100	2,703,800	0.074	200,900	23,334,800	0.039	912,000	533,000	0.038	20,300
AMC 2020	5,726,424	0.057	329,264	7,575,524	0.037	280,539	13,301,948	0.046	609,803	439,539	0.023	10,076
Difference in %	**-72%**	**69%**	**-54%**	**180%**	**-50%**	**40%**	**-43%**	**18%**	**-33%**	**-18%**	**-40%**	**-50%**

Notes for the Golder Estimate: Open Pit Mineral Resource has an effective date of 31 December 2013.
Source: Table 14-14 from Golder Associates (2014).
Notes for the AMC estimate: See notes on Table 14.3 with respect to the current estimate.
Sources: AMC Mining Consultants (Canada) Ltd.

14.16 Recommendations

The QP recommends:

- Drillholes should be re-evaluated / re-logged for oxidation to allow for the criteria to be coded into future block model estimations.
- Additional bulk density samples be taken in future drilling campaigns every 30 ft.
- Future updates of the block model include oxidation and other parameters that would support the determination of processing options. This will allow the Mineral Resources to be more accurately reported out with different COGs.

15 Mineral Reserve estimates

There are no Mineral Reserves on the Property.

16 Mining methods

As there are no Mineral Reserves, this section is not required.

17 Recovery methods

As there are no Mineral Reserves, this section is not required. Potential recovery methods are discussed in Section 13.

18 Project infrastructure

As there are no Mineral Reserves, this section is not required. Logistics and infrastructure are discussed in a summary fashion in Section 5.

19 Market studies and contracts

As there are no Mineral Reserves, this section is not required.

20 Environmental studies, permitting and social or community impact

As there are no Mineral Reserves, this section is not required.

21 Capital and operating costs

As there are no Mineral Reserves, this section is not required.

22 Economic analysis

As there are no Mineral Reserves, this section is not required.

23 Adjacent properties

To the authors’ knowledge the adjacent properties as of June 2020 were controlled by an affiliate of Barrick, and at the time of this report, no Mineral Resources have been identified on these properties. Additionally, no Mineral Resources described in this Report go beyond the boundaries of the properties controlled by the Companies.

24 Other relevant data and information

The Mineral Resources were estimated in the local mine grid.

To convert from Pinson Local Mine Grid to NAD 27 UTM Zone 11 Metres Units, the following transformation is required:

At Point (BLM BRASS CAP SECS 27/28/33/34 PINSON grid) E15229.6300, N10098.4000:

1 Scale = 0.30458350182.

2 Rotate = 0d 10' 03.7" clockwise.

3 Move = from Point E15229.6300, N10098.4000 (BLM BRASS CAP SECS 27/28/33/34 PINSON grid) to Point E479886.0530, N4553542.7910 (BLM BRASS CAP SECS 27/28/33/34 NAD27 UTM Z11 METRES grid).

Only the base point "BLM BRASS CAP SECS 28/27/33/34" is held with the surveyed values. It is the base point in both grids.

Table 24.1 shows the monuments / points that can be used for checking the coordinate conversion.

Table 24.1 Transformation check points

Pinson grid	27 UTM by survey	27 UTM transform check
CAP 27/28/33/34		CAP 27/28/33/34 (held)
E15229.6300 N10098.4000	E479886.0530 N4553542.7910	E479886.0530 N4553542.7910
CAP ¼ 31-32 E4690.0700	E476673.4010	E476673.4034
N7308.3200	N4552702.3650	N4552702.3779
CAP 21/22/27/28 E15204.5100	E473883.0550	E479883.0508
N15313.3100	N4555131.1830	N4555131.1821

Source: Osgood Mining Company LLC.

The Mineral Resource are reported in imperial units using the conversion factors showing in Table 24.2. Table 24.3 shows conversion from imperial to the metric system for reference.

Table 24.2 Conversion factors from metric to imperial system

Measure	Metric unit	Multiplication factor	Imperial unit
Length	Metre (m)	3.28084	Foot (ft)
Volume	Cubic metre (m^3)	35.3147	Cubic ft (ft^3)
Tonnage	Tonne (T)	1.10231	Ton (t)
Grade	Gram per metric tonne (g/t)	0.0291667	Troy ounce per short ton (opt)
Mass	Gram	0.0321507	Troy ounce
Density	Metric tonne per cubic metre (t/m^3)	0.031214	1 short ton per cubic foot ($sh.t/ft^3$)

Table 24.3 Conversion factors from imperial to metric system

Measure	Imperial unit	Multiplication factor	Metric unit
Length	Foot (ft)	0.3048	Metre (m)
Volume	Cubic ft (ft^3)	0.028317	Cubic metre (m^3)
Tonnage	Ton (t)	0.907185	Tonne (T)
Grade	Troy ounce per short ton (opt)	34.2857	Gram per metric tonne (g/t)
Mass	Troy ounce	31.10348	Gram
Density	Short ton per cubic foot ($sh.t/ft^3$)	32.0369	Metric tonne per cubic metre (t/m^3)

The QP is not aware of any other additional information or explanation that is necessary to make the Report understandable and not misleading.

25 Interpretation and conclusions

Gold mineralization at the Property comprises two main areas; the Underground and Open Pit areas. Both areas are sites of past production. The Mineral Resource estimates described in the report were prepared using Datamine software. They have been estimated by Ms Dinara Nussipakynova, P.Geo., of AMC, who takes responsibility for these estimates.

Using a 0.15 opt gold COG, Measured and Indicated Underground Resources are estimated at 620,000 tons grading 0.306 opt gold; and Inferred Mineral Resources are estimated at 1,676,000 tons grading 0.347 opt gold. The Underground area Mineral Resources are constrained within modeled underground stope shapes.

Two COGs are applied to the Open Pit area based on gold metal recovery. The low recovery zone COG is 0.014 opt Au. The high recovery zone COG is 0.007 opt Au. Measured and Indicated Open Pit area Resources are estimated at 22,554,000 tons grading 0.057 opt gold; and Inferred Mineral Resources are estimated at 1,388,000 tons grading 0.047 opt gold. The Open Pit area Mineral Resources were pit-constrained.

The metal price used in determining COGs for the Mineral Resources is $1,550/oz Au. A gold metallurgical recovery of 90% was used in establishing the underground COG. A metallurgical recovery of 40% was used in establishing the open pit COG for the low recovery zone and 80% was used for the high recovery zone.

The Property is subject to a number of royalty obligations.

Numerous data validation campaigns have been undertaken on the Property.

Drilling programs completed at the Property between 2005 and 2015 have included QA/QC monitoring programs which have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams. Some concerns have been highlighted, but the QP does not consider these issues to be material to the global, long term Mineral Resource estimate.

the Companies is presently in the process of reviewing potential options to mine material contained within the Mag and CX Open Pit areas and process this material as a heap leach operation; and to mine Underground Mineral Resources at Ogee and process material at a nearby autoclave facility via a toll treatment arrangement. Based on available data, the QP considers these approaches to be reasonable. Some concerns and gaps in the metallurgical information have been identified and recommendations made to address these. Gold recoveries between 48% to 82% for Mag Pit and 82% for the CX Pit are considered achievable using heap leach. Gold recoveries between 78% to 95% are also considered achievable using an autoclave for the refractory gold associated with the Ogee material.

25.1 Risk

25.1.1 Geological risk

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is a degree of uncertainty attributable to the estimation of Mineral Resources. Until resources are actually mined and processed, the quantity of mineralization and grades must be considered as estimates only. Any material change in quantity of Mineral Resources, mineralization, or grade may affect the economic viability of the project.

- The Mineral Resource estimate was not based on oxidation information. Collection and inclusion of oxidation data and other parameters that would support the determination of processing options could materially impact the COGs.
- Data used to inform the block model is historical in nature. Verification of the source of original data is challenging due to incomplete records. The past production on the Property mitigates some of this risk. Continued efforts should be made to verify the historical data.
- QA/QC monitoring programs have only been completed on the Property between 2005 and 2015. Insertion rates were low, CRMs showed poor precision and duplicate samples showed suboptimal performance. Despite the concerns highlighted above, the QP does not consider these issues to be material to the global, long term Mineral Resource estimate. The QP however cannot guarantee that there are no material impacts on the local scale.
- The number of bulk density measurements used in the block model is limited (153). Additional sampling may result in minor changes to the density and may affect the tonnage.

25.1.2 Metallurgical risk

- Metallurgical samples do not represent the grade variability of the deposit and test work should be undertaken on samples that represent the low- and high-grade variation of the mineralization. The lack of information on metallurgical performance of such samples remains a risk to the project.
- Deleterious elements (arsenic and mercury) are present in some zones at grades high enough to be a risk to the project. Additional test work on the deportment and fate of these elements is required to define the processes necessary to mitigate their impacts.
- Sample representativity should be improved. Metallurgical sampling has been localized to relatively small portions of the Mineral Resource. The metallurgical response of the samples is likely to represent the general behaviour of the zone, but sampling of at least one other area of each zone to confirm the metallurgical response will reduce uncertainty. Confirmatory testwork on targeted drilled samples is recommended to mitigate the risk.
- Many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. The QP believes this is a risk to gold recovery if it is not treated correctly.

25.2 Opportunities

25.2.1 Geological opportunities

The Pinson Mineral Resource presently excludes several zones of relatively continuous mineralization which were solely defined by drillhole assays that could not be supported by original certificates. Verification of assays in this region, or additional drilling to confirm these results may provide sufficient justification to classify Mineral Resources in these areas.

25.2.2 Metallurgical opportunities

- By developing a geometallurgical model of each of the underground and open pit resources, it is possible to optimize the choice of processing / recovery options.
  - Selective diversion of refractory feed to stockpile for toll treatment and non-refractory feed to conventional leaching.
  - Selective diversion of preg-robbing material (open pit Mineral Resource) to appropriate processing to improve recovery.
- Examine flotation of underground feed to reduce the mass of material to an autoclave circuit. The flotation concentrates with high sulphur and gold grades should reduce operating costs and increase throughput through the autoclave.
- Trial roasting as an alternative to autoclave pre-treatment (ahead of cyanide leach) as a method of treating refractory gold in Ogee feed. This takes advantage of the proximity of

sulphide roaster facilities in the region. Roasters could also be used to treat carbonaceous material so that preg-robbing issues would be prevented.

- Maximize the potential value of the resource by completing a techno-economic trade-off study looking at the roaster and autoclave options. This study should examine the demand for Pinson material from local roasters and autoclave facilities.

26 Recommendations

The QP's makes the following recommendations:

26.1 Overall project recommendation

A selectively assigned delineation core drilling program of 5,000 feet ($500k) is recommended in Indicated & Inferred areas of mineralization to support de-risking of the existing Open Pit Mineral Resources. A Phase 1 exploration drill program of 35,000 feet ($3.5M) utilizing RC-holes and core tails is recommended at Pinson underground to test areas of highest potential and provide a basis for preliminary development planning. A Phase 2 program ($8M) of underground development and delineation drilling, designed to delineate positive results from the Phase 1 program and further confirm existing Measured and Indicated Mineral Resources, would follow thereafter. The scale of Phase 2 is dependent on Phase 1 results.

Additional detailed recommendation by Section is given below. The total cost of the programs below is $0.35M.

26.2 Sample preparation, analyses, and security

26.2.1 Data validation

- Complete additional clean-up work on the Datashed database.

26.2.2 CRMs

- Purchase additional CRMs at the approximate COGs, average grades, and higher grades of the deposits.
- Include CRMs in every batch of samples submitted at a rate of at least 1 in every 20 samples (5%).
- Ensure that CRMs are monitored in real time on a batch by batch basis, and that remedial action is taken immediately as issues are identified.
- Ensure CRM warnings, failures and remedial action is documented.
- If pulps are available in areas relevant to the current Mineral Resource, the QP recommends that an investigation into analytical precision be completed. This would comprise selecting a number of mineralized intervals associated with poor performing CRMs and completing reanalysis of two separate sub-samples from each pulp using an umpire laboratory. CRMs should be included in this submission. Differences between the grades of the new pulp assays will allow assessment of subsampling variance and geological variance. Differences to the original samples may provide insight into the precision of the original laboratory.

26.2.3 Blanks

- The QP recommends that both coarse and pulp blanks are included in future exploration programs. Blank material should be analyzed prior to inclusion in QA/QC programs to ensure the material is below the appropriate analytical detection.
- The QP recommends that fine and coarse blank material be included in each batch. The weight of individual blank samples included in the sample stream should be consistent. Blank samples should comprise 5% of the total sample stream. Blank material should be included after recognized high grade samples.

26.2.4 Duplicates

- Field Duplicates, coarse duplicates and pulp duplicates should be regularly inserted into the sample stream.
- The QP recommends that further investigative work be completed to assess duplicate performance and sample bias.

26.2.5 Umpire samples / duplicates

- The QP recommends that if historical pulps are available in the areas of the current Mineral Resource, that umpire sampling be completed. Umpire samples should comprise 5% of total samples originally submitted.

26.3 Data verification

- Drillhole collars be re-surveyed if they can still be located on the ground.
- Missing original assay certificates, downhole survey logs, original geology, and alteration logs, as well as additional records on the density, should be located if possible.

26.4 Mineral processing

- Future testwork programs should be completed on a number of samples that represent the deposit's spatial variability of weathering profile, lithology, and gold grade, and that represent run-of-mine feed from progressive stages of the project.
- Conduct quantitative mineralogy (e.g., QEMScan) on selected samples that represent run-of-mine material from progressive stages of the project.
- Complete additional autoclave pre-treatment testwork on Ogee samples.
- Conduct comminution testwork on both underground and open pit samples.
- Conduct roaster pre-treatment testwork on Ogee samples, given the proximity of sulphide roaster facilities in the region. The roasting testwork could be trialed as an alternative to autoclave pre-treatment and can be used to treat carbonaceous material.
- Complete flotation testwork ahead of autoclave pre-treatment testwork to produce flotation concentrates with high sulphur and gold grades.
- Test the deportment of arsenic and mercury in the processing of the feed. This program should cover the CIL, heap leach, and pre-oxidation processes tested during the past test work program.
- Conduct additional column leach testwork on open pit samples. This testwork should be completed at varying crush sizes to determine the optimum crush size.
- Complete additional CIL testwork on open pit material.
- Test alternative options for dealing with the carbonaceous preg-robbing material:
  - Completing resin-in-leach testwork as an alternative to activated carbon.
  - Completing testwork where blinding agents such as kerosene are added to the bottle roll tests.
- Develop a geometallurgical block model for the Pinson material. This model should also include a financial model that determines the most economically viable process route for all blocks in the block model.

26.4.1 Geometallurgy

A geometallurgical block model should be developed for the Pinson material. This model should incorporate both Open Pit and Underground areas and include key inputs such as chemical assays (including gold, sulphur speciation, and carbon speciation), mineralogy and testwork parameters. This model would develop relationships between key parameters such as gold grade, sulphide grade, carbon grade and gold recovery. This model should also include a financial model that determines the most economically viable process route for all blocks in the block model. This financial model should include inputs such as gold price, gold grade, tested gold recovery, operating costs, and expected revenue from toll treatment. The model should also account for the capacity of the various process units (heap leach and autoclave) to avoid creating process bottlenecks.

26.5 Mineral Resource estimates

- Drillholes should be re-evaluated / re-logged for oxidation to allow for the criteria to be coded into future block model estimations.
- Additional bulk density samples be taken in future drilling campaigns every 30 ft.
- Future updates of the block model include oxidation and other parameters that would support the determination of processing options. This will allow the Mineral Resources to be more accurately reported out with different COGs.

27 References

References are listed by Section below.

Accessibility, climate, local resources, infrastructure, and physiography

Bureau of Land Management (Winnemucca Field Office) 2001, Granite Creek administrative draft environmental assessment N63-EA01-xx, prepared for Pinson Mining Company, p. 47.

History

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28 QP certificates

CERTIFICATE OF AUTHOR

I, Dinara Nussipakynova, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as Principal Geologist with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report titled "Getchell Project NI 43-101 Technical Report", with an effective date of 23 July 2020, (the Technical Report) prepared for the Issuer.

3 I am a graduate of Kazakh National Polytechnic University (B.Sc. and M.Sc. in Geology, 1987). I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #37412) and the Association of Professional Geoscientists of Ontario (License #1298). I have practiced my profession continuously since 1987 and have been involved in mineral exploration and mine geology for a total of 33 years since my graduation from university. My experience is principally in Mineral Resource estimation, database management, and geological interpretation.

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

4 I have visited the Pinson Project from 19-21 March 2019 for 3 days.

5 I am responsible for Sections 2 - 12, 14 - 16, 18 - 24, and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the Property that is the subject of the Technical Report. I co-authored the unpublished Technical Report entitled "Osgood Pinson Deposit NI 43-101 Technical Report" dated 1 November 2019.

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

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

Effective Date: 23 July 2020
Signing Date: 22 January 2021

Original signed and sealed by

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Dinara Nussipakynova, P.Geo.
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CERTIFICATE OF AUTHOR

I, Dr Paul Greenhill, FAusIMM (CP), of Melbourne, Victoria, do hereby certify that:

1 I am currently employed as a Principal Consultant with AMC Consultants Pty Ltd with an office at Level 29, 140 William Street, Melbourne Vic 3000 Australia.

2 This certificate applies to the technical report titled "Getchell Project NI 43-101 Technical Report", with an effective date of 23 July 2020, (the Technical Report) prepared for the Issuer.

3 I am a graduate of University of Tasmania in Hobart, Australia (Bachelors of Science (Hons) in 1981 and Doctor of Philosophy in 1986). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (License #204579). I have practiced my profession for 30 years since graduation and have relevant experience in the metallurgy of precious and base metals.

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

4 I have not visited the Pinson Project.

5 I am responsible for Sections 13, 17, and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

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Original signed by

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