P&E MINING CONSULTANTS INC.
Geologists and Mining Engineers

201 County Court Blvd., Suite 304
Brampton, Ontario
L6W 4L2

Tel: 905-595-0575
Fax: 905-595-0578
www.peconsulting.ca

PRELIMINARY ECONOMIC ASSESSMENT OF THE BACK FORTY PROJECT, MENOMINEE COUNTY, MICHIGAN, USA

FOR
AQUILA RESOURCES INC.

NI 43-101 & 43-101F1
TECHNICAL REPORT

Andrew Bradfield, P.Eng.
Jarita Barry, P.Geo.
David Burga, P.Geo.
Kebreab Habte, P.Eng., Golder Associates
Kenneth Kuchling, P.Eng.
Neil Lincoln, P.Eng., Lincoln Metallurgical
Manochehr Oliazadeh, P.Eng., Lycopodium Minerals Canada
David Penswick, P.Eng., Gibsonian
Eugene Puritch, P.Eng., FEC, CET
D. Gregory Robinson, P.Eng.
Yungang Wu, P.Geo.

P&E Mining Consultants Inc.
Report 329

Effective Date: October 14, 2019
Signing Date: September 16, 2020

TABLE OF CONTENTS

List of Tables

LIST OF FIGURES

1.0 SUMMARY

This National Instrument ("NI") 43-101 Technical Report was prepared by P&E Mining Consultants Inc. ("P&E") with input from Lycopodium Minerals Canada Ltd. ("Lycopodium"), Golder Associates Ltd. ("Golder") and various independent consultants for Aquila Resources Inc. ("Aquila") to provide an Updated Mineral Resource Estimate and summarize the results of a Preliminary Economic Assessment ("PEA") for the Back Forty Project ("the Project" or "the Property"), located in Menominee County, Michigan, USA. The Back Forty Property is 100% owned by Aquila Resources Inc. ("Aquila" or "the Company"). Aquila is a public, TSX listed, company trading under the symbol "AQA".

Aquila's Back Forty Deposit ("the Deposit") is a volcanogenic massive sulphide ("VMS") deposit located along the mineral-rich Penokean Volcanic Belt ("PVB") in Michigan's Upper Peninsula. The Project contains approximately 1.1 million ounces of gold and 1.2 billion pounds of zinc in the Measured and Indicated Mineral Resource classifications, with additional upside potential. For the purpose of this Technical Report, mineralized rock that is processed to recover zinc, copper, lead, gold and silver is referred to as "mineralized material".

A Feasibility Study on the Project was issued in September 2018 that studied open pit mining and on-site processing plants for treating oxide material to produce gold doré and sulphide material to produce zinc, copper, lead concentrates. The value proposition of the Project was based on mining the highest value material as soon as possible and treating this material through the process plants to maximize cash flow. This strategy is achieved by mining the mineralized material and either feeding the material directly to the process plant or stockpiling the material onsite for processing later per a feed schedule based on optimal economics and/or consistent feed for the operation.

The subject of this Technical Report and Preliminary Economic Assessment relates to an expansion of the open pit mining case (Phase 1) by proposing the development of an underground mine (Phase 2) associated with the Project after the open pit phase is complete. Before the open pit has been mined out, the development of an underground mine will commence to extend the life of mine of the Project. It should be noted that this is a preliminary economic analysis of a future underground option: the Company has not yet commenced the permitting process for a potential underground expansion, including technical and environmental impact studies needed to support this process.

While the value proposition and operating context is similar to the 2018 Feasibility Study, this PEA Technical Report assumes a number of key design changes including:

- As a result of an addition of an underground mine, the oxide and sulphide processing plants were resized to a lower throughput to align combined open pit and underground Mineral Resources to optimize the Project's economics. The oxide plant throughput has been reduced from 800 tpd to 350 tpd and the sulphide plant throughput has been reduced from 4,000 tpd to 2,800 tpd.

- New cost estimates were developed for the underground mine. The initial, sustaining capital and operating PEA costs for the open pit mine and process plants were derived from the 2018 Feasibility Study and were updated accordingly. The reduction in process plant throughput contributed to a $54 million decrease in initial capital expenditures.

- The oxide processing flowsheet was updated to include a SART plant for optimal doré quality, silver recovery, mercury management, and cyanide management. Cyanide consumption has been reduced by approximately one third versus the Feasibility Study.

- Process plant feed, stockpile management and sulphide process plant change-overs have been optimized to improve operability.

- Additional metallurgical testwork has been incorporated to assess blending options and process recovery performance and penalties.

- Updated permit conditions have been incorporated, including a double liner leak detection system under all waste rock storage areas and additional contact water storage volume.

Due to the inclusion of Inferred Mineral Resources in the underground mine plan, minimal metallurgical testwork being completed to validate the metallurgical response of the underground material in the process plant, and minimal geotechnical analysis and input to the underground mine design, this Technical Report is classified as a PEA. This PEA supersedes the 2018 Feasibility Study thereby replacing the former Mineral Reserves with a potentially extractable portion of the Mineral Resource.

1.1 PROPERTY DESCRIPTION AND LOCATION

Aquila controls approximately 1,304 hectares (3,222 acres) of private and public (State of Michigan) mineral lands located in Lake and Holmes Townships in Menominee County, Michigan. Approximately 1,019 hectares (2,517 acres) of these lands form a contiguous block of Aquila-controlled mineral rights. The Active Project Boundary encompasses approximately 479 hectares (1,183 acres). The Project is centred at latitude 45° 27’ N and longitude 87° 51’ W.

In addition to the key properties, Aquila has also purchased, leased, or optioned additional properties. These properties are either contiguous with the Key Parcels, may contain facilities utilized by the Company, are perceived to have exploration potential, or were purchased for other strategic purposes.

1.2 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Property area lies along the east bank of the Menominee River and consists of low, rolling hills with maximum topographic relief of 30 m and intervening wetland (in part prairie-savannah); mean elevation is approximately 200 to 300 masl. Vegetation is mostly immature hardwood-pine forest and swamp/prairie-savannah grasses; wetland areas also occur along creeks and secondary tributaries. The climate is temperate, allowing exploration, potential development, and potential mining activities to take place year-round. Regionally, July is the warmest month with a mean temperature of 19.7°C and January is the coldest month with a mean temperature of -15.4°C. On average, the region receives approximately 796 mm of precipitation annually.

The Property is located approximately 55 km south-southeast from Iron Mountain, and approximately 19 km west of Stephenson, Michigan, within the Escanaba River State Forest. Access from Stephenson is via County G12 Road, north on River Road, travelling approximately 5 km to the Project field office. A number of drill roads connect with River Road and cross the Property. Infrastructure on the Property includes a nearby power line and paved road access.

1.3 HISTORY

In 2004, a new company, Aquila Resources Corporation, was formed for the purpose of publicly listing the Project. In 2006, the Company was renamed Aquila Resources Inc. after a reverse take-over by JML Resources. In 2007, Aquila announced the approval to list on the Toronto Stock Exchange. During the period 2009 to 2014, Aquila entered into agreements and arrangements with Hudbay Minerals Inc. and REBgold Corporation, which eventually resulted in giving Aquila 100% ownership of the Back Forty Project.

The Company currently has three main subsidiaries, Aquila Resources Corp., Aquila Resources USA Inc., and Aquila Michigan Inc. (formerly known as HMI). The remaining subsidiaries are inactive. All subsidiaries are 100% owned.

In 2014, a Preliminary Economic Assessment was completed which contemplated an open pit and underground mining/processing operation at Back Forty.

On March 31, 2015, the Company closed a multi-level financing transaction with Orion Mine Finance (“Orion”) that included an equity private placement and a silver stream for total funding of $20.75 million (collectively, the “Orion Transaction”). Concurrent with the Orion Transaction, the Company completed the repurchase of two existing royalties on the Back Forty Project. As part of the Orion Transaction, pursuant to a silver purchase agreement (the “Silver Purchase Agreement”) dated March 31, 2015 between Orion Titheco Limited, the Company and Back Forty Joint Venture LLC, Orion acquired 75 per cent of Aquila’s life-of-mine (“LOM”) silver production from the Back Forty Project for gross proceeds of $17.25 million. Orion has advanced the first instalment of $6.5 million, the second instalment of $3.0 million, the third instalment totalling $3.375 million plus the $1.35 million land payment and the final installment of $2.376 million. In June 2016, the silver purchase agreement was amended to reduce the deposit owing by $625,000. In November 2016, the silver purchase agreement was amended to reduce the deposit owing by $14,000.

In July 2017, Orion sold a portfolio of royalties, streams and precious metal offtakes, including the Silver Purchase Agreement, to Osisko Gold Royalties Ltd. (“Osisko”).

On November 10, 2017, the Company completed a financing transaction with Osisko Bermuda Limited (“OBL”), a wholly owned subsidiary of Osisko pursuant to which OBL has agreed to commit $65 million to Aquila through a $10 million private placement and $55 million gold stream purchase agreement. In connection with the private placement, Osisko received the right to nominate one individual to the board of directors of Aquila and thereafter for such time as Osisko owns at least 10 per cent of the outstanding common shares. Osisko's nominee was appointed to the board of directors in November 2017.

Concurrent with the Strategic Investment, the parties have entered into a Gold Purchase Agreement (the "Gold Stream"), whereby OBL will provide the Company with staged payments totalling $55 million, payable as follows:

- $7.5 million on close of the Gold Stream (received November 2017);
- $7.5 million upon receipt by Aquila of all material permits required for the development and operation of the Project, and receipt of a positive Feasibility Study (received October 2018);
- $10 million following a positive construction decision for the Project (milestone amended in June 2020); and
- $30 million upon the first drawdown of an appropriate Project debt finance facility (reduced to $20 million in June 2020), subject to the COC Provision (as defined below).

Under the terms of the Gold Stream, OBL will purchase 18.5% of the refined gold from the Project (the "Threshold Stream Percentage") until the Company has delivered 105,000 ounces of gold (the "Production Threshold"). Upon satisfaction of the Production Threshold, the Threshold Stream Percentage will be reduced to 9.25% of the refined gold (the "Tail Stream"). In exchange for the refined gold delivered under the Gold Stream, OBL will pay the Company ongoing payments equal to 30% of the spot price of gold on the day of delivery, subject to a maximum payment of $600 per ounce.

On September 7, 2018 Aquila filed an open pit Feasibility Study Technical Report on SEDAR, with an effective date of August 1, 2018.

On October 5, 2018, Aquila received a payment of $7.4 million from an affiliate of Osisko under the Gold Purchase Agreement. This payment represents the second deposit of the total advance payment of US$55 million to be made by Osisko under the Gold Purchase Agreement. The payment, which was made net of a $100,000 capital commitment fee, follows receipt by Aquila of all material permits required for the development and operation of the Back Forty Project and the completion of the Back Forty Project Feasibility Study.

On June 28, 2019, the Company announced that its two largest shareholders, Orion Mine Finance (and its affiliated funds) ("Orion") and Osisko Gold Royalties Ltd. ("Osisko") completed a transaction whereby Orion purchased from Osisko all 49,651,857 common shares of the Company owned by Osisko (the "Transaction"). The Transaction was a small component of the share repurchase and secondary offering transaction first announced by Osisko on June 25, 2019. Orion now owns 97,030,609 common shares of Aquila representing approximately 28.7% of the outstanding common shares. Osisko remains a significant financial partner to Aquila as the holder of gold and silver streams on the Company's Back Forty Project.

On June 17, 2020, Aquila announced it entered into definitive agreements with Osisko to amend certain terms of the Gold Stream and the Silver Purchase Agreement in order to accelerate Aquila's access to a portion of the outstanding funding under the Gold Stream and to provide additional flexibility.

Under the terms of the amendments, Osisko will immediately advance $2.5 million (excluding transaction costs) of the remaining deposit under the Gold Stream to Aquila. Osisko will advance an additional $7.5 million upon Aquila achieving certain corporate and Project development milestones that are expected to be completed over the next 12 to 18 months. Osisko has also agreed to adjust certain milestone dates under the Gold Stream and Silver Purchase Agreement to align the streams with the current Project development timeline.

In exchange for Osisko agreeing to make the payments and milestone date changes described above, the remaining deposit available to Aquila under the Gold Stream will be reduced from $40 million to $35 million, of which $10 million is payable as described above, and the remaining $25 million will be payable pro-rata with drawdowns under a senior construction facility for the Company's Back Forty Project. The designated Gold Stream percentage remains unchanged at 18.5% until the delivery of 105,000 gold ounces to Osisko, upon which the stream will be reduced to 9.25%. Osisko will continue to pay 30% of the gold spot price on delivery, subject to a maximum payment of $600/oz. The Silver Purchase Agreement will be amended to increase the designated silver stream percentage from 75% to 85% of the number of payable silver ounces produced from Back Forty with no change to the ongoing price of $4/oz.

1.4 GEOLOGICAL SETTING AND MINERALIZATION

The Back Forty VMS Deposit is one of a number of deposits located throughout the Ladysmith-Rhinelander volcanic complex in northern Wisconsin and the Upper Peninsula of Michigan. The complex lies within the lower Proterozoic PVB, also known as the Wisconsin Magmatic Terranes. The PVB is part of the Southern Structural Sub-province of the Canadian Shield.

Published small-scale (1:250,000) geologic maps of northeastern Wisconsin indicate the area to the west of the Project area is underlain by the 1,760 to 1,870 Ma old Athelstane Quartz Monzonite, an intrusive complex composed of tonalite, granodiorite and granite. The plutonic complex is bounded on the north, east, and south by metavolcanic rocks of the Beecher Formation and contains numerous metavolcanic rock inclusions. The volcanics generally face outward from the margin of the intrusive complex. Dykes of Athelstane Quartz Monzonite extend a short distance into the Beecher Formation (Jenkins 1973).

The Beecher Formation consists of a stratigraphically lower, 3,000 m thick sequence of calc-alkaline andesite to dacite flows and an upper 300 m thick section of interbedded felsic ash, crystal tuff, lapilli tuff, coarser fragmental rocks, and locally black slates near the stratigraphic top of the formation. The Back Forty Deposit is hosted by a volcanic complex quite similar to the upper volcaniclastic section of the Beecher Formation. Zircons extracted from rhyolite crystal tuff and intrusive rhyodacite porphyry from Back Forty have yielded a uranium/lead age of 1,874 ±4 Ma (Schulz et al. 2008). This age is consistent with the published age of the Athelstane Quartz Monzonite. It is likely that the felsic sequence at Back Forty is a member of the Beecher Formation. The lateral extent of this volcanic centre is unknown at this time. However, drilling and gravity surveys indicate it is truncated to the west and north by Athelstane Quartz Monzonite, but likely extends further to the east and south, beneath Cambrian sandstone sediments.

Detailed core logging and lithogeochemical studies completed to date by Aquila have established at least four lithologic units within the portion of the felsic centre hosting the Back Forty mineralization. Regional deformation has produced a penetrative foliation; locally shears have been observed. The foliation is developed best in rhyolite crystal tuff units that have the strongest sericite alteration. In the fragmental units, clasts are commonly stretched parallel to foliation. In the bedded tuffaceous unit, schistosity is parallel to relict bedding.

Based on geologic relationships and apparent offsets, high angle, north-south striking faults were inferred striking through the central portion of the Back Forty Deposit. A detailed review of drill core and geotechnical data did not confirm these as major, through-going structures. A second set of west-southwest trending, high-angle faults were also previously interpreted. These faults in general parallel the axial plane of the anticlinal fold. The principal east-west fault has been confirmed by a review of drill core and geotechnical data, and appears to strike through the southern portion of the East Zone massive sulphide and continue west to form the northern boundary of the Hinge Zone massive sulphide, as well as the southern boundary of the Pinwheel Zone.

Mineralization at the Back Forty Deposit consists of discrete zones of: 1) zinc or copper-rich massive sulphide (±lead), which may contain significant amounts of gold and silver, 2) stockwork stringer and peripheral sulphide, which can be gold, zinc, and copper-bearing (±lead/silver), 3) precious metal-only, low-sulphide mineralization, and 4) oxide-rich, precious metal-bearing gossan.

To date, VMS-style mineralization has been identified within at least two stratigraphic levels within the felsic sequence at the Back Forty Deposit. Although the majority of rhyolitic rocks hosting the Deposit sulphide mineralization are indiscernible with respect to appearance, the two main rhyolites (rhyolites 1 and 2) have distinctive geochemical signatures as can be observed through aluminum-titanium and zirconium-titanium ratios. The Main Zone massive sulphide, which accounts for the vast majority of massive sulphide mineralization lies at the statigraphic boundary of these two rhyolite units. Rhyolite 1 lies stratigraphically below this sulphide horizon (footwall) while rhyolite 2 lies above the horizon (hanging wall). Another massive sulphide horizon, the Tuff Zone, is located at or near the upper contact of rhyolite 2 and the lower contact of an overlying package of tuffaceous and siliceous sediments. Another zone of massive sulphide mineralization, the Deep Zone, was identified as a possible third, lower mineralized horizon. Additional drill intercepts of massive sulphide mineralization have been encountered at depth and to the southwest (down plunge) of known mineralization. Due to limited follow-up drilling of these intercepts it is, at the current time, unknown as to how these fit in with the overall geology and stratigraphy of the Deposit. Massive sulphide refers to rocks composed of at least 80% sulphide, rather than the more common cut-off of 60% for massive sulphides. Semi-massive sulphide mineralization is considered to contain 10 to 80% sulphides.

The Main Zone is composed of three separate massive sulphide bodies (referred to as the East, Hinge, and South Limb Zones) that form parts of a plunging anticlinal structure and are considered the same horizon. These bodies are hosted by Rhyolite 1 (footwall) along and stratigraphically below their contact with Rhyolite 2 (hanging wall). These horizons are stacked, strata-bound massive sulphide bodies that are enveloped locally by stockwork and semi-massive sulphide mineralization. Pervasive sericite and disseminated pyrite alteration as well as variable silicification are abundant and extend outward for an undetermined distance. The Main Zone extends along strike for over 450 m in a west-southwest direction; it is up to 100 m wide and subcrops at its eastern end under thin (less than 10 m) glacial overburden or local Palaeozoic sandstone. The stockwork-stringer and peripheral sulphide envelope grades outward into a semi-conformable disseminated (less than 10%) pyritic halo that extends throughout the entire altered Rhyolite 1 host unit for an undetermined distance. The zone has been extensively disrupted by variably altered quartz feldspar porphyry ("QFP") intrusions.

The East Zone subcrops east of the Keweenawan dyke under glacial overburden, which is less than 10 m thick. Locally, erosional outliers of Palaeozoic sandstone are less than 0.5 m thick. The massive sulphide body is capped by a thin gossan (generally 3-5 m thick). At the top of the massive sulphide, directly underlying the gossan is a thin zone of copper-rich massive sulphide (often less than 1-2 m) which was likely enriched by means of late super-gene processes.

The Hinge Zone, in part offset by faulting, has been folded tightly into a cigar-shaped body that plunges moderately at approximately forty degrees to the southwest along the axial plane of the anticlinal fold; the South Limb is separated from the Hinge by a laterally persistent QFP dyke and remains open to the southwest. Further west, the horizon is apparently offset downwards again between Sections 435,225E and 435,200E. Between sections 435,200E and 435,100E, deformation of the Hinge horizon likely has resulted in tectonic thickening of this unit (up to approximately 70 m in the "hinge" area). Beyond Section 435,100E to the west, the Hinge horizon appears to pinch out against a QFP dyke.

The South Limb Zone is interpreted to represent the steeply-dipping southern fold limb of the anticline where it is steeply dipping to the south, while plunging to the west-southwest this interpretation is supported by lithogeochemical data. Locally, shearing is common, resulting in an overall uniform thickness and lens-shaped geometry.

The Hinge and South Limbs Zones are separated by large, variably-altered QFP dykes that have been intruded into the axial plane area of the anticlinal fold. These syn- or post-mineralization QFP intrusions have intruded, cut-off, and obliterated portions of both horizons. To the west, the model suggests that the South Limb may be pinching and swelling down plunge into a series of thin to thick lenses that occupy the south limb of the anticline. Drilling continues to support the above interpretation. The South Limb remains open along strike.

The Pinwheel Zone is located at the northwest end of the Deposit and is a shallow, isolated erosional remnant located structurally along the gently north-dipping northern limb of the anticlinal fold and is truncated to the south by the E-W fault. Limited geochemical data suggests that this unit is in fact located along the contact between rhyolite 1 and rhyolite 2 and is therefore likely the equivalent to the Main Zone massive sulphide and represent a 'faulted-up' portion of the north limb of this important massive sulphide horizon. Massive sulphide mineralization on strike of the Pinwheel Zone has been traced for roughly 700 m to the west-southwest where the gentle north-dip of the unit steepens. It should be noted however, that the massive sulphide mineralization is to some degree discontinuous and often has a 'stacked' geometry, and that numerous faults and shear zones have been encountered in the adjacent host rock. The geometry of this zone is likely complicated due to these structures.

The Pinwheel Zone is broken up in to two separate units based on spatial relationships and dominant mineralization types. The near-surface, gently north-dipping eastern-most portion of the Pinwheel Zone is referred to as the 'Pinwheel Cu-Rich Zone' due to the relative abundance of copper mineralization (predominantly pyrite + chalcopyrite) and subsequent lack of other base metals (zinc and lead) within the massive sulphide. The majority of the Cu-Rich Zone is capped by an overlying gossan that crops out on the Property along the southeast terminus of the zone. The Cu-Rich portion of the Pinwheel Zone represents the most copper-enriched massive sulphide located at the Back Forty Deposit and it is interpreted that the copper enrichment has a secondary, supergene association. It is possible, however, that this zone represents an original, high-temperature, copper rich portion of the VMS system. Along strike to the west-southwest, copper-dominant mineralization diminishes with a subsequent increase in the presence of zinc (sphalerite) and to a lesser extent lead (galena). This zone has been referred to as the 'Pinwheel Extension' or 'Pinwheel Zn-Rich Zone' and the variation in metal content with respect to the Cu-Rich portion is interpreted to be due, in part, to a lack of influence from secondary, super-gene processes.

The Deep Zone is located north of one of the QFP dykes, juxtaposed against the South Limb horizon. Recent geological and geochemical data interpretation suggests that the Deep Zone may be the down-dip continuation of the South Limb, where it has been folded and rotated. This interpretation leaves significant spatial potential for further resource discovery between the South Limb and the Deep Zone as well as down dip of the Deep Zone.

The Deep Zone is relatively enriched in copper compared to zones of the main horizon (East, Hinge, and South Limb) and suggests that a more copper-rich portion of this VMS system may occur at depth.

The Tuff Zone massive sulphide occurs at the south edge of the Deposit. Stratigraphic and structural data suggest this zone is located at a higher level in the volcanic sequence. In cross sections and three-dimensional models, the zone appears to have a bowl-shaped geometry possibly reminiscent of small relict depositional basin or local graben structure.

The Tuff Zone is hosted at or near the stratigraphically upper portions of the intensely sericitized and locally chlorite-altered Rhyolite 2 unit as well as within the lower portion of the overlying siliceous tuffaceous sediment unit. The Tuff Zone has been traced along strike to the southwest by drilling (parallel to the Main Zone) for roughly 25 m. The zone is predominantly steeply dipping to the south and occupies the southern limb of the anticlinal structure. Drilling intercepts down dip and at depth of the zone indicate shallowing and flattening of the unit that suggests proximity to a synclinal structure to the south. Massive sulphide mineralization of the Tuff zones appears preferentially developed within coarser grained tuffaceous units at or near the contact of rhyolite 2 and of the overlying tuffaceous and siliceous sediments. Overall sulphide content is less massive than that of the Main Zone (~60-80%) and is dominated by sphalerite, pyrite, and galena. The zone's thickness is typically on the order of a few metres. The horizon possibly subcrops in the northeast along Sections 435,175E and 435,150E but plunges southwest (to at least Section 435,000E) similar to the orientation of the massive sulphide horizons of the Main Zone.

1.5 DEPOSIT TYPE

The zinc-copper-lead-gold-silver bearing sulphide mineralization identified on the Property exhibits typical characteristics of VMS mineralization. VMS deposits form in a marine volcanic environment by the circulation of hot hydrothermal fluids near spreading centres. Cold seawater infiltrating ocean crust off-axis is progressively heated by hot magma underlying the rift zone. Heated and buoyant fluids leach metals from the surrounding rocks. Metallic sulphides precipitate at or near the rock-water interface as a result of rapid changes in Eh and pH triggered by rapid mixing with cold ambient seawater. Precipitated sulphides form massive mounds, fracture and cavity fills, as well as replacement textures. Metal zoning is common with copper-rich zones at or near the centre and zinc-rich zones at the fringes of a sulphide mound. Multiple events and zone refinement are common, often due to changes in the internal plumbing system.

1.6 EXPLORATION

Geophysical surveys including airborne EM, ground EM, gravity, and magnetic surveys have been the primary means of exploration over the life of the Project. To a lesser extent, geochemistry and geologic mapping have also been utilized to aid in exploration efforts.

Sparse outcrop mapping in the immediate Deposit area has yielded structural and geochemical data supporting the general Deposit model, although outcrop distribution does not allow for any delineation of mineralization.

A total of 680 geochemical whole rock analysis of drill core have been collected from host rocks at the Back Forty Deposit as well as from drilling peripheral to the Deposit area from 2002 to 2012 and have been compiled into a geochemical database. Additional whole rock samples have been collected from the 2015 to 2017 drill programs and are currently being added to the geochemical database. No traditional soil geochemical surveys have been undertaken in the Project area.

Extensive geophysical surveys have been completed over the immediate Project area and surrounding areas from 2002 to present. Geophysical surveys include two airborne magnetic/EM surveys and extensive ground surveys including HLEM (Max-Min), Pulse EM, magnetics and gravity as well as extensive downhole Pulse EM surveys completed during various drilling campaigns.

Two airborne electromagnetic and magnetic surveys have been flown over the Project area. In 2002, a GEOTEM, fixed wing electromagnetic and magnetic survey with north south 200 m spaced lines was flown over the area of the Back Forty discovery, and in 2007 a larger (500 square km), partially overlapping VTEM and magnetic survey was flown by Geotech Ltd. The VTEM survey line spacing was 100 m in the western portion of the block and 200 m in the eastern portion.

Previous ground geophysical surveys completed over the prospect area were conducted by initial operator MPC and include horizontal loop electro-magnetic (max-min), total field magnetics, and gravity. Ground and down-hole pulse electromagnetic surveys ("PEM") were conducted during the 2002 to 2003 drilling program. The ground and down-hole geophysical surveys were conducted by Crone Geophysics with interpretation provided by ACNC geophysicists. Four loops were laid out to locate extensions of the sulphide deposit.

Additional PEM surveys that were conducted in the immediate Back Forty Mineral Resource area were run during middle to late 2006 and 2007 with interpretation provided by Clark Jorgenson in 2007 and 2008. All electromagnetic responses were modelled with the "Maxwell" program developed by Electromagnetic Imaging Technology of Perth, Australia. A number of geophysical targets were tested successfully; other targets could not be explained through drilling.

Additional downhole Pulse EM surveys were completed during the 2009-2011 drill programs. The surveys were completed by Crone Geophysics and reviewed and interpreted by Hudbay geophysicists who aided in the initial delineation of the Back Forty Deposit at depths exceeding 650 m in the vertical direction.

Downhole surveys were also carried out following the 2016 drilling campaign and were completed by Abitibi Geophysics. Geophysicist, Dan Card has been overseeing the design and interpretation of these recent surveys, and has also recently reinterpreted the VTEM responses in the deposit are in conjunction with past and recently completed downhole PEM and Surface PEM.

Since most of the immediate Deposit area and prospective geologic trends adjacent to the Deposit are covered with glacial drift and Paleozoic sediments, and because cultural features (power lines, fences, etc.) are common and interfere with electromagnetic techniques, extensive gravity surveys have been conducted over the Deposit and surrounding area from the Project's inception through 2016.

In 2016, consolidation of land ownership peripheral to the Deposit allowed expansion of the detailed gravity grid to the northeast and southwest of the Deposit. Subsequent drill testing of the gravity anomaly extending southwest of the known Deposit resulted in the discovery of a new zone of massive sulphide mineralization – the 2016 Zone, which was the target of drill testing in 2017.

1.7 DRILLING

Drilling on the Property was conducted over several campaigns. Between 2002 and 2017, 624 drill holes totalled approximately 122,100 m. In addition to Mineral Resource delineation drilling associated with the expansion of the Back Forty Mineral Resource, focused drill efforts were also undertaken which included: drilling of exploration (geophysical) targets in the immediate vicinity of the Deposit area, drilling to support metallurgical testing programs, and geotechnical drilling to characterize the rock quality of the Deposit area.

The first drill program, conducted by ACNC, started in February 2002 and continued to late May 2003. The program consisted of 71 drill holes (20,600 m), from which approximately 7,600 assay samples and 340 whole-rock samples were collected.

The second drill program occurred in Q4 2006. This program delivered 13,190 m of core in 80 BTW sized holes. The majority of the drilling targeted the East and Pinwheel Zones.

The third drilling program was completed in 2007 with 118 drill holes totalling 27,800 m.

A fourth drill program in 2008 on targets distributed throughout the Mineral Resource area was completed in 2008 with 66 drill holes for 13,950 m.

From October 2009 to May 2010, another phase of drilling was mounted. For this program, IDEA Drilling completed the first 20 holes on the Project using NQ2 and the drill holes were oriented (totalling 1,327 m). IDEA Drilling subsequently completed 93 NQ3 split-tube oriented holes and one extension using BTW for a total of 8,681 m. IDEA Drilling also completed 11 drill holes outside the immediate Deposit area that were not used for the Mineral Resource Estimate (1,388 m). Boart Longyear completed 11 NQ3 split-tube oriented holes that were included in the Mineral Resource Estimate totalling 1,492 m. In addition, Boart Longyear completed five NQ3 "geotechnical" holes that targeted the conceptual open pit walls (971 m). The core from these holes was archived in its entirety, i.e., not cut and assayed, therefore they are not included in the current Mineral Resource Estimate.

Drilling from 2009 to 2010 outside the immediate Back Forty Deposit approximately 600 m to the east was targeted on ground magnetic and gravity anomalies. Anomalous zinc and gold mineralization in altered rhyolites and sediments was encountered in two drill holes. Drill hole PTL-1 intersected 10.0 m of 0.61% Zn, including one 1.5 m sample of 1.08% Zn. Drill hole PTL-2 encountered an interbedded sequence of flows and tuffaceous sediments including a chlorite-altered fragmental zone containing 26.5 m of 0.54% Zn, with smaller zones exceeding 1% Zn, a lower interval of tuffaceous sediments containing 12.5 m of 0.51% Zn, and an underlying siliceous breccia with 6 m of 1.1 g/t Au, including 1.5 m of 2.67 g/t Au. This suggests that prospective host rocks continue to the east of the Back Forty Deposit for at least 600 m. These two drill holes are not part of the Back Forty Mineral Resource Estimate.

78 holes were drilled during 2011. The programs included drilling 22 high-grade gold targets at depth, four geophysical targets, and 22 relatively shallow holes to delineate the Pinwheel Gossan Zone.

A total of 11 drill holes were completed to collect metallurgical samples, 12 for condemnation purposes east of the Mineral Resource and 5 drill holes to install monitoring wells for groundwater purposes. These additional 28 drill holes are not part of the current Mineral Resource Estimate.

Drilling in 2015 consisted of a total of 13 NQ sized drill holes totalling 1,775 m. The primary focus of the program consisted of 833 m of drilling in 9 metallurgical drill holes targeting sulphide mineralization within the open-pit portion of the Mineral Resource. Two drill holes from the 2015 drill program targeted Mineral Resource expansion of the Pinwheel Zone on a property that had previously been unavailable for drilling. The two drill holes intercepted zinc-rich massive sulphide and associated gold mineralization within the host rocks. An additional 2 drill holes targeted a geophysical anomaly peripheral to the Deposit area. No significant grades were reported in the two drill holes.

A total of 2,333 m was drilled in 13 holes in 2016. Geotechnical drilling consisted of 671 m of drilling in 3 drill holes evaluating rock quality in the south-western and south-eastern portion of the open pit Mineral Resource area as well as to test the rock mass quality along the proposed cut-off wall between the planned open pit and the Menominee River. One drill hole intercepted mineralization outside of the planned open pit extents. This drill hole was sampled and assayed as part of the 2017 drill program.

Four drill holes for 627 m were completed in 2016 to delineate and extend the known Mineral Resource outside of the planned open pit. An additional 6 drill holes totalling 1,195 m were drilled testing both airborne and recently identified ground geophysical anomalies proximal to the Back Forty Deposit.

A total of 24 drill holes totalling 6,001 m were completed between January and June of 2017. The drilling consisted of three independent programs including a geotechnical drilling program which characterized rock mass qualities for 'out of pit' Mineral Resource, a Mineral Resource delineation drilling program which included both infill drilling to convert Inferred Mineral Resources to Indicated Mineral Resources and step out drilling on known mineralization, as well as an exploration program evaluating geophysical anomalies. The geotechnical drilling program consisted of a total of 5 drill holes and 1,281.2 m total of drilling designed to evaluate the rock mass quality within the potential underground mining area including 3 drill holes in the Pinwheel area southwest of the planned open pit and 2 holes in the Main Zone and Deep Zone area below and southwest of the planned open pit. In addition to collecting geotechnical data a number of the geotechnical drill holes were also designed to intercept areas of Inferred mineralization within the Mineral Resource model in the vicinity of the Pinwheel Zone, Tuff Zone as well as the Deep Zone.

Mineral Resource delineation drilling consisted of a total of 10 drill holes as well as extensions of two holes for a total of 2,610 m. In addition to geological logging, geotechnical logging was completed on select drill holes due to a lack of geotechnical information within the Pinwheel portion of the potential underground mine area. Seven drill holes were designed to intercept Inferred Mineral Resources as well as to test the western, down-dip extension of the Pinwheel Massive sulphide. All drill holes encountered massive sulphide mineralization associated with the pinwheel massive sulphide. Two holes were designed to intercept Inferred mineralization located in the Deep Zone massive sulphide and adjacent Porphyry Margin Gold Zone. Both drill holes also encountered mineralization associated with the Tuff Zone massive sulphide and stringers as well as the 90 Gold Zone along the south margin of the proposed open pit.

A total of 9 drill holes totalling 2,110 m were drilled as part of an exploration program targeting a geophysical anomaly identified during 2016 and as follow-up on the newly discovered massive sulphide zone from the 2016 drill program. Given the limited drilling in this area mineralization has not been modelled and is not incorporated into the Mineral Resource Estimate.

Three drill holes totalling 633.27 m were drilled as part of an abbreviated exploration program in 2018. The drill program was designed to test the extents of the recently discovered 2016 Zone and another geophysical target peripheral to the known Deposit. The drill holes were completed after the current Mineral Resource Estimate was completed, and not are included in the Updated Mineral Resource Estimate.

The 2019 geomechanical drilling program consisted of a total of seven drill holes totalling 1,274.03 m. Drilling was designed to evaluate the rock mass quality within the west pit wall and to evaluate the rock quality on a potential crown pillar. The 2019 metallurgical sampling program consisted of a total of eight drill holes totalling 558.33 m targeting early mining within the open-pit portion of the Mineral Resource. Assays from the 2019 drill holes were not incorporated into the Updated Mineral Resource Estimate.

1.8 SAMPLE PREPARATION, ANALYSES AND SECURITY

It is P&E's opinion that sample preparation, security and analytical procedures for the Project drilling and sampling programs were adequate for the purposes of this Mineral Resource Estimate.

Based upon the evaluation of the QA/QC programs undertaken by Aquila, P&E concludes that the data are of good quality for use in the Back Forty Updated Mineral Resource Estimate.

1.9 DATA VERIFICATION

Based upon P&E's due diligence sampling and data verification, P&E concludes that the data are of good quality for use in the Back Forty Updated Mineral Resource Estimate.

1.10 MINERAL PROCESSING AND METALLURGICAL TESTING

Several historical metallurgical testwork campaigns have been completed on various samples related to the Project. The main objective of the metallurgical test work campaigns was to quantify the metallurgical response of the VMS mineralization and included several flotation and leaching studies, comminution and gravity tests. This work was used to established metallurgical domains (refer to Table 1.1) and direction for test conditions and to demonstrate variability throughout the Deposit. Metallurgical testing has generally focused on the three main sulphide mineralized zones (Main, Pinwheel and Tuff Zones) and the oxide portion of the Deposit.

TABLE 1.1
METALLURGICAL TYPES

No.	Major Zones	Name
1	Main	Main Zone Massive Sulphide
2	Pinwheel	Pinwheel Massive Sulphide Cu Rich
3	Pinwheel	Pinwheel Semi-Massive and Stringers
4	Pinwheel	Pinwheel Extension
5	Tuff	Tuff Zone
6	Oxides	Oxides
7	Pinwheel	Pinwheel Gossan Flotation
8	Pinwheel	Pinwheel Massive Sulphide Cu-Zn Rich

A series of metallurgical testing campaigns were completed from 2015 to 2019 in support of both the 2018 Feasibility Study and the current PEA. These metallurgical testwork programs were primarily conducted at SGS (Lakefield, Ontario) and dewatering and rheology work was conducted at Golder (Sudbury, Ontario). Filtration and sulphidization-acidification-recycling-thickening (“SART”) testwork was carried out by Tenova and BQE, respectively (Vancouver, British Columbia).

SGS’s Geostats group was engaged by Aquila to develop a drill plan for fresh sulphide material and assist with sample selection. Oxide and sulphide domain composites were created. Following sub-domain compositing, three sulphide master composites with the sub-domain composite material were created. The samples selected represent the spatial distribution, head grades and mineralization types of the Back Forty Deposit.

Comminution testwork included Bond ball work index (“BWI”), modified Bond ball work index (“ModBond”), abrasion index (“AI”), crusher work index (“CWI”) and SAG mill comminution (“SMC”) tests. Overall, the samples depicted a high degree of variability across the grindability characterization tests. Samples for SMC tests were considered soft to very hard with A x b ranging from 83.9 to 22.5. There was a broad range in the relative density, from 2.71 to 4.86 t/m^3. Within their own mineralized zones, there was relative consistency in both hardness and density of the samples. CWI samples covered the soft to moderately hard range of hardness within the SGS database, with CWI varying from 4.4 to 12.5 kWh/t. The average CWI was 7.9 kWh/t (classified as moderately soft). BWI results ranged from very soft to hard (9.1–18.9 kWh/t). While a relatively wide range of results are observed over the data as a full set, ranges are narrower by metallurgical type, with oxides being the most competent and Pinwheel being the least competent. ModBond samples covered very soft to very hard range of hardness in the SGS database, ranging from 9.2 to 20.8 kWh/t. Following the trend from other hardness characterization tests, the global set of data shows a significant relative standard deviation, while within each metallurgical type the data range was narrower. The AI values ranged from 0.285 g to 0.564 g, with an average value of 0.398 g, which is considered medium.

The metallurgical testwork program included flotation testwork to develop the flotation conditions and further optimize the historical results. The program aimed at minimizing the number of distinct metallurgical types from a processing perspective and to optimize reagent dosages with some consideration for alternatives. The approach taken to decrease the number of metallurgical types was to create variability composites within each of the main mineralized types (Main, Pinwheel and Tuff).

The main master composites were submitted for mineralogical analysis (QEMSCAN). The resulting modal analysis indicated that both the Pinwheel and Main zone master composites were dominated by chalcopyrite and sphalerite as value minerals and pyrite as the major gangue mineral. The Tuff Zone master composite was dominated by sphalerite and galena as value minerals, with a large amount of the gangue represented by quartz. Liberation data established the primary grind size for the main mineralized types.

Table 1.2 summarizes the relative mineralogical differences between Main, Pinwheel and Tuff material types.

TABLE 1.2 **MINERALOGY OF SULPHIDE MATERIAL (RELATIVE DIFFERENCES)**				
Material Type	**Mineral Liberation for Flotation (P_{80})**	**Chalcopyrite (Copper Mineral)**	**Galena (Lead Mineral)**	**Sphalerite (Zinc Mineral)**
Main	75 µm	Medium	-	High
Pinwheel	50-55 µm			
MS and Gossan		High	-	Trace
SM and Stringers		Low	-	Low
Extension		Medium	-	High
Tuff	60-65 µm	Trace	High	Medium

Note: *MS = massive sulphide, SM = semi-massive sulphide.*

All of the variability composites for each of the three metallurgical types were subjected to cleaner flotation testing. There were two main purposes of the tests: 1) to examine in greater depth the flotation responses of the individual samples making up each master composite in order to determine which of the individual samples were particularly problematic and to further explore optimization strategies; and 2) to understand the metallurgical responses over a range of samples.

In general, the target regrind size used in the most recent phase of testing for both copper and zinc rougher concentrate was 15–20 µm. Although not quantifiably tested historically, the general trend was a positive shift in the grade recovery relationship of a given composite that was subjected to a finer regrind (both bulk and zinc). This was further confirmed by mineralogical data that in general showed that the degree of free and liberated Cu-Sulphate, galena and sphalerite increased with decreasing particle size. This regrind target range was deemed suitable in consideration of grinding effort and the need to minimize overgrinding of the cleaner feed.

The metallurgical testwork program included an oxide testwork program to test various sub-domains within the metallurgical type, to determine suitable leach conditions and to acquire downstream data (oxide tailings filtration and SART). The approach taken was to subject all oxide sub-domains which made up the ultimate master composite to varying conditions in a series of bottle roll tests.

The main test conditions that were explored were primary grind size, cyanide concentration and oxygen addition. Other conditions examined were the addition of lead nitrate, test pH and leach time. Leaching of flotation tailings was also completed to investigate at a scoping level alternative gold recovery flowsheets.

Once universal test conditions were determined, a master composite was created by blending the representative sub-domains by what is understood to have been their appropriate in-situ proportions. A larger bulk leach was performed on the master composite to generate a global overall expected recovery as well as enough product to perform filtration testing and SART testing.

Tailings samples, representing Main, Pinwheel, Tuff and Oxide zones were subjected to settling and rheology testing.

Test results from 2016-2019 testwork programs and historical test results formed the basis for each of the metallurgical recovery equations. In general, there is a reasonable correlation between the head grade of the target base metal with the ultimate recovery to the concentrate. Target concentrate grades were selected based on the metallurgical performance of the samples for each material type for the financial analysis. For copper, the target concentrate grade varies for each copper containing material type and has a range of 17% to 22% Cu. Zinc concentrate targets vary from 50% to 55% Zn and the lead target concentrate grade is 35% Pb.

1.11 MINERAL RESOURCE ESTIMATE

All drilling and assay data were provided in the form of Excel data files by Aquila. The GEOVIA GEMS™ V6.8 database for the updated Mineral Resource Estimate, compiled by P&E, consisted of 741 drill holes totalling 128,670 m, of which 1,447 intersects totalling 17,201 m from 489 drill holes were used for the updated Mineral Resource Estimate.

The updated Mineral Resource Estimate with an effective date of October 14, 2019 is tabulated in Table 1.3. P&E considers the mineralization of Back Forty to be potentially amenable to Open Pit and Out of Pit (underground) extraction. Open pit model NSR cut-off values were $21/t for flotation and $22/t for leach material above 0 m EL, and $70/t below 0 m EL. Underground model NSR cut-off values ranged from $65/t to $68/t for flotation and $77/t for leach material.

54 and 58 mineralization wireframes were constructed for open pit and underground Mineral Resource Estimates, respectively. Block sizes in the models were 5.0 m x 2.5 m x 2.5 m (XYZ). For Mineral Resource estimation, P&E considers metallurgical ("met") type 2 and 7 to be the same material, and met type 4 and 8 to be the same material.

The pit-constrained Mineral Resource Estimate totalled 11.4 Mt of Measured and Indicated Mineral Resources at 1.87 g/t Au, 23.03 g/t Ag, 0.27% Cu, 0.22% Pb and 2.62% Zn. Pit-constrained Inferred Mineral Resources totalled 0.3 Mt at 3.13 g/t Au, 42.32 g/t Ag, 0.06% Cu, 0.56% Pb and 0.62% Zn. The underground Mineral Resource Estimate totalled 6.9 Mt of Measured and Indicated Mineral Resources at 1.93 g/t Au, 25.86 g/t Ag, 0.40% Cu, 0.32% Pb and 3.71% Zn. Underground Inferred Mineral Resources totalled 0.9 Mt at 3.88 g/t Au, 51.21 g/t Ag, 0.47% Cu, 0.45% Pb and 1.40% Zn.

Table 1.3 Back Forty Mineral Resource Estimate (1-7)															
Total Mineral Resource Estimate															
Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Flotable	Met1	Measured	12.4 +62.5	4,735	2.10	319.2	16.07	2,446.7	0.32	33.0	0.11	11.1	4.85	506.0
		Indicated		3,907	2.01	253.1	21.60	2,714.1	0.36	30.8	0.18	15.9	3.28	282.2
		M+I		**8,643**	**2.06**	**572.3**	**18.57**	**5,160.8**	**0.33**	**63.8**	**0.14**	**27.0**	**4.14**	**788.2**
		Inferred		373	3.26	39.1	38.77	465.2	0.55	4.5	0.41	3.4	1.02	8.4
	Met2	Measured	12.0 +62.0	428	1.96	26.9	73.43	1,010.5	2.42	22.8	0.02	0.2	0.11	1.0
		Indicated		156	2.82	14.2	62.00	311.5	1.33	4.6	0.04	0.1	0.08	0.3
		M+I		**584**	**2.19**	**41.1**	**70.37**	**1,322.0**	**2.13**	**27.4**	**0.02**	**0.3**	**0.10**	**1.3**
	Met3	Measured	12.0 +62.0	206	1.89	12.5	15.25	100.9	0.48	2.2	0.03	0.1	0.13	0.6
		Indicated		521	2.37	39.7	19.57	328.0	0.46	5.3	0.09	1.1	0.23	2.6
		M+I		**727**	**2.23**	**52.2**	**18.35**	**428.9**	**0.46**	**7.4**	**0.08**	**1.2**	**0.20**	**3.2**
		Inferred		65	5.19	10.9	30.89	64.6	0.35	0.5	0.25	0.4	0.20	0.3
	Met4	Measured	12.0 +62.0	232	1.13	8.4	30.24	225.9	0.55	2.8	0.54	2.8	6.43	33.0
		Indicated		1,802	1.30	75.3	24.65	1,428.3	0.53	21.2	0.41	16.4	5.53	219.6
		M+I		**2,035**	**1.28**	**83.8**	**25.28**	**1,654.1**	**0.54**	**24.0**	**0.43**	**19.2**	**5.63**	**252.6**
		Inferred		273	1.45	12.7	20.61	180.9	0.73	4.4	0.15	0.9	2.21	13.3
	Met5	Measured	13.1 +63.1	2,236	0.82	58.7	12.66	909.8	0.02	1.1	0.33	16.0	1.25	61.7
		Indicated		1,653	1.14	60.5	31.50	1,674.1	0.03	1.2	0.75	27.4	2.60	94.7
		M+I		**3,889**	**0.95**	**119.2**	**20.67**	**2,583.9**	**0.03**	**2.4**	**0.51**	**43.4**	**1.83**	**156.5**
		Inferred		99	3.02	9.6	121.26	387.5	0.05	0.1	1.09	2.4	3.30	7.2
	Sub Total	Measured	12.0 +12.4 +13.1 +62.0 +62.4	7,838	1.69	425.8	18.63	4,693.7	0.36	61.9	0.17	30.1	3.49	602.4
		Indicated		8,040	1.71	442.8	24.97	6,455.9	0.36	63.1	0.34	61.0	3.38	599.4
		M+I		**15,878**	**1.70**	**868.6**	**21.84**	**11,149.7**	**0.36**	**125.0**	**0.26**	**91.1**	**3.43**	**1,201.8**
		Inferred		811	2.78	72.4	42.14	1,098.2	0.53	9.5	0.39	7.0	1.64	29.2

TABLE 1.3
BACK FORTY MINERAL RESOURCE ESTIMATE (1-7)

TOTAL MINERAL RESOURCE ESTIMATE

Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
			+63.1											
Leachable	Met6	Measured	21.4 +71.4	607	5.76	112.3	37.73	735.9	0.05	0.7	0.13	1.7	0.20	2.7
		Indicated		1,786	2.26	129.5	39.47	2,267.0	0.04	1.6	0.28	11.0	0.41	16.0
		M+I		**2,393**	**3.14**	**241.8**	**39.03**	**3,003.0**	**0.04**	**2.3**	**0.24**	**12.7**	**0.35**	**18.7**
		Inferred		384	5.69	70.2	64.26	792.9	0.07	0.6	0.65	5.5	0.37	3.1
Total		Measured	12.0 +12.4 +13.1 +21.4 +62.0 +62.4 +63.1 +71.4	8,444	1.98	538.1	20.00	5,429.7	0.34	62.6	0.17	31.8	3.25	605.0
		Indicated		9,827	1.81	572.4	27.61	8,722.9	0.30	64.7	0.33	72.0	2.84	615.4
		M+I		**18,271**	**1.89**	**1,110.4**	**24.09**	**14,152.6**	**0.32**	**127.3**	**0.26**	**103.8**	**3.03**	**1,220.5**
		Inferred		1,194	3.71	142.5	49.24	1,891.2	0.38	10.1	0.47	12.5	1.23	32.3

Notes:

1) *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*
2) *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
3) *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4) *The Mineral Resources in this Technical Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
5) *Metallurgical type Oxide (all gold domains and leachable Gossans) is leachable, while all other metallurgical types are flotable.*
6) *The Mineral Resource Estimate was based on metal prices of US$1,375/oz gold, US$22.27/oz silver, US$1.10/lb zinc, US$3.19/lb copper and US$1.15/lb lead.*
7) *Open pit Mineral Resources were defined within the constraining pit design as per the 2018 Feasibility Study.*

1.12 MINING METHODS

The Back Forty mine plan presented in this Preliminary Economic Assessment is based on mining the highest value material as soon as possible and treating this material through the process plants to maximize cash flow. This strategy is achieved by mining the mineralized material and either feeding the material directly to the process plant or stockpiling the material onsite for processing later per a feed schedule based on optimal economics for the operation. The mine plan consists of a combined open pit and underground mining operation. Open pit mining will take place from Year 1 to Year 5. Underground development will be initiated in Year 5 and underground production mining will continue to Year 11.

A number of stockpiles, by material type, will facilitate the accelerated processing of higher-grade material and also manage fluctuations in process plant feed delivery from the two mining operations.

The Back Forty Project area consists of very subdued terrain and topography. The area, topography and climate are amenable to the conventional open pit mining operations proposed for the Project. The open pit mining operation will encompass a single open pit that will be mined with conventional mining equipment in three pushback phases. The underground mine will be developed beneath the open pit with a single decline access point located part way down the open pit ramp.

1.12.1 Open Pit Mining

The open pit design is based on the 2018 Feasibility Study. Minor modifications were made to standardize on 5 m high benches with a quadruple (4) bench configuration, resulting in a 20 m vertical distance between catch berms. For scheduling purposes, the Back Forty pit was subdivided into three phases. Mining commences in a small higher-grade pit and then expands outwards by pushing back the pit wall. This enables annual waste stripping quantities to be distributed to avoid high and low annual tonnage fluctuations.

Open pit mining operations will be carried out by Company personnel except for blasting. A blasting contractor will be used to supply the explosives, prepare the blasts, charge the holes, fire the blast, and inspect the area post-blast. The equipment fleet will consist of hydraulic excavators and front-end wheel loaders, both with 8 m^3 buckets, and 90 t capacity haul trucks, plus track dozers, graders, and support equipment.

A summary of the open pit mining schedule is shown in Table 1.4. Mineralized material may be delivered either to the primary crushers or placed into one of the stockpiles. Waste rock is either taken to a waste rock storage facility or used in tailings dam construction. A six month pre-production period is planned.

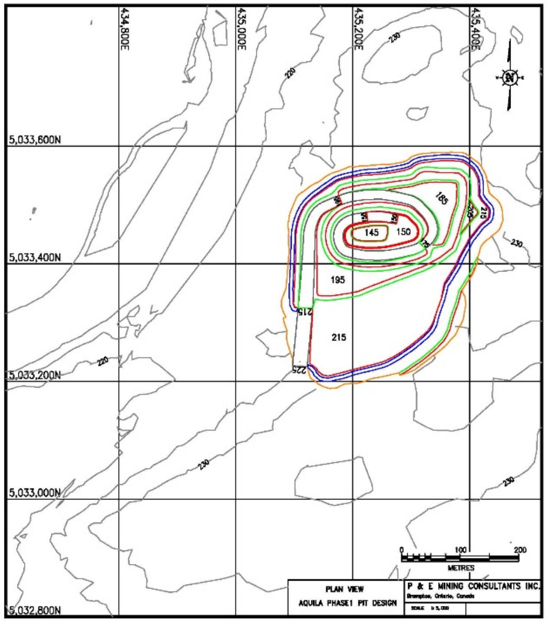
TABLE 1.4 OPEN PIT MINING SCHEDULE									
Type	**Units**	**Total**	**Year**						
			Y-1	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**
Overburden	kt	3,778	1,233	1,648	896	-	-	-
Waste Rock	kt	47,970	1,568	9,263	12,130	13,437	10,512	1,058
Total Waste	**kt**	**51,747**	**2,801**	**10,911**	**13,027**	**13,437**	**10,512**	**1,058**
Process Plant Feed Mining								
Total Sulphide	kt	8,815	73	2,236	1,647	1,406	2,678	776
Total Oxide	kt	1,317	126	353	327	157	309	45
Total Feed	**kt**	**10,132**	**199**	**2,589**	**1,974**	**1,563**	**2,987**	**821**
Total Material	**kt**	**61,880**	**3,000**	**13,500**	**15,000**	**15,000**	**13,500**	**1,879**
Strip ratio	w:o	5.1	14.1	4.2	6.6	8.6	3.5	1.3
Feed to Stockpiles	kt	6,961	199	1,995	1,609	575	1,953	629

Plan views of the three pit phases are shown in Figure 1.1. Mining will occur in several phases simultaneously in order to meet the requisite stripping and process plant feed delivery targets.

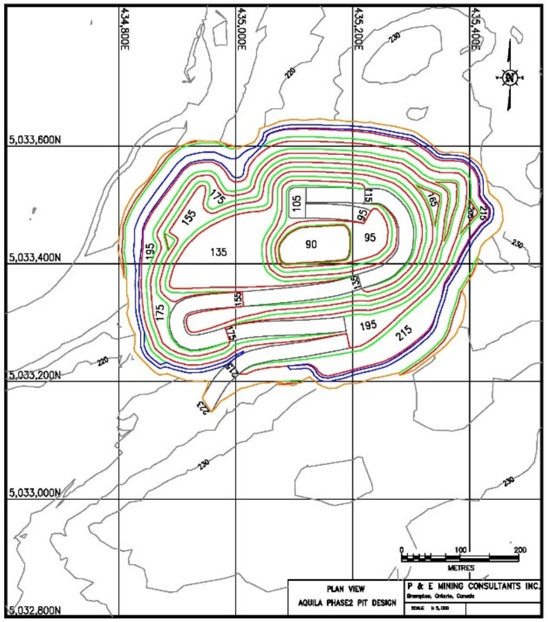
Mineralized material mining dilution is based on a selective mining unit ("SMU") model and is estimated at 22.3%, with 3% mining losses. Once excavated, the material is transported to either a stockpile or to one of the primary crushers, according to the material type (Main, Pinwheel, Tuff, Oxides).

FIGURE 1.1 OPEN PIT PHASES

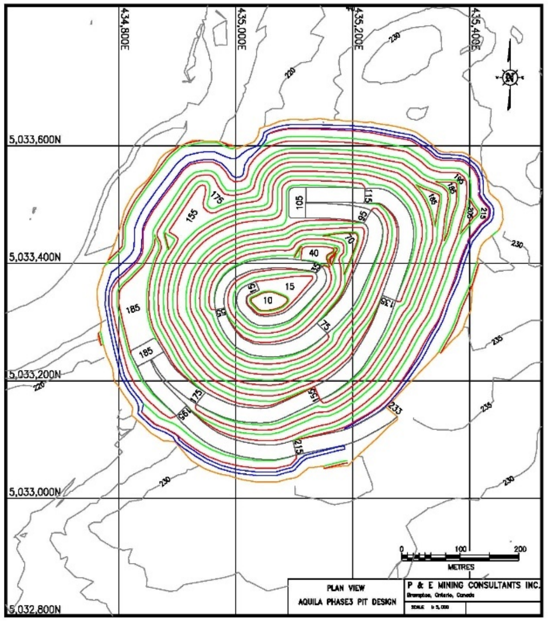
PHASE 1



PHASE 2



PHASE 3 (FINAL PIT)



1.12.2 Underground Mining

Extraction of the potentially economic portion of the underground Mineral Resource will be achieved by a combination of mechanized Cut and Fill ("CF") or Longhole ("LH") methods. CF mining is the dominant method, producing approximately 63% of mined tonnes, with LH producing the remaining 37% of tonnes. CF mining uses one of four stope sizes, and targets low-dipping material (dip less than 55°). LH mining uses one of two stope size subsets and orientations (transverse or longitudinal).

All waste and mineralized material development will be carried out using drill jumbos and mechanized bolting units, thus allowing for sharing of the equipment fleet between development and production assignments, allowing crews and machinery to perform production and/or development tasks in nearby mining areas while limiting machinery travel distances. Mineralized material will be extracted from the CF and LH stopes using 9 t and 14 t load-haul-dump ("LHD") units and loaded into 40 t underground trucks for transport to surface.

Access to the Deposit is via a 5 m by 5 m ramp from surface, with the underground portal located on the 187.5 m pit bench. All development and production material from underground is hauled to, and dumped at, a portal stockpile. From the stockpile, open pit trucks will transport the material to its final destination. Backfilling of the stope areas is achieved through the use of Pastefill ("PF"), delivered via two boreholes from the surface PF Plant. PF varies from 3-7% cement by mass, depending on application: higher cement contents are used for artificial sill pillars, lower cement contents are used otherwise. The PF system has a planned capacity of 2,300 tpd and the PF Plant is to be operated for 16-18 hours per day on average. All stoping areas are planned to be filled with pastefill.

The underground construction and development commences in Q1 of Year 5, with production beginning at the start of Q3 of Year 5. Commercial production is achieved midway through Q4 of Year 6. The production rate of the underground varies depending on development requirements, with a nominal commercial production rate of 2,300 tpd, increasing to a maximum of 3,200 tpd in Year 7, before decreasing slightly towards the end of mine life in Year 9 as CF mining areas are exhausted and the mine transitions to lower-value LH stopes. LH mining for the Back Forty Deposit uses a nominal 25 m floor-to-floor sublevel spacing, with 5 m drift heights.

The underground mine is equipped with a high-capacity pumping system capable of moving 109 L/s to surface if necessary. Ventilation is provided via three powered fresh air raises, with the portal and a single unpowered return air raise for exhaust. Electrical power is supplied initially at 15 kV, with step-down transformers distributed throughout the mine. The mine also has a small compressed air distribution system capable of providing 0.45 m^3/s at standard temperature and pressure.

Mining dilution is broken down into three types: Internal, External and Backfill. Average internal dilution is 13.6% by mass, average external dilution is 6.3% by mass, and average backfill dilution is 4.4% by mass. Overall mining recovery on a tonne-weighted basis is expected to be 93.4%.

The total mined and recovered portion of the underground Mineral Resource comprises 5,717 kt of mineralized material with an average Net Smelter Return ("NSR") value of US$109.24/t. Figure 1.2 presents a 3-D schematic of the underground mine layout at the end of the life-of-mine ("LOM"). The green stopes are active in the final year of mining, and the blue stoping areas are mined out and filled.

A total of 22,805 m of lateral development and 1,169 m of vertical development are required over the underground LOM.

FIGURE 1.2 UNDERGROUND MINE AT END OF LOM, 3-D SCHEMATIC

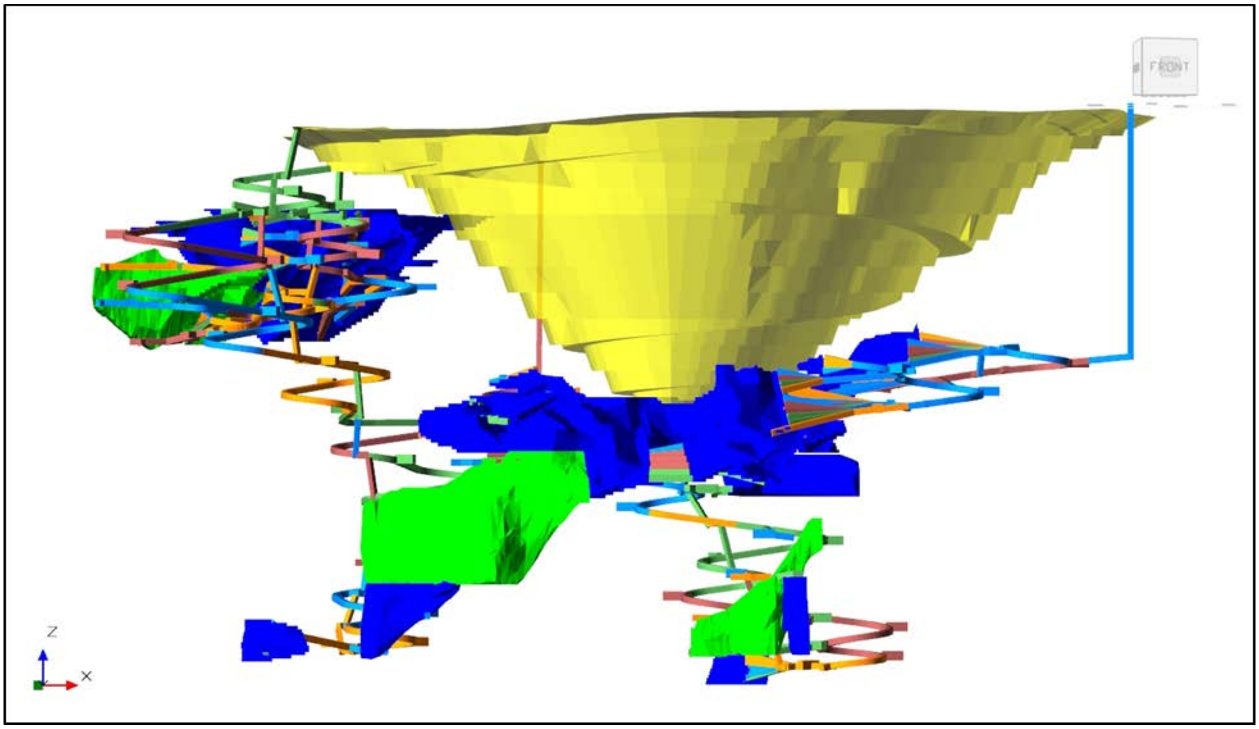


Table 1.5 shows the production tonnes from the Back Forty underground Mineral Resource by year and mining method. Units are in thousands of tonnes.

TABLE 1.5
PRODUCTION BY MINING TYPE BY YEAR (KT)

Type	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
LH	-	-	-	-	438	968	732	**2,138**
CF Type 1	-	98	503	520	268	-	-	**1,389**
CF Type 2	119	551	558	536	232	-	-	**1,996**
CF Type 3	1	18	43	47	13	-	-	**122**
CF Type 4	1	16	22	24	8	-	-	**72**
Total	**122**	**683**	**1,126**	**1,126**	**959**	**968**	**732**	**5,717**

Note: *CF1 = 7.5 m x 5.0 m, CF2 = 5.0 m x 5.0 m, CF3 = 4.0 m x 2.5 m, CF4 = 5.0 m x 2.5 m, Width x Height.*

1.13 PROCESS PLANT

Oxide mineralized material and sulphide mineralized material (Main, Pinwheel and Tuff material) are treated through separate process plants.

The oxide mineralized material will be processed via a cyanidation leach circuit to produce doré. Depending on the grades of copper, zinc and lead, the sulphide mineralized material will be processed via two stages of flotation to produce concentrates, i.e. either a copper and zinc concentrate, or a lead and zinc concentrate.

Sulphide mineralized material will be processed on a campaign basis based on the main material types that have a similar metallurgical response. As such the design of the sulphide plant is based on a flexible metallurgical flowsheet to process the main material types.

The oxide and sulphide flowsheets are based on proven unit operations in the industry.

The oxide plant has been designed for a throughput of 350 tpd (dry) at head grades of up to 8.0 g/t Au and 127 g/t Ag. The overall flowsheet includes the following steps:

- Three stage crushing using an open circuit jaw crusher, open-circuit secondary cone crusher and closed-circuit tertiary cone crusher.
- Grinding and classification.
- Pre-leach thickening.
- Cyanide leach.
- Vacuum filtration of leaching tailings.
- SART.
- Carbon-in-Column (“CIC”) gold adsorption.
- Carbon acid-washing, desorption and recovery (“ADR”).
- Smelting to produce doré.
- Cyanide destruction of the final wash filtrate from the vacuum filtration step.
- Tailings repulping and disposal to the Tailings Management Facility (“TMF”).

The sulphide plant has been designed for a nominal throughput of 2,800 tpd (dry), with varying copper, lead and zinc head grades. The overall flowsheet includes the following steps:

- Primary crushing.
- Coarse mineralized material stockpile and reclaim.
- Grinding and classification.
- Gravity concentration.
- Bulk rougher flotation to produce copper concentrate or lead concentrate depending on mineralized material campaign.
- Zinc rougher flotation.
- Bulk concentrate regrind (copper or lead concentrate).
- Zinc concentrate regrind.
- Bulk cleaner flotation, using three stages of cleaning (copper or lead concentrate).
- Zinc cleaner flotation, using two stages of cleaning.
- Bulk concentrate thickening and filtration (copper or lead concentrate).
- Zinc concentrate thickening and filtration.

- Tailings thickening and disposal in the common TMF.

Figure 1.3 presents an overall block flow diagram depicting the major unit operations incorporated in the selected process flowsheet.

The process plants will receive material based on a processing schedule where material will either come from the mine directly or from stockpiled material that was mined earlier.

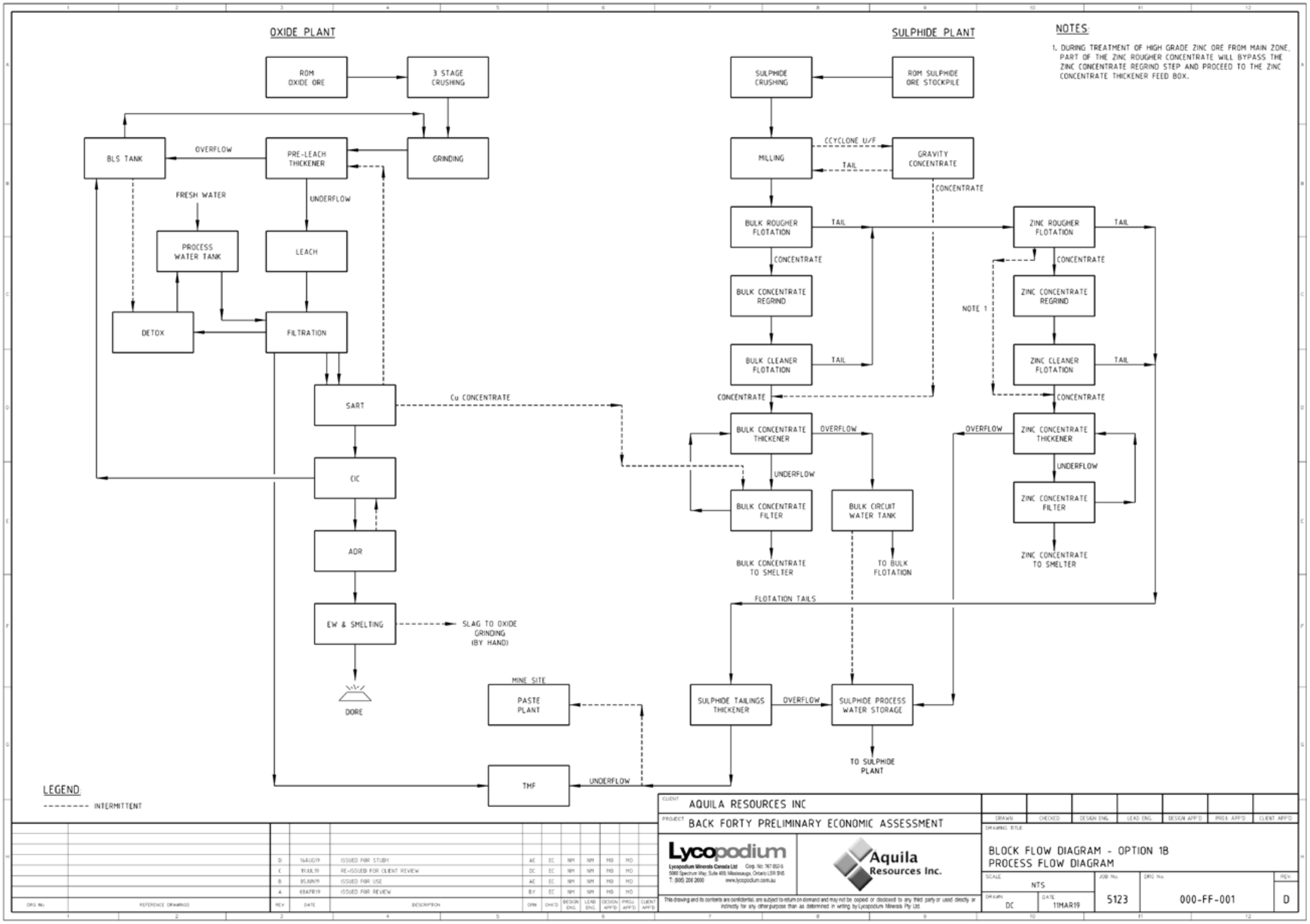
Stockpiled material will be stored according to the main material types (not blended) that have a similar metallurgical response in the plant, i.e. Main, Pinwheel, Tuff, and Oxide. Oxide material will be constantly fed to the Oxide Plant at 350 tpd. Depending on the processing schedule, sulphide material will be fed constantly per material type on a campaign basis to the Sulphide Plant at 2,800 tpd (Tuff), 3,500 tpd (Main) or 3,440 tpd (Pinwheel).

For stable process plant operations, the processing schedule has a minimum of one-month campaigning on a material type. When the feed material is changed from Main to Pinwheel and vice versa, the sulphide plant parameters are adjusted according to the metallurgical requirements and are considered relatively minor in nature. When the feed material is changed from Main or Pinwheel to Tuff and vice versa, then a complete clean out of the bulk flotation circuit is required to prevent contamination of the final concentrates.

Design parameters for the comminution circuit were sourced from testwork conducted at SGS during 2015 and 2017. Mineralized material characterization and comminution modelling was completed based on this testwork.

Aquila elected to pursue the vacuum filtration, SART and carbon adsorption flowsheet instead of the Merrill Crowe recovery circuit for this PEA. The primary driver for this decision is the need to remove copper from the circuit with a view to improving the quality of the doré bars. SART also allows for the removal of mercury and silver from the pregnant leach solution to further improve the quality of the doré product. The recovery of free cyanide and cyanide bound as weak acid dissociable metal complexes is expected to improve the economics of this flowsheet.

FIGURE 1.3 OVERALL PROCESS PLANT BLOCK FLOW DIAGRAM



Source: *Lycopodium Minerals Canada Ltd. (2019)*

1.14 SITE INFRASTRUCTURE

The overall site plan is shown in Figure 1.4 and includes major facilities of the Project including the open pit mine, mineralized material stockpiles, oxide and sulphide processing plants, TMF, waste rock facilities (“WRF”), cut-off wall (“COW”), contact water basin (“CWB”), non-contact water basins (“NCWB”), waste water treatment plant (“WWTP”), mine services, overburden stockpile and access road.

Prior to commencing underground mining, a paste backfill plant will be installed near the open pit mine to provide cemented paste for backfill requirements.

Access to the Project is from the east side of the Property off the existing County Road 356. Main access will be via the main security gate near the process plant.

Grid power will be provided from an incoming 138 kV high voltage (“HV”) line from the east side of the Project along the main access road.

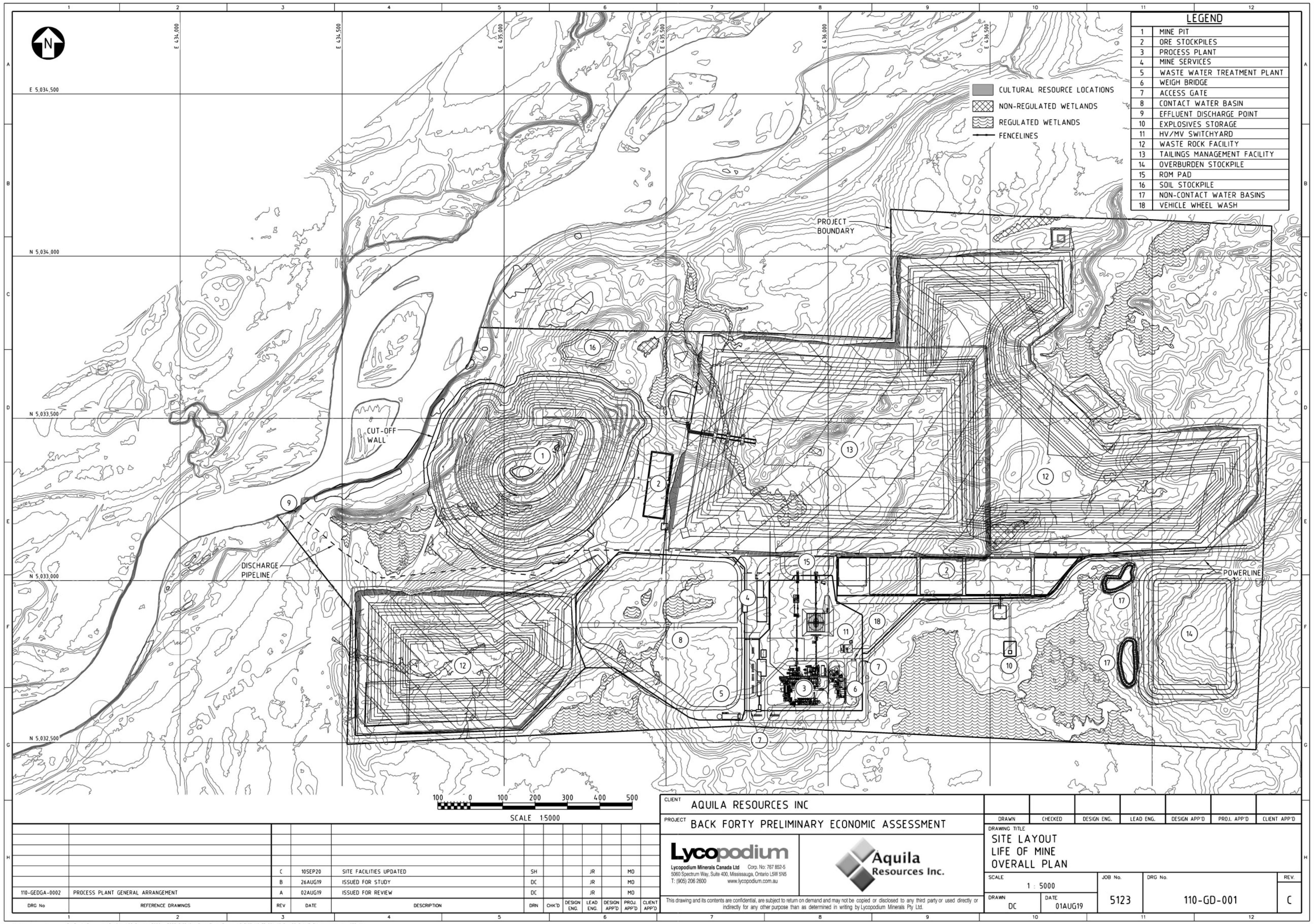
The site will be fenced to clearly delineate the Project area and deter access by unauthorized people.

1.15 MARKET STUDIES AND CONTRACTS

There are no material contracts or agreements in place as of the effective date of this Technical Report.

Statistics for metal markets have been taken from September 2019 analysis by BMO Capital Markets. Statistics for concentrate markets have been provided by Ocean Partners, who are specialist base metal concentrate traders.

FIGURE 1.4 OVERALL SITE PLAN



Source: *Lycopodium Minerals Canada Ltd. (2019)*

1.16 ENVIRONMENTAL STUDIES, PERMITS AND SOCIAL OR COMMUNITY IMPACT

Aquila currently holds several permits as required in Michigan's environmental regulations. The current permits that have been issued for the Back Forty Project include:

- Part 632 Mining Permit (MP 01 2016) for mining and beneficiation activities associated with the Back Forty Deposit.
- Part 632 Mining Permit (MP 01 2016) Amendment.
- National Pollutant Discharge Elimination System ("NPDES") Permit (MI0059945) for treated process wastewaters.
- Michigan Air Use Permit to Install ("PTI") (205-15) has been issued for the Project for emissions associated with construction and mining activities.
- PTI 205-15 Modification for the updated facilities.
- Part 301 Inland Lakes and Streams and Part 303 Wetlands Protection Permit (WRP011785).

In addition the permits listed above, a Dam Safety Permit for the CWB and TMF will be obtained prior to the construction and operation of the Back Forty Mine.

Wetlands have been extensively studied at the site and are documented in the MDEQ/USACE Joint Permit Application for: Wetland Protection; Inland Lakes and Streams; Floodplain (Foth et al., 2017). Wetlands of various sizes and classification encroach across the site. Although the mine and processing facilities have been located in a compact area, every effort has been made to avoid and minimize wetland impacts.

Aquatic surveys and assessments address aquatic biota and their habitats in the Menominee River, Shakey River, and Shakey Lakes systems. Original baseline sampling documented in the original Mining Permit Application (Foth, 2015) and Mining Permit Amendment Application (Foth, 2018) provides an understanding of presence and species of aquatic biota in and around the Project site. Prior to commencement of construction, additional baseline sampling is required under the Mining Permit. With understanding of aquatics provided by the original survey, an additional aquatics preconstruction survey is proposed.

On-going terrestrial flora monitoring to confirm baseline conditions, and address trends during construction and operations include annual observations of plant species along transects. Meander surveys through upland habitats and surveys within the established transects address the scope of upland vegetative surveys.

Terrestrial wildlife monitoring during the Project operations will include annual and semi-annual observations of amphibians, reptiles, birds, and mammals at designated survey sites previously studied. Observations will be documented and included with the Project's annual report. The fauna monitoring will be completed to confirm baseline conditions and document the trends and conditions of these resources during operations.

Over the life of the mine, the data and observations will be documented by qualified professionals and will assist in identifying trends in biota in and around the site. These trends, along with other media data such as groundwater and surface water quality and hydrologic parameters, will be used to evaluate whether an observed trend is related to the Project.

Undeveloped areas, such as the Project area, have very good air quality. The largest city with industrial activity is Menominee, Michigan – Marinette, Wisconsin, an area 30 miles south of the site with a combined population of approximately 20,000.

The Property and proposed development area have been investigated for the existence of cultural resources, and historical artefacts have been documented.

Feedback to this Project is continually requested and submitted by the public as part of Aquila's ongoing efforts to engage in operational transparency and information sharing with the local community and other affected stakeholders. This engagement strategy dates to almost a decade ago under previous Project proponents. The Back Forty Project represents many employment opportunities that have attracted interest of unions and business groups. As such, the Project has seen strong support from legislators and regulators. Aquila has developed a good working relationship with the Upper Peninsula Construction Council to ensure availability of local skilled labour.

Tribal engagement has been very important to the Project, especially considering the cultural resources near the site. Aquila plans to continue working with the Tribes to better understand their concerns and to find opportunities to work together on issues that are important to both parties such as communication on the preservation of and unanticipated discovery plan of historical artifacts.

Currently, there are four ongoing legal actions involving the Menominee Tribe, regarding the wetland and mine permits.

1.17 CAPITAL COSTS

1.17.1 Initial Capital Costs

The initial capital cost estimate for the Project is summarized in Table 1.6 by major area.

All costs are expressed in United States Dollars unless otherwise stated and are based on Q3 2019 pricing and deemed to have an overall accuracy of ±25%. The capital cost estimate conforms to Association for the Advancement of Cost Engineering International ("AACEI") Class 4 estimate standards as prescribed in recommended practice 47R11.

The initial capital cost estimate was based on an overall engineering, procurement and construction management ("EPCM") implementation approach and horizontal (discipline based) construction contract packaging. Equipment pricing was based on a combination of budget quotations and actual equipment costs from recent similar Lycopodium and P&E projects considered to be representative of the Project.

TABLE 1.6 CAPITAL ESTIMATE SUMMARY BY AREA (Q3 2019, ±25%)	
Item	**Capital Costs ($M)**
Construction Indirects	11.4
Oxide Plant	24.1
Sulphide Plant	57.5
TMF/WRFs	42.6
Infrastructure	34.2
Mining	23.6
EPCM	15.7
Owner costs	11.4
Subtotal	**220.6**
Contingency	29.9
Total	**250.4**

1.17.2 Sustaining Capital Costs

Capital expenditures for open pit mining incurred after Year -1 are considered sustaining capital and are estimated at $45.9M in Table 1.7. The majority of the sustaining capital consists of capital lease payments for the mining equipment. Given the life of the open pit, no equipment replacements are planned.

TABLE 1.7
OPEN PIT MINE SUSTAINING CAPITAL COSTS ($K)

Item	**Total ($k)**	**Year**						
		1	**2**	**3**	**4**	**5**	**6**	**7**
Equipment and Down Payments	**851**	851	0	0	0	0	0	0
Equipment Capital Leases	**39,824**	6,769	8,819	8,819	8,820	4,380	2,134	83
Mine Development	**3,228**	2,956	163	110	0	0	0	0
Freight and Spares	**2,034**	381	441	441	441	219	107	4
Total Mine Sustaining Capital	**45,937**	**10,956**	**9,423**	**9,370**	**9,261**	**4,599**	**2,241**	**87**

Initial capital costs for the underground mine are treated as sustaining capital costs for the Back Forty Project since open pit mining will be well underway by the time the underground mine is developed. Sustaining capital costs also include all costs associated with infrastructure, capital waste development (vertical and lateral), relevant equipment leasing costs (downpayments, legal fees, origination costs and mobilization costs), and the paste backfill plant. Total sustaining capital costs are estimated at $98.9M, as shown in Table 1.8.

TABLE 1.8
UNDERGROUND SUSTAINING CAPITAL COSTS ($K)

Item	Total ($k)	Year							
		4	5	6	7	8	9	10	11
Infrastructure	19,763	250	4,471	7,889	1,063	5,875	214	1	0
Equipment	32,198	100	9,011	9,990	9,241	3,265	542	0	50
Development	31,984	0	9,314	11,120	6,942	3,772	769	68	0
Paste Plant	15,000	15,000	0	0	0	0	0	0	0
Total	**98,946**	**15,350**	**22,796**	**28,999**	**17,246**	**12,912**	**1,524**	**69**	**50**

Other Project sustaining capital costs include subsequent TMF stage raises over the LOM and plant annual capital expenditures. The sustaining capital schedule over the life of mine is estimated at $69.3M as shown in Table 1.9.

TABLE 1.9
PROJECT SUSTAINING CAPITAL COSTS ($K)

Item	Total ($k)	Year				
		1	2	3	4	5
Cut-off Wall	**4,667**	4,667	0	0	0	0
TMF	**28,690**	17,039	4,580	5,257	0	1,813
SWRF	**9,233**	9,233	0	0	0	0
NWRF	**31,524**	10,616	20,907	0	0	0
Total Project Sustaining Costs	**69,320**	**38,728**	**23,623**	**5,155**	**0**	**1,813**

Mine closure costs, salvage value and rehabilitation costs are estimated at $75M.

A key aspect of mine closure is the backfilling of the open pit with waste rock. In addition, capping of the TMF is required along with topsoil placement in preparation for re-vegetation. These earthworks will occur during the last few years of the operation and extend two year beyond the end of processing. The total cost is estimated at $55M.

1.18 OPERATING COSTS

LOM operating costs are presented in Table 1.10.

TABLE 1.10
LOM OPERATING COSTS

Item	Total Cost ($ M)	Unit	Average Unit Cost

Open pit mining		$/t mined	3.03
Underground mining		$/t mined	50.31
Open pit mining	178	$/t processed	11.2
Underground mining	288	$/t processed	18.2
Process plant	310	$/t processed	19.5
G&A	46	$/t processed	2.9
Total	**821**	**$/t processed**	**51.8**

1.19 FINANCIAL EVALUATION

A Preliminary Economic Assessment of the Project has been conducted using an after-tax cash flow model. The model was structured using an EXCEL workbook. The economic analysis is presented for two macro-economic cases that are summarized in Table 1.11. The Base Case uses current (June 2020) consensus long term forecast metal prices, while the Spot Case uses prices at the time of writing (July 9, 2020).

The PEA was prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). Readers are cautioned that the PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

Input data was provided from a variety of sources, including the various consultants' contributions to this PEA, pricing obtained from external suppliers and contractors, and foreign exchange rates and Project specific financial data such as the expected taxation regime were received from Aquila.

TABLE 1.11
SUMMARY METRICS

Area	Item	Units	Base Case[1]	Spot Price[2]
Process Production	Total Process Feed	Mt	15.9	15.9
	Grade	g/t AuEq[4]	4.2	3.7
	Total Recovery and Payability	% of contained AuEq	74.3	73.4
	Payable Gold	koz Au	692	692
	Payable Gold Equivalent	koz AuEq	1,543	1,323
	Annual Gold Equivalent	koz AuEq	128	110
	Life of Mine	Years	12	12
	Throughput	tpd	Nominal 2,800 sulphides + 350 oxides	

Table 1.11 Summary Metrics				
Area	**Item**	**Units**	**Base Case[1]**	**Spot Price[2]**
	Total Tailings	Mt	14.4	14.4
Metal Price Deck	Gold	$/oz	1,485	1,998
	Zinc	$/lb	1.08	1.04
	Copper	$/lb	3.05	2.92
	Silver	$/oz	18.20	25.00
	Lead	$/lb	0.91	0.83
Revenue and OPEX	Gross Revenue	US$/t process feed	132	149
	NSR	US$/t process feed	113	130
	Total Site Opex	US$/t process feed	52	52
	Royalties	% of NSR	2.0	2.1
	EBITDA	US$/t process feed	59	75
	EBITDA margin	EBITDA / NSR	52	58
	C1 Cash Costs (co-product)[3]	US$/oz AuEq	733	854
	C1 Cash Costs (by-product)[3]	US$/oz Au	(82)	(29)
CAPEX	Initial Capital	US$ M	250	250
	Sustaining Capital	US$ M	214	214
	AISC (co-product)[3]	US$/oz AuEq	926	1,078
	AISC (by-product)[3]	US$/oz Au	397	462
Unlevered Returns[5]	Pre-Tax NPV 6% discount rate	US$ M	248	430
	Pre-Tax IRR	%	31.6	45.4
	Post-Tax NPV 6% discount rate	US$ M	176	316
	Post-Tax IRR	%	26.1	37.8
	Post-Tax Payback	Years	2.4	1.6

Notes:

1) *The Base Case macro-economic forecast assumes flat pricing that has been drawn from the consensus long term estimates of select banks as of August 4, 2020.*
2) *As at August 4, 2020.*
3) *C1 cash costs, which are intended to measure direct cash costs of producing paid metal, include all direct costs that would generate payable recoveries of metals for sale to customers, including mining of mineralized materials and waste, leaching, processing, refining and transportation costs, on-site administrative costs and royalties, net of by-product credits. C1 cash costs do not include depreciation, depletion, amortization, exploration expenditures, reclamation and remediation costs, sustaining capital, financing costs, income taxes, or corporate general and administrative costs not directly or indirectly related to the Project. C1 cash costs are divided by the number of ounces of gold estimated to be produced for the period to arrive at cash costs per gold ounce produced. AISC includes C1 cash costs, as defined above, plus exploration costs at the Project and sustaining capital expenditures (including additional tailings storage, permitting and customary improvements to the operations over the life of the Project). AISC is divided by the number of ounces of gold estimated to be produced for the period to arrive at AISC per gold ounce produced. EBITDA is earnings before interest, taxes, depreciation, and amortization.*
4) *Gold equivalent ounces were determined by calculating the total value of metals contained or produced and dividing that number by the gold price ($1,485/oz gold Base Case or $1,998/oz gold Spot Case). As the denominator is higher in the Spot Case, the gold equivalent is lower than at Base Case prices. Gold equivalent grade is calculated by dividing the number of gold equivalent ounces by the Mineral Resource size (tonnes).*

5) Project economics reflect the Company's gold and silver streaming agreements with Osisko Gold Royalties (see Aquila press release dated June 18, 2020). The PEA financial model includes $30 million of initial payments under the gold stream to be received during the design and construction period. The 2018 Feasibility Study did not include the impact of the gold streaming agreement.

Commercial terms for concentrate and doré have been based on guidance from the specialist metals traders, Ocean Partners, and are as follows:

Copper Concentrate

- The most cost-effective destination for concentrate treatment is in Eastern Canada, with a total cost of transport of approximately $52/t. Note that this cost, and transport costs discussed below, includes trucking of concentrate from the mine site to a rail head, rehandle then rail transportation to the final destination. The second lowest cost, in Western USA, would have an associated cost approximately $60/t higher.

- Concentrate grade would be adjusted to target the optimal economics per material type, ranging between 17 - 22% Cu and with an average of 18.5%. Copper payables would be calculated on a one unit deduction to a maximum of 96.5% and would average 94.1%. Treatment charges would include a base rate of $80/t, with penalties ranging from $4 - $10/t by material type (for mercury content) and average approximately $7/t. Refining charges would be $0.08/lb payable Cu.

- The grades of by-product Au and Ag would average 57 g/t and 738 g/t, respectively. These high grades would be expected to make Back 40 copper concentrate desirable and allow maximum payables of 96.3% Au and 90% Ag to be achieved. Refining charges would be $6/oz Au and $0.50/oz Ag.

Zinc Concentrate

- The most cost-effective destination for concentrate treatment is in Eastern Canada, with a total cost of transport of approximately $62/t. At present, this facility does not pay for precious metals though there has been discussion regarding addition of a circuit to recover these. Facilities that do pay for precious metals located in Western Canada or Europe have a transportation cost premium of approximately $70/t. Note that transport costs to Europe also include rehandle of concentrate at a port and shipping to the final destination.

- Concentrate grade would be adjusted to target the optimal economics per material type, ranging between 50 - 55% Cu and with an average of 53.9%. Zinc payables would be calculated on an eight unit deduction to a maximum of 85% and would average 84.8%. Treatment charges would include a base rate varying by material type from $200 - $220/t, with penalties ranging from $5 - $8/t by material type (for mercury, iron and silica content) and average $209/t. For the assumed long term zinc price of $1.09/lb, the standard escalation clause would result in a further charge of $8/t. There are no refining charges for zinc.

- Facilities in Western Canada, Europe, Korea and Japan currently pay for by-product gold and silver in excess of 1 g/t and 3 oz/t, respectively. There are no refining charges for by-product precious metals. Over the life of mine, approximately 78% of zinc concentrate would contain potentially payable levels of by-product precious metals, with 90% of potentially payable by-products contained in 55% of total concentrate and 50% of potentially payable by-products contained in just 20% of total concentrate. It is possible that zinc concentrate would be shipped to multiple destinations to optimize shipping costs and precious metals realizations.

Lead Concentrate

- Lead concentrate would be shipped to Western Canada, with a total cost of transport of approximately $136/t.

- Concentrate grade would average 35% Pb. Lead payables would be calculated on a three unit deduction to a maximum of 95% and would average 91.4%. Treatment charges would include a base rate of $160/t, with penalties for mercury content of $3/t. There are no refining charges for lead.

- Average by-product grades of 63 g/t Au and 1,183 g/t Ag would attract the maximum level of payability of 95%. Refining charges would be $20/oz Au and $1/oz Ag.

Doré

- Doré would command payables of 99.9% Au and 99% Ag.

- Freight costs would total $15k/t doré, while smelter charges would be $8k/t. Total charges would equate to $0.82/oz Au.

Aquila previously sold a stream that will comprise 85% of future silver production. The commercial arrangements associated with the stream included initial payments totalling $17.25M and a further $4/oz for silver delivered into the stream. The financial model does not include the initial payments as inflows since these have already been received.

Aquila also previously sold a stream that will comprise 18.5% of gold production to a cap of 105 koz into the stream (or approximately 568 koz total production). Thereafter, the stream reduces to 9.25% of total production. Over the life of mine, gold delivered into the stream is forecast to total 116 koz or 16.8% of total production. Gold stream payments included phased initial payments of $55M, of which $15M has been received to date and an additional $2.5M will be received prior to construction. The model reflects the final $30M deposit as an inflow during the construction period. The stream also makes provision for payment of 30% of the spot price, to a maximum of $600/oz, for gold delivered into the stream.

The streams are omitted from the calculation of tax obligations, with pre-tax revenues calculated based on the entirety of production sold at forecast spot prices.

Returns are most sensitive to gold and zinc prices, with a ±15% movement in prices having a 38 - 41% relative impact to the NPV. The impact of similar variation in copper prices is less than one third as much at 12%. Returns are relatively insensitive to variation in silver or lead prices. Project economics remain viable even with the entire suite of metals at 85% of the assumed long term price, with the Project generating an 8.0% IRR. Under the more optimistic pricing scenario, simple pay back could be achieved within 20 months.

1.20 CONCLUSIONS AND RECOMMENDATIONS

Based on the work undertaken to date, as summarized in this Technical Report, and the individual Qualified Persons conclusions listed in Section 25, the PEA has identified a viable future underground mining option for the Project.

Subject to ongoing Project funding and board approval, it is recommended that Aquila advance the PEA concepts and commence a Feasibility Study update phase including additional studies and site investigations set out in Section 26 at an estimated work program budget of $4M.

2.0 INTRODUCTION AND TERMS OF REFERENCE

This report was prepared to provide a National Instrument ("NI") 43-101 Technical Report and Preliminary Economic Assessment ("PEA") for the polymetallic (zinc + gold + copper + silver + lead) volcanogenic massive sulphide ("VMS") Back Forty Deposit (the "Project" or "Property") located in Menominee County, Michigan, USA. The Back Forty Property is 100% owned by Aquila Resources Inc. ("Aquila" or the "Company").

This Technical Report was prepared by P&E Mining Consultants Inc. ("P&E") at the request of Mr. Andrew Boushy, Senior VP Projects, and is considered current as of October 14, 2019.

Aquila is a public, TSX listed, company trading under the symbol "AQA", with its head office located at:

141 Adelaide Street West, Suite 520
Toronto, Ontario
Canada
M5H 3L5
Telephone: 647-943-5672

Aquila's Back Forty Project is an open pit VMS deposit with underground potential located along the mineral-rich Penokean Volcanic Belt ("PVB") in Michigan's Upper Peninsula. The Project contains approximately 1.2 billion pounds of zinc and 1.1 million ounces of gold in the Measured and Indicated Mineral Resource classifications, with additional upside potential. A Feasibility Study on the Project was issued in September 2018 that studied open pit mining and on-site processing plants for oxide and sulphide material. This Technical Report considers underground mining in addition to open pit mining. Currently Aquila is working to secure the final permits required to build and operate the Back Forty Project.

The purpose of this Technical Report is to provide an independent, NI 43-101 Updated Mineral Resource Estimate and Preliminary Economic Assessment on the Back Forty Project. P&E understands that this Technical Report may be used for internal decision-making purposes and will be filed as required under TSX regulations. The Technical Report may also be used to support public equity financings.

The current P&E Updated Mineral Resource Estimate presented in this Technical Report has been prepared in full conformance and compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines" as referred to in NI 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects and in force as of the effective date of this Technical Report.

Mr. Yungang Wu, P.Geo., and Mr. Eugene Puritch, P.Eng., FEC, CET of P&E, each a Qualified Person under the terms of NI 43-101, conducted a site visit of the Property on May 23, 2016. Mr. Wu conducted a subsequent site visit on November 13-14, 2017. A data verification sampling program was conducted as part of each on-site review.

Mr. Neil Lincoln, P.Eng., of Lincoln Metallurgical Inc., a Qualified Person under the terms of NI 43-101, visited the Property July 12, 2016 where he observed the drill core at the core storage area, and toured the site of the proposed mine, process plant and infrastructure.

Mr. Kebreab Habte, P.Eng., of Golder Associates Ltd., a Qualified Person under the terms of NI 43-101, visited the Property June 28, 2016 to carry out a site reconnaissance survey to become familiar with the site layout, drainage conditions, subsurface conditions, and to identify potential geotechnical risks.

Mr. David Penswick, P.Eng., of Gibsonian Inc., a Qualified Person under the terms of NI 43-101, visited the Property November 2, 2017 where he reviewed and confirmed aspects impacting the financial valuation.

2.1 SOURCES OF INFORMATION

In addition to the site visits, P&E held discussions with technical personnel from the Company regarding all pertinent aspects of the Project and carried out a review of available literature and documented results concerning the Property, including internal Company technical reports and maps, published government reports, Company letters, memoranda, public disclosure and public information, as listed in the References at the conclusion of this Technical Report. Sections from reports authored by other participating consultants have been summarized in this Technical Report, and are so indicated where appropriate. Table 2.1 presents the authors and co-authors of each section of the Technical Report, who acting as a Qualified Person as defined by NI 43-101, take responsibility for those sections of the Technical Report as outlined in the "Certificate of Author" attached to this Report.

TABLE 2.1
REPORT AUTHORS AND CO-AUTHORS

Qualified Person	Employer	Technical Report Section Responsibility
Mr. Andrew Bradfield, P.Eng.	P&E Mining Consultants Inc.	2, 3, 15, 24 and Co-author 1, 25, 26
Ms. Jarita Barry, P.Geo.	P&E Mining Consultants Inc.	11 and Co-author 1, 12, 25, 26
Mr. David Burga, P.Geo.	P&E Mining Consultants Inc.	4, 5, 6, 7, 8, 9, 10, 23 and Co-author 1, 25, 26
Mr. Kebreab Habte, P.Eng.	Golder Associates Ltd.	Co-author 1, 18, 21, 25, 26
Mr. Kenneth Kuchling, P.Eng.	P&E Mining Consultants Inc.	Co-author 1, 16, 18, 21, 25, 26
Mr. Neil Lincoln, P.Eng.	Lincoln Metallurgical Inc.	13, 20 and Co-author 1, 25, 26
Dr. Manochehr Oliazadeh, P.Eng.	Lycopodium Minerals Canada Ltd.	17 and Co-author 1, 18, 21, 25, 26
Mr. David Penswick, P.Eng.	Gibsonian Inc.	19, 22 and Co-author 1, 25, 26
Mr. Eugene Puritch, P.Eng., FEC, CET	P&E Mining Consultants Inc.	Co-author 1, 12, 14, 25, 26
Mr. D. Gregory Robinson, P.Eng.	P&E Mining Consultants Inc.	Co-author 1, 16, 21, 25, 26

TABLE 2.1 REPORT AUTHORS AND CO-AUTHORS		
Qualified Person	**Employer**	**Technical Report Section Responsibility**
Mr. Yungang Wu, P.Geo.	P&E Mining Consultants Inc.	Co-author 1, 12, 14, 25, 26

This Technical Report was prepared in accordance with the requirements of NI 43-101 and in compliance with Form NI 43-101F1 of the Ontario Securities Commission ("OSC") and the Canadian Securities Administrators ("CSA"). The Mineral Resource Estimate was prepared in compliance with the CIM Definitions and Standards on Mineral Resources and Mineral Reserves that were in force as of the effective date of this Technical Report.

2.2 UNITS AND CURRENCY

Unless otherwise stated, all units used in this Report are metric. Gold ("Au") and silver ("Ag") assay values are reported in grams of metal per tonne ("g/t"). Zinc ("Zn"), lead ("Pb") and copper ("Cu") assay values are reported in percent metal weight content ("%").

Quantities are generally stated in Système International d'Unités ("SI") metric units including metric tons ("tonnes", "t") and kilograms ("kg") for weight, kilometres ("km") or metres ("m") for distance, hectares ("ha") for area, grams ("g") and grams per tonne ("g/t") for gold grades ("g/t Au"). Gold and silver grades may also be reported in parts per million ("ppm") or parts per billion ("ppb"). Metal values are reported in percentage ("%"), grams per metric tonne ("g/t") and parts per billion ("ppb"). Quantities of gold and silver may also be reported in troy ounces ("oz") and quantities of copper in avoirdupois pounds ("lb"). Copper, lead and zinc metal assays are reported in percent ("%") or parts per million ("ppm"), whereas gold and silver assay values are reported in grams of metal per tonne (g/t) unless ounces per short ton ("oz/T") are specifically stated. Abbreviations and terminology are summarized in Table 2.2.

The US dollar is used throughout this Report unless otherwise specified. All metal prices are stated in US dollars.

The coordinate system used by Aquila for locating and reporting drill hole information is the Universal Transverse Mercator coordinate system ("UTM"), the datum used is NAD83, zone 16N. The coordinates for the approximate centre of the Property are latitude 45° 27' N, longitude 87° 51' W. Maps in this Report use either the UTM coordinate system or latitude and longitude.

Table 2.2 sets out terminology and abbreviations used in this Technical Report.

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
$	dollar
$M	dollars, millions
$/t	$/tonne

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
%	percent
AA	atomic absorption
AAS	atomic absorption spectrometry
AACEI	Association for the Advancement of Cost Engineering International
Accurassay	Accurassay Lab
ACNC	American Copper and Nickel Company, Inc.
Actlabs	Activation Laboratories Ltd.
ADR	acid-washing, desorption and recovery
Ag	silver
Ai	abrasion index
ALS Chemex	ALS Chemex Labs, now ALS Minerals
amsl	above mean sea level
Aquila	Aquila Resources Inc.
ARC	Aquila Resources Corporation
ARD/ML	acid rock drainage and/or metal leaching
the Arrangement	January 16, 2014, REBgold Corporation and Aquila closed a statutory plan of arrangement
asl	above sea level
Au	gold
Baker Steel	Baker Steel Capital Managers LLP
BFJV	Back Forty Joint Venture LLC
Bureau Veritas	Bureau Veritas Mineral Laboratories USA
BV	bed volume
BWI	bond ball mill work index
°C	degree Celsius
CAD$	Canadian dollar
CaO	calcium oxide
$CaSO_4$	gypsum
CCRS	Commonwealth Cultural Resources Group, Inc.
CDA	Canadian Dam Association
CDN	CDN Resources Laboratory
CF	cut and fill
CHTF	chloritic crystal tuff
CIC	carbon in column
CIL	carbon in leach
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CIP	carbon in pulp
cm	centimetre(s)
CMP	Cyanide Management Plan
CN	cyanide

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
the COC Provision	the Project may elect to forgo the final payment, in which case the Threshold Stream Percentage and Tail Stream will be reduced to 9.5% and 4.75%, respectively
the Company	Aquila Resources Inc.
conc	concentrate
COW	cut-off wall
CRF	cemented rock fill
CRM	certified reference material
CSA	Canadian Securities Administrators
CSM	Cutter Soil Mixing
CSPT	Chinese Smelter Purchase Team
Cu	copper
CWB	contact water basin
CWI	crusher work index
DDH	diamond drill hole
the Deposit	Back Forty Deposit
dmt	dry metric tonne(s)
DNR	Michigan Department of Natural Resources
DWT	drop weight test
ECOG	economic cut-off grade
EIA	Environmental Impact Assessment
EIAA	Environmental Impact Assessment Amendment
EGLE	Environment Great Lakes and Energy
EM	electromagnetic
EPCM	engineering, procurement and construction management
EW	electrowinning
FAR	fresh air raise
FEL	front-end loader
FOS	factor of safety
ft	foot / feet
g	gram(s)
g/t	gram(s) per tonne
G&T	G&T Metallurgical Services Ltd.
GCL	geosynthetic clay liner
Golder	Golder Associates Ltd.
GOSS	gossan
GPS	global positioning system
ha	hectare(s)
HBF	horizontal belt filter
HCl	hydrochloric acid
HCN	hydrogen cyanide

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
HDPE	high-density polyethylene
HMI	Hudbay Michigan Inc.
HS PF	high-strength pastefill
Hudbay	Hudbay Minerals Inc.
HV	high voltage
IBC	intermediate bulk container
ICP-AES	inductively coupled plasma-atomic emission spectrometry
ID	inverse distance
ID^3	inverse distance cubed
ID^2	inverse distance squared
IDEA Drilling	Idea Drilling Company
Inspectorate	Inspectorate America Corporation, now Bureau Veritas Mineral Laboratories USA
IP	induced polarization
ISO	International Organization for Standardization
Jumbo	electric-hydraulic powered development drill jumbo, typically with one or two drill booms
JV	joint venture
k	thousand(s)
K_{80}	primary grind size of 83 µm
kg	kilogram(s)
Km	kilometre(s)
KP	Knight Piesold Ltd.
kW	kilowatt
kWh/t	kilowatt hours per tonne
ktpm	kilotonnes per month
L	litre(s)
L/s	litres per second
lb	pound (weight)
LCS	leachate collection system
LDS	leachate detection system
Level	mine working level referring to the nominal elevation (m RL), eg. 4285 level (mine workings at 4285 m RL)
LH	longhole
LHD	load, haul and dump unit (underground loader)
LLCS	leachate and leakage collection sump
LOM	life of mine
LRS	liquid resistance starter
LS PF	low-strength pastefill
Lycopodium	Lycopodium Minerals Canada Ltd.
m	metre(s)

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
m^3	cubic metre(s)
Ma	millions of years
Mag	magnetic
MASU or MS	massive sulphide
max.	maximum
mbs	metres below surface
MCOG	marginal cut-off grade
MDEQ	Michigan Department of Environmental Quality
Met	metallurgical
MFDK	mafic dyke
MIBC	methyl isobutyl carbinol
min.	minimum
mm	millimetre
ModBond	modified Bond ball work index
Moz	million ounces
MPC	Minerals Processing Corporation
m/s	metres per second
MREC	Menominee River Exploration Company
MRHY	massive, aphyric rhyolite flows
m RL	metres relative level
Mt	mega tonne or million tonnes
Mtpa	million tonnes per annum
MW	megawatt
N–S	north–south, north to south
NaCN	sodium cyanide
NAD	North American Datum
NAG	non-acid generating
NCWB	non-contact water basin
NE	northeast
NI	National Instrument
NN	nearest neighbour
NOC	Notice of Coverage
NOI	Notice of Intent
NPDES	National Pollutant Discharge Elimination System
NPI	net profits interest
NPT	Northern Penokean Terrane
NPV	net present value
NREPA	Natural Resources and Environmental Protection Act
NRHP	National Register of Historic Places
NSR	net smelter return
NW	northwest

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
NWRF	north waste rock facility
OBL	Osisko Bermuda Limited
OMC	Orway Mineral Consultants
OP	open pit
OS	overburden stockpile
OSA	on-stream analysis
OSC	Ontario Securities Commission
Osisko	Osisko Gold Royalties Ltd.
Orion	Orion Mine Finance
the Orion Transaction	March 31, 2015, the Company closed a multi-level financing transaction with Orion that included an equity private placement and a silver stream for total funding of $20.75M
oz	ounce
P_{80}	80% percent passing
pa	per annum
P&E	P&E Mining Consultants Inc.
PAG	potentially acid generating
PAX	potassium amyl xanthate
Pb	lead
PC	Paterson and Cooke Canada Inc.
PEA	Preliminary Economic Assessment
PED	personal emergency device
PEM	pulse electromagnetic (survey)
P.Eng.	Professional Engineer
PF	pastefill
P.Geo.	Professional Geoscientist
PIPP	pollution incident prevention plan
Plug	artificial sill pillar
PoF	probability of failure
Portal	initial surface entrance prepared for ramp tunnel
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
the Production Threshold	is when the Company has delivered 105,000 oz of gold to Osisko Bermuda Limited
the Project	Back Forty Project
the Property	Back Forty Property
PTI	Michigan Air Use Permit to Install
PVB	Penokean Volcanic Belt
Q1, Q2, Q3, Q4	first quarter, second quarter, third quarter, fourth quarter of the year

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
QA/QC	quality assurance/quality control
QEMSCAN	quantitative evaluation of materials by scanning electron microscopy
QFP	quartz feldspar porphyry
QMS	quality management system
Ramp	tunnel excavated in downward (upward) inclination
RAR	return air raise
RATF	rhyolite ash tuff
RCTF	rhyolite crystal tuff
REBgold	REBgold Corporation
RF	revenue factor
RFID	radio frequency identification
ROM	run-of-mine
Ro Tail	rougher tail
RQD	rock quality designation
S	sulphur
SART	sulphidization, acidification, recycling and thickening
SCB	soil, cement, and bentonite
SCC	Standards Council of Canada
SE	southeast
SEDAR	System for Electronic Document Analysis and Retrieval
SESC	soil erosion and sedimentation control
SFST	sulphide stringer
SGS	SGS Canada Inc. and its subdivisions, e.g. SGS Mineral Services
SM	semi-massive sulphide
SMBS	sodium metabisulphite
SMC	SAG mill comminution
SMD	stirred media detritor
SMSS	semi-massive sulphide
SMU	selective mining unit
SPCC	spill prevention control and countermeasures
SSR	side slope risers
the Strategic Investment	OBL purchased 49,173,076 units of Aquila at a price of C$0.26 per unit for aggregate gross proceeds of $10 million
the Stream	a Gold Purchase Agreement between Osisko Bermuda Ltd. and Aquilla Resources Corp.
SW	southwest
SWRF	south waste rock facility
t	metric tonne(s)
T	short ton(s)
the Tail Stream	is when the Threshold Stream Percentage will be reduced to 9.25%

TABLE 2.2
TERMINOLOGY AND ABBREVIATIONS

Abbreviation	Meaning
	of the refined gold
TC	treatment charge
TCRC	treatment and refining charge
TFSD	tuffaceous sediments
the Threshold Stream Percentage	Osisko Bermuda Ltd. will purchase 18.5% of the refined gold from the Project
TMF	tailings management facility
tpd	tonnes per day
the Transaction	June 28, 2019, Orion purchased from Osisko all 49,651,857 common shares of the Company owned by Osisko
UG	underground
UTM	Universal Transverse Mercator grid system
VFD	variable frequency drive
VOIP	voice-over-internet-protocol
VMS	volcanogenic massive sulphide
VTEM™	versatile time domain electromagnetic (system)
w/v	weight by volume
w/w	weight by weight
WAD	weak acid dissociable
wmt	wet metric tonne(s)
WRF	waste rock facility (storage)
wt%	weight percent
WWTP	waste water treatment plant
Y or yr	year
Zn	zinc
$ZnSO_4$	zinc sulphate

3.0 RELIANCE ON OTHER EXPERTS

P&E has assumed that all of the information and technical documents listed in the References section of this Technical Report are accurate and complete in all material aspects. While P&E has carefully reviewed all of the available information presented, P&E cannot guarantee the accuracy and completeness of the documents listed in the References section of this Technical Report. P&E reserves the right, but will not be obligated, to revise the Technical Report and conclusions therein if additional information becomes known to P&E subsequent to the effective date of this Technical Report.

Copies of the tenure documents, operating licenses, permits, and work contracts were not reviewed. Information on tenure was obtained from Aquila and included a legal due diligence opinion supplied by Aquila's American legal counsel, Mr. Steven J. Tinti. P&E has relied upon tenure information from Aquila and has not undertaken an independent detailed legal verification of title and ownership of the Back Forty Project. P&E has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on, and believes it has a reasonable basis to rely upon Aquila to have conducted the proper legal due diligence.

Qualified Person Mr. Neil Lincoln has relied on Foth Infrastructure & Environment, LLC for information related to the environmental studies and permitting.

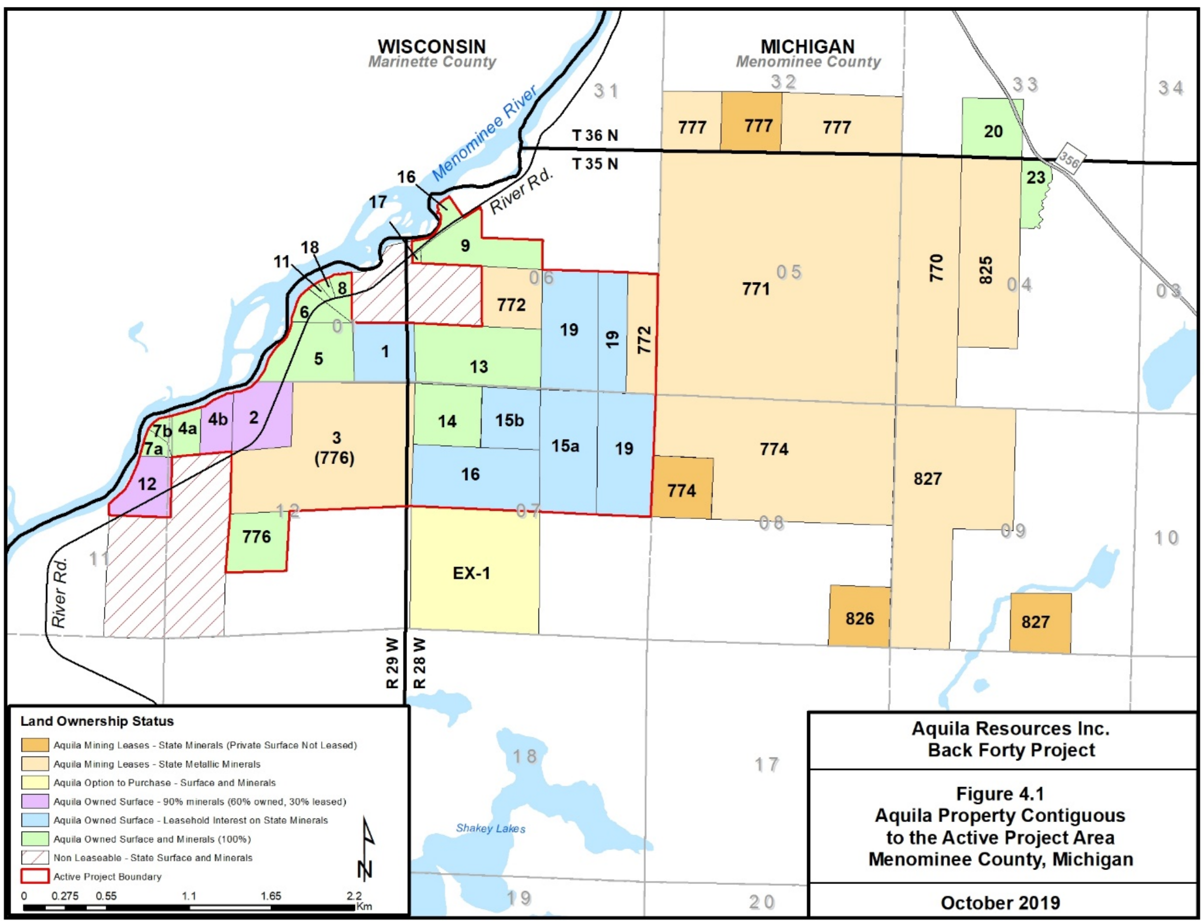
A draft copy of this Technical Report has been reviewed for factual errors by Aquila. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this Technical Report are given in good faith and in the belief that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

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 INTRODUCTION

Aquila controls approximately 1,304 hectares (3,222 acres) of private and public (State of Michigan) mineral lands located in Lake and Holmes Townships in Menominee County, Michigan. Approximately 1,019 hectares (2,517 acres) of these lands form a contiguous block of Aquila-controlled mineral rights (Figure 4.1). The Active Project Boundary encompasses approximately 479 hectares (1,183 acres) and is situated in portions of Sections 1, 11 and 12 in Township 35N, Range 29W, and portions of Sections 6 and 7, in T35N, R28W, in Lake Township, Menominee County, Michigan. The Project is centred at latitude 45° 27' N and longitude 87° 51' W.

FIGURE 4.1 BACK FORTY PROJECT PROPERTY

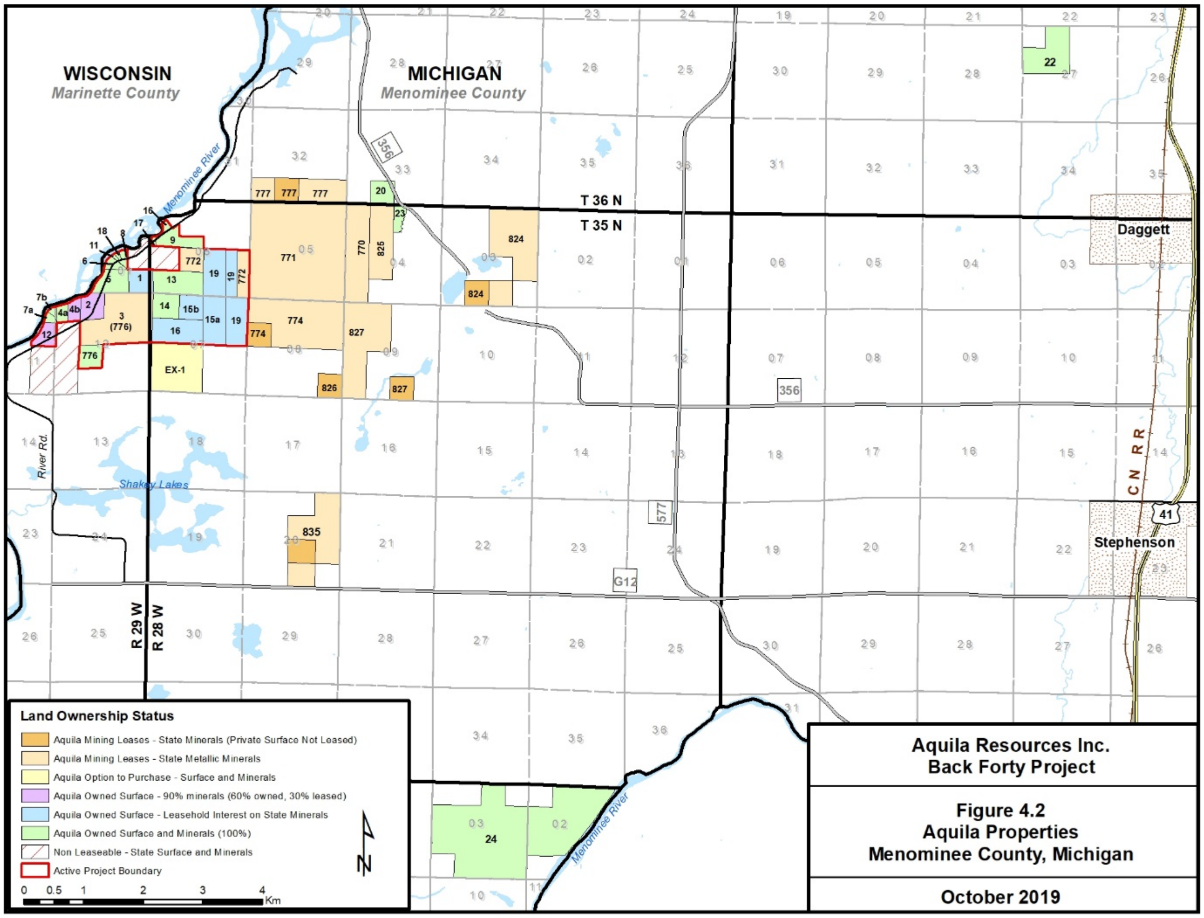


Source: *Aquila Resources Inc. (2019)*

Properties comprising the Active Project Area are currently 100% owned or controlled by Aquila through purchase of the Back Forty Joint Venture LLC ("BFJV") and Hudbay Michigan Inc. ("HMI"). Aquila properties comprising the contiguous parcels outside of the Active Project

Area are controlled by Aquila through metallic minerals leases with the State of Michigan. All Company properties are shown in Figure 4.2.

FIGURE 4.2 AQUILA PROPERTIES



Source: *Aquila Resources Inc. (2019)*

4.2 PROPERTY INTERESTS, TITLE, TAXES AND OTHER LEGAL OBLIGATIONS

The known Mineral Resource at the Project is covered by five parcels (Parcel numbers 1, 2, 3, 4b and 5 on Figure 4.1). Additional parcels that make up the balance of the Active Project Area are considered important for development purposes.

A title opinion was prepared for Aquila on October 14, 2019 by the law office of Steven J. Tinti. Based on records filed with the Menominee County Register of Deeds Office and the agreements examined, Mr. Tinti concluded that under Michigan law: Aquila has full rights to pursue its exploration plans on the parcels controlled by the Company that are within the Active Project Area; there are no legal impediments to Aquila pursuing its mineral exploration plans on these parcels. There are no legal impediments to Aquila pursuing its mineral exploration plans on these parcels. All Back Forty Project properties controlled by Aquila are described below.

4.2.1 Description of Properties

Key properties in the Active Project Area parcels, which contain known Mineral Resources or are considered to be important for development are described below and are shown in Figure 4.1. Parcel numbers are derived from the parcel descriptions in the title opinions.

- **Parcel 1 –** (16 hectares, 40 acres). 100% of the surface is owned by Aquila through purchase. The severed mineral estate owned by the State of Michigan is held under lease number M-00775. Property tax obligations for assessment year 2017 for this property were $1,331.43.

- **Parcel 2 –** (approximately 16 hectares, 39 acres). The surface is owned 100% by Aquila. This includes an Aquila ownership of a 20% mineral interest purchased from the surface owner and 40% mineral interest purchased from heirs of the mineral estate. Another 30% mineral interest is leased to Aquila by agreements with the heirs of the mineral estate. This gives the Aquila a total of 90% mineral interest in this property to date. The former surface owner is due a 3.5% NSR on open pit production and a 2.5% NSR on underground production. The leased mineral owners are due various NSR's ranging from 1% to 2%. Property tax obligations for assessment year 2017 for this property were $2,478.60.

- **Parcel 3 (776) –** (State Lease M-00776) – (97 hectares, 240 acres). 81 hectares (200 acres) of state surface and mineral estates in fee simple and 16 hectares (40 acres) of state mineral estate. This lease calls for minimum royalty payments (deductible from future production royalties) of $30/acre ($7,200) for year 2016 increasing by $5.00 per acre per year through year 2021. The rental payments increase to $55/acre after year 2021. Mineral production from the lease is subject to a sliding scale production royalty.

- **Parcel 4a –** (approximately 5 hectares, 13 acres) was acquired when Aquila exercised an Option to Purchase. The parcel consists of 100% private surface fee simple and mineral interest purchase. Property tax obligations for assessment year 2017 for this property were $5,598.03.

- **Parcel 4b –** (Approximately 7 hectares, 17 acres). The surface is owned 100% by the Aquila. This includes an Aquila ownership of a 20% mineral interest purchased from the surface owner and 40% mineral interest purchased from heirs of the mineral estate. Another 30% mineral interest is leased to the Aquila by agreements with the heirs of the mineral estate. This gives the Aquila a total of 90% mineral interest in this property to date. Property tax obligations for assessment year 2017 for this property were $2,268.19.

- **Parcel 5** - Government Lot 1 – (approximately 19 hectares, 47 acres, of private surface and mineral estate in fee simple) in T35N, R29W, Section 1. An option to purchase agreement for this property was executed in mid-2006. It called for annual option payments over a period of nine years. The final option payment for this property was made in 2015. There is no retained production royalty for the property owner. Property tax obligations for assessment year 2017 for this property were $4,805.76.

- **Parcel 6 –** (5 hectares, 11.5 acres) was acquired when Aquila exercised an Option to Purchase. The parcel consists of 100% private surface fee simple and mineral interest purchase with the former owner retaining a 1.5% NSR from open pit production and underground production. Property tax obligations for assessment year 2017 for this property were $3,002.72.

- **Parcel 13 –** (32 hectares, 80 acres) was acquired when Aquila exercised an Option to Purchase. This parcel consists of private fee simple surface and minerals now wholly owned by Aquila. The mineral estate for this parcel was previously listed under State of Michigan ownership. A 2010 title search led to the discovery that the mineral estate was held by the surface owner. The State has acknowledged this correction and the purchase agreement with the surface owner was amended to reflect the surface owner's mineral interest and retained royalty. The former surface owner has retained a NSR royalty equivalent to the State of Michigan royalty schedule. Property tax obligations for assessment year 2017 totalled $2,276.80.

- **Parcel 14 –** (16 hectares, 40 acres) was acquired when Aquila exercised an Option to Purchase. The parcel consists of 100% private surface fee simple and mineral interest purchase with the former owner retaining a 3.5% NSR from open pit production and a 2.5% NSR from underground production.

- **Parcels 15a and 15b –** (49 hectares, 120 acres) 100% of the surface is owned by Aquila through purchase. The mineral interest for these parcels is state owned and held under lease number M-00773. Property tax obligations for assessment year 2017 for parcels 15a and 15b totalled $5,207.48.

- **Parcel 16 –** (32 hectares, 80 acres) was acquired when Aquila exercised an Option to Purchase. The parcel consists of 100% private surface fee simple and mineral interest purchase with the former owner retaining a 2% NSR from both open pit and underground production. Property tax obligations for assessment year 2017 are included in the property tax bill with Parcel 15a.

- **Parcel 19 -** (81 hectares, 200 acres) 100% of the surface is owned by Aquila through purchase in 2016. The severed mineral estate is owned by the State of Michigan and held under state leases (M-00772 and M-00773). Property tax obligations for assessment year 2016 for all of parcel 19 totalled $4,973.65.

- **Parcel (772)** - State Lease M-00772 (32 hectares, 80 acres, of state surface and mineral estates in fee simple and 49 hectares, 120 acres, of state mineral estate). The severed mineral estate corresponds to Aquila owned private surface in a portion of parcel 19 (that portion residing in Section 6 of T35N, R28W). This lease calls for minimum royalty payments (deductible from future production royalties) of $30/acre ($6,000) for year 2016 increasing by $5.00 per acre per year through year 2021. The rental payments increase to $55/acre after year 2021. Mineral production from the lease is subject to a sliding scale production royalty.

- **Parcel (773)** - State Lease M-00773 (81 hectares, 200 acres, of state mineral estate). The severed mineral estate corresponds to Aquila owned private surface (parcels 15a, 15b, and the portion of parcel 19 residing in Section 7 of T35N, R28W). This lease calls for minimum royalty payments (deductible from future production royalties) of $30/acre ($6,000) for year 2016 increasing by $5.00 per acre per year through year 2021. The rental payments increase to $55/acre after year 2021. Mineral production from the lease is subject to a sliding scale production royalty.

- **Parcel (775)** - State Lease M-00775 (16 hectares, 40 acres, of state mineral estate) The severed mineral estate corresponds to Aquila owned private surface (parcel 1). This lease calls for minimum royalty payments (deductible from future production royalties) of $30/acre ($1,200) for year 2016 increasing by $5.00 per acre per year through year 2021. The rental payments increase to $55/acre after year 2021. Mineral production from the lease is subject to a sliding scale production royalty.

4.2.1.1 Peripheral Properties

In addition to the key properties, Aquila has also purchased, leased, or optioned additional properties. These properties are either contiguous with the Key Parcels, may contain facilities utilized by the Company, are perceived to have exploration potential, or were purchased for other strategic purposes. Figure 4.2 shows the locations and descriptions of the properties.

4.2.2 State of Michigan Metallic Mineral Leases

Michigan state leases (M00775, M00776, M00772 and M00773) in the Mineral Resource area were previously nominated by and awarded to Minerals Processing Corporation (“MPC”) as early as 2002 on a non-competitive basis. These and other state leases in the Project area originally held by MPC have been subsequently assigned to Aquila. The current leases call for a 10-year term that can be extended to 20 years by paying advance royalties.

Other terms include a one-time $1/acre minimum bonus bid, a rental rate commencing at $3/acre per year for the first five-years and $6/acre per year for years six through ten.

In the absence of mining operations, a minimum advance royalty payment (deductible from a production royalty) is due for years eleven through twenty. The advance royalty payment rate begins at $10.00/acre in the eleventh year and escalates by $5.00/acre per year until the twentieth year when the rate is $55.00/acre. If production occurs, a royalty must be paid to the State. A sliding scale production royalty with no deductions of incurred costs is utilized based on an “adjusted (indexed for inflation) sales value” per short ton of dry ore. For base and precious metals, it is calculated on a quarterly basis whereby the gross sales value (revenue received by the mine from a smelter or processor, i.e. “smelter return”) is divided by ore production which is then adjusted for inflation (using the producer price index for all commodities). The resulting adjusted sales value per short ton of ore is subject to the following rates: two percent on value less than $12/ton; this rate is increased by one percent for each $6.00 increase in the value above

$12.00 to a maximum of $71.99/ton; at or above $72/ton a seven percent rate applies. The State of Michigan allows for renegotiation of production royalties (rates and method of calculation) at any time during the term of the lease.

The Michigan Department of Natural Resources (“DNR”) has revised the current mining lease agreement which clarifies and expands the royalty schedule. The new lease format calls for rental rates and advance minimum royalty payments similar to the old lease but includes a much-improved production royalty schedule. The new production royalty is also based on “smelter return” that includes processor deductions for (1) base smelting and refining charges (2) sampling and/or assay charges assessed by the smelter (3) penalties for impurities that are deducted from the assay value of the ore (adjusted sales value). No deductions for operation of the mine, on-site enrichment of ore, or transportation to the smelter will be allowed in calculating smelter returns. None of these costs can be recouped by deductions against the adjusted sales value. The production royalty is calculated the same way as in the old lease for base and precious metals but uses a different royalty schedule that is shown in Table 4.1.

TABLE 4.1
STATE OF MICHIGAN MINERAL ROYALTY SCHEDULE

Adjusted Sales		Value US$ per ton	Underground Royalty Rate	Surface Royalty Rate	Adjusted Sales		Value US$ per ton	Underground Royalty Rate	Surface Royalty Rate
$0.01	-	$25	2%	2.50%	$250.01	-	$275	6%	6.50%
$25.01	-	50	2.40%	2.90%	$275.01	-	$300	6.40%	6.90%
$50.01	-	$75	2.80%	3.30%	$300.01	-	$325	6.80%	7.30%
$75.01	-	$100	3.20%	3.70%	$325.01	-	$350	7.20%	7.70%
$100.01	-	$125	3.60%	4.10%	$350.01	-	$375	7.60%	8.10%
$125.01	-	$150	4%	4.50%	$375.01	-	$400	8%	8.50%
$150.01	-	$175	4.40%	4.90%	$400.01	-	$425	8.40%	8.90%
$175.01	-	$200	4.80%	5.30%	$425.01	-	$450	8.80%	9.30%
$200.01	-	$225	5.20%	5.70%	$450.01	-	$475	9.20%	9.70%
$225.01	-	$250	5.60%	6.10%	$475.01	-	$500	9.60%	10.10%
					Above		$500.01	10%	10.50%

The new royalty rates are improved and more in line with industry standards. As with the old lease, royalty rates in the new lease agreement may be renegotiated any time during the lease term. Although the Project has no new state leases, it does have the option to renegotiate the production royalty in the older leases to the more favourable rates. A renegotiated production royalty is particularly important for leases M00775 and M00776 that control portions of the identified Mineral Resource. However, there is no guarantee that any negotiations with the state regarding modification of state lease production royalty rates will be successful.

4.2.3 Summary of Royalties

The royalties that apply to material planned to be mined include:

- The Michigan State royalty, which applies to approximately 36% of the total mineralized material (38% on a value basis). The royalty is calculated on a sliding scale that ranges from 2.5% to 10.5% of NSR for the open pit and 2.0% to 10.0% for the underground.

- The County royalty, which applies to 1% of the total mineralized material (1% on a value basis). This royalty is calculated in the same manner as the State royalty.

- The Ganzer Royalty, which applies to 12% of the total mineralized material (10% on a value basis). This royalty is a flat 3.5% of NSR.

Royalties aggregate to approximately 2.0% of NSR and are assumed to be paid following the year in which the material is mined.

4.3 ENVIRONMENTAL

To the best of knowledge and belief of P&E, after reasonable inquiry, P&E is not aware of any environmental litigation or pending fines associated with the Back Forty Project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY

The Property area lies along the east bank of the Menominee River and consists of low, rolling hills with maximum topographic relief of 30 m and intervening wetland (in part prairie-savannah); mean elevation is approximately 200 to 300 masl. Vegetation is mostly immature hardwood-pine forest (Figure 5.1) and swamp/prairie-savannah grasses; wetland areas also occur along creeks and secondary tributaries. The climate is temperate, allowing exploration, potential development, and potential mining activities to take place year-round.

FIGURE 5.1 TYPICAL LANDSCAPE OF BACK FORTY PROJECT

Source: *Tetra Tech Preliminary Economic Assessment (2014)*

5.2 CLIMATE

Climate information was obtained from the Midwest Regional Climate Center station in Stephenson, Michigan for the period of 1949 to 2004. Regionally, July is the warmest month with a mean temperature of 19.7°C and January is the coldest month with a mean temperature of -15.4°C.

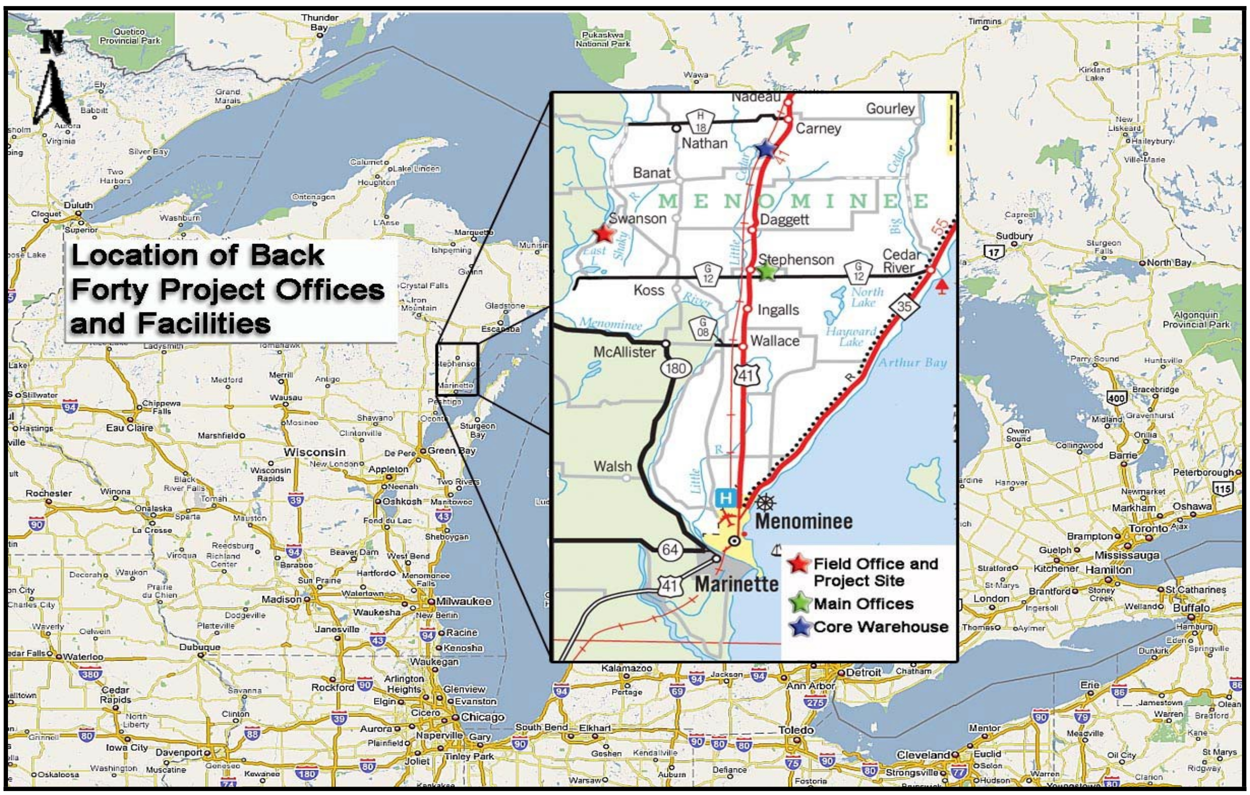
On average, the region receives approximately 796 mm of precipitation annually. Record high and low precipitation measurements were 1086 mm in 1959 and 568 mm in 1989. July and August are the wettest months, with average monthly precipitation of 92 mm, and February is the driest month with 27 mm of precipitation on average. The area receives an average of 154 cm of snowfall per year, with most snowfall occurring in January. There was no recorded snowfall during the months of June, July, or August over the periods of record.

An on-site meteorological station was operated from July 14, 2007 to July 13, 2009. During this time, ambient temperature averaged 5.8°C, with a low of -33.3°C on January 26, 2009 and a high of 35.2°C on July 31, 2007. Winds have been recorded 37.6% of the time, predominantly from the south southwest, southwest, northwest, and north-northwest sectors. The average wind speed for all directional sectors was 8.4 km/h. The local climatic conditions are not anticipated to impede an open pit mining operation at the Property and the expectation is to operate on a year-round basis.

5.3 ACCESS

The Property is located approximately 55 km south-southeast from Iron Mountain, and approximately 19 km west of Stephenson, Michigan, within the Escanaba River State Forest (shown in Figure 5.2). Access from Stephenson is via County G12 Road, north on River Road, travelling approximately 5 km to the Project field office. A number of drill roads connect with River Road and cross the Property. Infrastructure on the Property includes a nearby power line and paved road access.

FIGURE 5.2 LOCATION OF THE BACK FORTY PROJECT



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

5.4 SITE SUFFICIENCY

In the immediate area of the Back Forty Deposit, Aquila controls approximately 1,019 hectares (2,517 acres) of contiguous key parcels available for Project development. This is adequate to address the potential space requirements for mining operations, processing plant, overburden, and rock and tailings storage. The Project is also located within ready access to water and power needs, including a 138 kV transmission line that is currently proposed to service the operation from the northeast corner of the Project site.

6.0 HISTORY

In 2001 a private landowner hired a drilling company to construct a new domestic water well on his property in Michigan. The drillers collected drill cuttings from that well and brought them to a local geologist. Upon further inspection, the cuttings were found to represent a sphalerite-rich massive sulphide. This Property would become the key asset of the Back Forty Project. Subsequent assay of the cuttings confirmed the well had indeed penetrated 12 m (40 ft) of zinc-rich massive sulphide. Upon further surface investigation of the Property, bedrock exposures of favourable pyritic quartz-sericite schist were identified and an auriferous gossan that they interpreted to be capping massive sulphide mineralization at depth. This ultimately led to the acquisition of additional private and state mineral interests in the surrounding area.

In February of 2002, two diamond drill holes were completed along the eastern edge of the Property on state mineral leases. These holes targeted a 1.5 mGal gravity anomaly coincident with a strong max-min electromagnetic conductor. Although the first hole (108401) failed to intersect any significant mineralization, the second hole (108402) penetrated 37 m of massive sulphides grading 9.1% Zn and 5.7 g/t Au after penetrating the capping gossan that graded 21.9 g/t Au. The East Zone had been discovered.

Shortly thereafter, the Back Forty Joint Venture ("BFJV") was formed between the Menominee River Exploration Company ("MREC") and the American Copper and Nickel Company, Inc ("ACNC"), INCO's American subsidiary. ACNC could earn a 60% interest in the Project by spending $10 million over six years. After protracted negotiations, a purchase option was finally arranged for the Thoney property. In October 2002, drilling commenced and continued through early May of 2003. With up to five drill rigs operating, a total of 20,450 m in 71 holes were completed. This drilling partially delineated a zinc-copper-gold-silver rich VMS deposit. However, ACNC deemed the deposits potential size as too small to meet their minimum requirements of at least 20 Mt.

ACNC attempted unsuccessfully to sell its position in the BFJV. It still had not vested its 60% interest. By mid-2003, ACNC had negotiated with MREC an immediate withdrawal from the Project in exchange for a retained 7% net profits interest ("NPI") in any future deposits developed within the Project area. With ACNC out of the joint venture, MREC began seeking a new partner to help advance the Project. However, in early 2004, a new company, Aquila Resources Corporation ("ARC"), was formed for the purpose of becoming publicly listed with the Project. It was not until mid-2006 that JML Resources acquired 100% of the outstanding shares of ARC through a reverse take-over and was listed on the TSX Venture Exchange.

Once listed, the new company, renamed Aquila Resources Inc. (Aquila) raised additional exploration capital to restart drilling operations. By early September 2006, the Thoney Property was re-acquired and combined with adjacent parcels to become the Back Forty Property. Two drilling rigs were brought back to focus on drilling the shallow portions of the Deposit. An additional 14,600 m in 80 holes were completed by mid-November 2006, to fill in gaps in earlier drilling. In early 2007, Datamine International Ltd. was commissioned to conduct a Mineral Resource Estimate that included the 2006 drilling results. In April of 2007, Aquila announced the approval to list on the Toronto Stock Exchange.

Exploration drilling continued into 2008, resulting in 354 drill holes to be compiled into a new Mineral Resource Estimate. In 2008, SRK Consulting, Toronto, was retained to provide a new Mineral Resource Estimate. During 2008, SRK evaluated data from the Back Forty Property, including drilling, survey, core logging, assay and quality control procedures, data entry and management procedures, review of geological interpretation, and inspection of drill core. The initial Mineral Resource Estimate was released in January 2009.

In August of 2009, Hudbay Minerals Inc. ("Hudbay") entered into a Subscription, Option, and Joint Venture Agreement allowing Hudbay to earn a majority share of the Project and become the operator. Under the agreement, another phase of drilling started in the fall of 2009 and continued until June of 2010. The total number of drill holes increased to 478. Golder Associates, from Mississauga, Ontario were retained to calculate an updated Mineral Resource Estimate, which was released in October 2010.

In September 2010, Hudbay announced that, pursuant to the terms of a Subscription, Option, and Joint Venture Agreement with Aquila Resources Inc., Hudbay had exercised its option to earn a 51% joint venture interest in Aquila's Back Forty Project in Michigan's Upper Peninsula after expenditures of $10 million on the Project. Hudbay would be able to increase its ownership interest in the Project to 65% by completing a Feasibility Study and submitting a mine permit application to the State of Michigan.

On January 16, 2014, REBgold Corporation ("REBgold") and Aquila closed a statutory plan of arrangement (the "Arrangement"). The Arrangement required that:

- Aquila acquires 100% of the outstanding shares of REBgold in exchange for Aquila shares on a one-for-one basis.

- The acquisition of 100% of the shares of HudBay Michigan Inc. ("HMI"), effectively giving Aquila 100% ownership of the Back Forty Project.

- The non-brokered private placement of REBgold shares for gross proceeds of approximately $4.85 million (the "REBgold Financing"). Pursuant to the REBgold Financing, Baker Steel Capital Managers LLP ("Baker Steel"), on behalf of investment funds managed or controlled by it, invested $4.5 million of such gross proceeds. Proceeds from the REBgold Financing would be used for general working capital and to fund the next phase of development activity at Back Forty.

Pursuant to the REBgold Financing, REBgold issued a total of 37,300,385 shares at a price of $0.13 per share for gross proceeds of approximately $4.85 million. All of these shares were immediately exchanged for 37,300,385 Aquila shares pursuant to the Arrangement. In connection with the issuance of 2,285,000 REBgold shares for gross proceeds of $297,050 as part of the REBgold private placement, REBgold paid broker compensation consisting of (i) a cash commission equal to 7% of the gross proceeds related to such subscriptions, and (ii) non-transferable broker warrants (the "Broker Warrants") to purchase an aggregate of 159,950 REBgold shares (representing 7% of the REBgold shares related to such subscriptions) at a price of $0.15 per share for two years from the closing of the REBgold Financing. As a result of completion of the Arrangement, each Broker Warrant became exercisable for one Aquila share at a price $0.15 per share.

Immediately following completion of the Arrangement and related transactions, there were approximately 183 million common shares of Aquila outstanding and 27.6 million common shares exercisable through stock options, convertible debentures and warrants. Immediately prior to completion of the Arrangement and related transactions, there were 64,825,568 REBgold shares outstanding (including shares issued pursuant to the REBgold Financing). All of these shares were exchanged for Aquila shares pursuant to the Arrangement on a one-for-one basis.

Pursuant to the HMI Acquisition, Hudbay's 51% interest in the Back Forty Project was acquired in consideration for the issuance of 18,650,193 common shares of Aquila, future milestone payments tied to the development of the Back Forty Project and a 1% net smelter return royalty on production from certain land parcels in the Project. The net smelter return royalty was repurchased in conjunction with the Orion Transaction (see below). At the time, Baker Steel was Aquila's largest shareholder and owned or controlled 45,483,886 Aquila common shares or approximately 25% of the outstanding Aquila common shares. Hudbay owned or controlled 33,017,758 Aquila common shares or approximately 18% of the outstanding Aquila common shares. In connection with the completion of the Arrangement, REBgold, as a wholly-owned subsidiary of Aquila, had its shares delisted from the TSX Venture Exchange and ceased to be a reporting issuer.

In 2014, a Preliminary Economic Assessment was completed which contemplated an open pit mining/processing operation at Back Forty.

On March 31, 2015, the Company closed a multi-level financing transaction with Orion Mine Finance ("Orion") that included an equity private placement and a silver stream for total funding of $20.75 million (collectively, the "Orion Transaction"). Concurrent with the Orion Transaction, the Company completed the repurchase of two existing royalties on the Back Forty Project. As part of the Orion Transaction, Aquila issued 26,923,077 units at a price of $0.13 per unit for gross proceeds of $3.5 million, with each unit consisting of one common share and one-half common share purchase warrant. Each whole warrant allows the holder to purchase one common share at a price of $0.19 per common share for a term of three years. Also as part of the Orion Transaction, pursuant to a silver purchase agreement (the "Silver Purchase Agreement") dated March 31, 2015 between Orion Titheco Limited, the Company and Back Forty Joint Venture LLC, Orion acquired 75 per cent of Aquila's life-of-mine ("LOM") silver production from the Back Forty Project for gross proceeds of $17.25 million. Orion has advanced the first instalment of $6.5 million, the second instalment of $3.0 million, the third instalment totalling $3.375 million plus the $1.35 million land payment and the final instalment of $2.376 million. In June 2016, the silver purchase agreement was amended to reduce the deposit owing by $625,000. In November 2016, the silver purchase agreement was amended to reduce the deposit owing by $14,000.

The Company currently has three main subsidiaries, Aquila Resources Corp., Aquila Resources USA Inc., and Aquila Michigan Inc. (formerly known as HMI). The remaining subsidiaries are inactive. All subsidiaries are 100% owned.

On November 10, 2017, the Company completed a financing transaction with Osisko Bermuda Limited ("OBL"), a wholly owned subsidiary of Osisko Gold Royalties Ltd. (TSX and NYSE:

OR,) ("Osisko") pursuant to which OBL has agreed to commit $65 million to Aquila through a $10 million private placement and $55 million gold stream purchase agreement.

OBL purchased 49,173,076 units of Aquila at a price of C$0.26 per unit for aggregate gross proceeds of $10 million (the "Strategic Investment"). Each unit consists of one common share and one-quarter of one common share purchase warrant. Each whole warrant entitles the holder to purchase one common share of the Company for C$0.34 until May 10, 2021. Osisko also has the right to participate in any future equity or equity-linked financings to maintain its ownership level in Aquila. In connection with the private placement, Osisko received the right to nominate one individual to the board of directors of Aquila and thereafter for such time as Osisko owns at least 10% of the outstanding common shares. Osisko's nominee was appointed to the board of directors in November 2017.

Concurrent with the Strategic Investment, the parties have entered into a Gold Purchase Agreement (the "Stream"), whereby OBL will provide the Company with staged payments totalling $55 million, payable as follows:

- $7.5 million on close of the Streaming Transaction.
- $7.5 million upon receipt by Aquila of all material permits required for the development and operation of the Project, and receipt of a positive Feasibility Study.
- $10 million following a positive construction decision for the Project.
- $30 million upon the first drawdown of an appropriate Project debt finance facility, subject to the COC Provision (as defined below).

Under the terms of the Stream Agreement, OBL will purchase 18.5% of the refined gold from the Project (the "Threshold Stream Percentage") until the Company has delivered 105,000 ounces of gold (the "Production Threshold"). Upon satisfaction of the Production Threshold, the Threshold Stream Percentage will be reduced to 9.25% of the refined gold (the "Tail Stream"). In exchange for the refined gold delivered under the Stream, OBL will pay the Company ongoing payments equal to 30% of the spot price of gold on the day of delivery, subject to a maximum payment of $600 per ounce ("/oz").

In the event of a change of control of the Company prior to the advancement of the final $30 million under the Stream, the person or entity acquiring control over the Project may elect to forgo the final payment, in which case the Threshold Stream Percentage and Tail Stream will be reduced to 9.5% and 4.75%, respectively (the "COC Provision"). All other terms and conditions of the Stream will remain unchanged.

Pursuant to the Stream, the Company has agreed to pay a $200,000 capital commitment fee. The fee is payable as to 50% upon closing of the Stream transaction and 50% upon OBL funding the second deposit under the Stream. Aquila satisfied the initial $100,000 fee by way of the issuance of 478,781 common shares of the Company based upon the five-day volume weighted average price of the common shares prior to November 10, 2017.

On September 7, 2018 Aquila filed an open pit Feasibility Study Technical Report on SEDAR, with an effective date of August 1, 2018. The study concluded that the Project will produce approximately 1.1 Moz AuEq over a seven-year life. The study was limited to a sub-set of economically viable Mineral Resources that yielded optimal returns by open pit mining. There were additional economically viable Mineral Resources that could be exploited with a push back beyond the pit limits contemplated in the study. Alternatively, the incremental Mineral Resources could be exploited using underground methods. Salient metrics for the base case macro-economic forecast, which included prices of $1,300/oz for Au and $1.20/lb for Zn, are presented in Table 6.1.

TABLE 6.1
SUMMARY METRICS OF 2018 FEASIBILITY STUDY

Item	Unit	Value
Ore Mined	Mt	11.7
Payable Au	koz	512
Payable AuEq[1]	koz	468
Payable Zn	Mlbs	1,197
Payable ZnEq[1]	Mlbs	1,105
Gross Revenue	$/t ore	$123
Treatment Charge/Refining Charge	$/t ore	$15
Net Smelter Return	$/t ore	$108
Site Operating Costs	$/t ore	$31.88
Net Direct Cash Cost (C1)	$/lb Zn	($1.73)
Initial Capital	$M	$294
Total Investment[2]	$M	$480
Net All-in Sustaining Costs (AISC)	$/lb Zn	($1.34)
Post-Tax Net Present Value NPV6%	**$M**	**$208**
Post-Tax IRR	**%**	**28.2**
Post-Tax Cash Flow Index	NPV : Peak Investment	0.70x
Simple Payback	months	26

Notes:

1) *By-Products converted to equivalent Zn and Au using weighted average metal prices over LOM.*
2) *Total investment includes initial capital, sustaining capital and closure expenses.*

On October 5, 2018, Aquila received a payment of $7.4 million from an affiliate of Osisko under the Gold Purchase Agreement. This payment represents the second deposit of the total advance payment of US$55 million to be made by Osisko under the Gold Purchase Agreement. The payment, which was made net of a $100,000 capital commitment fee, follows receipt by Aquila of all material permits required for the development and operation of the Back Forty Project in Michigan and the completion of the Back Forty Project Feasibility Study.

On June 28, 2019, the Company announced that its two largest shareholders, Orion Mine Finance (and its affiliated funds) ("Orion") and Osisko Gold Royalties Ltd. ("Osisko") completed a transaction whereby Orion purchased from Osisko all 49,651,857 common shares of the Company owned by Osisko (the "Transaction"). The Transaction was a small component of the share repurchase and secondary offering transaction first announced by Osisko on June 25, 2019. Orion now owns 97,030,609 common shares of Aquila representing approximately 28.7% of the outstanding common shares. Osisko remains a significant financial partner to Aquila as the holder of gold and silver streams on the Company's Back Forty Project. Under its gold streaming agreement with the Company, Osisko remains committed to funding an additional US$40 million in staged payments to continue the development of the Back Forty Project.

6.1 HISTORICAL MINERAL RESOURCE ESTIMATES

Tetra Tech produced a Mineral Resource Estimate in a 2014 PEA using metal prices of US$1,456.36/oz gold, US$27.78/oz silver, US$3.64/lb copper, US$1.0125/lb lead, and US$0.96/lb zinc. The effective date of the Mineral Resource Estimate was February 4, 2013.

TABLE 6.2
FEBRUARY 4, 2013 MINERAL RESOURCE ESTIMATE

Resource Classification	Tonnes	Au (ppm)	Ag (ppm)	Cu (%)	Pb (%)	Zn (%)	NSR ($/t)	NSR zg ($/t)
Flotable Resources								
Measured	5,595,842	1.956	24.558	0.555	0.165	4.681	139.693	37.844
Indicated	7,614,303	1.538	19.707	0.220	0.255	2.587	85.727	42.847
Inferred	2,132,302	1.973	24.710	0.389	0.335	2.388	98.988	63.416
Measured + Indicated	**13,210,145**	**1.715**	**21.762**	**0.362**	**0.217**	**3.474**	**108.587**	**40.728**
Leachable Resource								
Measured	1,106,960	3.194	41.145	0.059	0.252	0.239	151.747	46.592
Indicated	816,942	5.527	45.870	0.195	0.278	0.228	263.598	51.689
Inferred	204,432	3.138	45.455	0.084	0.319	0.233	163.079	71.538
Measured + Indicated	**1,923,902**	**4.184**	**43.151**	**0.117**	**0.263**	**0.234**	**199.242**	**48.756**
Leachable + Flotable Resource								
Measured	6,702,803	2.160	27.297	0.473	0.180	3.947	141.684	39.288
Indicated	8,431,244	1.925	22.242	0.218	0.257	2.358	102.962	43.704
Inferred	2,336,734	2.075	26.525	0.362	0.334	2.200	104.595	64.127
Measured + Indicated	**15,134,047**	**2.029**	**24.481**	**0.331**	**0.223**	**3.062**	**120.112**	**41.748**

P&E produced a Technical Report and Updated Mineral Resource Estimate, with an effective date of February 6, 2018, as tabulated in Table 6.3. P&E considered the mineralization of the Back Forty Deposit to be potentially amenable to Open Pit and Out of Pit (underground) extraction. The Updated Mineral Resource Estimate formed the basis for the 2018 Feasibility Study that Aquila filed on SEDAR with an effective date of August 1, 2018.

The Mineral Resource Estimates noted in this section are superseded by the Updated Mineral Resource Estimate presented in Section 14 of this Technical Report.

TABLE 6.3 AUGUST 1, 2018 MINERAL RESOURCE ESTIMATE (1-6)														
Resource Area	**Metallurgy Type**	**Classification**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Zn (%)**	**Zn (Mlb)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**
Pit Constrained	Flotable	Measured	21	6,797	1.75	381	18.4	4,027	3.45	516.5	0.38	56.4	0.16	23.4
		Indicated	21	3,768	1.58	191	25.2	3,056	3.15	261.7	0.24	19.9	0.39	32.8
		M & I	**21**	**10,565**	**1.68**	**572**	**20.9**	**7,083**	**3.34**	**778.2**	**0.33**	**76.3**	**0.24**	**56.2**
		Inferred	21	71	1.01	2	30.7	70	2.98	4.7	0.14	0.2	0.37	0.6
	Leachable	Measured	22	553	5.61	100	34.8	618	0.19	2.4	0.05	0.6	0.13	1.5
		Indicated	22	1,777	2.15	123	39.6	2,263	0.41	16.1	0.03	1.3	0.29	11.5
		M & I	**22**	**2,330**	**2.97**	**223**	**38.5**	**2,881**	**0.36**	**18.5**	**0.04**	**1.9**	**0.25**	**13.0**
		Inferred	22	378	3.62	44	40.1	487	0.38	3.2	0.06	0.5	0.52	4.3
	Total	Measured	21+22	7,350	2.04	481	19.7	4,645	3.20	518.8	0.35	57.0	0.15	24.9
		Indicated	21+22	5,545	1.76	314	29.8	5,319	2.27	277.8	0.17	21.2	0.36	44.3
		M & I	**21+22**	**12,895**	**1.92**	**795**	**24.0**	**9,964**	**2.80**	**796.6**	**0.28**	**78.2**	**0.24**	**69.2**
		Inferred	21+22	448	3.21	46	38.6	557	0.79	7.9	0.07	0.7	0.49	4.9
Out of Pit	Flotable	Measured	70	556	1.79	32	26.8	480	5.32	65.2	0.33	4.0	0.41	5.0
		Indicated	70	3,059	1.84	181	26.2	2,577	4.23	285.4	0.51	34.3	0.30	20.3
		M & I	**70**	**3,615**	**1.83**	**213**	**26.3**	**3,057**	**4.40**	**350.7**	**0.48**	**38.4**	**0.32**	**25.3**
		Inferred	70	544	2.96	52	37.5	656	1.38	16.6	0.62	7.5	0.39	4.6
	Leachable	Measured	70	37	7.38	9	74.3	89	0.31	0.3	0.12	0.1	0.11	0.1
		Indicated	70	77	3.85	10	47.3	117	0.32	0.5	0.15	0.2	0.13	0.2
		M & I	**70**	**114**	**5.01**	**18**	**56.1**	**206**	**0.32**	**0.8**	**0.14**	**0.3**	**0.13**	**0.3**
		Inferred	70	137	5.93	26	81.0	356	0.42	1.3	0.16	0.5	0.49	1.5
	Total	Measured	70	593	2.14	41	29.8	569	5.01	65.5	0.32	4.1	0.39	5.1
		Indicated	70	3,135	1.88	190	26.7	2,694	4.14	286.0	0.50	34.6	0.30	20.5
		M & I	**70**	**3,729**	**1.93**	**231**	**27.2**	**3,262**	**4.28**	**351.5**	**0.47**	**38.7**	**0.31**	**25.7**
		Inferred	70	680	3.56	78	46.2	1,011	1.19	17.8	0.53	8.0	0.41	6.1
Total	Flotable	Measured	21+70	7,353	1.75	414	19.1	4,507	3.59	581.7	0.37	60.5	0.18	28.4
		Indicated	21+70	6,827	1.69	371	25.7	5,633	3.64	547.1	0.36	54.2	0.35	53.1

TABLE 6.3
AUGUST 1, 2018 MINERAL RESOURCE ESTIMATE (1-6)

Resource Area	Metallurgy Type	Classification	NSR Cut-off ($/t)	Tonnes (k)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)	Zn (%)	Zn (Mlb)	Cu (%)	Cu (Mlb)	Pb (%)	Pb (Mlb)
		M & I	**21+70**	**14,180**	**1.72**	**785**	**22.2**	**10,140**	**3.61**	**1,128.8**	**0.37**	**114.7**	**0.26**	**81.5**
		Inferred	21+70	615	2.74	54	36.7	726	1.57	21.2	0.57	7.7	0.38	5.2
	Leachable	Measured	22+70	590	5.72	109	37.3	707	0.20	2.6	0.05	0.7	0.12	1.6
		Indicated	22+70	1,854	2.22	132	39.9	2,380	0.41	16.7	0.04	1.6	0.29	11.7
		M & I	**22+70**	**2,444**	**3.07**	**241**	**39.3**	**3,087**	**0.36**	**19.3**	**0.04**	**2.2**	**0.25**	**13.4**
		Inferred	22+70	514	4.24	70	51.0	842	0.39	4.5	0.09	1.0	0.51	5.8
	Total	Measured	21+22+70	7,943	2.04	522	20.4	5,214	3.34	584.3	0.35	61.2	0.17	30.0
		Indicated	21+22+70	8,680	1.80	504	28.7	8,013	2.95	563.8	0.29	55.8	0.34	64.9
		M & I	**21+22+70**	**16,623**	**1.92**	**1,026**	**24.8**	**13,227**	**3.13**	**1,148.1**	**0.32**	**116.9**	**0.26**	**94.9**
		Inferred	21+22+70	1,129	3.42	124	43.2	1,568	1.03	25.7	0.35	8.7	0.44	11.0

Notes: *M = Measured Mineral Resource, I = Inferred Mineral Resource.*

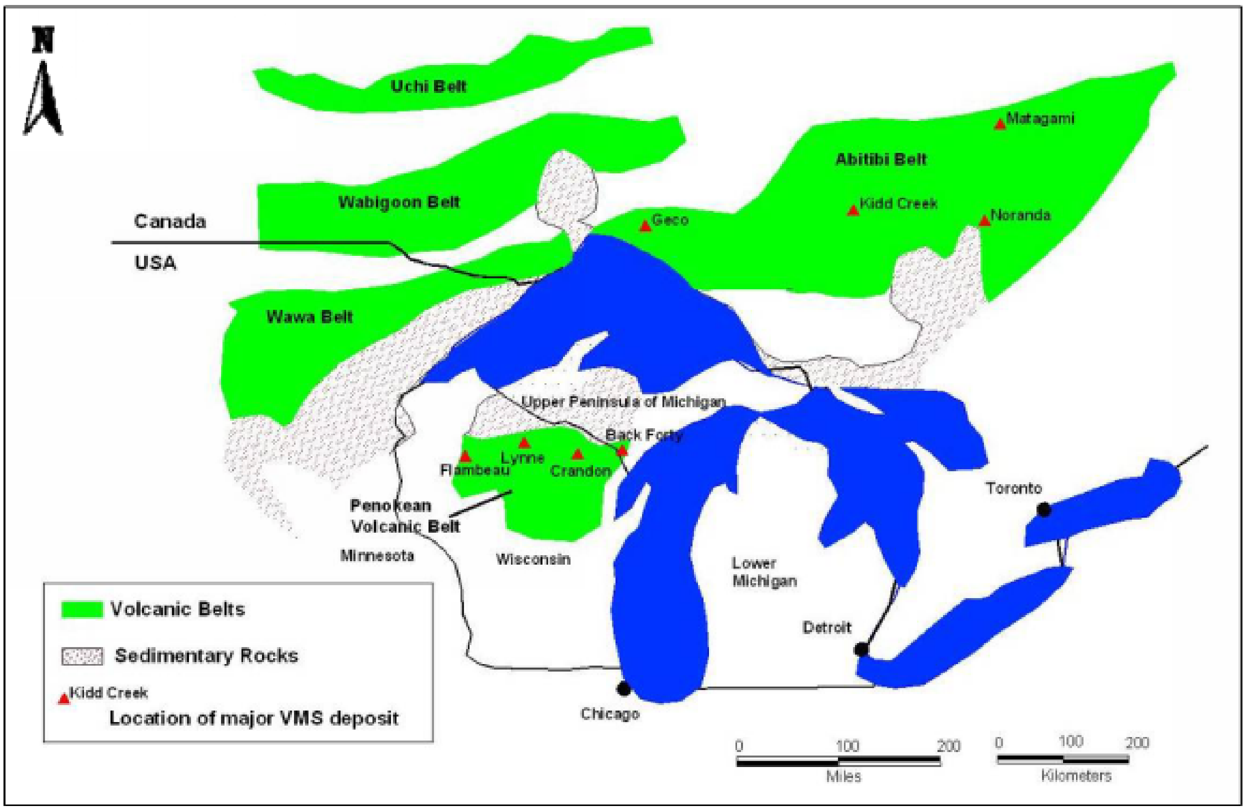
1) *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
2) *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3) *The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4) *Metallurgical type Oxide (all gold domains and leachable Gossans) is leachable, while all other metallurgical types are flotable.*
5) *The Mineral Resource Estimate was based on US$ metal prices of $1,375/oz gold, $22.27/oz silver, $1.10/lb zinc, $3.19/lb copper and $1.15/lb lead.*
6) *Mineral Resources were defined within a conceptual constrained pit shell.*

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Back Forty VMS Deposit is one of a number of deposits located throughout the Ladysmith-Rhinelander volcanic complex in northern Wisconsin and the Upper Peninsula of Michigan. The complex lies within the lower Proterozoic Penokean Volcanic Belt ("PVB"), also known as the Wisconsin Magmatic Terranes. The PVB is part of the Southern Structural Sub-province of the Canadian Shield (Figure 7.1).

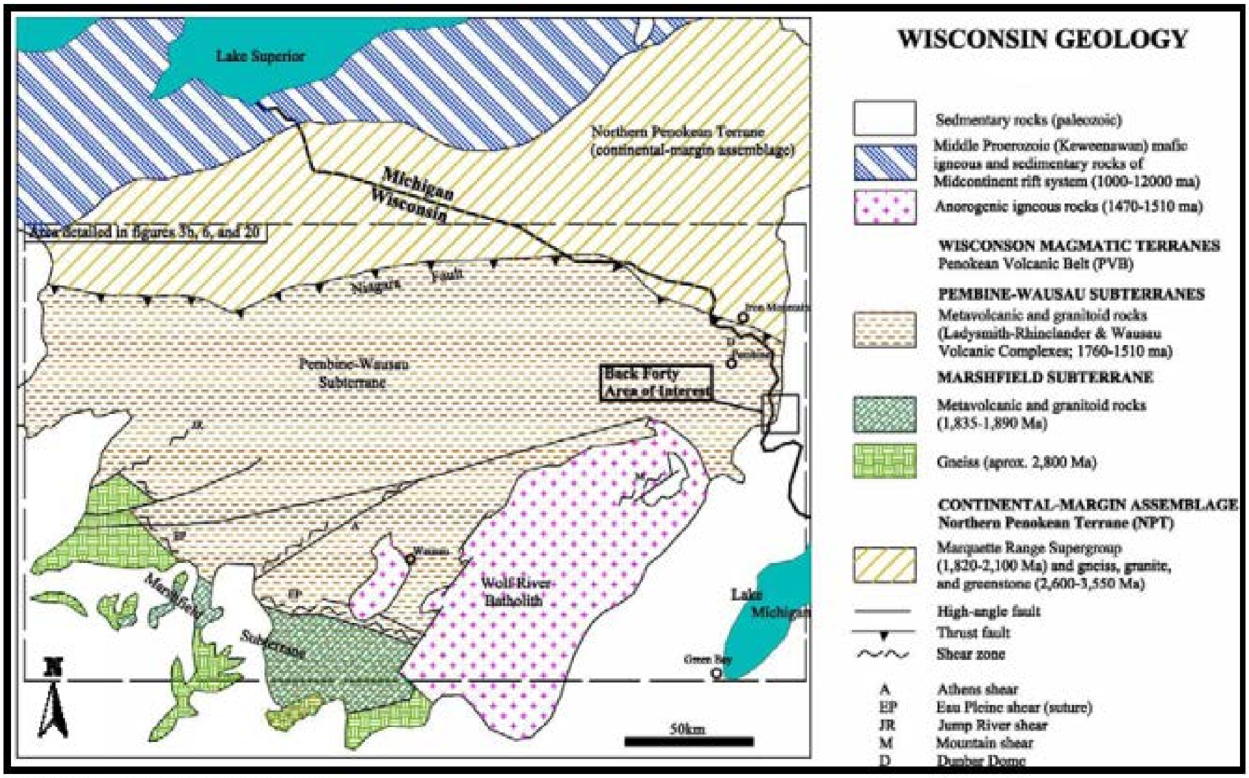
FIGURE 7.1 SCHEMATIC GEOLOGICAL MAP OF THE GREAT LAKES REGION SHOWING PRINCIPAL VOLCANIC BELTS



Source: Franklin and Thorpe (1982)

Sims et al. (1989) divided the PVB into the Pembine-Wausau and Marshfield subterranes, separated by the Eau Pleine Shear Zone. Each subterrane is characterized by volcanic island-arc-basin assemblages containing abundant calc-alkaline metavolcanic units and lesser amounts of sedimentary rocks; they generally lack major regional oxide-facies iron formations. Sims et al. (1989) established an Early Proterozoic age, ranging from 1,889 to 1,835 Ma (Figure 7.2). The PVB is in contact with another major terrane to the north designated the "Northern Penokean Terrane" (NPT). The contact between these terranes is marked by the Niagara Fault Zone, which is believed to be a paleosuture (Sims et al. 1989).

FIGURE 7.2 GEOLOGIC MAP OF NORTHERN WISCONSIN AND WESTERN MICHIGAN SHOWING MAJOR TERRANES



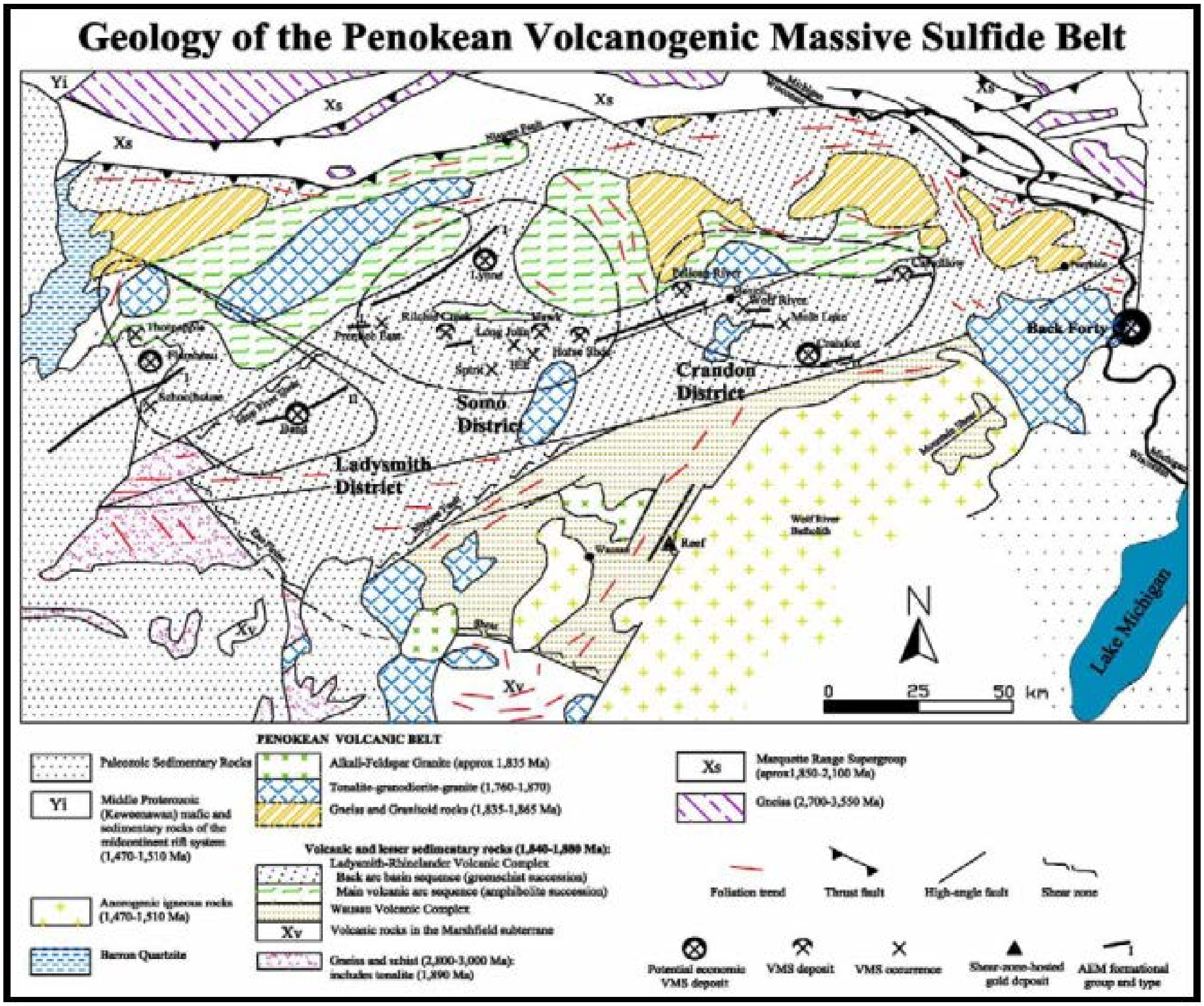
Source: Sims et al (1989)

The NPT is characterized in part by a thick turbidite platform sequence, which was deposited at a continental margin on Archean basement. Subordinate interbedded tholeiitic metavolcanics and major Superior-type oxide-facies iron formations occur within the package. This supracrustal sequence has been interpreted to correlate with the Marquette Range Supergroup in Michigan. Both terranes (NPT and PVB) have been affected by the Penokean Orogeny, which occurred from 1,900 to 1,840 Ma, and resulted in major folding and faulting, regional metamorphism, and emplacement of major granitic intrusions.

On the basis of regional gravity and magnetic data, three volcanic complexes have been defined in the PVB: 1) the Ladysmith-Rhinelander Volcanic Complex, which dominates the northern portion of the Pembine-Wausau Subterrane; 2) the Wausau Complex, to the south, which has been intruded over much of its extent by the Wolf River Batholith; and 3) the Eau Claire Complex, in the Marshfield Subterrane.

Geological, geophysical and geochemical data compiled since the 1960s define three depositional environments in the 1,880 to 1,860 Ma old Ladysmith-Rhinelander Complex, with each containing VMS mineralization: 1) a main volcanic-arc sequence, forming the structural core of the complex, 2) a laterally equivalent and/or possibly younger back-arc basin, volcanic-volcaniclastic succession that includes a series of mafic volcanic piles, and 3) major felsic volcanic centres in the back-arc basin and along the flanks of the main volcanic arc. The three mineral districts in the Ladysmith-Rhinelander Complex are defined by clustering of VMS deposits and occurrences as shown in Figure 7.3 (DeMatties 1994).

FIGURE 7.3 GEOLOGIC MAP OF THE PENOKEAN VOLCANIC BELT



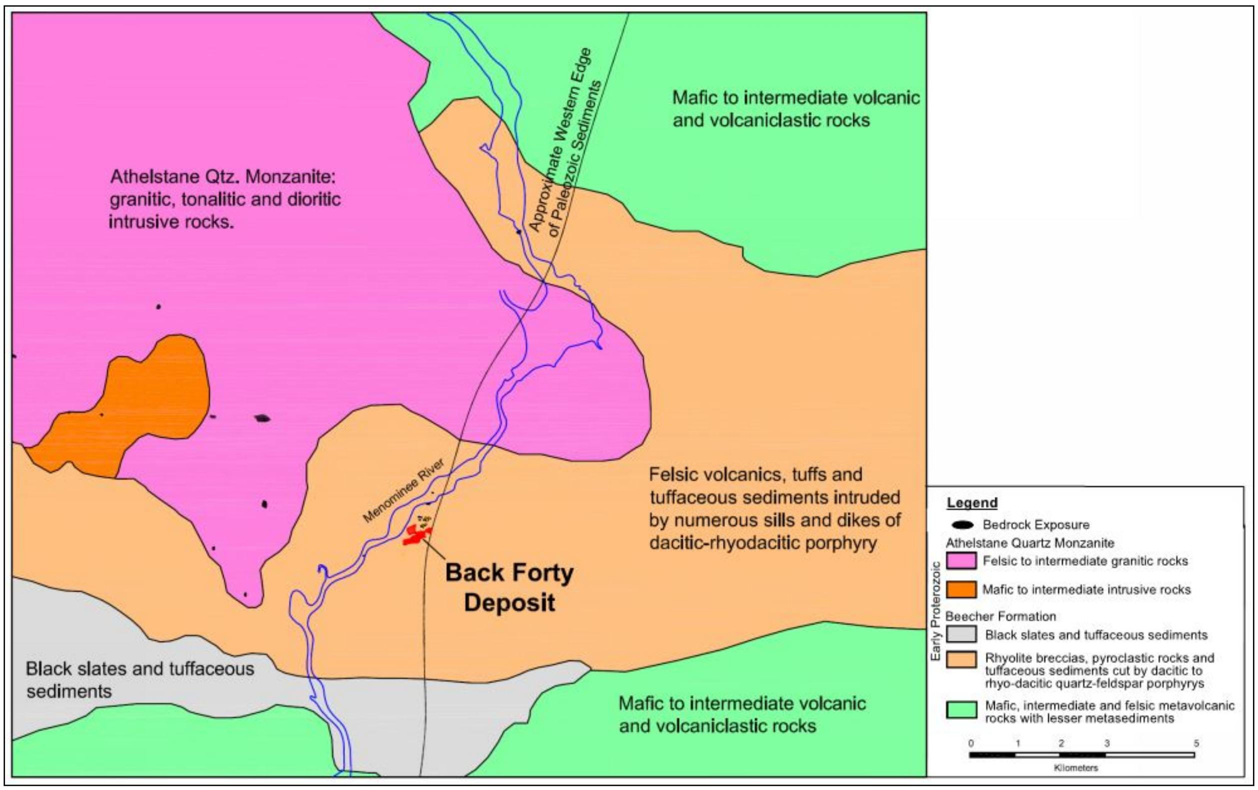
Source: DeMatties (1994)

The spatial distribution of the three districts appears to be linear, trending in an east-west direction (the so called "Highway 8" trend) and are separated from each other by 30 km to 50 km. However, a more complicated arrangement of individual deposits and occurrences is evident within each district. It is interesting to note the Back Forty deposit is isolated from the known districts; located at the east end of the belt and east of the Menominee River in Michigan. The nearest significant deposit is the Catwillow occurrence located at the east end of the Crandon District, approximately 50 km northwest of the Project. The distance from the Crandon District is significant given the average distance between districts of 30 km to 50 km. This distribution suggests strongly that the Back Forty Deposit lies within a new, but as yet unrecognized, district at the extreme east end of the belt. As a result, the Back Forty discovery has created a larger district-scale potential for Aquila.

7.2 DISTRICT GEOLOGY

Published small-scale (1:250,000) geologic maps of north-eastern Wisconsin indicate the area to the west of the Project area is underlain by the 1,760 to 1,870 Ma old Athelstane Quartz Monzonite, an intrusive complex composed of tonalite, granodiorite and granite. The plutonic complex is bounded on the north, east, and south by metavolcanic rocks of the Beecher Formation and contains numerous metavolcanic rock inclusions (Figure 7.4). The volcanics generally face outward from the margin of the intrusive complex. Dykes of Athelstane Quartz Monzonite extend a short distance into the Beecher Formation (Jenkins 1973).

FIGURE 7.4 BACK FORTY AREA GEOLOGIC BEDROCK MAP



Source: *Aquila Resources Inc. (2018)*

The Beecher Formation consists of a stratigraphically lower, 3,000 m thick sequence of calc-alkaline andesite to dacite flows and an upper 300 m thick section of interbedded felsic ash, crystal tuff, lapilli tuff, coarser fragmental rocks, and locally black slates near the stratigraphic top of the formation. The Back Forty Deposit is hosted by a volcanic complex quite similar to the upper volcaniclastic section of the Beecher Formation. Zircons extracted from rhyolite crystal tuff and intrusive rhyodacite porphyry from Back Forty have yielded a uranium/lead age of 1,874 ±4 Ma (Schulz et al. 2008). This age is consistent with the published age of the Athelstane Quartz Monzonite. It is likely that the felsic sequence at Back Forty is a member of the Beecher Formation. The lateral extent of this volcanic centre is unknown at this time. However, drilling and gravity surveys indicate it is truncated to the west and north by Athelstane Quartz Monzonite, but likely extends further to the east and south, beneath Cambrian sandstone sediments.

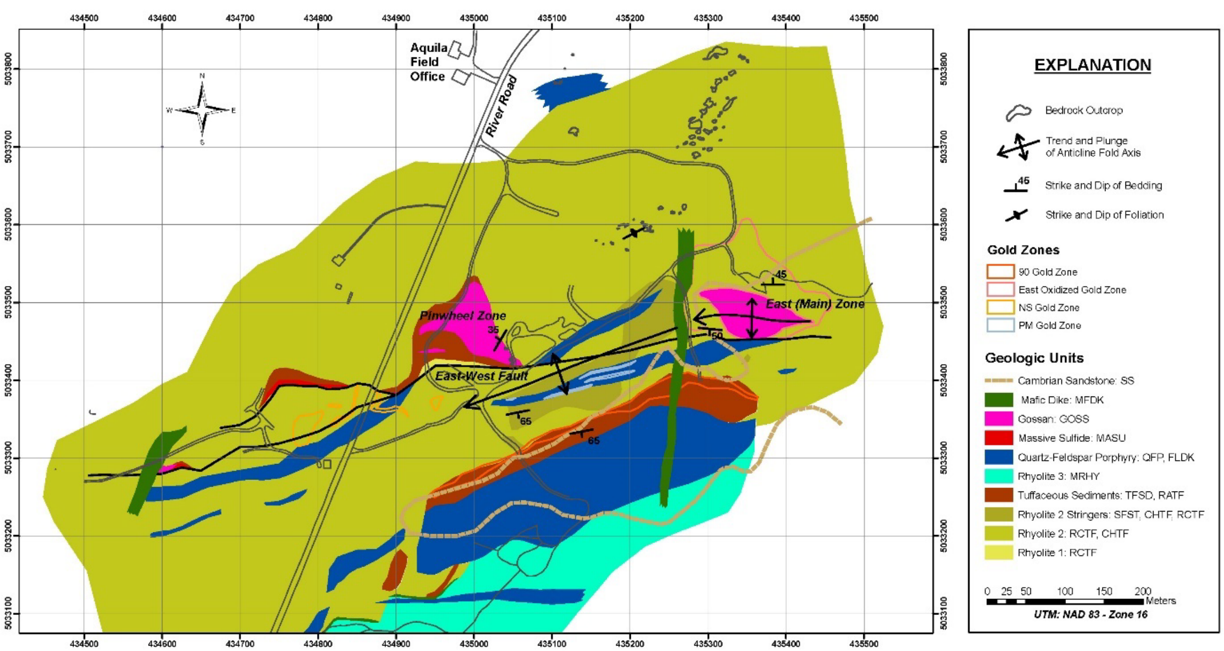
In the Menominee River valley, the PVB is unconformably overlain by erosional remnants of generally flat lying, Cambrian sandstone of the Munising Formation that coalesce into a coherent sandstone sheet approximately 600 m (1,969 feet) east of the Menominee River. The sandstone thickens and dips gently to the east, overlain by progressively younger sediments of the Michigan Basin further to the east.

The majority of western Menominee County is blanketed by an irregular thickness of unconsolidated sand, gravel, peat, and clay, deposited as the glaciers receded. Locally, water bearing sand and gravel formations are included in the glacial deposits. The thickest, most extensive deposits of sand and gravel occur along the Menominee River in the west-central part of Menominee County (Vanlier, 1963). In Lake Township, these deposits consist of predominantly glacial outwash sand and gravel and postglacial alluvium (Farrand, 1982).

7.3 PROPERTY GEOLOGY

Mineralization of the Back Forty Deposit is hosted within a succession of strongly altered felsic volcanic rocks interlayered with fine grained tuffaceous sediments which locally host strata-bound massive to semi-massive sulphide units. The volcanic stratigraphy has been intruded by a number of felsic to intermediate, syn-volcanic porphyry dykes and subsequently intruded by later dykes of mafic composition. The stratigraphy in the immediate deposit area is situated along an asymmetrical antiformal structure defined by a steeply dipping (65°) southern limb and a shallowly dipping (35°) north limb which steepens to the southwest. The antiformal structure plunges to the west-southwest at approximately 30°. Figure 7.5 shows the bedrock geology of the Back Forty Deposit.

FIGURE 7.5 BEDROCK GEOLOGY OF THE BACK FORTY PROJECT AREA



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

7.3.1 Lithology

Rhyolite crystal tuff ("RCTF") is the dominant lithology at the Project. Certain domains of the RCTF display a distinctive, often pervasive chlorite alteration. This rock is referred to as the chloritic crystal tuff ("CHTF"). Both of these units include quartz +/- feldspar phyritic rhyolite tuff, vitric tuff, pyroclastic or epiclastic breccias (or pseudobreccias) and other fragmental rocks. In other areas, particularly to the south of the deposit, distinctive massive, aphyric rhyolite flows ("MRHY") may represent a late felsic domal complex. Whole rock geochemistry and the observed geologic relationships indicate these rocks were formed in a volcanic environment. However, all of the lithologies have been pseudomorphically replaced by predominantly quartz with varying percentages of sericite, chlorite and pyrite; the result of being intensely altered, at least once, and recrystallized. In many cases, primary textures have been completely obscured. Elsewhere, pyroclastic textures are observed, but may in fact represent irregular alteration fronts producing pseudo-fragments. While these rocks are considered to be volcanic in origin, it is likely that some of this material may have been transported or reworked by gravity or water.

The RCTF and the CHTF are considered to be the same lithology except the CHTF shows moderate to strong chlorite alteration. In addition, the upper part of the CHTF commonly shows apparent clastic textures, suggesting it may have been transported and possibly reworked by epiclastic processes.

Volcanic-derived sedimentary units are interbedded within and often occupy the contacts between the felsic volcanic sequences. Tuffaceous sediments ("TFSD") consist of thin-bedded volcaniclastic sediments and tuffs interbedded with chert. Some rocks are very fine grained, foliated, and finely layered to massive sericite schist. This unit is composed almost entirely of sericite with minor amounts of quartz and may contain variable amounts of pyrite and base- and precious metal sulphides when associated with massive sulphide ("MASU"). Geochemically, this unit is very similar to RCTF and has been interpreted to be bedded rhyolite ash tuffs ("RATF"). Further analysis of this unit indicates it is found associated with deformed MASU and as thin units bounded by less deformed RCTF, so it may represent a sheared phase of RCTF.

Tuffaceous sediments ("TFSD") are similar to RATF but often show cherty or siliceous horizons interlayered with grey to white ash layers to produce a distinctive alternating pattern. This unit also includes lapilli tuff and volcaniclastic sediments. This unit can also be mineralized, hosting a relatively thin, undulating MASU layer with a very thin, gossan at the bedrock surface and low-sulphide, precious metal-bearing zones that are poorly defined.

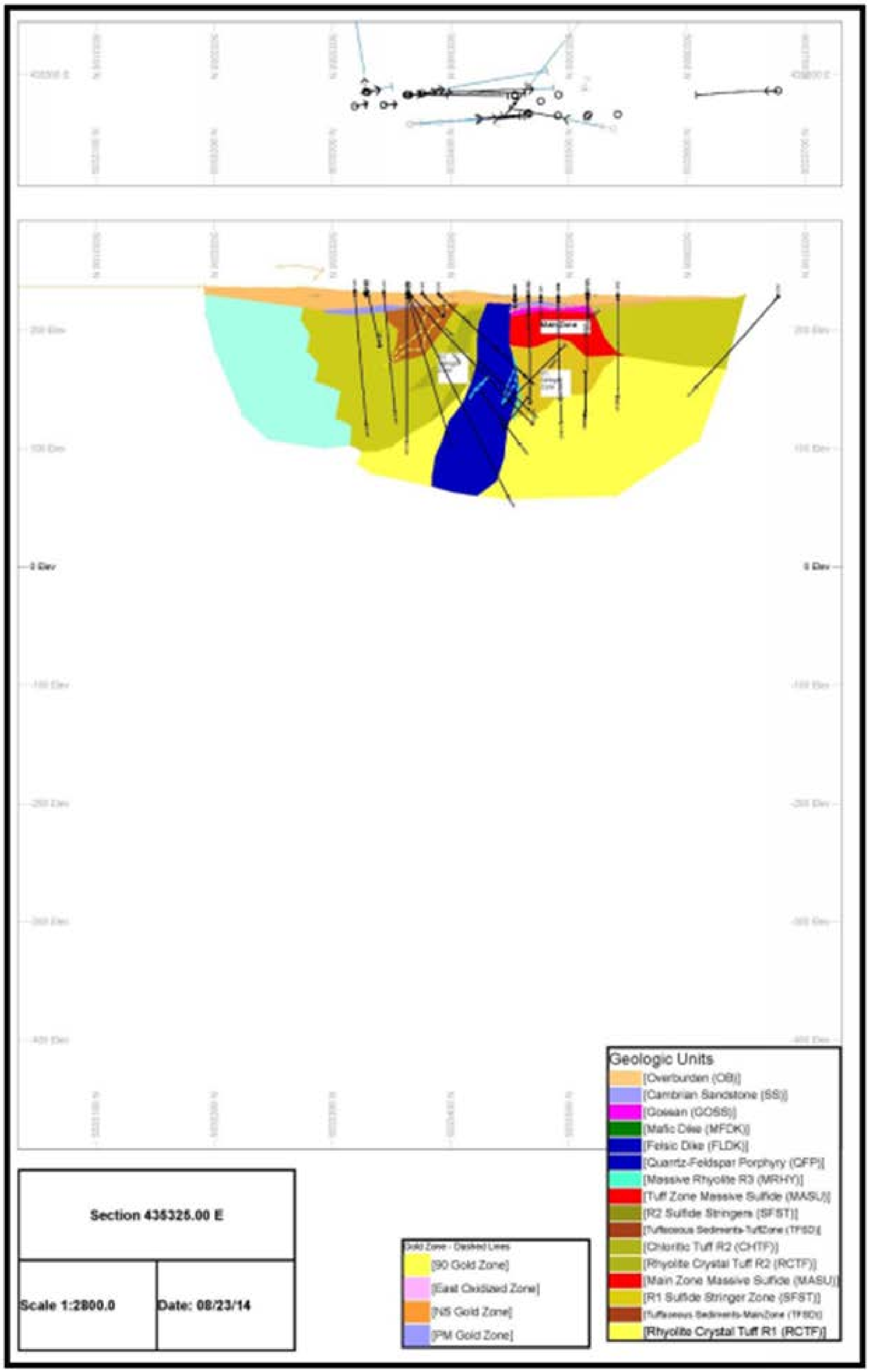
MASU and semi-massive sulphide ("SMAS") is typically associated with RATF and TFSD. On the Project, MASU refers to rocks composed of at least 80% sulphide and semi-massive sulphide is composed of 40% to 80% sulphide. Both also contain variable amounts of Zn, Cu, and Ag that occur most commonly as sulphides and Au found most commonly as finely divided native metal or as a natural amalgam. Sulphide stringer ("SFST") mineralization consists of 10 to 40% pyrite in veins with variable amounts of Cu, Ag, and Au that predominantly penetrates RCTF. Oxides and hydroxides of iron form a crudely bedded gossan ("GOSS") above MASU, where it is exposed at the bedrock surface. GOSS contains variable amounts of Au, Ag, and Cu and accessory minerals including chlorite and calcite. Certain intervals are characterized by significant amounts of finely laminated hematite and magnetite, and may represent an exhalative iron formation deposit.

Intruding the entire volcanic pile are several types of dykes and sills. Dacitic quartz feldspar porphyry (QFP) dykes and sills are the predominant intrusive rock. Intermediate to felsic dykes are locally abundant occurring as thin isolated intrusive units to multiple sheeted dykes that may represent reactivated chilled margins of QFP, or zones where QFP has partially to completely assimilated its host. These intrusive rocks have also been intensely altered, with sericite and biotite pseudomorphically replacing feldspar and hornblende, respectively. Thin, fault-bounded biotite lamprophyre dykes have been found in the southwest part of the Project Area. Several mafic dykes (MFDK), only moderately altered by chlorite, appear to cut all units and may represent the youngest intrusive in the Project Area.

Cambrian-age quartz sandstone overlies the east side of the known deposit and host rocks. This sandstone is a clast-supported quartz arenite, generally poorly cemented by calcite that grades downward into moderately silicified sandstone near the unconformity. Quaternary to recent alluvium blankets nearly the entire bedrock surface except for a relatively small area of Pre-cambrian rock exposures above 220 m in elevation. The alluvium consists of glacial till, muck, and sand, with post-glacial to recent alluvium within the Menominee River floodplain.

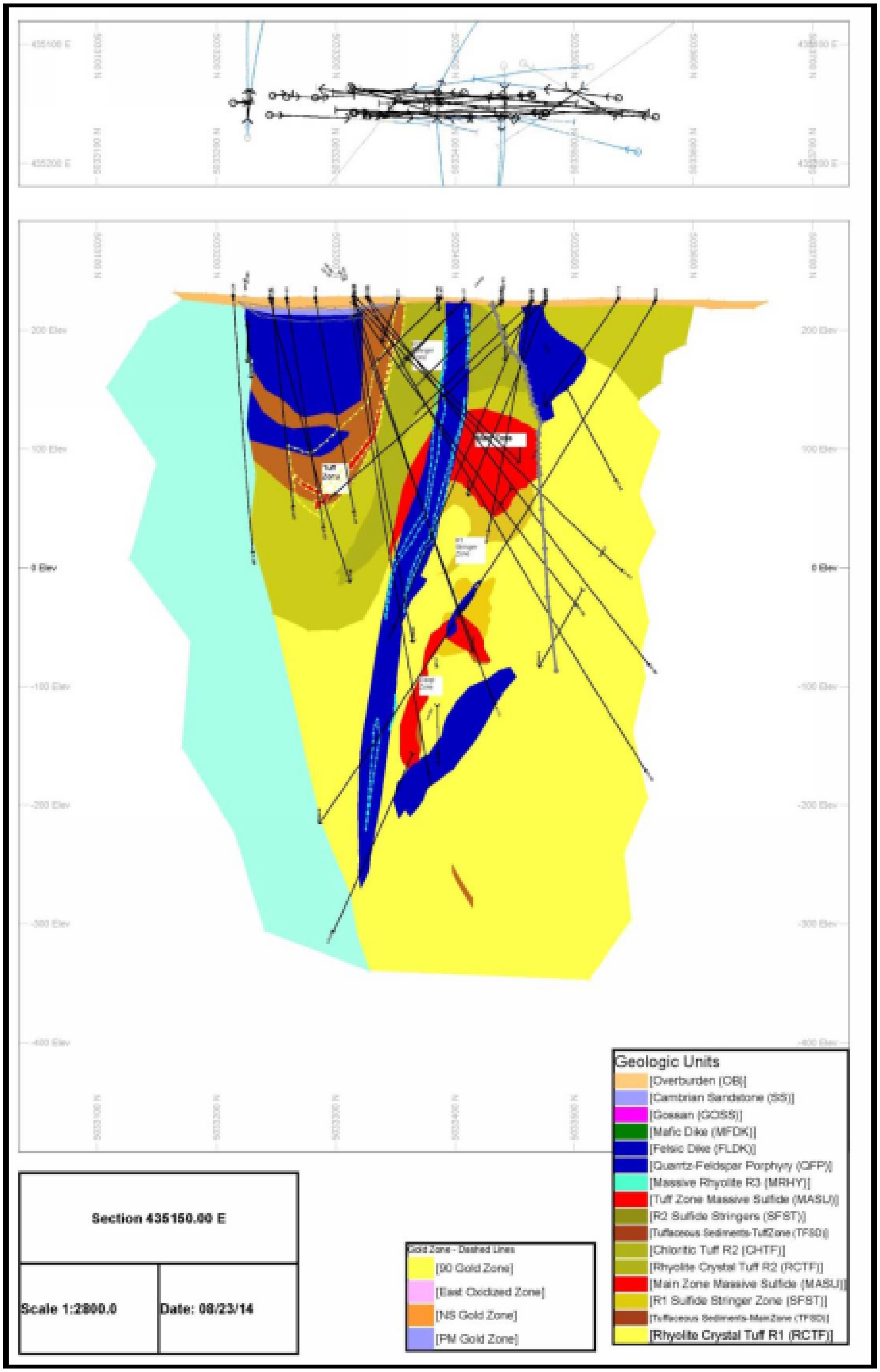
Three geologic cross-sections through the Deposit shown in Figures 7.6, 7.7 and 7.8.

FIGURE 7.6 EXAMPLE CROSS-SECTION (435325 E) THROUGH THE EASTERN PORTION OF THE DEPOSIT AREA



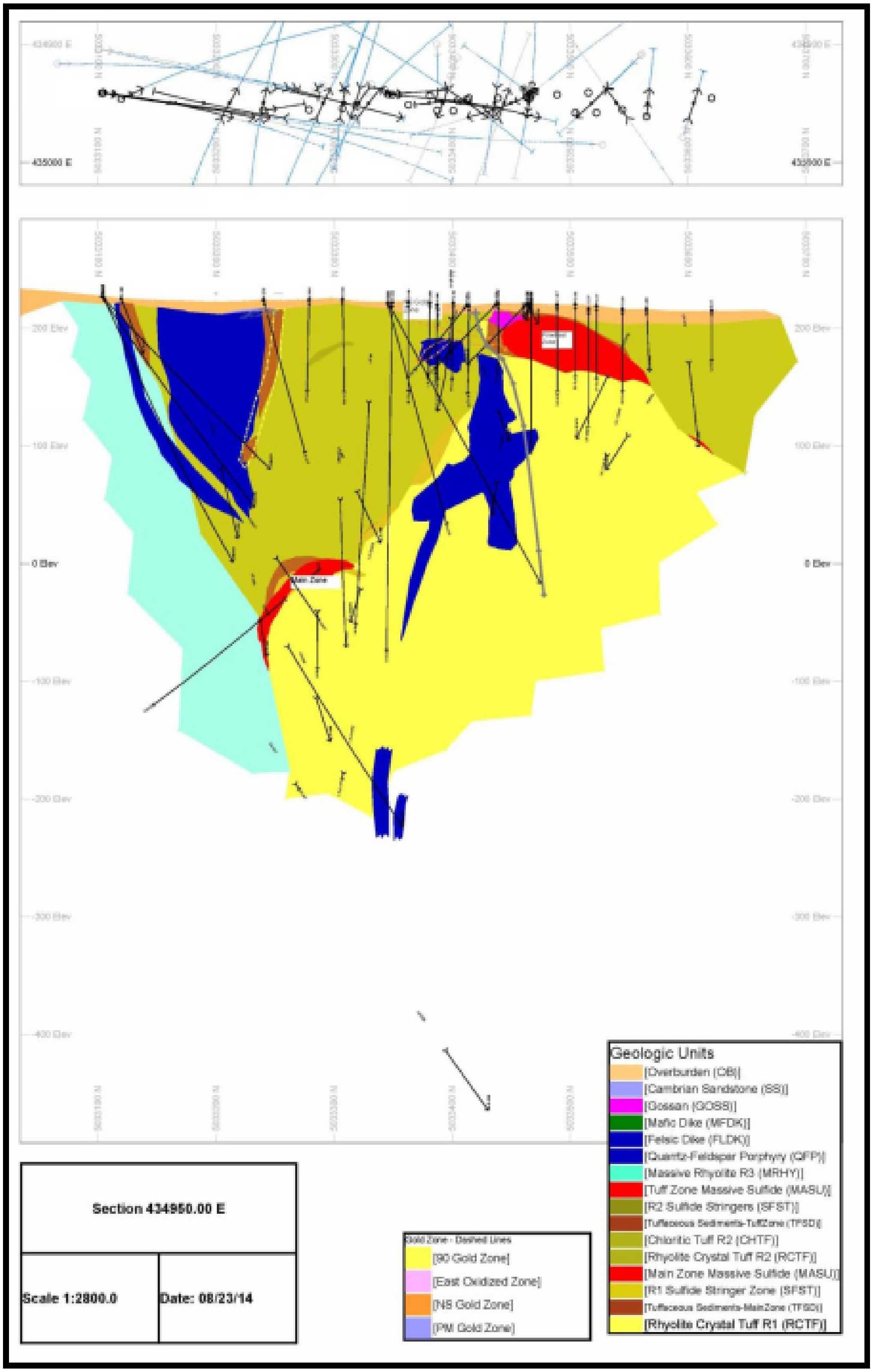
Source: *Tetra Tech Preliminary Economic Assessment (2014)*

FIGURE 7.7 EXAMPLE N-S CROSS-SECTION (435150 E) THROUGH THE CENTRAL PORTION OF THE DEPOSIT AREA



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

FIGURE 7.8 EXAMPLE N-S CROSS-SECTION (435150 E) THROUGH THE CENTRAL PORTION OF THE DEPOSIT AREA

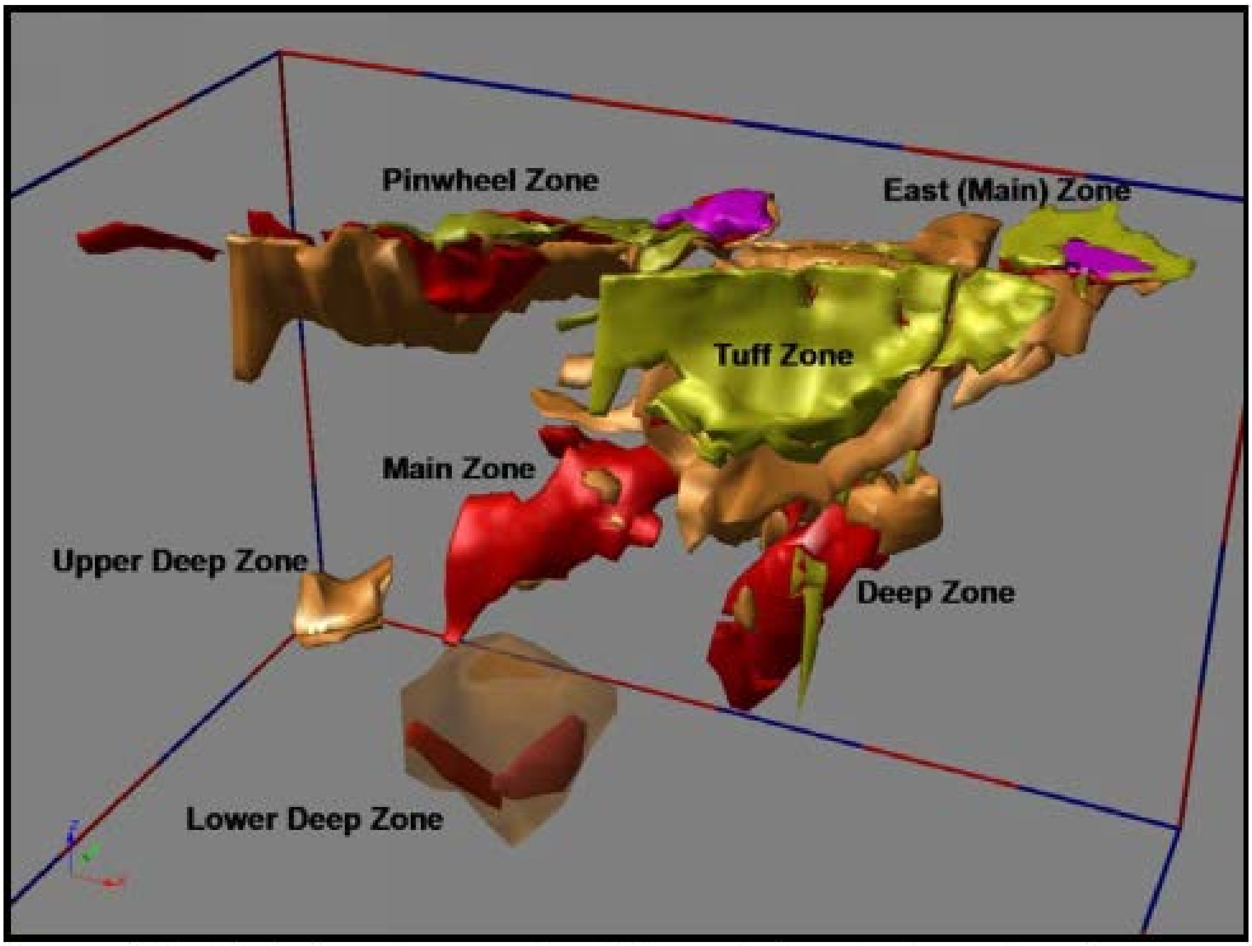


Source: *Tetra Tech Preliminary Economic Assessment (2014)*

7.4 MINERALIZED ZONES

Mineralization at the Back Forty Deposit consists of discrete zones of: 1) zinc or copper-rich massive sulphide (±lead), which may contain significant amounts of gold and silver, 2) stockwork stringer and peripheral sulphide, which can be gold, zinc, and copper-bearing (±lead/silver), 3) precious metal-only, low-sulphide mineralization, and 4) oxide-rich, precious metal-bearing gossan.

FIGURE 7.9 3-D MODEL OF THE MINERALIZED ZONES OF THE BACK FORTY DEPOSIT



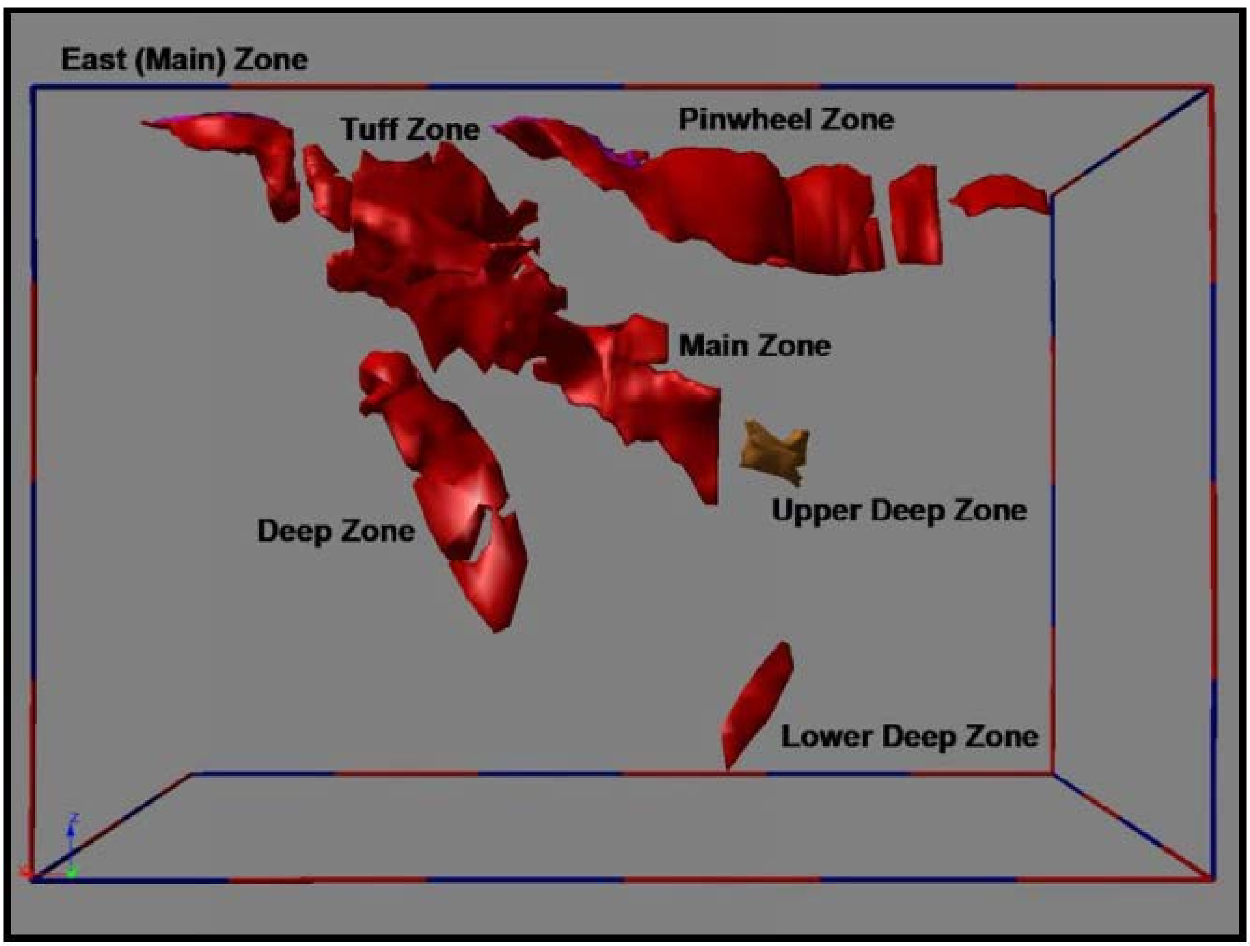
Note: Inclined view looking northwest; massive sulphide zones (red), stockwork and peripheral zones (tan), low-sulphur gold zones (yellow), and gossans (pink). Scale increments are 200 m.

Source: *Tetra Tech Preliminary Economic Assessment (2014)*

7.4.1 Massive Sulphide Mineralization

To date, VMS-style mineralization has been identified within at least two stratigraphic levels within the felsic sequence at the Back Forty Deposit (Figure 7.10). Although the majority of rhyolitic rocks hosting the Back Forty sulphide mineralization are indistinguishable with respect to appearance, the two main rhyolites (rhyolites 1 and 2) have distinctive geochemical signatures as can be observed through aluminium-titanium and zirconium-titanium ratios (Figure 7.11). The Main Zone massive sulphide, which accounts for the vast majority of massive sulphide mineralization lies at the statigraphic boundary of these two rhyolite units. Rhyolite 1 lies stratigraphically below this sulphide horizon (footwall) while rhyolite 2 lies above the horizon (hanging wall). Another massive sulphide horizon, the Tuff Zone, is located at or near the upper contact of rhyolite 2 and the lower contact of an overlying package of tuffaceous and siliceous sediments. A possible third massive sulphide horizon, the Deep Zone (Figure 7.10), may represent a lower mineralized zone. The general configuration of the massive sulphide horizons is shown in the 435,150 E cross-section (Figure 7.7). Additional drill intercepts of massive sulphide mineralization have been encountered at depth and to the southwest (down plunge) of known mineralization. Due to limited follow-up drilling of these intercepts it is, at the current time, unknown as to how these fit in with the overall geology and stratigraphy of the Deposit. In this section, massive sulphide refers to rocks composed of at least 80% sulphide, rather than the more common cut-off of 60% for massive sulphides. Semi-massive sulphide mineralization is considered to contain 10% to 80% sulphides.

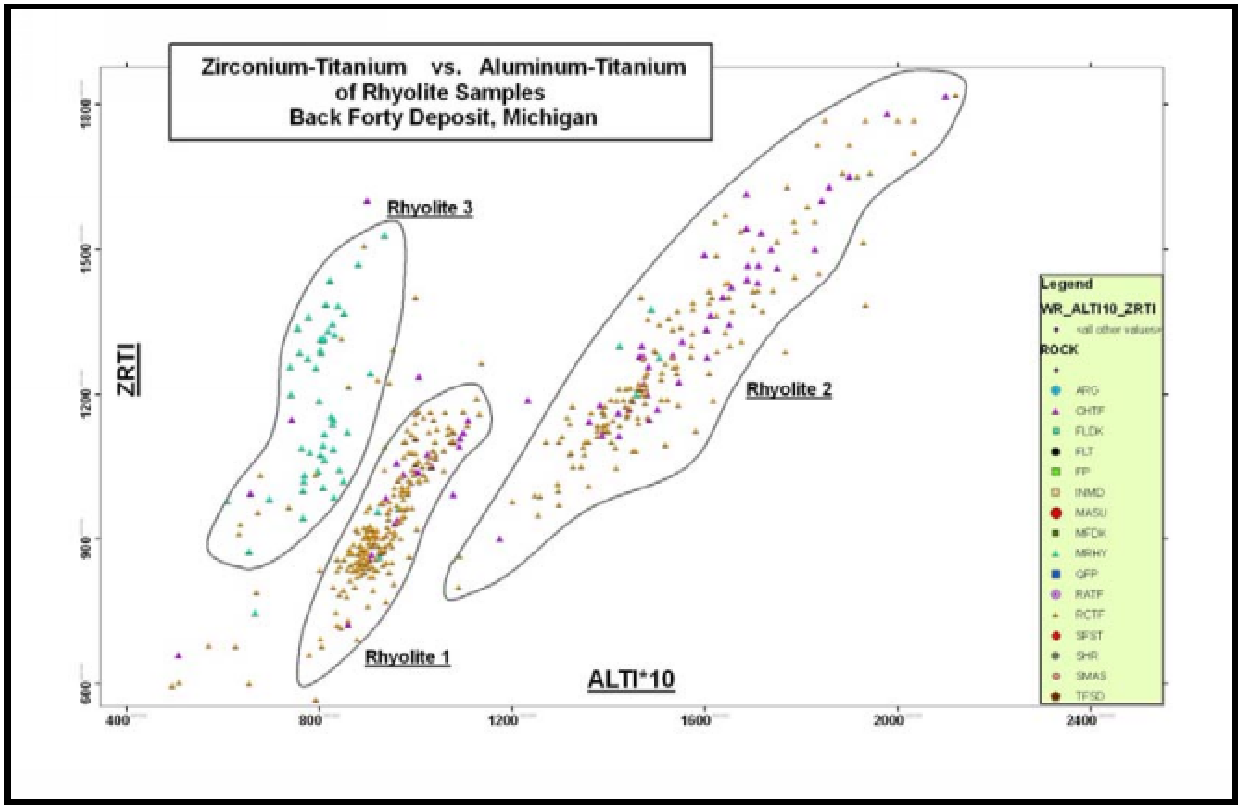
FIGURE 7.10 3-D MODEL OF THE MASSIVE SULPHIDE ZONES OF THE BACK FORTY DEPOSIT (LOOKING SOUTH)



Note: Scale increments are 200 m.

Source: *Tetra Tech Preliminary Economic Assessment (2014)*

FIGURE 7.11 ZIRCONIUM-TITANIUM VERSUS ALUMINIUM-TITANIUM RATIOS OF RHYOLITES HOSTING THE BACK FORTY DEPOSIT



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

7.4.1.1 Main Zone Massive Sulphide

The Main Zone is composed of three separate massive sulphide bodies (referred to as the East, Hinge, and South Limb Zones) that form a plunging antiform and are considered the same horizon. These bodies are hosted by Rhyolite 1 (footwall) along and stratigraphically below their contact with Rhyolite 2 (hanging wall). The strata-bound Main Zone is enveloped locally by stockwork and semi-massive sulphide mineralization. Pervasive sericite and disseminated pyrite alteration as well as variable silicification are abundant and extend outward for an undetermined distance. This zone extends along strike for over 450 m in a west-southwest direction; it is up to 100 m wide and subcrops at its eastern end under thin (less than 10 m) glacial overburden and local Paleozoic sandstone. The stockwork-stringer and peripheral sulphide envelope grades outward into a semi-conformable disseminated (less than 10%) pyritic halo that extends throughout the entire altered Rhyolite 1 host unit for an undetermined distance. The zone has been extensively disrupted by variably altered quartz feldspar porphyry ("QFP") intrusions.

The East Zone sub-crops east of a north-south-trending mafic dyke that appears to occupy a fault. West of the dyke the massive sulphide has been displaced downward approximately 20 m. This massive sulphide body is capped by a thin gossan (generally 3-5 m thick) and erosional outliers of Paleozoic sandstone. At the top of the massive sulphide, directly underlying the gossan is a thin zone of copper-rich massive sulphide (often less than 1-2 m) which was likely enriched by means of late supergene processes.

The Hinge Zone, in part offset by faulting, has been folded tightly into a cigar-shaped body that plunges moderately at approximately 40° to the southwest along the axial plane of the antiform; the South Limb is separated from the Hinge by a laterally persistent QFP dyke and remains open to the southwest. Further west, the horizon is apparently offset downwards again between Sections 435,225 E and 435,200 E. Between sections 435,200 E and 435,100 E, deformation of the Hinge Zone has apparently resulted in tectonic thickening of this unit up to approximately 70 m. Beyond Section 435,100 E to the west, the Hinge horizon appears to pinch out.

The South Limb Zone is interpreted to represent the southern limb of the antiform; dipping steeply to the south, while plunging to the west-southwest. This interpretation is supported by lithogeochemical data. Locally, shearing is common, resulting in an overall uniform thickness and lens-shaped geometry.

The Hinge and South Limb Zones are separated by large, variably-altered QFP dykes that appear to have intruded the axial plane of the antiform. These syn- or post-mineralization QFP intrusions have intruded, cut-off, and obliterated portions of both horizons. To the west, the model suggests that the South Limb may be pinching and swelling down plunge into a series of thin to thick lenses. Drilling continues to support the above interpretation. The South Limb remains open along strike.

7.4.1.2 Pinwheel Zone Massive Sulphide

The Pinwheel Zone occupies the northwest portion of the Deposit, located structurally along the gently north-dipping northern limb of the antiform and is truncated to the south by the E-W fault. Limited geochemical data suggests that this unit is in fact located along the contact between rhyolite 1 and rhyolite 2 and is therefore likely the equivalent to the Main Zone massive sulphide and represent a ‘faulted-up’ portion of the north limb of this important massive sulphide horizon. Massive sulphide mineralization on strike of the Pinwheel Zone has been traced for roughly 700 m to the west-southwest where the gentle north-dip of the unit steepens. It should be noted, however, that the massive sulphide mineralization is to some degree discontinuous and often has a ‘stacked’ geometry, and that numerous faults and shear zones have been encountered in the adjacent host rock. The geometry of this zone is likely complicated due to these structures.

The Pinwheel Zone is broken up in to two separate units based on spatial relationships and dominant mineralization types. The near-surface, gently north-dipping eastern-most portion of the Pinwheel Zone is referred to as the ‘Pinwheel Cu-Rich Zone’ due to the relative abundance of copper mineralization (predominantly pyrite + chalcopyrite) and subsequent lack of other base metals (zinc and lead) within the massive sulphide. The majority of the Cu-Rich Zone is capped by an overlying gossan that crops out on the Property along the southeast terminus of the zone. The Cu-Rich portion of the Pinwheel Zone represents the most copper-enriched massive sulphide mineralization located at the Back Forty Deposit and it is interpreted that the copper enrichment has a secondary, supergene association. It is possible, however, that this zone represents an original, high-temperature, copper rich portion of the VMS system. Along strike to the west-southwest, copper-dominant mineralization diminishes with a subsequent increase in the presence of zinc (sphalerite) and to a lesser extent lead (galena). This zone has been referred to as the ‘Pinwheel Extension’ or ‘Pinwheel Zn-Rich Zone’ and the variation in metal content with respect to the Cu-Rich portion is interpreted to be due, in part, to a lack of influence from secondary, super-gene processes.

7.4.1.3 Deep Zone Massive Sulphide

The Deep Zone is located north of one of the QFP dykes, juxtaposed against the South Limb Horizon. Recent geological and geochemical data interpretation suggests that the Deep Zone may be the down-dip continuation of the South Limb, where it has been folded and rotated. This interpretation leaves significant spatial potential for further Mineral Resource discovery between the South Limb and the Deep Zone as well as down dip of the Deep Zone.

The Deep Zone is relatively enriched in copper compared to zones of the Main Zone (East, Hinge, and South Limb) and suggests that a more copper-rich portion of this VMS system may occur at depth.

7.4.1.4 Tuff Zone Massive Sulphide

The Tuff Zone massive sulphide occurs at the south edge of the Deposit. Stratigraphic and structural data suggest this zone is located at a higher level in the volcanic sequence. In cross sections and three-dimensional models, the zone appears to have a bowl-shaped geometry possibly reminiscent of a small relict depositional basin or local graben structure.

The Tuff Zone is hosted at or near the stratigraphically upper portions of the intensely sericitized and locally chlorite-altered Rhyolite 2 unit as well as within the lower portion of the overlying siliceous, tuffaceous sediment unit. The Tuff Zone has been traced along strike to the southwest by drilling (parallel to the Main Zone) for roughly 250 m. The zone is predominantly steeply dipping to the south and occupies the southern limb of the antiform. Drilling intercepts down dip and at depth of the zone indicate shallowing and flattening of the unit that suggests proximity to a synclinal structure to the south. Massive sulphide mineralization of the Tuff Zones appears preferentially developed within coarser grained tuffaceous units at or near the contact of rhyolite 2 and of the overlying tuffaceous and siliceous sediments. Overall sulphide content is less massive than that of the Main Zone (~60%-80%) and is dominated by sphalerite, pyrite, and galena. The zone’s thickness is typically on the order of a couple of metres. The horizon possibly subcrops in the northeast along Sections 435,175 E and 435,150 E but plunges southwest (to at least Section 435,000 E) similar to the orientation of the massive sulphide horizons of the Main Zone.

7.4.2 Stockwork and Peripheral Sulphide Mineralization

Widespread and pervasive sulphide mineralization occurs throughout the host rocks and peripheral to the massive sulphide bodies at the Back Forty Deposit in the form of stockwork-stringer sulphides, massive to semi-massive discontinuous sulphide lenses, and disseminated sulphides. Geochemical data suggests that there are at least two distinct zones of peripheral sulphide mineralization that formed stratigraphically below each of the massive sulphide horizons and can be differentiated from each other by their relative location in the stratigraphy and relative metal content. Sulphide mineralization that occurs within rhyolite 1 appears to have a mineralogical affinity with the Main Zone and Deep Zone massive sulphides and consist of predominantly pyrite (commonly gold-bearing), minor to moderate sphalerite with variable amounts of chalcopyrite. Peripheral sulphide mineralization found within rhyolite 2 is similar mineralogically to the Tuff Zone massive sulphide unit that lies at or near the stratigraphic top of rhyolite 2. As with the associated Tuff Zone Massive sulphide, this zone is notably copper-poor and is instead relatively enriched in zinc and lead. As with the rhyolite 1 sulphide mineralization, it is locally gold-bearing but is also relatively enriched in silver. It is assumed that both peripheral sulphide zones were formed contemporaneous with the hydrothermal system associated with the formation of associated massive sulphide mineralization.

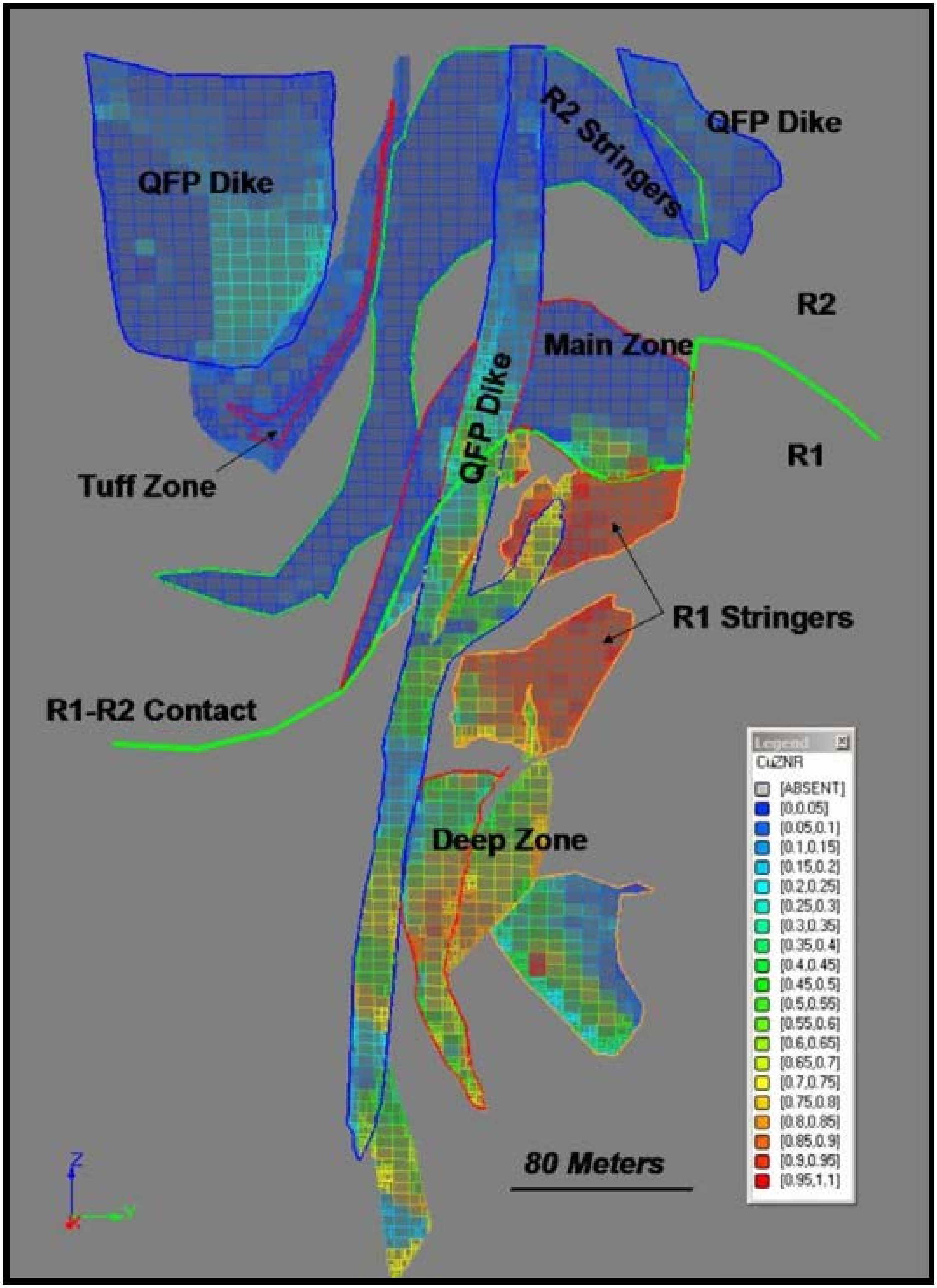
7.4.3 Copper Mineralization Associated with Sulphide Mineralization

Drilling has shown that the VMS mineralization of the Main and Tuff Zones is generally copper-poor. Copper mineralization; however, appears to be more prevalent in the lower stratigraphic horizons of the Deposit. Copper/Zinc ratios show a marked decrease that occurs at the rhyolite 1-rhyolite 2 contact (Figure 7.12) indicating that fluids responsible for the Main Zone massive sulphide formation and underlying peripheral sulphide mineralization (stringers, disseminations, etc. of rhyolite 1) were of higher temperature and carried more copper compared to those fluids associated with sulphide mineralization in the stratigraphically higher sulphide zones (Tuff Zone and Rhyolite 2 peripheral sulphides). Metal zonation of copper and zinc is also apparent within the massive sulphide units of the Main Zone and is often contiguous with similar metal zonation within the underlying peripheral sulphide mineralization. A preliminary look at the data suggests that high temperature copper-bearing hydrothermal fluids may have been focused at a number of different locations along the strike of the Main Zone horizon. Examples of this occur along the south-western portion of the East Zone near section 435,300 E, in proximity to the thickened portion of the Hinge Zone near section 435,150 E, and at the near surface, copper rich portion of the Pinwheel Zone near section 435,000 E.

The most consistent and highest-grade copper mineralization has been intersected in the Pinwheel Zone. As discussed in previous sections, the Pinwheel Zone was likely enriched in copper due to late genetic zone refining (supergene) processes and it is unknown whether the Pinwheel Zone originally represented a more copper-rich part of the VMS system.

Elevated copper values (above 0.5%) have been intersected in the Deep Zone at a depth of approximately 400 m. Mineralization consists of a deformed massive to semi-massive sulphide zone within the core of the antiform. Elevated copper values are related also to a late remobilization event that resulted in the formation of chalcopyrite-pyrite stockwork mineralization under the Deep Zone. This mineralization was intersected by drill hole 108470 between 393.2 m and 403.7 m. The sulphide mineralization is clearly late and overprints thin massive sulphide bodies and altered/mineralized quartz crystal tuff of the Rhyolite 1 unit; chalcopyrite veins, replacement blebs and wisps are common. The 24 m interval graded 1.42% copper and 0.84 g/t gold.

FIGURE 7.12 CROSS-SECTION THROUGH THE BACK FORTY DEPOSIT (SECTION 435,125 E) SHOWING CU/ZN VALUES FOR THE MINERALIZED ZONES WITHIN THE BLOCK MODEL



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

Late chalcopyrite-pyrite overprinting also occurs locally as found in the extension of borehole 108422, where a tuffaceous sedimentary rock unit was intersected. The hole intersected chalcopyrite-pyrite-pyrrhotite-quartz veins and stringers from 500 to 508 m; the veins and stringers disrupt and locally brecciate a section of delicately laminated exhalite consisting of alternating layers chert and pyrite.

7.4.4 Precious Metal-Rich Low-Sulphide Mineralization

In addition to gold values in both the massive and stockwork-stringer sulphide zones, significant gold and silver mineralization in surrounding low sulphur host-rocks has been identified. This mineralization typically contains less than 10% sulphide, although it may contain more locally. Host rocks include tuffaceous sediments and underlying rhyolite tuff (the 90 Zone), sheared rhyolite (NS Zone), and quartz-feldspar porphyry (PM Zone).

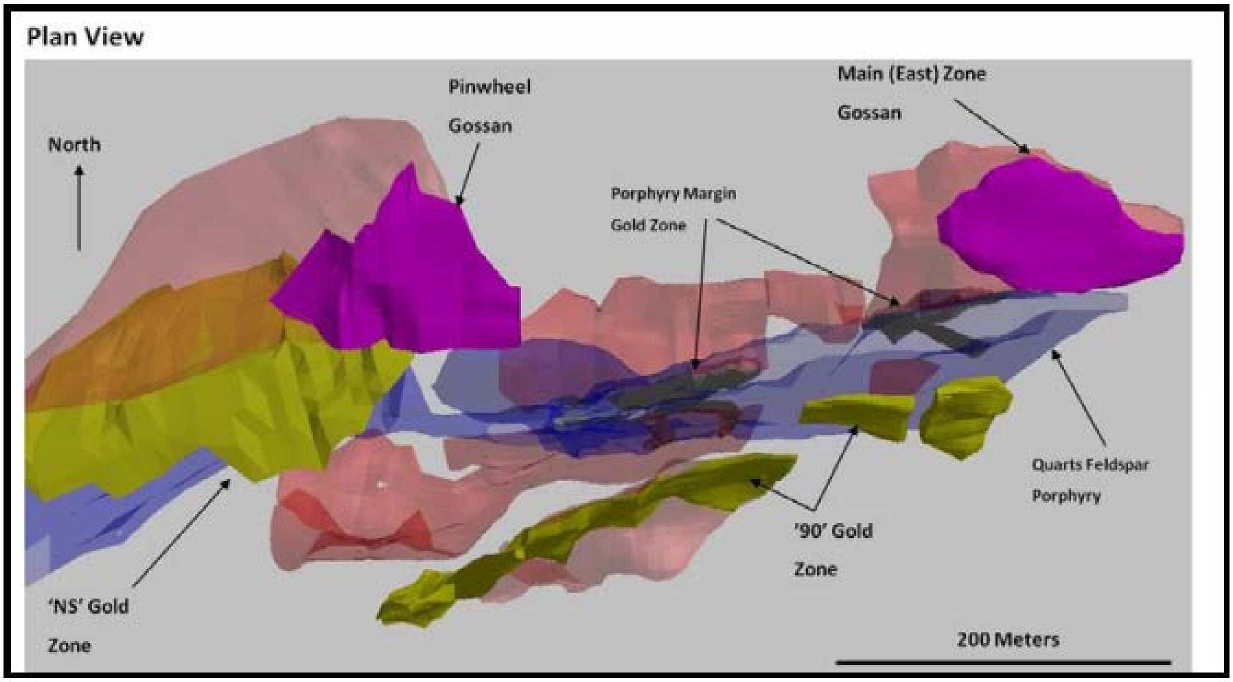
Gold mineralization of the “90” Zone occurs within the tuffaceous and siliceous sediments and underlying rhyolite 2 adjacent to and up dip of the Tuff Zone massive sulphide horizon. While gold grades vary somewhat throughout the zone, it appears that the higher-grade portions of the zone occur along the same horizon as the Tuff Zone, but up dip from massive sulphide mineralization where the host rock is typically increasingly silicified, altered, and fractured. The zone consists of 5%-10% disseminated pyrite with lesser arsenopyrite, chalcopyrite and galena. Gold mineralization is pervasive throughout the sediment package but in higher-grade portions of the zone, gold mineralization appears to be structurally controlled and likely cross cutting the section. Some near-surface gold may have been concentrated by supergene oxidation, although some high-grade gold occurs in unoxidized rocks as well. The 90 Zone extends from a depth of approximately 170 m to sub-crop just below the overburden and for 400 m on strike.

Locally, high-grade gold and silver of the NS Zone occurs in sheared rhyolites and rhyolite ash tuffs. Sulphide contents vary from trace to 5% and consist mainly of disseminated pyrite and galena with rare arsenopyrite. The mineralization is generally flat lying and shallow. Stratigraphically and structurally it appears to be a southward continuation of the Pinwheel Zone/Pinwheel Gossan across the east-west fault.

Gold mineralization is also hosted by the large QFP intrusion that occupies the hinge of the antiform (Porphyry Margin or PM Zone) (Figure 7.13). Gold mineralization in the PM Zone typically occurs near the contact of the intrusion in the east and central portions of the deposit area proximal to massive and heavily mineralized sulphide zones. Gold mineralization is associated closely with pervasive, moderate to intense silica alteration. Sulphide mineral assemblages include fine to medium-grained pyrite, arsenopyrite, galena, sphalerite, chalcopyrite, pyrrhotite and locally visible free gold. Sulphides occur as disseminated grains, blebs, veinlets and wisps; thicker sulphide veinlets have a chlorite halo.

In general assay data indicate good correlation between lead-copper and gold values for all zones.

FIGURE 7.13 PLAN VIEW OF THE BACK FORTY MINERALIZATION AND CROSS-CUTTING QFP DYKE HIGHLIGHTING GOLD ZONES



Note: *Gold zones are shown in yellow.*
Source: *Tetra Tech Preliminary Economic Assessment (2014)*

7.4.5 Gossan (Supergene) Mineralization

Near surface iron-oxide and precious metal-rich gossans cap the East Zone and the Pinwheel Zone. Both are consistently enriched in gold and in the case of the Pinwheel Gossan, silver and copper.

The East Gossan caps the East Zone Massive Sulphide. It subcrops under glacial overburden and at its eastern extent is capped by Paleozoic sandstone. The East Gossan is generally 3 m to 5 m thick, hematitic and gold-rich.

The Pinwheel Gossan caps the Pinwheel Zone massive sulphide in the west-central portion of the Deposit. The gossan locally exhibits a brecciated texture, probably formed by collapse of stratigraphic hanging wall rocks during volume reduction from supergene oxidation of the massive sulphide horizon. Botryoidal textures exhibited by iron minerals and poorly developed cellular boxwork have also been noted locally in the outcrop. In addition, columnar hematite/limonite observed in outcrop suggests supergene deposition above the water table, with the iron derived from adjacent massive pyrite (Blanchard, 1968). Remnant disseminated magnetite in the gossan suggests a cherty iron formation (exhalite) protolith in this portion of the zone.

7.4.6 Mineralization Encountered at Depth

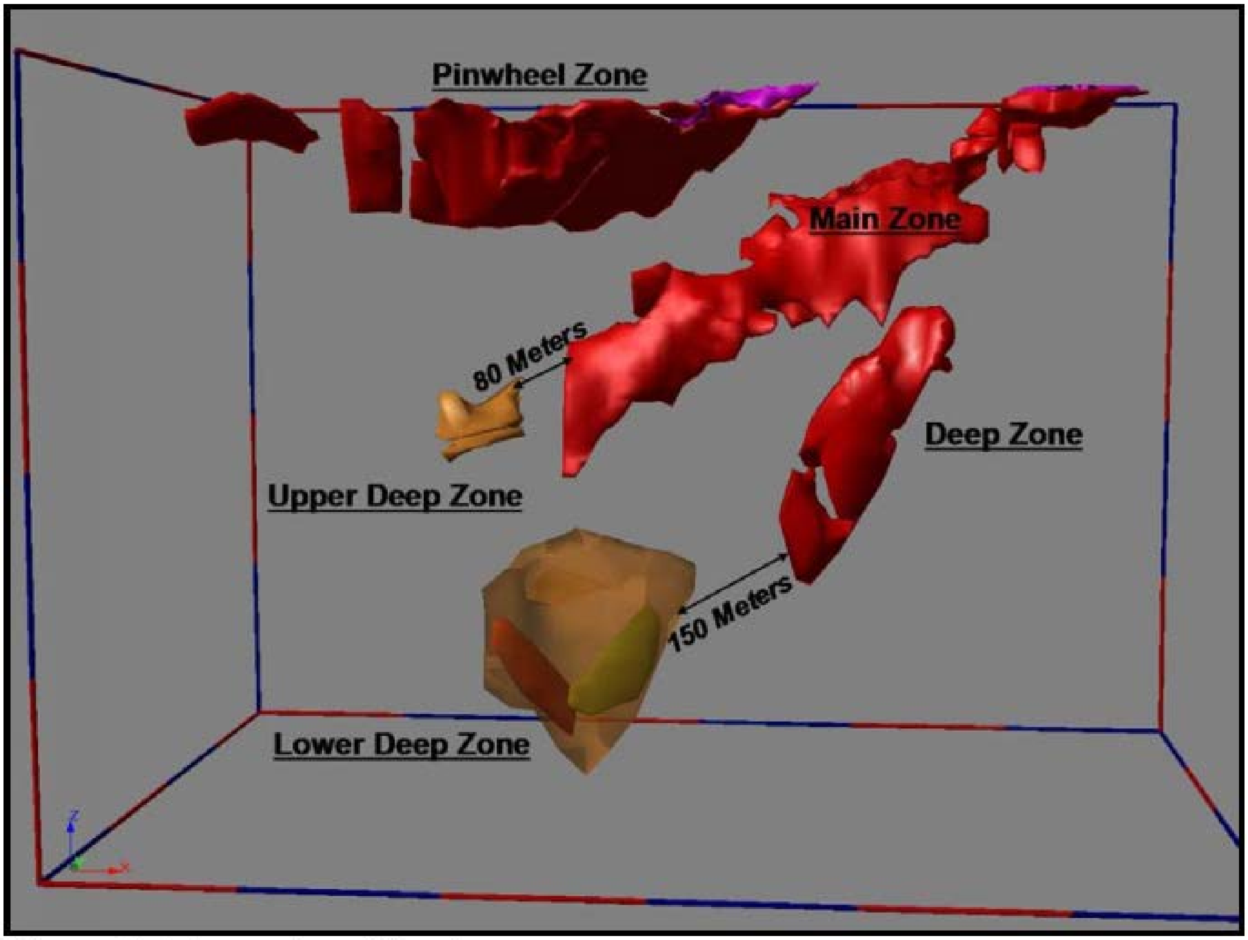
Drilling, primarily completed during 2010 and 2011, has encountered a number of mineralized zones at depth and along strike of known mineralization. Limited follow-up drilling, and a large gap in the drilling between these newly discovered zones and known mineralization, has made it difficult to correlate these newly discovered zones of mineralization to the current geologic model.

Mineralization has been encountered in two separate stratigraphic horizons which have tentatively been called the ‘Upper Deep’ and ‘Lower Deep’ zones. LK-479, drilled to the southwest of known mineralization and to a total depth of 911 m, encountered each of the two zones and represents the ‘discovery’ hole for each.

The upper mineralized section in drill hole LK-479 intercepted ~60 m of strongly altered rhyolite and tuffaceous and siliceous sediments with variable sulphide mineralization in the form of disseminated and stringer sulphides as well as two small intervals of high-grade massive sulphide. The mineralization was also cut by a quartz-feldspar porphyry dyke that contained PM-style gold mineralization. Assayed intervals (drilled thickness) within the section include: 12 m (366-278 m) of 1.25 g/t Au, 57.9 g/t Ag, 0.41% Pb, and 1.0% Zn and 12.8 m (407.3-420.1 m) of 4.27 g/t Au, 189.1 g/t Ag, 1.4% Pb, and 4.2% Zn. The metal content of this zone, as well as the spatial relationship to the tuffaceous and siliceous sediment unit would suggest that this zone may be related to the Tuff Zone type sulphide mineralization, however, limited and preliminary geochemical data indicates that the mineralization may reside at the contact of rhyolite 1 and 2 which would imply that this zone is related to Main Zone mineralization. It is also interesting that the high-grade massive sulphide encountered in this section appears to cross-cut bedding which would indicate that higher-grade mineralization may be related to a cross-cutting structure. A number of drill holes attempted to follow-up on this intercept and encountered similar mineralization in altered rhyolite and tuffaceous and siliceous sediments, but grades were typically less substantial than that of LK-479. The zone lies roughly 80 m along strike, to the southwest of the south limb of the Main Zone massive sulphide.

The lower mineralized section in drill hole LK-479 intercepted roughly 68 m of massive sulphide overlain by stockwork stringer type mineralization. Metal content of the massive sulphide was generally zinc-poor with a relative enrichment in copper which bears some similarities to the Deep Zone massive sulphide unit. The massive sulphide was also cut by a quartz-feldspar porphyry dyke containing PM Zone style gold mineralization. The upper portion of the massive sulphide intercept consisted of 45 m (747.4-792.4 m) of 0.72 g/t Au, 21.9 g/t Ag, 0.47% Cu, and 0.7% Zn. The lower portion of the massive sulphide contains local high-grade zinc including 13.46 m (802.34-815.8 m) containing 0.31 g/t Au, 17.7 g/t Ag, 0.36% Cu, and 3.7% Zn. The mineralized dyke (PM Zone) cutting through the massive sulphide consisted of 6.18 m (793.14-799.32 m) that assayed 6.4 g/t Au, 94.1 g/t Ag, and 1.82% Pb. Additional PM Zone intercepts were also reported further down the hole within a larger intercept of quartz-feldspar porphyry dyke. The two intercepts varied between 1 m and 3 m and assayed between 3 and 5 g/t Au. Massive sulphide and associated peripheral sulphide mineralization, along with local high-grade gold has been encountered in limited follow-up drilling including a section of strongly altered, quartz vein bearing rhyolite containing visible gold that was encountered in drill hole LK-484 containing 12 m (708-720 m) of 15.29 g/t Au, 66.5 g/t Ag and 0.77% Pb). Drill spacing at this time is inadequate to delineate the various mineralized zones. Mineralization associated with this zone is located roughly 150 m to the southwest, along strike of the known mineralization of the Deep Zone massive sulphide (Figure 7.14).

FIGURE 7.14 PLAN VIEW OF THE BACK FORTY MINERALIZATION AND CROSS-CUTTING QFP DYKE HIGHLIGHTING MINERALIZATION ENCOUNTERED AT DEPTH



Note: Scale increments are 200 meters

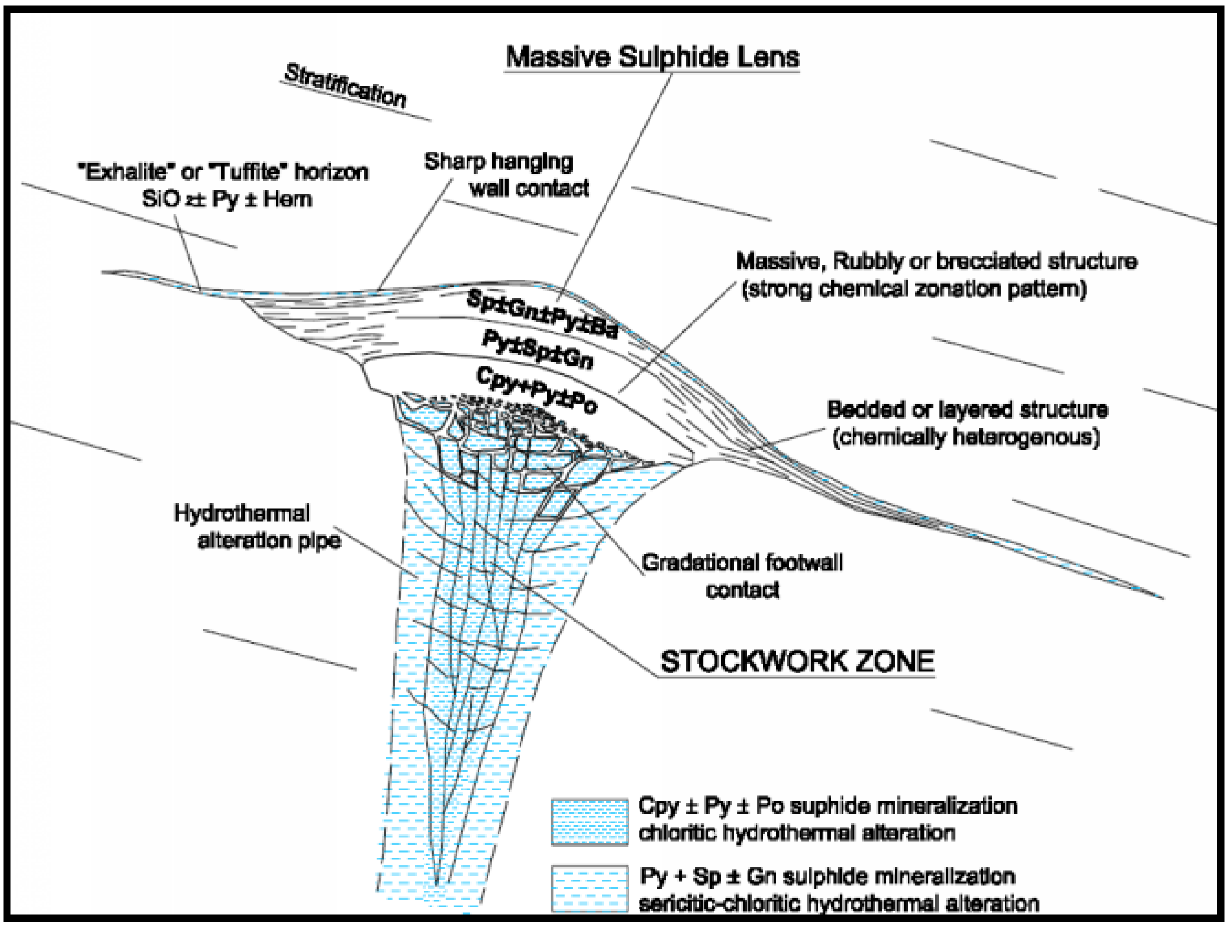
Source: *Tetra Tech Preliminary Economic Assessment (2014)*

8.0 DEPOSIT TYPES

Section 8.0 has been modified from Connolly et al. (2012).

The zinc-copper-gold-silver bearing sulphide mineralization identified on the Back Forty Property exhibits typical characteristics of VMS mineralization (Figure 8.1). This deposit type has been well documented in the literature since the early 1970s (Franklin et al. 1981) and the exploration model for the PVB was refined after the discovery of Flambeau (DeMatties et al. 1996).

FIGURE 8.1 SCHEMATIC CROSS-SECTION THROUGH A VMS MOUND

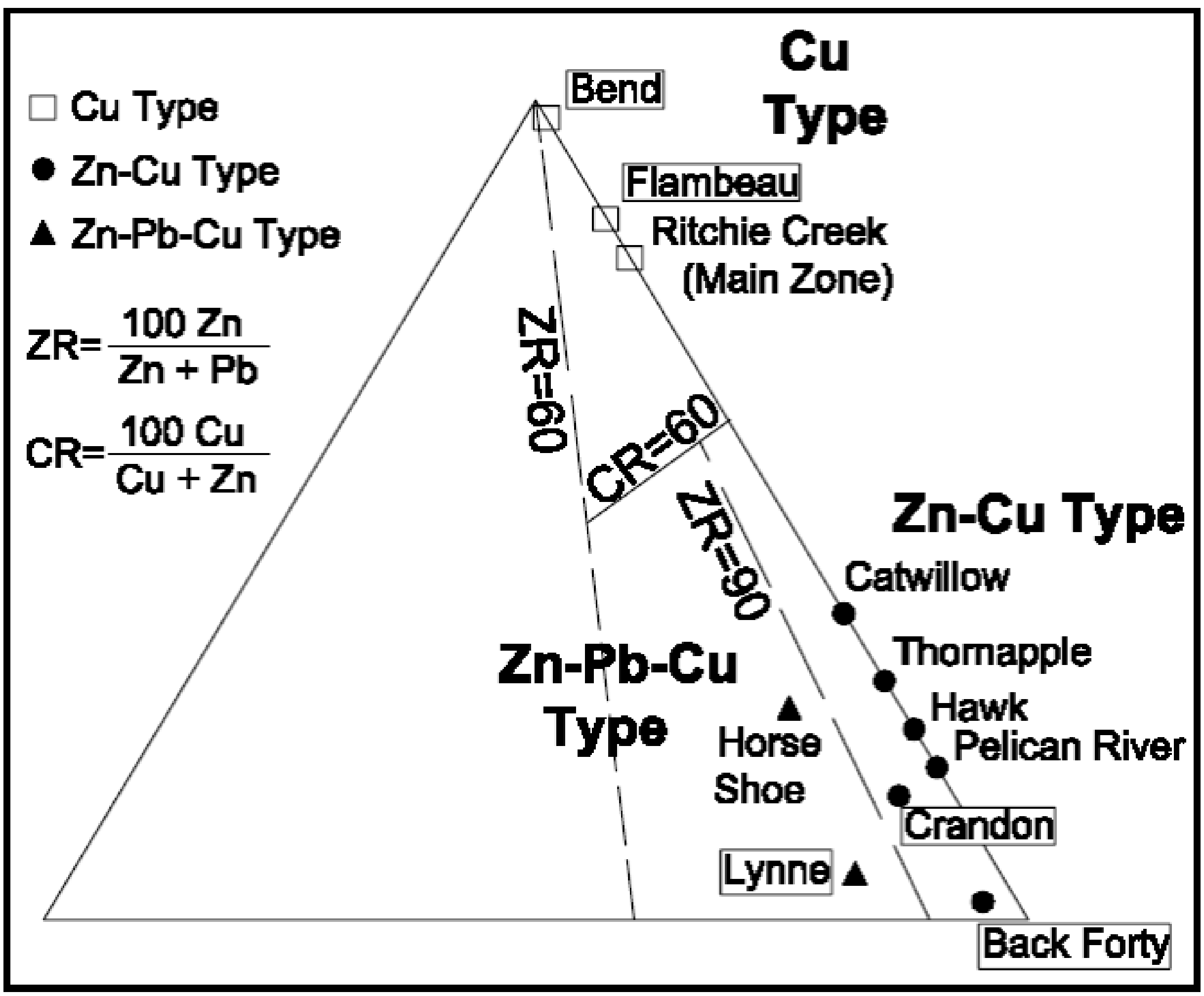


Source: *Tetra Tech Preliminary Economic Assessment (2014)*

VMS deposits form in a marine volcanic environment by the circulation of hot hydrothermal fluids near spreading centres. Cold seawater infiltrating ocean crust off-axis is progressively heated by hot magma underlying the rift zone. Heated and buoyant fluids leach metals from the surrounding rocks. Metallic sulphides precipitate at or near the rock-water interface as a result of rapid changes in Eh and pH triggered by rapid mixing with cold ambient seawater. Precipitated sulphides form massive mounds, fracture and cavity fills, as well as replacement textures. Metal zoning is common with copper-rich zones at or near the centre and zinc-rich zones at the fringes of a sulphide mound. Multiple emplacement and zone refinement are common, often due to changes in the internal plumbing system. VMS deposits are known in the Precambrian (e.g. Kidd Creek, deposits in the Noranda Camp), the Paleozoic (e.g. Rammelsberg), Mesozoic (e.g. Windy Craggy, Cyprus), and Cenozoic (e.g. Tag).

The lower Proterozoic rocks of the PVB are no exception and although only one deposit has been mined the belt hosts many small occurrences to large deposits. Together, approximately 14 known VMS deposits account for over 150 Mt of base and precious metal mineralization. The average deposit size in this district is approximately 2.5 Mt; this average is high compared to other VMS districts worldwide (DeMatties 1994). The identified VMS deposits are classified by metal content into three groups based on zinc and copper ratios (DeMatties 1994 and 1996). These include copper, zinc-copper, and zinc-lead-copper (Figure 8.2). Each group exhibits various styles of mineralization that include sheets, mounds, stacked lenses, and replacements. Calculated zinc and copper ratios for the Back Forty deposit place it in the zinc-copper group; which is the dominant group in the belt.

FIGURE 8.2 CLASSIFICATION OF VMS DEPOSITS BASED ON COPPER AND ZINC RATIOS



Source: *DeMatties (1994)*

9.0 EXPLORATION

Portions of Section 9.0 have been modified from Connolly et al. (2012).

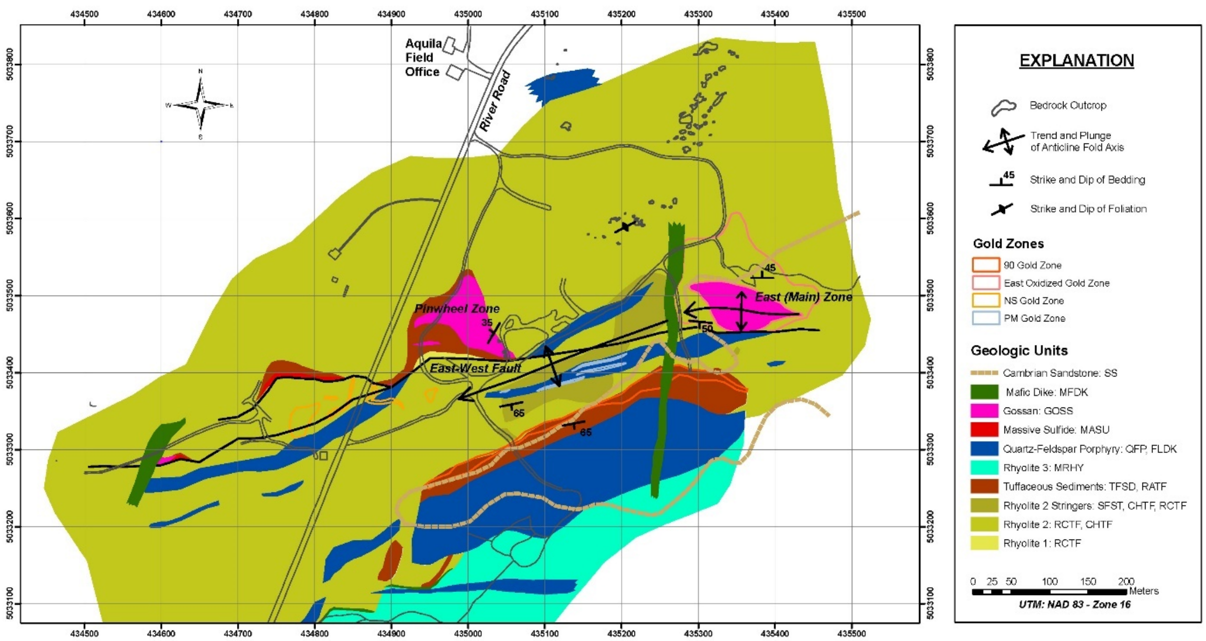
9.1 INTRODUCTION

Geophysical surveys including airborne EM, ground EM, gravity, and magnetic surveys have been the primary means of exploration over the life of the Project. To a lesser extent, geochemistry and geologic mapping have also been utilized to aid in exploration efforts.

9.2 SURFICIAL GEOLOGIC MAPPING

Sparse outcrop mapping in the immediate Deposit area has yielded structural and geochemical data supporting the general deposit model, although outcrop distribution does not allow for any delineation of mineralization. All rhyolite outcrops mapped in the area are highly pyrite-sericite-quartz altered. 90% of the whole-rock analyses of the Back Forty samples (mostly drill core) are considered intensely altered. Altered and mineralized outcrops extend north and west of the Back Forty Deposit for up to 500 m. Current sampling, whether from drilling or outcrop, has not identified the limit of strong pyrite-sericite-quartz alteration associated with the VMS mineralization. Indications are that mineralization may continue beyond the currently modelled resource. Figure 9.1 shows the bedrock geology of the immediate Project area overlain with Outcrop distribution.

FIGURE 9.1 BEDROCK GEOLOGICAL PROPERTY MAP WITH OUTCROP DISTRIBUTION



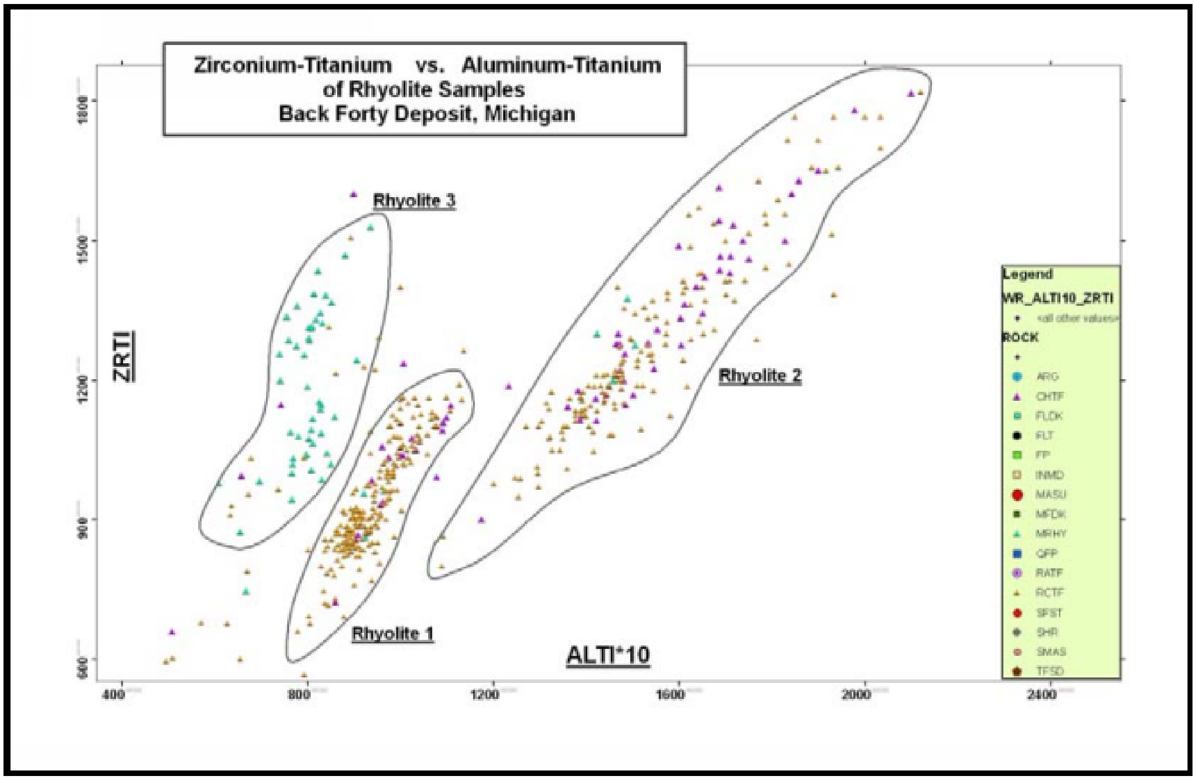
Source: *Tetra Tech Preliminary Economic Assessment (2014)*

9.3 GEOCHEMISTRY

A total of 680 geochemical whole rock analysis of drill core have been collected from host rocks at the Back Forty Deposit as well as from drilling peripheral to the Deposit area from 2002 to 2012 and have been compiled into a geochemical database. Additional whole rock samples have been collected from the 2015 to 2017 drill programs and are currently being added to the geochemical database. Major element geochemistry, specifically immobile element ratios (Al_2O_3/TiO_2 and ZrO_2/TiO_2), have been successfully implemented in the differentiation of the major host rhyolites (Figure 9.2) at the Project (which are often times visually indistinguishable from each other). Massive sulphide mineralization at the Back Forty Project typically occupies the boundaries between the major host rhyolites and further analysis of the growing geochemical database will likely yield increased understanding of the stratigraphy and geometry of the deposit and associated host rocks and will aid in exploration both near and peripheral to the known mineralization.

No traditional soil geochemical surveys have been undertaken in the Project area. However, monitor well samples, stream and other surface water samples have been taken as a part of baseline environmental studies required for mine permitting activities. These samples were analysed for a wide variety of trace elements related to establishing baseline environmental conditions, but have not been systematically compiled or analysed from an exploration perspective.

FIGURE 9.2 IMMOBILE ELEMENT PLOT - GEOCHEMICAL VARIATIONS OF RHYOLITES - BACK FORTY DEPOSIT



Source: *Tetra Tech Preliminary Economic Assessment (2014)*

9.4 GEOPHYSICS

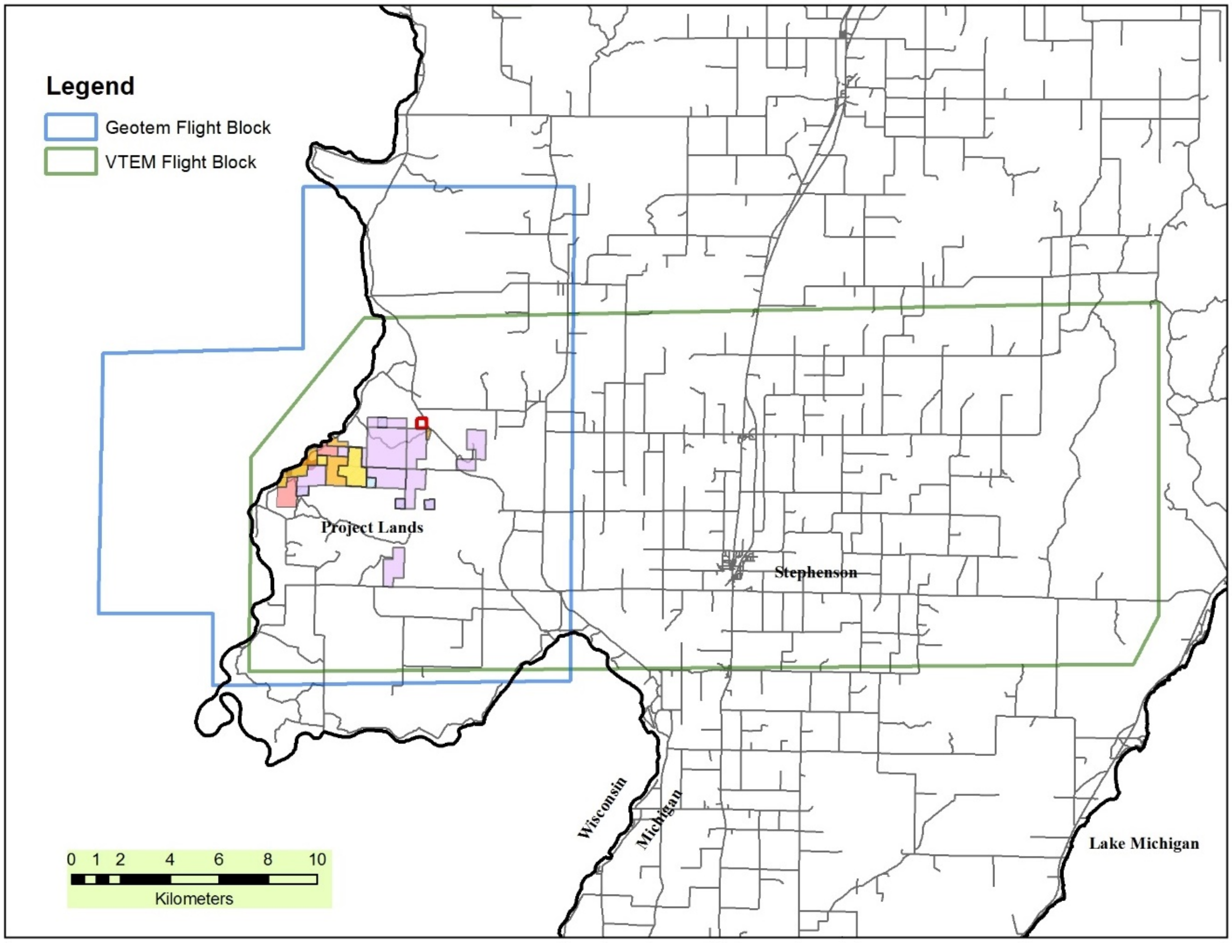
Extensive geophysical surveys have been completed over the immediate Project area and surrounding areas from 2002 to present. Geophysical surveys include two airborne magnetic/EM surveys and extensive ground surveys including HLEM (Max-Min), Pulse EM (PEM), magnetics and gravity as well as extensive downhole Pulse EM surveys completed during various drilling campaigns.

Results of these surveys and interpretations are available in reports and memos from Geotech Ltd., Dan Card, Crone Geophysics, Abitibi Geophysics and internal company reports.

9.5 AIRBORNE GEOPHYSICAL SURVEYS

Two airborne electromagnetic and magnetic surveys have been flown over the Project area (Figure 9.3). In 2002, a GEOTEM, fixed wing electromagnetic and magnetic survey with north south 200 m spaced lines was flown over the area of the Back Forty discovery, and in 2007 a larger (500 km^2), partially overlapping VTEM and magnetic survey was flown by Geotech Ltd. The VTEM survey line spacing was 100 m in the western portion of the block and 200 m in the eastern portion.

FIGURE 9.3 LOCATION OF AIRBORNE GEOPHYSICAL FLIGHT BLOCKS - BACK FORTY PROJECT AREA



Source: *Aquila Resources Inc. (2018)*

Multiple parties have reviewed and interpreted the airborne data from these surveys. Comprehensive reviews were conducted by HudBay geophysicists, and recently by independent geophysicist Dan Card. A number of strong, moderate, and weak conductive anomalies have been identified from these reviews, both in the vicinity of the Back Forty Deposit, to the east of the deposit under thickening Palaeozoic cover, and to the west of the Deposit in Wisconsin. Where roads cross these anomalies, and where access was possible, ground gravity surveys have been conducted over a number of these anomalies, and in some cases, have identified coincident gravity and conductive responses.

Since many of these anomalies are located under Paleozoic cover, they may have been subjected to extensive weathering and oxidation (gossan formation) which may have reduced, and in some cases even eliminated, much of the conductivity and resultant electromagnetic response. Therefore, even weak conductors may be of potential interest and should be targeted in future exploration campaigns.

9.6 GROUND AND DOWNHOLE GEOPHYSICAL SURVEYS

Previous ground geophysical surveys completed over the prospect area were conducted by initial operator MPC and include horizontal loop electro-magnetic (max-min), total field magnetics, and gravity. Ground and down-hole pulse electromagnetic surveys ("PEM") were conducted during the 2002 to 2003 drilling program. The ground and down-hole geophysical surveys were conducted by Crone Geophysics with interpretation provided by ACNC geophysicists. Four loops were laid out to locate extensions of the sulphide deposit.

Additional PEM surveys that were conducted in the immediate Back Forty Mineral Resource area were run during middle to late 2006 and 2007 with interpretation provided by Clark Jorgenson in 2007 and 2008. All electromagnetic responses were modelled with the "Maxwell" program developed by Electromagnetic Imaging Technology of Perth, Australia. A number of geophysical targets were tested successfully; other targets could not be explained through drilling.

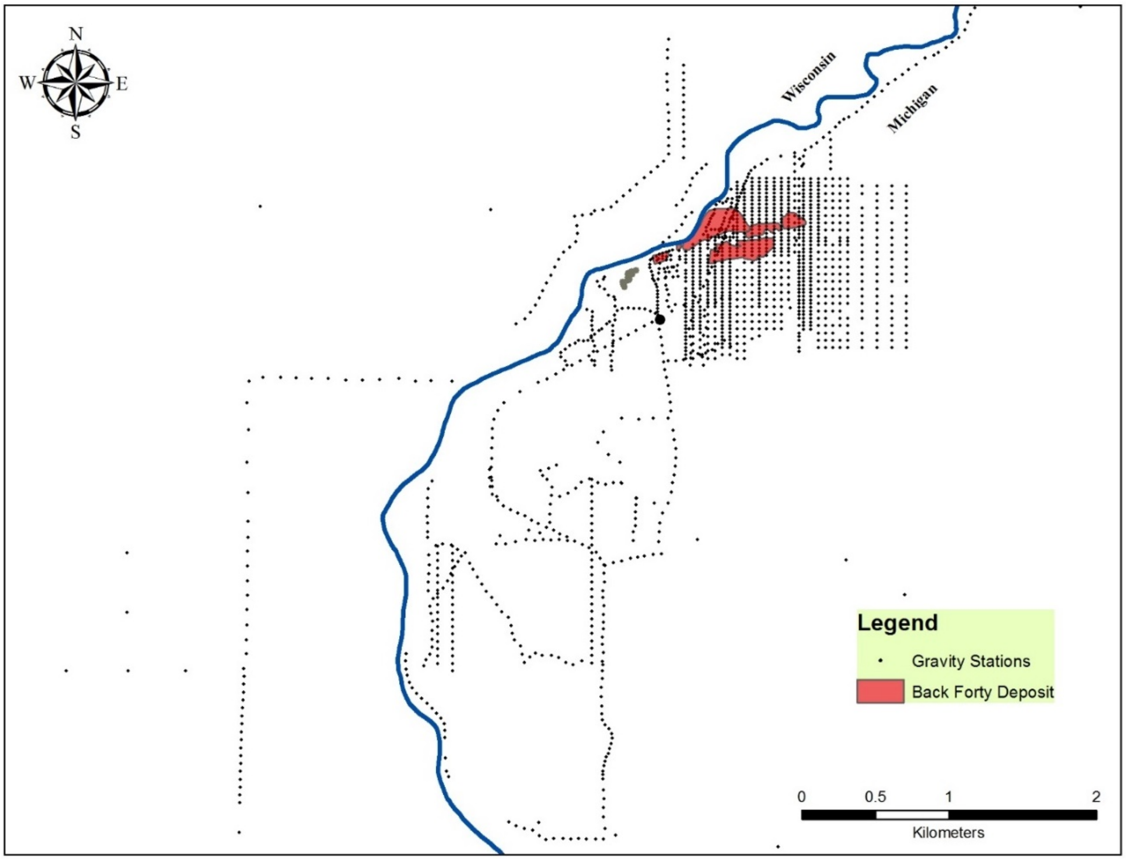
Additional downhole Pulse EM surveys were completed during the 2009-2011 drill programs. The surveys were completed by Crone Geophysics and reviewed and interpreted by Hudbay geophysicists who aided in the initial delineation of the Back Forty Deposit at depths exceeding 650 m in the vertical direction.

Downhole surveys were also carried out following the 2016 drilling campaign and were completed by Abitibi Geophysics. Geophysicist Dan Card has been overseeing the design and interpretation of these recent surveys, and has also recently reinterpreted the VTEM responses in the Deposit are in conjunction with past and recently completed downhole PEM and Surface PEM.

Since most of the immediate Deposit area and prospective geologic trends adjacent to the Deposit are covered with glacial drift and Paleozoic sediments, and because cultural features (power lines, fences etc.) are common and interfere with electromagnetic techniques, extensive gravity surveys have been conducted over the deposit and surrounding area from the Project's inception through 2016.

Figure 9.4 shows the location of gravity stations near the Back Forty Deposit. Regional gravity stations and some profiles are not shown.

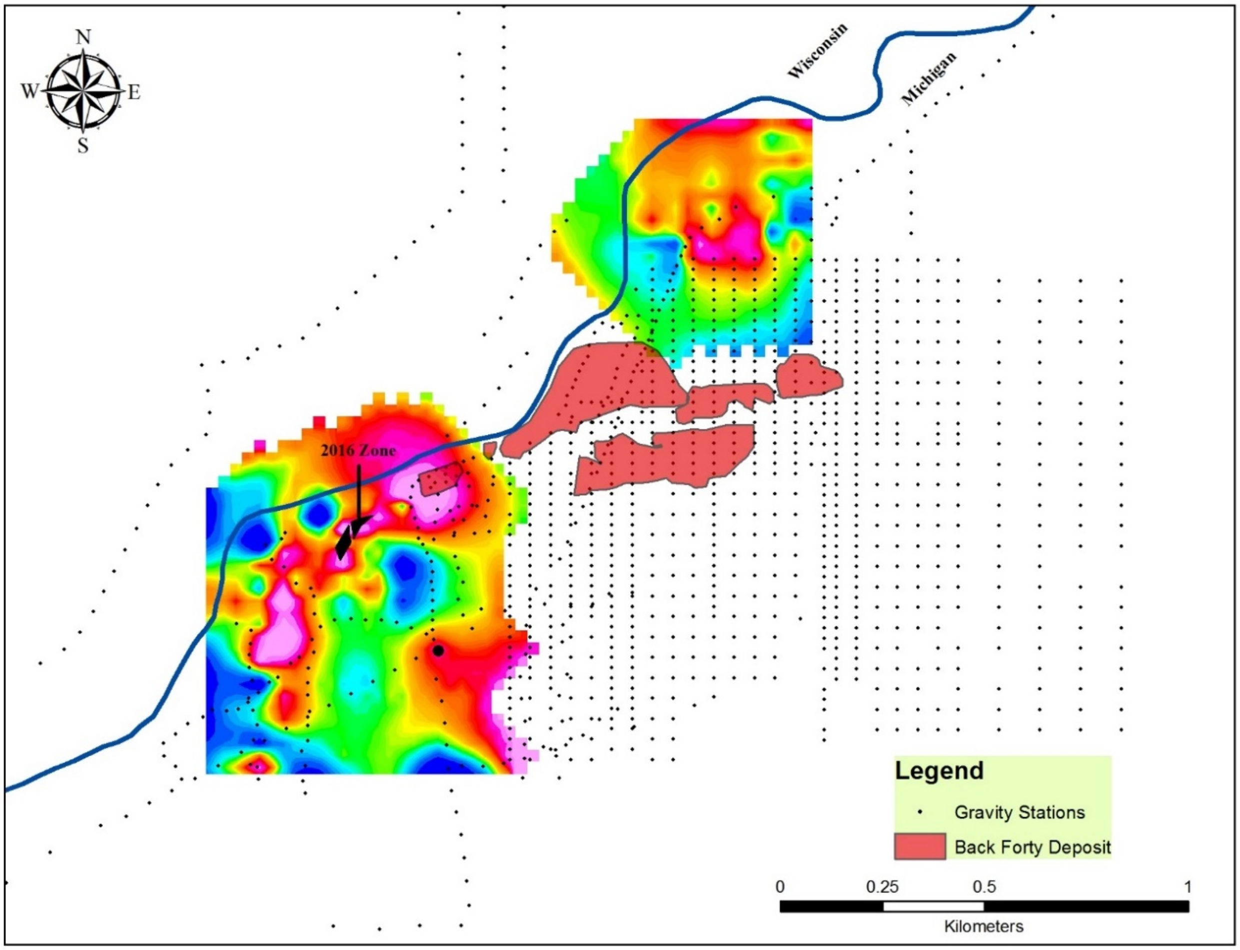
FIGURE 9.4 LOCATION OF GRAVITY STATIONS - BACK FORTY PROJECT AREA



Source: *Aquila Resources Inc. (2018)*

In 2016, consolidation of land ownership peripheral to the Deposit allowed expansion of the detailed gravity grid to the northeast and southwest of the deposit. Figure 9.5 shows a coloured image of this expanded gravity data. Subsequent drill testing of the gravity anomaly extending southwest of the known Deposit resulted in the discovery of a new zone of massive sulphide mineralization – the 2016 Zone, which was the target of drill testing in 2017.

FIGURE 9.5 EXPANDED DETAILED GRAVITY SURVEY SHOWING NEWLY DISCOVERED 2016 ZONE



Source: *Aquila Resources Inc. (2018)*

10.0 DRILLING

Portions of Section 10.0 are modified from Connolly et al. (2012).

10.1 INTRODUCTION

Drilling on the Property was conducted over several campaigns. Between 2002 and 2019, 642 boreholes totalling 124,580 m were drilled. In addition to resource delineation drilling associated with the expansion of the Back Forty Mineral Resource, focused drill efforts were also undertaken which included: The drilling of exploration (geophysical) targets in the immediate vicinity of the Deposit area, drilling to support metallurgical testing programs, and geotechnical drilling to characterize the rock quality of the Deposit area. Geotechnical soil borings are not described in this section. Additional drilling and sampling are described in more detail in the following sections of this Technical Report.

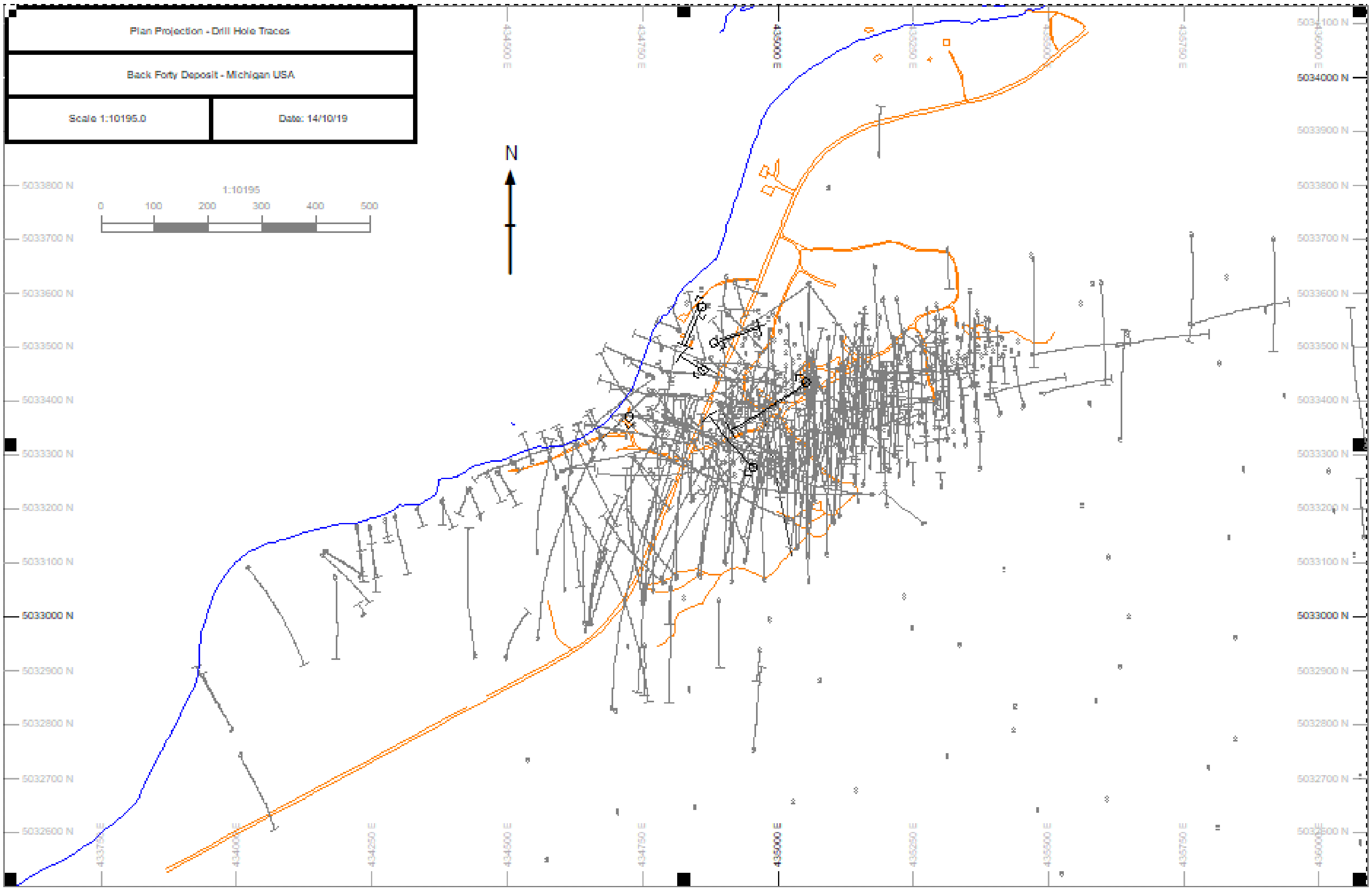
Figure 10.1 shows the traces of drill holes projected to surface. A year-by-year summary of drilling is summarized in Table 10.1.

TABLE 10.1
YEARLY SUMMARY OF DRILLING

Year	Drilling Company	Core Size	Number of Holes	Metres Drilled	Footage Drilled
2002	Klieman	NQ	5	448	1,469
2002	Klieman	BTW	2	325	1,067
2002	Major Midwest	NQ	41	11,201	36,741
2002	Salisbury and Associates	BDBGM	1	59	195
2003	Major Midwest	NQ	22	8,518	27,938
2006	IDEA Drilling	BTW	82	13,214	43,342
2007	Boart Longyear	BQ	22	5,063	16,608
2007	IDEA Drilling	BTW	64	12,434	40,782
2007	IDEA Drilling	BQ2	33	10,302	33,791
2008	Boart Longyear	NQ	20	3,126	10,253
2008	IDEA Drilling	BTW	49	8,871	29,098
2008	IDEA Drilling	NQ2	15	6,725	22,058
2009	IDEA Drilling	NQ2	18	1,260	4,132
2009	IDEA Drilling	NQ3	23	2,086	6,841
2010	Boart Longyear	NQ	5	633	2,075
2010	Boart Longyear	NQ3	11	1,830	6,002
2010	IDEA Drilling	BTW	0	48	158
2010	IDEA Drilling	NQ	1	76	250
2010	IDEA Drilling	NQ2	12	1,090	3,577
2010	IDEA Drilling	NQ3	70	6,836	22,421
2011	IDEA Drilling	NQ3	15	7,777	25,510
2011	IDEA Drilling	NQ2	35	6,202	20,344
2011	Boart Longyear	NQ3	5	2,754	9,032
2011	Coleman	NQ2	23	1,127	3,697
2015	IDEA Drilling	NQ	13	1,775	5,820
2016	Downing Drilling	NQ3	13	2,333	7,651
2017	IDEA Drilling	NQ3	13	2,399	7,868
2017	Downing Drilling	NQ3	11	3,603	11,816
2018	Downing Drilling	NQ3	3	633	2,078
2019	IDEA Drilling	HQ	12	1,832	6,011
Total			**642**	**124,580**	**408,625**

Note: *Extensions are not treated as separate drill holes.*

FIGURE 10.1 DRILL HOLE PLAN MAP SHOWING DRILL HOLE TRACES PROJECTED TO SURFACE



Source: *Aquila Resources Inc. (2018)*

10.1.1 2002-2003 Drilling Program

The first program, conducted by ACNC, started in February 2002 and continued to late May 2003. Drilling was contracted to Midwest Drilling Co., a Canadian company, using up to five Longyear 44 skid-mounted drill rigs. In addition, a small number of holes were drilled by Kleiman Pump and Well of Iron Mountain, Michigan. The program consisted of 71 drill holes (20,600 m), from which approximately 7,600 assay samples and 340 whole-rock samples were collected.

10.1.2 2006 Drilling Program

The second drill program occurred in Q4 2006. Drilling was completed by Idea Drilling Company ("IDEA Drilling") of Virginia, Minnesota, using two CS-1000 skid/trailer-mounted rigs. This program delivered 13,190 m of core in 80 BTW sized drill holes. The majority of the drilling targeted the East and Pinwheel Zones.

10.1.3 2007 Drilling Program

The third drilling program was completed in 2007. One hundred and eighteen drill holes (27,800 m) were completed by Boart Longyear of Wyeth, Virginia and IDEA Drilling. Boart Longyear drilled 22 NQ sized drill holes (5,060 m). IDEA Drilling utilized two drilling rigs, one of which drilled 33 NQ2 sized drill holes (10,300 m) while the other rig drilled 64 BTW sized drill holes (12,400 m). This drilling program tested a number of targets throughout the Mineral Resource area.

10.1.4 2008 Drilling Program

A fourth drill program was completed in 2008 using three drill rigs; one from Boart Longyear and two from IDEA Drilling. The former company completed fifteen NQ sized drill holes (2,600 m). IDEA Drilling completed 13 NQ2 sized drill holes (4,850 m) and 38 BTW sized drill holes (6,500 m). Similar to the previous year, drill targets were distributed throughout the Mineral Resource area.

10.1.5 2009-2010 Drilling Program

From October 2009 to May 2010, another phase of drilling was mounted. Two rigs were used: one from IDEA Drilling (trailer mounted Hagby) and one from Boart Longyear (skid-mounted 44). For this program, IDEA Drilling drilled the first 20 drill holes on the Project using NQ2 and the holes were oriented (totalling 1,327 m). IDEA Drilling subsequently completed 93 NQ3 split-tube oriented drill holes and one extension using BTW for a total of 8,681 m. IDEA Drilling also drilled 11 holes outside the immediate Deposit area that were not used for the updated Mineral Resource calculation (1,388 m). Boart Longyear completed 11 NQ3 split-tube oriented drill holes that were included in the updated Mineral Resource totalling 1,492 m. In addition, Boart Longyear completed five NQ3 "geotechnical" holes that targeted the conceptual open pit walls (971 m). The core from these drill holes was archived in its entirety, i.e., not cut and assayed, so they are not included in the updated Mineral Resource calculation.

Drilling from 2009 to 2010 outside the immediate Back Forty Deposit approximately 600 m to the east was targeted on ground magnetic and gravity anomalies. Anomalous zinc and gold mineralization in altered rhyolites and sediments was encountered in two drill holes. Drill hole PTL-1 intersected 10.0 m of 0.61% zinc, including one 1.5 m sample of 1.08% zinc. Drill hole PTL-2 encountered an interbedded sequence of flows and tuffaceous sediments including a chlorite-altered fragmental zone containing 26.5 m of 0.54% zinc, with smaller zones exceeding 1% zinc, a lower interval of tuffaceous sediments containing 12.5 m of 0.51% zinc, and an underlying siliceous breccia with 6 m of 1.1 g/t gold, including 1.5 m of 2.67 g/t gold. This suggests that prospective host rocks continue to the east of the Back Forty Deposit for at least 600 m. These two drill holes are not part of the Back Forty Mineral Resource Estimates.

10.1.6 2011 Drilling Program

78 holes were drilled in a series of drilling programs for the 2011 drilling program. The programs included drilling 22 high-grade gold targets at depth, four geophysical targets, and 22 relatively shallow drill holes to delineate the Pinwheel Gossan Zone.

A total of 11 holes were drilled to collect metallurgical samples, 12 for condemnation purposes east of the Mineral Resource and five drill holes to install monitoring wells for groundwater purposes. These additional 27 drill holes are not part of the updated Mineral Resource Estimate.

Ground conditions are generally good resulting in excellent core recovery: between 90% and 100%. NQ and BTW sized core from each hole were logged by Aquila personnel. Information collected includes lithology, structure, alteration, and mineralization. Rock quality designations (“RQD”) were also calculated over drilled intervals and specific gravity was measured on select samples. Oriented core from the 2009 to 2011 program were geotechnically logged in addition to geologically logged.

10.1.7 2015 Drilling Program

Drilling in 2015 consisted of a total of 13 NQ sized drill holes totalling 1,775 m. Drilling was completed with a track-mounted LF-90 drill rig by IDEA Drilling based out of Virginia Minnesota. Drilling operations were managed by Aquila staff with geological support including core logging, drill supervision, and Core-cutting/sampling by Great Lakes Exploration Inc. based out of Menominee, MI.

10.1.8 2015 Metallurgical Drilling Program

The primary focus of the 2015 program consisted of 833 m of drilling in nine metallurgical drill holes (MET-15-01 to MET-15-09) targeting sulphide mineralization within the open-pit portion of the Mineral Resource. Drill hole planning for the metallurgical drill program was completed by SGS Canada Inc. (Quebec, Quebec) with input from Aquila staff. Drill holes were designed to intercept all sulphide metallurgical domains and to provide sufficient sample material and spatial and mineralogical (grade) variability within each domain to support the necessary metallurgical testing requirements. Select intervals from the drill holes were sampled and submitted directly to SGS for assay and metallurgical testing. The remaining un-sampled material was reserved for potential future metallurgical use. All holes represented 'twin drill holes' from previous drilling campaigns and the assays from these drill holes were not incorporated into the updated Mineral Resource Estimate.

10.1.9 2015 Mineral Resource Drilling Program

Two drill holes (LK-15-508 and LK-15-509) from the 2015 drill program targeted Mineral Resource expansion of the Pinwheel Zone on property that had previously been unavailable for drilling. The two drill holes intercepted zinc-rich massive sulphide and associated gold mineralization within the host rocks. Significant assay results are tabulated in Table 10.2.

TABLE 10.2
2015 MINERAL RESOURCE DRILLING - SIGNIFICANT RESULTS

DDH	From (m)	To (m)	Interval* (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
LK-15-508	107.2	121.3	14.1	9.18	61.6	0.12	0.2	1.51
including	107.2	110.1	2.8	38.55	231.8	0.06	0.52	0.23
	110.1	118.3	8.2	0.71	14.5	0.16	0.13	2.48
including	110.1	112.4	2.3	1	31	0.08	0.37	4.96
	118.3	121.3	3	4.53	29.2	0.06	0.06	0.07
LK-15-509	136.5	138.1	1.6	0.62	25.9	0.08	0.11	4.25
	144	154	10	0.71	35.7	0.28	0.31	2.24
including	146.8	151	4.2	0.93	11.6	0.56	0.03	4.3

***** *Both holes drilled vertically. Interval is drilled thickness. True thickness is estimated to be approximately 80% of drilled thickness.*

10.1.10 2015 Exploration Drilling Program

An additional two drill holes (PHC-01 and PHC-02) from the 2015 drill program targeted a geophysical anomaly peripheral to the Deposit area. No significant grades were reported in the two drill holes, however, both holes encountered a 25-50 m thick section of finely bedded, exhalative sediments with disseminated pyrrhotite, pyrite and minor chalcopyrite and sphalerite hosted by altered fragmental felsic pyroclastic rocks similar to the Back Forty host rock sequence.

10.2 2016 DRILLING PROGRAM

A total of 2,333 m was drilled in 13 holes in 2016. Drilling was completed by George Downing Estate Drilling Ltd based out of Grenville-sur-la-Rouge, Quebec with a skid-mounted LF-90 drill rig. All drill core was NQ3 diameter and was oriented and utilized spit tubes to ensure maximum core recovery and to minimize the occurrence of mechanical fracturing in the core. Drilling consisted of geotechnical drilling to support open pit mine design, follow-up step-out drilling on the Pinwheel Zone, and exploration drilling focusing on evaluating recently identified geophysical anomalies to the southwest and along trend of the Updated Mineral Resource Estimate.

As a note, technical difficulties were encountered with respect to obtaining downhole surveys for a number of the drill holes completed during 2016. Drilling difficulties encountered during the drilling of LK-16-511 caused the casing to dislodge from bedrock and it was necessary for the hole to be abandoned and plugged without collecting downhole survey data. The current survey data for this hole consists of a single dip measurement near the top of the hole which was collected using the orientation tool and a compass reading (azimuth) of the position of the drill rig collected prior to the drill hole being abandoned. In addition, the survey tool malfunctioned for a number of the 2016 Mineral Resource and exploration drill holes. The drill holes were left open upon completion and survey data was collected at a later date during downhole geophysical surveys.

10.2.1 2016 Geotechnical Drilling Program

Geotechnical drilling consisted of 671 m of drilling in three drill holes (GT-06 to GT-08) evaluating rock quality in the south-western and south-eastern portion of the open pit Mineral Resource area as well as to test the rock mass quality along the proposed cut-off wall between the planned open pit and the Menominee River. The geotechnical drill program was planned by Golder. (Geotechnical logging and sampling were completed by Knight Piesold and geological logging was completed by Aquila Geologists.

GT-08 intercepted mineralization outside of the planned open pit extents. The drill hole was sampled and assayed as part of the 2017 drill program.

10.2.2 2016 Resource Drilling Program

An additional four drill holes (627 m) were completed to delineate and extend the known Mineral Resource outside of the planned open pit. Drilling was planned and managed by Aquila personnel with geologic support, including core logging, geotechnical logging, drill collar surveying, and drill supervision, by Great Lakes Exploration Inc.

Three drill holes (LK-16-511 to LK-16-513) targeted the north extension of the massive sulphide of the Pinwheel Zone. Drilling did not encounter massive sulphide but intercepted fine-grained rhyolite ash tuff with modest gold content interpreted to represent the equivalent to the Pinwheel sulphide horizon. LK-16-514 was drilled as a step-out hole on the southwestern extent of the Pinwheel massive sulphide. The drill hole encountered two intervals of high-grade gold mineralization within a discrete chlorite altered rhyolite as well as in fine grained rhyolite ash tuff interpreted to be the equivalent horizon of the Pinwheel massive sulphide. Due to the lack of drilling in the vicinity of the hole, true thickness of the gold mineralization is unknown. Significant assays results are tabulated in Table 10.3.

TABLE 10.3 **2016 MINERAL RESOURCE DRILLING - SIGNIFICANT RESULTS**								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
LK-16-511	107.1	110.3	3.2	1.78	20.48	0.06	0.04	0.13
LK-16-512	56	57.2	1.2	2.29	2.93	0.01	0.01	0.004
	141.57	142.67	1.1	1.76	13.02	0.02	0.03	0.06
LK-16-514	56	62	6	5.01	27.34	0.02	0	0.01
including	60.7	62	1.3	14.62	87.7	0.09	0.01	0.01
	125.2	127.1	1.9	10.01	155.76	0.17	1.07	0.11

***** *Both holes drilled vertically. Interval is drilled thickness. True thickness is estimated to be approximately 80% of drilled thickness.*

10.2.3 2016 Exploration Drilling Program

An additional six drill holes totalling 1,195 m were drilled testing both airborne and recently identified ground geophysical anomalies proximal to the Back Forty Deposit. The drill program was planned and managed by Aquila personnel with technical support including drill-rig supervision, geologic logging and sampling, drill collar surveying and core processing from consulting geologists and technicians provided by Great Lakes Exploration Inc.

LK-16-515 targeted a discrete gravity anomaly southwest of the Pinwheel Zone and encountered a new zinc-rich massive sulphide zone as well as deeper tuffaceous sediments containing moderate Ag, Zn, and Pb mineralization. LK-16-516 and LK-16-517 were drilled as an overcut and undercut of the discovery intercept, respectively. LK-16-518 targeted an airborne anomaly and intercepted a lower Tuffaceous Sediment Package at depth. Given the limited drilling in this area the mineralization has not been modelled and is not currently incorporated into the Updated Mineral Resource Estimate. Significant drill results are tabulated in Table 10.4.

TABLE 10.4 **2016 EXPLORATION DRILLING - SIGNIFICANT RESULTS**								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
LK-16-515*	25	27.5	2.5	2.94	6.87	0.03	0.03	0.01
	63	74.21	11.21	1.88	19.52	0.18	0.15	3.97
including	63	65.78	2.78	4.63	39.28	0.17	0.37	1.14
including	65.78	74.21	8.43	0.97	13	0.18	0.08	4.9
including	72.34	74.21	1.87	0.45	8.32	0.11	0.06	9.73
	156.5	161	4.5	0.29	37.28	0.01	0.61	2.07
LK-16-516*	76.18	89.2	13.02	0.81	23.64	0.08	0.42	4.92
including	77.5	86.85	9.35	0.84	10.7	0.09	0.1	6.7

TABLE 10.4 **2016 EXPLORATION DRILLING - SIGNIFICANT RESULTS**								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
including	80	86.85	6.85	0.8	11.92	0.08	0.11	7.72
including	83.15	86.85	3.7	0.6	10.71	0.06	0.11	9.79
LK-16-517	90.5	97.28	6.78	0.31	5.02	0.04	0.06	2.99
	127.9	128.4	0.5	0.59	304	0.06	4.71	0.01
LK-16-518	87.5	88.1	0.6	0.32	44.21	0.01	0.3	2.1
	172	187.68	15.68	0.1	18.61	0.01	0.26	0.76
including	173.5	174.2	0.7	0.21	52.66	0.03	0.57	3.72
including	183.1	184	0.9	0.11	72.58	<0.01	1.26	3.17
including	187	187.68	0.68	0.08	91.74	0.05	1.45	3.27

***** *Interval is drilled thickness. True thickness is estimated to be approximately 80% of drilled thickness for LK-16-515, LK-16-516 and LK-16-516. True thickness for LK-16-518 is unknown.*

10.3 2017 DRILLING PROGRAM

A total of 24 drill holes totalling 6,001 m were drilled between January and June of 2017. The drilling consisted of three independent programs including a geotechnical drilling program which characterized rock mass qualities for ‘out of pit’ Mineral Resource, a Mineral Resource delineation drilling program which included both infill drilling to convert Inferred Mineral Resources to Indicated Mineral Resources, and step out drilling on known mineralization, as well as an exploration program evaluating geophysical anomalies. Both a skid mounted LF-90 (Downing Drilling) and a trailer mounted Hagby (IDEA Drilling) were utilized to complete all drilling. All drill core was NQ3 and oriented utilizing split tubes.

10.3.1 2017 Geotechnical Drilling Program

The geotechnical drilling program consisted of a total of five drill holes (GT-09 – GT-13) and 1,281.2 m total of drilling designed to evaluate the rock mass quality within the potential underground mining area including three drill holes (GT-09 – GT-11) in the Pinwheel area southwest of the planned open pit and two holes (GT-12 and GT-13) in the Main Zone and Deep Zone area below and southwest of the planned open pit. The geotechnical drill program was planned, and geotechnical logging and sampling was carried out by Knight Piesold. Geological logging and sampling as well as collar surveying and core processing were completed by Aquila personnel and contract geologists/technicians supplied by Great Lakes Exploration Inc. Drilling was carried out by Downing Drilling.

In addition to collecting geotechnical data, a number of the geotechnical drill holes were also designed to intercept areas of Inferred mineralization within the Mineral Resource model in the vicinity of the Pinwheel Zone, Tuff Zone as well as the Deep Zone. Significant assay results are provided in Table 10.5.

TABLE 10.5 2017 GEOTECHNICAL DRILLING - SIGNIFICANT RESULTS								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
GT-08	19.64	24.5	4.86	1.88	214	1.12	0.2	0.03
	63	78.7	15.7	0.88	21	0.54	0.02	0.36
GT-09	55.5	57	1.5	9.379	126	0.16	0.65	0.04
	59.51	64.5	4.99	1.319	13	0.66	0.17	9.99
	76.5	89.3	12.8	2.241	9	0.3	0.04	0.16
GT-10	50.2	59.24	9.04	0.74	8	0.2	0.33	9.31
including	76.68	78.75	2.07	2.905	38	0.14	0.3	0.03
	114.23	128	13.77	1.381	14	0.04	0.08	3.82
including	114.23	117.7	3.47	1.651	13	0.07	0.05	6.93
	159.6	190	30.4	1.497	29	1.23	0.01	0.32
including	159.6	180.5	20.9	1.605	32	1.58	0.01	0.37
GT-11	42.31	50	7.69	3.627	35	0.12	0.12	10.07
	57.8	69.2	11.4	0.962	29	0.33	0.01	20.01
	80.5	82.57	2.07	6.643	15	0.1	0.22	0.08
GT-12	337	379.07	42.07	1.176	15	0.17	0.05	5.19
including	342	365.5	23.5	0.725	11	0.08	0.02	7.5
	378	399.4	21.4	11.655	50	0.44	0.41	1.29
including	379.07	385.34	6.27	28.333	60	0.51	0.64	3.24
GT-13	149.2	150.7	1.5	2.72	187	0.01	0.23	0.03
	163	167.52	4.52	0.83	255.01	0.01	0.14	0.04
including	164.3	166.15	1.85	1.64	531.65	0.02	0.26	0.05
	171	186.4	15.4	1.09	30.52	0.01	0.28	0.2
including	171.3	177	5.7	1.7	73.99	0.03	0.59	0.24
and	180.1	180.38	0.28	8.74	4.59	0	0.01	0.06
	187.4	189.9	2.5	1.43	257.36	0.14	9.06	23.5
	189.9	197	7.1	1.64	23.36	0.02	0.7	1.18
	343.4	359.07	15.67	1.72	2.2	0.11	0	0.14
including	356.5	359.07	2.57	4.22	1.68	0.08	0	0.09

***** *Interval is drilled thickness and does not represent true thickness. Estimated true thicknesses for individual holes are: GT-08 90%, GT-09 95%, GT-10 48% to 62%, GT-11 67%, GT-12 73%, GT-13 85%*

10.3.2 2017 Mineral Resource Drilling Program

Resource delineation drilling consisted of a total of 10 drill holes (LK-17-521 to LK-17-528, LK-17-531 and LK-17-533) as well as extensions of GT-12 and GT-13 for a total of 2,610 m total drilled by both IDEA Drilling and Downing Drilling. The program was planned jointly by P&E, Objectivity (Sudbury, Ontario), with input from Aquila personnel. Geological logging and sampling as well as collar surveying and core processing were completed by Aquila personnel and contract geologists/technicians from Great Lakes Exploration Inc. In addition to geological logging, geotechnical logging was completed by Great Lakes Exploration geologists on select drill holes (LK-17-521 to LK-17-527) due to a lack of geotechnical information within the Pinwheel portion of the potential underground mine area.

LK-17-521 through LK-17-527 were designed to intercept Inferred Mineral Resource material as well as to test the western, down-dip extension of the Pinwheel massive sulphide. All drill holes encountered massive sulphide mineralization associated with the Pinwheel massive sulphide.

LK-17-528 and LK-17-533 were designed to intercept inferred mineralization located in the Deep Zone massive sulphide and adjacent Porphyry Margin Gold Zone. Both drill holes also encountered mineralization associated with the Tuff Zone massive sulphide and stringers as well as the 90 Zone along the south margin of the proposed open pit.

Given the depth of the drill holes, drill hole deviation was monitored throughout drilling operations. LK-17-528 was terminated prior to reaching the planned final depth due to unexpected deviation.

Significant assay results for all drill holes are provided in Table 10.6.

TABLE 10.6
2017 RESOURCE DRILLING - SIGNIFICANT RESULTS

DDH	From (m)	To (m)	Interval* (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
LK-17-521	61.1	62.75	1.65	0.381	77	0.06	1.18	3.02
	68.52	74.49	5.97	0.719	53	0.18	0.37	2.21
	133	145.75	12.75	0.949	10	0.29	0.04	0.07
LK-17-522	134.65	150.05	15.4	0.943	10	0.26	0.05	0.8
including	134.65	137	2.35	0.975	35	0.29	0.17	4.1
LK-17-523	51	165	114	1.006	14	0.61	0.01	1.8
including	67.5	99	31.5	0.681	9	0.28	0.01	4.55
	120	165	45	1.333	15	0.99	0.01	0.39
including	133	137.5	4.5	2.126	22	2.07	0	0.24
LK-17-524	45.5	100.58	55.08	1.304	13	0.49	0.04	0.29
LK-17-525	142.88	143.55	0.67	0.838	32	0.04	0.46	22.42
	147.3	163.36	16.06	0.921	12	0.46	0.2	1.81
including	147.3	154.5	7.2	0.766	19	0.59	0.43	3.62
	166	170.63	4.63	1.83	8	0.21	0.06	0.13

TABLE 10.6 2017 RESOURCE DRILLING - SIGNIFICANT RESULTS								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
LK-17-526	155.7	172.5	16.8	0.76	8.73	0.11	0.16	4.27
including	156.36	160.25	3.89	0.85	17.41	0.18	0.41	14.17
including	170.37	172.5	2.13	0.59	14.86	0.16	0.26	6.83
LK-17-527	54.09	174.3	120.21	1.14	13.18	0.72	0.01	0.6
including	57	60	3	1.11	35	0.56	0.02	3.28
including	75	79.5	4.5	0.88	7	0.25	0.01	3.44
including	112.5	127	14.5	1.3	12.19	0.9	0.01	0.48
including	147	174.3	27.3	1.52	21.47	1.54	0.01	0.32
LK-17-528	163.94	166.33	2.39	1.23	112.96	0.05	1.12	13.61
	180.83	186.73	5.9	0.82	75.37	0.1	2.69	9.6
	192.5	198.5	6	0.98	65.75	0.06	2.53	8.95
including	193.74	196.19	2.45	1.09	130.41	0.09	5.31	19.79
	204.4	211.5	7.1	1.31	44.26	0.02	0.52	0.6
including	207	210	3	2.71	67	0.03	0.75	0.35
	291.37	295.16	3.79	2	15.27	0.57	0.05	2.02
including	292.87	294.14	1.27	4.59	28	1.14	0.04	1.37
LK-17-533	158	159.5	1.5	2.5	237	0.05	0.88	0.71
	227.65	268.17	40.52	0.38	3.56	0.02	0.1	2.31
including	227.65	228.46	0.81	4.29	73	0.03	2.6	14.14
including	239.3	244.33	5.03	0.76	3.5	0.04	0.08	10.01
including	256.5	261	4.5	0.59	1.33	0.04	0.02	2.88
	436.5	441	4.5	0.88	31.67	0.02	0.61	0.2
	442.6	455.84	13.24	1.18	16.55	0.59	0.09	0.84
including	442.6	444.51	1.91	1.25	19.1	0.02	0.48	4.89
including	444.51	455.84	11.33	1.17	16.12	0.68	0.02	0.16

***** *Interval is drilled thickness and does not represent true thickness. Estimated true thicknesses for individual holes are: LK-17-521 63%, LK-17-522 60%, LK-17-523 35%, LK-17-524 74%, LK-17-525 65%, LK-17-526 75%, LK-17-527 30%, LK-17-528 85%, LK-17-533 85%.*

10.3.3 2017 Exploration Drilling Program

A total of nine drill holes totalling 2,110 m total were drilled as part of an exploration program targeting a geophysical anomaly identified during 2016 and as follow-up on the newly discovered massive sulphide zone from the 2016 drill program. The drill program was planned and managed by Aquila personnel and geological logging and sampling as well as collar surveying was completed by contract geologists from Great Lakes Exploration Inc. Core processing, including core cutting and sampling was completed by Aquila personnel as well as contract technicians provided by Great Lakes Exploration Inc.

A downhole EM geophysical anomaly located to the south of the main Deposit area was tested by drill hole LK-17-520 and encountered thin lenses of massive sulphide coincident with the location of the geophysical target. Base and precious metal assays returned no significant results however, additional geophysical work will be completed to further evaluate the mineral potential in this area. An additional eight drill holes were completed to further define and extend the massive sulphide zone identified in 2016. Mineralization associated with this zone was extended approximately 35 m to the east-northeast and up-dip from the 2016 intercept. The zone was also extended approximately 70 m to the west-southwest and down-dip of the previous drilling completed in 2016. Given the limited drilling in this area mineralization has not been modelled and is not currently incorporated into the updated Mineral Resource Estimate. Significant results encountered during the 2017 exploration drilling program are tabulated below in Table 10.7.

Table 10.7
2017 Exploration Drilling - Significant Results

DDH	From (m)	To (m)	Interval* (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
K-17-529	55.11	57.57	2.46	0.5	12.94	0.09	0.05	6.49
LK-17-534	27.5	28.66	1.16	1.49	8	1.39	0.02	0.01
	112	114.5	2.5	0.15	38.2	0.02	0.67	0.02
including	113.5	114.5	1	0.24	70	0.02	1.31	0.01
	116.44	127.5	11.06	1.61	40.72	0.08	0.22	6.32
including	117.44	119.73	2.29	6.2	174.83	0.13	0.92	2.06
and	119.73	124.82	5.09	0.52	5.68	0.08	0.03	11.56
	233.5	235.5	2	0.55	28	0.75	0.01	0.12
	234.5	235.5	1	0.53	35	1.03	0.01	0.06
LK-17-535	64	78.85	14.85	0.84	13.23	0.29	0.04	0.2
including	71.5	78.85	7.35	1.05	16.94	0.46	0.02	0.13
	106.15	108	1.85	2.03	233.82	0.09	4.13	0.09
	108	116.38	8.38	1.49	29.55	0.08	0.2	7.61
including	108	113.38	5.38	1.85	25.58	0.13	0.17	11.46
LK-17-536	46.86	52.48	5.62	0.71	4.93	0.33	0.01	0.03
including	46.86	47.94	1.08	1.09	13	1	0.01	0.01
	77.5	79	1.5	2.73	8	0.01	0.06	0.07
	99.72	101	1.28	0.61	3	0	0.01	4.82
	127.5	132.03	4.53	0.57	5.47	0.01	0.05	3.19
including	129.2	132.03	2.83	0.79	7.02	0	0.07	4.27
LK-17-537	84.53	100.5	15.97	0.86	13.27	0.11	0.11	5.48
v	87.57	97.69	10.12	0.95	7.26	0.07	0.07	7.04
	113	131.09	18.09	1.26	11.87	0.03	0.1	6.37
including	120	122.34	2.34	3.72	22.64	0.05	0.25	4.48

TABLE 10.7 2017 EXPLORATION DRILLING - SIGNIFICANT RESULTS								
DDH	**From (m)**	**To (m)**	**Interval* (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
and	124	131.09	7.09	1.27	15.78	0.04	0.12	12.17

***** *Interval is drilled true thickness*

10.4 2018 EXPLORATION DRILLING PROGRAM

Three drill holes totalling 633.27 m were completed as part of an abbreviated exploration program in 2018. The drill program was planned and managed by Aquila. Geological logging was completed by Aquila personnel with support from a contract geologist from Great Lakes Exploration Inc. Collar surveying, core processing, including core cutting and sampling was completed by Aquila personnel. Drilling was carried out by Downing Drilling. The program was designed to test the extents of the recently discovered 2016 Zone and another geophysical target peripheral to the known Deposit. Drilling at the geophysical target intersected altered volcanics with anomalous zinc. The balance of the abbreviated program focussed on testing the extents of the 2016 Zone. One hole testing the southwest extension of the 2016 Zone intersected mineralization similar to the lower tuffaceous sediments. The drill holes were drilled after the current Mineral Resource Estimate was completed, and not are included in the Updated Mineral Resource Estimate.

10.5 2019 DRILLING

10.5.1 2019 Geomechanical Drilling Program

The geomechanical drilling program consisted of a total of seven drill holes totalling 1,274.03 m. Drilling was designed to evaluate the rock mass quality within the west pit wall (including KP-19-04, -05 and -06), and KP-19-01, -02, -02A and -03 were designed to evaluate the rock quality on a potential crown pillar. The geomechanical drill program was planned by KP, and geotechnical logging and sampling was carried out by KP. Geological logging and sampling as well as collar surveying and core processing were completed by Aquila personnel and a contract geologist supplied by Great Lakes Exploration Inc. Drilling was carried out by IDEA Drilling.

In addition to collecting geotechnical data, three of the geotechnical drill holes may have intersected areas of Inferred Mineral Resource mineralization within the Mineral Resource block model in the vicinity of the Pinwheel Zone, and NS Zone. None of the 2019 drill holes are included in the Updated Mineral Resource Estimate.

10.5.2 2019 Metallurgy Drilling Program

The primary focus of the 2019 program consisted of 558.33 m of drilling in eight metallurgical drill holes (MET-19-01 to MET-19-08) targeting early mining within the open-pit portion of the Mineral Resource. Drill hole planning for the metallurgical drill program was completed by Ben Chisolm of Aquila and Eric Quigley of Great Lakes Exploration, with input from Aquila staff.

Drill holes were designed to intersect the target intervals to provide sufficient sample material and spatial and mineralogical (grade) variability within each domain to support the necessary metallurgical testing requirements. Continuous samples (quarter core) from these drill holes were submitted to Minerals Processing for assaying. Composite samples (half core) were created from select intervals and sent to SGS for metallurgical testing. The remaining un-sampled material (quarter core) was reserved for potential future metallurgical testing. All holes represented are 'twin drill holes' from previous drilling campaigns. Assays from these drill holes were not incorporated into the Updated Mineral Resource Estimate.

10.6 GROUND CONDITIONS AND SURVEY DATA

Ground conditions are generally good resulting in excellent core recovery: between 90% and 100%. NQ and BTW sized core from each hole were logged by Aquila. Information collected includes lithology, structure, alteration, and mineralization. Rock quality designations ("RQD") were also calculated over drilled intervals and density was measured on select samples. Oriented core from the 2009 to 2011 program were geotechnically logged in addition to being geologically logged. Select oriented drill holes from 2016 and 2017 were also geotechnically logged.

All Project data are located using the local UTM survey grid using North American Datum 1983 (NAD-83), including the period prior to 2009, when coordinates were originally collected in NAD-27. Borehole collar location was determined with a high-resolution differential global positioning system ("GPS") unit (Locus system). Geologic, assay and directional survey data were compiled manually, entered into Microsoft Excel® spreadsheets, and then entered into Datamine Studio 3 for analysis and interpretation on plans, cross-sections, and 3-D wireframes.

Downhole borehole deviation was monitored during drilling, initially using a multi-shot Sperry Sun camera tool, and later using a Reflex™ FlexIt Smart tool. Both instruments determine azimuth deviation by magnetic methods. Downhole surveys were conducted approximately every 6 m for the length of the drill hole.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The following description has largely been taken from Martin, et al. (2014).

11.1 CORE SAMPLING

Boxed core samples from a secure area at the drill sites were transported daily to the nearby Aquila field office, located on the Property. Prior to 2009, core was geologically logged and marked for sampling and cutting and sampling took place at the field office facility. From 2009 to 2011, drill core was logged geotechnically and geologically, then marked for samples at the field office facility and transferred to a new core facility in Carney, Michigan (27 km away) for photographing, cutting, sampling and archiving. Virtually all the core of both mineralized and unmineralized material from all drill holes was sampled by sawing the core length-wise and retrieving half the split core for assaying. Typically, the drill core of mafic dykes were not sampled unless they exhibited visual mineralization.

Assay intervals are generally 1.5 m in length and honour geological boundaries. A total of 45,688 drill core samples have been collected and analyzed. Core samples were sent to one of five assay laboratories. In addition, 795 samples were sent for whole-rock analysis to one of two laboratories, and 711 samples were used for metallurgical testing. Table 11.1 shows a breakdown of the number of samples by drill hole, laboratory, and year.

Aquila used six primary laboratories for assaying core samples collected on the Project (refer to Table 11.1). ALS Chemex Labs (“ALS Chemex”), now known as ALS Minerals, Accurassay Lab (“Accurassay”) and Inspectorate America Corporation Analytical Laboratories (“Inspectorate America”) were used for primary drill core precious metal, base metal, and trace element analysis. Activation Laboratories Ltd. (“Actlabs”) of Ancaster, Ontario and SGS Mineral Services (“SGS”), formerly XRAL Laboratories, were used as umpire laboratories. Whole-rock major element analyses of drill core samples were primarily performed by SGS, with a small number by ALS Chemex.

ALS Chemex and SGS laboratories are accredited to International Organization for Standardization (“ISO”) 9001 by QMI-SAI Global and ISO 17025 by the Standards Council of Canada (“SCC”) for a number of specific test procedures, including fire assay for gold with atomic absorption (“AA”) and gravimetric finish, multi-element by inductively coupled plasma-atomic emission spectrometry (“ICP-AES”), and AA assays for silver, copper, lead and zinc. Accurassay is also accredited ISO 17025 by the SCC for a number of specific test procedures, including fire assay for gold with AA and test procedures including fire assay for gold with AA and gravimetric finish, and multi- element analysis using aqua regia and multi acid extraction and ICP-AES.

Assay samples were shipped from the Aquila field office in Michigan directly to ALS Chemex’s preparation facility in Elko, Nevada, Accurassay’s lab in Thunder Bay, Ontario, or Inspectorate America’s lab in Sparks, Nevada. ALS Chemex prepared samples were sent to Vancouver for analysis. Samples prepared by Accurassay were analyzed in Thunder Bay. Inspectorate America prepared and analyzed samples in the same facility in Sparks, Nevada. All pulps and coarse rejects were returned to the Project warehouse facilities in Michigan.

TABLE 11.1
SUMMARY OF ANALYTICAL SAMPLES BY YEAR

DDH	Samples	Laboratory Location	Year
108401-108471	7,545	ALS Chemex, Reno, NV and Vancouver, BC	2002-2003
LK-72 to LK-93	2,380	ALS Chemex, Thunder Bay, ON and Vancouver, BC	2006
LK-78 & LK-88 exts	235	Accurassay, Thunder Bay, ON	2006
LK-94 to LK-151	3,350	Accurassay, Thunder Bay, ON	2006
LK-152 to LK-186	5,320	Accurassay, Thunder Bay, ON	2007
LK-187PE to LK-197	1,575	SGS, Toronto, ON	2007
LK-198PE to LK-259	6,890	Accurassay, Thunder Bay, ON	2007
LK-260 to LK-323	6,818	Accurassay, Thunder Bay, ON	2008
LK- 324,325,326,329,332, 331,334,339,341,344,346	1,010	Actlabs, Thunder Bay, ON	2008
108 Series Infills	65	Actlabs, Thunder Bay, ON	2008
LK-327 to LK-354, later extensions and infills	3,615	Accurassay, Thunder Bay, ON	2008
LK-355 to LK-378	2190	Inspectorate America, Sparks, NV	2009
LK-379 to LK-478 and extensions (LK-85)	5552	Inspectorate America, Sparks, NV	2010
LK-479 to LK-506 and extensions	15,831	Inspectorate America, Sparks, NV	2010-2011
Total Assay	61,519	-	-
Whole Rock 2002-03	505	Neutron Activation Labs, Lakefield, ON	2002-2003
Whole Rock 2006	18	ALS Chemex	2006
Whole Rock 2006-08	222	SGS, Lakefield, ON	2006-2008
Whole Rock 2009-2010	12	SGS, Lakefield, ON	2009
Total Whole Rock	757	-	-
Metallurgical	605	G&T Metallurgical Services, Ltd. (G&T), Kamloops, BC	2007- 2008
Metallurgical	28	SGS, Lakefield	-
Metallurgical	78	Resource Development Inc. (RDI), Lakewood, CO	2011
Total Metallurgical	711	-	-
Sulphur and Carbon by Leco	21,183	MPC, Carney, MI	2011

11.2 SAMPLE PREPARATION AND ANALYSIS

11.2.1 2002 to 2003 Sampling Programs

The 2002 to 2003 drilling campaign generated approximately 8,000 drill core cut samples that were submitted to ALS Chemex for analyses. Samples were shipped to Elko, Nevada where they were prepared using standard sample preparation procedures (CRU-31). The crushed reject was mixed by splitting once on a Jones riffle and then by re-combining the two fractions (not a normal part of ALS Chemex's standard procedure). A minimum of 250 g of the crushed reject was split for pulverization (PUL-31) using a Jones riffle. ALS Chemex was instructed to clean the pulverizers with only pure silica sand after processing batches of highly-mineralized samples and before processing low-grade samples (near the contact between massive and stringer zones). Prepared pulps were then sent to Vancouver, British Columbia for analysis.

In Vancouver, all samples were analyzed for gold by fire assay with AA finish (code Au- AA25) on 30 g charges. The lower and upper detection limits for this package were 10 ppb to 100 ppm, respectively. Higher-grade samples were re-run with a gravimetric finish upon request. All samples were also analyzed using a multi-element package (code ME-ICP61). ME-ICP61 consisted of a four-acid digestion, "mineralized grade" inductively coupled plasma ("ICP") package with over-limit samples re-assayed for copper, lead, zinc and silver by four-acid digestion followed by AA (code AA62). A limited number of samples were analyzed for mercury by aqua regia digestion and flameless AA spectrometry (Hg-CV41 with detection limits of 0.01 to 100 ppm), and for total sulphur by Leco furnace and infrared spectroscopy (code S-IR08 with detection limits of 0.01 to 50%).

Density values for a limited number of core samples were determined using the pycnometer method on pulverized samples (code OAGRA08b). An additional 465 samples were sent to XRAL Laboratories (now SGS) for whole rock geochemical analysis by tetraborate fusion x-ray fluorescence spectrometry (code XRF103). All pulps and rejects were returned from ALS Chemex and stored indoors at MPC's Ropes Gold Mill facility in Humboldt, Michigan.

11.2.2 2006 Sampling Program

A total of 5,972 drill core cut samples were assayed in 2006. Samples from boreholes LK-72 to LK- 93 (2,270 samples) were sent to ALS Chemex. Samples from boreholes LK-94 to LK- 151, as well as samples from the extensions of boreholes LK-78, LK-80, LK-81, and LK- 88 (3,702 samples) were sent to Accurassay. Samples sent to ALS Chemex were shipped to the Thunder Bay preparation laboratory where the samples were crushed, split, and pulverized according to the standard procedure (PREP-31). The PREP-31 procedure included the following steps:

1. The entire sample was crushed until more than 70% of the samples passed a 2 mm screen.
2. The sample was split to 250 g using a riffle splitter.
3. The split was then pulverized to greater than 85% passing 75 μm mesh.

Prepared samples were shipped to Vancouver for assaying. All samples were analyzed for gold by 30 g gram fire assay and AA finish (code AA25). Detection limits for this method are 0.01 to 100 ppm. All assay results greater than 3 ppm gold were automatically re-run using 30 g fire assay with gravimetric finish (code Au-GRA32). The detection limit for these re-runs was 0.05 to 1,000 ppm gold.

All samples were also assayed for a suite of trace elements using a four-acid digestion followed by ICP-AES (Code ME-ICP61a, ME-ICP61a, or ME-OG62, depending on the expected levels). Samples sent to Accurassay were crushed to 90% passing 8 mesh, then a 250 g split was taken and pulverized to 90% passing 150 mesh (code ALP1). All samples were analyzed for gold by 30 g gram fire assay with AA finish (code ALFA1). The lower detection limit of this assay type was 5 ppb. All results greater than 5 ppm gold were re-analyzed by fire assay with gravimetric finish with a lower detection limit of 5 ppm (code ALFA3).

Low and intermediate grade samples were analyzed for a 32-element suite using a three-acid digestion followed by inductively coupled plasma-mass spectrometry (ICP-MS, code ICPMA). High-grade samples were analyzed using "mineralized grade" three-acid digestion with an AA finish. Coarse rejects and pulps were returned and stored in Aquila's indoor drill core storage facility in Daggett, Michigan.

11.2.3 2007 to 2008 Sampling Programs

In 2007 and 2008, Aquila continued to submit drill core cut samples to Accurassay for assaying using the same procedures described above. In 2007, a total of 12,210 samples were submitted for analysis. A total of 10,433 samples were analyzed in 2008. The reliability of Accurassay results was determined by submitting a number of samples to Actlabs for check assaying.

In 2007, 1,575 samples were sent to SGS laboratories in Don Mills, Ontario for mercury and trace metal analysis. In addition, 222 samples were submitted for whole rock analysis to SGS, Lakefield, Ontario.

In 2008, 1,075 samples from the regular sample stream were submitted to Actlabs in Thunder Bay, Ontario, in order to assess the quality and turn-around time of that lab in comparison to Accurassay.

At Actlabs up to 5 kg of the sample were dry crushed to 80%, -10 mesh, followed by a riffle split and pulverization of 250 g to 95%, -150 mesh (code RX1). Three types of analyses were performed: a gold and silver fire assay with a gravimetric finish (code 1A3); a multi-element, four-acid digestion followed by inductively coupled plasma-optical emission spectrometry (ICP-OES) (code 1F2); and an "mineralized grade" ICP scan for copper, lead, zinc, silver (code 8).
Samples were prepared at SGS using standard preparation procedures (PRP89) consisting of a dry crush of a sample less than 3 kg in weight to 75% passing 2 mm. This step was followed by pulverization of a 250 g split to 85% passing 75 μm. Gold was assayed either by using a standard fire assay procedure on 30 g charges with either an AA spectrometry (FAA313), or by gravimetric finish (FAG303). Base- and trace- metals were analyzed using either 32-element four-acid digestion with an ICPAES finish (code ICP40B), or a 50-element, four-acid digestion with ICPAES and ICP-MS finish (ICM40B). The latter package was used to test for indium, gallium, and other rare metals that might provide added value to the sulphide mineralization. Package ICP90Q was used to test potential higher-grade base metals. Cold vapour analysis was used to test for mercury (CVA14C).

11.2.4 2009 to 2010 Sampling Programs

During 2009 and 2010, a total of 7,742 drill core cut samples (6,885 intervals of core, along with 630 quality control samples, and 227 field splits) were sent to Inspectorate America in Sparks (Reno, Nevada) for analysis. At Inspectorate America, samples of up to 2.0 kg were dried for up to 24 hours, crushed and riffle split to approximately 250 g, and then pulverized to more than 90% -150 mesh (SP-RX-2K).

The samples were submitted for the standard GENX 10 package, which consists of a fire assay for gold (Au-1AT-AA), an ICP run (GNX10-AR-ICP) for metals silver, arsenic, bismuth, copper, molybdenum, lead, antimony, zinc (by aqua regia digestion, ICP analysis), and analysis for mercury (Hg-AR-TR). A fire assay-gravimetric finish (Au-1AT-GV) was performed on all gold results greater than 3 ppm. Higher-grade analyses by AA for zinc (Zn-AR-OR), copper (Cu-AR-OR), and lead (Pb-AR-OR) were run on over-limits (i.e. those values exceeding 10,000 ppm).

A total of 213 check (umpire) assays were performed by SGS in Toronto, Ontario on pulps initially analyzed by Inspectorate America during the 2009 to 2010 program. SGS analyzed gold by fire assay with gravimetric finish (FAG303) and 40 trace elements by ICP (ICP40B, multi-acid digestion). ICP over limits for silver were rerun using higher-grade analysis by AA (AAS21E) and for copper, lead, or zinc by ICP (ICP90Q).

In addition, 12 core samples were submitted to SGS in Lakefield, Ontario for whole rock analysis (XRF76C).

11.2.5 2010 to 2011 Sampling Programs

The 2010-2011 sampling program was a continuation of the 2009-2010 program. Throughout this period a total of 15,831 drill core cut samples (14,958 intervals of core, along with 873 quality control samples) were sent to Inspectorate America in Sparks (Reno, Nevada) for analysis. At Inspectorate America, samples of up to 2.0 kg were dried for up to 24 hours, crushed and riffle split to approximately 250 g, and then pulverized to more than 90% -150 mesh (SP-RX-2K).

The majority of samples during this program were submitted for fire assay for gold and standard 4 acid 30 element ICP analysis, however, the standard GENX 10 package (as described above) was utilized for a limited number of samples during the early part of the program. Furthermore, for metallurgical samples testing mineralized zones where only precious metals were of interest, the samples were submitted for fire assay for both gold and silver. For all samples, a fire assay-gravimetric finish (Au-1AT-GV) was performed on all gold results greater than 3 ppm. High-grade analyses by AA for zinc (Zn- AR-OR), copper (Cu-AR-OR), and lead (Pb-AR-OR) was run on over-limits (i.e. those values exceeding 10,000 ppm).

11.2.6 2015 Sampling Program

The 2015 sampling program consisted of sampling two exploration holes, which were drilled during a metallurgical drilling campaign (Table 11.2). A total of 120 drill core cut samples were submitted to Minerals Processing Corporation ("MPC") in Carney, MI to be prepped for analysis. Samples were dried for 24 hours, crushed and riffle split to approximately 250 g, and then pulverized to more than 90% -150 mesh.

Pulps were sent directly from the MPC laboratory to Inspectorate (Bureau Veritas) in Sparks, Nevada for analysis. All samples were submitted for fire assay for gold (FA430) with an AAS finish and a 34 element Aqua Regia digestion ICP analysis (AQ270). Gold assays exceeding 10 ppm and silver assays greater than 300 ppm were re-analyzed with fire assay and gravimetric finish (FA530-Au/Ag).

Nine of the samples were also submitted for XRF analysis for major oxides, Ba, C, S, Cu, Ni, Pb, Sr, Zn, and Zr for whole rock characterization.

Of the 120 total samples analyzed in 2015, nine of the samples were standards and two were blanks. These samples were inserted into the sample stream at a rate of roughly one standard/blank per 10 samples.

TABLE 11.2
SUMMARY OF 2015 SAMPLING PROGRAM

DDH	Samples	Laboratory Location	Year
MET-15-01 to MET-15-09*	181	SGS Lakefield, ON	2015
LK-15-508 to LK-15-509	120	Inspectorate (Bureau Veritas), Sparks, NV	2015

***** *Assays not included in assay database*

11.2.7 2016 to 2017 Sampling Programs

Boxed core samples from a secure area at the drill sites were transported daily to the nearby Aquila field office, located on the Property. Core was logged geotechnically and geologically, then marked for samples at the field office facility and transferred to the core facility in Carney, Michigan (27 km away) for photographing, cutting, sampling and archiving. Company-employed or -contracted geologists marked mineralized intersections for sampling and assaying and marked intersections were sampled by Aquila-employed geo-technicians.

Virtually all drill core from both mineralized and unmineralized intervals in all drill holes was sampled. Typically, core intersecting mafic dykes was not sampled unless it exhibited mineralization. Samples were sawn in half using a diamond core saw and one-half of the core was placed in sample bags and tagged with unique sample numbers, while the remaining half was returned to the core box for storage.

Assay intervals are generally 1.5 m in length and honour geological boundaries. A total of 2,724 drill core cut samples were collected and sent for analysis during the 2016 and 2017 drill programs at the Property.

Assay samples were transported from the Aquila field office in Michigan directly to the Minerals Processing Corporation ("MPC") laboratory of Carney, Michigan to be prepped for analysis. Samples were dried for 24 hours, crushed and riffle split to approximately 250 g, and then pulverized to more than 90% -150 mesh. One sample split was sent to Bureau Veritas Mineral Laboratories USA ("Bureau Veritas") (formerly Inspectorate America Corporation ("Inspectorate") in Sparks, Nevada) for analysis.

All remaining pulps and coarse rejects were returned to the Project warehouse facilities in Michigan.

All samples submitted for analysis were analyzed for gold, silver, copper, lead and zinc. Gold was determined using fire assay method (FA430) with an AAS finish. Silver, copper, lead and zinc were determined using either a three (aqua regia) or four acid digest with ICP-ES or ICP-MS finish. Gold assays exceeding 10 ppm and silver assays greater than 100 ppm were re-analyzed by fire assay method with a gravimetric finish.

Some samples were also analyzed by fusion lithium metaborate/lithium tetraborate X-ray fluorescence analysis (fusion XRF) to determine the whole rock composition of the drill core.

A number of samples were assayed at MPC, including pulp duplicates and over limits for gold and sulphur.

Analytical methods at MPC for gold, silver, copper, lead and zinc are by fire assay with an atomic absorption finish. Gold results for samples greater than 3.0 g/t Au and silver results for samples greater than 300 g/t Ag are analyzed by fire assay with a gravimetric finish.

Bureau Veritas is a leading provider of laboratory testing, inspection and certification, operating in 1,430 offices and laboratories in 140 countries. Bureau Veritas Minerals is ISO 9001 compliant and for selected methods, ISO 17025 compliant and has an extensive Quality Assurance/Quality Control ("QA/QC" or "QC") program to ensure that clients receive consistently high-quality data.

MPC is an ISO/IEC 17025:2005 certified analytical laboratory located in Carney, Michigan. Mr. Tom Quigley, former V.P. Exploration for Aquila, is also a shareholder and the current president of MPC.

11.3 BULK DENSITY

Between 2007 and 2011, Aquila completed approximately 825 bulk density measurements on drill core samples using a standard weight-in-water, weight-in-air determination. An electronic scale with an accuracy of ±0.1 g was used for weight measurements. The scale was mounted on a tripod standing over a large pail of water beneath the scale. Dry drill core pieces were weighed on the scale and then in a metal basket suspended in water. The metal basket was balanced on the scale to zero. The entire split-core interval that was sampled for assaying was weighed in two or more operations. The following procedures were used to ensure measurements were accurate:

- Core was washed to remove material remaining from diamond saw cutting.
- Core was dried after washing.
- Measurements were repeated periodically and checked to maintain accuracy of measurements.
- Accuracy of scale was checked and adjusted by using standard set of weights as part of standard procedures.

In addition, Aquila received bulk density measurement on core samples submitted for assaying as follows:

- 88 core bulk density measurements by G&T laboratory via the Stewart Group.
- 261 pycnometer (ALSG2) specific gravity measurements (including 41 repeat measurements) by Accurassay in Thunder Bay, Ontario.
- 37 specific gravity measurements by pycnometry by ALS Chemex (GA- GRA08b) in Vancouver.

11.4 SECURITY

For the 2009 to 2011 programs, drill core was removed directly from split-tube core barrels and placed in 3 m long aluminium core "V" rails. It was then transported by Aquila personnel directly to the Aquila field office located less than one mile away from the drill rig. These rails were placed inside the Project's core logging garage, which is secured by lock and alarm systems. The core was geotechnically logged in the logging garage and then transferred to wooden core boxes by Aquila personnel. Once transferred to wooden boxes, the core was geologically logged and marked for sampling. The wooden core boxes were then covered and transported to the Project's core warehouse in Carney, Michigan. The core was photographed, cut, sampled, and shelved at the Carney facility. The Carney core warehouse was secured by locks and alarm system. The core samples were packed and shipped from the Project's core warehouse facility by ground courier directly to Inspectorate America labs in Reno, Nevada.

Prior to 2009, all core was housed either in the outdoor fenced storage area at the field office or at the indoor Daggett core warehouse. As of 2011, all drill core from previous programs was transferred to the Project's core warehouse in Carney and photographed prior to archiving.

All digital computer files, including core photographs, logs, and data, are saved on a central server at the BFJV main office in Stephenson, Michigan. The entire contents of the server are backed-up onto a tape drive and removed from the premises on a daily basis.

11.5 QUALITY ASSURANCE/QUALITY CONTROL PROGRAM

Strict sampling and QA/QC protocol have been followed, including the insertion of standards and blanks in the sample stream on a regular basis. Sample intervals were typically 1.5 m in length.

The exploration work conducted by Aquila was carried out in general compliance with industry best practices with the aid of a QA/QC or QC program. All aspects of the exploration data acquisition and management, including mapping, surveying, drilling, sampling, sample security, and assaying and database management, were conducted under the supervision of appropriately qualified geologists.

The analytical QC data for the Project included both internal and external QC measures. Aquila used IS0-17025 and ISO-9001-accredited laboratories that implemented internal laboratory measures consisting of inserting QC samples (blanks and certified reference materials and duplicate pulps) within each batch of samples submitted for assaying.

The external analytical QC measures implemented by Aquila for all samples submitted for assaying included the insertion of QC samples (blanks and certified reference material) with each batch of core drilling samples, field splits, and submission of analyzed pulps for check (umpire) assays to an alternate laboratory.

11.5.1 2002-2008 Drill Programs

An evaluation of the 2002 to 2008 analytical QC measures and results can be found in previous Technical Reports (Keller 2009; Dematties 2007; Dematties, 2004). The aforementioned reports concluded that exploration data from the Project were acquired using adequate QC procedures that generally meet industry best practices for a Mineral Resource delineation-stage exploration property.

11.5.2 2009-2010 Drill Programs

QA/QC for the 2009 to 2010 sampling programs involved inserting external reference standards and blanks into the sample stream. Reference standards were purchased from CDN Resources Laboratory ("CDN"), Delta, BC, which prepares standards under the supervision of Duncan Sanderson, Certified Assayer of British Columbia. At CDN, material is dried, crushed, pulverized, and screened. The screened material is mixed for five days in a double-cone mixer. Splits are taken and sent to up to 15 laboratories for round robin assaying.

11.5.2.1 Performance of Standards

Certified Reference Materials ("CRMs" or "standards") chosen for use at the Project site included high- and low-grades of gold-only standards and several different multi-element standards that contain some or all of the following metals at varying grades: gold, silver, copper, lead, zinc. Table 11.3 shows the reference standards employed during the 2009 to 2011 program and the number of individual element analyses obtained. A total of 490 CRMs was used.

TABLE 11.3
MINERAL REFERENCE STANDARDS (2009-2010)

Standard	Number Used	Au Assays	Ag Assays	Cu Assays	Pb Assays	Zn Assays
CDN-CM-4	30	30	-	30	-	-
CDN-GS-2B	1	1	-	-	-	-
CDN-GS-2C	55	55	-	-	-	-
CDN-GS-2F	52	52	-	-	-	-
CDN-GS-3C	3	3	-	-	-	-
CDN-GS-3G	18	18	-	-	-	-
CDN-HLHZ	39	39	39	39	39	39
CDN-HLLC	5	5	5	5	5	5
CDN-HZ-3	83	83	83	83	83	83
CDN-ME-2	96	96	96	96	-	96
CDN-ME-3	12	12	12	12	12	12
CDN-ME-7	18	18	18	18	18	18
CDN-SE-2	78	78	78	78	78	78
Total	**490**	**490**	**331**	**361**	**235**	**331**

The performance of the 13 reference standards for all 5 elements was examined and data falling outside ± 3 standard deviations from the accepted mean value were considered failures. Standard data were within accepted limits of error with one exception. Standards in some batches consistently returned lead and zinc values greater than three standard deviations where drill core results were lead and zinc near 1%.

Discussions with Inspectorate determined that 1% lead and zinc values approach the upper limit of analysis for the ICP package, and are at the 1% lower limit for the AA range. Although original ICP analyses were largely within acceptable limits of error, many were slightly over 1%. These samples were subsequently sent for AA analysis. The AA analysis was found to be slightly out of calibration, which gave the results a high bias. Inspectorate addressed the over-limit reporting protocol, as well as the AA calibration issue, and adequately re-assayed all affected samples.

11.5.2.2 Performance of Blanks

Aquila used two blank samples (CDN-BL-2 and CDN-BL-3) prepared from barren granitic rock by CDN. These blanks were certified to contain low precious metals by D. Sanderson, a licensed assayer of British Columbia, after independent analysis of the material at ten commercial laboratories. Aquila also used bulk quartz sand as a blank. A total of 140 blank samples were used in 2009 to 2010.

For gold analyses, 132 samples measured at or below detection limit and another six measured in the less than 100 ppb range. Two gold analyses were anomalous. Silver performed similarly with all except 122 samples at or below detection limit, another 17 samples were less than 1 ppm, and one sample was considered anomalous. The single anomalous silver sample also returned anomalous levels of gold, lead, and zinc. All other base metal analyses were within acceptable limits for blank material. Based on the overall performance of the blanks, the author does not consider the anomalous samples to be of significant impact to the assay data.

11.5.2.3 Performance of Duplicates

Field splits of core samples provided an additional check on analytical consistency. The preparation of these samples involved breaking drill core from a selected sample into less than three centimetre-sized fragments, randomly mixing them, and splitting the fragments into two separate samples. While these core splits are not considered true field duplicates, they served as a general check on the reproducibility of the analyses. A total of 227 these pseudo-duplicates were used.

In addition to the pseudo-duplicates, a total of 213 replicate assays were also performed on pulps analyzed by Inspectorate America and subsequently by SGS in Toronto, Ontario.

A linear regression plot of gold, silver, copper, lead, and zinc for 213 replicate samples were plotted and comparative analyses are robust, with regressions for all five elements falling within acceptable limits of deviation.

Linear regression plots of gold, silver, copper, lead, and zinc for the 710 replicate samples were plotted and comparative analyses are robust, with regressions for all five elements falling within acceptable limits of deviation.

11.5.3 2010-2011 Drill Programs

Quality control for the 2010 to 2011 sampling programs was a continuation of the previous program and involved inserting external reference standards and blanks into the sample stream. Standards were purchased from CDN, Delta, BC, which prepares standards under the supervision of Duncan Sanderson, Certified Assayer of British Columbia. At CDN, drill core sample material is dried, crushed, pulverized, and screened. The screened material is mixed for five days in a double-cone mixer. Splits are taken and sent to up to 15 laboratories for round robin assaying.

11.5.3.1 Performance of Standards

Standards chosen for use at the Project site included high- and low-grades of gold-only standards and several different multi-element standards that contain some or all of the following metals at varying grades: gold, silver, copper, lead, zinc. Table 11.4 shows the reference standards employed during the 2010 to 2011 program and the number of individual element analyses obtained. A total of 978 mineral reference standards were used.

Table 11.4 Mineral Reference Standards (2010-2011)						
Standard	**Number Used**	**Assays**				
		Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
CDN-GS-3G	54	54	50	50	50	50
CDN-HZ-3	65	65	65	65	65	65
CDN-ME-2	664	664	653	653	653	653
CDN-ME-3	19	19	19	19	19	19
CDN-ME-7	13	13	13	13	13	13
CDN-ME-14	97	97	97	97	97	97
CDN-ME-16	66	66	66	66	66	66
Total	**978**	**978**	**963**	**963**	**963**	**963**

The vast majority of standards fell within acceptable limits. A total of 16 analyses for gold failed, 10 of which were for standards provisional for gold (CDN-HZ-3 and CDN-ME-14). Batches pertaining to certified standards that failed were not rerun due to other standards within the same batch passing. A total of eight standards failed for copper and four for lead. These were also considered to be either of no impact to the Mineral Resource Estimate, with other standards passing in the same batch or to be mismatched samples.

11.5.3.2 Performance of Blanks

Aquila used two blank samples (CDN-BL-2 and CDN-BL-3) prepared from barren granitic rock by CDN. These blanks were certified to contain low precious metals by D. Sanderson, a licensed assayer of British Columbia, after independent analysis of the material at ten commercial laboratories. Aquila also used bulk quartz sand as a blank. A total of 103 blank samples were used in 2010 to 2011.

For gold analyses, 89 samples measured at or below detection limit and another 13 measured in the less than 100 ppb range. Two gold analyses were anomalous. Silver performed similarly with 91 samples at or below detection limit, another 11 samples were less than 1 ppm, and one sample was considered anomalous. The anomalous silver sample also registered low. One copper analysis and 3 zinc analyses were anomalous. All lead analyses were within acceptable limits for blank material. Based on the performance of the blanks, the author does not consider the anomalous samples to significantly impact the integrity of the assay data.

11.5.3.3 Check Assays

A total of 710 check (umpire) assays were performed by SGS in Toronto, Ontario on pulps initially analyzed by Inspectorate during the late 2010-2011 program. SGS analyzed for gold by fire assay with gravimetric finish (FAG303) and 40 trace elements by ICP (ICP40B, multi-acid digestion). ICP over-limits for silver were rerun using analysis by AA (AAS21E) and for copper, lead, or zinc by ICP (ICP90Q).

Linear regression plots of gold, silver, copper, lead, and zinc for the 710 replicate samples were plotted and comparative analyses are robust, with regressions for all five elements falling within acceptable limits of deviation.

11.5.4 2015 Drill Program

Quality control for the 2015 sampling program was undertaken in a similar fashion to previous years and included the insertion of external reference standards and blanks into the sample stream. Standards were purchased from CDN, Delta, BC, which prepares standards under the supervision of Duncan Sanderson, Certified Assayer of British Columbia. At CDN, material is dried, crushed, pulverized, and screened. The screened material is mixed for five days in a double-cone mixer. Splits are taken and sent to up to 15 laboratories for round robin assaying.

11.5.4.1 Performance of Standards

Standards chosen for use at the Project site included high- and low-grade standards certified for some or all of the following metals at varying grades: gold, silver, copper, lead and zinc. Table 11.5 lists the reference standards employed during the 2015 program and the number of individual element analyses obtained. A total of 9 mineral reference standards were used to monitor QC for the 2 holes (LK-15-508 and LK-15-509) included in the current Mineral Resource Estimate.

TABLE 11.5
MINERAL REFERENCE STANDARDS (2015)

Standard	Number Used	Assays				
		Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
CDN-GS-3G	1	1	0	0	0	0
CDN-HZ-3*	3	3	3	3	3	3
CDN-ME-16	5	5	5	5	5	5
Total	**9**	**9**	**8**	**8**	**8**	**8**

***** *Provisional only for gold.*

All standards fell within acceptable limits for silver, copper and zinc and one out of three CDN-HZ-3 results for gold and one out five CDN-ME-16 results for zinc failed high. Batches corresponding to standard failures were not rerun due to other standards within the same batch passing.

11.5.4.2 Performance of Blanks

Aquila inserted two blanks only into the sample stream for the 2015 QA/QC program. Blank results for both gold and silver were at or very close to detection limit and both results for copper, lead and zinc displayed elevated results. A review of the internal laboratory QC for these 3 latter elements reveals multiple internal lab blank results falling at detection limit and the author does not consider the performance of these blanks to significantly impact the assay data.
Performance of Duplicates

No duplicates were assessed for the 2015 QC program as there were too few data to assess.

11.5.5 2016 Drill Program

11.5.5.1 Performance of Standards

CRMs chosen for use at the Project site included high- and low-grades of gold-only standards and several different multi-element standards that contain some or all of the following metals at varying grades: gold, silver, copper, lead, zinc. Table 11.6 shows the reference standards employed during the 2016 program and the number of individual element analyses obtained. A total of 80 standards were used.

TABLE 11.6
CERTIFIED REFERENCE MATERIALS USED IN 2016

2016 Standards	Number Used	Au Assays	Ag Assays	Cu Assays	Pb Assays	Zn Assays
CDN-GS-2J	6	6	-	-	-	-
CDN-GS-P7E	10	10	-	-	-	-
CDN-HZ-3	14	14	14	14	14	14
CDN-ME-1301	18	18	18	18	18	18
CDN-ME-14	20	20	20	20	20	20
CDN-ME-16	12	12	12	12	12	12
Total	**80**	**80**	**64**	**64**	**64**	**64**

The performance of the six CRMs for all five elements was examined and data falling outside ± 3 standard deviations from the accepted mean value were considered failures. Standard data were within accepted limits of error for silver, copper and zinc. The CDN-GS-P7E, CDN-HZ-3 and CDN-ME-16 CRMs returned one result each greater than three standard deviations for gold (a 4% failure rate). The CDN-MS-1301, CDN-ME-14 and the CDN-ME-16 standards respectively returned three, two and one result(s) greater than three standard deviations for lead (a 9% failure rate). Table 11.7 outlines CRM failure during the 2016 program.

TABLE 11.7
CERTIFIED REFERENCE MATERIALS FAILURES IN 2016

2016 Standards	Number Used	Au Failures	Ag Failures	Cu Failures	Pb Failures	Zn Failures
CDN-GS-2J	6	0	-	-	-	-
CDN-GS-P7E	10	1	-	-	-	-
CDN-HZ-3	14	1	0	0	0	0
CDN-ME-1301	18	0	0	0	3	0
CDN-ME-14	20	0	0	0	2	0
CDN-ME-16	12	1	0	0	1	0
Total Failures	**80**	**3**	**0**	**0**	**6**	**0**
Total % Failures		**4%**	**0%**	**0%**	**9%**	**0%**

11.5.5.2 Performance of Blanks

Aquila used the CDN-BL-3 blank prepared from barren granitic rock by CDN Resources Laboratories Ltd., of Langley British Columbia, during the 2016 program. The CDN-BL-3 blank is certified to contain low precious metals by D. Sanderson, a licensed assayer of British Columbia, after independent analysis of the material at ten commercial laboratories. A total of 13 blank samples were used in 2016.

All samples returned with values around detection limit for all elements.

11.5.5.3 Performance of Duplicates

No duplicates were assessed for the 2016 QC program.

11.5.6 2017 Drill Program

11.5.6.1 Performance of Standards

The standards used at the Project for the 2017 drilling were selected in similar fashion to that of the previous year, Table 11.8 summarizes the standards utilized during the 2017 program and the number of individual element analyses obtained. A total of 213 CRMs was used.

TABLE 11.8
CERTIFIED REFERENCE MATERIALS USED IN 2017

2017 Standards	Number Used	Au Assays	Ag Assays	Cu Assays	Pb Assays	Zn Assays
CDN-GS-2J	45	45	-	-	-	-
CDN-GS-5G	4	4	-	-	-	-
CDN-GS-P7E	38	38	-	-	-	-
CDN-GS-P7J	14	14	-	-	-	-
CDN-HZ-3	4	4	4	4	4	4
CDN-ME-1312	24	24	24	24	24	24
CDN-ME-1308	34	34	34	34	34	34
CDN-ME-14	27	27	27	27	27	27
CDN-ME-17	23	23	23	23	23	23
Total	**213**	**213**	**112**	**112**	**112**	**112**

The performance of the nine CRMs for all five elements was examined and data falling outside ± 3 standard deviations from the accepted mean value were considered failures. The CRM failure rate rose during the 2017 program; however, the majority of failures were not of concern since failures were isolated, with multiple other standards passing in the same batch. In the majority of cases, no further action was warranted. Table 11.9 summarizes standard performance for the 2017 program.

TABLE 11.9
CERTIFIED REFERENCE MATERIALS FAILURES IN 2017

2017 Standards	Number Used	Au Failures	Ag Failures	Cu Failures	Pb Failures	Zn Failures
CDN-GS-2J	45	3	-	-	-	-
CDN-GS-5G	4	0	-	-	-	-
CDN-GS-P7E	38	6	-	-	-	-
CDN-GS-P7J	14	0	-	-	-	-
CDN-HZ-3	4	2	0	2	2	0
CDN-ME-1312	24	0	3	1	6	7
CDN-ME-1308	34	1	1	4	2	6
CDN-ME-14	27	5	0	2	3	1
CDN-ME-17	23	6	4	1	3	6
Total Failures	**213**	**23**	**8**	**10**	**16**	**20**
Total % Failures		**11%**	**7%**	**9%**	**14%**	**18%**

Results for drill holes GT-10 and GT-11 returned with many failures for all elements, except silver, and pulps for these batches were re-analyzed at MPC to check the original results. Comparison of the original results verses the check assays carried out by MPC was made (see Figures 11.1 to 11.5) and confirmation of the original results was given. MPC results for these drill holes have been applied in the database.

FIGURE 11.1 DRILL HOLES GT-10 AND GT-11 CHECK ASSAYS FOR AU: BUREAU VERITAS VERSUS MPC

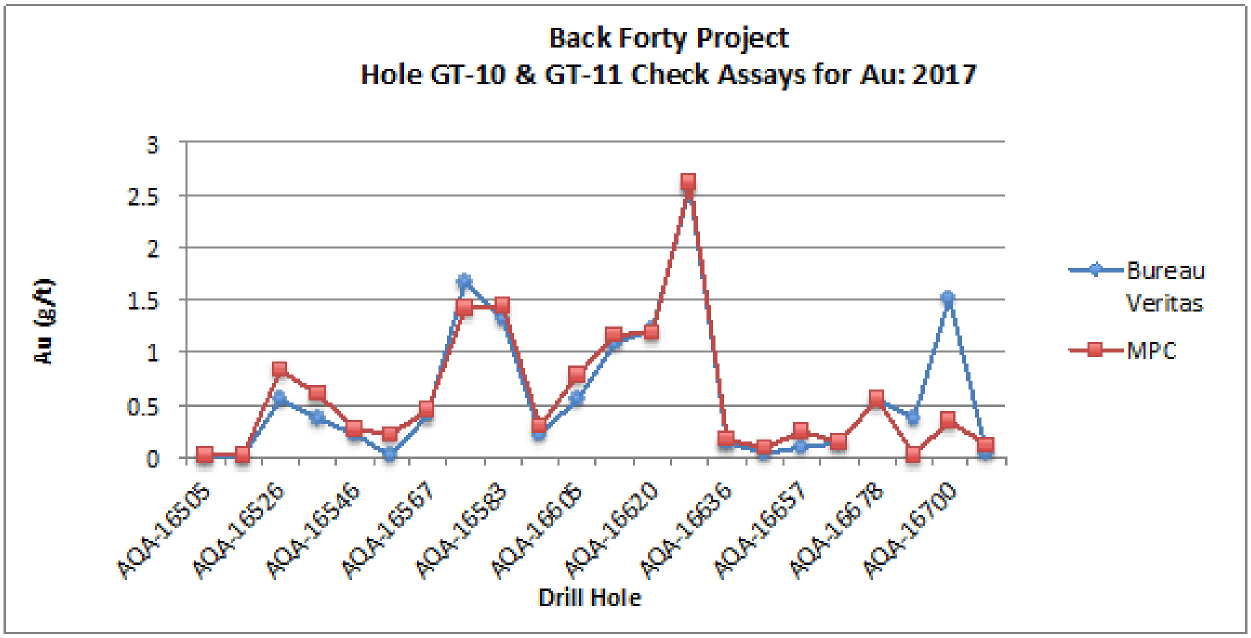


FIGURE 11.2 DRILL HOLES GT-10 AND GT-11 CHECK ASSAYS FOR AG: BUREAU VERITAS VERSUS MPC

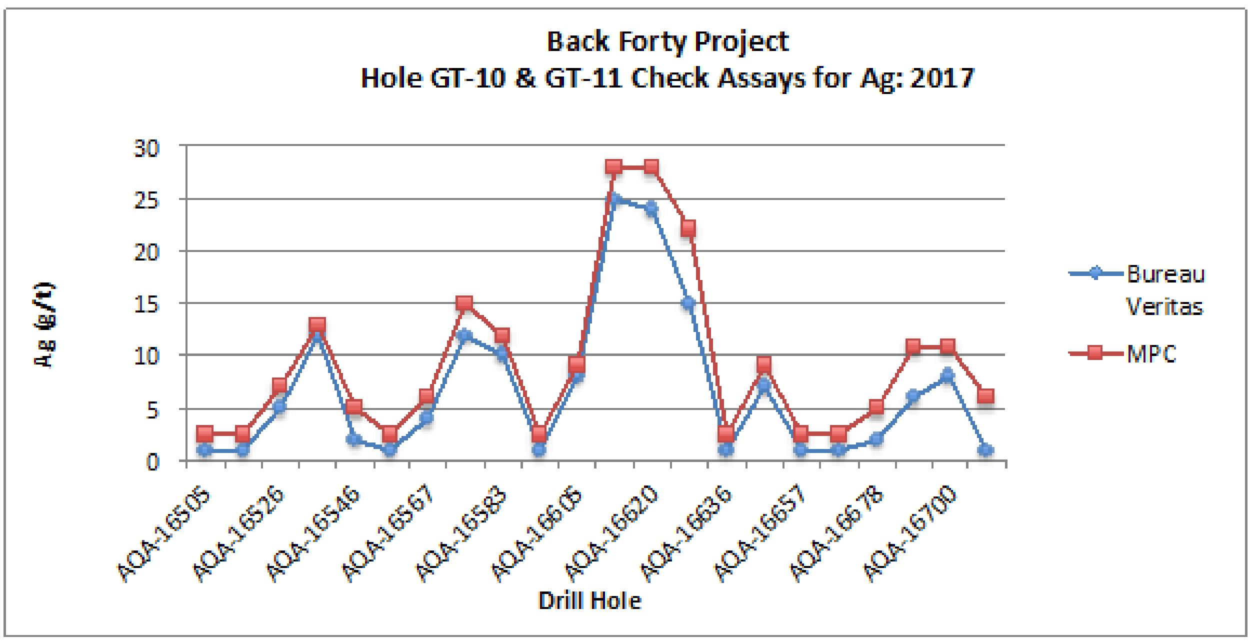


FIGURE 11.3 DRILL HOLES GT-10 AND GT-11 CHECK ASSAYS FOR CU: BUREAU VERITAS VERSUS MPC

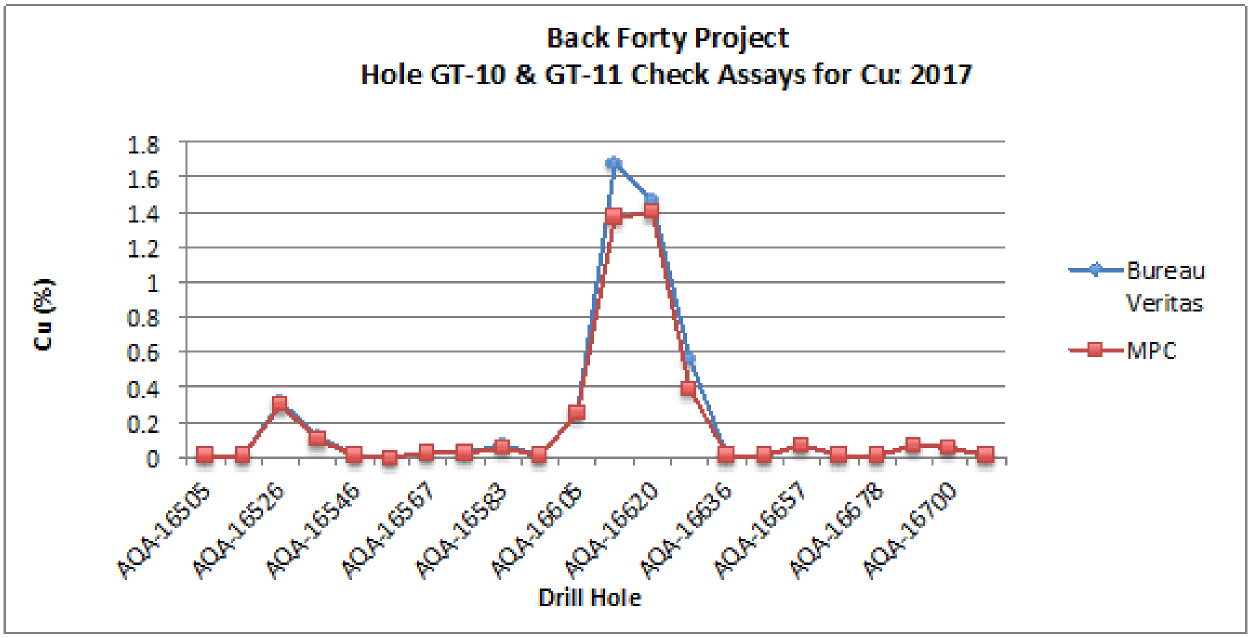


FIGURE 11.4 DRILL HOLES GT-10 AND GT-11 CHECK ASSAYS FOR PB: BUREAU VERITAS VERSUS MPC

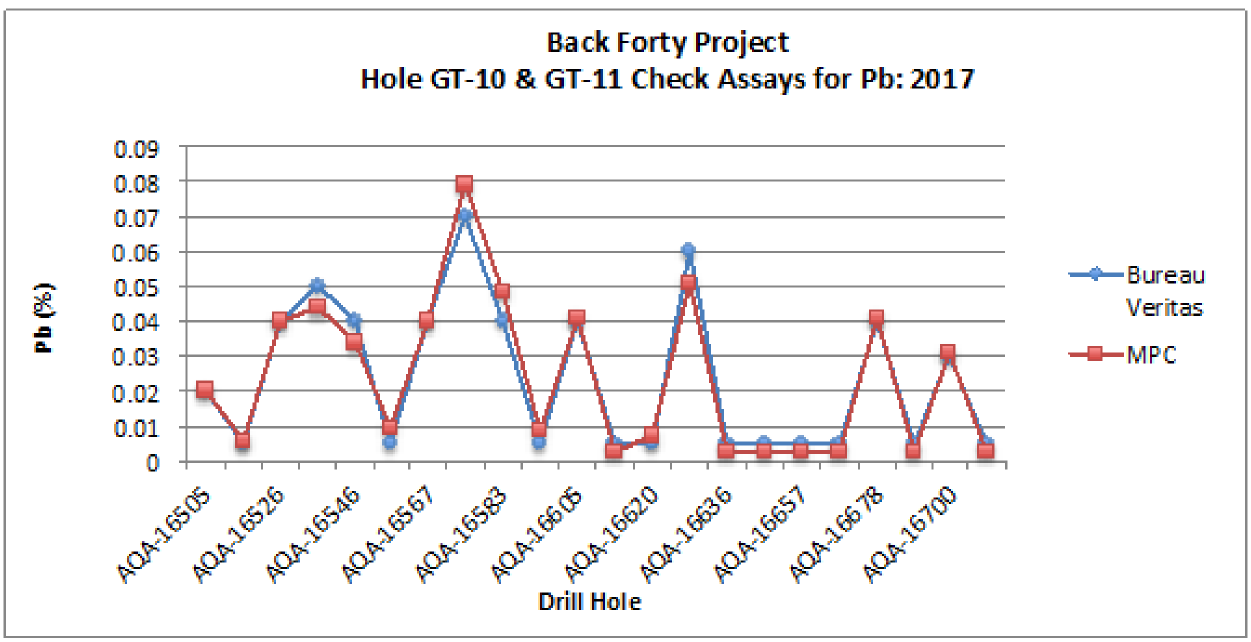
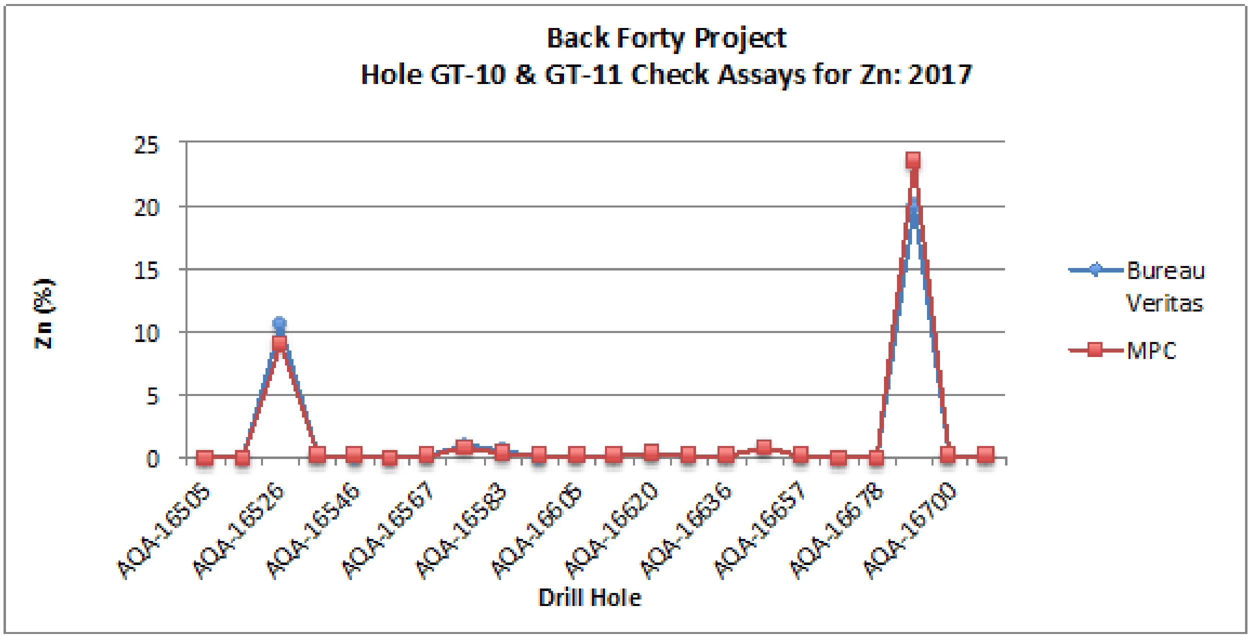


FIGURE 11.5 DRILL HOLES GT-10 AND GT-11 CHECK ASSAYS FOR ZN: BUREAU VERITAS VERSUS MPC



11.5.6.2 Performance of Blanks

Aquila used the CDN-BL-3 blank prepared from barren granitic rock by CDN in the 2017 program. The CDN-BL-3 blank is certified to contain low precious metals by D. Sanderson, a licensed assayer of British Columbia, after independent analysis of the material at ten commercial laboratories. A total of 12 blank samples were used in 2017.

All samples returned with values around detection limit for all elements, except for one result each for gold, silver, lead and zinc returning anomalous values. The anomalous results were still within acceptable limits and the author does not consider contamination to be an issue with the Mineral Resource Estimate data.

11.5.6.3 Performance of Duplicates

No duplicates were assessed for the 2017 QC program.

11.5.6.4 Performance of Check Assays

Aquila sent a number of assays for analysis to both Bureau Veritas and MPC laboratories throughout the 2016 and 2017 drill programs. A total of 108 gold results, 52 silver and 47 copper, lead and zinc results were examined. Comparison of the Bureau Veritas results verses the MPC results was made by P&E and results compare well between labs.

11.6 RECOMMENDATIONS AND CONCLUSIONS

P&E notes the direct interest in MPC laboratory that was held by the Company's VP Exploration during the 2016 to 2017 drill program. The VP Exploration is no longer affiliated with Aquila, therefore there is now no relationship between the laboratory and Company. Analyses carried out at MPC during the 2016 to 2017 program totalled less than 10% of all drill core samples analyzed and, of these samples, only around 3% were exclusively analyzed at MPC. All other samples were also analyzed at Bureau Veritas and comparison of these duplicate analyses is acceptable.

It is P&E's opinion that sample preparation, security and analytical procedures for the Project drilling and sampling programs were adequate for the purposes of the Updated Mineral Resource Estimate.

Based upon the evaluation of the QA/QC programs undertaken by Aquila, P&E concludes that the data are of good quality for use in the Back Forty Updated Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 DATABASE VERIFICATION MAY 2016 SITE VISIT

P&E conducted verification of the Back Forty Project drill hole assay database for gold, silver, zinc, copper and lead by comparison of the database entries with the assay certificates. The assay certificates were obtained in digital format directly from three assay laboratories:

- Inspectorate America of Sparks, Nevada.
- Accurassay of Thunder Bay, ON.
- ALS Chemex (now ALS Minerals) of Vancouver, BC.

Assay data ranging from 2002 through 2011 were verified for the Back Forty Project. 65% (7,543 out of 11,552) of the constrained drilling assay data were checked for gold, 58% for silver, 59% for zinc and 57% for copper and lead. A number of errors were encountered during verification of the Back Forty database and were corrected in the database utilized to calculate the current Mineral Resource Estimate.

12.2 SITE VISIT AND DUE DILIGENCE SAMPLING MAY 2016

The Project was visited by Mr. Yungang Wu, P.Geo., and Mr. Eugene Puritch, P.Eng. of P&E on May 23, 2016, for the purposes of completing a site visit and due diligence sampling. Mr. Wu and Mr. Puritch obtained information pertaining to general data acquisition procedures, core logging procedures and quality assurance/quality control ("QA/QC" or "QC").

Mr. Wu and Mr. Puritch collected 12 samples from 12 diamond drill holes and one sample from outcrop of the Pinwheel Gossan during the site visit. Samples were selected over a range of grades from the stored drill core and collected by taking a 1⁄4 split of the half core remaining in the core box. Samples were placed into plastic bags with unique tag identification, and were placed into a larger bag and transported by Mr. Puritch to AGAT Laboratories, Mississauga for both preparation and analysis.

AGAT is an independent lab that has developed and implemented at each of its locations a Quality Management System ("QMS") designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

AGAT maintains ISO registrations and accreditations, which provide independent verification that a QMS is in operation at the location in question. Most AGAT laboratories are registered or are pending registration to ISO 9001:2000.

Samples were analyzed for gold by fire assay with atomic absorption or gravimetric finish; silver, copper and zinc by 4-Acid Digest with ICP-OES finish; and lead by 4-Acid Digest with ICP-MS finish. Densities were also determined on all 13 of the samples.

Results of the site visit due diligence samples are presented in Figure 12.1 to Figure 12.5.

FIGURE 12.1 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR AU: MAY 2016

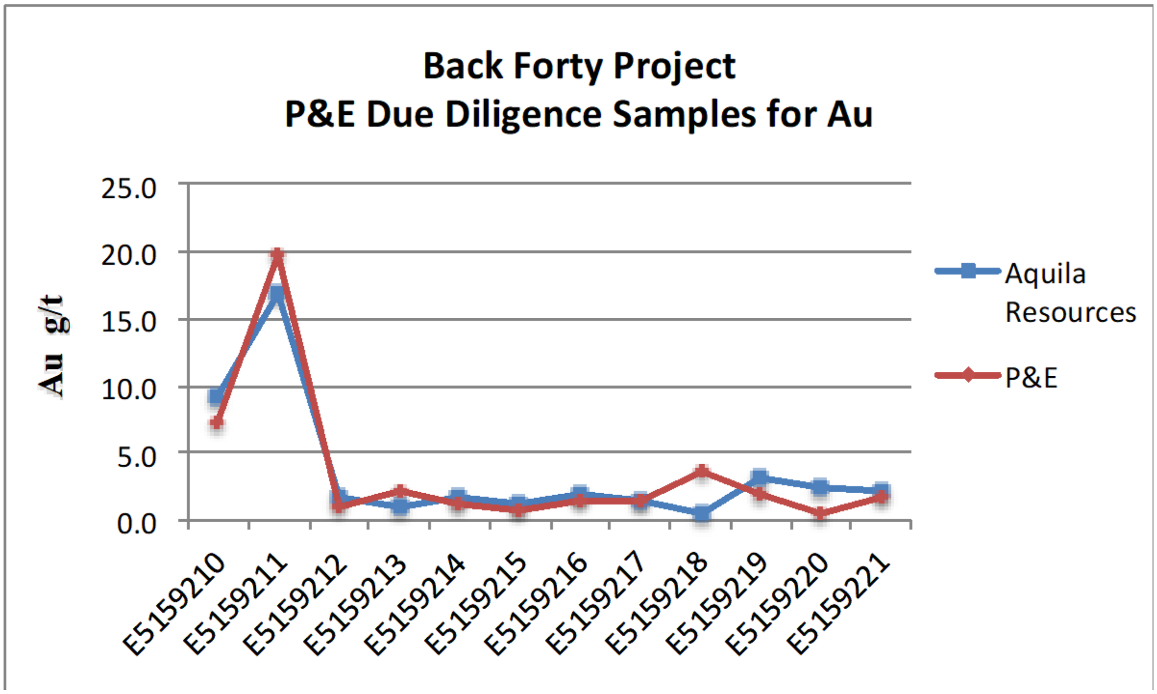


FIGURE 12.2 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR AG: MAY 2016

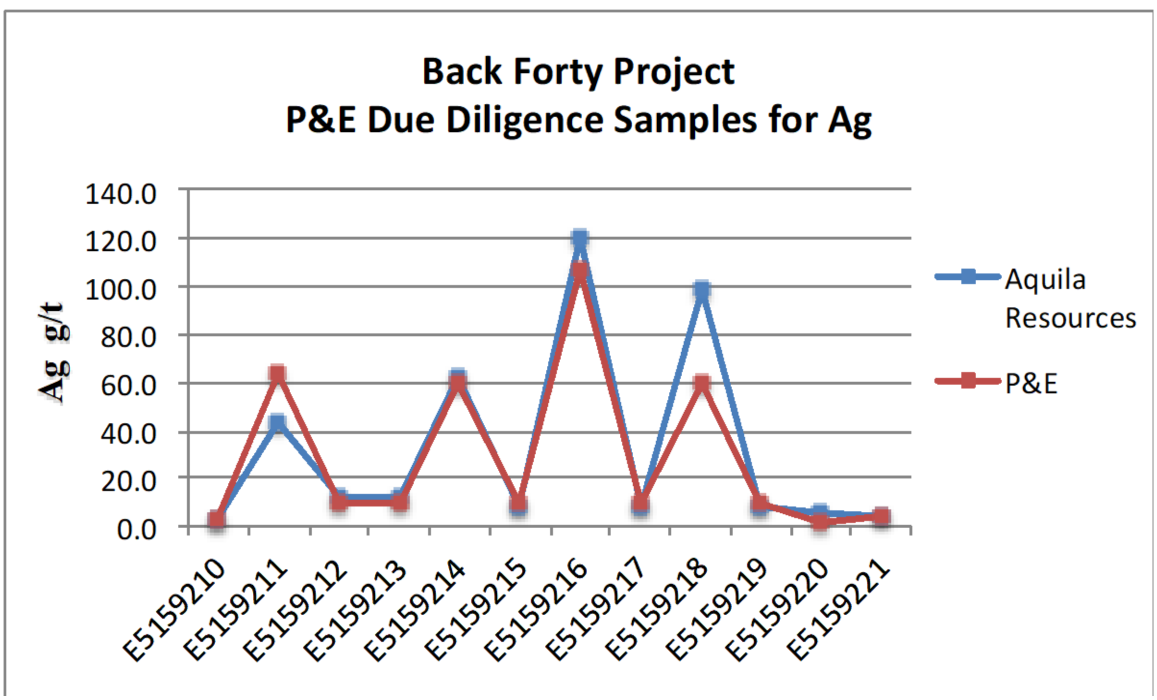


FIGURE 12.3 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR ZN: MAY 2016

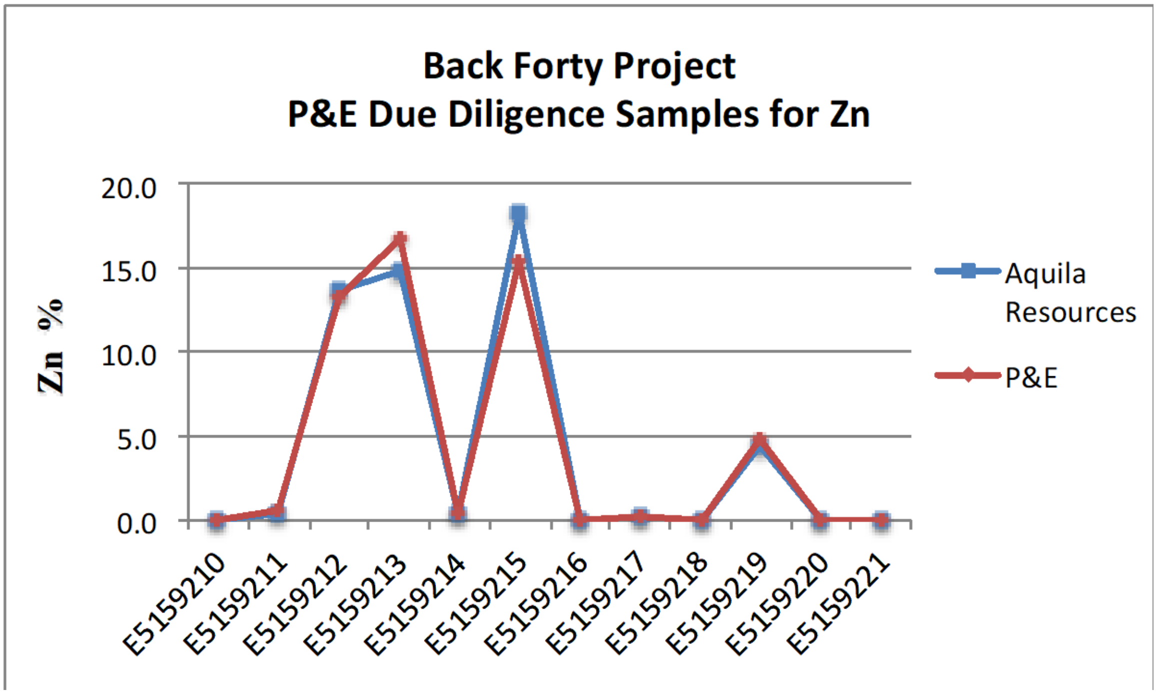


FIGURE 12.4 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR CU: MAY 2016

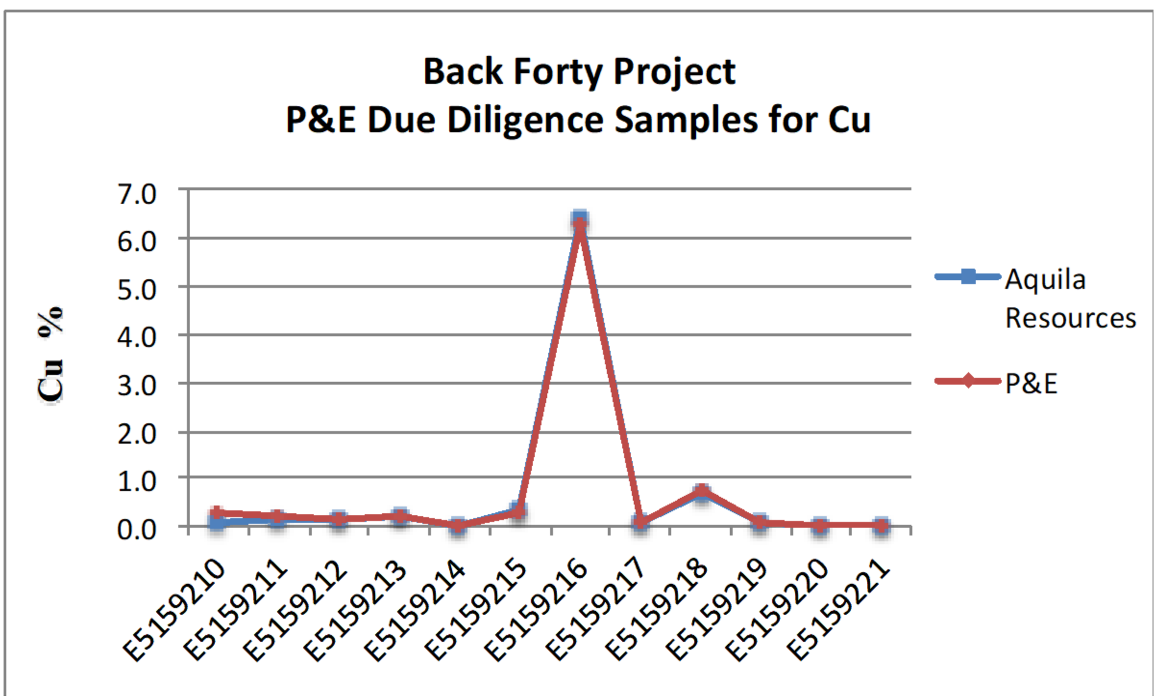
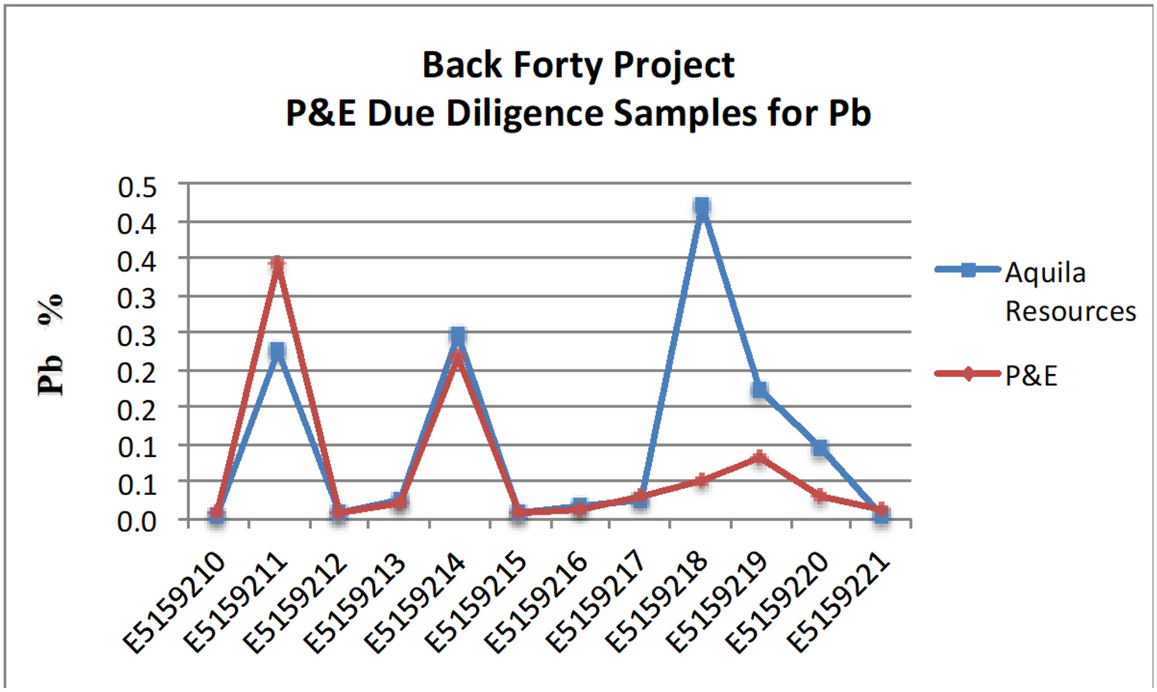


FIGURE 12.5 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR PB: MAY 2016



12.3 DATABASE VERIFICATION NOVEMBER 2017 SITE VISIT

P&E conducted verification of the Back Forty Project drill hole assay database for gold, silver, zinc, copper and lead by comparison of the database entries with the assay certificates. The assay certificates were obtained in digital format directly from two assay laboratories:

- Inspectorate America of Reno, Nevada.
- MPC, of Carney, Michigan.

Assay data ranging from 2016 through 2017 were verified for the Project. 96% (497 out of 517) of the constrained drilling assay data were checked and 53% (2,726 out of 5,141) of the overall data were checked against the original laboratory certificates from Inspectorate America and MPC. Seven errors were observed for gold and ten for copper. All errors were corrected in the database.

12.4 SITE VISIT AND DUE DILIGENCE SAMPLING NOVEMBER 2017

The Project was visited again by Mr. Yungang Wu, P.Geo., of P&E from November 13 to 14, 2017, for the purposes of completing a site visit and due diligence sampling. Mr. Wu obtained information pertaining to general data acquisition procedures, core logging procedures and QA/QC.

Mr. Wu collected 12 samples from nine diamond drill holes during the November 2017 site visit. Samples were selected over a range of grades from the stored drill core and collected by taking a 1⁄4 split of the half core remaining in the core box. Samples were placed into plastic bags with unique tag identification, and were placed into a larger bag for transport, via courier, to AGAT Laboratories, Mississauga for both preparation and analysis.

Samples were analyzed for gold by fire assay with atomic absorption or gravimetric finish; silver, copper, lead and zinc by 4-Acid Digest with ICP-OES or ICP-MS finish. Densities were also determined on all 12 of the samples.

Results of the November 2017 site visit due diligence sampling are presented in Figures 12.6 to 12.10.

12.5 RECOMMENDATIONS AND CONCLUSIONS

Based upon P&E's due diligence sampling and data verification, P&E concludes that the data are of good quality for use in the Back Forty Updated Mineral Resource Estimate.

FIGURE 12.6 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR AU: NOVEMBER 2017

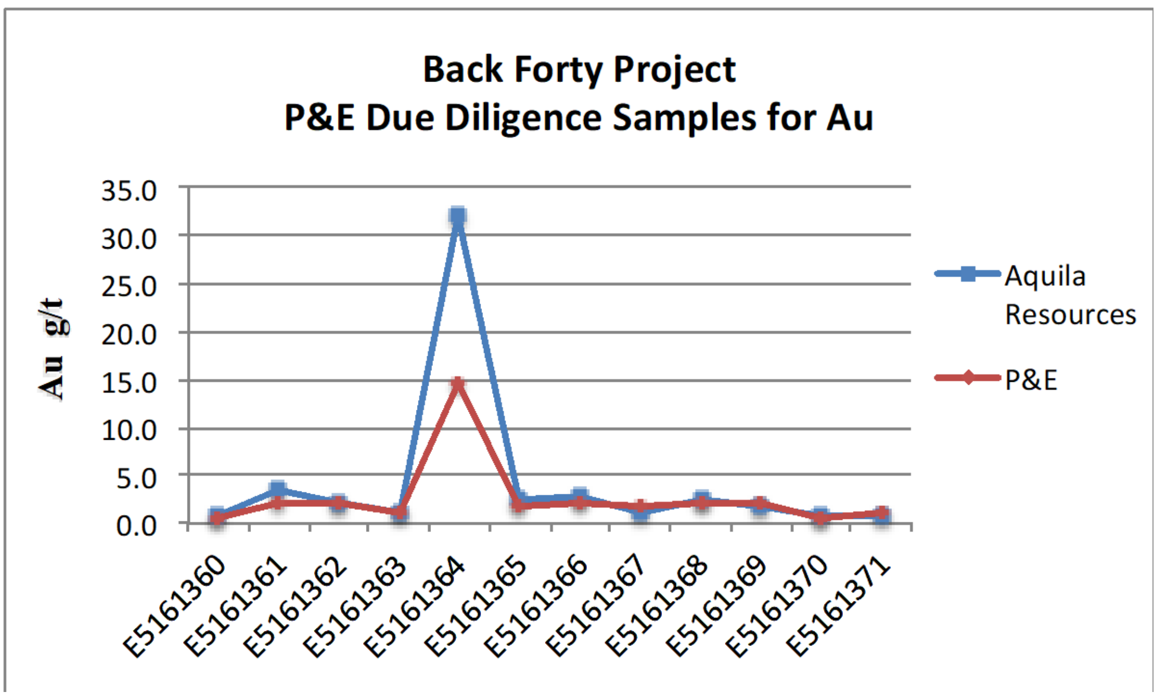


FIGURE 12.7 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR AG: NOVEMBER 2017

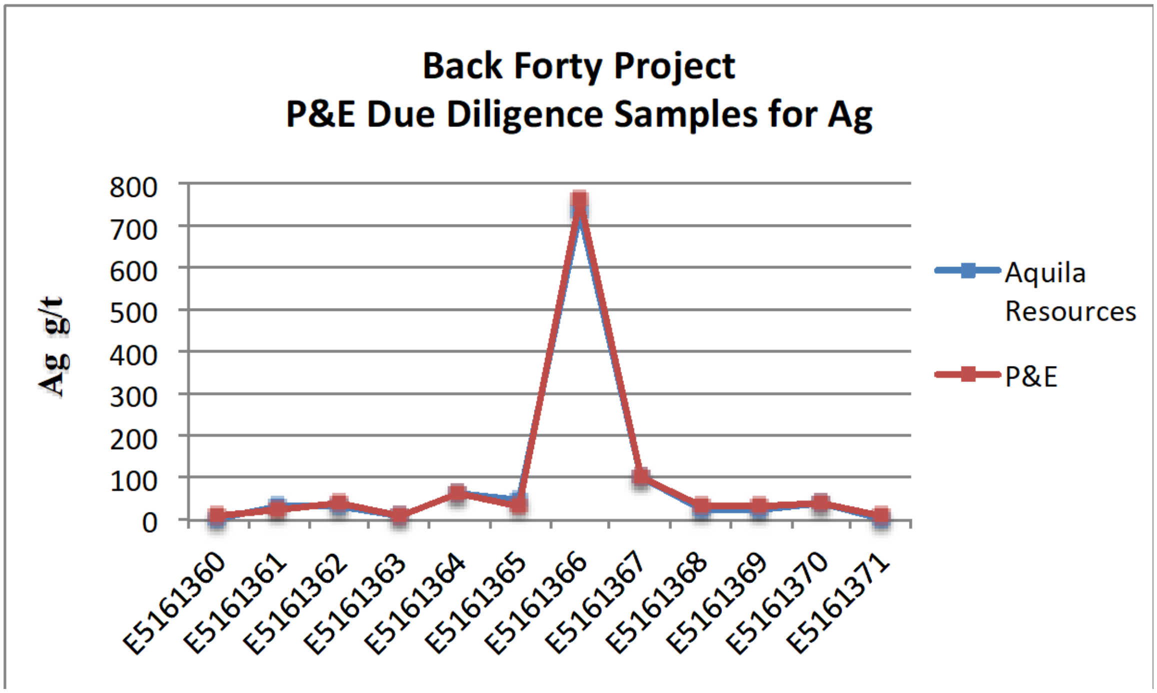


FIGURE 12.8 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR ZN: NOVEMBER 2017

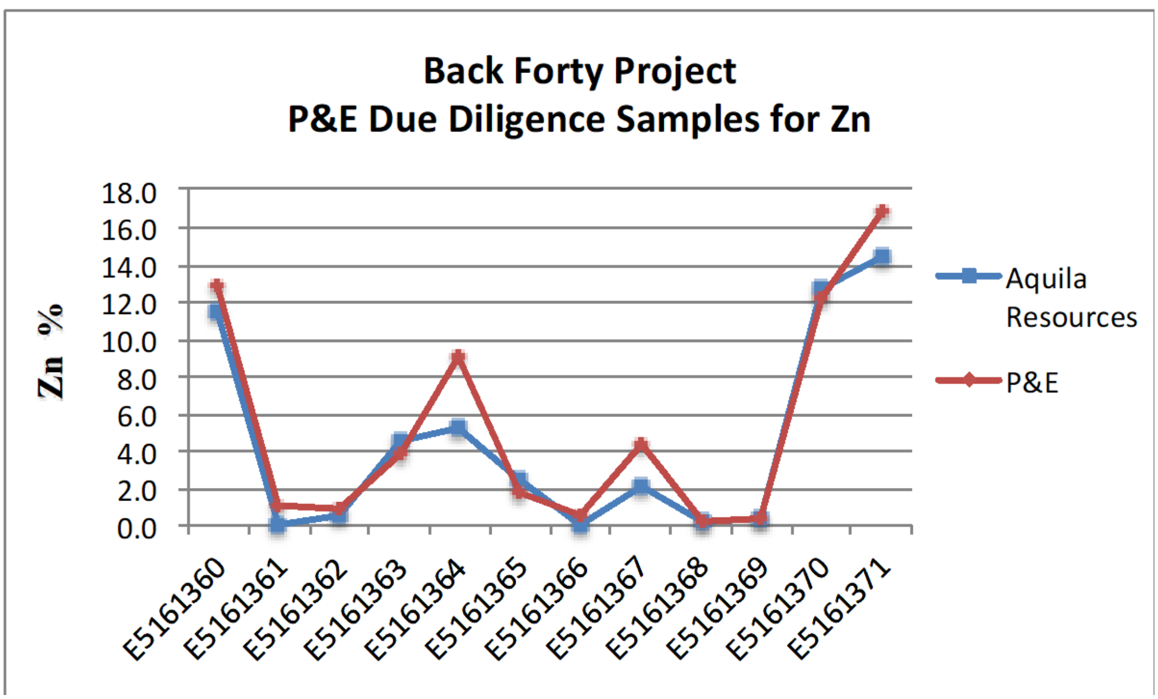


FIGURE 12.9 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR CU: NOVEMBER 2017

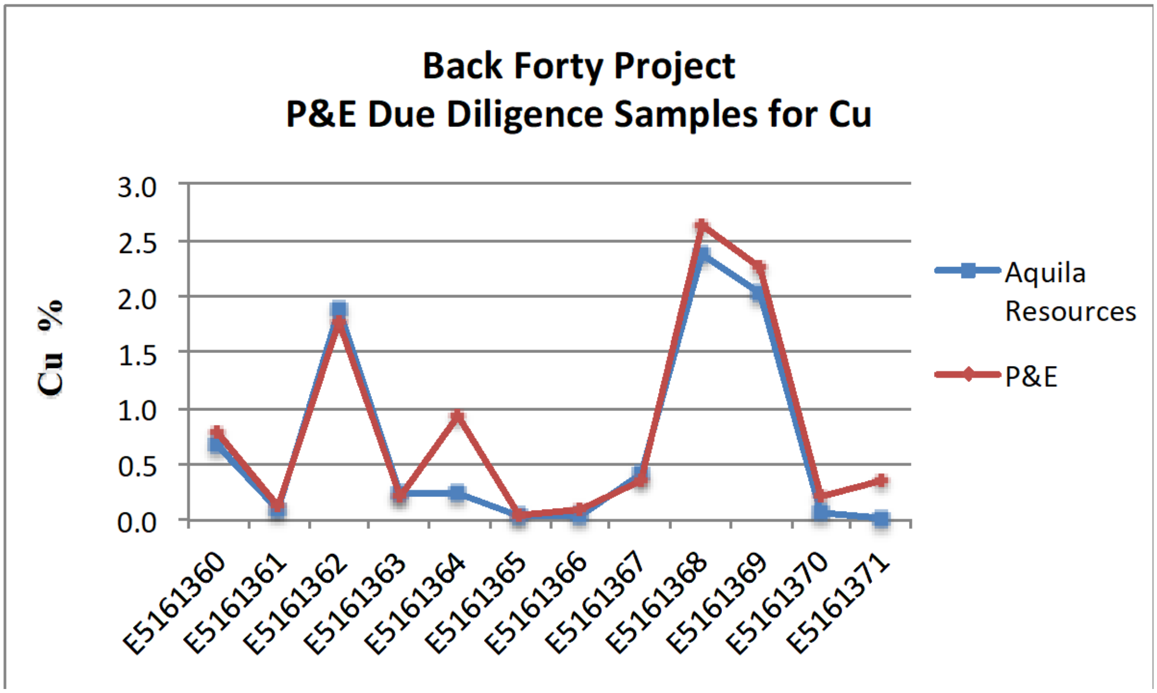
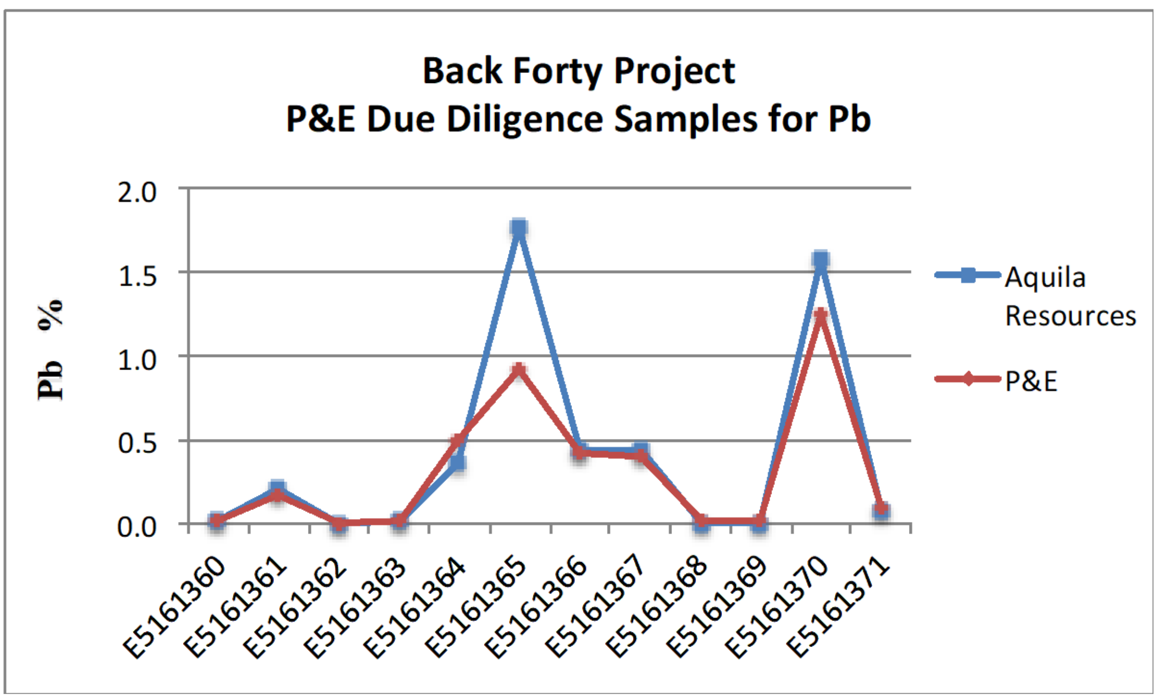


FIGURE 12.10 BACK FORTY PROJECT DUE DILIGENCE SAMPLE RESULTS FOR PB: NOVEMBER 2017



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The overall objective of the metallurgical testwork program was to quantify the metallurgical response of the Back Forty VMS. The testwork focused on three distinct sulphide mineralized material types (Main, Pinwheel and Tuff) and the oxide mineralized material of the deposit. The program was designed with the intent to develop the parameters for process design criteria for comminution, flotation, leaching, oxide tailings filtration, sulphidization-acidification-recycling-thickening ("SART") and cyanide destruction and tailings dewatering and rheology in the process plant.

The metallurgical testwork program was primarily conducted at SGS (Lakefield, Ontario) and dewatering and rheology work was conducted at Golder (Sudbury, Ontario). Filtration and SART testwork was carried out by Tenova and BQE, respectively (Vancouver, British Columbia).

The metallurgical testwork program was performed on the following composites and samples:
Grindability samples.

- Metallurgical domain composite master samples.
- Metallurgical variability composite samples.
- Individual core samples.

The samples are understood to have been selected to represent the spatial distribution, head grades and mineralization types of the Back Forty Deposit.

61 grindability samples were submitted to SGS to complete a suite of grinding characterization tests including BWI, ModBond and AI. In addition to these 61 samples, eight composites were selected from the PQ variability samples and compiled to complete CWI and SMC tests. Further to the aforementioned 69 samples, 24 additional samples were selected to complete SMC variability tests and complementary BWI and ModBond tests.

Several flotation and leaching studies and scoping comminution and gravity separation tests were performed from 2007 to 2012. This work was used to established metallurgical domains and direction for initial test conditions and to demonstrate variability throughout the Deposit.

Further investigative and optimization work was conducted from 2016 to 2018. This work was done on domain master composites, variability composites and individual core samples in an effort to optimize reagent consumption and flowsheet design, and to provide data which would be later used to generate recovery equations.

13.2 PREVIOUS TESTWORK

Several rounds of testing were undertaken at various laboratories prior to the most recent phase of study. A summary of the programs and results are described in the subsections below, as extracted from the original metallurgical testing reports from the individual laboratories, issued electronically.

Historical sample names and domains differ from the latest eight defined metallurgical types. However, historical composites usually fall under one of the current types. These latest eight metallurgical types are listed in Table 13.1.

TABLE 13.1 METALLURGICAL TYPES	
No.	**Name**
1	Main Zone Massive Sulphide
2	Pinwheel Massive Sulphide Cu Rich
3	Pinwheel Semi-Massive and Stringers
4	Pinwheel Extension
5	Tuff Zone
6	Oxides
7	Pinwheel Gossan Flotation
8	Pinwheel Massive Sulphide Cu-Zn Rich

13.2.1 G&T Metallurgical Services Ltd. 2007 (KM 1983)

In 2007, G&T Metallurgical Services Ltd. in Kamloops, B.C. (G&T) conducted a laboratory testwork program on seven composite samples reported as representing several of the domains from the sulphide mineralization of the Back Forty Deposit. The primary focus of the program was to scope metallurgical performance across several different metallurgical types. Mineralogy and rougher flotation tests were performed. Details and results of the program are summarized below.

13.2.1.1 Sample Composition

The head grades of the samples tested in the 2007 campaign are summarized in Table 13.2 by metallurgical ("Met") type.

TABLE 13.2 KM 1983 COMPOSITE HEAD GRADES										
Composite	**Met Type**	**Assays**								
		Cu (%)	**Pb (%)**	**Zn (%)**	**Ag (g/t)**	**Au (g/t)**	**As (g/t)**	**Sb (g/t)**	**Hg (g/t)**	**Cd (g/t)**
East Zone + Hinge	1	0.59	0.09	6.03	28	5.4	0.14	128	18	244
Pinwheel	2,3,8	4.04	0.05	0.13	86	1.9	0.14	128	45	14
South Limb	1	0.62	0.06	2.83	25	1.1	0.13	132	20	104
Tuff Zone	5	0.09	3.37	7.42	96	2.2	0.13	156	89	210
Gossan Pinwheel	6	0.80	0.09	0.03	88	16.8	0.084	178	51	4
Gossan East	6	0.04	0.09	0.02	13	19.5	0.12	174	3	4
Stringer Zone	1	0.27	0.10	0.69	13	2.7	0.11	60	6	28

13.2.1.2 Mineralogy

Mineralogical characterization was performed on all seven composites, at varying grind sizes, in an effort to scope required primary grind targets as well as calculate theoretical rougher flotation performance. Mineral composition and liberation results are summarized in Table 13.3 and Table 13.4.

TABLE 13.3 KM 1983 COMPOSITE MINERAL COMPOSITION					
Composite	**Composition (%)**				
	Chalco-pyrite	**Galena**	**Sphalerite**	**Pyrite**	**Non-sulphide Gangue**
East Zone + Hinge	1.9	<0.1	10.0	79.5	8.6
Pinwheel	11.5	<0.1	0.4	83.5	4.6
South Limb	1.8	<0.1	4.6	90.1	3.5
Tuff Zone	0.3	4.3	12.9	36.3	46.2
Gossan Pinwheel	2.1	0.1	0.1	0.9	96.8
Gossan East	0.1	0.1	<0.1	0.2	99.6
Stringer Zone	0.7	<0.1	1.1	39.6	58.6

TABLE 13.4
KM 1983 COMPOSITE MINERAL LIBERATION

Composite	K_{80} (µm)	Liberation (%)				
		Chalco-pyrite	Galena	Sphalerite	Pyrite	Non-sulphide Gangue
East Zone + Hinge	124	55	7	75	81	87
Pinwheel	147	13	41	35	51	53
South Limb	130	58	25	56	78	74
Tuff Zone	83	54	32	75	81	93
Gossan Pinwheel	85	39	<1	<1	36	96
Gossan East	103	56	<1	<1	76	99
Stringer Zone	102	65	<1	76	94	94

Based on the liberation data, it was concluded that chalcopyrite liberation was at levels adequate to achieve reasonable metallurgical response in rougher flotation for most of the composites. The Gossan Pinwheel and Pinwheel composites both exhibited lower chalcopyrite liberation and would require a finer primary grind.

Sphalerite liberation was generally suitable to provide adequate rougher flotation performance. Sphalerite liberation levels were low for the Pinwheel, Gossan Pinwheel, and Gossan East composites, and a finer primary grind would be required.

The Tuff Zone composite was the only sample containing appreciable levels of galena, and showed a low degree of galena liberation, even at a primary grind size (K_{80}) of 83 µm.

Size fractional analysis revealed a significant increase in liberation at finer particle sizes. This led to an initial primary grind size target P_{80} between 106 and 141 µm.

13.2.1.3 Rougher Flotation

All seven composites were subjected to scoping level rougher flotation tests. The flowsheet consisted of a primary grind, a 2-stage bulk rougher and a 3-stage zinc rougher on the bulk rougher tailings. Results are summarized in Table 13.5.

Test conditions were based on typical conditions for other similar mineralization. The bulk rougher was run at natural pH with only a collector (3418A) and a frother. The zinc rougher was run at an elevated pH with an activator ($CuSO_4$), a collector (SIPX) and a frother. The frother used in both cases was methyl isobutyl carbinol (“MIBC”).

Based on the results, both copper and zinc data indicated that the primary grind did not liberate enough of the mineralization. It was recommended that the primary grind size be among the test conditions to be more diligently optimized in future work.

TABLE 13.5 KM 1983 ROUGHER FLOTATION RESULTS SUMMARY										
Composite	Product	P_{80} (µm)	Assays				Distribution			
			Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Au (%)
East Zone + Hinge	Feed	124	0.6	0.1	6.0	4.4	100	100	100	100
	Bulk Ro Conc	-	2.2	0.1	2.6	11.5	79	38	9	54
	Zinc Ro Conc	-	0.3	0.1	44.2	4.0	6	13	86	11
Pinwheel	Feed	116	4.2	0.0	0.2	1.5	100	100	100	100
	Bulk Ro Conc	-	6.7	0.0	0.2	0.4	44	32	30	8
	Zinc Ro Conc	-	5.2	0.0	0.2	2.1	17	13	12	19
South Limb	Feed	130	0.6	0.0	3.0	1.2	100	100	100	100
	Bulk Ro Conc	-	3.6	0.0	1.7	3.3	71	16	7	34
	Zinc Ro Conc	-	0.5	0.0	12.0	1.3	8	11	44	12
Tuff Zone	Feed	111	0.1	3.7	8.7	1.1	100	100	100	100
	Bulk Ro Conc	-	0.3	10.6	12.9	3.0	71	94	48	85
	Zinc Ro Conc	-	0.1	0.6	15.2	0.4	15	4	51	11
Gossan Pinwheel	Feed	106	0.9	0.1	0.0	17.0	100	100	100	100
	Bulk Ro Conc	-	9.9	0.2	0.1	100	85	18	31	43
Gossan East	Feed	145	0.0	0.1	0.0	20.9	100	100	100	100
	Bulk Ro Conc	-	0.4	0.2	0.2	244	54	9	25	45
Stringer Zone	Feed	141	0.3	0.1	0.7	2.6	100	100	100	100
	Bulk Ro Conc	-	0.7	0.1	0.9	7.0	91	56	39	89
	Zinc Ro Conc	-	0.1	0.1	2.5	1.4	5	15	59	9

Note: *Ro Conc = Rougher Flotation Concentrate.*

13.2.2 G&T Metallurgical Services Ltd. 2008 (KM 2047)

In 2008, G&T conducted a more comprehensive testwork program on seventeen composite samples from the Back Forty Deposit. The samples originated from both the sulphide and oxide portions of the mineralization. The focus of this campaign was to develop further defined process conditions and develop the process flowsheet, as well as to demonstrate expected precious and base metal recoveries. This study also laid the foundations and guidance for future testwork. Details and results of the program are summarized below.

13.2.2.1 Sample Composition

The head grades of the samples tested in the 2008 campaign are summarized in Table 13.6.

TABLE 13.6 KM 2047 SULPHIDE AND OXIDE COMPOSITE HEAD GRADES								
Composite	**Met Type**	**Assays**						
		Cu (%)	**Pb (%)**	**Zn (%)**	**Fe (%)**	**Ag (g/t)**	**Au (g/t)**
90 Zone	6	0.04	0.11	0.02	2.2	114	12.0
Deep Zone	1	0.58	0.05	1.00	42.2	15	1.43
East Gossan	6	0.04	0.06	0.02	29.1	7	44.0
East Hinge Massive	1	0.32	0.07	4.29	40.0	19	2.68
East Zone and Hinge Low Zn	1	0.53	0.06	0.28	39.1	17	3.95
East Zone and Hinge Zn Rich	1	0.22	0.13	6.25	38.5	13	1.92
Hinge East Stringer	1	0.34	0.05	3.06	33.1	14	2.45
Main and South Zn Rich	1	0.10	0.12	7.55	38.0	14	1.2
Main Zone	1	0.31	0.06	3.33	31.7	15	1.97
Pinwheel Extension	4	0.60	0.35	4.42	38.7	18	0.95
Pinwheel Gossan	6	0.45	0.07	0.01	31.9	56	9.25
Pinwheel Sulphide Cu Rich	2,3	1.78	0.03	0.38	39.6	32	1.71
Porphyry Margin	6	0.06	0.80	0.22	4.6	68	4.54
South Limb	1	0.21	0.09	4.22	35.4	16	1.50
South Limb Pyritic	1	0.40	0.07	0.29	31.1	14	1.79
Stringer Zone Composite 2	1	0.38	0.06	0.08	20.4	10	2.92
Tuff Zone	5	0.05	2.16	9.97	22.1	87	1.60

13.2.2.2 Hardness

Two samples of the 17 from this program were submitted for Bond ball mill work index (“BWI”) evaluation. The samples were East Hinge Massive (9.6 kWh/t) and South Limb Pyritic (10.7 kWh/t). Both tests were conducted using a sieve closing aperture size of 150 mesh (106 µm). This is considered relatively soft material. Comparing laboratory grinding times to these samples indicated that the other 15 samples were also soft to moderately soft.

13.2.2.3 Mineralogy

Mineralogy was performed on nine of the 17 composites, at varying grind sizes, in an effort to scope required primary grind targets as well as estimate theoretical rougher flotation performance. Mineral composition and liberation results are summarized in Table 13.7 and Table 13.8.

TABLE 13.7
KM 2047 COMPOSITE MINERAL COMPOSITION

Composite	Composition (%)				
	Copper Sulphide	Galena	Sphalerite	Pyrite	Non-Sulphide Gangue
Deep Zone	1.78	0.05	1.59	88.6	8
East Zone and Hinge Low Zn	1.59	0.04	0.53	79.8	18
East Zone and Hinge Zn Rich	0.67	0.20	13.1	79.6	7
Main and South Zn Rich	0.30	0.14	11.7	75.2	13
Pinwheel Extension	1.63	0.43	6.70	78.5	13
Pinwheel Gossan	1.42	0.12	0.04	0.8	98
Pinwheel Sulphide Cu Rich	5.40	0.02	0.62	83.8	10
South Limb Pyritic	1.65	0.10	0.47	66.5	32
Stringer Zone Composite 2	1.23	0.03	0.13	46.0	53

TABLE 13.8
KM 2047 SULPHIDE COMPOSITE MINERAL LIBERATION

Composite	K_{80} (µm)	Liberation (%)				
		Copper Sulphide	Galena	Sphalerite	Pyrite	Non-Sulphide Gangue
Deep Zone	84	66	43	74	94	93
East Zone and Hinge Low Zn	119	63	56	59	91	91
East Zone and Hinge Zn Rich	123	58	66	76	82	96
Main and South Zn Rich	118	52	60	60	77	88
Pinwheel Extension	76	49	51	71	87	86
Pinwheel Gossan	66	68	42	56	69	99
Pinwheel Sulphide Cu Rich	128	32	35	42	81	88
South Limb Pyritic	76	59	49	56	90	95
Stringer Zone Composite 2	105	55	70	53	90	97

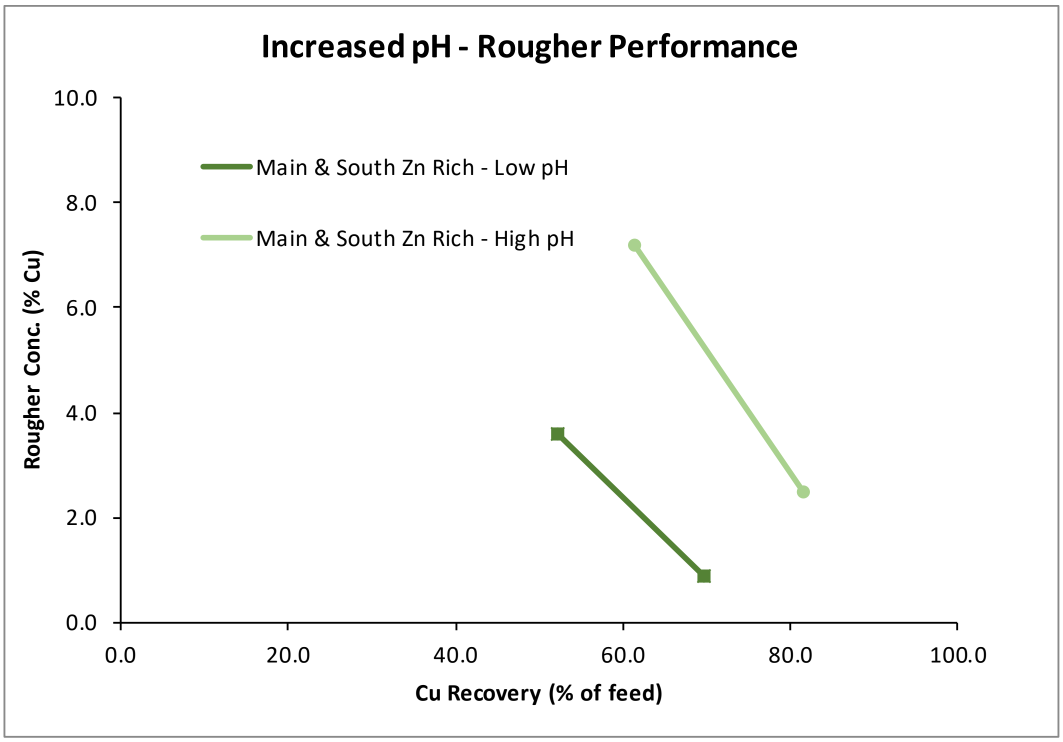
Copper sulphide liberation ranged from approximately 32% to 68%. In general, similar deposits would require at least 45% liberation to ensure positive rougher flotation performance. Back Forty is a more complex massive sulphide and it is expected that a higher level of liberation would be required. A fractional size analysis suggested a finer primary grind than was targeted in the 2007 study. The target primary grind size for rougher flotation was determined to be a P_{80} of 75 µm.

13.2.2.4 Rougher Flotation

A series of rougher flotation tests was conducted on eight of the 17 samples during the KM 2047 test program. Bulk rougher pH, collector dosage (3418A), depressant dosage ($ZnSO_4$, NaCN) and primary grind size were the key flotation conditions that were explored. Generally, optimization tests were first performed on samples deemed to be part of the Main Zone. Once condition trends were established, the two Pinwheel samples and the Tuff Zone samples were tested.

Initial tests were conducted at natural pH in the bulk roughers, but significant or typical upgrading was not achieved. It was found for the East Zone and Hinge Low Zn, East Zone and Hinge Zn Rich and Main and South Zn Rich samples that increasing the bulk rougher pH from natural to 10.0 shifted the rougher grade versus recovery curve significantly. To illustrate this trend, the Main and South Zn Rich concentrate grade versus recovery data with respect to copper at both high and low pH is displayed in Figure 13.1.

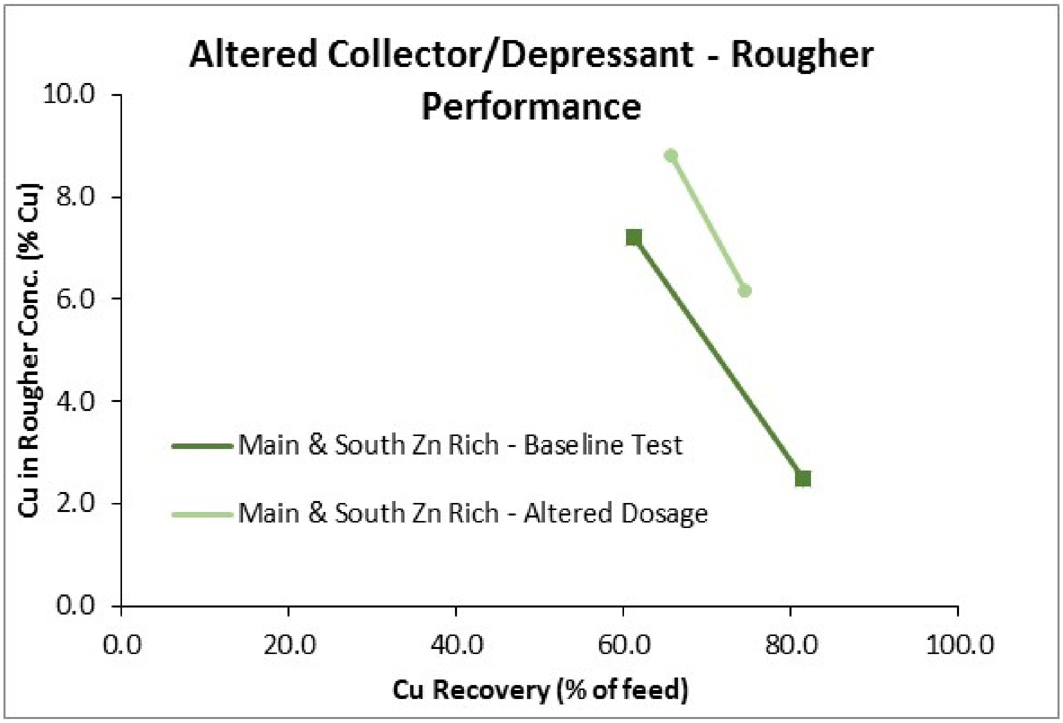
FIGURE 13.1 KM 2047 ROUGHER PERFORMANCE – VARYING PH



Source: *Aquila Resources Inc. (2020)*

Based on results from initial testing, four samples were subjected to additional rougher flotation tests: East Zone and Hinge Low Zn, East Zone and Hinge Zn Rich, Main and South Zn Rich and South Limb Pyritic. The objective was to optimize collector and depressant dosages. In general, collector dosage was decreased and depressant dosage was increased. The trend across the aforementioned samples was a positive shift in the rougher grade versus recovery curves. To illustrate this trend, the Main and South Zn Rich concentrate grade versus recovery data is displayed in Figure 13.2 with respect to copper.

FIGURE 13.2 KM 2047 ROUGHER PERFORMANCE – VARYING COLLECTOR AND DEPRESSANT DOSAGE



Source: *Aquila Resources Inc. (2020)*

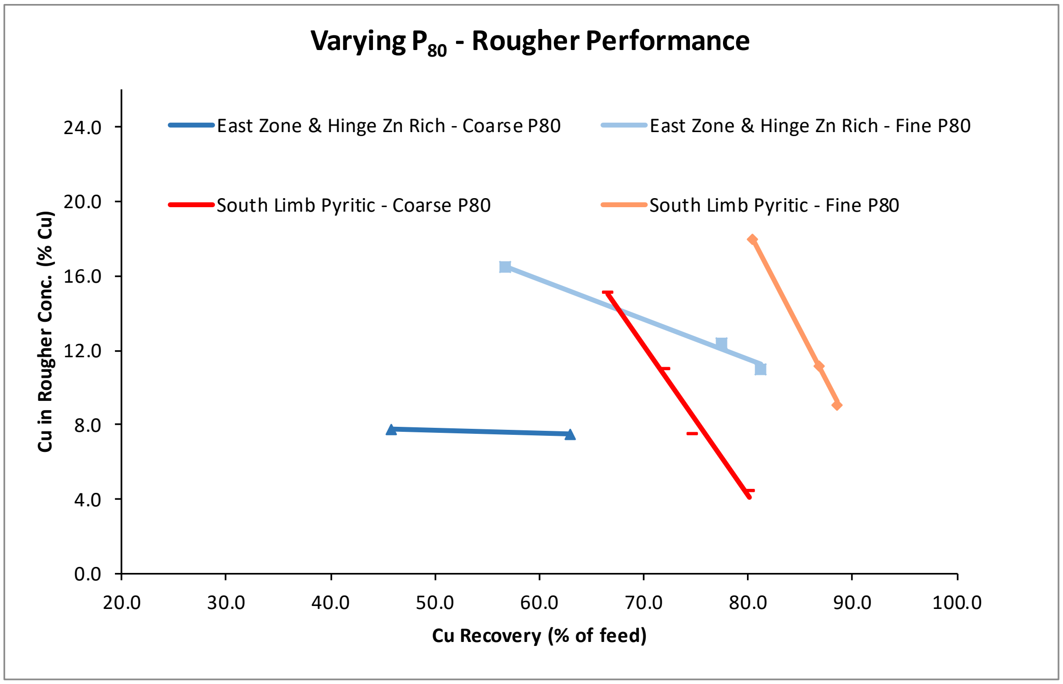
The next condition scoped in the rougher flotation stage of the KM 2047 test program was primary grind size. Seven of the eight samples were subjected to rougher flotation at both relatively coarse and fine grind sizes. As mineralogy would suggest, a significant and distinct shift in the rougher grade versus recovery curves for all seven samples was observed. To illustrate this trend, the East Zone and Hinge Zn Rich and South Limb Pyritic concentrate grade versus recovery data at both coarse and fine grind sizes is displayed in Figure 13.3 with respect to copper.

Rougher flotation testing from the KM 2047 program provided guidance, in general terms, of bulk rougher pH and primary grind size requirements. Gold recovery to the bulk rougher concentrate was also positively impacted by reduced primary grind size. Additionally, the rougher testing provided guidance on an individual sample basis on the balance between collector and depressant reagent dosages.

Where applicable, a zinc rougher flotation stage was also performed on the bulk rougher circuit tailings. Conditions for the zinc rougher flotation were those found typically in the industry. The primary grind was dictated by the bulk circuit, the collector used was a xanthate (SIPX) and the pH was 12.0. Positive upgrading and recoveries were achieved, although these are somewhat difficult to quantify as some zinc was lost to the bulk circuit under open test conditions (additional focus on zinc rejection in the bulk cleaner circuit would provide a better indication of zinc performance).

The Tuff Zone sample was subjected to only rougher flotation testing. The test was run at a finer grind size (P_{80} of 60 µm) and the bulk rougher was done at 7.5-8.0 pH. Since only one test was completed, collector and depressant dosages were not evaluated. The bulk (Pb) concentrate achieved a grade of 30% Pb at almost 90% lead recovery, while zinc concentrate was upgraded to over 46% Zn at almost 90% zinc recovery.

FIGURE 13.3 KM 2047 ROUGHER PERFORMANCE – VARYING PRIMARY GRIND SIZE



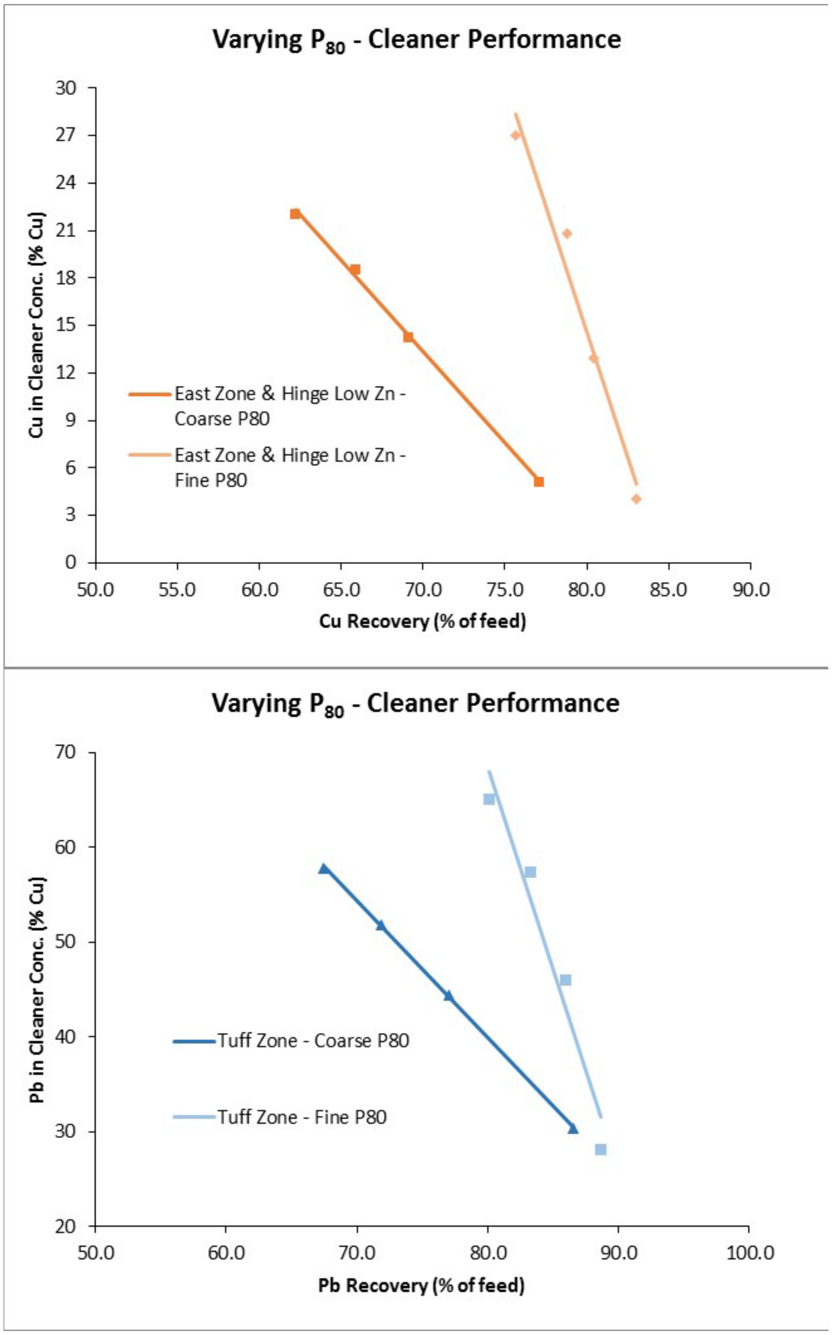
Source: *Aquila Resources Inc. (2020)*

13.2.2.5 Cleaner Flotation

A series of cleaner flotation tests was conducted on fourteen of the seventeen samples during the KM 2047 test program. The focus for the Main Zone samples was to optimize regrind P_{80}, cleaner collector dosage and cleaner depressant dosage. Pinwheel samples shared a similar objective; however, some results indicated a need to further optimize rougher metallurgy. The Tuff Zone sample was subjected to multiple cleaner flotation tests, optimizing both rougher and cleaner stage conditions.

Generally, the first condition to be explored in the cleaner flotation stage of the KM 2047 test program was regrind size. All fourteen samples were subjected to cleaner flotation at both relatively coarse and relatively fine bulk concentrate regrind sizes, with approximate P_{80} values of 15 to 50 µm. A significant and distinct shift in the cleaner grade versus recovery curves for all samples was observed for the bulk circuit. To illustrate this trend, the East Zone and Hinge Low Zn and Tuff Zone concentrate grade versus recovery data at both coarse and fine grind sizes is displayed in Figure 13.4. The zinc regrind P_{80} did not affect cleaner performance as dramatically.

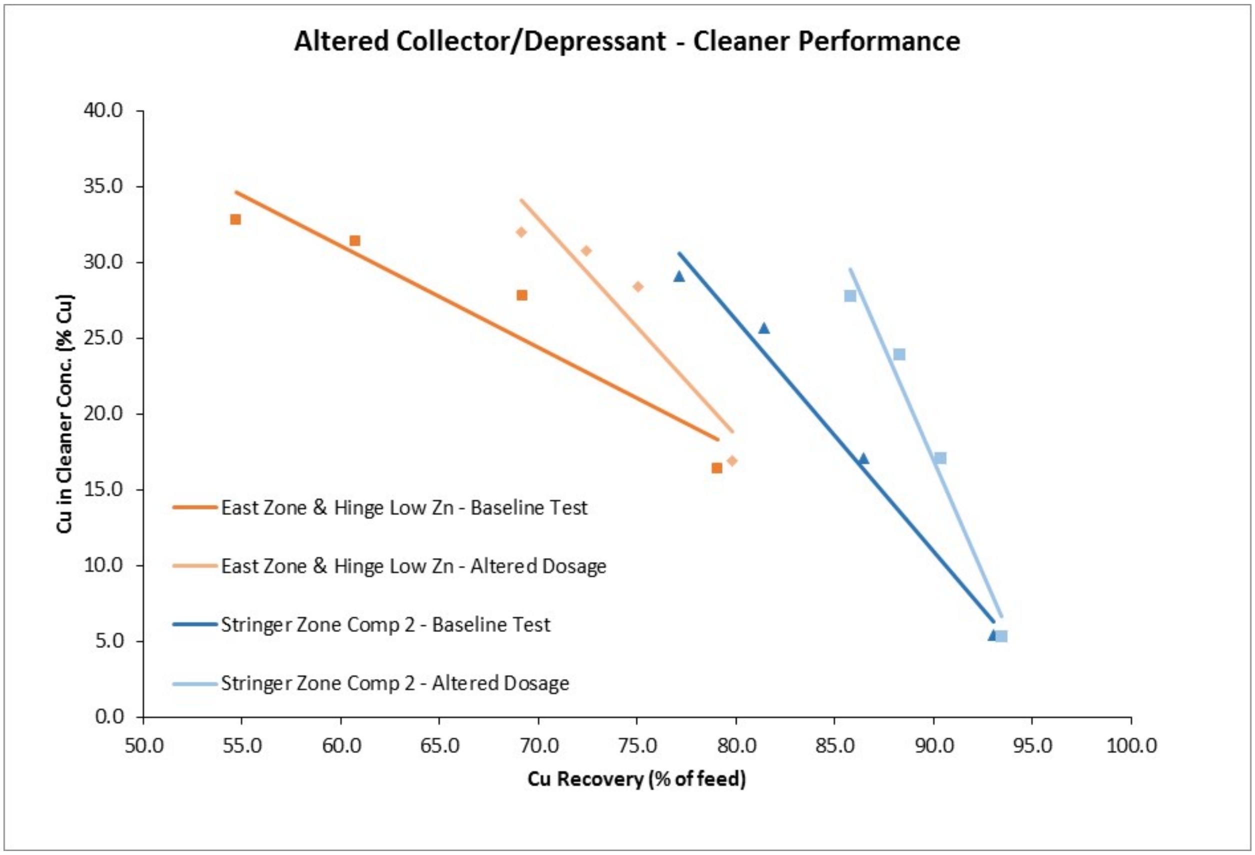
FIGURE 13.4 KM 2047 CLEANER PERFORMANCE – VARYING REGRIND SIZE



Source: *Aquila Resources Inc. (2020)*

Based on results from initial testing, several samples were subjected to additional cleaner flotation tests. The objective was to optimize collector and depressant dosages. In general, collector dosages were altered in the cleaner stages. There was a positive shift in the cleaner grade versus recovery curves. To illustrate this trend, the East Zone and Hinge Low Zn and Stringer Zone Comp 2 concentrate grade versus recovery data with respect to copper is displayed in Figure 13.5.

FIGURE 13.5 KM 2047 CLEANER PERFORMANCE – VARYING COLLECTOR AND DEPRESSANT DOSAGE

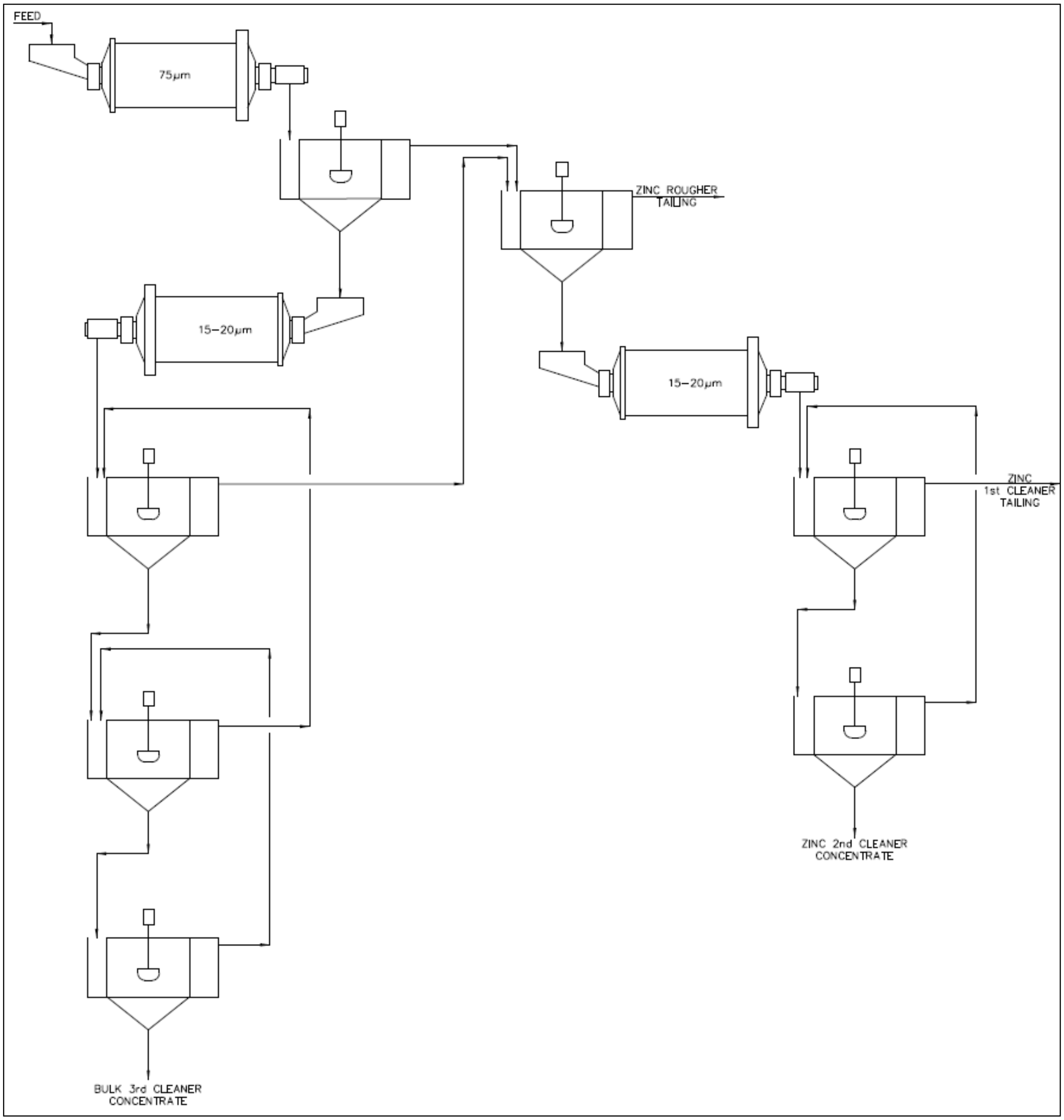


Source: *Aquila Resources Inc. (2020)*

13.2.2.6 Locked Cycle Flotation

Locked cycle flotation testing was conducted on 13 of the 17 composites. The other four samples were considered to be from the oxide portion of the Deposit and were not considered amenable to flotation. The basis for the test conditions was previous rougher and cleaner tests throughout the program. The testwork flowsheet for the locked cycle tests is illustrated in Figure 13.6. Primary and regrind sizes are quoted at the 80% passing size.

FIGURE 13.6 KM 2047 FLOWSHEET – LOCKED CYCLE TESTS



Source: *Aquila Resources Inc. (2020)*

Six of the 13 samples were subjected to one locked cycle test: South Limb, East Hinge Massive (2 tests, minor dosage change), East Hinge Stringer, Main Zone, Pinwheel Sulphide Cu Rich and Pinwheel Extension. The copper concentrate grade of the Pinwheel samples averaged over 18% Cu with 79% copper recovery. Only the Pinwheel Extension sample attempted to recover a zinc concentrate, and the result was the production of a zinc concentrate grading over 56% Zn with 88% zinc recovery. An additional four samples, all of which ultimately fell into metallurgical type 1, were subjected to locked cycle test(s) and performed even more favourably. On average, copper concentrate grades reached over 28% Cu with over 80% copper recovery, while zinc concentrate grades were over 54% Zn with over 96% zinc recovery.

Each of the remaining three samples was subjected to 2 to 3 locked cycle tests – slightly tweaking reagent dosage where necessary. All samples where the copper head grade was greater than 0.25% achieved bulk concentrate grades higher than 25% Cu with recoveries ranging from 75% to 88% copper. All samples where the zinc head grade was greater than 0.30% Zn achieved zinc concentrate grades higher than 50% Zn with recoveries ranging from 72% to 95% zinc.

The Tuff Zone sample, which was the only lead dominated sample, also performed well under locked cycle conditions. While this sample certainly contained much higher than expected head grades for both lead and zinc (2.2% Pb and 11.1% Zn), results showed potential for this part of the Back Forty Deposit. A lead concentrate grade of 65% Pb was achieved at 86% lead recovery and a zinc concentrate grade of 54% Zn was achieved at 93% zinc recovery.

13.2.2.7 Oxide Whole Material Cyanidation

Duplicate whole material cyanidation tests were carried out on the four oxide composites. The results of the tests are displayed in Table 13.9. These tests were carried out over 48 hours with intermediate sampling at 6 and 24 hours.

TABLE 13.9
KM 2047 OXIDE CYANIDATION RESULTS

Composite	K_{80} (µm)	Reag. Cons. (kg/t)		Extraction (%)		Calc. Head (g/t)	
		NaCN	CaO	Au	Ag	Au	Ag
90 Zone	87	0.5	0.5	95.1	30.7	11.3	107
90 Zone	87	0.3	1.1	93.8	29.7	10.2	101
East Gossan	71	1.0	1.8	93.5	57.3	33.4	5
East Gossan	71	0.5	2.5	95.4	34.6	25.9	5
Pinwheel Gossan	66	1.1	1.3	88.3	40.4	10.9	58
Pinwheel Gossan	75	1.8	3.7	91.8	38.8	11.9	47
Porphyry Margin	-	0.5	0.7	94.2	32.4	4.6	68
Porphyry Margin	-	1.0	0.8	93.6	34.1	5.2	74

Note: *Reag. Cons. = reagent consumption, Calc. Head = calculated head grade.*

The average 48-hour gold extraction across all eight tests was 93%. The Pinwheel Gossan composite produced slightly lower 48-hour gold extraction, at roughly 90%. This compares to an average 48-hour gold extraction of 94% for the other three composites.

Sodium cyanide consumption ranged from 0.3 to 1.8 kg/t. The highest consumption was observed for the Pinwheel Gossan composite, which contained elevated copper content. Further optimization work would be required.

13.2.2.8 Sulphide Zinc Rougher Tailings Cyanidation

A number of flotation test tailings streams were also subjected to cyanidation bottle roll testing. The results of these tests are summarized in Table 13.10.

TABLE 13.10
KM 2047 SULPHIDE TAILINGS CYANIDATION RESULTS

Composite	Reag. Cons. (kg/t)		Extraction (%)		Residue (g/t)		Head Grade (g/t)	
	NaCN	CaO	Au	Ag	Au	Ag	Au	Ag
East Hinge Massive	1.2	1.1	29.5	50.2	0.8	3	1.1	6
East Hinge Stringer	1.0	2.0	40.6	50.6	0.4	3	0.7	0.6
East Zone and Hinge Low Zn	1.7	1.6	30.3	75.6	1.0	2	1.4	9
East Zone and Hinge Zn Rich	0.7	0.9	41.3	29.3	0.5	6	0.7	7
Main and South Zn Rich	1.5	1.3	3.7	43.6	0.6	3	0.6	5
Main Zone	0.9	1.6	25.8	40.5	0.4	3	0.6	5
Pinwheel Sul Cu Rich	3.5	7.2	38.5	42.6	0.4	6	0.6	11
South Limb	1.1	1.0	27.8	36.6	0.5	4	0.7	6
Stringer Zone Comp 2	0.6	1.6	27.8	55.6	0.8	1	1.1	3
Tuff Zone	1.3	1.2	32.6	44.4	0.2	5	0.3	9

Note: *Reag. Cons. = reagent consumption, Calc. Head = calculated head grade.*

Cyanidation gold extraction for the zinc rougher tailings, on average, was almost 30%. The average calculated head grades were 0.8% Au.

13.2.2.9 Gravity Concentration Tests

A series of gravity concentration tests was carried out on the four high-grade gold (oxide) composites. A 2 kg sample of the feed was ground to a P_{80} of between 70-100 µm prior to being processed in a 3-inch laboratory scale Knelson concentrator. The concentrate from the Knelson concentrator was then hand panned to upgrade the gold content in the final gravity concentrate. The results of the gravity tests are summarized in Table 13.11.

TABLE 13.11
KM 2047 GRAVITY CONCENTRATION RESULTS

Composite	Gravity Product	Mass % of Feed	Assays		Recovery	
			Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
90 Zone	Pan Conc	0.5	798	5,220	33	19
90 Zone	Kn Conc	3.4	148	1,422	45	38
East Gossan	Pan Conc	0.2	3,593	480	24	11
East Gossan	Kn Conc	2.6	423	69	33	19

Table 13.11 KM 2047 Gravity Concentration Results						
Composite	**Gravity Product**	**Mass % of Feed**	**Assays**		**Recovery**	
			Au (g/t)	**Ag (g/t)**	**Au (%)**	**Ag (%)**
Pinwheel Gossan	Pan Conc	2.4	247	1,520	38	41
Pinwheel Gossan	Kn Conc	8.7	87	514	47	49
Porphyry Margin	Pan Conc	0.8	166	2,100	19	21
Porphyry Margin	Kn Conc	3.8	73	677	41	32

The average gold recovery to the Knelson concentrate was about 42%. On average, approximately 5% of the feed mass was recovered as Knelson concentrate with an average grade of 182 g/t Au. The pan concentrate contained, on average, approximately 29% of the feed gold units. On average, about 1% of the feed mass was recovered into the pan concentrate which contained an average of roughly 1,200 g/t Au and 2,300 g/t Ag.

13.2.2.10 Minor Element Assays

Minor element determinations were carried out on the locked cycle flotation test concentrates from several of the composites. The mercury content in the zinc concentrate, on average for the blended samples, was 320 g/t Hg. The average mercury content in the copper concentrates of the blended samples was 43 g/t Hg. Cadmium levels were also elevated in the bulk concentrate, at 150 g/t Cd. Combined arsenic plus antimony concentrations in the bulk concentrate were as high as about 2,800 g/t for the South Limb blend sample.

13.2.3 SGS Canada Inc. 2009

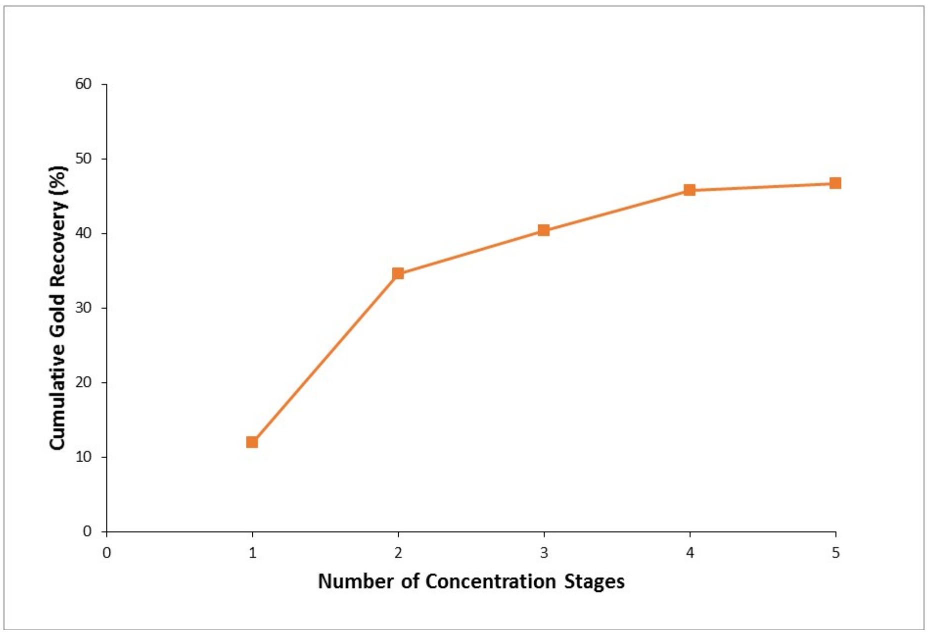
In 2009, one sample (East Gossan, met type 6) was sent to SGS Lakefield to explore the sample's metallurgical response to gravity separation, via a laboratory-scale Knelson concentrator, as well as to cyanide leach of gravity separation middlings and tailings. The head sample assays were calculated from test product assays; gold and silver assays were calculated to be 36.2 g/t Au and 6.1 g/t Ag, respectively.

13.2.3.1 Gravity Separation

The entire sample, at 100% minus 10 mesh, was pulped with water and passed through a Knelson concentrator, from which a Knelson concentrate and a Knelson tailings were collected. The Knelson concentrate was ground in a laboratory rod mill to a P_{80} of 211 µm. The ground Knelson concentrate was upgraded on a Mozley shaking table, from which a concentrate, middlings and tailings products were collected. The Mozley products were dried and samples riffled out for gold and silver assay. A sample of the Mozley middlings was also submitted for a multi-element ICP scan.

The first stage Knelson tailings was ground in a laboratory rod mill to a P_{80} of approximately 75 µm. The mill discharge was passed through the Knelson concentrator. The Knelson concentrate was further upgraded on the Mozley table, collecting a Mozley concentrate, middlings and tailings. The Mozley products were assayed as in the first pass. The Knelson tailings from the second pass was re-passed through the Knelson concentrator and the Knelson concentrate was treated as above. The third pass Knelson tailings was ground in a laboratory rod mill to a P_{80} of 27 µm. The mill discharge was passed through the Knelson concentrator and the Knelson concentrate upgraded on the Mozley shaking table as in the previous stages. The fourth pass Knelson tailings was re-passed through the Knelson concentrator, and the Knelson concentrate was upgraded on the Mozley table as in the previous stages. Figure 13.7 illustrates the cumulative gold recovery by stage of gravity concentration. Cumulative gold recovery after five passes was ~47%, at a concentrate grade of ~20,000 g/t Au.

FIGURE 13.7 SGS 2009 EAST GOSSAN – GOLD RECOVERY BY GRAVITY CONCENTRATION STAGE



Source: *Aquila Resources Inc. (2020)*

13.2.3.2 Cyanide Leach

A proportionally weighted sample of the Mozley middlings and tailings and final Knelson tailings was prepared for a bottle roll test. The sample was leached for 48 hours with 0.5 g/L NaCN at pH 10.5 – 11.0. At the end of the 48 hours the sample was filtered, collecting a pregnant leach solution and a residue. The residue was washed several times with water. The solutions and residue were submitted for gold and silver analysis. Results are displayed in Table 13.12.

TABLE 13.12 SGS 2009 EAST GOSSAN – CYANIDE LEACH RESULTS								
Composite	**Reag. Cons. (kg/t)**		**Extraction (%)**		**Residue (g/t)**		**Head Grade (g/t)**	
	NaCN	**CaO**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
East Gossan	0.17	1.83	92.1	49.9	1.63	2.4	20.6	4.8

Note: *Reag. Cons. = reagent consumption, Calc. Head = calculated head grade.*

13.2.3.3 Overall Recovery

The overall recoveries of gold and silver by gravity separation followed by cyanidation of the combined middlings and tailings were 95.8% and 59.4%, respectively. Refer to Table 13.13.

TABLE 13.13 SGS 2009 EAST GOSSAN – OVERALL RECOVERIES						
Composite	**Gravity Conc.**		**CN Leach**		**Overall Recovery (%)**	
	Distribution (%)		**Extraction (%)**		**Gravity + Cyanide**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**
East Gossan	46.6	18.9	92.1	49.9	95.8	59.4

Note: *Conc. = concentration, CN = cyanide.*

13.2.4 G&T Metallurgical Services Ltd. 2010 (KM 2575)

In 2010, G&T conducted a testwork program on 10 composite samples. Seven of the samples were distinctly sulphides, two samples were transitional although largely oxide and the remaining sample was an oxide. The purpose of this program was to generate tailings for environmental testing. Samples were subjected to flotation, cyanidation and gravity separation. The chemical compositions of the samples are displayed in Table 13.14.

TABLE 13.14 KM 2575 COMPOSITE HEAD GRADES								
Composite	**Met Type**	**Assays**						
		Cu (%)	**Pb (%)**	**Zn (%)**	**Fe (%)**	**S (%)**	**Ag (g/t**	**Au (g/t)**
Main Zone 111	1	0.36	0.69	6.90	37.7	40.2	13	2.85
Main Zone 4% Zn	1	0.87	0.10	3.02	40.7	41.7	18	2.08
Main Zone 6.5% Zn	1	0.26	0.69	6.90	37.7	40.2	72	3.44
Main Zone 9% Zn	1	0.15	0.24	10.4	37.9	40.2	15	1.27
Tuff Zone	5	0.04	0.80	3.31	5.3	7.1	27	1.06
Pinwheel Zone Composite 1	8	1.79	0.03	0.76	42.6	47.3	23	1.75
Pinwheel Zone Hi-Zn	8	0.35	0.93	9.05	37.7	44.4	29	0.88
Pinwheel Gossan Comp 1	6/7	0.55	0.06	0.04	32.0	2.4	88	5.0

TABLE 13.14 KM 2575 COMPOSITE HEAD GRADES								
Composite	Met Type	Assays						
		Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Ag (g/t	Au (g/t)
Pinwheel Gossan Comp 2	6/7	1.49	0.07	0.02	31.1	1.4	47	9.4
NS Zone Composite 1	6	0.04	0.08	0.01	1.3	0.2	39	5.2

13.2.4.1 Cleaner Flotation

Three cleaner flotation tests were conducted on Main Zone samples. One test was conducted on the Main Zone 111 sample and two tests were conducted on a blend of Main Zone 6.5% Zn (75%) and Main Zone 9% Zn (25%). In addition to base metal recovery, the zinc rougher tailings were subjected to pyrite flotation.

Since the purpose of these tests was to generate tailings for environmental testing, no optimization took place. Zinc circuit performance was excellent in the three tests. For the two tests on the blend sample, the copper circuit had difficulty upgrading, likely due to the relatively low head grade (<0.3% Cu).

13.2.4.2 Locked Cycle Flotation

Nine locked cycle flotation tests were conducted, one on each of nine composites (NS Zone composite 1 was not tested because it was an oxide sample). For all the Main Zone composites (x4), the Tuff Zone composite (x1) and two of the Pinwheel composites (high sulphur content), the purpose of the unoptimized tests was the generation of tailings for environmental testing.

For the seven aforementioned composites, some of the copper concentrates did not achieve optimal performance from either an upgrading or a recovery perspective. However, there were indications that improvement could be achieved. On the other hand, zinc and lead circuits performed favourably. The only lead composite (Tuff Zone) achieved a lead 3rd cleaner concentrate grade of 53% Pb at 76% lead recovery. All seven zinc concentrates had greater than 48% Zn grade and all but two had a better than 90% zinc recovery. The single anomaly was the zinc recovery of the Pinwheel Zone Comp 1, at less than 40%.

The other two composites, Pinwheel Gossan Comps 1 and 2, were subjected to a slightly different flowsheet. Both composites did not contain zinc so that part of the test was not completed. At the time of this testing program, it was believed (or at least considered) that transitional material like these composites (i.e., oxide to sulphide) would be processed through both the flotation and leach circuits. The purpose of these two locked cycle tests was to evaluate copper recoveries to flotation but also to generate tailings for more detailed cyanidation bottle roll tests. Despite lower sulphide content, the bulk flotation circuit performed favourably. Both tests produced marketable concentrates (20+% Cu) and both recovered over 76% of the copper in the feed. Results of the locked cycle testing are summarized in Table 13.15.

TABLE 13.15
KM 2575 LOCKED CYCLE FLOTATION TEST RESULTS

Composite	Met Type	Product	Assays					Distribution				
			Cu (%)	Pb (%)	Zn (%)	Au (g/t	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Au (%)	Ag (%)
Main 111	1	Bulk Conc.	31.4	0.12	1.12	167	451	65.1	1.8	0.2	50.1	36.1
		Zinc Conc.	1.16	0.08	55.4	5	33	17.7	9.2	89.4	11.1	19.7
		Feed	0.41	0.06	3.93	2.88	11	100.0	100.0	100.0	100.0	100.0
Main 4% Zn	1	Bulk Conc.	30.9	0.13	0.88	40	305	75.5	21.8	0.6	47.7	39.6
		Zinc Conc.	2.09	0.03	52.9	2	49	9.9	11.0	74.2	5.5	12.5
		Feed	1.05	0.02	3.55	2.12	20	100.0	100.0	100.0	100.0	100.0
Main 6.5% Zn	1	Bulk Conc.	12.8	27.1	7.42	21	3,657	70.5	71.1	1.7	22.3	77.2
		Zinc Conc.	0.29	0.42	55.8	2	51	11.4	7.8	93.4	15.5	7.6
		Feed	0.32	0.68	7.53	1.66	84	100.0	100.0	100.0	100.0	100.0
Main 9% Zn	1	Bulk Conc.	9.2	14.9	15.6	68	606	20.0	41.0	0.7	77.3	47.9
		Zinc Conc.	0.31	0.20	56.7	0	17	22.9	18.5	93.8	18.1	47.5
		Feed	0.25	0.20	11.4	2.13	31	100.0	100.0	100.0	100.0	100.0
Tuff Zone Comp 2	5	Bulk Conc.	0.9	53.1	7.10	82	1,689	15.3	76.4	1.9	79.6	67.2
		Zinc Conc.	0.19	0.55	56.3	1	38	21.3	5.0	92.1	5.2	9.5
		Feed	0.05	0.65	3.55	1.01	25	100.0	100.0	100.0	100.0	100.0
PW Gossan Comp 1	Transition	Bulk Conc.	20.7	0.11	0.03	116	2676	81.6	10.3	3.5	52.7	67.6
		Feed	0.56	0.02	0.02	4.88	88	100.0	100.0	100.0	100.0	100.0
PW Zone Comp 1	8	Bulk Conc.	10.4	0.13	7.35	7	145	28.3	9.9	33.3	21.7	24.8
		Zinc Conc.	3.88	0.61	48.6	13	212	1.8	7.6	37.6	7.0	6.2
		Feed	1.73	0.06	1.04	1.69	31	100.0	100.0	100.0	100.0	100.0
PW Zone High Zn	8	Bulk Conc.	10.9	34.2	15.1	11	788	50.2	70.1	3.3	27.0	42.8
		Zinc Conc.	0.60	0.98	55.1	1	66	20.5	15.0	90.7	18.4	27.1
		Feed	0.46	1.04	9.68	0.90	39	100.0	100.0	100.0	100.0	100.0
PW Gossan Comp 2	Transition	Bulk Conc.	39.6	0.29	0.06	119	1,053	76.3	24.0	16.5	42.9	62.2
		Feed	1.76	0.04	0.01	9.40	57	100.0	100.0	100.0	100.0	100.0

Note: *met = metallurgical.*

13.2.4.3 Pinwheel Gossan Cyanide Leach

The rougher tailings from each cycle (x5) of the Pinwheel Gossan flotation locked cycle tests was subjected to a bottle roll cyanidation leach test. The test conditions were constant (i.e., maintained pH, CN concentration, leach time (48 h) and pulp density).

The feed to the cyanidation tests remained fairly consistent in terms of precious metal grades, at 2 and 31 ppm and 5 and 21 ppm for gold and silver, respectively, for comp 1 and comp 2. Average results are displayed in Table 13.16.

TABLE 13.16
KM 2575 PW GOSSAN FLOTATION TAILINGS – CYANIDATION LEACH RESULTS

Composite	Au Recovery (%)				
	Bulk Conc	Bulk Tail	CN*	CN**	Total
PW Gossan Comp 1	53	47	80	38	91
PW Gossan Comp 2	43	57	48	27	70

The Bulk Concentrate ("Conc" and Bulk Tail columns represent flotation results. CN* represents the stage recovery from the leach feed (bulk flotation tailings), CN** represents recovery to pregnant solution relative to the flotation feed, and the Total column shows global recovery to both flotation concentrate and cyanidation leach solution.

Silver recoveries for PW Gossan Comp 1 were approximately 70% of cyanidation feed, while for PW Gossan Comp 2 this number was much lower, at approximately 5%.

13.2.4.4 NS Zone Gravity and Cyanide Leach

The NS Zone composite was subjected to gravity separation and cyanide leach testing. The gravity testing included passing the slurried sample through a Knelson concentrator and then panning the Knelson concentrate. Results from this testing are summarized in Table 13.17 and Table 13.18.

TABLE 13.17
KM 2575 NS ZONE – GRAVITY CONCENTRATION RESULTS

Product	Weight		Assays (g/t)		Recovery (%)	
	g	%	Au	Ag	Au	Ag
Pan Conc	7.1	0.4	314	698	24.5	7.3
Pan Tail	41.7	2.1	36.1	163	16.5	10.0
Knelson Tail	1,948.6	97.6	2.76	29	59.0	82.7
Feed	1,997.4	100	4.56	34	100	100

As can be seen above, 24.5% of the gold and 7.3% of the silver from the head sample reported to the pan concentrate. The pan tailings and Knelson tailings were combined and subjected to a cyanidation bottle roll test. The total gold and silver recoveries were 96% and 46%, respectively.

TABLE 13.18
KM 2575 NS ZONE – GRAVITY TAILINGS LEACH RESULTS

Composite	Pan Conc	Grav. Tail	CN*	CN**	Total
NS Zone Composite 1	Au Recovery (%)				
	24	76	94	72	96
	Ag Recovery (%)				
	7	93	42	39	46

CN* *represents recovery from bottle roll feed and CN** represents recovery relative to flotation feed.*

This composite was also submitted for a whole material cyanidation test at the same P_{80} feed size. The recoveries for gold and silver in that test were roughly 95% and 50%, respectively.

13.2.5 SGS Canada Inc. 2010 (12338-001)

In 2010, SGS Lakefield was commissioned to complete a testwork program on three oxide samples. The program included grindability, cyanide leaching, carbon-in-leach, carbon-in-pulp and cyanide destruction testing. Direct analysis of the head samples was not completed, but the average calculated head grades from test products are shown in Table 13.19.

TABLE 13.19
12338-001 CALCULATED HEAD GRADES

Composite	Met Type	Calc. Head Grade	
		Au (g/t)	Ag (g/t)
90 Zone	6	1.97	38.1
PM Zone	6	2.62	41.4
East Gossan	6	41.0	8.7

Note: *met = metallurgical.*

13.2.5.1 Bond Ball Mill Grindability

A Bond ball mill grindability test was performed at 200 mesh of grind (75 µm) on the East Gossan sample. The test results are summarized in Table 13.20. The sample was categorized as having medium hardness, with a ball mill work index of 15 kWh/t.

Table 13.20 12338-001 Bond Ball Mill Grindability Test Results					
Mesh of Grind	**F_{80} (µm)**	**P_{80} (µm)**	**Grams per Revolution**	**BWI (kWh/t)**	**Hardness Percentile**
200	1,016	58	1.26	15.0	55

13.2.5.2 Cyanide Leach

Bottle roll cyanidation tests were completed on the three composites. Each sample was ground in a laboratory rod mill. The mill discharges were pulped to 33% solids, the pH was adjusted to 10.5-11.0 with hydrated lime and 0.5 g/L NaCN was added. The pH and NaCN levels were maintained throughout the leaching period. Total leach time was 48 hours. A summary of the results is tabulated in Table 13.21.

Table 13.21 12338-001 Summary of Bottle Roll Test Results									
Composite	**Feed Size**	**Reag. Cons. (kg/t)**		**Extraction (%)**		**Residue (g/t)**		**Head Grade (g/t)**	
	K_{80} (µm)	**NaCN**	**CaO**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
90 Zone	114	0.21	0.38	86.3	37.2	0.26	22.5	1.90	35.8
PM Zone	83	0.57	0.38	94.9	51.4	0.13	20.2	2.53	41.6
East Gossan	58	0.28	1.33	95.9	44.8	1.69	4.91	41.7	8.9

Note: *Reag. Cons. = reagent consumption.*

Gold extractions ranged from 86% to 96%, with final residues grading 0.13 – 1.69 g/t Au. Silver extractions ranged from 37% to 51%, with residues grading 4.91 – 22.5 g/t Ag. The NaCN consumptions ranged from 0.21 – 0.57 kg/t. Lime consumptions ranged from 0.38 – 1.33 kg/t.

13.2.5.3 Carbon-in-Leach

The composite samples were ground in a laboratory rod mill and the discharge was pulped to 33% solids in glass bottles. The pH was adjusted to 10.5 – 11.0 with hydrated lime. 10 g/L of carbon and 0.5 g/L NaCN were added to the bottles, which were then placed on rolls for 48 hours. The pH and NaCN levels were maintained throughout the leaching period. After 48 hours, the carbon was screened from the pulp and the samples were filtered. The barren solutions, residues and carbon samples were submitted for gold and silver analysis. The residues were also submitted for particle size analysis. The results are summarized in Table 13.22.

13.2.5.4 Carbon-in-Pulp

The composite samples were ground and leached without carbon, as described above. After 48 hours of leaching, 10 g/L of carbon was added to the slurry and allowed to contact with the pulp for five hours. The barren solutions, residues and carbon samples were submitted for gold and silver analysis. The residues were also submitted for particle size analysis. The results are summarized in Table 13.22.

The gold and silver extractions were similar for the carbon-in-leach (“CIL”) and carbon-in-pulp (“CIP”) tests. The 90 Zone composite gold extractions were 85-86% and the silver extractions were 34 – 36%. The PM Zone composite gold extractions were 95 – 96% and the silver extractions were 48 – 52%. The East Gossan composite gold extractions were 95% and the silver extractions were 65 – 69%.

It should be noted that the P_{80} levels for the 90 Zone and the PM Zone samples were significantly higher when these composites were subjected to standard bottle roll tests than they were when CIL and CIP tests were completed. The recoveries for both gold and silver (for both composites) were similar with the two methodologies. This indicates that, at least for these sub-domains, bulk leach is likely more effective for precious metal recovery.

TABLE 13.22
12338-001 SUMMARY OF CIL AND CIP BOTTLE ROLL TEST RESULTS

Composite	Feed Size K_{80} (µm)	Reag. Cons. (kg/t)		Extraction (%)		Residue (g/t)		Loaded Carbon (g/t)		Final Barren (g/t)		Head Grade Calc. (g/t)	
		NaCN	CaO	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
90 Zone – CIL	47	1.07	0.51	85.1	36.4	0.28	24.3	85.7	724	<0.01	0.39	1.84	38.2
90 Zone – CIP	52	1.02	0.53	85.8	34.1	0.24	25.0	77.8	631	0.01	0.77	1.65	37.9
PM Zone – CIL	49	1.54	0.45	95.4	52.2	0.13	19.6	147	1126	0.01	0.72	2.74	41.0
PM Zone – CIP	44	1.32	0.58	96.1	48.5	0.10	20.7	133	917	0.02	1.42	2.55	40.2
East Gossan – CIL	71	0.47	1.57	95.2	65.4	1.91	3.1	2086	311	0.11	0.10	39.4	8.8
East Gossan - CIP	65	0.83	1.58	95.3	69.4	1.91	2.5	1833	236	0.24	0.30	40.7	8.0

Note: *CIL = carbon in leach, CIP = carbon-in-pulp, Reag. Cons. = reagent consumption.*

13.2.5.5 Cyanide Destruction

In order to generate solutions for cyanide destruction testing, a 10 kg charge of each of the three samples was prepared. Each sample was ground to a target P_{80} of 75 µm in a laboratory rod mill. Carbon-in-pulp tests were carried out under the sample conditions as outlined above. After the five-hour contact time, a representative sample of the pulp was taken and filtered. The barren leach solutions, washed residues and carbon samples were submitted for gold and silver assay. The residue was also submitted for particle size analysis. The remainder of the leached pulps were submitted for cyanide destruction testwork.

The barren solution from leaching the 90 Zone composite contained 420 ppm CN_T, 220 ppm CN_{WAD}, 54 ppm CNS, 19 ppm Cu, 64 ppm Fe and 4 ppm Zn. The results indicated that it was possible to obtain detoxified product containing less than 1 ppm residual CN_T by treating the pulp at approximately pH 8.5 with 1 hour retention time, using 3.08 g equivalent SO_2, 2.08 g hydrated lime and 0.56 g Cu (added as copper sulphate) per gram CN_{WAD} in the feed.

The pulp from leaching the PM Zone composite contained 907 ppm CN_T, 520 ppm CN_{WAD}, 140 ppm CNS, 49 ppm Cu, 134 ppm Fe and 17 ppm Zn. The results indicated that it was possible to obtain detoxified product containing less than 1 ppm residual CN_T by treating the pulp at approximately pH 8.5 with 88 minutes retention time, using 3.92 g equivalent SO_2, 2.16 g hydrated lime and 0.56 g Cu (added as copper sulphate) per gram CN_{WAD} in the feed.

The pulp from leaching the East Gossan composite contained 274 ppm CN_T, 240 ppm CN_{WAD}, 30 ppm CNS, 21 ppm Cu, 8 ppm Fe and 1 ppm Zn. The results indicated that it was possible to obtain detoxified product containing less than 1 ppm residual CN_{WAD} by treating the pulp at approximately pH 8.5 with 61 minutes retention time, using 5.11 g equivalent SO_2, 3.42 g hydrated lime and 0.14 g Cu (added as copper sulphate) per gram CN_{WAD} in the feed. However, the CN_T was still 5.8 ppm owing to the presence of residual Fe (1.99 ppm). A higher copper addition would likely be required to achieve <1 ppm residual CN_T in the CND product.

13.2.6 G&T Metallurgical Services Ltd. 2011 (KM 2775)

In 2011, G&T conducted a short testwork program on one sample from the Main Zone. First, composites were prepared from drill core samples which would target composites similar in chemical composition to Main Zone Composite 1 (from the KM 2047 program). Then, flotation testing was completed as in KM 2047 (test 111) to generate tailings that would be subjected to pyrite flotation. This investigation explored various collectors to maximize gold and silver recoveries to the pyrite concentrate. Cyanidation bottle roll tests were then completed on the pyrite concentrate to determine the potential gold and silver recoveries to pregnant solution. The assay results of the composite samples used in the KM 2775 and 'original' composite used in the KM 2047 program are displayed in Table 13.23.

All of the metallurgical testing was completed on Main Zone Composite 3.

TABLE 13.23 KM 2775 - COMPOSITE ASSAYS										
Composite	Assays									
	Cu (%)	Cu (ox) (%)	Cu (CN) (%)	Pb (%)	Zn (%)	Fe (%)	Au (g/t)	Ag (g/t)	S (%)	C (%)
Main Zone Comp 1 (KM 2047)	0.31	0.004	-	0.07	3.33	31.7	3.84	15	-	-
Main Zone Comp 2	1.38	0.03	0.16	0.06	2.33	43.2	4.49	11	50.2	0.10
Main Zone Comp 3	1.23	0.02	0.19	0.08	4.12	39.6	4.29	11	50.6	0.09

13.2.6.1 Cleaner Flotation

Five cleaner flotation tests were conducted on Main Zone Composite 3. The flowsheet was very similar to test 111 from the G&T KM 2047 program. However, the purpose was to generate tailings for pyrite flotation, so there were a few differences:

- Only the first bulk and first zinc cleaner stages were completed.
- The zinc rougher tailings were subjected to a 3-stage pyrite rougher flotation.
- The first pyrite rougher concentrate and the combination of the second and third pyrite rougher concentrates were subjected to bottle roll cyanidation tests.

Although flotation performance was different from Main Zone Composite 1, acceptable first cleaner concentrates were observed for the both the bulk and zinc circuits. The five cleaner tests featured a 3-stage rougher pyrite circuit, testing three key variables: collector type (PAX and 5688), collector dosage and slurry pH. Gold and silver grades and recoveries from the fifth flotation test are displayed in Table 13.24.

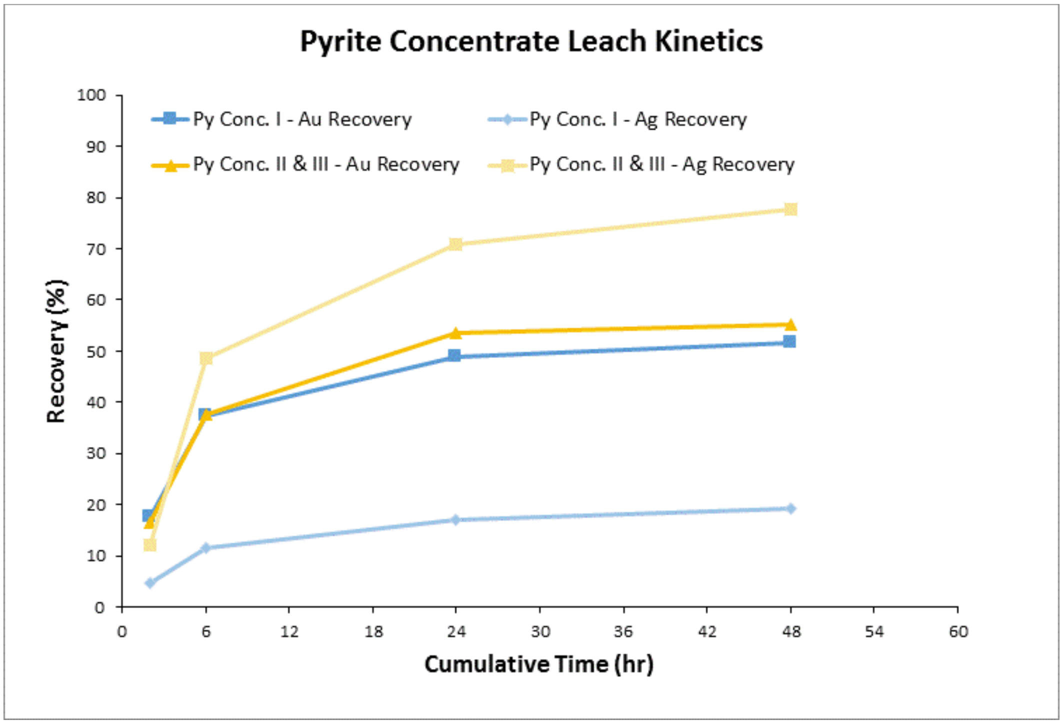
TABLE 13.24 KM 2775 - PYRITE ROUGHER CONCENTRATE GENERATION				
Stream	Grade		Recovery	
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
Feed	3.69	11	100	100
Bulk Concentrate	23.6	63	31.5	27.0
Bulk 1st Cleaner Tail	3.06	13	10.2	14.4
Zinc Concentrate	7.00	23	14.0	14.8
Zinc 1st Cleaner Tail	9.71	17	8.0	4.4
Pyrite Ro Concentrate	3.10	7	7.1	7.0
Pyrite Ro Tails	1.69	6	29.3	32.3

Note: *Ro Concentrate = Rougher Concentrate, Ro Tails = Rougher Tails.*

13.2.6.2 Cyanide Leach

Two cyanide leach tests (bottle roll tests) were performed on the pyrite rougher concentrates from the final cleaner flotation test. The samples tested were pyrite rougher concentrate I and pyrite rougher concentrate II + III. Leach kinetics are presented in Figure 13.8.

FIGURE 13.8 KM 2775 - PYRITE ROUGHER CONCENTRATE LEACH KINETICS



Source: *Aquila Resources Inc. (2020)*

Leach recoveries for gold were both over 50% after 48 hours of leaching, and silver recovery was variable. The global recoveries of gold and silver to the pregnant leach solution were 3.8% and 2.2%, respectively. These results indicated that a substantial improvement in pyrite flotation metallurgy would be required to be considered a viable opportunity.

13.2.7 Resource Development Inc. (RDi) 2011

In 2011, Resource Development Inc. was sent two oxide samples to study the comminution characteristics, cyanidation leach conditions and kinetics, amenability to gravity concentration of flotation tailings as well as potential to concentrate precious metals via flotation. The two samples were from the East Gossan and Pinwheel Gossan portions of the Deposit.

13.2.7.1 Sample Composition

Assays for the two oxides samples are shown in Table 13.25.

TABLE 13.25
RDI 2011 - SAMPLE ASSAYS

Composite	Met Type	Assays											
		Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Hg (g/t)	C_{tot} (%)	C_{org} (%)	C_{inorg} (%)	S_{tot} (%)	$S_{sulphide}$ (%)	$S_{sulphate}$ (%)
East Gossan	6	25.2	10.2	0.022	0.056	0.065	1.62	1.97	0.29	1.68	0.46	0.09	0.37
PW Gossan	6	5.59	75.2	0.36	0.030	0.023	8.96	3.15	0.19	2.96	0.77	0.31	0.46

Note: *met = metallurgical, tot = total, org = organic, inorg = inorganic.*

13.2.7.2 Comminution Characteristics

Both samples were submitted for both Bond ball mill work index and abrasion index tests. The results are displayed in Table 13.26.

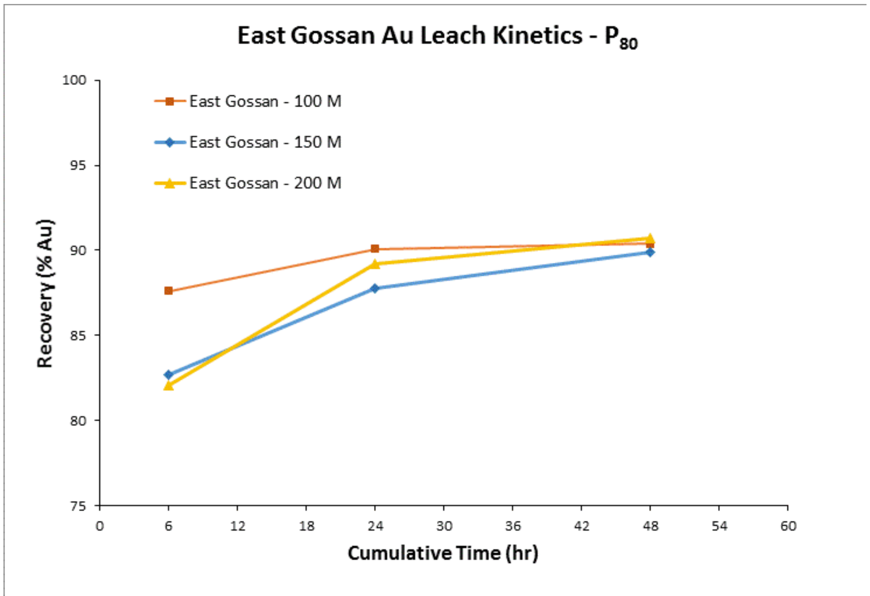
TABLE 13.26
RDI 2011 - COMMINUTION TEST RESULTS SUMMARY

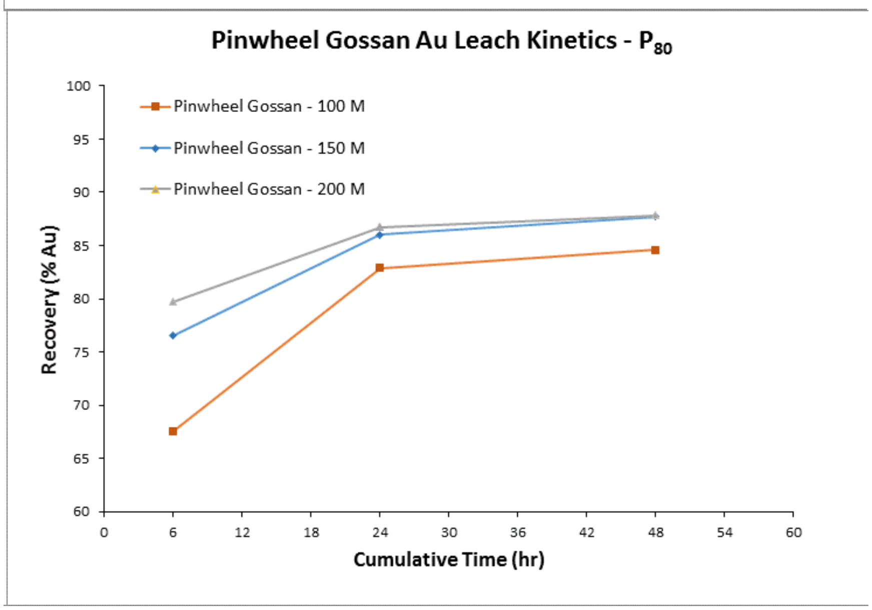
Composite	BWI (kWh/t) @100M	Ai (g)
East Gossan	15.5	0.053
Pinwheel Gossan	12.0	0.029

13.2.7.3 Cyanide Leach

A total of 20 bottle roll tests were completed on the head samples (10 per sample). Four key variables were evaluated: feed grind size, NaCN concentration, pre-aeration and pulp density. Figure 13.9, Figure 13.10 and Figure 13.11 illustrate the results.

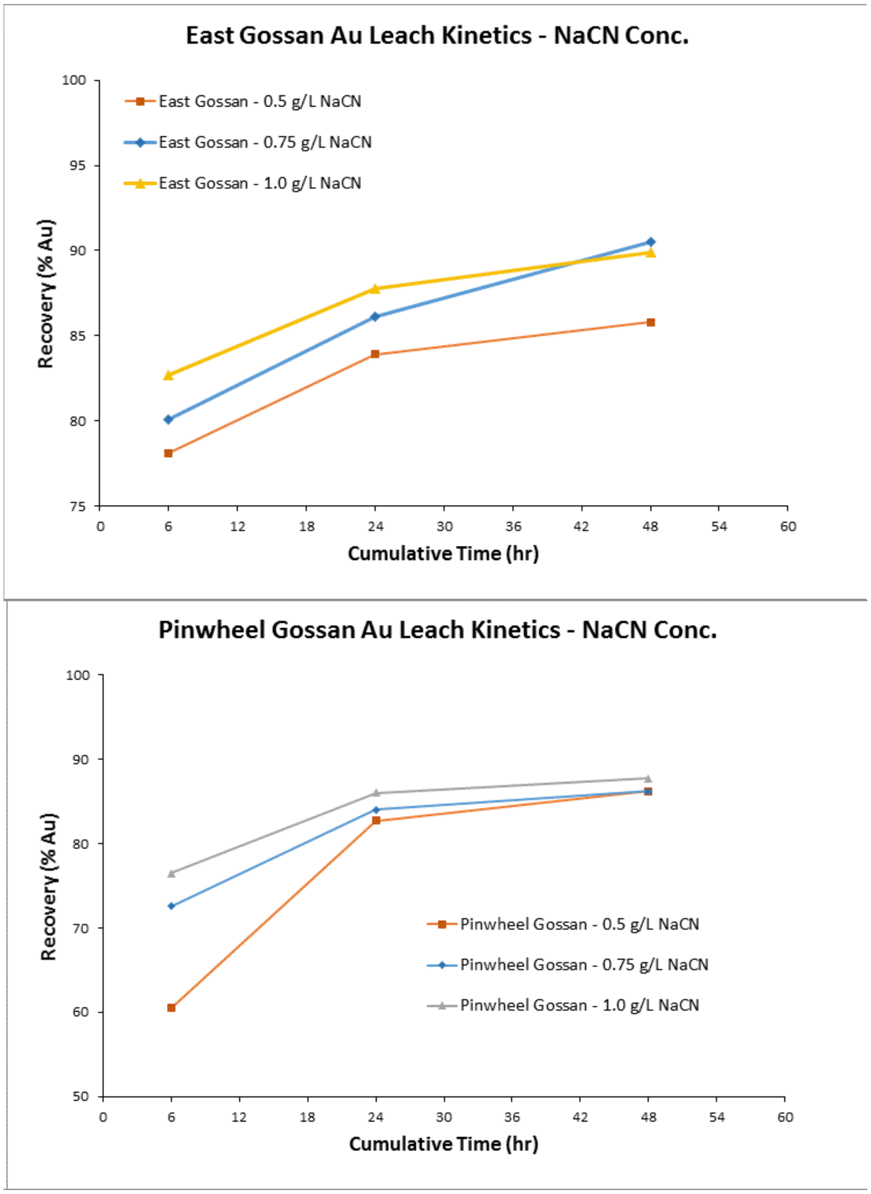
FIGURE 13.9 RDI 2011 –EFFECT OF GRIND SIZE ON CYANIDE LEACH





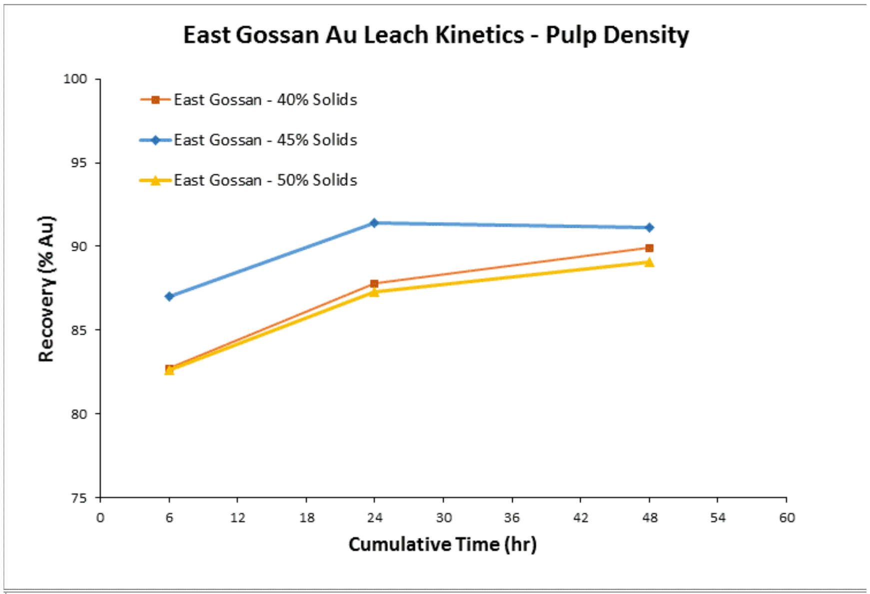
Source: *Aquila Resources Inc. (2020)*

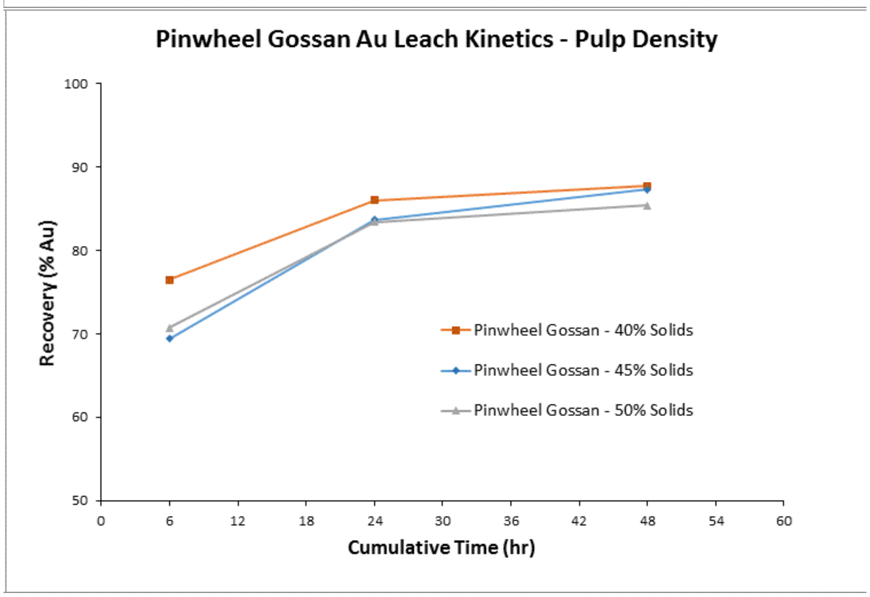
FIGURE 13.10 RDI 2011 - EFFECT OF NACN CONCENTRATION ON CYANIDE LEACH



Source: *Aquila Resources Inc. (2020)*

FIGURE 13.11 RDi 2011 - EFFECT OF PULP DENSITY ON CYANIDE LEACH





Source: *Aquila Resources Inc. (2020)*

Three primary grind sizes (P_{80}) were investigated for each of the two composites: 100 M, 150 M and 200 M (150, 106 and 74 µm, respectively). The East Gossan sample achieved, counterintuitively, faster leach kinetics for the coarsest grind, but the ultimate gold recovery to the pregnant solution at 48 hours was the same for all grind sizes. The Pinwheel Gossan sample achieved increasing kinetics with decreasing grind size, as expected, with the coarser size achieving a lower overall gold recovery.

The East Gossan and Pinwheel Gossan samples were subjected to bottle roll tests at three different NaCN concentrations: 0.5 g/L, 0.75 g/L and 1.0 g/L. The results indicated faster gold leaching kinetics as NaCN concentration was increased. This held true for both samples. However, final extractions did not differ conclusively. The East Gossan sample had similar final extractions for the two higher concentrations, with a lower extraction at the lowest NaCN concentration. The Pinwheel Gossan sample had all three final extractions within the test margin of error.

The two samples were also subjected to bottle roll tests at three different pulp densities: 40%, 45% and 50% solids. As can be seen in Figure 13.11, there was little evidence to suggest an impact on leach kinetics or gold recoveries over the range of pulp densities tested. Although not illustrated, a similar remark can be made for the effect of pre-aeration.

13.2.8 Resource Development Inc. (RDi) – 2012

In 2012, RDi was sent five oxide composites and three sulphide composites to be subjected to series of metallurgical tests. The samples were shipped as drill core to be composited and are reported to have ultimately represented their sub-domain within the pit.

13.2.8.1 Oxide Sample Composition

The five oxide samples are understood to have represented the PM Zone, 90 Zone, NS Zone, PW Gossan and East Gossan domains. Head assays for the five samples are displayed in Table 13.27.

TABLE 13.27
RDI 2012 - OXIDE COMPOSITE ASSAYS

Composite	Met Type	Assays								
		Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Hg (g/t)	S_{tot} (%)	$S_{sulphide}$ (%)	$S_{sulphate}$ (%)
PM Zone	6	1.99	17.6	0.04	0.25	0.30	0.10	1.48	1.48	<0.02
90 Zone	6	1.99	43.8	0.03	0.31	0.39	3.27	4.91	4.91	<0.02
NS Zone	6	3.02	50.0	0.50	0.04	0.04	<0.05	3.89	3.89	<0.02
PW Gossan	6	4.80	69.1	1.29	0.08	0.02	2.89	2.38	2.38	<0.02
East Gossan	6	21.5	4.5	0.02	0.05	0.02	1.53	0.35	0.33	0.06

13.2.8.2 Sulphide Sample Composition

The three sulphide composites are understood to have represented the Pinwheel sulphide, Main Zone and Main Zone - Other domains. Head assays for the three samples are displayed in Table 13.28.

TABLE 13.28 RDI 2012 - SULPHIDE SAMPLE ASSAYS										
Composite	**Met Type**	**Assays**								
		Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Fe (%)**	**S_{total} (%)**	**$S_{sulphide}$ (%)**	**$S_{sulphate}$ (%)**
PW Sulphide	2	2.87	55.2	1.56	0.02	0.62	25.0	33.2	33.2	0.07
Main	1	2.15	14.9	0.35	0.03	4.26	34.6	44.9	44.9	0.11
Main - Other	1	1.38	16.4	0.03	0.03	1.28	6.26	8.24	8.24	<0.02

13.2.8.3 Oxide Sample Cyanide Leaching

Each of the five oxide samples was subjected to 2 cyanide leach tests, both at pH 11 and 45% solids density. For each sample, one test was conducted at a maintained CN concentration of 0.75 g/L (main.) while the other test allowed the initial 0.75 g/L concentration to naturally decrease (decay) throughout the test duration. The results are summarized in Table 13.29.

TABLE 13.29 RDI 2012 - SUMMARY OF BOTTLE ROLL TEST RESULTS								
Composite	**Reag. Cons. (kg/t)**		**Extraction (%)**		**Residue (g/t)**		**Head Grade (g/t)**	
	NaCN	**CaO**	**Au**	**Ag**	**Au**	**Ag**	**Au**	**Ag**
PM Zone (decay)	0.29	4.48	92.0	43.8	0.13	10.4	1.63	18.6
PM Zone (main.)	0.35	5.00	91.8	45.7	0.13	9.8	1.64	18.1
NS Zone (decay)	0.87	2.99	42.2	0.7	1.64	61.8	2.87	62.3
NS Zone (main.)	2.61	2.99	42.9	2.1	1.70	47.6	3.00	46.6
East Gossan (decay)	0.24	8.55	92.6	57.4	1.61	2.2	22.0	5.2
East Gossan (main.)	0.16	8.99	90.8	59.4	2.05	2.0	22.5	5.0
90 Zone (decay)	0.29	3.52	85.2	37.9	0.29	27.4	1.96	44.4
90 Zone (main.)	0.25	3.66	82.9	44.2	0.34	23.0	2.01	41.4
PW Gossan (decay)	0.87	8.15	30.3	0.5	2.95	89.8	4.25	90.2
PW Gossan (main.)	2.88	4.24	38.4	0.7	2.54	59.0	4.13	59.4

Note: *Reag. Cons. = reagent consumption.*

The recovery difference between maintaining and allowing the CN concentrations to decrease naturally appears to have been minimal. The gold and silver extractions for three composites (PM Zone, East Gossan and 90 Zone) were significantly higher (85-92% and 38-60% respectively) than the extractions for the other two samples (30-42% and 0.5-2.1%, respectively).

It should be noted that both of the poorer performing samples had high copper content, relative to the other three samples tested.

13.2.8.4 Sulphide Sample Flotation

Each of the three sulphide samples was subjected to one open circuit cleaner flotation test. The results are summarized in Table 13.30.

The PW Sulphide composite did not produce marketable copper or zinc concentrates.

It appears from the results above that the regrind sizes in both the copper (Main sample) and zinc (Main and Main Other samples) circuits were too coarse. Rougher performance was reasonable, but upgrading in the cleaners was not as efficient, indicating that the minerals were not sufficiently liberated. Note that the regrind size P_{80} target was 44 µm, but the actual regrind size was not measured in the tests.

TABLE 13.30
RDI 2012 - CLEANER FLOTATION SUMMARY

Sample	Product	Wt. (%)	Assays					Distribution				
			Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Pb (%)	Cu (%)	Zn (%)	Au (%)	Ag (%)	Pb (%)
Pinwheel Sulphide	Cu 3rd Cl Conc.	4.1	8.28	4.86	5.30	222	-	23.0	32.4	11.2	18.7	-
	Cu Ro. Conc.	20.5	6.18	2.56	4.42	184	-	85.7	85.3	46.8	77.2	-
	Zn 3rd Cl Conc.	0.6	5.30	4.60	6.20	216	-	2.1	4.5	1.9	2.6	-
	Zn Ro. Conc.	6.0	2.22	0.84	2.51	81.2	-	9.0	8.1	7.7	9.9	-
	Rougher Tail	73.5	0.11	0.06	1.20	8.6	-	5.3	6.6	45.5	12.9	-
	Calc. Feed	100.0	1.48	0.62	1.94	48.9	-	100.0	100.0	100.0	100.0	-
Main Other	Cu 1st Cl. Conc.	0.4	3.78	9.62	112	1417	21.1	21.3	3.4	42.9	41.1	50.2
	Cu Ro. Conc.	2.6	0.94	3.79	22.8	310	4.67	34.0	8.7	56.3	57.8	71.6
	Zn 3rd Cl. Conc.	1.1	0.23	53.8	1.40	32	0.23	3.4	49.7	1.4	2.4	1.4
	Zn Ro. Conc.	4.0	0.30	23.6	2.17	30.3	0.35	16.5	82.0	8.2	8.6	8.2
	Rougher Tail	93.4	0.04	0.11	0.40	5.0	0.04	49.5	9.3	35.5	33.5	20.2
	Calc. Feed	100.0	0.07	1.14	1.05	13.9	0.17	100.0	100.0	100.0	100.0	100.0
Main	Cu 2nd Cl. Conc.	0.4	26.5	1.76	121	425	0.72	35.4	0.2	26.3	16.1	9.4
	Cu Ro. Conc.	3.0	8.88	3.59	32.2	148	0.35	80.2	2.6	47.3	37.9	30.8
	Zn 3rd Cl. Conc.	3.6	0.10	52.9	0.40	11.6	0.01	1.1	45.7	0.7	3.6	0.7
	Zn Ro. Conc.	11.7	0.28	33.6	1.06	17.8	0.02	9.9	95.0	6.2	17.9	7.9
	Rougher Tail	85.3	0.04	0.12	1.10	6.0	0.02	9.9	2.5	46.5	44.2	61.3
	Calc. Feed	100.0	0.33	4.14	2.02	11.6	0.03	100.0	100.0	100.0	100.0	100.0

13.3 SAMPLE SELECTION – 2018 FEASIBILITY STUDY AND CURRENT PEA

13.3.1 Introduction

A series of testing campaigns were conducted from 2015 to 2019 in support of both the 2018 Feasibility Study as well as the current PEA. All data presented below are considered valid as part of the current flowsheet configuration.

13.3.2 Comminution Samples

Three different sizes of samples were selected for the comminution data collection: halved, quartered and whole NQ core samples, for a total of 69 composites and master composites. The whole diameter NQ samples were chosen to complete crusher work index tests (Bond low energy impact tests) which require 10-20 specimens minus 3" plus 2" per test sample.

Samples were selected and composited to represent the variability of the Back Forty Deposit from several perspectives including pyrite (and other mineral content), lithology, metallurgical type, and metal grades.

13.3.3 Metallurgical Samples

The SGS Geostats group selected metallurgical samples for themetallurgical program. The mandate was to create oxide and sulphide domain composites which included a drill plan for fresh sulphide material. Following sub-domain compositing, they were to create three sulphide master composites with the sub-domain composite material. More specifically, the goals of the sample selection were:

- Validation of the sample quantity.
- Tonnage and LOM.
- Existing geological domain validation.
- Material quantity from each domain to be mined in the LOM.
- Au-Ag zones (oxides) validation and composite generation.
- Cu-Pb-Zn zones (sulphides) validation and composite generation.
- Sampling list and DDH planning.
- Creation of master composites from domain composites.

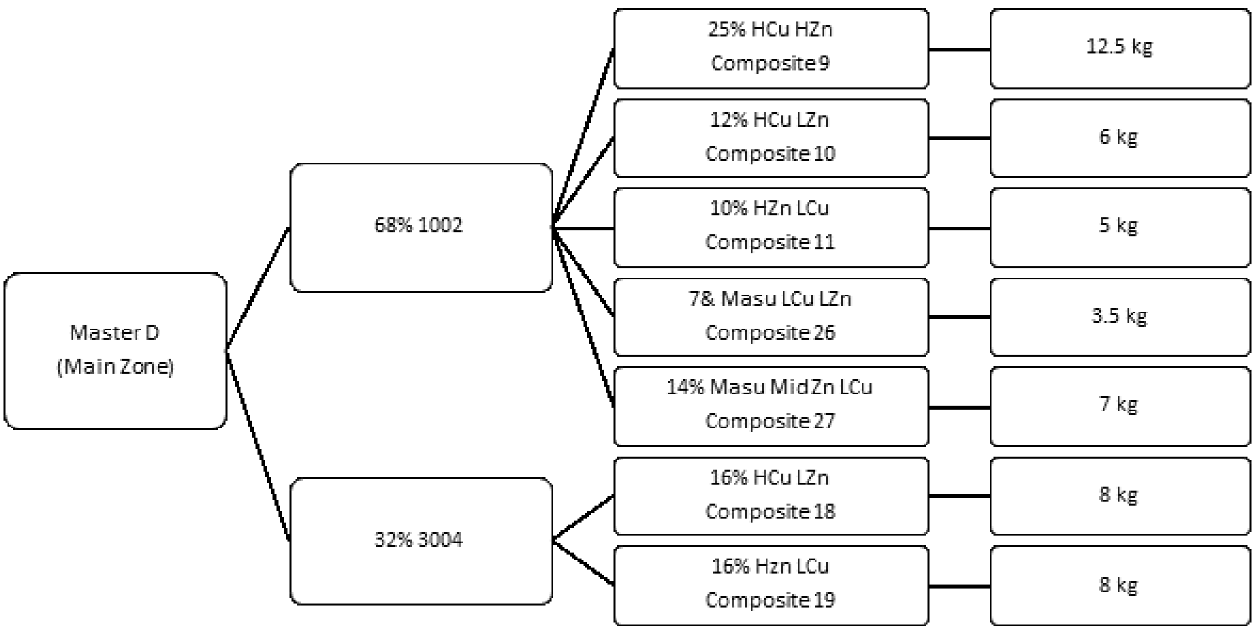
The final samples and metal grades can be found in Table 13.31.

TABLE 13.31
METALLURGICAL FEASIBILITY SAMPLES

Composite	Zone Description	Metal-lurgical Type	Assays					
			Cu (%)	Pb (%)	Zn (%)	S (%)	Au (g/t)	Ag (g/t)
1	Near Surface Low Sulphur	6	0.02	<0.01	0.01	1.51	3.22	30.6
2	Pinwheel Gossan Low Sulphide	6	0.09	0.03	<0.01	0.45	5.42	14.8
3	East Gossan	6	0.01	0.00	0.02	0.23	14.7	6.33
4	East Oxide	6	0.03	<0.01	0.18	5.21	0.74	<10
5	90 Zone Low Sulphide	6	0.03	0.12	0.25	3.27	1.81	23.1
6	Porphyry Low Sulphur	6	0.07	0.31	1.67	5.49	3.97	24.4
9	Main Zone High Cu, High Zn	1	0.26	0.13	11.9	44.7	2.26	14.3
10	Main Zone High Cu, Low Zn	1	0.95	0.01	0.37	50.6	3.41	17.6
11	Main Zone High Zn, Low Cu	1	0.09	<0.01	11.0	48.3	1.88	<10
12	Pinwheel High Cu, High Zn	8	2.77	0.00	7.07	50.0	1.02	34.0
13	Pinwheel High Cu, Low Zn	2	3.76	0.00	0.11	48.9	2.54	68.0
14	Tuff Zone Massive Sulphide	5	0.07	0.80	2.99	7.24	2.93	95.5
15	Pinwheel Sulphides and Stringers	3	0.36	<0.01	0.11	16.7	1.20	<10
16	Tuff Zone Stringers High Pb, Low Zn	5	0.03	0.44	2.46	8.09	0.46	8.17
17	Tuff Zone Stringers Low Pb, Low Zn	5	0.03	0.14	0.04	4.46	2.12	30.8
18	Main Sulphide Stringers High Cu, Low Zn	1	1.16	<0.01	0.08	33.2	1.39	16.4
19	Main Sulphide Stringers High Zn, Low Cu	1	0.10	0.02	5.13	16.7	1.87	10.2
20	Main Sulphide Stringers Low Grade	1	0.07	<0.01	0.46	4.96	1.15	<10
23A	Pinwheel Gossan Low Sulphur	Ox / Trans	0.64	0.04	0.01	2.35	8.20	476
23B	Pinwheel Gossan Massive Sulphide	Sul / Trans	4.45	0.06	<0.01	35.0	2.23	302
23.1 (70%A, 30%B)	Pinwheel Gossen Cu Rich and Transitional	Trans	1.82	0.05	0.01	12.5	6.35	422
24	Near Surface Quartz-Feldspar Porphyry (QFP)	6	0.01	0.01	0.92	0.43	5.15	130
25	Pinwheel Massive Sulphide Low Grade	3	0.62	0.00	0.15	34.5	1.76	9.55
26	Main Zone Massive Sulphide Low Cu and Zn	1	0.12	0.00	0.40	52.6	1.54	<10
27	Main Zone Massive Sulphide Mid Zn and Low Cu	1	0.08	0.01	2.93	52.2	1.93	1.71
28	90 Zone Flotation	6	0.02	0.26	0.35	3.85	1.42	38.7

From the samples above, four master composites were created; Main, Pinwheel and Tuff zones as well as oxide. An illustration of master composite (Main Zone) compilation is shown in Figure 13.12.

FIGURE 13.12 MASTER COMPOSITE COMPILATION



Source: *Aquila Resources Inc. (2020)*

13.4 COMMINUTION CIRCUIT CHARACTERIZATION TESTWORK

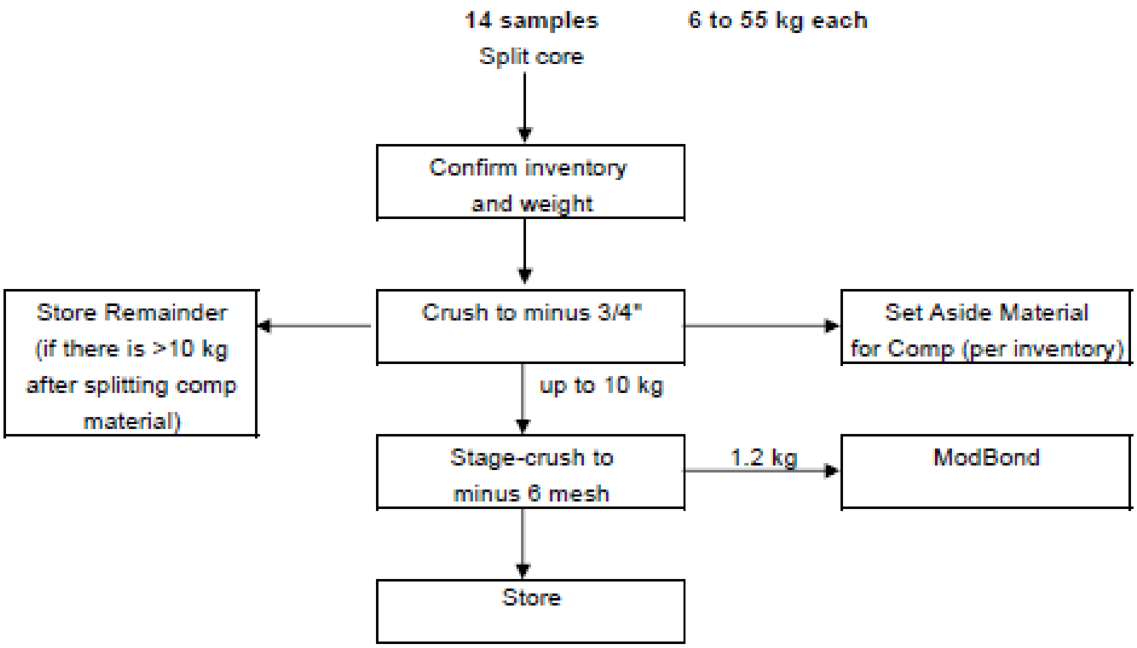
The testing consisted of grindability testwork to characterize the competency, hardness and abrasion of the Back Forty Deposit material.

Two shipments of drill cores were sent to the SGS, Lakefield site, from July 2015 to December 2015. The shipments included sixty-nine samples which were subjected to a series of tests: Bond ball work index (BWI, 16), modified Bond ball work index (ModBond, 61) and abrasion index (AI, 9). Further composites were created by compiling unused material and were submitted for SMC tests (9). A majority of the material used for this phase was from past drilling programs and was sent as quartered NQ core. As new drill core became available, more material was sent to SGS to conduct crusher work index tests (CWI, 6); full NQ was used for that purpose only.

Several sample preparation instructions were used depending on the mass of material available and the tests to which the material was to be subjected. Figure 13.13 illustrates one of the sample preparation instructions.

The samples submitted for Bond ball mill work index testing were also submitted for the ModBond test to establish the ModBond - BWI correlation parameters.

FIGURE 13.13 COMMINUTION SAMPLE PREPARATION EXAMPLE



Source: *Aquila Resources Inc. (2020)*

13.4.1 Overall Grindability Testwork Results

The summary of the results of the first phase of grindability tests for the comminution variability samples is shown in Table 13.32. The following discussion is a summary of the results from the SGS grindability report.

Overall, the sample depicted a large amount of variability across the grindability characterization tests. The relative standard deviation of test results within each series ranged from 16 to 54, which is considered broad. Further variability grindability tests are therefore recommended to better understand the nature of this variability.

TABLE 13.32
SUMMARY OF GRINDABILITY TEST STATISTICS (2016 RESULTS)

Statistics	Rel. Density		CWI	JK Parameters			BWI (kWh/t)		ModBond (kWh/t)		Ai
	CWI	SMC	(kWh/t)	A x b	t_a	DWi	@150M	@200M	@150M	@200M	(g)
Results Available	6	9	6	9	9	9	11	5	61	61	9
Average	3.85	3.46	7.9	45.3	0.3	8.8	12.9	13.8	13.0	13.3	0.398
Std. Dev.	0.64	0.84	2.9	24.5	0.1	2.5	3.2	2.3	3.1	2.7	0.107
Rel. S.D. (%)	16	24	37	54	32	28	25	17	24	20	27
Min	2.99	2.71	4.4	83.9	0.5	5.3	9.1	10.6	8.3	9.2	0.285
10^{th} Percentile	-	-	-	-	-	-	-	-	9.6	10.4	-
25^{th} Percentile	-	-	-	-	-	-	-	-	10.4	11.2	-
Median	4.54	2.96	7.6	29.4	0.3	9.8	12.8	14.4	12.4	12.7	0.365
75^{th} Percentile	-	-	-	-	-	-	-	-	14.8	14.7	-
90^{th} Percentile	-	-	-	-	-	-	-	-	17.5	17.1	-
Max	4.68	4.86	12.5	22.5	0.2	12.3	18.9	16.7	21.7	20.8	0.564

13.4.2 SMC Tests

The SMC test is an abbreviated version of the standard JK Drop Weight Test performed on rocks from a single size fraction (-16.0/+13.2 mm in this case). The SMC test was performed on a total of nine samples.

The SMC test results are preferably calibrated against reference samples submitted to the standard JKTech drop weight tests ("DWT") in order to consider the natural 'gradient of hardness' by size, which can vary from one sample to another. As no suitably sized core was available and subsequently no DWT results were available from the Back Forty Deposit, the SMC results were calibrated against the JKTech database. The samples fell in the soft to very hard range of hardness, with A x b ranging from 83.9 to 22.5. There was a broad range in the relative density, from 2.71 to 4.86. Of the nine samples tested, three were Main Zone, one was Pinwheel, two were Tuff Zone and three were oxide. Within their own mineralized zones, there was relative consistency in both hardness and density of the samples.

SMC test results are typically calibrated against a full JK Drop Weight Test. In this case, the results were calibrated against the JKTech database. It is recommended that calibrations against full JK DWTs be completed on Back Forty samples to validate this method.

13.4.3 Crusher Work Index Tests

The Bond low-energy impact test determines the Bond Crusher Work Index ("CWI"), which can be used with Bond's third-theory of comminution to calculate power requirements for crusher sizing. Six samples, with between three and twenty rocks in the range of 2 inches to 3 inches, were shipped to SGS Vancouver for the completion of the Bond low-energy impact test.

Some samples were well below the 20 rocks recommended by SGS in order to account for the variability of the samples. However, this was all of the material available. As such, it is recommended that in these cases the results be further validated on samples of 20 specimens or more. The results from sub-zones were used to calculate the overall zone CWI data (e.g., PW SFST and PW MASU to determine the overall PW CWI). The samples covered the soft to moderately hard range of hardness within the SGS database, with CWI varying from 4.4 kWh/t to 12.5 kWh/t. The average CWI was 7.9 kWh/t, which was classified as moderately soft.

13.4.4 Bond Ball Mill Grindability Tests

The Bond ball mill grindability test is widely accepted and used in the mining industry for comminution circuit sizing. The samples submitted for this test were selected based on the ModBond test results in order to cover the whole range of hardness of the variability samples. ModBond tests were completed on 61 samples.

The closing screen size should be selected to reflect the plant product. Initially, the tests were performed at a closing size of 150 mesh (106 µm) on six selected samples. However, the results of flotation testing (conducted concurrently with comminution tests) revealed that a finer primary grind was required across all metallurgical types. As a result, the closing size was reduced to 200 mesh of grind (75 µm) for the remaining five tests. In addition, it was requested that SGS determine the BWI at the finer size from the initial tests completed at a closing size of 150 mesh. To accomplish this, the following work was completed:

- A second set of tests (5) was conducted at both closing screen sizes (150 and 200 mesh).
- ModBond calibrations were conducted for the five samples at both closing sizes.
- The correlation between ModBond at both closing sizes was determined.
- Using this correlation, the ModBond results at 200 mesh for the initial six samples were estimated based on their ModBond values at 150 mesh.

The five samples evaluated at both closing screen sizes showed that the closing screen size had very limited effect on the BWI, for the samples tested. The difference between the closing screen sizes was within the experimental error for four of the samples, at between -0.2 and 0.3 kWh/t. The softest sample tested showed an increase in BWI from 9.2 kWh/t at the coarser closing size to 10.6 kWh/t at the finer size.

The BWI testing was conducted on a total of eleven samples (the aforementioned initial six and subsequent five). The Ball mill Work indices (BWI) ranged from very soft, with a BWI of 9.1 kWh/t, to hard, with a BWI of 18.9 kWh/t. While a relatively wide range of results are observed over the data as a full set, ranges are narrower by metallurgical type, with oxides being the most competent and Pinwheel being the least competent.

13.4.5 ModBond Tests

The ModBond test consists of a single batch test, which is calibrated against standard Bond ball mill grindability test results. The first twenty-nine samples (i.e., from the July 2015 shipment) were calibrated at 150 mesh, and the six full Bond test results were used to accomplish this.

The ModBond tests for the second set (i.e., from the December 2015 shipment, thirty-two samples) were calibrated at both 150 and 200 mesh against the five full Bond test results from the same set of samples. A calibration between the ModBonds at the two closing sizes was developed, and this calibration was then used to estimate the ModBond results for the initial twenty-nine samples at 200 mesh.

The samples covered the very soft to very hard range of hardness in the SGS database, ranging from 9.2 kWh/t to 20.8 kWh/t. Following the trend from other hardness characterization tests, the global set of data shows a significant relative standard deviation, while within each metallurgical type the data range was narrower. Again, oxides on average were the most competent while Pinwheel Zone samples were the least competent.

The average ModBond results were similar at the two closing screen sizes, at 13.0 kWh/t and 13.3 kWh/t at 150 mesh and 200 mesh, respectively. This was classified as moderately soft in the SGS database. The coarser closing size was shown to have a slightly wider range of ModBond results.

13.4.6 Bond AbrasionTests

A total of nine composite samples was submitted for the Bond abrasion test. The Abrasion Index ("AI") ranged from 0.285 g to 0.564 g, corresponding to the medium to abrasive range of the SGS database. The average AI was 0.398 g.

13.4.7 Additional Comminution Testing

In 2017, 24 samples were shipped to SGS Lakefield. All 24 samples were subjected to SMC testing and five samples were subjected to Bond ball mill tests. The results are summarized in Table 13.33.

TABLE 13.33
SUMMARY OF GRINDABILITY TEST STATISTICS, 2017 RESULTS

Statistics	Rel. Density	JK Parameters			BWI (kWh/t)
		A x b	t_a	DWi	@ 150M
Results Available	24	24	24	24	5
Average	3.42	42.9	0.32	10.5	11.7
Std. Dev.	0.69	16.4	0.06	1.8	1.7
Rel. S.D. (%)	20	38	18	17	15
Min	2.77	85.6	0.49	6.8	8.8
10th Percentile	-	-	-	-	-
25th Percentile	-	-	-	-	-
Median	3.13	35.7	0.30	12.0	12.4
75th Percentile	-	-	-	-	-
90th Percentile	-	-	-	-	-
Max	4.83	27.0	0.25	12.3	13.0

13.5 FLOTATION TESTWORK

13.5.1 Introduction

The objectives of the flotation testwork campaign were to develop the flotation conditions and further optimize the historical results. This was intended to minimize the number of distinct metallurgical types from a processing perspective and to optimize reagent dosages with some consideration for alternatives.

Individual samples from two shipments were selected by Aquila, with support from SGS Geostat, to make up 26 variability composites. The approach taken to decrease the number of metallurgical types was to create variability composites within each of the main mineralization types (Main, Pinwheel and Tuff). The variability composite selection is understood to have considered the factors as listed in section 13.3.2 above. In order to create master composites, the variability composites were prepared and then specific proportions of each were riffled out and blended. This created three master composites (Main, Pinwheel and Tuff). The variability composite samples were conserved as much as possible and retained separately. Thus, if acceptable metallurgical performance from any of the master composites was not attained, then adequate sample would be available to complete optimization testing on the variability composites (creating additional metallurgical types). Individual core samples were also stored for possible future evaluation and to confirm that recoveries were in line with of the average.

Reagent optimization occurred once the level at which each mineralization type needed to be evaluated was determined. That is, reagent optimization was completed either at the master composite level or the variability composite level (or a combination of two or more variability composites).

13.5.2 Master Composite Mineralogy

After the three master composites were prepared and ground to a target grind size of P_{80} of 75 µm, they were submitted for mineralogical analysis with QEMSCAN, on four size fractions. Table 13.34 displays the overall modal mineralogy of each master composite.

TABLE 13.34
MODAL MINERALOGY - MASTER COMPOSITE SAMPLES

Mineral	Mineral Mass (%)		
	Pinwheel	Main	Tuff
Pyrite/Marcasite	72.5	71.3	13.0
Sphalerite	2.31	9.89	4.24
Chalcopyrite	6.31	1.37	0.10
Galena	0.00	0.06	0.73
Arsenopyrite	0.08	0.07	0.30
Other Sulphides	0.21	0.28	0.13
Fe Oxides	0.02	0.24	0.04
Other Oxides	0.03	0.02	0.02
Quartz	6.11	7.38	55.8
K-Feldspar	0.83	0.60	3.08
Plagioclase	0.01	0.17	0.06
Sericite/Muscovite	7.46	4.66	17.0
Chlorite/Clays	1.02	2.17	4.37
Talc	1.49	0.11	0.04
Biotite	0.43	0.15	0.23
Other Silicates	0.04	0.15	0.05
Dolomite	0.06	0.48	0.25
Calcite	0.34	0.18	0.11
Sulphates	0.62	0.56	0.41

TABLE 13.34 **MODAL MINERALOGY - MASTER COMPOSITE SAMPLES**			
Mineral	**Mineral Mass (%)**		
	Pinwheel	**Main**	**Tuff**
Other	0.12	0.12	0.12

The above modal analysis indicates that both the Pinwheel and Main zone master composites were dominated by chalcopyrite and sphalerite as value minerals and pyrite as the major gangue mineral. The Tuff Zone master composite was dominated by sphalerite and galena as value minerals, with a large amount of the gangue represented by quartz.

Table 13.35 and 13.36 display the liberation data of the target minerals for the Pinwheel master composite at four different size fractions and for the overall sample for the minerals of interest: Cu-sulphides, galena and sphalerite. Cu-Sulphide liberation was poor in the -106/+75 µm range, at approximately 33% (free plus liberated). This increased to over 74% for particles finer than 20 µm. Sphalerite liberation (free plus liberated) was better, at almost 74% for particles -106/+75 µm and over 87% at the finest size fraction. The global free plus liberated particles for pyrite in this sample was over 93%.

TABLE 13.35
SUMMARY OF PINWHEEL MASTER COMPOSITE CU-SULPHIDE LIBERATION

Mineral	Combined	+106 µm	-106/+75 µm	-75/+20 µm	-20 µm
Free Cu-Sulph	41.6	6.90	28.0	39.8	62.4
Lib Cu-Sulph	8.15	5.31	4.98	7.21	11.9
Midds Cu-Sulph	11.6	9.44	14.4	12.7	9.38
Sub Midds Cu-Sulph	19.6	34.4	26.6	20.2	10.1
Locked Cu-Sulph	19.1	44.0	26.1	20.1	6.26
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

TABLE 13.36
SUMMARY OF PINWHEEL MASTER COMPOSITE SPHALERITE LIBERATION

Mineral	Combined	+106 µm	-106/+75 µm	-75/+20 µm	-20 µm
Free Sphal	69.4	50.5	62.5	73.0	75.6
Lib Sphal	9.79	3.38	11.4	9.18	11.8
Midds Sphal	9.30	4.87	10.6	10.4	7.42
Sub Midds Sphal	6.12	12.3	9.26	5.07	3.01
Locked Sphal	5.43	28.9	6.12	2.35	2.20
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

For the Main master composite, both minerals of interest were relatively well liberated. Table 13.37 shows that Cu-Sulphides were 80% and over 91% liberated at -106/+75 µm and -20 µm, respectively. Table 13.38 shows that sphalerite liberation was even better, at almost 89% in the -106/+75 µm size fraction and over 94% for particles finer than 20 µm. This would suggest that zinc performance for the Main Zone would be excellent, from a mineralogical perspective. The global free plus liberated particles for pyrite in this sample was over 98%.

TABLE 13.37
SUMMARY OF MAIN MASTER COMPOSITE CU-SULPHIDE LIBERATION

Mineral	Combined	+106 µm	-106/+75 µm	-75/+2 0 µm	-20 µm
Free Cu-Sulph	78.4	35.3	73.8	85.1	76.5
Lib Cu-Sulph	7.18	1.42	6.15	3.72	15.0
Midds Cu-Sulph	3.63	0.00	2.91	3.58	4.79
Sub Midds Cu-Sulph	3.31	8.90	5.09	3.10	1.60
Locked Cu-Sulph	7.45	54.4	12.0	4.47	2.10
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

TABLE 13.38
SUMMARY OF MAIN MASTER COMPOSITE SPHALERITE LIBERATION

Mineral	Combined	+106 µm	-106/+75 µm	-75/+20 µm	-20 µm
Free Sphal	88.3	47.9	85.3	91.0	90.7
Lib Sphal	3.63	2.96	3.37	3.91	3.43
Midds Sphal	2.47	5.26	3.77	1.77	2.17
Sub Midds Sphal	2.70	12.3	4.04	1.71	2.14
Locked Sphal	2.92	31.6	3.56	1.58	1.51
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Tables 13.39 and 13.40 display the liberation of the Tuff master composite sample minerals of interest, galena and sphalerite. Galena liberation at the -106/+75 µm size fraction was less than 54% and increased to ~87% in the minus 20 µm size fraction. As with the Main sample, the Tuff sample had a relatively high degree of sphalerite liberation. Sphalerite liberation degrees of approximately 82% and 93% were measured at the -106/+75 µm and -20 µm ranges, respectively. Silica liberation was not evaluated for this sample.

TABLE 13.39
SUMMARY OF TUFF MASTER COMPOSITE GALENA LIBERATION

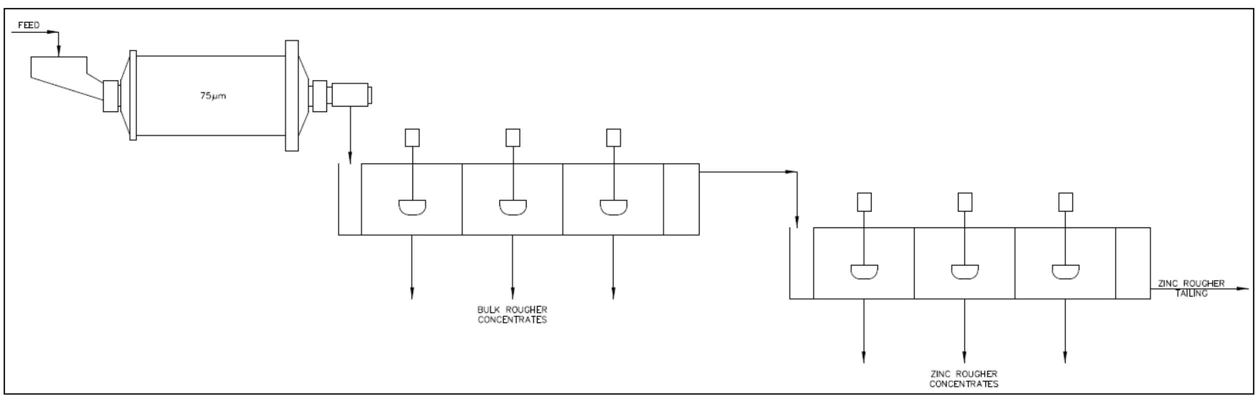
Mineral	Combined	+106 µm	-106/+75 µm	-75/+20 µm	-20 µm
Free Galena	68.2	14.6	52.9	69.4	77.9
Lib Galena	5.72	1.10	1.45	3.35	9.03
Midds Galena	6.01	0.11	4.63	4.19	8.41
Sub Midds Galena	6.09	0.44	10.3	9.88	3.13
Locked Galena	14.0	83.8	30.7	13.2	1.50
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

TABLE 13.40 SUMMARY OF TUFF MASTER COMPOSITE SPHALERITE LIBERATION					
Mineral	**Combined**	**+106 µm**	**-106/+75 µm**	**-75/+20 µm**	**-20 µm**
Free Sphal	80.3	33.8	77.5	85.6	83.2
Lib Sphal	6.39	0.92	4.32	5.37	9.69
Midds Sphal	3.72	3.15	5.18	4.14	2.54
Sub Midds Sphal	4.21	17.3	5.92	2.84	2.80
Locked Sphal	5.34	44.9	7.11	2.04	1.73
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

13.5.3 Master Composite Rougher Testing

To scope the optimal target primary grind size, two tests were completed on each of the master composites. The target grind sizes were P_{80} values of 75 and 106 µm. Figure 13.14 illustrates the flowsheet to which the composites were subjected. Primary grind is quotes as 80% passing size.

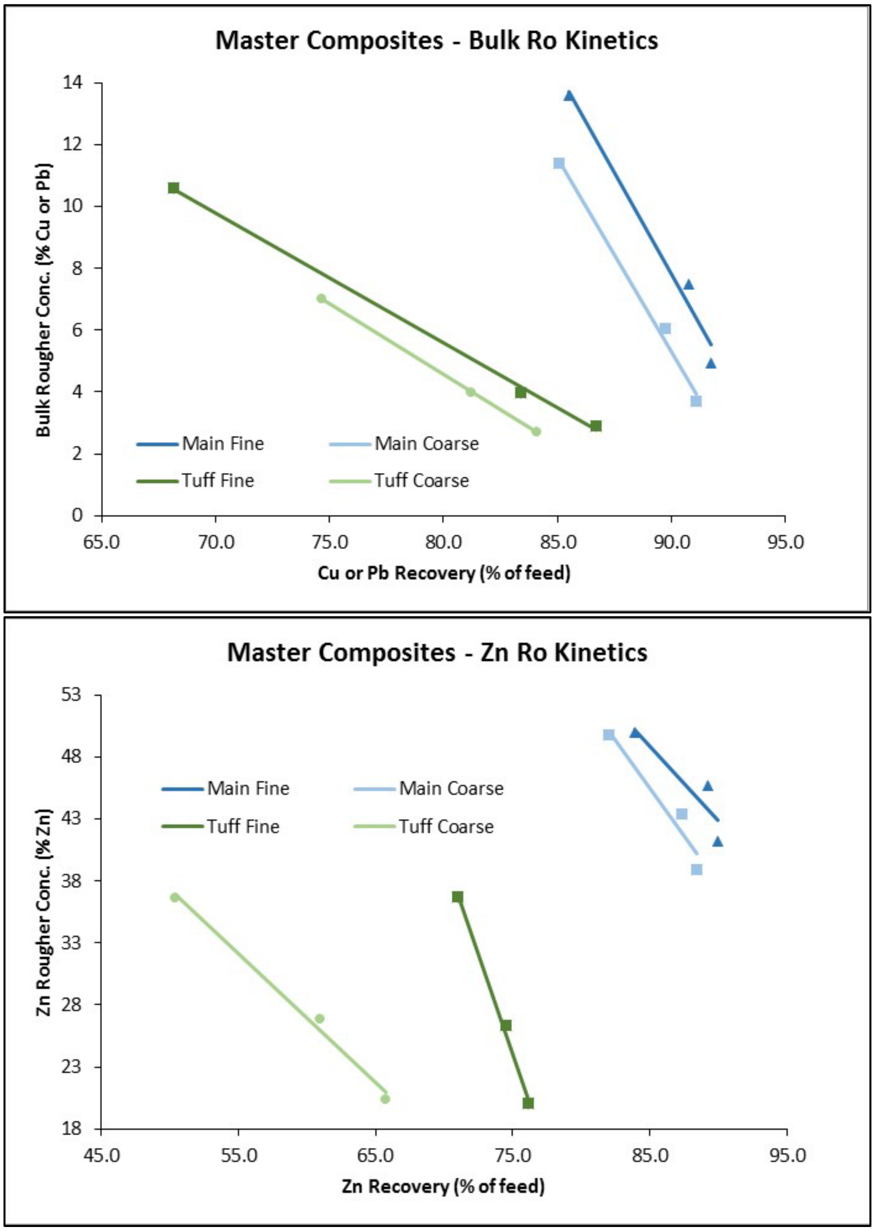
FIGURE 13.14 ROUGHER FLOTATION FLOWSHEET



Source: *Aquila Resources Inc. (2020)*

The rougher flotation flowsheet consisted of a primary grind followed by six stages of flotation; three targeting copper followed by three targeting zinc. The results of four of the six tests are illustrated in Figure 13.15. The Main and Tuff zones showed a clear improvement with decreased flotation feed size, which confirms historical results. The Pinwheel master composite performed poorly at both feed sizes, which was indicated by mineralogy. (Results of the Pinwheel tests are not included in Figure 13.15.)

FIGURE 13.15 MASTER COMPOSITE ROUGHER FLOTATION PERFORMANCE



Source: *Aquila Resources Inc. (2020)*

One additional rougher test was completed on each of the Main and Tuff composites. The results were similar to those obtained with the fine grind tests discussed above. It was deemed appropriate to move ahead with cleaner flotation testing for the Main and Tuff master composites.

The Pinwheel master composite was subjected to six rougher tests, in addition to the fine and coarse primary grind test. Only one test upgraded to an acceptable rougher concentrate grade, but the recovery was less than 40%.

13.5.4 Master Composite Cleaner Testing

15 cleaner flotation tests were run on the three master composites, with five on each of the Main, Pinwheel and Tuff zones.

13.5.4.1 Main Zone Cleaner Flotation

Initial cleaner tests on the Main Zone master composite began using rougher conditions from the aforementioned rougher tests as well as cleaner conditions used in the historical work. While results for the first tests were reasonable, it appeared as though an inability to upgrade the concentrate in the bulk circuit was a result of over-collection of gangue material (likely pyrite) at the rougher stage. A 50% reduction in collector in the rougher stages confirmed this suspicion; a decrease in mass pull, increase in rougher concentrate grade and similar rougher recovery confirmed over-collection had been occurring. Subsequently, cleaner upgrading of the concentrate was more efficient.

Although the best bulk and zinc results did not occur in the same single test, enough information was collected to proceed with locked cycle testing on the Main Zone master composite.

13.5.4.2 Tuff Zone Cleaner Flotation

The approach taken for the Tuff Zone cleaner tests was similar to the approach mentioned above for the Main Zone; rougher conditions were used for the rougher stages and historical conditions were used for the cleaner stages. As with the Main Zone, the Tuff Zone master composite exhibited difficulty upgrading in the bulk cleaner circuit. It was again suspected that over-collection had been happening in the rougher circuit. However, while the results with decreased collector improved in a similar fashion to the Main Zone, there was still significant difficulty upgrading. Marginally improved results led to the conclusion that the variability samples (which comprised the master composite) would need to be evaluated independently.

13.5.4.3 Pinwheel Zone Cleaner Flotation

Although the rougher flotation results for the Pinwheel Zone did not produce significant upgrading and a very large mass pull in the bulk rougher was observed, several cleaner tests were conducted. Attempts to improve performance, including finer primary grind size, decreased collector dosage and increased cleaner pH (to suppress pyrite), were all unsuccessful. As a result, it was determined that the individual variability composites making up the master composite should be tested individually as well as submitted for mineralogy, with the goal of identifying the poor-performing constituents.

13.5.4.4 Pinwheel Variability Composite Mineralogy

Due to poor flotation performance of the Pinwheel master composite, the variability composites (four), which made up the master composite were submitted for mineralogy. The variability composites were designated VC-12, VC-13, VC-15 and VC-25. Additionally, two test products (copper and zinc concentrates) from one of the Pinwheel master composite tests were also submitted for mineralogy.

Table 13.41 displays the modals, or mineral masses, contained in the six samples evaluated, while Table 13.42 presents the degree of liberation of these samples.

TABLE 13.41
PINWHEEL VARIABILITY COMPOSITE HEAD AND CONCENTRATE MODAL MINERALOGY

Sample		VC-12	VC-13	VC-15	VC-25	Cu Conc.	Zn Conc.
Fraction		**-300/+3 µm**	**-300/+3 µm**	**-300/+3 µm**	**-300/+3 µm**	**-300/+3 µm**	**-300/+3 µm**
Mass Size Dist. (%)		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
Mineral Mass (%)	Pyrite/Marcasite	74.7	83.2	33.3	66.6	63.5	53.2
	Sphalerite	12.9	0.1	0.2	0.3	2.3	35.5
	Chalcopyrite	9.1	9.2	1.4	2.3	27.9	8.6
	Quartz	0.3	0.2	23.7	12.4	4.4	0.3
	Sericite/Muscovite	0.2	0.8	29.6	13.8	0.0	0.1
	Talc	0.1	4.0	0.4	0.1	0.5	0.3
	Chlorite/Clays	0.2	0.6	3.7	1.3	0.1	0.1
	Other	2.6	2.0	7.7	3.2	1.3	2.1
Total		**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

TABLE 13.42
PINWHEEL VARIABILITY COMPOSITES AND CONCENTRATES -
CU-SULPHIDE AND PYRITE LIBERATION

Sample	VC-12	VC-13	VC-15	VC-25	Cu Conc.	Zn Conc.
Free + Liberated Cu-Sulphide	67.3	41.5	81.0	80.2	66.7	44.3
Midds and Sub-Midds Cu-Sulphide	22.5	35.0	12.3	11.1	27.9	41.2
Locked Cu-Sulphide	10.2	23.5	6.7	8.7	5.4	14.5
Total	100.0	100.0	100.0	100.0	100.0	100.0
Free + Liberated Pyrite	93.1	92.1	96.6	97.7	81.7	75.8
Midds and Sub-Midds Pyrite	6.5	7.6	2.8	2.0	17.6	23.0
Locked Pyrite	0.4	0.3	0.6	0.3	0.7	1.2
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Table 13.42 indicates significant mineralogical differences among the samples of the Pinwheel Zone, and further metallurgical type distinctions are likely required. For example, the copper and gangue mineral proportions as well as copper sulphide and pyrite liberations were similar between VC-15 and VC-25, and as such the optimal flotation conditions to allow efficient

separation will likely be similar for the two materials. Also, only one of the samples contained appreciable levels of sphalerite.

For the concentrates from the master composite cleaner test, pyrite was the main dilutant in both cases. While there was some dilution from the other mineral of interest in each concentrate, there was over 63% pyrite present in the bulk concentrate and over 53% in the zinc concentrate. Thus, strategies to target pyrite rejection will be required, most likely across all sub-domains of the Pinwheel mineralization.

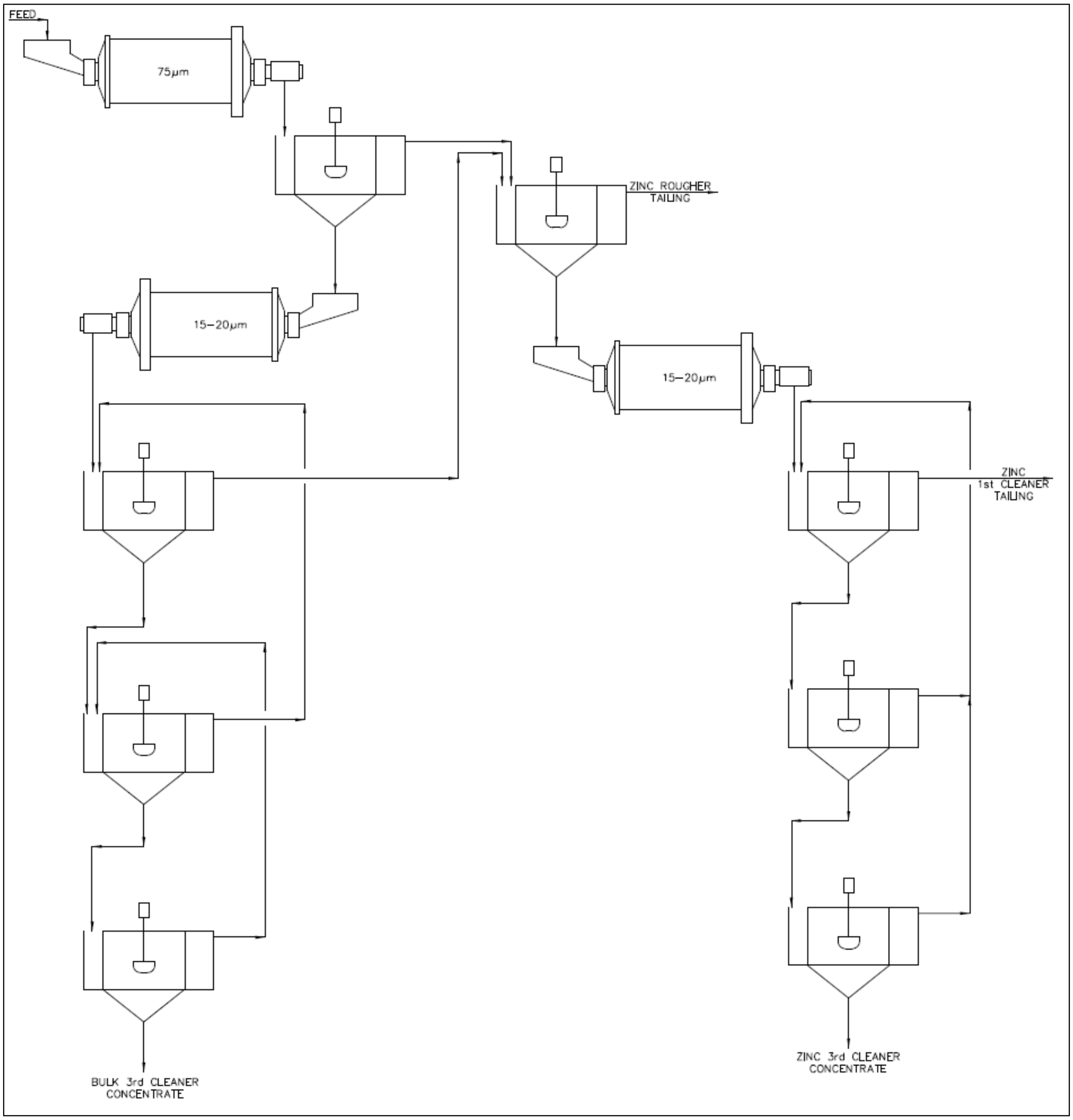
The degree of liberation of copper sulphides was relatively high in three of the four samples studied, especially for VC-15 and VC-25, and the liberation of pyrite was high in all cases including the concentrates, at >90% free and liberated. This indicates that efficient separation of pyrite from copper sulphides should be achievable from a mineralogically limiting perspective.

13.5.4.5 Master Composite Locked Cycle Testing

As described above, both the Tuff Zone and Pinwheel Zone master composites were deemed not suitable to advance to locked cycle testing. However, the Main Zone master composite was subjected to one locked cycle test. The flowsheet for the locked cycle tests is depicted in Figure 13.16. Primary grind and regrind sizes are quoted as 80% passing size.

In the five cleaner tests discussed previously, both positive and negative results were observed in both the copper and zinc stages. However, positive results in both circuits did not occur in the same open circuit test. The results of the locked cycle test, which simulates recirculating streams, were very positive. Copper recovery was 84% at a grade of over 25% Cu and zinc recovery was 95% at a grade of almost 55% Zn.

FIGURE 13.16 LOCKED CYCLE FLOTATION FLOWSHEET



Source: *Aquila Resources Inc. (2020)*

13.5.4.6 Variability Composite Cleaner Testing

All of the variability composites for each of the three metallurgical types were subjected to cleaner flotation testing. There were two main purposes of the tests: 1) to examine in greater depth the flotation responses of the individual samples making up each master composite in order to determine which of the individual samples were particularly problematic and to further explore optimization strategies; and 2) to understand the metallurgical responses over a range of samples.

14 composites were tested: seven Main Zone, four Pinwheel Zone and three Tuff Zone. Additional composites of interest were also tested. The head assays of these composites are presented in Table 13.31.

13.5.4.7 Main Zone Variability Cleaner Flotation

Seven Main Zone variability samples that comprised the master composite were tested. Because the master composite performed relatively well, these tests focussed on optimization. The samples tested were composites 9, 10, 11, 18, 19, 26 and 27.

A total of 13 tests were run on the seven samples, with no sample being tested more than three times. Samples with appreciable head grades of the target minerals performed comparably to the master composite.

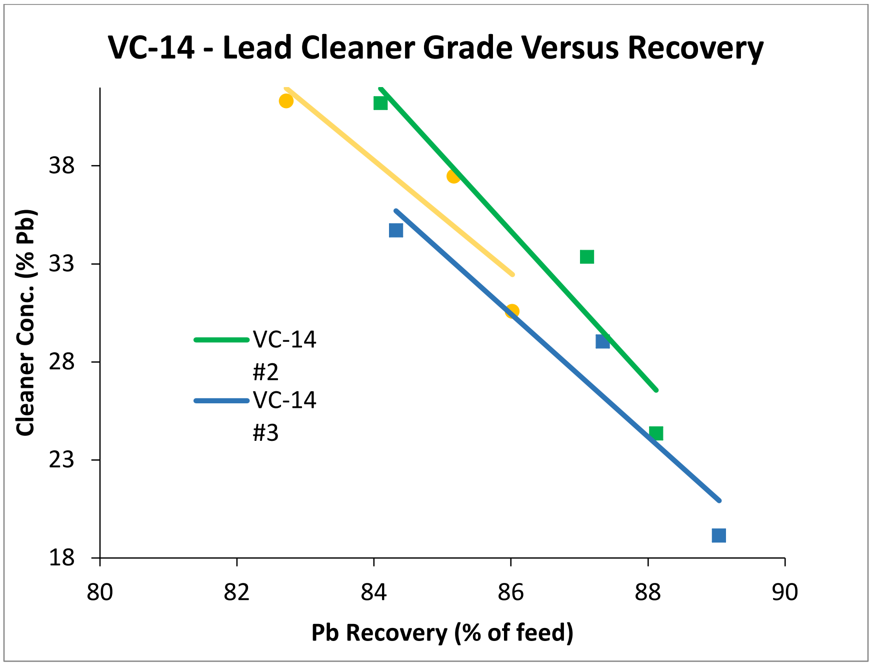
13.5.4.8 Tuff Zone Variability Cleaner Flotation

Three Tuff Zone variability samples that comprised the master composite were tested. The samples tested were composites 14, 16 and 17.

Historical data on testing on this zone, although on a limited number of high-grade samples, indicated that a lead concentrate grade of over 50% Pb was achievable and as such that was the initial target of the 2016 campaign. However, it became obvious that concentrate grades for these samples (which were deemed more representative of the zone) were not able to achieve reasonable recoveries at concentrate grades over 50%. As a result, a lower target concentrate grade of 30-40% Pb was set.

The focus of the Tuff Zone cleaner flotation testing became improving upon the grade recovery relationship as much as possible by varying the test conditions, such as primary and regrind targets and reagent schemes. Figure 13.17 illustrates three of the various curves for composite 14 with respect to lead.

FIGURE 13.17 GRADE RECOVERY RELATIONSHIP FOR COMPOSITE 14



Source: *Aquila Resources Inc. (2020)*

A similar series of tests was conducted on all three composites.

Zinc performance in composites 14 and 16 was positive (50% Zn concentrate grades with 80% zinc recovery). Composite 17 did not contain appreciable zinc.

In an effort to minimize the number of distinct metallurgical types from a processing perspective, composites 14 and 16 were blended at a ratio understood to be representative of that within the Tuff Zone. This blend was understood to have represented the massive sulphide and high lead grade stringers. Results indicated that processing a blend of these samples was possible. Details are shown in Table 13.43.

TABLE 13.43
TUFF ZONE BLENDED COMPOSITE PERFORMANCE

Composite	Pb Recovery at 35% Pb	Zn Recovery at 50% Zn
14	86.7	81.7
16	70*	85*
Blend – Expected	72.7	85.0
Blend - Actual	72.6	79.9*

* *extrapolated value*

In addition to the testing described above, three other samples were examined. One sample was a composite, namely composite 28, and the other two were individual core samples. Composite 28 was a ‘transitional’ sample and was composed of core that was along the boundary between the Tuff and oxide zones. Considering the composite’s low lead head grade (0.26% Pb), it responded reasonably well to flotation. A concentrate grade of over 33% Pb was achieved with over 66% lead recovery. The two core samples, which were each subjected to one test, achieved lead grades around 30% Pb with lead recoveries around 75%.

13.5.4.9 Pinwheel Zone Variability Cleaner Flotation

The four Pinwheel composites that comprised the master composite were subjected to a total of fourteen cleaner flotation tests. Variability composite 12 represented met type 8 (as listed in Table 13.1), composite 13 represented met types 2 and 7 and composites 15 and 25 represented met type 3.

Mineralogy on composite 12 showed that liberation of Cu-sulphide minerals (liberated + free) was only around 67%. The main dilutant was pyrite. This indicates that this material will require a finer primary grind size and a production of a rougher concentrate that maximizes copper recovery but minimizes pyrite content, which would negatively impact cleaner flotation performance. A total of six tests was run on this composite. Variables explored included primary grind size, depressant dosage, collector dosages, flotation time and bulk cleaner pH. From these results, a primary grind size target P_{80} of 55-60 µm was established. The best test data yielded a copper concentrate of over 20% Cu at over 80% copper recovery. Zinc performance was difficult to evaluate in open circuit because a substantial amount of zinc reported to the bulk circuit. However, a zinc scavenger stage was performed on all bulk 1st cleaner tailings, with stage zinc recovery of over 90%, on average.

Like composite 12, mineralogy on composite 13 indicated that Cu-sulphide liberation was going to limit flotation performance. Liberation of Cu-sulphides in composite 13 was only 41% (free and liberated), and over 23% were locked (not recoverable). For the five tests completed on this composite, primary grind size, depressant dosage, collector dosages, flotation time and bulk cleaner pH were explored. The results were not as positive as with composite 12. The best result yielded a bulk concentrate grade of just over 16% Cu with a copper recovery of around 76%. Composite 13 was considered zinc barren.

Composites 15 and 25, representing the semi-massive sulphide and stringers portion of the Pinwheel Zone (met type 3), responded much better to flotation than the massive sulphide composites 12 and 13. Not only did composites 15 and 25 contain less pyrite, but liberation of Cu-sulphide minerals was higher, at around 81% for both samples. Both samples achieved over 20% Cu concentrate grades and 75% copper recoveries in the first test. These composites were considered to be zinc barren.

13.5.5 Variability Composite Locked-Cycle Testing

Five of the variability composites were subjected to locked cycle testing, of which two were from the Main Zone, two were from the Tuff Zone and one was from the Pinwheel Zone.

13.5.5.1 Main Zone Variability Composite Locked Cycle Flotation

The two Main Zone composites selected for locked cycle testing were composites 9 and 10. It is understood that these two samples were selected due to the fact that they were the only composites that contained appreciable amounts of both copper and zinc. Table 13.44 displays the results of these locked cycle tests as well as a comparison to batch cleaner test performance.

In all cases (i.e., both circuits of both composites), locked cycle test results were superior to the comparable open circuit test results. Note that the zinc head grade of composite 10 was less than 0.4% Zn, which likely explains the low zinc grade of the zinc concentrate.

TABLE 13.44
MAIN ZONE VARIABILITY COMPOSITE LOCKED CYCLE FLOTATION RESULTS

Sample	Product	Test	Assays				Distribution			
			Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (%)	Ag (%)
VC -9	2nd Cu Cl. Conc.	Batch	22.7	5.5	145	418	57.7	0.4	49.8	27.2
	3rd Cu Cl. Conc.	LCT	21.9	5.57	118	482	74.6	0.4	52.9	34.7
	2nd Zn Cl. Conc.	Batch	0.18	55.4	1.57	15.1	11.0	89.4	10.2	23.6
	2nd Zn Cl. Conc.	LCT	0.18	54.2	0.72	14.6	9.2	96.8	7.3	23.9
VC-10	2nd Cu Cl. Conc.	Batch	29.6	1.2	53.7	343	84.9	9.2	48.0	51.9
	3rd Cu Cl. Conc.	LCT	28.1	0.85	48.5	311	86.0	7.4	49.8	54.6
	1st Zn Cl. Conc.	Batch	1.8	29.7	18.9	118	1.7	76.3	5.5	5.8
	2nd Zn Cl. Conc.	LCT	2.56	32.9	15.1	136	10.1	86.3	4.6	7.2

13.5.5.2 Tuff Zone Variability Composite Locked Cycle Flotation

Composites 14 and 16 from the Tuff Zone samples were selected for locked cycle testing. It is understood that these two samples were selected because they contained appreciable amounts of both lead and zinc and also because they represented both the massive sulphide and stringer domains within the zone. Table 13.45 displays the results of the locked cycle tests as well as a comparison to batch cleaner test performance.

TABLE 13.45
TUFF ZONE VARIABILITY COMPOSITE LOCKED CYCLE FLOTATION RESULTS

Sample	Product	Test	Assays				Distribution			
			Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Au (%)	Ag (%)
VC -14	3rd Pb Cl. Conc.	Batch	41.2	6.4	120	3211	84.1	3.4	73.6	58.0
	3rd Pb Cl. Conc.	LCT	50.0	4.9	231	2510	88.1	2.3	85.9	36.0
	2nd Zn Cl. Conc.	Batch	0.5	51.2	6.0	159	2.8	82.1	11.3	8.8
	2nd Zn Cl. Conc.	LCT	0.5	47.5	3.6	795	4.0	93.0	5.5	47.2
VC-16	3rd Pb Cl. Conc.	Batch	28.1	7.2	26.0	562	71.6	3.1	60.0	52.0
	3rd Pb Cl. Conc.	LCT	29.7	5.8	28.7	52.2	78.8	2.3	65.7	55.1
	2nd Zn Cl. Conc.	Batch	0.5	48.8	1.2	27.0	4.9	85.7	10.8	9.9
	2nd Zn Cl. Conc.	LCT	0.3	47.7	0.54	52.2	4.2	95.6	6.4	27.9

In all cases, locked cycle test results were superior to the comparable open circuit test results. In the case of the lead circuit, both concentrate lead grade and lead recovery were better in closed circuit, while for the zinc circuit the concentrate decreased somewhat to the benefit of zinc recovery.

13.5.5.3 Pinwheel Zone Variability Composite Locked Cycle Flotation

From the Pinwheel Zone, composite 12 was chosen to be subjected to locked cycle testing. This was largely because it was the only sample that contained appreciable amounts of both copper and zinc. Results from this test are displayed in table 13.46 and are compared to one of the batch tests performed on the same sample.

Results for the copper circuit were similar in both locked cycle mode and batch mode, with production of a low copper grade bulk concentrate at good copper recovery. Future emphasis should be placed on improving concentrate grade to enable marketability. Of note is that the regrind size in the locked cycle test was coarser than target, and this could be investigated further as a cause of poor concentrate quality.

Zinc metallurgy in locked cycle mode was considerably better than that in open circuit testing.

TABLE 13.46
PINWHEEL ZONE VARIABILITY COMPOSITE LOCKED CYCLE FLOTATION RESULTS

Sample	Product	Test	Assays				Distribution			
			Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (%)	Ag (%)
VC -12	1st Cu Cl. Conc.	Batch	13.4	7.4	4.5	128	87.4	18.8	66.8	76.7
	3rd Cu Cl. Conc.	LCT	13.1	5.4	3.5	114	89.8	14.5	65.7	68.7
	2nd Zn Cl. Conc.	Batch	1.5	43.6	0.8	36.4	6.1	71.4	8.3	12.8
	2nd Zn Cl. Conc.	LCT	1.1	47.5	0.5	29.4	4.9	81.6	5.9	11.3

13.5.6 Additional Variability Composite Testing

In 2017, Aquila identified that the majority of the samples tested previously, as discussed above, were either relatively high or relatively low grade. As such, additional flotation testwork was completed with the goal of evaluating the metallurgical performance of samples at various grades for all metallurgical types.

13.5.6.1 Main Zone Additional Variability Composite Flotation

The additional testing on the Main Zone material targeted samples of various zinc head grades in the range not previously studied. Six composites and six individual samples were tested in open circuit mode. Each baseline batch cleaner flotation test targeted a primary grind size P_{80} of <75 µm and regrind size P_{80} of <20 µm and consisted of bulk and zinc rougher stages of 4-8 minutes each, 3 stages of bulk circuit cleaning, and 2 stages of zinc circuit cleaning.

The tests on Main Zone composites 1 and 4 struggled to reach the bulk concentrate grade target of 25% Cu. This was likely due to the very low head grades of 0.08% Cu of those composites. The other composites tested performed well with respect to copper metallurgy.

Only Main Zone composite 3 did not achieve a grade of >48% Zn at >75% zinc recovery in the final zinc concentrate. This composite contained a low to medium zinc head grade of 2.01% Zn.

In addition to batch flotation, one composite was subjected to a locked cycle test. The sample chosen had head grades that were previously not evaluated in locked cycle mode. The comparison of the open circuit test results with the locked cycle test results for this composite (M1 – C3) is displayed in Table 13.47.

TABLE 13.47
MAIN ZONE ADDITIONAL FLOTATION COMPARISON

Sample	Product	Test	Assays				Distribution			
			Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (%)	Ag (%)
M1 – C3	2nd Cu Cl. Conc.	Batch	26.7	1.0	59.5	287	66.1	0.7	44.7	37.2
	3rd Cu Cl. Conc.	LCT	25.5	4.29	52.1	275	83.4	2.0	50.2	41.5
	2nd Zn Cl. Conc.	Batch	0.3	43.2	0.82	28.0	4.5	83.8	3.3	19.4
	2nd Zn Cl. Conc.	LCT	0.4	41.0	0.82	25.9	5.7	90.7	3.7	18.2

Table 13.47 demonstrates the increase metallurgical performance from the batch test to the locked cycle test. Copper recovery improved by over 17% and zinc recovery improved by almost 7%, at similar concentrate grades.

13.5.6.2 Tuff Zone Additional Variability Composite Flotation

The Tuff Zone, or met type 5, had six distinct (and one blended) samples that were tested prior to 2017. The Aquila Resources gap analysis showed that there was a lack of zinc data in the 0.2 – 1.6% Zn range of head grades and that there were gaps in lead head grades as well.

In total, seven composites and six individual samples were subjected to batch flotation tests, and one composite was also tested in locked cycle mode. A comparison of the locked cycle versus batch test results for Tuff Zone composite 1.1 is displayed in Table 13.48.

TABLE 13.48
TUFF ZONE ADDITIONAL FLOTATION COMPARISON

Sample	Product	Test	Assays				Distribution			
			Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Au (%)	Ag (%)
M5 – C1.1	2nd Pb Cl. Conc.	Batch	40.7	3.7	47.8	685	59.4	0.9	60.4	34.9
	3rd Pb Cl. Conc.	LCT	36.7	4.9	31.8	753	87.3	2.1	77.9	71.2
	2nd Zn Cl. Conc.	Batch	0.2	48.7	0.5	21.0	1.5	87.0	4.4	7.4
	2nd Zn Cl. Conc.	LCT	0.4	51.5	0.6	32.6	4.3	96.3	6.1	13.7

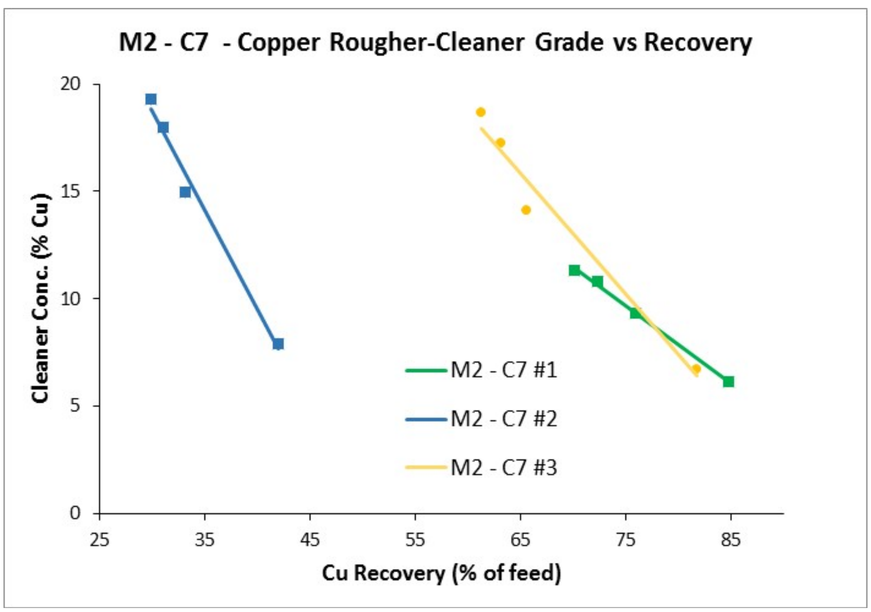
Table 13.48 demonstrates a significant improvement in metallurgical performance from the batch test to the locked cycle test. Although lead concentrate grades were somewhat different, interpolation of the batch test data to the equivalent concentrate grade indicates a lead recovery in the low 60 percent range, resulting in an increase of over 20%. Even at a higher concentrate grade, zinc recovery increased by over 9%.

13.5.6.3 Pinwheel Zone Additional Variability Composite Flotation

The majority of the testwork completed during the additional variability testing campaign was done on Pinwheel Zone material. As previously stated, it is understood that the Pinwheel Zone was divided into three distinct metallurgical types. These three metallurgical types were the Pinwheel massive sulphides Cu rich (met type 2 + 7), Pinwheel semi-massive and stringers (met type 3), and Pinwheel massive sulphide Cu-Zn (met type 8). A total of 20 composites were prepared for this phase of the Pinwheel Zone material testing.

Metallurgical type 2 + 7 had an additional nine composites with a wide range of copper head grades tested during the additional variability testing campaign. These nine samples are understood to have represented the same metallurgical type as variability composite 13 from the 2016 testwork program. These samples generally behaved similarly to VC-13, with challenging metallurgy, likely due to similar reasons such as high pyrite content and poor liberation of Cu-sulphide minerals. Twenty-six batch tests were completed on these samples. An example of the optimization results is depicted in Figure 13.18, with respect to copper.

FIGURE 13.18 GRADE RECOVERY RELATIONSHIP FOR M2 – C7



Source: *Aquila Resources Inc. (2020)*

Figure 13.18 illustrates the significant impact that relatively minor condition changes had on metallurgical type 2 samples. The first test was conducted at the coarsest primary (P_{80} of 65 µm) and regrind (P_{80} of 26 µm) sizes as well as with the highest collector dosage. It did not achieve a suitable concentrate copper grade. The second test utilized finer primary grind (P_{80} of 32 µm) and regrind (P_{80} of 13 µm) and utilized less collector, thereby generating a higher concentrate copper grade but at the expense of copper recovery. The final test was intermediate in terms of grind size and collector use, and also increased the pH in the rougher circuit, which ultimately generated the best results. These samples were low in zinc grade and therefore the zinc circuit was not operated.

Metallurgical type 3 contains low-grade copper, so six composites, with head grades ranging from 0.23 – 0.81% Cu, were each subjected to one batch flotation test. All but one composite achieved a bulk concentrate grade over 20% Cu and over 70% copper recovery, which is in line with the only other two composites representing this metallurgical type previously tested, VC-15 and VC-25. These samples were low in zinc grade and therefore the zinc circuit was not operated.

Five samples representing metallurgical type 8 were subjected to a total of eight batch tests. The corresponding variability composite from 2016 was VC-12 and these five samples generally outperformed VC-12. Copper grades were considerably higher, at somewhat reduced copper recoveries, and in four of the samples, zinc concentrate grades were higher.

Two locked cycle tests were conducted on the Pinwheel sub-domains. The first composite tested was a master composite of met type 2. The second composite tested was one from met type 8, namely, M8 – C2. A comparison of their performances relative to that of their respective batch tests is shown in Table 13.49.

TABLE 13.49
PINWHEEL ZONE ADDITIONAL FLOTATION COMPARISON

Sample	Product	Test	Assays				Distribution			
			Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (%)	Ag (%)
M2 – MC	3rd Cu Cl. Conc.	Batch	18.1	-	9.1	235	60.5	-	43.0	5.7
	3rd Cu Cl. Conc.	LCT	17.1	-	8.1	239	77.8	-	48.9	53.2
M8 – C2	2nd Cu Cl. Conc.	Batch	23.8	4.8	16.7	338	63.1	2.4	27.0	26.9
	3rd Cu Cl. Conc.	LCT	23.9	6.9	18.1	342	76.7	4.2	33.0	35.3
	2nd Zn Cl. Conc.	Batch	1.0	50.0	1.3	51.7	8.7	85.9	7.3	14.1
	2nd Zn Cl. Conc.	LCT	0.9	53.9	1.7	71.8	7.5	90.0	8.5	20.1

Both the copper concentrates, of similar final concentrate grade as in the respective batch test, had much higher copper recoveries under locked cycle conditions, with an average increase of 15%. Only the M8 – C2 composite contained appreciable zinc, which improved in both grade and recovery by 4%.

13.5.7 Regrind Size Selection

In general, the target regrind size used in the most recent phase of testing for both copper and zinc rougher concentrate was 15 – 20 µm.

Although not quantifiably tested historically, the general trend was a positive shift in the grade recovery relationship of a given composite that was subjected to a finer regrind (both bulk and zinc). This was further confirmed by aforementioned mineralogical data that in general showed that the degree of free and liberated Cu-Sulphate, galena and sphalerite increased with decreasing particle size. This regrind target range was deemed suitable in consideration of grinding effort and the need to minimize overgrinding of the cleaner feed.

13.5.8 Zinc Regrind Bypass

A series of 14 zinc rougher kinetic tests were conducted on a total of seven samples; a Main Zone master composite, three Main Zone variability composites, two Tuff variability composites and one Pinwheel variability composite. The intention of the tests was to demonstrate that a significant portion of the zinc rougher concentrate did not require regrinding and subsequent cleaner flotation. Results are displayed in Table 13.50.

TABLE 13.50
ZINC ROUGHER KINETICS

Sample	Head Grade (% Zn)	Recovery (% Zn)	Rougher Concentrate Grade (% Zn)
MZ – MC	5.53	91.4	52.3
VC – 9	11.9	83.6	54.7
VC – 10	0.37	-	-
VC - 19	5.13	71.6	47.7
VC – 12	7.07	62.9	51.2
VC – 14	2.99	76.1	46.3
VC - 16	2.46	59.8	29.6

As can be seen above, the samples with the five highest head grades achieved zinc recoveries of 71.6% to 91.4% at a rougher concentrate grade range of 46.3% to 54.7% Zn. The grades and recoveries reported from either the first zinc rougher stage or the first plus the second zinc rougher stage.

Additionally, the Main Zone master composite sample was subjected to a test where the remaining rougher concentrate (stages three to five) were subjected to regrinding and cleaner flotation for further upgrading. The resulting final concentrate (first and second roughers plus cleaner product) achieved 95.6% zinc recovery at a grade of 52.0% zinc.

13.5.9 Concentrate Minor Elemental Analysis

Batch and locked cycle test concentrates were submitted for minor element analysis. Concentrates from both the 2016 and 2017 testwork campaigns were included. Table 13.51 presents the average of the data collected by concentrate for the three metallurgical types (Main, Pinwheel and Tuff).

TABLE 13.51
SUMMARY OF CONCENTRATE MINOR ELEMENTAL ANALYSIS

Met Type	Conc	Assay (%)								Assay (ppm)							
		As	Cl	Fe	MgO	Pb	Sb	SiO_2	Zn	Bi	Cd	Cl + F	Co + Ni	F	Hg	Se	Te
Main	Cu	0.01	0.001	-	-	2.00	0.02	-	2.29	<200	56	-	-	50	30	<30	<100
Pinwheel	Cu	0.10	0.002	-	-	0.49	0.04	-	2.53	<30	135	-	-	<50	78.4	<30	<100
Main	Zn	-	-	11.1	0.05	0.06	-	1.07	-	-	2,900	<40	<40	-	252	<30	-
Tuff	Zn	-	-	8.01	0.14	0.41	-	5.85	-	-	2,360	<120	<44	-	420	<30	-
Pinwheel	Zn	-	-	10.6	0.03	0.02	-	0.70	-	-	2,700	<30	<27	-	319	<30	-
Tuff	Pb	0.08	-	-	-	-	0.24	-	4.86	<20	-	<180	-	-	64.4	-	-

Note: *Met = metallurgical, Conc = concentrate.*

13.6 OXIDES TESTWORK

13.6.1 Introduction

The objectives of the oxide testwork program were as follows:

- To test various sub-domains within the metallurgical type.
- Determine suitable leach conditions.
- Acquire downstream data (oxide tailings filtration, SART).

The approach taken was to subject all oxide sub-domains which made up the ultimate master composite to varying conditions in a series of bottle roll tests.

The main test conditions that were explored were primary grind size, cyanide concentration and oxygen addition. Other conditions examined were the addition of lead nitrate, test pH and leach time. Also scoped was the leaching of flotation tailings as well as alternative recovery methods, namely CIL and CIP.

Once universal test conditions were determined, a master composite was created by blending the representative sub-domains by what is understood to have been their appropriate in-situ proportions. A larger bulk leach was performed on the master composite to generate a global overall expected recovery as well as enough product to perform filtration testing and SART testing.

Table 13.52 highlights the samples from Table 13.31 that are understood to be considered as part of the oxide metallurgical type.

TABLE 13.52 OXIDE SAMPLES									
Composite	**Zone Description**	**Met Type**	**Assays**						
			Cu (%)	**Pb (%)**	**Zn (%)**	**S (%)**	**Au (g/t)**	**Ag (g/t)**
1	Near Surface Low Sulphur	6	0.02	<0.01	0.01	1.51	3.22	30.6
2	Pinwheel Gossan Low Sulphide	6	0.09	0.03	<0.01	0.45	5.42	14.8
3	East Gossan	6	0.01	0.00	0.02	0.23	14.7	6.33
4	East Oxide	6	0.03	<0.01	0.18	5.21	0.74	<10
5	90 Zone Low Sulphide	6	0.03	0.12	0.25	3.27	1.81	23.1
6	Porphyry Low Sulphur	6	0.07	0.31	1.67	5.49	3.97	24.4
23.1 (70%A, 30%B)	Pinwheel Gossen Cu Rich and Transitional	Trans	1.82	0.05	0.01	12.5	6.35	422
24	Near Surface Quartz-Feldspar Porphyry (QFP)	6	0.01	0.01	0.92	0.43	5.15	130
28	90 Zone Flotation	6	0.02	0.26	0.35	3.85	1.42	38.7

Note: *Met = metallurgical.*

13.6.2 Bottle Roll Tests

A total of 72 bottle roll tests were completed to study the conditions mentioned above. In all cases, sub-samples were taken and conditions maintained at 2, 6, 24 and 48 hours (except for 72 hour tests). All tests were performed on 1 kg samples, with the exception of one test which was performed on a 20 kg sample to generate samples for downstream testwork. A summary of the results at the selected standard conditions is displayed in Table 13.53.

TABLE 13.53 SUMMARY OF CYANIDATION BOTTLE ROLL TEST RESULTS						
Composite	**Extraction (%)**		**Residue (g/t)**		**Head Grade (g/t)**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**
1	97	91	0.09	3.0	3.27	32.5
2	92	68	0.57	6.0	6.74	14.8
3	94	55	0.83	2.1	14.0	4.64
4	72	56	0.22	1.9	0.78	4.36
5	85	48	0.22	11.8	1.45	22.8
6	91	45	0.32	14.6	3.63	26.6

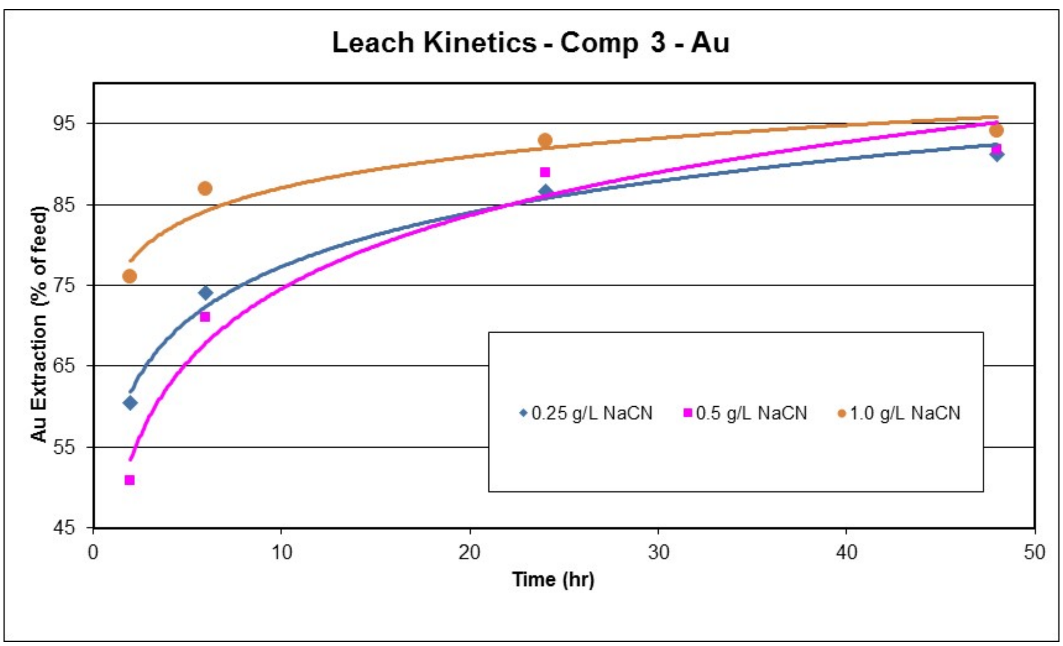
TABLE 13.53 SUMMARY OF CYANIDATION BOTTLE ROLL TEST RESULTS						
Composite	**Extraction (%)**		**Residue (g/t)**		**Head Grade (g/t)**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**
24	97	85	0.15	21.3	5.38	142
28	86	39	0.26	22.9	1.86	37.2
Master Comp.	92	64	0.30	13.1	3.55	36.3

13.6.2.1 Cyanide Concentration

Eight of the oxide samples were subjected to a bottle roll test at three different cyanide concentrations. The concentrations were 0.25, 0.5 and 1.0 g/L NaCN.

Figure 13.19 is an example of the trend of the leaching kinetics generally observed across the oxide samples. While the advantage with respect to gold extraction kinetics of higher NaCN concentration was more pronounced in the earlier stages of testing than the final hours, there was nonetheless a measurable increase in both gold and silver recoveries at the end of the test. It was determined that although more NaCN would be consumed at a concentration of 0.5 g/L NaCN and higher, somewhat higher recoveries of gold and silver would be expected.

FIGURE 13.19 LEACH KINETICS AT VARYING NACN CONCENTRATIONS



Source: *Aquila Resources Inc. (2020)*

13.6.2.2 Primary Grind Size

The same eight samples that were part of the NaCN concentration evaluation were subjected to bottle roll tests at varying primary grind sizes. Four samples did not leach significant additional precious metals at the finer grind sizes tested. The other four samples, representing over 46% of the master composite, did achieve higher recoveries at finer grind sizes.

13.6.2.3 Oxygen Addition

All eight oxide samples were subjected to bottle roll tests using both air and oxygen for aeration while keeping all other conditions constant. In addition, a master composite (comprised of all eight samples) was tested using both air and oxygen.

For gold, there was no improvement in recoveries while using oxygen. Silver showed some modest recovery improvements when using oxygen for aeration. Oxygen addition was not pursued further.

13.6.2.4 Other Conditions

Other conditions that were scoped included the addition of lead nitrate, test pH and test duration.

Two samples were subjected to bottle roll tests using lead nitrate as an additional reagent to evaluate if the material responded with increased gold dissolution. Gold recoveries in both cases were identical to those tests without the use of lead nitrate.

Two bottle roll tests were completed to determine the effect, if any, of lowering the maintained pH during the test. The results for the two tests were inconclusive.

A further two tests were completed using a leach time of 72 hours (versus 48 hours for all other tests). Both tests recovered slightly more gold but no additional silver.

13.6.3 Leach Product Filtration (Via Vacuum Filtration)

A 20 kg oxide master composite was blended from the individual samples that represented the sub-domains of the oxide mineralized material and was subjected to a bottle roll test. Unlike previous bottle roll tests, this test utilized a synthetic solution which was modelled to represent oxide process water. This test was completed to generate enough samples for downstream testwork (i.e., filtration, SART testing). The cyanidation bottle roll product was split into multiple charges to conduct filtration testing.

In order to establish filtration parameters, initial tests were conducted on the cyanidation leach product. First, tests were conducted to determine filter cloth and flocculant selection. Once the appropriate filter cloth and flocculant (dosage of 20 g/t) was selected, tests to optimize cake loading were conducted. The cake/solids loading was determined to be approximately 19 kg/m^2 which resulted in a cake thickness of 12 mm and an ultimate cake moisture of approximately 20%.

Subsequent testing was conducted using established parameters described above with the intent of evaluating metal recoveries (wash efficiency) and required wash ratios. To achieve this, three filtrations on the leaching product were conducted; initial filtration, 1st wash with barren leach solution and 2nd wash with synthetic process water. Almost all wash efficiencies were >97% with a wash ratio of approximately 0.8 for the 1st wash and a range of 0.4 - 0.8 for the 2nd wash.

13.6.4 SART (Sulphidization, Acidification, Recycling, Thickening)

SART testing was conducted on pregnant leach solution (PLS) from the above described filtration tests. The primary driver for selecting the SART process was the removal of copper from final doré product. Table 13.54 displays the benefit of SART over an alternative PLS treatment, namely Merrill-Crowe precipitation.

TABLE 13.54
MODELLED DORÉ QUALITY

Flowsheet	Doré Quality (wt.%)		
	Au	Ag	Cu
Merrill-Crowe	7.5	32.7	50.6
SART on PLS	96.0	0.4	3.5

It should be noted that copper, silver, mercury and other elements that are removed via SART in precipitate form as a by-product concentrate of the process.

13.6.5 Cyanide Destruction

The objective of the cyanide destruction testwork was to reduce the CN_{WAD} content of the final MC-3 barren solution to less than 1 mg/L to satisfy environmental requirements. Cyanide destruction was conducted in a single batch stage to establish basic operating parameters and then in a series of continuous tests varying certain parameters to achieve the objective and optimize the conditions. The sample used for this testing was the product of a bottle roll test that was subjected to Merrill-Crowe precipitation (to removed precious and base metals). This procedure simulates a worst-case scenario for the cyanide destruction process.

For the initial batch test (CND-1) performed at a feed CN_{WAD} level of 374 mg/L, the CN_{WAD} level was reduced to 0.31 mg/L with reagent dosages of 3.33 grams SO_2 equivalent (added as sodium metabisulphide) and no copper per litre of feed solution.

For the continuous testing the retention times ranged from 55 minutes to 88 minutes, with SO_2 equivalent additions ranging from 1.57 grams to 3.65 grams per litre of feed solution. Copper, zinc and lime additions ranged from no copper to 0.054 grams of copper per litre of feed solution (added as copper sulphate), no zinc to 0.055 grams of zinc per litre of feed solution (added as zinc sulphate) and 0.88 grams of lime to 2.18 grams of lime per litre of feed solution (added as calcium hydroxide).

The varied parameters resulted in CN_{WAD} levels in the cyanide destruction barren solution ranging from 47.5 mg/L to a low of <0.1 mg/L. The only test to achieve a final CN_{WAD} solution level of < 1 mg/L was completed with a retention time of 85 minutes and reagent additions of 3.65 grams of SO_2 equivalent, 0.055 grams of zinc, 2.18 grams of lime and no copper per litre of feed solution.

13.6.6 Tailings Leach Testwork

13.6.6.1 Gravity Recovery and Sulphide Tailings Testwork

A screening level metallurgical test work program was completed in 2019 with the objective of investigating a gravity recovery circuit at the sulphide ball mill and cyanide leaching of sulphide tailings to extract gold (post-base metal recovery). Some historical testwork was completed to investigate leaching sulphide tailings but was not considered feasible at the time.

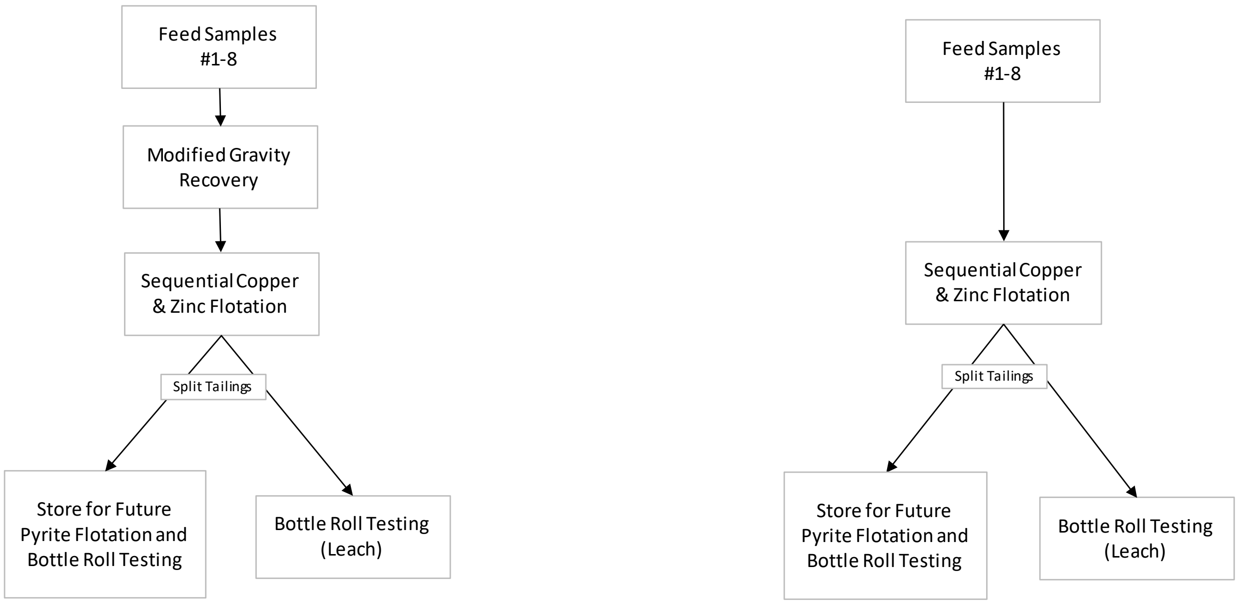
This testing was completed on sulphide material types and a blend of sulphide and oxide material. Two Main Zone, two Pinwheel and one Oxide composite were created. The main drivers behind composite selection were copper and gold grades. Composite head assay data for base and precious metals are summarized in Table 13.55.

TABLE 13.55
COMPOSITE HEAD ASSAYS

Composite	Assay (%, g/t)				
	Cu	Zn	Au	Ag	S
MZ1	0.85	0.09	3.91	< 10	36.90
MZ2	0.34	14.50	2.10	10.90	45.60
PW1	5.26	0.02	3.04	70.30	45.80
PW2	0.80	0.23	1.84	15.90	32.30
Ox	0.03	0.06	7.99	44.24	1.62

Eight samples containing sulphides were subjected to metallurgical recovery processes as per the testing flow chart illustrated in Figure 13.20. A ninth sample (Oxide only) was subjected to a cyanidation bottle roll test (leaching).

FIGURE 13.20 METALLURGICAL TESTING FLOW CHART



Source: *Aquila Resources Inc. (2020)*

Two samples were subjected to gravity recoverable gold ("EGRG") testing and further modeled by FLSmidth. One sample was predominately Main Zone and the other Pinwheel Zone, and both were blended with oxides. Predicted gravity recoverable gold results ranged 7.5-11.6% (refer to Table 13.56).

TABLE 13.56
EGRG MODELLING RESULTS FROM FLSMIDTH

Plant Feed TPH	Composite	Unit Size	Gravity Feed TPH	% Circulating Load Treated	Au Rec. %	Knelson Conc. kg/day
167	MZ + Ox	QS30	125	25.1	10.4	1300
167	PW + Ox	QS30	125	25.1	7.5	1300
100	MZ + Ox	QS30	125	41.7	11.6	1300
100	PW + Ox	QS30	125	41.7	8.6	1300

All samples were subjected to a flotation test and a laboratory gravity test followed by flotation. The intention of this test was to further evaluate the potential of gold recovery via gravity separation and to quantify the impact on the flotation kinetics.

These test results are summarized in Table 13.57 and Table 13.58.

TABLE 13.57
FLOTATION VERSUS GRAVITY + FLOTATION SUMMARY – MAIN ZONE

Test	Product	Weight %	Assay (%, g/t) Cu	Zn	Au	Ag	Distribution (%) Cu	Zn	Au	Ag
	Mozley Con	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MZ1 - 1	Bulk 3rd Clnr Con	2.07	32.3	0.82	39.3	384.0	77.3	15.4	35.4	49.8
No Gravity	Zn 2nd Clnr Con	0.72	5.97	8.11	39.3	239.0	4.98	53.0	12.3	10.8
	Zn Ro Tail	92.9	0.12	0.03	1.06	4.60	12.9	24.5	42.9	26.8
	Head (Calc.)	100.00	0.86	0.11	2.29	15.9	100.0	100.0	100.0	100.0
	Mozley Con	0.20	0.00	0.00	95.1	131.3	0.00	0.00	7.08	1.37
MZ1 - 3	Bulk 3rd Clnr Con	1.94	32.4	0.65	41.5	295.0	76.0	12.7	30.7	30.5
Gravity	Zn 2nd Clnr Con	0.62	11.4	10.7	36.7	320.0	8.60	67.4	8.71	10.6
	Zn Ro Tail	95.5	0.09	0.02	1.18	10.00	10.6	16.4	43.0	50.9
	Head (Calc.)	100.00	0.83	0.10	2.62	18.8	100.0	100.0	100.0	100.0
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MZ2 - 1	Bulk 3rd Clnr Con	0.83	28.4	6.13	179.0	613.0	67.6	0.36	61.0	35.5
No Gravity	Zn 2nd Clnr Con	23.1	0.29	56.3	1.4	19.9	19.18	92.0	13.4	32.0
	Zn Ro Tail	71.5	0.04	1.06	0.75	6.14	8.60	5.36	22.0	30.6
	Head (Calc.)	100.00	0.35	14.1	2.44	14.3	100.0	100.0	100.0	100.0
	Mozley Con	0.27	0.00	0.00	199.4	219.4	0.00	0.00	22.2	4.83
MZ2 - 2	Bulk 3rd Clnr Con	0.51	27.9	4.41	225.0	836.0	43.4	0.16	47.0	34.5
Gravity	Zn 2nd Clnr Con	22.5	54.2	0.52	22.8	33.4	37.9	85.9	4.80	41.7
	Zn Ro Tail	72.1	0.06	2.48	0.80	2.00	12.8	12.6	23.7	11.7
	Head (Calc.)	100.00	0.33	14.2	2.43	12.3	100.0	100.0	100.0	100.0

TABLE 13.58
FLOTATION VERSUS GRAVITY + FLOTATION SUMMARY – PINWHEEL ZONE

Test	Product	Weight %	Assay (%, g/t) Cu	Zn	Au	Ag	Distribution (%) Cu	Zn	Au	Ag
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PW1 - 1	Bulk Ro Con	36.12	8.11	0.07	4.37	116.4	57.1	69.5	51.9	53.0
No Gravity										
	Head (Calc.)	100.00	5.13	0.04	3.04	79.3	100.0	100.0	100.0	100.0
	Mozley Con	0.26	0.00	0.00	210.5	293.0	0.00	0.00	16.4	0.97
PW1 - 2	Moz + 3rd Cl Con	33.93	9.78	0.05	6.08	109.4	64.7	53.9	60.6	46.7
Gravity										
	Head (Calc.)	100.00	5.13	0.03	3.40	79.6	100.0	100.0	100.0	100.0
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PW2 - 1	Bulk 3rd Clnr Con	2.18	28.5	4.85	39.2	222.0	80.2	57.2	49.8	32.6
No Gravity	Zn 2nd Clnr Con	0.23	13.6	18.8	46.0	630.0	3.97	23.0	6.07	9.62
	Zn Ro Tail	90.5	0.06	0.01	0.54	5.00	6.55	6.86	28.5	30.6
	Head (Calc.)	100.0	0.83	0.24	1.89	18.6	100.0	100.0	100.0	100.0
	Mozley Con	0.27	0.00	0.00	44.5	44.7	0.00	0.00	6.34	0.85
PW2 - 2	Bulk 3rd Clnr Con	2.08	28.3	5.28	41.5	233.0	77.4	63.1	45.6	34.3
Gravity	Zn 2nd Clnr Con	0.12	7.32	8.56	45.0	600.0	1.20	6.11	2.96	5.27
	Zn Ro Tail	90.8	0.07	0.02	0.53	5.00	8.23	11.5	25.4	32.1
	Head (Calc.)	100.0	0.80	0.21	2.07	17.6	100.0	100.0	100.0	100.0

Comparing the results with and without gravity separation ahead of flotation, similar trends were observed for both Main Zone composites. For MZ1 and MZ2, the gold reporting to the Mozley concentrate was 7.1 and 22.2%, respectively. Assuming that the gravity concentrate is to be combined with the copper cleaner concentrate, MZ1 achieves a 2.4% increase in gold recovery and MZ2 achieved 8.2% increase in gold recovery. However, both composites had similar gold losses in tailings under the two different tests. Both composites also had a decrease in gold recovered to their respective zinc concentrates. This indicates that most of the gold recovered via gravity concentration was material that would have floated in the zinc circuit. Further investigation would be required to quantify the impact of gravity on the circuit downstream of Main Zone copper flotation based on these results.

Pinwheel composite PW1 had poor metallurgical performance. The results were similar to Pinwheel VC-13 from the 2016 test campaign. Through mineralogy, it was determined that VC-13 had a fine grain association between pyrite and chalcopyrite and contained some talc which can impact flotation performance. It is believed that PW1 has similar characteristics, as talc was visually present in the core sample, and fine grain association is a general issue throughout the open pit Pinwheel Zone. A series of flotation tests altering reagents and test conditions help increase flotation performance, however, limited composite availability prevented further testing.

Despite the above-mentioned issues with PW1, a comparison can still be made between PW1-1 copper rougher concentrate and PW1-2 Mozley + 3rd copper cleaner concentrate as they have similar mass pulls, copper grades and recoveries. PW1-1 gold recovery was 51.9% while PW1-2 gold recovery was 60.5%. Unfortunately, with an open circuit, highly variable mass pulls and very low zinc grades, it is not possible to compare gold final tailings grades.

Pinwheel composite PW2 had much better metallurgical performance. Without gravity recovery, PW2-1 gold recovery to copper concentrate was 49.8%. With gravity recovery, ultimate copper concentrate gold recovery was 51.9%. As with the Main Zone composites, gravity recovery resulted in less gold reporting the zinc concentrate. However, unlike the Main Zone composites, gold to final tailings was over 3% lower in the test with gravity recovery.

In general, there was an increase in gold recovery to the copper concentrate when the composites were subjected to gravity separation. To determine the impact of gravity recovery on the zinc concentrate and global gold recovery, more testing is required. Specifically, locked-cycle testing would provide definitive impacts on all product streams.

As was completed for the sulphide only composites, the four sulphide blended with oxide composites (MZ1+Ox, MZ2+Ox, PW1+Ox and PW2+Ox) were each subjected to two test procedures. One test was flotation using a standard set of conditions established under previous phases of testing. In the other test, the sample was first subjected to one pass through a laboratory scale Knelson concentrator with the concentrate then run over a Mozley table. The Knelson tailings and Mozley tailings were combined and subjected to flotation under the same conditions as the first test. The intent of these two tests on each sample was to evaluate the amount gold that could be removed prior to flotation as well as quantify the impact to gold recovered in flotation concentrates. These test results are summarized in Table 13.59 and Table 13.60.

TABLE 13.59
FLOTATION VERSUS GRAVITY + FLOTATION SUMMARY – MAIN ZONE + OXIDE

Test	Product	Weight	Assay (%, g/t)				Distribution (%)			
		%	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MZ1-Ox - 1	Bulk 3rd Clnr Con	1.66	31.5	1.02	111.0	708.0	85.8	7.66	46.1	67.3
No Gravity	Zn 2nd Clnr Con	1.02	1.09	5.45	21.0	149.0	1.82	25.1	5.35	8.69
	Zn Ro Tail	94.1	0.06	0.13	1.70	2.00	8.79	55.3	40.0	10.8
	Head (Calc.)	100.00	0.61	0.22	4.00	17.5	100.0	100.0	100.0	100.0
	Mozley Con	0.38	0.00	0.00	206.0	233.7	0.00	0.00	19.1	4.69
MZ1-Ox - 2	Bulk 3rd Clnr Con	1.53	32.00	0.97	90.40	794.0	81.4	12.5	33.4	63.4
Gravity	Zn 2nd Clnr Con	0.93	4.06	7.13	28.7	247.0	6.28	56.0	6.45	12.0
	Zn Ro Tail	93.6	0.06	0.03	1.58	2.00	8.56	22.9	35.7	9.8
	Head (Calc.)	100.00	0.60	0.12	4.14	19.2	100.0	100.0	100.0	100.0
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MZ2-Ox - 1	Bulk 3rd Clnr Con	0.65	28.3	2.62	375.0	1551.0	73.8	0.2	54.4	51.9
No Gravity	Zn 2nd Clnr Con	17.49	0.29	54.0	3.24	33.1	20.2	96.6	12.6	29.6
	Zn Ro Tail	72.5	0.01	0.18	1.55	2.00	2.89	1.33	24.9	7.42
	Head (Calc.)	100.0	0.26	9.79	4.66	19.8	100.0	100.0	100.0	100.0
	Mozley Con	0.17	0.00	0.00	717.0	914.3	0.00	0.00	28.2	5.97
MZ2-Ox - 3	Bulk 3rd Clnr Con	0.62	27.2	4.80	249.0	1478	70.4	0.3	35.5	35.0
Gravity	Zn 2nd Clnr Con	16.3	0.13	53.8	0.8	24.2	8.88	92.0	2.97	15.1
	Zn Ro Tail	71.6	0.03	0.19	1.41	10.00	8.42	1.4	23.3	27.5
	Head (Calc.)	100.2	0.24	9.52	4.33	26.0	100.0	100.0	100.0	100.0

TABLE 13.60
FLOTATION VERSUS GRAVITY + FLOTATION SUMMARY – PINWHEEL ZONE + OXIDE

Test	Product	Weight %	Assay (%, g/t) Cu	Zn	Au	Ag	Distribution (%) Cu	Zn	Au	Ag
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PW1-Ox - 1	Bulk Ro Con	64.3	5.11	0.05	5.27	90.5	96.7	79.3	90.6	92.6
No Gravity										
	Head (Calc.)	100.0	3.40	0.04	3.74	62.8	3.26	20.7	9.36	7.44
	Mozley Con	0.37	0.00	0.00	77.0	284.2	0.00	0.00	6.59	1.53
PW1-Ox - 2	Moz + Bulk Ro Con	62.96	5.35	0.05	4.92	91.1	95.6	80.6	72.2	84.0
Gravity										
	Head (Calc.)	100.00	3.53	0.04	4.29	68.3	100.0	100.0	100.0	100.0
	Mozley Con	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
PW2-Ox - 1	Bulk 3rd Clnr Con	1.66	27.0	2.67	109.0	676.0	87.0	31.7	54.3	50.0
No Gravity	Zn 2nd Clnr Con	0.14	5.35	35.8	50.0	15.00	1.40	34.6	2.0	0.1
	Zn Ro Tail	92.8	0.04	0.02	1.14	10.00	6.49	15.3	31.8	41.4
	Head (Calc.)	100.0	0.52	0.17	3.40	23.2	100.0	100.0	100.0	100.0
	Mozley Con	0.24	0.00	0.00	185.7	290.3	0.00	0.00	13.6	3.06
PW2-Ox - 2	Bulk 3rd Clnr Con	1.55	27.5	5.02	78.3	571	80.8	58.3	36.8	38.6
Gravity	Zn 2nd Clnr Con	0.09	7.48	9.0	50.0	200.0	1.31	6.2	1.40	0.8
	Zn Ro Tail	91.8	0.05	0.03	1.25	10.0	9.04	17.2	34.7	40.0
	Head (Calc.)	100.0	0.55	0.16	3.52	25.9	100.0	100.0	100.0	100.0

Comparing the results with and without gravity separation ahead of flotation, similar trends are observed for both the Main Zone + Oxide composites. For MZ1+Ox and MZ2+Ox, the gold reporting to the Mozley concentrate was 19.1% and 28.2%, respectively. Assuming that the gravity concentrate is to be combined with the copper cleaner concentrate, MZ1+Ox achieves a 6.4% increase in gold recovery and MZ2+Ox achieves a 9.3% increase in gold recovery. This is a higher increase than observed for the Main Zone sample absent of oxide. However, unlike the sulphide only composites, both blended composites had lower gold losses in tailings under the two different test scenarios. Further investigation would be required to quantify the impact of gravity on the circuit downstream of Main Zone + Oxide copper flotation based on these results.

The PW1+Ox composite did not achieve reasonable or comparable metallurgical performances for either test. Comparing copper rougher concentrates, which have similar mass pulls, copper grades and copper recoveries, there is a decrease in gold recovery with the addition of gravity separation of over 10%. Although not shown in Table 13.60, cleaner kinetics and recoveries were much lower for the test where gravity separation was completed. Other composites in this campaign displayed a similar response. The reason for this is believed to be the grinding conditions. For the tests where gravity was not applied, composites were ground in the laboratory mill with reagents present for the duration of the grind. Where gravity concentration was applied, the composites were ground without reagents, subjected to gravity separation, then ground for the last five minutes with reagents present. Two of the four composites that contain sulphide material were subjected to repeat tests where the grind time post-gravity separation was lengthened. In both cases, copper flotation kinetics were improved very similarly to those sample that ground with reagents for the full duration of the grind. Due to lack of material and overall poor metallurgical performance, PW1+Ox was selected to be repeated.

While a significant amount of gold was recovered to the PW2+Ox Mozley concentrate, 13.6%, this composite also failed to achieve a higher net gold recovery to the copper concentrate with the addition of gravity concentration.

The increases in gold recovery with gravity separation for the eight composites are shown in Table 13.61.

TABLE 13.61
IMPACT OF GRAVITY CONCENTRATION TO COPPER CONCENTRATE

	Gold Recovery, %			
Composite	w/o Gravity	w/ Gravity	Net Increase	Note
MZ1	35.4	37.7	2.3	
MZ2	61.0	69.2	8.2	
PW1	51.9	60.6	8.7	
PW2	49.8	52.0	2.1	
MZ1+Ox	46.1	52.5	6.4	
MZ2+Ox	54.4	63.7	9.3	
PW1+Ox	90.6	72.2	-18.4	Poor metallurgical performance
PW2+Ox	54.3	50.4	-4.0	Repeat test data pending

Cyanidation bottle roll testing was completed on the tailings of all flotation tests as well as on the one oxide composite. The raw data from the cyanidation bottle rolls on composites not subjected to gravity recovery are presented in Table 13.62.

TABLE 13.62
CYANIDATION BOTTLE ROLL RESULTS

Composite	Test	Extraction, % Au	Residue, g/t Au	Head Grade, g/t Au
Oxide	CN-19	93.0	0.54	7.7
MZ1	CN-1	36.8	0.84	1.33
MZ2	CN-3	40.0	0.64	1.07
MZ1+Ox	CN-5	66.9	0.69	2.09
MZ2+Ox	CN-7	71.1	0.6	2.07
PW1	CN-9	28.9	1.56	2.19
PW2	CN-11	49.7	0.33	0.66
PW1+Ox	CN-13	89.9	0.09	0.89
PW2+Ox	CN-15	87.1	0.33	2.56

There is potential that the sulphide material contains preg-robbing minerals or that the high sulphur content of the flotation tailings may impact the dissolution kinetics of the oxide portion of the blended material.

Table 13.63 summarizes the cyanide consumption for the nine composites tested.

TABLE 13.63
BOTTLE ROLL TEST – CYANIDE CONSUMPTIONS

Composite	Test	NaCN Addition kg/t	CN_{wad} in PLS mg/L	Recycled CN kg/t	NaCN Consumed kg/t
Oxide	CN-19	1.85	250	0.31	1.54
MZ1	CN-1	3.32	512	0.63	2.69
MZ2	CN-3	4.40	749	0.92	3.48
MZ1+Ox	CN-5	3.58	424	0.52	3.06
MZ2+Ox	CN-7	3.78	749	0.92	2.86
PW1	CN-9	4.94	799	0.98	3.96
PW2	CN-11	3.59	774	0.95	2.64
PW1+Ox	CN-13	5.00	924	1.13	3.87
PW2+Ox	CN-15	5.34	1150	1.41	3.93

The four columns in Table 13.63 represent, from left to right, total cyanide addition during the cyanidation bottle roll test, CNwad in the PLS after 48 hr, cyanide recovered via SART and final calculated cyanide consumption. The consumptions from the samples align with previous phases of testing where lower cyanide consumptions were observed for oxide composites (0.75 - 1.5 kg/t) and higher consumptions for sulphide composites (2 - 4 kg/t).

In summary:

- In general gold recoveries increased when gravity was included in the test.
- It is not known how much of the increased gold recovery via gravity separation affects gold recovered to the zinc concentrate or the pregnant leach solutions.
- In general, gold dissolutions for sulphide and oxide blended composites were less than the dissolution where sulphide and oxide material are leached separately.
- In general, cyanide consumption increased when leaching sulphide and oxide material together.

Also of note, there was a testing campaign completed in 2019 in which final flotation tailings was subjected a sulphide/pyrite flotation stage followed by cyanidation bottle roll leaching. In general, the sulphide/pyrite flotation stage did not achieve significant gold upgrading, however, the recoveries in the bottle roll tests were much better than without the flotation stage. This led to similar overall gold recoveries as reported when cyanidation was completed on whole tailings samples. Tailings sulphide/pyrite flotation conditions were not optimized.

It is recommended to complete additional metallurgical testwork to further evaluate cyanide leaching sulphide flotation tailings to economically recover additional gold, such as:

- Testing additional Main and Pinwheel samples.
- Understanding the degree of pre-robbing or cyanide consuming materials.
- Investigating the impact of grind size on gold recoveries (tailing regrind circuit).

13.7 TAILINGS DEWATERING AND RHEOLOGY

13.7.1 Introduction

Tailings dewatering and rheology was completed by Golder Associates at their laboratory in Sudbury, Ontario, Canada. A total of four samples were sent from SGS to Golder, representing the Main, Pinwheel, Tuff and oxide zones of the Back Forty Deposit. A total of approximately 50 kg of each sample in slurry form was sent.

Samples were generated by a combination of using stored tailings from metallurgical testing as well as completing 10 kg flotation tests for the purpose of tailings generation. The tailings were blended to match the master composites used for flotation and oxide testwork.

The scope of the tests performed included flocculant selection, settling tests and rheological characterization.

13.7.2 Flocculant Selection

The first stage of settling tests was assessing the potential for thickening through the use synthetic polymers (flocculants). Several types of flocculants were screened in order to test a range of parameters such as charge density and molecular weight. The typical types of flocculants considered are anionic and non-ionic polymers.

The full suite of tests was completed on the oxide and Tuff tailings. Only confirmatory tests were completed on the Main and Pinwheel tailings.

In selecting the most effective flocculant, several factors were examined, such as initial settling velocity, overflow clarity, flocculant size and structure as well as underflow density. The overflow clarity was measured in NTU (Nephelometric Turbidity Unit) where lower numbers indicate clearer water. The screening for the oxide and Tuff samples indicated that the anionic flocculant, AN926VHM was best suited for this material.

13.7.3 Settling Tests

The next step was to optimize flocculant dosage and feed solids. The dosage represents the amount of flocculant (polymer) in grams for each tonne of material. The tests were carried out in 1 L vessels. Underflow solids were raked after settling for 45 minutes.

The optimal conditions from the screening results were carried forward to larger scale 4 L tests to more accurately determine the underflow density. Based on the results obtained, it was determined that a feed density of 15 weight percent ("wt%") solids and a flocculant dosage of 30 to 40 g/t would provide good settling characteristics. Table 13.64 shows the summary of the 4 L tests.

TABLE 13.64
4 L SETTLING TEST SUMMARY

Material Type	Flocculant Dosage (g/t)	Feed Solids (wt% solids)	Overflow Clarity After 45 Minutes (NTU)	Calculated Underflow Density After 2 Hours (wt% solids)	Calculated Underflow Density After 24 Hours (wt% solids)	Measured Underflow Density After 24 Hours (wt% solids)
Oxide	40	15	49	60	61	62
Main	30	15	41	73	73	73
Pinwheel	30	15	53	69	69	69
Tuff	30	15	2	60	60	60

13.7.4 Rheological Characterization

Rheological testing was carried out to evaluate flow and handling properties. These tests provide an indication regarding the behaviour of the material in the course of mixing, slump adjustment, pumping, flowing and also while sitting idle. Rheological characterization provides data for the selection of process equipment such as mixers, pumps and pipelines. A full suite of rheology testwork was completed for oxide, Pinwheel and Tuff tailings. It is anticipated that the Main tailings will exhibit similar rheological behaviour to Pinwheel tailings.

13.7.4.1 Slump Versus Solids Content

To gauge sensitivity to water additions, small increments of water were added to the bulk sample. After each addition, slump and solids content was determined. This generates a relationship between slump and solids content which is typically used to determine the degree of process control required to maintain slump control of the final product.

13.7.4.2 Static Yield Stress Testing

Yield stress is defined as the minimum force required to initiate flow. Static yield stress was determined by using a very slow moving (0.2 RPM) vane spindle attached to a torque spring. The spindle was immersed in the sample and measurements were taken at various solids contents. From these data points, yield stress curves were developed.

13.7.4.3 Water Bleed and Yield Stress Versus Time

Moisture retention testing was carried out to assess the water bleed properties of the thickened tailings while sitting idle in test beakers. Two slump consistencies were tested at four-time intervals. At each interval the water bleed and yield stress were measured.

13.7.4.4 Plug Yield Stress

Plug yield stress analysis was performed to determine if consolidation has occurred throughout a cross-section of idle thickened tailings material, as may be present in a pipeline cross-section. Two slump consistencies of material were allowed to sit idle for two hours, and a specially designed vane spindle was immersed at three depths to measure yield stress.

13.7.4.5 Viscosity and Dynamic Yield Stress Determination

Viscosity testing provides bench scale flow properties and fluid characterization. Dynamic viscosity and yield stress data is essential for mixer, pump and pipeline design. In order to compare or duplicate viscosity results of non-Newtonian fluids, it is important to test according to the same conditions. Test conditions and parameters such as cycle time and instrument sensor configuration are critical to producing usable data from bench scale viscometers.

The yield stress determined through this testing is referred to as dynamic yield stress since it is extrapolated from dynamic shear stress data to zero shear. The instrument sensor rotated inside the cup which contained the sample and torque measurements were recorded at several incremental speeds or shear rates.

13.8 METAL RECOVERY EQUATIONS

13.8.1 Methodology

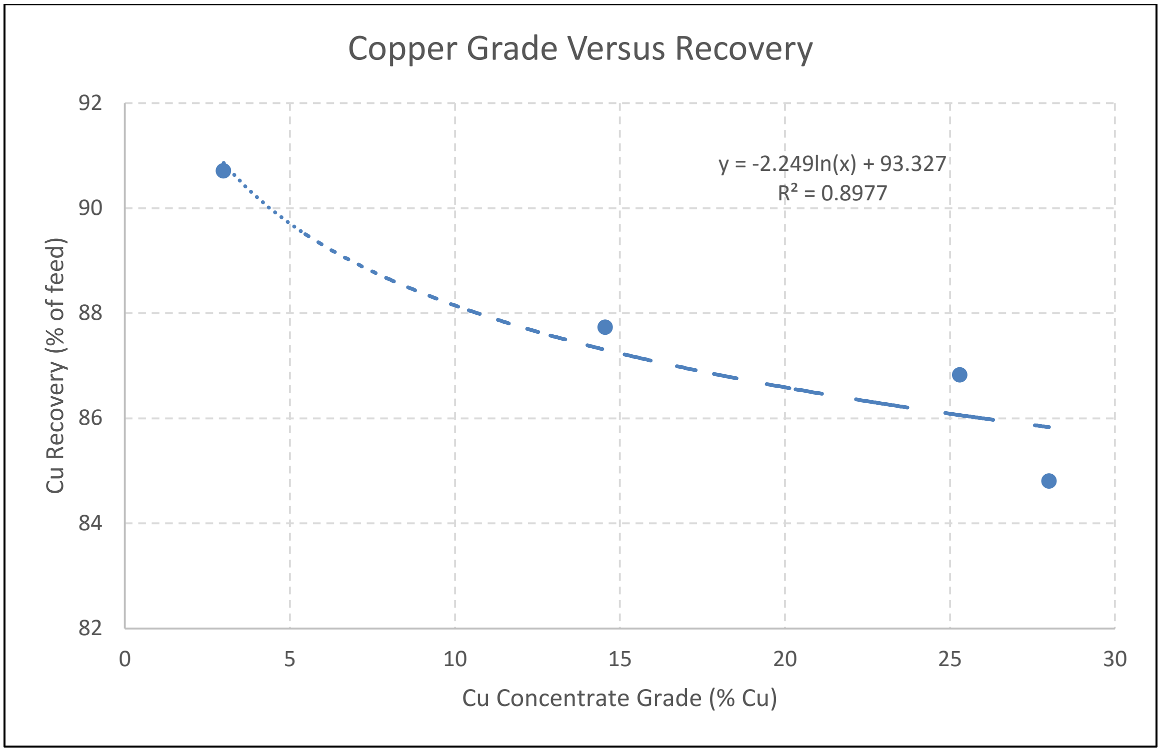
Metal recovery equations have been based on a total of 108 metallurgical tests and are summarized in Table 13.65. Type 4 is assumed to have the same metallurgical response as type 8. Metallurgical (“met”) types 2 and 7 will be blended through the plant and the equations are thus based on average of all tests for the two met types. All other met types will be treated through the oxide and sulphide plants separately, with oxides leached through a separate leach circuit, while the sulphide met types will be campaigned through a flotation plant.

TABLE 13.65
SUMMARY OF TESTWORK TO SUPPORT RECOVERY EQUATIONS

Metallurgical Type	Description	Total Tests		
		Batch	Locked Cycle	Total
Type 1	Main	27	14	41
Type 2 / 7	Pinwheel - Cu Rich	13	3	16
Type 3	Pinwheel - Stringers	8	0	8
Type 5	Tuff	20	5	25
Type 4 / 8	Pinwheel - Zn Rich	6	3	9
Sub-total Sulphides		74	25	99
Type 6	Oxides	9	0	9
Total	**Oxide + Sulphides**	**83**	**25**	**108**

For the each of the various sulphide tests, recovery was recorded for the Bulk Rougher Concentrate and each of the three stages of Cleaner Concentrate. These results were plotted and regressed in order to calculate recovery as a function of concentrate grade. Figure 13.21 provides an example of the regression analysis for one of the Main Zone batch tests.

FIGURE 13.211 EXAMPLE OF CONCENTRATE GRADE – RECOVERY REGRESSION



Concentrate Grade	Recovery		
	Cu (%)	Au (g/t)	Ag (g/t)
28.00	84.80	45.06	32.83
25.29	86.82	46.69	33.92
14.55	87.73	48.36	36.60
2.99	90.70	53.35	43.50

Source: *Aquila Resources Inc. (2020)*

The reported recovery from batch tests was adjusted to reflect the impact of a locked cycle. The assumed incremental recovery was calculated based on the comparison of average recovery for all batch tests with the average for all locked cycle tests for each met type. Table 13.66 summarizes the incremental recovery assumed for the batch test results for each of the various met types (note that the incremental recovery varies as a function of concentrate grade and the values provided are for the defined optimal concentrate grade for each met type).

TABLE 13.66
INCREMENTAL RECOVERY TO ACCOUNT FOR LOCKED CYCLE TESTING

Met Type	Pb / Cu	Zn	Au	Ag	Comment
Type 1	6.6%	4.0%	8.7%	8.1%	
Type 2 / 7	16.6%	n/a	0.0%	0.0%	L/C does not include PM recovery
Type 3	n/a	n/a	n/a	n/a	no L/C tests for Type 3
Type 5	10.0%	12.2%	2.8%	8.6%	
Type 8	9.0%	19.0%	2.8%	11.1%	

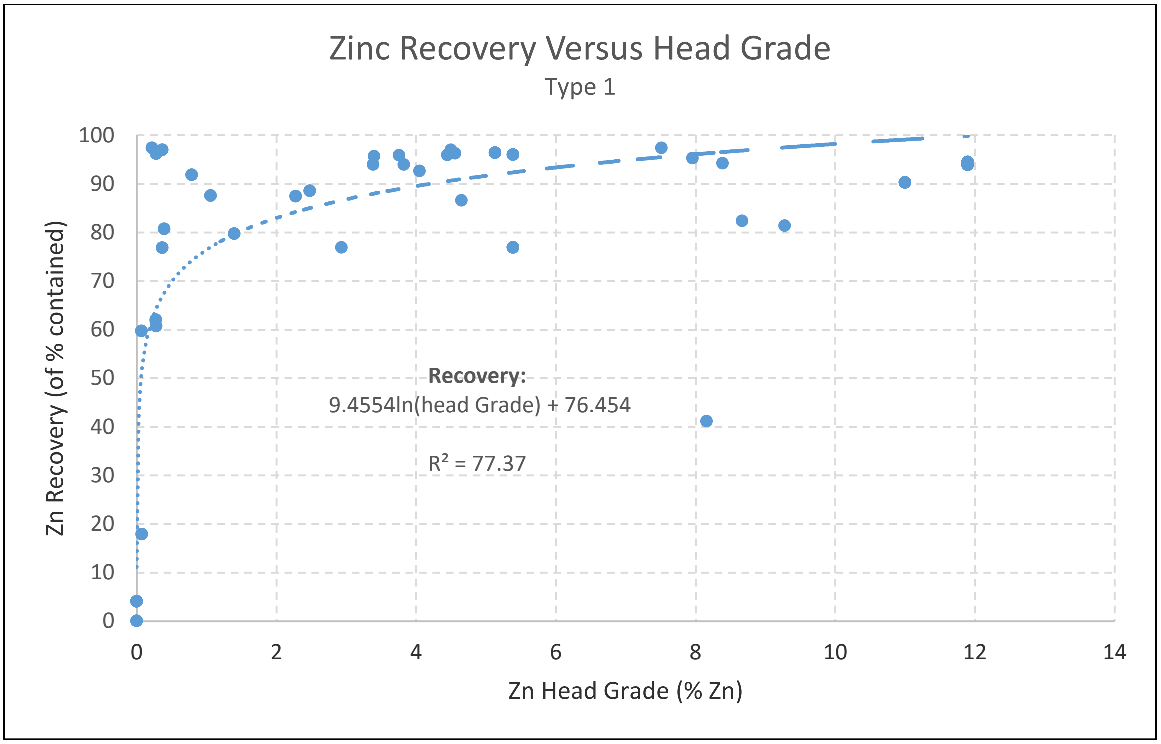
Note: *met = metallurgical.*

The regression equations and factors for incremental recovery were then used to calculate the recovery for each test at the following range of concentrate grades:

- Copper: 10% Cu – 22% Cu.
- Zinc: 48% Zn – 55% Zn.
- Lead: 30% Pb – 40% Pb.

Tests were weighted to reflect the total proportion of each met type of similar base metal grade to the particular test. For example, the 41 tests of Type 1 were expanded to a set of 96 total Cu recovery datapoints. Test number 20 (SGS ’16 – F VC9 No. 1) with a grade of 0.26% is representative of only 1% of the total material (in the grade bin 0.2 – 0.3% Cu) and was only used once. However, Test number 21 (SGS ’16 – F VC10 No. 2) with a grade of 0.95% Cu is representative of 3.9% of the total material (grade bin 0.9 – 1.0% Cu) and was thus included four times. The calculated recoveries for the entire dataset were then plotted and regressed (with recovery expressed as a function of head grade) to generate equations at different concentrate grades. Figure 13.22 provides an example of the resulting equation for Type 1 Zn, at a concentrate grade of 55% Zn.

FIGURE 13.222 TYPE 1 ZN RECOVERY (55% ZN CONCENTRATE GRADE)



Source: *Aquila Resources Inc. (2020)*

In general, as concentrate grade increases, the treatment charges decrease and metal payability increases, though recovery of metal decreases. The optimal concentrate grade can thus vary according to met type. For each met type, the various concentrate grades were tested and the following selected as generating optimal economics at the Base Case metal price assumptions:

- Type 1: 18% Cu Concentrate/55% Zn Concentrate.
- Type 2 and 7: 17% Cu Concentrate.
- Type 3: 20% Cu Concentrate.
- Type 5: 35% Pb Concentrate/53% Zn Concentrate.
- Type 8: 22% Cu Concentrate/50% Zn Concentrate.

For Type 6 (Oxide), the equation was simply based on a regression of recovery and sample head grade.

13.8.2 Results

For the flotation concentrates, a total of eight products were considered. This included four copper concentrates for met types 1, 2 + 7, 3 and 8, three zinc concentrates for met types 1, 5 and 8, as well as a lead concentrate for met type 5. Where applicable, gold and silver credits were also evaluated. Type 4 is assumed to have the same recovery equation as type 8. The complete list of recovery equations is displayed in Table 13.67.

TABLE 13.67
RECOVERY EQUATIONS

Met Type	Conc.	Conc. Quality	Recovery Equation
1	Cu	18% Cu	Cu Rec. = $495.7x^5 - 1776.0x^4 + 2490.0x^3 - 1723.6x^2 + 594.7x + 6.44$
			Au Rec. = 5.72ln(x) + 48.7
			Ag Rec. = 4.05ln(x) + 32.5
	Zn	55% Zn	Zn Rec. = 10.4ln(x) + 75.4
			Au Rec. = 1.27ln(x) + 8.80
			Ag Rec. = 1.81ln(x) + 13.1
2 & 7	Cu	17% Cu	Cu Rec. = -0.19ln(x) + 79.7
			Au Rec. = 6.93ln(x) + 38.3
			Ag Rec. = 7.79ln(x) + 12.3
3	Cu	20% Cu	Cu Rec. = 8.28ln(x) + 86.1
			Au Rec. = 8.44ln(x) + 54.9
			Ag Rec. = 2.18ln(x) + 40.6
4 & 8	Cu	22% Cu	Cu Rec. = 7.98ln(x) + 82.1
			Au Rec. = 47.6ln(x) + 48.9
			Ag Rec. = 20.6ln(x) – 18.0
	Zn	50% Zn	Zn Rec. = 14.6ln(x) + 68.2
5	Pb	35% Pb	Pb Rec. = 6.16ln(x) + 85.9
			Au Rec. = 1.90ln(x) +74.0
			Ag Rec. = 1.23ln(x) + 57.8
	Zn	53% Zn	Zn Rec. = 5.22ln(x) + 87.3
			Au Rec. = -5.40ln(x) + 6.53
			Ag Rec. = 0.66ln(x) + 21.2
6	Doré	-	Au Rec. = 6.73ln(x) + 82.1
			Ag Rec. = 4.74ln(x) + 50.7

Note: *Met = metallurgical, Conc = Concentrate.*

14.0 MINERAL RESOURCE ESTIMATE

14.1 INTRODUCTION

The purpose of this Technical Report section is to provide an updated Mineral Resource Estimate for the Aquila Resources Inc. Back Forty Project, Michigan, USA. The update includes Open Pit (“OP”) and Underground (“UG”) Mineral Resource models which are defined by the pit design in the Feasibility Study "Back Forty Project, Michigan, USA, Feasibility Study, NI 43-101 Technical Report" with an effective date of August 1, 2018. The update was mainly performed for the UG Mineral Resource located outside of the open pit design; while the model of the OP Mineral Resource was constrained to the pit design from the Feasibility Study. The Mineral Resource Estimate presented herein is reported in accordance with the Canadian Securities Administrators’ National Instrument 43-101 and has been estimated in conformity with the generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral Resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent Mineral Resource Estimates.

This Mineral Resource Estimate was based on information and data supplied by Aquila, and was undertaken by Yungang Wu, P.Geo., and Eugene Puritch, P.Eng. of P&E, independent Qualified Persons in terms of NI 43-101. The effective date of this Mineral Resource Estimate is October 14, 2019.

14.2 DATABASE

All drilling and assay data were provided in the form of Excel data files by Aquila. The GEOVIA GEMS™ V6.8 database for this Mineral Resource Estimate, compiled by P&E, consisted of 741 drill holes totalling 128,670 m, of which 1,447 intersects totalling 17,201 m from 489 drill holes were used for the Mineral Resource Estimate. A drill hole plan is shown in Appendix A.

The database contained assays for Au, Ag, Zn, Cu, Pb and S as well as other lesser elements of non-economic importance. The number of assays utilized for grade interpolation for the elements of economic interest are presented in Table 14.1.

TABLE 14.1 **BACK FORTY ASSAY DATABASE SUMMARY**			
Element	**Total No. of Database Assays**	**No. of Assays Used for Open Pit Mineral Resource Estimate**	**No. of Assays Used for Underground Mineral Resource Estimate**
Au	63,850	12,226	9,740
Ag	62,876	12,120	9,658
Cu	62,023	11,847	9,446
Pb	61,952	11,820	9,433
Zn	61,915	11,827	9,439
S	31,822	6,711	5,355

Note: *some assays were used for both open pit and underground models.*

All drill hole survey and assay values are expressed in metric units, with grid coordinates in the NAD 83, Zone 16N UTM system.

14.3 DATA VERIFICATION

Verification of Au, Ag, Cu, Zn and Pb assay database was performed by P&E against original laboratory, electronically-issued certificates from ALS Chemex, Vancouver, BC, Accurassay Laboratories, Thunder Bay, and Inspectorate America Corporation, Sparks, Nevada. A total of approximately 60% of the wireframe constrained assays were checked. Unchecked assays were due to laboratory certificates not being made available to P&E. Some minor errors were noticed and corrected in the Gems database.

P&E also validated the Mineral Resource database by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. Some very minor errors were noted and corrected in the database. P&E believes that the supplied database is suitable for Mineral Resource estimation.

14.4 DOMAIN INTERPRETATION

The mineralization wireframes of the open pit model remain unchanged from the 2018 Feasibility Study, which were created from the surface down to approximately -500 m elevation with NSR \$/tonne ("\$/t") cut-off values as shown in Table 14.2. The mineralization domains were defined by continuous mineralized structures, lithology along strike and down dip, and assay intervals equal to or greater than the NSR \$/t cut-off. The NSR values were derived from the assumptions listed in Table 14.2.

TABLE 14.2 NSR $/T CUT-OFF VALUES USED FOR THE OPEN PIT MODEL WIREFRAMES						
Open Pit Elevation (m)	**Processing Method**	**Processing ($/t)**	**TMF and Stockpiling ($/t)**	**UG Mining ($/t)**	**G&A ($/t)**	**NSR Cut-off ($/t)**
Above 0 m EL	Flotation	15	2	0	4	21
	Leach	16	2	0	4	22
Below 0 m EL	Flotation	15	2	49	4	70

Note: *TMF = tailings management facility.*

The mineralization wireframes for the underground model were generated with NSR $/t cut-off values as shown in Table 14.3. Different NSR cut-offs were applied based on the metallurgical type. The design pit for the 2018 Feasibility Study was used as a guideline to ensure that the wireframes of the underground model partially overlapped with the wireframes of open pit model. The wireframes of the underground model were created at least 50 m above the design pit and down to approximately -500 m elevation.

TABLE 14.3 NSR $/T CUT-OFF VALUES USED FOR THE UNDERGROUND MODEL WIREFRAMES						
Cost Area	**Metallurgical Type**					
	Main	**Pinwheel MS Cu / Gossan**	**Pinwheel Semi-Massive / Stringers**	**Pinwheel MS Cu / Zn**	**Tuff**	**Oxide**
Processing $/t	14.21	11.39	11.07	14.23	14.61	23.70
G&A $/t	2.19	2.19	2.19	2.19	2.19	2.19
TMF and WRF $/t	0.98	0.98	0.98	0.98	0.98	0.98
UG Mining $/t	50.00	50.00	50.00	50.00	50.00	50.00
NSR Cut-off $/t	68	65	65	68	68	77

Note: *MS = massive sulphide, TMF = tailings management facility, WRF = waste rock storage facility.*

Fifty-four (54) and fifty-eight (58) mineralization wireframes were constructed for open pit and underground Mineral Resource Estimates, respectively. The wireframes were created from successive cross-sectional polylines on east-facing vertical sections with a variable 10 m to 25 m spacing. In some cases, mineralization below the cut-off values in Table 14.2 and 14.3 were included for the purpose of maintaining zonal continuity. On each section, polyline interpretations were digitized from drill hole to drill hole, but not typically extended more than 25 m into untested territory. The minimum constrained sample length for interpretation was 2.0 m. The resulting Mineral Resource domains were used as constraining boundaries during Mineral Resource estimation, for rock coding, statistical analysis and compositing limits. The 3-D domains are presented in Appendix B.

Surfaces of topography and faults, and solids of Mafic and QFP dykes were provided by Aquila, while an overburden surface was created by P&E using drill hole logs.

14.5 ROCK CODE DETERMINATION

A unique rock code was assigned for each mineralized domain in the Mineral Resource model. Six (6) metallurgical types were defined based on the metallurgical tests provided by Aquila, of which oxides, including all gold domains (coded as type 6), will be potentially leachable, while all other types will be potentially flotable. The rock codes, metallurgical types and codes can be seen in Table 14.4.

TABLE 14.4
MODEL CODES USED FOR THE MINERAL RESOURCE ESTIMATE

Model	Domains	Rock Type	Metallurgical Type	Metallurgical Code
OP & UG	MainMS1	1010	Main Zone	1
OP & UG	MMS1ZNHG	1015	Main Zone	1
OP & UG	MainMS2	1020	Main Zone	1
OP & UG	MainMS3	1030	Main Zone	1
OP & UG	PWSMSSCU	1040	PWMSCu	2
OP & UG	PWMSZnCu	1041	PWMS Cu/Zn	4
OP & UG	PWMSCR	1042	PWSMSS	3
OP & UG	PWMS2	1050	PWMS Cu/Zn	4
OP & UG	PWMS3	1060	PWMS Cu/Zn	4
OP & UG	TFMS	1070	Tuff Zone	5
OP & UG	DeepMS1	1080	Main Zone	1
OP & UG	DeepMS2	1090	Main Zone	1
OP & UG	DDeepMS1	1100	Main Zone	1
OP & UG	DDeepMS2	1110	Main Zone	1
OP & UG	PWMS4	1120	PWMS Cu/Zn	4
OP & UG	PWMS5	1130	PWMS Cu/Zn	4
OP & UG	90AU	2010	Oxide	6
OP & UG	NSAU	2020	Oxide	6
OP & UG	PMAU1	2030	Oxide	6
OP & UG	PMAU2	2040	Oxide	6
OP & UG	PMAU3	2050	Oxide	6
OP & UG	PMAU4	2060	Oxide	6
OP & UG	PMAU5	2070	Oxide	6
OP & UG	PWGossan	2080	PW Gossan_ Flotation	2
OP & UG	PWGossanLeach	2086	Oxide	6
OP & UG	EastGossan	2090	Oxide	6
OP & UG	EAST_OX	2100	Oxide	6
OP & UG	PWGSN2	2200	PW Gossan_ Flotation	2
OP & UG	MainSTR	3000	Main Zone	1

TABLE 14.4
MODEL CODES USED FOR THE MINERAL RESOURCE ESTIMATE

Model	Domains	Rock Type	Metallurgical Type	Metallurgical Code
UG	MainSTR0	3001	Main Zone	1
OP	MainSTR (sub-domain)	3005	Main Zone	1
OP & UG	MainSTR1	3010	Main Zone	1
OP & UG	MainSTR2	3020	Main Zone	1
OP & UG	MainSTR3	3030	Main Zone	1
OP & UG	MainSTR4	3040	Main Zone	1
OP & UG	MainSTR5	3050	Main Zone	1
OP & UG	MainSTR6	3060	Main Zone	1
UG	MainSTR6 (sub-domain)	3065	Main Zone	1
OP & UG	MainSTR7	3070	Main Zone	1
OP & UG	MainSTR8	3080	Main Zone	1
OP & UG	MainSTR9	3090	Main Zone	1
OP & UG	MainSTR10	3100	Main Zone	1
OP & UG	MainSTR11	3110	Main Zone	1
OP & UG	MainSTR12	3120	Main Zone	1
OP & UG	PWSTR1	3130	PW Stringer	3
OP & UG	PWSTR2	3140	PW Stringer	3
OP	DeepSTR1	3150/3151	Main Zone	1
OP	DeepSTR2	3155	Main Zone	1
OP	DeepSTR3	3156	Main Zone	1
UG	DeepSTR1	3150	Main Zone	1
UG	DeepSTR2	3151	Main Zone	1
UG	DeepSTR3	3152	Main Zone	1
UG	DeepSTR4	3153	Main Zone	1
UG	DeepSTR5	3154	Main Zone	1
UG	DeepSTR6	3155	Main Zone	1
OP & UG	UpDeep	3160	Tuff Zone	5
OP & UG	TFSTR1	3170	Tuff Zone	5
OP & UG	TFSTR2	3180	Tuff Zone	5
OP & UG	TFSTR3	3190	Tuff Zone	5
OP & UG	TFSTR4	3200	Tuff Zone	5
OP & UG	TFSTR5	3210	Tuff Zone	5
OP & UG	TFSD	3220	Tuff Zone	5
OP & UG	TFSTR6	3230	Tuff Zone	5
OP & UG	Air	0	Air	
OP & UG	OVB	10	Overburden	
OP & UG	Waste	99	Waste	
OP & UG	Mafic East	4001	Waste dyke	

TABLE 14.4 **MODEL CODES USED FOR THE MINERAL RESOURCE ESTIMATE**				
Model	**Domains**	**Rock Type**	**Metallurgical Type**	**Metallurgical Code**
OP & UG	Mafic West	4002	Waste dyke	
OP & UG	Main QFP	4003	Waste dyke	
OP & UG	QFP North	4004	Waste dyke	
OP & UG	QFP South	4005	Waste dyke	
OP & UG	QFP Deep	4006	Waste dyke	
OP & UG	QFP West	4007	Waste dyke	
OP & UG	Sandstone	4008	Waste	
OP & UG	FLDK_N	4009	Waste dyke	
OP & UG	QFP-PW	4010	Waste dyke	
OP & UG	QFP-PW2	4011	Waste dyke	
OP & UG	QFP-SMRHY	4012	Waste dyke	

Note: *OP = open pit, UG = underground.*

14.6 COMPOSITING

The basic statistics of all constrained assays and sample lengths are presented in Table 14.5.

TABLE 14.5 **BASIC STATISTICS OF ALL CONSTRAINED ASSAYS AND SAMPLE LENGTHS**								
Model	**Variable**	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**Cu (%)**	**Pb (%)**	**S (%)**	**Length (m)**
Open Pit	Number of samples	12,226	12,120	11,827	11,847	11,820	6,711	12,233
	Minimum value	0.00	0.05	0.00	0.00	0.00	0.00	0.01
	Maximum value	375.06	29,766.85	45.86	20.27	25.40	58.60	4.50
	Mean	2.22	31.66	2.34	0.28	0.23	19.54	1.39
	Median	0.96	10.00	0.27	0.07	0.04	11.10	1.50
	Variance	43.09	89,169.84	23.19	0.61	0.60	329.25	0.08
	Standard Deviation	6.56	298.61	4.82	0.78	0.77	18.15	0.29
	Coefficient of Variation	2.95	9.43	2.06	2.77	3.37	0.93	0.21
UG	Number of samples	9,740	9,658	9,439	9,446	9,433	5,355	9,744
	Minimum value	0.00	0.05	0.00	0.00	0.00	0.01	0.15
	Maximum value	375.06	29,766.85	45.86	20.27	25.40	58.60	3.81
	Mean	2.55	36.13	2.81	0.33	0.26	22.03	1.40
	Median	1.17	11.00	0.33	0.09	0.04	14.71	1.50

TABLE 14.5 BASIC STATISTICS OF ALL CONSTRAINED ASSAYS AND SAMPLE LENGTHS								
Model	**Variable**	**Au (g/t)**	**Ag (g/t)**	**Zn (%)**	**Cu (%)**	**Pb (%)**	**S (%)**	**Length (m)**
	Variance	51.81	110,510.04	27.50	0.66	0.73	356.09	0.07
	Standard Deviation	7.20	332.43	5.24	0.81	0.86	18.87	0.27
	Coefficient of variation	2.82	9.20	1.87	2.51	3.34	0.86	0.19

As shown in Figure 14.1 and 14.2, approximately 68% and 69% of the constrained sample lengths were 1.5 m in length for the open pit and underground model, respectively, with an overall average of 1.4 m. In order to regularize the assay sampling intervals for grade interpolation, a 1.5 m compositing length was selected for the drill hole intervals that fell within the constraints of the above-mentioned Mineral Resource domains. The composites were calculated for Au, Ag, Zn, Cu, Pb and S over 1.5 m lengths starting at the first point of intersection between assay data hole and hanging wall of the 3-D zonal constraint. The compositing process was halted upon exit from the footwall of the aforementioned constraint. Un-assayed intervals and below detection limit assays were set to 0.001 g/t or % for Au, Ag, Zn, Cu and Pb and nil for S. Any composites that were less than 0.5 m in length were discarded so as not to introduce any short sample bias in the interpolation process. The constrained composite data were extracted to point files for a capping study. The composite statistics are summarized in Table 14.6.

FIGURE 14.1 CONSTRAINED SAMPLE LENGTH DISTRIBUTIONS OF OPEN PIT MODEL

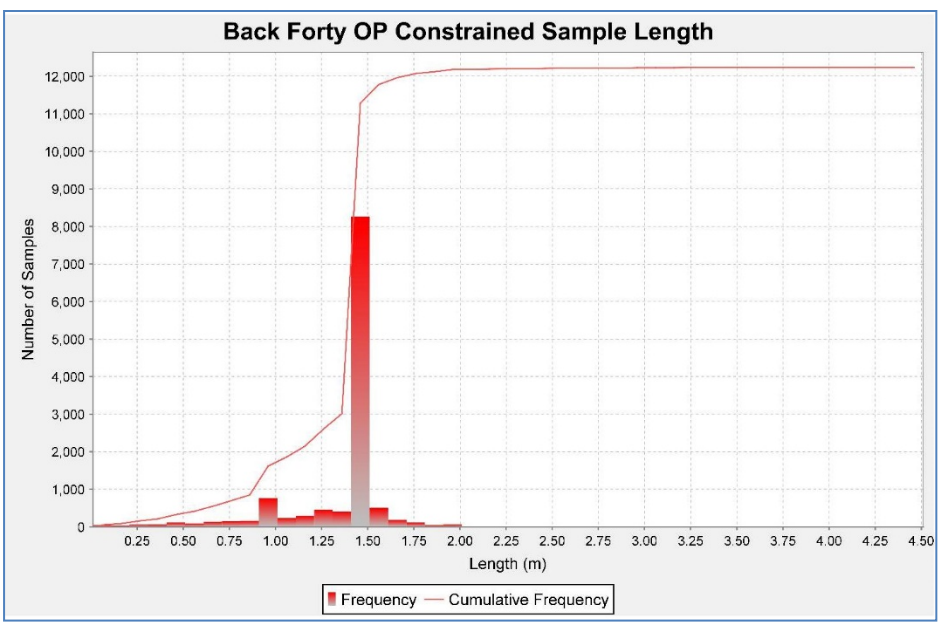


FIGURE 14.2 CONSTRAINED SAMPLE LENGTH DISTRIBUTIONS OF UNDERGROUND MODEL

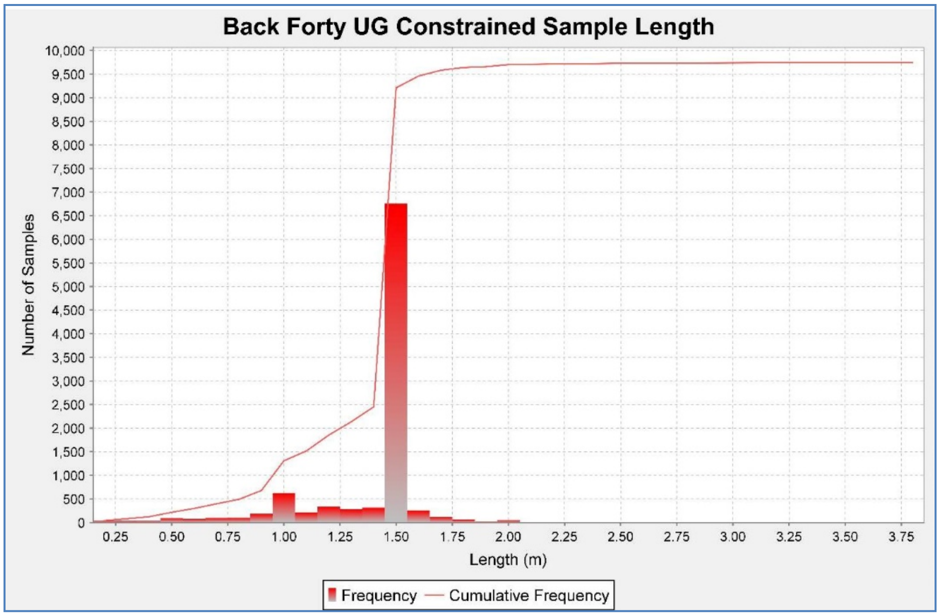


TABLE 14.6
SUMMARY STATISTICS OF COMPOSITES

Model	Variable	Au Comp (g/t)	Ag Comp (g/t)	Zn Comp (%)	Cu Comp (%)	Pb Comp (%)	S Comp (%)
Open Pit	Number of samples	11,648	11,648	11,648	11,648	11,648	10,649
	Minimum value	0.001	0.001	0.001	0.001	0.001	0.001
	Maximum value	151.55	7,469.27	44.87	14.70	15.87	56.60
	Mean	2.08	26.57	2.18	0.27	0.20	12.79
	Median	0.98	10.00	0.26	0.06	0.04	4.34
	Variance	24.52	13,483.12	20.04	0.48	0.37	285.97
	Standard Deviation	4.95	116.12	4.48	0.69	0.61	16.91
	Coefficient of Variation	2.38	4.37	2.05	2.61	2.99	1.32
UG	Number of samples	9,284	9,284	9,284	9,284	9,284	9,311
	Minimum value	0.001	0.001	0.001	0.001	0.001	0.001
	Maximum value	151.63	6,775.88	44.87	14.34	15.86	56.6
	Mean	2.42	30.38	2.66	0.31	0.23	13.20
	Median	1.20	11.21	0.33	0.08	0.04	3.49
	Variance	29.97	13,396.26	24.50	0.55	0.48	311.67
	Standard Deviation	5.47	115.74	4.95	0.74	0.69	17.65
	Coefficient of variation	2.26	3.81	1.86	2.39	2.99	1.34

14.7 GRADE CAPPING

Grade capping was investigated on the 1.5 m composite values in the database within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Log-normal histograms of Au, Ag, Zn, Cu, Pb and S composites were generated for each mineralized zone and the selected resulting graphs are exhibited in Appendix C for the open pit model and in Appendix D for the underground model. The statistics of capped composites are summarized in Table 14.7 and the grade capping values are detailed in Table 14.8 and 14.9. The capped composites were utilized to develop variograms and for block model grade interpolation search parameters.

TABLE 14.7
SUMMARY STATISTICS OF CAPPED COMPOSITES

Model	Variable	Au Cap (g/t)	Ag Cap (g/t)	Zn Cap (%)	Cu Cap (%)	Pb Cap (%)	S Cap (%)
Open Pit	Number of samples	11,648	11,648	11,648	11,648	11,648	10,649
	Minimum value	0.001	0.001	0.001	0.001	0.001	0.001
	Maximum value	70.00	640.00	32.00	14.34	10.30	56.60
	Mean	1.99	23.07	2.16	0.26	0.20	12.74
	Median	0.98	10.00	0.26	0.06	0.04	4.34
	Variance	14.49	2,061.23	19.50	0.42	0.30	284.70
	Standard Deviation	3.81	45.40	4.42	0.65	0.55	16.87
	Coefficient of Variation	1.91	1.97	2.04	2.49	2.80	1.32
UG	Number of samples	9,284	9,284	9,284	9,284	9,284	9,311
	Minimum value	0.001	0.001	0.001	0.001	0.001	0.001
	Maximum value	70.00	750.00	40.00	14.34	10.30	56.60
	Mean	2.31	26.68	2.62	0.30	0.22	13.15
	Median	1.20	11.21	0.33	0.08	0.04	3.49
	Variance	17.91	2848.73	23.72	0.49	0.37	310.17
	Standard Deviation	4.23	53.37	4.87	0.70	0.60	17.61
	Coefficient of variation	1.83	2.00	1.86	2.31	2.77	1.34

TABLE 14.8
CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL

AU CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value Au (g/t)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
MainMS1	1010	1427	30	4	2.06	2.02	1.57	1.30	99.7
MainMS2	1020	621	30	2	3.12	3.12	1.21	1.20	99.7
MainMS3	1030	939	15	11	1.81	1.68	1.78	1.34	98.8
PWSMSSCu	1040	288	9	2	1.83	1.73	1.18	0.75	99.3
PWMSZNCU	1041	224	4	1	0.82	0.79	1.16	0.77	99.6
PWMSCR	1042	276	6	4	1.91	1.86	0.69	0.57	98.6
PWMS2	1050	226	8	3	1.32	1.22	1.47	1.04	98.7
PWMS3	1060	184	6	1	1.50	1.49	0.59	0.54	99.5
TFMS	1070	77	No Capping	0	1.90	1.90	1.06	1.06	100.0
DeepMS1	1080	199	10	2	2.27	2.23	0.90	0.80	99.0
DeepMS2	1090	128	15	2	1.90	1.81	1.49	1.23	98.4
DDeepMS	1100-1110	24	No Capping	0	6.41	6.41	1.43	1.43	100.0
PWMS4	1120	23	No Capping	0	2.27	2.27	0.97	0.97	100.0
PWMS5	1130	31	No Capping	0	1.35	1.35	1.17	1.17	100.0
90AU	2010	1266	30	2	2.18	2.17	1.66	1.60	99.8
NSAU	2020	521	35	7	4.75	4.24	2.22	1.59	98.7
PMAU	2030-2070	359	40	5	3.76	3.41	2.36	1.87	98.6
PWGossan	2080	95	30	1	5.12	4.66	1.74	1.29	98.9
PWGossanLeach	2086	96	No Capping	0	6.69	6.69	1.37	1.37	100.0
EastGossan	2090	100	70	3	18.03	16.00	1.45	1.12	97.0
EAST_OX	2100	148	No Capping	0	0.82	0.82	1.53	1.53	100.0
PWGSN2	2200	24	No Capping	0	2.19	2.19	1.01	1.01	100.0
MainSTR	3000	1224	20	2	1.60	1.56	1.62	1.31	99.8
MainSTR1-12	3010-3120	525	10	5	1.08	0.99	2.00	1.45	99.0
PWSTR	3130-3140	194	20	2	1.70	1.58	2.30	1.91	99.0

TABLE 14.8 CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL									
AU CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Au (g/t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
Deep STRs	3150-3155	80	No Capping	0	1.46	1.46	1.09	1.09	100.0
TFSTRs	3160-3230	2349	10	9	0.90	0.88	1.50	1.26	99.6

Note: *CoV = Coefficient of Variation*

TABLE 14.8 CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL									
AG CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Ag (g/t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	1427	100	12	15.33	14.69	1.30	0.95	99.2
MainMS2	1020	621	150	7	19.22	18.83	1.31	1.21	98.9
MainMS3	1030	939	180	12	24.13	22.84	1.59	1.29	98.7
PWSMSSCu	1040	288	70	5	23.36	15.16	4.61	0.79	98.3
PWMSZNCU	1041	224	120	6	31.55	29.60	1.14	0.92	97.3
PWMSCR	1042	276	640	1	84.15	78.89	1.89	1.37	99.6
PWMS2	1050	226	65	4	15.82	15.15	1.07	0.88	98.2
PWMS3	1060	184	No Capping	0	17.59	17.59	0.55	0.55	100.0
TFMS	1070	77	300	1	107.17	101.19	0.99	0.78	98.7
DeepMS1	1080	199	60	2	12.72	12.59	0.87	0.82	99.0
DeepMS2	1090	128	100	1	20.18	19.90	0.89	0.82	99.2
DDeepMS	1100-1110	24	No Capping	0	49.05	49.05	0.86	0.86	100.0
PWMS4	1120	23	70	1	34.69	21.18	2.18	0.92	95.7
PWMS5	1130	31	70	4	89.22	28.41	2.79	0.87	87.1
90AU	2010	1266	400	17	40.90	38.14	2.13	1.79	98.7

Table 14.8 Capped Composite Values of the Open Pit Model									
Ag Capping									
Domains	Rock Type	Total No. of Composites	Capping Value Ag (g/t)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
NSAU	2020	521	350	12	65.01	38.78	5.19	1.82	97.7
PMAU	2030-2070	359	350	4	36.93	36.02	1.81	1.71	98.9
PWGossan	2080	95	400	8	202.64	109.99	3.80	1.08	91.6
PWGossanLeach	2086	96	210	2	50.11	44.77	1.46	1.05	97.9
EastGossan	2090	100	No Capping	0	5.07	5.07	0.94	0.94	100.0
EAST_OX	2100	148	20	1	1.70	1.46	2.92	1.90	99.3
PWGSN2	2200	24	120	2	40.64	33.91	1.39	1.00	91.7
MainSTR	3000	1224	100	1	7.05	6.62	2.81	1.39	99.9
MainSTR1-12	3010-3120	525	250	3	19.01	17.80	2.45	2.03	99.4
PWSTR	3130-3140	194	200	2	18.34	16.42	2.55	1.91	99.0
Deep STRs	3150-3155	80	100	1	25.85	20.25	2.37	0.96	98.8
TFSTRs	3160-3230	2349	260	14	17.94	16.97	2.41	2.00	99.4

Note: *CoV = Coefficient of Variation*

Table 14.8 Capped Composite Values of the Open Pit Model									
Zn Capping									
Domains	Rock Type	Total No. of Composites	Capping Value Zn (%)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
MMS1ZNLG	1010	507	4	4	0.22	0.20	3.03	2.62	99.2
MainMSZNHG	1015	921	32	5	8.31	8.29	0.85	0.83	99.5
MainMS2	1020	621	No Capping	0	4.77	4.77	1.23	1.23	100.0
MainMS3	1030	939	32	1	6.45	6.44	0.95	0.94	99.9
PWSMSSCu	1040	288	2	3	0.30	0.26	2.31	1.21	99.0

TABLE 14.8
CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL

ZN CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Zn (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
PWMSZNCU	1041	224	No Capping	0	7.39	7.39	0.98	0.98	100.0
PWMSCR	1042	276	2	2	0.10	0.10	3.19	2.64	99.3
PWMS2	1050	226	No Capping	0	2.52	2.52	1.88	1.88	100.0
PWMS3	1060	184	10	3	0.83	0.79	2.36	2.20	98.4
TFMS	1070	77	No Capping	0	11.58	11.58	0.70	0.70	100.0
DeepMS1	1080	199	13	1	1.33	1.31	2.13	2.08	99.5
DeepMS2	1090	128	2	2	0.29	0.20	3.50	1.77	98.4
DDeepMS	1100-1110	24	No Capping	0	1.54	1.54	1.33	1.33	100.0
PWMS4	1120	23	No Capping	0	0.41	0.41	1.51	1.51	100.0
PWMS5	1130	31	No Capping	0	1.90	1.90	1.00	1.00	100.0
90AU	2010	1266	6	11	0.41	0.37	3.06	2.27	99.1
NSAU	2020	521	4	3	0.23	0.16	6.32	3.23	99.4
PMAU	2030-2070	359	7	4	0.62	0.50	3.19	2.12	98.9
PWGossan	2080	95	No Capping	0	0.01	0.01	1.15	1.15	100.0
PWGossanLeach	2086	96	No Capping	0	0.01	0.01	1.10	1.10	100.0
EastGossan	2090	100	No Capping	0	0.02	0.02	1.71	1.71	100.0
EAST_OX	2100	148	No Capping	0	0.02	0.02	4.12	4.12	100.0
PWGSN2	2200	24	No Capping	0	0.03	0.03	0.76	0.76	100.0
MainSTR	3000	1224	10	5	0.76	0.75	2.02	1.94	99.6
MainSTR1-12	3010-3120	525	10	4	0.75	0.73	2.23	2.08	99.2
PWSTR	3130-3140	194	No Capping	0	0.12	0.12	1.45	1.45	100.0
Deep STRs	3150-3155	80	3	3	0.61	0.38	3.13	1.71	96.3
TFSTRs	3160-3230	2349	20	2	1.11	1.11	1.60	1.55	99.9

Note: *CoV = Coefficient of Variation*

Table 14.8 Capped Composite Values of the Open Pit Model									
Cu Capping									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Cu (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	1427	No Capping	0	0.33	0.33	1.08	1.08	100.0
MainMS2	1020	621	No Capping	0	0.41	0.41	1.19	1.19	100.0
MainMS3	1030	939	No Capping	0	0.23	0.23	1.42	1.42	100.0
PWSMSSCu	1040	288	2	1	0.47	0.46	0.93	0.72	99.7
PWMSZNCU	1041	224	4	4	0.54	0.50	1.73	1.27	98.2
PWMSCR	1042	276	No Capping	0	2.58	2.58	1.00	1.00	100.0
PWMS2	1050	226	No Capping	0	0.32	0.32	0.88	0.88	100.0
PWMS3	1060	184	No Capping	0	0.94	0.94	0.75	0.75	100.0
TFMS	1070	77	No Capping	0	0.08	0.08	1.02	1.02	100.0
DeepMS1	1080	199	No Capping	0	0.43	0.43	1.08	1.08	100.0
DeepMS2	1090	128	No Capping	0	0.59	0.59	0.68	0.68	100.0
DDeepMS	1100-1110	24	No Capping	0	0.23	0.23	1.00	1.00	100.0
PWMS4	1120	23	No Capping	0	0.32	0.32	0.96	0.96	100.0
PWMS5	1130	31	No Capping	0	0.26	0.26	1.28	1.28	100.0
90AU	2010	1266	No Capping	0	0.02	0.02	1.40	1.40	100.0
NSAU	2020	521	3	4	0.12	0.09	5.68	3.45	99.2
PMAU	2030-2070	359	No Capping	0	0.07	0.07	1.88	1.88	100.0
PWGossan	2080	95	5	3	0.94	0.74	2.29	1.64	96.8
PWGossanLeach	2086	96	No Capping	0	0.13	0.13	1.50	1.50	100.0
EastGossan	2090	100	No Capping	0	0.02	0.02	2.18	2.18	100.0
EAST_OX	2100	148	No Capping	0	0.01	0.01	2.02	2.02	100.0
PWGSN2	2200	24	No Capping	0	0.32	0.32	0.88	0.88	100.0
MainSTR	3000	1224	2	3	0.19	0.19	1.44	1.35	99.8
MainSTR1-12	3010-3120	525	No Capping	0	0.24	0.24	1.47	1.47	100.0
PWSTR	3130-3140	194	No Capping	0	0.25	0.25	1.15	1.15	100.0
Deep STRs	3150-3155	80	No Capping	0	0.87	0.87	0.94	0.94	100.0

TABLE 14.8 CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL									
CU CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Cu (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
TFSTRs	3160-3230	2349	No Capping	0	0.02	0.02	1.39	1.39	100.0

Note: *CoV = Coefficient of Variation*

TABLE 14.8 CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL									
PB CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Pb (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	1427	3	4	0.10	0.10	3.17	2.64	99.7
MainMS2	1020	621	No Capping	0	0.06	0.06	3.11	3.11	100.0
MainMS3	1030	939	2.5	7	0.22	0.21	2.18	1.81	99.3
PWSMSSCu	1040	288	0.6	1	0.07	0.06	2.37	1.10	99.7
PWMSZNCU	1041	224	No Capping	0	0.65	0.65	1.98	1.98	100.0
PWMSCR	1042	276	No Capping	0	0.02	0.02	0.85	0.85	100.0
PWMS2	1050	226	No Capping	0	0.11	0.11	1.44	1.44	100.0
PWMS3	1060	184	No Capping	0	0.07	0.07	0.96	0.96	100.0
TFMS	1070	77	No Capping	0	3.31	3.31	0.83	0.83	100.0
DeepMS1	1080	199	No Capping	0	0.09	0.09	1.67	1.67	100.0
DeepMS2	1090	128	No Capping	0	0.11	0.11	1.39	1.39	100.0
DDeepMS	1100-1110	24	No Capping	0	0.66	0.66	1.14	1.14	100.0
PWMS4	1120	23	No Capping	0	0.17	0.17	1.95	1.95	100.0
PWMS5	1130	31	No Capping	0	0.18	0.18	0.67	0.67	100.0
90AU	2010	1266	3	13	0.26	0.25	2.23	1.84	99.0
NSAU	2020	521	2	2	0.08	0.07	3.70	2.95	99.6

TABLE 14.8
CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL

PB CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value Pb (%)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
PMAU	2030-2070	359	6	3	0.58	0.56	1.91	1.83	99.2
PWGossan	2080	95	No Capping	0	0.03	0.03	2.62	2.62	100.0
PWGossanLeach	2086	96	No Capping	0	0.05	0.05	2.02	2.02	100.0
EastGossan	2090	100	No Capping	0	0.02	0.02	1.21	1.21	100.0
EAST_OX	2100	148	No Capping	0	0.01	0.01	1.63	1.63	100.0
PWGSN2	2200	24	No Capping	0	0.13	0.13	0.57	0.57	100.0
MainSTR	3000	1224	1.5	1	0.04	0.04	4.78	2.92	99.9
MainSTR1-12	3010-3120	525	2.5	5	0.16	0.15	2.95	2.47	99.0
PWSTR	3130-3140	194	No Capping	0	0.05	0.05	2.61	2.61	100.0
Deep STRs	3150-3155	80	2	1	0.36	0.19	4.93	2.10	98.8
TFSTR	3160-3230	2349	4.5	3	0.29	0.29	1.74	1.66	99.9

Note: *CoV = Coefficient of Variation*

TABLE 14.8
CAPPED COMPOSITE VALUES OF THE OPEN PIT MODEL

S CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value S (%)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
MS Zones	1010-1130	2453	No capping	0	37.79	37.79	0.39	0.39	100.0
AU Zones	2010-2200	1432	30	15	3.77	3.67	1.42	1.28	99.0
Stringers	3000-3220	2887	No capping	0	11.88	11.88	0.79	0.79	100.0
Dykes	4001-4007	3877	15	36	0.97	0.88	2.90	2.31	99.1

Note: *CoV = Coefficient of Variation*

TABLE 14.9
CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL

AU CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value Au (g/t)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
MainMS1	1010	457	15	4	2.71	2.64	1.00	0.86	99.1
MainMS1 ZNHG	1015	909	22	3	1.86	1.77	1.91	1.36	99.7
MainMS2	1020	568	30	2	3.37	3.33	1.25	1.15	99.6
MainMS3	1030	960	20	9	1.91	1.83	1.76	1.46	99.1
PWSMSSCu	1040	235	10	2	2.10	1.98	1.13	0.76	99.1
PWMSZNCU	1041	210	4	1	0.87	0.83	1.12	0.73	99.5
PWMSCR	1042	280	7	3	2.01	1.97	0.71	0.59	98.9
PWMS2	1050	200	7	3	1.38	1.26	1.49	1.01	98.5
PWMS3	1060	175	5	2	1.52	1.50	0.59	0.51	98.9
TFMS	1070	90	No Capping	0	1.83	1.83	0.89	0.89	100.0
DeepMS1	1080	205	10	2	2.26	2.22	0.89	0.79	99.0
DeepMS2	1090	134	15	2	2.00	1.91	1.41	1.18	98.5
DDeepMS	1100-1110	27	No Capping	0	5.76	5.76	1.53	1.53	100.0
PWMS4	1120	21	No Capping	0	2.42	2.42	0.93	0.93	100.0
PWMS5	1130	26	No Capping	0	1.59	1.59	1.25	1.25	100.0
90AU	2010	768	30	3	3.20	3.17	1.42	1.36	99.6
NSAU	2020	372	55	5	6.43	6.00	1.94	1.53	98.7
PMAU	2030-2070	310	40	5	4.30	3.89	2.22	1.76	98.4
PWGossan	2080-2086	31	15	1	5.77	3.87	2.20	0.86	96.8
EastGossan	2090	100	70	3	18.44	16.41	1.44	1.13	97.0
EAST_OX	2100	140	No Capping	0	0.83	0.83	1.55	1.55	100.0
PWGSN2	2200	27	No Capping	0	2.03	2.03	1.05	1.05	100.0
MainSTR	3000	292	10	1	1.91	1.83	1.30	0.93	99.7
MainSTR0	3001	553	20	1	2.20	2.14	1.49	1.19	99.8
MainSTR1-12	3010-3120	248	15	4	1.85	1.73	1.71	1.41	98.4
PWSTR	3130-3140	97	20	3	2.89	2.49	2.02	1.51	96.9

Table 14.9 Capped Composite Values of the Underground Model									
Au Capping									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Au (g/t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
Deep STRs	3150-3155	106	No Capping	0	1.59	1.59	0.97	0.97	100.0
TFSTR2	3180	237	8	2	1.11	1.09	1.33	1.23	99.2
TFSTR3	3190	1026	10	3	0.99	0.97	1.40	1.23	99.7
TFSTR4	3200	372	No Capping	0	0.86	0.86	0.72	0.72	100.0
TFSTRs	3160-3230	108	16	1	2.20	2.10	1.61	1.38	99.1

Note: *CoV = Coefficient of Variation*

Table 14.9 Capped Composite Values of the Underground Model									
Ag Capping									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Ag (g/t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	457	80	2	12.72	12.66	0.89	0.86	99.6
MainMS1 ZNHG	1015	909	100	10	17.35	16.31	1.37	0.93	98.9
MainMS2	1020	568	130	10	20.45	19.65	1.30	1.12	98.2
MainMS3	1030	960	180	11	25.58	24.16	1.60	1.27	98.9
PWSMSSCu	1040	235	60	5	25.17	14.97	4.73	0.70	97.9
PWMSZNCU	1041	210	120	5	32.77	30.74	1.16	0.92	97.6
PWMSCR	1042	280	450	6	101.79	77.51	3.12	1.23	97.9
PWMS2	1050	200	80	2	16.39	15.90	1.07	0.92	99.0
PWMS3	1060	175	No Capping	0	17.33	17.33	0.57	0.57	100.0
TFMS	1070	90	300	1	95.62	93.73	0.85	0.80	98.9
DeepMS1	1080	205	65	1	12.69	12.59	0.85	0.82	99.5

Table 14.9 Capped Composite Values of the Underground Model									
Ag Capping									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Ag (g/t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
DeepMS2	1090	134	100	1	19.88	19.61	0.90	0.84	99.3
DDeepMS	1100-1110	27	No Capping	0	50.76	50.76	0.82	0.82	100.0
PWMS4	1120	21	70	1	35.72	20.92	2.21	0.97	95.2
PWMS5	1130	26	135	2	95.59	34.89	3.00	1.22	92.3
90AU	2010	768	600	7	58.37	56.56	1.89	1.73	99.1
NSAU	2020	372	400	11	88.51	51.39	4.55	1.69	97.0
PMAU	2030-2070	310	400	3	42.24	41.06	1.82	1.70	99.0
PWGossan	2080-2086	31	750	1	231.29	214.88	1.13	0.96	96.8
EastGossan	2090	100	No Capping	0	5.11	5.11	0.93	0.93	100.0
EAST_OX	2100	140	25	1	1.92	1.71	2.78	2.05	99.3
PWGSN2	2200	27	150	1	36.47	32.01	1.47	1.12	96.3
MainSTR	3000	292	55	4	8.80	8.54	1.31	1.19	98.6
MainSTR0	3001	553	100	2	9.09	8.06	3.14	1.37	99.6
MainSTR1-12	3010-3120	248	250	5	33.48	30.42	2.00	1.64	98.0
PWSTR	3130-3140	97	200	2	29.03	23.45	2.53	1.61	97.9
Deep STRs	3150-3155	106	100	1	24.75	20.52	2.20	1.00	99.1
TFSTR2	3180	237	260	5	43.23	41.65	1.38	1.26	97.9
TFSTR3	3190	1026	200	6	15.66	15.18	2.00	1.79	99.4
TFSTR4	3200	372	No Capping	0	4.78	4.78	0.99	0.99	100.0
TFSTRs	3160-3230	108	520	2	83.64	80.79	1.60	1.51	98.1

Note: *CoV = Coefficient of Variation*

TABLE 14.9
CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL

ZN CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Zn (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MMS1	1010	457	4	4	0.23	0.20	3.52	2.77	99.1
MainMSZNHG	1015	909	40	1	8.46	8.45	0.84	0.83	99.9
MainMS2	1020	568	No Capping	0	5.20	5.20	1.16	1.16	100.0
MainMS3	1030	960	No Capping	0	6.34	6.34	0.97	0.97	100.0
PWSMSSCu	1040	235	2	3	0.31	0.28	1.74	1.22	98.7
PWMSZNCU	1041	210	No Capping	0	8.16	8.16	0.90	0.90	100.0
PWMSCR	1042	280	2	2	0.11	0.09	3.67	2.66	99.3
PWMS2	1050	200	16	7	2.78	2.52	1.78	1.55	96.5
PWMS3	1060	175	7	5	0.88	0.76	2.29	1.89	97.1
TFMS	1070	90	No Capping	0	10.37	10.37	0.83	0.83	100.0
DeepMS1	1080	205	13	1	1.31	1.28	2.16	2.16	99.5
DeepMS2	1090	134	2	2	0.29	0.20	3.44	1.76	98.5
DDeepMS	1100-1110	27	No Capping	0	1.84	1.84	1.11	1.11	100.0
PWMS4	1120	21	No Capping	0	0.46	0.46	1.39	1.39	100.0
PWMS5	1130	26	No Capping	0	2.41	2.41	0.79	0.79	100.0
90AU	2010	768	6	6	0.36	0.30	4.48	2.74	99.2
NSAU	2020	372	7	2	0.27	0.19	6.29	3.82	99.5
PMAU	2030-2070	310	7	2	0.55	0.49	2.84	2.03	99.4
PWGossan	2080-2086	31	No Capping	0	0.01	0.01	1.48	1.48	100.0
EastGossan	2090	100	No Capping	0	0.02	0.02	1.68	1.68	100.0
EAST_OX	2100	140	No Capping	0	0.02	0.02	3.99	3.99	100.0
PWGSN2	2200	27	No Capping	0	0.03	0.03	0.72	0.72	100.0
MainSTR	3000	292	9	1	1.22	1.20	1.54	1.46	99.7
MainSTR0	3001	553	10	4	0.72	0.70	2.38	2.28	99.3
MainSTR1-12	3010-3120	248	10	4	1.13	1.07	2.05	1.87	98.4
PWSTR	3130-3140	97	2	2	0.30	0.19	3.61	1.89	97.9

TABLE 14.9 **CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL**									
ZN CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Zn (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
Deep STRs	3150-3155	106	8	1	0.60	0.54	2.93	2.41	99.1
TFSTR2	3180	237	10	2	1.05	1.02	2.03	1.89	99.2
TFSTR3	3190	1026	15	3	1.47	1.45	1.46	1.32	99.7
TFSTR4	3200	372	7	2	1.00	0.98	1.38	1.29	99.5
TFSTRs	3160-3230	108	No Capping	0	1.21	1.21	2.39	2.39	100.0

Note: *CoV = Coefficient of Variation*

TABLE 14.9 **CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL**									
CU CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Cu (%t)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	457	No Capping	0	0.52	0.52	0.88	0.88	100.0
MainMS1 ZNHG	1015	909	No Capping	0	0.24	0.24	1.03	1.03	100.0
MainMS2	1020	568	No Capping	0	0.44	0.44	1.14	1.14	100.0
MainMS3	1030	960	No Capping	0	0.25	0.25	1.40	1.40	100.0
PWSMSSCu	1040	235	2	1	0.52	0.50	0.89	0.66	99.6
PWMSZNCU	1041	210	2.5	4	0.56	0.48	1.71	1.00	98.1
PWMSCR	1042	280	No Capping	0	2.52	2.52	1.00	1.00	100.0
PWMS2	1050	200	No Capping	0	0.35	0.35	0.82	0.82	100.0
PWMS3	1060	175	No Capping	0	0.96	0.96	0.75	0.75	100.0
TFMS	1070	90	No Capping	0	0.07	0.07	1.03	1.03	100.0
DeepMS1	1080	205	2	3	0.43	0.41	1.06	0.90	98.5

TABLE 14.9
CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL

CU CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value Cu (%t)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
DeepMS2	1090	134	No Capping	0	0.58	0.58	0.71	0.71	100.0
DDeepMS	1100-1110	27	No Capping	0	0.23	0.23	1.02	1.02	100.0
PWMS4	1120	21	No Capping	0	0.32	0.32	0.98	0.98	100.0
PWMS5	1130	26	No Capping	0	0.23	0.23	1.47	1.47	100.0
90AU	2010	768	No Capping	0	0.02	0.02	1.49	1.49	100.0
NSAU	2020	372	1	6	0.14	0.07	5.68	2.36	98.4
PMAU	2030-2070	310	No Capping	0	0.08	0.08	1.98	1.98	100.0
PWGossan	2080-2086	31	No Capping	0	1.00	1.00	1.62	1.62	100.0
EastGossan	2090	100	No Capping	0	0.02	0.02	2.17	2.17	100.0
EAST_OX	2100	140	No Capping	0	0.01	0.01	1.96	1.96	100.0
PWGSN2	2200	27	No Capping	0	0.31	0.31	0.82	0.82	100.0
MainSTR	3000	292	No Capping	0	0.14	0.14	1.35	1.35	100.0
MainSTR0	3001	553	2	5	0.28	0.26	2.00	1.30	99.1
MainSTR1-12	3010-3120	248	No Capping	0	0.27	0.27	1.50	1.50	100.0
PWSTR	3130-3140	97	No Capping	0	0.35	0.35	0.93	0.93	100.0
Deep STRs	3150-3155	106	4	1	0.72	0.71	1.07	1.01	99.1
TFSTR2	3180	237	No Capping	0	0.02	0.02	1.64	1.64	100.0
TFSTR3	3190	1026	No Capping	0	0.03	0.03	1.19	1.19	100.0
TFSTR4	3200	372	No Capping	0	0.03	0.03	1.62	1.62	100.0
TFSTRs	3160-3230	108	No Capping	0	0.03	0.03	1.34	1.34	100.0

Note: *CoV = Coefficient of Variation*

TABLE 14.9 CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL									
PB CAPPING									
Domains	**Rock Type**	**Total No. of Composites**	**Capping Value Pb (%)**	**No. of Capped Composites**	**Mean of Composites**	**Mean of Capped Composites**	**CoV of Composites**	**CoV of Capped Composites**	**Capping Percentile**
MainMS1	1010	457	No Capping	0	0.05	0.05	1.83	1.83	100.0
MainMSZNHG	1015	909	2	6	0.13	0.12	3.05	2.31	99.3
MainMS2	1020	568	0.7	10	0.06	0.05	3.08	2.33	98.2
MainMS3	1030	960	3	5	0.23	0.23	2.20	1.88	99.5
PWSMSSCu	1040	235	0.6	1	0.07	0.06	2.58	1.23	99.6
PWMSZNCU	1041	210	7	2	0.70	0.68	1.95	1.89	99.0
PWMSCR	1042	280	No Capping	0	0.02	0.02	0.95	0.95	100.0
PWMS2	1050	200	0.5	3	0.11	0.10	1.42	0.91	98.5
PWMS3	1060	175	No Capping	0	0.07	0.07	0.96	0.96	100.0
TFMS	1070	90	No Capping	0	3.04	3.04	0.87	0.87	100.0
DeepMS1	1080	205	No Capping	0	0.09	0.09	1.64	1.64	100.0
DeepMS2	1090	134	No Capping	0	0.10	0.10	1.40	1.40	100.0
DDeepMS	1100-1110	27	No Capping	0	0.71	0.71	1.10	1.10	100.0
PWMS4	1120	21	No Capping	0	0.18	0.18	1.95	1.95	100.0
PWMS5	1130	26	No Capping	0	0.20	0.20	0.71	0.71	100.0
90AU	2010	768	3	14	0.28	0.25	2.66	2.09	98.2
NSAU	2020	372	2	2	0.09	0.08	3.65	2.91	99.5
PMAU	2030-2070	310	6	4	0.66	0.64	1.87	1.74	98.7
PWGossan	2080-2086	31	No Capping	0	0.03	0.03	1.03	1.03	100.0
EastGossan	2090	100	No Capping	0	0.03	0.03	1.19	1.19	100.0
EAST_OX	2100	140	No Capping	0	0.01	0.01	2.61	2.61	100.0
PWGSN2	2200	27	No Capping	0	0.12	0.12	0.72	0.72	100.0
MainSTR	3000	292	0.5	8	0.05	0.04	3.42	2.65	97.3
MainSTR0	3001	553	1	2	0.06	0.05	4.65	2.08	99.6
MainSTR1-12	3010-3120	248	2.5	3	0.28	0.25	2.48	1.94	98.8

TABLE 14.9
CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL

PB CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value Pb (%)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
PWSTR	3130-3140	97	No Capping	0	0.08	0.08	2.34	2.34	100.0
Deep STRs	3150-3155	106	2	1	0.34	0.21	4.62	1.93	99.1
TFSTR2	3180	237	4	1	0.35	0.33	2.08	1.74	99.6
TFSTR3	3190	1026	3	6	0.39	0.39	1.46	1.33	99.4
TFSTR4	3200	372	0.7	4	0.10	0.09	1.49	1.22	98.9
TFSTRs	3160-3230	108	No Capping	0	0.48	0.48	1.96	1.96	100.0

Note: *CoV = Coefficient of Variation*

TABLE 14.9
CAPPED COMPOSITE VALUES OF THE UNDERGROUND MODEL

S CAPPING

Domains	Rock Type	Total No. of Composites	Capping Value S (%)	No. of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
MS	1010-1130	2350	No capping	0	38.01	38.01	0.39	0.39	100.0
AU Zones	2010-2200	987	30	15	3.76	3.60	1.63	1.44	98.5
Stringers	3000-3220	2087	No capping	0	12.51	12.51	0.81	0.81	100.0
Dykes	4001-4007	3887	15	37	0.98	0.89	2.87	2.29	99.0

Note: *CoV = Coefficient of Variation*

14.8 SEMI-VARIOGRAPHY

A semi-variography study was performed as a guide to determining a grade interpolation search strategy. Omni, along strike, down dip and across dip semi-variograms were attempted for each zone and each element (Au, Ag, Zn, Cu and Pb) using capped composites. Selected variograms are attached in Appendix E for the open pit model and in Appendix F for the underground model.

Continuity ellipses based on the observed ranges were subsequently generated and used as the basis for estimation search ranges, distance weighting calculations and Mineral Resource classification criteria.

14.9 BULK DENSITY

A total of 1,487 bulk density measurements were provided by Aquila, of which 892 measurements were analyzed for sulphur. It appeared that the bulk density correlated well with the sulphur content. Figure 14.3 and Figure 14.4 show a correlation between bulk density and sulphur for massive sulphide (including semi-massive sulphide) and non-massive sulphide mineralization (stringers and gold zones), respectively. There were more sulphur assays available compared to bulk density measurements, and the correlation between density and sulphur was good. P&E estimated the bulk density of the mineralization blocks, except blocks of the gossan domain, using linear regression of density and sulphur, which was interpolated with composites. A uniform bulk density of 3.18 t/m^3, an average of 53 samples of gossan, was applied to all domains of gossan, since the measured bulk density values correlated poorly with sulphur in the gossan. Bulk densities of 2.0 t/m^3 and 2.7 t/m^3 were employed for overburden and waste, respectively.

FIGURE 14.3 CORRELATION BETWEEN BULK DENSITY AND SULPHUR FOR MASSIVE SULPHIDE (INCLUDING SEMI-MASSIVE SULPHIDE)

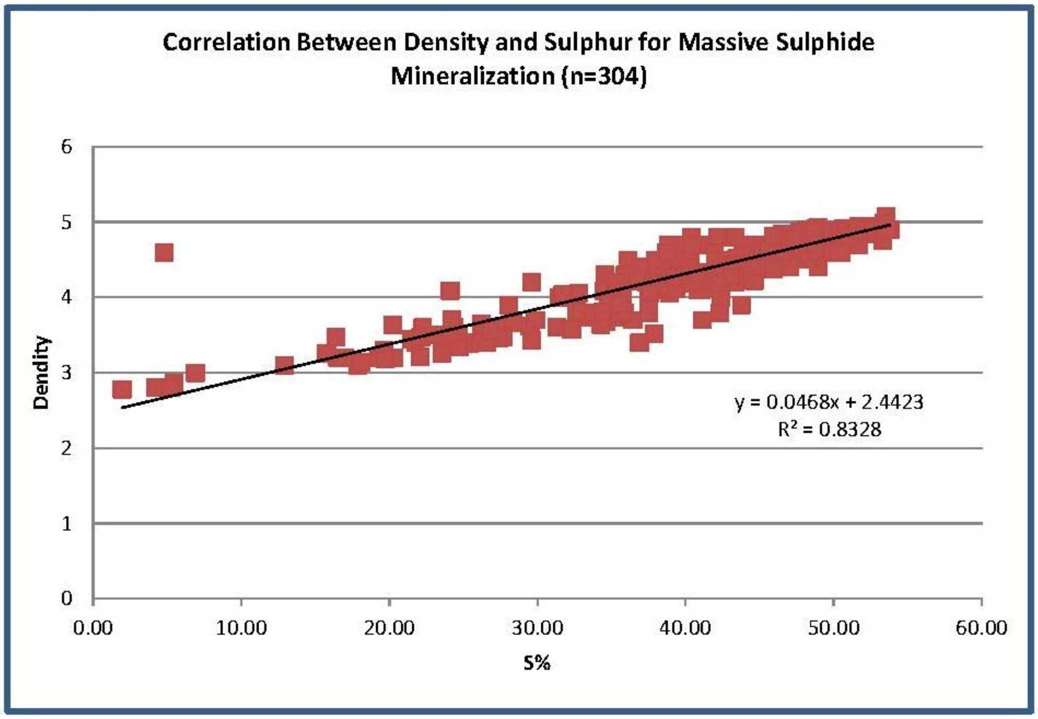
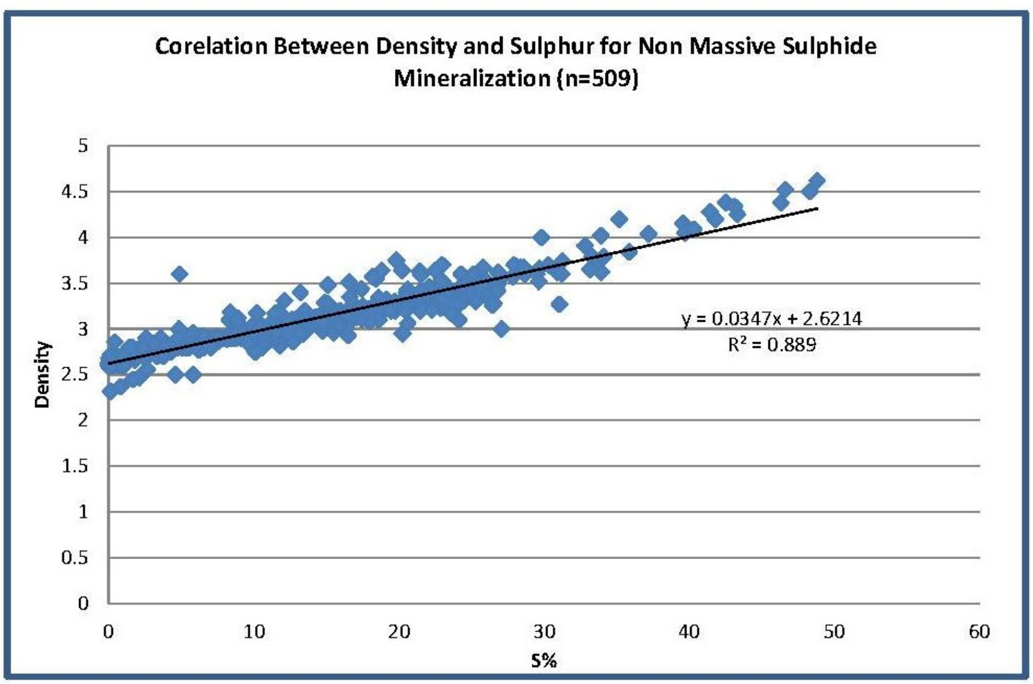


FIGURE 14.4 CORRELATION OF BULK DENSITY AND SULPHUR FOR NON-MASSIVE SULPHIDE MINERALIZATION (STRINGERS AND GOLD ZONES)



Yungang Wu, P.Geo. and Eugene Puritch, P,Eng. of P&E collected 13 verification samples during their site visit on May 23, 2016. The samples were analyzed at AGAT Laboratories in Mississauga, and an average bulk density of 3.63 t/m^3 was attained with a variance between 2.65 to 4.61 t/m^3.

Yungang Wu, P.Geo. of P&E collected 12 additional verification samples during his site visit on November 13, 2017. The samples were analyzed at AGAT Laboratories in Mississauga, and an average bulk density of 3.97 t/m^3 was attained with a variance between 2.82 to 4.91 t/m^3.

14.10 BLOCK MODELING

The block models of the Back Forty open pit and underground Mineral Resource Estimate were constructed using GEOVIA GEMS™ V6.8 modelling software, and the block model origin and block size are tabulated in Table 14.10. The block model consists of separate model attributes for estimated grade of Au, Ag, Cu, Zn, Pb and S, rock type, volume percent, bulk density, metallurgical type, NSR value and classification.

TABLE 14.10
BLOCK MODEL DEFINITION FOR OPEN PIT AND UNDERGROUND MINERAL RESOURCE ESTIMATE

Direction	Origin	No. of Blocks	Block Size (m)
X	434,353.5	240	5.0
Y	5,032,905.5	350	2.5
Z	240	300	2.5
Rotation	0^o		

All blocks in the rock type block model were initially assigned a waste rock code of 99, corresponding to the surrounding country rocks. All mineralized domains and waste dykes were used to code all blocks within the rock type block model that contain 1% or greater volume within the domains. These blocks were assigned their appropriate individual rock codes as indicated in Table 14.4. The overburden and topographic surfaces were subsequently utilized to assign rock code 10 and 0, corresponding to overburden and air respectively, to all blocks 50% or greater above the respective surfaces.

A volume percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining domains. As a result, the domain boundary was properly represented by the volume percent model ability to measure individual infinitely variable block inclusion percentages within that domain. The minimum coding percentage of the mineralized block was set to 1%.

The Au and Ag grade blocks were interpolated with Inverse Distance Cubed (“ID^3”), while Zn, Cu, Pb and S were interpolated with Inverse Distance Squared (“ID^2”). Multiple passes were executed for the grade interpolation to progressively capture the sample points in order to avoid over-smoothing and preserve local grade variability. Search ranges were based on the variograms and search ellipse orientations were aligned with the strike and dip directions of each domain/sub-domain accordingly. Grade blocks were interpolated using the parameters in Tables 14.11 and 14.12 for open pit and underground models, respectively.

TABLE 14.11
BLOCK MODEL INTERPOLATION PARAMETERS FOR THE OPEN PIT MODEL

Elements	Pass	Dip Range (m)	Strike Range (m)	Across Dip Range (m)	Max No. of Samples per Hole	Min No. of Samples	Max No. of Samples
Au	I	5-25	10-25	5-10	2	5	15
	II	10-35	20-40	10-20	2	3	15
	III	20-75	40-80	20-40	2	1	15
Ag	I	15-35	15-35	10-20	2	5	15
	II	25-55	25-55	15-35	2	3	15
	III	50-110	50-110	30-70	2	1	15
Zn	I	15-40	15-35	5-25	2	5	15
	II	25-65	25-55	10-40	2	3	15
	III	60-130	60-115	25-100	2	1	15
Cu	I	15-35	15-35	10-25	2	5	15
	II	25-60	25-60	15-40	2	3	15
	III	60-120	60-120	30-80	2	1	15
Pb	I	15-30	20-40	10-25	2	5	15
	II	25-50	30-65	15-45	2	3	15
	III	60-100	60-130	30-90	2	1	15
S	I	15-40	15-35	5-25	2	5	15
	II	25-65	25-55	10-40	2	3	15
	III	60-130	60-115	25-100	2	1	15

TABLE 14.12
BLOCK MODEL INTERPOLATION PARAMETERS FOR THE UNDERGROUND MODEL

Elements	Pass	Dip Range (m)	Strike Range (m)	Across Dip Range (m)	Max No. of Samples per Hole	Min No. of Samples	Max No. of Samples
Au	I	5-25	10-30	5-30	2	5	15
	II	10-40	20-50	10-50	2	3	15
	III	20-80	40-100	20-100	2	1	15
Ag	I	15-35	10-30	10-20	2	5	15
	II	25-55	20-50	15-35	2	3	15
	III	50-110	40-100	30-70	2	1	15
Zn	I	15-40	15-35	5-30	2	5	15
	II	25-65	25-55	10-50	2	3	15
	III	50-130	50-110	25-100	2	1	15
Cu	I	15-35	15-35	10-25	2	5	15
	II	25-60	25-60	15-40	2	3	15

TABLE 14.12 **BLOCK MODEL INTERPOLATION PARAMETERS FOR THE UNDERGROUND MODEL**							
Elements	**Pass**	**Dip Range (m)**	**Strike Range (m)**	**Across Dip Range (m)**	**Max No. of Samples per Hole**	**Min No. of Samples**	**Max No. of Samples**
	III	60-120	60-120	30-80	2	1	15
Pb	I	15-30	20-40	10-25	2	5	15
	II	25-50	30-65	15-45	2	3	15
	III	60-100	60-130	30-90	2	1	15
S	I	15-40	15-35	5-30	2	5	15
	II	25-65	25-55	10-50	2	3	15
	III	50-130	50-110	25-100	2	1	15

Selected cross-sections and plans of the Au and Zn grade blocks are presented in Appendix G and Appendix H, respectively. Selected cross-sections and plans of the NSR block values are presented in Appendix I.

The NSR values of the model blocks were manipulated for each metallurgical type and are presented in Table 14.4.

A bulk density model of the mineralization was manipulated using the linear regression of bulk density and sulphur. The regression formula used for massive and semi-massive sulphide was "Density=0.0468 x S% + 2.4423", while "Bulk Density = 0.0347 x S% +2.6214" was used for non-massive sulphide mineralization (stringer and gold zones). A uniform bulk density of 3.18 t/m^3 was utilized for Gossan domains. The resulting average densities of each metallurgical type are presented in Table 14.13, at zero NSR value basis for In-pit and Out-of-pit (relative to the pit design used for the 2018 Feasibility Study).

TABLE 14.13 **AVERAGE DENSITY BY METALLURGY TYPE**		
Metallurgy Type	**In-Pit Bulk Density (t/m^3)**	**UG Bulk Density (t/m^3)**
Main Zone (1)	4.04	3.70
PWMS and Gossan (2)	3.79	3.26
PW SMSS and Stringer (3)	3.59	3.43
PWMS Cu/Zn (4)	3.52	4.04
Tuff Zone (5)	2.89	2.96
Oxide (6)	2.78	2.81

14.11 MINERAL RESOURCE CLASSIFICATION

In P&E's opinion, the drilling, assaying and exploration work of the Back Forty Deposit supports this Mineral Resource Estimate and are sufficient to indicate a reasonable potential for economic extraction and thus qualify it as a Mineral Resource under the CIM definition standards. The Mineral Resources were classified as Measured, Indicated and Inferred based on the geological interpretation, semi-variogram performance and drill hole spacing. The Measured Mineral Resources were classified for the blocks interpolated by the grade interpolation Pass I in Table 14.12, above, which used at least five composites from a minimum of three drill holes; the Indicated Mineral Resources were defined for the blocks interpolated by the grade interpolation Pass II, which used at least three composites from a minimum of two holes; and Inferred Mineral Resources were categorized for all remaining grade populated blocks within the mineralized domains. The search ellipse passes for Au grade interpolation were used for classification of all Au domains (oxide metallurgical type), while the search ellipse passes for Zn grade interpolation were used for classification of all other domains. The classified blocks have been adjusted and smoothed to reasonably reflect their distribution. Selected classification block cross-sections and plans are attached in Appendix J.

14.12 NSR CALCULATION

The Mineral Resource Estimates were derived from applying NSR cut-off values to the block models and reporting the resulting tonnes and grades for potentially mineable areas. The following parameters were used to calculate the NSR values that determine the net block revenue for open pit and underground potentially economic portions of the constrained mineralization. The metal prices are the same as those used in the 2018 Feasibility Study.

NSR Value Calculation

Au Price	US$1,375/oz
Ag Price	US$22.27/oz
Cu Price	US$3.19/lb
Pb Price	US$1.15/lb
Zn Price	US$1.10/lb
Au Process Recovery	Variable 5.9% to 98.0%
Ag Process Recovery	Variable 11.3% to 91.2%
Cu Process Recovery	Variable 37.1% to 96.0%
Pb Process Recovery	Variable 63.1% to 91.7%
Zn Process Recovery	Variable 26.7% to 98.0%
Cu Concentrate Freight	$80/t conc.
Zn Concentrate Freight	$25/t conc.
Cu Smelter Treatment Charge	$70/t conc.
Zn Smelter treatment Charge	$220/t conc.
Cu Concentrate Insurance	$35/t conc.
Zn Concentrate Insurance	$10/t conc.
Concentrate Moisture	8%
Cu Refining Charge	$0.07/lb
Au Refining Charge	$6/oz
Ag Refining Charge	$0.40/oz
Au Smelter Payable	96.5%
Ag Smelter Payable	94.0%
Cu Smelter Payable	96.5%

Zn Smelter Payable	85.0%

Open Pit NSR Cut-Off Calculation

Open pit NSR cut-off values were determined from financial modelling after the 2018 Feasibility Study was completed.

Leach Total Cost	$21.4/t processed
Flotation Met Type 1 Total Cost	$12.4/t processed
Flotation Met Type 2,3,4,7,8 Total Cost	$12.0/t processed
Flotation Met Type 5 Total Cost	$13.1/t processed

Underground NSR Cut-Off Calculation

Underground Mining	$50/t mined
Leach Total Cost	$71.4/t processed
Flotation Met Type 1 Total Cost	$62.4/t processed
Flotation Met Type 2,3,4,7,8 Total Cost	$62.0/t processed
Flotation Met Type 5 Total Cost	$63.1/t processed

Design Pit

The open pit design was from the 2018 Feasibility Study.

14.13 MINERAL RESOURCE ESTIMATE

The resulting Mineral Resource Estimate, as of the effective date of this Technical Report, is tabulated in Table 14.14. P&E considers the mineralization of Back Forty to be potentially amenable to Open Pit and Underground extraction which are constrained by the pit design from the 2018 Feasibility Study. Note that P&E considers met type 2 and 7 to be the same material, and met type 4 and 8 to be the same material.

TABLE 14.14
BACK FORTY MINERAL RESOURCE ESTIMATE (1-7)

PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE

Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Flotable	Met1	Measured	12.4	3,696	2.13	253.1	14.26	1,694.0	0.32	26.06	0.08	6.2	4.92	400.9
		Indicated		1,162	2.07	77.5	15.96	596.5	0.24	6.06	0.12	3.0	4.03	103.3
		M+I		**4,858**	**2.12**	**330.7**	**14.66**	**2,290.5**	**0.30**	**32.12**	**0.09**	**9.2**	**4.71**	**504.2**
		Inferred		24	0.65	0.5	16.20	12.4	0.12	0.06	0.23	0.1	2.36	1.2
	Met2	Measured	12.0	419	1.97	26.6	74.51	1,004.6	2.43	22.47	0.02	0.2	0.11	1.0
		Indicated		111	3.02	10.8	74.04	265.0	1.46	3.58	0.03	0.1	0.08	0.2
		M+I		**531**	**2.19**	**37.4**	**74.41**	**1,269.6**	**2.23**	**26.05**	**0.02**	**0.2**	**0.10**	**1.2**
	Met3	Measured	12.0	176	1.82	10.3	15.57	88.2	0.48	1.86	0.02	0.1	0.12	0.5
		Indicated		141	1.36	6.2	14.10	64.1	0.36	1.12	0.03	0.1	0.13	0.4
		M+I		**318**	**1.62**	**16.5**	**14.91**	**152.3**	**0.43**	**2.99**	**0.02**	**0.2**	**0.12**	**0.9**
	Met4	Measured	12.0	93	0.94	2.8	29.83	89.5	0.73	1.49	0.07	0.1	5.02	10.3
		Indicated		41	1.02	1.3	27.72	36.6	0.85	0.77	0.07	0.1	4.65	4.2
		M+I		**134**	**0.96**	**4.2**	**29.19**	**126.1**	**0.76**	**2.26**	**0.07**	**0.2**	**4.91**	**14.5**
	Met5	Measured	13.1	2,142	0.78	53.6	12.01	827.2	0.02	1.05	0.30	14.0	1.17	55.0
		Indicated		1,243	0.96	38.5	28.85	1,153.0	0.03	0.87	0.75	20.5	2.38	65.4
		M+I		**3,386**	**0.85**	**92.1**	**18.19**	**1,980.3**	**0.03**	**1.92**	**0.46**	**34.6**	**1.61**	**120.4**
		Inferred		9	1.49	0.4	66.85	20.0	0.03	0.01	0.58	0.1	4.85	1.0
	Sub Total	Measured	12.0 +12.4 +13.1	6,527	1.65	346.5	17.65	3,703.4	0.37	52.94	0.14	20.6	3.25	467.8
		Indicated		2,699	1.55	134.4	24.37	2,115.2	0.21	12.40	0.40	23.8	2.91	173.4
		M+I		**9,226**	**1.62**	**480.9**	**19.62**	**5,818.7**	**0.32**	**65.34**	**0.22**	**44.4**	**3.15**	**641.2**
		Inferred		33	0.88	0.9	30.47	32.4	0.10	0.07	0.33	0.2	3.06	2.2
Leachable	Met6	Measured	21.4	535	5.44	93.6	34.81	598.5	0.05	0.57	0.13	1.5	0.20	2.4
		Indicated		1,641	2.09	110.3	38.37	2,024.9	0.03	1.15	0.28	10.0	0.41	14.7
		M+I		**2,176**	**2.91**	**203.9**	**37.50**	**2,623.4**	**0.04**	**1.71**	**0.24**	**11.6**	**0.36**	**17.0**
		Inferred		231	3.45	25.6	44.01	327.0	0.06	0.28	0.59	3.0	0.27	1.4

TABLE 14.14 BACK FORTY MINERAL RESOURCE ESTIMATE (1-7) PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE														
Metallurgy Type	**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Total	Measured	12.0	7,062	1.94	440.1	18.95	4,302.0	0.34	53.51	0.14	22.1	3.02	470.1
	Indicated	+12.4	4,341	1.75	244.7	29.67	4,140.1	0.14	13.55	0.35	33.8	1.97	188.1
	M+I	+13.1	**11,403**	**1.87**	**684.8**	**23.03**	**8,442.0**	**0.27**	**67.05**	**0.22**	**55.9**	**2.62**	**658.2**
	Inferred	+21.4	264	3.13	26.6	42.32	359.4	0.06	0.35	0.56	3.3	0.62	3.6

Notes 1-7 *at base of table.*

TABLE 14.14 BACK FORTY MINERAL RESOURCE ESTIMATE (1-7) UNDERGROUND MINERAL RESOURCE ESTIMATE															
Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Flotable	Met1	Measured	62.4	1,040	1.98	66.1	22.52	752.7	0.30	6.9	0.21	4.9	4.59	105.1
		Indicated		2,745	1.99	175.6	24.00	2,117.6	0.41	24.8	0.21	12.9	2.96	178.9
		M+I		**3,784**	**1.99**	**241.7**	**23.59**	**2,870.3**	**0.38**	**31.7**	**0.21**	**17.8**	**3.40**	**284.0**
		Inferred		350	3.44	38.6	40.30	452.8	0.57	4.4	0.42	3.2	0.93	7.2
	Met2	Measured	62.0	9	1.21	0.3	21.13	5.9	1.72	0.3	0.02	0.0	0.10	0.0
		Indicated		45	2.33	3.4	32.17	46.5	1.00	1.0	0.08	0.1	0.09	0.1
		M+I		**54**	**2.14**	**3.7**	**30.38**	**52.4**	**1.12**	**1.3**	**0.07**	**0.1**	**0.09**	**0.1**
	Met3	Measured	62.0	30	2.31	2.2	13.34	12.8	0.45	0.3	0.06	0.0	0.20	0.1
		Indicated		380	2.74	33.5	21.61	263.9	0.49	4.1	0.12	1.0	0.26	2.2
		M+I		**410**	**2.71**	**35.7**	**21.01**	**276.7**	**0.49**	**4.4**	**0.12**	**1.0**	**0.26**	**2.3**
		Inferred		65	5.19	10.9	30.89	64.6	0.35	0.5	0.25	0.4	0.20	0.3

Table 14.14 Back Forty Mineral Resource Estimate (1-7)															
Underground Mineral Resource Estimate															
Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
	Met4	Measured	62.0	139	1.26	5.6	30.51	136.4	0.43	1.3	0.86	2.6	7.38	22.6
		Indicated		1,761	1.31	74.0	24.57	1,391.7	0.53	20.4	0.42	16.3	5.55	215.4
		M+I		**1,900**	**1.30**	**79.6**	**25.01**	**1,528.1**	**0.52**	**21.7**	**0.45**	**19.0**	**5.68**	**238.0**
		Inferred		273	1.45	12.7	20.61	180.9	0.73	4.4	0.15	0.9	2.21	13.3
	Met5	Measured	63.1	94	1.68	5.1	27.45	82.6	0.04	0.1	0.96	2.0	3.25	6.7
		Indicated		410	1.67	22.0	39.57	521.0	0.04	0.4	0.76	6.9	3.25	29.4
		M+I		**503**	**1.67**	**27.1**	**37.31**	**603.6**	**0.04**	**0.5**	**0.80**	**8.9**	**3.25**	**36.1**
		Inferred		90	3.18	9.2	126.88	367.5	0.05	0.1	1.15	2.3	3.14	6.2
	Sub Total	Measured	62.0 +62.4 +63.1	1,311	1.88	79.3	23.50	990.3	0.31	9.0	0.33	9.5	4.66	134.6
		Indicated		5,341	1.80	308.5	25.28	4,340.7	0.43	50.7	0.32	37.2	3.62	426.0
		M+I		**6,651**	**1.81**	**387.8**	**24.93**	**5,331.0**	**0.41**	**59.7**	**0.32**	**46.7**	**3.82**	**560.6**
		Inferred		778	2.86	71.4	42.63	1,065.9	0.55	9.4	0.40	6.8	1.57	27.0
Leachable	Met6	Measured	71.4	72	8.10	18.7	59.46	137.4	0.08	0.1	0.10	0.2	0.17	0.3
		Indicated		145	4.11	19.2	51.89	242.2	0.15	0.5	0.31	1.0	0.42	1.3
		M+I		**217**	**5.43**	**37.9**	**54.39**	**379.6**	**0.13**	**0.6**	**0.24**	**1.2**	**0.34**	**1.6**
		Inferred		153	9.08	44.6	94.91	465.9	0.09	0.3	0.73	2.4	0.52	1.7
Total		Measured	62.0 +62.4 +63.1 +71.4	1,382	2.21	98.0	25.37	1,127.7	0.30	9.1	0.32	9.7	4.43	134.9
		Indicated		5,486	1.86	327.7	25.98	4,582.8	0.42	51.2	0.32	38.2	3.53	427.3
		M+I		**6,868**	**1.93**	**425.7**	**25.86**	**5,710.6**	**0.40**	**60.3**	**0.32**	**47.9**	**3.71**	**562.2**
		Inferred		930	3.88	116.0	51.21	1,531.8	0.47	9.7	0.45	9.2	1.40	28.7

***Notes 1-7** at base of table.*

TABLE 14.14
BACK FORTY MINERAL RESOURCE ESTIMATE (1-7)

TOTAL MINERAL RESOURCE ESTIMATE

Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Flotable	Met1	Measured	12.4 +62.5	4,735	2.10	319.2	16.07	2,446.7	0.32	33.0	0.11	11.1	4.85	506.0
		Indicated		3,907	2.01	253.1	21.60	2,714.1	0.36	30.8	0.18	15.9	3.28	282.2
		M+I		**8,643**	**2.06**	**572.3**	**18.57**	**5,160.8**	**0.33**	**63.8**	**0.14**	**27.0**	**4.14**	**788.2**
		Inferred		373	3.26	39.1	38.77	465.2	0.55	4.5	0.41	3.4	1.02	8.4
	Met2	Measured	12.0 +62.0	428	1.96	26.9	73.43	1,010.5	2.42	22.8	0.02	0.2	0.11	1.0
		Indicated		156	2.82	14.2	62.00	311.5	1.33	4.6	0.04	0.1	0.08	0.3
		M+I		**584**	**2.19**	**41.1**	**70.37**	**1,322.0**	**2.13**	**27.4**	**0.02**	**0.3**	**0.10**	**1.3**
	Met3	Measured	12.0 +62.0	206	1.89	12.5	15.25	100.9	0.48	2.2	0.03	0.1	0.13	0.6
		Indicated		521	2.37	39.7	19.57	328.0	0.46	5.3	0.09	1.1	0.23	2.6
		M+I		**727**	**2.23**	**52.2**	**18.35**	**428.9**	**0.46**	**7.4**	**0.08**	**1.2**	**0.20**	**3.2**
		Inferred		65	5.19	10.9	30.89	64.6	0.35	0.5	0.25	0.4	0.20	0.3
	Met4	Measured	12.0 +62.0	232	1.13	8.4	30.24	225.9	0.55	2.8	0.54	2.8	6.43	33.0
		Indicated		1,802	1.30	75.3	24.65	1,428.3	0.53	21.2	0.41	16.4	5.53	219.6
		M+I		**2,035**	**1.28**	**83.8**	**25.28**	**1,654.1**	**0.54**	**24.0**	**0.43**	**19.2**	**5.63**	**252.6**
		Inferred		273	1.45	12.7	20.61	180.9	0.73	4.4	0.15	0.9	2.21	13.3
	Met5	Measured	13.1 +63.1	2,236	0.82	58.7	12.66	909.8	0.02	1.1	0.33	16.0	1.25	61.7
		Indicated		1,653	1.14	60.5	31.50	1,674.1	0.03	1.2	0.75	27.4	2.60	94.7
		M+I		**3,889**	**0.95**	**119.2**	**20.67**	**2,583.9**	**0.03**	**2.4**	**0.51**	**43.4**	**1.83**	**156.5**
		Inferred		99	3.02	9.6	121.26	387.5	0.05	0.1	1.09	2.4	3.30	7.2
	Sub Total	Measured	12.0 +12.4 +13.1 +62.0 +62.4 +63.1	7,838	1.69	425.8	18.63	4,693.7	0.36	61.9	0.17	30.1	3.49	602.4
		Indicated		8,040	1.71	442.8	24.97	6,455.9	0.36	63.1	0.34	61.0	3.38	599.4
		M+I		**15,878**	**1.70**	**868.6**	**21.84**	**11,149.7**	**0.36**	**125.0**	**0.26**	**91.1**	**3.43**	**1,201.8**
		Inferred		811	2.78	72.4	42.14	1,098.2	0.53	9.5	0.39	7.0	1.64	29.2

TABLE 14.14
BACK FORTY MINERAL RESOURCE ESTIMATE (1-7)

TOTAL MINERAL RESOURCE ESTIMATE

Metallurgy Type		**Classi-fication**	**NSR Cut-off ($/t)**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**	**Ag (g/t)**	**Ag (koz)**	**Cu (%)**	**Cu (Mlb)**	**Pb (%)**	**Pb (Mlb)**	**Zn (%)**	**Zn (Mlb)**
Leachable	Met6	Measured	21.4 +71.4	607	5.76	112.3	37.73	735.9	0.05	0.7	0.13	1.7	0.20	2.7
		Indicated		1,786	2.26	129.5	39.47	2,267.0	0.04	1.6	0.28	11.0	0.41	16.0
		M+I		**2,393**	**3.14**	**241.8**	**39.03**	**3,003.0**	**0.04**	**2.3**	**0.24**	**12.7**	**0.35**	**18.7**
		Inferred		384	5.69	70.2	64.26	792.9	0.07	0.6	0.65	5.5	0.37	3.1
Total		Measured	12.0 +12.4 +13.1 +21.4 +62.0 +62.4 +63.1 +71.4	8,444	1.98	538.1	20.00	5,429.7	0.34	62.6	0.17	31.8	3.25	605.0
		Indicated		9,827	1.81	572.4	27.61	8,722.9	0.30	64.7	0.33	72.0	2.84	615.4
		M+I		**18,271**	**1.89**	**1,110.4**	**24.09**	**14,152.6**	**0.32**	**127.3**	**0.26**	**103.8**	**3.03**	**1,220.5**
		Inferred		1,194	3.71	142.5	49.24	1,891.2	0.38	10.1	0.47	12.5	1.23	32.3

Notes:

1) *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.*
2) *P&E is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant factors that may materially affect the Mineral Resource Estimate.*
3) *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
4) *The Mineral Resources in this Technical Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
5) *Metallurgical type Oxide (all gold domains and leachable Gossans) is leachable, while all other metallurgical types are flotable.*
6) *The Mineral Resource Estimate was based on metal prices of US$1,375/oz gold, US$22.27/oz silver, US$1.10/lb zinc, US$3.19/lb copper and US$1.15/lb lead.*
7) *Open pit Mineral Resources were defined within the constraining pit design as per the 2018 Feasibility Study.*

Mineral Resources are sensitive to the selection of a reporting NSR cut-off value. The sensitivities of the NSR cut-off values are demonstrated in Table 14.15 and 14.16 for pit-constrained and underground Mineral Resources, respectively.

TABLE 14.15
PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Flotable	Met1	Measured	300	459	4.14	24.72	0.36	0.15	13.09
			250	770	3.35	21.45	0.34	0.13	11.91
			200	1,256	2.85	19.28	0.33	0.12	10.26
			150	1,873	2.61	17.90	0.34	0.10	8.42
			100	2,723	2.44	16.46	0.36	0.09	6.41
			12.4	3,696	2.13	14.26	0.32	0.08	4.92
			0	3,708	2.12	14.22	0.32	0.08	4.90
		Indicated	300	115	6.12	29.56	0.27	0.24	8.75
			250	223	4.47	24.29	0.24	0.21	9.08
			200	350	3.66	21.77	0.23	0.18	8.53
			150	528	3.09	19.68	0.23	0.16	7.48
			100	699	2.81	18.45	0.24	0.14	6.29
			12.4	1,162	2.07	15.96	0.24	0.12	4.03
			0	1,171	2.06	15.87	0.24	0.12	4.00
		Inferred	150	0.1	1.06	10.47	0.13	0.05	6.54
			100	5	0.99	10.94	0.20	0.09	4.80
			12.4	24	0.65	16.20	0.12	0.23	2.36
			0	24	0.63	15.84	0.12	0.22	2.29
	Met2	Measured	300	33	2.39	196.44	6.06	0.02	0.05
			250	62	2.37	161.54	5.26	0.02	0.08
			200	111	2.33	132.83	4.45	0.02	0.09
			150	189	2.25	108.21	3.67	0.02	0.09
			100	324	2.08	85.77	2.84	0.02	0.10
			12	419	1.97	74.51	2.43	0.02	0.11
			0	419	1.97	74.51	2.43	0.02	0.11

TABLE 14.15
PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
		Indicated	300	13	11.33	117.75	2.25	0.04	0.02
			250	18	8.93	116.75	2.79	0.03	0.02
			200	27	6.92	115.92	2.88	0.03	0.03
			150	38	5.61	111.99	2.62	0.03	0.03
			100	65	4.25	99.07	2.06	0.02	0.05
			12	111	3.02	74.04	1.46	0.03	0.08
			0	111	3.02	74.04	1.46	0.03	0.08
	Met3	Measured	200	0.4	5.30	26.96	1.03	0.02	0.15
			150	6	4.50	24.27	0.77	0.03	0.13
			100	35	3.08	19.80	0.71	0.03	0.12
			12	176	1.82	15.57	0.48	0.02	0.12
			0	179	1.80	15.41	0.47	0.02	0.12
		Indicated	150	1	4.09	21.82	0.76	0.06	0.15
			100	14	2.97	21.05	0.69	0.05	0.20
			12	141	1.36	14.10	0.36	0.03	0.13
			0	145	1.33	13.85	0.35	0.03	0.13
	Met4	Measured	300	0.2	1.67	31.17	1.54	0.01	10.83
			250	4	1.20	32.05	1.55	0.01	9.55
			200	17	1.14	30.67	1.19	0.03	8.60
			150	41	1.02	29.60	0.94	0.05	7.61
			100	66	0.98	31.21	0.85	0.06	6.27
			12	93	0.94	29.83	0.73	0.07	5.02
			0	93	0.94	29.83	0.73	0.07	5.02
		Indicated	300	0.1	1.62	33.84	2.94	0.01	6.29
			250	2	1.48	49.83	2.53	0.01	5.56
			200	5	1.54	48.93	2.08	0.02	5.21
			150	13	1.39	37.79	1.33	0.04	5.52
			100	35	1.07	28.44	0.93	0.06	5.01

Table 14.15
Pit-Constrained Mineral Resource Estimate Sensitivity

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
			12	41	1.02	27.72	0.85	0.07	4.65
			0	41	1.02	27.72	0.85	0.07	4.65
	Met5	Measured	300	5	1.53	110.64	0.09	4.67	16.44
			250	7	2.06	105.64	0.09	3.81	13.28
			200	13	2.73	84.31	0.07	2.52	8.47
			150	40	2.69	62.06	0.05	1.45	4.47
			100	139	1.94	43.76	0.04	0.86	2.77
			13.1	2,142	0.78	12.01	0.02	0.30	1.17
			0	2,161	0.77	11.94	0.02	0.30	1.16
		Indicated	300	115	1.91	132.98	0.12	4.64	14.71
			250	130	2.00	128.37	0.11	4.31	13.72
			200	146	2.10	123.46	0.10	3.95	12.62
			150	173	2.13	114.09	0.09	3.49	11.08
			100	237	1.99	95.04	0.08	2.71	8.62
			13.1	1,243	0.96	28.85	0.03	0.75	2.38
			0	1,249	0.96	28.75	0.03	0.75	2.38
		Inferred	300	1	1.24	102.34	0.05	1.12	13.32
			250	3	1.12	82.66	0.04	1.07	11.51
			200	4	1.20	82.52	0.04	1.01	10.77
			150	4	1.31	84.91	0.04	0.96	9.77
			100	8	1.49	69.62	0.03	0.63	5.56
			13.1	9	1.49	66.85	0.03	0.58	4.85
			0	9	1.49	66.85	0.03	0.58	4.85
Leachable	Met6	Measured	300	119	16.95	39.98	0.04	0.04	0.07
			250	142	15.16	41.19	0.04	0.05	0.08
			200	171	13.45	43.35	0.05	0.06	0.09
			150	217	11.32	46.42	0.07	0.07	0.10
			100	284	9.27	45.25	0.06	0.09	0.12

TABLE 14.15
PIT-CONSTRAINED MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
			21.4	535	5.44	34.81	0.05	0.13	0.20
			0	561	5.21	33.47	0.05	0.13	0.20
		Indicated	300	95	11.78	98.03	0.09	0.33	0.41
			250	125	10.28	97.54	0.08	0.42	0.43
			200	172	8.70	93.27	0.08	0.43	0.44
			150	262	6.87	86.55	0.07	0.40	0.42
			100	438	5.04	75.44	0.06	0.38	0.41
			21.4	1,641	2.09	38.37	0.03	0.28	0.41
			0	1,762	1.97	36.16	0.03	0.27	0.39
		Inferred	300	33	10.77	78.99	0.09	0.67	0.19
			250	44	9.55	77.13	0.09	0.72	0.19
			200	60	8.30	72.64	0.09	0.72	0.18
			150	81	7.07	68.28	0.08	0.73	0.16
			100	122	5.51	60.75	0.07	0.71	0.22
			21.4	231	3.45	44.01	0.06	0.59	0.27
			0	242	3.31	42.27	0.05	0.57	0.26

TABLE 14.16
UNDERGROUND MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Flotable	Met1	Measured	300	60	5.12	44.17	0.32	0.53	9.86
			250	139	3.71	35.89	0.31	0.43	9.31
			200	282	2.92	31.07	0.29	0.36	8.28
			150	517	2.44	28.36	0.29	0.31	6.84
			100	839	2.11	25.00	0.31	0.24	5.36
			62.4	1,040	1.98	22.52	0.30	0.21	4.59

TABLE 14.16
UNDERGROUND MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
			0	1,189	1.83	20.42	0.29	0.19	4.10
		Indicated	300	107	5.10	44.14	0.28	0.47	9.07
			250	215	3.87	35.71	0.27	0.39	9.08
			200	462	3.08	32.00	0.25	0.38	7.82
			150	977	2.59	31.28	0.33	0.33	5.81
			100	1,973	2.20	27.60	0.40	0.26	3.86
			62.4	2,745	1.99	24.00	0.41	0.21	2.96
			0	3,066	1.86	22.53	0.40	0.20	2.70
		Inferred	300	38	16.51	82.59	0.20	0.78	0.56
			250	56	12.86	72.25	0.57	0.65	0.49
			200	63	11.92	70.04	0.54	0.65	0.57
			150	114	7.64	65.64	0.66	0.62	0.92
			100	215	4.92	49.64	0.62	0.51	0.99
			62.4	350	3.44	40.30	0.57	0.42	0.93
			0	428	2.94	35.51	0.53	0.37	0.85
	Met2	Measured	150	0.2	1.95	29.35	2.89	0.02	0.06
			100	3	1.42	22.47	2.01	0.02	0.08
			62	9	1.21	21.13	1.72	0.02	0.10
			0	9	1.19	20.59	1.67	0.02	0.11
		Indicated	150	4	6.32	34.16	0.44	0.16	0.02
			100	15	3.75	53.96	0.79	0.10	0.03
			62	45	2.33	32.17	1.00	0.08	0.09
			0	58	2.02	28.85	0.85	0.07	0.08
	Met3	Measured	200	0.2	7.14	40.30	0.16	0.19	0.33
			150	0.4	6.00	31.76	0.19	0.16	0.31
			100	7	3.37	16.34	0.42	0.09	0.20
			62	30	2.31	13.34	0.45	0.06	0.20
			0	45	1.83	11.52	0.43	0.05	0.20

TABLE 14.16
UNDERGROUND MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
		Indicated	300	6	13.45	125.04	0.22	0.47	0.29
			250	13	10.96	88.72	0.22	0.40	0.26
			200	24	9.27	59.44	0.19	0.29	0.22
			150	41	7.39	50.52	0.30	0.24	0.20
			100	134	4.34	32.77	0.47	0.17	0.21
			62	380	2.74	21.61	0.49	0.12	0.26
			0	606	2.11	17.40	0.45	0.11	0.30
		Inferred	300	5	9.89	49.69	0.25	0.46	0.22
			250	21	9.13	45.76	0.24	0.42	0.21
			200	24	8.90	43.63	0.24	0.40	0.21
			150	30	8.04	40.86	0.27	0.37	0.20
			100	48	6.33	36.23	0.30	0.30	0.20
			62	65	5.19	30.89	0.35	0.25	0.20
			0	70	4.95	30.04	0.34	0.23	0.20
	Met4	Measured	300	3	0.72	13.99	0.43	0.20	18.06
			250	19	0.81	23.31	0.48	1.21	14.53
			200	53	0.83	31.54	0.50	1.36	12.20
			150	81	0.83	32.37	0.48	1.26	10.81
			100	108	1.18	33.18	0.44	1.05	8.95
			62	139	1.26	30.51	0.43	0.86	7.38
			0	150	1.24	29.28	0.42	0.80	6.90
		Indicated	300	113	0.78	44.72	0.35	2.09	18.75
			250	228	0.81	38.38	0.38	1.49	16.51
			200	373	0.83	34.69	0.38	1.18	14.50
			150	557	0.88	31.70	0.40	0.97	12.38
			100	1,006	1.25	29.52	0.50	0.65	8.40
			62	1,761	1.31	24.57	0.53	0.42	5.55
			0	2,094	1.25	22.67	0.49	0.37	4.81

TABLE 14.16
UNDERGROUND MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
		Inferred	250	0.1	0.67	15.01	0.34	0.22	14.45
			200	4	0.67	15.48	0.31	0.36	11.67
			150	12	0.74	23.62	0.30	0.68	9.68
			100	89	1.31	24.54	0.90	0.24	3.57
			62	273	1.45	20.61	0.73	0.15	2.21
			0	301	1.40	20.42	0.69	0.15	2.10
	Met5	Measured	300	8	2.04	94.42	0.06	3.46	12.73
			250	12	2.83	86.85	0.06	2.91	10.22
			200	18	3.28	73.89	0.06	2.23	7.55
			150	25	3.11	63.47	0.06	1.81	6.30
			100	43	2.46	43.90	0.06	1.36	4.88
			63.1	94	1.68	27.45	0.04	0.96	3.25
			0	120	1.44	23.29	0.04	0.84	2.77
		Indicated	300	21	3.19	134.60	0.09	3.29	8.42
			250	40	3.43	120.29	0.07	2.53	6.45
			200	61	3.28	104.46	0.07	2.05	5.51
			150	139	2.39	68.78	0.05	1.23	5.11
			100	268	1.95	50.54	0.05	0.90	4.10
			63.1	410	1.67	39.57	0.04	0.76	3.25
			0	493	1.48	35.05	0.04	0.71	2.86
		Inferred	300	26	7.10	273.31	0.08	2.29	4.73
			250	33	5.82	233.37	0.07	2.06	5.79
			200	38	5.45	223.43	0.07	1.96	5.24
			150	45	4.91	206.65	0.06	1.78	4.64
			100	76	3.50	144.73	0.05	1.30	3.52
			63.1	90	3.18	126.88	0.05	1.15	3.14
			0	92	3.11	124.23	0.05	1.13	3.09
Leachable	Met6	Measured	300	38	12.31	76.43	0.05	0.03	0.06

TABLE 14.16
UNDERGROUND MINERAL RESOURCE ESTIMATE SENSITIVITY

Metallurgical Type		Classification	NSR Cut-off ($/t)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
			250	43	11.63	74.07	0.05	0.04	0.07
			200	48	10.96	72.07	0.05	0.06	0.09
			150	53	10.16	70.02	0.06	0.07	0.11
			100	61	9.25	66.46	0.07	0.09	0.15
			71.4	72	8.10	59.46	0.08	0.10	0.17
			0	88	6.79	50.64	0.08	0.09	0.17
		Indicated	300	20	10.29	94.29	0.13	0.46	0.07
			250	33	8.67	75.02	0.13	0.36	0.12
			200	46	7.60	67.65	0.12	0.36	0.26
			150	69	6.31	64.61	0.13	0.38	0.50
			100	112	4.84	55.41	0.15	0.34	0.46
			71.4	145	4.11	51.89	0.15	0.31	0.42
			0	190	3.35	46.56	0.14	0.27	0.37
		Inferred	300	67	16.49	154.36	0.12	0.94	0.28
			250	80	14.70	145.69	0.12	0.92	0.28
			200	90	13.59	134.86	0.11	0.90	0.32
			150	106	12.10	120.85	0.11	0.84	0.39
			100	128	10.49	104.62	0.10	0.77	0.44
			71.4	153	9.08	94.91	0.09	0.73	0.52
			0	193	7.39	78.94	0.09	0.66	0.69

14.14 CONFIRMATION OF ESTIMATE

The block models were validated using a number of industry standard methods including visual and statistical methods. Visual examination of composites and block grades on successive plans and cross-sections were performed on-screen in order to confirm that the block models correctly reflect the distribution of local composite grade values. The review of estimation parameters included:

- Number of composites used for estimation;
- Number of drill holes used for estimation;
- Mean distance to sample used;
- Number of interpolation passes used to estimate grade; and
- Mean value of the composites used.

Comparisons of mean grade values of composites with the block models by mineralization type at zero grade are presented in Table 14.17.

The comparisons show that the average grades of the block models to be slightly different to the average grades of capped composites used for the grade estimations. These are most likely due to the smoothing by the grade interpolation process. The block model values will be more representative than the capped composites due to 3-D spatial distribution characteristics of the block models.

TABLE 14.17
AVERAGE GRADE COMPARISON OF COMPOSITE GRADE VALUES WITH BLOCK MODELS

Model	Mineralization	Data Type	Au (g/t)	Ag (g/t)	Zn (%)	Cu (%)	Pb (%)
Open Pit	Massive Sulphide Zones	Composites	2.03	25.37	4.41	0.50	0.19
		Capped Composites	1.97	23.34	4.39	0.50	0.19
		Block Model IDW*	1.84	24.61	4.11	0.47	0.28
		Block Model NN**	1.86		4.09		
	Gold Zones	Composites	3.72	47.79	0.33	0.09	0.23
		Capped Composites	3.46	37.45	0.28	0.07	0.22
		Block Model IDW	2.90	39.21	0.36	0.06	0.29
		Block Model NN	2.93		0.35		
	Stringer Zones	Composites	1.16	15.18	0.92	0.12	0.20
		Capped Composites	1.12	14.21	0.90	0.12	0.19
		Block Model IDW	1.13	16.88	0.91	0.13	0.25
		Block Model NN	1.13		0.90		
UG	Massive Sulphide Zones	Composites	2.14	27.54	4.61	0.52	0.21
		Capped Composites	2.07	24.29	4.58	0.51	0.20
		Block Model IDW	1.92	28.16	4.55	0.50	0.36
		Block Model NN	1.91		4.52		
	Gold Zones	Composites	4.79	57.08	0.32	0.08	0.27
		Capped Composites	4.46	47.80	0.26	0.06	0.25
		Block Model IDW	4.09	52.94	0.32	0.06	0.29
		Block Model NN	4.13		0.32		
	Stringer Zones	Composites	1.49	19.24	1.12	0.14	0.24
		Capped Composites	1.43	18.06	1.09	0.13	0.23
		Block Model IDW	1.53	22.16	1.14	0.16	0.30
		Block Model NN	1.52		1.13		

***** *block model grades were interpolated using Inverse Distance Cubed for Au and Ag, while Inverse Distance Squared was used for Zn, Cu and Pb.*
****** *block model grades were interpolated using the Nearest Neighbour method.*

Comparisons of the grade-tonnage curve of the Au grade model interpolated with Inverse Distance Cubed (“ID^3”) and Nearest Neighbour (“NN”) on a global resource basis for all zones are presented in Figures 14.5 and 14.6.

Figures 14.7 and 14.8 present the comparisons of the grade-tonnage curve on a global resource basis for the Zn grade block model interpolated with ID^2 and NN for all zones.

FIGURE 14.5 AU GRADE-TONNAGE CURVE FOR ID^3 AND NN INTERPOLATION FOR ALL ZONES OF OPEN PIT MODEL

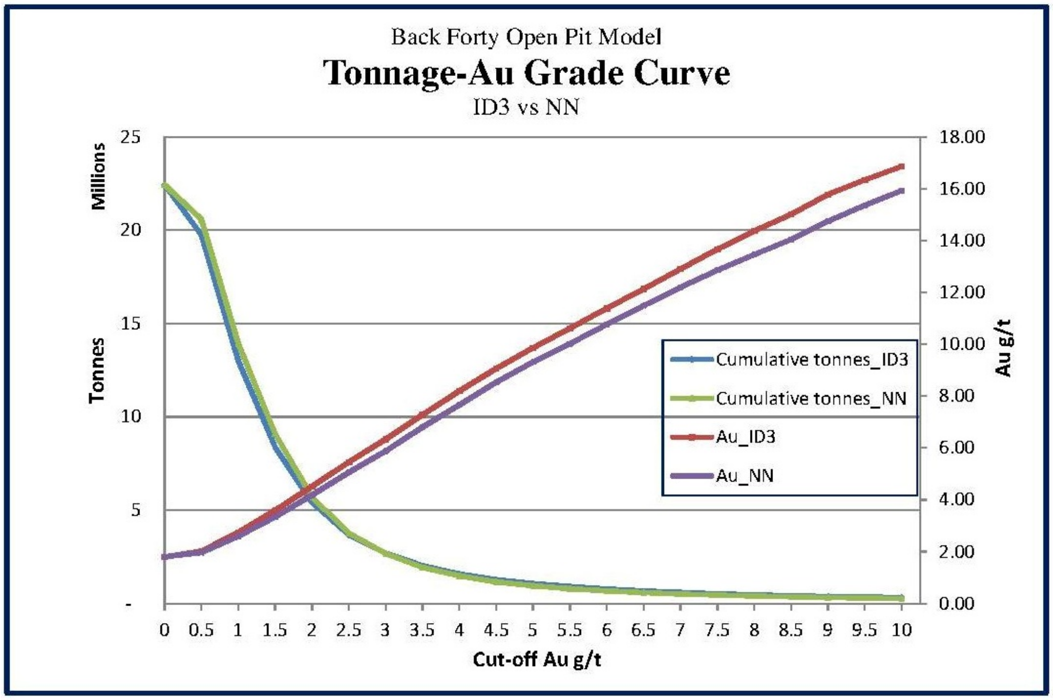


FIGURE 14.6 AU GRADE-TONNAGE CURVE FOR ID^3 AND NN INTERPOLATION FOR ALL ZONES OF UNDERGROUND MODEL

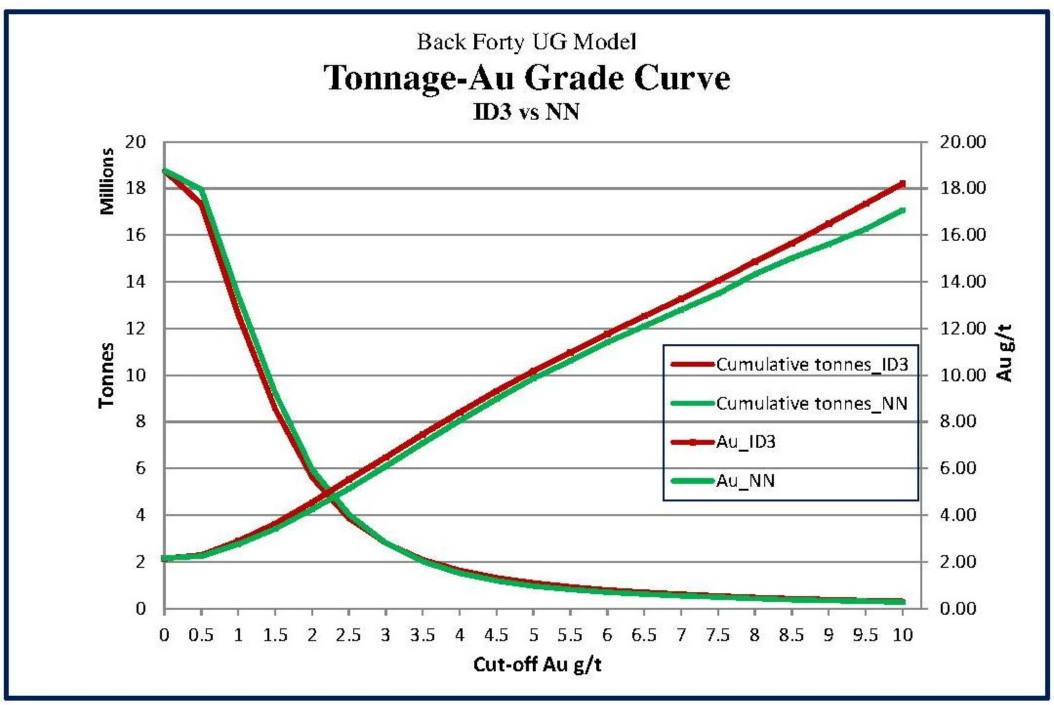


FIGURE 14.7 Zn GRADE-TONNAGE CURVE FOR ID^2 AND NN INTERPOLATION FOR ALL ZONES OF OPEN PIT MODEL

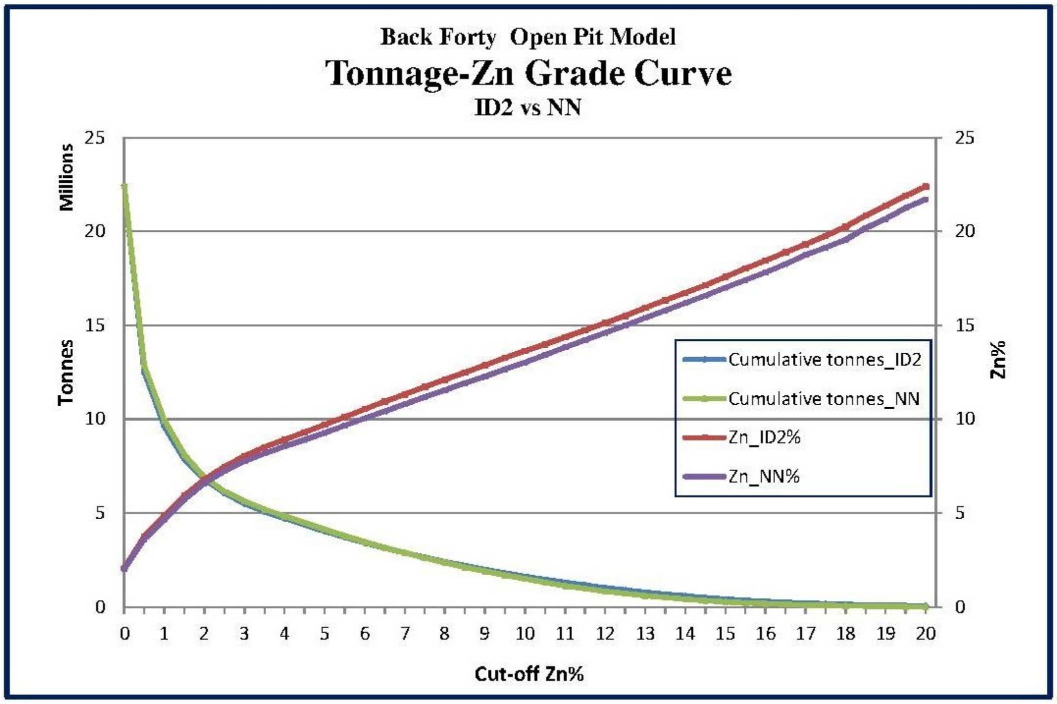
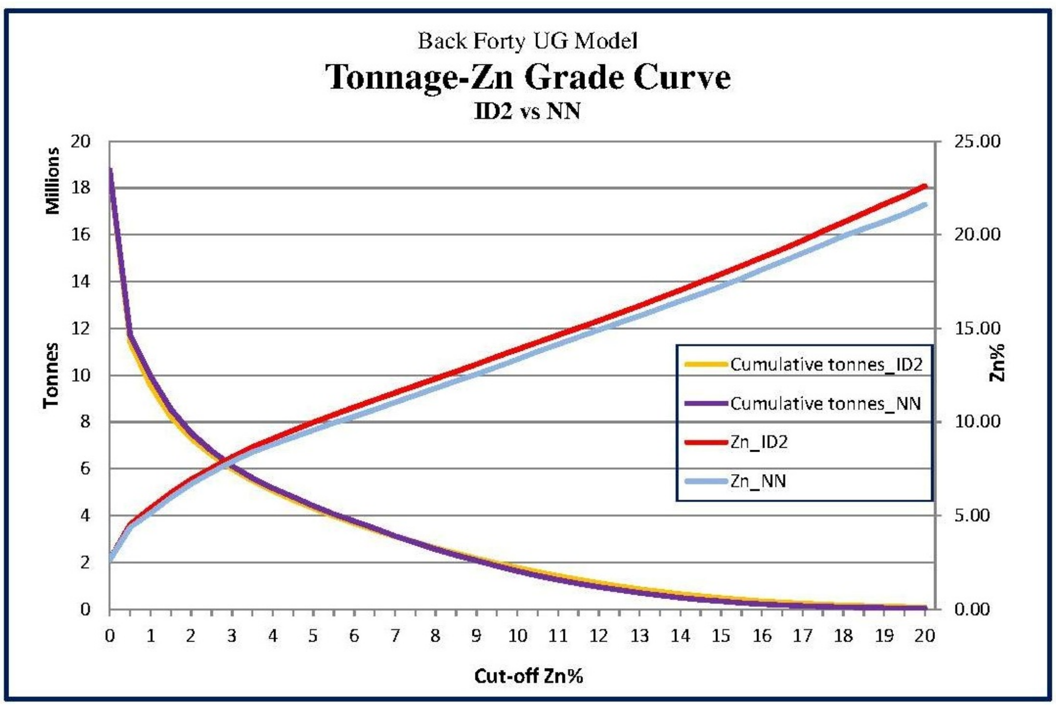


FIGURE 14.8 Zn GRADE-TONNAGE CURVE FOR ID^2 AND NN INTERPOLATION FOR ALL ZONES OF UNDERGROUND MODEL



Au local trends of all zones were evaluated by comparing the ID^3 and NN estimate against Au Composites and Capped Composites. As shown in Figures 14.9 to 14.14, Au grade interpolations with ID^3 and NN agreed well.

FIGURE 14.9 AU GRADE SWATH EASTING PLOT OF OPEN PIT MODEL

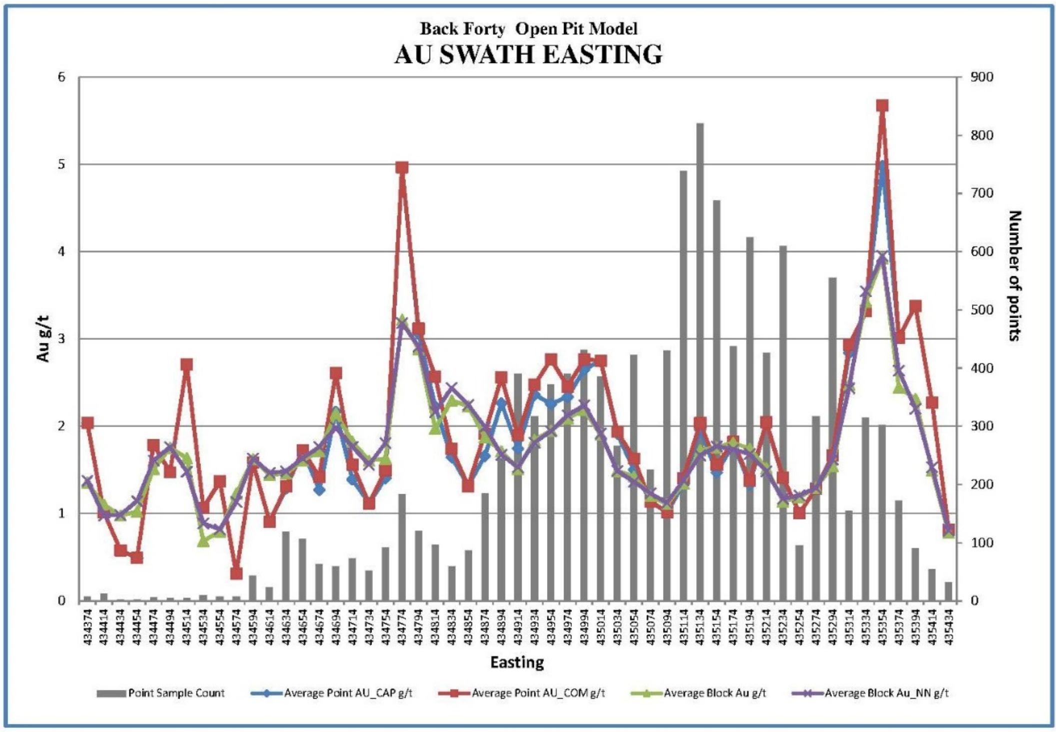


FIGURE 14.10 AU GRADE SWATH NORTHING PLOT OF OPEN PIT MODEL

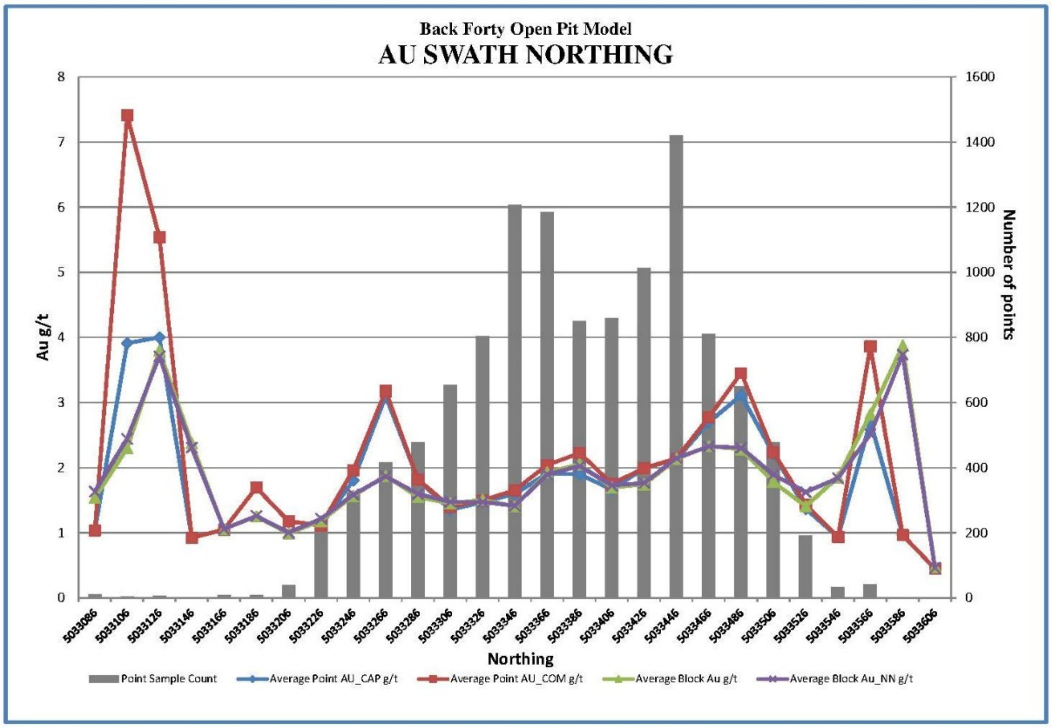


FIGURE 14.11 AU GRADE SWATH ELEVATION PLOT OF OPEN PIT MODEL

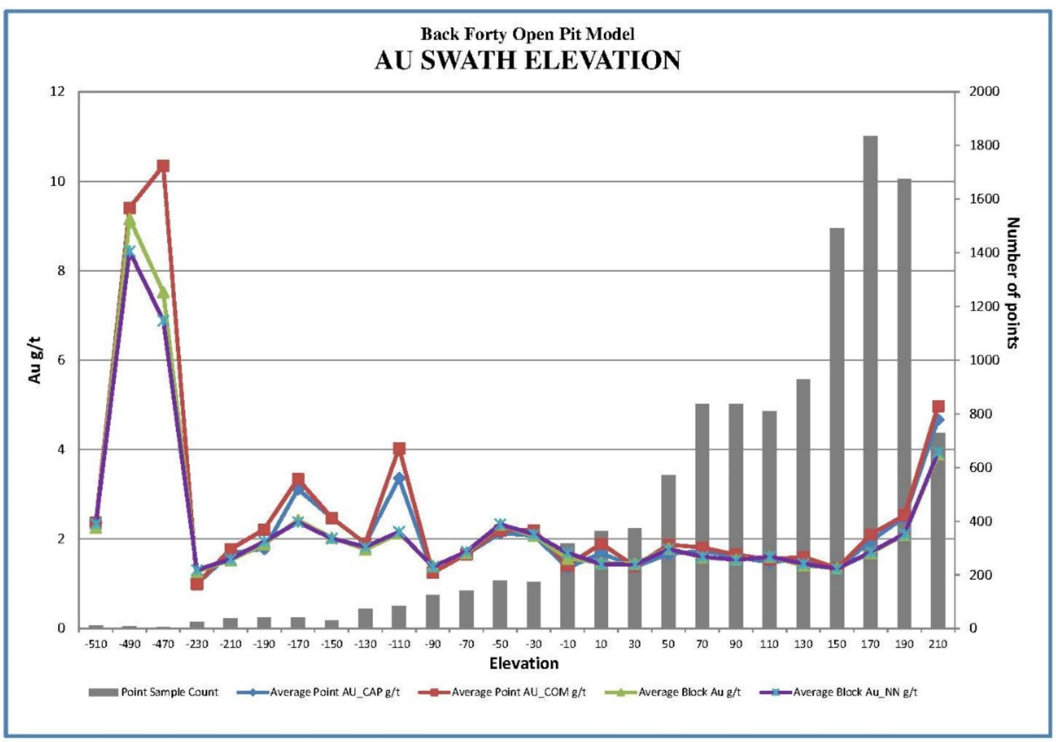


FIGURE 14.12 AU GRADE SWATH EASTING PLOT OF UNDERGROUND MODEL

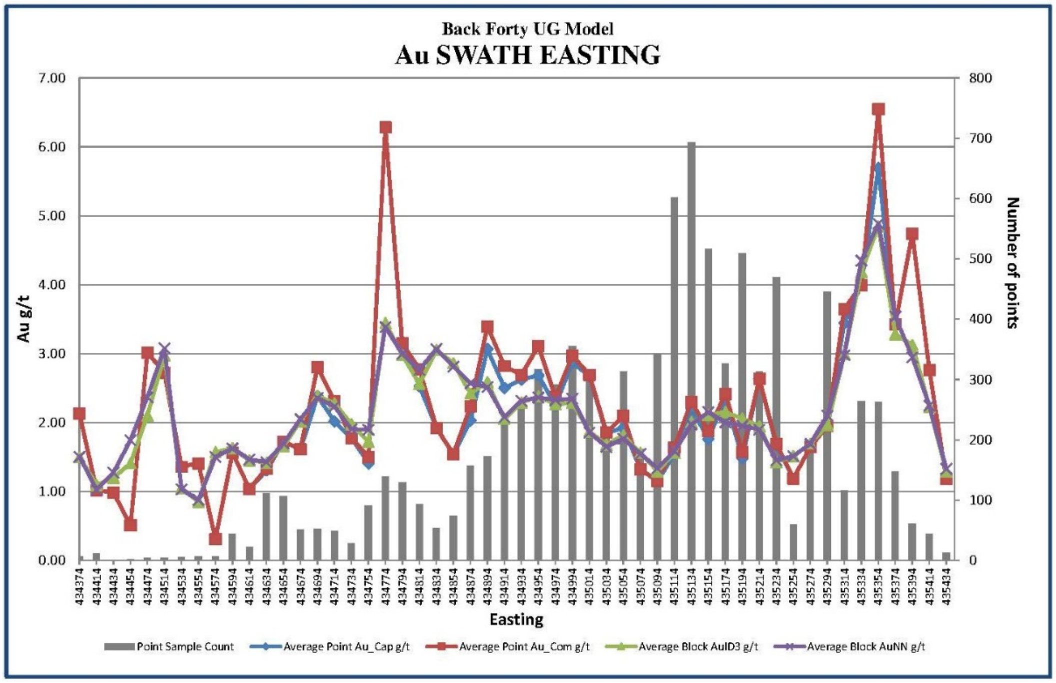


FIGURE 14.13 AU GRADE SWATH NORTHING PLOT OF UNDERGROUND MODEL

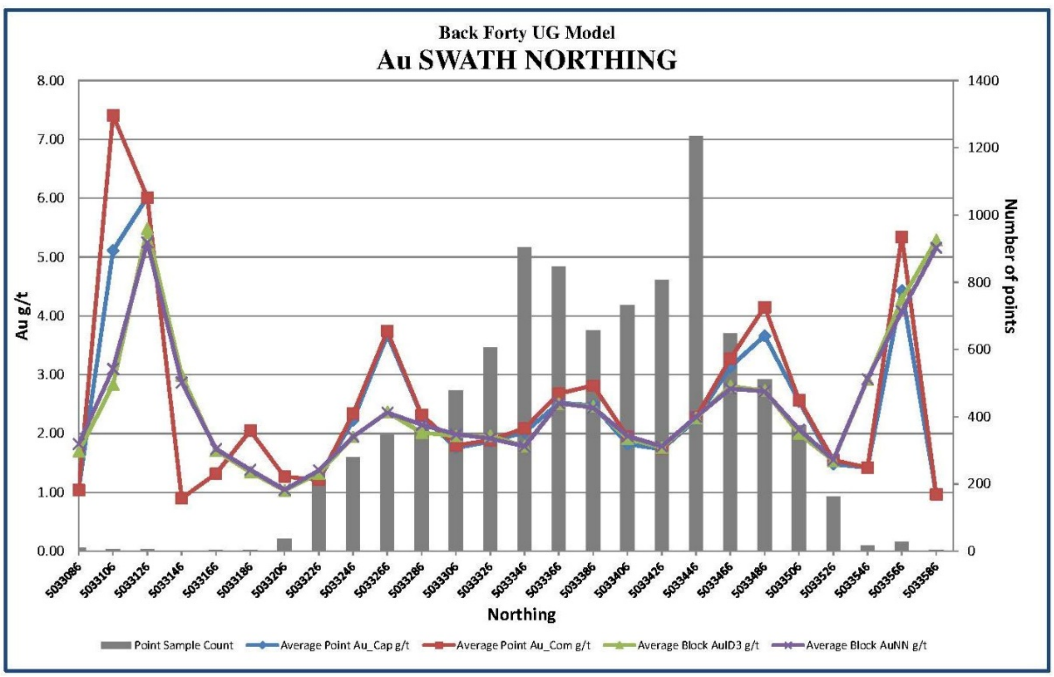
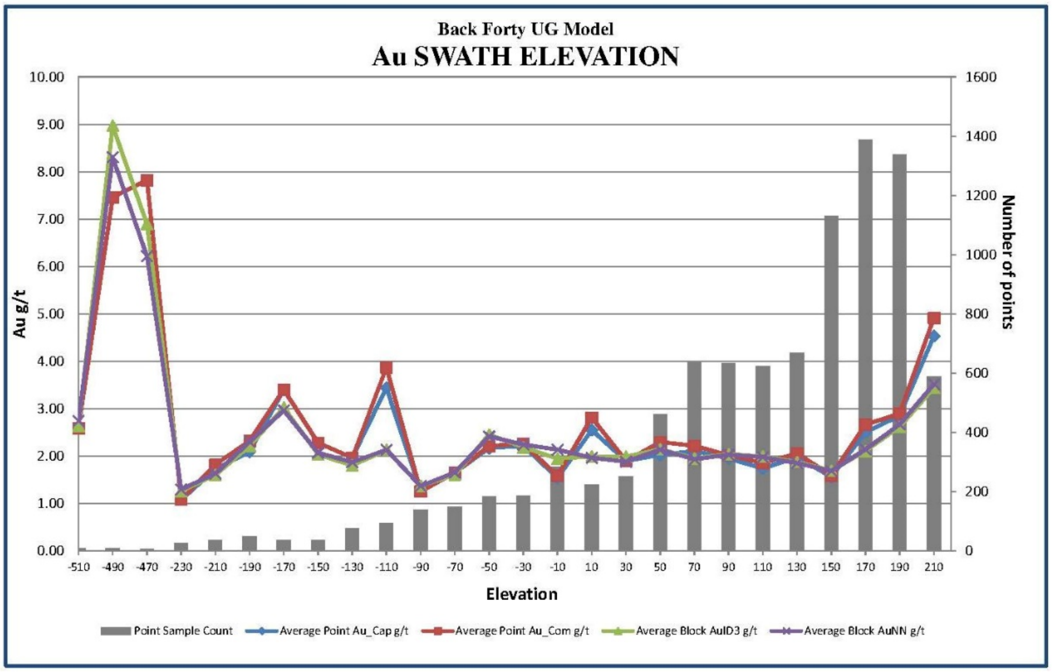


FIGURE 14.14 AU GRADE SWATH ELEVATION PLOT OF UNDERGROUND MODEL



Zn local trends of all zones were evaluated by comparing the ID^2 and NN estimate against Zn Composites and Capped Composites. As shown in Figures 14.15 to 14.20, Zn grade interpolations with ID^2 and NN agreed well.

FIGURE 14.15 ZN GRADE SWATH EASTING PLOT OF OPEN PIT MODEL

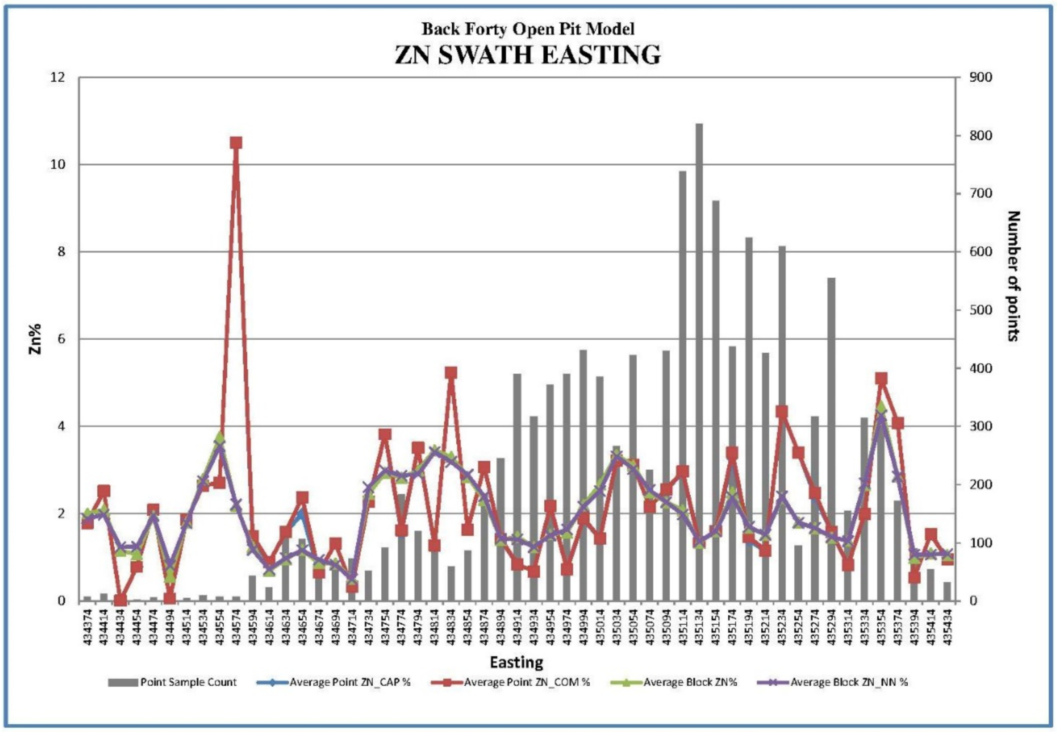


FIGURE 14.16 ZN GRADE SWATH NORTHING PLOT OF OPEN PIT MODEL

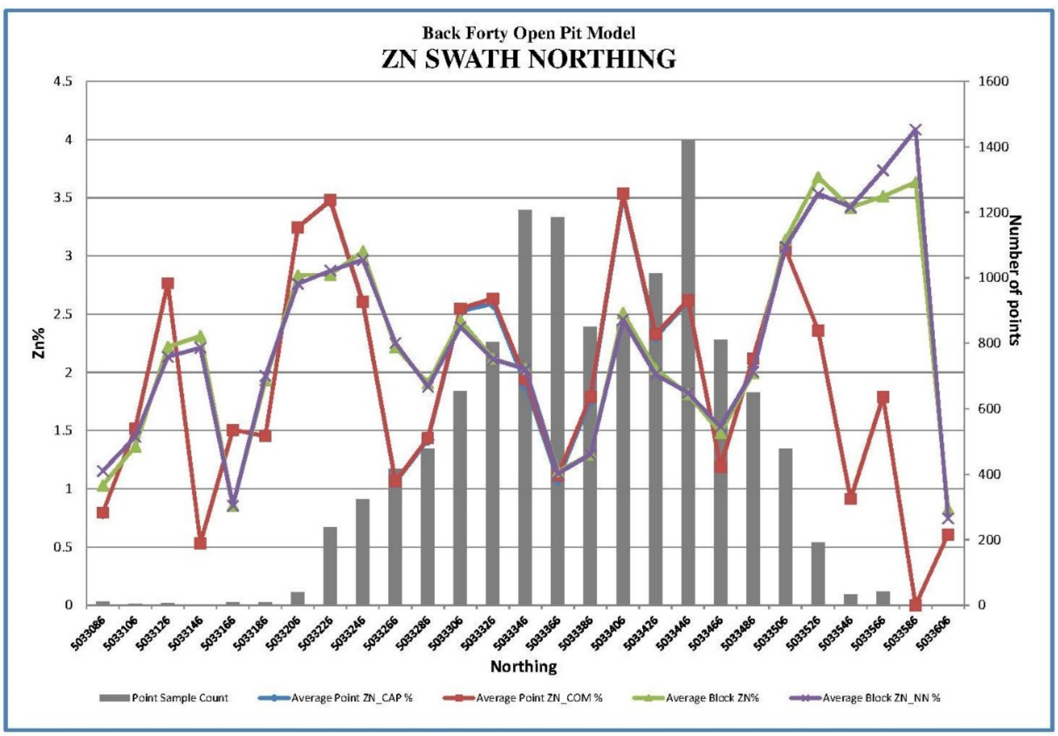


FIGURE 14.17 ZN GRADE SWATH ELEVATION PLOT OF OPEN PIT MODEL

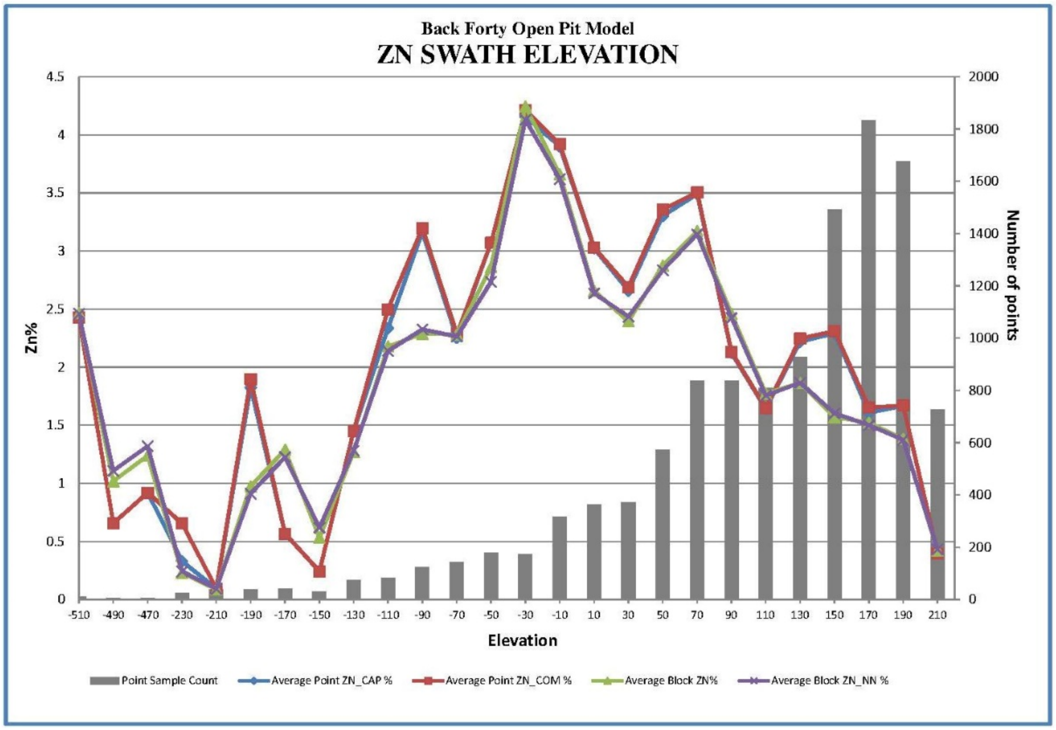


FIGURE 14.18 ZN GRADE SWATH EASTING PLOT OF UNDERGROUND MODEL

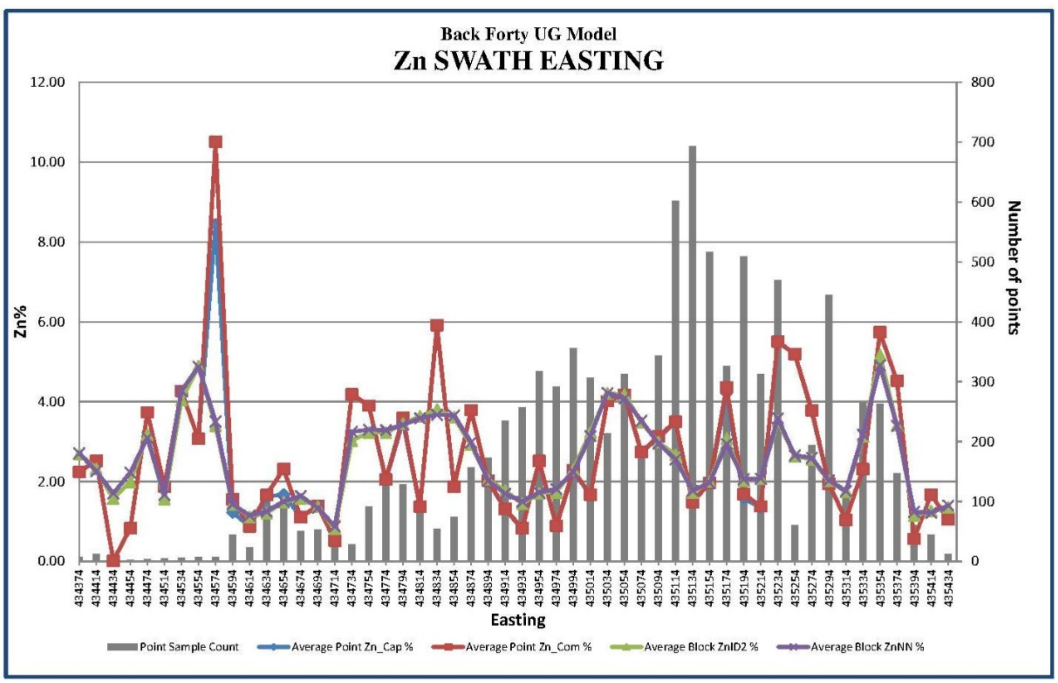


FIGURE 14.19 ZN GRADE SWATH NORTHING PLOT OF UNDERGROUND MODEL

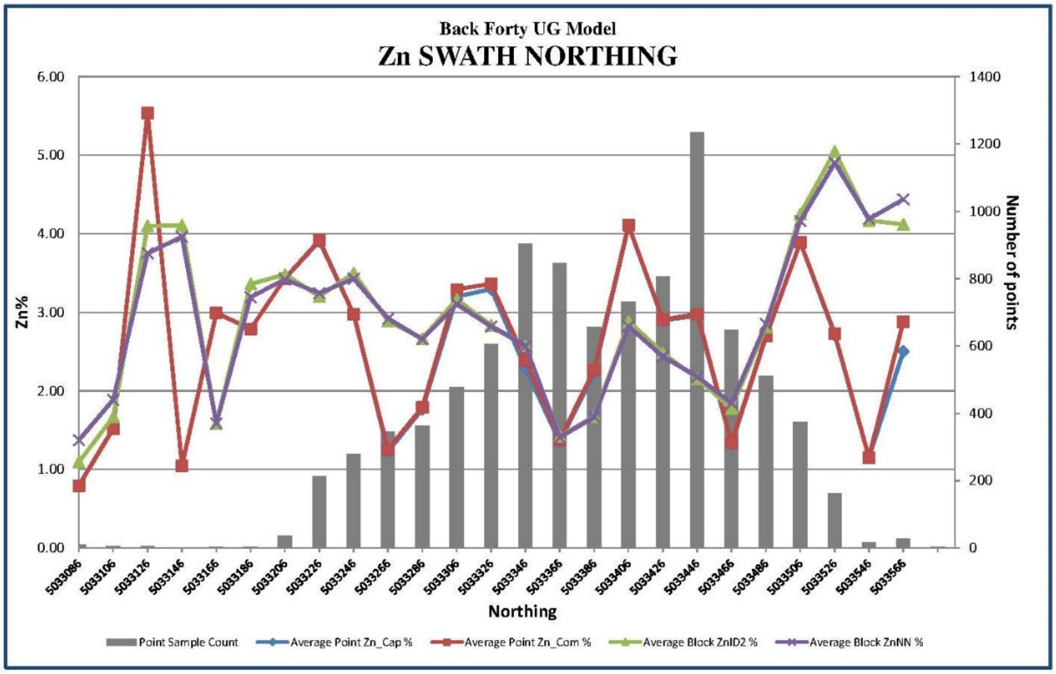
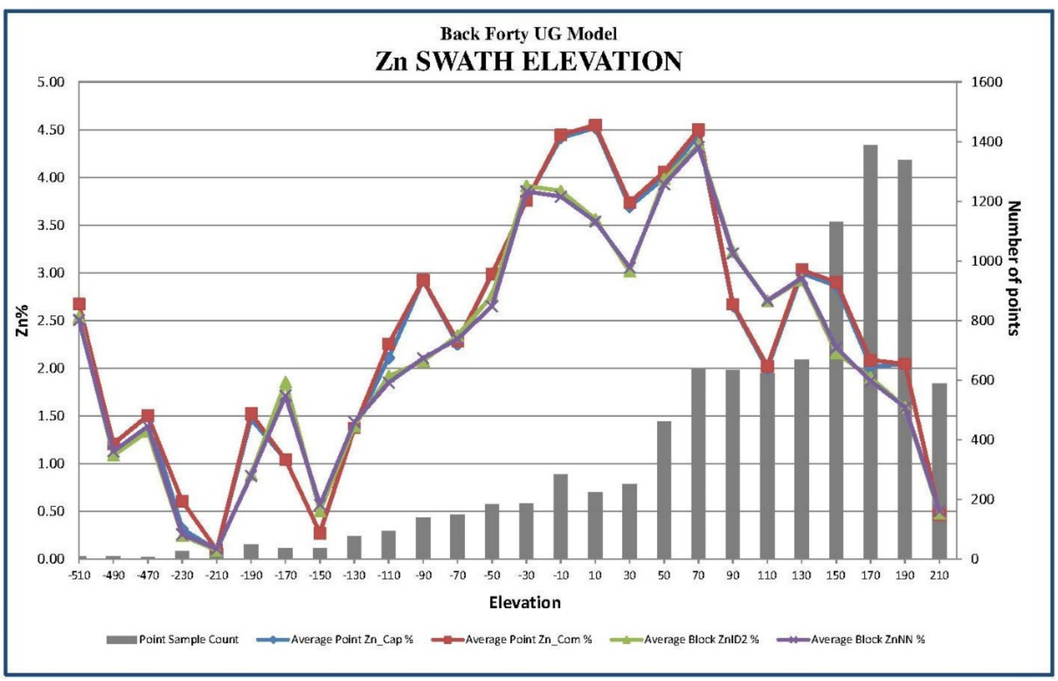


FIGURE 14.20 ZN GRADE SWATH ELEVATION PLOT OF UNDERGROUND MODEL



15.0 MINERAL RESERVE ESTIMATE

There are no stated Mineral Reserves for the Back Forty Project.

According to NI 43-101 guidelines, a Preliminary Economic Assessment is considered preliminary in nature and includes the use of Inferred Mineral Resources which are considered too speculative geologically to apply economic considerations that would enable them to be classified as Mineral Reserves.

16.0 MINING METHODS

The value proposition of the Project is based on mining the highest value material as soon as possible and treating this material through the process plants to maximize cash flow. This strategy is achieved by mining the mineralized material and either feeding the material directly to the process plant or stockpiling the material onsite for processing later per a feed schedule based on optimal economics for the operation. Open pit mining will take place from Year 1 to Year 5. Underground development will be initiated in Year 5 and underground production mining will continue to Year 11 subject to additional environmental permitting.

A series of grade blending stockpiles, by material type, will serve to prioritize the processing of higher-grade material and also manage fluctuations in process plant feed delivery from the two mining operations.

The Back Forty Project area consists of very subdued terrain and topography. The area, topography and climate are amenable to the conventional open pit mining operations proposed for the Project. The open pit mining operation will encompass a single open pit that will be mined with conventional mining equipment in three pushback phases. The underground mine will be developed beneath the open pit with a single decline access point located part way down the open pit ramp.

16.1 OPEN PIT MINING

16.1.1 Open Pit Geotechnical Studies

Golder Associates Ltd. was retained by Aquila in February 2016 to complete a pit slope stability design study. The slope design study included drilling five geotechnical drill holes in bedrock and five geotechnical drill holes in the overburden for the cut-off wall design.

The results of the study are documented in Golder report “1546541 Back Forty Project Feasibility Pit Slope Design”, dated November 11, 2016.

16.1.2 Overburden Slope

Overburden seepage and slope stability analyses were carried out on the critical section of the open pit slope. This location coincided with the maximum thickness of the overburden, together with the minimum distance of the open pit shell to the Menominee River.

Based on the configuration used for the overburden seepage and slope stability assessment, a 21° overall slope (i.e. batter face angle (“BFA”) 27°) is recommended with a 4 m bench for every 10 m of vertical height.

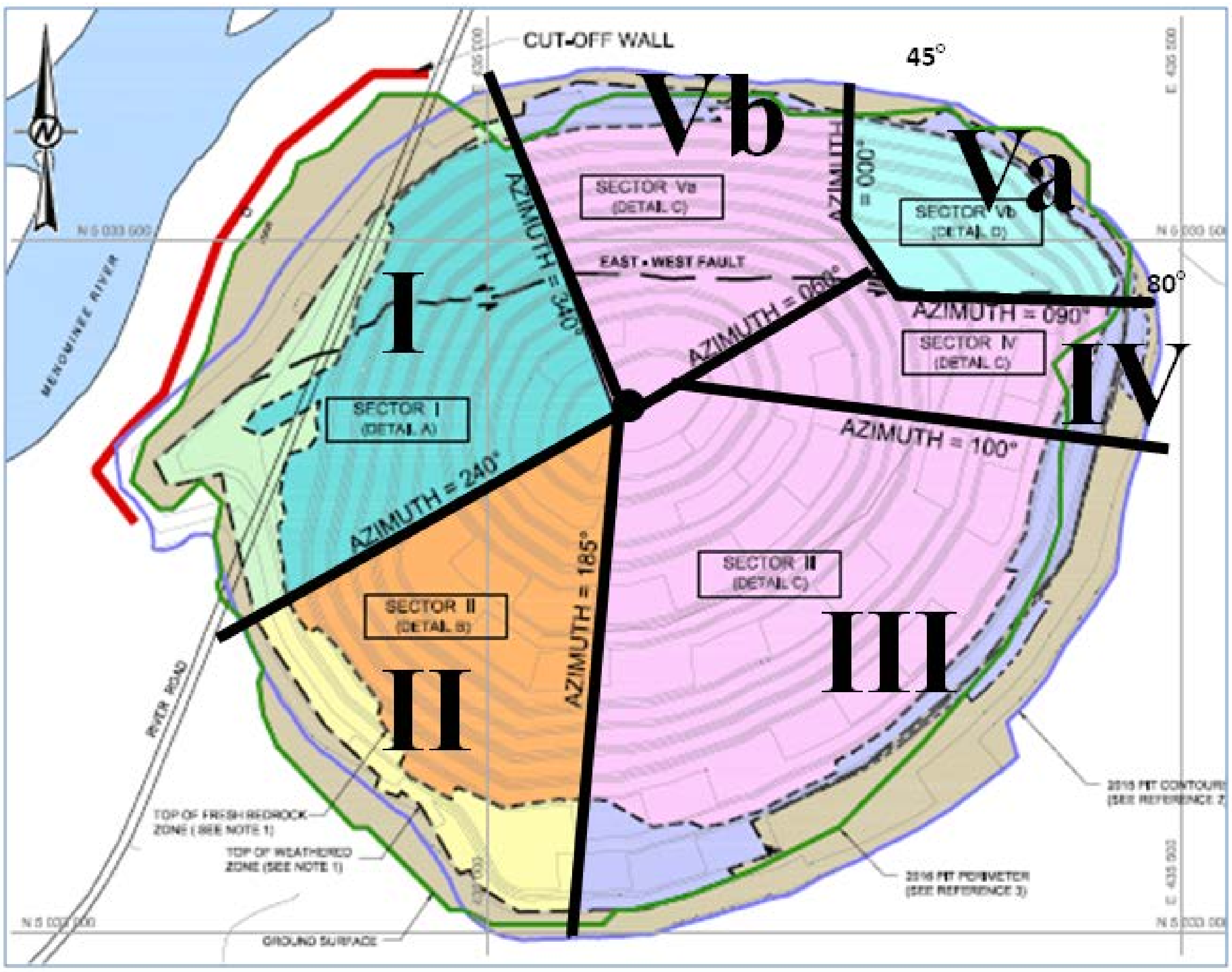
16.1.3 Rock Slopes

Recommended slope configurations for the Weathered and Fresh Bedrock Zones, obtained from the kinematic analyses are summarized in Table 16.1. The pit wall sectors are shown in Figure 16.1.

The recommended slope angles are considered achievable (or optimal) based on the available data. Implementation of proper blasting control techniques on all benches will be critical to the success in achieving the recommended design parameters.

The pit slope stability assessments originally considered 10 m and 20 m high benches. However, to provide flexibility for mine planning, slope criteria were also provided for 7.5 m, 15 m, and 22.5 m high benches.

FIGURE 16.1 OPEN PIT SLOPE DESIGN SECTORS



Source: *Golder Associates (2016)*

In the overburden, a slope of 2:1 (27°) with a 4 m berm every 10 m of height is the requirement. Since the overburden thickness is variable around the pit and relatively thin, an average 24° slope was used for the pit design.

A Selective Mining Unit ("SMU") block height was determined to be 5.0 m in the Main and Tuff Zones. In the Pinwheel and Oxide Zones it was determined to be 2.5 m in order to maintain selective process plant feed mining capability. For mining efficiency purposes, it is assumed that in waste rock areas a 5.0 m bench height will be mined. However, there will be the opportunity to mine waste rock at a 10.0 m bench height.

A multiple (4) bench configuration has been designed, resulting in a 20 m vertical distance between catch berms. For pit wall sectors that do not incorporate a haul ramp, a 15 m wide geotechnical berm is included if the wall height exceeds 160 m.

TABLE 16.1
PIT SLOPE CRITERIA

Design Sector	Azimuth (°)	Dip Direction	Bench Face Angle (°)	Bench Height (m)	Berm Width (m)	Inter-ramp Angle (°)
Weathered Zone						
I to II	185-340	005-160	65	10	6.5	42
			65	20	11.0	44.5
III to V	340-185	160-005	65	10	6.5	44.5
			65	20	13.0	44.5
Fresh Bedrock						
I	240-340	060-160	70	20	9.0	51
II	185-240	005-060	65	20	8.5	48
III	100-185	280-005	70	20	8.5	52
IV	60-100	240-280	70	20	8.5	52
Va	340-60	160-240	70	20	8.5	52
Vb	000-90	180-270	70	20	11.0	48

16.1.4 Groundwater Studies

Groundwater modelling was undertaken as part of the environmental impact assessment ("EIA") process and a modelling report was provided in Foth Infrastructure and Environment LLC, USA, "Groundwater Modelling Back Forty Project, Project I.D.: 14A021, October 2015".

The three main hydrogeological units included in the model are unconsolidated Quaternary glacial till and outwash deposits, the underlying Cambrian sandstone, and Precambrian bedrock.

A three-dimensional groundwater flow model was constructed to analyze the groundwater flow system, estimate groundwater inflow rates to the open pit and underground workings, and estimate potential impacts associated with the Project.

To evaluate the impact of open pit excavation and subsequent groundwater inflow to and dewatering from the pit, and to provide an estimate of open pit inflow rates, the groundwater model was adapted to simulate pit excavation and dewatering. Drain boundary conditions were assigned to each layer of the model with drain elevations assigned to represent approximate pit floor elevations for mine production Years 1 through 5 (i.e. the life of mine ("LOM") for the modelled open pit).

16.1.5 Open Pit Optimization

The Back Forty open pit design was developed in a two-step process.

1. A pit optimization analysis was completed, and an optimized pit shell was selected to be used as the basis for the pit design.

2. An operational pit design was prepared that incorporates catch benches, detailed pit wall slopes based on geotechnical assessment, and truck haulage ramps.

16.1.5.1 Pit Optimization Parameters

The first step in the pit design process is pit optimization. A series of pit optimization analyses were completed to help select the optimal pit as part of the 2018 Feasibility Study, and these remained the same for this Technical Report. Table 16.2 summarizes the key optimization parameters used. As noted previously, cost estimates for various process plant feed types shown in this table were preliminary in nature. The processing rate assumption used in the Feasibility Study was higher than used in this Technical Report.

The individual block model revenue values are provided by the NSR parameters described in Section 14.12.

TABLE 16.2
KEY PIT OPTIMIZATION PARAMETERS

Type	Units	Unit Costs							
OP Mining Overburden	$/t	$2.19							
OP Mining Waste Rock	$/t	$2.92							
OP Mining – increment ($0.02/5 m vertical)	$/t/m	$0.004							
OP Process Plant Feed	$/t mill feed	$2.92							
Process Plant Feed Types	**Units**	**1**	**2**	**8**	**3**	**7**	**4**	**5**	**6**
		Main Zone Mass Sulp	**Pinwheel MSCR**	**Pinwheel Zn-Cu**	**Pinwheel SMSS**	**Pinwheel Gossan Float**	**Pinwheel Ext**	**Tuff Zone**	**Oxide (All Types)**
Processing - Flotation	$/t mill feed	$18.13	$19.54	$19.54	$19.54	$16.08	$19.54	$21.77	n/a
Processing - Leaching	$/t mill feed	n/a	n/a	n/a	n/a	n/a	n/a	n/a	$26.90
+ G&A Cost ($8.15M/yr)	$/t mill feed	$4.63	$4.63	$4.63	$4.63	$4.63	$4.63	$4.63	$4.63
Processing Rates									
Nominal Processing rate - Flotation	tpd	2,800	2,800	2,800	2,800	2,800	2,800	2,800	
Nominal Processing rate - Flotation	Mtpa	1.02	1.02	1.02	1.02	1.02	1.02	1.02	
Processing rate - Leaching	Mtpa								0.13
Discount Rate (for optimizer)	%	0% and 8%							

Note: *OP = Open Pit, Mass Sulp = massive sulphide, MSCR = massive sulphide copper rich, SMSS = semi-massive sulphide and stringers domain..*

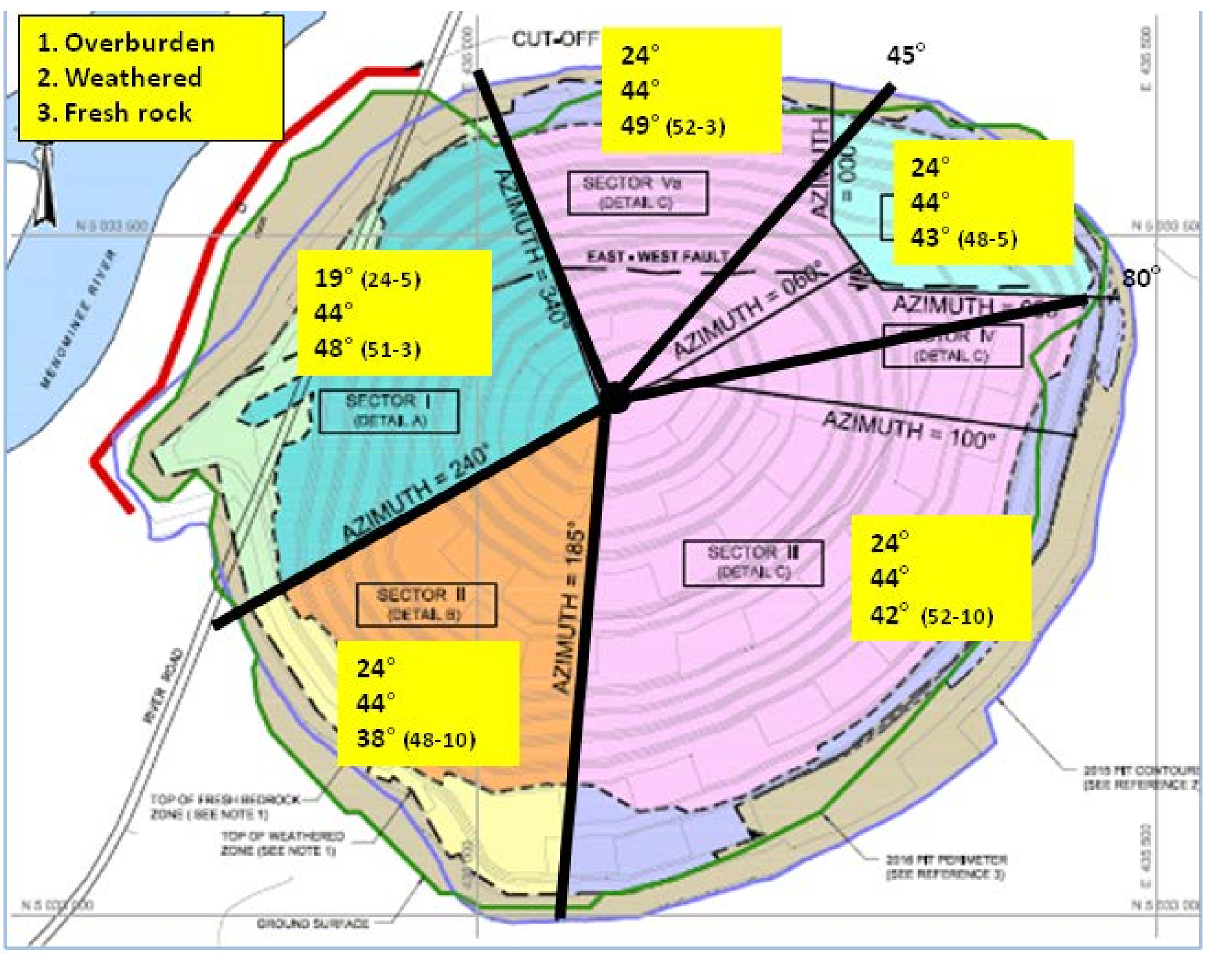
The Back Forty Deposit extends out beneath the Menominee River. It is necessary to preclude mining from encroaching upon the river, therefore in the optimization analysis, a pit wall constraint was applied along the northwest wall of the pit. The pit crest was constrained to remain at least 46 m (150 ft) from the 100-year flood water level line. Along the other sides of the pit, no boundary constraint was applied, and the pit shell was free to expand as needed to recover economic material.

Figure 16.2 describes the pit slope criteria used in optimization. In some pit sectors the slopes used in optimization were modified (slightly flattened) from the initial design inter-ramp slopes to account for the eventual placement of truck haulage ramps in the final pit design.

Along the northwest river wall, the optimization overburden slope was flattened to 19° to account for the cut-off wall design that would be used in this area. The other sectors of the pit used a 24° overburden slope.

Beneath the overburden in the upper weathered rock zone, the pit slopes were set at 44° around the entire pit perimeter. In the underlying fresh rock, the optimization slope angles ranged from 38° to 49° depending on whether the pit wall sector would eventually incorporate a truck haulage ramp.

FIGURE 16.2 OPTIMIZATION SLOPES (ADJUSTED FOR RAMPS)



16.1.5.2 Pit Optimization Results

The pit optimization analysis examined a series of revenue factors, ranging from 3% to 100% in order to assess the sensitivity of the pit size to metal prices. A revenue factor applies the same percent change in metal price to all five payable metals (Au, Ag, Zn, Pb, Cu), and hence to the NSR value, simultaneously.

The results of the pit optimization are presented in a series of graphs that examine the pit size at different revenue factors ("RF"). Low revenue factors would represent small pits that would be economic at low metal prices, consisting of either high grades, low strip ratios, or both. Higher revenue factor pits will be larger in size since higher metal prices can make marginally-economic material economic, thereby expanding the size of the pit.

Figure 16.3 presents a graph of pit net present value ("NPV") versus RF. The graph shows that beyond a RF of 52%, the NPV does not change dramatically and reaches a peak at a RF of 100%. The NPV is used to compare options and is based on revenue and operating costs only. It does not incorporate capital costs, taxes, or a closure cost and therefore does not represent the real Project NPV (which is described in Section 22).

Figure 16.4 presents a graph of pit tonnage versus RF. The graph shows that beyond a RF of 54%, the process plant feed tonnage does not increase dramatically. There is a significant increase in waste tonnage beyond a RF of 52%.

The boundary limit placed by the Menominee River constrains the optimized pit in the northwest corner.

FIGURE 16.3 PIT NPV VERSUS REVENUE FACTOR

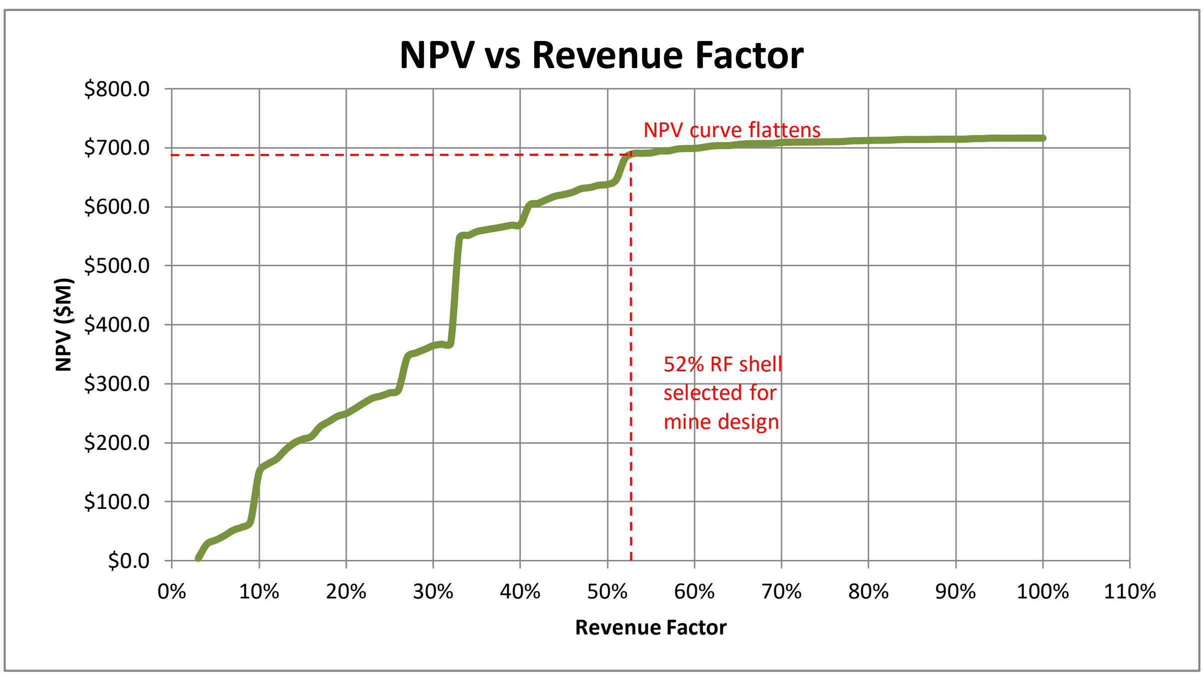
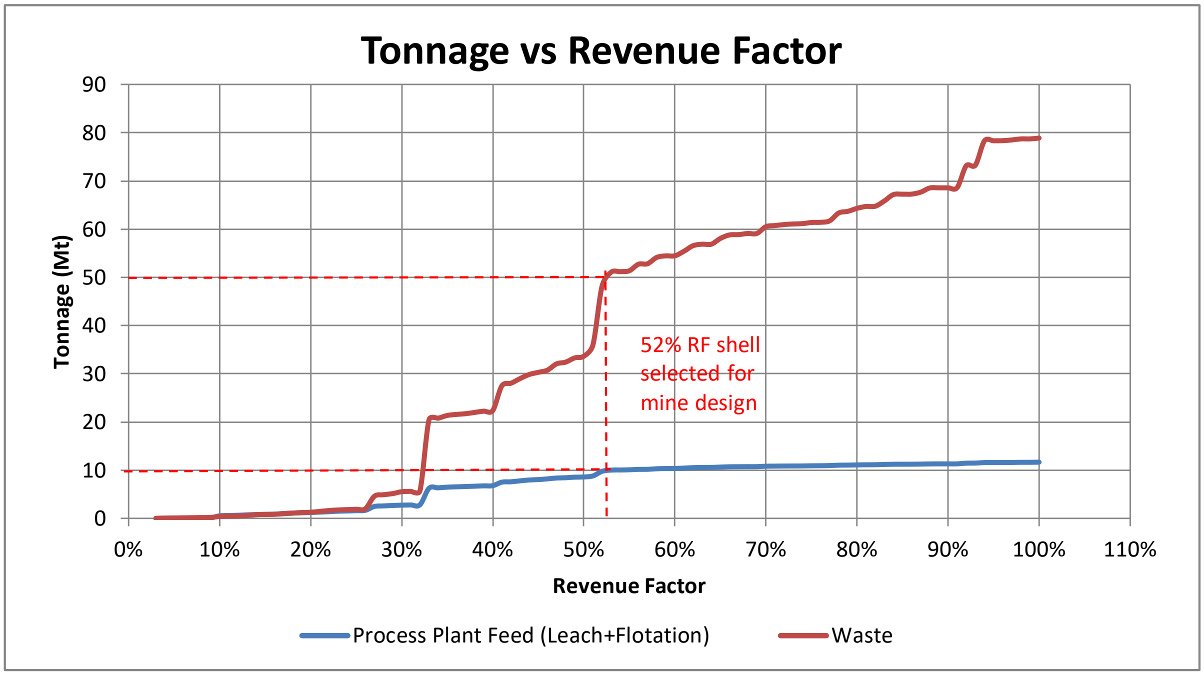


FIGURE 16.4 PIT TONNAGE VERSUS REVENUE FACTOR



16.1.6 Pit Design

The open pit design is based on the 2018 Feasibility Study design. Minor modifications were made to standardize on 5 m high benches with a quadruple (4) bench configuration, resulting in a 20 m vertical distance between catch berms. The operational pit design incorporates catch benches, truck haul ramps, and inter-ramp angles based on geotechnical analyses.

The haul ramp is based on a width of 25 m, providing a 21 m running width, a 1 m ditch, and a 3 m wide safety berm on the outside of the ramp. The ramp gradient was maintained at 10% (H:V) or less. Near the base of the pit, the truck haul ramp has been narrowed to single lane (16 m) to maximize process plant feed recovery at depth.

The resulting final pit design is shown in Figure 16.5. Since the mineralized zones dip into the north wall of the pit, in order to maximize the process plant feed recovery along the north wall the truck haul ramps were positioned along the south and east walls. This resulted in two switchbacks along the travel route.

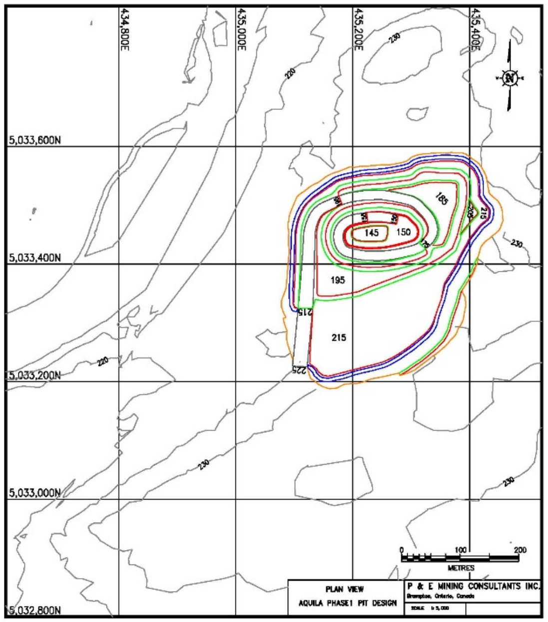
16.1.7 Pit Phases

For production scheduling purposes, the Back Forty pit was subdivided into three mining phases. Mining commences in a small higher-grade pit and then expands outwards by pushing back the pit wall. This enables annual waste rock stripping quantities to be distributed to avoid significant fluctuations in the annual tonnages mined.

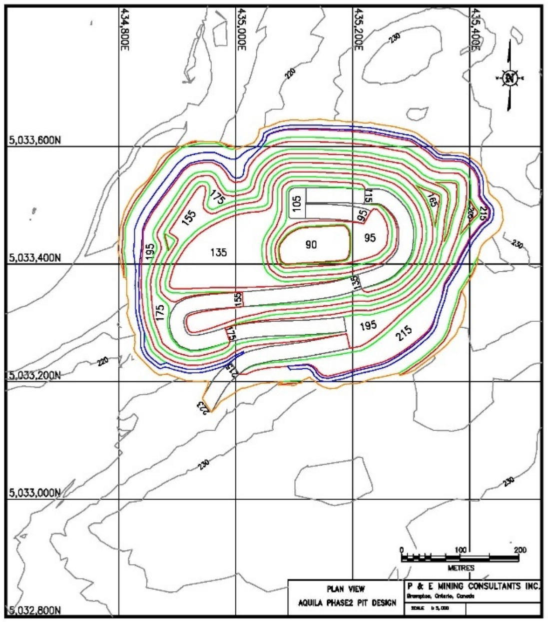
The three pit phases (cumulative) are shown in Figure 16.5.

FIGURE 16.5 OPEN PIT PHASES

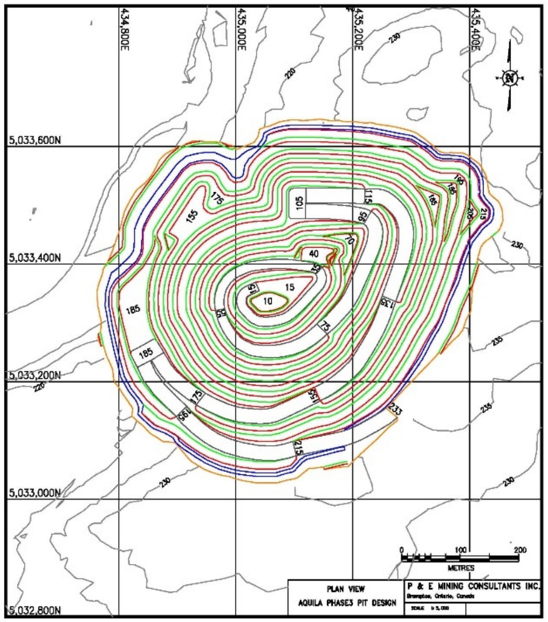
PHASE 1



PHASE 2



PHASE 3 (FINAL PIT)



16.1.8 Dilution and Mining Loss

The Mineral Resource model described in Section 14 of this Technical Report was converted to a sub-cell block model with a primary block size of 5.0 x 2.5 x 2.5 m (XYZ), and subdivided into smaller sub-cells in narrower portions of the Deposit and along the wireframe fringes. This block model is considered an undiluted model and contains 10.4 Mt of potential process plant feed in the pit-constrained Measured and Indicated Mineral Resource classifications.

The Selective Mining Unit (SMU) is deemed as the smallest block size that can be mined by the chosen equipment. For mine planning purposes SMU sizes were examined to assess potential dilution impacts. The use of a smaller SMU size decreases the amount of dilution but would increase the mining cost due to the use of smaller equipment.

In order to minimize dilution, a 2.5 x 2.5 x 2.5 m block size was used in Oxide and Pinwheel process plant feed types. For the Main Zone and Tuff Zone process plant feed, a block size of 5 m x 5 m x 5 m was used.

The Mineral Resource model blocks were re-blocked into SMU blocks. This step results in averaging of metal grades and dilution of grade where waste blocks are composited together with process plant feed blocks. For defining process plant feed and waste material for mine planning purposes, the NSR cut-off criteria were then applied to re-blocked SMU blocks values.

In situations where an SMU was re-blocked with very few mineralized blocks, the resulting SMU block may have been below cut-off value and designated as a waste block, resulting in a loss of process plant feed. In other situations where an SMU block incorporates more mineralization and less waste, the resulting SMU block may be above NSR cut-off value and would be designated as economic, resulting in a process plant feed tonnage gain albeit at a diluted grade.

The SMU re-blocking process was undertaken for the entire block model resulting in both process plant feed gains and losses. No additional dilution or loss criteria were applied beyond what was introduced during the SMU re-blocking step.

The estimated process plant feed tonnage in the SMU re-blocked model was 11.65 Mt (Measured and Indicated Mineral Resources), or approximately 112% of the undiluted tonnage (= 11.65 Mt ÷ 10.4 Mt). This equates to approximately 12% net dilution. This "net" dilution, includes both process plant feed losses and gains.

To estimate the process plant feed loss, a calculation was made using the sub-cell model for isolated sub-blocks above cut-off value but not included in the SMU model. An estimated process plant feed tonnage of 1.07 Mt was reported outside the SMU model, and inside the mineralized wireframes, indicative of a 10.3% loss (= 1.07 Mt ÷10.4 Mt).

The combined results of a 10.3% feed loss and 12% net dilution indicated the true dilution would be 22.3% (12%+10.3%) due to the SMU re-blocking process. This is reasonable given the narrow nature of some of the mineralized zones in the pit.

16.1.9 Open Pit Production Schedule

Separate production schedules have been developed for mining and processing. The mining schedules define the annual tonnages of process plant feed and waste rock that must be moved. Process plant feed will be delivered either to the primary crushers or placed onto one of the stockpiles. Waste rock is either taken to one of waste rock storage facilities ("WRF") or used for the construction of the tailings management facility ("TMF").

16.1.10 Stockpiling Strategy

The underlying premise to the mine to stockpile to process plant strategy is mining and accessing high value material as soon as possible and processing the high value material to maximize cash flow. The NSR cut-off value used to define high grade versus low grade varies from year to year in order to maximize NPV and head grades processed during the year.

Mineralized material will be treated based on main material types (Main, Pinwheel, Tuff, Oxides) that have a similar metallurgical response in the process plants. As the material types are mined, the material is kept separate and is either delivered directly to the primary crushers or to one of the stockpiles. A processing schedule defines the sequence of processing and has a different grade profile compared to the mining grade profile. The operating context is further described in Section 17.

The main stockpiles and metallurgical feed types are summarized in Table 16.3.

TABLE 16.3
STOCKPILING STRATEGY

Stockpile 1 - Type 1 HG/LG Main
Stockpile 2 - Type 2, 3, 4, 7, 8 Pinwheel
Stockpile 3 - Type 5 Tuff
Stockpile 4 - Type 6 HG/LG Oxide

Note: *HG = high grade, LG = low grade.*

16.1.10.1 Sulphide Plant (Flotation) Feed Types

Flotation feeds comprised the majority of the process plant feed types. Only Type 6 is an oxide type. Metallurgical testwork indicates that each of the flotation types is preferably campaigned separately through the process plant and this is the case for the Main, Tuff, and Pinwheel feeds. The five individual Pinwheel feeds have been grouped into three categories that must be processed separately, with Types 2 and 7 (Massive Sulphides, Cu Rich) combined, Types 4 and 8 (Massive Sulphides, Zn Rich) combined, and Type 3 (Stringers) remaining separate. When one of the flotation feed types is being processed, any different flotation feeds that are mined must be placed into an appropriate stockpile for future processing. At certain periods in time, the flotation plant would switch feed types and either reclaim from the stockpiles and/or accept direct delivery from the mine.

16.1.10.2 Oxide Plant (Leach) Feed Type

Oxide feed is the only material directed to the leaching circuit. This feed type has been sub-divided into high-grade and low-grade stockpiles. The processing schedule will target high-grade leach feed and defer the processing of lower-grade feed.

16.1.11 Open Pit Mining Schedule

The mining schedule has been decoupled from the processing schedule in order to both optimize mining efficiency and accelerate delivery of higher value material to the process plant. As a result, open pit mining is completed after five years (including six months of pre-stripping prior to commissioning of the process plants). After the five year open pit mine life, approximately 2.5 years of mill feed is available on stockpiles. This strategy not only optimizes Project NPV, but provides feed to keep the process plants at full capacity until the underground mine achieves full production.

In general, the mine scheduling criteria were to:

- Meet or exceed the flotation and leach feed delivery targets;
- Avoid building up excessively large feed stockpiles;
- Minimize the pre-strip tonnage and duration;
- Provide sufficient waste rock in pre-production to build the starter dams for the TMF; and
- Avoid large fluctuations in annual tonnages moved, or in equipment requirements.

Some of the key aspects of the mining schedule are shown graphically in Figures 16.6 and 16.7. Table 16.4 presents the life-of-mine (“LOM”) open pit production schedule.

Figure 16.6 indicates that the peak mining rate occurs in Years 2 and 3, with a total of 15 Mtpa of material moved. The mined head grades by year by feed type are shown in Table 16.5.

TABLE 16.4 **OPEN PIT MINING SCHEDULE**								
Type	**Units**	**Total**	**Year**					
			Y-1	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**
Overburden	kt	3,778	1,233	1,648	896	-	-	-
Waste Rock	kt	47,970	1,568	9,263	12,130	13,437	10,512	1,058
Total Waste	**kt**	**51,747**	**2,801**	**10,911**	**13,027**	**13,437**	**10,512**	**1,058**
Process Plant Feed Mining								
Type 1 ROM	kt	2,261	-	418	168	833	672	170
Type 2/7 ROM	kt	165	-	115	50	-	-	-
Type 3 ROM	kt	51	-	-	43	8	-	-
Type 4/8 ROM	kt	13	-	-	13	-	-	-
Type 5 ROM	kt	286	-	-	-	47	238	-
Type 6 ROM	kt	396	-	61	90	99	124	21
Type 1 S/Pile	kt	2,774	7	858	191	207	1,025	487
Type 2/7 S/Pile	kt	428	-	239	188	1	-	-
Type 3 S/Pile	kt	269	-	33	236	-	-	-
Type 4/8 S/Pile	kt	100	-	48	52	0	-	-
Type 5 S/Pile	kt	2,468	65	525	706	309	743	119
Type 6 S/Pile	kt	922	126	292	237	58	185	24
Total Feed	**kt**	**10,132**	**199**	**2,589**	**1,974**	**1,563**	**2,987**	**821**
Total Material	**kt**	**61,880**	**3,000**	**13,500**	**15,000**	**15,000**	**13,500**	**1,879**
Strip ratio	w:o	5.1	14.1	4.2	6.6	8.6	3.5	1.3
Feed to S/Piles	kt	6,961	199	1,995	1,609	575	1,953	629

Notes: *2/7 = 2 and 7, ROM = run of mine, S/piles = stockpiles, w:o = waste:ore ratio.*

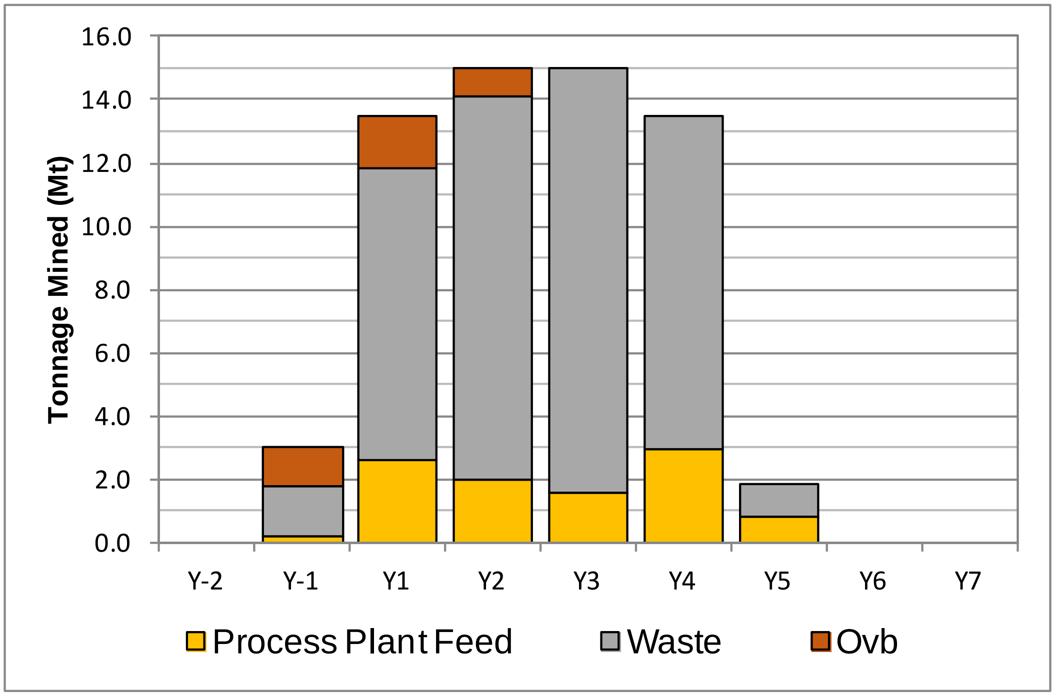
The NSR cut-off values by met type used to define process plant feed and waste rock are as follows:

Type 1	=	$26/t.
Type 2/7	=	$28/t.
Type 3	=	$24/t.
Type 4/8	=	$22/t.
Type 5	=	$28/t.
Type 6	=	$46/t.

TABLE 16.5 MINED HEAD GRADES BY FEED TYPE								
Feed Type	**Units**	**Total**	**Year**					
			Y-1	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**
Type 1	kt	5,035	7	1,276	358	1,040	1,697	657
Cu	%	0.28	0.01	0.38	0.21	0.23	0.28	0.21
Zn	%	4.73	0.01	4.59	4.61	3.05	5.31	6.30
Pb	%		n/a	n/a	n/a	n/a	n/a	n/a
Au	g/t	2.15	37.53	2.39	2.99	1.91	2.02	1.52
Ag	g/t	14.2	7.3	14.3	15.4	12.0	15.9	12.9
Type 2 / 7	kt	593		354	238	1		
Cu	%	2.23		2.55	1.75	0.92		
Zn	%			n/a	n/a	n/a		
Pb	%			n/a	n/a	n/a		
Au	g/t	2.54		2.97	1.90	1.92		
Ag	g/t	76.1		82.5	66.7	22.3		
Type 3	kt	321		33	279	8		
Cu	%	0.55		0.54	0.55	0.44		
Zn	%			n/a	n/a	n/a		
Pb	%			n/a	n/a	n/a		
Au	g/t	1.83		1.62	1.85	1.86		
Ag	g/t	19.4		25.5	18.8	14.4		
Type 4 / 8	kt	113		48	65	0		
Cu	%	0.92		1.08	0.80	0.57		
Zn	%	3.96		3.68	4.18	1.41		
Pb	%			n/a	n/a	n/a		
Au	g/t	1.05		1.13	0.99	1.40		
Ag	g/t	28.5		33.8	24.7	15.7		
Type 5	kt	2,754	65	525	706	357	982	119
Cu	%		n/a	n/a	n/a	n/a	n/a	n/a
Zn	%	1.80	0.76	1.17	1.19	1.22	2.82	2.11
Pb	%	0.50	0.28	0.21	0.32	0.27	0.89	0.58
Au	g/t	0.88	1.22	0.97	0.89	0.84	0.85	0.66
Ag	g/t	19.5	8.0	8.5	15.9	21.6	28.7	13.1
Type 6	kt	1,317	126	353	327	157	309	45
Cu	%		n/a	n/a	n/a	n/a	n/a	n/a
Zn	%		n/a	n/a	n/a	n/a	n/a	n/a
Pb	%		n/a	n/a	n/a	n/a	n/a	n/a
Au	g/t	4.36	10.90	4.33	3.58	3.12	2.92	6.21
Ag	g/t	47.2	7.8	33.7	50.6	49.8	73.9	47.5
Total Sulphide	**kt**	**8,815**	**73**	**2,236**	**1,647**	**1,406**	**2,678**	**776**
Total Oxide	**kt**	**1,317**	**126**	**353**	**327**	**157**	**309**	**45**
Total	**kt**	**10,132**	**199**	**2,589**	**1,974**	**1,563**	**2,987**	**821**

Figure 16.6 shows the mined material type by year. Overburden is mainly mined in Year -1 to Year 2. The majority of the material moved consists of waste rock. The process plant feed mining rate is initially slightly higher to allow low-grade separation and to maximize the value of the feed to be processed.

FIGURE 16.6 TOTAL ANNUAL OPEN PIT MINED MATERIAL TYPE



Stockpile re-handling and blending will commence in Year 1 and will continue for the life of the Project. Although physical mining activities (open pit and underground) will cease in Year 11, processing will continue into Year 13 from stockpiled material (see Figure 16.7 and Table 16.6). Over the life of the entire Project, approximately 10.4 Mt of stockpiled process plant feed will be re-handled. Approximately 33% of the total stockpiling re-handling will occur during the open pit life.

FIGURE 16.7 STOCKPILE RE-HANDLING

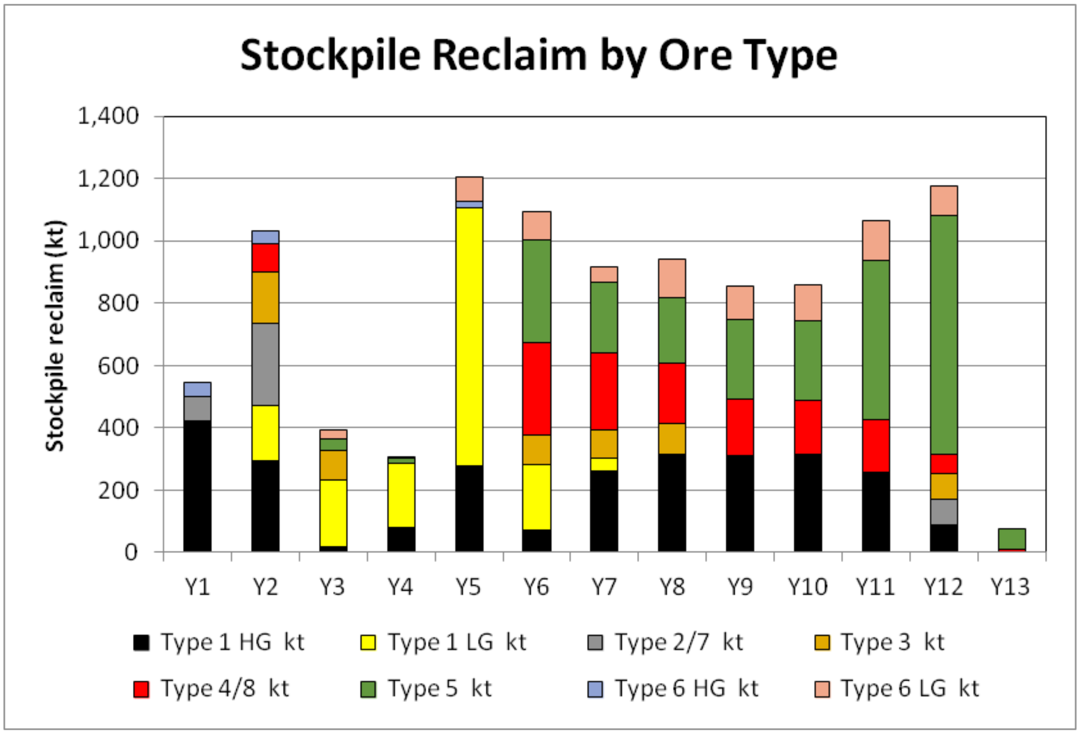


TABLE 16.6
STOCKPILE RECLAIM ACTIVITY

Met Type	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Type 1 HG	kt	2,707	423	293	19	80	277	70	261	315	310	315	256	87	-
Type 1 LG	kt	1,681	0	178	212	206	830	214	40	-	-	-	-	1	-
Type 2/7	kt	428	79	264	-	-	-	-	-	-	-	-	-	85	-
Type 3	kt	625	-	166	97	-	-	93	92	98	-	-	-	80	-
Type 4/8	kt	1,430	-	91	-	-	-	295	245	196	184	172	172	64	11
Type 5	kt	2,680	-	-	38	17	-	332	229	208	256	256	511	767	67
Type 6 HG	kt	104	43	38	0	4	19	-	-	-	-	-	-	-	-
Type 6 LG	kt	818	-	-	28	-	80	91	48	124	106	118	128	95	-
Total	**kt**	**10,474**	**546**	**1,031**	**395**	**307**	**1,207**	**1,095**	**916**	**941**	**856**	**860**	**1,066**	**1,178**	**78**

Note: *Y = year.*

16.1.12 Open Pit Mining Practices

Open pit mining will be undertaken using conventional mining equipment, and will follow similar practices used at other operations in the North America. The mining fleet will include blasthole drills, excavators, and haul trucks. Various support equipment will be required, such as dozers, graders, water trucks, and light vehicles for maintenance personnel and mine supervision. A computerized mine dispatch system will be installed to assist with proper truck allocation to mining areas and tipping points.

In order to improve mining selectivity and reduce dilution, two different bench heights will be used. The Oxide and Pinwheel Zones can be narrow and closely spaced and hence they will be mined using a 2.5 m high bench height. In the Main and Tuff Zones and in large waste areas away from the mineralized zones, bench heights of 5.0 m will be used to minimize unit mining costs. In waste areas near the oxide zones, some waste will be mined on a 2.5 m bench height as part of the process plant feed/waste separation step.

Smaller sized excavators will be used to mine the 2.5 m high benches. There will be higher unit mining costs for this selective mining, but this is offset by the benefits of reduced dilution and less waste rock delivered to the process plant.

The initial mining period will be used to acquire knowledge and experience with the rock mass and the mineralized zones, and will be used to assess the blasting patterns, blast vibration control, grade control methods, rock mechanics, and groundwater seepage rates. One would expect several months of operation to optimize the mining systems with site-specific knowledge.

16.1.13 Drilling and Blasting

Drilling and blasting will be a combined owner-operated and contracted operation. The owner will provide the blasthole drilling operations, including the drills and drilling manpower. A blasting contractor will be used to supply the explosives, prepare the blasts, charge the holes, fire the blast, and inspect the area post-blast.

All rock will be blasted and only the overburden is considered free-digging. Table 16.7 specifies the drilling and blasting parameters assumed for the Back Forty mining operation.

TABLE 16.7
DRILL AND BLAST DESIGN

Items	Units	Leach Feed	Flotation Feed (2.5 m Bench)	Flotation Feed (5.0 m Bench)	Waste Rock
In-situ rock density	t/m^3	2.80	3.69	3.69	2.78
Bench height	m	2.50	2.50	5.00	5.00
Subgrade drill (15%)	m	0.38	0.38	0.75	0.75
Total hole length	m	2.88	2.88	5.75	5.75
Hole diameter	mm	76.20	76.20	177.80	177.80

TABLE 16.7 DRILL AND BLAST DESIGN					
Items	**Units**	**Leach Feed**	**Flotation Feed (2.5 m Bench)**	**Flotation Feed (5.0 m Bench)**	**Waste Rock**
Powder factor	kg/t	0.30	0.30	0.30	0.30
Explosives in hole	kg	7.1	7.1	77.1	77.1
Rock blasted per hole	t	23.6	23.6	257.0	257.0
Drill pattern burden	m	1.7	1.4	3.4	3.9
Drill pattern spacing	m	2.0	1.8	4.1	4.8
Explosives density	kg/m^3	900	900	900	900
Explosives required	m^3	0.01	0.01	0.09	0.09
Powder column	m	1.73	1.73	3.45	3.45
Stemming column	m	1.15	1.15	2.30	2.30
% Powder column	%	60%	60%	60%	60%

Drilling will be done using blast hole diameters of 76 mm to 178 mm. To control blast vibration levels in the area of the cut-off wall, smaller blast holes may be used in the vicinity and these will be selected based on measured vibration levels.

Blast holes may be both dry and wet, however since it is expected that holes will be wet, an emulsion explosive will be used predominantly. The explosive contractor will have a mix plant on site and deliver the explosive into the blast hole. The assumed explosives powder factor will be approximately 0.3 kg/t for all materials.

In the mineralized zones, the blast hole spacing will be in the range of 1.7 to 2.0 m for Pinwheel and Oxide Zones and 3.4 to 4.1 m for Main and Tuff Zones. These holes would also be used for grade control to define process plant feed and waste contacts.

16.1.14 Grade Control Drilling

It is assumed that grade control will mainly be done by assaying the blast holes in the vicinity of the mineralized zones.

For the 2.5 m benches, the blast hole spacing of 2 m will provide tight grade control. However, there may be areas around the pit where additional targeted grade control drilling is required. An additional drill rig is included to account for this activity. Grade control holes drilled at multiple bench height depths will provide for advanced mine planning and blast layout optimization.

16.1.15 Loading and Hauling

The primary waste loading units will be a front-loading shovel with an 8 m^3 bucket. The same unit will be used for loading the Main and Tuff Zones, while greater selectivity will be achieved in the Pinwheel and Oxide Zones with 5 m^3 bucket backhoe excavators.

A large wheel loader with an 8 m^3 bucket will be used to support both the process plant feed and waste mining activities. An additional 8 m^3 wheel loader will be used at the stockpiles to load trucks when needed.

Once the underground mine is operational, process plant feed and waste will be placed next to the portal. From there, a front-end loader and mine trucks will be used to transport these materials to their required locations. This activity will persist for the entire underground mine life from Year 5 to 11. The annual quantities are relatively small, in the range of 1.0 Mtpa.

All material will be hauled using a fleet of 90 t capacity haul trucks.

Table 16.8 summarizes the loading and hauling fleet requirements for the LOM. As material movement quantities and haulage distances fluctuate, there may be corresponding changes in the major equipment fleet requirements.

16.1.16 Stockpile Handling

Four stockpiles will be required, with space for separate high grade and low grades areas, and different Pinwheel types. Most of these stockpiles will be located several hundred metres away from the crusher area due to a restricted mine site footprint.

When the decision is made to campaign a specific process plant feed type, the primary crusher would be fed by both direct feed from the mine (if that feed type is available at the mining face) as well as material reclaimed from the corresponding stockpile. Due to the distances from the stockpiles to the crusher, it is expected that most material must be loaded onto trucks and hauled to the crusher.

Stockpile re-handling will be done using a front-end wheel loader and 90 t haul trucks.

16.1.17 Pit Dewatering

Water will normally enter the pit due to pit wall seepage and precipitation events. In extreme storm events, the open pit is designated as the emergency water containment facility. For example, if the TMF or the Contact Water Basin ("CWB") are inundated and exceed their capacities, then spillways will direct excess water into the pit.

Under normal circumstances, the expected average water handling requirement is 30 m^3 per hour. A pumping system sized for 30 m^3 per hour will be installed and used to pump water to surface. This water is deemed contact water and will be retained on site.

Under the extreme storm events, the pit bottom may flood temporarily. The site water management system directs excess water from extreme storm events into the open pit. The mining equipment will relocate to upper benches in such circumstances. Should the mine operation be shutdown entirely, the process plant can continue to operate with feed delivered from stockpiles. The pumping system will drain the pit over several days (depending on the size of the storm event and available capacity for management, treatment, and discharge of mine dewatering water at the CWB and water treatment plant).

16.1.18 Auxiliary Pit Services

The primary mining operations will be supported by a fleet of support equipment consisting of bulldozers with ripper attachments, graders, water trucks, maintenance vehicles, and service vehicles. A list of major and support equipment for auxiliary services is provided in Table 16.8. Beyond year 7, the equipment is mainly required for hauling underground material and stockpile operations.

TABLE 16.8
MINE EQUIPMENT REQUIREMENTS

Item	Year						
	Y-1	Y1	Y2	Y3	Y4	Y5	>Y6
Drill, 90 mm, Crawler, Percussion	1	2	3	2	2	1	
Drill, 165 mm, Crawler, DTH	2	3	3	3	3	2	
Stemming Truck, 15 t	1	1	1	1	1	1	
Hydraulic Shovel, 8 m^3 (C6015)	1	2	2	2	2	1	
Excavator, 5 m^3 (C374F)	1	1	2	1	1	1	1
Haul Truck 90 t (K785)	3	8	13	13	13	10	1
Personnel van/bus	1	1	1	1	1	1	
Dozer (D8T)	3	3	3	3	3	3	1
Mechanic and Welding Truck	1	1	1	1	1	1	1
Excavator, 4 m^3 (CAT 336E)	1	1	1	1	1	1	
Fuel and Lube Truck	1	1	1	1	1	1	
Grader (GD655)	2	2	2	2	2	2	1
Light plant	4	4	4	4	4	4	1
Pickup truck	10	10	10	10	10	10	2
Pit Water Pumps Diesel	3	3	3	3	3	3	
Wheel Loader 4 m^3 (C966)	1	1	1	1	1	1	
Water truck (40 t, 6,500 gallon)	1	1	1	1	1	1	1
Wheel Loader 8 m^3 (WA800)		2	2	2	2	2	2

16.1.19 Waste Storage Facilities

There are several types of waste material that will be mined from the open pit. These consist of topsoil, overburden, and waste rock. A portion of the waste material will be used for construction fill or as reclamation material.

Overburden will be used to build some of the initial earthworks such as roads, pads, dams, and process plant site fill. Waste rock will be used in the TMF starter dams and dam raises, as described in Section 18.

16.1.19.1 Topsoil

There is a thin layer of topsoil over the open pit and this material will be dozed into piles and recovered, with quantities as shown in Table 16.9. This material will be placed into several topsoil stockpiles and will be used as reclamation material.

TABLE 16.9
TOPSOIL VOLUMES FROM MINING ACTIVITIES

Source	Quantity (m^3)
Pit Phase 1	17,000
Pit Phase 2	19,500
Pit Phase 3	10,600
Under Stockpile 1	13,100
Under Stockpile 2	2,000
Haulroad Phase 1	7,600
Haulroad Phase 2	0
Haulroad Final	200
Haulroad to SWRF	1,100
Total	**71,100**

16.1.19.2 Overburden

Overburden stripped from the upper benches of the open pit will be used for construction fill with excess placed into the overburden stockpile located on the east side of the Project area.

At the end of mine life the overburden will be re-handled and used as reclamation material.

16.1.19.3 Waste Rock

72% of the 48 Mt of waste rock has been characterized as potentially acid generating (“PAG”). The remaining 28% of the waste rock has been characterized as non-acid generating (“NAG”). When it is possible to mine and segregate NAG waste rock separately from PAG waste rock, it will be preferentially designated as construction fill for roads and the TMF containment structure.

Waste rock will be placed into three different areas. Waste rock will be used in the ongoing construction and raising of the TMF dams. The remaining waste rock will be placed into the North Waste Rock Facility (“NWRF”) and the South Waste Rock Facility (“SWRF”).

According to the current reclamation plan, at the cessation of mining, waste rock will be re-handled from the WRFs to backfill the open pit.

16.1.20 Open Pit Support Facilities

The mine support facilities are described in Section 18 (Infrastructure) and will consist of the following:

- Truck maintenance shop with three service bays;
- Mine supervision and technical office space;
- Truck wash station;
- Explosive storage area for blasting agents and blasting supplies;
- Fuelling station;
- Tire station;
- Warehouse for parts storage; and
- Laydown area for spares tire, buckets and large components.

16.1.21 Open Pit Manpower

The mining manpower will consist of mine supervision and technical support as well as mine operators and shift supervision. Table 16.10 lists the total open pit manpower requirements.

The office and technical support staff will work five 8-hour days per week with weekends off. Grade control technicians will be required to work on a 7-day cycle to provide daily coverage in the mine.

The mine operators will work 12-hour day and night shifts on an 8-day work cycle, consisting of 4 days on and 4 days off. This equates to 48 hours per work cycle over 45.6 cycles per year (365 ÷ 8 days) or 2,189 h/yr.

Office staffing levels will remain fixed each year, however, mine operator requirements will fluctuate with tonnage targets and mine productivity changes.

TABLE 16.10 OPEN PIT MINING MANPOWER REQUIREMENTS							
Occupation	**Year**						
	Y-1	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**>Y6**
Driller	3	15	18	14	13	2	
Driller Helper	2	9	9	8	8	1	
Stemming Operator	1	1	1	1	1	1	
Truck Drivers	10	30	49	48	47	36	4
Excav 1 Operators	3	5	5	5	5	3	
Excav 2 Operators	2	3	4	3	2	2	1
HD Mechanic	4	19	25	24	23	4	3
Pit services (dewatering)	2	2	2	2	2	2	
Grader Operator	8	8	8	8	8	8	2
Dozer Operator	12	12	12	12	12	12	2
Water Truck Operator	4	4	4	4	4	4	2
Utility Operators	10	30	49	48	47	36	4
Mine Superintendent	1	1	1	1	1	1	
Mine Gen Foremen	1	1	1	1	1	1	
Mine Foremen	4	4	4	4	4	4	
Mine Clerk	1	1	1	1	1	1	
Equipment Trainer	1	1	1	1	1	1	
Maintenance Gen Foreman	1	1	1	1	1	1	
Maintenance Foreman	4	4	4	4	4	4	
Planner	1	1	1	1	1	1	
Welder	2	2	2	2	2	2	1
Gas Mechanic	2	2	2	2	2	2	1
Tireman	1	1	1	1	1	1	1
Partsman	1	1	1	1	1	1	
Laborer	3	3	3	3	3	3	
Chief Mine Engineer	1	1	1	1	1	1	
Senior OP Engineer	1	1	1	1	1	1	
Geologist	2	2	2	2	2	2	
Surveyor	1	1	1	1	1	1	
Survey Tech	2	2	2	2	2	2	
Mine Tech	1	1	1	1	1	1	
Grade Control Tech	4	4	4	4	4	4	1
Total	**90**	**147**	**176**	**168**	**164**	**114**	**18**

16.2 UNDERGROUND MINING

The underground portion of the Back Forty Deposit is complex. The same metallurgical ("MET") types as described in the open pit mining section of this Technical Report will be encountered underground. The underground portion of the Deposit consists of a series of lenses encompassing an area roughly 680 m long, 420 m wide and extending to a depth of roughly 440 m below surface topography. Individual mined lenses vary from 2-942 kt in size.

Extraction of the underground Mineral Resource will be achieved by a combination of mechanized Cut and Fill ("CF") or Longhole ("LH") methods. CF mining is the dominant method, producing approximately 63% of mined tonnes, with LH producing the remaining 37% of tonnes. CF mining uses one of four stope sizes, and LH mining uses one of two stope size subsets and orientations (transverse or longitudinal).

All waste and mineralized material development will be undertaken using jumbo drills and mechanized bolting units, thus allowing for sharing of the equipment fleet between development and production assignments, allowing crews and machinery to perform production and/or development tasks in nearby mining areas while limiting machinery travel distances. Mineralized material will be extracted from the CF and LH stopes using load-haul-dump ("LHD") units and loaded directly into underground trucks for transport to surface.

Access to the Deposit is via a ramp from surface, with the underground portal located on the 187.5 m pit bench. All development and production material from underground is hauled to, and dumped at, a portal stockpile. From the stockpile, open pit trucks will transport the material to its final destination. Backfilling of the stope areas is achieved through the use of Pastefill ("PF"), delivered via two boreholes from the surface PF Plant. PF varies from 3-7% cement by mass, depending on application: higher cement contents are used for artificial sill pillars, lower cement contents are used otherwise. The PF system has a planned capacity of 2,300 tpd and the PF Plant is to be operated for 16-18 hours per day on average. All stoping areas are planned to be filled with pastefill. Rockfill is not planned to be utilized.

The underground mine is equipped with a high-capacity water pumping system capable of moving 109 L/s to surface if necessary. Ventilation is provided via three powered fresh air raises, with the portal and a single unpowered return air raise for exhaust. Electrical power is supplied initially at 15 kV, with step-down transformers distributed throughout the mine. The mine also has a small compressed air distribution system capable of providing 0.45 m^3/s at standard temperature and pressure.

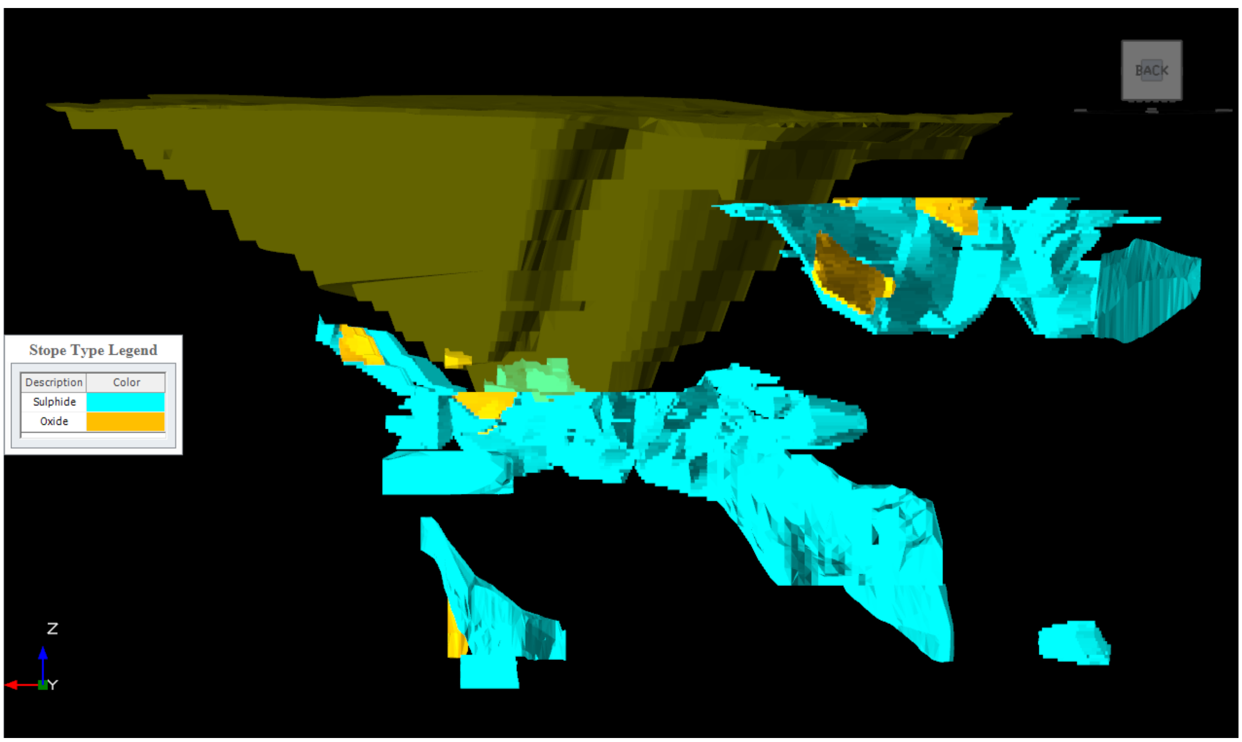
The underground commences construction and development in Q1 of Year 5, with production beginning at the start of Q3 of Year 5. Commercial production is achieved midway through Q4 of Year 6. The production rate of the underground operation varies depending on development requirements, with a nominal commercial production rate of 2,300 tpd, increasing to a maximum of 3,200 tpd in Year 7, before decreasing slightly towards the end of mine life in Year 9 as CF mining areas are exhausted and the mine transitions to lower-value LH stopes.

The total mined and recovered portion of the Deposit comprises 5,717 kt of material with an average Net Smelter Return ("NSR") value of US$109.24/t. Table 16.11 shows the tonnage distribution by MET type and Figure 16.8 shows the location of the oxide and sulphide material.

TABLE 16.11 POTENTIALLY MINEABLE PORTION OF THE MINERAL RESOURCE									
	Mineral Resource			**Average Material Metal Contents**					**NSR**
	MET	**Type**	**Tonnes (k)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**US$/t**
Potentially Mineable Mineral Resource	1	Sulphide	3,133	1.66	21.76	0.33	3.07	0.19	102.58
	2	Sulphide	-	-	-	-	-	-	-
	3	Sulphide	387	2.37	14.69	0.36	0.25	0.11	75.83
	4	Sulphide	1,029	1.65	17.50	0.67	0.88	0.08	83.66
	5	Sulphide	295	1.77	47.45	0.04	2.98	0.57	112.27
	6	Oxide	160	6.96	66.09	0.10	0.23	0.52	287.07
	7	Sulphide	-	-	-	-	-	-	-
	8	Sulphide	713	0.59	22.19	0.28	8.39	0.67	150.89
	Total		**5,717**						**109.24***

* *Average NSR is derived from MET stream blending, not weighted averages, MET = metallurgical.*

FIGURE 16.8 UNDERGROUND MINING AREAS, VIEW LOOKING SOUTH



16.2.1 Design Methodology

The Back Forty underground mine design was driven largely by the following parameters:

- Production rate of 2,300 tpd minimum.

- Cut-Off Grade ("ECOG") of $75/t for Sulphides and $85/t for Oxides.
- CF mining for high-value or low-dipping material.
- LH mining for lower-value material.
- An operating pit in close proximity to the underground mine.
- Paste backfill for all stoping areas.
- Mineralized and waste material transfer to surface by truck.
- Numerous mining faces for maximum scheduling flexibility.

Since the Deposit is composed of groups of lenses with mining limits defined by NSR economics, rather than a single continuous geological deposit, the development was optimized to access each lens in an economically efficient manner. Certain stoping areas are large enough to sustain repetitive levels (primarily LH mining areas); most are not. As such, each area was designed and evaluated as a single stoping block prior to being incorporated into the overall mine design to ensure maximum economic benefit to the Project. The infrastructure of the mine (main access ramp, ventilation system, mine services) was then designed around the optimized mining areas to best serve their needs. This was found to be more economic than using repetitive level designs with large lateral extents to reach widely distributed stopes.

Primary services (ventilation, dewatering, power) are run in the main ramp areas to each mining zone, and each specific stope has services provided via auxiliary lines (auxiliary vent duct, 100 mm pipe instead of 200 mm, lower voltage power lines). An underground maintenance shop was located near the centroid of the mining areas, just off the main haulage route, to minimize equipment travel distances while minimally impacting trucking routes. The dewatering system is designed to handle several times the expected inflows, and is set up to allow individual areas of the mine to function in isolation if necessary. Stopes are provided with their own power centres, re-muck bays, sumps and other infrastructure, independent of the main ramp. Some stopes share services where they are positioned closely enough together that separate accesses would be less economic, but the majority have private accesses. This modular design allows for many available mining faces that can operate largely independently of each other, and de-risks the production plan, as an issue in any individual stope is unlikely to result in a significant loss to production.

Mining methods were selected by evaluating the geometry of each specific mining area, and determining which method, or set of methods, provided the maximum economic benefit from the area. Most CF mining areas use multiple opening sizes to maximize tonnage recovered, whereas most LH mining areas normally use a single size of stope to maximize productivity. Only one mining area is planned to use both CF and LH methods in close proximity.

In general, CF mining was selected for high-value material where calculated dilution from LH mining was considered unacceptable, or for low-dipping areas where LH mining would result in low recovery. LH mining was selected for lower-value material where bulk mining methods can maximize tonnages while minimizing costs, or in areas where repetitive level designs can be easily supported by stope geometries (high-dipping areas where one or both of the under-cut or over-cut accesses could be shared with another stope). In general, lower-value LH material is planned to be mined using transverse methods, and higher-value LH material is planned to be mined using longitudinal retreat methods.

All mining areas will be filled with PF. Development waste was determined to be uneconomic to use as fill in CF mining areas due to transport and placement costs, and LH mining primarily takes place towards the end of mine life when the quantity of waste available underground is minimal. Therefore, all material from production and development will be transported to surface, and all underground voids requiring fill will be filled with PF. The PF cement content and strength will be determined on a location-by-location basis, with higher cement contents used in areas where artificial sill pillars are required to support mining operations, and lower cement contents otherwise. Cement contents will range from 3-7% by mass.

No material handling system, other than the PF lines, has been included in the design. The economic impacts of blending of multiple MET types in passes, the additional costs of rehandling material, along with the significant capital expenditures for passes and load-outs were determined to be greater than the impacts of on-level loading. Each stoping area is therefore set up to allow on-level loading of trucks within a short distance of a re-muck bay or attack ramp.

The result of the above is a mine with great operational flexibility and minimized capital costs due to its modular design.

16.2.2 Geotechnical Considerations

Detailed geotechnical work has not been performed on the underground portion of the Deposit, however a geotechnical characterization program has been carried out for the open pit by Knight Piesold Ltd. ("KP"), which involved significant geotechnical drilling, sampling, and evaluation. Evaluation included packer testing, far-field stress condition modelling, RQD and rockmass quality analyses, and mining-induced deformation estimates. It is P&E's opinion that sufficient work has been done related to the open pit portion of the Deposit to evaluate the underground portion at a PEA level of study.

The Deposit is divided into two zones: the Pinwheel Zone on the west side of the pit near the Menominee River; and the Main Zone to the East and below the pit (includes the Oxide Zone and the Tuff Zone). Faulting is prevalent in the Pinwheel Zone (four major faults), while only one significant fault structure exists deeper in the Main Zone. Water inflows in the Pinwheel Zone are expected to be moderate, while inflows in the Main Zone are expected to be minimal. The Pinwheel Zone has two major and three minor joint sets, while the Main Zone has one set of each type. The rock of the Pinwheel Zone is generally categorized as Fair, with lower and more variable rock mass quality than the Main Zone. The Main Zone is generally categorized as Good.

The Back Forty Deposit is affected by three pillars of varying thicknesses, as shown in Figure 16.9:

1. Pinwheel Pillar (affecting the Pinwheel Zone near the Menominee River).
2. Main Pillar (affecting the Main Zone).
3. Pit Pillar (affecting areas around and under the open pit).

Due to the quantity of mineralized material below the open pit that is affected by the Pit Pillar, an artificial sill pillar ("Plug") comprised of compacted cemented rock fill ("CRF") will be emplaced in the bottom of the pit after mining. The Plug will allow mining to proceed up to the original bottom of the open pit, allowing mining of material that would otherwise have been excluded from the mine plan.

Since pit backfilling operations overlap with underground mining operations as of Q4 of Year 9, KP has recommended an additional pit backfill Exclusion Zone extending 50 m laterally and below the level of backfill in the open pit to minimize the potential for fluid ingress from the pit into the underground mine.

The extents and interactions of the Pinwheel, Main and Pit Pillars; Plug; and Exclusion Zone over time are shown in Figures 16.10 to 16.12.

FIGURE 16.9 PILLAR THICKNESS

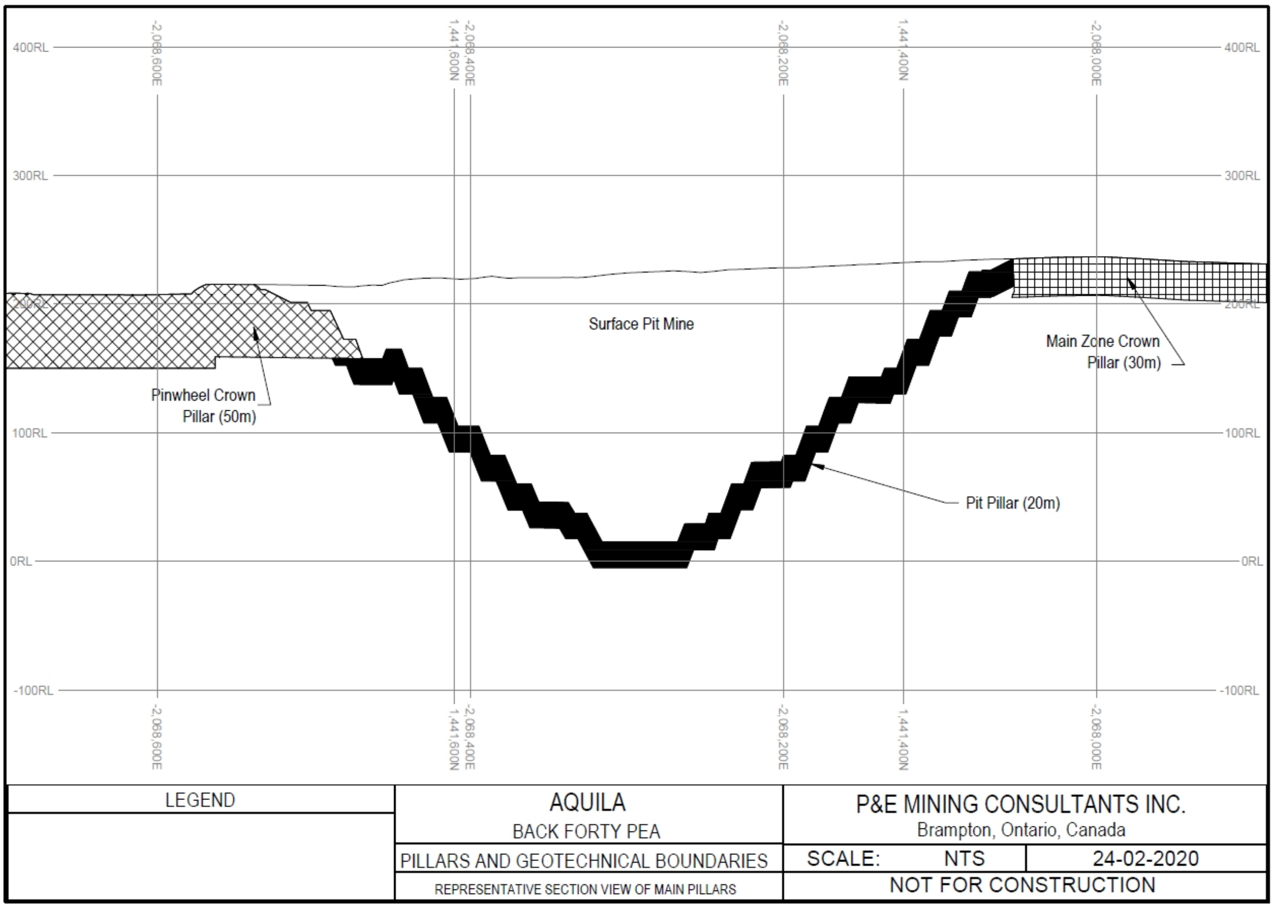


FIGURE 16.10 OPEN PIT FILL EXCLUSION ZONE AT END OF YEAR 9

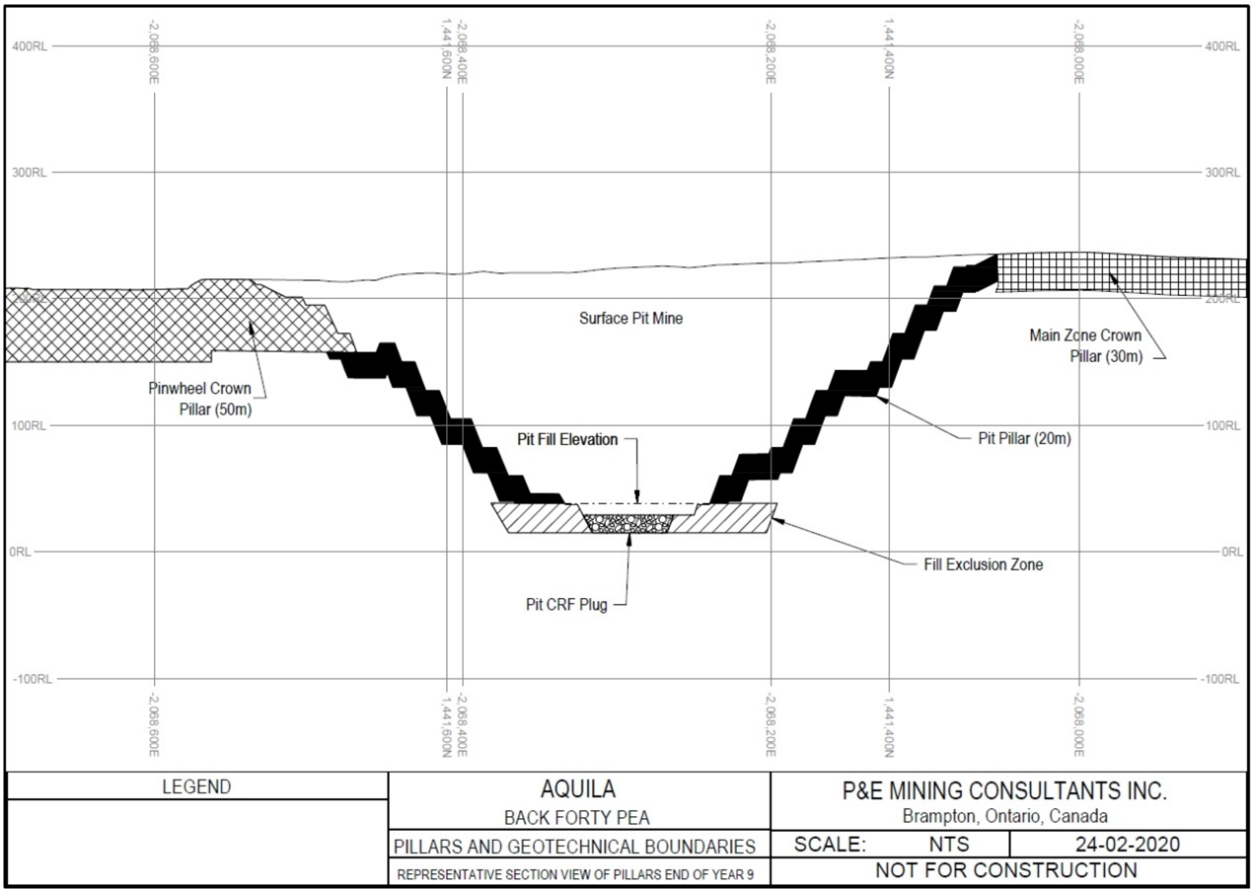


FIGURE 16.11 OPEN PIT FILL EXCLUSION ZONE AT END OF YEAR 10

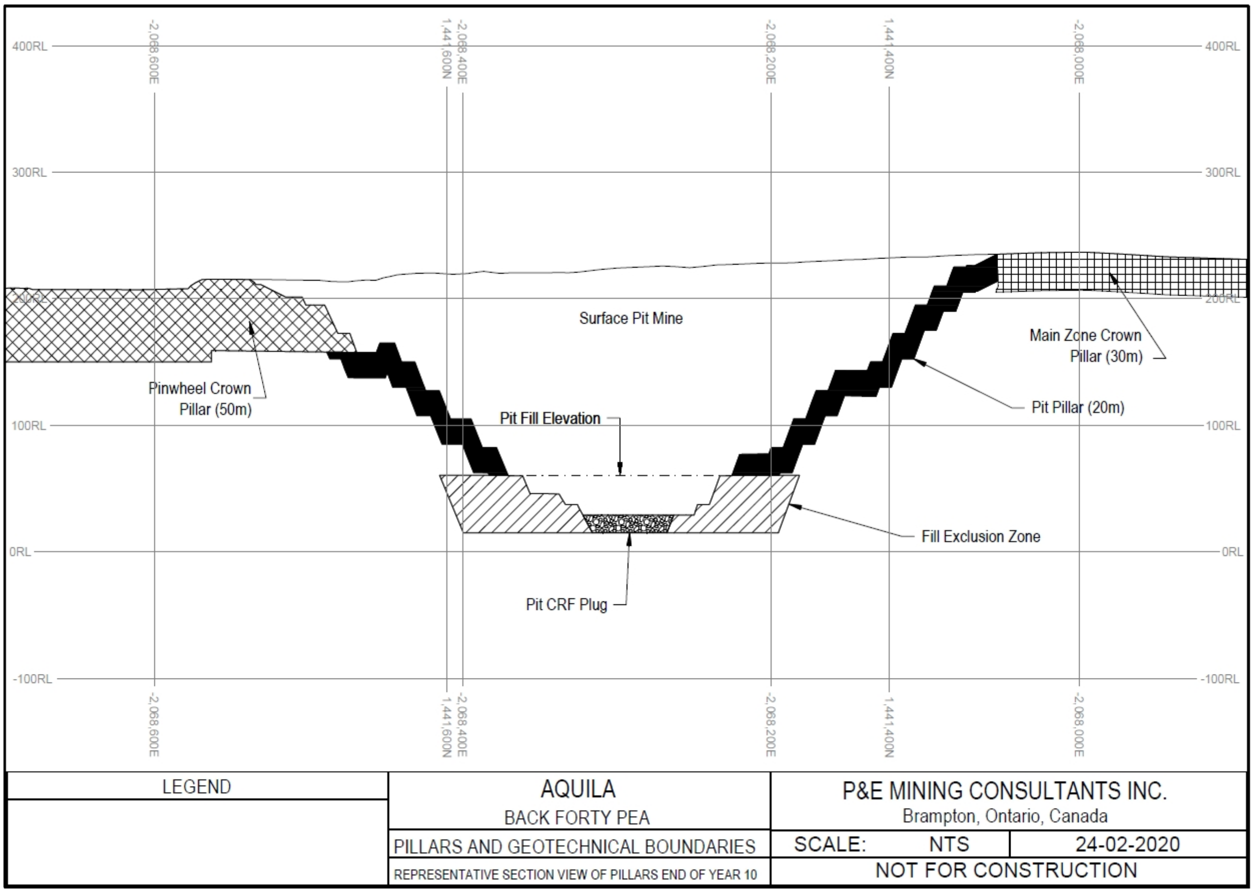
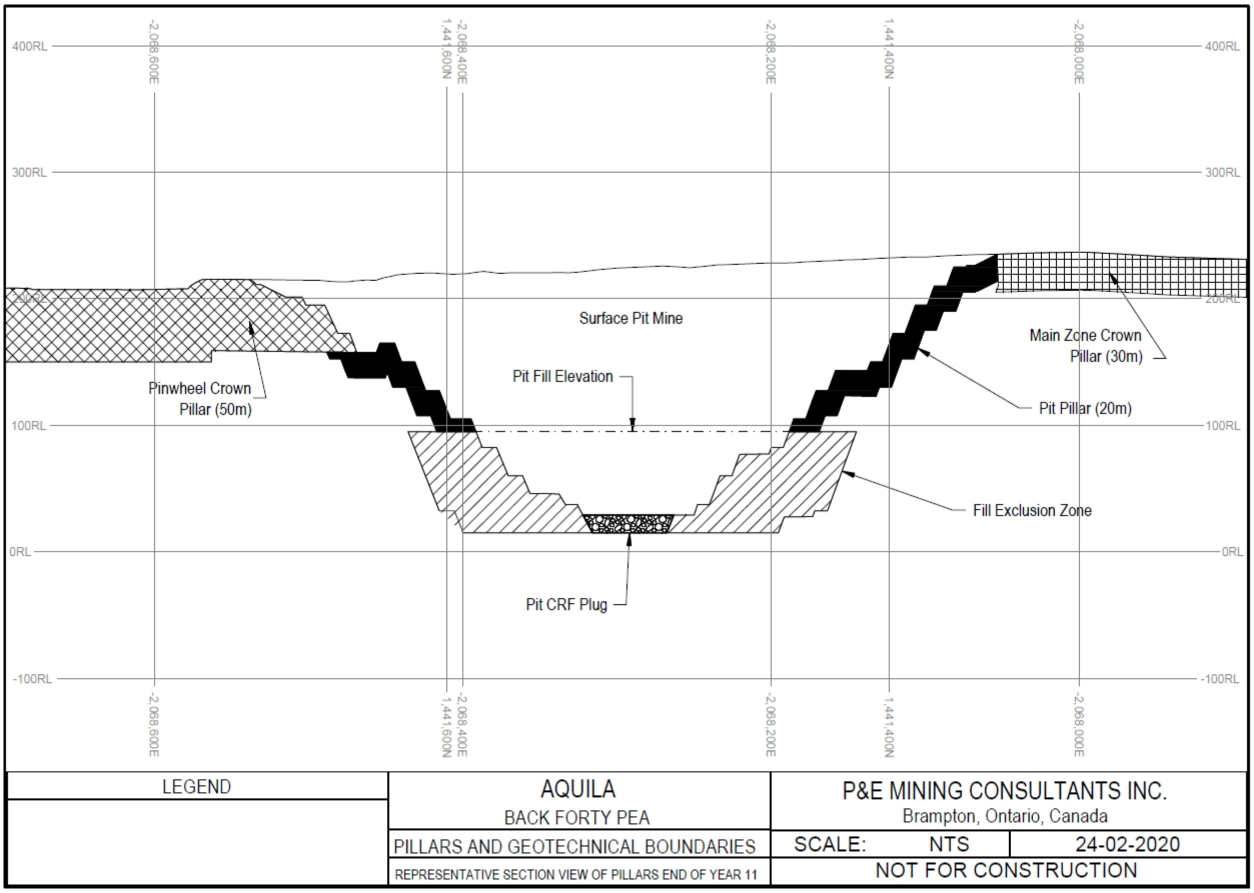


FIGURE 16.12 OPEN PIT FILL EXCLUSION ZONE AT END OF YEAR 11



A combination of empirical methods (Critical Scaled Span method), numerical modelling in Rocscience's RS2, and beam theory (for artificial sill pillars) were used to recommend pillar thicknesses for these designs. For all pillars, a Probability of Failure ("PoF") of 5% was used when determining the design pillar thickness. Further details for pillar calculations can be found in KP's report "NB19-00836 – Crown Pillar Recommendations" dated November 28, 2019.

While KP's evaluation was robust given the information provided, there are significant quantities of mineralized tonnes contained in pillar areas. Recovery of those tonnes has not been evaluated, and further study of these areas is recommended.

16.2.2.1 Pinwheel Pillar

The Pinwheel Pillar is intersected by numerous faults, which are partially responsible for the requirements of a pillar with a minimum thickness of 50 m. Due to a lack of bathymetry data on the Menominee River, P&E has used a 60 m thick pillar below the river area to allow for a water depth of 10 m. P&E and KP both recommend the collection of bathymetry data for the river to improve the accuracy of crown pillar estimates. Once past the river bank, the pillar is returned to its original thickness of 50 m, and extends to the northwest edge of the open pit. No mining within the Pinwheel Pillar is included in the underground mine plan.

16.2.2.2 Main Pillar

The Main Pillar is not intersected by the faults found deeper in the Main and Tuff Zones, and therefore its minimum thickness is less than that of the Pinwheel Pillar, at 30 m. This crown pillar extends around the pit, excepting the NW wall area, where the Pinwheel Pillar supersedes it. No mining within the Main Pillar is included in the underground mine plan.

16.2.2.3 Pit Pillar

The Pit Pillar extends in all directions around the final open pit, except where it intersects the Main Pillar or the Pinwheel Pillar, where these pillars supersede it. Due to the sizes of openings, this pillar has a thickness of 20 m where it abuts CF mining operations, or 30 m where it abuts LH mining operations. To maximize recovered tonnes, only CF mining operations have been planned to abut this pillar, minimizing its impact on the mine plan.

Below the pit floor, mining has been scheduled within the Pit Pillar. These areas will be mined out and filled after the installation of the CRF Plug, which will form an artificial sill pillar of sufficient integrity to allow their extraction.

No mining is planned within 20 m lateral offset of the exposed final pit walls to maximize wall integrity for the eventual deposition of tailings and waste rock in the open pit.

16.2.2.4 Pit Backfill Exclusion Zone

Backfilling of the exhausted open pit is scheduled to begin in Q4 of Year 9. Due to the potential for ingress of fluid from tailings into the underground through fractures in the rock mass around the open pit, KP has recommended an Exclusion Zone extending 50 m laterally around and below any area containing tailings backfill. The CRF Plug supersedes the Exclusion Zone for areas beneath the Plug, allowing for the recovery of in-situ material situated there (see Figures 16.10 and 16.11).

There is slightly more than one year of time between the emplacement of the Plug and beginning of open pit backfilling. This period will be used to extract the mining areas situated directly below and adjacent to the Plug, moving the mining fronts away from the area influenced by the Exclusion Zone, and further reducing any potential risk to the underground mine associated with tailings in the open pit. No tonnes are expected to be lost in the Exclusion Zone in the current mine plan.

16.2.2.5 Cemented Rockfill Pit Plug

An artificial sill pillar comprised of roller-compacted CRF will be emplaced in the bottom of the open pit. As per KP's recommendations, this pillar will be 15 m thick and have a strength of 4-5 MPa. This strength will be achieved by using a binder content of 6-8% cement, similar to the cement content of the high-strength PF in use for artificial sill pillars in underground mining blocks. Based on pit volumes, the Plug will require a total volume of approximately 115,000 m^3 of CRF. Placement of the Plug is expected to take approximately three months at a rate of 2,000 tpd, with another three months allotted for curing to its ultimate strength.

Placement of the plug will begin in Q4 of Year 7, and the Plug will be cured by the end of Q2 of Year 8. The Plug will support mining operations inside the area that would previously have been part of the Pit Pillar. The Plug will support mining operations in the areas below it of up to 10 m span using either CF or LH methods. P&E has planned for only CF mining to take place below the Plug to allow better control of opening sizes. The 10 m allowable span exceeds the largest CF opening planned for the underground mine, however for additional safety, the largest span in the area influenced by the Plug will be 5 m.

Once the Plug is in place, testing of the Plug will be undertaken prior to the commencement of pit backfilling to confirm its strength, permeability and other geotechnical properties, and to modify the Pit Backfill Exclusion Zone adjacent to the Plug appropriately.

16.2.3 Stope Design Recommendations

Table 16.12 shows KP's recommendations for stope sizing in the various underground mining zones at the Back Forty Deposit. KP has assumed that any area with a dip shallower than 55° will be mined by CF methods, and that LH mining areas will be on 30 m level spacings with 5 m drift heights. These recommendations have been incorporated into the mine plan. The term "Long support" in the table refers to tendons greater than 2.4 m in length, however specific tendon types (water-inflated connectable bolts, cable bolts, etc.) have not been recommended. P&E has assumed cable bolts will be used for all long support applications.

TABLE 16.12
STOPE SIZE RECOMMENDATIONS

Domain	Mining Method	HW-FW Span	Strike Length (min 55° Stope Dip)	Comments
Main Zone and Tuff Zone	Open Stoping (Transverse)	14 m	15 m	Larger strike length of 20 m may be possible in CHTF (Chlorite Tuff) unit.
	Open Stoping (Longitudinal)	8 m	15 m	Long support may be required on a case by case basis to manage wedges.
	Cut-and-Fill	8 m	N/A	Long support may be required on a case by case basis to manage wedges in the Main Zone.
		10 m	N/A	Long support required in stope back.
Pinwheel Zone	Cut-and-Fill	5 m	N/A	Long support may be required on a case by case basis to manage wedges.

16.2.3.1 Ground Support

No current ground support designs or recommendations exist. As the Pinwheel Zone is expected to have Fair rock quality and the Main Zone is expected to have Good rock quality, P&E has assumed that ground support requirements for similar openings in similar rock will be sufficient.

Permanent development (ramps, main levels, etc.) is a minimum of 5.0 m W x 5.0 m H and is expected to use 2.4 m resin rebar of 22 mm diameter with 100 mm opening 6-gauge screen on the walls and back. Where conditions permit, the rebar in the walls will be reduced to 1.8 m in length. Shotcrete and/or straps will be used on a case-by-case basis as necessary.

Temporary development (attack ramps or FW accesses for LH mining) will be a maximum of 5.0 m W x 5.0 m H and is expected to use 2.4 m resin rebar and screen in the back, with 1.8 m long friction bolts (split sets) of 39 mm diameter in the walls, with screen on the walls. Shotcrete and/or straps will be used on a case-by-case basis as necessary.

For large intersections, long support will be used on a 1.8 m x 1.8 m grid pattern.

LH stopes, when necessary, will have location-specific long-support patterns. Costing for the supports was based on rings of 6 m grouted 15 mm diameter twin-strand bulbed cables.

CF mining areas will use the same support standards as temporary development. For CF Type 1 mining areas where the span is 7.5 m, long support may be used in specific locations of poorer ground conditions.

Vertical development will generally be supported, as the vent raises are used for emergency escapeways. Raises driven through standard LH methods will be supported with 2.4 m resin rebar and screened during the construction of ladders and landings. For longer raises driven by Alimak, support will be installed during driving of the raise, and an escapeway will be installed during stripping of the raise.

16.2.3.2 Pastefill Strength

Backfill strength requirements vary depending on whether future mining will happen above, below or laterally adjacent to the pastefill. As such, a range of PF strengths was recommended by KP (refer to memo “NB18-00168 Evaluation of Paste Fill Strengths”, dated March 21, 2018). Testing of a sample of Main Zone tailings was performed by Paterson and Cooke Canada Inc. (“PC”) in 2017 (refer to “ARB-32-0213 Backfill Test Work Report”, dated December 21, 2017) and determined that a binder content of 3% of solids mass would provide sufficient strength for laterally adjacent mining (using paste as a wall) or overhand mining above paste (using it as a floor), and pastefill using a binder content of 5% of solids mass would provide sufficient strength to undercut an artificial sill pillar constructed of that pastefill (subject to reasonable pillar thickness and span). For additional safety, P&E has used binder contents of 5-7% and a maximum span of 7.5 m where undercutting is necessary. P&E has assumed all binder will be Portland (Grey) Cement.

16.2.4 Development

Waste development in the Back Forty underground is comprised of lateral and vertical development of varying sizes. All lateral development is fully mechanized, using electro-hydraulic jumbos and bolters for drilling and support operations. Vertical development is semi-mechanized, since Alimak drilling, loading, support and scaling is manual, and since ventilation drop raises will generally be manually supported to allow the installation of escapeways.

Mineralized development is included with production tonnages, as CF operations are analogous to development operations. For LH mining areas, mineralized sill development costs are included in the mining cost per tonne. Like waste development, this development is fully mechanized. For smaller openings, specialized low-profile equipment has been included in the fleet for this purpose.

16.2.4.1 Lateral Development

Table 16.13 shows lateral development by type.

TABLE 16.13
TOTAL LOM LATERAL DEVELOPMENT

Development Type	Development Profile*	Metres
Attack Ramp (Slashed)	4.5 m W x 5.0 m H	6,420
Attack Ramp (Driven)	4.5 m W x 5.0 m H	2,232
Level Access	5.0 m W x 5.0 m H	3,193
Sills and Crosscuts	4.5 m W x 5.0 m H	1,094
Main Ramp	5.0 m W x 5.0 m H	7,016
Electrical	5.0 m W x 5.0 m H	230
Maintenance Shop	6.0 m W x 7.0 m H	205
Sump	4.5 m W x 5.0 m H	235
Magazine	5.0 m W x 5.0 m H	42
Pump Station	6.0 m W x 7.0 m H	104
Re-muck Bay	5.0 m W x 7.0 m H	1,128
Truck Turn-Around	5.0 m W x 5.0 m H	40
Refuge Station	5.0 m W x 5.0 m H	92
Ventilation Access (Lateral)	4.5 m W x 5.0 m H	774
Total Lateral Metres		**22,805**

***** *W = width, H = height.*

16.2.4.2 Vertical Development

Table 16.14 shows vertical development by type.

TABLE 16.14 TOTAL LOM VERTICAL DEVELOPMENT		
Development Type	**Development Profile***	**Metres**
Vent Drop Raise (Vertical)	4.0 m W x 4.0 m L	765
Alimak (Vertical)	4.0 m W x 4.0 m L	404
Total Lateral Metres		**1,169**

* *W = width, L = length.*

16.2.5 Mining Methods

Production of mineralized material will be achieved using either Cut-and-Fill ("CF") or Longhole Open Stoping ("LH") methods.

16.2.5.1 Cut and Fill Mining

The Back Forty Deposit contains many mining areas with low-dipping and/or high-value mineralized material. This material is generally not amenable to LH mining for several reasons:

1. For low-dipping material (dip less than 55°) it is difficult to ensure material moves down the footwall of a stope, which often results in increased mining losses and/or increased dilution. This exacerbates the impact of (2) below.

2. High-value material is more significantly reduced in value through dilution than lower value material, meaning more selective methods are required to maximize the value of extracted material.

3. Material of different MET types are often found adjacent and/or contiguous to each other in the Deposit. Significant value loss is likely to occur if materials of one MET type are processed as a different MET type. The ability to be selective and segregate materials of different MET types is crucial to the maximizing the value of the Deposit. Visual inspection on a daily basis of CF areas will allow much greater control of this issue.

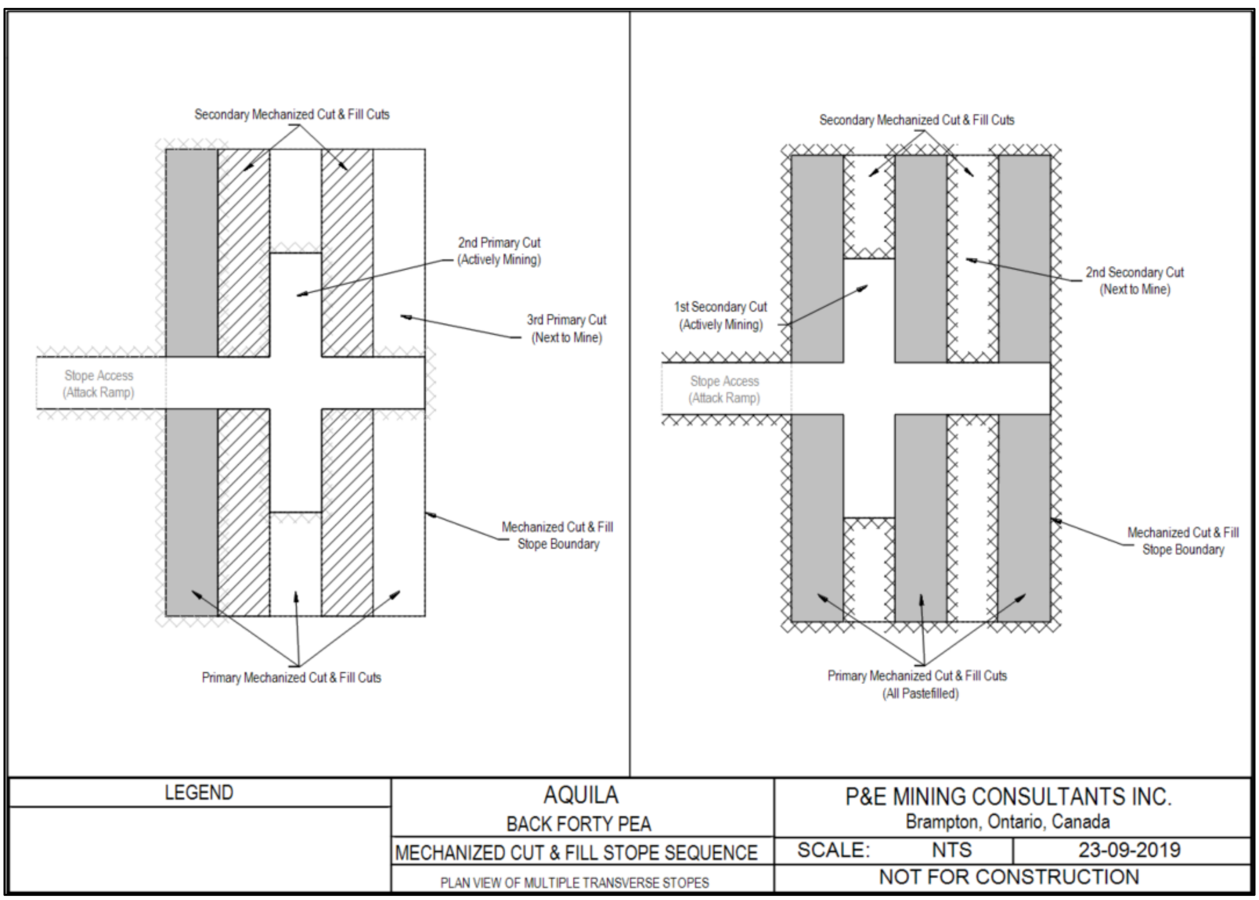
Due to the lateral extents of many of the mineralized zones in the Back Forty Deposit, standard vertically progressing CF methods are not optimal for extracting the mineralized material due to the cost of waste development for attack ramps. Lateral progression of cuts (drifts) on each sub-level is necessary to efficiently extract the mineralized material while limiting expenditures on waste development and minimizing dilution and mining losses. As such, mechanized CF mining was selected as the primary mining method for the Deposit. The mining method is similar to Drift and Fill mining on a larger scale. Approximately 63% of the Deposit will be mined with CF methods.

CF mining for the Back Forty Deposit uses one of four different opening sizes to drive a series of drifts through the mineralized zone and extract the mineralized material (see Table 16.15). These drifts are generally oriented along strike of the mineralized material where possible to maximize extraction from a single face but can be oriented at any angle to the mineralized material as necessary.

Table 16.15 **CF Mining Opening Dimensions**		
Sub-Type	**Opening Width**	**Opening Height**
CF 1	7.5 m	5.0 m
CF2	5.0 m	5.0 m
CF 3	4.0 m	2.5 m
CF 4	5.0 m	2.5 m

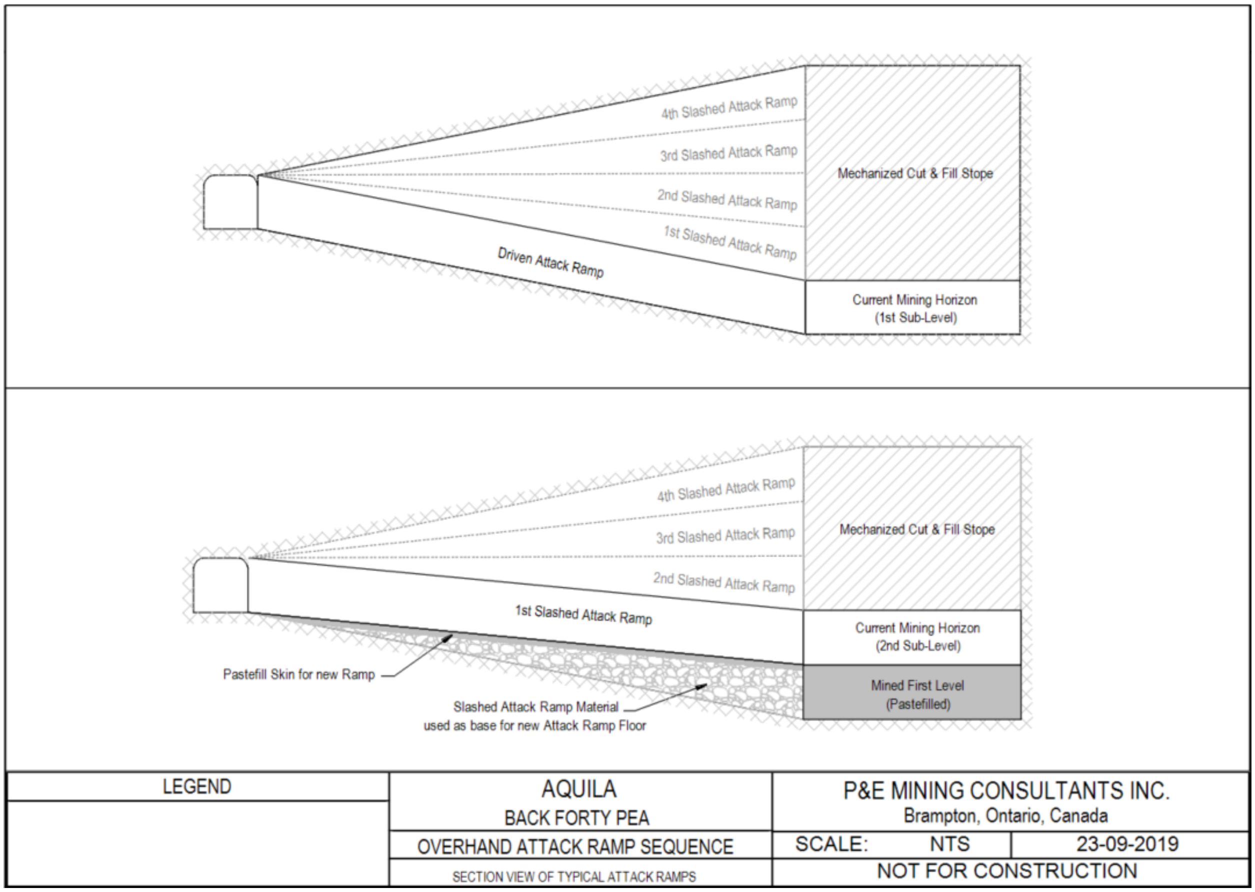
An access drive for each cut will be driven on a transverse perpendicular heading through the mineralized material to allow access to the mining drifts. A stope will be divided into primary and secondary cuts on each sub-level (sub-levels are nominally 5 m high, but can vary as necessary), and primary drifts will be mined out, leaving secondary drifts as rib pillars. This is shown in Figure 16.13.

Figure 16.13 Mechanized CF Mining Sequence Plan View



Once drifts are mined out, they will be backfilled with PF. Once the PF in adjacent drifts has cured, the secondary cuts may be mined out, using the fill as a rib pillar. When all drifts on a sub-level have been mined out, the transverse access will be backfilled with PF, then the attack ramp (developed in waste to access the stope) will be blasted into the void of the previously excavated attack ramp, and the process will be repeated at the next sub-level. Nominally, mining levels are 25 m apart (five 5 m sub-levels), varying ±5 m depending on stope geometries. A small amount of PF may be used to raise the floor of the attack ramp to the correct elevation and create a working surface. Figure 16.14 shows this progression.

FIGURE 16.14 MECHANIZED CF MINING SEQUENCE SECTION VIEW



For narrow stopes, no lateral progression of drifts is necessary, and therefore no access drive in mineralized material is needed. For this case, mining is identical to narrow-vein CF mining with PF. All CF mining at the Back Forty Deposit progresses in an overhand fashion. Where it is necessary to mine under fill, artificial sill pillars will be left in place during the extraction of the mining level above. These sill pillars will comprise two sub-levels (10 m vertical extent) of 5-7% cement by mass PF.

A significant advantage of CF mining at Back Forty is that equipment used for waste development, and for production CF mining, is largely interchangeable (as are the skill sets of the personnel operating the equipment). As such, the flexibility introduced to the mining schedule (ability to ramp production rates up in times of low development requirements, ability to move personnel from development to production and back, etc.) is of considerable impact and benefit to the mine.

16.2.5.2 Longhole Mining

The Back Forty Deposit contains several mining areas with the necessary geometry to allow for LH mining (dip of 55° or more; continuous mineralization of one major MET type; no major waste inclusions; sufficient thickness, strike and vertical extent to allow for ring drilling; sufficient value to support FW waste development in the case of transverse stoping). LH mining is generally less expensive on a unit basis than CF mining methods, but results in higher dilution and greater mining loss than the more selective CF method, and requires a different skill set from CF mining or development. For areas with large quantities of lower-value mineralized material, LH mining is an effective method of extraction that avoids incurring a significant reduction in the value of the material. Transverse LH mining has the additional benefit of allowing mining to continue in nearby stopes while completed stopes are filled, whereas CF mining incurs a delay when attack ramps are re-developed. In general, LH mining at the Back Forty Deposit has been scheduled near the end of the underground mine life. Approximately 37% of the mineralized material tonnage will be mined by LH methods.

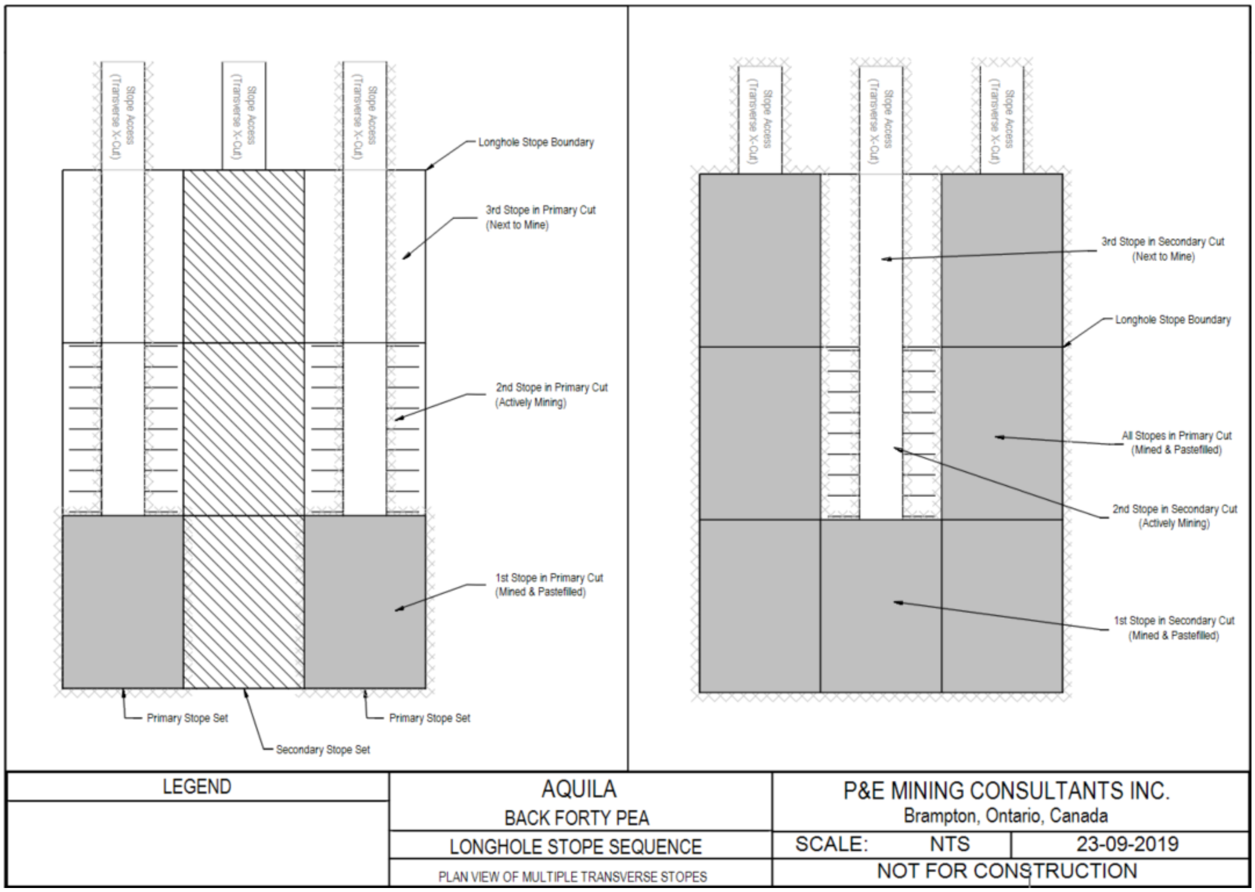
LH mining for the Back Forty Deposit uses a nominal 25 m floor-to-floor level spacing, with 5 m drift heights and a maximum width of 14 m, which is within the recommendations laid out in Section 16.2.3. The level spacing is reduced slightly from the 30 m maximum to align better with the spacing of the more common CF mining areas, and to reduce possible overbreak of the stopes by reducing their hydraulic radius. In specific areas, the level spacing may vary by ±5 m, depending on the geometry of the target mining area. In general, stopes are transverse-oriented and of the maximum width; in a small minority of areas they may be longitudinal, with a maximum width of 7.5 m.

Transverse LH mining will take place with mining commencing on even-numbered crosscuts. Once stopes are mined out, they will be filled with PF (strength to be determined on a location-by-location basis) and allowed to cure while other stopes are mined. Once curing is completed, adjacent stopes can begin mining, with the cured PF acting as a rib pillar for laterally adjacent stopes, or as a floor for stopes above. Once all primary crosscuts are exhausted, secondary crosscuts will be mined using the same methods and sequences.

Longitudinal LH mining, where applicable, will take place on a simple retreat method along the access crosscut. During curing of PF, drilling operations may take place in the adjacent stope.

Figure 16.15 shows two stages of a transverse mining sequence, the second of which is identical to a longitudinal retreat mining sequence once the primary stopes are mined out.

FIGURE 16.15 LONGHOLE MINING SEQUENCE PLAN VIEW



16.2.6 Backfill

Pastefill ("PF") will be the primary backfill method for the underground portion of the Back Forty Deposit. Prior to filling a stope, a bulkhead will be installed to retain the PF in the desired area. PF will be generated in a surface plant located between the process plant and the open pit and will be pumped along insulated lines to surface boreholes, where it will flow under gravity through a distribution system to its final deposition location in a stope. Four surface boreholes will be drilled, one providing PF to the Pinwheel Zone, and the other to provide PF to the Main and Tuff Zones, with a redundant borehole for each Zone in addition to the primary hole. Main PF distribution lines use Schedule 80 pipe, up to the start of the attack ramp for the stope, beyond which Schedule 40 pipe is used. Once filling of an area is complete, water will be used to flush the line, and will flow back to the nearest sump prior to entering the primary dewatering system.

16.2.6.1 High-Strength Pastefill

High-Strength Pastefill ("HS PF") will be comprised of 5%-7% binder by solids mass. This material will be used for areas where future mining will undercut the PF to create an artificial sill pillar. These pillars are a minimum of two sub-levels (10 m) thick, creating a pillar of greater than 1:1 aspect for increased stability.

HS PF makes up 25% of backfill by mass in the underground portion of the Back Forty Deposit.

16.2.6.2 Low-Strength Pastefill

Low-Strength Pastefill ("LS PF") will be comprised of 3% binder by solids mass. This material will be used for areas where the PF will only be exposed laterally or used as a floor for overhand mining.

LS PF makes up 75% of backfill by mass in the underground portion of the Back Forty Deposit.

16.2.6.3 Other Fill

Where possible, in LH mining areas, waste material from nearby development can be dumped into open secondary stopes, rather than being trucked to surface. Due to scheduling limitations, this form of filling will likely comprise a very small portion of total fill tonnes, as LH mining operations occur towards the end of the schedule when most waste development is already complete. As such, P&E has estimated costs assuming that all waste is transported to surface and all underground voids requiring filling are filled with PF.

It is also possible that, near the end of mine life, tailings with minimal binder content could be pumped into underground voids (where no future mining will take place) via the PF deposition system. Again, P&E has assumed that all UG stope voids requiring filling are filled with PF with a binder content of at least 3% by solids mass.

16.2.7 Productivity Estimates

The Back Forty Deposit uses two primary mining methods: Cut and Fill ("CF"), and Longhole ("LH"). Both methods have subsets (CF mining is divided into four types by opening size, LH mining is divided into longitudinal and transverse mining). For all calculations, it is assumed that there are 19.5 useable hours in a day (divided into two shifts), with the remaining 4.5 daily hours allotted to line-up meetings, blast clearing, travel to/from workplaces, etc.

To estimate productivities, several different sets of first-principle models were developed to determine:

- Face productivity for the unit operations:
  - Each CF opening size.
  - Each LH stope size.
  - Each lateral development size.

- Operations group total productivity:
  - CF production.
  - LH production.
  - Lateral development.

- Equipment utilization and availability.
- Operational considerations (traffic, parts delivery, stand-downs, etc.).

Face productivities were calculated using machine productivities provided by Sandvik and cross-checked against other operations using similar mining methods.

Once productivities were calculated, weighted averages of tonnes/metres of each type were calculated, and average productivities of operations groups were calculated as follows:

- CF mining (tonne-weighted average of the 4 sub-methods): 1,600 tpd.
- LH mining (tonne-weighted average of the 2 sub-methods): 1,100 tpd.

CF mining and lateral development were limited by the productivity of rock bolters. CF mining advances at a maximum rate of 110 m per month per face (except CF Type 4 which is limited to 70 m per month due to opening geometries).

LH mining was limited by the productivity of drills (the lack of mobility of LH drills versus CF/development equipment is a significant factor in their productivity).

Lateral development headings are normally scheduled at a maximum of 110 m per month, except for high-priority ramp development, which is scheduled at 125 m per month. The additional productivity can be achieved by “batching” services installation (skipping services installation until no further rounds can be cycled without extending services, then halting development of the heading until services are caught up). Vertical development rates for Alimak development are scheduled independently of the other operational groups, as this development will be contracted out. Vertical development via LH methods is included in the LH group productivity calculation.

To determine the productivity of a stope, each stope was analyzed for its geometry, quantity and position of accesses, and its mining method. Cycle-time analyses were performed to determine the productivity of each access point into the stope, which varied from 100-420 tpd per access. During scheduling, the number of active accesses was then multiplied by the productivity per access to get an overall stope productivity.

Overall mine productivity was calculated by determining the minimum number of groups required to meet minimum production requirements of 2,300 tpd and maintain development advances sufficient to continue that production uninterrupted for the life of mine. It was determined that three groups would be sufficient during initial development and steady-state production. After Q4 of Year 8, development is sufficiently advanced that the resources of the dedicated development group could be reduced to half-time operations before eventually being allocated to LH mining.

16.2.8 Personnel Estimates

The Back Forty Underground uses a 14-on/14-off shift schedule, with four rosters (two on site, two offsite). Personnel work an entire 14-day period on either night shift, or day shift, alternating with each 28-day cycle. Shifts are nominally 12 hours in length. Table 16.16 shows the typical complement of mining personnel on a roster.

TABLE 16.16 TYPICAL HOURLY MINING PERSONNEL ROSTER							
Drillers	**Blasters**	**Ground Support**	**Services**	**Loading**	**Trucking**	**Construction and Backfill**	**Auxiliary**
6	6	8	8	4	5	8	8

Technical, Management, and Support Services staff work standard weekly dayshifts, with minor exceptions where overnight coverage is required.

Table 16.17 shows the average number of personnel on payroll for the underground mine.

TABLE 16.17 UNDERGROUND SITE PERSONNEL BY DEPARTMENT BY YEAR						
Year	**Total Personnel**	**Hourly Mine**	**Other Mine**	**Maintenance**	**Geology**	**Engineering**
Year 5	198	76	60	42	11	9
Year 6	273	144	60	42	13	14
Year 7	285	156	60	42	13	14
Year 8	273	144	60	42	13	14
Year 9	273	144	60	42	13	14
Year 10	261	132	60	42	13	14
Year 11	218	96	60	42	11	9

16.2.9 Mining Schedule

The sections below show the breakdown of production and development by period and type.

Figures 16.16 to 16.22 show the progression of mining at the end of each year from Year 5 to Year 11. Green stoping areas are active during the year, blue stoping areas are mined out and filled, grey stoping areas are developed and partially produced, but not active during the year.

FIGURE 16.16 UG MINE AT END OF YEAR 5, 3-D SCHEMATIC

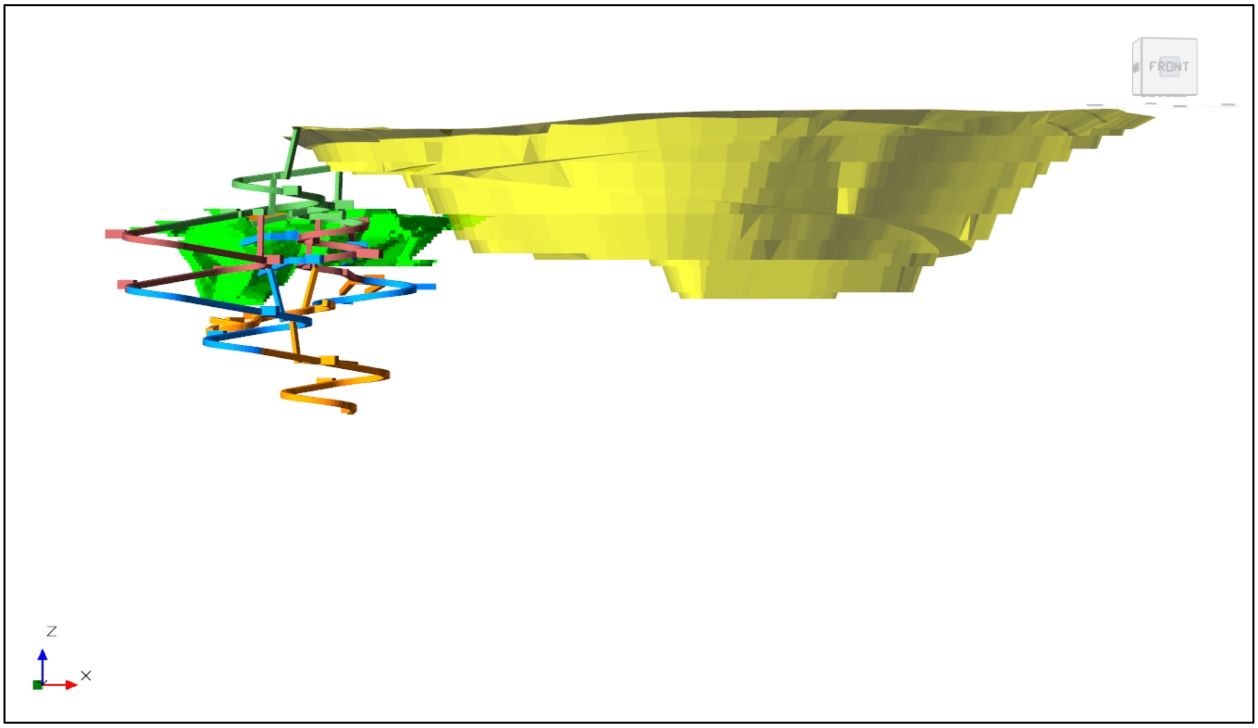


FIGURE 16.17 UG MINE AT END OF YEAR 6, 3-D SCHEMATIC

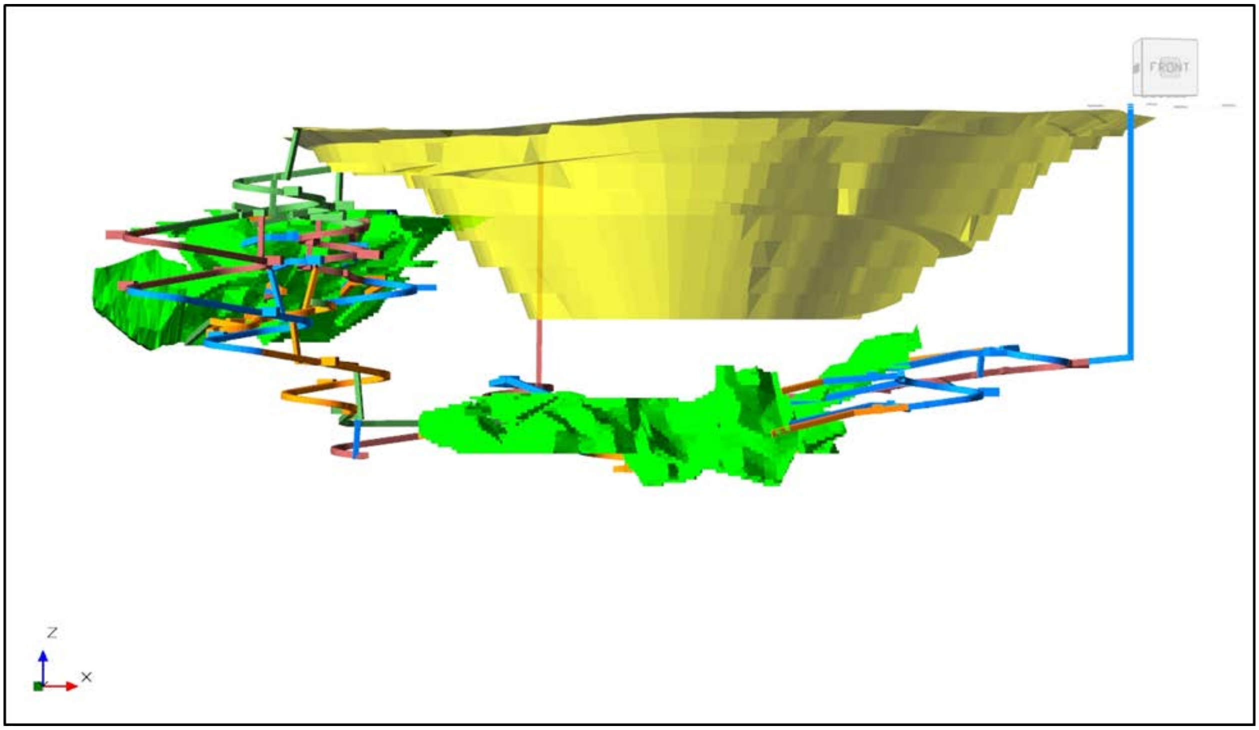


FIGURE 16.18 UG MINE AT END OF YEAR 7, 3-D SCHEMATIC

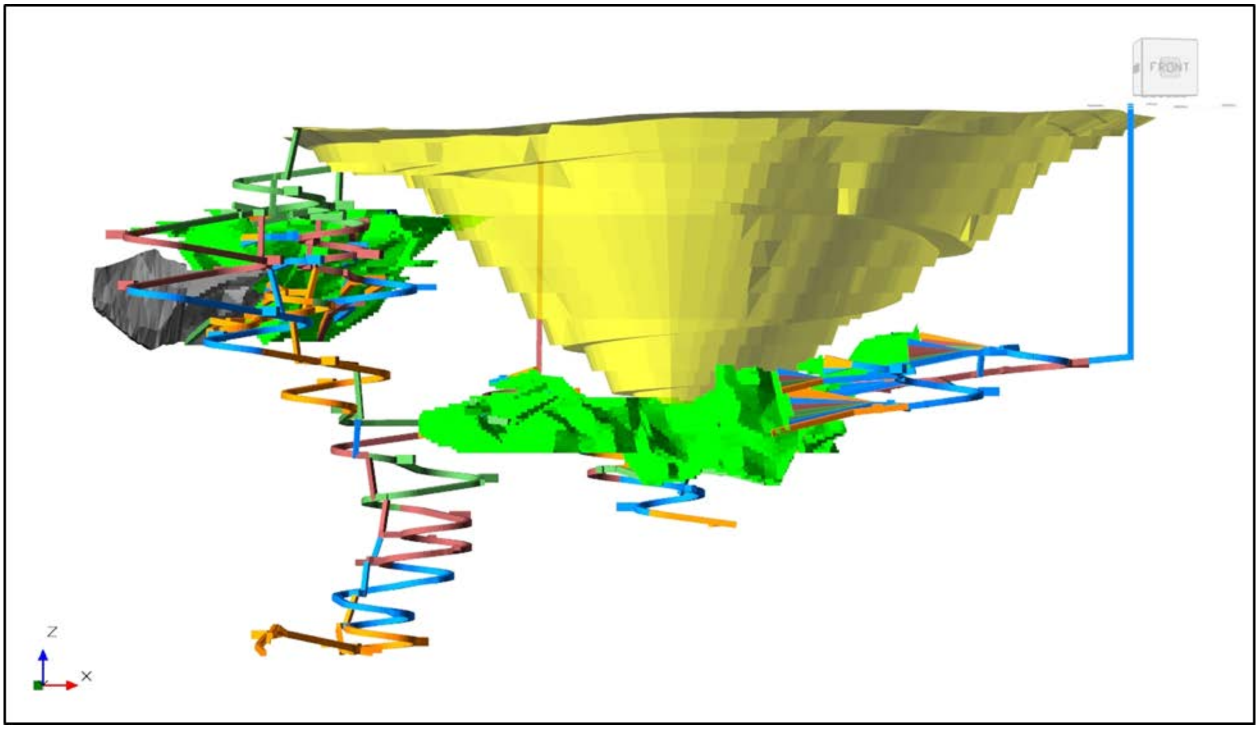


FIGURE 16.19 UG MINE AT END OF YEAR 8, 3-D SCHEMATIC

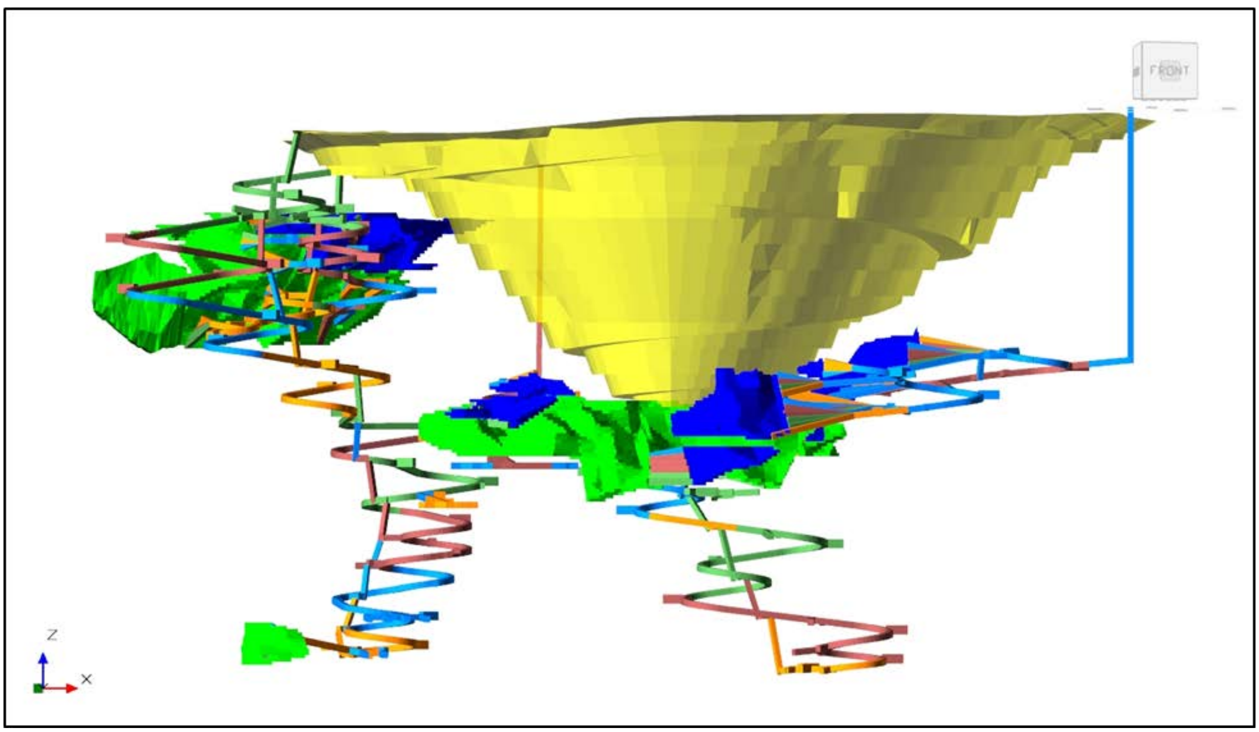


FIGURE 16.20 UG MINE AT END OF YEAR 9, 3-D SCHEMATIC

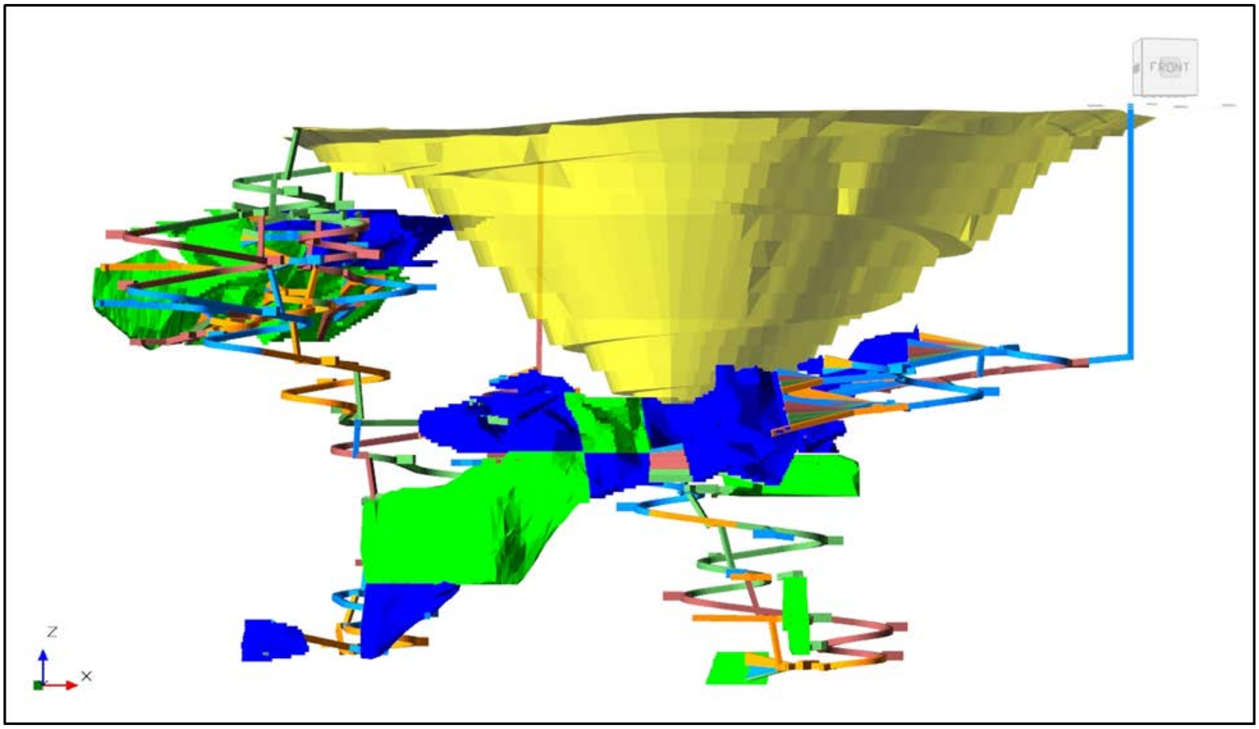


FIGURE 16.21 UG MINE AT END OF YEAR 10, 3-D SCHEMATIC

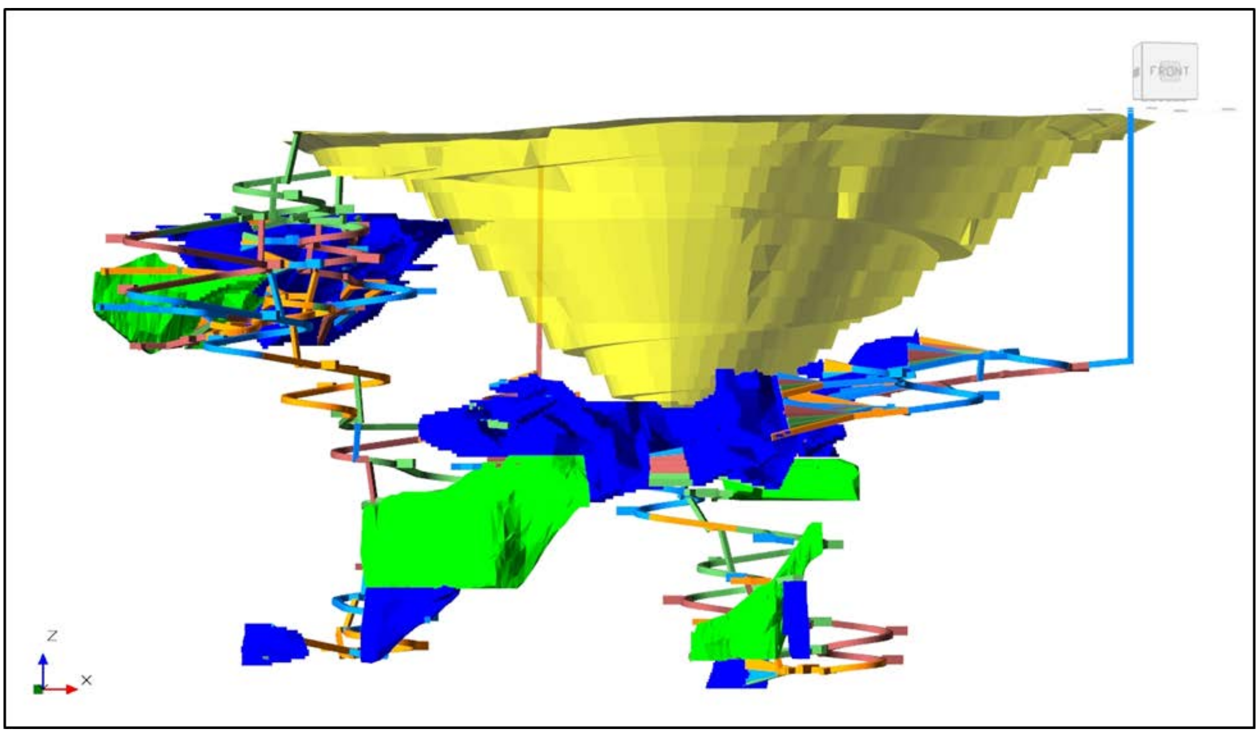
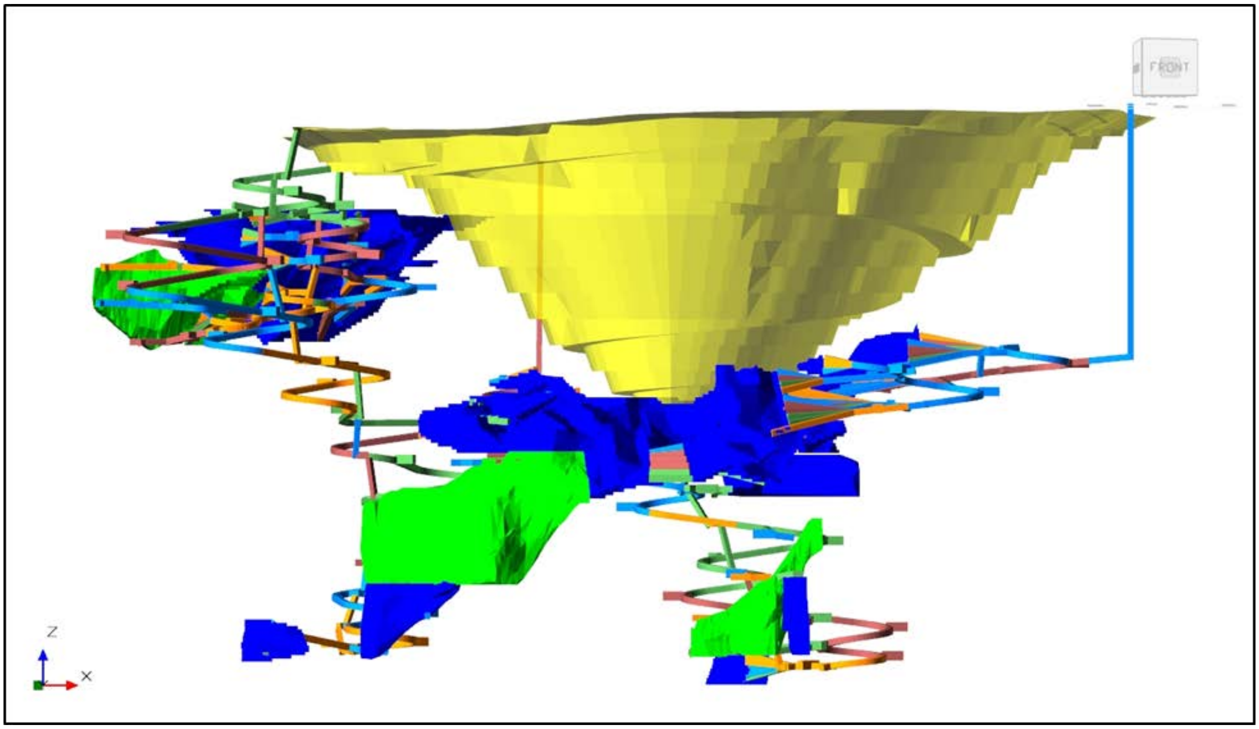


FIGURE 16.22 UG MINE AT END OF YEAR 11, 3-D SCHEMATIC



16.2.9.1 Development

Table 16.18 shows the metres of lateral development in the Back Forty underground Deposit by year. Note that there is no development in Year 11.

TABLE 16.18
LATERAL DEVELOPMENT BY TYPE BY YEAR (METRES)

Development Type	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Attack Ramp (Slashed)	34	664	2,673	2,250	799	-	**6,420**
Attack Ramp (Driven)	316	1,140	576	141	60	-	**2,232**
Level Access	343	722	430	640	852	206	**3,193**
Sills and Crosscuts	-	-	-	347	532	215	**1,094**
Main Ramp	2,345	1,978	1,803	842	49	-	**7,016**
Electrical	59	88	44	29	10	-	**230**
Maintenance Shop	-	205	-	-	-	-	**205**
Sump	36	77	42	45	30	5	**235**
Magazine	-	42	-	-	-	-	**42**
Pump Station	40	36	16	12	-	-	**104**
Re-muck Bay	296	276	192	204	144	16	**1,128**
Truck Turn-Around	-	-	12	16	12	-	**40**
Refuge Station	20	52	20	-	-	-	**92**
Ventilation Access (Lateral)	287	244	191	51	-	-	**774**
Total Lateral Metres	**3,775**	**5,524**	**6,000**	**4,576**	**2,487**	**442**	**22,805**

Table 16.19 shows the vertical development in the Back Forty underground Deposit by year.

TABLE 16.19
VERTICAL DEVELOPMENT BY TYPE BY YEAR (METRES)

Development Type	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Vent Drop Raise (Vertical)	297	104	212	153	-	-	**765**
Alimak (Vertical)	-	404	-	-	-	-	**404**
Total Vertical Metres	**297**	**508**	**212**	**153**	**-**	**-**	**1,169**

16.2.9.2 Production

Table 16.20 shows the production tonnes from the Back Forty underground Deposit by year and mining method. Units are in thousands of tonnes.

TABLE 16.20
PRODUCTION BY MINING TYPE BY YEAR (KT)

Type	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
LH	-	-	-	-	438	968	732	**2,138**
CF Type 1	-	98	503	520	268	-	-	**1,389**
CF Type 2	119	551	558	536	232	-	-	**1,996**
CF Type 3	1	18	43	47	13	-	-	**122**
CF Type 4	1	16	22	24	8	-	-	**72**
Total	**122**	**683**	**1,126**	**1,126**	**959**	**968**	**732**	**5,717**

16.2.9.3 Mining Within 50 m of Pit Operations

As previously noted, underground operations occur concurrently with open pit backfilling operations. The underground has been scheduled around pit backfilling operations such that there is always at least a 50 m distance to open pit backfill elevations. Due to the high number of available faces, this offset distance has no impact to the production rate or total tonnes recovered from the underground portion of the Deposit.

16.2.10 Mine Services

Mine services include ventilation, electrical, communications, dewatering and compressed air.

16.2.10.1 Ventilation

The primary ventilation system for the Back Forty underground mine is comprised of four 4.0 m x 4.0 m Alimak raises and one 5.0 m W x 5.0 m H ramp portal. Three raises (Pinwheel, Main, East) will be provided with primary ventilation fans and function as Fresh Air Raises ("FARs"), while the fourth (Central) will be a Return Air Raise ("RAR"), and the Portal will function as an exhaust route. Primary fans for the FARs will be 2.13 m in diameter (similar to Hurley 84-43-900 models) and powered by either 225 kW or 150 kW motors, depending on location (the Pinwheel Zone, due to its shallower depth, requires less power to achieve its required flows). Variable Frequency Drives ("VFDs") will be installed with each primary fan. For noise mitigation, these fans will be installed in bulkheads underground, while surface infrastructure will be comprised only of plenum buildings, propane-fired mine air heaters, and their associated controls and supports. For redundancy, one or more 1.22 m diameter, 112 kW, auxiliary fans will be installed in each FAR fan bulkhead (one in the Pinwheel Zone, two each on Main and East). In the event of a major primary fan failure, these fans can be activated to maintain the mine airflow during required repairs.

The Back Forty Deposit is expected to require a maximum of 313 m^3/s of fresh air during regular operations to maintain sufficient airflows in all operational areas. Fresh air will be forced to the deeper areas of the mine and allowed to exhaust upwards to surface. This arrangement ensures that fresh air supply to the underground is isolated from the ramp in the event of a machine fire, providing a safe egress for personnel and entrance for mine rescue teams if necessary. For dust control purposes, maximum airspeed in any lateral drift is roughly 25 km/hr.

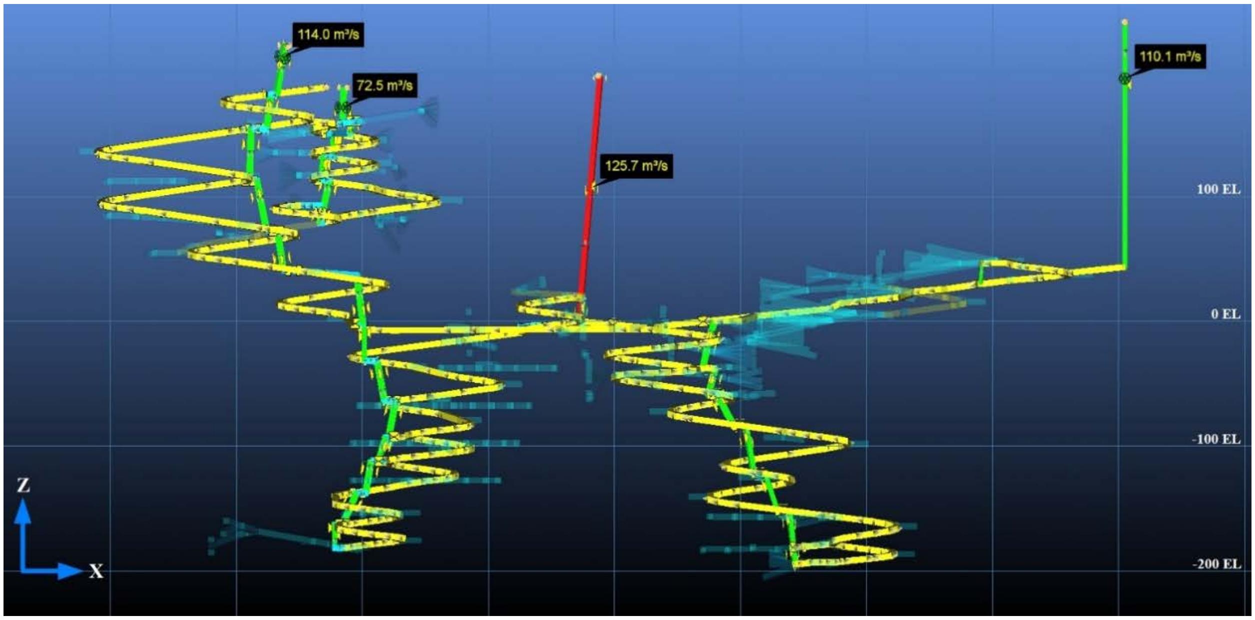
Internal raises will be driven through a combination of Alimak (for raises greater than 40 m long) or drop raise methods (raises less than 40 m long), with all raises being 4.0 m W x 4.0 m L (for Alimak raises this will be accomplished by slashing the raise after driving a pilot). Drop raising is the dominant method used to drive raises.

Mining levels will be ventilated via auxiliary ventilation drawing from the ramp and pushing fresh air to the face via auxiliary ducting. Due to the geometry of the Deposit, some auxiliary ventilation runs will be quite long (greater than 600 m) and will require specialized rigid or semi-rigid ducting (similar to G+ Speed Air ducting) to facilitate mining operations at the furthest extents. Ducting from the ramp to the mining area will be 1.22 m diameter, beyond which it will be one of 1.22 m, 1.07 m or 0.91 m diameter, depending on the opening size of the mining method being used.

Auxiliary fans will generally be the same diameter as the ducting and powered by 75 kW or 112 kW motors. Where necessary, booster fans will be installed in rigid or semi-rigid ducting to increase the flow rate on long auxiliary runs.

A snapshot of the underground ventilation system Ventsim™ model can be seen in Figure 16.23.

FIGURE 16.23 UNDERGROUND VENTSIM VENTILATION MODEL, LOOKING NORTH



16.2.10.2 Electrical

The electrical system for the Back Forty underground mine will use 15.0 kV main supply lines connected to level load centres that will transform the voltage to 1,000 V or 600 V depending on applications. These voltages have been selected to minimize the line losses during transmission.

Main lines will be run in the ramp for the Pinwheel Zone and initial development towards the Main Zone. Once the Main Zone is under development, a supply line will be run down a borehole or vent raise to reduce transmission distances.

Load centres are expected to be 750 kVA in size, sufficient for approximately two mining areas to be run from one centre, or one mining area and one pump station. The load centres are modular in nature to allow relocation as necessary. A 1.5 MVA substation will be installed near the portal for initial development and to power the Pinwheel Zone upper areas during initial mining. It is expected that 18 load centres will be required at the maximum extents of the underground mine.

Table 16.21 shows the average and peak power draws by year.

TABLE 16.21 ELECTRICAL LOAD DETAILS							
Year	**Ventilation (kW)**	**Dewatering (kW)**	**Compressors (kW)**	**Fleet (kW)**	**Line Losses (kW)**	**Average Load (kW)**	**Peak Load* (kW)**
5	531	13	167	917	55	1,684	3,885
6	799	152	223	1,618	96	2,889	5,090
7	1,006	279	223	1,705	104	3,318	5,518
8	899	324	167	1,705	103	3,199	5,400
9	658	371	167	1,705	102	3,004	5,205
10	440	386	195	1,705	101	2,827	5,028
11	354	380	112	1,426	85	2,356	4,557

Note: *This is the maximum connected load if everything in the mine were turned on at once.*

16.2.10.3 Communications and Controls

Communications infrastructure will include a fibre-optic backbone for a wireless internet system. This system will allow real-time tracking and communication of assets underground. Radio Frequency Identification ("RFID") tracking of equipment will also be used. Additional leaky-feeder radio systems will be installed for general communication and backup blasting control. Telephone systems to important locations will use Voice-Over-Internet-Protocol ("VOIP")-type phones running on the internet system.

A traffic management system will be installed to minimize delays in the busiest parts of the ramp (near the surface portal, the Pinwheel/Main intersection, and the main connector drive).

Cap lamps will be equipped with RFID beacons and the Personal Emergency Device ("PED") system to allow personnel tracking and limited one-way communications in event of an emergency.

Due to the complex nature of the Deposit, it is envisioned that detailed data-gathering and information tracking will be necessary to ensure optimal underground operations. Software such as Centric (or similar) will be used to monitor equipment for maintenance and dispatch reasons, monitor and relay geological mineralized material/waste calls and MET designations, transfer designs to operators and machinery, and perform administrative- or safety-related tasks. Real-time reports from team leaders, engineers and geologists will also be used to maximize the agility of the operations process and maximize production and profitability from the underground mine.

16.2.10.4 Dewatering

All active mining faces will be dewatered using compressed air pumps (similar to Wilden PX series pumps) pumping to 100 mm pipes to level sump cut-outs.

The dewatering system for the Back Forty underground uses five pump stations to pump water from underground seepage and operations to the surface. Water from faces is collected in small level sumps and either allowed to cascade via gravity to a pump station or pumped up to a transfer line which then flows via gravity to a pump station. Sumps are expected to use compressed air pumps.

There are five major pump stations in the Back Forty underground: one in the Pinwheel Zone and four in the Main Zone. The system is designed around a normal pumping requirement of 13 L/s on a 33% duty cycle. Maximum output of the system is 109 L/s, with 27 L/s coming from the Pinwheel Zone and 82 L/s coming from the Main Zone. Main transfer lines are 200 mm diameter. In each pump station, a settling sump will be used to segregate clean water from dirty water for re-use in mining operations.

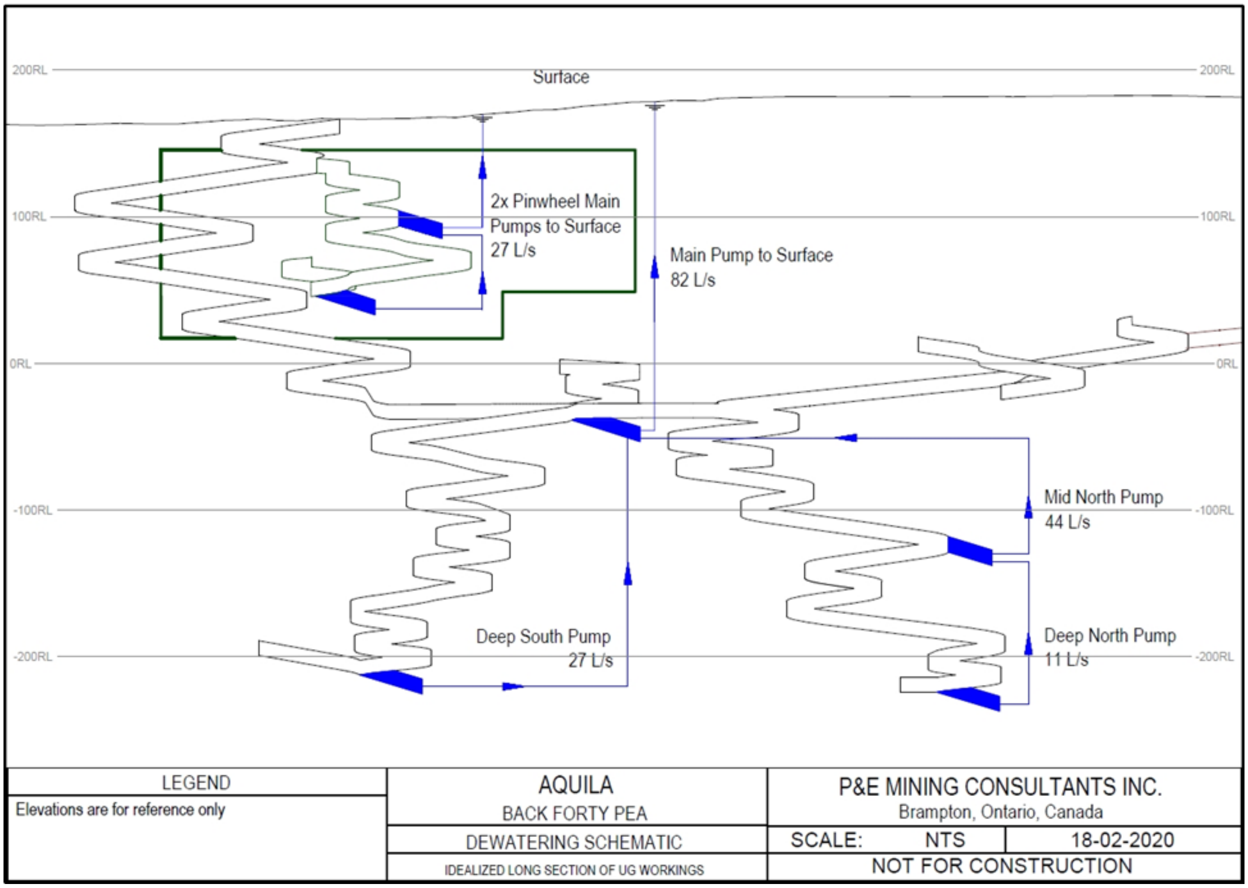
Due to the proximity of the Menominee River to the underground, and the fractures expected in the Pinwheel Crown Pillar, the Pinwheel Zone pump station has been designed to function as a stand-alone station and is not connected to the Main Zone. In the event of a significant water inflow, the Pinwheel Zone can be isolated and used as an overflow area. This pump station is designed with submersible pumps, with the electrical load centres located higher in the mine, which will allow the station to keep functioning in the event of flooding.

While the Main Zone is not as near to the Menominee River as the Pinwheel Zone, the portal ramp does go through the Main Pillar on the river side of the pit, and it is possible that there will be openings from the pit into the underground. A breach in the cut-off wall on the NW side of the pit would also pose a significant flooding risk to the Main Zone. Pump stations are located such that sudden rapid inflow will be dealt with as soon as possible, preventing flooding from reaching deeper into the mine. The smallest pump station (Deep North) is sized to handle an entire mine's worth of expected inflows, with all other stations significantly oversized from that to prevent any flooding from progressing that far. Additionally, the mine is designed with floodable areas to allow water control in the event of a catastrophic system failure. Table 16.22 and Figure 16.24 give details about the system. All pump stations except the Pinwheel Zone and Deep North pump stations are equipped with a small submersible pump (37 kW or less) and a large centrifugal pump. The Pinwheel Zone pump station is equipped with large submersible pumps only.

TABLE 16.22
PUMP STATION DETAILS

Zone	Station	Flow Rate (L/s)	Total Installed Power (kW)	Main Pump Type (Expected)
Pinwheel	Pinwheel Main	27	149	Eliminator SBGT3S100-4T4-S
Main	Main Pump Stn	82	529	Flyght 11WAHC
Main	Deep South	27	138	Flyght 10RAHC
Main	Mid North	44	93	Flyght 9RCLO
Main	Deep North	11	149	Flyght 7CHC

FIGURE 16.24 UNDERGROUND DEWATERING SYSTEM SCHEMATIC



16.2.10.5 Compressed Air

While most UG equipment is electro-hydraulic in nature, a compressed air system has been included in mine services for the purposes of flexibility to allow for pneumatic door controls, face dewatering, jack-leg drilling during construction operations, and general operational flexibility. It is estimated that the maximum required flow will be 0.45 m^3/s (946 cfm), including a 25% loss to leakage. This will be handled by two 93 kW compressors (similar to Sullair TS20-125), with a third in reserve. The compressors will be installed with two on surface and one underground near the shop.

16.2.10.6 Refuges, Egress, Additional Underground Infrastructure

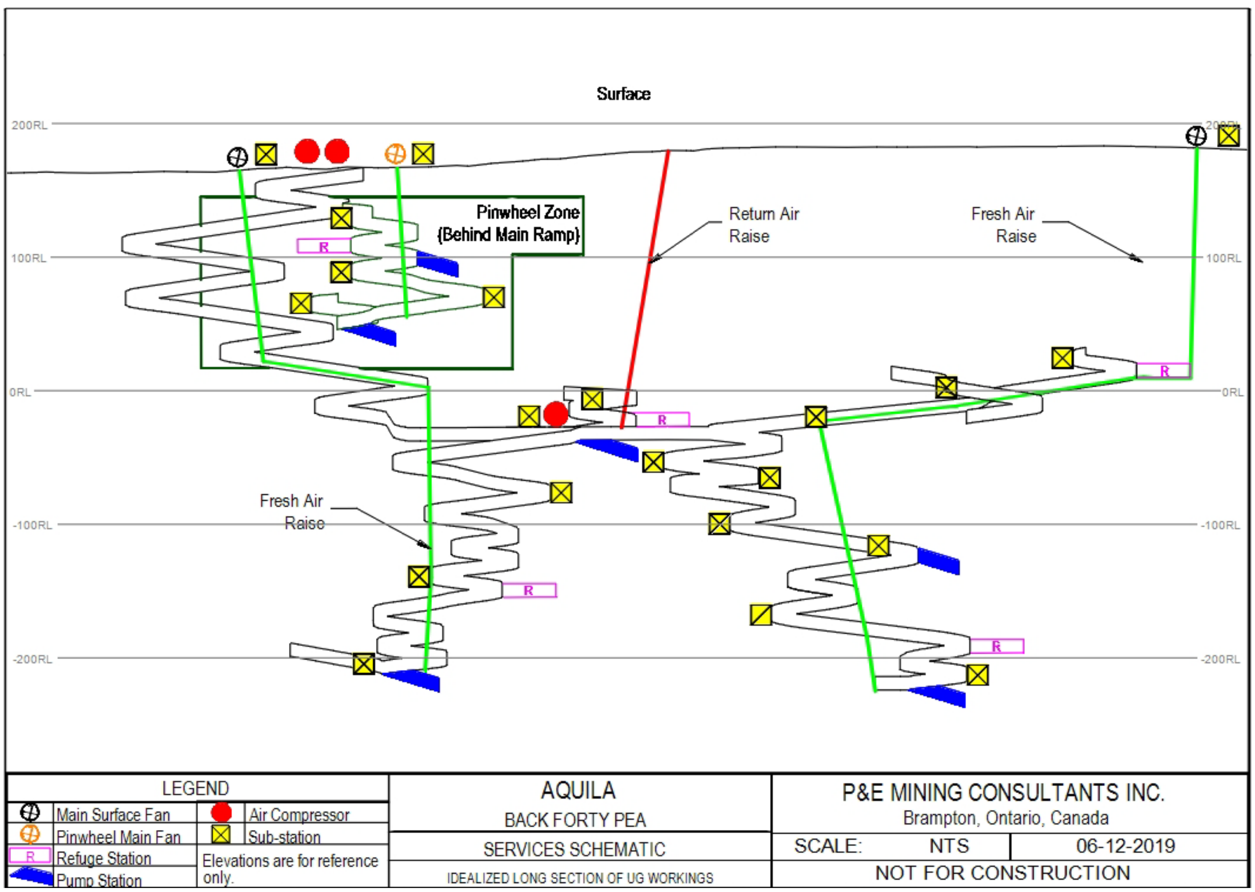
The underground mine will be fitted with five permanent refuge stations (see Figure 16.25. These stations will generally be used as lunch rooms, but will be fitted with the requirements to function as refuge stations (double airlock doors, first aid supplies, food and water, communications, etc.).

To facilitate emergency egress from the mine, each FAR will be equipped with a ladderway built to the specifications of regulatory requirements. Where the vertical extent of an escapeway exceeds 100 vertical metres, a mechanical hoist will be installed in compliance with MSHA requirements.

An underground maintenance shop will be constructed off the main connector drive near the bottom of the RAR. This shop will be equipped to perform all normal operational maintenance on all pieces of underground equipment, and will contain warehousing for parts, as well as an underground office. Major rebuilds will take place at surface facilities, on- or off-site. The location of the shop was selected for several reasons: it is roughly central to the mine, located just off the primary travel route for all equipment, and in the event of a fire in the shop, the RAR will exhaust any escaping smoke/fumes directly to surface, preventing them from entering the rest of the mine.

The mine will have underground explosives magazines located in the Pinwheel Zone during initial development and mining, and then off the main connector drift later in mine life. In each case, two magazines will be developed, one for initiating systems and one for bulk explosives.

FIGURE 16.25 UNDERGROUND SERVICES SCHEMATIC



16.2.11 Equipment Inter-Operability

Wherever possible, equipment has been selected to allow inter-operability between development and production mining. Lateral development fleets and CF mining fleets are entirely interchangeable, with the exception of CF Type 4, which has a lower opening height. Equipment for this size of opening is generally used as support equipment for ancillary jobs in the mine when not in use for production purposes.

16.2.11.1 Lateral Development Equipment

Lateral development uses a two-boom jumbo (similar to Sandvik DD421) for face drilling, a mechanized explosive loader (similar to a Getman A64) for loading, a mechanized bolter (similar to Sandvik DS421) for ground support, a 14 t-class LHD (similar to Sandvik LH514) for mucking, and 40 t trucks for haulage (similar to Sandvik TH540). Services will be installed using a scissor-deck or pipe handler (similar to Getman A64 platforms for interchangeability of parts and ease of operator training).

16.2.11.2 Vertical Development Equipment

Vertical development for longer raises (over 40 m) is done using Alimak methods by a contractor. For shorter raises, a longhole drill (similar to Sandvik DU311) will be used for drilling, with loading being performed by a mechanized loader similar to those used for lateral development. Support of the raises, where necessary, will be done by hand using jack-leg drills.

A V30-type reamer will be purchased to drill slots for the raises and, eventually, for longhole stopes. The suggested LH drill is compatible with this type of head.

16.2.11.3 Cut and Fill Mining Equipment

CF mining equipment is largely the same as Lateral Development equipment and, as such, units are interchangeable within the fleet. The exception is small scale CF mining (CF Type 4).

CF Type 4 mining will use specialized low-profile equipment to maximize productivity from the low back heights. Drilling will be done with a single-boom jumbo (similar to a Sandvik DL211L), loading will be done using ancillary equipment (forklift) to bring in a cassette-type pneumatic loader. Mucking will be done with a low-profile LHD (similar to a Sandvik LH209). Ground support will be installed using a low-profile bolter (similar to a Sandvik DS211L). This type of mining makes up roughly 9% of overall mining tonnes in the Deposit. The low-profile equipment, while planned for use on this specific operation, has the ability to reach the back heights of regular CF mining and waste development areas. When not required, it can be used to supplement those operations, or to perform construction or support tasks elsewhere in the mine.

16.2.11.4 Longhole Mining Equipment

LH mining equipment is largely the same as Lateral Development equipment, with the exception of the drill and the explosive loader, which are the same as the units used for Vertical Development.

16.2.11.5 Ancillary and Support Equipment

In addition to the equipment mentioned above, support equipment for the mine includes:

- Pickup trucks (similar to Toyota/Miller Landcruiser).
- Diamond drills (similar to Boart LM55).
- Modular tractor (similar to MineCat MC100 platforms).
- Service truck, fuel truck and lube truck (similar to Getman A64 platforms).
- Shotcreter and transmixer (similar to Getman A64 platform).
- Grader (similar to CAT 16M3).

As previously mentioned, certain modular platforms have been selected for their flexibility: the MineCat MC100 can be configured as a cable reeler, personnel carrier, forklift, explosive loader, scissor lift, shotcreter and more, as can the Getman A64. The Getman is a larger platform, and more suitable to the larger scale mining areas, whereas the MineCat is more suitable to smaller scale areas or general ancillary operations.

16.2.11.6 Diamond Drilling

Due to the varying geometry of the numerous mineralized lenses in the Back Forty underground mine, significant definition drilling will be necessary to predict the economic extents of stopes during the planning stage. As such, two small diamond drills (similar to a Boart Longyear LM55) have been included in the auxiliary equipment for the underground. These drills will be used to delineate the rough mineable shapes prior to driving stope-specific waste development.

CF mining allows for ongoing stope delineation by drilling and sampling test holes from inside the mining area, but this level of access does not exist with LH mining methods. As such, delineation drilling requirements will be higher later in mine life when production is at a maximum and the mining method in use is more weighted to LH mining.

It is expected that delineation drilling in mineralized zones will be in the form of a grid on 25 m centres. P&E has included 30% extra holes to further improve delineation in the more severely irregular mineralized zones in the Back Forty underground, resulting in an average grid spacing of approximately 17 m centre-to-centre.

16.2.11.7 Fleet Summary

Table 16.23 gives a summary of the primary underground production and development fleet, by year and type, for the Back Forty Deposit.

TABLE 16.23 PRIMARY FLEET BY YEAR AND EQUIPMENT TYPE							
Equipment Type	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**
2-Boom Jumbo	2	4	4	4	4	4	3
Low-Profile Jumbo	2	2	2	2	2	2	2
Explosives Loader	2	4	4	4	4	4	3
Bolter	2	4	4	4	4	4	3
Low-Profile Bolter	2	2	2	2	2	2	2
Standard Scissor Deck	3	5	5	5	5	5	4
Pipe Handler	1	1	1	1	1	1	1
14 t LHD	2	4	4	4	4	4	3
Low-Profile 9 t LHD	2	2	2	2	2	2	2
40 t Haul Truck	3	4	5	5	5	5	4
Longhole Production Drill	1	2	3	3	3	3	3
Alimak (Contractor)	0	1	1	0	0	0	0

Table 16.24 gives a summary of the auxiliary supporting equipment for the underground fleet, by year and type, for the Back Forty Deposit.

TABLE 16.24 AUXILIARY FLEET DETAILS AND QUANTITIES BY YEAR							
Equipment Type	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**
Pickup	6	10	11	10	10	10	8
Diamond Drill	1	2	2	2	2	2	2
Tractor/Manitou	2	4	4	4	4	4	3
Grader	2	2	2	2	2	2	2
Fuel Truck	1	1	1	1	1	1	1
Lube Truck	1	1	1	1	1	1	1
Shotcrete Machine	2	2	2	2	2	2	2
Transmixer	2	2	2	2	2	2	2
Service Truck	1	1	1	1	1	1	1
Bus	1	1	1	1	1	1	1

16.2.12 Material Handling

For the purposes of this PEA, it is assumed that all waste and mineralized material is taken to surface for processing or stockpiling, and all PF is generated from process plant tailings. Where possible, underground development waste will be disposed of in open longhole stopes instead of being trucked to surface, however this is expected to be a minor percentage of total tonnes.

16.2.12.1 Mineralized Material Handling

Mineralized material is loaded from the face to either a re-muck bay or directly to a truck. Due to the geometry of the Back Forty Deposit, orepasses are generally inefficient and non-economic, so direct loading is the standard method of filling trucks. Backs will be elevated at the intersection of the re-muck bay and the access drift to allow truck loading at grade.

Mineralized material is segregated underground by MET type and is trucked to surface to corresponding stockpiles outside the UG portal in the pit. From this point it is rehandled into surface trucks for transport to its final destination.

In general, each mining level will have its own re-muck bay, and additional re-mucks will be available in the ramp and in access development areas. These additional re-mucks may be used for temporary storage of material awaiting geological assays to categorize it as mineralized material or waste, or as turn-around for equipment. Re-muck back heights are sufficient for trucks to dump in them if necessary.

16.2.12.2 Waste Rock Handling

Waste rock is handled similarly to mineralized material and is deposited in a separate stockpile near the pit portal prior to re-handling into a surface truck for transport to its final destination.

16.2.13 Cut-off Grades

Cut-Off Grades for the Deposit are derived from an NSR function. The function uses cost inputs for five payable metals, determines their relative recoveries in one of five metal streams, and then subtracts penalties and smelting/processing costs to determine the expected value of the processed mineralized material. Information on recoveries can be found in Section 13 of this Technical Report, while further details on the smelting/refining aspects of the function can be found in Section 19.

16.2.13.1 Economic Cut-off Grade

Rather than creating specific Economic Cut-Off Grade ("ECOG") values for each combination of mining method, MET type and NSR stream, two general estimates of ECOGs for the underground portion of the Deposit were used: US$85 per tonne for Oxides (MET Type 6) and US$75 per tonne for Sulphides (all other MET types).

A summary of the components making up the ECOG values can be seen in Table 16.25.

Table 16.25 Economic Cut-off Grade Estimate	
Component	**Cost per Tonne**
Average Direct Mining Cost	$33.00
Development, Rehab, Construction, etc.	$8.75
Indirect Costs	$16.25
G&A	$4.75
Processing	$11.90
Sulphide ECOG	**$74.65 (used $75.00)**
Additional Oxide Processing Cost	$10.00
Oxide ECOG	**$84.65 (used $85.00)**

The initial estimate of the average direct mining cost is shown in Table 16.26.

Table 16.26 Average Direct Mining Cost Initial Estimate		
Method	**Direct Mining Cost per Tonne**	**Estimated Portion of Tonnes**
LH Transverse	$28.46	15%
LH Longitudinal	$29.56	5%
CF Type 1	$27.88	35%
CF Type 2	$31.88	15%
CF Type 3	$34.05	15%
CF Type 4	$51.85	15%
Weighted Average	**$33.00**	**100%**

16.2.13.2 Stope Optimization and Marginal Cut-off Grades

Once the two ECOG values were determined, a computerized stope optimization routine was run on the Deposit to identify economic areas to investigate. These areas were then checked for preliminary viability as mining blocks by estimating development, production, indirect and G&A costs associated with their extraction. If they were profitable, mining shapes were constructed around the blocks, taking into account the specific mining methods and costs for the block, and incorporating any contiguous material that exceeded the marginal cost of mining and processing for the block as a whole. Marginal Cut-Off Grades ("MCOG") were further used once individual profitable stopes were identified to maximize profitable material recovered in each stope.

The final shapes were checked for viability using actual designed development, and the MCOG was revised and the shape rebuilt if profit from the final mining area was less than profit from the preliminary area. Each mining block in the Deposit has had this check performed on it to maximize the value of the UG mine. An example of the optimization can be seen in Table 16.27.

TABLE 16.27 EXAMPLE STOPE OPTIMIZATION TABLE FOR A SULPHIDE STOPE				
Units	**Item**	**Value**	**Item**	**Value**
$ / t	Original ECOG	75	Mining MCOG	59.89
t	Original Tonnes	231,294	Additional Tonnes	214,541
$ / t	Original Shape NSR	178.77	New Shape NSR	121.56
m	Est Dev Length	865	New Dev Length	1,155
$ / m	Est Dev Cost	2,540	New Dev Cost	2,540
$ / t	Est Direct Mining Cost	33.12	New Direct Mining Cost	26.96
$ / t	Est Processing Cost	11.90	New Processing Cost	11.90
t / d	Est Stope Prod Rate	382	New Stope Prod Rate	468
t / d	Est Mine Prod Rate	2,300	New Mine Prod Rate	2,605
$ / d	Est Indirect Cost per Day	35,000	New Indirect Cost per Day	35,000
$ / d	Est G&A Cost per Day	2,500	New G&A Cost per Day	2,500
$ x 1000	Initial Stope NSR Value	41,348	New Stope NSR Value	54,197
$ x 1000	Initial Extraction Cost Est	16,382	New Extraction Cost	26,675
$ / t	Initial Profit / t	107.94	New Profit / t	61.73
$ x 1000	Initial Profit	24,967	New Profit	27,522

16.2.14 Mining Dilution

To recover mineralized material from the Back Forty Deposit, some non-economic material will be mined. Any recovered material with an NSR less than the MCOG for a stope is considered to be dilution, and material with an NSR between the MCOG and the ECOG for a stope is considered marginal mineralized material (e.g. mineralized material that is only being extracted since its value exceeds the variable cost of extraction and processing since the fixed costs of access are already sunk for the stope). Dilution is broken down into three types: Internal, External and Backfill, which are further explained in the following sub-sections. Average internal dilution is 13.6% by mass, average external dilution is 6.3% by mass, and average backfill dilution is 4.4% by mass.

16.2.14.1 Internal Dilution

Internal dilution is non-economic material that cannot be readily avoided in the extraction sequence and has a negative impact on the overall value of a stope.

In the case of LH mining methods, a small waste lens may exist within an LH stope. The overall value of extracting a slightly diluted stope is greater than leaving behind material that would otherwise be isolated by the waste rock.

In the case of CF methods, small areas of sub-MCOG material within the main stope may be taken to maintain access to larger areas further along strike. It may be possible to leave these areas as pillars when mining occurs, however, they are considered to be mined for the purposes of this PEA.

All internal dilution within the mining shape is included when the shape is interrogated against the block model to create the average NSR of the shape prior to external dilution.

16.2.14.2 Longhole External Dilution

After constructing the optimized mining shapes for a stope being extracted using LH mining methods, the final shape was extruded 1.0 m in all directions to account for overbreak while blasting. This material is considered to be external dilution and is primarily comprised of waste material, however, it may contain mineralized material with a positive NSR.

For narrow longitudinal LH stopes (roughly 7.5 m wide), this 1.0 m likely over-estimates actual dilution, however, for the purposes of this PEA it was deemed better to over-estimate and maintain a consistent diluting skin across all LH mining areas.

Overall average external dilution of LH stopes is 10.0% by mass.

16.2.14.3 Cut and Fill External Dilution

After constructing the optimized mining shapes for a stope being extracted using CF mining methods, the final shape was extruded 0.25 m in all lateral directions to account for overbreak outside of the mining shape while blasting. This material is considered to be external dilution, and is primarily comprised of waste material, however, it may contain mineralized material with a positive NSR.

Overall average external dilution of CF stopes is 4.2% by mass.

16.2.14.4 Backfill External Dilution

For areas where multiple laterally- or vertically-adjacent stopes are mined, the overbreak of primary stopes will intersect future mining panels. This overbreak is comprised of mineralized material for primary stopes. For secondary stopes, the overbreak will intersect PF. This dilution is comprised entirely of zero-value material. Table 16.28 shows the average backfill dilution for each mining method.

TABLE 16.28
BACKFILL DILUTION RATIOS BY MINING TYPE

Method	LH	CF1	CF2	CF3	CF4
PF Dilution by Mass	5.3%	3.7%	3.8%	4.0%	6.9%

16.2.15 Mining Recovery

During the extraction process, a percentage of the mineable material is not recovered. This can be due to leaving material in a stope due to geometries or geotechnical requirements, losses due to poor blasting practices, losses during transport, or misallocation of mineralized material/waste calls. Different mining methods have different expected recoveries, with LH mining generally having a lower recovery than CF mining due to its geometries and remote mucking requirements. Mining recoveries of each type are detailed in the following subsections.

Overall mining recovery on a tonne-weighted basis is expected to be 93.4%.

16.2.15.1 Longhole Mining Loss

Mining recovery in LH stopes is expected to be approximately 93% for normal transverse stoping. This number is based on experience in other LH operations.

16.2.15.2 Cut and Fill Mining Loss

CF mining of different sizes is expected to have slightly different mining recovery, varying from 93-95%. In general, it is assumed that as the openings decrease in size, the loss will decrease. This is based on the assumption that smaller, more selective, mining opening will be used in narrower, higher value mining areas, and therefore more attention will be paid to recovering every possible tonne from the stope to avoid leaving behind high value material. Table 16.29 shows the recoveries by opening size. The decrease in recovery for CF Type 4 is a result of it often being an uppers slash into a void below, rather than a true lateral advance CF method.

Table 16.29
CF Recovery Loss by Mining Type

Opening Size	Mining Recovery
CF 1 (7.5 m W x 5.0 m H)	93%
CF 2 (5.0 m W x 5.0 m H)	94%
CF 3 (4.0 m W x 5.0 m H)	95%
CF 4 (5.0 m W x 2.5 m H)	94%

Note: *W = width, H = height.*

Total average recovery from CF mining is 93.7%. This number is somewhat lower than for regular CF mining in veins since the geometry of most of the mineralized zones is sufficiently irregular, that selective recovery near the edges of mining areas may be more difficult than in CF vein operations.

16.2.16 Net Smelter Return Impacts

The NSR of material at Back Forty is sensitive to several factors, resulting in significant variations between in-situ values and mined values. A straight weighted average is therefore inappropriate to use for valuing the material. The impacts, and the methods of dealing with those impacts, are described in the sub-sections below.

16.2.16.1 Impact of Dilution on NSR

The functions governing the NSR of a block of material in the Back Forty Deposit are very complex, and are dealt with elsewhere in this Technical Report. It should be noted, however, that weighted averages of multiple material types blended together normally does not result in an NSR of the blend being the weighted average of the NSR of the individual components: it is normally less. As such, when calculating final diluted NSRs of mining blocks, the metal contents are used to determine an average metal content for the fully diluted (Internal, External and Backfill diluted) stope, and then the NSR of that average material is then recalculated using the appropriate NSR stream (dependent on material MET type). An example of this process is shown in Table 16.30.

TABLE 16.30
IMPACT OF WEIGHTED AVERAGE VERSUS RECALCULATION ON NSR

For a Stope of Met Type 5	Internally Diluted	External Dilution	Backfill Dilution	Fully Diluted (Average)	Fully Diluted (Recalculated)
Mined Tonnes	88	8	4	100	100
Au g/t	1.99	0.98	0	1.83	1.83
Ag g/t	45.04	8.56	0	40.32	40.32
Cu % by mass	0.04	0.06	0	0.04	0.04
Zn % by mass	1.50	0.25	0	1.34	1.34
Pb % by mass	0.88	0.32	0	0.80	0.80
NSR $/t	99.59	38.13	0.00	90.69	90.22

Material that does not fall within one of the MET classes (waste, or PF dilution) is apportioned to those classes on a mass-weighted basis. Material outside of the eight MET classes does not contain any measurable content of the payable metals of the Back Forty Deposit, so the original metal content of the applicable MET class is spread across the new total tonnage assigned to that MET type. An example of this process is shown in Table 16.31.

TABLE 16.31
IMPACT OF WASTE AND PASTEFILL DILUTION ON NSR

Item	MET Type	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	NSR ($/t)
Original Shape Material Content	1	70	2.28	59.09	0.05	1.70	1.02	88.64
	5	18	2.80	19.32	0.05	0.28	0.17	92.36
	Waste	8	0.00	0.00	0.00	0.00	0.00	0.00
	Paste	4	0.00	0.00	0.00	0.00	0.00	0.00
Final Material Content	1	80	2.01	52.00	0.04	1.50	0.90	76.83
	5	20	2.46	17.00	0.04	0.25	0.15	81.49

16.2.16.2 Impact of Multiple MET Types on NSR

Many stopes contain more than one MET type within the stope, but are predominately one type. For CF stopes, it should generally be possible to separate the various types during the mining sequence and send them to the appropriate stockpiles prior to blending and processing. For LH stopes, this is much more difficult, and is unlikely to be operationally feasible.

Since CF mining is the dominant method used in the Back Forty underground mine, NSR calculations have been standardized to assume that individual MET types can be separated prior to processing, and NSRs of individual stopes have been calculated under this assumption.

An example of the effect of this assumption for a stope comprised of 100 t of 80/20 blend of two Sulphide types being processed as the dominant type can be seen in Table 16.32.

TABLE 16.32
IMPACT OF MET TYPE SEGREGATION ON NSR

Material MET Type	1	5	Blended
Processed as MET Type	1	5	1
Tonnes	80	20	100
Au g/t	2.01	2.46	2.10
Ag g/t	52.00	17.00	45.00
Cu % by Mass	0.04	0.04	0.04
Zn % by Mass	1.50	0.25	1.25
Pb % by Mass	0.90	0.15	0.75
NSR for MET = 1	76.83	-	74.29
NSR for MET = 5	-	81.49	-
Blended Value ($x1000)	-	-	7.43
Separated Value ($x1000)	6.15	1.63	7.78

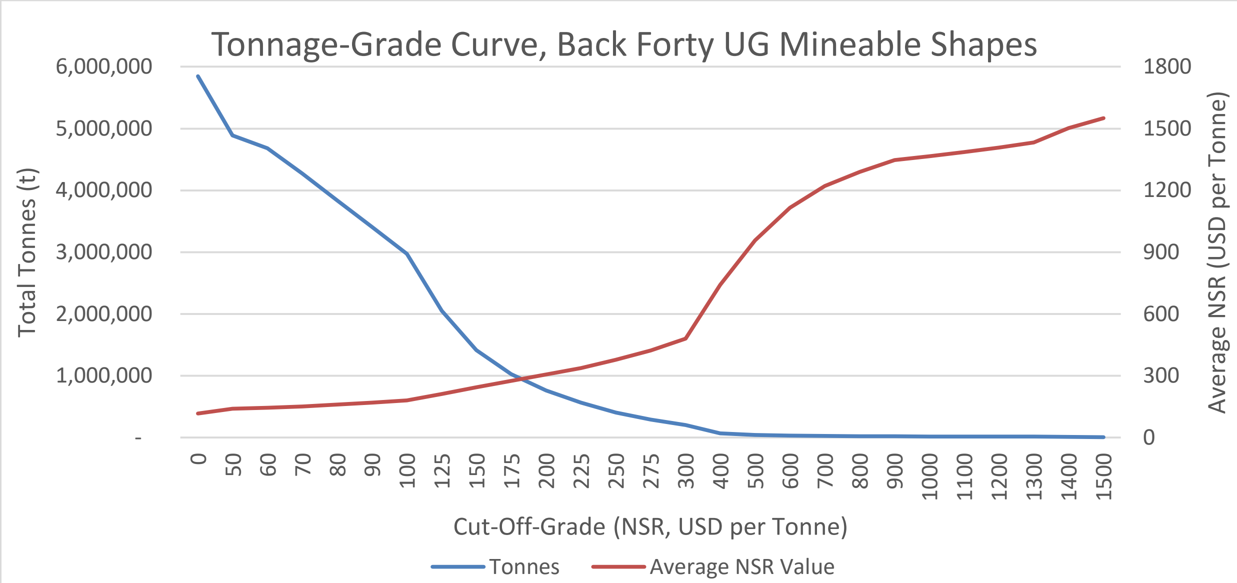
It should be noted that processing material of a different MET type in a particular MET stream is often likely to have some negative impacts on overall recovery that would further reduce the actual NSR of the blended material beyond the simple effects shown in the table. It is for these reasons that the assumption of separation of MET types at the mining face is maintained.

All stopes being mined via LH are comprised entirely of a single MET type, eliminating the impact of this assumption on LH mining areas.

16.2.17 Potentially Mineable Resource

The Back Forty underground Deposit is expected to yield a total of 5,717 kt of potentially mineable Mineral Resource at US$ 109.24 NSR per tonne. A breakdown of these tonnes by MET type can be seen in Table 16.11. The following subsections detail the methods and calculations used to determine those values. Figure 16.26 shows the tonnage-grade curve of the potentially mineable shapes.

FIGURE 16.26 UNDERGROUND MINEABLE SHAPE TONNAGE-GRADE CURVE



16.2.17.1 Potentially Mineable Mineral Resource Calculation

To calculate the potentially mineable portion of the underground Mineral Resource, the following two-stage process was followed:

1. Calculate contents of viable in-situ diluted stopes.
   a. Generate initial target shapes using an algorithm (Deswik Stope Optimizer) using ECOG values of US$75 per tonne for Sulphides and US$85 per tonne for Oxides.
   b. Eliminate shapes in crown pillars.
   c. Check remaining initial shapes for viability.
   d. Draw mineable shapes around the remaining target shapes, including marginal mineralized material with NSR above the MCOG value for the mining method used in extracting the shape. Include internal dilution as necessary to generate a shape that is operationally viable.
   e. Externally dilute the mineable shape by expanding the shape as previously described. Table 16.33 shows a summary of potentially mineable Mineral Resources at this stage.

2. Calculate recoverable portion of diluted stopes.
   a. Add PF dilution as previously described.
   b. Apply mining losses as previously described.
   c. Apply pillar losses related to Pit Pillar offset as previously described.
   d. Check final shapes for viability.
   e. Eliminate any stopes that cannot be recovered due to pit backfill schedule.
   f. Eliminate any stopes that are marginally profitable but displace more profitable mineralized material in the schedule and result in a probable negative impact to

the NPV of the UG mine. Table 16.34 shows a summary of Potentially Mineable Mineral Resources at this stage.

Under the mine plan described in this Technical Report, no stopes were eliminated by steps 2(e) and 2(f).

The total quantity of in-situ material in the In-Situ Diluted stopes is 5,849 kt.

The total quantity of material that can potentially be recovered from planned stopes is 5,717 kt. Total losses are 400 kt of fully diluted material.

TABLE 16.33
IN-SITU STOPE MATERIAL

			MET	Type	Tonnes (k)	Material Metal Contents Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
In-Situ Diluted Stopes	Internally Diluted Stopes	NSR > ECOG Material (Economic)	1	Sulphide	2,258	2.13	27.63	0.39	4.38	0.25
			2	Sulphide	-	-	-	-	-	-
			3	Sulphide	155	4.41	24.02	0.37	0.22	0.18
			4	Sulphide	695	2.19	22.86	0.88	1.13	0.09
			5	Sulphide	215	2.43	64.23	0.05	4.07	0.75
			6	Oxide	98	11.69	108.01	0.16	0.35	0.85
			7	Sulphide	-	-	-	-	-	-
			8	Sulphide	575	0.72	27.92	0.34	10.77	0.86
			99	Waste	-	-	-	-	-	-
		MCOG < NSR < ECOG Material (Marginal)	1	Sulphide	354	1.52	22.83	0.45	0.69	0.16
			2	Sulphide	-	-	-	-	-	-
			3	Sulphide	106	1.75	12.94	0.53	0.36	0.09
			4	Sulphide	176	1.27	13.75	0.59	0.77	0.07
			5	Sulphide	23	1.08	31.94	0.03	1.75	0.45
			6	Oxide	2	1.85	45.60	0.21	0.34	0.50
			7	Sulphide	-	-	-	-	-	-
			8	Sulphide	29	0.67	14.63	0.32	3.08	0.15
			99	Waste	-	-	-	-	-	-
		NSR < MCOG Material (Internal Dilution)	1	Sulphide	101	1.52	15.04	0.36	0.46	0.11
			2	Sulphide	-	-	-	-	-	-
			3	Sulphide	68	1.27	11.72	0.44	0.39	0.09
			4	Sulphide	50	1.06	14.69	0.37	0.81	0.08
			5	Sulphide	9	0.72	31.70	0.02	1.10	0.54
			6	Oxide	6	1.12	46.39	0.19	0.45	0.43
			7	Sulphide	-	-	-	-	-	-
			8	Sulphide	17	0.54	14.03	0.30	2.35	0.12
			99	Waste	544	-	-	-	-	-
	External Dilution		1	Sulphide	59	1.79	21.27	0.41	2.34	0.17
			2	Sulphide	-	-	-	-	-	-
			3	Sulphide	12	1.81	13.88	0.42	0.38	0.11
			4	Sulphide	8	1.58	20.89	0.51	0.93	0.11
			5	Sulphide	6	1.22	28.85	0.03	2.70	0.51
			6	Oxide	3	8.11	86.39	0.12	0.69	0.63
			7	Sulphide	-	-	-	-	-	-
			8	Sulphide	5	0.67	20.69	0.29	8.03	0.52
			99	Waste	276	-	-	-	-	-

Table 16.34 **Recoverable Stope Material**									
				Average Material Metal Contents					
		MET	**Type**	**Tonnes (k)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Zn (%)**	**Pb (%)**
Fully Diluted Stopes	**In-Situ Diluted Stopes**	1	Sulphide	2,771	2.02	26.42	0.39	3.72	0.23
		2	Sulphide	-	-	-	-	-	-
		3	Sulphide	340	2.87	17.78	0.44	0.30	0.13
		4	Sulphide	929	1.95	20.67	0.79	1.04	0.09
		5	Sulphide	253	2.21	59.28	0.05	3.72	0.71
		6	Oxide	109	10.82	102.83	0.16	0.36	0.81
		7	Sulphide	-	-	-	-	-	-
		8	Sulphide	626	0.71	26.87	0.33	10.16	0.81
		99	Waste	820	-	-	-	-	-
			Paste	268	-	-	-	-	-
	Losses	1	Sulphide	190	2.03	26.66	0.40	3.70	0.23
		2	Sulphide	-	-	-	-	-	-
		3	Sulphide	20	2.87	17.76	0.44	0.30	0.13
		4	Sulphide	57	1.96	20.88	0.82	1.02	0.09
		5	Sulphide	16	2.22	60.81	0.04	3.79	0.71
		6	Oxide	6	11.35	109.25	0.17	0.36	0.88
		7	Sulphide	-	-	-	-	-	-
		8	Sulphide	38	0.71	26.87	0.33	10.16	0.81
		99	Waste	55	-	-	-	-	-
			Paste	18	-	-	-	-	-
Potentially Recoverable Stope Material (Blended and Diluted)		1	Sulphide	3,133	1.66	21.76	0.33	3.07	0.19
		2	Sulphide	-	-	-	-	-	-
		3	Sulphide	387	2.37	14.69	0.36	0.25	0.11
		4	Sulphide	1,029	1.65	17.50	0.67	0.88	0.08
		5	Sulphide	295	1.77	47.45	0.04	2.98	0.57
		6	Oxide	160	6.96	66.09	0.10	0.23	0.52
		7	Sulphide	-	-	-	-	-	-
		8	Sulphide	713	0.59	22.19	0.28	8.39	0.67

16.2.17.2 Potentially Mineable Mineral Resource Summary

A summary of the potentially mineable portion of the underground Mineral Resource can be seen in Table 16.35. There are no MET Type 2 or 7 in the potentially mineable portion of the underground Mineral Resource. All tonnages are fully diluted and recovered blended into the MET types on a tonne-weighted basis.

TABLE 16.35 POTENTIALLY MINEABLE PORTION OF THE MINERAL RESOURCE									
				Average Material Metal Contents					NSR
	MET	**Type**	**Tonnes (k)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Zn (%)**	**Pb (%)**	**US$/t**
Potentially Mineable Mineral Resource	1	Sulphide	3,133	1.66	21.76	0.33	3.07	0.19	102.58
	2	Sulphide	-	-	-	-	-	-	-
	3	Sulphide	387	2.37	14.69	0.36	0.25	0.11	75.83
	4	Sulphide	1,029	1.65	17.50	0.67	0.88	0.08	83.66
	5	Sulphide	295	1.77	47.45	0.04	2.98	0.57	112.27
	6	Oxide	160	6.96	66.09	0.10	0.23	0.52	287.07
	7	Sulphide	-	-	-	-	-	-	-
	8	Sulphide	713	0.59	22.19	0.28	8.39	0.67	150.89
	Total		**5,717**						**109.24***

***** *Average NSR is derived from MET stream blending, not weighted averages.*

The total potentially mineable portion of the underground Mineral Resource is 5,717 kt at an average NSR of US$ 109.24 per tonne. It should be noted that, due to stockpiling and blending on surface, the realized NSR of the tonnes produced from the underground portion of the Deposit may not exactly match the calculated NSR.

16.3 COMBINED OPEN PIT AND UNDERGROUND

Open pit mining will extend from Year -1 to Year 5. Underground development will be initiated in Year 5 and underground mining will continue to Year 11. Remaining stockpiles will be processed in Year 12 and a partial Year 13. The combined mining schedule is shown in Table 16.36.

TABLE 16.36
COMBINED OPEN PIT AND UNDERGROUND MINING SCHEDULE

Material Source	Units	Total	Year											
			Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Open Pit														
Waste Rock	kt	51,747	2,801	10,911	13,027	13,437	10,512	1,058	-	-	-	-	-	-
Process Plant Feed	kt	10,132	199	2,589	1,974	1,563	2,987	821	-	-	-	-	-	-
Sub-Total	**kt**	**61,880**	**3,000**	**13,500**	**15,000**	**15,000**	**13,500**	**1,879**	**-**	**-**	**-**	**-**	**-**	**-**
Underground														
Waste Rock	kt	1,322					-	306	405	264	187	128	32	-
Process Plant Feed	kt	5,717	-	-	-	-	-	122	683	1,126	1,126	959	968	732
Sub-Total	**kt**	**7,039**	**-**	**-**	**-**	**-**	**-**	**427**	**1,088**	**1,390**	**1,313**	**1,088**	**1,000**	**732**
Combined														
Waste Rock	kt	53,069	2,801	10,911	13,027	13,437	10,512	1,364	405	264	187	128	32	-
Process Plant Feed	kt	15,849	199	2,589	1,974	1,563	2,987	943	683	1,126	1,126	959	968	732
Total	**kt**	**68,918**	**3,000**	**13,500**	**15,000**	**15,000**	**13,500**	**2,307**	**1,088**	**1,390**	**1,313**	**1,088**	**1,000**	**732**

17.0 RECOVERY METHODS

17.1 PROCESS DESIGN

The Project is based on mining the highest value material as soon as possible and treating this material through the process plant to maximize cash flow. This strategy is achieved by a mine to stockpile to process plant strategy.

Oxide mineralized material and sulphide mineralized material (Main, Pinwheel and Tuff material) are treated through separate process plants.

The oxide mineralized material will be processed via a cyanidation leach circuit to produce doré. Depending on the grades of copper, zinc and lead, the sulphide mineralized material will be processed via two stages of flotation to produce concentrates, i.e. either a copper and zinc concentrate, or a lead and zinc concentrate.

Sulphide mineralized material will be processed on a campaign basis based on the main material types that have a similar metallurgical response. As such the design of the sulphide plant is based on a flexible metallurgical flowsheet to process the main material types.

The oxide and sulphide flowsheets are based on proven unit operations in the industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layouts provide ease of access to all equipment for operating and maintenance requirements, whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the process plants are:

- Nominal throughput of 2,800 tpd sulphide mineralized material (also known as flotation mineralized material) and 350 tpd oxide mineralized material (also known as leach mineralized material).

- Crushing circuit availability of 75% supported by the use of surge bins and dedicated feeders for choke feeding cone crushers for optimum crushing performance and wear minimization.

- Oxide and sulphide process plant availability of 91.3% through the use of standby equipment in critical areas and reliable grid power supply.

- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if and when required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork.

17.1.1 Selected Process Flowsheet

The oxide process plant has been designed for a throughput of 350 tpd (dry) at head grades of up to 8.0 g/t Au and 127 g/t Ag. The overall flowsheet includes the following steps:

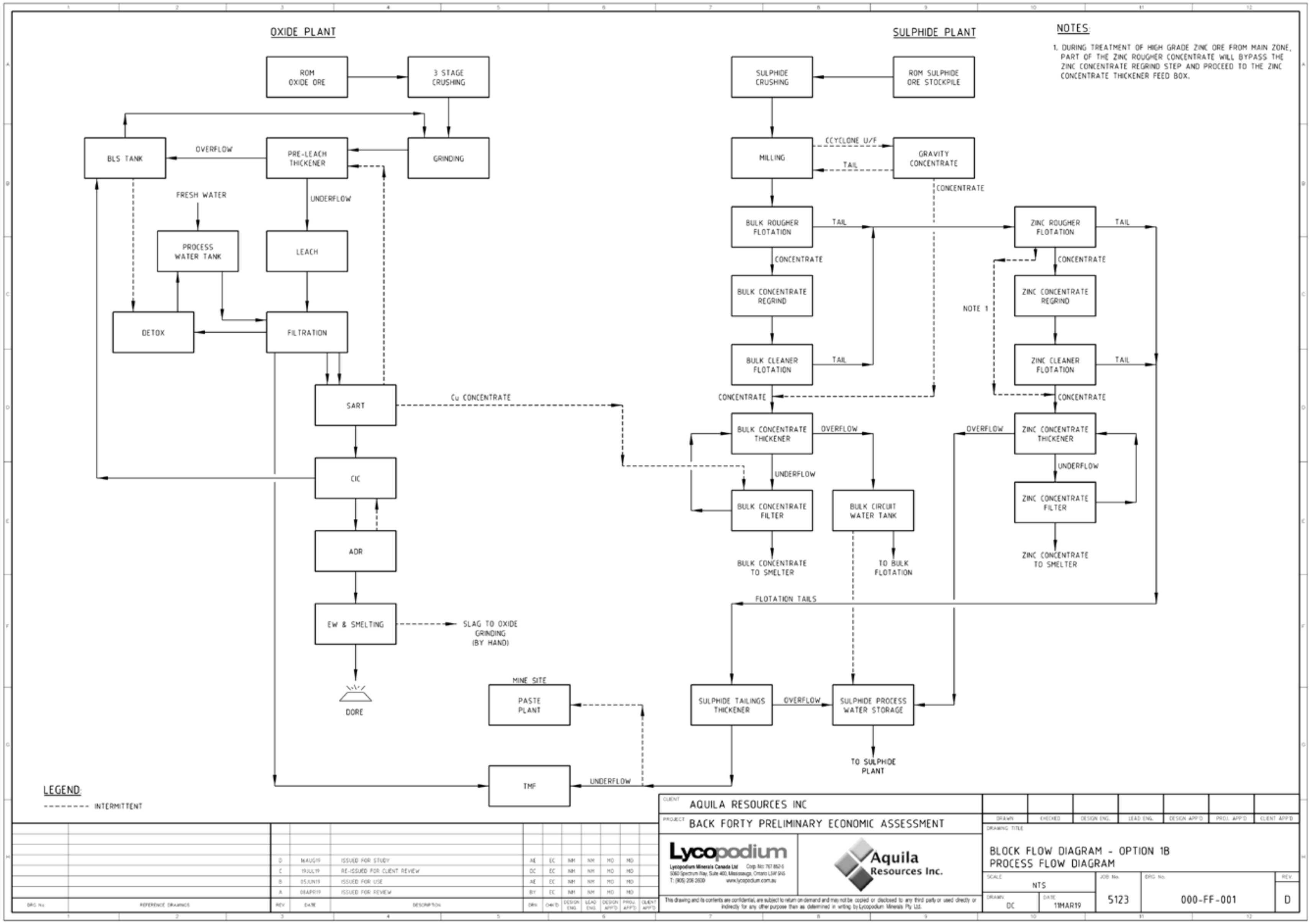
- Three stage crushing using an open circuit jaw crusher, open-circuit secondary cone crusher and closed-circuit tertiary cone crusher.
- Grinding and classification.
- Pre-leach thickening.
- Cyanide leach.
- Vacuum filtration of leaching tailings.
- Sulphidization, Acidification, Recycle and Thickening (“SART”).
- Carbon-in-Column (“CIC”) gold adsorption.
- Carbon acid-washing, desorption and recovery (“ADR”).
- Smelting to produce doré.
- Cyanide destruction of the final wash filtrate from the vacuum filtration step.
- Tailings repulping and disposal to the TMF.

The sulphide process plant has been designed for a nominal throughput of 2,800 tpd (dry), with varying copper, lead and zinc head grades. The overall flowsheet includes the following steps:

- Primary crushing.
- Coarse mineralized material stockpile and reclaim.
- Grinding and classification.
- Gravity concentration.
- Bulk rougher flotation to produce copper concentrate or lead concentrate depending on mineralized material campaign.
- Zinc rougher flotation.
- Bulk concentrate regrind (copper or lead concentrate).
- Zinc concentrate regrind.
- Bulk cleaner flotation, using three stages of cleaning (copper or lead concentrate).
- Zinc cleaner flotation, using two stages of cleaning.
- Bulk concentrate thickening and filtration (copper or lead concentrate).
- Zinc concentrate thickening and filtration.
- Tailings thickening and disposal in the common TMF.

Figure 17.1 presents an overall block flow diagram depicting the major unit operations incorporated in the selected process flowsheet.

FIGURE 17.1 OVERALL PROCESS PLANT BLOCK FLOW DIAGRAM



Source: *Lycopodium Minerals Canada Ltd. (2019)*

17.1.2 Operating Context

Oxide material and sulphide material will be treated through separate process plants. The process plants will receive material based on a processing schedule where material will either come from the mine directly or from nearby stockpiled material that was mined earlier.

Stockpiled material will be stored according to the main material types (not blended) that have a similar metallurgical response in the plant, i.e. Main, Pinwheel, Tuff, HG Oxide and LG Oxide. Oxide material will be constantly fed to the Oxide Plant at 350 tpd. Depending on the processing schedule, sulphide material will be fed constantly per material type on a campaign basis to the Sulphide Plant at a nominal rate of 2,800 tpd.

Table 17.1 summarizes the relative mineralogical differences between Main, Pinwheel and Tuff material types.

TABLE 17.1
MINERALOGY OF SULPHIDE MATERIAL (RELATIVE DIFFERENCES)

Material Type	Mineral Liberation for Flotation (P_{80})	Chalcopyrite (Copper Mineral)	Galena (Lead Mineral)	Sphalerite (Zinc Mineral)
Main	75 µm	Medium	-	High
Pinwheel	50-55 µm			
MS and Gossan		High	-	Trace
SM and Stringers		Low	-	Low
Extension		Medium	-	High
Tuff	60-65 µm	Trace	High	Medium

Note: *MS = massive sulphide, SM = semi-massive sulphide.*

For stable process plant operations, the processing schedule has a minimum of one month campaigning on a material type. When the feed material is changed from Main to Pinwheel and vice versa, the sulphide plant parameters are adjusted according to the metallurgical requirements and are considered relatively minor in nature. When the feed material is changed from Main or Pinwheel to Tuff and vice versa, then a complete clean out of the bulk flotation circuit is required to prevent contamination of the final concentrates. Table 17.2 summarizes the key operational changes in the sulphide plant when changing between material campaigns.

TABLE 17.2 PROCESS PLANT CHANGEOVER SCENARIOS			
Scenario	**Main to Pinwheel**	**Main to Tuff**	**Pinwheel to Tuff**
Material Hardness	13.3 to 12.4 kWh/t	13.3 to 15.6 kWh/t	12.4 to 15.6 kWh/t
Grind Size	75 to 55 µm	75 to 65 µm	55 to 65 µm
Ball Load	minor increase	increase	increase
Mill Speeds	minor increase	increase	increase
Bulk Rougher:			
pH and Frother	minor change	minor change	minor change
Collector	reduce	reduce	reduce
Bulk Regrind:			
Mass Recovery	150-250% increase	150% increase	50% decrease
Regrind size	20 to 15 µm	20 to 15 µm	no change
Bulk Cleaner:			
pH and Frother	minor change	minor change	minor change
Collector	reduce	reduce	reduce
Zinc Rougher			
pH and Frother	minor change	minor change	minor change
Collector	reduce	reduce	reduce
Zinc Regrind:			
Mass Recovery	50% decrease	50% decrease	minor change
Zinc Cleaner:			
pH and Frother	minor change	minor change	minor change
Collector	reduce	reduce	reduce

17.1.3 Key Process Design Criteria

The key process design criteria listed in Table 17.3 and Table 17.4 form the basis of the detailed process design criteria and mechanical equipment list. It is noted that the process plant feed grades are based on a processing schedule that is different to the mine production plan in Section 16 due to the use of stockpiles. Metal recovery equations are summarized in Section 13.8.

TABLE 17.3
KEY PROCESS DESIGN CRITERIA – OXIDE PROCESS PLANT

Parameter	Units	Value	Source
Plant Throughput	tpd	350	Aquila
Head Grade – Design	g/t Au	8.02	Aquila
	g/t Ag	127	Aquila
Design Leach Recovery – Au[1]	%	92.4	Lycopodium
Design Leach Recovery – Ag[1]	%	83.1	Lycopodium
Crushing Plant Availability	%	75.0	Lycopodium
Plant Availability	%	91.3	Lycopodium
Mineralized Material Specific Gravity	-	2.7	Lycopodium
Bond Crusher Work Index (CWI)	kWh/t	10.7	Testwork
Bond Ball Mill Work Index (BWI)	kWh/t	18.2	Testwork
SMC Axb		23.2	Testwork
Bond Abrasion Index (AI)	g	0.467	Testwork
Grind Size (P_{80})	µm	75	Testwork
Pre-Leach Thickener Solids Loading	$t/m^2.h$	1.14	Lycopodium
Leach Circuit Residence Time	hrs	48	Testwork
Leach Slurry Density	% solids (w/w)	46	Testwork
Number of Leach Tanks		5	Lycopodium
Sodium Cyanide Addition	kg NaCN/t	1.00	Testwork
Lime Addition[2]	kg/t	0.88	Testwork
Oxygen Source		Air	Lycopodium
Air Addition Rate	$Nm^3/h/m^3$	0.20	Lycopodium
Pregnant Solution Recovery Method		Vacuum Filtration	Aquila
Copper and Silver Recovery Method		SART	Aquila
Gold Recovery Method		CIC, ADR	Aquila
Cyanide Destruction Method		SO_2 / Air	Aquila
Tailing Disposal		Pumped to TMF	Aquila
Tailings Discharge Slurry Density	% solids (w/w)	55	Lycopodium

Notes:
1) *At design head grade of 8.0 g/t Au and 127 g/t Ag.*
2) *100% CaO basis.*
3) *w/w = % solids by weight.*

TABLE 17.4
KEY PROCESS DESIGN CRITERIA – SULPHIDE PLANT

Parameter	Units	Value	Source
Nominal Plant Throughput	tpd	2,800	Aquila
Plant Throughput when treating Tuff	tpd	2,800	Aquila
Plant Throughput when treating Main	tpd	3,500	Aquila
Plant Throughput when treating Pinwheel	tpd	3,440	Aquila
Pinwheel mineralized material Type - Head Grade (High Cu, Low Zn Mineralized material)	% Cu (w/w)	2.33	Aquila
	% Pb (w/w)	0.00	Aquila
	% Zn (w/w)	0.02	Aquila
Main mineralized material Type - Head Grade (High Zn, Low Cu Mineralized material)	% Cu (w/w)	0.21	Aquila
	% Pb (w/w)	0.13	Aquila
	% Zn (w/w)	9.91	Aquila
Tuff Mineralized material Type - Head Grade (High Pb Mineralized material)	% Cu (w/w)	0.02	Aquila
	% Pb (w/w)	0.54	Aquila
	% Zn (w/w)	1.62	Aquila
Crushing Plant Availability	%	75.0	Lycopodium
Plant Availability	%	91.3	Lycopodium
Mineralized material Specific Gravity - Pinwheel	-	3.26 to 4.75	Testwork
Mineralized material Specific Gravity - Main	-	3.26 to 4.78	Testwork
Mineralized material Specific Gravity - Tuff	-	2.94 to 3.00	Testwork
Bond Crusher Work Index (CWI) - Pinwheel	kWh/t	8.2	Testwork
Bond Crusher Work Index (CWI) - Main	kWh/t	14.6	Testwork
Bond Crusher Work Index (CWI) - Tuff	kWh/t	23.6	Testwork
Bond Ball Mill Work Index (BWI) - Pinwheel	kWh/t	12.4	Testwork
Bond Ball Mill Work Index (BWI) - Main	kWh/t	13.3	Testwork
Bond Ball Mill Work Index (BWI) - Tuff	kWh/t	15.6	Testwork
SMC Axb – Pinwheel		35.7	Testwork
SMC Axb – Main		37.4	Testwork
SMC Axb – Tuff		28.8	Testwork
Bond Abrasion Index (AI) – Pinwheel	g	0.288	Testwork
Bond Abrasion Index (AI) – Main	g	0.374	Testwork
Bond Abrasion Index (AI) – Tuff	g	0.544	Testwork
Grind Size (P_{80}) – Pinwheel	µm	55	Testwork
Grind Size (P_{80}) – Main	µm	70	Testwork
Grind Size (P_{80}) – Tuff	µm	60	Testwork
Gravity Concentration – Cyclone UF split	%	27	Lycopodium
Bulk Rougher Residence Time – Lab	min	7	Testwork
Bulk Cleaner 1 Residence Time – Lab	min	10	Testwork
Bulk Cleaner 2 Residence Time – Lab	min	7	Testwork
Bulk Cleaner 3 Residence Time – Lab	min	7	Testwork
Zinc Rougher Residence Time – Lab	min	7	Testwork
Zinc Cleaner 1 Residence Time – Lab	min	8	Testwork

TABLE 17.4 **KEY PROCESS DESIGN CRITERIA – SULPHIDE PLANT**			
Parameter	**Units**	**Value**	**Source**
Zinc Cleaner 2 Residence Time – Lab	min	7	Testwork
Bulk Regrind Mill Product Size (P_{80})	µm	15 - 20	Testwork
Zinc Regrind Mill Product Size (P_{80})	µm	15 - 20	Testwork
Bulk Concentrate Production Rate – Design1	t/h	6.5	Calculated
Zinc Concentrate Production Rate – Design1	t/h	21.8	Calculated
Bulk Concentrate Thickener Solids Loading	$t/m^2.h$	0.20	Lycopodium
Zinc Concentrate Thickener Solids Loading	$t/m^2.h$	0.20	Lycopodium
Bulk Filter Solids Loading	$kg/m^2.h$	200	Lycopodium
Zinc Filter Solids Loading	$kg/m^2.h$	200	Lycopodium
Copper Concentrate SG - Design	-	4.2	Lycopodium
Zinc Concentrate SG - Design	-	4.0	Lycopodium
Lead Concentrate SG - Design	-	6.0	Lycopodium
Concentrate Moisture - Design	-	8.0	Lycopodium
Tailings Thickener Type		High Compression	Golder
Tailings Thickener Solids Loading	$t/m^2.h$	0.94	Golder
Tailings Slurry Density – Pinwheel	% solids (w/w)	72.0	Golder
Tailings Slurry Density – Main	% solids (w/w)	76.0	Golder
Tailings Slurry Density – Tuff	% solids (w/w)	65.0	Golder

Note: *1) Maximum production rate at design head grades, calculated using relationship between head grade and concentrate mass recovery from testwork.*

17.1.3.1 Comminution

Design parameters for the comminution circuit were sourced from testwork conducted at SGS during 2015 and 2017. Orway Mineral Consultants ("OMC") carried out mineralized material characterization and comminution modelling based on this testwork. Design A x b values were derived from the 15th percentile ranking of specific energies determined for each individual mineralized material type.

Major observations and conclusions from the mineralized material characterization were as follows:

- There is significant variability in competency between the samples tested. The JK A and b parameters measure resistance to impact breakage and can be used to classify the mineralized material in terms of competency – the lower the product of A x b the greater the resistance to breakage.

- The Oxide and Sulphide Tuff mineralized material exhibit the highest competency with an A x b less than 30 indicating that these samples are extremely competent. The Sulphide Main and Sulphide Pinwheel mineralized material, with an A x b value in the 30 to 40 range, have a moderate competency.

- In general, more variability is observed with the Main A x b and minimal variability for the Tuff A x b. It was observed a small difference between the 75^{th} and 85^{th} percentile for Tuff and Pinwheel A x b. (less than 2%), but about 20% for Main Axb.

- The high competency of the Oxide samples (A x b less than 30) preclude the possibility of utilizing a single stage SAG mill, since the mill will be very energy inefficient. A multi-stage crushing circuit would provide a more energy efficient comminution circuit.

- Significant variability in mineralized material abrasiveness is evident. With the exception of the Pinwheel samples, all samples are considered moderately abrasive (AI more than 0.4) and above average wear rates and media consumption are anticipated.

- Significant variability in hardness and grindability is evident between the samples tested. The mineralized material types with a BWI greater than 15 kWh/t (85^{th} percentile) are considered hard whilst Pinwheel mineralized material with a BWI of 12.4 kWh/t (85% percentile) are considered only moderately hard.

- A consistent shape in the variability trends in the modified BWI (modBond) (highly correlated to the Standard Bond BWI) were found for all mineralized material types, in which the Tuff mineralized material is offset from other two softer mineralized material types. There is a small difference between the 75^{th} and the 85^{th} percentile for BWI for all mineralized material types (less than 5%).

17.1.3.2 Cyanide Leach and Metal Recovery

Design parameters for the leach circuit design were sourced from testwork conducted at SGS during 2015 and 2017 and 2018 at BQE.

Several trade-off studies were completed to examine alternative circuit configurations for the leach and recovery of precious metals. These included:

- Air versus oxygen.
- Standard Merrill-Crowe versus hybrid Merrill Crowe versus flotation Merrill Crowe.
- Leach residence times of 24, 36 and 48 hours.
- Vacuum filtration followed by SART as an alternative to a Merrill-Crowe flowsheet.

The 48-hour leaches gave the highest NPVs, with very little difference between air and oxygen at the longer residence time. Aquila elected to pursue the vacuum filtration, SART and carbon adsorption flowsheet instead of the Merrill Crowe recovery circuit for this study. The primary driver for this decision is the need to remove copper from the circuit with a view to improving the quality of the doré bars. SART also allows for the removal of mercury and silver from the pregnant leach solution to further improve the quality of the doré product. The recovery of free cyanide and cyanide bound as weak acid dissociable metal complexes is expected to improve the economics of this flowsheet.

17.1.3.3 Flotation Circuit

The flotation circuit has been designed to treat a range of mineralized material types.

The flotation circuit configuration, residence times, reagent addition rates and concentrate mass recoveries have been selected based on the metallurgical testwork conducted at SGS.

A relationship between head grade and concentrate mass recovery was developed for each flotation stage to enable the engineering mass balance to be developed.

17.1.3.4 Ancillary Testwork

No concentrate regrinding concentrate thickening or concentrate filtration testwork has been completed as part of this study. All values are based on similar projects from the Lycopodium database and/or vendor advice.

Oxygen uptake testing and leach slurry rheology has not been completed for the oxide samples. This work should be completed prior to the next phase design.

17.1.3.5 Tailings Thickening

Tailings thickener parameters were sourced from testwork conducted by Golder in 2016 using a combined oxide tailings sample from the SGS testwork and three separate sulphide samples representing Pinwheel, Main and Tuff mineralized material, again from the SGS testwork.

17.2 OXIDE CRUSHING CIRCUIT

A modular three-stage crushing circuit will be used to produce a product with a P_{80} of 8 mm, suitable for feeding a single stage ball mill.

17.2.1 Oxide Primary Crushing

The Run-of-Mine ("ROM") loader will collect mineralized material from the ROM oxide stockpile and feed it into the coarse mineralized material bin via a grizzly. A water spray/glycol system will be installed at the coarse mineralized material bin for dust suppression purposes. The static grizzly will have a spacing of 600 mm to prevent the ingress of oversize mineralized material. A mobile rock breaker will be provided to assist in breaking down oversize material retained on the static grizzly. Mineralized material will be withdrawn from the ROM bin by a variable speed vibrating grizzly with a 40 mm aperture size. Oversize from the grizzly will report directly to the primary jaw crusher, which will operate in open circuit. Crushed product from the jaw crusher, along with vibrating grizzly undersize, will report to the oxide transfer conveyor.

The transfer conveyor will be fitted with a weightometer to monitor and control the crushing rate via the vibrating grizzly variable speed control. A static magnet will remove metal from the crushed mineralized material stream. Tramp metal will be removed manually. A metal detector will be installed near the head end of the transfer conveyor as a secondary safeguard.

The primary crushing area will be serviced by the primary dust collector, which will be comprised of a series of extraction hoods, ducting and a bag house. Dust collected from this system will be discharged onto the transfer conveyor.

A sump pump will be provided in the crushing area for spillage and run-off control.

17.2.2 Secondary and Tertiary Crushing and Screening

Primary crushed mineralized material will report to a secondary crusher feed bin from where mineralized material will be fed to the secondary crusher using a vibrating feeder. Secondary crusher product with a P_{80} of 14 mm will be conveyed to a double deck screen. This screen will be fitted with 16 mm aperture panels on the top deck and a 12 mm panels on the bottom deck. Oversize from the top and second decks will report to the tertiary crusher feed bin. This oversize material will be fed to the tertiary cone crusher using a vibrating feeder. Tertiary crushed product with a P_{80} of 10.6 mm will be conveyed back to the screen feed conveyor.

The screen feed conveyor will be equipped with a metal detector which will automatically stop the belt and indicate the location of metal via a paint marker. Tramp metal will then need to be manually removed from the belt.

Undersize product from the double deck screen, at a P_{80} of 8 mm, will report to the oxide fine mineralized material product conveyor which will discharge it into the fine mineralized material bin.

The secondary crushing area will be serviced by the secondary dust collector, which will be comprised of a series of extraction hoods, ducting and a bag house. Dust collected from this system will be discharged onto the screen product conveyor.

17.3 OXIDE GRINDING CIRCUIT

Mineralized material from the oxide fine mineralized material bin will be delivered to a skid mounted oxide ball mill using the oxide reclaim feeder. This variable speed belt feeder will discharge onto the oxide mill feed conveyor. The conveyor will be equipped with a weightometer for measuring and controlling the mill feed rate.

A pulse dust collection system will be installed at the transfer point between the feeders and the covered conveyor to reduce airborne dust around the fine mineralized material reclaim area.

The oxide grinding circuit will receive mineralized material at a nominal top size of 12 mm with a P_{80} passing size of 8 mm. The circuit will consist of a single-stage ball mill in closed circuit with a cyclone cluster. The ball mill will be a 2.90 m diameter x 4.60 m EGL overflow mill, with a 450 kW fixed speed motor. The mill will operate at a 32% ball charge. Mineralized material will be fed to the ball mill at a controlled rate, nominally 16.0 dry t/h, and water will be added to the feed chute to achieve the desired milling density. Hydrated lime will also be added to the mill feed to ensure adequate mixing and contact with the mineralized material surfaces.

Product from the oxide ball mill will discharge over a trommel, with oversize reporting to the scats bunker, where it will be periodically removed by skid-steer loader and returned to the circuit via the clean-up hopper. Trommel undersize will flow, by gravity, to the oxide cyclone feed hopper where it will be further diluted to achieve the required cyclone feed density. The oxide cyclone feed pump will deliver slurry to the cyclone cluster. Cyclone underflow will return to the oxide ball mill, while cyclone overflow will flow by gravity to the oxide trash screen. The cyclone cluster will consist of two operating 10-inch cyclones plus one installed spare and at least one blank nozzles.

The oxide trash screen will be a vibrating screen designed to remove foreign material prior to thickening. Trash will report to the trash bin which will be periodically emptied. Screen undersize will flow by gravity to the pre-leach thickener.

A vertical spindle sump pump will service the area. The concrete floor under the mill area will slope to the sumps to facilitate cleanup. Grinding media for the oxide ball mill will be introduced by use of a dedicated kibble and the grinding building gantry crane.

17.4 PRE-LEACH THICKENING AND LEACHING

Slurry from the oxide trash screen undersize will gravity flow to the pre-leach thickener feed box, where it will be mixed with dilute flocculant to increase particle settling rates. The pre-leach thickener will be a 5 m diameter high-rate thickener. Thickener underflow at 55% solids (w/w) will be pumped to the leach feed distribution box using the pre-leach thickener underflow pump. A pressure pipe sampler and secondary vezin sampler will be installed to provide a sample of leach feed for monitoring purposes. Thickener overflow will gravity flow to the oxide process water tank for re-use. An oxide process water pump will be provided to distribute process water around the plant.

A single vertical spindle sump pump will service the pre-leach thickener area for spillage clean-up. Spillage and wash down collected by the sump pump will be returned to the thickener feed box.

Pre-leach thickener underflow will report to the leach feed distribution box. The slurry from the leach feed distribution box will flow by gravity to the first leach tank. If the first leach tank is offline, the slurry will be diverted to the second leach tank, via a dart valve system.

The leach circuit will consist of five, mechanically agitated, leach tanks operating in series. The volume of this leach circuit will allow a nominal residence time of 48 hours at a feed rate of 16.0 t/h. Each leach tank will have a live volume of 246 m^3.

Sodium cyanide, for gold and silver dissolution, will be added to the leach circuit via the cyanide ring main and dosing valves. The primary cyanide dosing point will be the leach feed distribution box, with a further addition point located in the third leach tank. The operating pH of the leach circuit will be maintained between 10.5 and 11.0 to maintain the protective alkalinity of the circuit and prevent the loss of cyanide to gaseous hydrogen cyanide. This will be achieved by the addition of lime slurry to the oxide ball mill feed chute. However, provision for further lime addition to the leach feed distribution box has also been made. Additionally, process water will be added to the leach feed distribution box to achieve the desired leach slurry density of 46% solids (w/w).

Low pressure air will provide the amount of oxygen required for leaching and will be supplied via a dedicated low-pressure air blower, with air delivered through the hollow agitator shafts.

Pneumatic actuated gates, located within the leach inter-stage launders, will allow any of the leach tanks to be bypassed for maintenance purposes. One gate will divert slurry to the following leach tank while the second gate will allow slurry flow to the subsequent leach tank.

An inline ‘shark fin’ style sampler and a secondary vezin sampler will be installed on the discharge of the last leach tank for process monitoring purposes. Slurry exiting the leach circuit will flow by gravity to the horizontal belt filter feed box.

The leach circuit will be serviced by one vertical spindle sump pump, which will return spillage to the feed distributor of the leach circuit.

17.5 VACUUM FILTRATION AND OXIDE TAILINGS

A horizontal belt filter (“HBF”) will be used to recover pregnant leach solution for further treatment and to produce a barren filter cake for tailings disposal. Two stages of washing will be used to minimize the amount of dissolved gold in the moist filter cake. The second filtrate from the first washing stage will be combined with the first filtrate (pregnant leach solution) to form the feed solution to the downstream gold and cyanide recovery unit operations. Filtrate from the final (second) washing stage (third filtrate) will be recycled via a cyanide detoxification step to the oxide process water tank.

Filtration rates will be enhanced by the addition of dilute flocculant to the leached slurry in the HBF Feed Tank. The filtration rate basis is 225 kg/h/m^2 which yields a filter area requirement of 72 m^2. A final filter cake moisture of 20% w/w was determined under these conditions during laboratory testing and are considered for the filter design.

At the expected solution displacement fractions of 85% for both washes and 75% for the initial filtration step the total theoretical recovery of gold is 99.4%.

Oxide process water is used for the final (second) wash, while barren leach solution is used for the first wash. Barren leach solution is the effluent from the carbon-in-column adsorption unit operation, while the oxide process water consists of the cyanide detoxification effluent combined with raw make-up water.

Filter cake tailings will be repulped in a dedicated mixing tank, using process water from the sulphide plant, before it is pumped to the TMF for final disposal. An inline ‘shark fin’ style sampler and a secondary vezin sampler will be installed on the tailings discharge for metallurgical account purposes. A vertical spindle sump pump will be provided in the HBF area to return spillage to the circuit.

17.6 CYANIDE DESTRUCTION

The purpose of the cyanide destruction step is to ensure that the solution used in the final cake wash contains the least amount of cyanide in order to reduce the load of cyanide reporting to the TMF. The detoxification step provides a bleed-off for other cyanide species, such as thiocyanide and ferrocyanides which may otherwise accumulate indefinitely within the barren leach solution closed loop.

The final wash solution from the HBF will report to the cyanide destruction feed box, along with process water for a bleed of barren solution if required. Cyanide destruction will be carried out using the SO_2 / air process. In the SO_2 / air process, sodium metabisulphite, air, copper sulphate (catalyst) and hydrated lime will be added to oxidize residual free and weak acid dissociable (“WAD”) cyanide to cyanate.

Copper sulphate may also not always be required due to the presence of sufficient copper ions in the slurry. Provision has been made for a dedicated copper sulphate reagent mixing and distribution system for this duty.

The cyanide destruction circuit will consist of two mechanically agitated tanks, operating in parallel, to provide a total residence time of 1.5 hours. Air will be sourced from the low-pressure air header. The cyanide destruction circuit will reduce the residual cyanide contained within the tailings stream to below 1 mg/L (g/m^3) WAD cyanide at the reactor discharge.

Effluent leaving the cyanide destruction reactors will be pumped to the oxide process water tank. A process water sampler consisting of a pressure pipe sampler and a secondary vezin sampler will be installed to allow monitoring of WAD cyanide levels in the cyanide destruction discharge to the oxide process water tank. The WAD cyanide levels of the cyanide destruction discharge sample will be measured using the on-site laboratory.

The cyanide destruction circuit will be serviced by a dedicated vertical spindle sump pump. Any spillage within this area will be returned to the cyanide destruction feed box.

17.7 SART

The SART process will be used to remove copper, silver and mercury from the solution ahead of the gold recovery steps.

Pregnant liquor filtrate from the HBF, combined with the first wash filtrate, will report to the acidification tank. Sulphuric acid will be added to this reactor to lower the pH to approximately 4.5 units. At this pH the weak silver, copper and mercury cyanide complexes disassociate. Sodium hydrosulphide is also added to this reactor to provide the sulphur required to precipitate the metal ions as insoluble metal-sulphides.

The effluent from the acidification tank will gravity flow to the downstream copper thickener. Flocculant will be added to this stream to facilitate the growth and settling of the metallic sulphide particles (principally copper sulphide, Cu_2S). A portion of the thickener underflow will be recycled to the acidification tank as seeding for the precipitate particles forming in the acidification tank. The remainder of the copper thickener underflow will be pumped to a copper filter feed tank where caustic will be added to neutralize the precipitate slurry ahead of the filtration step.

The precipitate filters will be plate and frame filters fitted with filter cloths. The copper rich sulphide precipitate will be bagged and stored. Filtrate from the copper press will be recycled back to the acidification tank.

Supernatant solution from the copper thickener will report to the neutralization tank, where milk of lime and/or sodium hydroxide (caustic) will be added to raise the solution pH back to above 10 units. This step will cause the dissolved hydrogen cyanide ("HCN") formed during the preceding acidification step, to disassociate to CN^- which can then be re-used for leaching. Gypsum ($CaSO_4$) will precipitate at this pH and will be removed in the subsequent gypsum thickener. The gypsum thickener underflow will be pumped back to the leach section so it can be disposed of with the HBF tailings cake. The overflow from the gypsum thickener will be pumped to the carbon adsorption columns for gold recovery.

A poppet sampler will be installed on the barren solution line so that shift and spot samples can be taken for metallurgical accounting and process monitoring purposes.

17.8 CARBON-IN-COLUMN ADSORPTION

The gold and cyanide rich solution (pregnant leach solution) from the SART process' gypsum thickener will be fed to the Carbon-in-Column ("CIC") plant for gold recovery by adsorption onto activated carbon. Solution will flow upwards through the columns, thus fluidizing the carbon inventory in each column into an expanded bed. The columns are staggered to allow the solution to flow by gravity through all the columns.

The CIC plant will consist of six individual columns each loaded with 0.5 t of activated carbon. Every second day loaded carbon will be pumped from tank No.1 to the gold stripping operation (ADR). The carbon inventory in each downstream column will then be pumped sequentially to its upstream column. The carbon transfer sequence is completed when eluted carbon is returned to the 6th column from the ADR Plant.

The CIC circuit was designed for a targeted gold concentration of 0.006 mg/l in the effluent stream returning to the barren leach solution tank. A relatively high loading is expected due to the enhanced adsorption kinetics inherent to the design of columns and the relative absence of competing silver and copper cyanide complexes.

17.9 ACID-WASHING, DESORPTION AND REGENERATION (ADR)

Loaded carbon, at an average rate of 1.75 t/week, will be processed in a 0.5 t/batch carbon stripping circuit.

The stripping circuit will consist of separate acid wash and elution columns. A cold acid wash will be utilized. Following acid washing, gold will be desorbed from the carbon utilizing a pressure Zadra elution process. The elution circuit will be designed to complete one strip per 14-hour period.

17.9.1 Acid Wash

A cold acid wash sequence will be required to remove accumulated, inorganic carbonaceous material from the carbon pores surface. The acid wash column fill sequence will be initiated by taking the first carbon column offline and pumping its entire content to the loaded carbon screen. Carbon will gravitate from the loaded carbon screen directly into the acid wash column while the underflow from this screen will return to the carbon transport water system. Once the acid wash column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will utilize a 3% w/w hydrochloric acid ("HCl") solution. This dilute acid will be prepared by the addition of oxide process water and commercially available hydrochloric acid solution (32% weight by volume ("w/v")), into the hydrochloric acid tank. The acid wash sequence will involve the direct injection of the dilute acid solution into the column by the hydrochloric acid dosing pump, via the feed manifold located beneath the column. Once the required amount of acid has been added to the column, the acid pump will be stopped and the carbon will be allowed to soak for a period of one hour.

Upon completion of the acid soak, the acid wash cycle will be initiated by pumping dilute acid solution through the column for a period of 0 to 1 hour depending on the amount of calcite and gypsum to be removed. The acid solution will be recycled to the dilute acid tank. After completion of this step the acid rinse/neutralization step will be initiated. During the rinse cycle, two bed volumes ("BV") of water, at 2 BV/h, will be pumped through the column. The first BV will include a caustic injection, to neutralize the acid waste, whilst the last BV will be comprised of a raw water rinse only. Neutralized acid waste and displaced solution from both the acid rinse and wash steps will be drained to the acid wash sump from where it will be discharged to either the leach feed distribution box or the repulped tailings tank.

The sequence will conclude with carbon being pumped to the elution column. Water, for carbon transfer between the acid wash and elution columns, will be supplied from the slime cone via the carbon transport water pump.

17.9.2 Elution

The elution sequence will commence with the injection of raw make-up water into the strip solution tank, along with the simultaneous addition of cyanide and caustic solutions. Fixed amounts of cyanide and caustic will be added to achieve a 1% w/v NaOH and 0.2% w/v NaCN strip solution. The pre–heating period will then commence. During this period, the strip solution will be circulated through the first heat exchanger to pre-heat it up to 95°C. Upon completion of pre-heating, the elution sequence will commence, and gold will be stripped from the carbon. During this stripping time barren eluate, from the strip solution tank, will be pumped, through the heat recovery heat exchanger, picking up residual heat from the eluate leaving the elution column.

The pre-heated, incoming eluate will then pass through the primary heat exchanger to elevate the eluate temperature to 135°C prior to entering the base of the column. An electrical elution heater will provide the heat to the heat exchangers. A temperature probe will monitor the temperature of eluate exiting the column, which will be used to control the heater output. Eluate will flow up through the carbon bed and out of the top of the column, passing through the recovery heat exchanger via the elution discharge strainers to the flash tank. Initially, eluate emerging from the heat exchanger will be directed to the pregnant eluate tank. Pregnant eluate will flow by gravity from the flash tank through the electrowinning ("EW") cells, with the barren eluate exiting the EW cells being pumped back to the strip solution tank.

A total of 16 bed volumes of strip solution will be cycled through this closed-circuit comprising of the strip column and EW cells. More bed volumes may be required if the carbon contains higher than expected silver and copper quantities. Upon completion, heating will cease, and cooling water will be injected into the circulating stream for a period of 0.5 hours. This cooling water will displace a portion of the strip solution, which will be bled from the circuit to the leach feed distribution box. Upon completion of the cool down sequence, the eluted carbon will be pumped to the carbon regeneration kiln de-watering screen.

17.9.3 Carbon Regeneration

Eluted carbon will be transferred from the elution column to the carbon regeneration circuit using a transfer pump. The carbon slurry will be directed to the carbon dewatering screen, allowing the removal of the excess of water prior to the carbon discharge into the carbon regeneration kiln feed hopper. The dewatering screen undersize will flow, by gravity, back to the slimes cone.

Carbon will be withdrawn from the carbon regeneration kiln feed hopper, via the kiln screw feeder, and discharged directly to the carbon regeneration kiln, at a rate of 10 kg/h. Within the electrically heated horizontal rotary kiln, the carbon will be heated to 700°C to remove volatile organic foulants from the carbon pores surface, thereby restoring the carbon activity.

Re-activated carbon leaving the kiln will discharge directly to the carbon quench tank, where it will be submerged into water and rapidly cooled. From the quench tank, carbon will be pumped to the carbon desliming sieve bend. The oversize carbon will be pumped back to the CIC columns via a regenerated carbon bin and transfer pump. In turn, the sieve bend undersize will discharge into the slimes cone. Fresh carbon will also be added to the CIC circuit from the quench tank.

17.10 ELECTROWINNING AND GOLD ROOM

Soluble gold and silver will be recovered from the pregnant eluate by electrowinning onto stainless steel cathodes. The electrowinning ("EW") circuit will consist of one electrowinning cell, containing 16 cathodes. An electrical current rectifier assisting the electrowinning cell will supply the current necessary to electroplate the dissolved gold onto the cathode.

Once the elution pre-heating cycle has been completed, the electrowinning sequence will be initiated by diverting strip solution through the closed loop of the elution column and the EW cell. During the electrowinning cycle, the electrowinning cell discharge will be continuously returned to the strip solution tank, by gravity flow.

Upon completion of the electrowinning process, gold sludge on the plated cathodes will be washed off the cathodes with a high-pressure cathode washer. The gold bearing sludge will be recovered to a sludge hopper, from where it will be filtered using a pressure filter. The filter cake will be dried overnight in a small oven.

Once the product has cooled, it will be mixed with fluxes and loaded into the electric arc furnace for smelting. The fluxes will react with base metal oxides to form a slag, whilst the gold and silver will remain as molten metal. The molten metal will be poured into moulds, to form doré ingots, which will be cleaned, assayed, stamped and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the oxide ball mill feed chute by hand.

One 18 litre smelting furnace will be required to process the expected mass of precipitate for the design head grade case at a frequency of one pour per week.

Fume extraction and scrubbing equipment will be provided to remove noxious gases from the smelting furnace. In addition to this, fresh air fans will be provided to ensure there is adequate ventilation inside the gold room.

A sump pump, complete with gold trap, will be installed in the gold room to remove any hose down or spillage.

The gold room will be an 'inside out' design such that the inside of the gold room presents smooth walls for easy cleaning. While it is expected that the SART process will remove mercury from the pregnant solution, thus negating the need for a retort, the floors and sumps in the gold room will nevertheless be coated to minimise impregnation of mercury in concrete areas.

Auxiliary equipment for the gold room will also include:

- Flux bin, platform scale, flux mixing table;
- Gold pouring cascade trolley and slag cart;
- Bullion moulds and bullion cleaning table; and
- Bullion balance.

17.11 SULPHIDE CRUSHING CIRCUIT

ROM mineralized material will be fed into the coarse mineralized material bin above the primary crusher via a front-end loader which will collect mineralized material from the ROM stockpiles or direct tip from the mine haul trucks. A water/glycol spray system will be installed at the coarse mineralized material bin for dust suppression purposes. The coarse mineralized material bin will include a static grizzly with a spacing of 600 mm to prevent the ingress of oversize material. A mobile rock breaker (shared with the oxide coarse mineralized material bin) will be provided to assist in breaking down oversize material retained on the static grizzly. Mineralized material will be withdrawn from the ROM bin by a variable speed vibrating grizzly with a 90 mm aperture size. Oversize from the grizzly will report directly to the primary jaw crusher, which will operate in open circuit. Crushed product from the jaw crusher, along with vibrating grizzly undersize, will report to the sulphide transfer conveyor. The transfer conveyor will be fitted with a weightometer to monitor and control the crushing rate via the vibrating grizzly variable speed control. Discharge from the transfer conveyor reports to an inline covered mineralized material stockpile. The primary crushing area will be serviced by the primary dust collector, which will be comprised of a series of extraction hoods, ducting and a bag house. Dust collected from this system will be discharged onto the transfer conveyor.

Raw water for dust suppression in the ROM and crushing areas will be provided by a dedicated crusher raw water booster tank and pumps.

A sump pump will be provided in the crushing area for spillage and run-off control.

17.12 SULPHIDE GRINDING CIRCUIT

Mineralized material from the covered sulphide mineralized material stockpile will be delivered to the sulphide SAG mill using the sulphide reclaim feeders. These variable speed apron feeders will discharge onto the sulphide mill feed conveyor. The conveyor will be equipped with a weightometer for measuring and controlling the mill feed rate.

A pulse insertable dust collection system will be installed at the transfer point between the feeders and the conveyor to reduce airborne dust around the fine mineralized material reclaim area.

The sulphide grinding circuit will receive mineralized material at a nominal top size of 208 mm with a P_{80} passing size of 100 mm. The circuit will consist of a SAG mill and a ball mill in closed circuit with a cyclone cluster.

The SAG mill will be a 6.10 m diameter x 3.60 m EGL mill with a 2,200 kW variable speed motor. The SAG mill will operate with 7.2% to 15.0% ball charge. Mineralized material will be fed to the SAG mill at a controlled rate, nominally 127.8 dry t/h, and sulphide process water added to the feed chute to achieve the desired milling density (70% solids). Hydrated lime will also be added to the mill feed to ensure adequate mixing and contact with the mineralized material surfaces while providing the correct alkalinity to the slurry. Product from the sulphide SAG mill will discharge over a trommel with the oversize reporting to the scats bunker where it will be periodically removed by the skid-steer loader and returned to the circuit via the clean-up hopper. A diverter gate is installed to divert the oversize to a future pebble crushing circuit. The pebble crushing circuit may be potentially required for processing the harder Tuff mineralized material in the future and layout space has been allowed for installation of this circuit. Trommel undersize will flow by gravity to the sulphide cyclone feed hopper where it will be further diluted to achieve the required cyclone feed density.

The sulphide cyclone feed pump will deliver slurry to the cyclone cluster. Cyclone underflow will flow by gravity to the sulphide ball mill, while cyclone overflow will report to the sulphide trash screen. Also, part of the cyclone underflow will be intermittently diverted to the gravity concentration circuit described below.

The ball mill will be a 4.50 m diameter x 6.85 m EGL overflow mill, with a 2,200 kW fixed speed motor. The mill will operate with between 16.3% and 36% ball charge. Product from the sulphide ball mill will discharge over a trommel, with oversize reporting to the rejects bin. Trommel undersize will gravitate back to the sulphide cyclone feed hopper to be classified again.

Grinding media for the sulphide mills will be introduced by use of a dedicated kibble and the grinding building maintenance crane. Three vertical spindle sump pumps, one located at the feed end of the mills, another at the discharge end of the mills, and the third one nearby the sulphide cyclone feed hopper will service the area. The concrete floor under the mill area will slope to the sumps to facilitate clean-up. In addition, a gold trap will be located in the gravity concentration area.

17.13 GRAVITY CONCENTRATION

Part of the underflow stream from the classifying cyclones will be diverted to the gravity concentration circuit, which consists of a gravity (scalping) feed screen and a centrifugal gravity concentrator. The aim of the gravity circuit is the recovery of any free coarse gold and gold associated with coarse sulphide particles. Sulphide process water will be added into the cyclone cluster launder to facilitate flow to the gravity scalping screen. This screen will remove coarse particles and trash from the slurry stream prior to the gravity concentrator. The screen oversize will be directed to the sulphide ball mill feed chute while the screen undersize will pass through the centrifugal gravity concentrator.

The gravity concentrator will operate in a semi-continuous mode where the cycle consists of 30 minutes of continuous concentration followed by a batch flush of the accumulated concentrate into the gravity concentrate hopper. Raw water will be used for fluidization after passing through dedicated filters. The low mass gravity concentrate will be periodically pumped to the bulk concentrate thickener where it will combine with the bulk flotation concentrate. The gravity tailings will report to the sulphide ball mill feed chute.

17.14 SULPHIDE TRASH SCREENING

Cyclone overflow will gravitate to a vibrating trash screen located in an area adjacent to the cyclone cluster for removal of foreign material and coarse particles prior to flotation. Trash will report to the trash bunker which will be periodically removed for emptying. The trash screen undersize will gravitate to the first stage of the flotation plant, the bulk rougher conditioner tank.

A cross-cut sampler will be installed on the screen undersize (flotation feed) line to take a sample to the On-Stream Analysis ("OSA") package for process control purposes. The screen undersize/flotation feed OSA return stream will be pumped back to the bulk rougher conditioner after analysis and sampling using the sample return pump.

17.15 BULK ROUGHER FLOTATION

Zinc sulphate and sodium cyanide will be added into the bulk rougher conditioner tank for zinc depression. Additional lime can also be added here to adjust the pH. The copper/lead/silver collector, 3418A, will be metered into the conditioner tank using a dedicated pump. Frother will also be dosed into the bulk rougher feed box using a dedicated pump. Process water can be added if required to dilute the feed to the appropriate slurry density.

The bulk rougher flotation cells will consist of six 15 m^3 forced air tank cells in series. Bulk rougher concentrate (copper or lead) will gravitate into the bulk regrind cyclone feed hopper, while the bulk rougher tailings will flow, by gravity, to the zinc rougher conditioner tank No. 1. A pressure pipe sampler will be installed on the discharge of the bulk rougher concentrate pumps to take a sample to the OSA for process control purposes. The bulk rougher concentrate OSA return stream will be pumped back to the bulk rougher conditioner, after analysis and sampling, using the sample return pump No. 1.

Copper or lead minerals will be recovered in the bulk rougher circuit and the tailings will flow, by gravity, to the zinc rougher conditioner tank.

The capability to dose 3418A collector and frother into the head of bulk rougher flotation cell No. 3 will be provided so that stage collector and frother additions can be used if required.

The flotation building gantry crane will be used for all maintenance lifting functions within the flotation area.

A vertical spindle sump pump will service this area for spillage clean-up.

17.16 BULK REGRIND

Bulk rougher concentrate (copper or lead) will normally report to the bulk regrind cyclone feed hopper. Lime will be added to this hopper, along with sulphide process water for slurry density control. The slurry will be pumped to the bulk regrind cyclone clusters by the bulk regrind cyclone feed pump. The cyclone underflow will gravitate to the bulk regrind mill where water will be added to achieve the desired milling density. In turn, the bulk regrind cyclone overflow will flow, by gravity, to the bulk cleaner conditioner tank.

The bulk regrind mill will be a stirred media detritor ("SMD"). Grinding will be achieved via attrition and abrasion of the particles in contact with the high speed, small, circulating media (ceramic beads). Mill discharge will flow by gravity back to the regrind cyclone feed hopper for classification in the regrind cyclones.

The regrind media will be introduced via the bulk regrind media hopper. The bulk media hoist will be installed to allow filling of the bulk regrind media hopper from bulk bags. A davit crane will be supplied at the bulk regrind cyclone tower for removal of cyclones for maintenance.

A vertical spindle sump pump will service this area for spillage clean-up.

17.17 BULK CLEANER FLOTATION

Bulk cleaner flotation (copper or lead) will consist of three stages of closed-circuit cleaning.

Bulk regrind cyclone overflow will flow, by gravity, to the bulk cleaner conditioner tank. Zinc sulphate and sodium cyanide will be added to this tank for zinc depression. Lime slurry can also be added here for trimming of pH if required. The capability to add process water to dilute the slurry to the desired density will also be provided. The collector, 3418A, will be added to the conditioner tank by a dedicated dosing pump. Frother will be dosed to the bulk cleaner 1 flotation feed box by a dedicated pump, if required.

The bulk cleaner 1 flotation cells will consist of seven 5.0 m^3 trough cells in series. Bulk cleaner 1 concentrate will report to the bulk cleaner 1 concentrate froth pump, while the bulk cleaner tailings will flow to the bulk cleaner tailings froth pump. Dual launders will be provided for the first bank of the cleaner 1 flotation cells such that part or the total mass of this concentrate can be diverted to the bulk final concentrate hopper during times of high-grade mineralized material (massive copper mineralized material from the Pinwheel Zone), if required.

The bulk cleaner 1 concentrate will be pumped to the bulk cleaner 2 flotation cells by the bulk cleaner 1 concentrate pumps. A pressure pipe sampler will be installed on the discharge line of the pump to take a sample to the OSA for process control purposes.

The bulk cleaner tailings will be pumped to the zinc rougher conditioner tank No. 1 by the bulk cleaner tailings pumps. A pressure pipe sampler will be installed on the discharge line of these pumps to take a sample to the OSA for process control purposes.

Lime slurry and collector 3418A will be added to the bulk cleaner 2 flotation feed box where these will mix with the bulk cleaner 1 concentrate. The bulk cleaner 2 flotation cells will consist of five 5.0 m^3 trough cells in series. Bulk cleaner 2 concentrate will flow to the bulk cleaner 2 concentrate froth pump, while the bulk cleaner 2 tailings will flow to the bulk cleaner conditioner tank. Dual launders will be provided such that part, or the total of this concentrate, can be diverted to the bulk final concentrate hopper during times of high-grade mineralized material, if required.

The bulk cleaner 2 concentrate will be pumped to the bulk cleaner 3 flotation cells by the bulk cleaner 2 concentrate pumps. A pressure pipe sampler will be installed on the discharge line of these pumps to take a sample to the OSA for process control purposes.

Lime slurry and collector 3418A will be added to the bulk cleaner 3 flotation feed box, if required, where these will mix with the bulk cleaner 2 concentrate. The bulk cleaner 3 flotation cells will consist of eight 1.5 m^3 trough cells in series, arranged in two banks. Bulk cleaner 3 concentrate will report to the bulk final concentrate froth pump, while the bulk cleaner 3 tailings will flow, by gravity, to the bulk cleaner 2 feed box.

The bulk cleaner 3 concentrate will be pumped to the bulk concentrate thickener by the bulk final concentrate pumps. A pressure pipe sampler will be installed on the discharge line of these pumps to take a sample to the OSA for process control and metallurgical accounting purposes.

A vertical spindle sump pump will service this area for spillage clean-up.

17.18 BULK CONCENTRATE THICKENING AND FILTRATION

Bulk final concentrate (copper or lead) will be pumped to the 8 m diameter high rate bulk concentrate thickener, along with filtrate return from the bulk filtration area. Flocculant stock solution will be further diluted with bulk circuit water in an in-line static mixer prior to addition to the concentrate thickener. Thickener overflow will flow, by gravity, to the bulk circuit water tank. From there the bulk thickener overflow will be reticulated to the bulk regrind and bulk cleaners' bulk circuit water pump. Excess bulk circuit water can be directed to the sulphide process water tank, if the plant water balance dictates.

Bulk concentrate thickener underflow, at approximately 60% solids w/w, will be pumped to the agitated bulk concentrate filter feed tank by the bulk concentrate thickener underflow pump. This tank will provide 12 hours of surge capacity between the thickener and the bulk concentrate filter.

A vertical spindle sump pump will be provided to return spillage to the bulk concentrate thickener.

Thickened bulk concentrate (copper or lead) will be pumped batch-wise to the bulk concentrate filter using the bulk filter feed pumps. The filter will remove water from the concentrate to meet the target moisture of approximately 8% w/w using a series of pressing and air blowing steps. After the desired filtration time, the filter press will open and discharge bulk concentrate directly to the floor of the concentrate shed. Following discharge of concentrate, the filter cloth will be washed prior to the next cycle using raw water from the bulk filter cloth wash tank. Filtrate from the bulk concentrate filter will be hydraulically returned to the bulk concentrate thickener.

A vertical spindle sump pump will be provided to return any spillage from the bulk filter area to the bulk concentrate thickener.

A front-end loader ("FEL") will be used to remove the bulk concentrates from beneath the filter press and transfer them to the adjacent concentrate storage areas. Concentrates will be normally loaded into concentrate containers by the FEL when required. During campaigns processing high-grade lead mineralized material from the Tuff Zone, the lead concentrate will be transferred by a FEL to a hopper feeding the lead concentrate bagging system.

Concentrate transport trucks will report to a weighbridge for mass measurement prior to leaving site. The trucks will also pass through a wheel/undercarriage wash system to ensure any residual concentrate is removed. Water from the wheel/undercarriage wash system will be returned to the zinc concentrate thickener by the wheel wash sump pump.

17.19 ZINC ROUGHER FLOTATION

Bulk rougher tailings and bulk cleaner 1 tailings will combine in zinc rougher conditioner tank No. 1. Lime slurry will be added to achieve the desired pH. A cross-cut sampler will be installed on the conditioner feed box discharge line to take a sample to the OSA for process control purposes. The zinc rougher feed, tailings from bulk cleaners 1, 2 and 3 and zinc rougher concentrate OSA return streams will be pumped back to the zinc rougher conditioner tank No. 1, after analysis and sampling, using sample return pump No. 7.

Slurry from zinc rougher conditioner tank No. 1 will flow, by gravity, into conditioner tank No. 2 via an overflow launder. Copper sulphate, for zinc activation, will be added at the slurry entry point, while SIPX collector will be added on the other side of the tank adjacent to the overflow launder. Process water can be added to conditioner tank No. 2 if required to achieve the desired discharge density. Frother will be added to the zinc rougher feed box using a dedicated pump.

The zinc rougher flotation cells will consist of six 20 m^3 forced air tank cells in series. Zinc rougher concentrate will flow, by gravity, into the zinc regrind cyclone hopper. A pressure pipe sampler will be installed on the discharge of the zinc rougher concentrate pumps to take a sample to the OSA for process control purposes.

The zinc rougher tailings will gravitate to the flotation tailings hopper. An in-line 'shark fin' style sampler will take a sample to the OSA for process control purposes.

The circuit capability to dose SIPX and frother into the head of zinc rougher flotation cell No. 3 will be provided so that stage collector and frother additions can be used if required.

During campaigns part of the zinc rougher concentrate can be diverted to bypass the zinc regrind stage and pumped directly to the zinc concentrate thickener. The two first zinc rougher flotation cells will be provided with dual launders to facilitate the concentrate bypass operation.

A vertical spindle sump pump will service this area for spillage clean-up.

17.20 ZINC REGRIND

Zinc rougher concentrate will discharge by gravity into the zinc regrind cyclone feed hopper. Lime will be added to this hopper, along with process water for slurry density control. During treatment of high-grade massive copper mineralized material from the Pinwheel Zone, the zinc regrind mill will be used for copper regrinding.

The slurry will be pumped from the hopper to the zinc regrind cyclone clusters by the regrind cyclone feed pumps. The cyclone underflow will gravitate to the zinc regrind mill where water will be added to achieve the desired milling density. In turn, the zinc regrind cyclone overflow will flow, by gravity, to the zinc cleaner conditioner tank.

A SMD will be used for regrinding zinc rougher concentrate. Grinding will be achieved via attrition and abrasion of the particles in contact with the high speed, small, circulating media (ceramic beads). Mill discharge will flow, by gravity, back to the regrind cyclone feed hopper for classification in the regrind cyclones.

Media will be introduced via the zinc regrind media hopper. The zinc media hoist will be installed to allow filling of the zinc regrind media hopper from bulk bags. A davit crane will be supplied at the zinc regrind cyclone tower for removal of cyclones for maintenance.

A vertical spindle sump pump will service this area for spillage clean-up.

17.21 ZINC CLEANER FLOTATION

Zinc cleaner flotation will consist of two stages of closed-circuit cleaning.

Zinc regrind cyclone overflow will flow, by gravity, to the zinc cleaner conditioner tank. Copper sulphate will be added to the conditioner tank to activate any new zinc surfaces exposed by regrinding. Lime slurry can be added for trimming of the pH if required. SIPX and frother will also be added prior to zinc cleaner flotation.

During treatment of high-grade copper mineralized material from the Pinwheel Zone, the zinc regrind and zinc cleaner circuits will be used for copper regrind and cleaner flotation, therefore the ability to add zinc sulphate and sodium cyanide to the zinc cleaner conditioner tank will also be provided.

The zinc cleaner 1 flotation cells will consist of one bank of four 10 m^3 trough cells followed by one bank of four 10 m^3 trough cells in series (8 cells in total). Zinc cleaner 1 concentrate will flow to the zinc cleaner 1 concentrate froth pump, while the zinc cleaner tailings will report to the zinc cleaner tailings froth pump. Dual launders will be provided for the first bank of the cleaner 1 flotation cells such that part or the total mass of this concentrate can be diverted to the zinc final concentrate froth pump during times of high-grade mineralized material, if required.

The zinc cleaner 1 concentrate will be pumped to the zinc cleaner 2 flotation cells by the zinc cleaner 1 concentrate froth pump. A pressure pipe sampler will be installed on the discharge line of the zinc cleaner 1 concentrate froth pump to take a sample to the OSA for process control purposes.

The zinc cleaner tailings will be pumped to the flotation tailings hopper by the zinc cleaner tailings pump. A pressure pipe sampler will be installed on the discharge line of the pump to take a sample to the OSA for process control purposes.

Lime slurry and SIPX will be added to the zinc cleaner 2 flotation feed box where they will mix with the zinc cleaner 1 concentrate. The zinc cleaner 2 flotation cells will consist of nine 5 m^3 trough cells in series, arranged in two banks. Zinc cleaner 2 concentrate will flow, by gravity, to the zinc cleaner 2 concentrate froth pump, while the zinc cleaner 2 tailings will report to the zinc cleaner conditioner tank.

The zinc final concentrate will be pumped to the zinc concentrate thickener by the zinc final concentrate froth pump. A pressure pipe sampler will be installed on the discharge line of the pump to take a sample to the OSA for process control and metallurgical accounting purposes.

A vertical spindle sump pump will service this area for spillage clean-up.

17.22 ZINC CONCENTRATE THICKENING AND FILTRATION

Zinc final concentrate will be pumped to the 12 m diameter high rate zinc concentrate thickener, along with filtrate return from the zinc filtration area. Flocculant stock solution will be further diluted with sulphide process water in an in-line static mixer prior to addition to the concentrate thickener. Thickener overflow will gravitate to the sulphide process water tank for re-use in the process.

Zinc concentrate thickener underflow, at approximately 60% solids (w/w), will be pumped to the agitated zinc concentrate filter feed tank by the zinc concentrate thickener underflow pump. This tank will provide 12 hours of surge capacity between the thickener and filter. Zinc concentrate will be pumped to the zinc concentrate filter by the zinc filter feed pump.

A vertical spindle sump pump will be provided to return spillage to the zinc concentrate thickener.

Thickened zinc concentrate will be pumped batch wise to the zinc concentrate filter using the zinc filter feed pump. The filter will remove water from the concentrate to meet the target moisture content of approximately 8% (w/w) using a series of pressing and air blowing steps. After the desired filtration time, the filter press will open, and discharge zinc concentrate directly to the floor of the concentrate shed.

Following discharge of concentrate, the filter cloth will be washed prior to the next cycle using raw water from the zinc filter cloth wash tank. Filtrate from the zinc concentrate filter will be hydraulically returned to the zinc concentrate thickener. A sump pump will be provided to return any spillage from the zinc filter area to the zinc concentrate thickener.

A FEL will be used to remove the zinc concentrates from beneath the filter press and transfer them to the adjacent concentrate storage areas. Concentrates will be loaded into concentrate containers by the FEL when required.

Concentrate transport trucks will report to a weighbridge for mass measurement prior to leaving site. The trucks will also pass through a wheel/undercarriage wash system to ensure any residual concentrate is removed.

Water from the wheel/undercarriage wash system will be returned to the zinc concentrate thickener by the wheel wash sump pump.

17.23 SULPHIDE TAILINGS AND SULPHIDE PROCESS WATER

Zinc rougher and zinc cleaner tailings will combine with the OSA tailings return streams and several intermittent reagent sump pump streams to the flotation tailings hopper which will be in the flotation building.

Flotation tailings will be transferred to the 15 m diameter high compression type sulphide tailings thickener by the flotation tailings pump. An in-line ‘shark fin’ style sampler will be used to take a sample to the OSA for process control and metallurgical accounting purposes.

Sulphide tailings thickener underflow, at approximately 65% to 76% solids (w/w), will be pumped to the TMF by the sulphide tailings thickener underflow pump. Part of these sulphide tailings will be diverted to service the paste plant when the underground mine will be developed.

Sulphide tailings thickener overflow will flow, by gravity, to the sulphide process water tank, along with zinc thickener overflow, any excess bulk circuit water, tailings decant water, and raw water as make-up if required. This process water will be distributed to the sulphide grinding circuit, bulk and zinc roughers, zinc regrind mill, zinc cleaners, oxide tailings repulping, and sulphide plant service points by the sulphide process water pump.

A vertical spindle sump pump will be provided to return spillage to the sulphide tailings thickener.

17.24 REAGENTS AND CONSUMABLES

Reagent mixing for both the sulphide and oxide process plants will be located in a central location as a number of reagents are used in both plants, with the exception of flocculants, which will be located nearby each of the thickeners.

17.24.1 Sodium Cyanide

Sodium cyanide will be delivered as briquettes in double skinned bulk bags or contained in boxes and stored in the reagent shed. Oxide process water will be added to the agitated cyanide mixing tank. Caustic Soda (sodium hydroxide) will be added to provide protective alkalinity to avoid generation of hydrogen cyanide gas. Bags will be lifted into the cyanide bag breaker, located on top of the tank, using the reagents building overhead crane. The solid reagents will fall into the tank and be dissolved in water to achieve the required concentration. After mixing for a pre-set time period, cyanide solution will be transferred to the cyanide storage tank using the cyanide transfer pump.

Cyanide will be delivered to the flotation and leach circuits using the cyanide circulation pump and a ring main system. Actuated control valves will provide the required cyanide flowrates at several locations around the two plants.

The cyanide mixing area will be ventilated using the cyanide area roof fan.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.24.2 Sodium Metabisulphite (SMBS)

SMBS will be delivered in powder form in bulk bags and stored in the reagent shed. Oxide process water will be added to the agitated SMBS mixing tank. Bags will be lifted into the SMBS bag breaker, located on top of the tank, using the SMBS lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required concentration. After mixing for a pre-set time period, SMBS solution will be transferred to the SMBS storage tank using the SMBS transfer pump.

SMBS will be delivered to the cyanide destruction tanks using the SMBS circulation pumps via a ring main pipe system.

An extraction fan will be provided over the SMBS mixing tank to remove any SO_2 gas that may be generated during mixing. The SMBS mixing area will be ventilated using the SMBS area roof fan.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.24.3 Copper Sulphate

Copper sulphate will be delivered in powder form in bulk bags and stored in the reagent shed. Oxide process water will be added to the agitated copper sulphate mixing tank. Bags will be lifted into the copper sulphate bag breaker, located on top of the tank, using the copper sulphate lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required dosing concentration. Copper sulphate solution will be transferred to the copper sulphate storage tank using the copper sulphate transfer pump.

Copper sulphate will be delivered to the flotation and cyanide destruction circuits using the copper sulphate circulation pump and ring main.

The copper sulphate mixing area will be ventilated using the copper sulphate area roof fan.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.24.4 Oxide Plant Flocculant

Flocculant will be delivered to site in shrink wrapped, 25 kg bags on a 36-bag pallet, and stored in the reagent shed. A vendor supplied mixing and dosing system will be installed, which will include flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant will be loaded into the flocculant storage hopper by hand. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with oxide process water. Flocculant solution, at 0.25% w/v will be agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant will be transferred to the flocculant storage tank using the flocculant transfer pump.

Flocculant will be dosed to the HBF, pre-leach, copper and gypsum thickeners using dedicated variable speed helical rotor style pumps. Flocculant will be further diluted to approximately 0.025% w/v prior to the addition point.

A dedicated vertical spindle sump pump will be provided in this area.

17.24.5 Activated Carbon

Activated Carbon will be delivered to site in 1.2 tonne bags and will be added manually via the quench tank to top-up the batch inventory as required. Adding it through the quench tank allows for some pre-attritioning when it is pumped to the desliming/sizing screen. This screen will remove fines produced in this way prior to its addition to the CIC circuit.

17.24.6 Sodium Hydrosulphide

Sodium hydrosulphide solution, at 40% w/v strength, will be delivered to site in standard 210 L (55 gallon) drums. It will be added as pure product to the SART acidification tank by an air operated drum pump directly connected to the 55-gallon drum. Sodium hydrosulphide vapours will be collected by a local suction chamber and treated through the SART scrubber.

17.24.7 Sulphuric Acid

Concentrated sulphuric acid (98% w/w) will be delivered in isotainers to site. A centrifugal pump will be used to transfer the concentrated acid to a partially filled sulphuric acid tank. From here the diluted acid will be pumped to the SART acidification tank. Raw Water will be used to dilute the concentrated sulphuric acid.

17.24.8 Sodium Hydroxide (Caustic Soda)

Sodium hydroxide pearls or beads will be delivered in 25 kg bags to site. Oxide process water will be used to fill the caustic soda mixing tank to a predetermined level before adding the pearls through a bag breaker into the agitated tank. A set number of bags will be added while continuously stirring the caustic solution to ensure complete dissolution of all pellets.

Caustic soda will be delivered to the various end users through a centrifugal pump pressurised header line with T-offs to each user. Control valves will be used to add a predetermine flow rate to each end-user.

17.24.9 Hydrochloric Acid

Hydrochloric acid will be delivered to site in 1 m^3 totes. It will be pumped into the dilute HCL tank through an air-operated drum pump. This pump will deliver a set predetermined volume of the 32% w/v neat acid into a pre-filled tank. The diluted acid (3% w/v) will be pumped through the acid-washing column by a dedicated centrifugal pump.

The hydrochloric tote, dilute acid tank and acid-wash column will be placed inside a dedicated bunded area that will contain any spillage and will prevent contact with cyanide bearing solutions. A sump pump in this bunded are will discharge any spillage as well as the spent acid solution, after completion of the acid wash sequence, to the tailings repulp tank.

17.24.10 Hydrated Lime

Hydrated lime will be delivered to site in a tanker and will be pneumatically conveyed from the tanker to the 170-tonne lime storage silo. The lime will be extracted from the lime storage silo via a rotary valve and screw feeder, and discharge into the lime slurry mixing and storage tank. Oxide process water will also be added to the slurry mixing and storage tank to achieve the desired lime density.

The lime slurry from the lime storage tank will be distributed throughout the process plant by the lime slurry circulation pump and a ring main, with take-offs distributing lime to the process as required.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.24.11 3418A (Collector)

3418A will be delivered in intermediate bulk containers ("IBC") or boxes, and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. 3418A will be dosed as received, without dilution. Multiple diaphragm style dosing pumps will deliver the reagent to the required locations within the flotation circuit. Top-up of the permanent bulk box will be carried out manually as required.

17.24.12 MIBC (Frother)

MIBC will be delivered in bulk containers or boxes (IBC) and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. MIBC will be dosed as received, without dilution. Multiple diaphragm style dosing pumps will deliver the reagent to the required locations within the flotation circuit. Top up of the permanent bulk box will be carried out manually as required.

A single air operated diaphragm sump pump will be provided for spillage control in the 3418A and MIBC areas, which will be located adjacent to each other.

17.24.13 SIPX (Collector)

SIPX will be delivered in pellet form in bulk bags within boxes and stored in the reagent shed. Raw water will be added to the agitated SIPX mixing tank. Bags will be lifted into the SIPX bag breaker, located on top of the tank, using the SIPX lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required dosing concentration (10% w/v). SIPX solution will be transferred to the SIPX storage tank using the SIPX transfer pump.

Both the mixing and storage tanks will be ventilated using the SIPX tank fan to remove carbon disulphide gas.

SIPX will be delivered to the flotation circuit using the SIPX circulating pump and a ring main system. Actuated control valves will provide the required SIPX flowrates at various locations around the zinc flotation circuit.

The SIPX mixing area will be ventilated using the SIPX area roof fan.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.24.14 Zinc Sulphate

Zinc sulphate will be delivered in powder form in bulk bags and stored in the reagent shed. Raw water will be added to the agitated zinc sulphate mixing tank. Bags will be lifted into the zinc sulphate bag breaker, located on top of the tank, using the zinc sulphate lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required concentration (20% w/v). Zinc sulphate solution will be transferred to the zinc sulphate storage tank using the zinc sulphate transfer pump. The mixing tank will be ventilated using the zinc sulphate tank fan.

Zinc sulphate will be delivered to the sulphide plant using the zinc sulphate circulating pump and a ring main system. Actuated control valves will provide the required zinc sulphate flowrates at various locations around the circuit.

The zinc sulphate mixing area will be ventilated using the zinc sulphate roof fan.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.24.15 Sulphide Plant Flocculant

Powdered flocculant for the sulphide process plant will be delivered to site in 25 kg bulk bags and stored in the reagent shed. A vendor supplied mixing and dosing system will be installed, which will include flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant will be loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with raw water. Flocculant solution, at 0.25% (w/v) will be agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant will be transferred to the flocculant storage tank using the flocculant transfer pump.

Flocculant will be dosed to the bulk concentrate thickener, zinc concentrate thickener, and tailings thickener using variable speed helical rotor style pumps. Flocculant will be further diluted to approximately 0.025% w/v just prior to the addition point.

A dedicated vertical spindle sump pump will be provided in this area.

17.24.16 Anti-scalant

Anti-scalant will be delivered in liquid form in bulk totes and stored in the reagent shed until required. Permanent bulk boxes will be installed to provide storage capacity local to each dosing point. Anti-scalant will be dosed neat, without dilution. Positive displacement style dosing pumps will deliver the anti-scalant to the required locations around the oxide and sulphide process plants. Top up of the permanent bulk boxes will be carried out manually as required. Sulphamic acid will be used to descale the elution heat exchangers as required.

17.25 WATER CIRCUITS

Process water used in the plant will be sourced from different locations. However, a strategy of use and re-use of water will prioritize the use of tailings decant water and then open pit (mine) dewatering water or treated effluent water at the raw water tank.

17.25.1 Raw Water

Raw water supplied from the open pit dewatering system will be delivered to the raw water settling tanks for gross removal of solids and to an oil/water separator for gross removal of hydrocarbons. The raw water transfer pump will pump sediment and oil free water to the raw water tank. Approximately 47 m^3/h of raw water is required on average with a peak intermittent consumption of 62 m^3/h.

Raw water will be used for the following duties:

- Low pressure gland water, using the low-pressure gland water pump;
- General process uses in the crushing, stockpile, gravity concentration and filtration areas via the raw water pump;
- Reagent make-up via the raw water pump;
- Oxide and sulphide process water make-up via the raw water pump; and
- Cooling water, via the raw water pump.

17.25.2 Fresh Water

Fresh water will be supplied by a number of fresh water supply pumps remote from the plant. The fresh water pumps will pump directly to the fresh water tank. Fresh water will be used to supply the potable water treatment plant via the potable water treatment plant feed pump and the fire water tank via the fire water supply pump. In addition, fresh water will be supplied as contingency to raw water for first fills and if insufficient site water resources are available for re-use. Approximately 2.5 m^3/h of fresh water is required on average with a peak intermittent consumption of 3.3 m^3/h.

17.25.3 Potable Water

The potable water treatment plant will be a vendor package. The plant potable water tank will be used to store potable water for use in the OSA, site buildings and site ablutions. A separate safety shower water tank and ring main system will be installed to provide water to the safety showers and drinking fountains around the plant.

17.25.4 Fire Water

Fire water will be delivered using a vendor package which will include a fire water pump, a fire water jockey pump and a diesel fire water pump.

17.25.5 TMF Decant Water and Contact Water Basin

Oxide and sulphide tailings will be pumped to the tailings management facility ("TMF"). TMF decant water will be pumped to the sulphide process water tank by the pontoon-mounted decant return pumps. Excess decant water can be diverted to the Contact Water Basin ("CWB"), if required. An uninstalled spare decant pump will also be provided. Leachate from TMF will be pumped to the CWB by TMF leachate collection pump. An uninstalled spare leachate pump will also be provided. Water/leachate will be pumped to the CWB by two NWRF and one SWRF leachate pumps. Three uninstalled spare pumps will also be provided for NWRF and SWRF leachate collection.

Water from the mineralized material blending area will be transferred to the CWB by the contact water pump.

Water from mineralized material stockpile pad 1 and 2 will be transferred to CWB by the mineralized material stockpile sump pumps. Also, collection water along the haul road will be transferred to CWB by sump pumps.

The CWB will collect excess mine dewatering water, excess TMF decant, TMF leachate, NWRF and SWRF leachate, mineralized material stockpile pad water, haul road water, plant site water, plant diesel area sump pump discharge, mine truck wash water, as well as streams returned from the waste water treatment plant ("WWTP"). A single contact water basin will contain two cells. Water from the CWB will be pumped to the WWTP by the WWTP feed pump. Treated water from this WWTP will be collected in a treated water tank and underflow from the WWTP clarifier will be discharged to the sulphide tailings thickener. Treated water from treated water tank will be distributed for re-use at the mine, or discharged to the environment, or supplied as contingency to raw water for first fills and if insufficient site water resources are available for re-use.

17.26 SERVICES AND UTILITIES

17.26.1 On-stream Analysis System

The performance of the flotation circuit will be monitored by a dedicated On-Stream Analysis ("OSA") system, to allow the operator to make air, level or reagent changes based on real time assays. All the major streams will be monitored by the OSA. Analysis will include percent solids, copper, iron, lead, zinc and silver assays.

Cumulative shift samples for laboratory analysis will also be collected via the OSA sampling system. The system will have a stand-alone control, calibration and reporting system but will have the capacity to provide assay data to the plant control system if required.

Process streams that will be analysed are listed as follows:

- Screen undersize/flotation feed;
- Bulk rougher concentrate;
- Zinc rougher feed;
- Zinc rougher concentrate;
- Zinc rougher tailings;
- Bulk cleaner 1 concentrate;
- Bulk cleaner 2 concentrate;
- Bulk final concentrate;
- Bulk cleaner 1 tailings;
- Bulk cleaner 2 tailings;
- Bulk cleaner 3 tailings;
- Zinc cleaner feed;
- Zinc cleaner 1 concentrate;
- Zinc final concentrate;
- Zinc cleaner 1 tailings;
- Zinc cleaner 2 tailings; and
- Flotation tailings.

Samples will be collected using a combination of sample pumps, pressure pipe samplers and linear samplers as required. Samples will be logically combined after analysis and returned to the bulk rougher conditioner tank, the flotation tailings hopper, bulk cleaner 2 feed, bulk concentrate thickener, zinc cleaner 2 feed, zinc concentrate thickener and zinc rougher conditioner tank No. 1 using vertical spindle style pumps.

17.26.2 High- and Low-Pressure Air

High-pressure air at 700 kPa(g) will be provided by two high-pressure air compressors, operating in a lead-lag configuration. The entire high-pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via the main plant air receiver, with additional receivers in the crushing, grinding and concentrate filtration areas.

Low-pressure air at 130 kPa(g) for the leach tanks and at 50 kPa(g) for cyanide destruction tanks will be supplied by a dedicated blower. A second blower will supply air at 50 kPa(g) to the flotation circuit.

17.27 SAMPLING AND METALLURGICAL ACCOUNTING

17.27.1 Oxide Plant

A weightometer on the oxide transfer conveyor will measure the instantaneous and totalized shift crushed mineralized material tonnage. A weightometer on the oxide mill feed conveyor will measure the instantaneous and totalised shift oxide mill feed tonnes.

A density and flowmeter on the tailings line will allow the dry tonnage of solids pumped to the TMF to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer.

Automatic samplers will be installed in the following locations:

- Pre-leach thickener feed sampler;
- Leach tank discharge sampler;
- Final Tailings sampler; and
- Solution samples will be taken manually from around the circuit e.g. the pregnant and barren solutions and cyanide destruction discharge.

Regular 'gold and silver in circuit' surveys will allow reconciliation of precious metals in feed compared to doré production.

17.27.2 Sulphide Plant

A weightometer on the sulphide transfer conveyor will measure the instantaneous and totalized shift crushed mineralized material tonnage delivered to the sulphide crushed mineralized material stockpile. A weightometer on the sulphide mill feed conveyor will measure the instantaneous and totalised shift sulphide mill feed tonnes.

A density and flow meter on the tailings line will allow the dry tonnage of solids pumped to the TMF to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer.

All other metallurgical and process sampling will be carried out via the OSA. Sampling stations will be included as part of the OSA vendor package.

Weigh frames will be included as part of the filter vendor supply so that the batch weight of bulk and zinc concentrates can be determined. Together with the concentrate moistures, this will allow the daily concentrate production to be calculated. A reconciliation can then be made with the calculated production rates based on stream assays, feed tonnage and the three-product formula.

Ad hoc manual sampling of the concentrate stockpiles will be carried out to determine required blend ratios for loading of concentrate containers. Concentrate containers will be sampled manually prior to leave site. Concentrate trucks and containers will be weighed before and after leaving site to determine the tonnage of concentrate trucked.

17.28 ENERGY REQUIREMENTS

The process plant plants will be operated using electricity provided by grid power. Due to the different characteristics of the Pinwheel, Main and Tuff material types, different power usage is required. Annual power usage is summarized in Table 17.5.

TABLE 17.5
ANNUAL POWER USAGE

Usage Site	Annual Power Consumption (kWh)
Oxide Plant	8,195,330
Sulphide Plant (processing Pinwheel)	50,508,320
Sulphide Plant (processing Main)	48,804,413
Sulphide Plant (processing Tuff)	55,446,070

17.29 CONSUMABLES

Annual process plant consumables are summarized in Tables 17.6 and 17.7

TABLE 17.6
ANNUAL CONSUMABLES – OXIDE PLANT

Equipment/Area	Item	Consumption
Primary Jaw Crusher	Fix Jaw	3.2 sets/year
	Swing Jaw	2.0 sets/year
	Upper Cheek Plate	0.9 sets/year
	Lower Cheek Plate	1.4 sets/year
Grizzly Feeder	Bars	4.6 sets/year
Secondary Crusher	Mantle	5.3 sets/year
	Bowl Liner	5.3 sets/year
Tertiary Crusher	Mantle	5.3 sets/year
	Bowl Liner	5.3 sets/year
Screening	Top Deck	1.4 sets/year
	Bottom Deck	0.9 sets/year
Ball Mill	Liners	0.6 sets/year
	Grinding Media	1.54 kg/t
Cyclone Spare Parts	-	1.0 set/year
Trash Screen Spares	-	2.0 sets/year
Pre-leach and Leach	Flocculant	0.04 kg/t
	Sodium Cyanide	1.00 kg/t
	Hydrated Lime	1.26 kg/t
Filtration	Filter Cloth	1.8 sets/year

TABLE 17.6
ANNUAL CONSUMABLES – OXIDE PLANT

Equipment/Area	Item	Consumption
	Flocculant	0.02 kg/t
SART	Sulphuric Acid	1.88 kg/t
	Sodium Hydrosulphide	0.10 kg/t
	Sodium Hydroxide	0.02 kg/t
	Flocculant	0.008 kg/t
	Hydrated Lime	1.52 kg/t
	Antiscalant	2.62 t/year
ADR	Carbon Safety Screen Panels	1.0 sets/y
	Carbon Dewatering Screen Panels	1.0 sets/y
	Loaded Carbon Screen Panels	1.0 sets/y
	Carbon Desliming Screen Panels	1.0 sets/y
Acid Washing	Hydrochloric Acid	0.07 t/strip
Elution and Electrowinning	Sodium Cyanide	6.4 kg/strip
	Sodium Hydroxide (Caustic)	32.0 kg/strip
	Activated Carbon	0.014 kg/t
	Antiscalant	0.0002 kg/t
Cyanide Destruction	Sodium MetaBisulphite	0.52 kg/t
	Copper Sulphate	Not required
	Hydrated Lime	0.28 kg/t
	Antiscalant	2.84 t/y
Gold Room	Borax	13 kg/smelt
	Silica Sand	6 kg/smelt
	Nitre - Sodium Nitrate	2 kg/smelt
	Soda Ash	2 kg/smelt
	Crucibles	20 pours/unit

TABLE 17.7 ANNUAL CONSUMABLES – SULPHIDE PLANT				
Equipment/Area	**Item**	**Consumption Main**	**Consumption Pinwheel**	**Consumption Tuff**
Primary Jaw Crusher	Fix Jaw	9.9 sets/year	7.2 sets/year	15.8 sets/year
	Swing Jaw	5.3 sets/year	3.9 sets/year	8.5 sets/year
	Upper Cheek Plate	2.6 sets/year	2.0 sets/year	3.9 sets/year
	Lower Cheek Plate	4.6 sets/year	3.9 sets/year	7.9 sets/year
Grizzly Feeder	Bars	3.9 sets/year	3.3 sets/year	5.9 sets/year
SAG Mill	Liners	1.2 sets/year	1.1 sets/year	1.4 sets/year
	Grinding Media	0.40 kg/t	0.37 kg/t	0.82 kg/t
Ball Mill	Liners	1.1 sets/year	1.2 sets/year	1.9 sets/year
	Grinding Media	0.59 kg/t	0.63 kg/t	1.05 kg/t
Cyclone Spare Parts	-	1.0 set/year	1.0 set/year	1.0 set/year
Flotation	Trash Screen Spares	2.0 sets/year	2.0 sets/year	2.0 sets/year
	Zinc Sulphate Heptahydrate	0.11 kg/t	0.25 kg/t	0.14 kg/t
	Sodium Cyanide	0.04 kg/t	0.08 kg/t	0.05 kg/t
	3418A	0.13 kg/t	0.08 kg/t	0.07 kg/t
	Copper Sulphate Pentahydrate	0.52 kg/t	0.65 kg/t	0.43 kg/t
	SIPX	0.05 kg/t	0.07 kg/t	0.04 kg/t
	Hydrated Lime	5.61 kg/t	7.41 kg/t	4.16 kg/t
	MIBC	0.10 kg/t	0.11 kg/t	0.10 kg/t
Bulk Regrind Mill	Liners	0.5 sets/year	0.5 sets/year	0.5 sets/year
	Grinding Media	0.05 kg/t	0.05 kg/t	0.05 kg/t
	Bulk Regrind Cyclone Cluster Spare Parts	1.0 sets/year	1.0 sets/year	1.0 sets/year
Zinc Regrind Mill	Liners	0.5 sets/year	0.5 sets/year	0.5 sets/year
	Grinding Media	0.05 kg/t	0.05 kg/t	0.05 kg/t
	Zinc Regrind Cyclone Cluster Spare Parts	1.0 sets/year	1.0 sets/year	1.0 sets/year
Concentrate Thickening	Bulk Concentrate Flocculant	0.03 kg/t	0.03 kg/t	0.03 kg/t
	Zinc Concentrate Flocculant	0.03 kg/t	0.03 kg/t	0.03 kg/t
Tailings Thickening	Flocculant	0.03 kg/t	0.03 kg/t	0.03 kg/t
	Antiscalant	21.5 t/year	21.5 t/year	21.5 t/year

17.30 PROCESS PLANT PERSONNEL

The personnel for the oxide and sulphide process plants will consist of management, technical support, shift supervision, laboratory staff, operators and maintenance staff. The management and technical support staff will work five 8-hour days per week with weekends off. Shift supervision, laboratory staff, shift operators and maintenance will work 12-hour day and night shifts on a 7-day work cycle, using four rotating crews.

Annual process plant personnel requirements are provided in Table 17.8.

TABLE 17.8
ANNUAL PROCESS PLANT PERSONNEL REQUIREMENTS

Position	Classification	Employees per Team	Number of Teams	Number of Employees
Process Manager	M	1	1	1
Sulphide Process Plant				
Plant Metallurgist	SD	0.5	1	0.5
Shift Supervisor	SS	1	4	4
Crusher Operators	SS	1	4	4
Milling Operators /Control Room	SS	1	4	4
Flotation / Regrind Operators	SS	1	4	4
Concentrate Dewatering Operators	SS	1	4	4
Concentrate Loader Operator	SD	1	1	1
Reagent / Daycrew / Relief Operators	SS	1	1	1
Oxide Process Plant				
Plant Metallurgist	SD	0.5	1	0.5
Shift Supervisor	SS	1	4	4
Crusher Operators	SS	1	4	4
Milling / Leach / Filtration Operators	SS	1	4	4
SART/Detox/CIC/ADR Operators	SS	1	4	4
Goldroom Operators	SS	1	4	4
Reagent / Daycrew / Relief Operators	SS	1	1	1
Laboratory				
Analysts - Oxide	SS	0.5	4	2
Analysts - Sulphide	SS	0.5	4	2
Sample Prep	SS	1	2	2
Maintenance				
Maintenance Manager	M	1	1	1
Maintenance Planner	SD	1	1	1
Maintenance Supervisors	SD	1	1	1
Boilermakers / Fitters	SS	1	4	4
Trades Assistants	SS	1	4	4
Electricians	SS	1	4	4
Instrument Technician	SD	1	1	1
Total				**67**

Note: *Classifications: M = Manager, SD = Day shift only, SS = Rotating night shift / day shift, Management and Day shift are weekday staff.*

18.0 PROJECT INFRASTRUCTURE

18.1 OVERALL SITE

The overall Project site plan is shown in Figure 18.1 and includes major facilities including the open pit mine, mineralized material stockpiles, oxide and sulphide process plants, tailings management facility ("TMF"), waste rock facilities ("WRF"), cut-off wall ("COW"), contact water basin ("CWB"), non-contact water basins ("NCWB"), waste water treatment plant ("WWTP"), mine services, overburden stockpile and access road.

Prior to commencing underground mining, a paste backfill plant will be installed near the open pit mine to provide cemented paste for backfill requirements.

Access to the Project is from the east side of the Property from the existing County Road 356. Main access will be via the main security gate near the process plant.

Grid power will be provided from an incoming high voltage ("HV") line from the east side of the Project along the main access road.

The site will be fenced to clearly delineate the Project area and deter access by unauthorized people.

18.2 ROADS

18.2.1 Access to Site

Access to the Project is via a new gravel road that will connect from the existing County Road 356 (under the jurisdiction of the Menominee County Road Commission) to the east side of the Project. The new road will be approximately 3 km in length and 11 m wide (lanes and shoulders). The access road will be designed for all weather and all-season access to the Project and have a 55 kph speed limit. Once Project and mine development commences, the existing River Road will be abandoned.

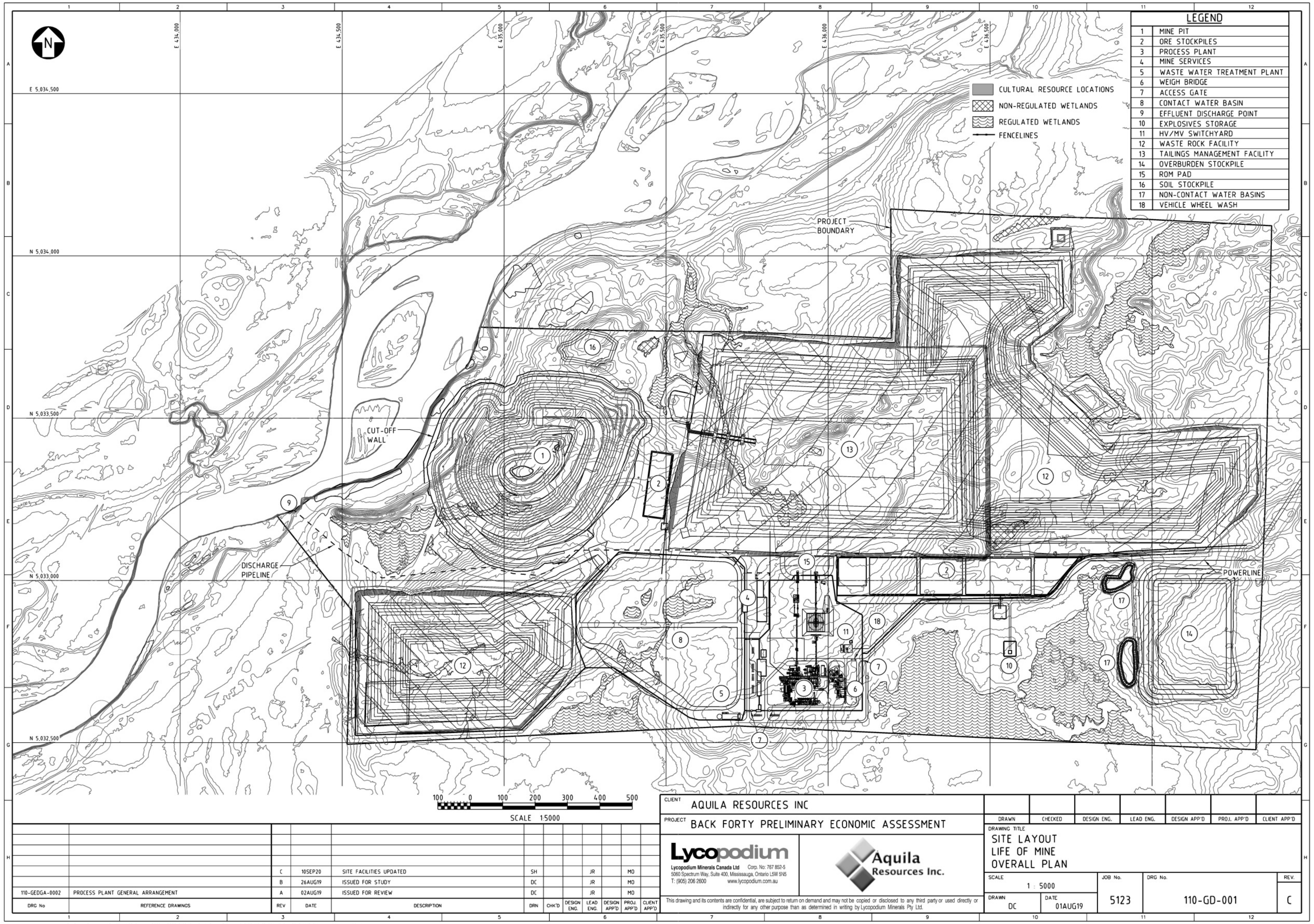
18.2.2 Project Site Roads

Project internal roads will provide access between the administration area, process plant facilities and mine services area. These roads will generally be 6 m wide and will be constructed flush with bulk earthworks pads to ensure that storm water sheet flow is achieved across the site, thereby avoiding the need for deep surface drains and culvert crossings within the Project area.

18.2.3 Other Access Routes

A number of new access routes will be constructed to access infrastructure such as the TMF and WRFs, overburden stockpile, NCWBs, and fresh water supply pumps remote from the process plant site. These access routes will be cleared and graded natural earth tracks. Exact routing will be determined during construction of the Project to best fit local terrain and vegetation density.

FIGURE 18.1 OVERALL PROJECT SITE PLAN



Source: *Lycopodium Minerals Canada Ltd. (2019)*

18.3 POWER SUPPLY

Site power will be provided from a high voltage ("HV") line at the Project's boundary that will be provided by the local power authority. A peak demand of 9.4 MW with an average load of 7.5 MW for full production is required for the facility. When underground mining commences, addition power is required, i.e. peak demand and average load increases to 13.0 MW and 10.5 MW, respectively.

The sulphide SAG and ball mills at the process plant are the largest loads. The sulphide SAG mill has been specified with a variable speed drive and sulphide ball mill with liquid resistance starter ("LRS") to reduce the load surge during start-up.

18.3.1 Electrical Distribution

The process plant electrical system is based on 13.8 kV, 1,200 A, 60 Hz distribution. The 138 kV feed from the local power authority will be stepped down to 13.8 kV at the main substation and will supply the main 13.8 kV switchgear housed in the switchroom of the main substation. 13.8 kV supply will be stepped down to 4.16 kV at the sulphide process plant substation for the plant large loads. Separate 13.8 kV/480 V distribution transformers at the process plant various substations will be fed from the plant main 13.8 kV switchgear.

The following substations with switchrooms will be provided:

- Main Substation.
- Process plant Services and Reagents.
- Feed Preparation.
- Oxide Plant.
- Sulphide Plant Grinding.
- Sulphide Flotation (two switchrooms).

Switchrooms will house 13.8 kV switchgear (main substation only), 4.16 kV switchgear (sulphide plant grinding only), 480 V motor control centres ("MCC"), area VVVF drives, plant control system cabinets, plant lighting transformers, various distribution boards and UPS power distribution.

13.8 kV overhead power lines will provide power to various remote facilities. Pole mounted transformers will step down the voltage at each location and supply an outdoor 480 V switchboard local to each equipment area.

18.3.2 Electrical Buildings

Electrical buildings will be pre-fabricated 'flat pack' panel buildings to minimize installation time on site. Buildings will be installed on a structural framework over 2 m above ground level to allow for bottom entry of cables into electrical cabinets. The electrical buildings will be installed with HVAC units and suitably sealed to prevent ingress of dust.

18.3.3 Transformers and Compounds

The main transformer 138 kV / 13.8 kV will be of ONAN / ONAF cooling configuration. Process plant large loads distribution transformer 13.8 kV / 4.16 kV and all the 13.8 kV / 480 V distribution transformers will be of ONAN cooling configuration and vector group Dyn11.

Fire-rated concrete walls will be constructed around the transformers.

18.4 FUEL SUPPLY

Diesel fuel will be stored on site near the mine services area for heavy and light vehicle refuelling.

The diesel fuel storage and delivery system will be a vendor package consisting of the following equipment:

- Diesel unloading pump;
- Diesel storage tank;
- Diesel supply pumps;
- Light vehicle bowsers; and
- Heavy vehicle bowsers.

A vertical spindle sump pump will be provided to remove any rain water from the diesel fuel bund area.

18.5 PASTE BACKFILL PLANT

Prior to underground mining, a paste backfill plant will be installed adjacent to the open pit. The paste backfill plant will be provided as a vendor supply package and installed in a building.

Preliminary paste testwork was completed and a paste recipe of 78% tailings and 2.4% cement has been used as a basis for paste backfill requirements for the underground mine. The peak monthly tailings requirement for paste backfill for the underground mine is approximately 47,000 t/month, i.e. 1,560 tpd tailings (dry basis). Operating at 18 hrs/day, the resulting paste plant capacity is 130 tph paste backfill (approximately 2,300 tpd paste backfill).

During normal process plant operations, thickened tailings from the sulphide tailings thickener will be pumped to the TMF. When paste is required, the thickened tailings will be pumped with the same positive displacement pumps via a separate pipeline from the tailings thickener directly to a continuous paste mixer at the paste plant. Cement will be supplied to the paste mixer from a 180 t cement silo located at the paste plant. Paste backfill from the mixer will discharge to a hopper and flow via gravity to the underground mine.

Treated water from the WWTP and power from the site 13.8 kV system will be provided for the operation of the paste backfill plant.

18.6 SUPPORT BUILDINGS

The following support buildings will be provided for the facility:

- Oxide Crushing Buildings (primary crushing and secondary crushing/screening).
- Oxide Grinding Building.
- Oxide Plant Belt Filter Building.
- Oxide Plant SART Building.
- Oxide Plant ADR Building.
- Sulphide Primary Crushing Building.
- Sulphide Crushed Mineralized Material Stockpile Cover.
- Sulphide Grinding Building.
- Flotation Building.
- Concentrate Dewatering/Loadout Building.
- Reagent Building.
- Waste Water Treatment Building.
- Plant Workshop and Maintenance Building.
- Change Facility.
- Laboratory.
- MCC/Electrical Rooms.
- Main Administration Building.
- Mine Maintenance Office.
- Mine Truck Wash Down Building.
- Mine Truck Shop.
- Main Warehouse.
- Explosives Storage and Handling.
- Fuel Station.
- Main Gatehouse.
- Paste Backfill Plant Building (will be installed prior to underground mining commencing and located near the open pit).

18.7 CUT-OFF WALL (COW)

On its west side, the open pit will be adjacent to the Menominee River. A 427 m long COW will be constructed between the pit and the river as shown on Figure 18.2. The purpose of the COW will be to reduce the flow of water from the Menominee River into the pit during mining.

The planned COW will comprise a soil, cement, and bentonite ("SCB") mix, constructed using the Cutter Soil Mixing ("CSM") technique. This construction technique will provide a COW having a hydraulic conductivity of 1×10^{-8} m/sec or lower. The COW will be a minimum of 1.0 m thick and it will extend through the overburden soils, terminating at least 0.5 m (and at most 2.0 m) into the underlying weathered bedrock.

18.8 MINE WASTE MANAGEMENT FACILITIES

Over the LOM, the Project will generate 49.28 Mt (24.96 Mm^3) of waste rock, 14.80 Mt of tailings, and 3.78 Mt (2.15 Mm^3) of overburden. Approximately 11.95 Mt (6.15 Mm^3) of the tailings will be disposed on the surface and the remaining tailings (2.85 Mt) will be disposed in the underground workings in the form of paste backfill.

The oxide process plant will generate a total of 1.48 Mt of tailings of which approximately 1.18 Mt (0.81 Mm^3) will be disposed on the surface and the remainder will be used as a paste backfill in the underground workings. The sulphide process plant will generate a total of 13.33 Mt of tailings of which approximately 10.76 Mt (5.34 Mm^3) will be disposed on the surface and the remainder will be used as a paste backfill in the underground workings.

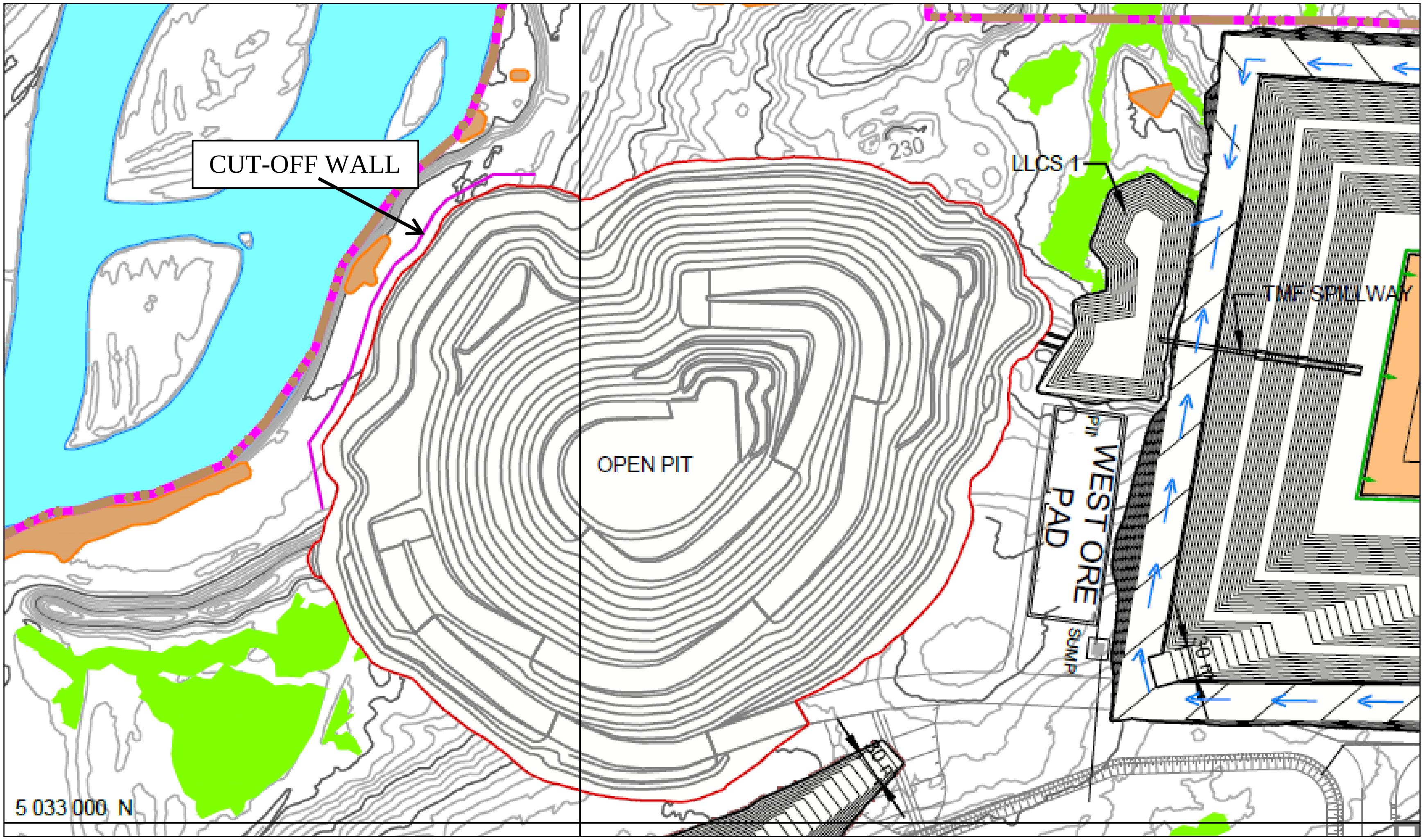
The tailings are potentially acid generating ("PAG"). Foth (2015b) estimated that 77% of the waste rock is PAG and metal leaching, and 23% of the waste rock is non-acid generating ("NAG"). For the purpose of this study, all of the waste rock is assumed to be potentially acid generating ("PAG").

The oxide and sulphide tailings streams will be co-disposed together with the waste rock in the TMF. The TMF currently has capacity to contain 5.91 Mm^3 of waste rock and 4.90 Mm^3 of tailings. Options are available for the disposal of the additional 1.25 Mm^3 tailings, such as filtering and stacking in the crown over the TMF or disposal in another small standalone surface tailings facility.

During closure, the open pit is planned to be backfilled with waste rock. The total volume of waste rock required for backfilling is approximately 19.06 Mm^3. During operations, this waste rock will be temporarily stored in two facilities referred to as the South Waste Rock Facility ("SWRF") and the North Waste Rock Facility ("NWRF").

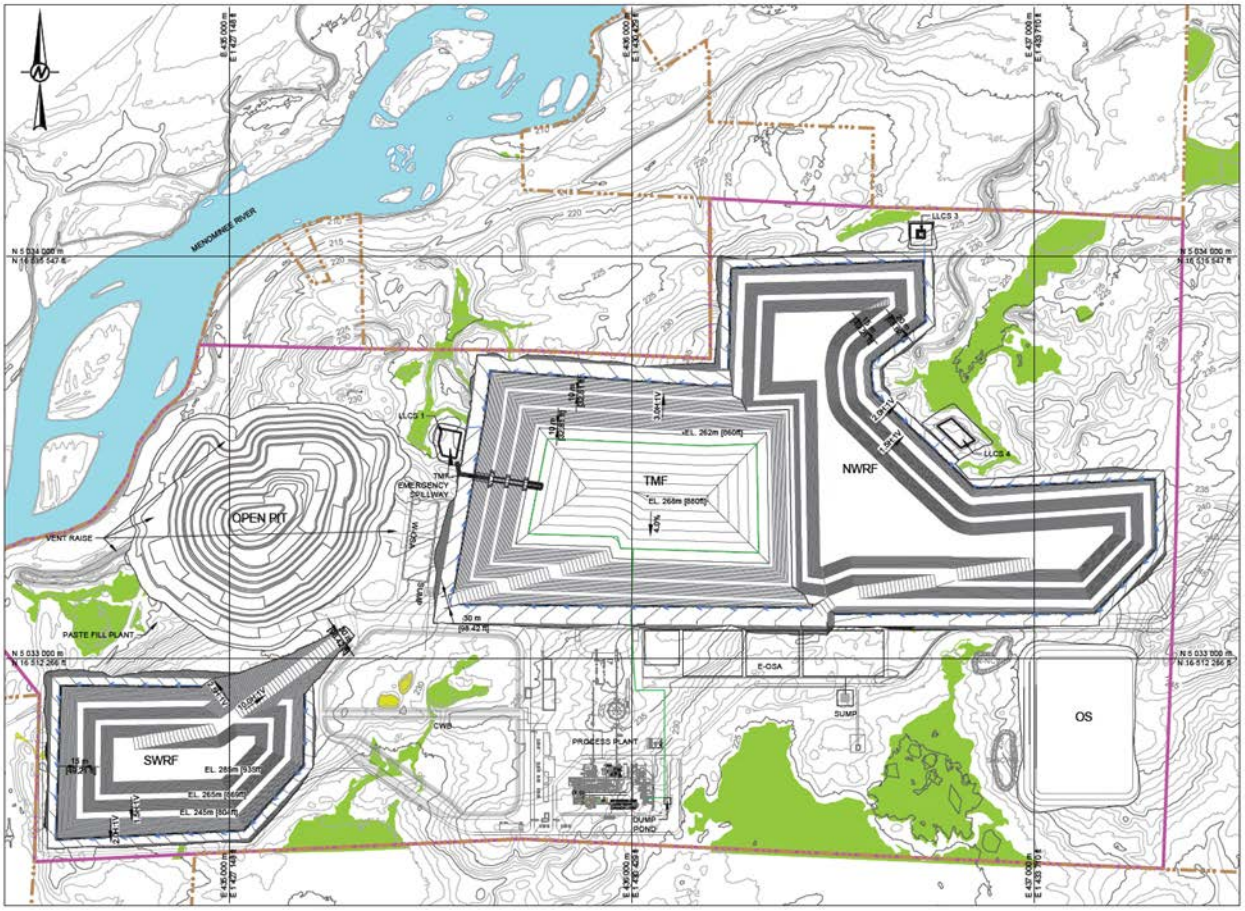
The overburden produced during operation will be deposited separately in a facility referred to as the Overburden Stockpile ("OS"). The general arrangement plan of these waste management facilities is shown on Figure 18.3 depicting the conditions following capping of the TMF with waste rock.

FIGURE 18.2 GENERAL ARRANGEMENT PLAN OF THE CUT-OFF WALL



Note: *The cut-off wall is the magenta line along the northwest corner of the pit, approximately parallel to the Menominee River*
Source: *Golder Associates (2019)*

FIGURE 18.3 GENERAL ARRANGEMENT PLAN OF THE TMF, WRFS AND OS



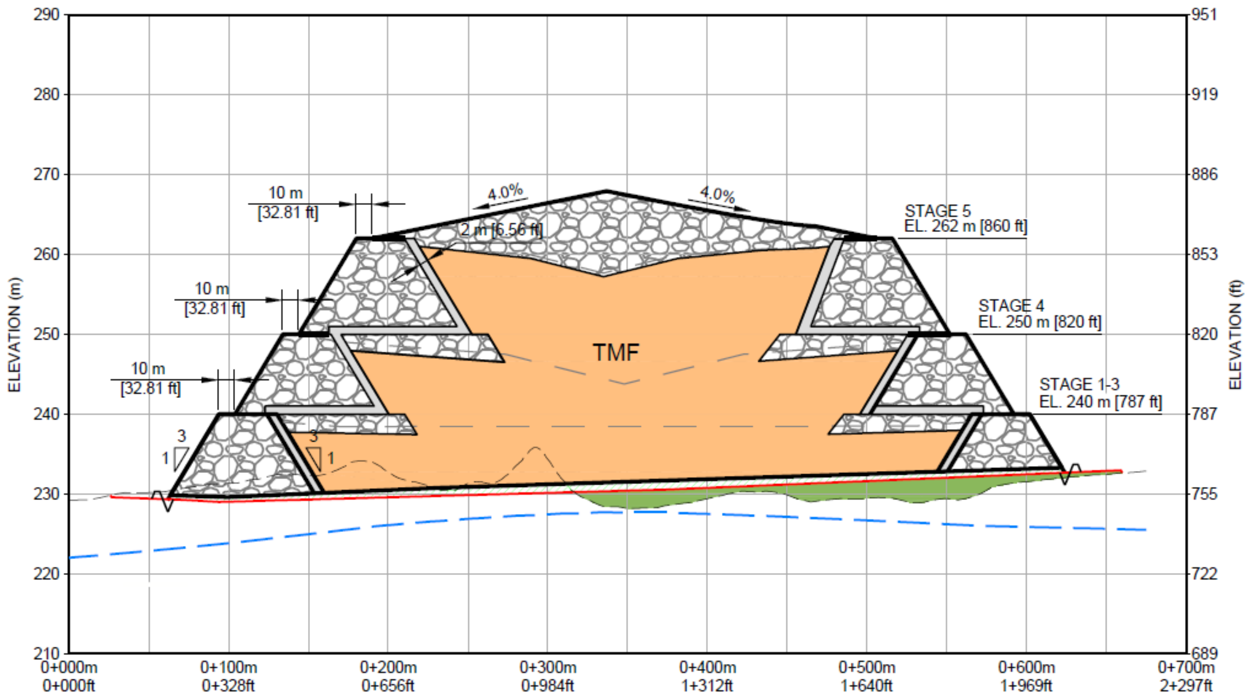
Source: *Golder Associates (2019)*

18.8.1 Tailings Management Facility

The TMF will be located east of the open pit and north of the process plants. The start-up TMF will cover a total footprint area of 27.8 ha and it will provide storage for 20 months. The Ultimate TMF will cover a total footprint area of 50.2 ha and it will have a maximum height of 36 m at the end of operation and a maximum height of 42 m after closure.

The TMF will be contained on all sides by a Perimeter Wall comprising a prism of waste rock at least 30 m thick, which will be raised continually in 10 m lifts using the upstream construction method (Figure 18.4). Approximately 10.0 m wide berms will be provided between each stage, and the interior and exterior side slopes of the Perimeter Wall between the berms will be graded to 3H:1V. The Perimeter Wall will be free draining. To retain tailings particles, the interior side slope of the Perimeter Wall will incorporate a 2.0 m thick zone of transition and filter materials comprising crushed waste rock and screened overburden soil, respectively. Below the Perimeter Wall at the bottom each lift, a heavy geotextile will be installed inside a crushed waste rock layer to filter the tailings.

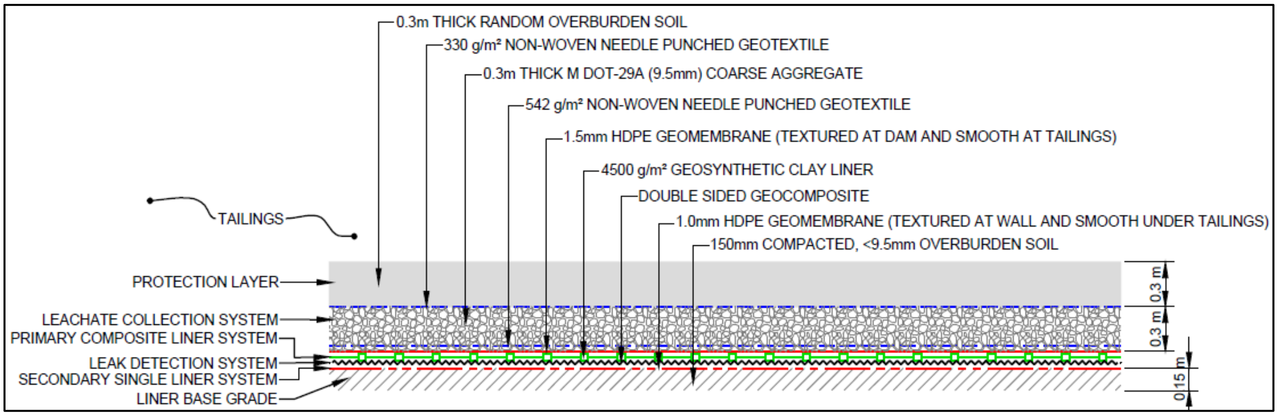
FIGURE 18.4 TYPICAL CROSS-SECTION OF THE TMF



Source: *Golder Associates (2019)*

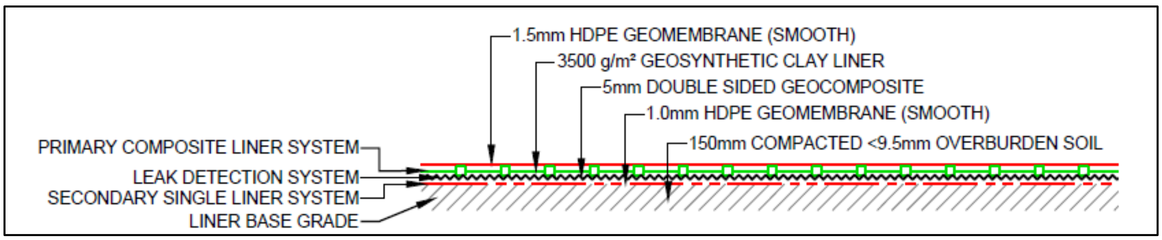
The footprint of the TMF will be covered with a double liner system with a leak collection system as shown on Figure 18.5. The liner system will consist of a composite primary liner (consisting of a high-density polyethylene ("HDPE") geomembrane over a geosynthetic clay liner ("GCL") and a single secondary liner. A drainage aggregate will be provided above the primary liner to collect and remove the tailings leachate. A geocomposite drain will be provided between the primary and secondary liners to collect and remove any leakage from the primary liner. The base of the TMF will be sloped with a 1% minimum gradient towards the northwest corner to allow the leachate and leakage to drain by gravity. A perimeter berm and ditch system will be provided around the perimeter of the TMF to collect run-off from the exterior side slope of the Perimeter Wall and to convey the leachate (including any leakage from the base of the TMF) into an external sump (referred to as the Leachate and Leakage Collection Sump No. 1 ("LLCS1") to be located at the northwest corner of the facility. The sump will be constructed below existing ground level and it will be lined with a double liner system as shown on Figure 18.6. The sump will be provided with a pumping and pipeline system to regularly convey the collected solution to the CWB.

FIGURE 18.5 DOUBLE LINER SYSTEM OF THE TMF BASE



Source: *Golder Associates (2019)*

FIGURE 18.6 DOUBLE LINER SYSTEM OF LEACHATE AND LEAKAGE COLLECTION SUMPS



Source: *Golder Associates (2019)*

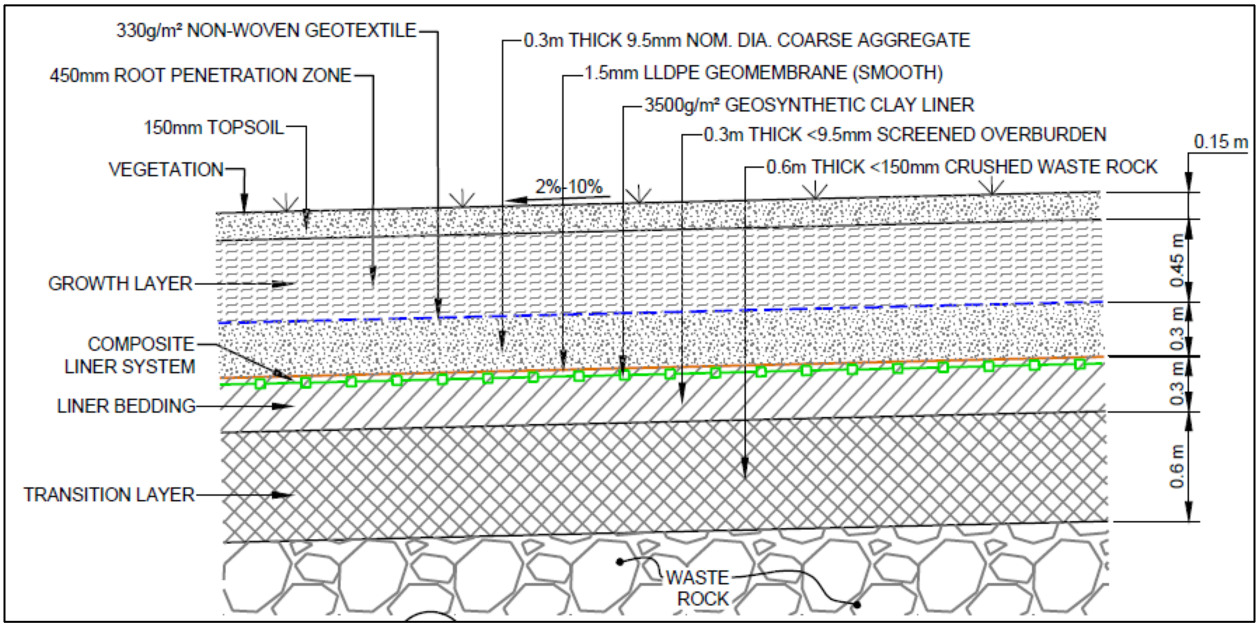
Both the oxide and sulphide tailings will be dewatered separately in high compression thickeners to target solids contents ranging between 65% and 76%, depending on the mineralized material type. The water recovered from the dewatering process will be circulated back to the respective process plants. The thickened tailings will be transported separately from each thickener to the TMF for disposal through carbon steel pipelines and pumped using piston pumps. The thickened tailings will be discharged from the interior of the TMF Perimeter Wall through a series of spigots in an HDPE header pipe. The supernatant water from the TMF will be pumped back to the process plants for re-use or to the CWB for treatment and release to the environment. A floating pump barge will be used to reclaim the TMF supernatant water from the interior of the TMF. Access to the floating pump barge will be through a floating walkway.

The maximum volume of supernatant water allowed within the TMF will reduce from 150,000 m^3 to 50,000 m^3 as the facility gets raised and the available space declines. An emergency spillway will be provided on the side of the TMF to safely convey a design extreme storm event (i.e., the 24-hour, probable maximum flood (“PMF”)). The spillway will discharge into the LLCS1, which will overflow into the open pit if necessary.

At closure, the TMF supernatant water will be drained and a waste rock crown with 4% side slopes will be placed over the tailings beach and the decant area to create a stable landform that will readily shed run-off water. The exterior surfaces of the TMF will be covered with multi-layer closure cover systems as shown in Figure 18.7 and Figure 18.8. The closure cover system will include a composite liner on flat and gently sloping surfaces, such as the TMF bench and the TMF crown, and a single liner on steeply sloping surfaces, such as the side slopes between the

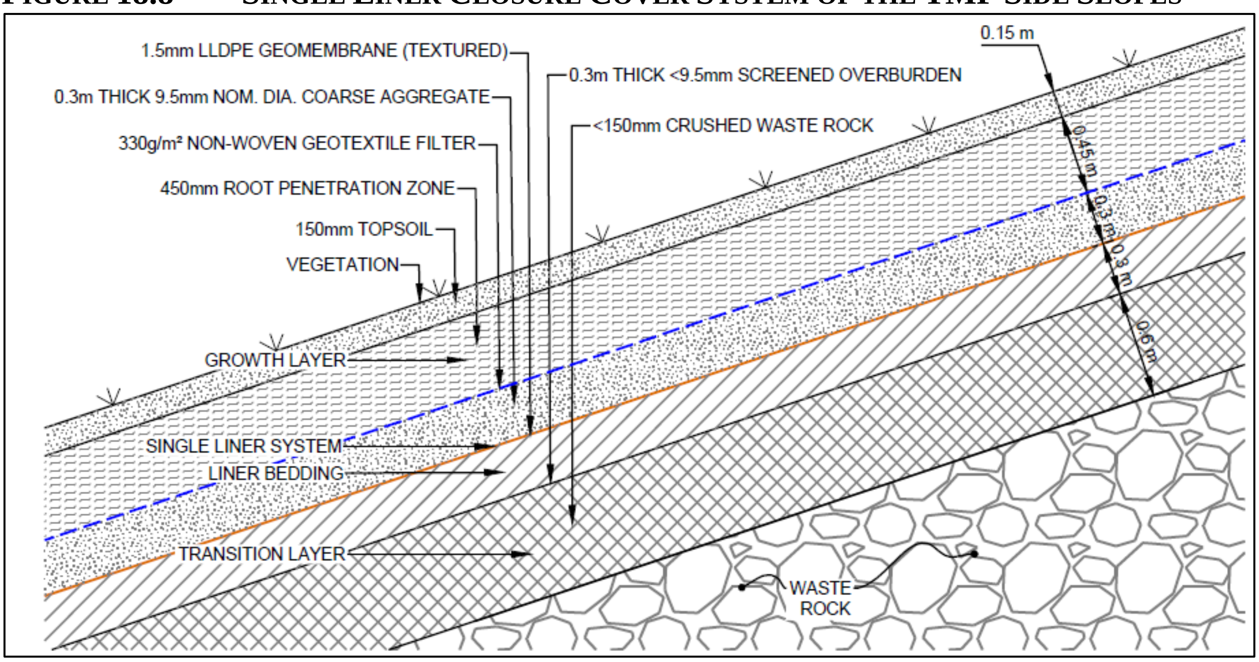
TMF benches, to limit infiltration into the TMF. A vegetation cover will be established over the closure cover systems. Drainage ditches and chutes will be provided to convey surface run-off water from the reclaimed TMF into the open pit via an external sedimentation pond.

Figure 18.7 Composite Liner Closure Cover System of the TMF Benches and Crown



Source: *Golder Associates (2019)*

Figure 18.8 Single Liner Closure Cover System of the TMF Side Slopes



Source: *Golder Associates (2019)*

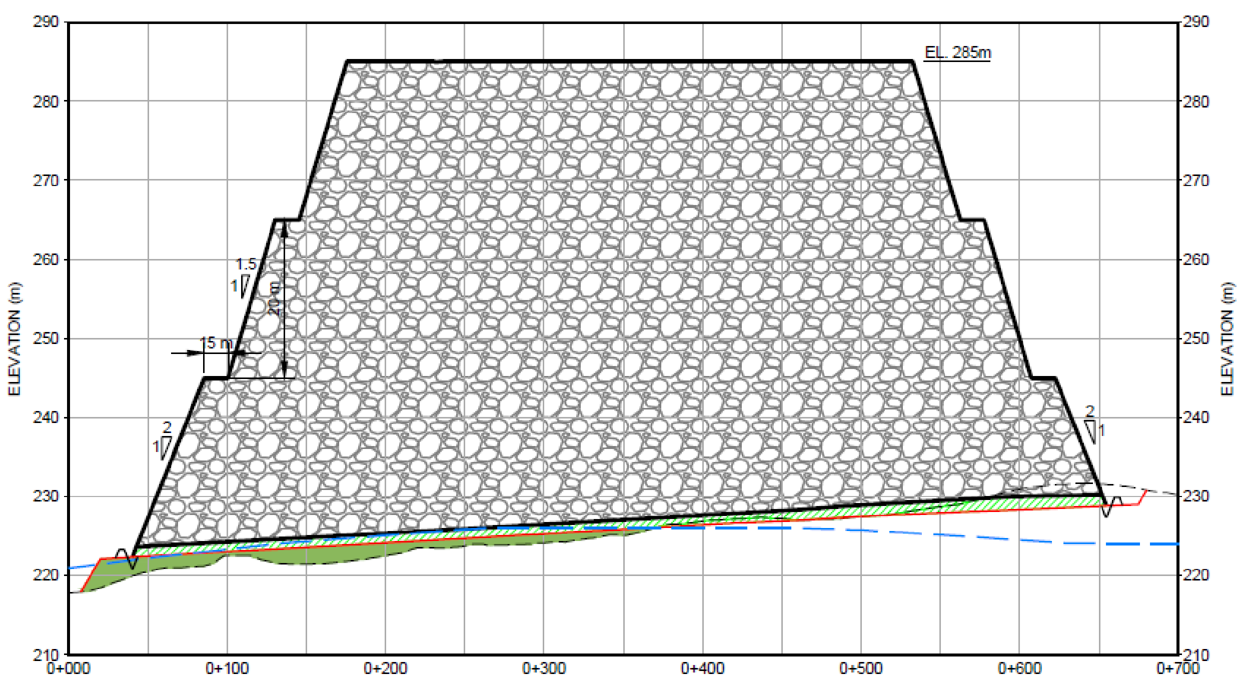
18.8.2 Waste Rock Storage Facilities

During operations, approximately 20.15 Mm^3 waste rock will be temporarily stored in the SWRF and NWRF. The SWRF will be located at the southeast corner of the open pit and the NWRF will be located east of the TMF. During closure, the waste rock from these facilities will be used for creating a crown on top of the TMF (1.09 Mm^3) and for backfilling the open pit (19.06 Mm^3).

For start-up, a total footprint area of 14.8 ha will be developed for the SWRF. This start-up facility will have a capacity to store 2.9 Mm^3 waste rock during the first 9 months of operation. The ultimate SWRF will have a maximum storage capacity of 6.51 Mm^3, a maximum height of 65.0 m and it will cover a total footprint area of 22.7 ha. The ultimate NWRF will have a maximum storage capacity of 13.64 Mm^3, a maximum height of 52.0 m and it will cover a total footprint area of 47.7 ha.

The waste rock will be deposited in 20.0 m high benches. The side slopes of the first bench will be graded to have a 2H:1V gradient for stability. The remaining benches will be deposited at angle of repose, which is estimated to be 1.5H:1V. Approximately 15.0 m wide berms will be provided after each bench for stability, with the exception of the first berm at the NWRF, which will be 20.0 m wide. A typical cross-section of the WRFs is presented in Figure 18.9.

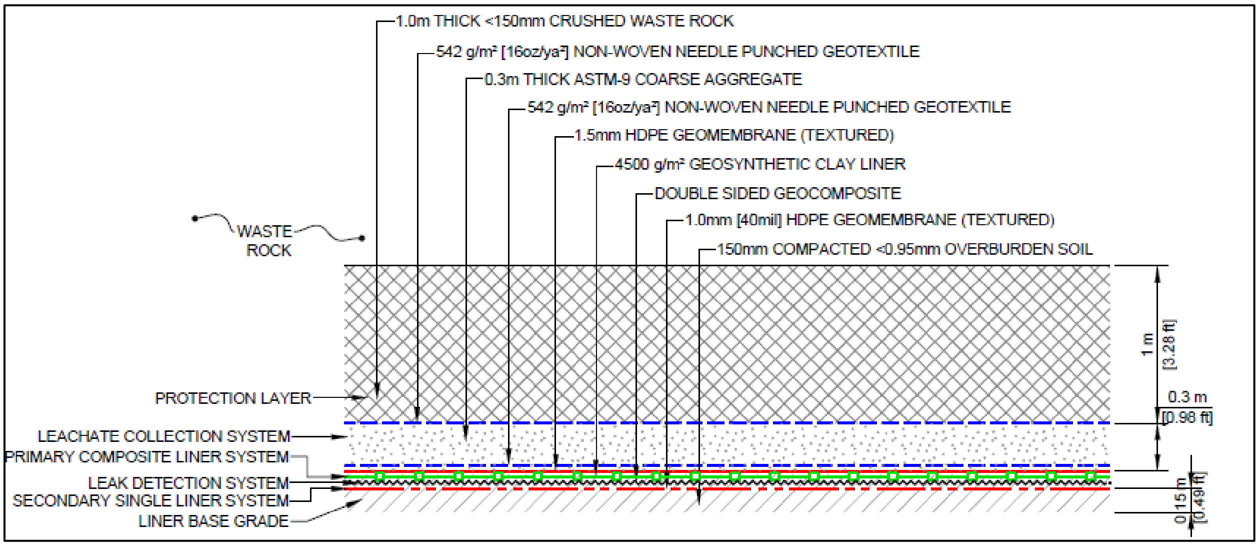
FIGURE 18.9 TYPICAL CROSS-SECTION OF THE WRFS



Source: *Golder Associates (2019)*

The footprint of the WRFs will have a double liner system similar to the liner system of the TMF as shown on Figure 18.10. The liner system will consist of a composite primary liner and a single secondary liner. A drainage aggregate will be provided above the primary liner to collect and remove leachate from the waste rock. A geocomposite drain will be provided between the primary and secondary liners to collect and remove any leakage from the primary liner. The bases of the WRFs will be sloped with a 1% minimum gradient and the perimeters of the WRFs will be encircled by systems of berms and ditches to allow the leachate and any leakage to drain by gravity into lined sumps. The SWRF will be provided with one sump and the NWRF will be provided with two sumps. The sumps will be constructed below ground and they will be lined with a double liner system as shown on Figure 18.6. The sumps will be provided with pumping and pipeline systems to regularly convey the collected solution to the CWB.

FIGURE 18.10 DOUBLE LINER SYSTEM OF THE WRFS BASE



Source: *Golder Associates (2019)*

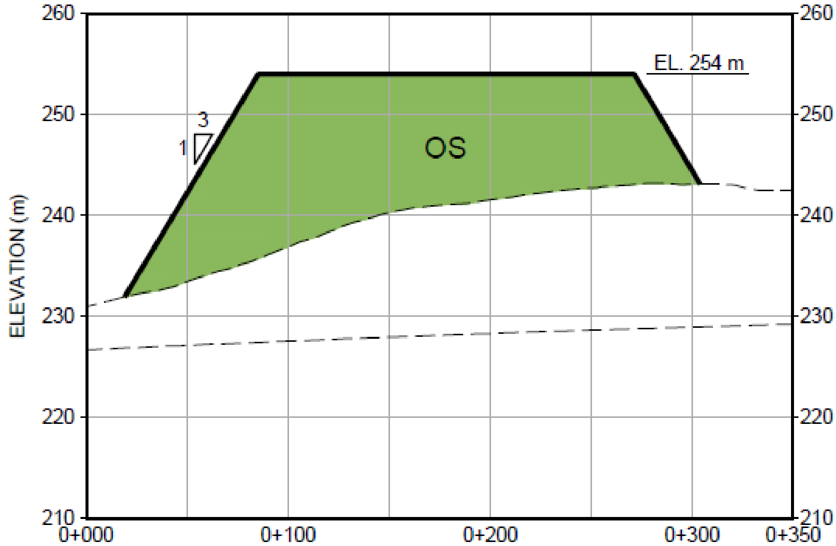
At closure, both WRFs will be removed. The footprint of the WRFs will be revegetated once the liner systems are removed and the sumps are backfilled.

18.8.3 Overburden Stockpile

The overburden from the open pit will be temporarily stored at the OS. The OS will be located south of the NWRF and east of the process plant. During mine operation, the overburden will be used as an earth fill material for roads, ramps and pads, as a filter material at the interior side slopes of the TMF, as a drainage layer at the base of the TMF and WRFs, and as a liner protection layer at the TMF base. The remaining overburden will be used during mine closure as a cover material at the TMF and the open pit.

The ultimate OS will cover a total footprint area of 10.8 ha and it will have a maximum capacity of 1.50 Mm^3. The maximum height of the stockpile will be 29.0 m. Side slopes of the stockpile will be graded to an overall profile of 3H: 1V for stability and to reduce erosion. A typical cross-section of the OS is shown in Figure 18.11.

FIGURE 18.11 TYPICAL CROSS-SECTION OF THE OS



Source: *Golder Associates (2019)*

18.9 WATER CIRCUITS

18.9.1 Raw Water Supply

Raw water supplied from the open pit dewatering system will be delivered to the raw water settling tanks for gross removal of solids and water storage. The raw water settling tanks can be individually isolated and taken out of service for periodic removal of accumulated solids; removed solids will be sent for ultimate disposal at the TMF. The suction of the raw water settling tanks pump will be above a silt level. Raw water is then pumped to an oil/water separator for gross removal of hydrocarbons and then pumped to the raw water tank for storage and distribution. Oil from the oil/water separator will be stored in waste oil totes and sent off site for ultimate disposal. There is an option to divert treated water from the waste water treatment plant via the treated water tank to the raw water tank.

18.9.2 Fresh Water

Fresh water will be obtained from a local groundwater well(s) to be drilled on or adjacent to the Property. The definition of site investigations and testing requirements to confirm groundwater well locations, sustainability of yield, and water quality is pending. Local permitting for groundwater supply will be completed prior to construction.

Fresh water will be supplied by the fresh water well pumps from the local groundwater well to an atmospheric vented fresh and fire water tank. Fresh water will be provided from the fresh and fire water tank for use as fire water and feed to the potable water treatment plant. In addition, fresh water will be supplied as contingency to raw water in the event that insufficient site water resources are available for re-use.

18.9.3 Fire Water

Fire water will be supplied from the fresh and fire water tank which has a reserve of 144 m^3 for fire water storage. Fire water supply will be via a vendor fire water pumping package which will include a fire water pump, a fire water jockey pump and a diesel fire water pump.

Fire water will be piped to all main facilities via buried underground fire water ring mains around each of the facilities. In addition, all buildings will be equipped with hose cabinets and supplemented with handheld fire extinguishers of two types: general purpose extinguishers for inside plant areas, and dry type extinguishers for inside electrical and control rooms. Ancillary buildings will be provided with automatic wet sprinkler systems throughout the buildings.

18.9.4 Potable Water Supply

Fresh water will be supplied from the fresh and fire water tank to the potable water treatment plant (vendor package). The fresh and fire water tank has a dedicated live capacity of 30 m^3 for potable water. The potable water treatment plant will be designed to local drinking water guidelines. The treatment plant is expected to include multimedia filtration for reduction of turbidity, followed by ultraviolet disinfection for primary disinfection, and the addition of sodium hypochlorite for secondary disinfection. Treatment residuals from the potable water treatment plant (e.g., multimedia filtration backwash), will be sent to the sulphide tailings thickener for ultimate disposal within the TMF. Treated potable water from the potable water treatment plant will be stored in the plant potable water tank and the safety shower water tank. Treated potable water from the plant potable water tank will be distributed via the plant potable water pump in a piping ring main to serve all potable water users in all facilities, such as the process plant and mine offices, change house, maintenance shop, main administration building and On-Stream Analysis ("OSA"). Treated potable water from the safety shower water tank will be distributed via the safety shower water pumps to drinking fountains, eye wash stations, and safety showers.

Potable water piping in the plant area will either be buried below the frost line, routed through heated buildings or heat traced and insulated. Manual drain points will be included to allow emptying of pipelines should conditions dictate.

18.9.5 Process Water

Process water used in the oxide and sulphide process plants will be sourced from different locations. However, a strategy of use and re-use of water will prioritize the use of tailings decant water and raw water or treated effluent water at the raw water tank. Process water piping in the plant area will be routed through heated buildings and provided with manual drain points to allow emptying of pipelines should conditions dictate.

18.9.6 Treated Water

Contact water from the CWB will be treated prior to discharge. Treated effluent will be delivered from the WWTP water tank to the Menominee River via the treated effluent pump and pipeline.

18.9.7 Mine Water Supply

Mine water for use in road dust suppression and the mine truck wash will be supplied via re-use of treated effluent. Treated effluent will be delivered from the WWTP treated water tank to a standpipe for the mine water truck. Treated effluent will be delivered from the WWTP treated water tank to the mine truck wash via the treated water pump.

When underground mine development commences, two (one on duty, one redundant) 8” heat traced water lines will be provided to supply water services. The water will be treated water from the WWTP.

18.10 MINE AIR SERVICES

Air services for the mining area will be provided by the mine services air compressor, mine services air dryer, plus fine and coarse filtration. Two receivers will be provided, namely the mine services air receiver and the fuel area air receiver.

18.11 SEWAGE TREATMENT

Sewage generated within the Project site will be collected via an underground sanitary sewer network to a common location where it will be treated by an above-grade, mechanical sewage treatment plant (vendor package). Treated sewage effluent will be discharged to the environment subject to further regulatory permit requirements. Sludge generated as a by-product of the treatment of sewage will be disposed off-site by a licensed contractor.

18.12 SITE WATER MANAGEMENT AND EFFLUENT TREATMENT

The main objectives for water management for the Project are to provide collection and treatment of flows and run-off from contact areas, and to divert run-off from non-contact areas to the greatest extent possible.

Contact areas include the open pit, haul roads, WRFs, mineralized material stockpiles, process plant area, TMF and CWB.

Non-contact areas include topsoil stockpile and OS, perimeter roads around the WRFs and TMF and undisturbed areas.

Water management measures include site grading, berms, ditches, and culverts for the diversion and collection of contact and non-contact water, and basins for the sedimentation, equalization, and accumulation of contact and non-contact water.

18.12.1 Contact Water Management and Treatment

Contact water will be generated during operations from a variety of sources: net precipitation inputs and groundwater inflow to the open pit and net precipitation inputs to haul roads, WRFs, mineralized material stockpiles, the TMF, and the process plant area, as well as the CWB itself. Contact water will also be generated as effluents from the process plant, mine truck wash, and process plant diesel area sump. Surface run-off that has come in contact with these areas will be collected via a series of ditches and channels as required and directed preferentially via gravity, or where required, via pumping to the CWB. As discussed in this section mine dewatering water will be directed to the raw water settling tanks and an oil/water separator prior to re-use in the process as raw water. Excess mine dewatering water that cannot be re-used in the process plant will be directed to the CWB from the raw water settling tanks via the raw water settling tanks pump.

Contact water ditches will convey flows from the contact areas to the CWB. The contact water ditches will be lined ditches (HDPE underlay or other impervious lined alternatives) and designed to convey the 1 in 100-year storm event without overspill into the natural environment.

Contact water culverts will convey contact water below roads. The inlet and outlet of the culvert crossings will be provided with appropriate erosion protection in the form of riprap with geotextile and HDPE underlay or other impervious lined alternatives.

Contact water from the CWB will be treated prior to discharge. Treated water from the WWTP will be discharged to the environment meeting National Pollutant Discharge Eliminate System ("NPDES") permit requirements.

Both short-duration rainfall events and long-duration rain-on-snowmelt events have been evaluated for the CWB storage capacity design. The CWB will have an emergency spillway, the purpose of which is to allow the release of extreme flood events to the open pit without overtopping the CWB berms. The spillway will comprise a pond level control weir and a spillway channel. The emergency overflow spillway will be routed to the open pit.

The CWB embankments must have an adequate factor of safety ("FOS") for the conditions under which the basin will operate. Table 18.1 summarizes the minimum FOS values for static and seismic assessment of embankment structures recommended in the Canadian Dam Association ("CDA") Guidelines.

TABLE 18.1
CDA MINIMUM FOS FOR SLOPE STABILITY

Loading Condition	Minimum FOS
End of Construction (before reservoir filling)	1.3 (upstream and downstream slopes)
Long Term (steady-state seepage, normal reservoir level)	1.5 (downstream slopes)
Earthquake (pseudo-static)	1.0 (upstream and downstream slopes)

A water balance has been generated to reflect the latest Project configuration described within this Technical Report, to estimate the magnitude and extent of any water surplus or deficit conditions, on a site-wide basis, based on average, dry, and wet climatic conditions at the site. The water balance is based on the phased build-out with respect to the site development and includes the following inputs:

- Site layouts and footprints for major site features.
- Mine plan inputs (e.g., mineralized material and waste rock movements, explosives powder factor).
- Process mass balance inputs (e.g., water demands, wastewater generation, tailings production rate, tailings characteristics).
- Climate inputs (e.g., precipitation, evaporation).
- Physical and hydrological inputs (e.g., topography, watersheds/catchments, run-off coefficients).
- Hydrogeological inputs (e.g., groundwater inflows to the open pit).
- Geochemistry inputs (e.g., weathering behaviour of tailings and waste rock materials).
- Discharge permit requirements (e.g., limitations on quantity and or quality of treated effluent to be discharged to the environment).

At this time, it is expected that treatment for heavy metals or other oxyanions will be required. The primary process for metals removal from the wastewater will be a low-density sludge ("LDS") lime-based neutralization process. Additional metal removal will be achieved via sulphide precipitation. The LDS treatment process is a technique to produce sludge by precipitating metal hydroxides. It involves the addition of lime slurry to a reactor vessel(s) to increase the pH of the feed water, facilitating the reaction to precipitate soluble metals present in the feed water. Sulphide precipitation involves the addition of a sulphide reagent to produce metal sulphide sludge. Sulphide precipitation allows for lower dissolved metal concentrations to be achieved in the wastewater as compared to lime-based neutralization alone. Feed water, following lime neutralization and sulphide precipitation, is then flocculated through the addition of a polymer flocculent and then fed to the clarifier. Hydroxide and sulphide precipitated particles settle out as sludge within the clarifier. Underflow from the clarifier will be sent back to the sulphide tailings thickener. Overflow from the clarifier is pH-adjusted using carbon dioxide following clarification. pH-adjusted water is then sent to filtration and the filtrate is further treated with sulfuric acid to optimize the pH prior to mercury polishing. Following filtration and mercury polishing, water is sent to the treated water tank. From the treated water tank, treated water is reused as process water for reagent make-down within the WWTP, for haul road dust suppression, and for the mine water truck and mine truck wash. Excess treated water will be discharged to the environment at the Menominee River according to the NPDES discharge permit.

The WWTP will be housed in a treatment building which will include, at a minimum, reactor tanks, transfer pumps, chemical dosing systems, and ancillary equipment. The CWB, carbon dioxide tank, lime silo, as well as the clarifier and clarifier underflow pump(s) will be located outside the water treatment building.

18.12.2 Non-Contact Water Management

Non-contact water ditches and culverts will divert clean surface run-off away from the contact areas. Water collected through the non-contact water ditches and culverts at the OS will be conveyed to the NCWBs. The non-contact water ditches and culverts will be designed to convey a 1 in 100-year, 24-hour storm event without overspill into the contact water areas. The non-contact water drainage system does not require HDPE underlay (or other impervious lined alternatives).

The NCWBs are designed in accordance with the Michigan Department of Environmental Quality ("MDEQ") Stormwater Management Guidebook.

Run-off from the topsoil stockpiles is to be managed with a combination of silt fencing and buffer strips to reduce flow velocity and remove sediment.

19.0 MARKET STUDIES AND CONTRACTS

19.1 CONTRACTS

There are no material contracts or agreements in place as of the effective date of this Technical Report.

19.2 METAL PRICES AND MARKET OUTLOOK

The Base Case macro-economic forecast assumes a flat pricing that has been drawn from the consensus long term estimates of North American equity analysts as of August 4, 2020. The Spot Case is based on prices at the time of writing. Prices for the two cases are presented in Table 19.1 (in order of economic contribution):

TABLE 19.1
METAL PRICE CASES

Metal	Units	Base Case	Spot Price
Gold	US$/oz	1,485	1,998
Zinc	US$/lb	1.08	1.04
Copper	US$/lb	3.05	2.92
Silver	US$/oz	18.20	25.00
Lead	US$/lb	0.91	0.83

Zinc and gold are the most important metals that will be produced at Back Forty, generating over 80% of total revenue. Under the Base Case long term price forecast, precious metals comprise 45% of total revenue.

Over 50% of payable precious metals will be contained in the three concentrates produced, with the copper concentrate containing approximately 75% of payable gold in concentrate and 60% of payable silver in concentrate.

The following is a brief discussion of the market outlook for the three most important metals (gold, zinc and copper) and zinc, copper and lead concentrates. Statistics for metal markets have been taken from September 2019 analysis by BMO Capital Markets. Statistics for concentrate markets have been provided by Ocean Partners, who are specialist base metal concentrate traders.

19.2.1 Gold

Mine supply of gold is forecast to be on a downward trend over the first half of the current decade due to falling mined grades and Mineral Reserve tonnage depletion at many existing mines.

Gold’s status as a ‘safe-haven’ led to further price rises as the Coronavirus pandemic swept the world during the first half of 2020. As gold prices continue to rise, this could provide some incentive for mining companies to either lengthen the life of existing assets by increasing the gold price used to calculate Mineral Reserves or develop new green-field projects. However, thus far the response has been muted. Mining companies are likely to be cautious about building higher gold prices into Mineral Reserve calculations after doing this when the gold price spiked in 2011 and then incurring significant impairment charges when prices subsequently fell. Exploration budgets in the gold mining industry are reported to be increasing. However, many proposed new projects are either technically challenging or located in countries with a difficult legislative environment. There will therefore inevitably be a significant time lag before increased exploration activity translates into higher mine output.

To date, the main outcome of higher gold prices has been a spike in corporate M&A activity, such as the recent mergers of Randgold with Barrick and Newmont with Goldcorp. Although such activity may increase the output and cost competiveness of individual companies, the total amount of gold available to the overall market remains unchanged. With western mining companies increasingly focussed on returning value to shareholders and Chinese output constrained by stricter environmental controls, there appears to be limited scope for global gold mine output to reverse its forecast decline over the short to medium term.

Purchases of gold jewellery by the general public are likely to be inhibited by recent price rises. However, gold's attractiveness as a financial asset will more than compensate for subdued consumer demand over the coming years. Data from the World Gold Council shows that by the end of June 2020, global holdings of gold-backed exchange traded finds ("ETFs") had reached an all-time high level of 3,621 tonnes.

Low interest rates and poor bond yields mean that gold became an increasingly attractive option to investment managers during 2019 and 2020. In addition, rising tensions in the Middle East, the prospect of a trade war between the US and China and the recent Coronavirus outbreak have all proved supportive for the gold price while the value of industrial metals has faltered. Gold is also an attractive option for emerging economies that are looking to diversify their strategic reserves away from US dollars. It is understood that the central banks of countries such as China, Russia and Kazakhstan have been significant gold purchasers over recent years.

The strong fundamentals described may be already reflected in the prevailing gold price. However, they are unlikely to reverse over the short to medium term.

19.2.2 Zinc

The rapid expansion of the Chinese economy has supported world growth in total zinc demand over the past decade. During this period, consumption in China grew at around 6% per annum while demand elsewhere remained broadly level. Overall global demand for zinc grew at an average of approximately 2.4% per annum over the last 10 years.

As the Chinese economy transitions towards one where consumer demand rather than infrastructure investment and exports is the primary source of economic growth, annual increases in zinc consumption are expected to slow. Slower growth in China will only be partially mitigated by accelerating demand in India and ASEAN countries such as Thailand, Vietnam and Indonesia.

The traditional uses of zinc are under threat from demand substitution. New less zinc intensive corrosion resistant alloys, coatings and application methods are becoming increasingly prevalent. In recent years zinc die-castings have also lost market share to aluminum and plastic as manufacturers seek to reduce the weight of the vehicles. High copper prices also mean that some brass components such as joints and valves have been replaced by plastics.

It is expected that slowing demand growth from China and the threat of demand substitution will be to some extent mitigated by new applications for zinc. Rising global populations and climate change are expected to be the driving forces for increased use of zinc as a micro-nutrient in fertilizer. Zinc supplements and zinc fortified foods also offer long term solutions to the problem of zinc deficiency in the diets of infants and children. The potential development of zinc-based battery technologies represents significant upside potential for zinc demand. The future need for batteries in transport applications, for the storage of renewable energy, and in the telecoms sector, mean that this is currently a particularly active area of research. Overall, global demand for zinc is forecast to grow steadily over the next decade.

The global economic slowdown caused by the Coronavirus crisis is forecast to see global zinc metal consumption register a third consecutive annual fall during 2020. Even though the pandemic will also lead to a decline in mine production, the zinc metal market is still forecast to register a substantial surplus during 2020. As economic activity recovers during 2021 and 2022 the zinc metal consumption is forecast to resume an upward trend leading to smaller deficits in the zinc metal market. By 2023, lower mined grades and Mineral Reserve depletion at existing mines mean that some supply from mining projects that are currently in development will be required in order to satisfy forecast market requirements. If the global economy recovers more slowly than expected from the disruption caused by Coronavirus, then further closures of uneconomic mines or those that are unable to sell complex concentrate qualities in an over-supplied market could potentially bring forward the time when the zinc metal market moves back into deficit.

An oversupplied market is expected to lead to zinc prices trending below long-term averages over the next few years before recovering towards the middle of the decade.

19.2.3 Copper

In a similar scenario to that described for zinc, global growth rates for global copper demand are also expected to slow in the coming years as the Chinese economy transitions towards one where consumer demand rather than infrastructure investment and exports is the primary source of economic growth. Wood Mackenzie forecasts that global refined copper consumption will grow at an average of 1.7% p.a. over the first half of this decade. This figure compares with an estimated average growth rate of 3.2% p.a. over the previous 10-years.

Slowing demand growth rates should be seen in the context of a much higher ‘base’ compared to earlier periods. The Wood Mackenzie forecast implies that even after accounting for a decline during 2020 due to the Coronavirus pandemic, global copper demand will still grow by an average of 428 kt/a between 2020 and 2025. In an environment when many old mines are experiencing lower mined grades and Mineral Reserve depletion, the mining industry faces a considerable challenge keeping pace with this demand.

Over the longer term the global transition to a 'low carbon' economy is expected to be supportive for copper consumption. Demand will be boosted by the electric cabling requirements associated with renewable energy sources such as wind and solar power and the extra copper required to build electric vehicles and the associated charging infrastructure. Increased focus on preventing the spread of viruses following the recent Coronavirus pandemic may also mean that its anti-microbial properties see copper used more widely as a building material.

The commissioning and ramp-up of new mine capacity such as Spence, Grasberg Block Cave, Quebrada Blanca, Kamoa-Kakula and Quellaveco is forecast to keep the global market adequately supplied with mined copper over the period 2021 to 2023 with consequent downward pressure on copper prices. However, a significant 'supply-gap' is forecast to open from 2024 unless new mine capacity is developed. Although the potential 'upside' for copper prices is limited by the threat of demand substitution from aluminium, the long-term copper price still must be high enough to incentivise mining companies to build sufficient new capacity to fill the looming 'supply gap' in the copper market. On this basis, market surpluses are expected to keep prices below these levels over the near to mid-term which is likely to discourage mining companies from investing in new copper mining projects. Given the extended time-scales required to develop copper mining projects, a company that is able to bring new capacity on-line around the middle of this decade is likely to benefit from a positive price environment.

19.3 CONCENTRATE MARKETING

19.3.1 Zinc Concentrate Market

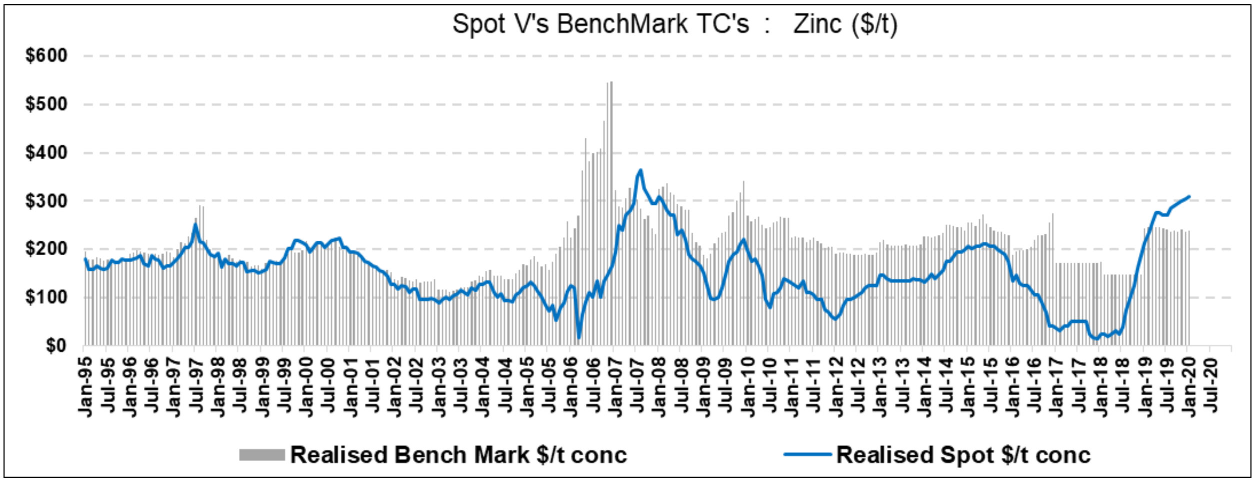
The global zinc concentrate market operated at a surplus during 2019 as mine output increased at locations such as Antamina and Glencore's Queensland operations while new projects including New Century and Gamsberg also ramped up production. Spot treatment charges ("TCs") climbed steadily during the year and by March were in excess of the annual benchmark level of $245/t. Typical spot terms for 'standard quality' concentrate reached $315/t by February 2020 with more challenging materials attracting significantly higher terms.

The 2020 benchmark TC was set at $299.75/t, reflecting expectations that the zinc concentrate market would remain in surplus during the year. However, the market outlook changed rapidly from March 2020 as mine production in Latin America and elsewhere was curtailed by the introduction of measures required to contain the Coronavirus pandemic. Reductions in the zinc metal price during this period also led to more extended production cuts as higher cost mines were placed on care and maintenance and construction projects were delayed. Typical spot TC terms for 'standard quality' concentrate fell to around $160/t in May before recovering slightly during June.

The global zinc concentrate market is now forecast to be close to balanced during 2020 and 2021. Spot TCs are expected to climb gradually as miners in Latin America and elsewhere resume operations. However, it is likely that mine production levels will continue to be constrained by measures required to contain the spread of the Coronavirus for the foreseeable future. Spot TCs are therefore expected to remain below the 2020 benchmark level. Chinese smelters are operating under increased environmental constraints and as a result any concentrate with a lower zinc grade or elevated levels of residue forming and/or penalty elements are likely to attract less favourable terms now and in the future.

The differential in historical Benchmark and Spot TCs is presented in Figure 19.1.

FIGURE 19.1 HISTORICAL SPOT – BENCHMARK DIFFERENTIALS FOR ZINC CONCENTRATE



Source: *Ocean Partners (2019)*

Back Forty Zinc Concentrates

Every smelter has a preferred range of concentrate qualities which maximize their economics. On the other hand, every smelter will struggle with one or more deleterious elements – whether these be impurities they cannot handle, or by-products they cannot recover.

Based on Back Forty metallurgical testwork, zinc concentrates produced from the Main and Pinwheel Zones are expected to be low in silver (<100 g/t), with no other significant by-products (i.e. copper, germanium, indium) other than gold. Some Western smelters may credit gold when the content is above 1.00-1.25 g/t, depending on market conditions.

Mercury, cadmium, silica and iron are the most significant deleterious elements in the expected quality specification and can potentially be subject to penalties and in some cases outright rejection of the quality. One can assume that pressure on the production, handling and disposal of Hg will only continue to increase over the coming years.

One should also note the fact that all smelters, and especially the Chinese smelters, are now operating under increased environmental constraints and as a result any concentrate with a lower zinc grade or elevated levels of residue forming (e.g. Fe, SiO_2, MgO, CaO) and/or penalty elements are likely to attract less favourable terms now and in the future. This is especially true in surplus concentrate market conditions.

Considering current Chinese import restrictions/recommendations, it is important that the cadmium limit not exceed 0.3%, so as not to preclude China as a potential market. It is also important to note that under the current trade war shipments of US origin zinc concentrates are subject to a 25% tariff on import to China.

The expected zinc concentrates from Main and Pinwheel should be saleable under any market conditions in both Western and Far East markets.

The zinc concentrate from the Tuff material is expected to be more complex than the Main and Pinwheel Zones. The higher silver and gold content will be attractive to Western smelters with capacity to recover precious metals. The quality may struggle in balanced to oversupplied markets in the Chinese market, with the Chinese generally trying to avoid silver-bearing concentrates. The quality will not obtain a gold payment under any market conditions in China.

The key challenge in western markets, from a quality perspective, is the very high silica content (>5%). This can severely restrict potential outlets, likely eliminating those with limited capacity to handle residues, and will definitely be reflected in the treatment charges and penalties offered by those who are capable of treating these concentrates. The most logical outlet logistically for Back Forty's zinc concentrates should be Teck's Trail integrated zinc/lead complex in the interior of British Columbia. Trail was traditionally fed by local mines. As mines closed, such as Sullivan, Trail was forced to diversify and did so largely internally through Teck's Red Dog Mine in Alaska, Pend Oreille Mine in Washington State as well as other local mines. Teck very much widened its scope of possible feeds through significant investments in precious metal and by-product (indium, germanium) recovery, specifically allowing it to compete in the Mexican, Peruvian and Bolivian markets, targeting by-product containing concentrates. As with China, elevated levels of residue forming (e.g. Fe, SiO_2, MgO, CaO) and/or penalty elements are likely to attract less interest and less favourable terms now and in the future.

The other two Canadian zinc smelters, CEZ and Flin Flon, offer a similar logistic advantage over other custom smelters overseas, however, these facilities have no precious metal or minor element recovery and limited but certainly some ability to handle anything other than the standard impurities they saw over the decades from local mines. That said, rumours on investments in precious metals recovery continue to swirl around at CEZ. HudBay is currently planning to close the Flin Flon zinc plant in 2022 and therefore Flin Flon should be excluded as a possible outlet.

Zinc concentrates can be shipped in bulk to the Far East market through the port of Vancouver in Canada.

In Asia, Korea Zinc, without any doubt one of the premier custom zinc smelting groups in the world, is an obvious potential outlet. Their two zinc smelters, combined with an expanded, and state-of-the-art, lead plant and associated copper plant are incredibly flexible, efficient and well-run.

In general, Japanese smelters have above-average precious metals recovery. Unlike Korea Zinc, and other custom smelters, the Japanese ted to be extremely intolerant of impurities such as silica and mercury.

Three factors remain important when considering the Chinese market.

- The Chinese have a low tolerance for silver and gold content in concentrates. The Chinese rarely pay for silver, except in tight markets, and never for gold. Despite this limitation on precious metals, many Chinese smelters are very keen on and very good at recovering other by-products like copper, indium and germanium from zinc concentrates.

- Chinese customs limits/recommendations on the cadmium (<0.3%), mercury (<600 ppm) and arsenic (0.6%) content for imported zinc concentrates.

- Under the current US/China trade war, shipments of US origin zinc concentrates are subject to tariffs on import to China.

Back Forty's zinc concentrates can reach the European Market though bulk handling facilities in Quebec City and Three Rivers.

In Europe, Nyrstar has some of the most flexible plants, and one of the most inflexible plants in the region, from a quality perspective. Balen (Belgium) and Auby (France) are, and always have been, custom smelters which were originally engineered specifically to treat a very wide range of qualities and extract maximum value from those concentrates. These two plants have a reasonable tolerance to Fe and SiO_2 compared to their sister plant at Budel (Netherlands). Budel is quite unique in that it is a zero-residue producing plant, as mandated by the government. Everything that goes into the plant must go out as a product. Residue generating elements such as iron and SiO_2 are generally severely frowned upon unless they bring significant other by-product values in the concentrate.

Commercial terms for zinc concentrate that have been assumed in the evaluation of Back Forty include:

<u>Delivery:</u>
CIF North America or parity

<u>Treatment Charge:</u>
TC: $200-220/dmt (assume additional $20 for Tuff Zone)
Basis $2,000
Escalation +$0.02 /-$0.02

<u>Refining Charges:</u>
There are traditionally no refining charges for precious metals in zinc concentrates

<u>Payables:</u>
Zn: Pay for 85% (minimum deduction of 8 units) of the contained zinc at the LME Settlement quotation (price) for Special High Grade Zinc averaged over the agreed quotational period.

Ag: Deduct 3.0 oz/dmt of the contained silver and pay for 70% of the balance at daily US$ LBMA Spot quotation for silver, as provided by CME-TR (CME-Thomson Reuters) averaged over the quotational period.

Au: Deduct 1.00 g/dmt of the contained gold and pay for 70% of balance at a price equal to the mean of the daily US$ London Bullion morning and afternoon quotations for gold as published in the London “Metal Bulletin averaged over the quotational period..

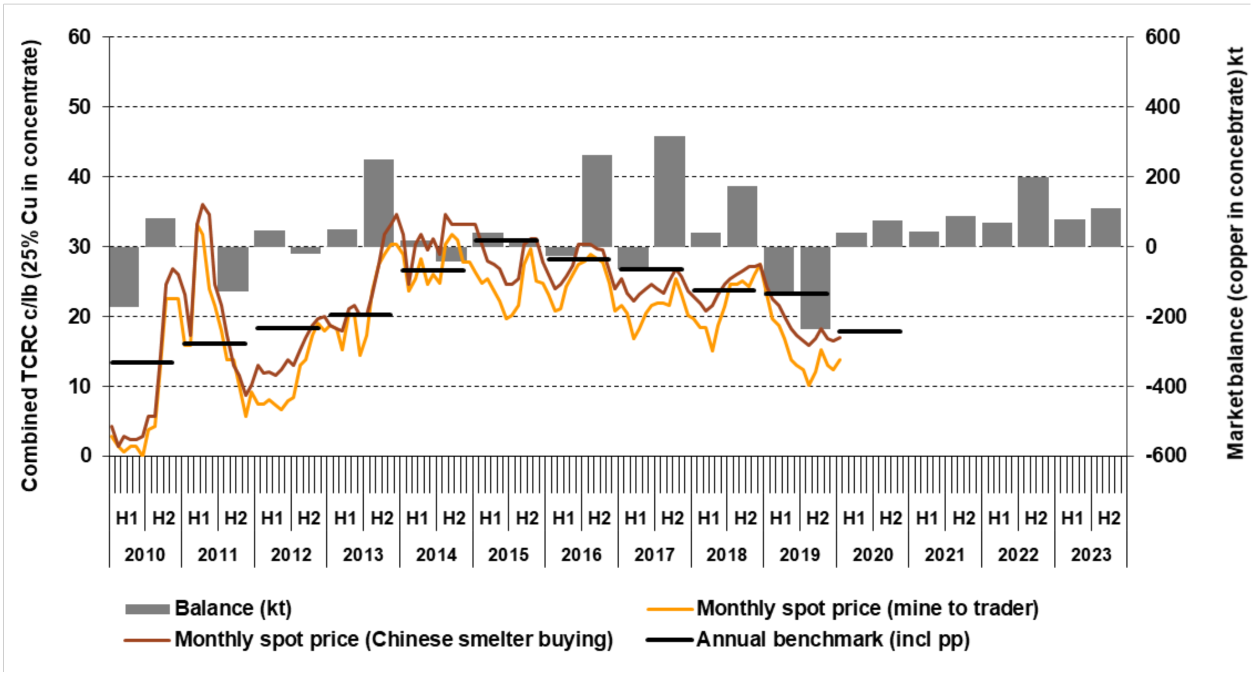
Penalties:
Fe: $1.50-2.00 / dmt for each 1.0% that the Fe exceeds 9.0%
SiO2: $1.50-2.00 / dmt for each 1.0% that the SiO2 exceeds 3.5%
Cd: $1.50-2.00 / dmt for each 0.10% that the Cd exceeds 0.40%
Hg: $1.50-2.00 / dmt for each 100 ppm that the Hg exceeds 250 ppm

19.3.2 Copper Concentrate Market

Treatment and refining charges (“TCRCs”) for copper concentrate reached multi-year lows during 2019. This was the result of a significant deficit in the copper concentrate market as mine supply fell by 1.9% while expansion of Chinese smelting capacity saw global demand for concentrate rise by 1.8%. Monthly average buying terms for clean concentrate from Chinese smelters fell from $84/t and 8.4 cents/lb in January to $57/t and 5.7 cents/lb in December. This decline was also reflected in the TCRCs for sales of clean concentrate from miners to traders which fell from a monthly average of $80/t and 8.0 cents/lb in January to $43/t and 4.3 cents/lb by December. Expectations of continued tightness in global copper concentrate markets were reflected in the 2020 benchmark for long term contracts which was set at $62/t and 6.2 cents/lb, the lowest level since 2011.

Spot TCRCs for copper concentrate briefly rose above the benchmark level early in 2020 as smelter utilization rates in China were hit by the initial Coronavirus outbreak in that country. However, Chinese economic activity quickly returned to normal levels while mine production elsewhere in the world was curtailed by the pandemic. Spot TCRCs fell back with the terms for sales of clean concentrate from miners to traders returning to the low levels seen during 2019 (Figure 19.2).

FIGURE 19.2 SUPPLY – DEMAND BALANCE VERSUS SPOT AND BENCHMARK COPPER SMELTING TERMS



Source: *Ocean Partners (2019)*

The global copper concentrate market is forecast to be better supplied from 2021. Production from new mines such as Spence, Grasberg Block Cave, Quebrada Blanca and Quellaveco will become available meaning that TCRCs are therefore expected to rise to from the low levels seen during 2019 and 2020 to a more normal range of $80/t and 8.0 cents/lb to $100/t and 10.0 cents/lb. Factors including falling acid prices, increased costs of environmental compliance for smelters and the growing power of the Chinese Smelter Purchase Team ("CSPT") mean that spot TCRCs are unlikely to return to the very low levels seen in 2010 and 2012.

Commercial terms for copper concentrate that have been assumed in the evaluation of Back Forty include:

<u>Delivery:</u>
CIF North America or parity

<u>Payables:</u>

Cu: Min	Max	Payable
Below	15%	96.5%, subject to minimum deduction of 1.5 units
15%	17%	96.5%, subject to minimum deduction of 1.2 units
17%	20%	96.5%, subject to minimum deduction of 1.1 units
20%	29%	96.5%, subject to minimum deduction of 1.0 unit

Ag: If the final Ag content is greater than 30 g/dmt, pay for 90% of the full content

Au: Pay according to the following schedule for deliveries to China:

Nil	if less than or equal to 1 g/dmt
90%	if over 1 g/dmt and up to and including 3 g/dmt
92%	if over 3 g/dmt and up to and including 5 g/dmt
93%	if over 5 g/dmt and up to and including 8 g/dmt
94%	if over 8 g/dmt and up to and including 10 g/dmt
95%	if over 10 g/dmt and up to and including 15 g/dmt
95.5%	if over 15 g/dmt

Back Forty copper concentrate is expected to reach grades in excess of 100 g/t Au and average approximately 50 g/t over the life of mine. At these higher grades, it may be possible to achieve a payable as high as 96.5%.

Treatment Charge:
$80/dmt

Refining Charges:
Cu: $0.08/lb payable Cu
Ag: $0.50/oz payable Ag
Au: $6.00/oz payable Au

Penalties:

Pb+Zn:	$2.00/dmt for each 1.0% > 3%
Hg:	$0.20/dmt for each 1 ppm > 10 ppm
Cd:	$2.00/dmt for each 100 ppm > 200 ppm

For the average grade of Back Forty copper concentrate, penalties are expected to average $7/t.

It should be noted that the stated average mercury content is comfortably below the current Chinese import limit of 100 ppm. For sales to European smelters, penalties are usually charged when the Hg level exceed 10-15 ppm. Japanese smelters are facing increasing issues disposing of mercury bearing waste and are therefore unlikely to be interested in purchasing material containing 10 to 20 ppm Hg.

19.3.3 Lead Concentrate Market

Unlike the zinc and copper concentrate markets, there is no “Benchmark” for lead concentrates. This is since there are too many different quality tiers, each with their own unique market fundamentals, within the overall lead concentrate market, to try to generalize. The differentiation between tiers has increased dramatically over the years and, as a result, comparisons to historical terms can be very misleading.

Spot TCs for lead concentrates did not decline as rapidly as those for zinc following COVID related mine production cuts during the first half of 2020. Typical terms for low to medium silver grade material fell from around $180/t in January and February to $140/t by June. Silver RCs associated with these terms have been in the region of $1.50/oz. It is thought that low levels of economic activity in much of the world combined with declining metal prices constrained the availability of scrap batteries during the period. Smelters were therefore more dependent on lead concentrates to meet production targets. Stricter environmental regulations have led to closures of many small secondary lead plants in China. This has created an opportunity for some of the larger primary smelters to invest in battery recycling facilities. As these smelters obtain an increasing proportion of lead units from secondary sources, they will increasingly be focussed on deriving by-product revenue from concentrates. Zinc (>7%) copper (>2%) and antimony (2.5%) typically received additional credit in the Chinese market. It is also expected that high grade lead concentrates will be favoured as they are required for blending with secondary materials in the smelter feed.

The forecast grade of Back Forty lead concentrates – relatively low lead (<40%) and silver (≤1,000 g/t) and relatively high gold (>20 g/t) is a unique quality. While the combination of grades may not be attractive to some smelters, the tonnage produced will be low and there should be little difficulty in blending output from Back Forty concentrate with other concentrates in a smelter's supply position given sufficient notice.

With the closure of Glencore's Belledune smelter, Teck's Trail facility is the only primary lead smelter remaining in Canada or the US. Trail has the capacity to recover precious metals and some by-products. Other overseas alternatives capable of treating precious metal-bearing lead concentrates are potentially Umicore, Berzelius, and Korea Zinc. China is also an alternative, however, at Au grades greater than 20 g the concentrate will currently be classified as a gold concentrate and be subject to VAT, hurting the economics of import into the Chinese market.

Commercial terms for lead concentrate that have been assumed in the evaluation of Back Forty include:

<u>Delivery:</u>
CIF Canada or parity

<u>Treatment Charge:</u>
TC: $160/dmt

<u>Refining Charge:</u>
Ag: $1.03/payable oz
Au: $20/payable oz

<u>Payable Metals:</u>

Pb	Pay 95% of the lead content subject to a minimum deduction of 3 units
Ag	Pay 95% of the silver content subject to a minimum deduction of 50 g
Au	Pay 95% of the gold content subject to a minimum deduction of 1 g

<u>Penalties:</u>
Cl+F: US$1.50-2.00 / dmt for each 100 ppm that the Cl+F exceeds 300 ppm
Hg: US$1.50-2.00 / dmt for each 10 ppm that the Hg exceeds 50 ppm
For the average grade of Back Forty lead concentrate, penalties are expected to average $3/t.

20.0 ENVIRONMENTAL STUDIES, PERMITS, AND SOCIAL OR COMMUNITY IMPACTS

20.1 ENVIRONMENTAL STUDIES

Aquila currently holds several permits as required in Michigan's environmental regulations. The current permits that have been issued for the Back Forty Project include:

- A Part 632 Mining Permit (MP 01 2016) for mining and beneficiation activities associated with the Back Forty Deposit.
- A Part 632 Mining Permit (MP 01 2016) Amendment.
- A National Pollutant Discharge Elimination System ("NPDES") Permit (MI0059945) for treated process wastewaters.
- A Michigan Air Use Permit to Install ("PTI") (205-15) has been issued for the Project for emissions associated with construction and mining activities.
- A PTI 205-15 Modification for the updated facilities.
- A Part 301 Inland Lakes and Streams and Part 303 Wetlands Protection Permit (WRP011785).
- A Dam Safety Permit application for the construction of the contact water basin ("CWB") and tailings management facility ("TMF") regulated under the Natural Resources and Environmental Protection Act ("NREPA"), Part 315 is under development. The Dam Safety Permit is currently the final major permit required prior to the construction and operation of the Back Forty Mine.

The following environmental baseline and engineering studies, initiated in 2007, have been completed or are currently underway. They are documented in the Environmental Impact Assessment (Foth, 2015b).

20.1.1 Geologic and Related Geotechnical Studies

Throughout the life of the Project, to date, various types of geologic and geotechnical drilling programs have been completed. These programs include:

- Metallurgical drilling: to extract additional mineralized and non-mineralized material in specifically targeted zones for related studies such as grindability, minerals processing and liberation, and waste rock characterization.
- Exploration drilling: to characterize economic mineralization, geologic structures, and lithologies.

- Geotechnical drilling: for sterilization and subsurface integrity in the vicinity of proposed mine infrastructure.

- Geotechnical drilling: in the vicinity of the proposed mine development, to complete the pit wall stability analysis.

- Geotechnical drilling: in the vicinity of proposed mine development, to aid in the design of the cut-off wall between the open pit and adjacent Menominee River.

Regional and local geology is well described and is consistent with known geology in the Upper Midwest region. The geotechnical investigations address the engineering data needs for the planned facilities.

20.1.2 Groundwater and Surface Water Hydrology and Quality

Beginning in 2008, the following study was commissioned to characterize regional groundwater and surface water conditions surrounding the Project:

- Environmental Resources Management, September 2011. *Hydrogeology Report Environmental Baseline Studies*, Appendix D-1.

A total of 25 surface water monitoring stations were included in the baseline study.

Concurrent with surface water studies, the baseline groundwater hydrology and water quality for the various hydrostratigraphic units and the potential impact of mine development were characterized to support the groundwater section of the environmental impact assessment ("EIA") and documented in the following reports:

- Environmental Resources Management, September 2011. *Hydrogeology Report Environmental Baseline Studies*, Appendix D-1.

- Foth Infrastructure and Environment, LLC, June 2015. *Subsurface Geotechnical Investigation for Mine Facilities*, Appendix D-2.

- Foth Infrastructure and Environment, LLC, July 2015. *Precambrian Bedrock Hydrogeological Report*, Appendix D-3.

- Foth Infrastucture and Environment, LLC, November 2015. *Groundwater Modeling Report*, Appendix D-4.

Aquila completed detailed field programs to characterize and document hydraulic characteristics of the Precambrian bedrock in a localized area around the proposed mine. This work was completed in mid-2011. The data complements the groundwater understanding for the Project and supports the mine inflow and groundwater quality modelling efforts.

In 2018, Aquila commissioned a confirmation baseline study to update the local and regional groundwater and surface water conditions for the Project. The study includes surface water analytical sampling, local and regional streamflow monitoring, and monthly measurement of groundwater levels in accordance with mine permit conditions. Additionally, in 2018 Aquila began the preconstruction monitoring program defined by the wetland permit conditions. The wetland monitoring program includes the installation of piezometers and continuous data loggers along transects perpendicular to predicted groundwater drawdown contours, wetland stage and flow monitoring using flumes and controlled survey techniques, water chemistry monitoring, soil permeability, and the calibration of the groundwater model and wetland drawdown analysis with an updated comprehensive baseline assessment.

20.1.3 Geochemical Characterization of Water Rock and Tailings

Potentially reactive materials, such as mineralized material, waste rock, overburden, and peripheral rock has been characterized and identified in the *Geochemical Investigation Report* (Foth, 2015b). The Project has followed industry standards in relation to the characterization of waste streams, to assess the potential for acid rock drainage and/or metal leaching ("ARD/ML") from the anticipated waste materials. Static, kinetic, and mineralogical testing programs have been completed, and the data from these studies are being used to aid in the development of engineering plans for waste and water management facilities and reclamation plans for the Project.

13 lithologies have been identified at the site. The ARD/ML assessment of potential waste rock examined (using primary and duplicate samples) 481 static samples and 40 kinetic and mineralogical samples. 11 tailings samples representing early process plant tailings streams, a duplicate, and composite streams from two alternative mine designs were also tested using the same static, kinetic, and mineralogical test methods.

Static testing results showed four lithologies (felsic dike, mafic dike, quartz feldspar porphyry, and Cambrian sandstone) are not expected to generate acidity. Eight lithologies (chloritic crystal tuff, massive sulfide, massive flow rhyolite, rhyolite ash tuff, rhyolite crystal tuff, sulfide stringer zone, semi-massive sulfide, and tuffaceous/exhalative sediments) underwent kinetic testing, representing 77% of the total anticipated waste rock. Low pH leachate was observed in those kinetic tests. The final lithology is the gossan, which was not tested because it will be completely removed as process plant feed. Tailings samples are expected to generate acidic leachate. Further, kinetic testing showed waste rock and tailings readily leaching a variety of metals. These results indicate the waste management structures will have to address the potentially acidic and metal leaching characteristics of the stored materials.

Studies have been commissioned to model water quality within various facilities including the North Waste Rock Facility ("NWRF"), South Waste Rock Facility ("SWRF"), TMF, open pit, mineralized material stockpiles, and CWB. Water quality is modeled to support project activities such as the development of a limestone amendment plan, the detailed design of the wastewater treatment plant ("WWTP"), and the EIA. The evaluation and modeling are based on the designs described in the *Feasibility Design of Tailings and Waste Rock Management Facilities Report* (Golder, 2018) and the geochemical characterization described in the *Geochemical Investigation Report* (Foth, 2015b). The assumptions described in *Water Quality Models for Open Pit and Tailings and Waste Rock Management Facilities* (Foth, 2015b) and *Tailings Management Facility Geochemical Model* (Foth, 2018a) apply to these models as well.

Water quality modeling predicts that, with the addition of limestone, circumneutral pH can be maintained during all modeled time periods. This conclusion is consistent with results of previous modeling efforts (Foth, 2015b).

20.1.4 Wetlands

Wetlands have been extensively studied at the site and are documented in the MDEQ/USACE Joint Permit Application for: Wetland Protection; Inland Lakes and Streams; Floodplain (Foth et al., 2017). Wetlands of various sizes and classification encroach across the site. Although the mine and processing facilities have been located in a compact area, every effort has been made to avoid and minimize wetland impacts.

Initial wetland delineations were completed in 2010 and 2011. Additional delineation and wetland identification updates took place in 2017, 2018, and 2019. In 2018, after receiving the wetland permit (WRP011785), pre-construction wetland monitoring and reporting commenced. Various studies were completed during the wetland application process, and updates of associated plans are being administered as data is collected and the pre-construction timeline is executed. Studies include:

- Comprehensive Baseline Assessment and Monitoring Plan.
- Revision of the MODFLOW groundwater model.
- Wetland Mitigation and Monitoring Plan.
- Wetland Restoration Plan.
- Stream Mitigation and Monitoring Plan.
- Secondary Wetland Impacts Analysis.
- Identification of feasible and prudent alternatives Floristic Quality Assessment.
- Comprehensive Adaptive Management Plan.
- Long-term management plan and third-party stewardship agreement.
- Unanticipated discovery plan for cultural, historical, and/or archeological resources.

The primary controls to minimize wetland impacts are defined in the Adaptive Management Plan. The wetland mitigation plan identifies preservation and restoration as the proposed mitigation method for those wetlands unavoidably impacted by the Project. A 507.74 acre preservation parcel with 294.24 acres of wetlands has been proposed to the Department of Environment Great Lakes and Energy (“EGLE”) and conditions for mitigation, restoration, and preservation are written in the Wetland Permit. Continuous monitoring of on-site wetland hydrology and the reporting of performance standards will gauge direct and secondary wetland impacts due to pit dewatering during operations and will be used to calibrate the groundwater model.

20.1.5 Aquatic Biology and Terrestrial Vegetation and Wildlife

Aquatic surveys and assessments address aquatic biota and their habitats in the Menominee River, Shakey River, and Shakey Lakes systems. Original baseline sampling documented in the original Mining Permit Application (Foth, 2015) and Mining Permit Amendment Application (Foth, 2018) provides an understanding of presence and species of aquatic biota in and around the Project site. Prior to commencement of construction, additional baseline sampling is required under the Mining Permit. With understanding of aquatics provided by the original survey, an additional aquatics preconstruction survey is proposed. The elements of the aquatic survey program moving forward are:

- Water quality sampling.
- Sediment sampling.
- Fish community assessment.
- Macroinvertebrate community assessment.
- Habitat assessment.
- Periphyton community assessment.
- Phytoplankton and zooplankton community assessment, mussel tissue testing.

Baseline conditions for terrestrial flora and fauna are fully documented in the EIA (Foth, 2015b) and Environmental Impact Assessment Amendment ("EIAA") (Foth, 2018b). The results of these studies provide comparative information for future surveys during preconstruction, construction, operation, and post closure phases of the Project.

On-going terrestrial flora monitoring to confirm baseline conditions, and address trends during construction and operations will include annual observations of plant species along transects. Meander surveys through upland habitats and surveys within the established transects will address the scope of upland vegetative surveys.

Terrestrial wildlife monitoring during the Project operations will include annual and semi-annual observations of amphibians, reptiles, birds, and mammals at designated survey sites previously studied. Observations will be documented and included with the Project's annual report. The fauna monitoring will be completed to confirm baseline conditions and document the trends and conditions of these resources during operations.

Over the life of the mine, the data and observations will be documented by qualified professionals and will assist in identifying trends in biota in and around the site. These trends, along with other media data such as groundwater and surface water quality and hydrologic parameters, will be used to evaluate whether an observed trend is related to the Project.

20.1.6 Air Quality and Meteorology

In 2007, Aquila commissioned a study to document baseline air quality and meteorological conditions at the site, documented in Foth (2015a). Two years of quarterly air quality and meteorological monitoring data were collected, beginning in Q3 2007. Wind speed and direction, temperature, solar radiation, precipitation, and particulate matter 10 µm and less (PM10) were collected. These data were used in evaluating the potential impacts of the Project.

Undeveloped areas, such as the Project area, have very good air quality. The largest city with industrial activity is Menominee, Michigan – Marinette, Wisconsin, an area 50 km south of the site with a combined population of approximately 20,000.

20.1.7 Cultural and Historical Resource Studies

The Property and proposed development area have been investigated for the existence of cultural resources and historical artefacts documented in:

- Commonwealth Cultural Resources Group, Inc, ("CCRS") July 2015. Archaeological Investigations of the Aquila Resources Inc., Back Forty Project Area.

- 106 Group, September 2015. Phase 1 Archaeological Resources Survey for the Back Forty Project – Private Land North.

- 106 Group, November 2019. Phase I Survey for the Back Forty Project: Waste Rock Facilities and East Corridor.

Pre-field research provided baseline information relating to known or postulated cultural resources within the Sensitivity Study Area and considered a large area extending on both Michigan and Wisconsin sides of Menominee River. This area had a historical presence of Native Americans and the Menominee River was significant as a water source and for transportation. 27 previously undocumented potential archaeological sites were discovered as a result of CCRG's 2009 survey of portions of the Back Forty Project Area. These ranged from mounds possibly containing human remains, ancient campsites and homesteads, to lithic scatter and debitage flakes. Recommendations on the significant finds are potentially eligible to the National Register of Historic Places ("NRHP"). The identified resources are classified as potentially eligible, not eligible, not cultural, or unevaluated where archaeologists could not determine whether the site artifacts were of cultural origin. Eight sites were potentially NRHP eligible, and 10 sites were unevaluated. The remaining 9 sites were classified either not eligible or not cultural. All sites potentially NRHP eligible or unevaluated that were recommended by CCRG be excluded from ground-disturbing activities have been avoided in the Project footprint along with a 30 m buffer around the site. The later studies did not identify any additional potentially eligible sites within the updated facility footprint or expanded eastern corridor.

20.2 KEY ENVIRONMENTAL PROTECTION ISSUES

The following items represent key environmental protection issues that will be incorporated into the engineering design of the Project, to avoid or mitigate potential impacts. They have been identified from similar mining operations, the regulations, and the environmental priorities of the region.

20.2.1 Open Pit Proximity to the Menominee River

The Back Forty Deposit is located adjacent to the Menominee River, which creates a boundary line between the states of Michigan and Wisconsin. The mineralized material will be extracted from the open pit, located in close proximity to the river (approximately 46 m). Concerns include mine dewatering impacts on river flow, structural integrity of the pit wall between the river and open pit mine, and potential for post-reclamation impacts on water quality in the river. The design, operations, reclamation, and continued monitoring program accounts for those concerns.

Design and construction of the cut-off wall is based on mature technology. Its presence will greatly reduce the hydraulic conductivity between the backfilled pit and the river. Modelling and technical issues are addressed in the current permit applications, with additional requirements and submittals to be reviewed and determined by regulatory agencies.

20.2.2 Management of Waste Rock and Tailings and Site Reclamation

The waste rock and tailings that will be generated from the Project will contain sulphide mineralization and the potential for ARD/ML. The engineering plans for storage and management of waste materials are required as part of the treatment and containment plan. These plans are based on common industry standards and practice, including engineering designs to mitigate impacts from ARD/ML by incorporating appropriate liners, covers, and leak detection systems. Specially designed storage facilities including liners and monitoring requirements will prevent environmental impacts during mining activities and after mining ceases. Waste handling methods are carefully considered to maintain the structural integrity of both the backfilled pit and the remaining TMF.

20.2.3 Archaeological Artifacts

As noted in Section 20.1.7, cultural resources including archaeological artifacts have been documented at the site (CCRS, 2015). Of the professionally identified and documented artifacts and sites, none have been established as eligible for NRHP. Mining structures will be sited to avoid or mitigate impacts to cultural resources located in the vicinity of the mine operations and have been recommended as sites of interest as “potentially eligible” or “unevaluated” for NRHP. These defined archeological sites have been provided a 30 m buffer within the Project area. Once construction commences, adherence to an unanticipated archaeological discovery plan will be followed.

20.2.4 Wetlands Protection

The Wetland Permit considers wetland protection, inland lakes and streams, and the floodplain. Extensive modelling has been prepared to estimate the Project impact on wetlands. Monitoring and mitigating strategies will be in place to adaptively manage unanticipated effects.

20.2.5 Listed and Protected Species and Sensitive Habitat in the Area

Vegetative and wildlife studies have been conducted and a small number of threatened, endangered, and special concern species have been identified in the area (Foth, 2015a).

The Menominee River sustains several aquatic species in the vicinity of the proposed WWTP discharge location. A mussel relocation plan has been prepared to address the mussels in the discharge area. Additionally, the NPDES permit contains an acute toxicity test required using mussels, as well as the common water flea (*Ceriodaphnia dubia*) and fathead minnow (*Pimephales promelas*).

In preparation for construction, an ETSC evaluation and relocation plan for listed terrestrial species will be prepared for approval in accordance with permit conditions. Prior to clearing and grubbing, a vegetative survey will be conducted to identify any listed species specimens within the area. If appropriate, specimens will be relocated to a suitable community outside the Project area.

20.2.6 Particulate Emissions

Extracting, hauling, and handling large quantities of rock (overburden, waste rock, mineralized material), and processing and transporting mineralized material and concentrates, will generate air emissions including particulate matter. To mitigate potential impacts, emissions controls will be provided by air pollution control equipment (baghouses, spray bars, etc.) and practices outlined in a fugitive dust control plan. Part 55 of NREPA, Act No. 451 of 1994 as amended (Part 55) Air Pollution Control provides the requirements for the facility emissions and the framework for the air permit needed for the Project. The air permit application (Foth, 2015d) provides information on the air pollution control equipment and emissions have been demonstrated as acceptable according to applicable rules. Particulate deposition has been evaluated on effects on water quality and soils and has been demonstrated as acceptable according to applicable rules.

20.2.7 Waste Disposal

Back Forty mineralized material is potentially reactive and acid generating. The State of Michigan has specific requirements for acceptable methods of mine waste management (Part 632 of NREPA, Act No. 451 of 1994 as amended (Part 632)). The rules addressing treatment and containment of materials including waste rock and tailings are contained in R 425.409.

As the mine is developed, waste rock will be placed in the lined disposal facilities. As the Deposit is accessed and processing begins, tailings will be placed in a lined disposal facility. Limestone amendment may be required at disposal facilities to maintain circumneutral pore water/leachate. According to the current reclamation plan, a portion of waste rock will be backfilled to the open pit after cessation of mining. Residual waste rock not used for backfilling the open pit and tailings will remain in the lined disposal facility for tailings and will be capped with an engineered cover. This cover will minimize infiltration from precipitation and reduce the duration of final drain-down and contact water management/treatment during reclamation.

20.3 SITE MONITORING

Monitoring is required throughout operations and after mine closure. A groundwater and surface water monitoring network extending beyond the mine site, initiated in baseline environmental studies, will be supplemented to provide operational monitoring in accordance with the NREPA Part 632 Mining Permit. The site monitoring program for the life-of-mine is fully documented in:

- Foth, January 2020 “Environmental Monitoring Plan Sampling and Analysis Plan Quality Assurance Project Plan”

Monitoring includes groundwater, surface water, wetland hydrology, biological monitoring, soil testing, mine facility monitoring, and post-closure monitoring. Environmental protection measures include impervious surfaces, liners, Leachate Collection System (“LCS”), Leachate Detection System (“LDS”), Side Slope Risers (“SSR”), soil erosion control, and covers. Annual reports are required with any maintenance and corrective action noted. After cessation of operations, some, or all, of the monitoring network will remain in place and monitoring will continue. Post-closure monitoring and maintenance activities shall continue until a request to modify the post-closure monitoring plan is approved by the MDEQ. Financial assurance considers these costs.

20.4 WATER MANAGEMENT

Contact water is anticipated to be generated during operations from a variety of sources:

- Groundwater inflow into the open pit.
- Net precipitation inputs: haul roads, WRFs, mineralized material stockpiles, the TMF, the process plant area, the open pit, as well as the CWB itself.
- Effluents from the mine truck wash, and process plant diesel area sump.

Surface run-off that has come in contact with these areas will be collected via a series of ditches and channels as required and directed preferentially via gravity, or where required, via pumping to the CWB.

As discussed in Section 18.8, mine dewatering water will be directed to the raw water settling tanks and will undergo an oil/water separator procedure prior to re-use in the process as raw water. Excess mine dewatering water that cannot be re-used in the process will be directed to the CWB from the raw water settling tanks via the raw water settling tanks pump. As discussed in Section 18.9, contact water from the CWB will be treated prior to discharge. Water will be discharged to the environment meeting NPDES permit requirements.

A groundwater model was prepared to estimate mine water inflow as the pit develops (Foth, 2015a). A collection system and WWTP will be designed to accommodate anticipated and modeled operating and climate conditions. The systems will be constructed and put into operation early in the Project, during construction. Water will be discharged to the environment meeting NPDES permit requirements.

Upon cessation of operations, water management and treatment needs will be reduced. Mine facilities will be reclaimed (either removed or converted to other uses) and contact water sources requiring treatment will be limited to leachate from the closed TMF. Post-closure treatment, monitoring, and ongoing maintenance of the TMF and subsequent leachate will likely be required for a period of time after mine closure. Any modifications to the closure and continued monitoring methods of the TMF and its effluents will be regulated and approved by EGLE. Financial assurance put forth by the Company considers these ongoing costs.

20.5 PROJECT PERMITTING REQUIREMENTS

As with any proposed project, a number of permits are required or may be required prior to construction of the Project. The following permits have been approved:

Mining Permit No. MP 01 2016 issued December 28, 2016 addresses the overall proposed Project, and based on the permit application, how Part 632 requirements will be met. The application included information, as listed in the rules under R425.201, is as follows:

- Permit application form.
- Permit application fee.
- Environmental impact assessment.
- Mining, reclamation, and environmental protection plan.
- Contingency plan.
- Financial assurance plan.
- List of other necessary permits and licenses.
- Organization report.

Amendments to Mining Permit MP 01 2016 were approved on December 12, 2019. These amendments addressed several conditions which consider the updated facility design, additional monitoring requirements, and an updated financial assurance model.

The EIA and EIAA developed as part of the Mine Permit and Mine Permit Amendment applications provided the basis upon which to describe the potential environmental impact of the proposed Project. Impacts on the following topics were addressed:

- Topography and drainage.
- Soils.
- Geology and hydrology (including groundwater and surface water).
- Water supply (public and private).
- Wetlands and floodplains.
- Natural, wild, and scenic rivers.
- Wilderness areas.
- Flora and fauna.
- Threatened and endangered species.
- Cultural, historical, archaeological resources.
- Air quality, meteorology, and climatology.
- Aesthetic resources.
- Noise, light, and seismicity.

- Feasible and prudent alternatives analysis.

The Project was issued NPDES Permit MI0059945 on April 5, 2017 for discharge to surface water. In the application, a water balance for the facility, a preliminary wastewater treatment plant design, and predicted influent and effluent concentrations were described as well as a summary of the State and Federal requirements and how they will be fulfilled.

The Project was issued Michigan Air Use Permit - Permit to Install 205-15 on December 28, 2016. The application (Foth, 2015d) included an emissions inventory, demonstration of compliance with applicable standards by dispersion modelling, description of pollution control equipment, and a fugitive dust control plan. As part of the environmental impact analysis, an air deposition model was conducted to estimate impacts from particulate matter deposition.

The Michigan Air Use Permit – Permit to Install 205-15A was issued as a modification for the updated Project facilities on December 12, 2019. The updated permit added several new conditions, which mostly relate to mercury in the retort and refining furnace operations.

The Project was issued the Part 301 Inland Lakes and Streams and Part 303 Wetlands Protection Permit (Wetland Permit WRP011785) on June 4, 2018 authorizing the excavation and fill of material within 11.22 acres of wetlands and up to 17.17 acres of secondary impacts due to reductions of hydrology.

A Dam Safety Permit is required pursuant to the Part 315 rules promulgated under NREPA for any facility over 6 feet in height and over 5 acres are impounded during the design flood which is applicable to the CWB and TMF facilities designed for the Project. The Dam Safety program administered by EGLE-WRD focuses on ensuring that dams are properly constructed, inspected and maintained, and are adequately prepared for potential emergencies. A Dam Safety Permit for the Project facilities was submitted to EGLE in November 2018 and subsequent review determined that preconstruction studies required as part of WRP011785 including the updated groundwater modelling efforts calibrated using the 2019 baseline hydrologic data, and the revision of the secondary impacts analysis is required prior to EGLE issuance of the Dam Safety Permit. Data collection and modelling efforts beginning in 2019 to be completed in 2020 will address this requirement.

As the Project has previously developed through Feasibility Study design and the mine plan refined, the permits above may need to be amended. Michigan has a well-defined process to amend Mining Permits, as established in Public Act No. 162, and other permits.

Other required permits and plans include:

- Part 91 soil erosion and sedimentation control (“SESC”) permit and approved SESC plan along with a Notice of Coverage (“NOC”) to obtain NPDES permit authorization pursuant to Part 31, Water Resources protection of the NREPA for stormwater management during construction and Notice of Intent (“NOI”) for storm water management during operations.

- Spill Prevention Control and Countermeasures ("SPCC") Plan for the fuel storage area that conforms to 40 CFR 112 compliant with the Part 5 rules promulgated pursuant to Part 31 of the NREPA.

- Pollution Incident Prevention Plan ("PIPP") to address potential spillage of fuel, salt, and other polluting materials in compliance with R 324.2001 through R 324.2009.

- Cyanide Management Plan ("CMP") that complies with applicable state and federal standards.

- Limestone Amendment Plan for the WRF and TMF.

- The plans and specifications for the TMF and WRF and the operations plan for the TMF and WRF, including any EGLE-approved modifications.

- Design certifications of liners, covers, and leachate collection systems.

- WWTP Engineering Design Plans.

- The Company continues to evaluate other local requirements that may be applicable.

20.6 MINE CLOSURE

The mining, reclamation, and environmental protection plan required under rule R425.201 requires a strategy for the proposed final reclamation activities, including the anticipated schedule, sequence, and duration of reclamation as required under R425.204 – Reclamation Plan.

The reclamation plan was submitted with the Mining Permit Application (Foth, 2015b) and included the following closure procedures and post-closure monitoring plans:

- Anticipated final land contours.

- Proposed final land use.

- Soil erosion and sedimentation structures that will remain after reclamation. In addition to mitigating structures, all disturbed surfaces will be stabilized and re-vegetated. Site re-vegetation will be reclaimed to achieve a self-sustaining ecosystem appropriate for the region, considering short- and long-term climate change.

- Plans and schedules for removing or stabilizing waste rock facilities, water management ponds, tailings disposal facilities, overburden banks, open pit banks and walls, Project area roads and mine site infrastructure.

- Final disposition of all toxic and hazardous wastes, refuse, and tailings managed in a manner that protects the environment and all applicable laws.

- The open pit will be backfilled with waste rock and amended to maintain circumneutral pore water. This configuration has been successfully completed at the Flambeau Mine in Wisconsin. The Project groundwater model shows groundwater/surface water that will flow towards the Menominee River after site reclamation will meet water quality criteria for the Menominee River.

- An engineered cover system will be installed on the TMF to prevent precipitation infiltration. Leachate will be collected and treated until the process is deemed no longer necessary by regulatory agencies.

Reclamation cost estimates are the basis of the financial assurance requirements of Part 632 as covered under rule R425.301. Financial assurance applies to all mining and reclamation activities covered under the Mining Permit and are to be sufficient to cover the cost to administer and hire a third party to implement the reclamation, remediation, and post-closure monitoring required. The financial assurance cost estimate includes costs for reclamation, remediation of contamination of air, surface water, and groundwater along with administrative oversight, monitoring, fees, and reasonable contingencies.

Financial assurance consists of an approved assurance instrument or combination of instruments covering at least 75% of the required amount. Unless modified by forfeiture, transfer of ownership, or other specified circumstance, financial assurance requirements continue through reclamation until the final release upon termination of the mining permit.

20.7 SOCIAL AND COMMUNITY IMPACTS

Feedback to this Project is continually requested and submitted by the public as part of Aquila's ongoing efforts to engage in operational transparency and information sharing with the local community and other affected stakeholders. This engagement strategy dates to almost a decade ago under previous Project proponents. Engagement events and opportunities deployed include, but are not limited to:

- Participation in meetings with local units of government.
- Community Advisory Group meetings.
- Customized meetings to explain Project objectives and proposals.
- Geology-based educational outreach activities.
- Meeting with Menominee Alliance for Progress.
- Outreach to Native American Communities.
- Grant-writing for community advancement.
- Multiple scholarship awards.
- Contributions/sponsorships of local organizations and events.
- Volunteer positions or other participation methods at local festivals/fairs.
- Open houses.
- Site Tours.
- Educational field trips for organizations and universities.

The Back Forty Project represents many employment opportunities that have attracted interest of unions and business groups. As such, the Project has seen strong support from legislators and regulators. Aquila has developed a good working relationship with the Upper Peninsula Construction Council to ensure availability of local skilled labour.

20.7.1 Tribal Relationships

Tribal engagement has been very important to the Project, especially considering the cultural resources near the site. Outreach to local Tribes, including the Menominee Indian Tribe of Wisconsin, began in June of 2010; Aquila plans to continue working with the Tribes to better understand their concerns and to find opportunities to work together on issues that are important to both parties such as communication on the preservation of and unanticipated discovery plan of historical artifacts.

Currently, there are four ongoing legal actions involving the Menominee Tribe, which opposes the Project:

1. A federal lawsuit against the U.S. Army Corps of Engineers and the U.S. Environmental Protection Agency claiming the agencies failed to take "primary responsibility" for wetland permitting. The Tribe’s claims were dismissed by the federal trial and appellate courts. The Project anticipates that the Tribe will seek further review by the United States Supreme Court, but the likelihood of the Court granting further review is low.

2. An appeal to the state circuit court of the Michigan Department of Environment, Great Lakes and Energy’s (“EGLE”) final decision and order approving the Project’s initial mine permit.

3. An administrative contested case challenging EGLE’s approval of amendments to the Project’s mine permit.

4. An administrative contested case on EGLE’s approval of the Project’s wetland permit.

21.0 CAPITAL AND OPERATING COSTS

21.1 INITIAL CAPITAL COST

21.1.1 Summary

The capital estimate for the Project is summarized in Table 21.1 by major area.

All costs are expressed in United States Dollars unless otherwise stated and are based on Q3 2019 pricing and deemed to have an overall accuracy of ±25%. The capital cost estimate conforms to Association for the Advancement of Cost Engineering International ("AACEI") Class 4 estimate standards as prescribed in recommended practice 47R11.

The capital cost estimate was based on an overall engineering, procurement and construction management ("EPCM") implementation approach and horizontal (discipline based) construction contract packaging. Equipment pricing was based on a combination of budget quotations and actual equipment costs from recent similar Lycopodium and P&E projects considered representative of the Project.

TABLE 21.1
INITIAL CAPITAL ESTIMATE SUMMARY BY AREA
(Q3 2019, ±25%)

Item	Capital Costs ($M)
Construction Indirects	11.4
Oxide Plant	24.1
Sulphide Plant	57.5
TMF/WRFs	42.6
Infrastructure	34.2
Mining	23.6
EPCM	15.7
Owner costs	11.4
Subtotal	**220.6**
Contingency	29.9
Total	**250.4**

Table 21.2 lists the exchange rates that have been used in the compilation of the estimate.

TABLE 21.2 CURRENCY EXCHANGE RATES	
Currency	**1 USD =**
CAD	C$1.33
AUD	A$1.41
EUR	€0.88
ZAR	R14.34

Foreign currency exposure is shown in Table 21.3.

TABLE 21.3 FOREIGN CURRENCY EXPOSURE	
Currency	**% of Capital Estimate**
USD	~90
CAD	~8
AUD	~1
EUR	~1

21.1.2 Open Pit Mine Capital Costs

The open pit mine capital cost has been subdivided into four areas; (i) equipment (ii) capital leases, (iii) other mine development and (iv) freight and spares.

The open pit mine capital cost estimate is mainly developed from first principles, determining quantities and applying unit pricing. Unit pricing information is derived from in-house databases as well as vendor quotations for major items.

All costs are in US dollars. Table 21.4 summarizes the initial open pit mine capital cost incurred in the two pre-production years.

TABLE 21.4 OPEN PIT MINE INITIAL CAPITAL COST			
Capital Costs	**Year -2 ($k)**	**Year -1 ($k)**	**Total ($k)**
Equipment and Down Payments	2,096.5	1,527.0	3,623.5
Equipment Capital Leases		4,523.5	4,523.5
Mine Development		15,052.9	15,052.9
Freight and Spares	104.8	302.5	407.3
Total Mine Capital	**2,201.3**	**21,405.9**	**23,607.2**

21.1.2.1 Open Pit Mining Equipment

The procurement of open pit mining equipment assumes that capital leasing arrangements will be available for all equipment. This may be through the vendors or via third parties. Capital leasing will lower the initial capital costs by deferring the equipment purchase costs over time, although the interest component will increase the total cost for the unit. The leasing input assumptions are:

- Down payment of 10% at the time of procurement.
- 60-month lease period.
- 6% interest rate.
- 0% finance fee.

The initial equipment procurement requirements are summarized in Table 21.5.

21.1.2.2 Open Pit Mine Development

The details for the open pit mine development activities are shown in Table 21.6. This includes capitalized pre-stripping undertaken in Year -1, and well as the construction of haul roads, topsoil recovery, stockpile preparation, and water control.

Some earthworks are deferred into production Year 1, and are therefore not part of the initial capital cost. For example, construction of the second stockpile pad near the pit is deferred into Year 1.

The mine pre-stripping activity will be undertaken in Year -1 by the mine fleet. The costs for this activity are estimated in the operating cost model; however, those costs are allocated to pre-production capital. Some of the materials pre-stripped from the open pit will be used in various construction activities across the site. Therefore, the cost to deliver this material to the final destination is included in the mining cost, however any specific costs needed for special processing or placement are not included in the mining cost.

TABLE 21.5 OPEN PIT MINE INITIAL EQUIPMENT				
Equipment	**Buy or Lease**	**Unit Cost ($k)**	**Year -2 and -1 Units**	**Total Cost ($k)**
Drill, 90 mm, Crawler, Percussion	Lease	63.0	2	126.0
Drill, 165 mm, Crawler, DTH	Lease	131.5	3	394.5
Stemming Truck, 15 t	Lease	2.8	1	42.8
Hydraulic Shovel, 10 m^3 (C6018)	Lease	320.0	2	640.0
Excavator, 5 m^3 (C374F)	Lease	95.0	1	95.0
Haul Truck 90 t (K785)	Lease	138.5	8	1,108.0
Personnel van/bus	Lease	10.0	1	10.0
Dozer (D8T)	Lease	85.0	3	255.0
Mechanic and Welding Truck	Lease	39.4	1	39.4
Excavator, 4 m^3 (CAT 336E)	Lease	30.8	1	30.8
Fuel and Lube Truck	Lease	87.0	1	87.0
Grader (GD655)	Lease	30.5	2	61.0
Light plant	Buy	25.0	4	100.0
Pickup truck	Lease	4.0	10	40.0
Pit Water Pumps Diesel	Buy	50.0	3	150.0
Wheel Loader 4 m^3 (C966)	Lease	45.5	1	45.5
Water truck (40 ton 6,500 gallon)	Lease	78.5	1	78.5
Wheel Loader 10 m^3 (WA800)	Lease	160.0	2	320.0
Initial Equipment Capital				**3,623.5**

TABLE 21.6 INITIAL OPEN PIT MINE DEVELOPMENT	
ITEM	**Year -1 ($k)**
Pre-Stripping (from opex)	7,115.0
Stockpile Pad 1 cut and fill	188.6
Stockpile Pad 1 liner pad	3,325.0
Stockpile Pad 1 lined ditching	224.4
Stockpile Pad 1 sump (excavate)	124.0
Stockpile Pad 1 sump (liner)	241.8
Stockpile Pad 1 pipeline	50.0
Stockpile Pad 1 pump	50.0
Crusher Retaining wall	946.9
Crusher Ramp Retaining wall	1,506.4
Compactor Rental (ea)	180.0
Topsoil Recov - Year -1	85.1
Topsoil Recov - Stockpile Pad 1	65.6
Topsoil Recov - Haulroad MR1	10.9

TABLE 21.6 **INITIAL OPEN PIT MINE DEVELOPMENT**	
ITEM	**Year -1 ($k)**
Topsoil Recov - Haulroad MR2	4.5
Topsoil Recov - Crusher Ramp	8.6
Topsoil Recov - Crusher Pad	5.4
Haulroad MR1 Cut and Fill	87.0
Haulroad MR2 Cut and Fill	33.0
Road capping HaulroadMR1	36.8
Road capping Haulroad MR2	16.1
Road capping Haulroad MR3	143.9
Road capping - Crusher Pad	20.1
Road capping - Haulroad Crusher Ramp	33.7
Pit Pumping equipment	100.0
Pit Pipelines	70.0
Office Equip plus Software	200.0
Radio Communications + GPS	100.0
Survey Equipment	80.0
Sub-total	**15,052.9**

21.1.3 Process Plant and Associated Infrastructure Capital Cost

21.1.3.1 Estimating Methodology

The capital costs for the process plant and infrastructure capital are based on the facilities described in Sections 17 and 18 of this Technical Report, prepared by Lycopodium with input from Golder on the TMF and WRFs, and Hatch on water management and the waste water treatment plant. The purpose of the capital cost estimate is to provide substantiated costs which can be utilized to assess the preliminary economics of the Project.

Process plant general arrangement drawings were produced at a PEA level to permit the assessment of preliminary engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the processing plant and infrastructure.

The layouts were based on the 2018 Feasibility Study phase concepts for the Project.

Unit rates from the 2018 Feasibility Study were updated to reflect the current contractor pricing in the Upper Peninsula region in Michigan.

Updated budget pricing for equipment and infrastructure facilities were obtained from suitable suppliers and contractors. All in packages for oxide process plant circuits have been incorporated to allow for lower construction cost and ease of construction assembly, overall aiding to lower capital expenditures. Several buildings in the Project have been considered under a lease to own option as a mechanism to offset capital during the initial phases of construction.

21.1.3.2 Project Implementation Strategy

The implementation strategy for the Project is based on an EPCM implementation approach and horizontal discipline-based contract packaging. Horizontal packages are for the earthworks, building works, concrete works, field erected tankage, structural, mechanical and piping installation, electrical and instrumentation supply and installation. An experienced engineering firm will be engaged to provide engineering and procurement ("EP") services for the development of the process plant and the associated infrastructure. An experienced engineering firm will also be engaged to provide construction management ("CM") services as part of an integrated team with Aquila for the development of the process plant and the associated infrastructure.

21.1.3.3 Quantity Development

The Project works were quantified for the defined scope of work and to enable the application of rates to determine costs. Allowances for earthworks compaction, waste, rolling margin and the like are included in the build-up of unit costs.

Quantity information was derived from a combination of sources and categorized to reflect the maturity of design information as follows:

- Study Engineering: includes quantities derived from concept or preliminary engineering for the purpose of the study.

- Estimated: Includes quantities derived from sketches or redline mark-ups of previous Project drawings/data by estimating or similar projects.

- Factored: Quantities derived from percentages applied as a factor derived based on experience.

The estimate is an amalgamation of engineering, material take offs ("MTO's") and in-house benchmarks. The level of accuracy and detail in the estimate varies based on the engineering progress of the given scope and discipline. Major quantities are summarized in Table 21.7.

TABLE 21.7 MAJOR QUANTITY SUMMARY			
Description	**Quantity**	**Unit**	**MTO Type**
Earthwork			
Backfill	215,964	m^3	MTO based on current layout
Excavation	23,098	m^3	MTO based on current layout
Strip topsoil	368,397	m^3	MTO based on current layout
Concrete			
Concrete	12,220	m^3	MTO based on current layout
Steel Work			
Structural Steel*	979	t	MTO based on current layout
Grating	2,814	m^2	MTO based on current layout
Cladding	5,383	m^2	MTO based on current layout
Platework			
Platework	200	t	Detailed MTO as per equipment list
Tankage			
Tanks	138	t	Detailed MTO as per equipment list
Piping			
HDPE Piping, overland + tailings	42,445	m	MTO based on current layout
CS Piping	1	lot	Factored
Electrical			
Cabling	1	lot	Factored
Power Lines (Overhead and incoming to site)	1	lot	MTO's based on current layout
Buildings			
Buildings, only pre-engineered type	4,263	m^2	MTO based on current layout

***** *Structural steel includes for concentrate loadout building*

21.1.3.4 Pricing Basis

Table 21.8 summarizes the source of costs included in the capital cost estimate.

TABLE 21.8 **SUPPLY COST SOURCE**			
Description	**Allowance / Factored (%)**	**Historical Pricing (%)**	**Budgetary Quote (%)**
General	0	61	39
Earthworks	0	0	100
Concrete	1	4	95
Steelwork	4	11	85
Platework/ Tankage	10	11	31
Mechanical	2	7	78
Piping	0	0	100
Electrical	97	3	0
Instrumentation and Control	92	0	8
Buildings and Architectural	26	3	63
Mining	0	100	0
Owners Costs	0	0	0

21.1.3.5 Earthworks

Quantities for plant site bulk earthworks and roads were prepared based on the process plant layout drawings and overall site plan. Rates were solicited from local earthworks contractors who have undertaken similar works in the state.

21.1.3.6 Concrete

Quantities for concrete works were established using the 3-D model and general arrangement drawings prepared for the study and preliminary engineering. Rates for concrete works were solicited from local civil contractors to be of a suitable size and experience and currently undertaking work in Michigan. Unit rates include the set up and operation of the batch plant, delivery of concrete to the forms, formwork, rebar supply and installation, placing concrete and any finishing requirements. Rates and quantities were prepared on a composite per cubic metre basis for each specific type of concrete assembly.

21.1.3.7 Steelwork

Structural steel quantities were established using the 3-D model and general arrangement drawings prepared for the study. Material take-off for individual structural element was prepared for the process plant site.

Rates for structural steel fabrication, packaged ready for sea freight were solicited from local fabricators and overseas suppliers from SE Asia. The scope included the preparation of workshop fabrication drawings, marking plans and bolt lists. Site installation hours were based on Lycopodium’s experience and supported by contractors undertaking work in Michigan.

21.1.3.8 Platework and Shop Fabricated Tanks

Platework and tankage quantities were calculated using the sizing provided in the plant mechanical equipment list prepared for the study as the basis. Rates for the supply and fabrication of platework, packaged ready for road/sea freight were solicited from fabricators in North America and Asia. The scope included the preparation of workshop fabrication drawings, marking plans and bolt lists. Site installation hours were based on Lycopodium's experience and supported by contractors undertaking work in Michigan.

21.1.3.9 Field Erected Tanks

Field erected tankage quantities (generally tanks from 3.1 m to 8.2 m in diameter) and shop fabricated tanks (generally up to 3 m in diameter) were calculated using the sizing provided in the plant mechanical equipment list prepared for the study as the basis. Rates for the supply and fabrication of field erected tankage, packaged for sea freight, freight to site and site installation were solicited from contractors in North America and SE Asia. The scope included the preparation of workshop fabrication drawing.

21.1.3.10 Conveyors

Budget pricing was obtained from experience vendors for the supply of conveying systems based on quotations from vendors. The freight to site and installation costs was estimated by Lycopodium.

21.1.3.11 Equipment

Generally, major equipment packages are quoted by vendors within the last six months. Mechanical installation hours were assessed using Lycopodium's database of experience in North America.

21.1.3.12 Pipework

In plant piping was factored as a percentage of the mechanical cost in each plant area. MTO's were developed for overland HDPE piping. Pricing was based on historical rates.

21.1.3.13 Electrical / Instrumentation

Electrical and instrumentation were factored as a percentage of the mechanical installed cost in each plant area.

21.1.3.14 Erection and Installation

In addition to the discipline by discipline assessment of erection/installation costs, an estimate was made for heavy lift cranes and contractor site establishment.

21.1.3.15 Engineering, Procurement and Construction Management (EPCM)

The EPCM estimate was prepared on a deliverables-basis for the proposed for the Project scope of work and equipment lists. Duration based costs such as project and construction management were estimated using the Project implementation schedule as a guide.

21.1.3.16 Spares

The cost of capital spares was factored, considering 3% of the equipment supply costs.

21.1.3.17 Transport

Transport costs are included in the estimate and were factored based on the supply cost of material and equipment to account for transport and logistics to site. In general, the freight allowance used was 5% of equipment supply cost. In certain cases, mechanical equipment was quoted, vendors provided separate freight costing to site, and therefore these rates were utilized.

21.1.3.18 Installation Costs

Installation costs are further divided between direct labour, equipment and construction indirect costs.

The labour component reflects the cost of the direct workforce required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour, inclusive of overtime premiums, statutory overheads, and payroll burden.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes cranes, vehicles, small tools, consumables, and PPE.

Construction indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, equipment and labour mobilization and demobilization, Rest and Recreation ("R&R"), meals and accommodation costs.

The labour gang rates, and equipment costs estimated for each major trade commodity are shown in Table 21.9.

TABLE 21.9 STANDARD DIRECT LABOUR GANG RATES			
Item	**Direct Labour ($/h)**	**Equipment ($/h)**	**Total Hourly Rate ($/h)**
Earthworks	72	87	159
Concrete Installation	76	11	87
Steelwork	96	40	136
Platework	105	37	142
Field Erected Tanks	105	37	142
SMP (excl Mills)	97	28	125
Mill Erection	105	37	142
Piping	108	27	135
Electrical and Instrumentation	93	14	107
Building Installation	95	29	124

21.1.4 TMF and WRFs

Golder established the scope and quantities for the TMF and WRFs. Project rates were provided by Lycopodium which was provided by local contractors/vendors currently undertaking similar work in Michigan. Liner rates were based on quotations for fully supplied and installed liner based on quotations from three USA vendors.

21.1.5 Owner's Costs

The Owner's costs include:

- Owner's Project management team;
- Owner's team expenses;
- Project insurances;
- Cost of preproduction labour, training and operational readiness.
- First fills (grinding media, lubricants, fuel, and reagents).
- Opening stocks.
- Process plant mobile equipment.
- Consumable, insurance and commissioning spares.
- Vendor representative and training costs for the process plant.
- Maintenance tools and equipment.
- Office equipment and furniture.

21.1.6 Escalation

The financial evaluation has been conducted using real terms, so no escalation has been applied.

21.1.7 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- The base date for the bulk of pricing for the estimate is third quarter 2019 (Q3 2019).

- Prices of materials and equipment with an imported content have been converted to USD at the rates of exchange stated in this report section. All pricing received has been entered into the estimate utilizing native currencies wherever possible.

- The bulk earthworks commodity rates that include imported material are based on the assumption that suitable construction/fill materials will be available from borrow pits within 2 km of the work fronts.

- Engineering quantities and rates for the TMF, WRFs, access road, water diversions, COW, water management areas have been provided by Golder and Hatch. Sub-consultant design costs related to these work scopes for Project implementation have also been included.

- There is no allowance for unforeseen blasting in the bulk earthworks cost estimates, given the results of the site geotechnical investigation.

- The estimate allows for supply of structural steel from SE Asia and platework from the region.

- It has been assumed mobile equipment purchased and used by the owner's construction team will be handed over to the owner's operations team upon completion of construction. No allowance for additional mobile equipment has been made for operations.

21.1.8 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Permits and licences.
- Project sunk costs.
- Exchange rate variations.

21.1.9 Contingency

An amount of contingency has been provided in the initial capital cost estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed Project cost.

Contingency has been applied to the estimate on a line-by-line basis as a deterministic allowance by assessing the level of confidence in each of the defining inputs to the item cost, these being scope, supply costs and installation costs, and then applying an appropriate weighting to each of the three inputs.

Scope, supply costs of equipment and materials, and installation costs were separately categorized and attributed as a potential contingent sum as a percentage of the capital, i.e. range from 5% (if engineering was completed or contract award price) to 25% (if an allowance was applied). The resulting contingency per line item was a weighted average of these inputs.

The level of definition and pricing basis per discipline is summarized in Table 21.8

The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

Resulting contingencies per discipline is summarized in Table 21.10. The overall resultant contingency for the initial capital for the Project is 14%. No contingency has been applied to the sustaining capital costs.

TABLE 21.10
CONTINGENCY PER DISCIPLINE

Discipline	Contingency (%)
General	11
Earthworks	16
Concrete	12
Steelwork	10
Platework	11
Piping	23
Electrical	13
Instrumentation	17
Buildings	11
Mining	5
Owners Costs	17
Management Costs (EPCM)	15
Average	**14**

21.2 SUSTAINING CAPITAL COSTS

21.2.1 Open Pit Mine Sustaining Capital

Capital expenditures incurred after Year -1 are considered sustaining capital and are detailed in Table 21.11. The majority of the sustaining capital consists of capital lease payments for the mining equipment. Given the life of the open pit, no equipment replacements are planned.

TABLE 21.11 OPEN PIT MINE SUSTAINING CAPITAL COSTS ($K)								
Item	**Total ($k)**	**Year**						
		1	**2**	**3**	**4**	**5**	**6**	**7**
Equipment and Down Payments	**851**	851	0	0	0	0	0	0
Equipment Capital Leases	**39,824**	6,769	8,819	8,819	8,820	4,380	2,134	83
Mine Development	**3,228**	2,956	163	110	0	0	0	0
Freight and Spares	**2,034**	381	441	441	441	219	107	4
Total Mine Sustaining Capital	**45,937**	**10,956**	**9,423**	**9,370**	**9,261**	**4,599**	**2,241**	**87**

21.2.2 Underground Mine Sustaining Capital

Initial capital costs for the underground mine are treated as sustaining capital costs for the Back Forty Project since open pit mining will be well underway by the time the underground mine is developed.

Underground sustaining cost estimates are based on:

- First-principles estimates.
- Detailed design of the underground mine.
- Contractor budgetary quotes for specialist operations (Alimak).
- Vendor pricing and finance terms for equipment.
- Cost databases for capital equipment.

Costs have an accuracy of ±30% and do not include any contingency (a contingency is included elsewhere in the financial model). Underground equipment is assumed to have no salvage value at the end of mine life.

Underground mining and development equipment is assumed to be leased from the manufacturer. Leasing costs for underground equipment were generated from a term sheet provided by Sandvik Mining. The terms were as follows:

- Downpayment 15% of purchase price.
- Origination fee of 0.60% of purchase price.
- Initial US$1,200 contract legal fee.
- Monthly payment of 2.434% of purchase price per month for 36 months.

These terms apply to all mobile mining equipment manufactured by Sandvik, and it was indicated that equipment manufactured by Getman could also be acquired under the same terms. For other ancillary mobile equipment, these same terms have been assumed. All other machinery for the underground not listed in the fleet summary in Section 16 is assumed to be purchased without any financing.

The Back Forty underground mine is expected to be developed in Q1 Year 5, and is scheduled to be mined over six and a half years. The underground mine is considered to be a separate project from the surface mine, but shares many of the surface facilities (processing, offices, etc). As such, only the costs solely associated with the underground mine are included in this Technical Report section. No expansion of surface infrastructure is expected to be required specifically for the underground mine, outside of a small pushback and contouring of the open pit at the 185 m bench and above to create sufficient room for temporary stockpiles and traffic around the underground portal.

Sustaining capital costs also include all costs associated with infrastructure, capital waste development (vertical and lateral), relevant equipment leasing costs (down payments, legal fees, origination costs and mobilization costs), and the paste backfill plant.

Total underground mine sustaining capital costs are estimated at $98.9M, as shown in Table 21.12.

TABLE 21.12
UNDERGROUND MINE SUSTAINING CAPITAL COSTS ($K)

Item	Total ($k)	Year							
		4	5	6	7	8	9	10	11
Infrastructure	19,763	250	4,471	7,889	1,063	5,875	214	1	0
Equipment	32,198	100	9,011	9,990	9,241	3,265	542	0	50
Development	31,984	0	9,314	11,120	6,942	3,772	769	68	0
Paste Plant	15,000	15,000	0	0	0	0	0	0	0
Total	**98,946**	**15,350**	**22,796**	**28,999**	**17,246**	**12,912**	**1,524**	**69**	**50**

A breakdown of sustaining capital infrastructure costs is presented in Table 21.13. The cemented rockfill ("CRF") plug is included in infrastructure, as it is neither equipment nor development. Costs for the plug are incurred in Year 8.

TABLE 21.13
UNDERGROUND MINE SUSTAINING CAPITAL INFRASTRUCTURE COSTS

Item	Cost ($k)
Mining Infrastructure	3,455
Maintenance Infrastructure	3,052
Surface Infrastructure	2,758
Dewatering Infrastructure	1,278
Underground Infrastructure	4,220
CRF Plug at Open Pit Bottom	5,000
Total	**19,763**

A breakdown of underground mine sustaining capital costs for equipment for the Back Forty underground mine is presented in Table 21.14.

TABLE 21.14
UNDERGROUND MINE SUSTAINING CAPITAL EQUIPMENT COSTS

Equipment Type	Downpayments ($k)	Lease Capital Payments ($k)
Drill Jumbos	863	4,578
Scissor Decks and Bolters	1,099	5,793
Production Drills	430	2,280
Explosives Loaders	257	1,348
LHDs	790	4,189
Haul Trucks	718	3,809
Auxiliary	926	4,764
Equipment Mobilization and Demobilization	353	
Total	**32,198**	

Underground mine sustaining capital development costs for Back Forty are estimated at $32.0M, comprised of $28.2M for lateral and $3.8M for vertical development, as presented in Table 21.15 and Table 21.16, respectively.

TABLE 21.15
UNDERGROUND MINE SUSTAINING CAPITAL LATERAL DEVELOPMENT COSTS

Development Type	Quantity (m)	Direct Cost ($/m)	Total Cost ($k)
Main Ramp	7,016	2,509	17,604
Electrical Cut-Outs	230	2,509	577
Maintenance Shop	205	4,130	847
Sump	235	2,180	512
Magazine	42	2,509	105
Pump Station	104	4,130	430
Re-muck Bay	1,128	3,167	3,572
Truck Turn-Around	40	2,509	100
Refuge Station	92	2,509	231
Ventilation Access (Lateral)	774	2180	1,687
Day Works and Sundries	-	-	2,567
Totals	**9,866**		**28,233**

TABLE 21.16 UNDERGROUND MINE SUSTAINING CAPITAL VERTICAL DEVELOPMENT COSTS			
Development Type	**Quantity (m)**	**Direct Cost ($/m)**	**Total Cost ($k)**
Drop Raise	765	1,949	1,492
Alimak Raise	404	4,750	1,919
Day Works and Sundries	-	-	341
Totals	**1,169**		**3,752**

21.2.3 Non-Mining Sustaining Capital

Other Project sustaining capital costs include subsequent TMF stage raises over the life of mine and process plant annual capital expenditures. The sustaining capital schedule over the life of mine is estimated as shown in Table 21.17.

TABLE 21.17 OTHER PROJECT SUSTAINING CAPITAL COSTS ($K)						
Item	**Total ($k)**	**Year**				
		1	**2**	**3**	**4**	**5**
Cut-off Wall	**4,667**	4,667	0	0	0	0
TMF	**28,690**	17,039	4,580	5,257	-	1,813
SWRF	**9,233**	9,233	0	0	0	0
NWRF	**31,524**	10,616	20,907	0	0	0
Total Project Sustaining Costs	**69,320**	**38,728**	**23,623**	**5,155**	**0**	**1,813**

21.2.4 Project Closure Costs

Project closure costs, salvage value and rehabilitation costs were estimated at $75M as presented in Table 21.18.

TABLE 21.18 PROJECT CLOSURE COSTS	
Cost Item	**Total Cost ($M)**
Mine Closure Costs	55.0
Rehabilitation Costs	29.5
Salvage Revenue	(9.8)
Total	**75.7**

A key aspect of mine closure is the backfilling of the open pit with waste rock. In addition, capping of the tailings facility is required along with topsoil placement in preparation for re-vegetation. These earthworks will occur during the last few years of the operation and extend two year beyond the end of processing. The total cost is estimated at $55M, as shown in Table 21.19.

TABLE 21.19
MINE CLOSURE AND BACKFILLING ANNUAL COST

Item	Units	Total	Year					
			10	**11**	**12**	**13**	**14**	**15**
Loading	$(k)	**12,927**	276	1,816	1,560	2,707	3,284	3,284
Hauling	$(k)	**17,266**	388	2,499	1,957	3,388	4,517	4,517
Services/Roads/Dumps	$(k)	**19,867**	658	4,053	3,893	3,789	3,738	3,738
General, Supervision and Technology	$(k)	**2,362**	111	450	450	450	450	450
Allowance	$(k)	**2,621**	72	441	393	517	599	599
Total Pit Closure Cost	**$(k)**	**55,044**	**1,505**	**9,260**	**8,253**	**10,850**	**12,588**	**12,588**

21.3 OPERATING COSTS

LOM operating costs are summarized in Table 21.20.

TABLE 21.20 LOM OPERATING COSTS			
Item	**Total Cost ($ M)**	**Unit**	**Average Unit Cost**
Open pit mining		$/t mined	3.03
Underground mining		$/t mined	50.31
Open pit mining	178	$/t processed	11.2
Underground mining	288	$/t processed	18.2
Process plant	310	$/t processed	19.5
G&A	46	$/t processed	2.9
Total	**821**	**$/t processed**	**51.8**

21.3.1 Open Pit Mine Operating Cost

Mine operating costs are derived from a combination of first principle calculations with an in-house equipment database for all major and supporting equipment operating parameters, and include fuel, consumables, labour ratios, and general parts costs. The annual open pit mine operating cost is summarized in Table 21.21 and Table 21.22 and averages at $3.03/t mined over the production years 1 to 5 of the open pit. If year -1, pre-production, is included, the average unit mining cost is $3.00/t mined.

Annual mineralized material tonnes, waste tonnes and, loading and hauling hours are calculated based on the capacities of the loading and hauling fleet. These tonnes and hours provide the basis for drilling, blasting, and support fleet inputs. Based on the tonnes scheduled, a requirement for production drilling hours is calculated based on blast hole size and pattern, bench height, material density and drill penetration rate.

An estimate for blasting supplies, initiation systems and blasting accessories is provided on a per hole basis. Drilling and blasting inputs (pattern area, powder factor, etc.) have been included.

Fleet requirements for loading, hauling and support are derived from the loading and hauling operating hours. Operating hours for a support fleet of dozers, front-end loaders, graders, service and welding trucks, etc., are estimated to derive the support fleet requirements.

The diesel fuel price assumed is $ 0.71/L ($ 2.69/gallon).

All equipment costs are based on estimated fuel consumption rates, consumables costs, ground-engaging tools (“GET”) estimate, and general parts and preventative maintenance costs on a per-hour or per-metre interval basis.

A 5% allowance has been included in the operating costs to account for miscellaneous expenditures that don’t fit into the other major cost categories. The allowance is not a contingency, it is costs that are expected to be incurred as part of open pit mining operations.

Operating labour man-hours are categorized for the different labour categories such as operators, mechanics, electricians, etc. The mining cost also includes costs for all mine salaried staff, consumables, and software and fleet management systems’ licensing and maintenance.

TABLE 21.21 ANNUAL OPEN PIT MINE OPERATING COSTS															
Item	**Units**	**Year 1 to End**	**Year**												
			1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**
Direct Mining Costs (by Activity)															
Drilling	$(k)	27,060	6,481	7,545	6,405	5,968	661								
Blasting	$(k)	22,147	4,959	6,054	5,533	4,973	628								
Loading	$(k)	18,164	3,574	3,905	3,752	3,361	679	463	477	466	410	398	389	280	10
Hauling	$(k)	63,124	9,194	14,722	14,848	14,473	2,637	1,126	1,231	1,290	1,005	982	959	633	25
Services/Roads/Dumps	$(k)	28,075	4,399	4,354	4,353	4,358	776	1,581	1,396	1,364	1,361	1,366	1,370	1,376	20
General, Supervision and Technology	$(k)	11,313	2,552	2,552	2,552	2,552	629	68	68	68	68	68	68	68	
Allowance	$(k)	8,494	1,558	1,957	1,872	1,784	301	162	159	159	142	141	139	118	3
Total Open Pit Mining Cost	**$(k)**	**178,377**	**32,717**	**41,089**	**39,315**	**37,469**	**6,311**	**3,400**	**3,331**	**3,347**	**2,987**	**2,954**	**2,925**	**2,475**	**57**
Direct Mining Costs (by Cost Element)		**Year 1 to end**													
Operating Labour	$(k)	38,361	7,037	8,750	8,159	7,921	1,345	787	787	787	715	715	715	644	
Maintenance Labour	$(k)	14,880	2,357	2,871	2,785	2,700	317	603	603	603	532	532	532	446	
Supervision and Technical	$(k)	10,574	2,378	2,378	2,378	2,378	586	68	68	68	68	68	68	68	
Non-Energy Consumption and Parts	$(k)	74,849	14,289	18,347	17,370	16,245	2,519	897	960	986	866	848	831	670	20
Fuel	$(k)	28,242	4,670	6,356	6,322	6,012	1,078	584	625	614	534	521	510	400	14
Electric Power	$(k)	1,806	219	219	219	219	54	219	110	110	110	110	110	110	
Leases and Outside Services	$(k)	1,172	210	210	210	210	112	80	20	20	20	20	20	20	20
Allowance	$(k)	8,494	1,558	1,957	1,872	1,784	301	162	159	159	142	141	139	118	3
Total Open Pit Mining Cost	**$(k)**	**178,377**	**32,717**	**41,089**	**39,315**	**37,469**	**6,311**	**3,400**	**3,331**	**3,347**	**2,987**	**2,954**	**2,925**	**2,475**	**57**

TABLE 21.22
ANNUAL OPEN PIT MINE UNIT OPERATING COSTS

Item	Units	Year 1 to End	Year												
			1	2	3	4	5	6	7	8	9	10	11	12	13
Direct Mining Costs (by Activity)															
Drilling	$/t mined	0.46	0.48	0.50	0.43	0.44	0.35								
Blasting	$/t mined	0.38	0.37	0.40	0.37	0.37	0.33								
Loading	$/t mined	0.31	0.26	0.26	0.25	0.25	0.36								
Hauling	$/t mined	1.07	0.68	0.98	0.99	1.07	1.40								
Services/Roads/Dumps	$/t mined	0.48	0.33	0.29	0.29	0.32	0.41								
General, Supervision and Technology	$/t mined	0.19	0.19	0.17	0.17	0.19	0.33								
Allowance	$/t mined	0.14	0.12	0.13	0.12	0.13	0.16								
Total OP Mining Cost (mined material)	**$/t mined**	**3.03**	**2.42**	**2.74**	**2.62**	**2.78**	**3.36**			UG hauling and stockpile operations				Stockpile operations	
Total OP Mining Cost (incl. stockpiles)	$/t mined	2.34	2.33	2.56	2.55	2.71	1.80	1.56	1.44	1.48	1.54	1.59	1.63	2.10	0.73
Total OP Mining Cost	**$/t Feed**	**17.96**	**12.64**	**20.82**	**25.16**	**12.54**	**7.69**								
Direct Mining Costs (by Cost Element)		**Year 1 to End**													
Operating Labour	$/t mined	0.65	0.52	0.58	0.54	0.59	0.72								
Maintenance Labour	$/t mined	0.25	0.17	0.19	0.19	0.20	0.17								
Supervision and Technical	$/t mined	0.18	0.18	0.16	0.16	0.18	0.31								
Non-Energy Consumption and Parts	$/t mined	1.27	1.06	1.22	1.16	1.20	1.34								
Fuel	$/t mined	0.48	0.35	0.42	0.42	0.45	0.57								
Electric Power	$/t mined	0.03	0.02	0.01	0.01	0.02	0.03								
Leases and Outside Services	$/t mined	0.02	0.02	0.01	0.01	0.02	0.06								
Allowance	$/t mined	0.14	0.12	0.13	0.12	0.13	0.16								
Total Operating Mining Cost	**$/t mined**	**3.03**	**2.42**	**2.74**	**2.62**	**2.78**	**3.36**								
Total Operating Mining Cost	**$/t Feed**	**17.96**	**12.64**	**20.82**	**25.16**	**12.54**	**7.69**								

21.3.2 Underground Mine Operating Cost

Underground mine operating cost estimates are based on:

- First-principles calculations.
- Detailed design and scheduling of the underground mine.
- Contractor budgetary quotes for specialist operations (Alimak).
- Vendor pricing and finance terms for equipment.
- Vendor estimates of productivity and operational costs.
- Cost databases for consumables.
- Salary surveys and comparables.

Total operating costs for mining the underground portion of the Back Forty Deposit are estimated at $287.6M over the Life of Mine. These costs include development, mining, backfilling, hauling, power, salaries and lease payments. A breakdown of the underground operating costs can be seen in Table 21.23. Per-tonne costs use 5,717 kt recovered tonnes as the divisor.

TABLE 21.23
LOM UNDERGROUND MINING OPERATING COSTS

Item	Cost ($k)	Cost per LOM Recovered Tonne ($/t)
Operating Development	24,180	4.23
Production	158,686	27.76
Definition Drilling	5,240	0.92
Equipment Leasing	4,480	0.78
Other Indirect Costs	95,020	16.62
Totals	**287,606**	**50.31**

Operating development costs include all direct costs associated with the development of attack ramps from the level to the mineralized material, all costs associated with driving through waste between mineralized lenses due to discontinuous stope geometries, and all costs associated with developing footwall accesses and cross-cuts for LH stopes. Operating development costs are presented in Table 21.24 and total $24.2M over the LOM.

Table 21.24 LOM Underground Operating Development Costs by Type			
Development Type	**Quantity (m)**	**Direct Cost ($/m)**	**Total Cost ($k)**
Attack Ramp Waste (Slashed)	6,420	1,047*	6,719
Attack Ramp Waste (Driven)	2,232	2,180	4,867
Level Accesses	3,193	2,509	8,011
Sills and Cross-Cuts	1,094	2,180	2,384
Days Works and Sundries	-	-	2,198
Totals	**12,939**		**24,180**
***** *Cost per equivalent metre for slashed material.*			

Production costs include all direct costs associated with the drilling, blasting, extraction and transport of mineralized material, and backfilling of stopes. Production costs are presented in Table 21.25, segregated by mining method and type, and total $158.7M over the LOM.

Table 21.25 LOM Production Costs by Underground Mining Method			
Mining Method	**Mass (t, k)**	**Direct Cost ($/t)**	**Total Cost ($k)**
CF Type 1 (7.5 m W x 5.0 m H)	1,389	26.27	36,501
CF Type 2 (5.0 m W x 5.0 m H)	1,996	30.18	60,226
CF Type 3 (4.0 m W x 5.0 m H)	122	32.30	3,943
CF Type 4 (5.0 m W x 2.5 m H)	72	43.19	3,097
LH	2,138	25.69	54,919
Totals	**5,717**	**27.76**	**158,686**

Indirect costs include all costs not associated with production or development operations. These costs include salaried personnel (technical services, geology, management, UG construction, training, and fixed-plant maintenance, etc) directly associated with the underground mine, power costs, and other G&A costs such as software, consultants, and Personal Protective Equipment ("PPE"). Indirect costs are presented in Table 21.26 and total $104.7M over the LOM.

TABLE 21.26 LOM UNDERGROUND INDIRECT OPERATING COSTS	
Item	**Cost ($k)**
Staff Salaries	77,059
General and Administrative	1,950
Equipment Lease Interest Payments	2,398
Equipment Rebuild and Overhauls	2,083
Equipment Fleet Power Costs	7,604
Primary Ventilation Power Costs	1,573
Primary Pumping Power Costs	1,172
Primary Ventilation Propane Heating Costs	5,661
Definition Drilling	5,240
Total	**104,740**

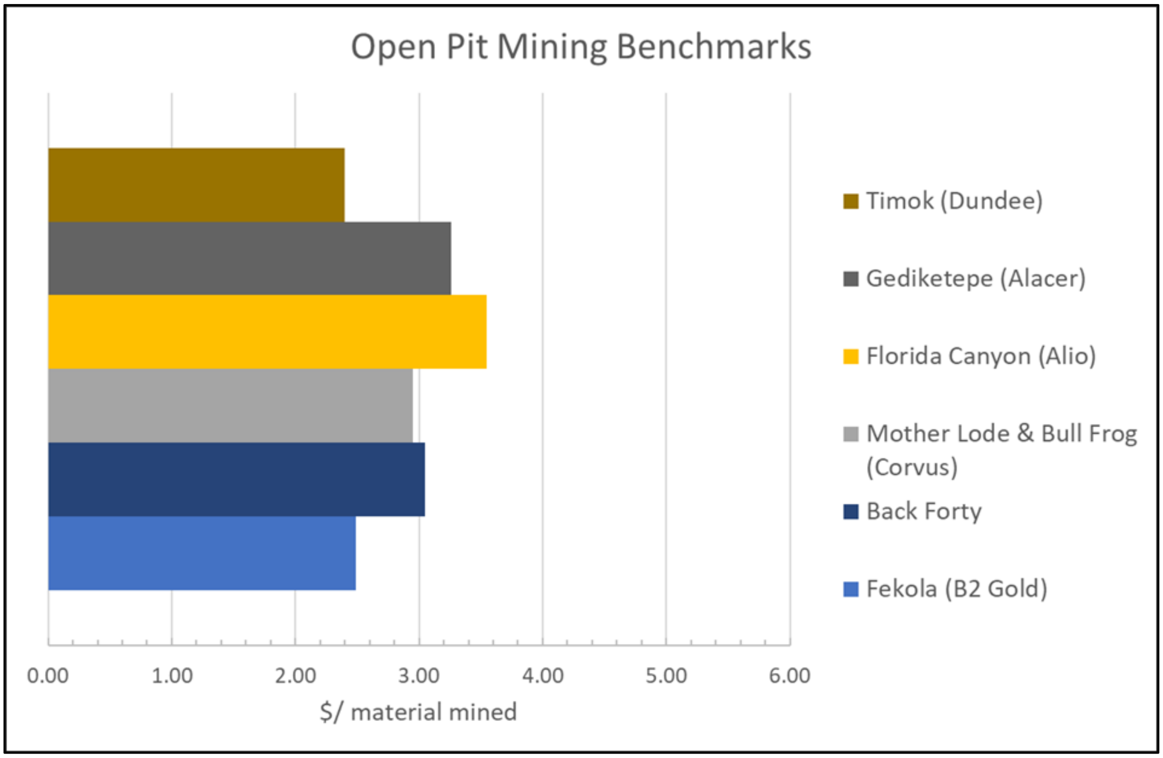
A summary of LOM costs to extract the underground portion of the Back Forty Deposit is presented in Table 21.27. All per tonne costs are calculated for the LOM production of the Back Forty underground mine (5,717 kt).

TABLE 21.27 UNDERGROUND LOM COST SUMMARY			
Type	**Item**	**Cost ($k)**	**LOM Unit Cost ($/t)**
CAPEX	Lateral Development	28,833	4.94
	Vertical Development	3,752	0.66
	Infrastructure	19,763	3.46
	Equipment Leasing and Mobilization	32,598	5.63
	Capital Total	**83,946**	**14.68**
OPEX	Lateral Development	24.180	4.23
	Production	158,686	27.76
	Definition Drilling	5,240	0.92
	Equipment Leasing	4.480	0.78
	Power	10,349	1.81
	Indirect Costs	84,671	14.81
	Operating Total	**287,606**	**50.31**
Total		**371,551**	**64.99**

21.3.3 Benchmark Mining Costs

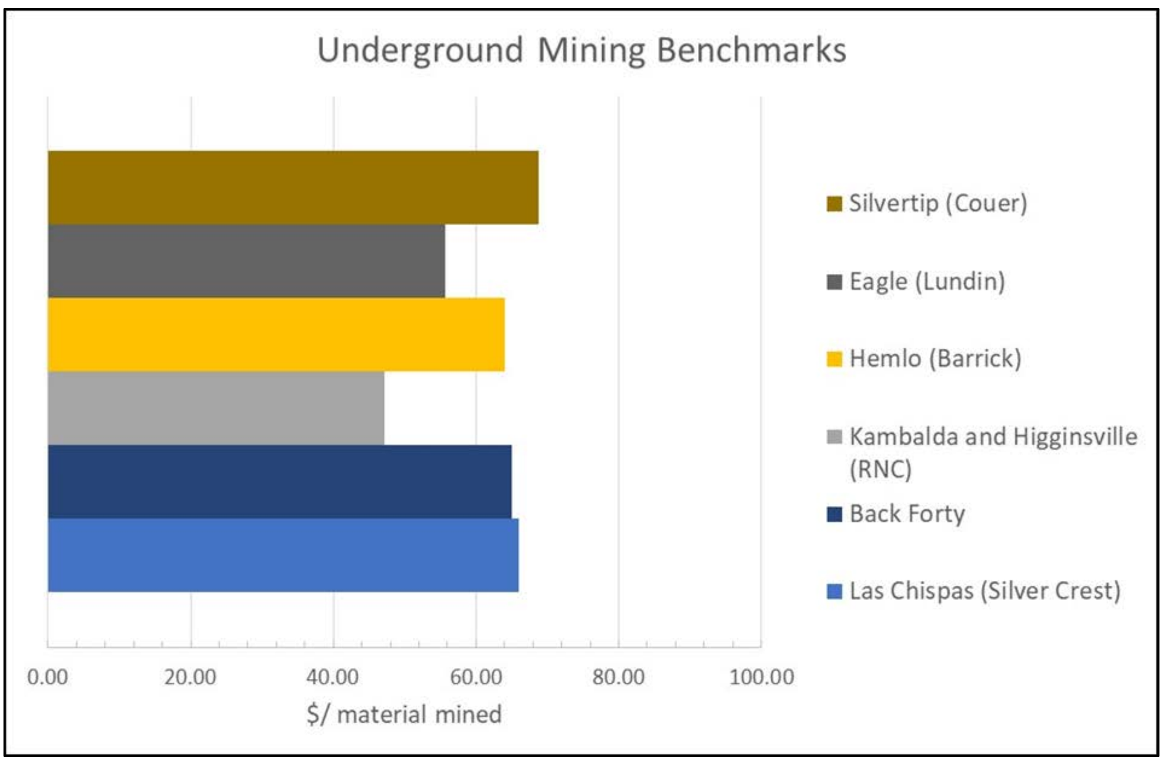
Mining operating costs similar to the Project were benchmarked using public information and are summarized in Figure 21.1 and Figure 21.2.

FIGURE 21.1 OPEN PIT MINING OPERATING COST BENCHMARKS



Source: *Aquila Resources Inc. (2020)*

FIGURE 21.2 UNDERGROUND MINING OPERATING COST BENCHMARKS



Source: *Aquila Resources Inc. (2020)*

21.3.4 Process Plant Operating Costs

The typical annual process plant operating costs per mineralized material processed are summarized in Table 21.28 and Table 21.29. All costs are expressed in United States Dollars unless otherwise stated and are based on Q3 2019 pricing and deemed to have an overall accuracy of ±25%. The process plant operating costs have been developed based on an annual oxide throughput of 127,750 tpy and an annual sulphide throughput of 1,022,000 tpy. The sulphide mineralized material consists of five major different types – Main, Pinwheel MS Cu/Gossan, Pinwheel Semi-Massive/Stringers, Pinwheel MS Cu/Zn, and Tuff.

TABLE 21.28
OXIDE PROCESS PLANT OPERATING COSTS

Cost Centre	All Material Types	
	Year ($k)	Material ($/t)
Operating Consumables	1,549	12.13
Plant Maintenance	607	4.76
Laboratory	79	0.62
Power	573	4.48
Labour	1,962	15.36
Total Process Plant	**4,771**	**37.35**

The process operating costs were developed by Lycopodium with input from Aquila and are based on typical industry standards applicable to polymetallic and precious metals processing plants.

Quantities and cost data were compiled from:

- Metallurgical testwork.
- Supplier quotations.
- Advice from Aquila.
- Lycopodium data.
- First principles.

TABLE 21.29
SULPHIDE PROCESS PLANT OPERATING COSTS

Cost Centre	Main		Pinwheel MS Cu/Gossan		Pinwheel SM/Stringers		Pinwheel MS Cu/Zn		Tuff	
	Year ($k)	**Material ($/t)**	**Year ($k)**	**Material ($/t)**	**Year ($k)**	**Material ($/t)**	**Year ($k)**	**Material ($/t)**	**Year ($k)**	**Material ($/t)**
Operating Consumables	7,971	7.80	5,574	5.45	5,462	5.34	8,382	8.20	8,229	8.05
Plant Maintenance	1,140	1.12	1,140	1.12	1,140	1.12	1,140	1.12	1,140	1.12
Laboratory	27	0.03	27	0.03	27	0.03	27	0.03	27	0.03
Power	3,153	3.09	3,263	3.19	3,263	3.19	3,263	3.19	3,581	3.50
Labour	2,520	2.47	2,520	2.47	2,520	2.47	2,250	2.47	2.520	2.47
Total Process Plant	**14,811**	**14.49**	**12,524**	**12.25**	**12,412**	**12.15**	**15,332**	**15.00**	**15,497**	**15.16**

21.3.4.1 Qualifications and Exclusions

The process plant operating cost estimate includes all direct costs associated with the Project to allow production of doré, bulk concentrate (copper or lead) and zinc concentrate. Each cost estimate is presented with the following exclusions:

- Process plant operating cost battery limits are the ROM pad ahead of the crushing circuits to the TMF. All costs associated with areas beyond the battery limits of the Project are excluded. Concentrate hauling and rehandling costs are excluded but accounted for in the economic analysis in Section 22.
- All taxes and import duties (included in the Section 22 economic analysis).
- Any impact of foreign exchange rate fluctuations.
- Any business interruption costs.
- Any escalation beyond the date of the estimate.
- First fill and opening stocks costs (included in the capital cost estimate).
- Tailings storage, rehabilitation or closure costs (included in the capital cost estimate).
- Land lease or other compensation costs.
- Product costs (transportation, refining, marketing and insurance), (included in the economic analysis).
- Licence fees or royalties (included in the economic analysis).
- Plant rehabilitation costs.
- Union fees.
- No contingency allowance.

21.3.4.2 Basis of Operating Cost Estimate

The process plant operating costs have been calculated based on labour, consumable, power, maintenance, mobile equipment and the onsite laboratory costs.

A description of each cost category is provided in the following sections.

Process Plant Labour

The personnel for the oxide and sulphide process plants will consist of management, technical support, shift supervision, laboratory staff, operators and maintenance staff. The total labour compliment required for the process plant is 67 employees. Management and technical support staff will work five 8-hour days per week with weekends off. Shift supervision, laboratory staff and shift operators and maintenance will work 12-hour day and night shifts on a 7-day work cycle, consisting of 4 days on and 3 days off. Table 21.30 summarizes the annual labour costs for each position.

TABLE 21.30 **PROCESS PLANT LABOUR COSTS**			
Position	**Classification**	**Number of Employees**	**Annual Labour Cost ($/year)**
Process Manager	M	1	159,376
Sulphide Process Plant			
Plant Metallurgist	SD	0.5	50,000
Shift Supervisor	SS	4	318,752
Crusher Operators	SS	4	233,750
Milling Operators /Control Room	SS	4	233,750
Flotation / Regrind Operators	SS	4	233,750
Concentrate Dewatering Operators	SS	4	233,750
Concentrate Loader Operator	SD	1	58,438
Reagent / Daycrew / Relief Operators	SS	1	58,438
Oxide Process Plant			
Plant Metallurgist	SD	0.5	50,000
Shift Supervisor	SS	4	318,752
Crusher Operators	SS	4	233,750
Milling / Leach / Filtration Operators	SS	4	233,750
SART/Detox/CIC/ADR Operators	SS	4	297,502
Goldroom Operators	SS	4	233,750
Reagent / Daycrew / Relief Operators	SS	1	58,438
Laboratory			
Analysts - Oxide	SS	2	116,875
Analysts - Sulphide	SS	2	116,875
Sample Prep	SS	2	116,875
Maintenance			
Maintenance Manager	M	1	100,938
Maintenance Planner	SD	1	58,438
Maintenance Supervisors	SD	1	79,688
Boilermakers / Fitters	SS	4	297,502
Trades Assistants	SS	4	233,750
Electricians	SS	4	297,502
Instrument Technician	SD	1	58,438
Total		**67**	**4,482,825**

***** *Classifications: M = Manager, SD = Day shift only, SS = Rotating night shift / day shift, Management and Day shift are weekday staff*

Labour rates are based on similar operations in the region and provided by Aquila. The labour rate includes 25% for burdens, overtime and bonuses where applicable.

Consumables

Consumables include all reagents, wear parts, and consumable materials in the process plant:

- Comminution wear consumables or steel consumables (crusher liners, mill liners and grinding media) were evaluated for each mineralized material type due to the variations in equipment operating conditions. Crusher liner, SAG mill liner, ball mill liner as well as steel ball consumption rates are based on in-house calculations and simulations using metallurgical testing results.

- Laboratory metallurgical testwork results were used wherever possible to determine the individual reagent consumption rates. In the absence of testwork data, reagent consumption rates are assumed based on first principle calculations, Lycopodium's in-house database and experience and generally accepted practice within the industry. A detailed description of the reagents required for the process plant is presented in Section 17 Recovery Methods of this Technical Report.

- Consumables and reagents prices are based on data obtained through relevant supplier quotes, and compared among competing vendors operating in the region.

- Gasoline and diesel fuel consumption rates for the mobile equipment are based on first principles calculations and Lycopodium experience. A gasoline and diesel price of $2.69/gal and $2.95/gal, respectively, used in the estimate were provided by Aquila.

- Waste water treatment plant and potable water consumables were provided by Hatch.

- Laboratory costs are allocated on a per sample basis, assuming the laboratory is owned and with minimal tests undertaken by a third party. Quotes for laboratory tests were sourced from recognized laboratories operating in North America and experienced in running on-site laboratories.

Power

The process plant power consumption was calculated based on the installed power excluding standby equipment. Electrical load factors and utilization factors of each electrical motor on the equipment list for the process plant and services are applied to the installed power to arrive at the annual average power draw which is then multiplied by total hours operated per annum and the electricity price to obtain the plant power cost. Power consumptions and costs per process plant area for each metallurgical type are summarized in Table 21.31.

An average unit power cost of $0.06435/kWh was used in the calculations and was provided by Aquila.

TABLE 21.31 SUMMARY OF PROCESS PLANT POWER COSTS		
Metallurgical Mineralized Material Type	**Annual Power Consumption (kWh)**	**Total Annual Power Cost ($k)**
Main	56,999,743	3,726
Pinwheel	58,703,651	3,836
Tuff	63,641,400	4,153

Maintenance

Annual maintenance material costs are estimated by applying factors to the ex-works mechanical equipment cost in each area of the process plant. The factors applied are based on Lycopodium's database and experience and are average costs over the life of the mine. As such, actual repair costs may be lower during the initial years but rise later. An overall weighted average factor of 4.6% is applied to the mechanical equipment supply cost ex-works for the process plant areas. The estimated annual maintenance cost for process plant and mobile equipment is $1.4M.

Mobile Equipment

The operating cost for light vehicles and mobile equipment includes gasoline and diesel fuel consumption, spares and tires, and maintenance parts, but excludes vehicle lease costs. The fuel costs are included in the consumables cost centre while the other operating costs are included in the overall maintenance materials cost centre.

Process Plant Laboratory

The operating cost associated with the process plant laboratory and assaying is estimated based on the anticipated number of samples required per shift, per day, per month and per year to operate both the oxide and the sulphide process plants. The forecasted requirement will be around 11,300 samples from the process plant per year. The cost for each sample averages $8.50/sample. The total estimated annual laboratory operating cost is $105k. Sample and assay costs associated with mine grade control samples and water management samples are captured separately in the mine and wastewater treatment plant operating cost estimates.

21.3.5 General and Administration Costs

General and administration ("G&A") labour is estimated at $1.2M and is summarized in Table 21.32.

TABLE 21.32 G&A LABOUR COMPENSATION		
Item	**Number of Employees**	**Total ($/year)**
General Manager	1	187,500
HR Manager	1	71,876
Senior Accountant	1	71,876
Payroll Clerk	1	43,750
Reception/Accounts Payable/ Office Manager	1	43,750
OHS Manager	1	106,251
Environmental Officer	1	87,501
Safety Officer	1	71,876
Purchasing Manager	1	106,251
Logistics Officers/Purchasing	1	71,876
Warehouse Supervisor	1	71,876
Security	4	287,503
Total	**15**	**1,221,883**

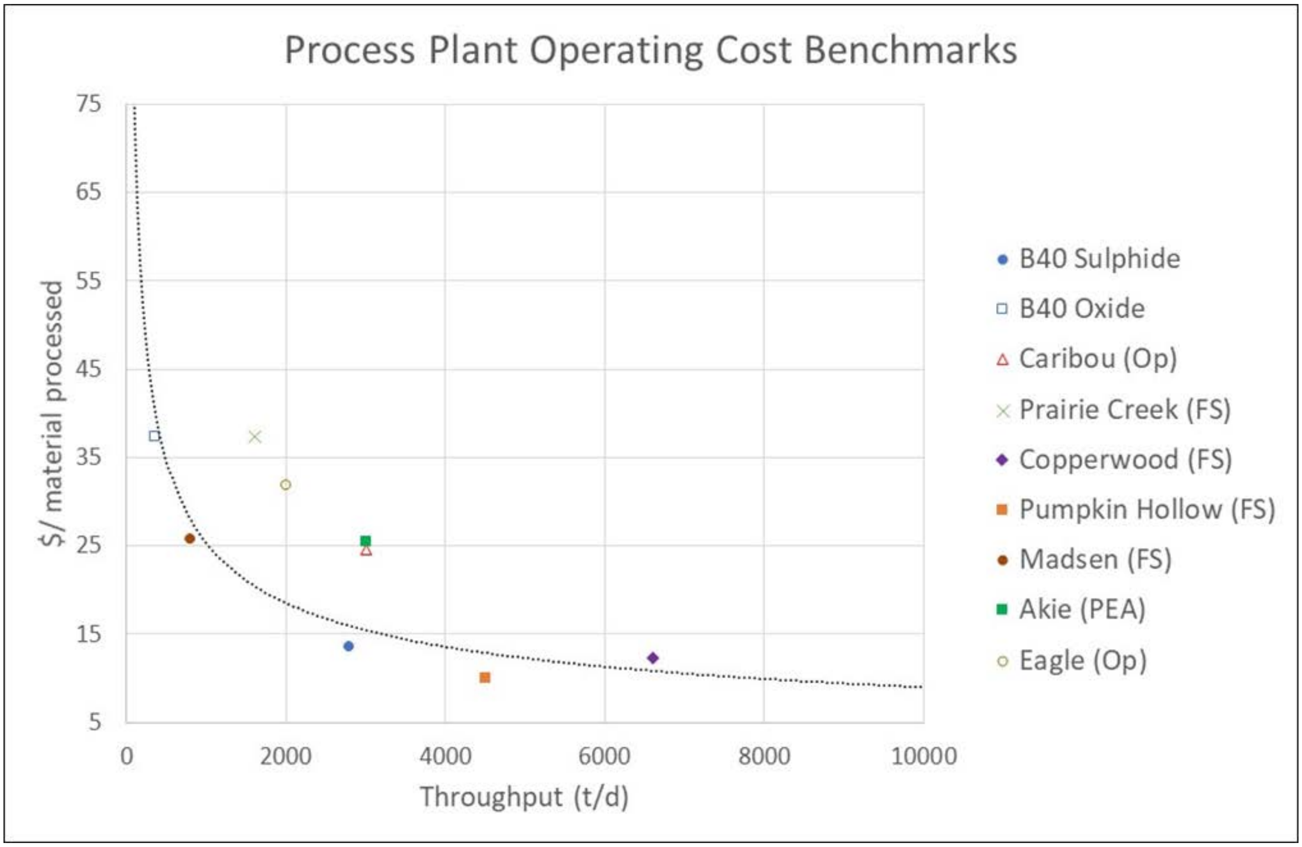
The labour rate includes 25% for burdens, overtime and bonuses where applicable.

The total estimated annual G&A expense is $2.4M.

21.3.6 Process Plant Benchmark Costs

Process plant operating costs similar to the Project were benchmarked using public information and are summarized in Figure 21.3.

FIGURE 21.3 PROCESS PLANT OPERATING COST BENCHMARKS



Source: *Aquila Resources Inc. (2020)*

21.3.7 Water Treatment Plant Operating Costs

Waste water treatment plant ("WWTP") operating costs were derived from a combination of first principle calculations with an in-house equipment database for all major and supporting equipment operating parameters, and include power, consumables, rentals, labour, and general maintenance and parts costs. Table 21.33 summarizes the reagent requirements and costs for the WWTP.

TABLE 21.33
REAGENT REQUIREMENTS AND COSTS FOR THE WWTP

Reagent	Specifications	Unit Cost ($/kg)	Annual Consumption (kg/year)	Annual Cost ($/year)
Dry Polymer	Anionic, dry polymer	6.36	2,240	14,426
Hydrated Lime	92% $Ca(OH)_2$	0.18	888,355	162,540
Metclear MR2405	-	10.74	22,400	240,490
Sulfuric Acid	93%	0.40	21,495	8,600
Carbon Dioxide	-	0.36	808,000	289,264
Subtotal				**715,150**

The chemical consumption presented in Table 21.33 does not take into account variable water quality within the CWB over the LOM. The chemical consumption is based on the design water quality and the nominal treated water flow (i.e. design flow operating for 8,000 hours/year).

Where possible, Project-specific budgetary quotes for reagents were obtained from local vendors based on the estimated annual consumption, delivery type (i.e. tote, bulk, pallet, etc.), and Project location. Project-specific quotes for hydrated lime, carbon dioxide, and the sulphide (Metclear MR2405, ("MR2405")) were obtained from Graymont Western Lime, Praxair, and Suez (previously GE), respectively. In-house reference costs were used for dry polymer and sulfuric acid.

Within the WWTP, the only equipment that is assumed to be rented is the CO_2 Vendor Package (including the CO_2 Tank and CO_2 Vaporizer). Rental costs for the CO_2 Vendor Package are presented in Table 21.34 and do not include the supply of carbon dioxide.

TABLE 21.34
CO_2 VENDOR PACKAGE RENTALS AND COSTS

Equipment	Specifications	Unit Cost ($/month)	Annual Cost ($/year)
CO_2 Vendor Package	34 t horizontal CO_2 Tank	1,200	14,400
Sub-Total			**14,400**

The main consumables used in the WWTP include cartridge filters (0.1 and 0.5 micron), Multimedia Filter ("MMF") media (sand, anthracite, and garnet) and MERSORB media.

The cartridge filters have a finite service life as determined by the solids load on the filters. Complete replacement of the 0.1 micron cartridge filters is anticipated to be required approximately every three (3) weeks; complete replacement of the 0.5 micron cartridge filters is anticipated to be required approximately every two (2) weeks. Frequency will depend upon the operation of both the clarifier and multi-media filters. Spent cartridge filters will be disposed of off-site as hazardous waste.

The MERSORB media has a finite service life as determined primarily by the mercury load on the media. Complete replacement of the 47,175 kg of MERSORB media is anticipated every three years. Complete replacement of the media will be a recurring capital cost and is included in the operating cost estimate. Within the operating cost estimate, one third of the MERSORB media cost is included in the yearly operating cost. During years in which MERSORB is not replaced, it is assumed that MERSORB media within the Mercury Removal Filters will not need to be topped up to compensate for losses due to attrition and thus no additional MERSORB media is accounted for. Spent MERSORB media will be slurried and disposed of within the TMF via truck. Costs for transporting the MERSORB media via truck to the TMF have not been included in this operating cost estimate.

The MMF's 1/2/3 contain sand, anthracite, and garnet that is expected to be replaced every five (5) years. Complete replacement of the media will be a recurring cost and is included in the operating cost estimate. Within the operating cost estimate, one fifth of the MMF media (sand, anthracite, and garnet) cost is included in the yearly operating cost.

Consumable requirements and costs are summarized in Table 21.35.

TABLE 21.35
REAGENT REQUIREMENTS AND COSTS

Consumable	Unit Cost ($)	Annual Consumption	Annual Cost ($/year)
0.1 Micron Cartridge Filters	183.20/unit	1912 units/year	350,278
0.5 Micron Cartridge Filters	424.00/unit	154 units/year	65,296
MMF Media, Anthracite	1,687/m^3	3 m^3/year	4,863
MMF Media, Sand	1,373.39/m^3	2 m^3/year	2,639
MMF Media, Garnet	3,531.47/m^3	1 m^3/year	3,393
Mersorb LW Media	12.79/kg	15725 kg/year	201,067
Sub-Total			**627,500**

Note that operator requirements for monitoring/inspection storm water management infrastructure as outlined in the Back Forty NPDES permit (reference document MI005945) have not been included in this labour estimate.

21.3.8 Hazardous Waste Disposal Requirements and Costs

The WWTP will generate the following waste and residual streams:

- Clarifier sludge.
- Spent MERSORB media.
- Spent cartridge filters.
- Spent MMF media.
- MMF Backwash.

Sludge from the clarifier underflow will be returned to the sulphide tailings thickener in the process plant and ultimately disposed of in the TMF. Thus, no hazardous waste disposal costs have been quantified for the clarifier underflow sludge.

Spent MMF and MERSORB will be disposed in the TMF. Thus, no hazardous waste disposal costs have been quantified for the medias.

Spent cartridge filters are assumed to be disposed off-site and to be considered hazardous waste. Quotes from local waste management organizations were not able to be obtained. As such, hazardous waste disposal costs have not been included in the WWTP estimate.

Backwash from MMF 1/2/3 will be returned to the CWB. Costs associated with dredging and storage of dredged CWB material have not been included in the operating cost estimate.

21.3.9 Tailings Management Facility and Waste Rock Facility Operating Costs

21.3.9.1 Structural Maintenance

The annual structural maintenance is estimated at $183,600. The following structures in the TMF and WRFs are assumed to require annual maintenance:

- Repair damaged non-contact water diversion ditches.
- Repair damaged perimeter ditches of TMF and WRFs.
- Cleanout the leachate collection and leak detection system.
- Snow ploughing.

21.3.9.2 External Monitoring Program

The annual cost of the monitoring and inspection is estimated at $75,000. The TMF and WRFs are assumed to require the following monitoring and inspection by independent consultants:

- Annual environmental monitoring.
- Annual geotechnical inspection of facilities.
- Annual aerial survey of TMF and WRFs.
- Annual bathymetric survey of TMF Decant Area.

22.0 ECONOMIC ANALYSIS

22.1 SUMMARY

The economic analysis is presented for two macro-economic cases that are summarized in the following section. The Base Case uses current (June 2020) consensus long term forecast metal prices, while the Spot Case uses prices at the time of writing (July 9, 2020). The Project scope is described in the previous sections of this Technical Report, key elements of which are summarized below and are presented in Table 22.1.

- An open pit mine that will provide mineralized material at a faster rate than required to satisfy the process plants. The costs resulting from associated rehandle are more than compensated by benefits including the accelerated production of higher value material.
- An underground mine that will begin producing feed following depletion of the open pit.
- A 350 tpd circuit for leaching oxide material.
- A nominal 2,800 tpd circuit for processing the three different sulphide material types separately in campaigns rather than as a blend.

TABLE 22.1
SUMMARY METRICS

Area	Item	Units	Base Case[1]	Spot Price[2]
Process Production	Total Process Plant Feed	Mt	15.9	15.9
	Grade	g/t AuEq[4]	4.2	3.7
	Total Recovery and Payability	% of contained AuEq	74.3	73.4
	Payable Gold	koz Au	692	692
	Payable Gold Equivalent	koz AuEq	1,543	1,323
	Annual Gold Equivalent	koz AuEq	128	110
	Life of Mine	Years	12	12
	Throughput	tpd	Nominal 2,800 sulphides + 350 oxides	
	Total Tailings	Mt	14.4	14.4
Metal Price Deck	Gold	$/oz	1,485	1,998
	Zinc	$/lb	1.08	1.04
	Copper	$/lb	3.05	2.92
	Silver	$/oz	18.20	25.00
	Lead	$/lb	0.91	0.83
Revenue and OPEX	Gross Revenue	US$/t process feed	132	149
	NSR	US$/t process feed	113	130

TABLE 22.1 SUMMARY METRICS				
Area	**Item**	**Units**	**Base Case[1]**	**Spot Price[2]**
	Total Site Opex	US$/t process feed	52	52
	Royalties	% of NSR	2.0	2.1
	EBITDA	US$/t process feed	59	75
	EBITDA margin	EBITDA / NSR	52	58
	C1 Cash Costs (co-product)[3]	US$/oz AuEq	733	854
	C1 Cash Costs (by-product)[3]	US$/oz Au	(82)	(29)
CAPEX	Initial Capital	US$ M	250	250
	Sustaining Capital	US$ M	214	214
	AISC (co-product)[3]	US$/oz AuEq	926	1,078
	AISC (by-product)[3]	US$/oz Au	397	462
Unlevered Returns[5]	Pre-Tax NPV 6% discount rate	US$ M	248	430
	Pre-Tax IRR	%	31.6	45.4
	Post-Tax NPV 6% discount rate	US$ M	176	316
	Post-Tax IRR	%	26.1	37.8
	Post-Tax Payback	Years	2.4	1.6

Notes:

1) *The Base Case macro-economic forecast assumes flat pricing that has been drawn from the consensus long term estimates of select banks as of August 4, 2020.*
2) *As at August 4, 2020.*
3) *C1 cash costs, which are intended to measure direct cash costs of producing paid metal, include all direct costs that would generate payable recoveries of metals for sale to customers, including mining of mineralized materials and waste, leaching, processing, refining and transportation costs, on-site administrative costs and royalties, net of by-product credits. C1 cash costs do not include depreciation, depletion, amortization, exploration expenditures, reclamation and remediation costs, sustaining capital, financing costs, income taxes, or corporate general and administrative costs not directly or indirectly related to the Project. C1 cash costs are divided by the number of ounces of gold estimated to be produced for the period to arrive at cash costs per gold ounce produced. AISC includes C1 cash costs, as defined above, plus exploration costs at the Project and sustaining capital expenditures (including additional tailings storage, permitting and customary improvements to the operations over the life of the Project). AISC is divided by the number of ounces of gold estimated to be produced for the period to arrive at AISC per gold ounce produced. EBITDA is earnings before interest, taxes, depreciation, and amortization.*
4) *Gold equivalent ounces were determined by calculating the total value of metals contained or produced and dividing that number by the gold price ($1,485/oz gold Base Case or $1,998/oz gold Spot Case). As the denominator is higher in the Spot Case, the gold equivalent is lower than at Base Case prices. Gold equivalent grade is calculated by dividing the number of gold equivalent ounces by the Mineral Resource size (tonnes).*
5) *Project economics reflect the Company's gold and silver streaming agreements with Osisko Gold Royalties (see Aquila press release dated June 18, 2020). The PEA financial model includes $30 million of initial payments under the gold stream to be received during the design and construction period. The 2018 Feasibility Study did not include the impact of the gold streaming agreement.*

22.2 ASSUMPTIONS

22.2.1 Macro-Economic Forecasts

The Base Case macro-economic forecast assumes a flat pricing that has been drawn from the consensus long term estimates of North American equity analysts as of August 4, 2020. The Spot Case is based on prices at the time of writing. Prices for the two cases are presented in Table 22.2 (in order of economic contribution).

TABLE 22.2 METAL PRICE CASES			
Metal	**Units**	**Base Case**	**Spot Price**
Gold	US$/oz	1,485	1,998
Zinc	US$/lb	1.08	1.04
Copper	US$/lb	3.05	2.92
Silver	US$/oz	18.20	25.00
Lead	US$/lb	0.91	0.83

22.2.2 Realization

Commercial terms for concentrate and doré have been based on guidance from the specialist metals traders, Ocean Partners, and are as follows:

22.2.2.1 Copper Concentrate

- The most cost-effective destination for treatment is in Eastern Canada, with a total cost of concentrate transport of approximately $52/t. Note that this cost, and transport costs discussed below, includes trucking of concentrate from the mine site to a rail head, rehandle then rail transportation to the final destination. The second lowest cost, in Western USA, would have an associated cost approximately $60/t higher.

- Concentrate grade would be adjusted to target the optimal economics per material type, ranging between 17 – 22% Cu and with an average of 18.5%. Copper payables would be calculated on a one-unit deduction to a maximum of 96.5% and would average 94.1%. Treatment charges would include a base rate of $80/t, with penalties ranging from $4 - $10/t by material type (for mercury content) and average approximately $7/t. Refining charges would be $0.08/lb payable Cu.

- The grades of by-product Au and Ag would average 57 g/t and 738 g/t, respectively. These high grades would be expected to make Back Forty copper concentrate desirable and allow maximum payables of 96.3% Au and 90% Ag to be achieved. Refining charges would be $6/oz Au and $0.50/oz Ag.

22.2.2.2 Zinc Concentrate

- The most cost-effective zinc concentrate destination for treatment is in Eastern Canada, with a total cost of transport of approximately $62/t. At present this facility does not pay for precious metals. Facilities that do pay for precious metals located in Western Canada or Europe have a transportation cost premium of approximately $70/t. Transport costs include hauling bulk zinc concentrate from the mine site, rehandling and transportation to the final destination.

- Concentrate grade would be adjusted to target the optimal economics per material type, ranging between 50 – 55% Cu and with an average of 53.9%. Zinc payables would be calculated on an eight unit deduction to a maximum of 85% and would average 84.8%. Treatment charges would include a base rate varying by material type from $200 - $220/t, with penalties ranging from $5 - $8/t by material type (for mercury, iron and silica content) and average $209/t. For the assumed long term zinc price of $1.09/t, the standard escalation clause would result in a further charge of $8/t. There are no refining charges for zinc.

- Facilities in Western Canada, Europe, Korea and Japan currently pay for by-product gold and silver in excess of 1 g/t and 3 oz/t, respectively. There are no refining charges for by-product precious metals. Over the life of mine, approximately 78% of zinc concentrate would contain potentially payable levels of by-product precious metals, with 90% of potentially payable by-products contained in 55% of total concentrate and 50% of potentially payable by-products contained in just 20% of total concentrate. It is possible that zinc concentrate would be shipped to multiple destinations to optimize shipping costs and precious metals realizations.

22.2.2.3 Lead Concentrate

- Lead concentrate would be shipped to Western Canada, with a total cost of transport of approximately $136/t. Transport costs include hauling bagged lead concentrate from the mine site, rehandling and transport to the smelter.

- Concentrate grade would average 35% Pb. Lead payables would be calculated on a three unit deduction to a maximum of 95% and would average 91.4%. Treatment charges would include a base rate of $160/t, with penalties for mercury content of $3/t. There are no refining charges for lead.

- Average by-product grades of 63 g/t Au and 1,183 g/t Ag would attract the maximum level of payability of 95%. Refining charges would be $20/oz Au and $1/oz Ag.

22.2.2.4 Doré

- Doe would command payables of 99.9% Au and 99% Ag.

- Freight costs would total $15k/t doré, while smelter charges would be $8k/t. Total charges would equate to $0.82/oz Au.

22.2.3 Gold and Silver Streaming

Aquila previously sold a stream that will comprise 85% of future silver production. The commercial arrangements associated with the stream included initial payments totalling $17.25M and a further $4/oz for silver delivered into the stream. The financial model does not include the initial payments as inflows as these have already been received.

Aquila also previously sold a stream that will comprise 18.5% of gold production to a cap of 105 koz into the stream (or approximately 568 koz total production). Thereafter, the stream reduces to 9.25% of total production. Over the life of mine, gold delivered into the stream is forecast to total 115 koz or 16.3% of total production. Gold stream payments included phased initial payments of $55M, of which $15M has been received to date and an additional $2.5M will be received prior to construction. The model reflects the final $30M deposit as an inflow during the construction period. The stream also makes provision for payment of 30% of the spot price, to a maximum of $600/oz, for gold delivered into the stream.

The streams are omitted from the calculation of tax obligations, with pre-tax revenues calculated based on the entirety of production sold at forecast spot prices.

22.2.4 Operational Performance

Key performance and operating assumptions include the following:

- The open pit will be mined at a rate faster than that required to satisfy the process plants, with surplus mineralization being stockpiled. Benefits of this approach include accelerated delivery of the highest value material to the process plants, flexibility for process operations and lower overall operating costs due to economies of scale. These benefits more than compensate for the cost of rehandle. The open pit mining rate selected was the one found to yield optimal economic returns.

- Development of the underground mine will be deferred until completion of open pit mining. While the mining rate from underground will be lower than nameplate throughput for the process plants, supplemental feed from stockpiles will allow the process plants to continue operating at capacity and thus achieve the lowest unit costs.

- Separate and adjacent process plants will be operated concurrently to recover precious metals from oxide mineralization and base and precious metals from sulphide mineralization.

- The oxide leach circuit will have a throughput of 350 tpd, with an initial throughput at 60% of nameplate rising to nameplate by month 11. This process plant will produce doré that will then be shipped to a refinery.

- The sulphide process plant circuit will have a nominal design throughput of 2,800 tpd, with an initial throughput at 60% of nameplate capacity rising to nameplate by month 10. This circuit will process three different types of mineralization, which will not be mixed, but campaigned separately. The additional cost of rehandle associated with this approach is more than offset by the benefits of processing a single feed type at any one time, and results in lower capital intensity, improved recovery and lower operating costs. In general, the process plants will be fed with the highest value material available, with higher value material on a stockpile prioritized over lower value ROM arisings.

- Sulphide material from the Main and Pinwheel Zones is softer (as measured by Bond Work Index) than that from the Tuff Zone. Material from the Main Zone also requires a less fine grind than the Tuff. As plant throughput will be constrained by the energy required to achieve target grind and the plant has been designed to achieve the nominal throughput of 2,800 tpd with material from the Tuff Zone, material from the Main and Pinwheel Zones can be processed at higher rates. The capacity for Main material will be 25% higher (or 3,500 tpd) while Pinwheel will be 23% higher (or 3,440 tpd).

- The scheduling increment used for the financial model was one month. For each month, the sulphide material type selected for processing was that with the highest value, provided the combined total of existing stockpiles and ROM arisings was equal to or greater than the plant throughput (i.e. the minimum duration for campaigns was one month).

- Closure costs, including those associated with backfilling the open pit void, are incurred following completion of open pit and underground mining operations.

22.2.5 Metallurgical Recovery

The mathematical equations used to model metallurgical recovery (see Section 13.8.2 of this Technical Report) were generated from regression analysis of testwork data. For all metals, this analysis showed a strong correlation between the grade of sample tested and recovery, with recovery increasing as a function of grade. Multiple functions were tested to model the relationship(s) between grade and recovery. The logarithmic and polynomial functions ultimately used were selected based on having the best fit of the data as measured by the coefficient of determination or r2. As these functions are not linear, the recovery estimate for the average grade that will be processed in a given month is not necessarily equal to the sum of recovery estimates for the individual blocks of material. In practical terms, this reflects that recovery from a stream of varying material grades will be higher than if the grade were kept constant.

Over the life of mine, the difference between process recovery estimated based on the monthly average grade and the block-by-block grades is as follows:

- Gold: monthly average understates recovery by 1.28%.
- Zinc: monthly average understates recovery by 1.92%.
- Copper: monthly average understates recovery by 0.56%.
- Silver: monthly average understates recovery by 2.32%.
- Lead: monthly average understates recovery by 3.52%.

On a value weighted basis, the aggregate understatement of process recovery equates to 1.72% for the Base Case forecast metal prices (1.76% for the Spot Price scenario).

Since the average grade treated by the process plants will vary over the course of a month, and operating set-points for the process plants will be adjusted based on the grade and material type being treated, and use of the monthly average grade to forecast recovery is considered overly conservative. Process recoveries were subsequently adjusted by the midpoint of variance between the two methods, or 0.89% overall on a value weighted basis (Table 22.3).

TABLE 22.3
METALLURGICAL RECOVERY

Metal	Contained Metal (koz / lbs)	Recovery (%)			Recovered Metal (koz / lbs)
		Base	Increment	Total	
Gold	1,000	73.8	0.64	74.4	744
Zinc	1,027,632	90.9	0.96	91.9	944,074
Copper	111,773	81.0	0.28	81.2	90,792
Silver	12,035	66.1	1.16	67.2	8,090
Lead	34,271	82.0	1.76	83.7	28,689

22.2.6 Royalties and Payments

The royalties that apply to material planned to be mined include:

- The Michigan State royalty, which applies to approximately 36% of the total mineralized material (38% on a value basis). The royalty is calculated on a sliding scale that ranges from 2.5% to 10.5% of NSR for the open pit and 2.0% to 10.0% for the underground.
- The County royalty, which applies to 1% of the total mineralized material (1% on a value basis). This royalty is calculated in the same manner as the State royalty.
- The Ganzer Royalty, which applies to 12% of the total mineralized material (10% on a value basis). This royalty is a flat 3.5% of NSR.

Royalties aggregate to approximately 2.0% of NSR and are assumed to be paid when material is mined.

22.2.7 Financial

NPV is reported using a discount rate of 6%, which reflects that in excess of 45% of gross revenue is generated from precious metals (which are typically afforded a discount rate of 5% for a low risk geography such as Michigan). NPV is expressed in real, Q3 2019 terms. The start date for discounting is the commencement of Project construction. It is expected that no material expenditures will be made prior to this date.

Results were calculated on a post-tax basis, and incorporate the current US fiscal regime as follows:

- The state Nonferrous Metallic Minerals Extraction Severance Tax has been applied in lieu of state income taxes. The severance tax rate of 2.75% is applied to taxable mineral value, which is the NSR.

- The federal income tax rate of 21%, which is applied to pre-tax income that includes the state income taxes as a deduction. Other deductions include an allowance for depletion, depreciation of capital investment and amortization of closure expenses.

22.3 RESULTS

There is an anticipated 24 month period for construction of the sulphide process plant circuit, oxide leach circuit and all associated infrastructure. Following construction, treatment of material by metallurgical type is as summarized in Table 22.4, with payable metal presented in Table 22.5.

Figure 22.1 illustrates LOM production, payable production and cash flow for the Project. The following is highlighted:

- The peak funding requirement is $222M, which comprises the initial capital cost less pre-production inflows from the gold stream from Osisko net of payments to HudBay. The mine plan targets highest grades from the outset, leading to cash positive operations from the outset and simple payback within 2.4 years.

- Payable output totals 1.5 Moz AuEq over the 12-year life, averaging 128 koz AuEq per annum.

- Cumulative free cash flow reaches a peak of $350M before closure expenses reduce it to $298M. Approximately 63% of gross closure expenses are due to backfilling of the open pit.

- Detailed production and financial metrics are provided in Table 22.6.

TABLE 22.4
PROCESS PLANT FEED

Main Zone	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Material Processed	ktonnes	8,175	841	639	1,065	958	1,278	319	532	532	745	745	426	95	0
Grade Cu	% Cu	0.30	0.42	0.29	0.22	0.28	0.25	0.26	0.24	0.25	0.38	0.36	0.38	0.39	0.00
Grade Au	g/t Au	1.96	2.98	2.66	1.85	2.26	1.65	1.56	1.75	2.05	1.66	1.52	1.35	1.32	0.00
Grade Ag	g/t Ag	17.1	16.4	14.0	11.2	16.4	14.1	17.3	25.8	26.7	20.9	16.8	19.3	19.7	0.0
Grade Zn	% Zn	4.09	5.60	4.14	2.83	6.96	4.78	3.25	4.38	4.44	3.04	1.74	1.90	1.90	0.00
Pinwheel Zone [1]	**Units**	**Total**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**
Material Processed	ktonnes	3,156	194	629	105	0	0	524	419	419	210	210	210	221	18
Grade Cu	% Cu	0.32	2.84	1.38	0.53	0.00	0.00	0.32	0.30	0.48	0.47	0.77	0.89	0.90	0.89
Grade Au	g/t Au	0.64	3.63	1.79	1.85	0.00	0.00	1.51	1.38	1.55	1.03	1.29	1.36	1.84	1.36
Grade Ag	g/t Ag	11.5	88.0	48.8	18.6	0.0	0.0	21.0	20.6	17.4	16.8	16.7	16.6	30.0	16.6
Grade Zn	% Zn	0.90	0.00	0.67	0.00	0.00	0.00	4.68	5.23	2.50	3.97	1.31	0.42	0.10	0.41
Tuff Zone	**Units**	**Total**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**
Material Processed	ktonnes	3,050	0	0	85	256	0	341	256	256	256	256	511	767	69
Grade Pb	% Pb	0.19	0.00	0.00	0.21	0.88	0.00	0.47	0.48	0.51	0.48	0.48	0.48	0.48	0.48
Grade Au	g/t Au	0.36	0.00	0.00	0.84	0.89	0.00	0.90	0.96	1.15	0.98	0.98	0.98	0.98	0.98
Grade Ag	g/t Ag	8.3	0.0	0.0	8.4	29.7	0.0	19.1	20.0	28.4	21.8	21.8	21.8	21.8	21.8
Grade Zn	% Zn	0.71	0.00	0.00	1.22	2.89	0.00	1.73	1.84	2.04	1.84	1.84	1.84	1.84	1.84
Oxide	**Units**	**Total**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**
Material Processed	ktonnes	1,477	104	128	128	128	128	128	128	128	128	128	128	95	0
Grade Au	g/t Au	0.84	13.30	9.76	3.69	4.78	3.77	4.09	5.79	2.28	2.50	2.33	2.22	2.22	0.00
Grade Ag	g/t Ag	8.9	32.0	63.7	55.9	96.4	44.6	48.7	59.1	37.5	37.3	36.5	36.7	36.7	0.0

Notes:
1. Combination of Material Types 2, 3, 4, 7 & 8. Only Types 4 & 8 contain economic levels of Zn

TABLE 22.5
PAYABLE METAL

Copper Concentrate	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Concentrate	ktonnes	223	43	48	12	12	14	10	10	13	17	19	15	10	1
Payable Cu	000 lbs	85,762	15,458	17,595	4,586	4,512	5,298	4,068	4,036	5,454	6,656	7,661	6,152	4,017	270
Payable Au	000 oz	391	59	51	39	40	38	25	28	33	26	26	16	10	1
Payable Ag [1]	000 oz	4,769	509	740	331	473	358	367	478	400	359	311	253	186	4

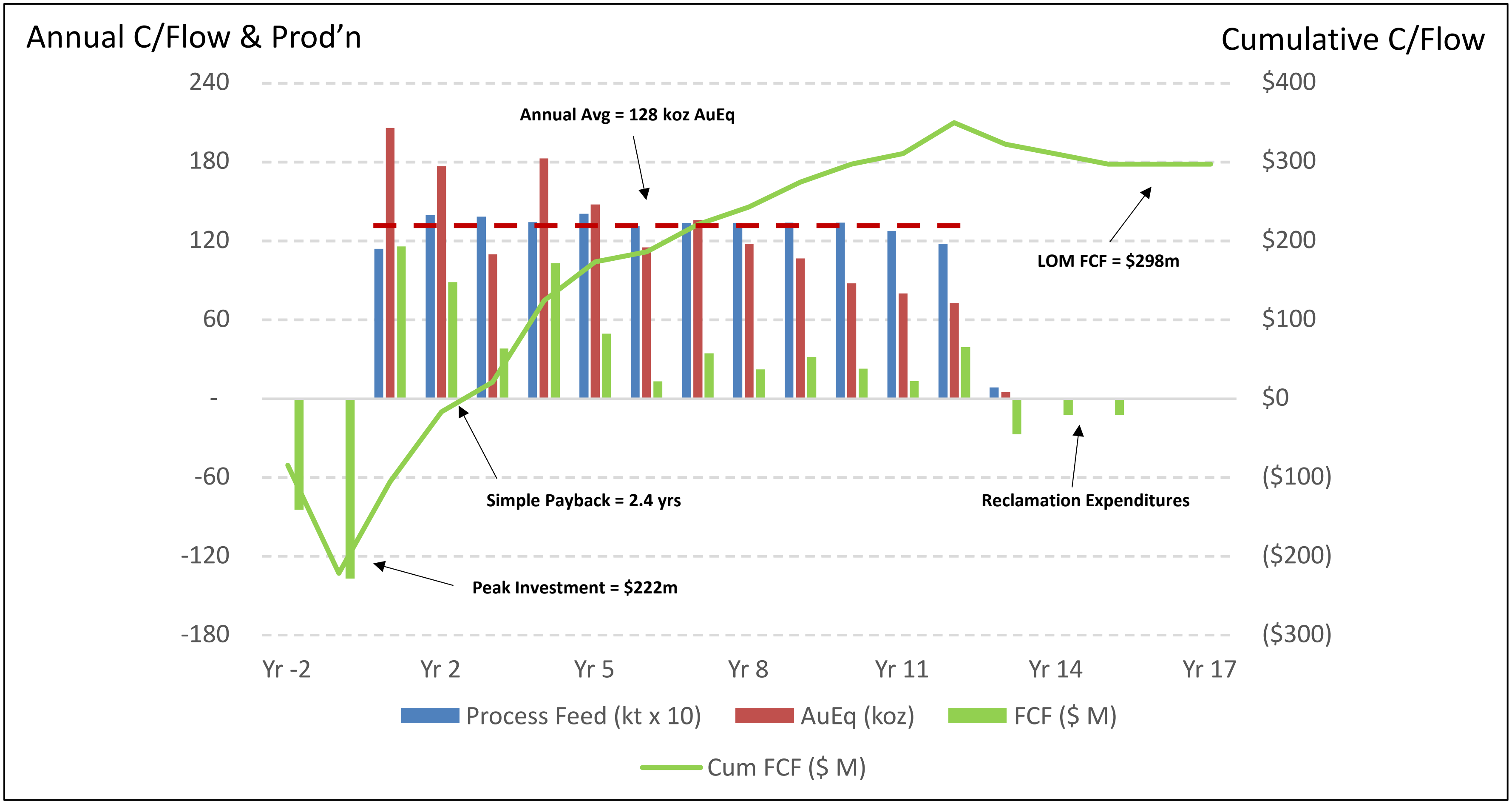
Lead Concentrate	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Concentrate	ktonnes	37	0	0	0	6	0	4	3	3	3	3	6	9	1
Payable Pb	000 lbs	26,230	0	0	287	3,944	0	2,709	2,041	2,187	2,072	2,072	4,144	6,216	558
Payable Au	000 oz	70	0	0	2	5	0	7	6	7	6	6	12	18	2
Payable Ag	000 oz	1,287	0	0	13	143	0	123	97	140	106	106	212	318	29

Zinc Concentrate	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Concentrate	ktonnes	794	81	52	50	131	104	72	87	67	59	32	29	27	2
Payable Pb	000 lbs	800,914	83,835	53,158	51,701	134,006	107,663	68,855	85,232	66,513	59,209	31,804	29,558	27,142	2,237
Payable Au	000 oz	25	4	3	3	2	2	1	1	2	2	2	1	1	0
Payable Ag	000 oz	170	0	0	0	13	0	13	10	20	14	24	34	39	3

Dore	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Dore	tonnes	7	1.4	1.3	0.5	0.7	0.5	0.6	0.8	0.3	0.3	0.3	0.3	0.2	0.0
Payable Au	000 oz	206	43	39	14	18	14	15	22	8	9	8	8	6	0
Payable Ag [1]	000 oz	33	1	4	3	6	3	3	3	2	2	2	2	2	0

Notes:
1. 90% of recoverable Ag in oxide ore recovered by SART process to Cu conc

FIGURE 22.1 PRODUCTION AND CASH FLOW



Source: *Gibsonian (2020)*

TABLE 22.6
DETAILED PROJECT METRICS

Item	Units	Total	Pre-Prod'n	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14
Mill Feed Processed	ktonnes	15,858		1,139	1,395	1,382	1,341	1,405	1,312	1,335	1,335	1,338	1,338	1,274	1,178	86	0
Payable Cu	Mlb	86		15	18	5	5	5	4	4	5	7	8	6	4	0	0
Payable Pb	Mlb	26		0	0	0	4	0	3	2	2	2	2	4	6	1	0
Payable Zn	Mlb	801		84	53	52	134	108	69	85	67	59	32	30	27	2	0
Payable Au	koz	692		107	93	58	66	54	49	57	50	43	42	37	34	2	0
Payable Ag	koz	6,260		511	744	348	635	361	505	589	562	482	443	501	545	36	0
Payable AuEq [1]	koz	1,543		206	177	110	183	148	115	136	118	107	88	80	73	5	0
Gross Revenue																	
Copper Concentrate	$ M	$801		$127	$125	$67	$68	$67	$47	$51	$62	$56	$61	$43	$27	$2	$0
Lead Concentrate	$ M	$127		$0	$0	$3	$11	$0	$13	$10	$12	$10	$11	$22	$32	$3	$0
Zinc Concentrate	$ M	$899		$96	$61	$60	$148	$119	$76	$94	$74	$67	$37	$34	$30	$2	$0
Dore	$ M	$268		$56	$50	$18	$24	$18	$20	$29	$11	$12	$12	$11	$8	$0	$0
Total	$ M	$2,095		$279	$236	$147	$251	$205	$155	$184	$159	$145	$121	$109	$98	$7	$0
Treatment and Refining																	
Copper Concentrate	$ M	$44		$8	$9	$3	$3	$3	$2	$2	$3	$3	$4	$3	$2	$0	$0
Lead Concentrate	$ M	$14		$0	$0	$0	$2	$0	$1	$1	$1	$1	$1	$2	$3	$0	$0
Zinc Concentrate	$ M	$252		$26	$16	$16	$41	$33	$23	$28	$21	$19	$10	$10	$9	$1	$0
Dore	$ M	$0		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total	$ M	$310		$34	$26	$19	$46	$36	$26	$31	$25	$23	$15	$15	$14	$1	$0
Net Smelter Return	**$ M**	$1,785		$245	$210	$129	$205	$169	$129	$153	$133	$122	$106	$94	$84	$6	$0
Operating Costs																	
Open Pit Mining	$ M	$178		$33	$41	$39	$37	$6	$3	$3	$3	$3	$3	$3	$2	$0	$0
Underground Minng	$ M	$288		$0	$0	$0	$0	$19	$41	$54	$54	$46	$41	$33	$0	$0	$0
Processing	$ M	$310		$21	$23	$24	$24	$27	$28	$28	$28	$28	$28	$28	$21	$1	$0
G&A	$ M	$46		$4	$4	$4	$4	$4	$4	$4	$4	$4	$4	$4	$4	$0	$0
Total Operating Costs	$ M	$821		$57	$68	$67	$65	$56	$76	$90	$90	$81	$76	$67	$27	$2	$0
Gross C1 Cash Costs [2]	$ / oz	$733		$443	$529	$781	$607	$618	$887	$888	$973	$982	$1,037	$1,020	$576	$584	$0
Capital Costs																	
Initial	$ M	$250	$250	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining	$ M	$222		$53	$35	$15	$10	$45	$32	$18	$13	$2	$0	$0	$0	$0	$0
Closure [3]	$ M	$75		$0	$0	$0	$0	$0	$0	$0	$0	$0	$2	$9	$8	$31	$25
Royalties & Taxes																	
Royalties & Payments [4]	$ M	$11	($29)	$12	$8	$1	$5	$4	$1	$2	$2	$2	$2	$1	$0	$0	$0
Cash State Severance Tax	$ M	$49		$7	$6	$4	$6	$5	$4	$4	$4	$3	$3	$3	$2	$0	$0
Cash Federal Income Taxes	$ M	$59		$0	$5	$4	$16	$11	$4	$4	$2	$2	$1	$1	$6	$0	$0
Cash Flow																	
Pre-Tax	$ M	$405	($222)	$123	$99	$46	$125	$65	$21	$43	$28	$37	$27	$17	$48	($27)	($25)
Post-Tax	$ M	$298	($222)	$116	$89	$38	$103	$49	$13	$34	$22	$32	$23	$13	$39	($27)	($25)

Notes:
1. AuEq caculated based on consensus prices
2. Divisor for Gross C1 is AuEq
3. Closure Payments for years 15 - 30 included in yr14 total
4. Includes initial payments from Au stream (inflow of $30m) and payments to Hudbay (outflow of $7.5m)

22.4 COMPOSITION OF RETURNS

22.4.1 Returns by Source and Classification of Mineral Resources

Figures 22.2 and 22.3 illustrate the composition of cashflow by source and classification of Mineral Resource. There are no Inferred Mineral Resources contained within the open pit mine plan. The Measured and Indicated ("MI") open pit Mineral Resources generate approximately 80% of total undiscounted cash flow, rising to 87% of NPV.

Points to note regarding the underground Mineral Resource include:

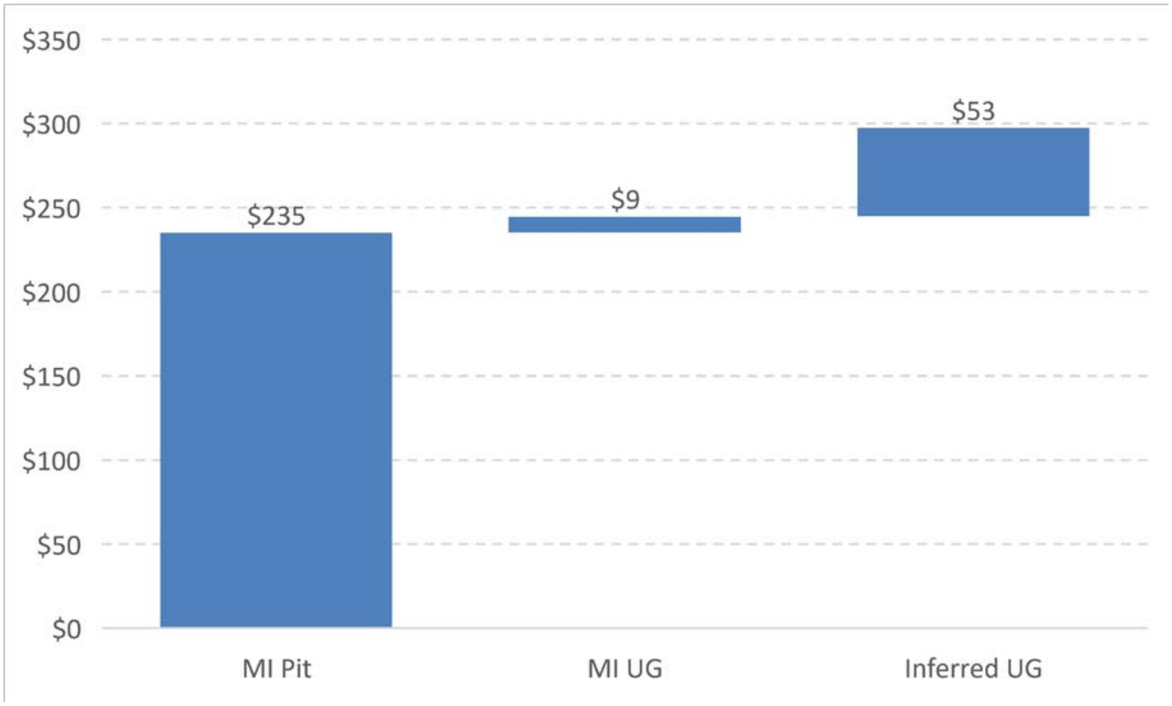
- The underground mine is accretive on an undiscounted basis if limited to only current MI Mineral Resources. However, the MI-only underground mine is marginally dilutive to NPV.

- Underground MI represents approximately 86% of the tonnage of underground Measured, Indicated and Inferred ("MII") Mineral Resources.

The underground deposit is open. The contribution of current underground Inferred Mineral Resources to overall Project value suggests that Mineral Resource additions at depth could result in non-linear increases to overall NPV.

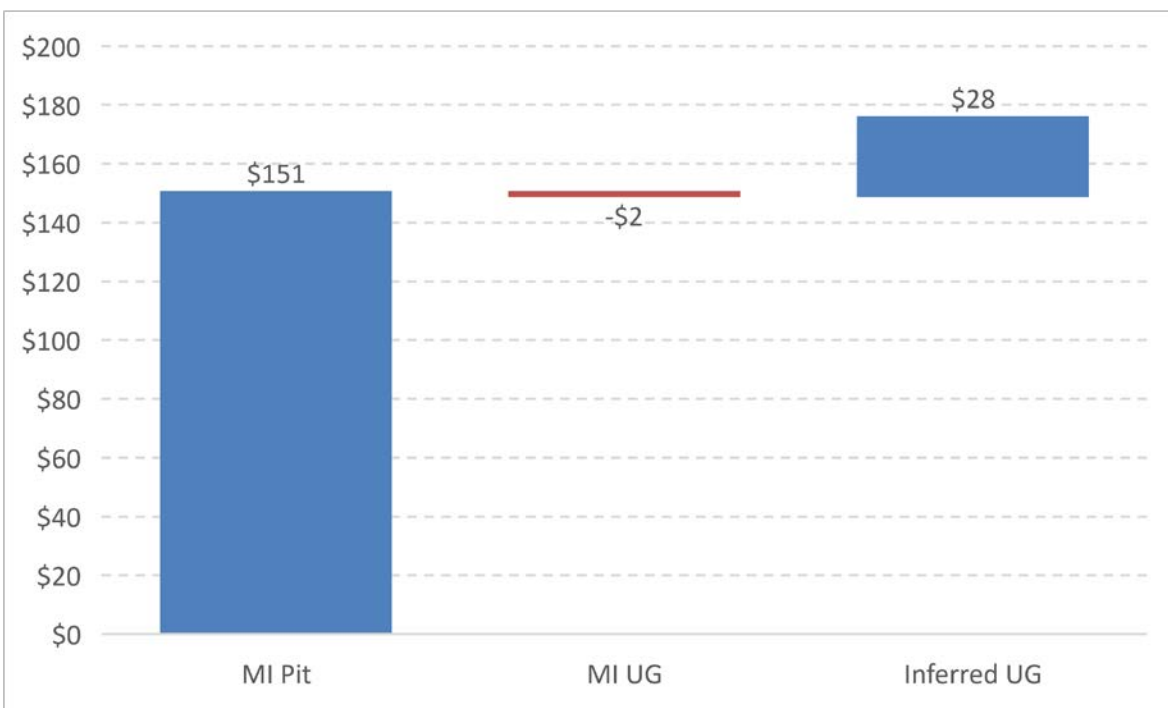
This last point underscores the critical mass required to achieve acceptable returns with underground mining. In Section 16, it is noted that the open pit was limited to a RF of 52%. Beyond this RF, incremental Mineral Resources of approximately one million tonnes could be extracted with an expanded open pit. Section 16 notes that the yield curve for this incremental material flattened. Another reason for not including this material in the open pit plan was to ensure sufficient Mineral Resources remained so that critical mass could be achieved with the underground operation. Additional underground Mineral Resources at depth could therefore also allow the open pit to be expanded beyond RF 52%.

FIGURE 22.2 COMPOSITION UNDISCOUNTED CASH FLOWS



Source: *Gibsonian (2020)*

FIGURE 22.3 COMPOSITION OF NPV

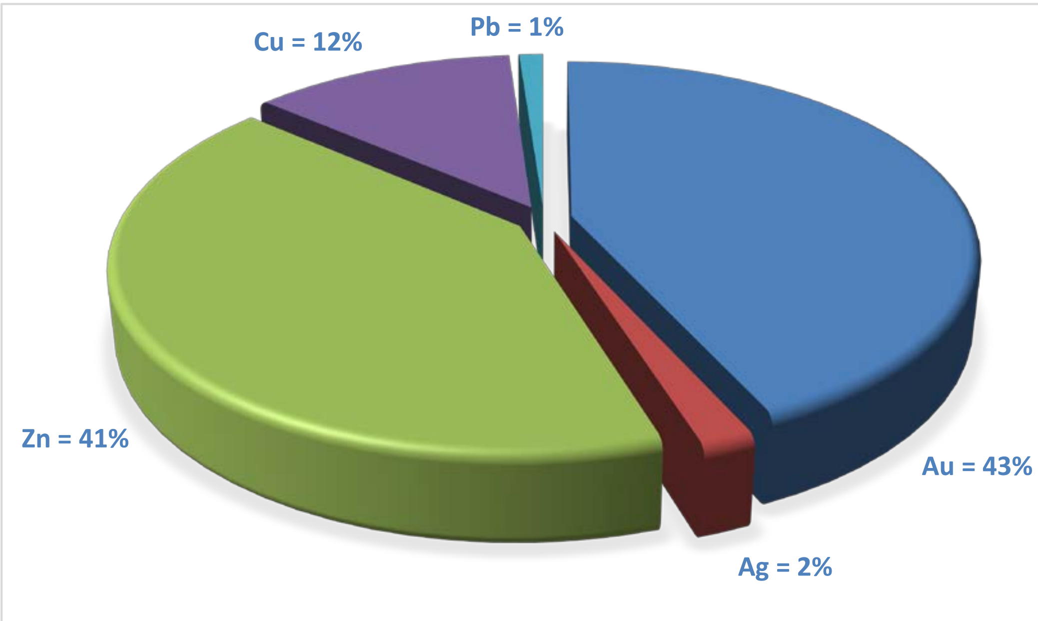


Source: *Gibsonian (2020)*

22.4.2 Revenue by Metal

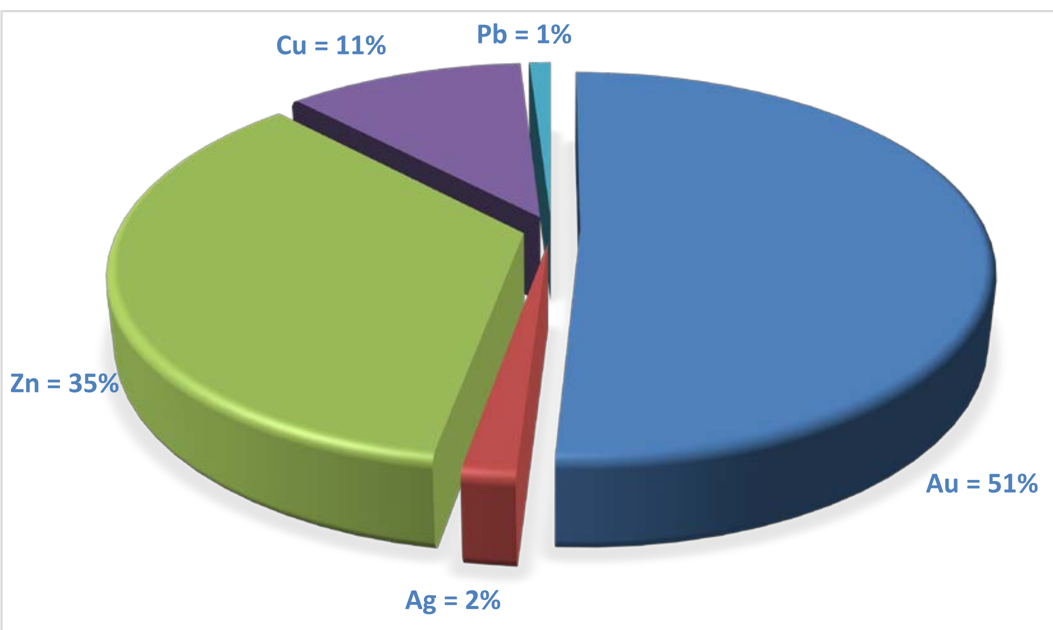
Figures 22.4 and 22.5 illustrate the composition of revenue (including the impact of streaming) of the various saleable metals that will be produced. Under the Base Case long term price forecast, precious metals comprise 45% of revenue, rising to 53% under the Spot Case.

FIGURE 22.4 REVENUE BY METAL – BASE CASE



Source: *Gibsonian (2020)*

FIGURE 22.5 REVENUE BY METAL – SPOT CASE



Source: *Gibsonian (2020)*

22.5 SENSITIVITY ANALYSIS

22.5.1 Macro-Economic

Tables 22.7 to 22.9 summarize the sensitivity of returns and payback to a ± 15% variation in the metal price assumptions. It can be seen that returns are most sensitive to gold and zinc prices, with a ±15% movement in prices having a 38 - 41% impact to the NPV. The impact of similar variation in copper prices is less than one third as much at 12%. Returns are relatively insensitive to variation in silver or lead prices.

Project economics remain viable even with the entire suite of metals at 85% of the assumed long term price, generating an 8.0% IRR. Under the more optimistic pricing scenario, simple pay back could be achieved within 20 months.

TABLE 22.7
SENSITIVITY OF NPV TO METAL PRICE ASSUMPTIONS

		Base Case		
Post-Tax NPV 6% ($ M)		**-15%**	**0%**	**+15%**
Gold	($1262 - $1708)	$103	$176	$248
Zinc	($0.92 - $1.24)	$108	$176	$244
Copper	($2.59 - $3.51)	$155	$176	$198
Silver	($15.47 - $20.93)	$176	$176	$176
Lead	($0.77 - $1.05)	$175	$176	$178
All Metals		**$12**	**$176**	**$337**

TABLE 22.8
SENSITIVITY OF IRR TO METAL PRICE ASSUMPTIONS

		Base Case		
Post-Tax IRR		**-15%**	**0%**	**+15%**
Gold	($1262 - $1708)	19.1%	26.1%	32.2%
Zinc	($0.92 - $1.24)	20.0%	26.1%	31.3%
Copper	($2.59 - $3.51)	24.1%	26.1%	28.2%
Silver	($15.47 - $20.93)	26.1%	26.1%	26.1%
Lead	($0.77 - $1.05)	26.1%	26.1%	26.2%
All Metals		**8.0%**	**26.1%**	**38.7%**

TABLE 22.9
SENSITIVITY OF SIMPLE PAYBACK TO METAL PRICE ASSUMPTIONS

Post-Tax Simple Payback (years)		-15%	Base Case 0%	+15%
Gold	($1262 - $1708)	3.2	2.4	1.9
Zinc	($0.92 - $1.24)	3.0	2.4	2.0
Copper	($2.59 - $3.51)	2.8	2.4	2.1
Silver	($15.47 - $20.93)	2.4	2.4	2.4
Lead [1]	($0.77 - $1.05)	2.4	2.4	2.5
All Metals		**3.8**	**2.4**	**1.6**

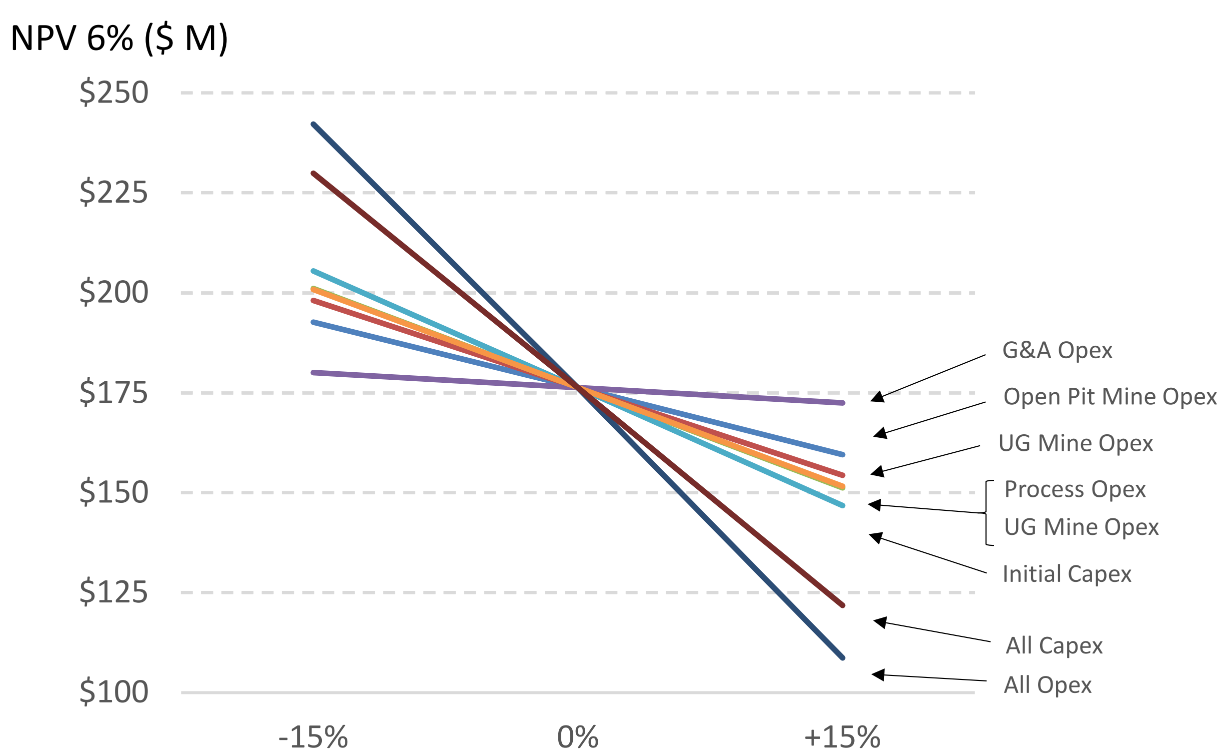
Notes:

1. Payback increases with higher lead price as no lead produced during payback period. Increased alue of subsequent production impacts depletion allowance and associated tax payments

22.5.2 Costs

The 'Spider Graph' presented in Figure 22.6 illustrates the impact of variation in costs to post-tax NPV. Returns are more sensitive to overall operating costs than capital costs, though initial capital has the highest individual impact.

FIGURE 22.6 SENSITIVITY TO COSTS



Source: *Gibsonian (2020)*

22.5.3 Operating Assumptions

Returns are impacted by the following key assumptions:

- **Adjustment to process recovery estimates,** to take account of the difference between that forecast on a block-by-block grade basis compared to the average grade over the course of a month. In the worst case scenario, where no adjustment is made, the NPV would decrease by 5% to $168M, while IRR would decrease by 0.8% to 25.4% and simple payback would increase by approximately one month to 2.5 years. The improvement that would result with an adjustment of 100% is roughly symmetric, with NPV climbing by $9M (to $185M) and IRR increasing to 26.9%. There is no material change to simple payback for the entire range from 0% - 100%.

- **Backfilling the open pit upon closure**. There is anecdotal evidence that jurisdictions previously insisting on backfilling following the completion of mining activity have begun reconsidering that stance after accounting for the environmental impact of backfilling, including carbon footprint of the additional mining activity. In the event it was not necessary to backfill the open pit, NPV would increase to $194M (10% improvement). Given the timing of backfill expenditures, there is no material impact to IRR or simple payback.

- **Payability of precious metals in zinc concentrate.** In the event that no credit was realized for precious metals in zinc concentrate, NPV would fall by 11% to $157M (IRR = 24.4%, simple payback = 2.7 years).

- **Losses during change of campaigns.** It has been assumed that no material process recovery losses would be experienced when changing campaigns from one sulphide material type to another. In the event of losses equivalent to 8 hours of production were experienced when changing to and from Tuff material (i.e. when converting the copper circuit to lead), life of mine payable output would reduce by 0.11% or 1.8 koz AuEq and the associated reduction to NPV would be $1M. The impact of changing to and from Pinwheel mineralization would be less as it shares the same copper circuit with Main Zone material. In the even losses equivalent to 4 hours production were experienced during campaign changes, payable output would reduce by 0.06% or 1.0 koz AuEq, while NPV would reduce by $0.6M.

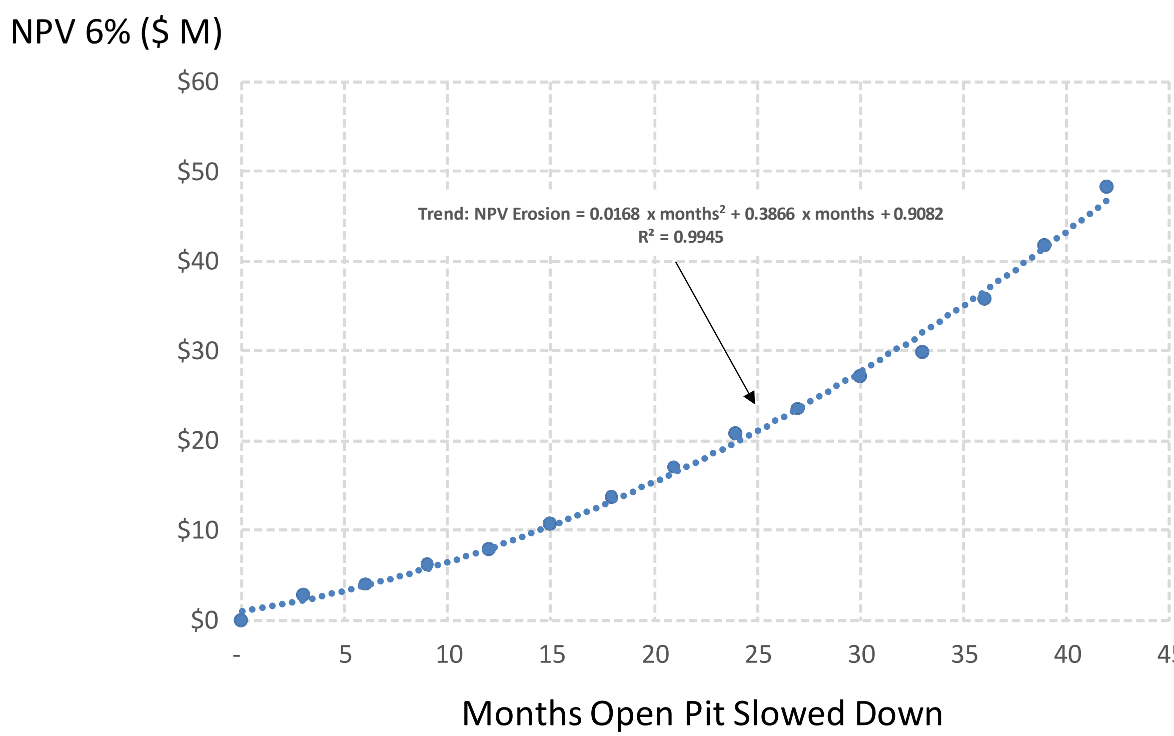
- **Timing of Royalty Payments.** It has been assumed that royalties would be paid when material is mined. A total of 10.5 Mt will be stockpiled, representing 66% of total process feed. The average residence time in stockpiles will be 32 months. In the event that royalties were paid when material was processed, the deferral would increase NPV by approximately $2.0M.

- **High Grade Stockpiles**. To facilitate prioritization of the highest value material to process plants, separate high-grade and low-grade stockpiles are planned for both Main Zone and Oxide material. It is entirely conceivable this segregation could be achieved with a single stockpile that had separate zones for high-grade and low-grade material. In the event that this is not possible, and all stockpile material is blended for both material types, the impact would be as follows:

- o Blending Main Zone stockpiles: NPV reduces by $1.4M (0.8%), IRR reduces by 0.6% and simple payback increases by one month.
- o Blending Oxide stockpiles: NPV reduces by $9.8M (5.6%), IRR reduces by 3.5% and simple payback increases by 9 months.
- o Blending both stockpiles: NPV reduces by $11.6M (6.6%), IRR reduces by 4.0% and simple payback increases by 9 months.

- **Mine Speed Factor.** The stockpiles are in large measure a product of mining the open pit at a rate faster than is required to minimally satisfy the process plants. The current plan is to complete open pit mining in 50 months following the start-up of the process facilities (this is the same mining plan employed in the 2018 Feasibility Study). This is approximately 70% faster than the 'speed' required to satisfy the sulphide process plant, as the 8.8 Mt sulphides mined from the open pit will require approximately 90 months to process (with both cases requiring a further 6 months of pre-stripping). Figure 22.7 illustrates the erosion in NPV that occurs as the open pit is slowed down in increments of 3 months. Note that beyond 30 months (or an 80 month open pit plan), the process plant begins to run at less than full capacity for some months as there is insufficient material of one type to maintain a campaign of a month duration – in the event that campaigns of shorter duration were considered, the erosion of value would not be as significant.

FIGURE 22.7 EROSION OF VALUE FROM SLOWING OPEN PIT



Source: *Gibsonian (2020)*

23.0 ADJACENT PROPERTIES

Section 23.0 has been modified from Connolly et al. (2012).

P&E is not aware of any relevant work on properties immediately adjacent to the Project.

The Back Forty Deposit is hosted within the Penokean Volcanic Belt ("PVB") which extends from the upper peninsula of Michigan to the western portion of north central Wisconsin and which hosts a number of VMS deposits and occurrences which have seen varied extents of exploration and development activity over the course of the past few decades. In addition to the deposits of the PVB, the Back Forty Deposit is situated proximal to intrusive and extrusive rocks associated with the Mid-Continent Rift ("MCR"). The MCR hosts a number of copper and copper-nickel deposits. This section summarizes the non-ferrous mining and exploration activities associated with these terranes proximal to the Back Forty Deposit. Table 23.1 summarizes the non-ferrous properties of the PVB by commodity, company and location. The table does not include the prolific native copper mining district of the Keweenawan Peninsula in the northern part of the Upper Peninsula. The Qualified Person has not been able to verify the information noted in Table 23.1 and the mineralization of the non-ferrous properties is not necessarily indicative of the mineralization at the Project.

Currently, only the Eagle Mine is in operation within the region. The Eagle Mine, approximately 105 km north of the Back Forty Deposit, is hosted by rocks of the MCR and is currently producing nickel and copper as the primary metals.

Past producing mines within the region include the Ropes Gold Mine and White Pine Copper Mine located in the upper peninsula of Michigan and the Flambeau Mine, located in Wisconsin. The Flambeau Mine constitutes the only VMS deposit of the PVB that has gone into commercial production, producing copper, gold and silver. The mine concluded operations in the 1990s.

Presently, there are six advanced exploration properties in the region including Michigan's Upper Peninsula, Wisconsin, Minnesota, and Southern Ontario. These include VMS style deposits of the PVB located in Wisconsin (Crandon and Lynne) as well as copper and copper-nickel deposits associated with the MCR.

Aquila is currently exploring two deposits within the PVB which are both located in Wisconsin. The Bend deposit, located in Taylor County, is a copper and gold-rich VMS deposit. Initial discovery and exploration work was completed in the 1980s by the Jump River Joint Venture. Aquila has been intermittently exploring the Property since 2010. The Reef deposit, located in Marathon County, was initially discovered and explored by INCO, then Noranda in the 1980s. Aquila began acquiring land and has been intermittently exploring the Deposit since 2009.

TABLE 23.1
NON-FERROUS PROPERTIES OF THE GREAT LAKES REGION

Property	Commodity	Affiliated Company	Location	Distance From Back Forty (km)
Operating				
Eagle Mine	Ni/Cu	Lundin	Upper Peninsula of Michigan	105
Former Operations				
Flambeau	Ni/Cu	Kennecott (Rio Tinto)	Wisconsin	160
White Pine	Cu	Highland Copper	Upper Peninsula of Michigan	135
Ropes	Au	Callahan Mining Co.	Upper Peninsula of Michigan	105
Advanced Exploration				
Back Forty	Zn/Au/Cu/Ag	AQA	Upper Peninsula of Michigan	-
Copperwood	Cu	Highland Copper	Upper Peninsula of Michigan	137
Tamarack	Ni/Cu	Rio Tinto	Minnesota	275
Nokomis	Ni/Cu/PGM	Duluth Metals	Minnesota	260
Crandon	Zn/Cu	2 WI Tribes	Wisconsin	65
Thunder Bay North	Ni/Cu/PGM	Magma Metals Ltd.	Southern Ontario	260
Lynne	Zn/Pb/Cu/Ag	Noranda	Wisconsin	115
Exploration Properties				
Peninsula	Au	MPC	Upper Peninsula of Michigan	65
Reef	Au	AQA	Wisconsin	85
Bend	Cu/Au	AQA	Wisconsin	135

24.0 OTHER RELEVANT DATA AND INFORMATION

Several Project key opportunities and risks were identified during the PEA as follows.

24.1 OPPORTUNITIES

- Increased gold process recovery: There is value in further investigating leaching sulphide flotation tailings to economically recover additional gold. Previous scoping metallurgical testwork and cost analysis investigated various options, at a high level, to extract gold from flotation tailings and was favourable at a US$1,600/oz gold price.

- Contract mining: The current mine operations plan is based on an owner-operated mine fleet. Contract mining may be an option to offset initial mine capital costs and mitigate any risks associated with training, operational readiness and the availability of experienced mine personnel.

- Contract process plant operations and maintenance: The current process plant operations plan is based on owner-operating and maintaining the process plant. An operations and maintenance contract may be an option to mitigate any risks associated with training, operational readiness and the availability of experienced plant operators and maintenance personnel.

- Tailings Management: An option may exist to purchase land for additional tailings storage infrastructure due to potential underground mine expansion.

24.2 RISKS

- Commodity prices are considered a major risk to the Project, affecting the Project economics.

- Permitting is ongoing and any delays to the permitting process will affect the Project schedule.

- Employing experienced mining and process plant operators is considered a risk. The Project is based on mining a VMS deposit and treating major metallurgical material types through separate oxide and sulphide process plants.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 PROPERTY DESCRIPTION AND LOCATION

Aquila controls approximately 1,304 hectares (3,222 acres) of private and public (State of Michigan) mineral lands located in Lake and Holmes Townships in Menominee County, Michigan. Approximately 1,019 hectares (2,517 acres) of these lands form a contiguous block of Aquila-controlled mineral rights. The Active Project Boundary encompasses approximately 479 hectares (1,183 acres). The Project is centred at latitude 45° 27’ N and longitude 87° 51’ W.

In addition to the key properties, Aquila has also purchased, leased, or optioned additional properties. These properties are either contiguous with the Key Parcels, may contain facilities utilized by the Company, are perceived to have exploration potential, or were purchased for other strategic purposes.

The Property area lies along the east bank of the Menominee River and consists of low, rolling hills with maximum topographic relief of 30 m and intervening wetland (in part prairie-savannah); mean elevation is approximately 200 to 300 masl. Vegetation is mostly immature hardwood-pine forest and swamp/prairie-savannah grasses; wetland areas also occur along creeks and secondary tributaries.

The Property is located approximately 55 km south-southeast from Iron Mountain, and approximately 19 km west of Stephenson, Michigan, within the Escanaba River State Forest. Access from Stephenson is via County G12 Road, north on River Road, travelling approximately 5 km to the Project field office. A number of drill roads connect with River Road and cross the Property. Infrastructure on the Property includes a nearby power line and paved road access.

To P&E’s knowledge, there are no other significant factors or risks that may affect access, title, or the right or ability to perform work on the Property.

25.2 GEOLOGICAL SETTING AND MINERALIZATION

The Back Forty VMS Deposit is one of a number of deposits located throughout the Ladysmith-Rhinelander volcanic complex in northern Wisconsin and western Michigan. The complex lies within the lower Proterozoic PVB, also known as the Wisconsin Magmatic Terranes. The PVB is part of the Southern Structural Sub-province of the Canadian Shield.

Mineralization at the Back Forty Deposit consists of discrete zones of: 1) zinc or copper-rich massive sulphide (±lead), which may contain significant amounts of gold and silver, 2) stockwork stringer and peripheral sulphide, which can be gold, zinc, and copper-bearing (±lead/silver), 3) precious metal-only, low-sulphide mineralization, and 4) oxide-rich, precious metal-bearing gossan.

25.3 DRILLING AND SAMPLE VERIFICATION

Aquila sent a number of assays for analysis to both Bureau Veritas and MPC laboratories throughout the 2016 and 2017 drill programs. A total of 108 gold results, 52 silver and 47 copper, lead and zinc results were examined. Comparison of the Bureau Veritas results verses the MPC results was made by P&E and results compare well between labs.

P&E notes the direct interest in MPC laboratory that was held by the Company's VP Exploration during the 2016 to 2017 drill program. The VP Exploration is no longer affiliated with Aquila, so now there is no relationship between the laboratory and Company. Analyses carried out at MPC during the 2016 to 2017 program totalled less than 10% of all drill core samples analyzed and, of these samples, only around 3% were exclusively analyzed at MPC. All other samples were also analyzed at Bureau Veritas and comparison of these duplicate analyses is acceptable.

It is P&E's opinion that sample preparation, security and analytical procedures for the Project drilling and sampling programs were adequate for the purposes of the Updated Mineral Resource Estimate.

Based upon the evaluation of the QA/QC programs undertaken by Aquila, P&E concludes that the data are of good quality for use in the Back Forty Updated Mineral Resource Estimate.

25.4 MINERAL PROCESSING AND METALLURGICAL TESTING

Various metallurgical test work campaigns were completed to quantify the metallurgical response of the VMS mineralization and included several flotation and leaching studies, comminution and gravity tests. This work was used to established metallurgical domains and direction for test conditions and to demonstrate variability throughout the Deposit. Metallurgical testing has generally focused on the three main sulphide mineralized zones (Main, Pinwheel and Tuff Zones) and the oxide portion of the Deposit.

Test results from 2016-2019 testwork programs and historical test results formed the basis for each of the metallurgical recovery equations. In general, there is a reasonable correlation between the head grade of the target base metal with the ultimate recovery to the concentrate. Target concentrate grades were selected based on the metallurgical performance of the samples for each material type for the financial analysis. For copper, the target concentrate grade varies for each copper containing material type and has a range of 17% to 22% Cu. Zinc concentrate targets vary from 50% to 55% Zn and the lead target concentrate grade is 35% Pb.

25.5 MINERAL RESOURCE ESTIMATE

P&E considers the mineralization of Back Forty to be potentially amenable to Open Pit and Out of Pit (underground) extraction. Open pit model NSR cut-off values were \$21/t for flotation and \$22/t for leach material above 0 m EL, and \$70/t below 0 m EL. Underground model NSR cut-off values ranged from \$65/t to \$68/t for flotation and \$77/t for leach material.

The pit-constrained Mineral Resource Estimate totalled 11.4 Mt of Measured and Indicated Mineral Resources at 1.87 g/t Au, 23.03 g/t Ag, 0.27% Cu, 0.22% Pb and 2.62% Zn. Pit-constrained Inferred Mineral Resources totalled 0.3 Mt at 3.13 g/t Au, 42.32 g/t Ag, 0.06% Cu, 0.56% Pb and 0.62% Zn. The underground Mineral Resource Estimate totalled 6.9 Mt of Measured and Indicated Mineral Resources at 1.93 g/t Au, 25.86 g/t Ag, 0.40% Cu, 0.32% Pb and 3.71% Zn. Underground Inferred Mineral Resources totalled 0.9 Mt at 3.88 g/t Au, 51.21 g/t Ag, 0.47% Cu, 0.45% Pb and 1.40% Zn.

25.6 MINING METHODS

The mine plan is based on open pit mining from Year 1 to Year 5 utilizing conventional open pit mining equipment. Underground development will be initiated in Year 5 and underground production mining will continue to Year 11, utilizing cut and fill and sublevel longhole methods, with cemented paste backfill. The process plant will be fed with stockpiled material during production Year 12 and a partial Year 13.

A series of grade blending stockpiles, by material type, will serve to prioritize the processing of higher-grade material and also manage fluctuations in process plant feed delivery from the two mining operations.

The open pit design is based on the 2018 Feasibility Study design. Minor modifications were made to standardize on 5 m high benches with a quadruple (4) bench configuration, resulting in a 20 m vertical distance between catch berms. For scheduling purposes, the Back Forty pit was subdivided into three phases. Mining commences in a small higher-grade pit and subsequently expands outwards by pushing back the pit wall. This enables annual waste stripping quantities to be distributed to avoid high and low annual tonnages.

25.7 RECOVERY METHODS

Oxide and sulphide mineralized material (Main, Pinwheel and Tuff Zones) are treated through separate process plants.

The oxide mill feed will be processed via a cyanidation leach circuit to produce doré. Depending on the grades of copper, zinc and lead, the sulphide mill feed will be processed via two stages of flotation to produce concentrates, i.e. either a copper and zinc concentrate, or a lead and zinc concentrate.

Sulphide mill feed will be processed on a campaign basis based on the main material types that have a similar metallurgical response. As such, the design of the sulphide process plant is based on a flexible metallurgical flowsheet to process the main material types.

25.8 SITE INFRASTRUCTURE

The initial Project includes an open pit mine, mineralized material stockpiles, oxide and sulphide processing plants, tailings management facility (“TMF”), waste rock storage facilities (“WRF”), cut-off wall (“COW”) to control water seepage from the Menominee River to the open pit, contact water basin (“CWB”), non-contact water basins (“NCWB”), waste water treatment plant (“WWTP”), mine services, overburden stockpile and access roads.

Prior to commencing underground mining, a paste backfill plant will be installed near the open pit mine to provide cemented paste backfill for mined-out stopes.

Access to the Project is from the east side of the Property off the existing County Road 356. Main access will be via the main security gate near the process plant.

Grid power will be provided from an incoming 138 kV high voltage ("HV") line from the east side of the Project along the main access road.

25.9 ENVIRONMENTAL STUDIES, PERMITS AND SOCIAL OR COMMUNITY IMPACT

Aquila currently holds several permits as required in Michigan's environmental regulations. The current permits that have been issued for the Back Forty Project include:

- Part 632 Mining Permit (MP 01 2016) for mining and beneficiation activities associated with the Back Forty Deposit.
- Part 632 Mining Permit (MP 01 2016) Amendment.
- National Pollutant Discharge Elimination System ("NPDES") Permit (MI0059945) for treated process wastewaters.
- Michigan Air Use Permit to Install ("PTI") (205-15) has been issued for the Project for emissions associated with construction and mining activities.
- PTI 205-15 Modification for the updated facilities.
- Part 301 Inland Lakes and Streams and Part 303 Wetlands Protection Permit (WRP011785).
- Dam Safety Permit application for the construction of the CWB and TMF regulated under NREPA, Part 315 is under development.

Currently, there are four ongoing legal actions involving the Menominee Tribe, regarding the wetland and mine permits.

25.10 ECONOMIC ANALYSIS

Open pit mining costs have been estimated to average \$3.03/t material mined or \$11.22/t total material processed over the LOM. Underground mining costs have been estimated to average \$50.31/t material mined or \$18.16/t total material processed over the LOM. Processing costs (\$19.55/t material processed) and site G&A (\$2.90/t material processed) contribute to a total LOM average cost estimated at \$52/t material processed.

Initial capital costs are estimated at \$250M and include a 14% contingency. Sustaining capital costs are estimated at \$297M for mining equipment lease capital costs, underground mine infrastructure, paste backfill plant, COW, TMF, WRFs and closure costs.

Using bank consensus long term metal pricing of US\$1,485/oz Au, US\$1.08/lb Zn, US\$3.05/lb Cu, US\$18.20/oz Ag and US\$0.91/lb Pb, the Project has an estimated pre-tax NPV at a 6% discount rate of \$248M and an IRR of 31.6%. After-tax NPV and IRR are estimated at \$176M and 26.1%, respectively. Simple payback is 2.4 years. Post-tax NPV: peak investment ratio is 0.80.

Project economics are most sensitive to gold and zinc prices. In terms of Project costs, Project economics are more sensitive to overall operating costs than capital costs.

The PEA has highlighted several opportunities to improve Project economics and reduce identified risks. These include opportunities to optimize mining and processing plants, including contract operations instead of owner-operated, and to develop the underground mine as an expansion opportunity.

26.0 RECOMMENDATIONS

Subject to ongoing Project funding and board approval, it is recommended that Aquila commence an updated Feasibility Study followed by basic engineering of the Project in line with the preliminary Project implementation plan including additional studies and site investigations listed below.

26.1 GEOMECHANICAL DESIGN INPUT

- Collect bathymetry data for the Menominee River in the area of the Project.
- Complete drilling or probing to confirm the bedrock profile below the Menominee River where it overlies the proposed underground workings in the Pinwheel Zone.
- Complete additional geomechanical drill holes with oriented core or televiewer surveys in the northern end of the Pinwheel Zone and in the immediate vicinity of a potential underground portal. The drill holes in the Pinwheel Zone also provide an opportunity to collect additional hydrogeological data between the river and the open pit.
- Develop a 3-D numerical model to evaluate the interaction between the open pit and potential underground workings. The model should consider the relative sequencing of, and interactions between, open pit and underground mining. The impact of mining on the cut-off wall and the proposed crown pillars should also be considered.
- The influence of underground stopes in close proximity to the open pit slopes and the stability of those slopes should be evaluated using 2-D or 3-D numerical models as appropriate.

26.2 MINING

- Complete trade-off studies to examine the use of contract mining versus owner operated mining for both open pit and underground operations.
- During the early stages of open pit mining, undertake blast vibration studies to assess vibration levels. This will provide data that can be used for open pit blast design in potentially sensitive areas.
- Complete an economic evaluation for underground mining deeper extensions of the Deposit.

26.3 WASTE WATER TREATMENT

Subject to ongoing discussions with the regulator, it is recommended that the following be considered in upcoming Project phases:

- Updates to the estimated WWTP influent quality including refinements to estimated groundwater inflow and quality and source terms based on further metallurgical testing and geochemical characterization.

- Review of the proposed mercury removal technology included in the design of the WWTP. This review may include, but is not limited to, vendor engagement, benchmarking, and test work on a synthetic sample of WWTP influent.

- Review of the estimated WWTP influent quality against the 2019 Surface Water Quality Final Acute Value for sulphate to determine whether sulphate mitigation is required.

- Review of aquatic toxicology literature for the three ridge mussel and the plan for whole effluent toxicity testing using glochidia and juvenile life stages of the three ridge mussel as required in the NPDES permit. Confirm whether changes to the WWTP design criteria, specifically treated effluent quality parameters and concentration limits as defined in the NPDES permit, and therefore the WWTP design, will be required to prevent failure of whole effluent toxicity tests.

- Review of the regulatory and technical feasibility of on-site disposal of WWTP residuals within the TMF, including a review of the potential risk of voiding the Bevill Amendment to the Resource Conservation and Recovery Act ("RCRA") which allows for the disposal of mining wastes as non-hazardous waste.

26.4 METALLURGICAL TESTWORK

- Complete additional testwork to further evaluate leaching flotation tailings to recover additional gold.

- Review flotation recovery performance due to mixing of mineralized types at interfaces as part of mining operations:
  - Prioritized based on spatial proximity
  - Oxides interface with Pinwheel
  - Pinwheel sub-domain interfaces
  - Blocks that contain significant lead (Pb>Cu).

- Review potential of stockpile oxidation losses/impacts and incorporate into recovery equations in the financial model.

- Review stockpile incidental mixing and process plant feed losses and/or error proportionate to probabilities.

- Complete additional cyanide destruction testwork to confirm process design of the oxide process plant.

- Complete cemented paste backfill testwork.

- Continue with concentrate quality testwork and analysis.

26.5 PROCESS PLANT AND ASSOCIATED INFRASTRUCTURE

- Advance and update the sulphide and oxide process plant design to Feasibility Study level.
- Update surface infrastructure design to Feasibility Study level.
- Complete a preliminary paste backfill plant design.
- Advance main access road design concepts.
- Develop a contract with the local electrical power utility.

26.6 OPERATIONAL READINESS

- Continue to develop operational readiness plans.

26.7 TAILINGS MANAGEMENT AND WASTE ROCK FACILITIES

The subsequent stages of the Project should include the following tasks related to the TMF, WRFs, and OS:

- A trade-off study should be carried out to identify the optimal disposal option for the excess 1.25 Mm^3 of tailings.

- A detailed land survey, and bathymetric survey where required, should be carried out within the footprint areas of the TMF, WRFs, and OS.

- Supplementary geotechnical investigation should be carried out at the TMF, WRFs and OS.

- Condemnation drilling should be carried out within the footprint areas of the TMF, WRFs, OS to ensure the facilities are not sterilizing potential Mineral Resources.

- A borrow investigation should be carried out to identify potential sources of crushed aggregates for the TMF and WRFs close to the Project site.

- Long-term chemical compatibility testing should be carried out between the predicted leachate and the proposed GCL.

- Liner puncture testing should be carried out using the proposed coarse aggregate and the cushion geotextile.

- Additional tailings thickening tests such as flow loop testing, pilot scale thickening testing, and vendor testing should be completed on representative tailings samples.

26.8 CUT-OFF WALL

A technically and economically feasible CSM wall design was developed to control seepage from the Menominee River to the open pit. The following additional work is required prior to the construction of the COW:

- Complete a geophysical study in the area between the COW and the ultimate open pit rim to determine if there are boulders and bedrock troughs that might impact the reliability of the selected technology.

- Finalize the alignment of the COW location with consideration to optimizing the mine design and extraction of the Deposit.

- Carry out additional borehole drilling program if the final alignment of the COW is far from the existing borehole data.

- Carry out a Cone Penetration Testing ("CPT") investigation and confirm the stability analysis of the river bank near the Project site.

26.9 BUDGET

The proposed budget for a recommended work program is estimated at $4M and is summarized in Table 26.1.

TABLE 26.1
RECOMMENDED WORK PROGRAM BUDGET

Scope	Cost ($ M)
Process Plant and Infrastructure Engineering	1.80
Mining Study (Open Pit and Underground)	0.50
Geotechnical Site Investigation	0.50
Provisional Drilling	0.25
Hydrogeology Model Update	0.05
Paste Testwork and Backfill Plant Design	0.30
Water Quality and Water Balance Modelling	0.05
Metallurgical Testwork	0.30
Deformation and Numerical Modelling	0.10
General and Administration	0.15
Total	**4.00**

27.0 REFERENCES

Bryan et al., 2014. Preliminary Economic Assessment of the Back Forty Deposit, Michigan, USA. Prepared by Tetra Tech MM Inc. for Aquila Resources Inc. Effective date July 23, 2014.

Connolly, et al., 2012. Preliminary Economic Assessment Technical Report on the Back Forty Deposit, Menominee County, Michigan, USA. Prepared for Aquila Resources Inc. April 26, 2012.

DeMatties, Theodore A. 2007. An Evaluation of the Back Forty Volcanogenic Massive Sulfide (VMS) Deposit, Menominee County, Michigan, USA, Technical Report. Cambridge Minnesota. Independent technical report prepared for Aquila Resources (USA) Inc. and filed on SEDAR.

Foth Infrastructure & Environment, LLC, 2015a. Environmental Impact Assessment. October 2017.

Foth Infrastructure & Environment, LLC, 2015b. Mining Permit Application. October 2017.

Foth Infrastructure & Environment, LLC, 2015c. State of Michigan National Pollutant Discharge Permit Application for Discharges to Surface Waters. October 2015.

Foth Infrastructure & Environment, LLC, 2015d. Michigan Air Use Permit – Permit to Install. October 2015.

Foth Infrastructure & Environment, LLC, Stantec Consulting Services Inc., King & MacGregor Environmental, Inc., 2017. Permit Application – Back Forty Project. MDEQ/USACE Joint Permit Application for: Wetland Protection, Inland Lakes and Streams, Floodplain, MDEQ Submittal No. 2NN-5PE0-MT3W. October 2017.

Franklin, J.M., and Thorpe, R.I., 1982. Comparative metallogeny of the Superior, Slave and Churchill provinces in Hutchinson, R.W., Spence, C.D., and Franklin, J.M., Precambrian sulfide deposits (H.S.Robinson Memorial Volume): Geological Association of Canada Special Paper 25, p3-90.

Golder Associates Ltd., 2016. "Feasibility Pit Slope Design – Back Forty Project", Prepared for Aquila Resources Inc. November 11, 2016.

Golder Associates Ltd., "Cutter Soil Mixing Cut-off Wall Feasibility Design – Back Forty Project- Draft", December 15, 2017.

Golder Associates Ltd., "Feasibility Design of Tailings and Waste Rock Management Facilities – Back Forty Project- Draft", March 12, 2017.

Golder Associates Ltd., "Tailings Management Facility, Waste Rock Facilities, Open Pit Co-Disposal, Ore Storage Areas and Overburden Stockpile - Preliminary Economic Assessment Design, Lower Throughput Case, Back Forty Project, Michigan - Draft", August 23, 2019.

G&T Metallurgical Services Ltd., “Preliminary Metallurgical Assessment of the Back Forty Deposit”, June 13, 2007.

G&T Metallurgical Services Ltd., “Flowsheet Development Metallurgical Program – Back Forty Project”, November 17, 2008.

G&T Metallurgical Services Ltd., “Locked Cycle and Cyanidation Bottle Roll Testing for the Back Forty Project”, March 30, 2010.

G&T Metallurgical Services Ltd., “Metallurgical Assessment of the Back Forty Deposit”, January 14, 2011.

Lincoln et al., 2018. Back Forty Project, Michigan, USA – Feasibility Study, NI 43-101 Technical Report. Prepared by Lycopodium Minerals Canada Ltd. Effective date February 6, 2018.

Knight Piesold Ltd., 2018. Back Forty Project – Underground Mine Design – Preliminary Paste Fill Strength Requirements. Prepared for Aquila Resources Inc. March 21, 2018.

Knight Piesold Ltd., 2018. Back Forty Project – Prefeasibility Underground Mine Recommendations – Updated Stope Dimensions. Prepared for Aquila Resources Inc. June 12, 2018.

Knight Piesold Ltd., 2019. Back Forty Project – Numerical Hydrogeological Model Update. Prepared for Aquila Resources Inc. June 21, 2019.

Knight Piesold Ltd., 2019. Back Forty Project – Crown Pillar Recommendations. Prepared for Aquila Resources Inc. November 28, 2019.

Ocean Partners, “Market Review and Analysis – Back Forty Project”, April 11, 2015.

Paterson & Cooke Canada Inc., 2017. Backfill Prefeasibility Study, Test Work Report. Prepared for Aquila Resources Inc. December 21, 2017.

Puritch et al., 2018. Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA. Prepared by P&E Mining Consultants Inc. Effective date February 6, 2018.

Resource Development Inc, “Metallurgical Testing of Back Forty Oxide Ores”, June 28, 2011.

Resource Development Inc, “Metallurgical Testing of Back Forty Oxide and Sulfide Ores”, June 29, 2012.

SGS Lakefield Research Ltd., “East Gossan Ore Zone Testwork Results”, February 11, 2009.

SGS Canada Inc., “Metallurgical Testing of Three Gold-Bearing Samples from the Back Forty Project”, July 14, 2010.

SGS Lakefield Research Ltd., “Benefication and Cyanidation Testing on Samples from the Back Forty Deposit”, September 6, 2016.

SGS Lakefield Research Ltd., “Metallurgical Variability Testing on Samples from the Back Forty Deposit”, February 15, 2018.

Sims, P.K., Van Schmus, W.R., Schulz, K.J., and Peterman, Z.E., 1989. Tectonostratigraphic evolution of the Early Proterozoic Wisconsin magmatic terranes of the Penokean Orogen: Canadian Journal of Earth Sciences, v. 26, p. 2145-2158.

Tetra Tech, 2014. Preliminary Economic Assessment of the Back Forty Project, Michigan, USA. Prepared for Aquila Resources Inc. Effective date July 23, 2014.

Tinti, Steven J., 2019. Title Opinion on Certain Menominee County, Michigan Real Estate. Prepared for Aquila Resources. October 14, 2019.

Tinti, Steven J., 2019. Title Opinion. Prepared for Aquila Resources. October 15, 2019.

28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON
ANDREW BRADFIELD, P. ENG.

I, Andrew Bradfield, P. Eng., residing at 5 Patrick Drive, Erin, Ontario, Canada, N0B 1T0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the Technical Report titled “Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA”, (The “Technical Report”) with an effective date of October 14, 2019.
3. I am a graduate of Queen’s University, with an honours B.Sc. degree in Mining Engineering in 1982. I have practiced my profession continuously since 1982. I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM. 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.

   I have practiced my profession continuously since 1982. My summarized career experience is as follows:
   - Various Engineering Positions – Palabora Mining Company, 1982-1986
   - Mines Project Engineer – Falconbridge Limited, 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited, 1987-1990
   - Independent Mining Engineer, 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc, 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc, 1996-1999
   - Independent Mining Engineer, 1999-2001
   - Principal Mining Engineer – SRK Consulting, 2001-2003
   - COO – China Diamond Corp, 2003-2006
   - VP Operations – TVI Pacific Inc, 2006-2008
   - COO – Avion Gold Corporation, 2008-2012
   - Independent Mining Engineer, 2012-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 2, 3, 15 and 24 and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a “Qualified Person” for a Technical Report titled “Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA”, with an effective date of February 6, 2018.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signing Date: September 16, 2020

{SIGNED AND SEALED}
[Andrew Bradfield]

____
Andrew Bradfield, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 4 Creek View Close, Mount Clear, Victoria, Australia, 3350, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for a total of 15 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875), Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Geologist, Foran Mining Corp. 2004
   - Geologist, Aurelian Resources Inc. 2004
   - Geologist, Linear Gold Corp. 2005-2006
   - Geologist, Búscore Consulting 2006-2007
   - Consulting Geologist (AusIMM) 2008-2014
   - Consulting Geologist, P.Geo. (APEGBC/AusIMM) 2014-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 11 and co-authoring Sections 1, 12, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018, and for a Technical Report titled "Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA", with an effective date of February 6, 2018.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[Jarita Barry]

___

Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, Canada, do hereby certify that:

1. I am an independent geological consultant contracted by P & E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Exploration Geologist, Cameco Gold 1997-1998
   - Field Geophysicist, Quantec Geoscience 1998-1999
   - Geological Consultant, Andeburg Consulting Ltd. 1999-2003
   - Geologist, Aeon Egmond Ltd. 2003-2005
   - Project Manager, Jacques Whitford 2005-2008
   - Exploration Manager – Chile, Red Metal Resources 2008-2009
   - Consulting Geologist 2009-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 4, 5, 6, 7, 8, 9, 10 and 23 and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018, and for a Technical Report titled "Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA", with an effective date of February 6, 2018.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[David Burga]

____
David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
KEBREAB HABTE, P.ENG.

I, Kebreab Berhane Habte, M.Sc.(Eng.), P.Eng., residing at 22 Nature Court, Hamilton, Ontario, Canada, do hereby certify that:

1. I am employed as a Senior Geotechnical Engineer with consulting firm Golder Associates Ltd. located at 6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada, L5N 7K2.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of the University of Asmara, Eritrea with a Bachelor of Science degree in Soil and Water Conservation (1998) and a graduate from the University of KwaZulu-Natal, South Africa with a Master of Civil Engineering degree (2004). I have worked as an engineer for more than 16 years since graduating. I am a professional engineer in good standing currently licensed by the Professional Engineers Ontario (PEO) in Canada (#100174660).

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

   My relevant experience for the purpose of the Technical Report is:
   - Geotechnical Engineer and then Senior Geotechnical Engineer, Golder Associates Ltd. 2008-Present
   - Geotechnical Engineer, Golder Associates (UK) Ltd. 2007-2008
   - Geo-environmental Design Engineer, Golder Associates Africa (Pty) Ltd. 2005-2007
   - Civil Engineering Lecturer, Durban University of Technology 2004-2005
4. I have visited the Property that is the subject of this Technical Report on June 28, 2016 for one day.
5. I am responsible for co-authoring Sections 1, 18.7, 18.8, 21.1.4, 21.3.9, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[Kebreab Berhane Habte]

____

Kebreab Berhane Habte, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
KENNETH KUCHLING, P.ENG.

I, Kenneth Kuchling, P. Eng., residing at 33 University Ave., Toronto, Ontario, Canada, M5J 2S7, do hereby certify that:

1. I am a senior mining consultant with KJ Kuchling Consulting Ltd. located at #1903-33 University Ave, Toronto, Ontario, Canada, contracted as a senior mining associate by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of McGill University with a Bachelor degree in Mining Engineering (1980) and a graduate from the University of British Columbia with a Master of Engineering degree in Mining Engineering (1984). I have practiced my profession continuously as a mining engineer since my graduation from university in 1980 and with P&E Mining Consultants Inc. since 2009. My relevant work experience for the purpose of the Technical Report is 12 years as an independent mining consultant in commodities such as gold, copper, potash, diamonds, molybdenum, tungsten, and bauxite. I am a professional engineer in good standing currently licensed by the Professional Engineers of Ontario (PEO) in Canada (no. 100173556).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Associate Mining Engineer, P&E Mining Consultants Inc. 2011 – Present
   - Mining Consultant, KJ Kuchling Consulting Ltd. 2000 – Present
   - Senior Mining Engineer, Diavik Diamond Mines Inc., 1997 – 2000
   - Senior Mining Consultant, KJ Kuchling Consulting Ltd., 1995 – 1997
   - Senior Geotechnical Engineer, Terracon Geotechnique Ltd., 1989 - 1995
   - Chief Mine Engineer, Mosaic, Esterhazy K1 Operation. 1985 – 1989
   - Mining Engineering, Syncrude Canada Ltd. 1980 – 1983
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1, 16, 18, 21, 25 and 26 of this Technical Report.
6. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[Kenneth Kuchling]

____
Kenneth Kuchling, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
NEIL LINCOLN, P.ENG.

I, Neil Lincoln, B.Sc., P.Eng., residing at 383 Allan Street, Oakville, Ontario, Canada, L6J 3P6, do hereby certify that:

1. I am an independent Consulting Metallurgist and Study Manager of Lincoln Metallurgical Inc.
2. This certificate applies to the Technical Report titled “Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA”, (The “Technical Report”) with an effective date of October 14, 2019.
3. I am a graduate of the University of the Witwatersrand, South Africa with a Bachelor of Science in Metallurgy and Materials Engineering (Minerals Process Engineering) degree (1994). I have practiced my profession continuously as a metallurgist for 24 years. I am a professional engineer in good standing currently licensed by the Professional Engineers Ontario (PEO) in Canada (no. 100039153).

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

   My relevant experience for the purpose of the Technical Report is:

   - Consulting Metallurgist and Study Manager, Lincoln Metallurgical Inc. 2019-current
   - VP Business Development and Studies, Lycopodium Minerals 2011-2019
   - Senior Process Engineer and Study Manager, Hatch 2010-2011
   - Senior Metallurgist and Study Manager, SNC-Lavalin 2006-2010
4. I have visited the Property that is the subject of this Technical Report on October 24, 2017, and inspected the proposed open pit, process plant, tailings storage facility and mine infrastructure areas.
5. I am responsible for authoring Sections 13 and 20 and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a “Qualified Person” for a Technical Report titled “NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA”, with an effective date of August 1, 2018.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[Neil Lincoln]

____
Neil Lincoln, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
MANOCHEHR OLIAZADEH, P.ENG.

I, Manochehr Oliazadeh, Ph.D., P.Eng., residing at 976 Cristina Court, Mississauga, Ontario, Canada, L5J 4S1, do hereby certify that:

1. I am a principal process engineer working for Lycopodium Minerals Canada Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of Leeds University with a Doctor of Philosophy (Honours) degree in Minerals Engineering (1990). I have worked as a process engineer for a total of 28 years since graduating. I am a principal process engineer currently licensed by the Professional Engineers of Ontario (License No 100119302).

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

   My relevant experience for the purpose of the Technical Report is:

   - Manager of Process, Lycopodium Minerals Canada 2017-Present
   - Director BD/Projects, Middle East, Hatch Associates 2016-2017
   - Regional Director, Mineral Processing, Hatch Associates 2012-2016
   - Interim Manager, Minerals Processing, Hatch Associates 2011-2012
   - Senior Process Engineer, Hatch Associates 2007-2011
   - Technical Deputy, KE Company 2003-2005
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 17 and co-authoring Sections 1, 18, 21, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[Manochehr Oliazadeh]

____
Manochehr Oliazadeh, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
DAVID PENSWICK, P.ENG.

I, David Penswick, BSc Eng., MSc Eng., P.Eng., residing at #71-1 Elsie Lane, Toronto, Ontario, Canada, do hereby certify that:

1. I am an independent mining consultant with Gibsonian Inc., Toronto, Ontario, Canada.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of Queens University with a Bachelor of Science degree in Mining Engineering (1989) and I am a graduate of the University of Witwatersrand in Johannesburg, South Africa with a Master of Science degree in Mining Engineering (1993). I have practiced my profession continuously as a mining engineer in various capacities since 1989. I am a professional engineer in good standing currently licensed by the Professional Engineers Ontario (PEO) in Canada (license no. 100111644).

   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.

   I have been continuously self-employed as a consultant since 2002. The bulk of my engagements during this period have included mine design and/or evaluation and are thus relevant for the purpose of this Technical Report.
4. I have visited the Property that is the subject of this Technical Report on November 2, 2017 and inspected various elements pertaining to the forecast overall economic performance of the project.
5. I am responsible for authoring Sections 19 and 22 and co-authoring Sections 1, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signed Date: September 16, 2020

{SIGNED AND SEALED}
[David Penswick]

____

David Penswick P.Eng.

CERTIFICATE OF QUALIFIED PERSON
EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene J. Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, Canada, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's Degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy. 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.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:

   - Mining Technologist - H.B.M.& S. and Inco Ltd., 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, 1995-2004
   - President – P&E Mining Consultants Inc, 2004-Present

4. I have visited the Property that is the subject of this Technical Report on May 23, 2016 to review both the Mineral Resource Estimate and open pit engineering.
5. I am responsible for co-authoring Sections 1, 12, 14, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018, and for a Technical Report titled "Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA", with an effective date of February 6, 2018.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019

Signed Date: September 16, 2020

{SIGNED AND SEALED}

[Eugene Puritch]

____

Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON
D. GREGORY ROBINSON, P. ENG.

I, David Gregory Robinson, P.Eng., residing at 1236 Sandy Bay Road, Minden, Ontario, Canada, K0M 2K0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of Dalhousie University, Queens University and Cornell University, and Professional Engineer of Ontario (License No. 100216726).

   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.

   I have practiced my profession continuously since 2008. My summarized career experience is as follows:

• Lead Mining Engineer, P&E Mining Consultants	Aug 2017 - Present
• Mine Engineer, Lac des Iles Mine, North American Palladium	May 2016 – Jun 2017
• Senior Underground Engineer, Phoenix Gold, Rubicon Minerals	Sep 14 – Jan 2016
• Mine Engineer, Diavik Diamond Mine, Rio Tinto Diamonds	Sep 2011 – Sep 2014
• Mine Engineer, Bengalla Mine, Rio Tinto Coal and Allied	Dec 2008 – Sep 2011
• EIT, Creighton Mine, Vale-Inco	May2008 - Dec, 2008

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1, 16, 21, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had no prior involvement with the Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 14, 2019
Signing Date: September 16, 2020

{SIGNED AND SEALED}
[D. Gregory Robinson]

____

D. Gregory Robinson, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
YUNGANG WU, P.GEO.

I, Yungang Wu, P. Geo., residing at 3246 Preserve Drive, Oakville, Ontario, Canada, L6M 0X3, do hereby certify that:

1. I am an independent consulting geologist contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment of the Back Forty Project, Menominee County, Michigan, USA", (The "Technical Report") with an effective date of October 14, 2019.
3. I am a graduate of Jilin University, China, with a Master's Degree in Mineral Deposits (1992). I am a geological consultant and a registered practising member of the Association of Professional Geoscientists of Ontario (Registration No. 1681).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is as follows:
   - Geologist –Geology and Mineral Bureau, Liaoning Province, China 1992-1993
   - Senior Geologist – Committee of Mineral Resources and Reserves of Liaoning, China 1993-1998
   - VP – Institute of Mineral Resources and Land Planning, Liaoning, China 1998-2001
   - Project Geologist–Exploration Division, De Beers Canada 2003-2009
   - Mine Geologist – Victor Diamond Mine, De Beers Canada 2009-2011
   - Resource Geologist– Coffey Mining Canada 2011-2012
   - Consulting Geologist 2012-Present
4. I have visited the Property that is the subject of this Technical Report on May 23, 2016 and on November 13-14, 2017 to review items pertaining to the Mineral Resource Estimate.
5. I am responsible for co-authoring Sections 1, 12, 14, 25 and 26 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "NI 43-101 Technical Report, Back Forty Feasibility Study, Michigan, USA", with an effective date of August 1, 2018, and for a Technical Report titled "Updated Mineral Resource Estimate and Technical Report on the Back Forty Project, Michigan, USA", with an effective date of February 6, 2018.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

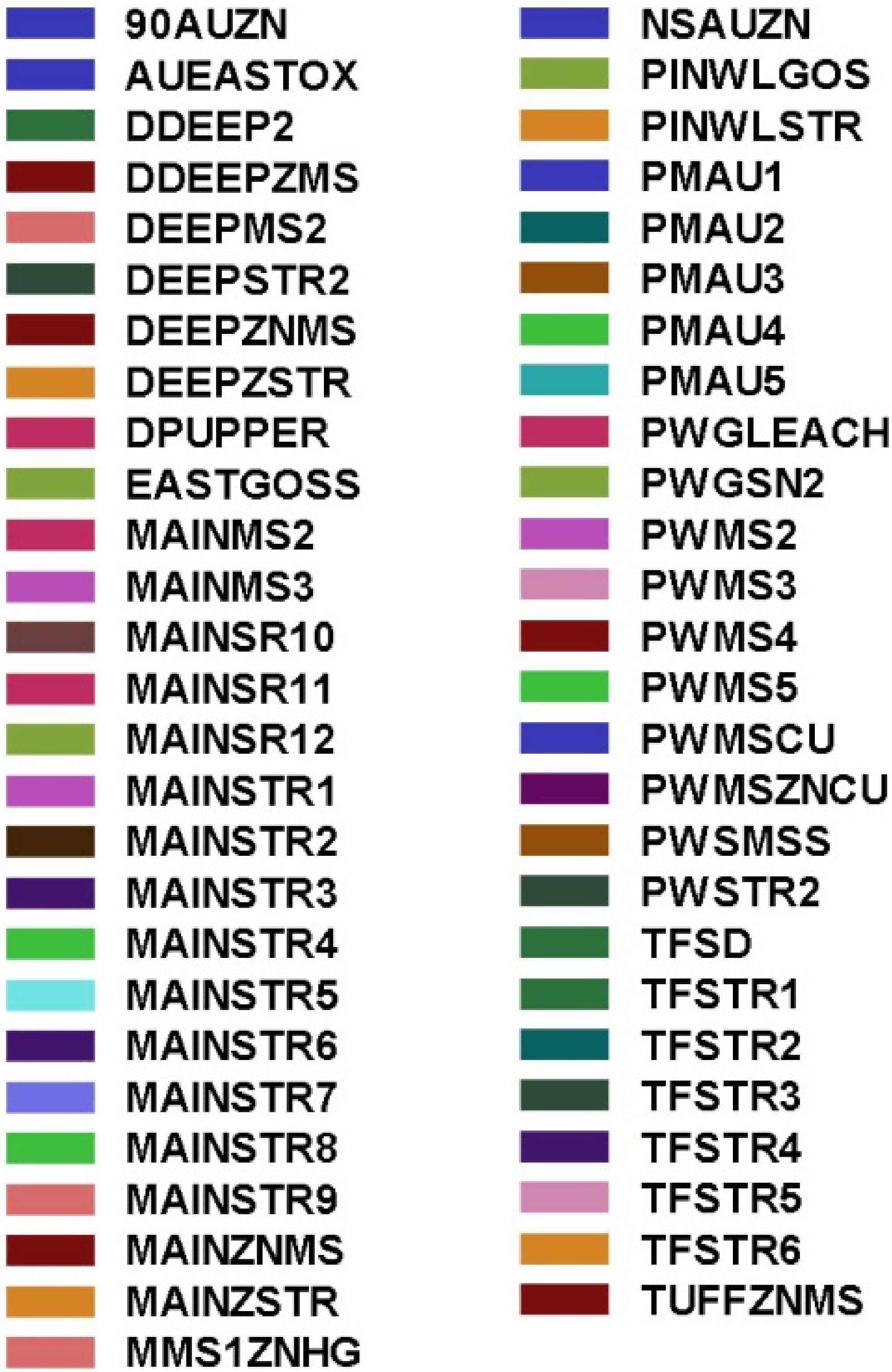
Effective Date: October 14, 2019
Signing Date: September 16, 2020

{SIGNED AND SEALED}
[Yungang Wu]

____

Yungang Wu, P.Geo.

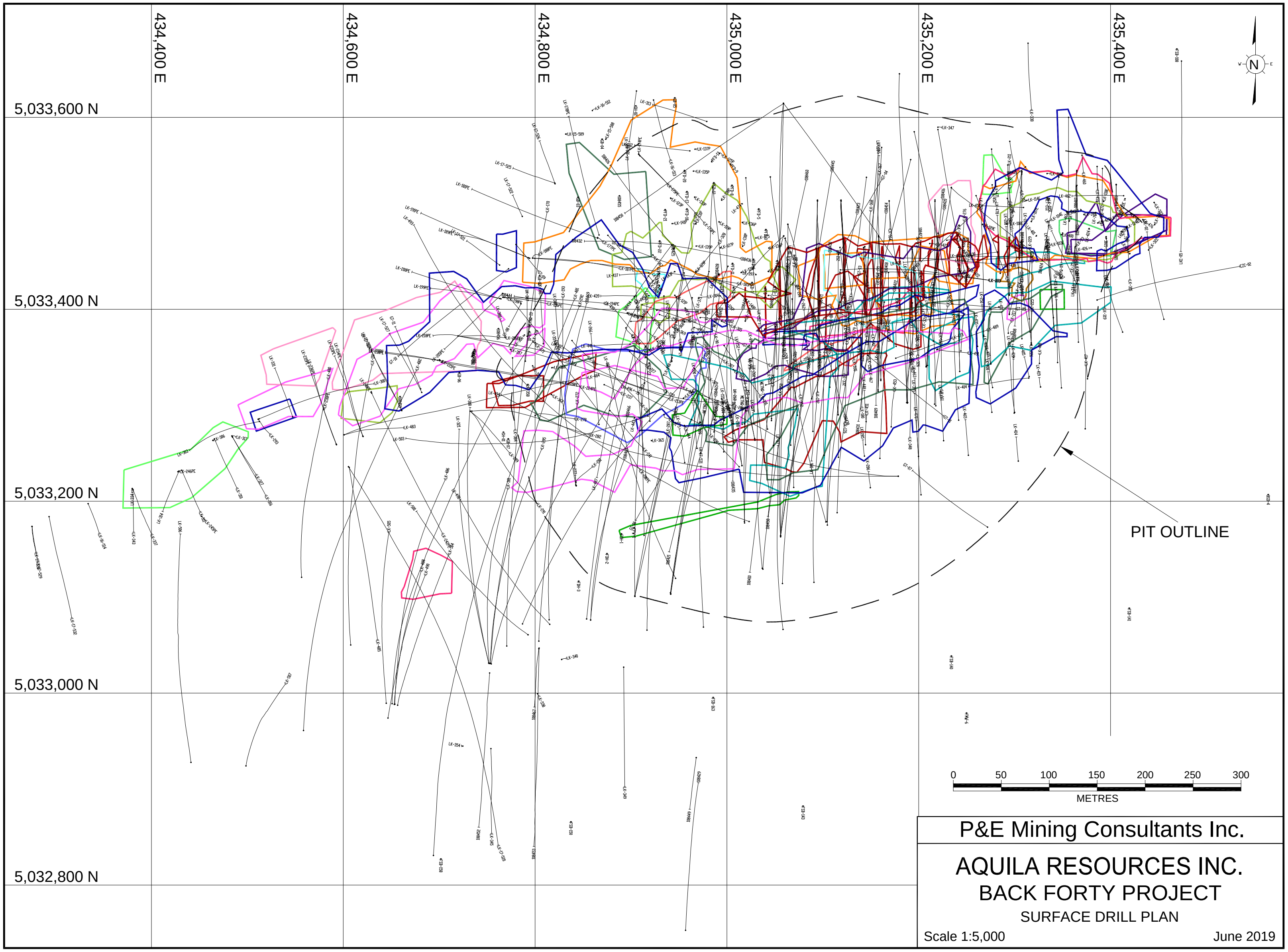
OPEN PIT DOMAINS LEGEND FOR APPENDICES



UNDERGROUND DOMAINS LEGEND FOR APPENDICES

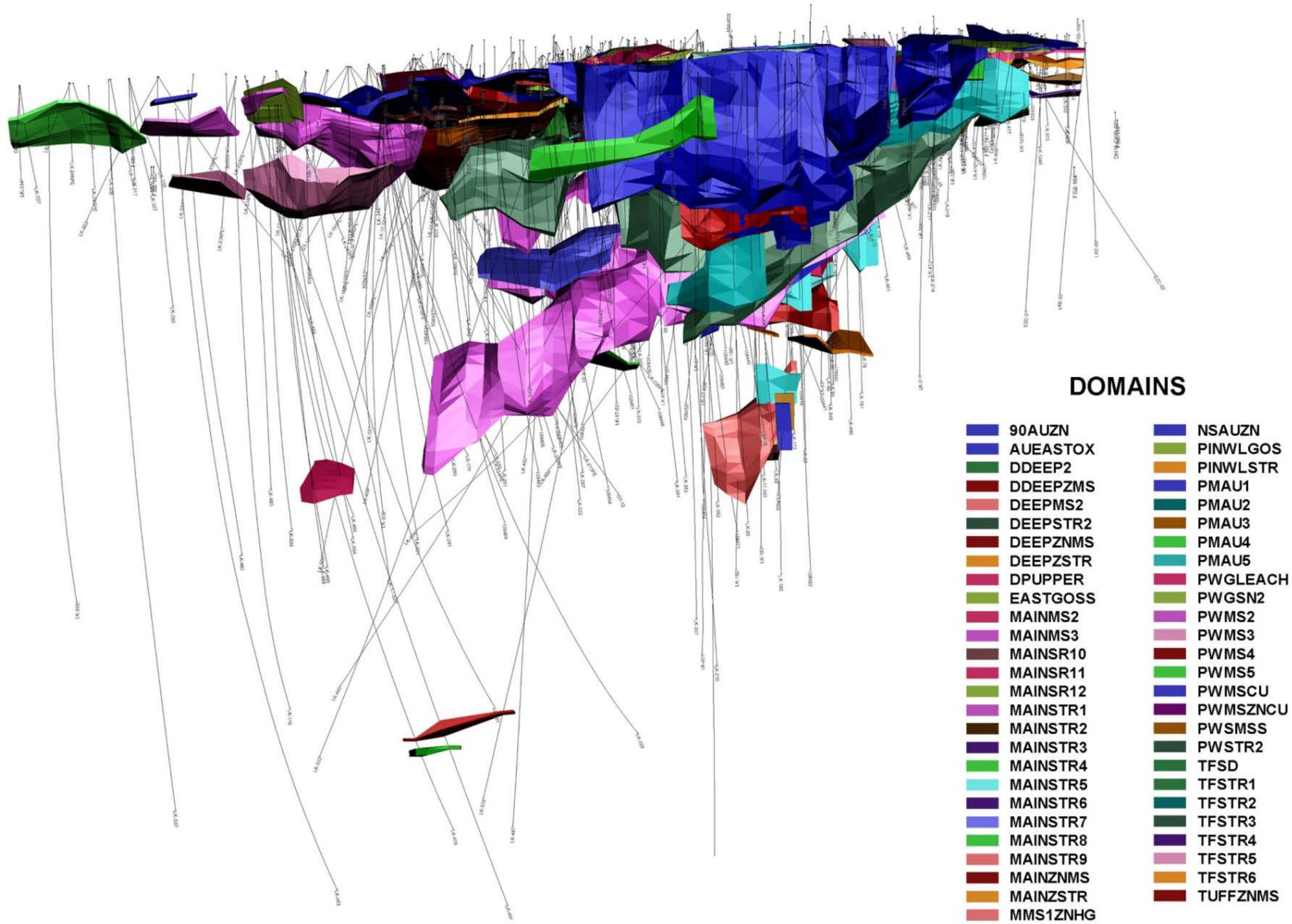
90AUZN
AUEASTOX
DDEEP2
DDEEPZMS
DEEPMS2
DEEPSTR2
DEEPSTR3
DEEPSTR4
DEEPSTR5
DEEPSTR6
DEEPZNMS
DEEPZSTR
DPUPPER
EASTGOSS
MAINMS2
MAINMS3
MAINSR10
MAINSR11
MAINSR12
MAINSTR0
MAINSTR1
MAINSTR2
MAINSTR3
MAINSTR4
MAINSTR5
MAINSTR6
MAINSTR7
MAINSTR8
MAINSTR9
MAINZNMS
MAINZSTR
NSAUZN
PINWHLMS
PINWLGOS
PINWLSTR
PMAU1
PMAU2
PMAU3
PMAU4
PMAU5
PWGLEACH
PWGSN2
PWMS2
PWMS3
PWMS4
PWMS5
PWMSCU
PWMSZNCU
PWSMSS
PWSTR2
TFSD
TFSTR1
TFSTR2
TFSTR3
TFSTR4
TFSTR5
TFSTR6
TUFFZNMS

APPENDIX A SURFACE DRILL HOLE PLAN

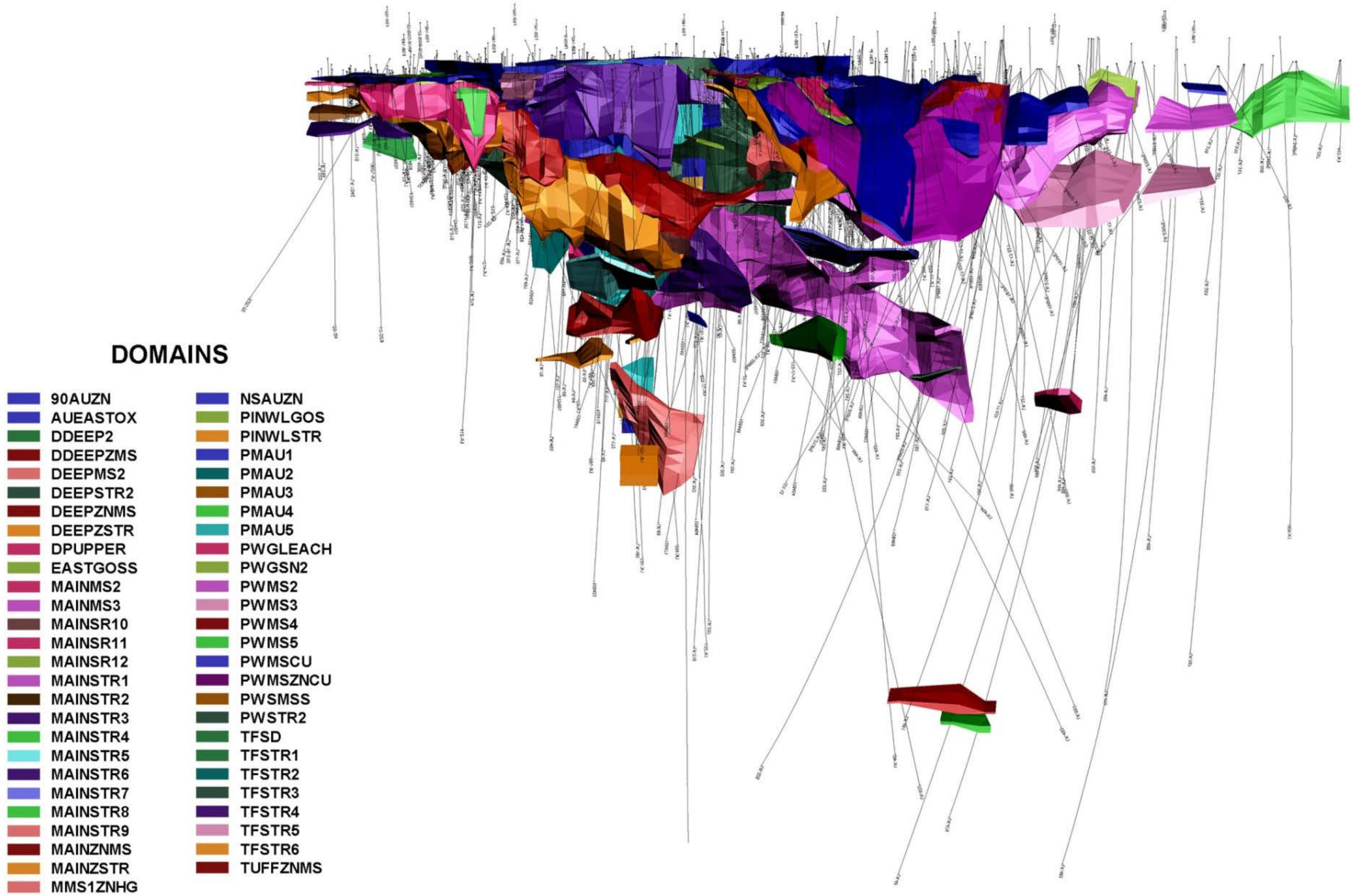


APPENDIX B 3-D DOMAINS

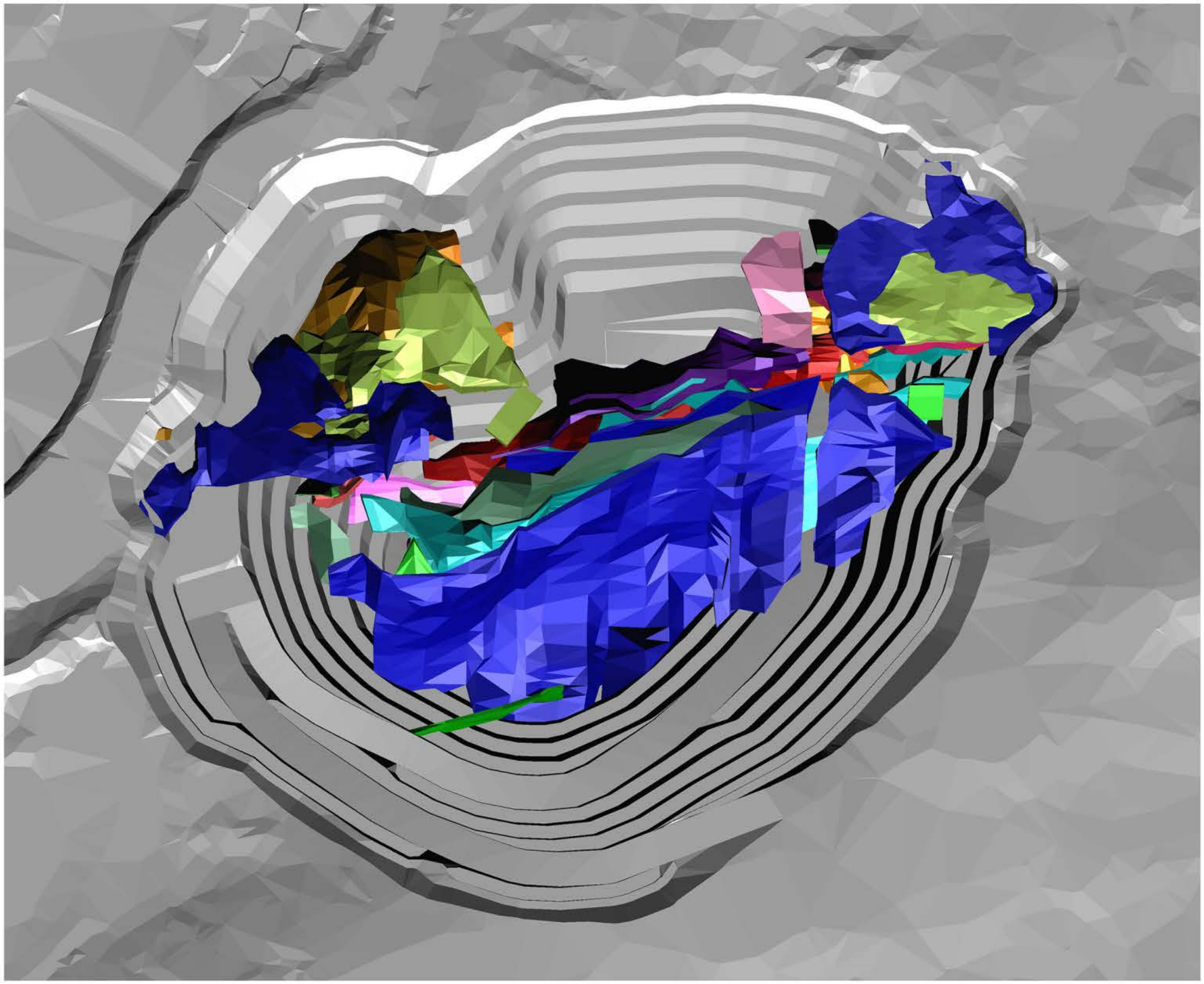
BACK FORTY PROJECT - OPEN PIT 3D DOMAINS
LOOKING NORTH



BACK FORTY PROJECT - OPEN PIT 3D DOMAINS
LOOKING SOUTH



BACK FORTY PROJECT
3D PIT DESIGN

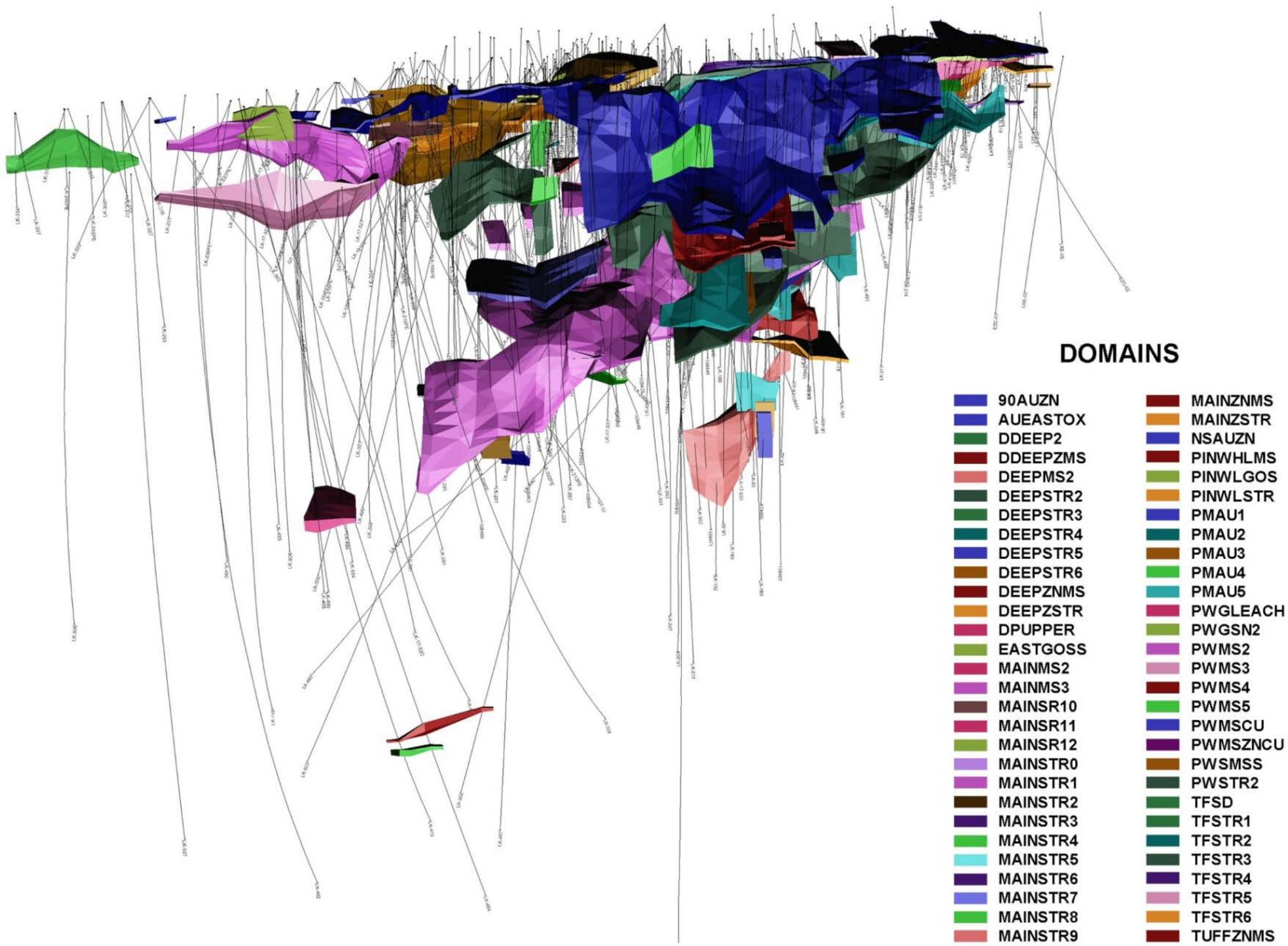


P&E Mining Con
Aquila Resources

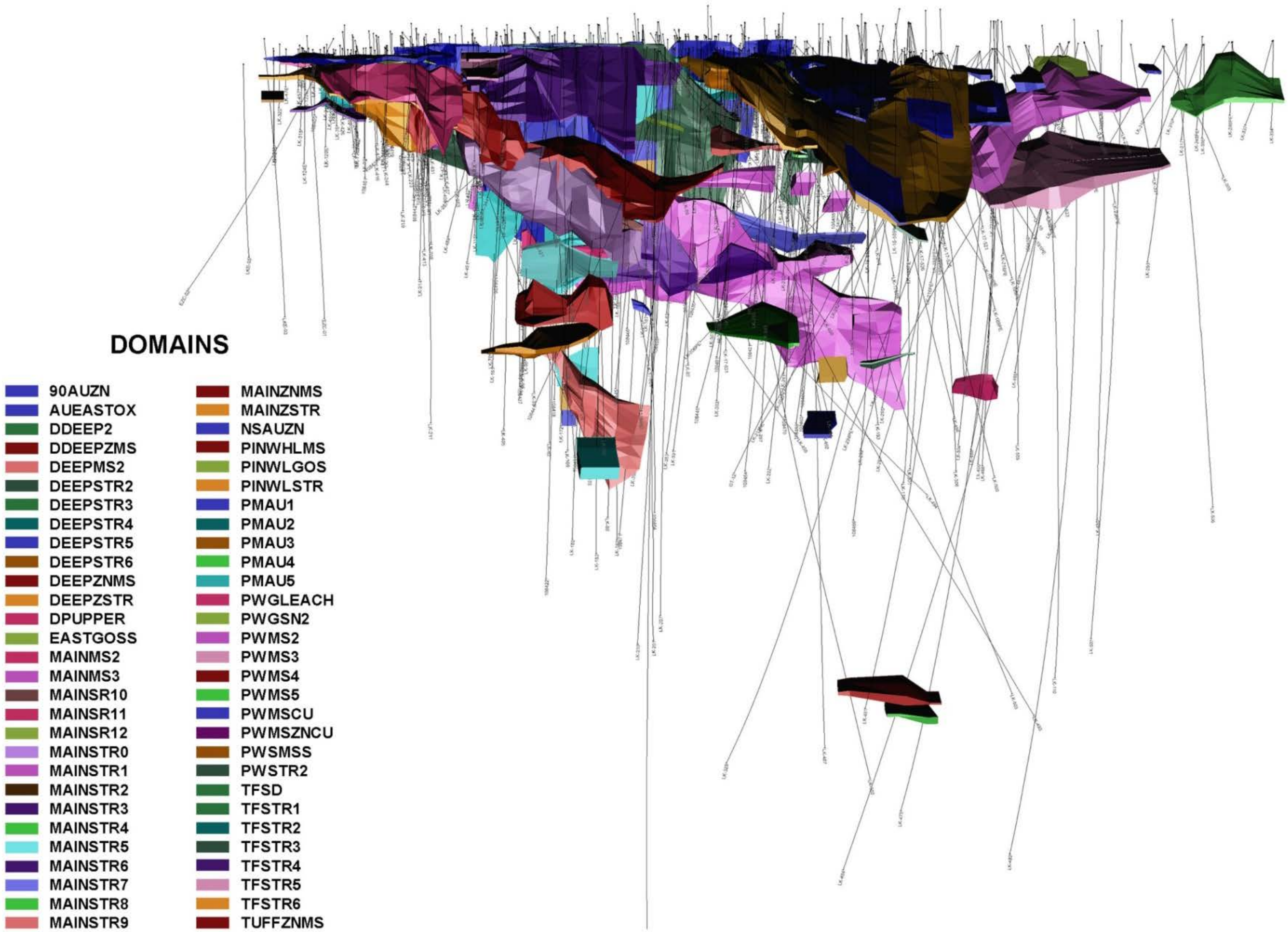
DOMAINS

90AUZN MAINMS3 MAINSTR8 PMAU4 PWSTR2

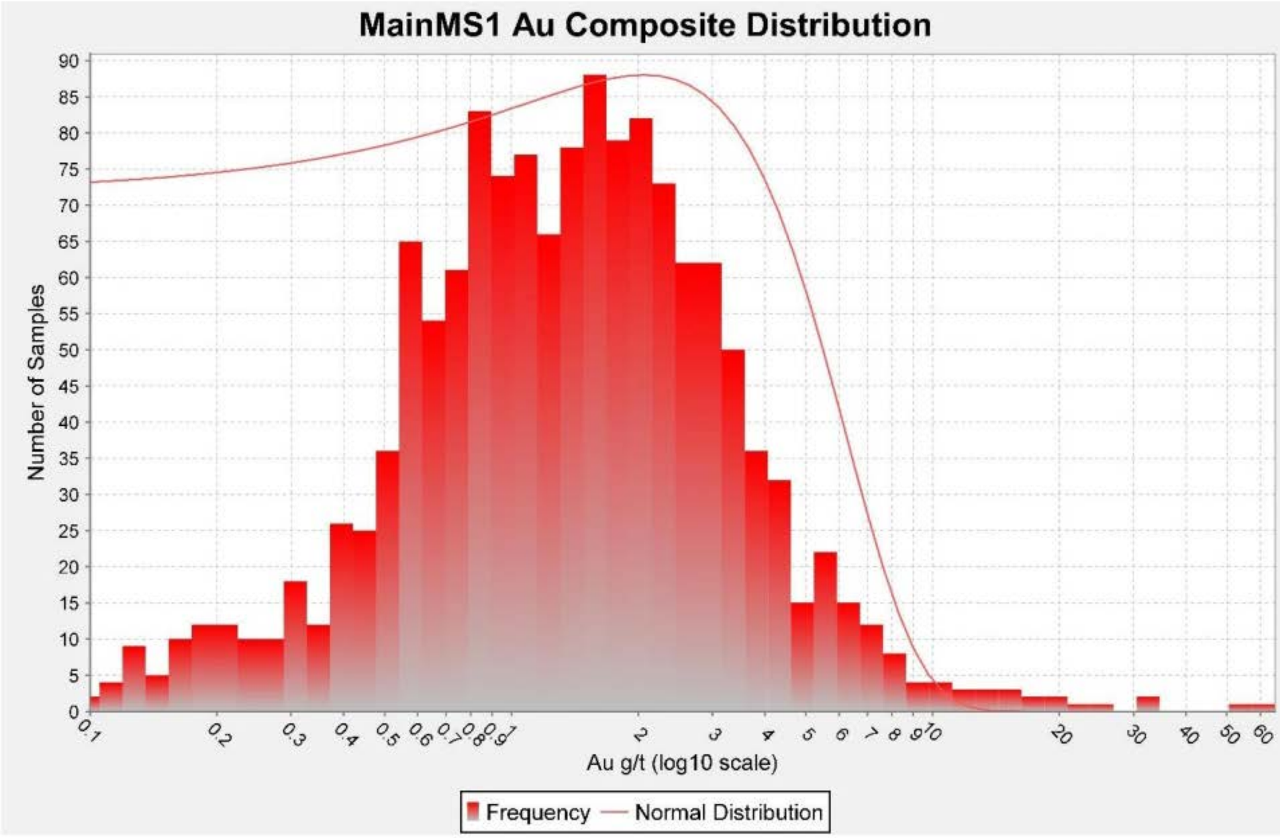
BACK FORTY PROJECT - UNDERGROUND 3D DOMAINS
LOOKING NORTH



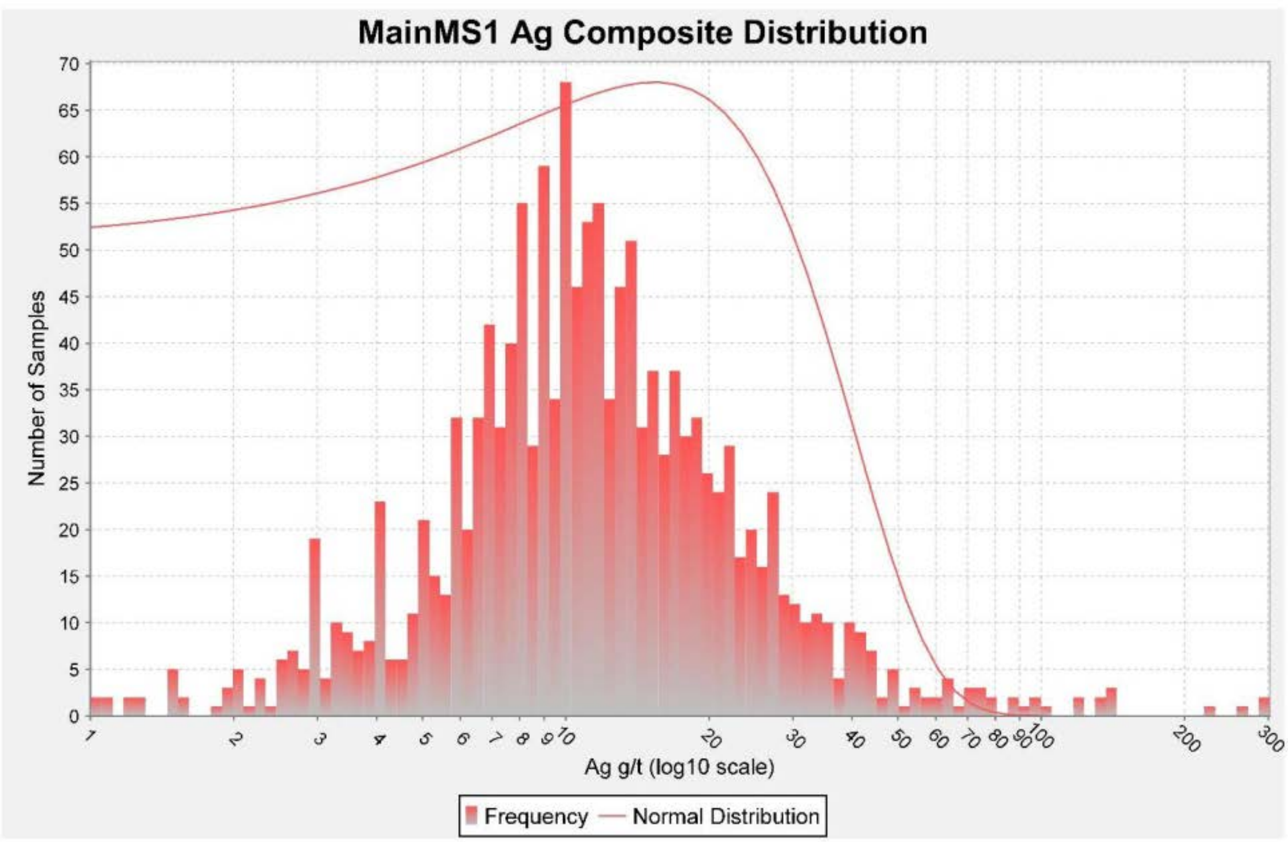
BACK FORTY PROJECT - UNDERGROUND 3D DOMAINS
LOOKING SOUTH



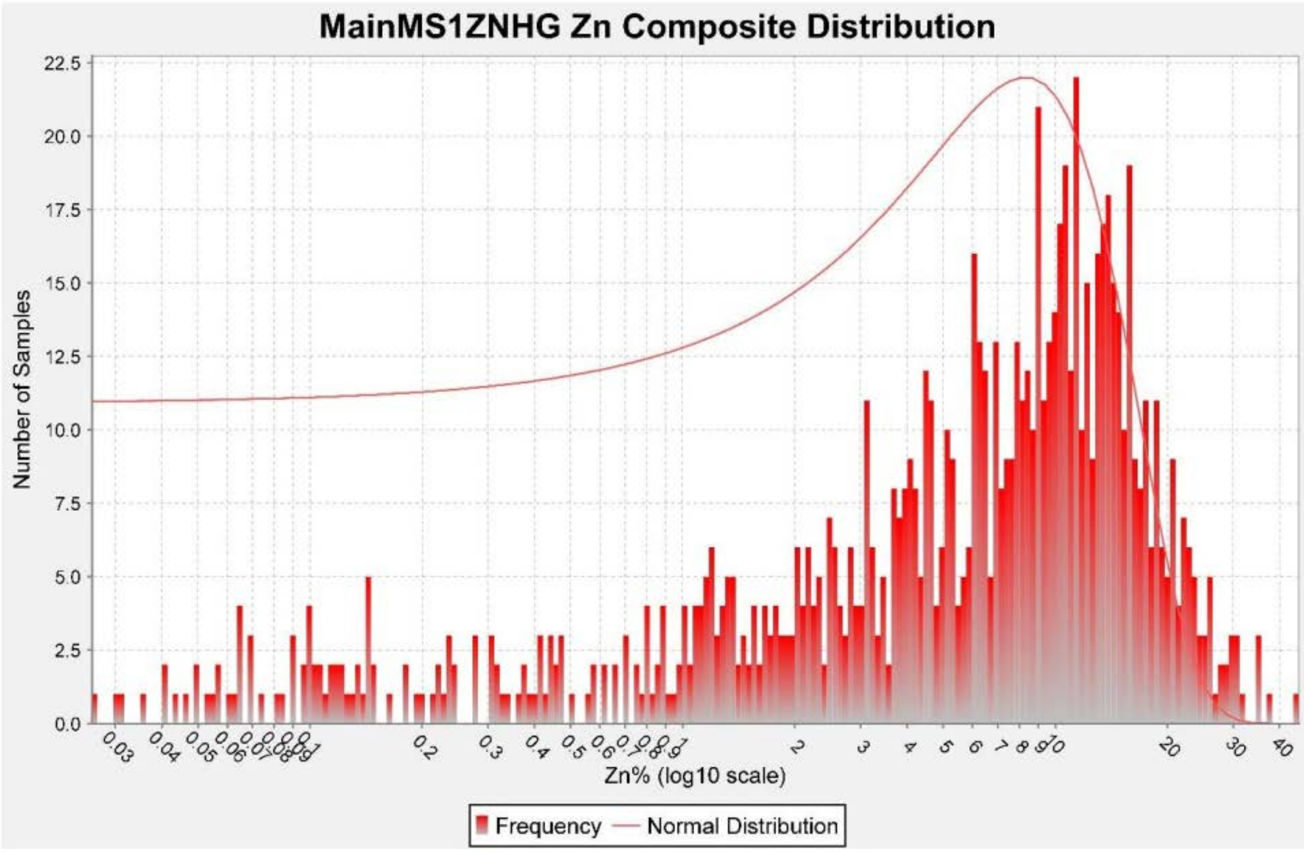
APPENDIX C LOG NORMAL HISTOGRAMS OF OPEN PIT MODEL



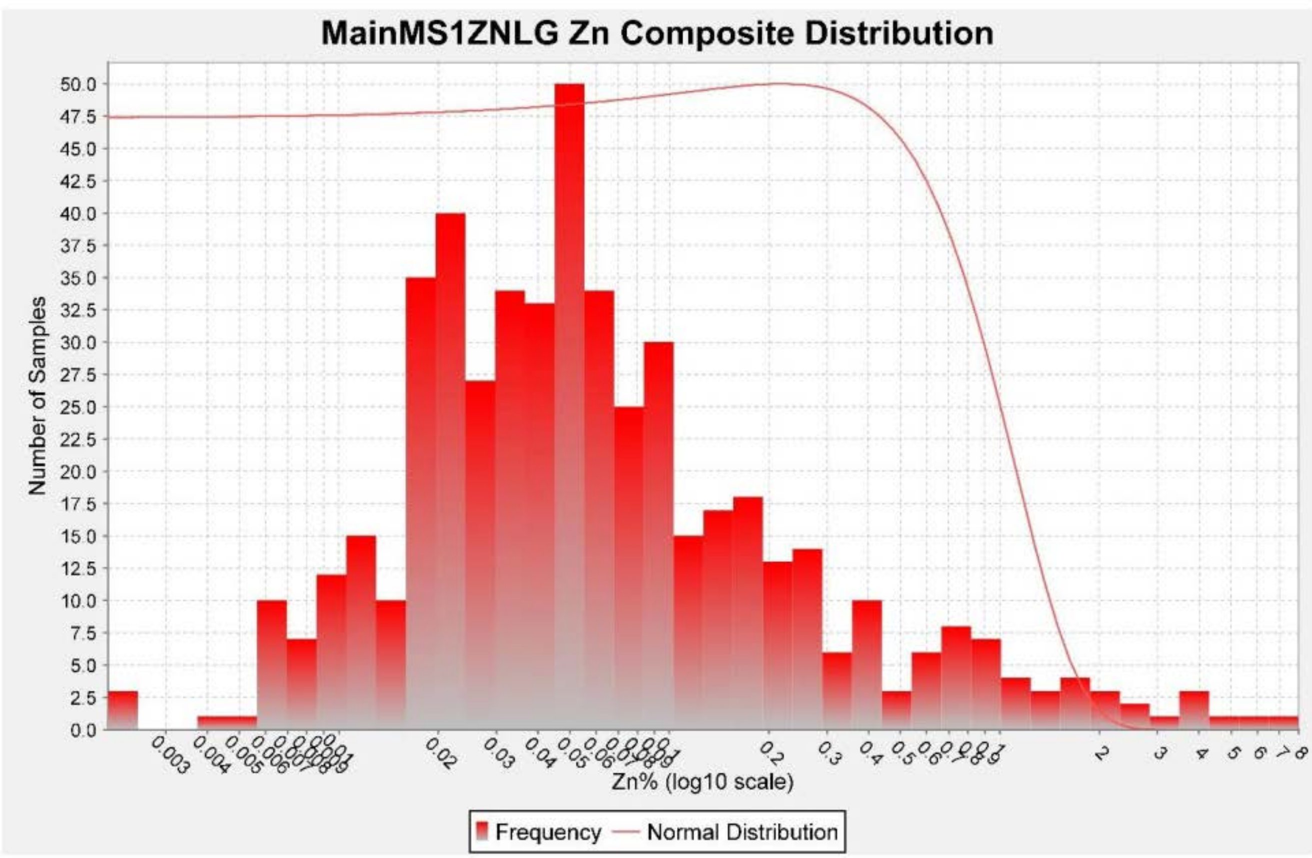
MainMS1 Au Composite Distribution
Number of Samples
0
5
10
15
20
25
30
35
40
45
50
55
60
65
70
75
80
85
90
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1
2
3
4
5
6
7
8
9
10
20
30
40
50
60
Au g/t (log10 scale)
Frequency
Normal Distribution



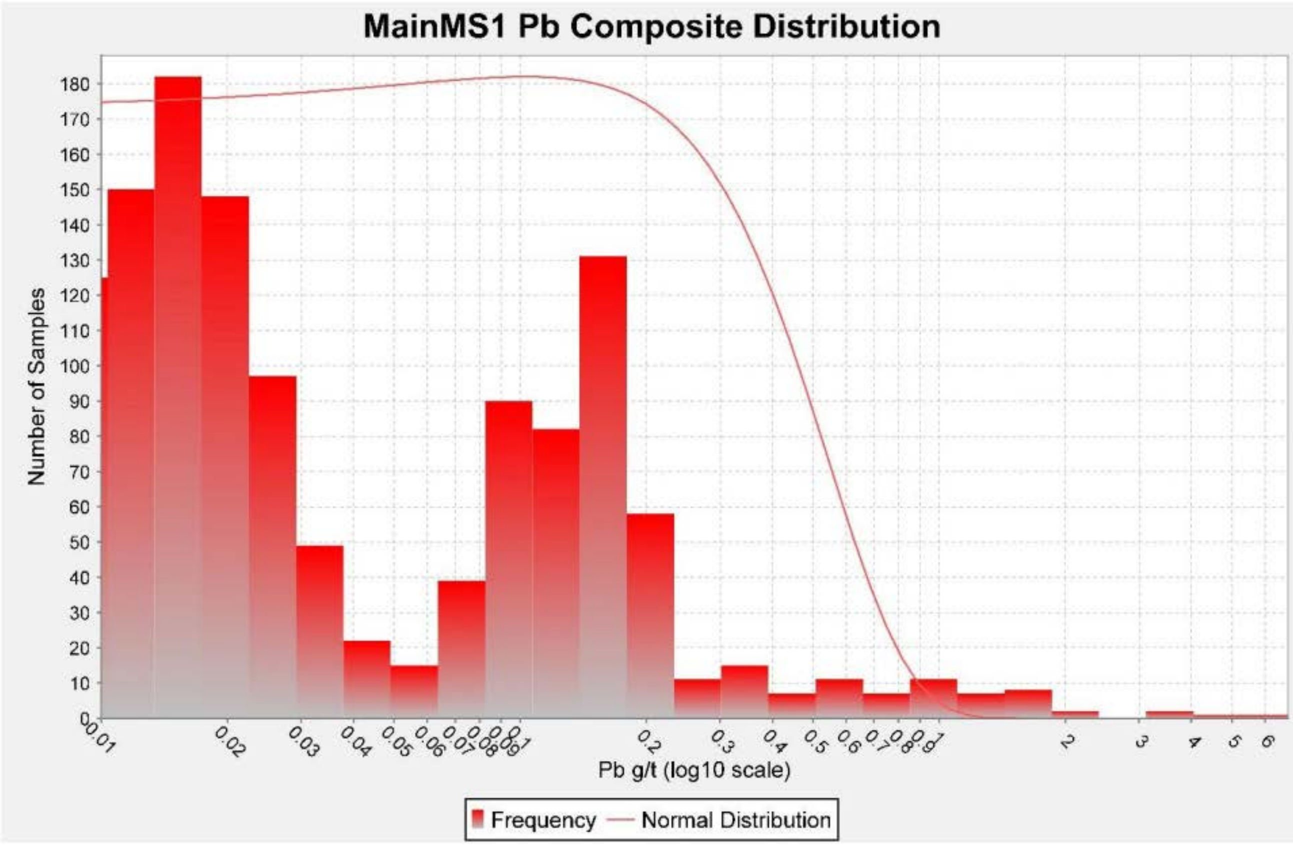
MainMS1 Ag Composite Distribution
Number of Samples
0
5
10
15
20
25
30
35
40
45
50
55
60
65
70
1
2
3
4
5
6
7
8
9
10
20
30
40
50
60
70
80
90
100
200
300
Ag g/t (log10 scale)
Frequency
Normal Distribution

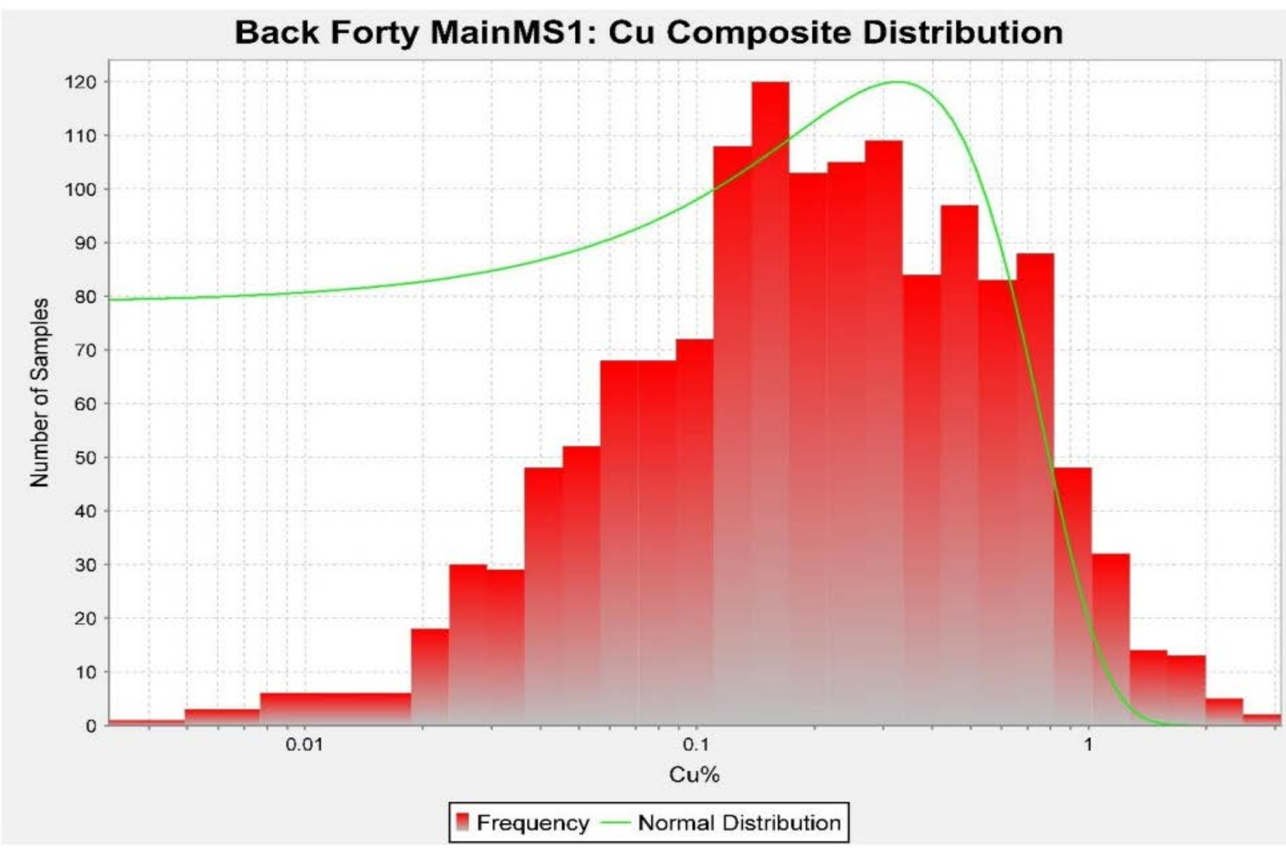


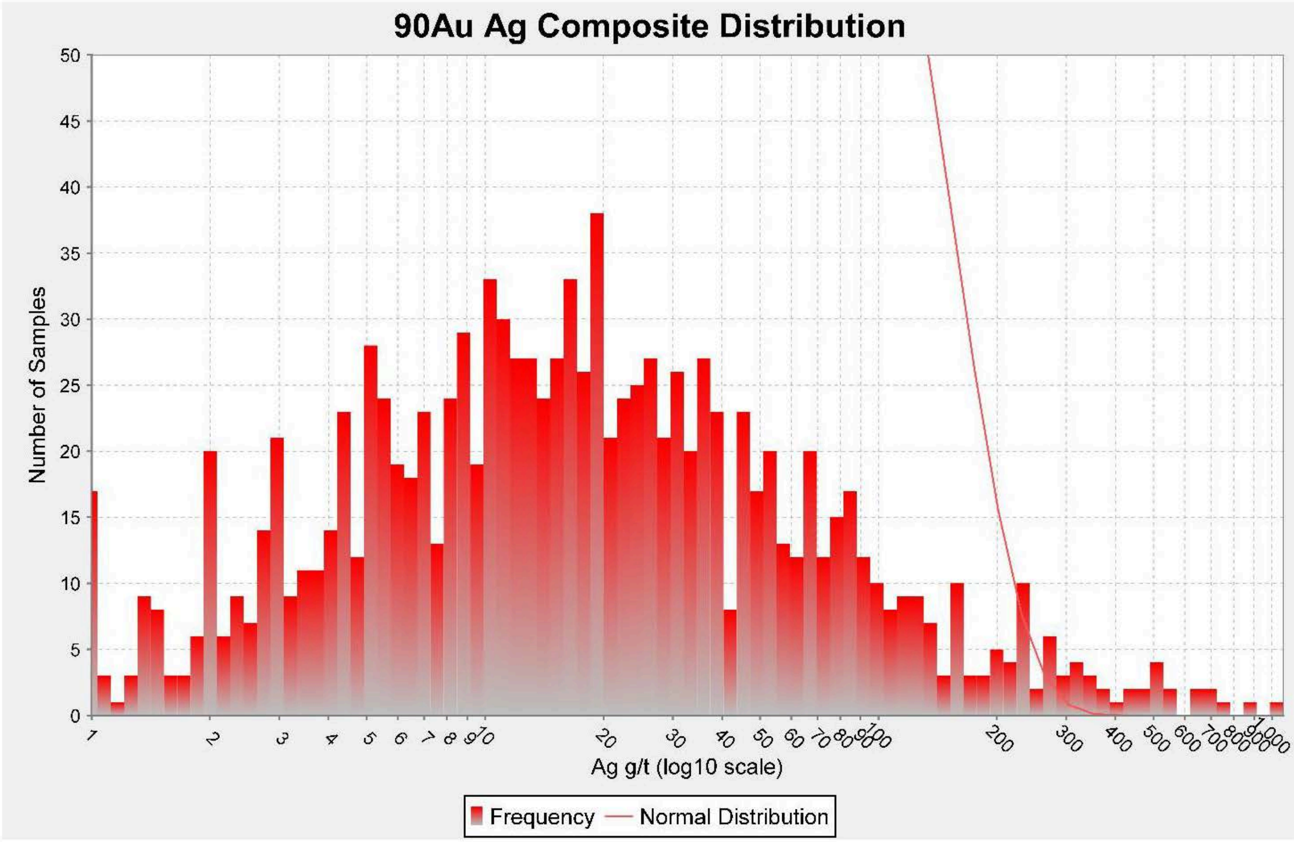
MainMS1ZNHG Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution



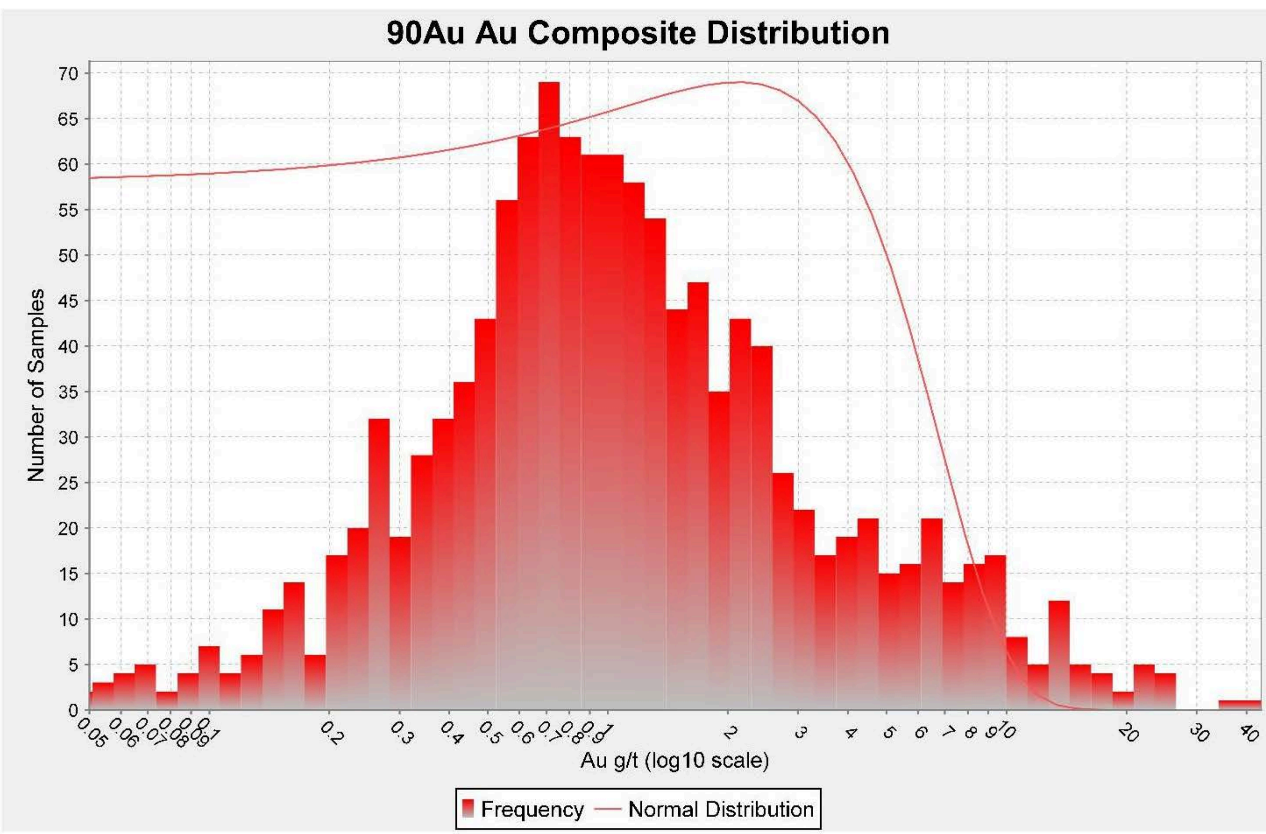
MainMS1ZNLG Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution


MainMS1 Pb Composite Distribution
Number of Samples
Pb g/t (log10 scale)
Frequency
Normal Distribution

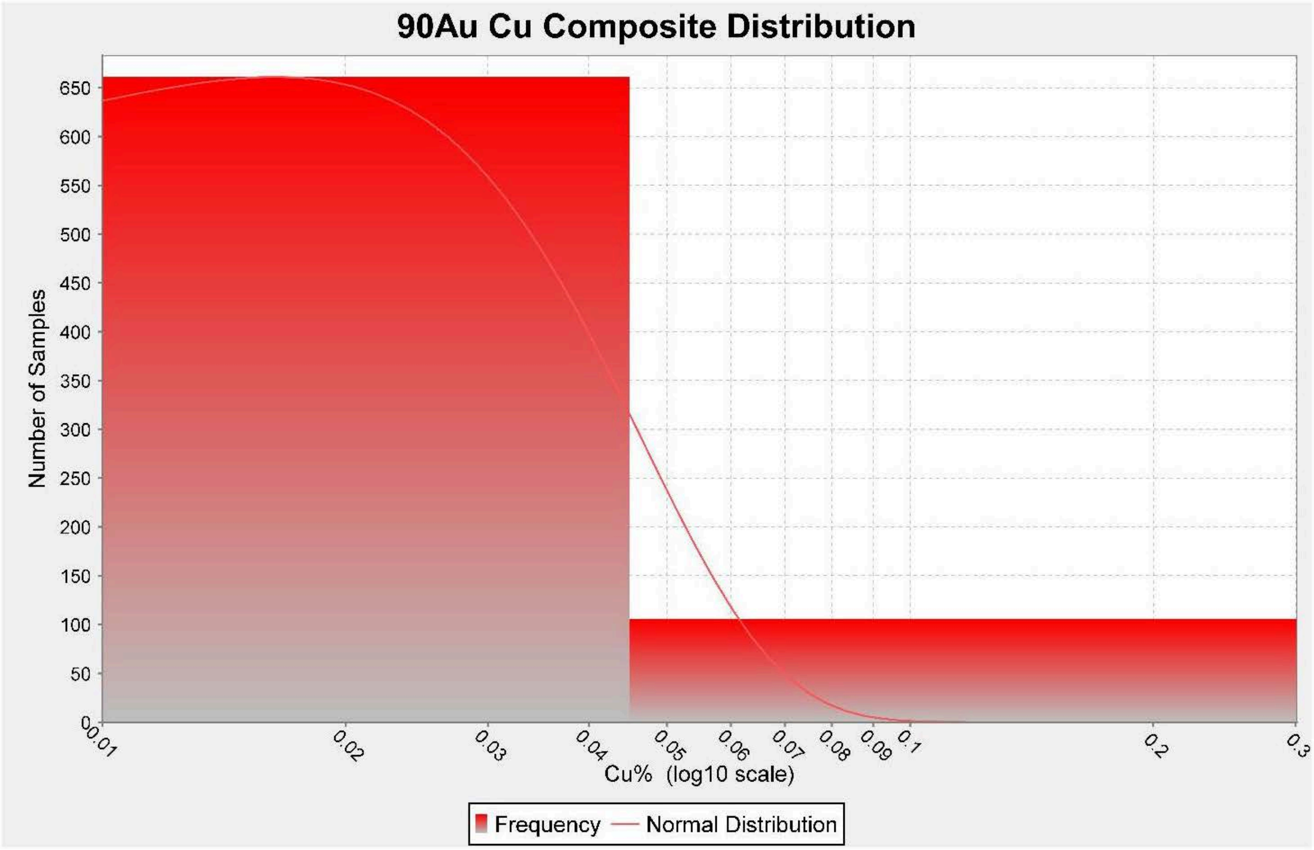

Back Forty MainMS1: Cu Composite Distribution
Number of Samples
Cu%
Frequency
Normal Distribution

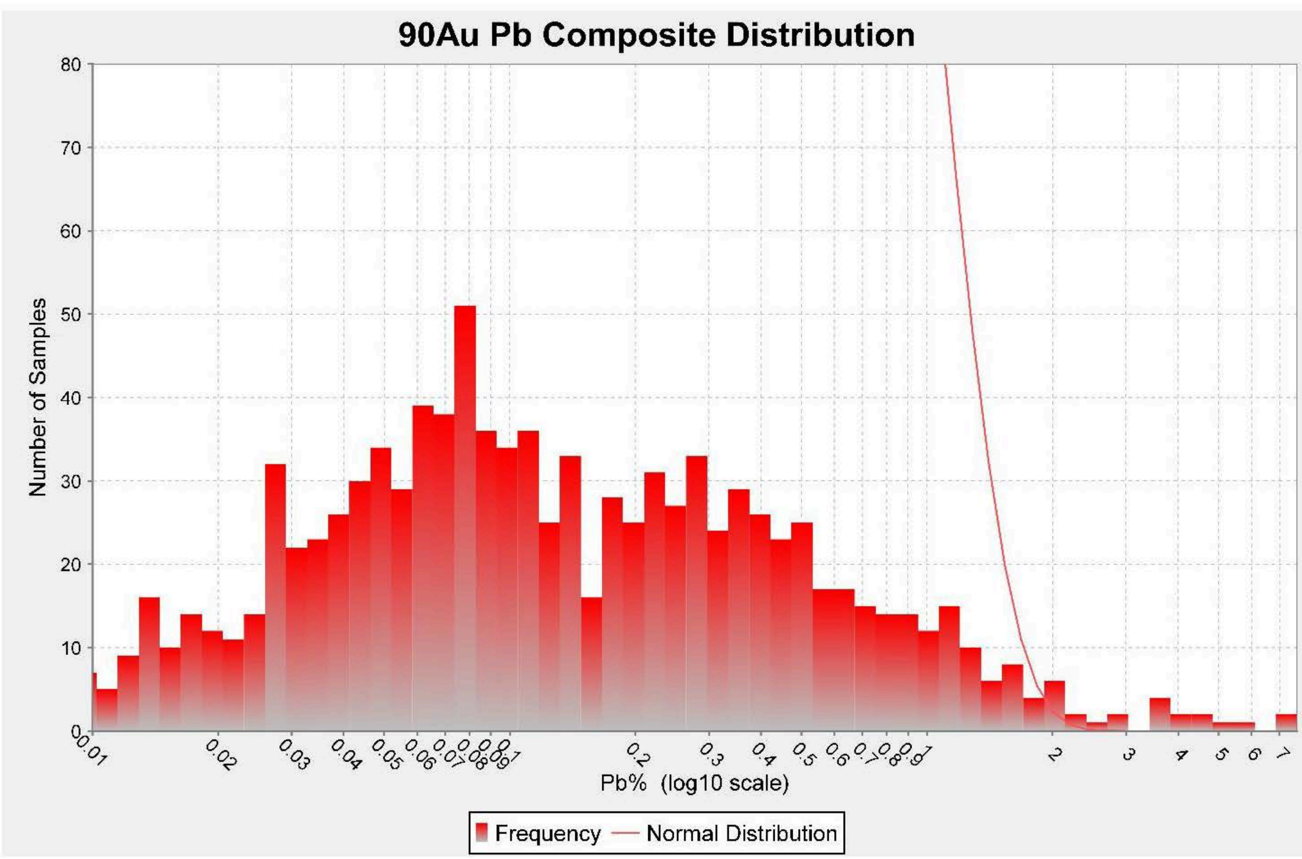


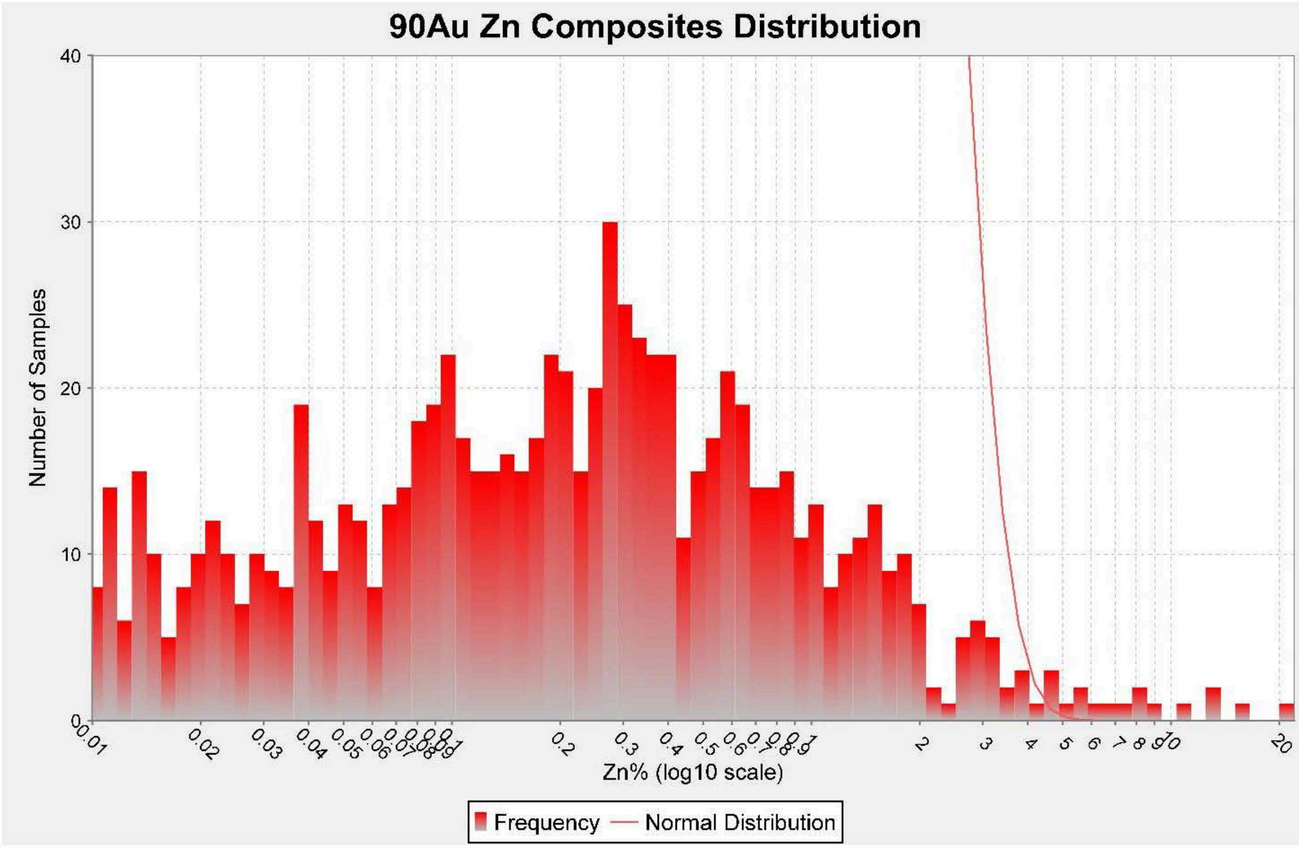
90Au Ag Composite Distribution
Number of Samples
Ag g/t (log10 scale)
Frequency
Normal Distribution

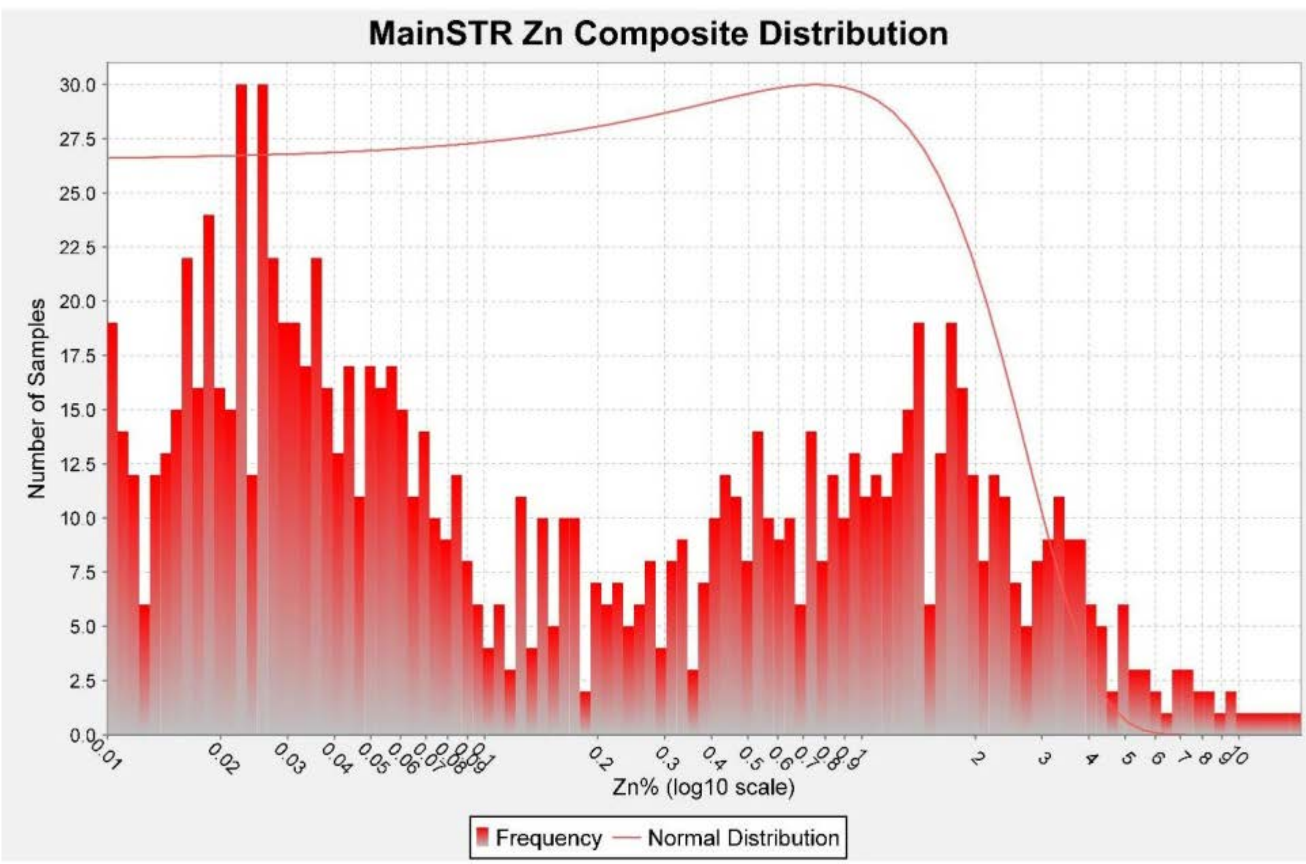


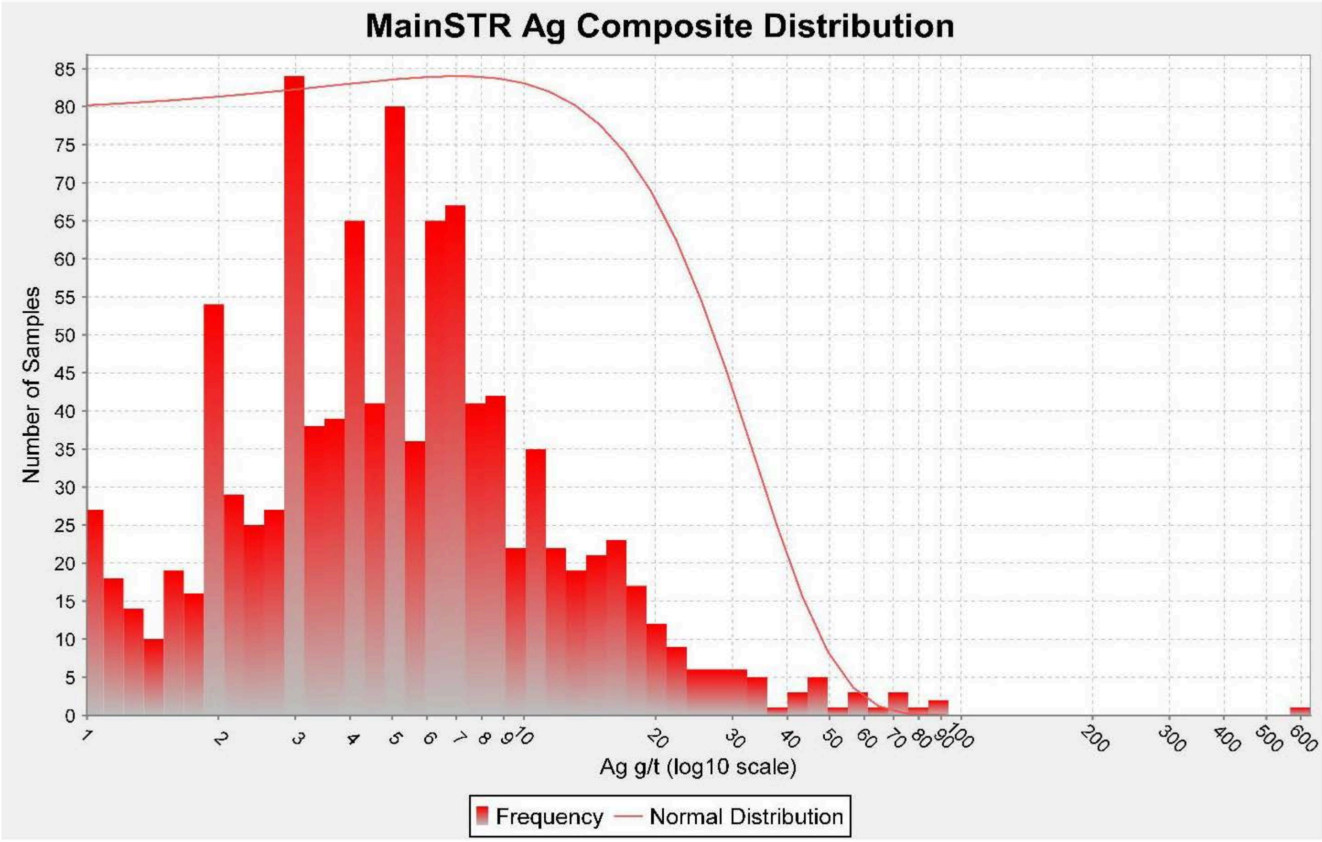
90Au Au Composite Distribution
Number of Samples
Au g/t (log10 scale)
Frequency
Normal Distribution

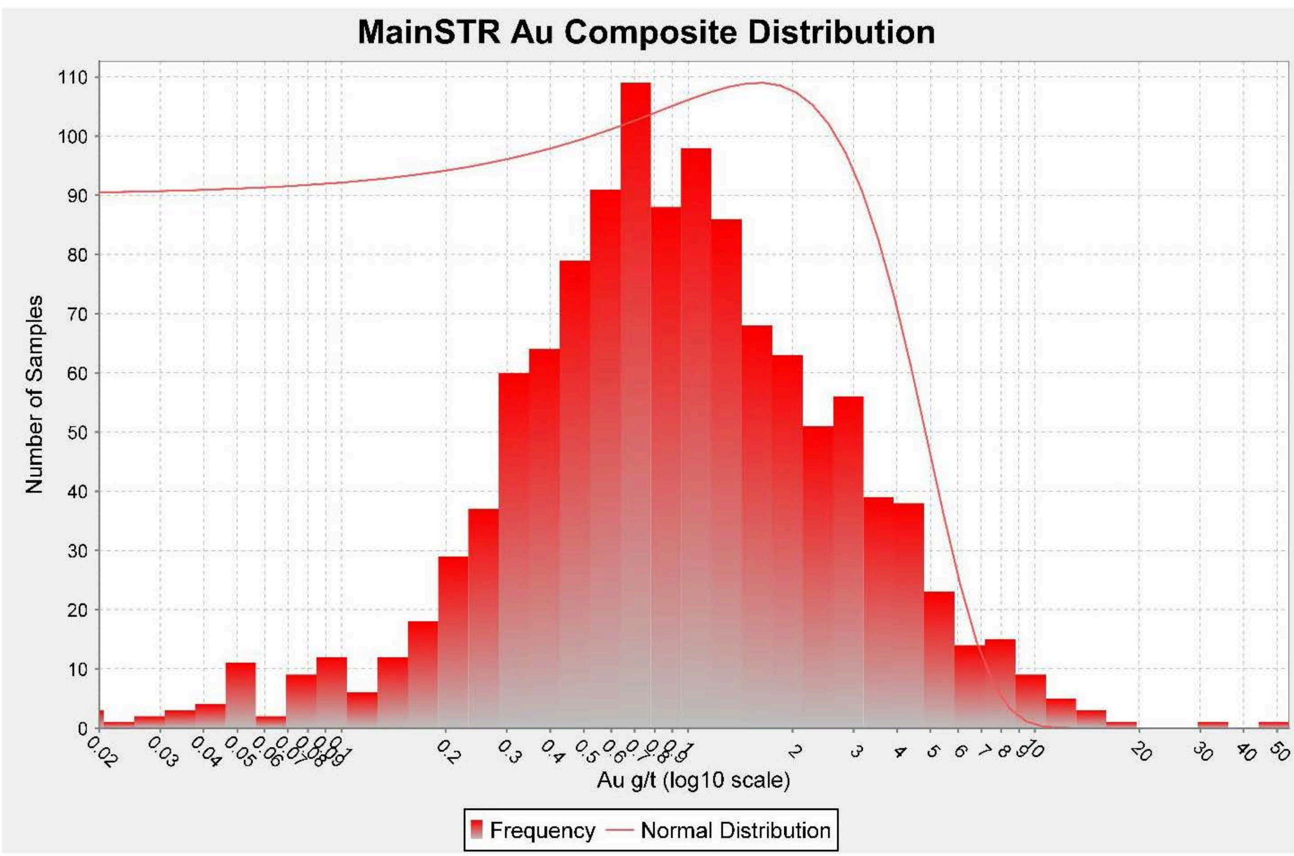

90Au Cu Composite Distribution
Number of Samples
Cu% (log10 scale)
Frequency
Normal Distribution

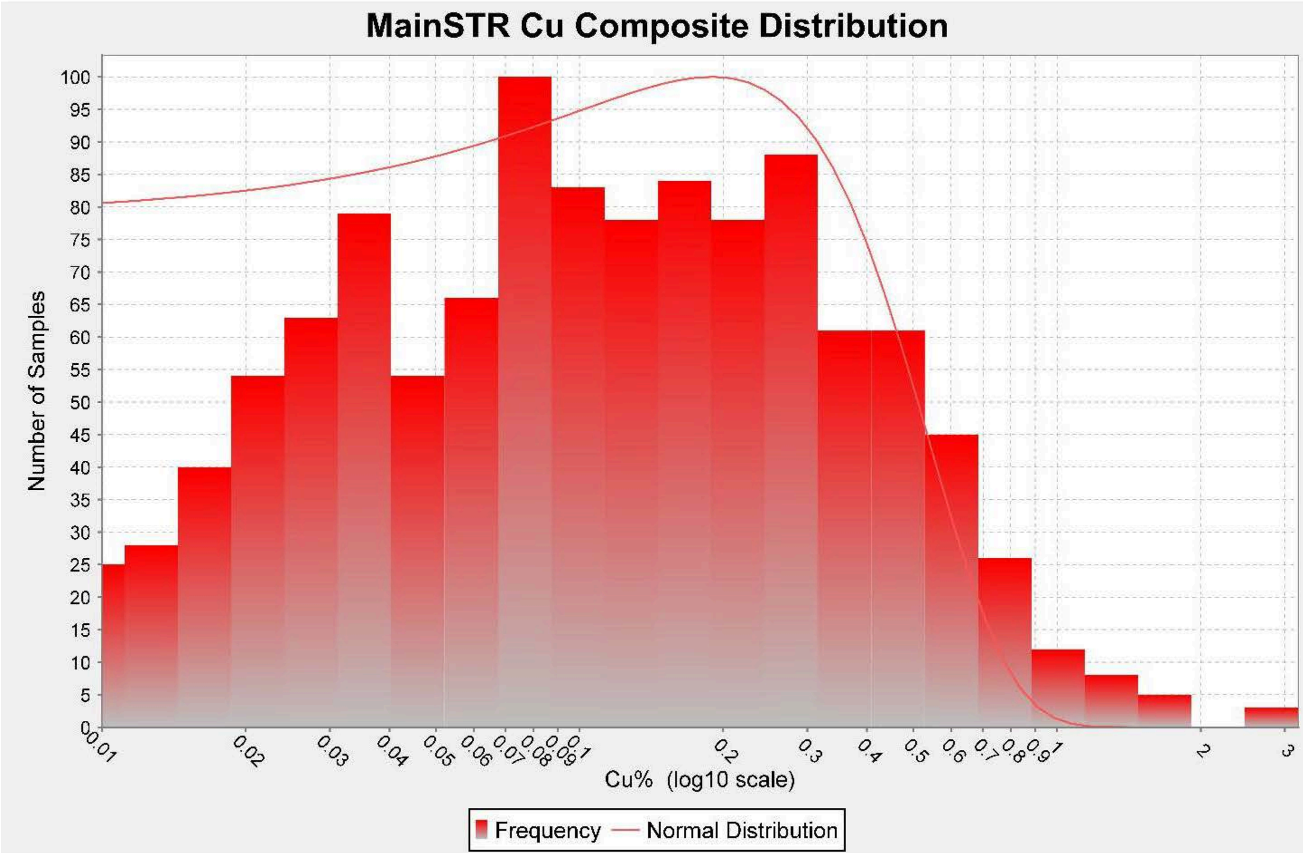

90Au Pb Composite Distribution
Number of Samples
Pb% (log10 scale)
Frequency
Normal Distribution

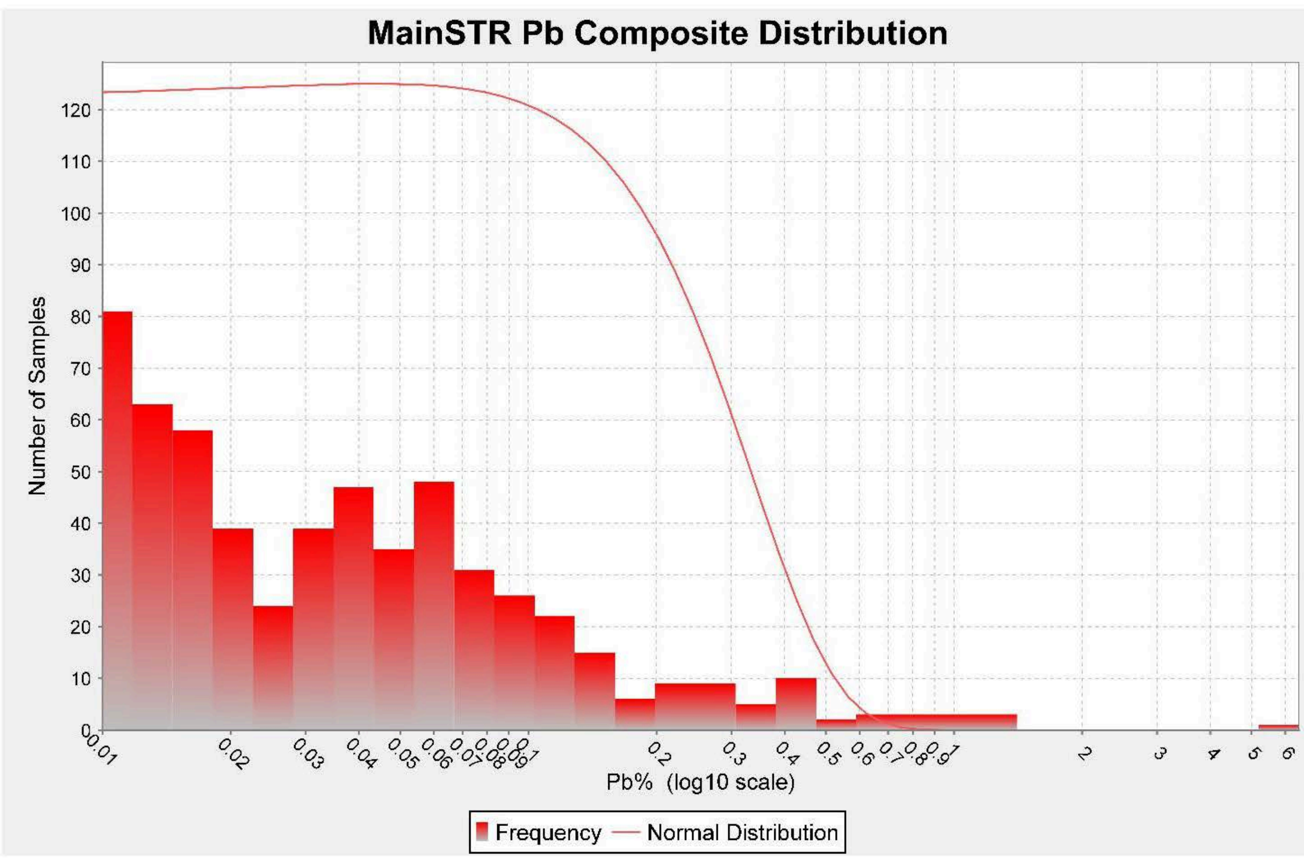

90Au Zn Composites Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution


MainSTR Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution

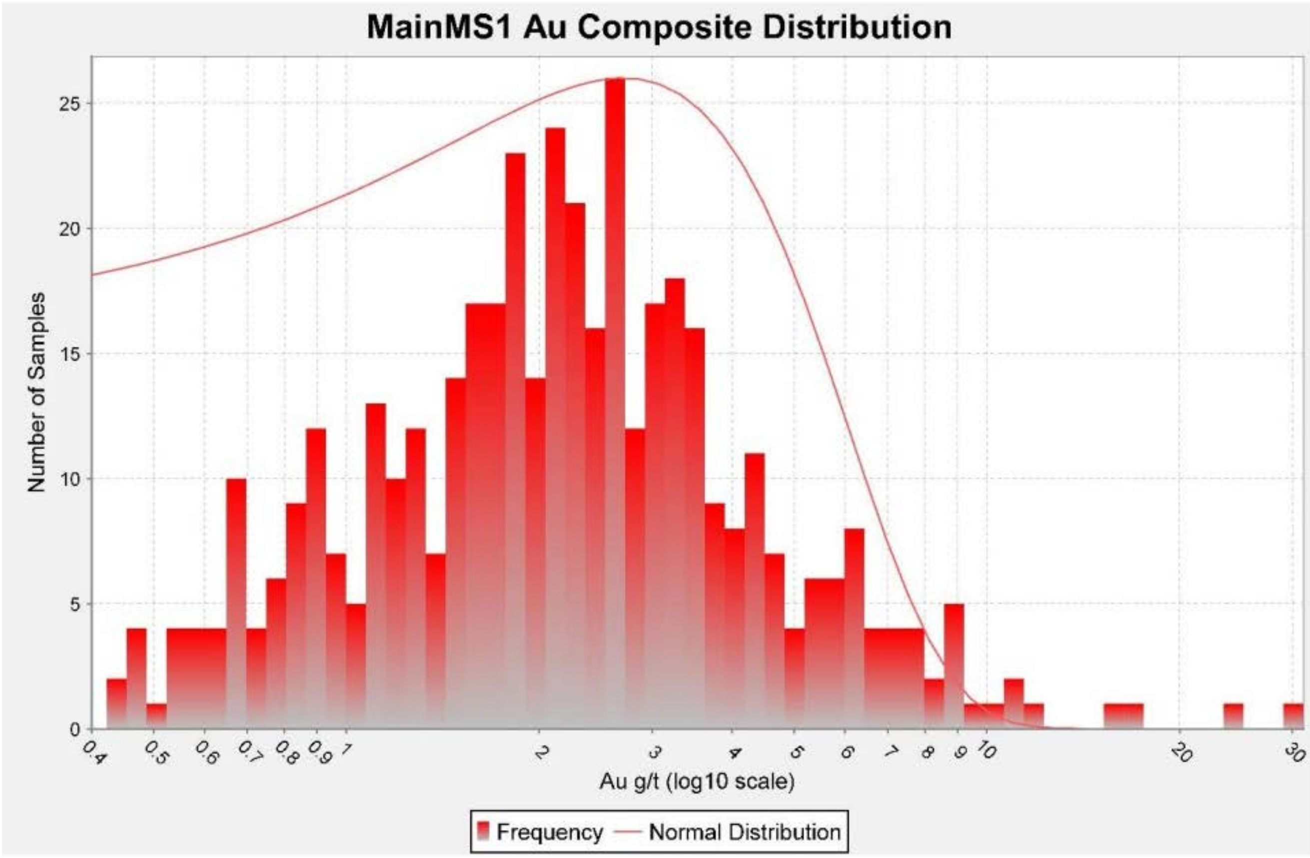

MainSTR Ag Composite Distribution
Number of Samples
Ag g/t (log10 scale)
Frequency
Normal Distribution

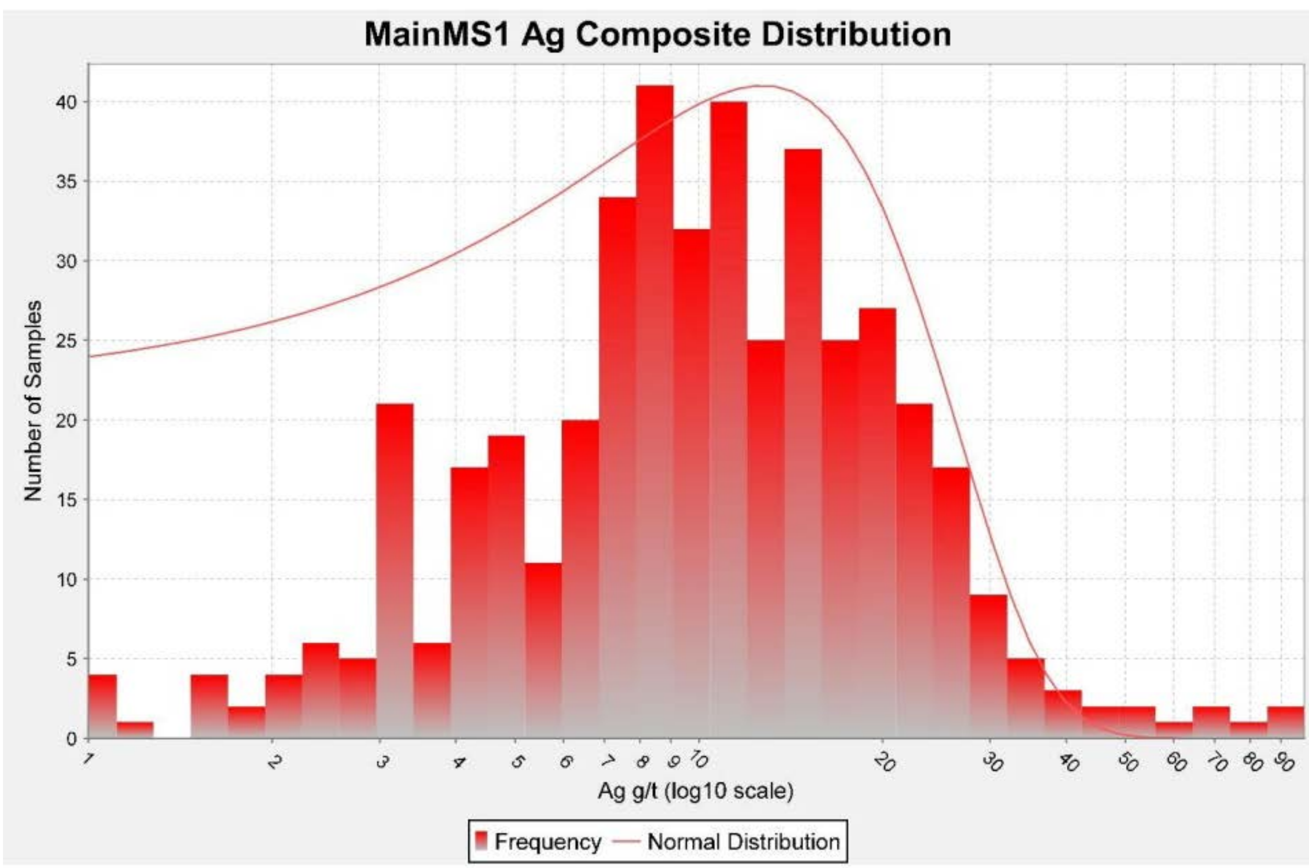

MainSTR Au Composite Distribution
Number of Samples
Au g/t (log10 scale)
Frequency
Normal Distribution

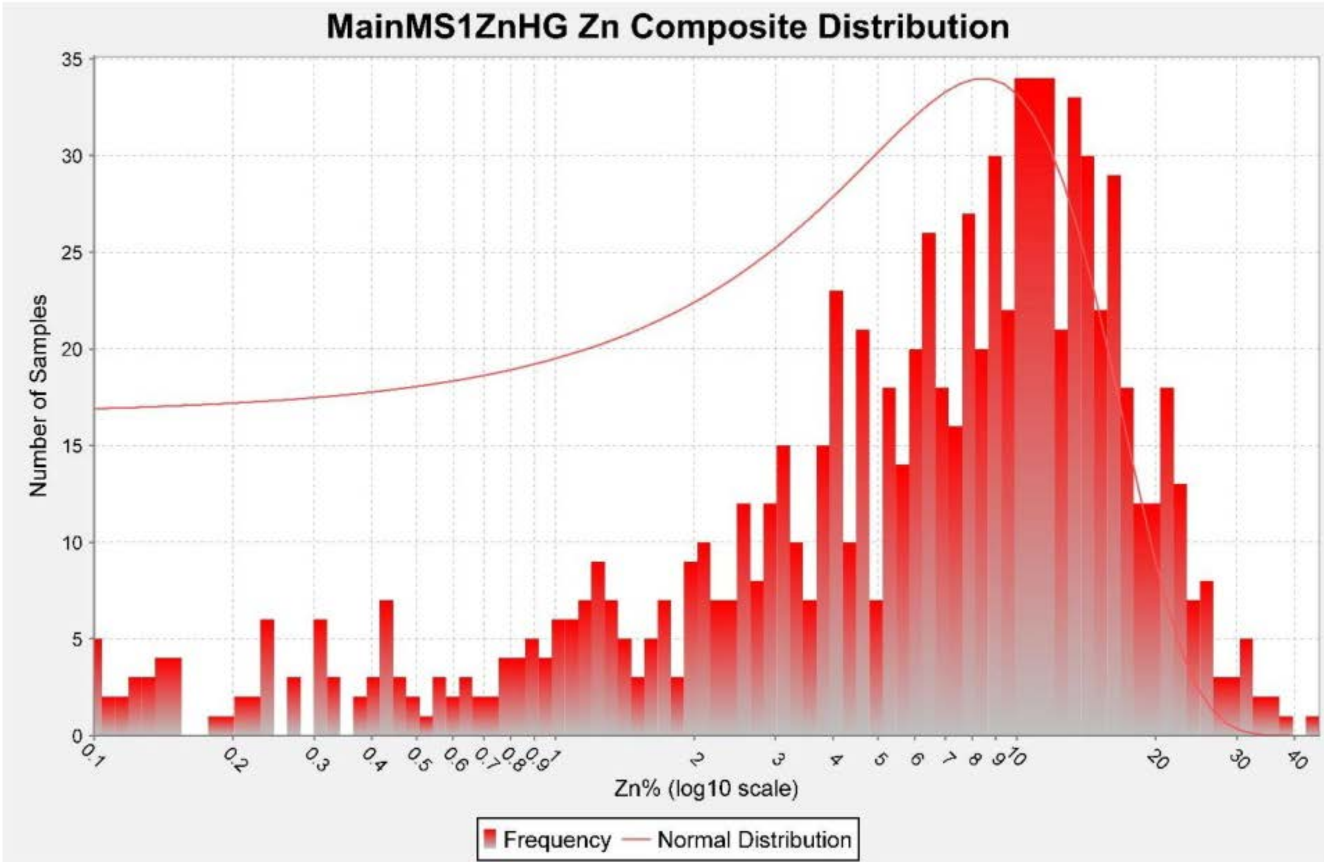

MainSTR Cu Composite Distribution
Number of Samples
Cu% (log10 scale)
Frequency
Normal Distribution

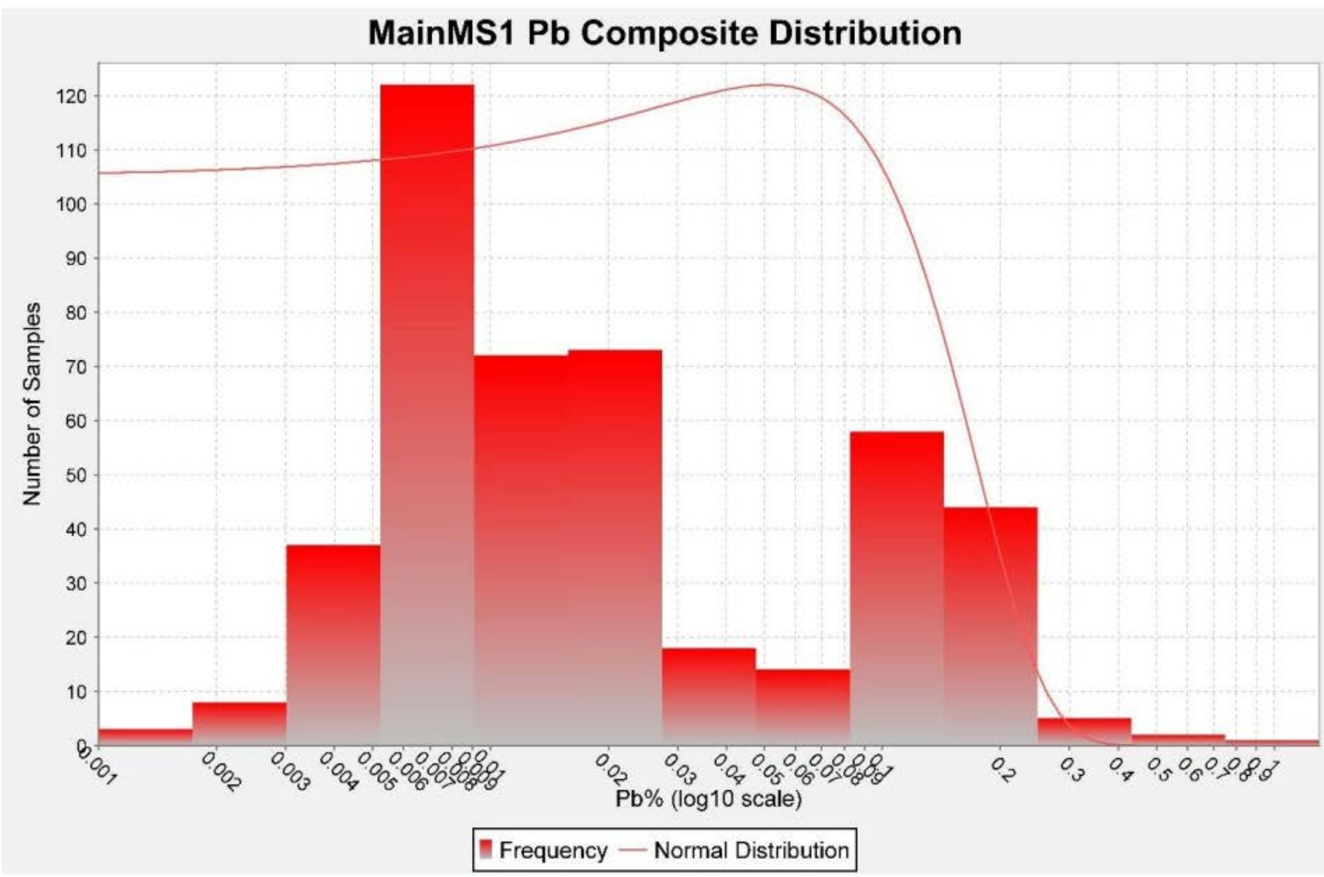

MainSTR Pb Composite Distribution
Number of Samples
Pb% (log10 scale)
Frequency
Normal Distribution

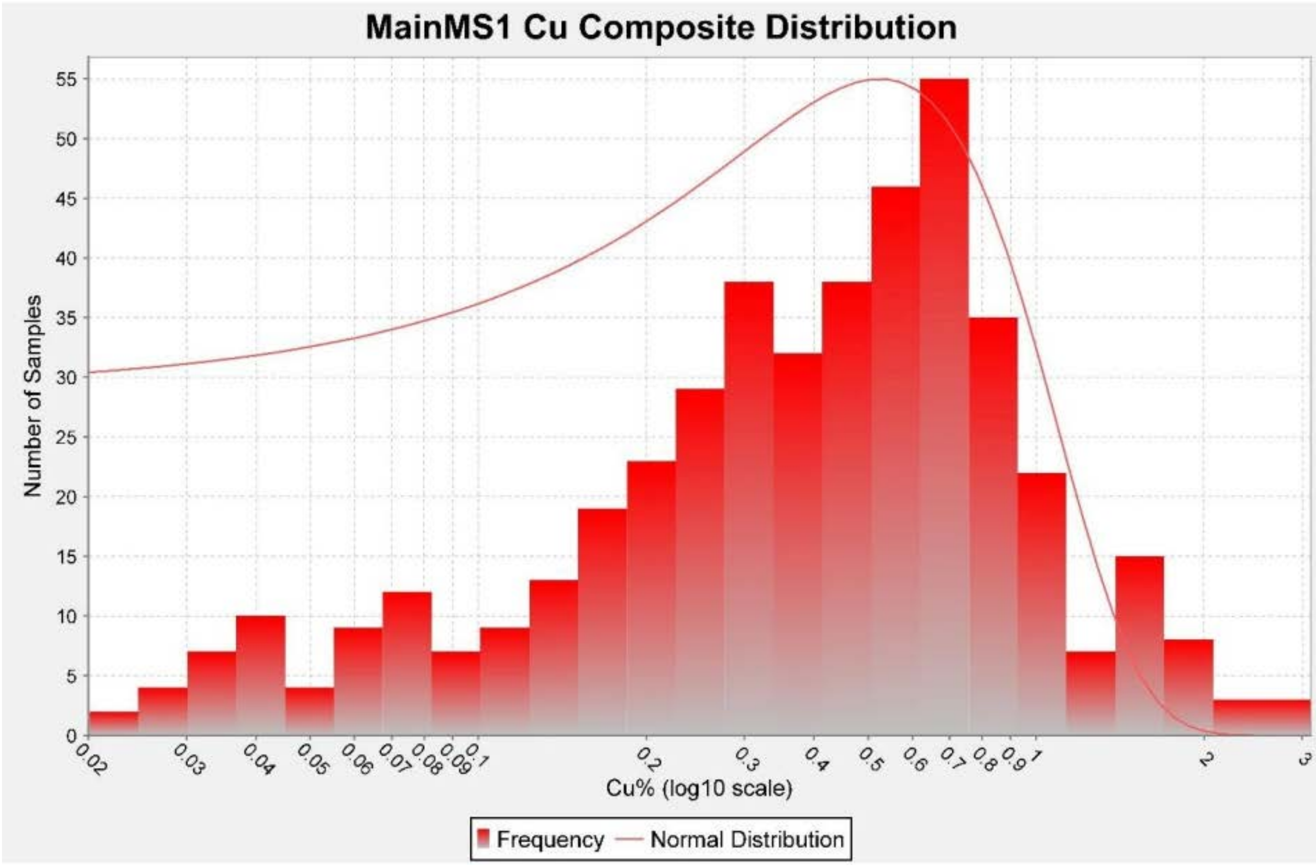
APPENDIX D LOG NORMAL HISTOGRAMS OF UNDERGROUND MODEL


MainMS1 Au Composite Distribution
Number of Samples
Au g/t (log10 scale)
Frequency
Normal Distribution

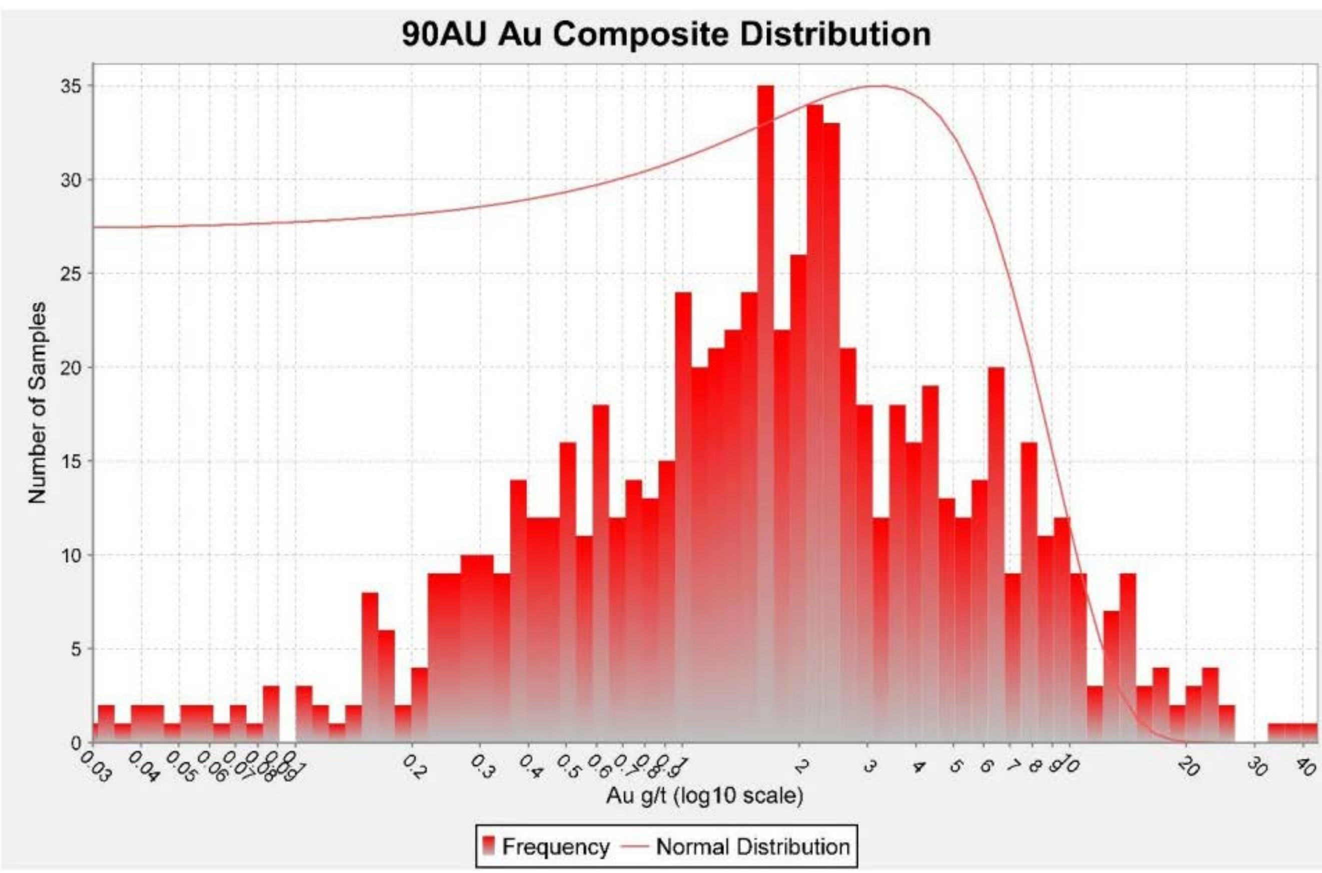

MainMS1 Ag Composite Distribution
Number of Samples
Ag g/t (log10 scale)
Frequency
Normal Distribution


MainMS1ZnHG Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution

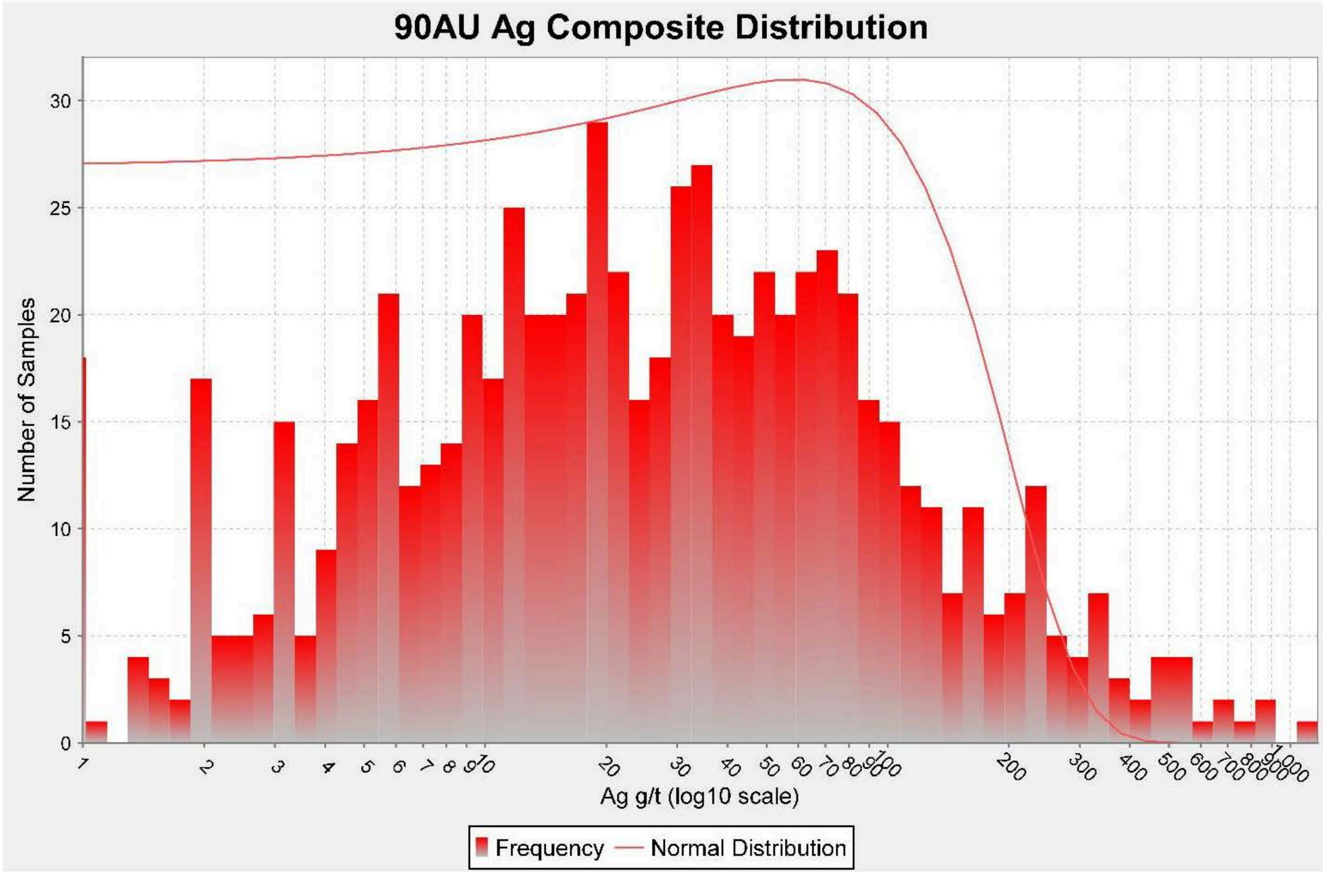

MainMS1 Pb Composite Distribution
Number of Samples
Pb% (log10 scale)
Frequency
Normal Distribution

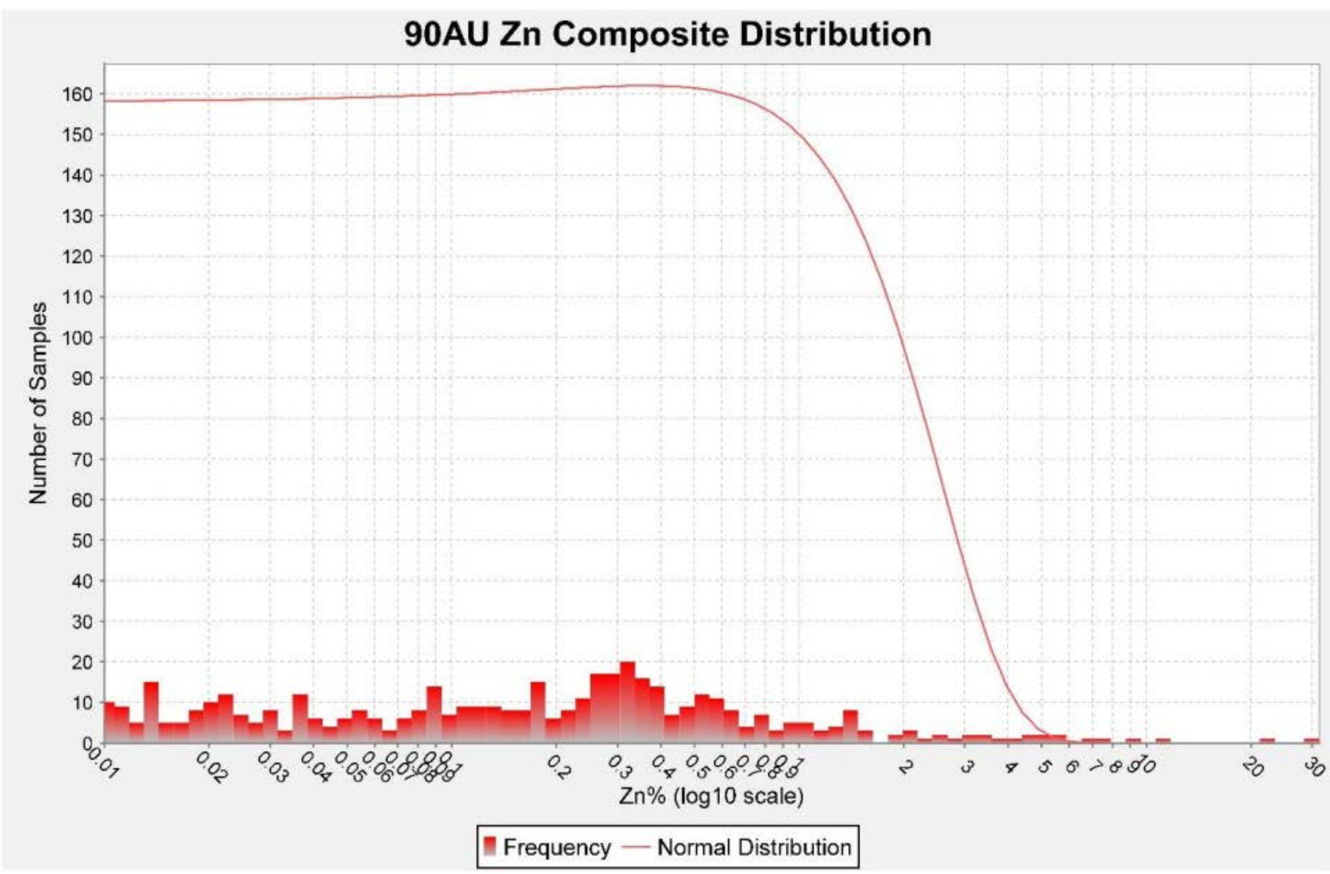


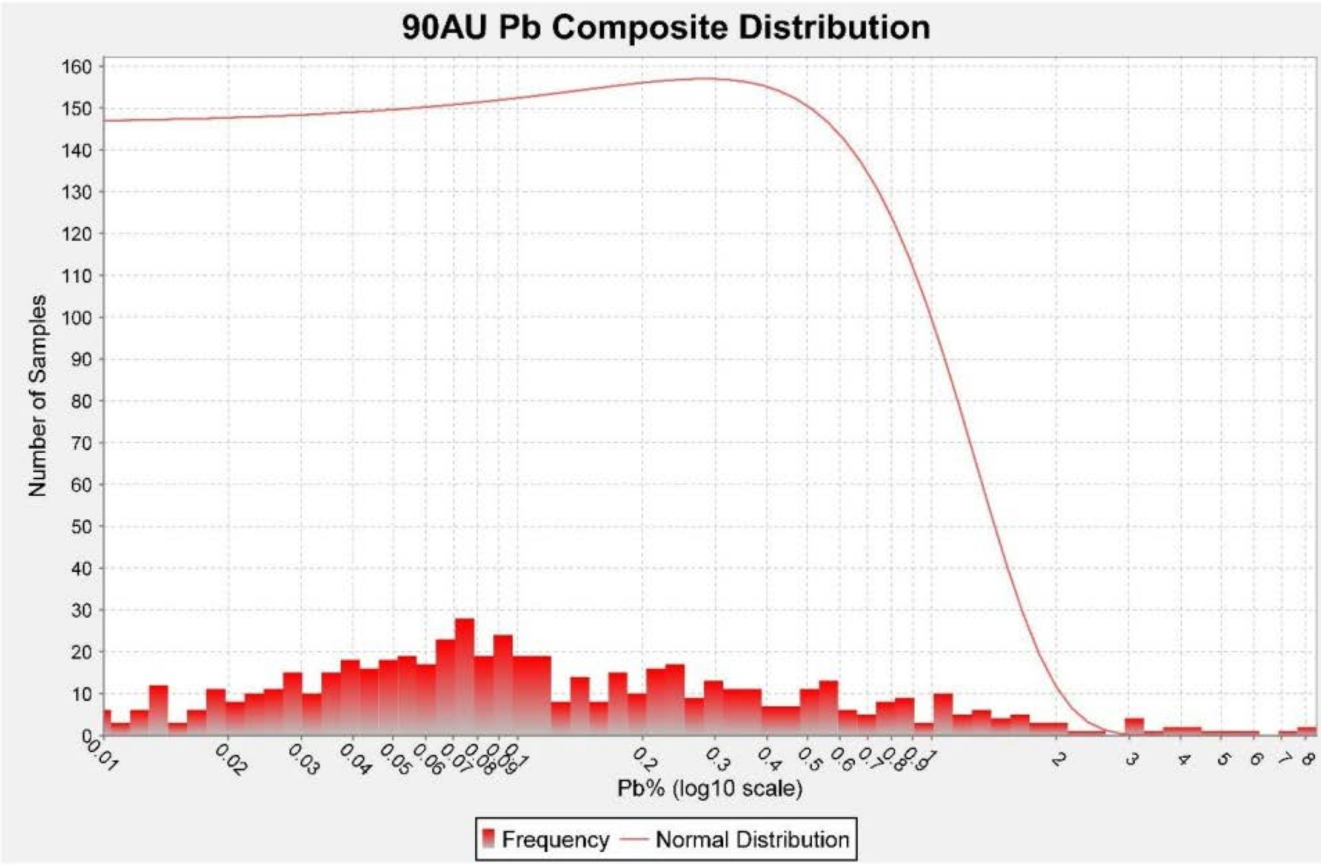
MainMS1 Cu Composite Distribution
Number of Samples
0
5
10
15
20
25
30
35
40
45
50
55
0.02
0.03
0.04
0.05
0.06
0.07
0.08
0.09
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1
2
3
Cu% (log10 scale)
Frequency
Normal Distribution

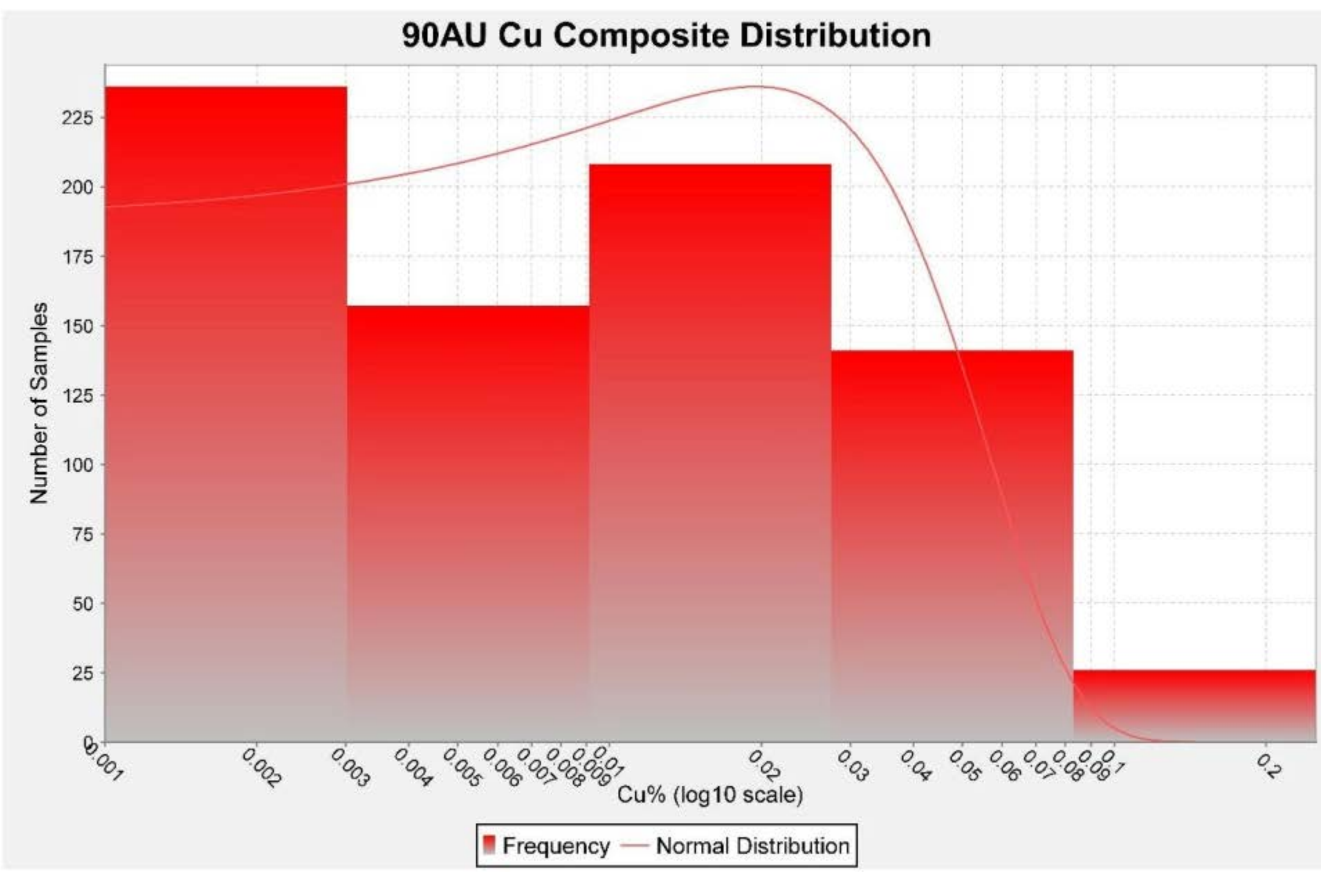


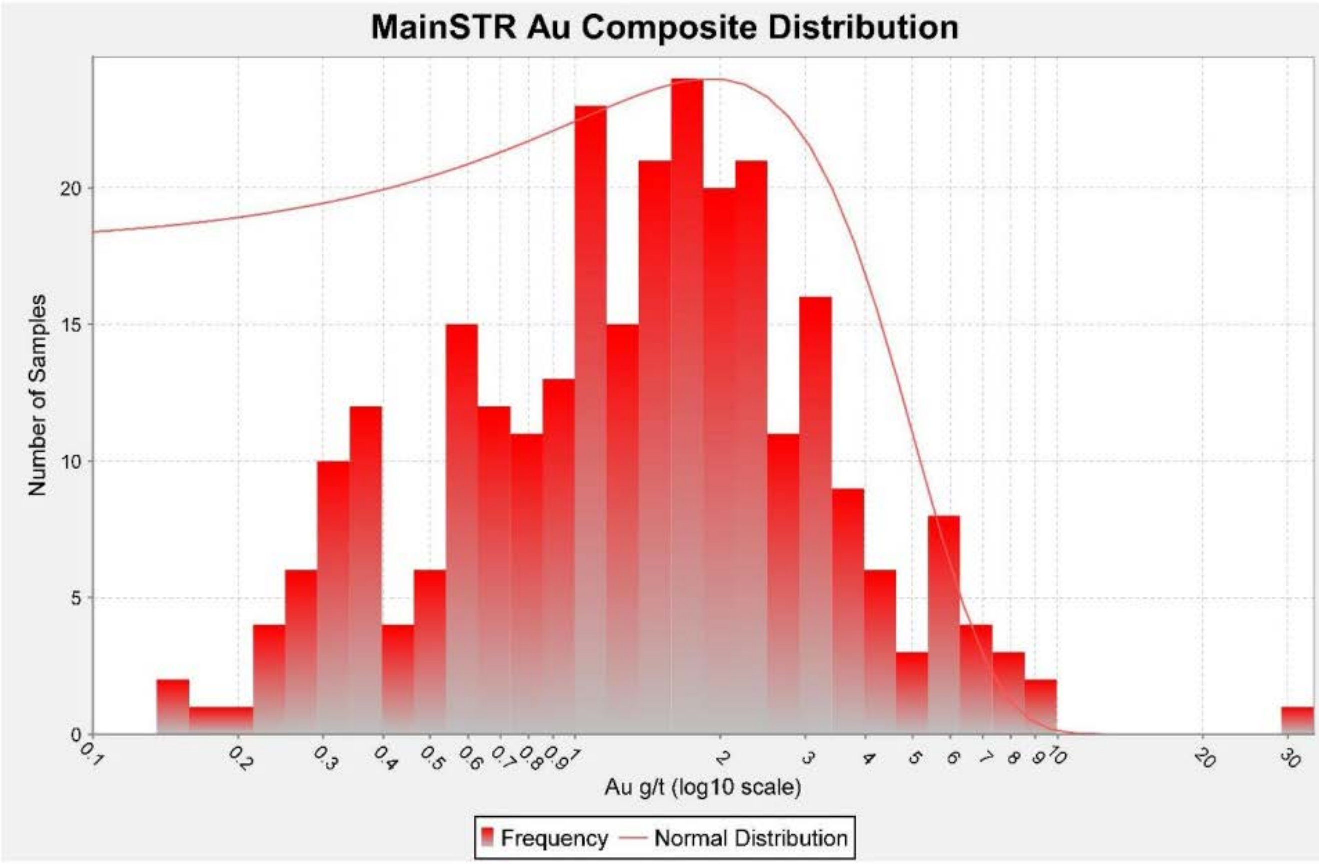
90AU Au Composite Distribution
Number of Samples
0
5
10
15
20
25
30
35
0.03
0.04
0.05
0.06
0.07
0.08
0.09
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1
2
3
4
5
6
7
8
9
10
20
30
40
Au g/t (log10 scale)
Frequency
Normal Distribution

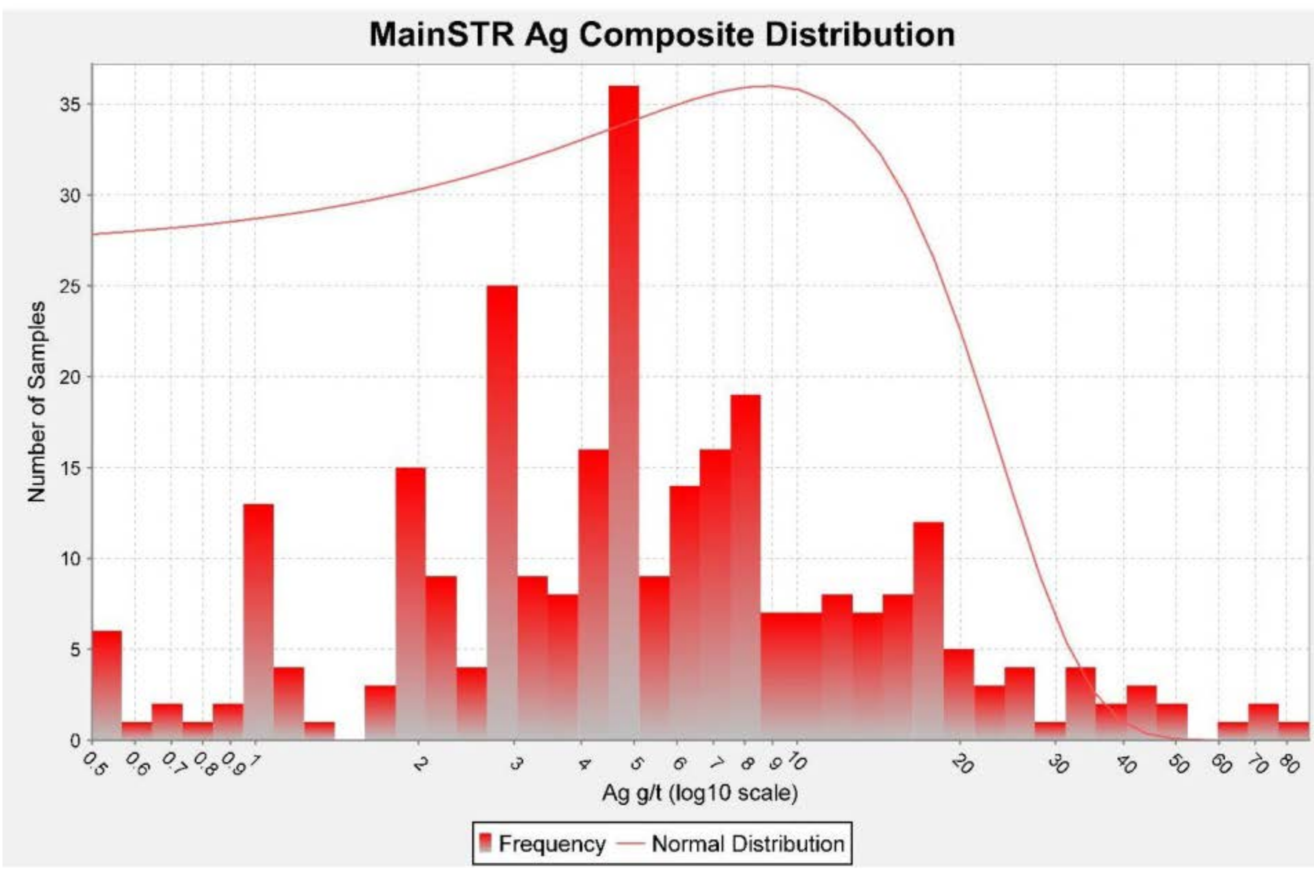

90AU Ag Composite Distribution
Number of Samples
Ag g/t (log10 scale)
Frequency
Normal Distribution

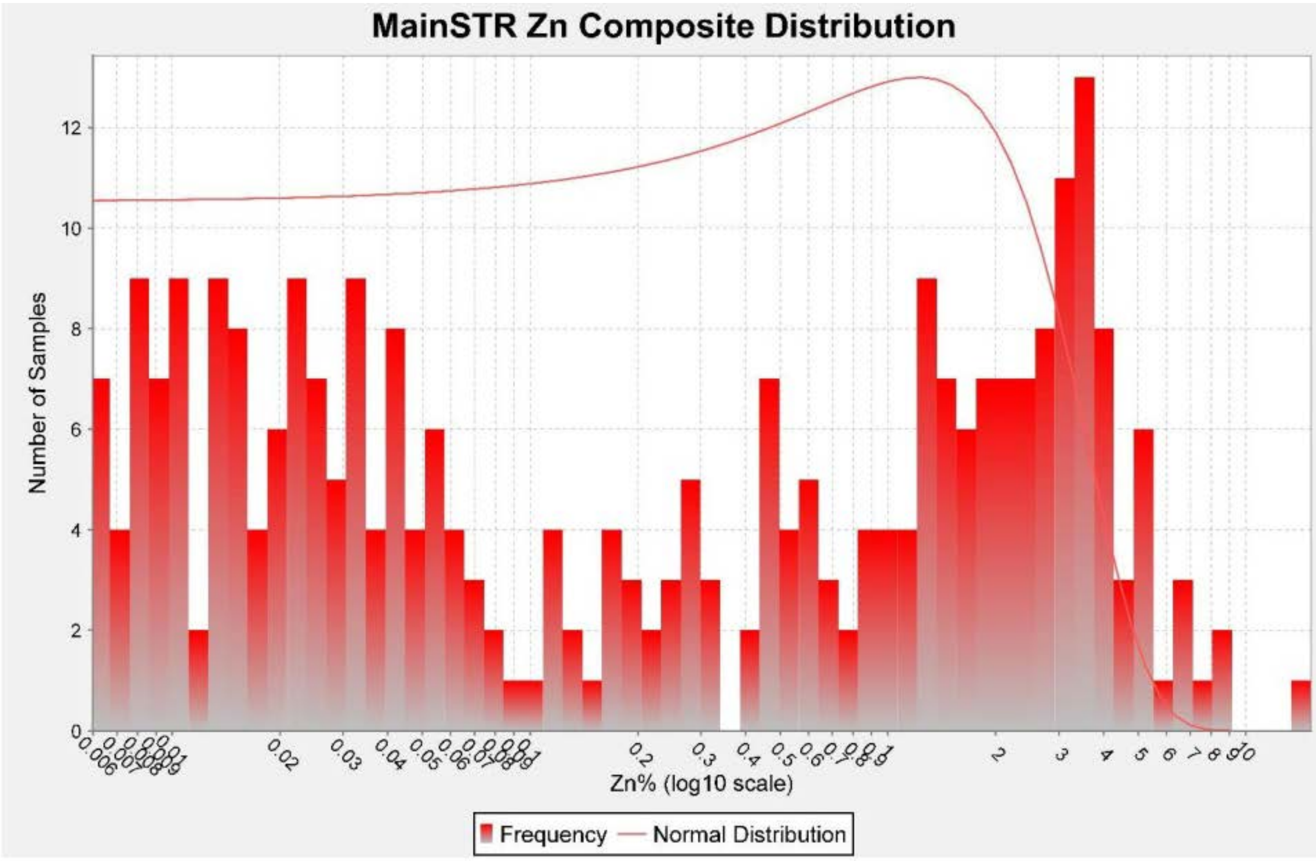

90AU Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution

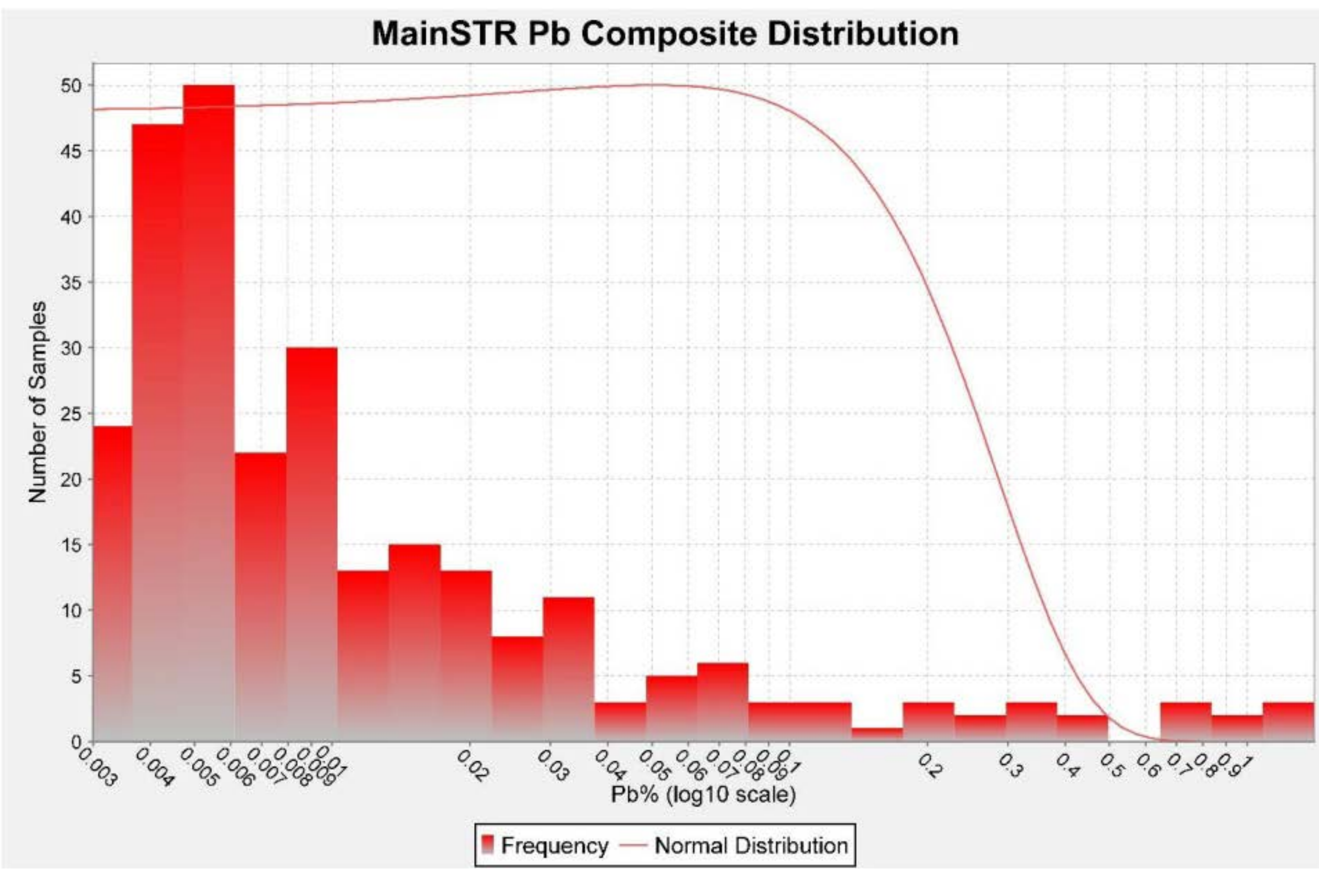

90AU Pb Composite Distribution
Number of Samples
Pb% (log10 scale)
Frequency
Normal Distribution

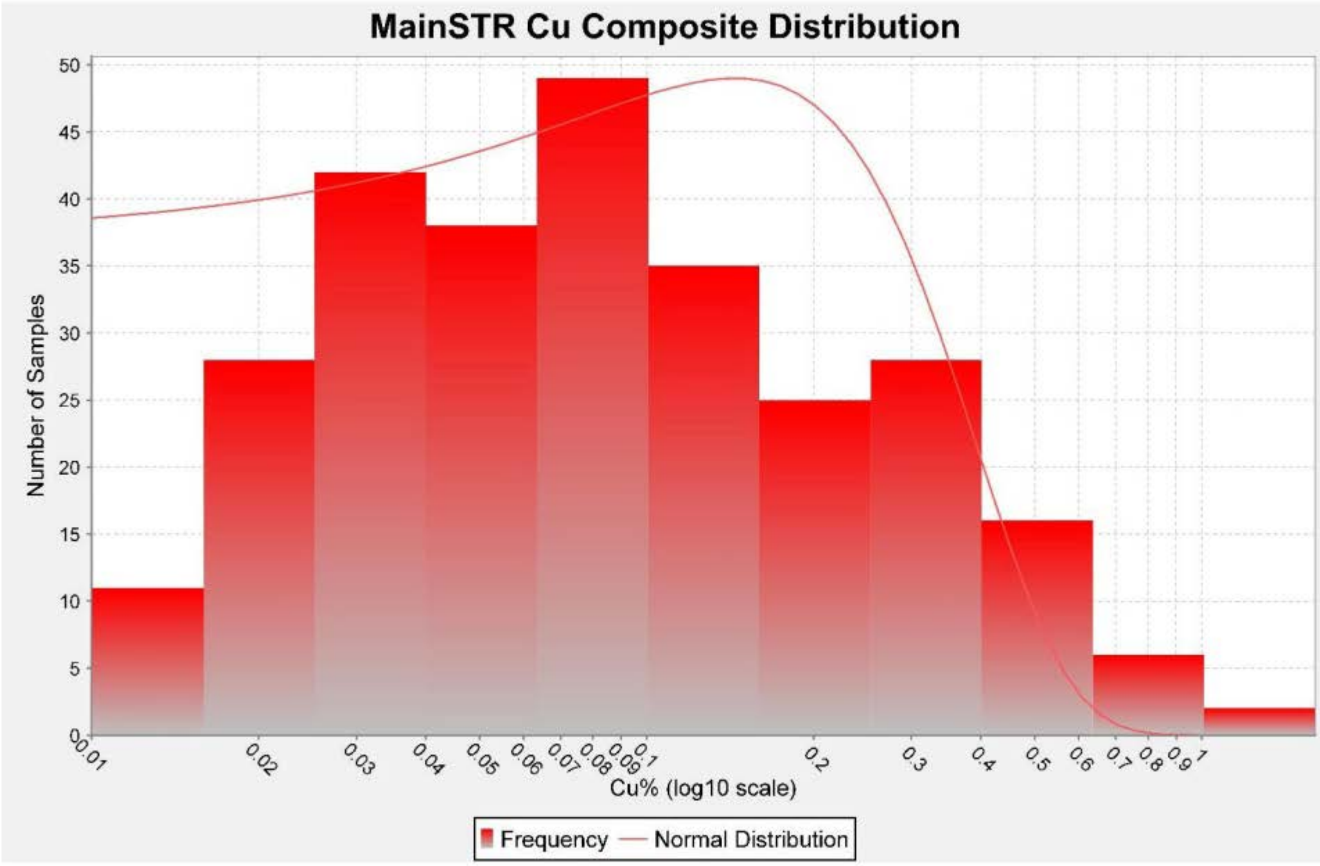

90AU Cu Composite Distribution
Number of Samples
Cu% (log10 scale)
Frequency
Normal Distribution


MainSTR Au Composite Distribution
Number of Samples
Au g/t (log10 scale)
Frequency
Normal Distribution

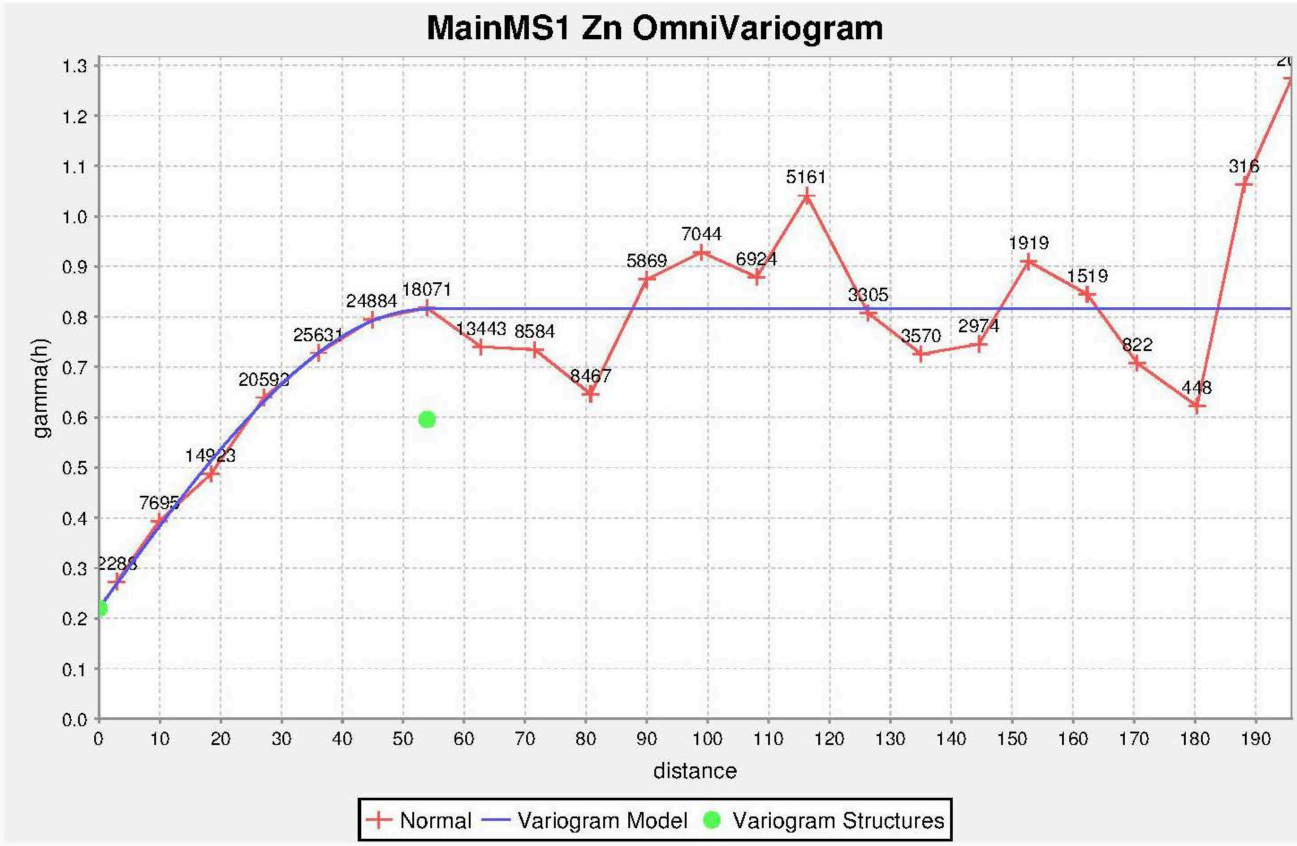

MainSTR Ag Composite Distribution
Number of Samples
Ag g/t (log10 scale)
Frequency
Normal Distribution


MainSTR Zn Composite Distribution
Number of Samples
Zn% (log10 scale)
Frequency
Normal Distribution

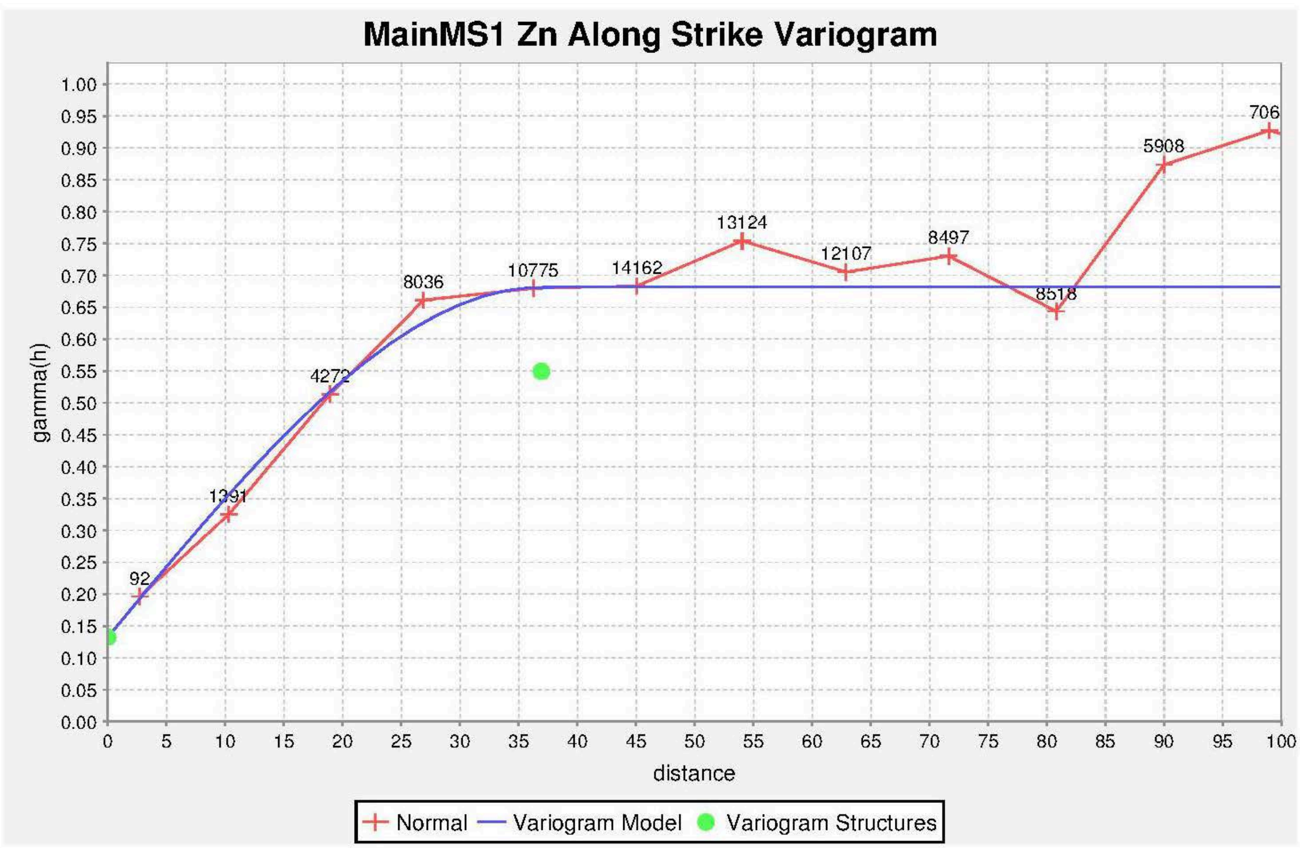

MainSTR Pb Composite Distribution
Number of Samples
Pb% (log10 scale)
Frequency
Normal Distribution


MainSTR Cu Composite Distribution
Number of Samples
Cu% (log10 scale)
Frequency
Normal Distribution

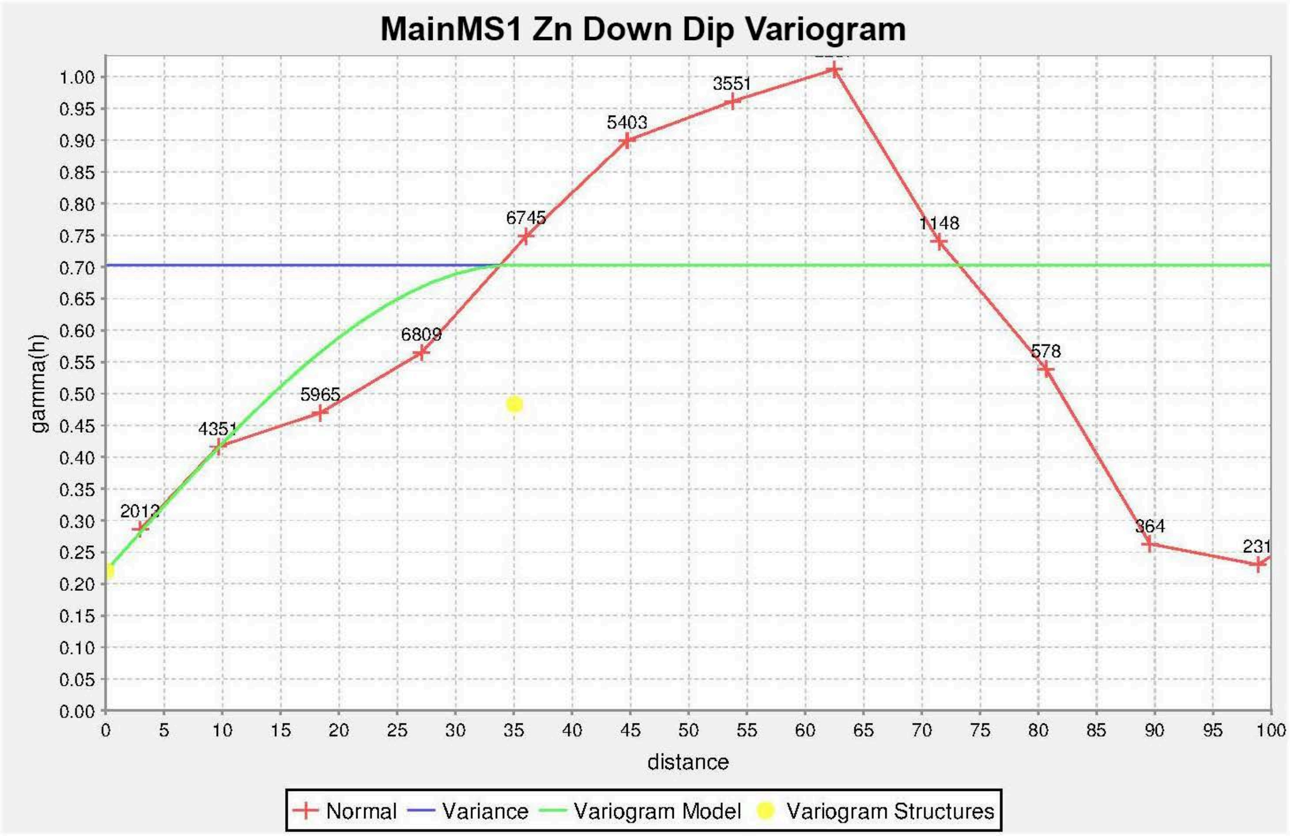
APPENDIX E VARIOGRAMS OF OPEN PIT MODEL

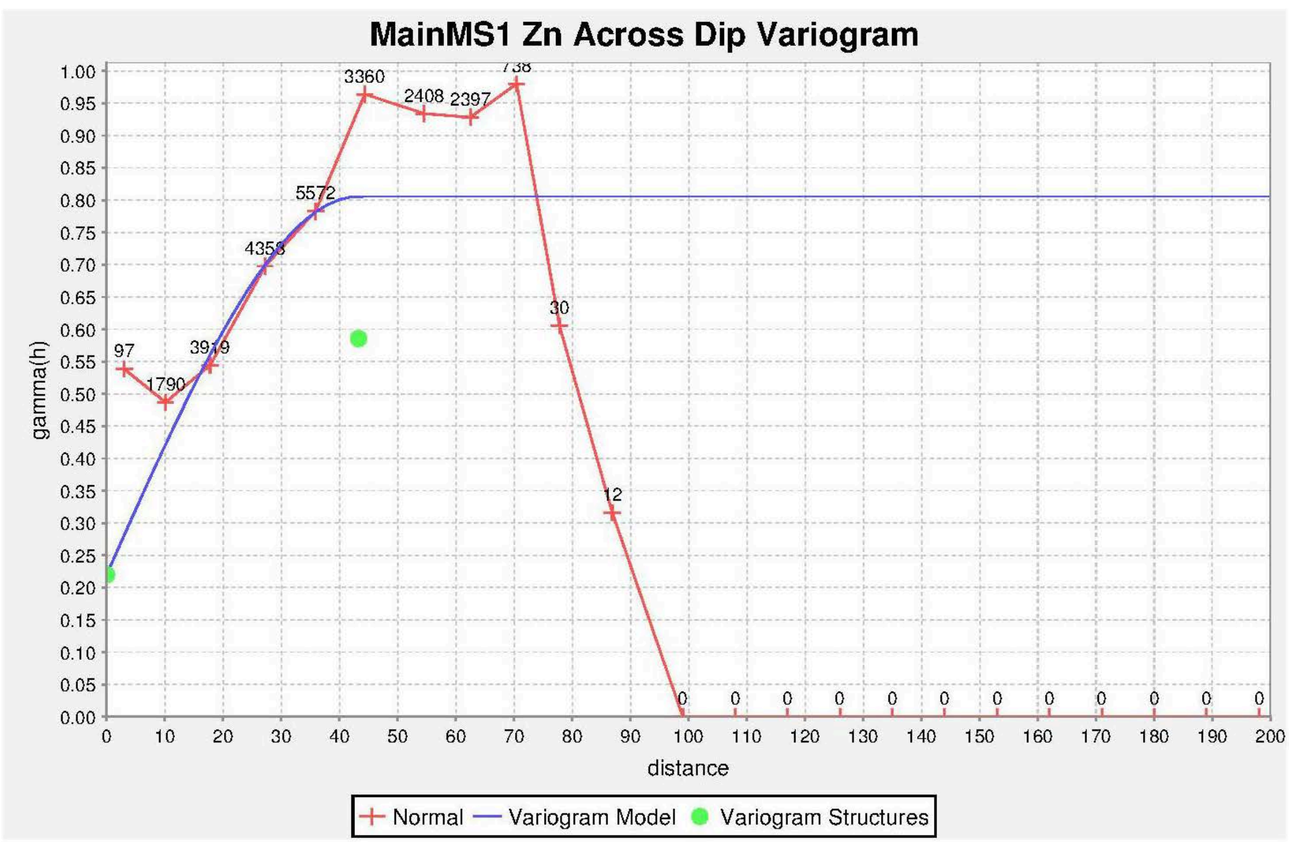


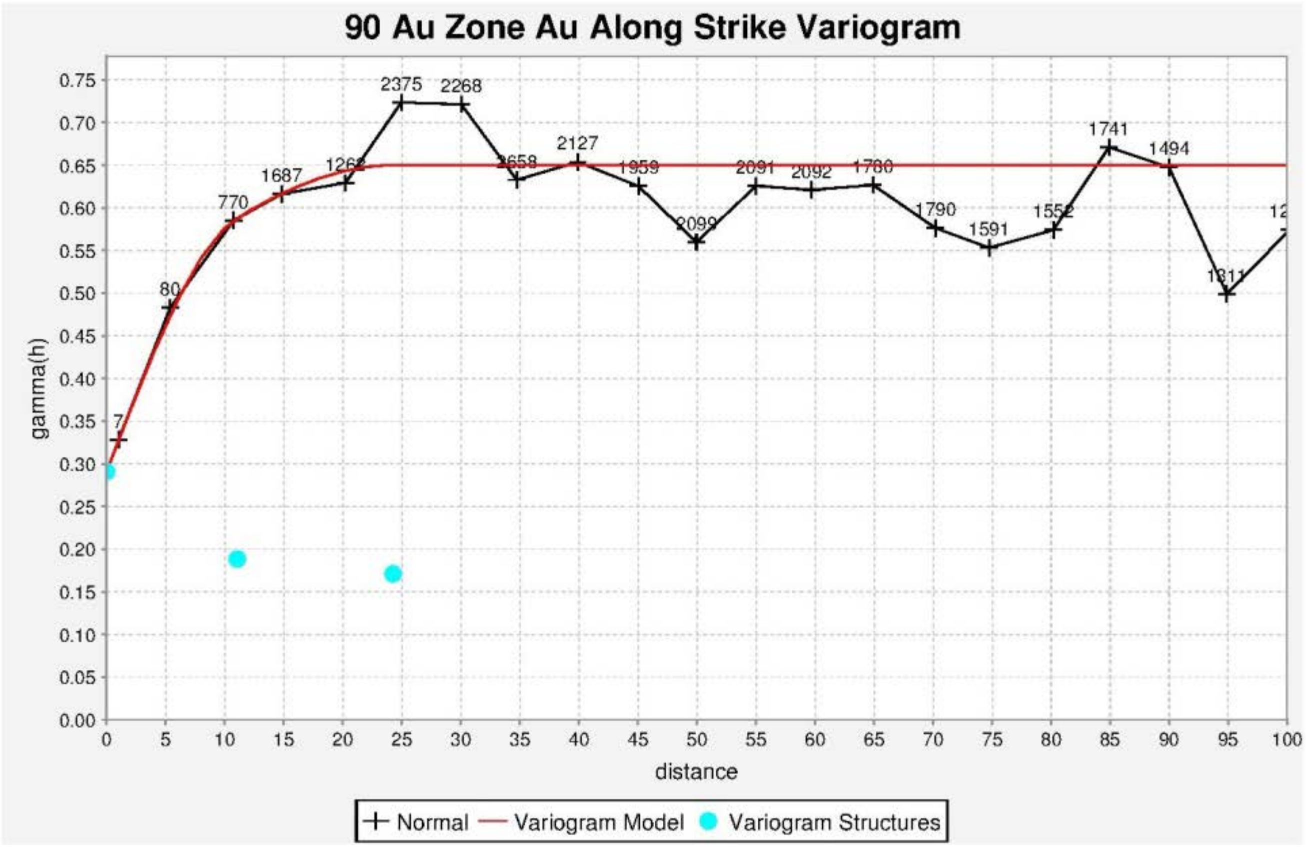
MainMS1 Zn OmniVariogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

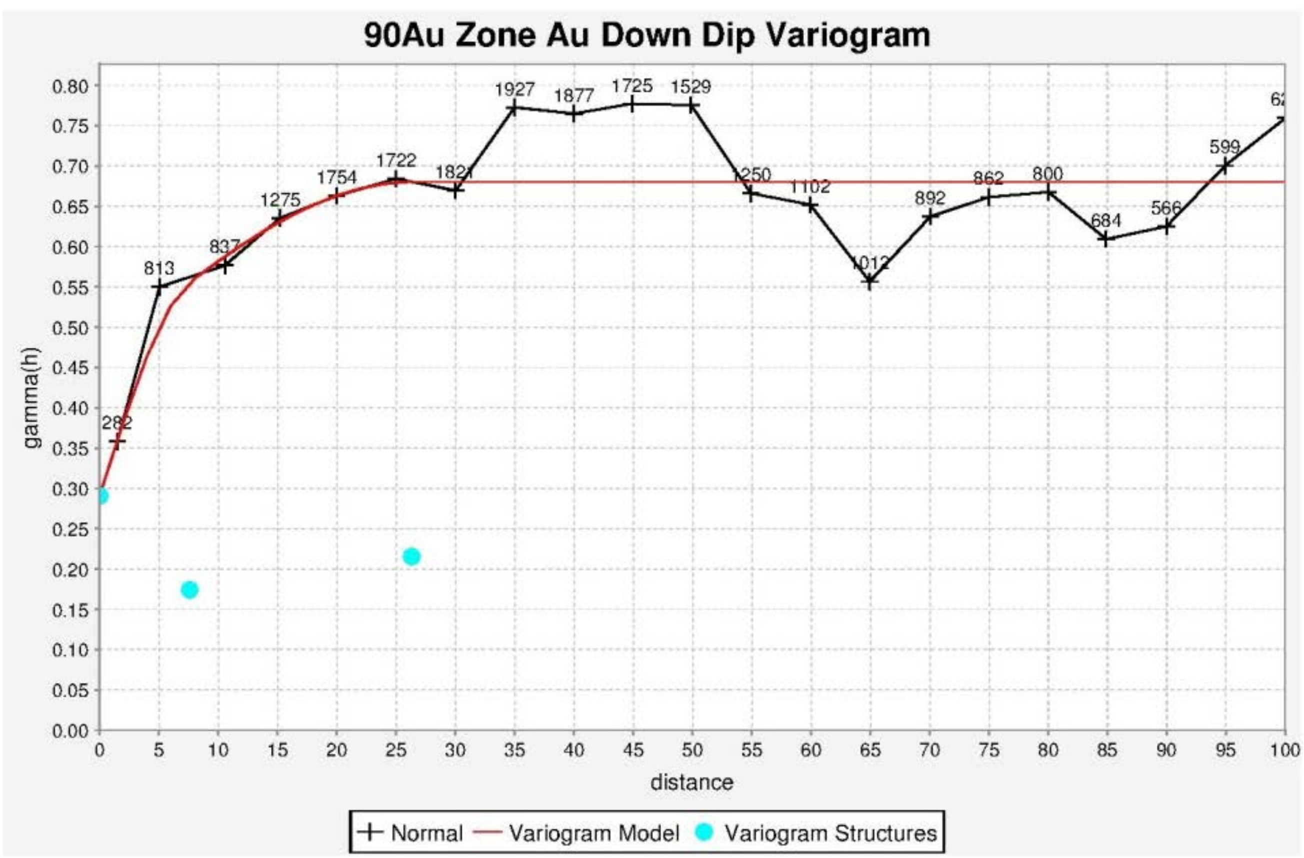


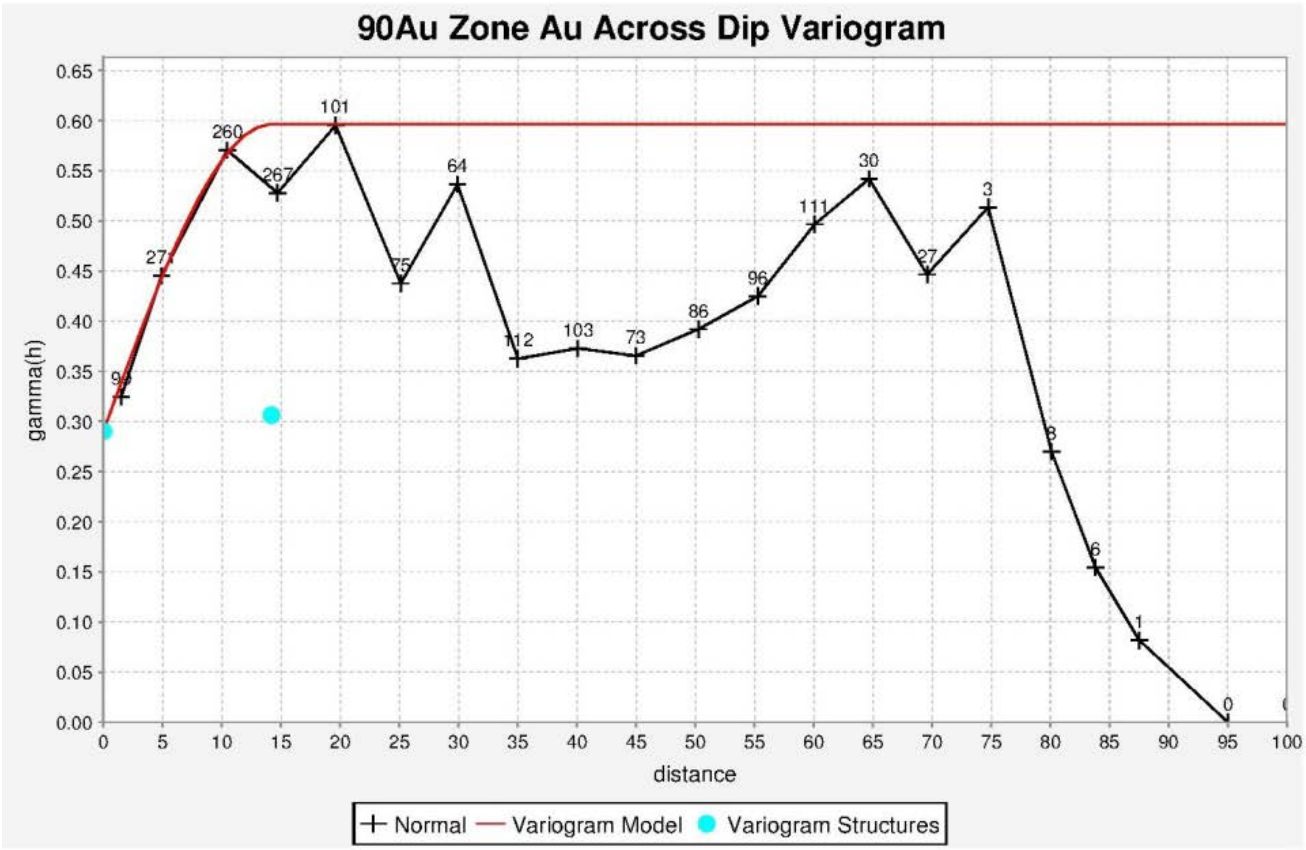
MainMS1 Zn Along Strike Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures


MainMS1 Zn Down Dip Variogram
gamma(h)
distance
2012
4351
5965
6809
6745
5403
3551
1148
578
364
231
Normal
Variance
Variogram Model
Variogram Structures

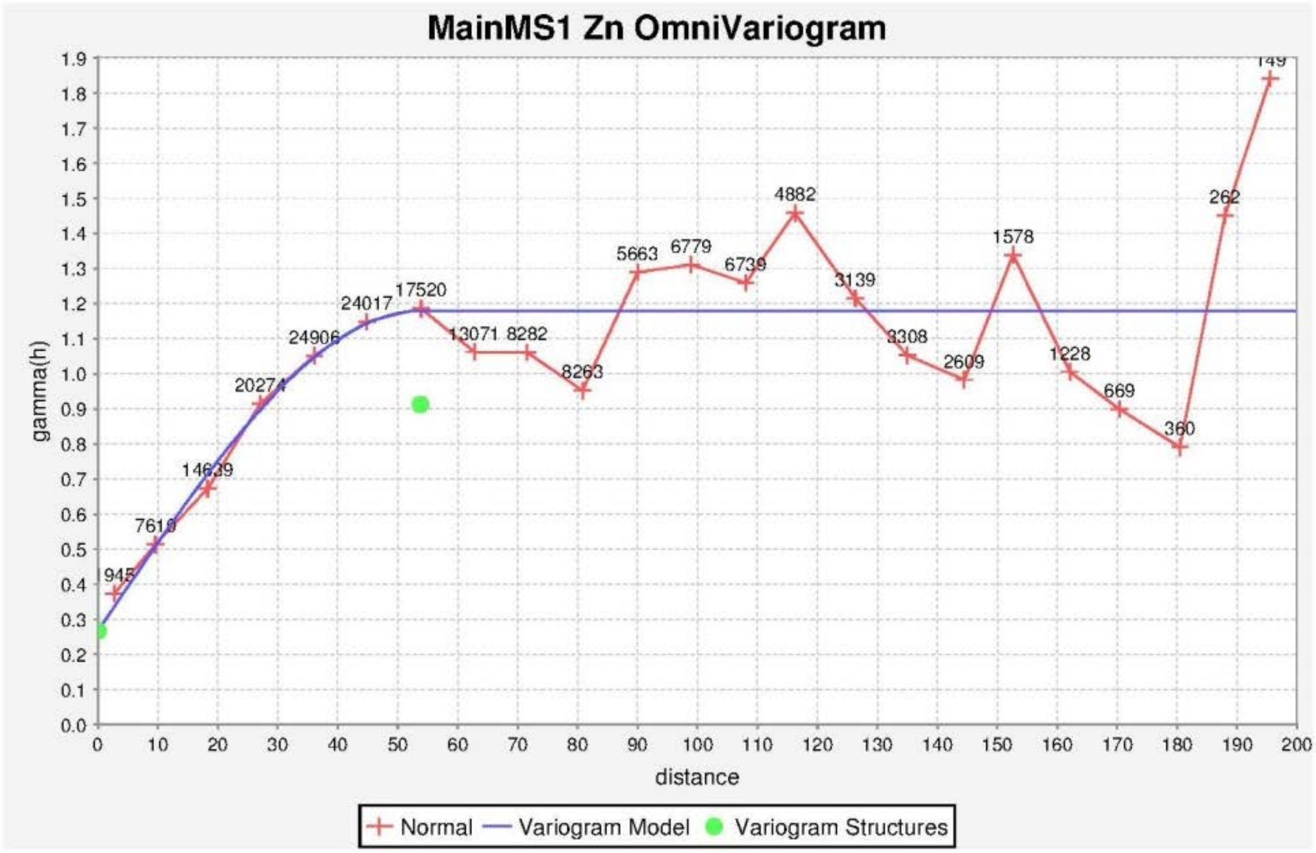

MainMS1 Zn Across Dip Variogram
gamma(h)
distance
97
1790
3919
4358
5572
3360
2408
2397
30
12
Normal
Variogram Model
Variogram Structures

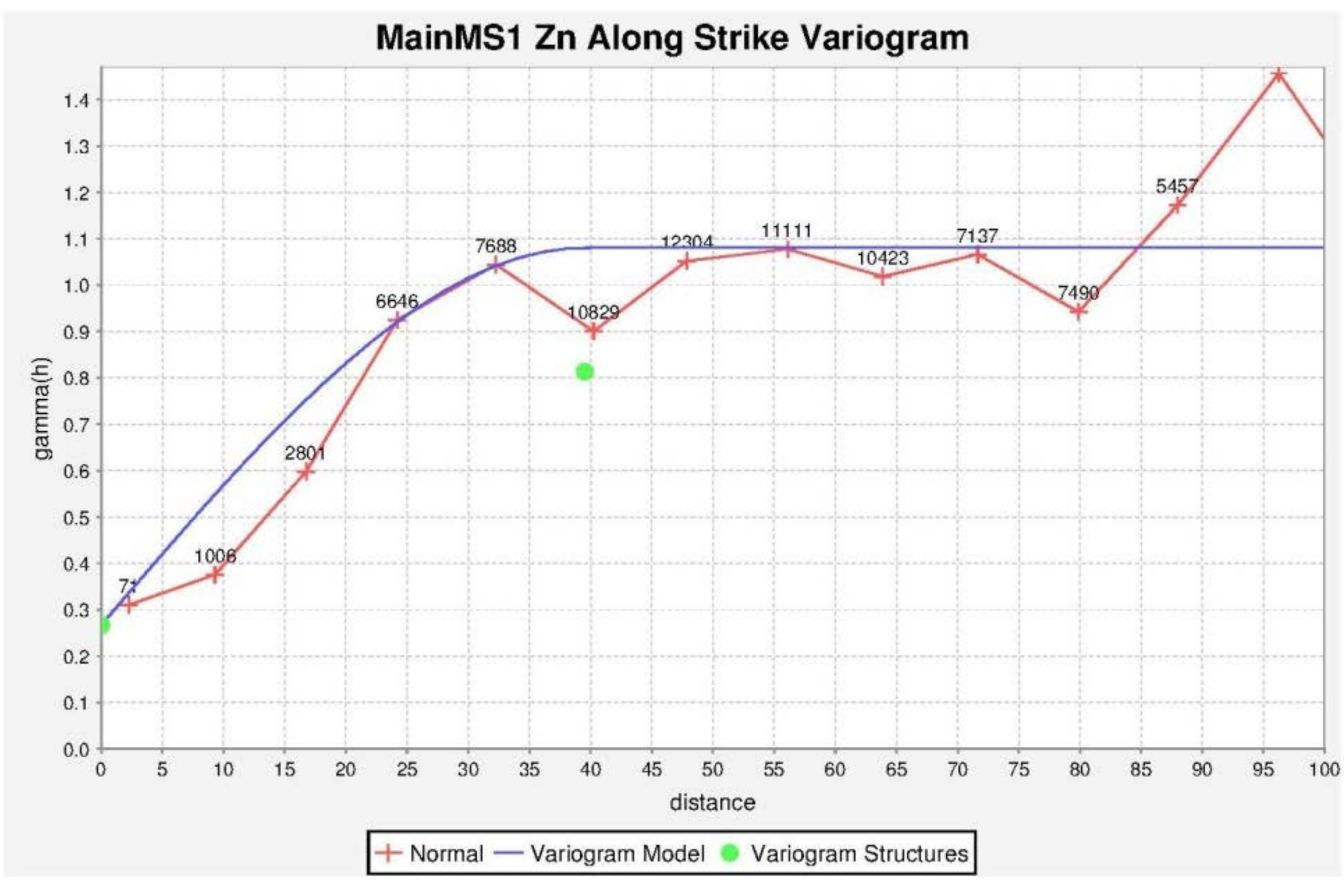

90 Au Zone Au Along Strike Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

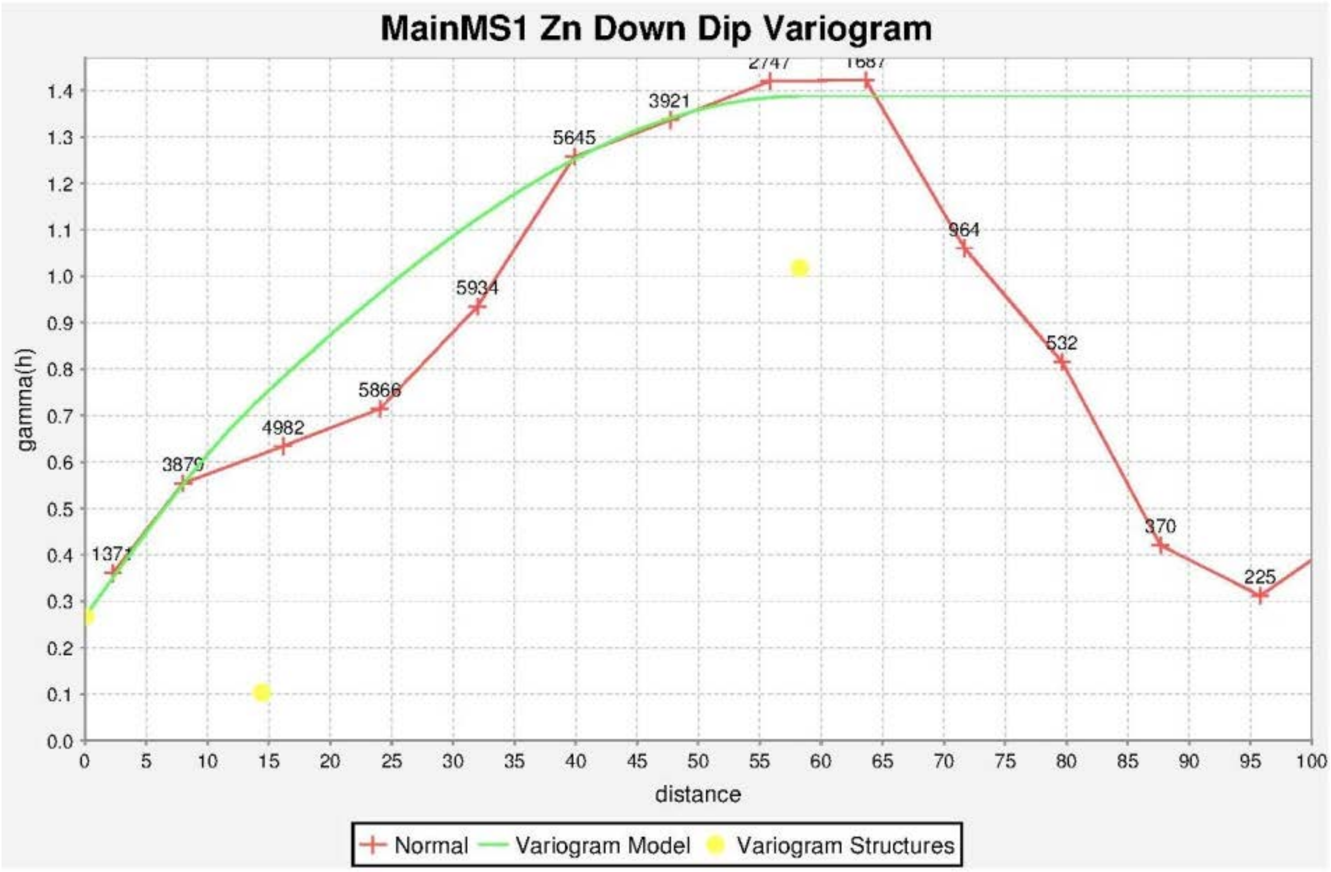

90Au Zone Au Down Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

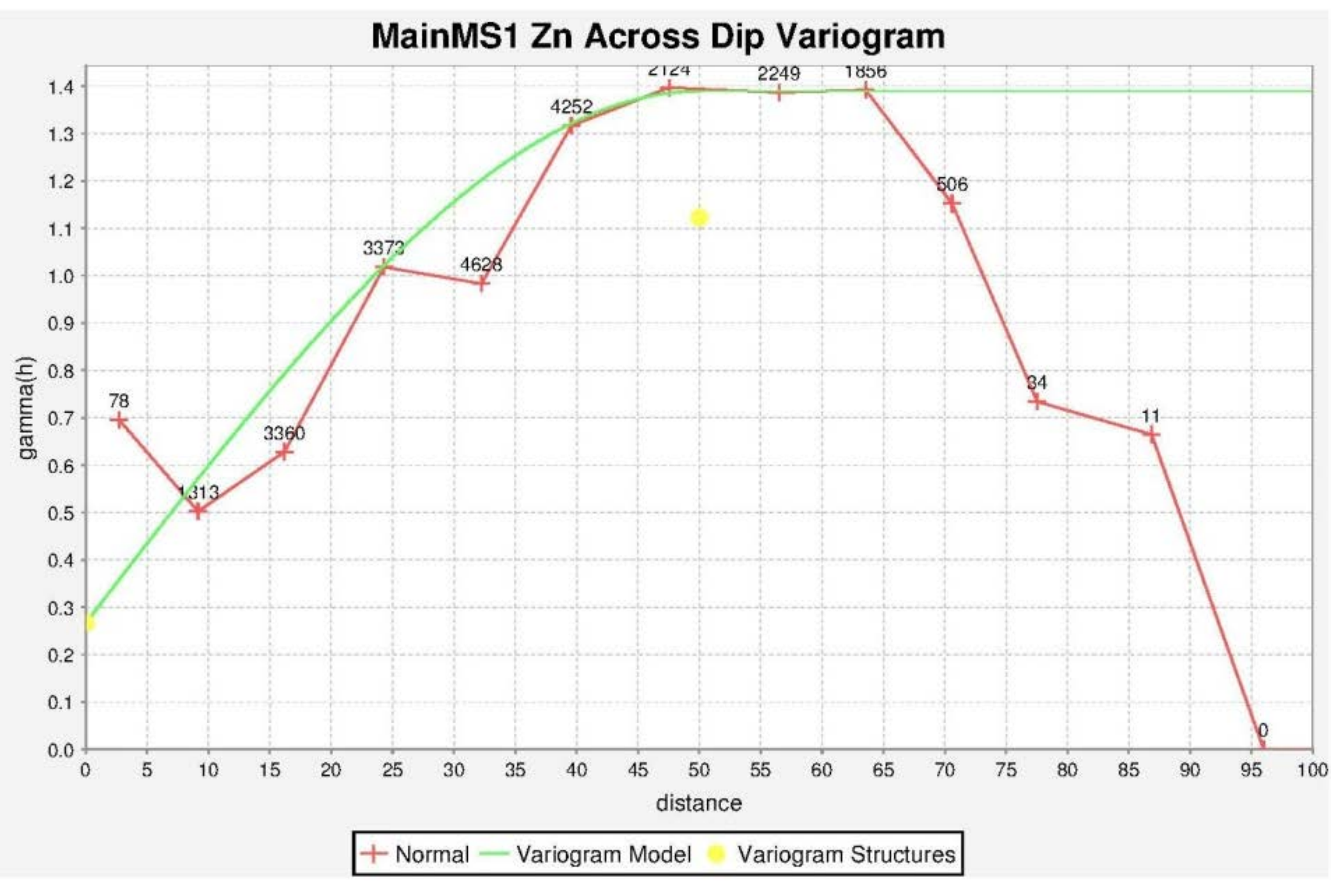

90Au Zone Au Across Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

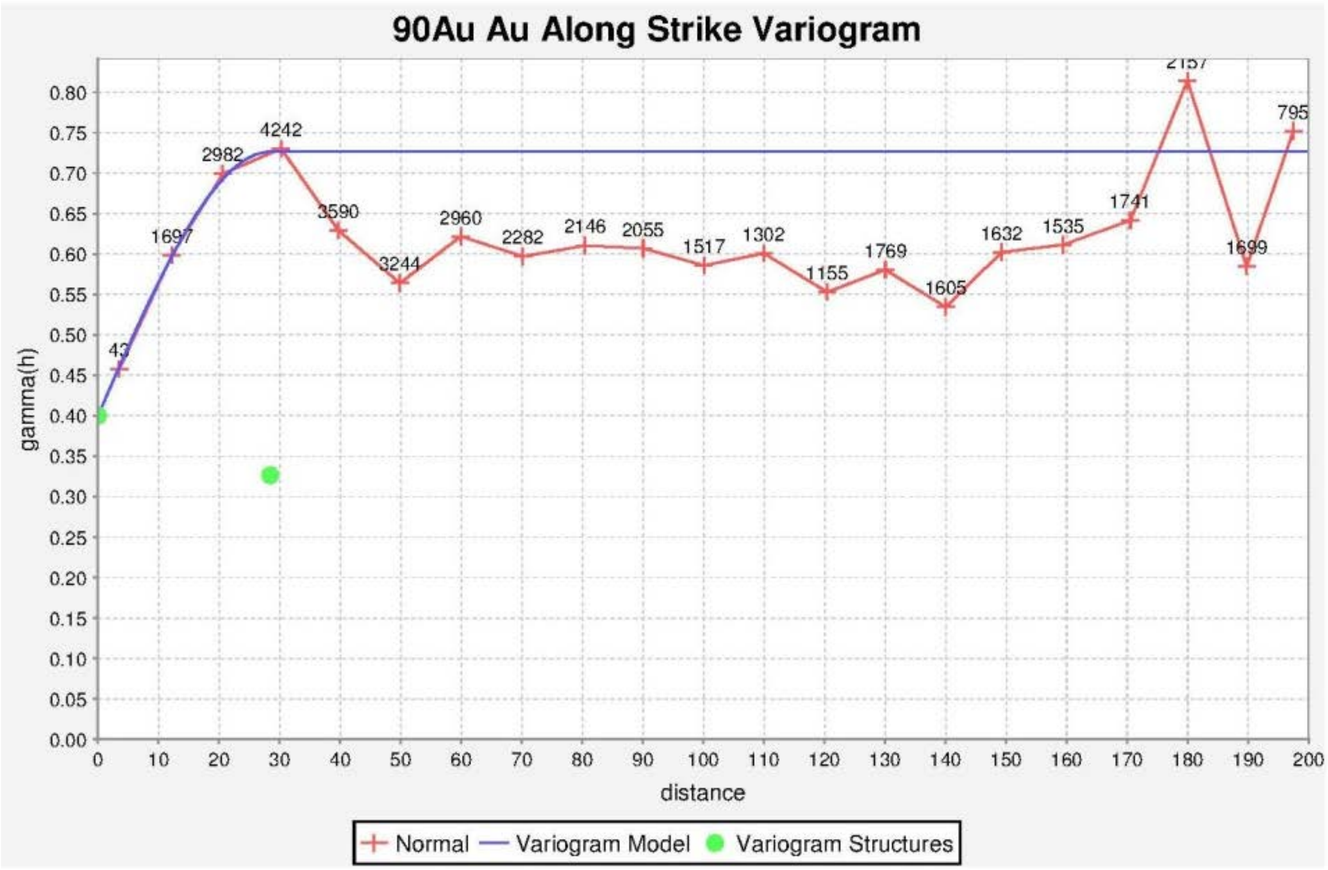
APPENDIX F VARIOGRAMS OF UNDERGROUND MODEL

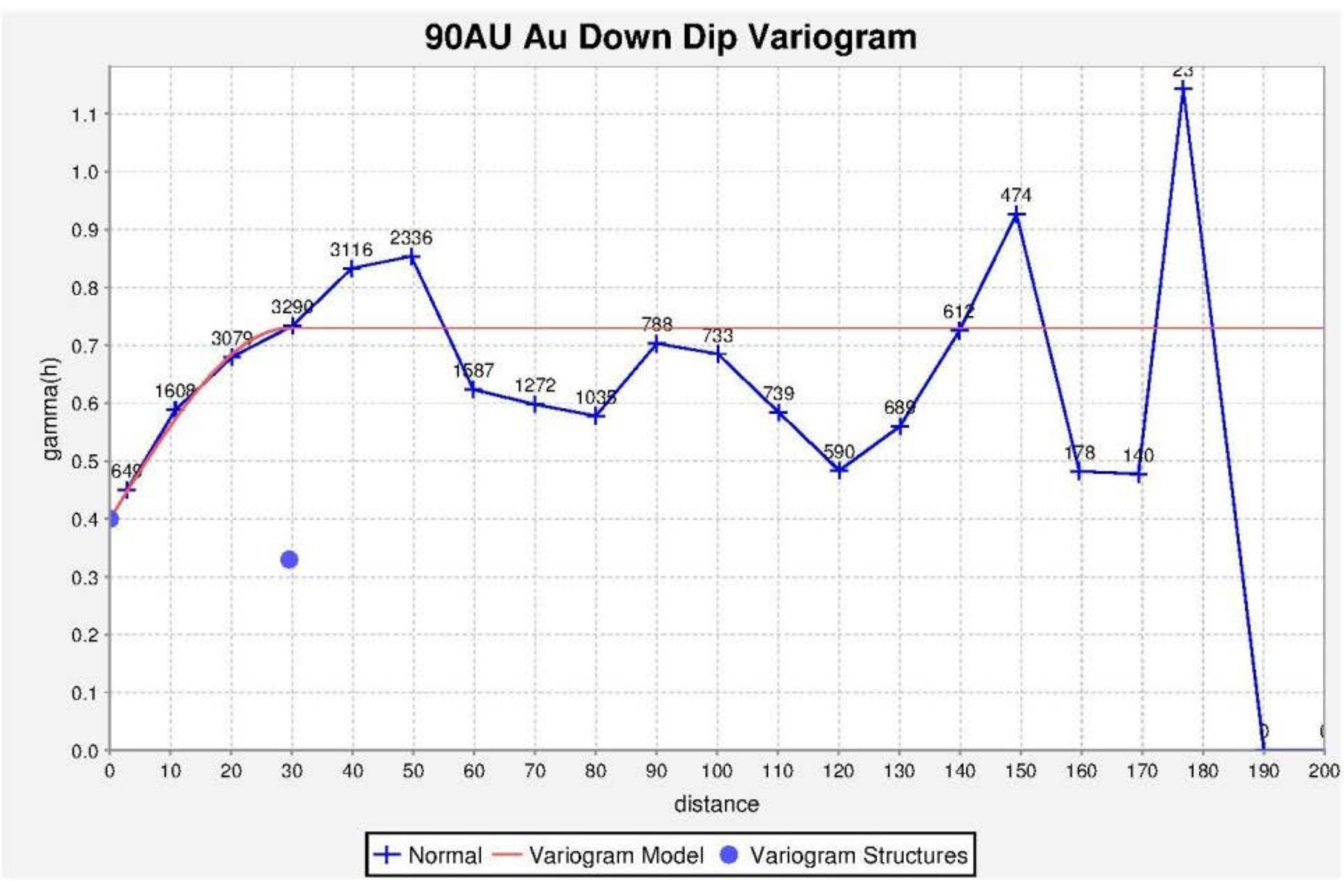

MainMS1 Zn OmniVariogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

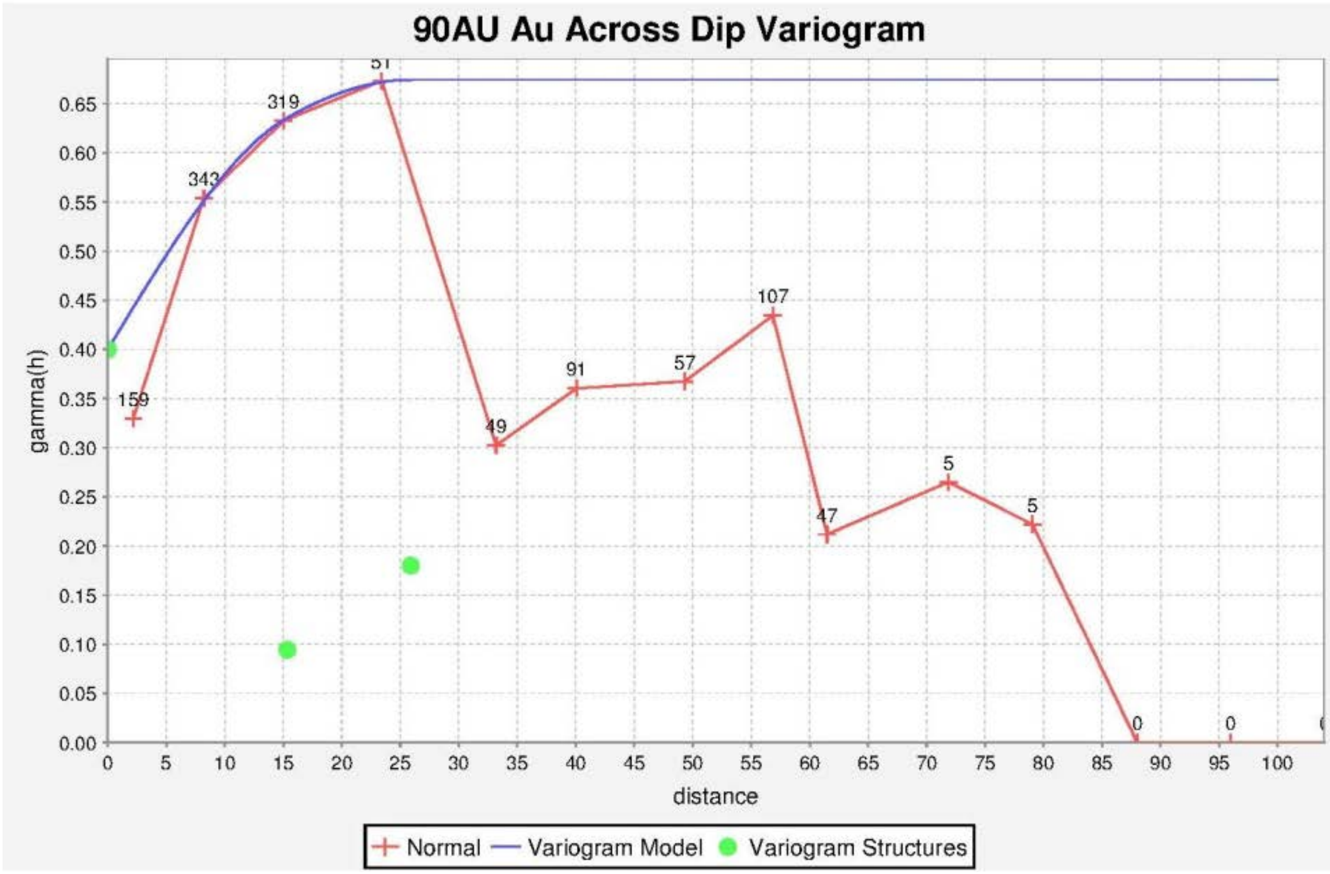

MainMS1 Zn Along Strike Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures


MainMS1 Zn Down Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

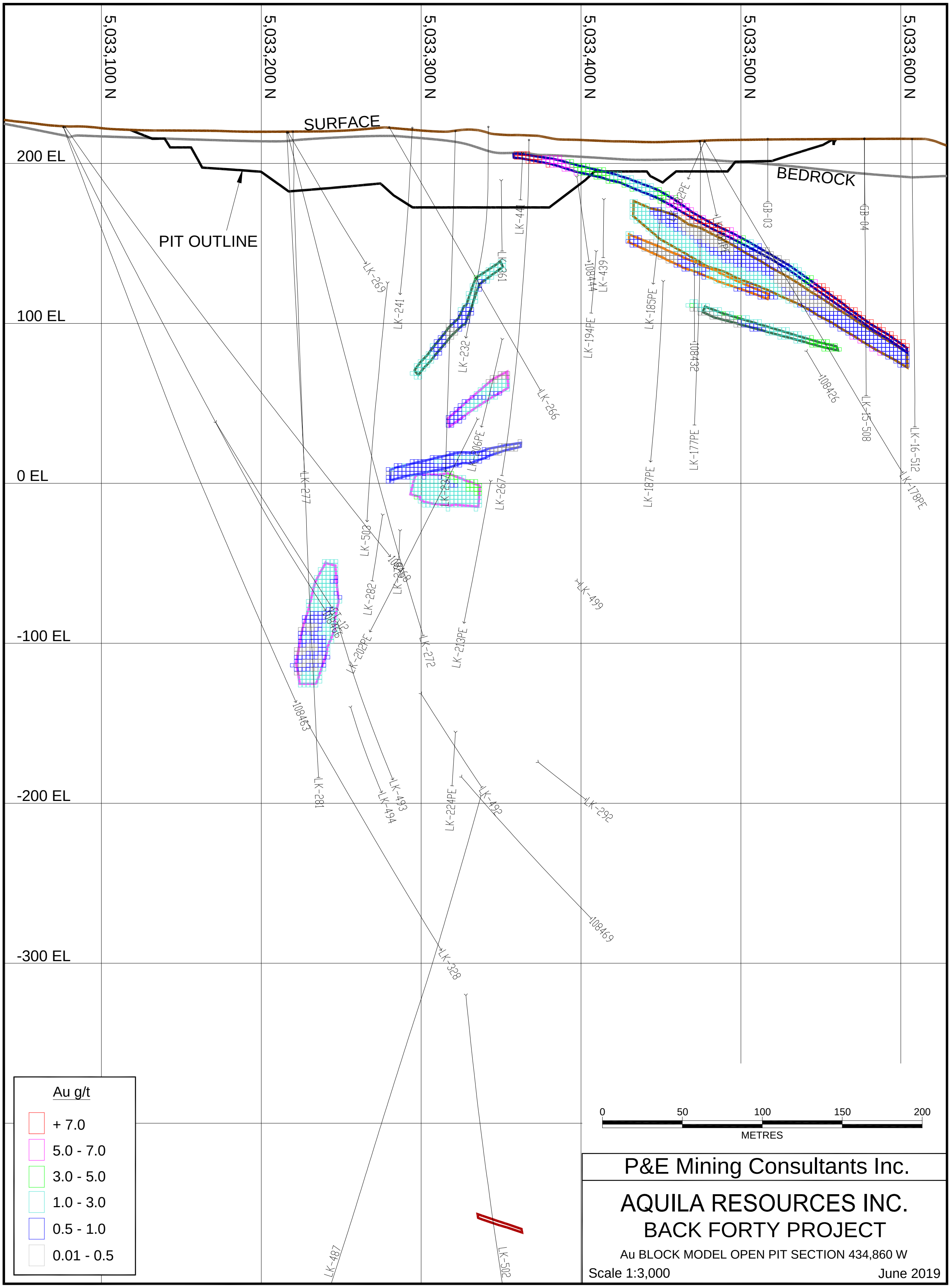

MainMS1 Zn Across Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

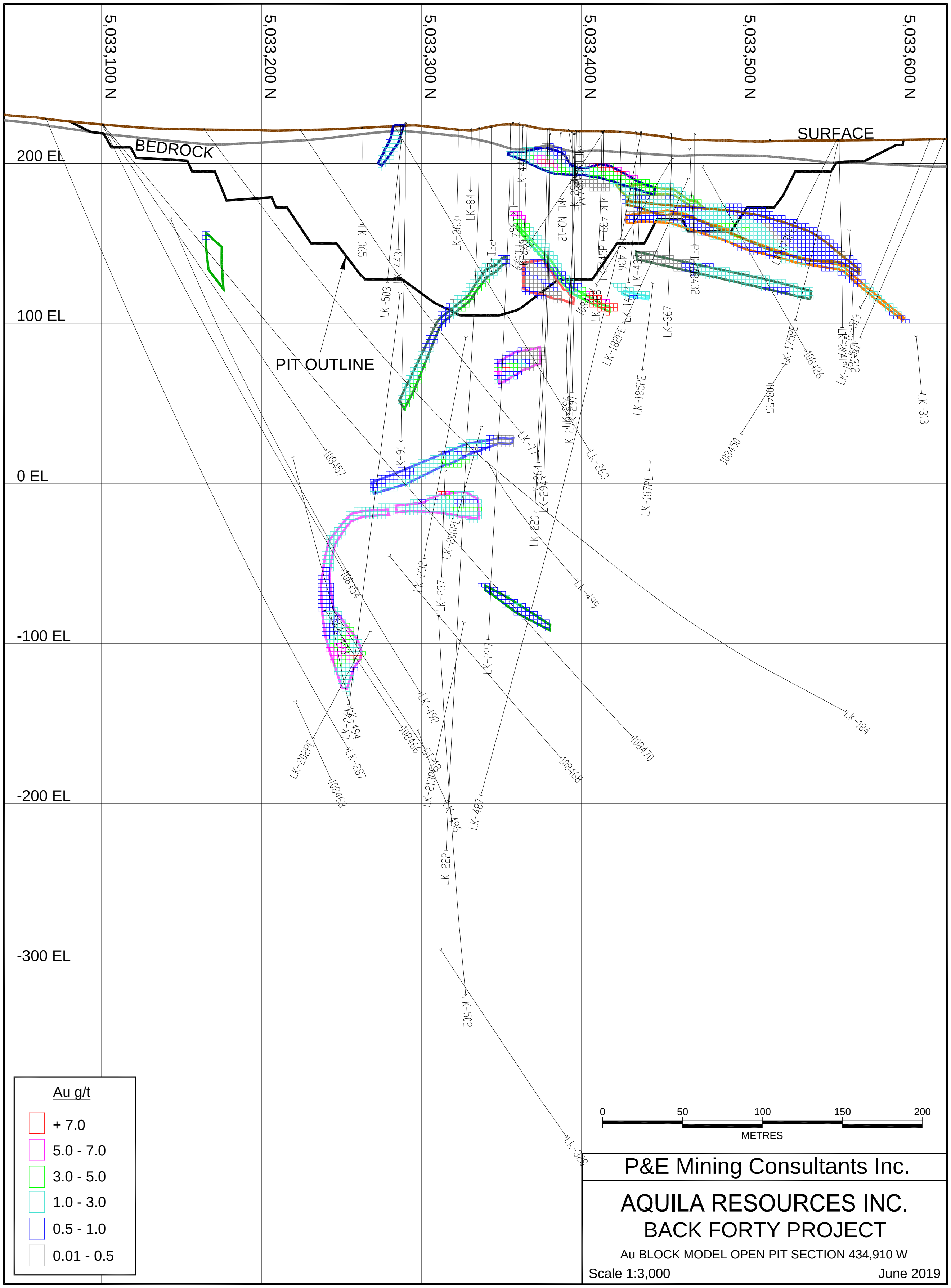

90Au Au Along Strike Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures


90AU Au Down Dip Variogram
gamma(h)
distance
Normal
Variogram Model
Variogram Structures

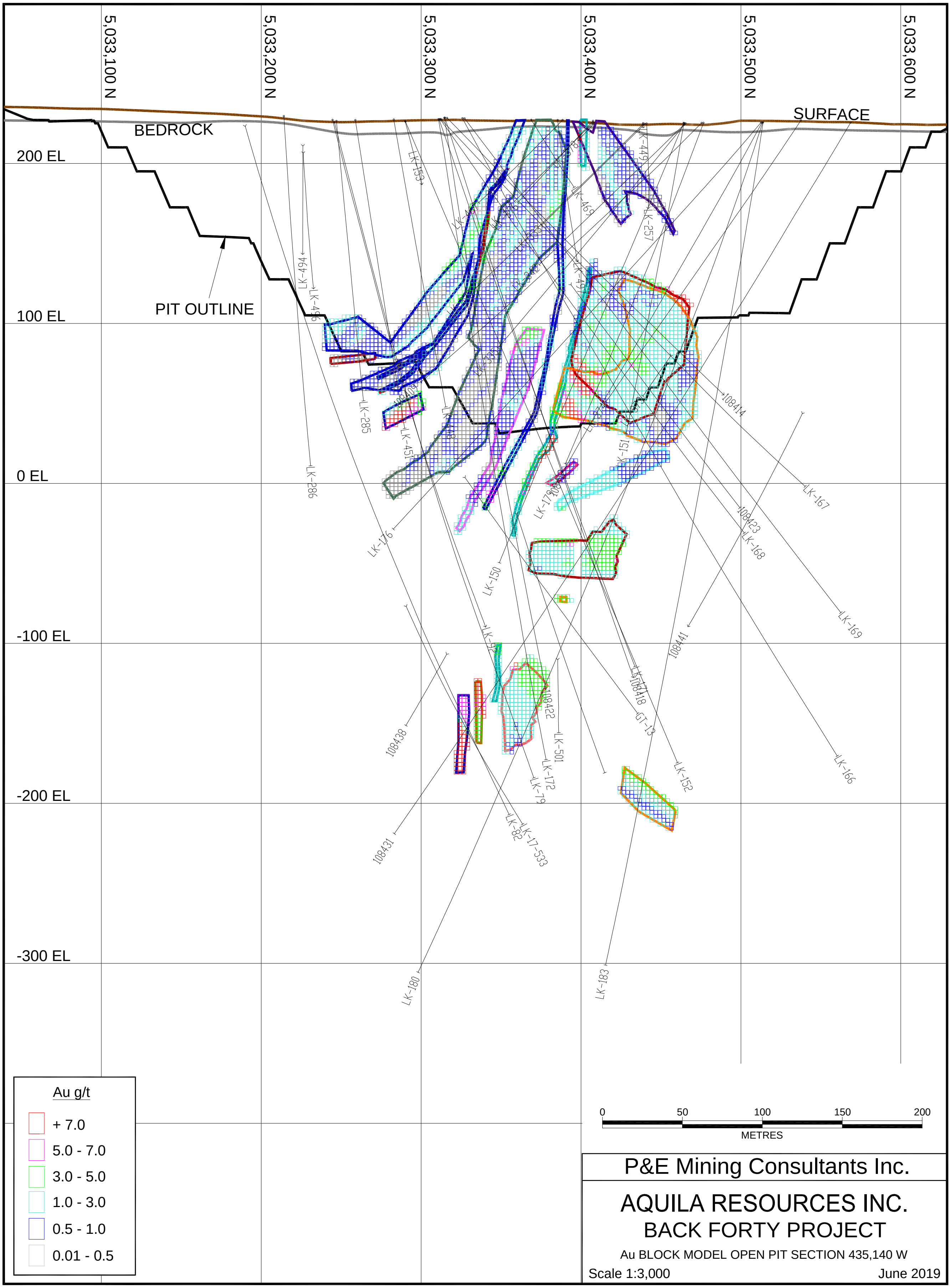

90AU Au Across Dip Variogram
gamma(h)
distance
0.00
0.05
0.10
0.15
0.20
0.25
0.30
0.35
0.40
0.45
0.50
0.55
0.60
0.65
0
5
10
15
20
25
30
35
40
45
50
55
60
65
70
75
80
85
90
95
100
159
343
319
49
91
57
107
47
5
5
0
0
Normal
Variogram Model
Variogram Structures

APPENDIX G AU BLOCK MODEL CROSS SECTIONS AND PLANS


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT SECTION 434,860 W
Scale 1:3,000
June 2019



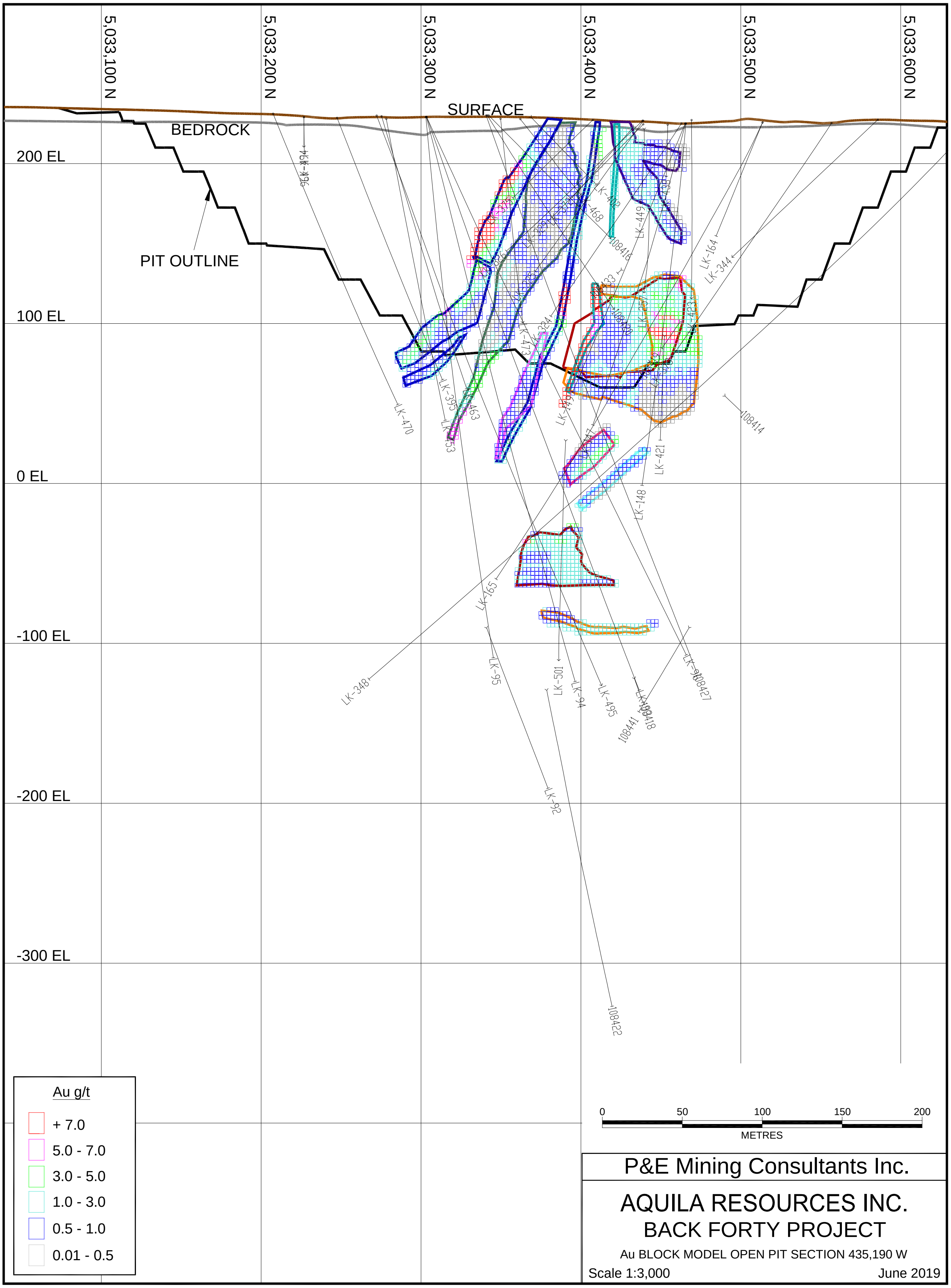
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT SECTION 434,910 W
Scale 1:3,000
June 2019



P&E Mining Consultants Inc.

AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT SECTION 435,140 W

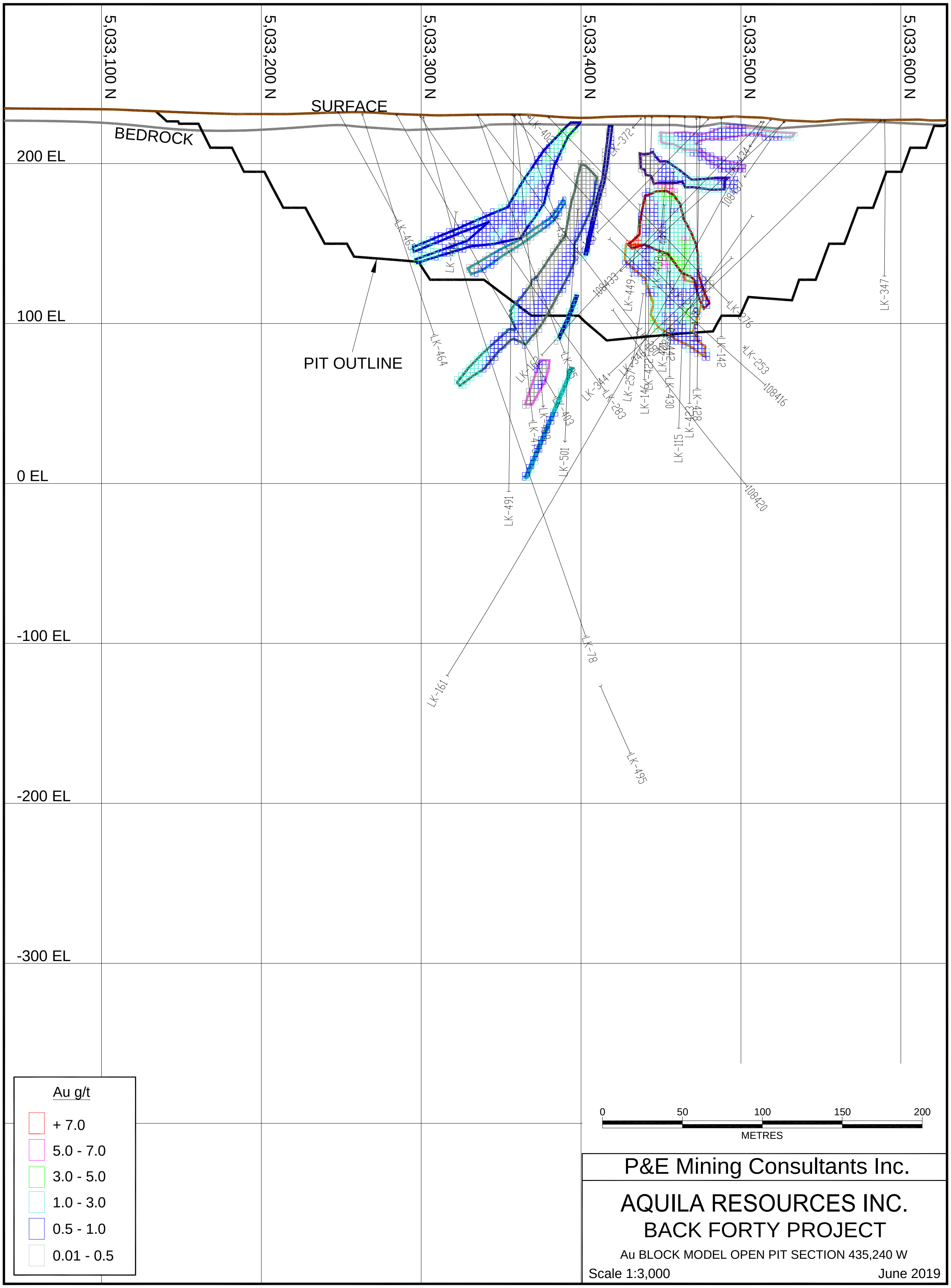
Scale 1:3,000 June 2019



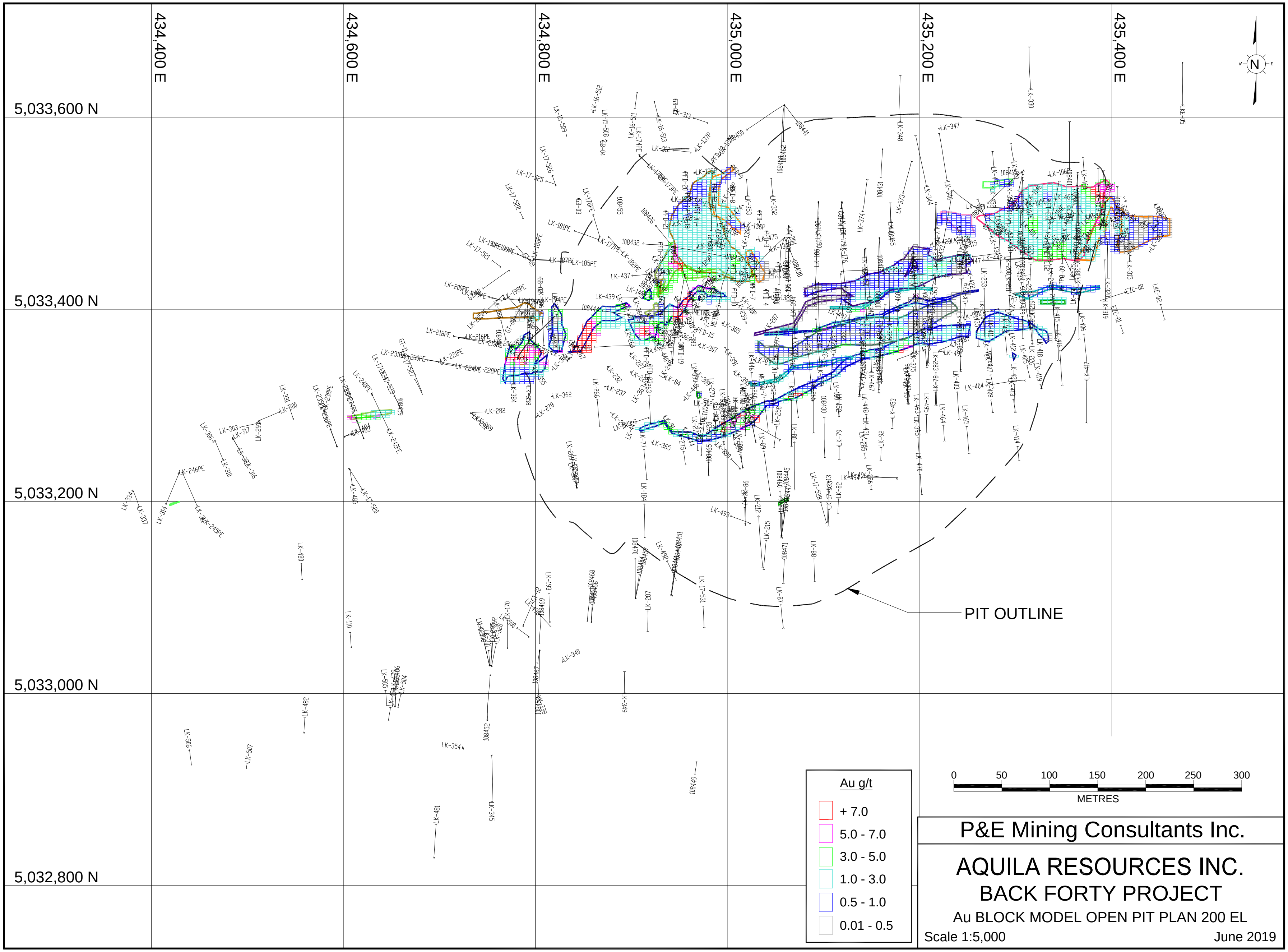
P&E Mining Consultants Inc.

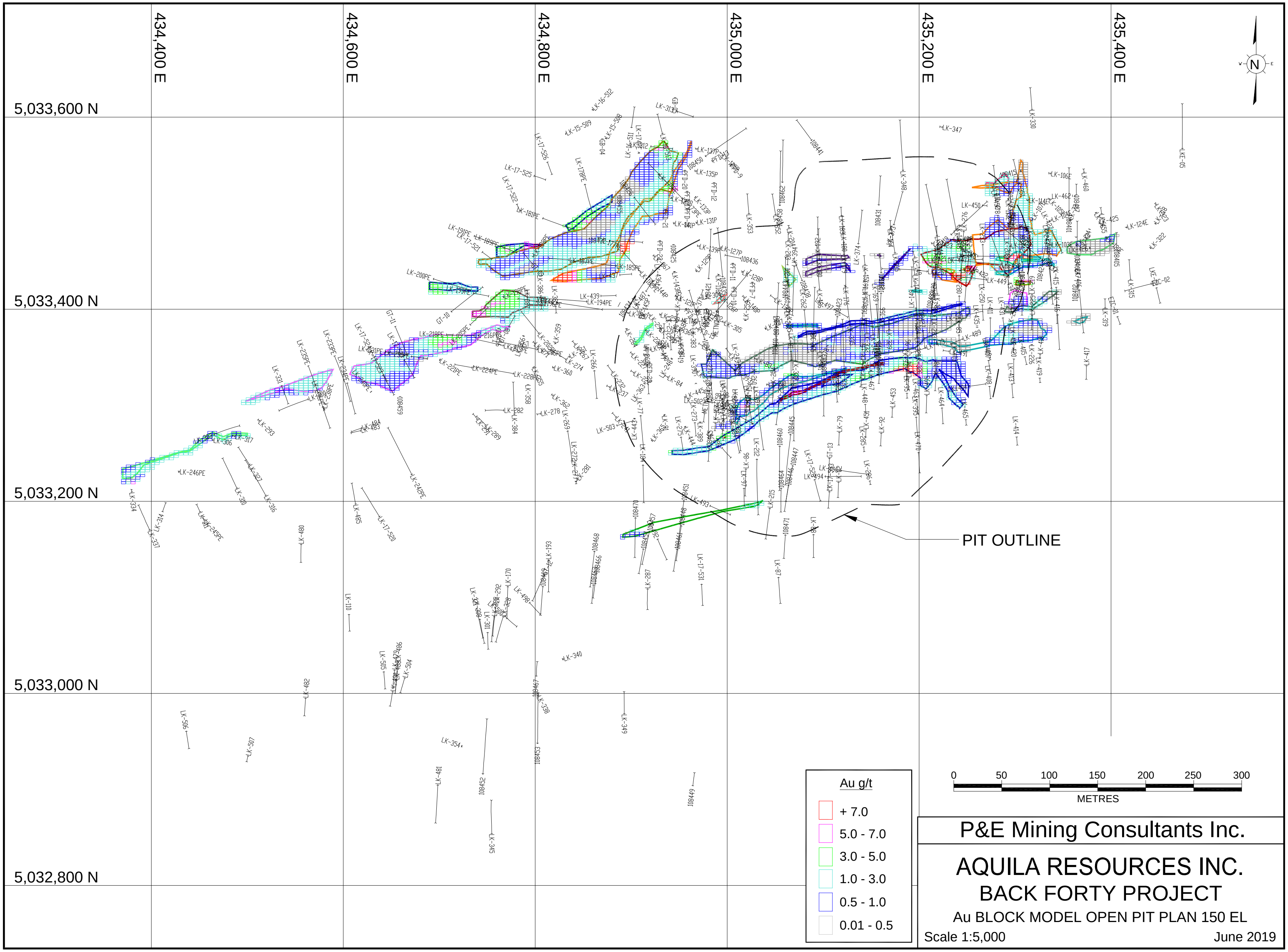
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT SECTION 435,190 W

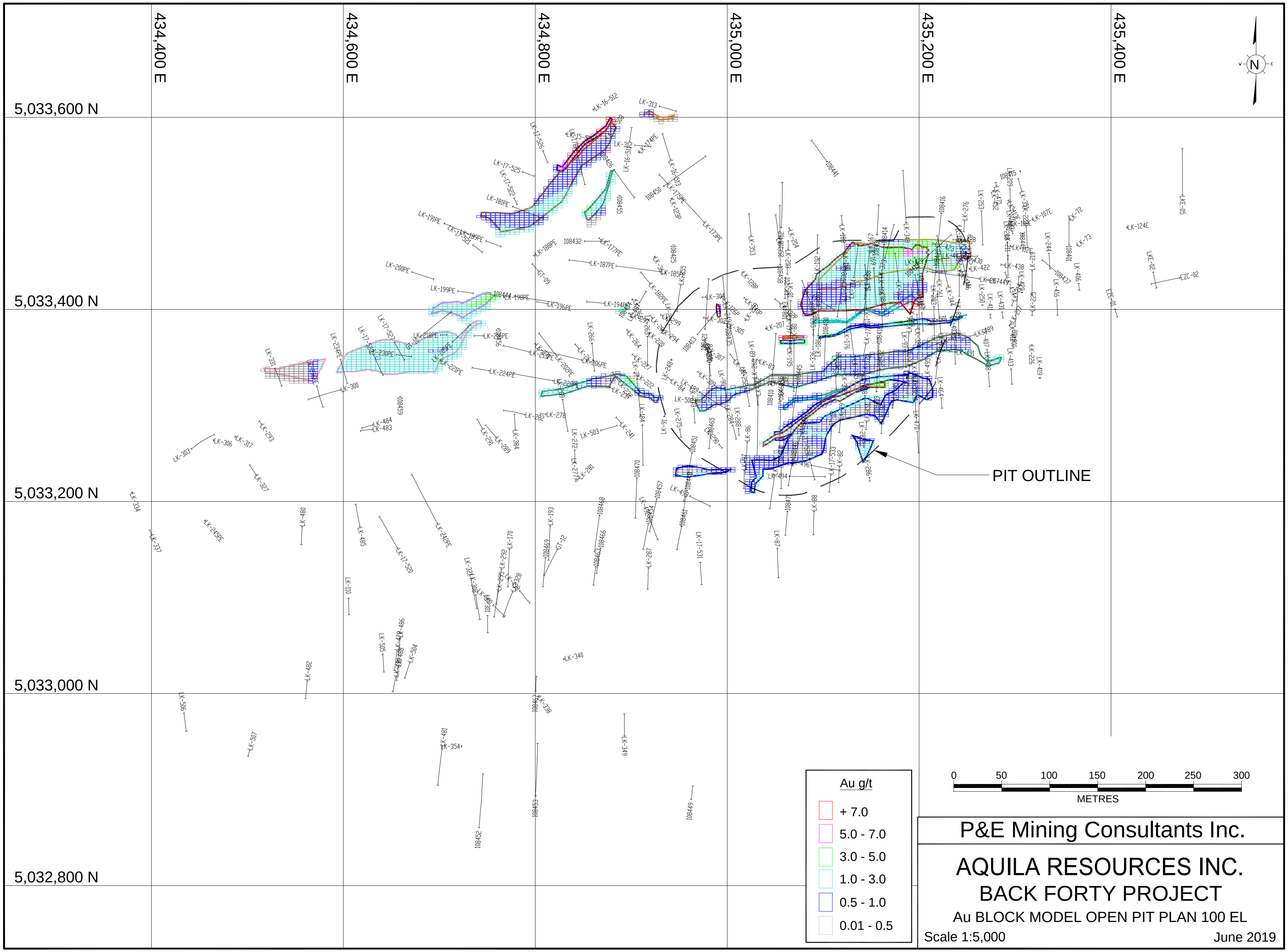
Scale 1:3,000 June 2019

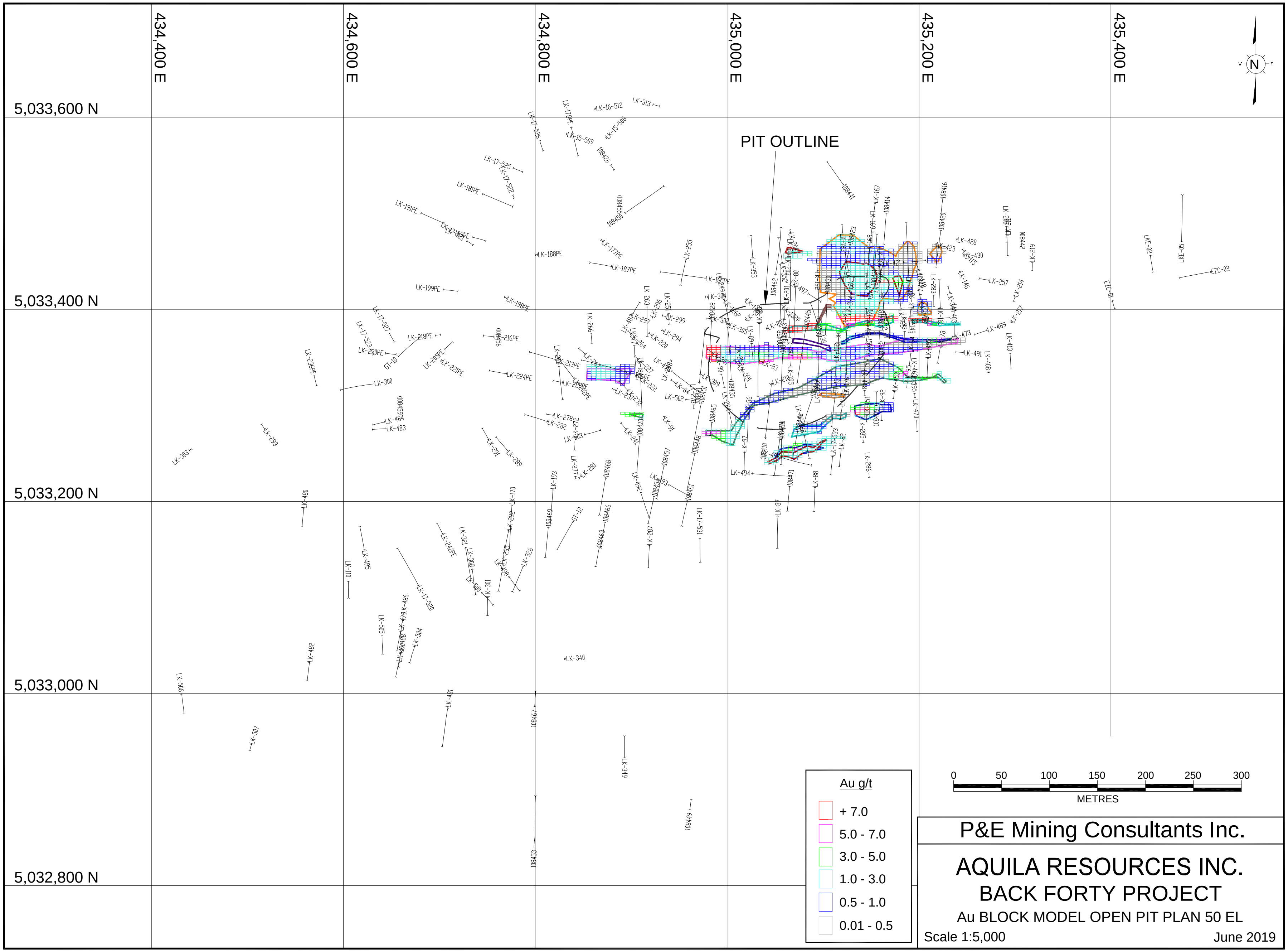


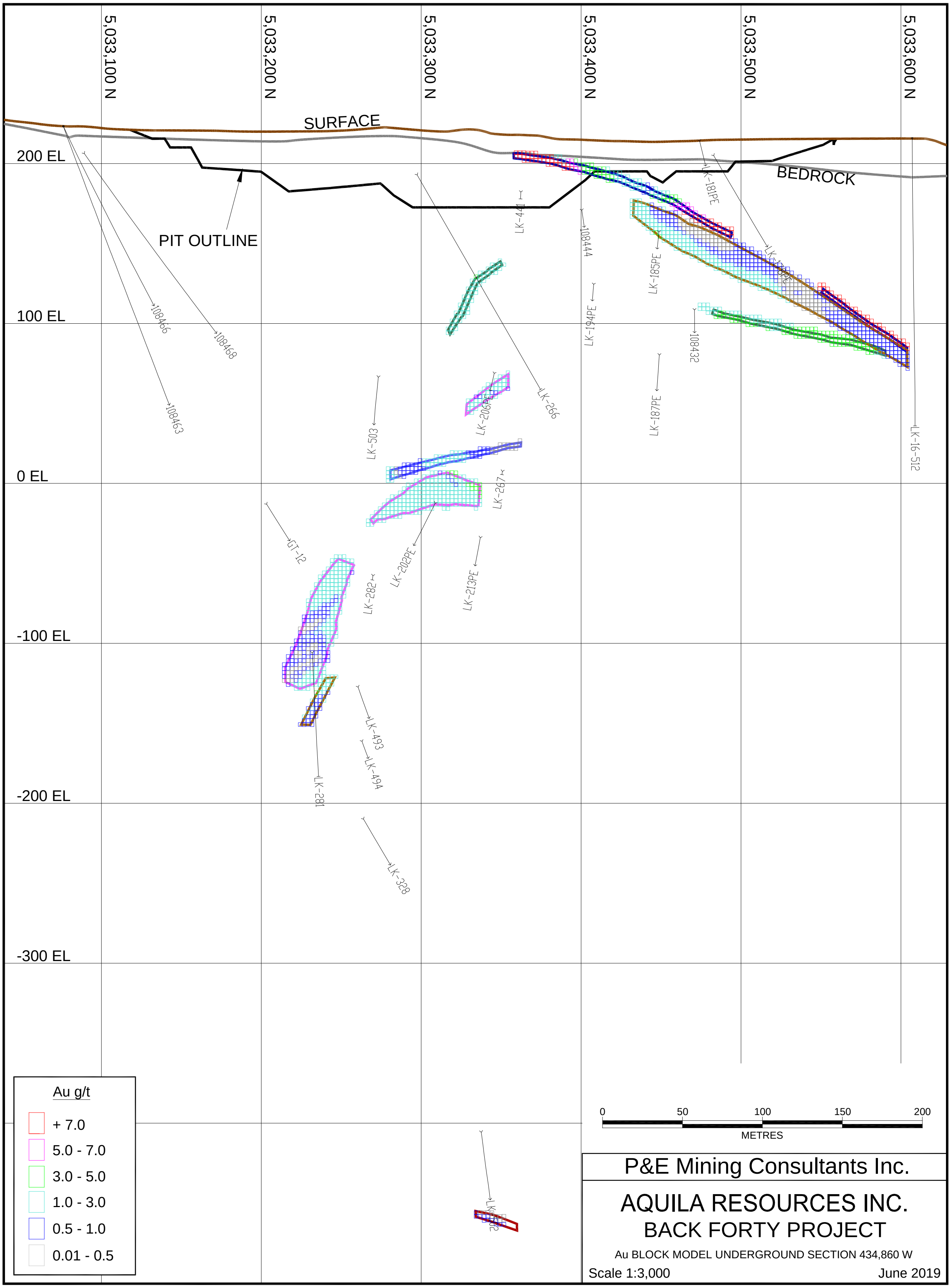
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-464
LK-491
LK-501
LK-78
LK-161
LK-495
LK-347
LK-142
LK-253
LK-276
108416
108420
108433
LK-115
LK-428
LK-423
LK-430
LK-283
LK-344
LK-372
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT SECTION 435,240 W
Scale 1:3,000
June 2019

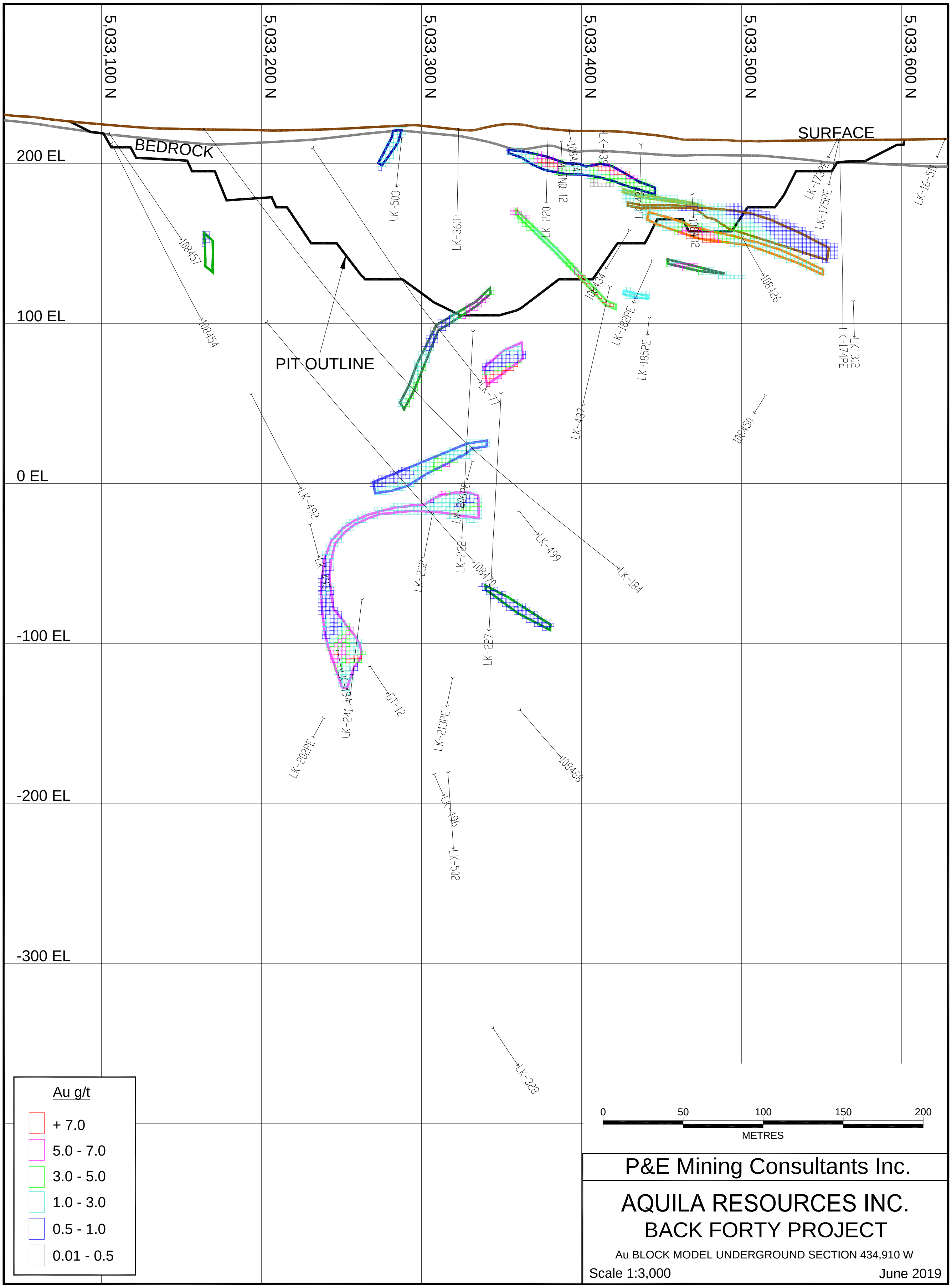

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT PLAN 200 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT PLAN 150 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT PLAN 100 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL OPEN PIT PLAN 50 EL
Scale 1:5,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND SECTION 434,860 W
Scale 1:3,000
June 2019

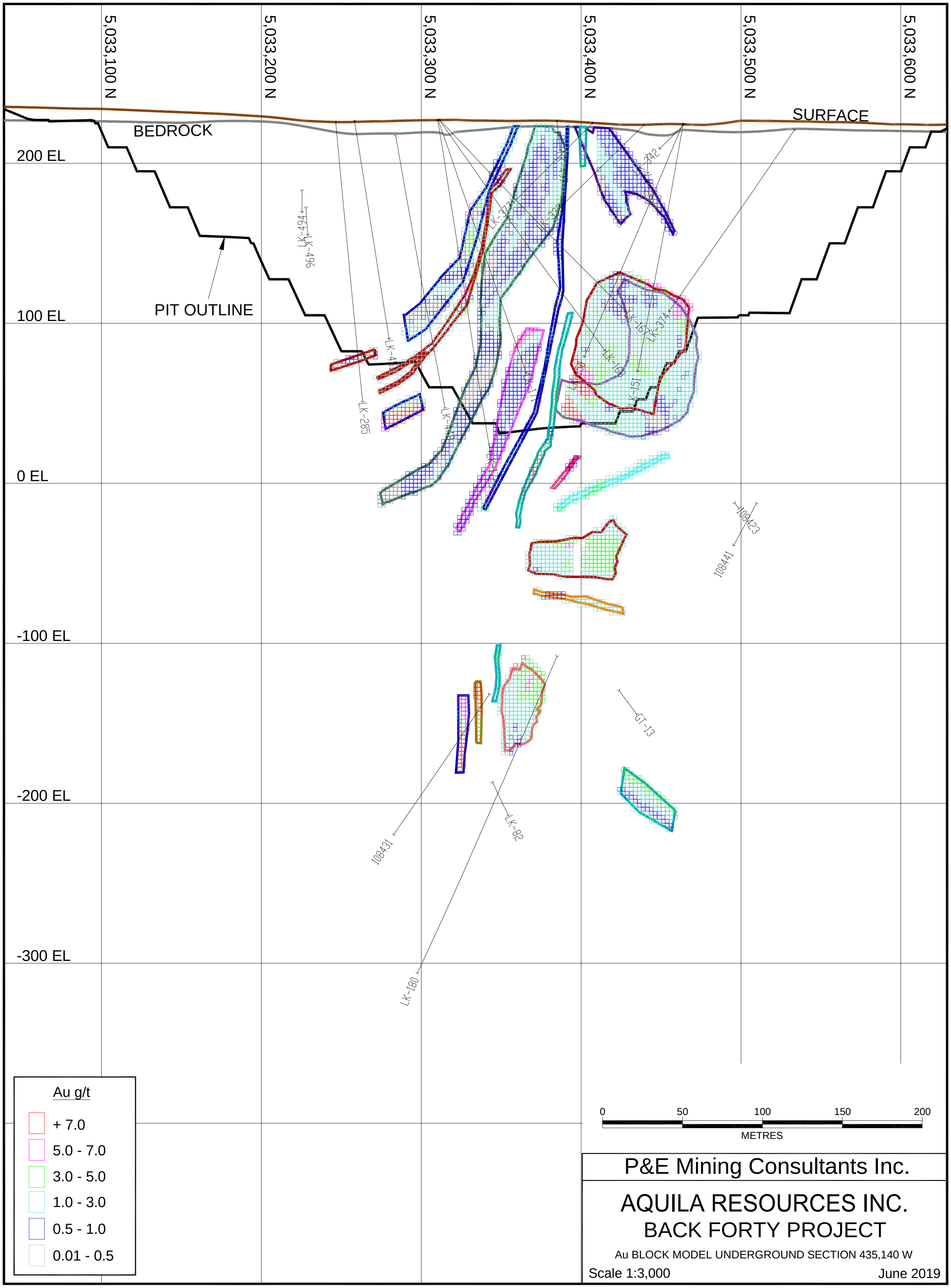


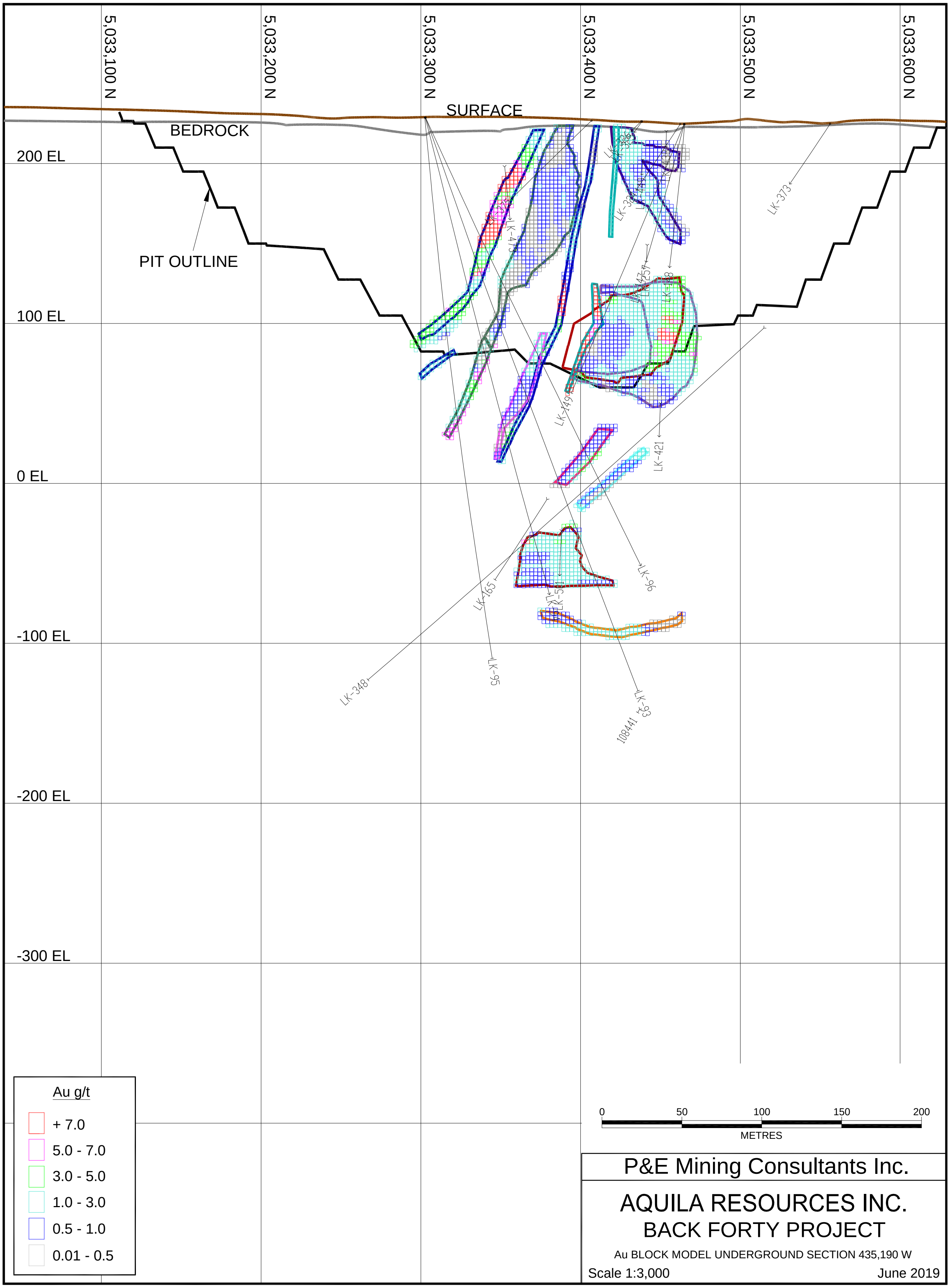
P&E Mining Consultants Inc.

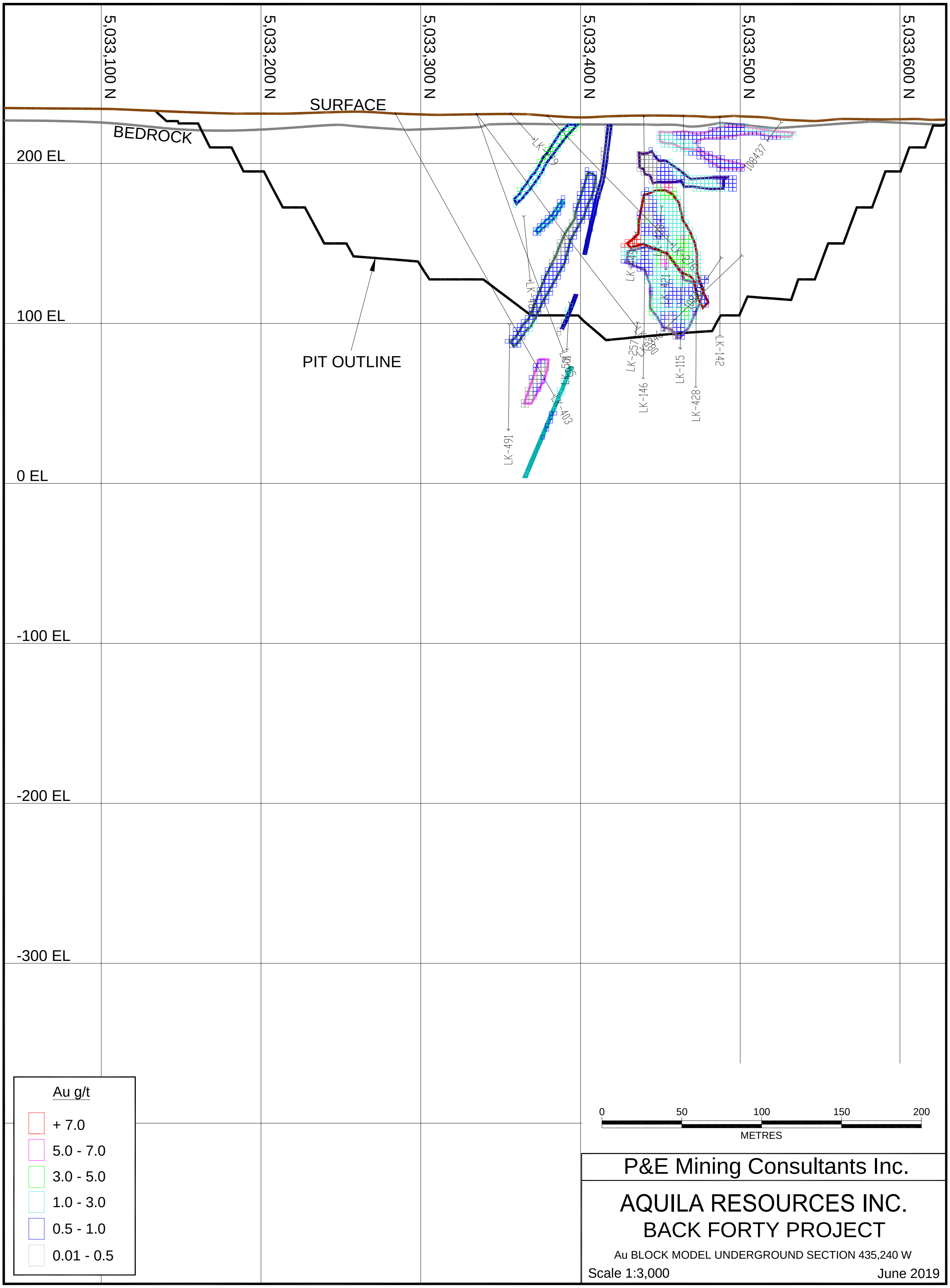
AQUILA RESOURCES INC.
BACK FORTY PROJECT

Au BLOCK MODEL UNDERGROUND SECTION 434,910 W

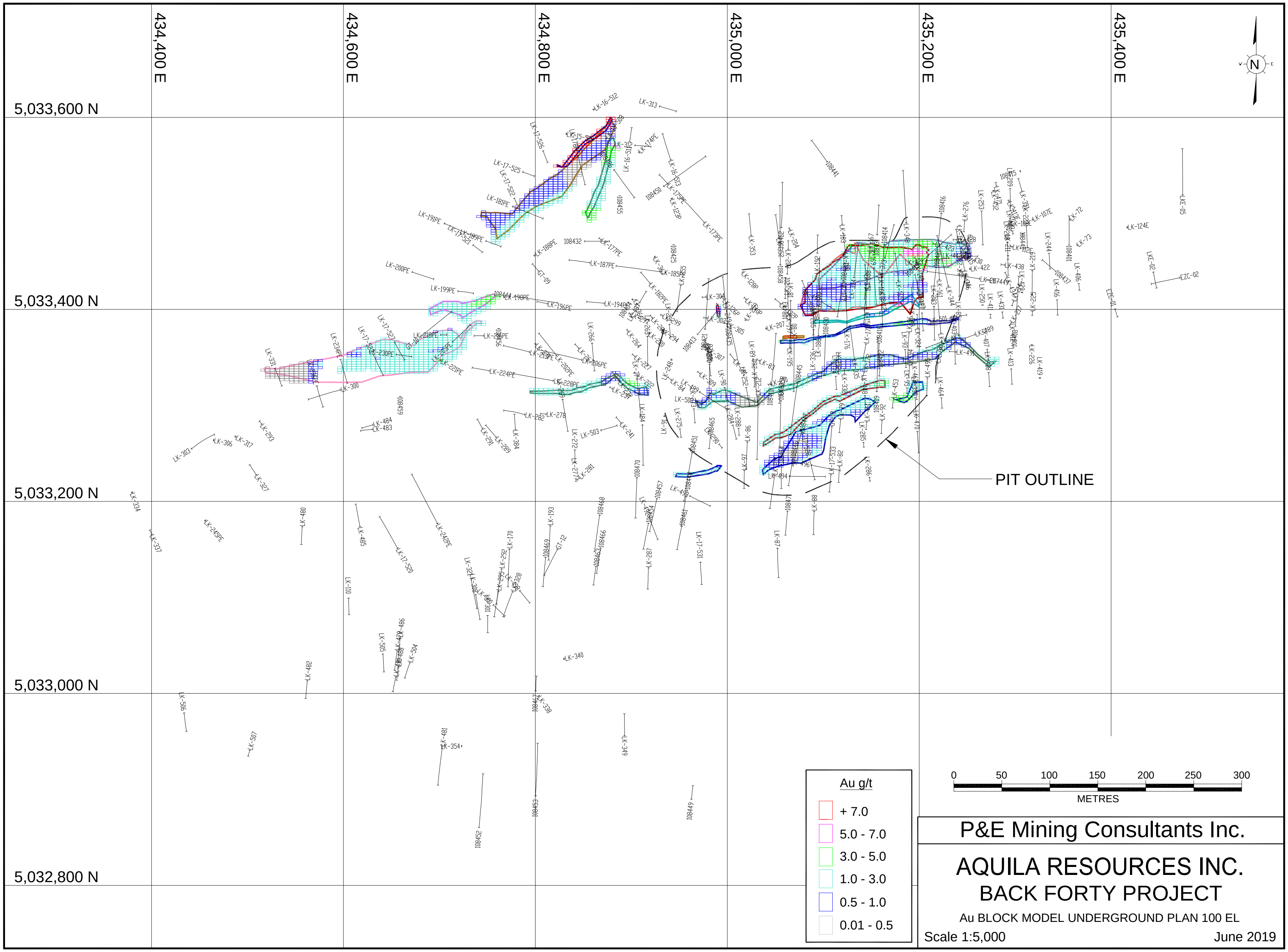
Scale 1:3,000 June 2019

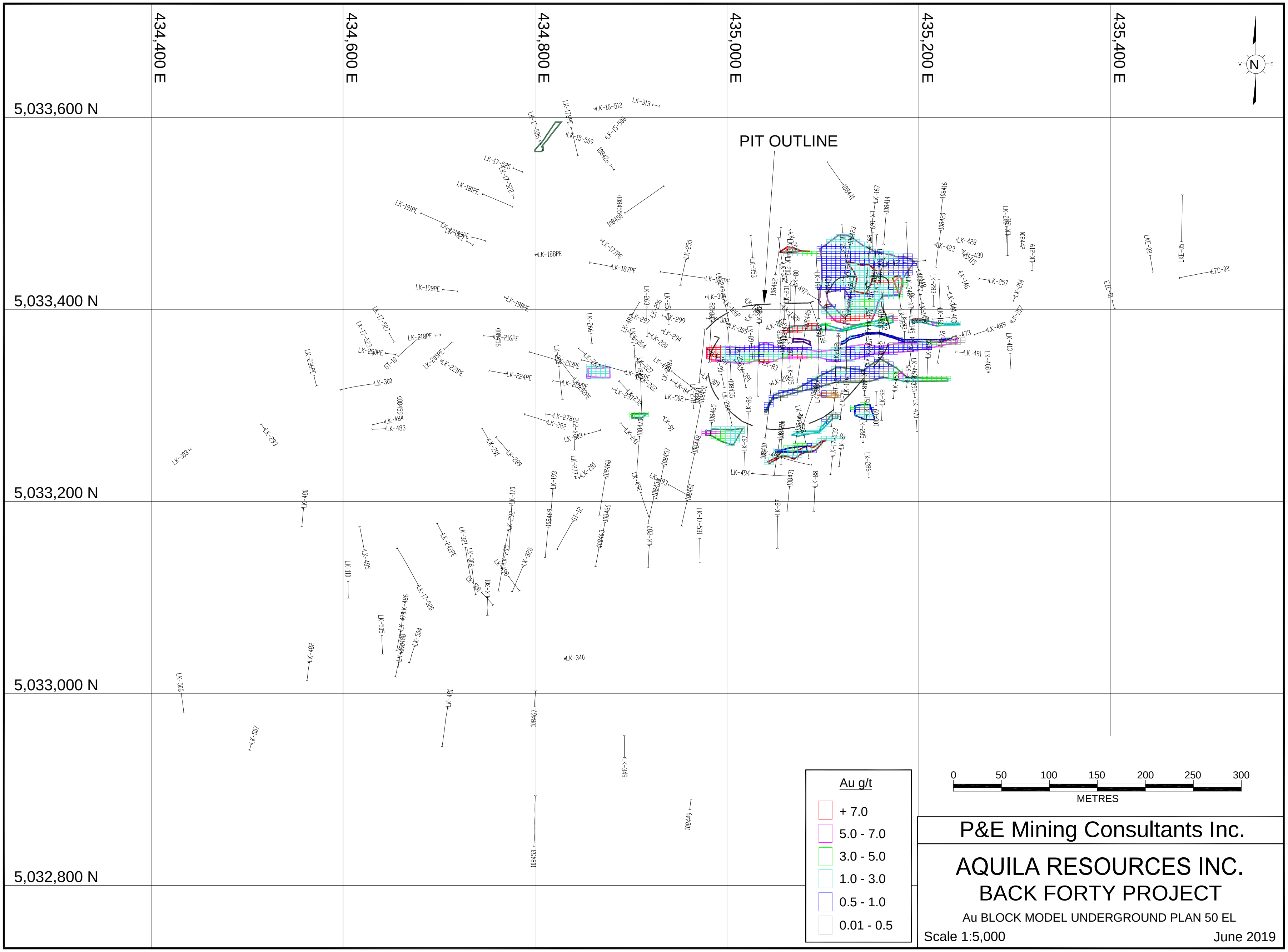

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
LK-494
LK-496
LK-285
LK-82
LK-180
108431
108441
108423
GT-13
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND SECTION 435,140 W
Scale 1:3,000
June 2019

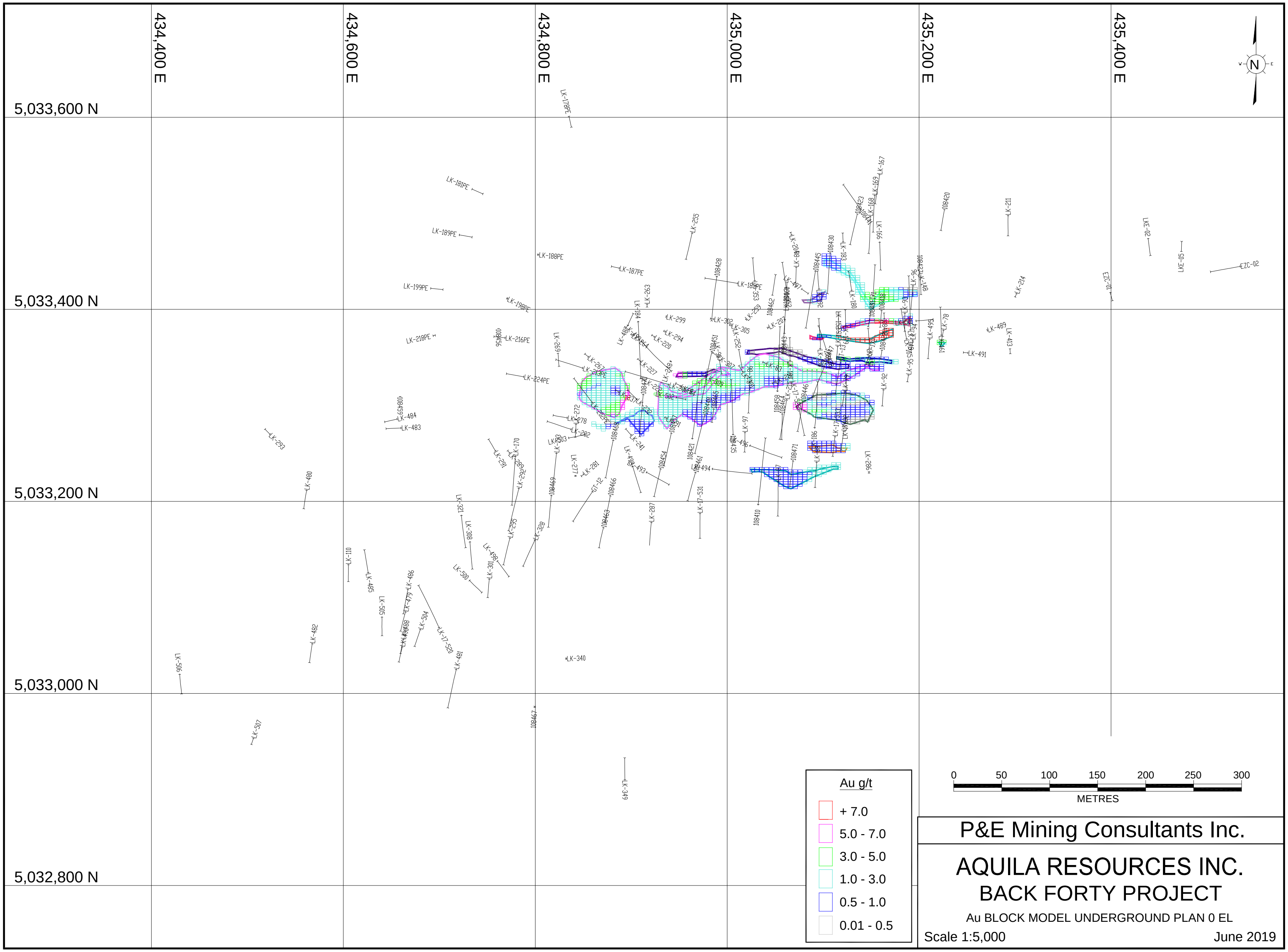

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
PIT OUTLINE
100 EL
0 EL
-100 EL
-200 EL
-300 EL
LK-373
LK-421
LK-96
LK-165
LK-348
LK-95
LK-93
108441
LK-149
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND SECTION 435,190 W
Scale 1:3,000
June 2019



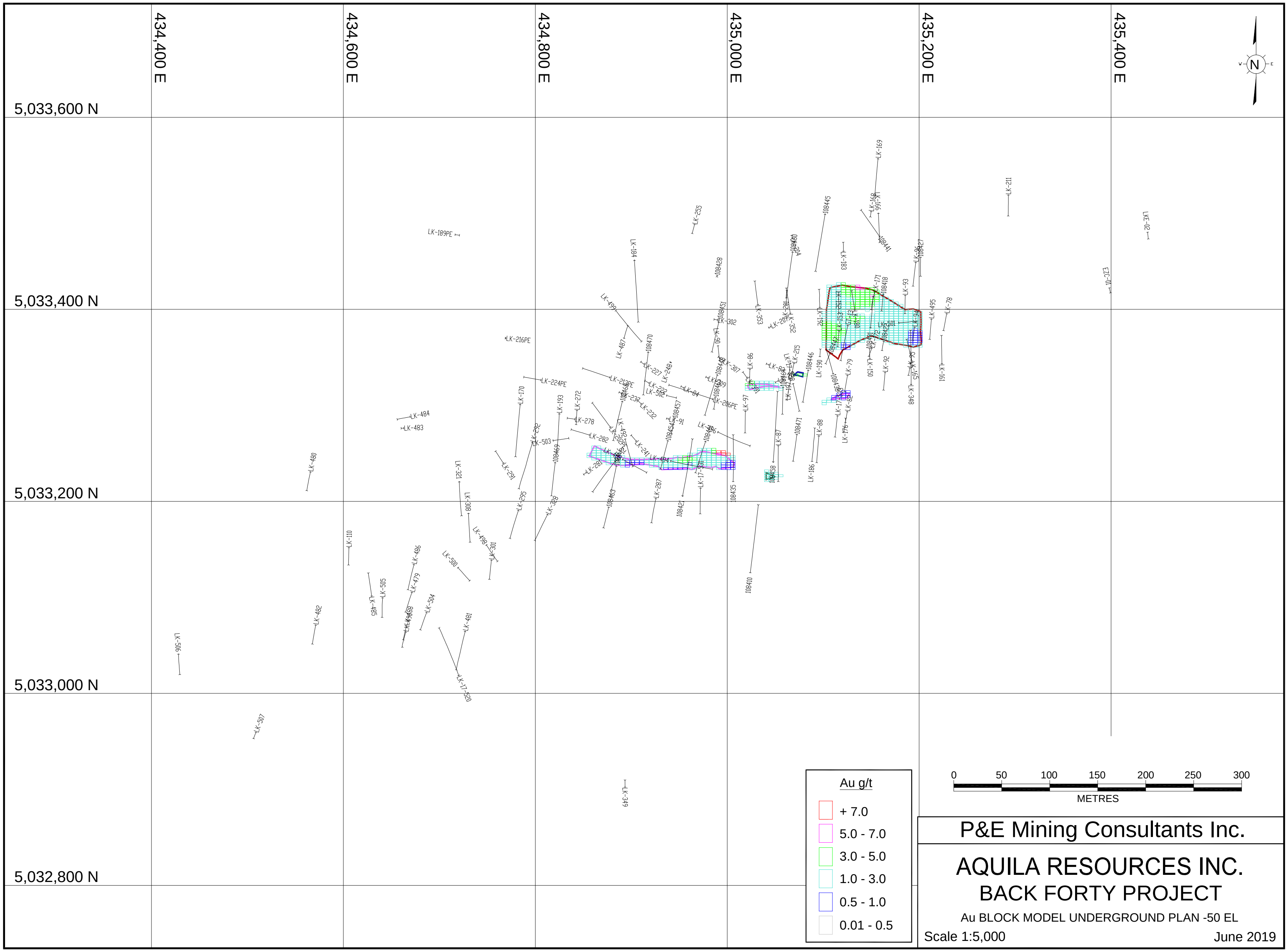
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-491
LK-403
LK-257
LK-146
LK-115
LK-428
LK-142
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND SECTION 435,240 W
Scale 1:3,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND PLAN 100 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
N
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND PLAN 50 EL
Scale 1:5,000
June 2019

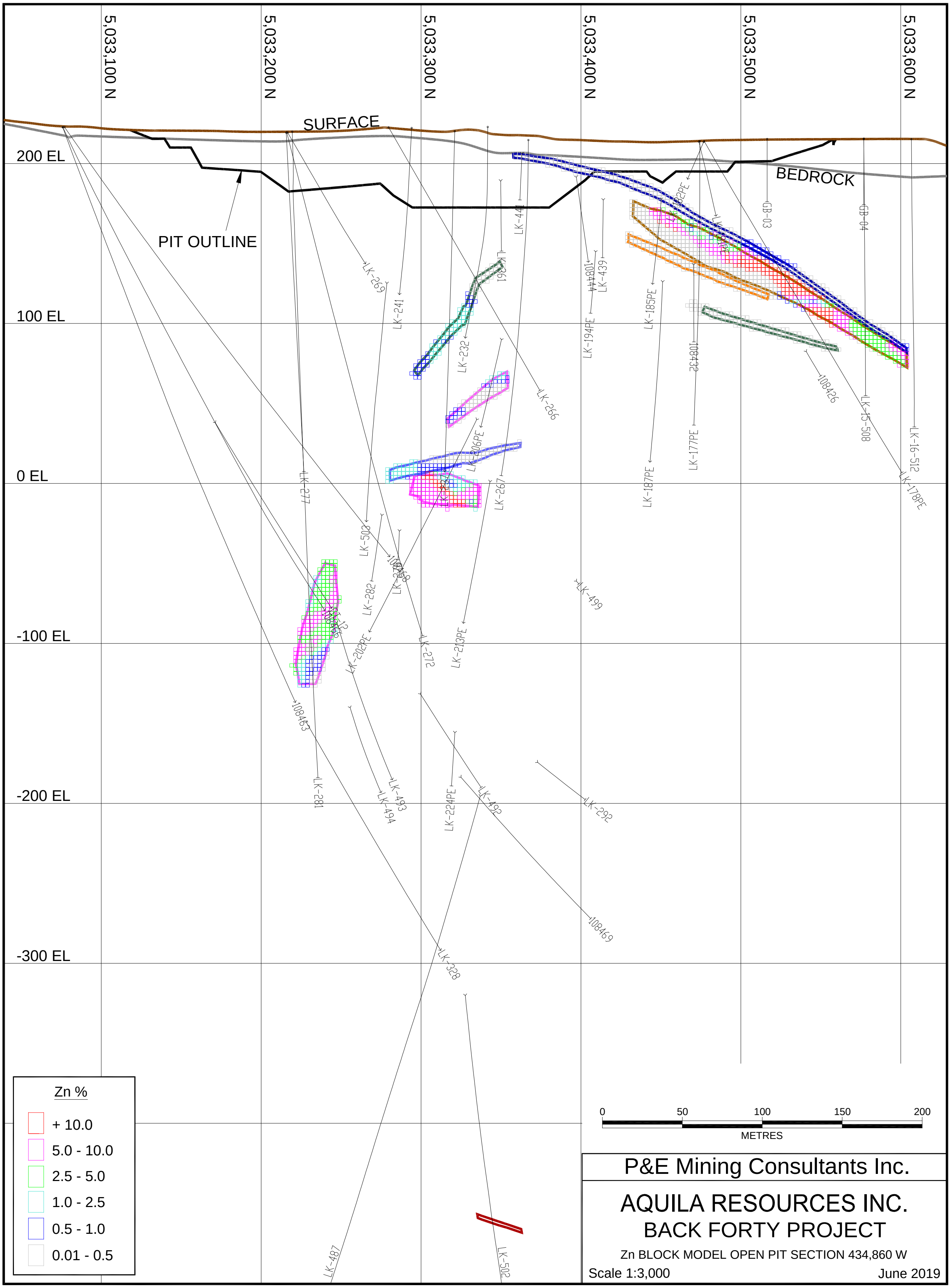


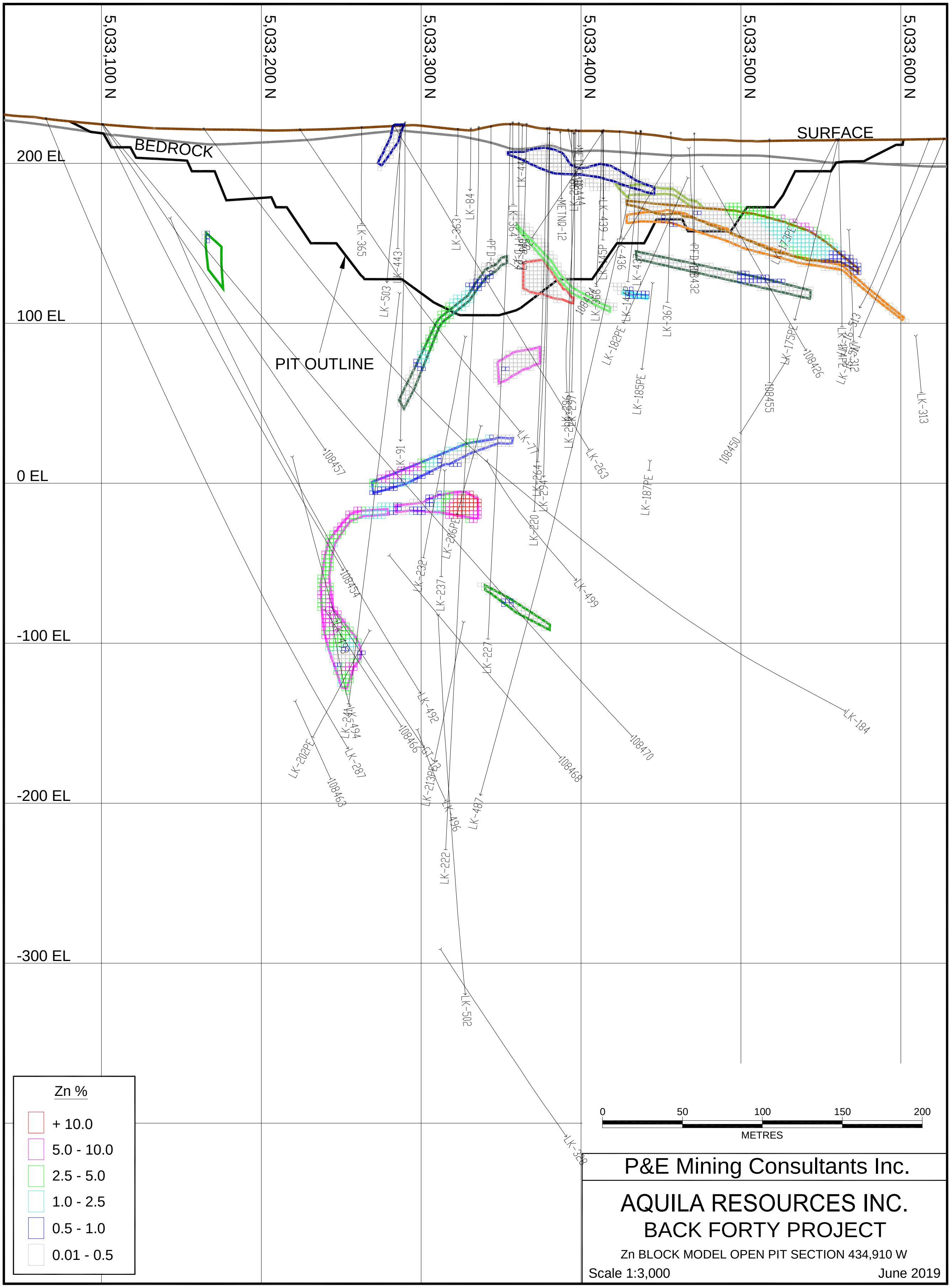
434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND PLAN 0 EL
Scale 1:5,000
June 2019

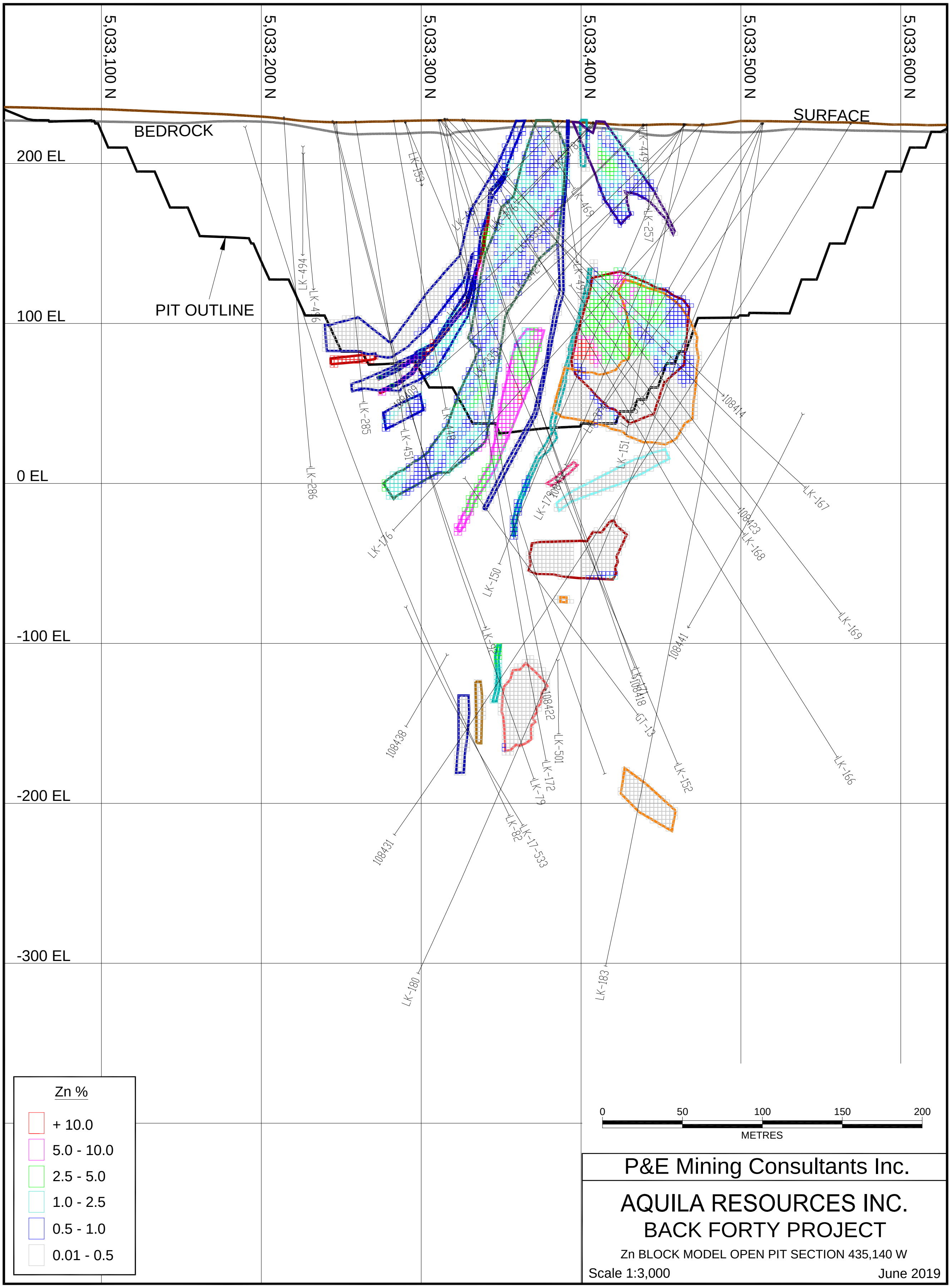


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
Au g/t
+ 7.0
5.0 - 7.0
3.0 - 5.0
1.0 - 3.0
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Au BLOCK MODEL UNDERGROUND PLAN -50 EL
Scale 1:5,000
June 2019

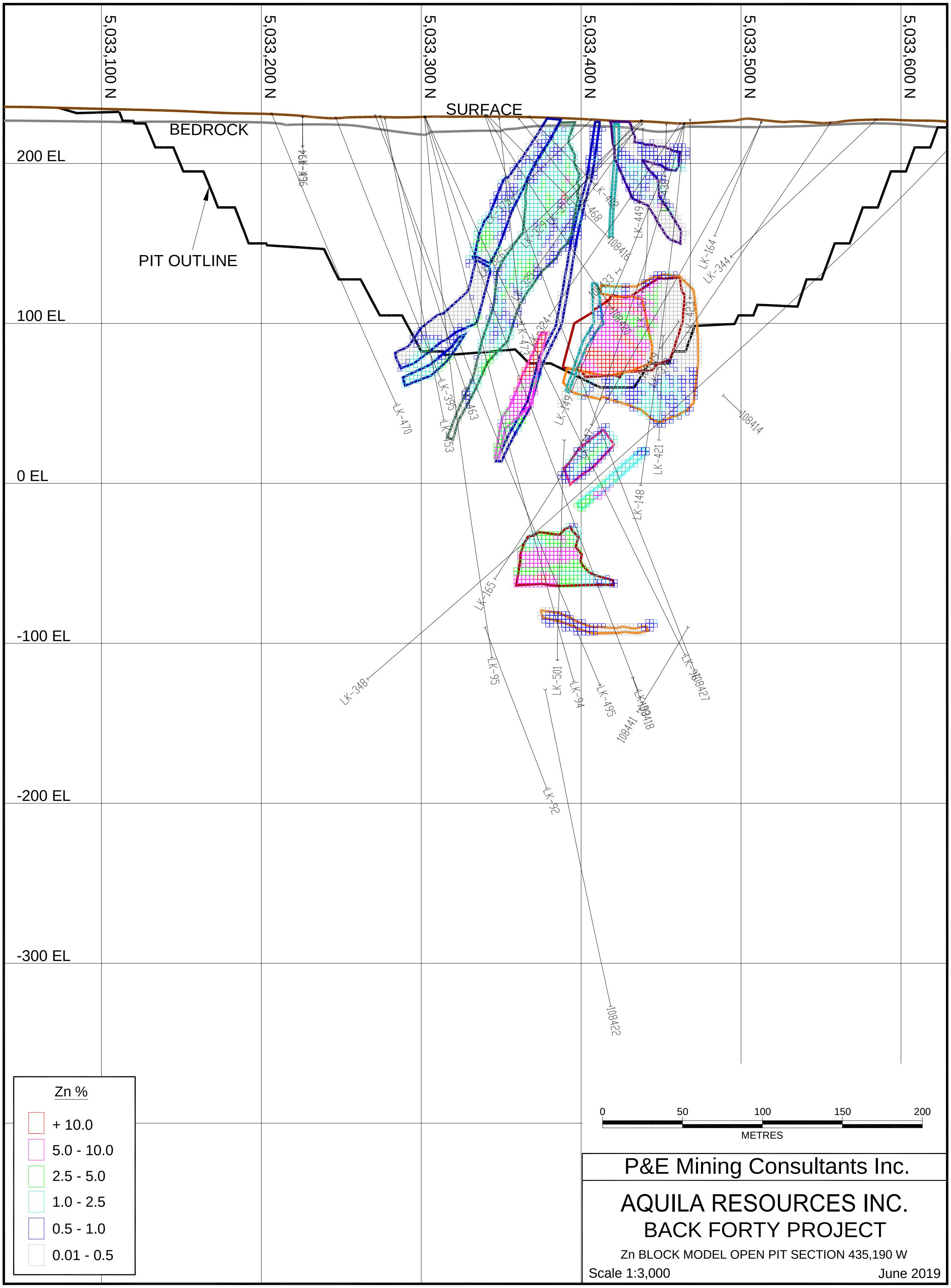
APPENDIX H ZN BLOCK MODEL CROSS SECTIONS AND PLANS

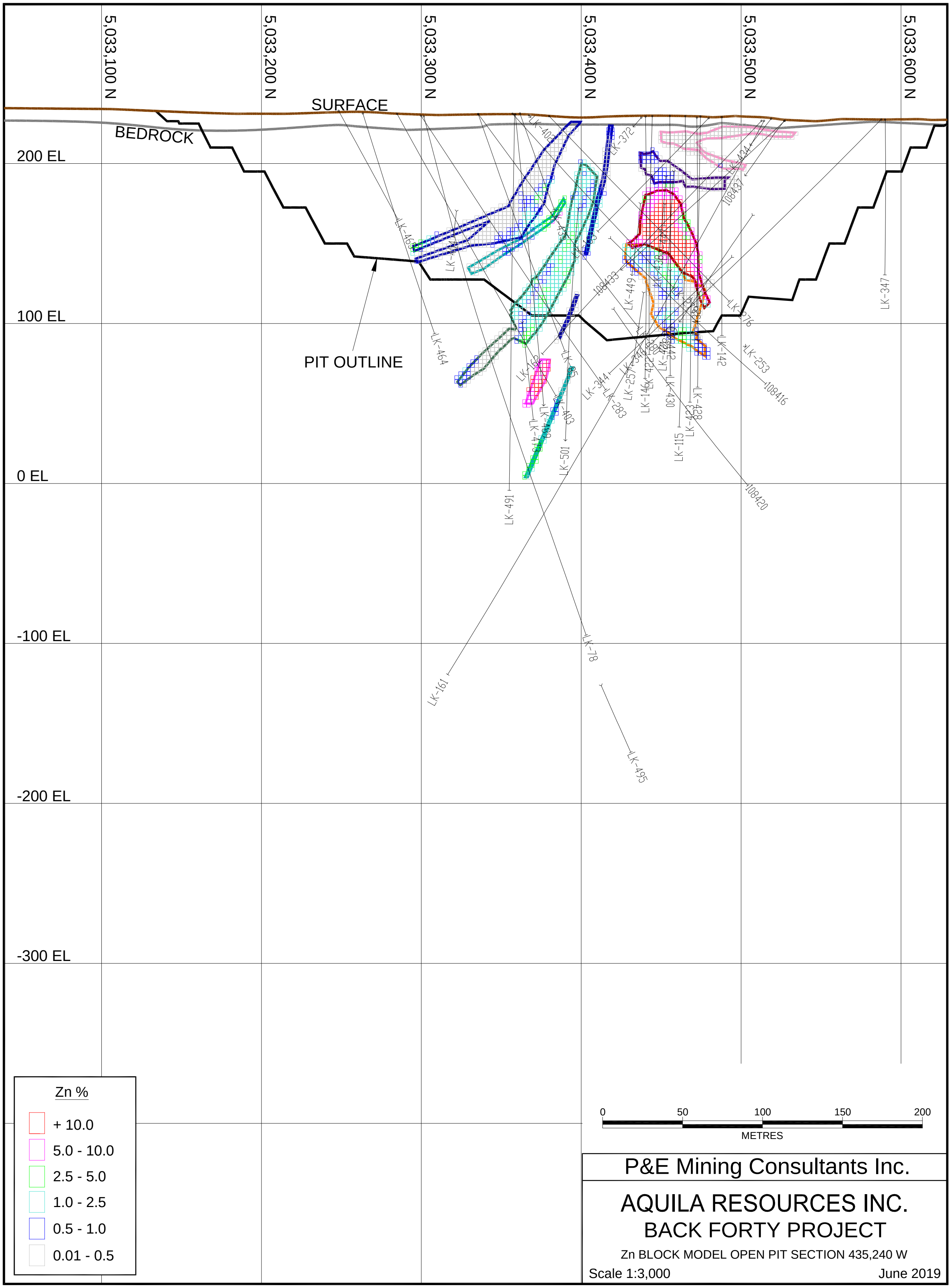

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
SURFACE
BEDROCK
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT SECTION 434,860 W
Scale 1:3,000
June 2019

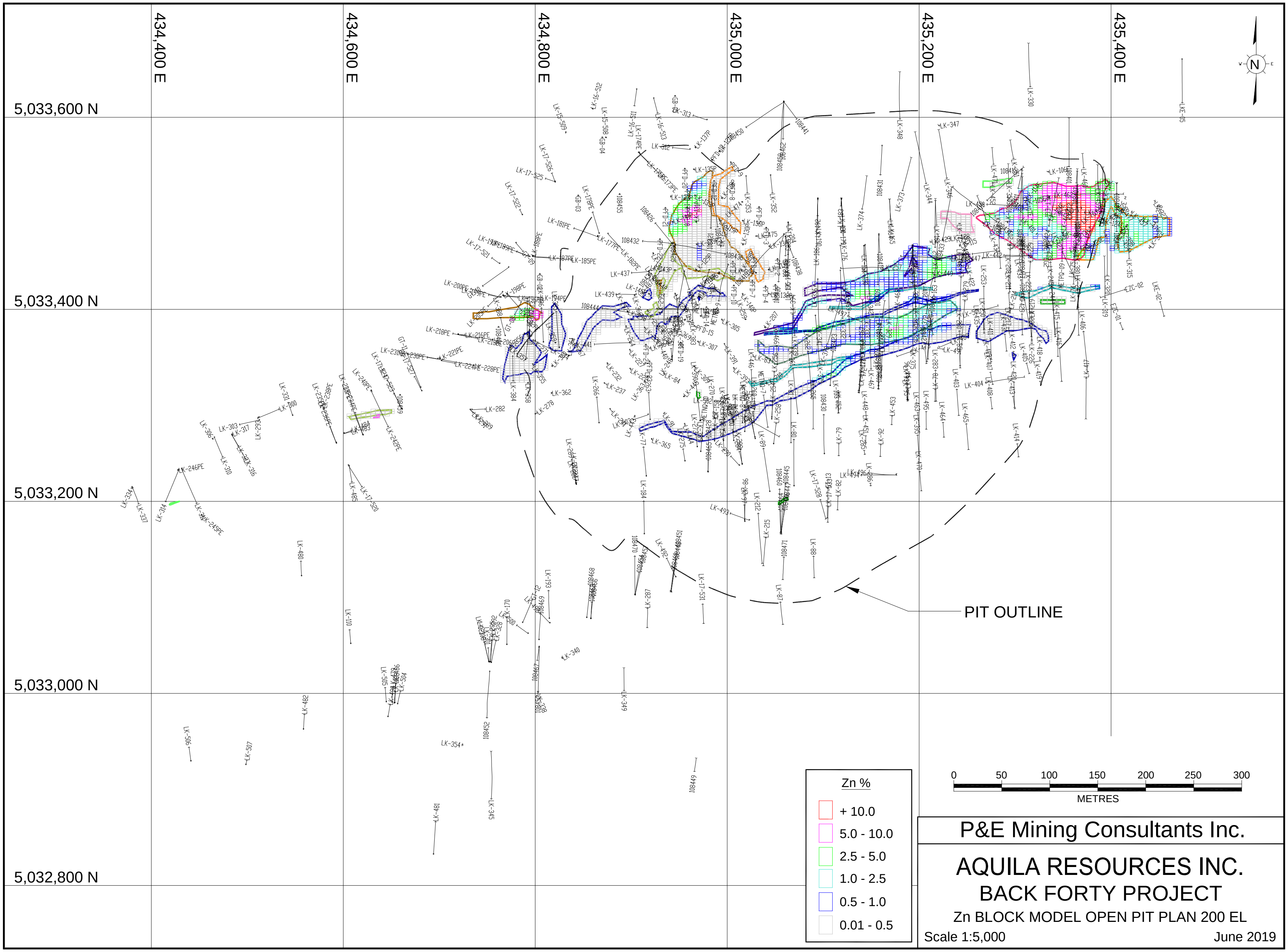

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT SECTION 434,910 W
Scale 1:3,000
June 2019

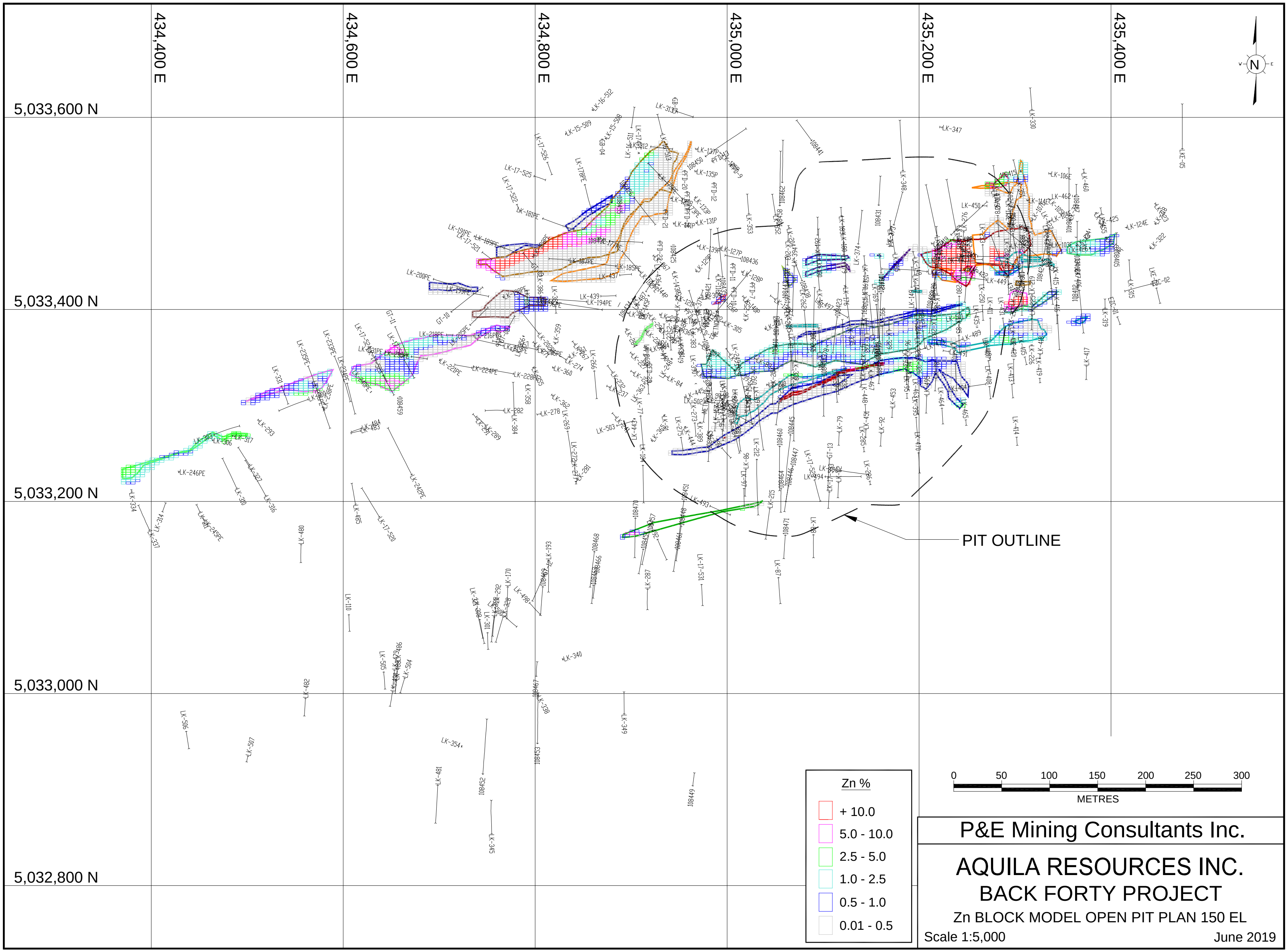


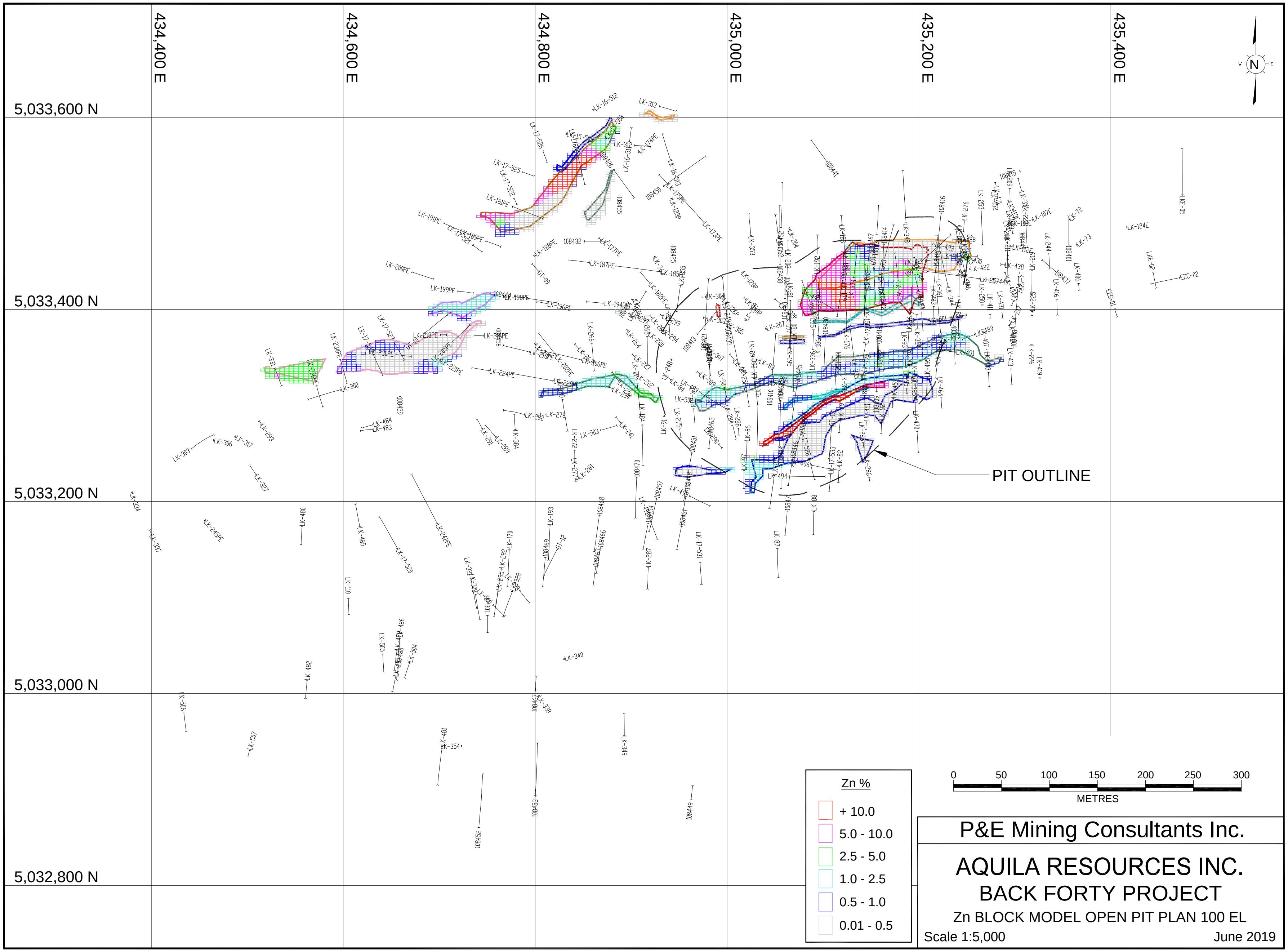
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
PIT OUTLINE
0 EL
-100 EL
-200 EL
-300 EL
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT SECTION 435,140 W
Scale 1:3,000
June 2019

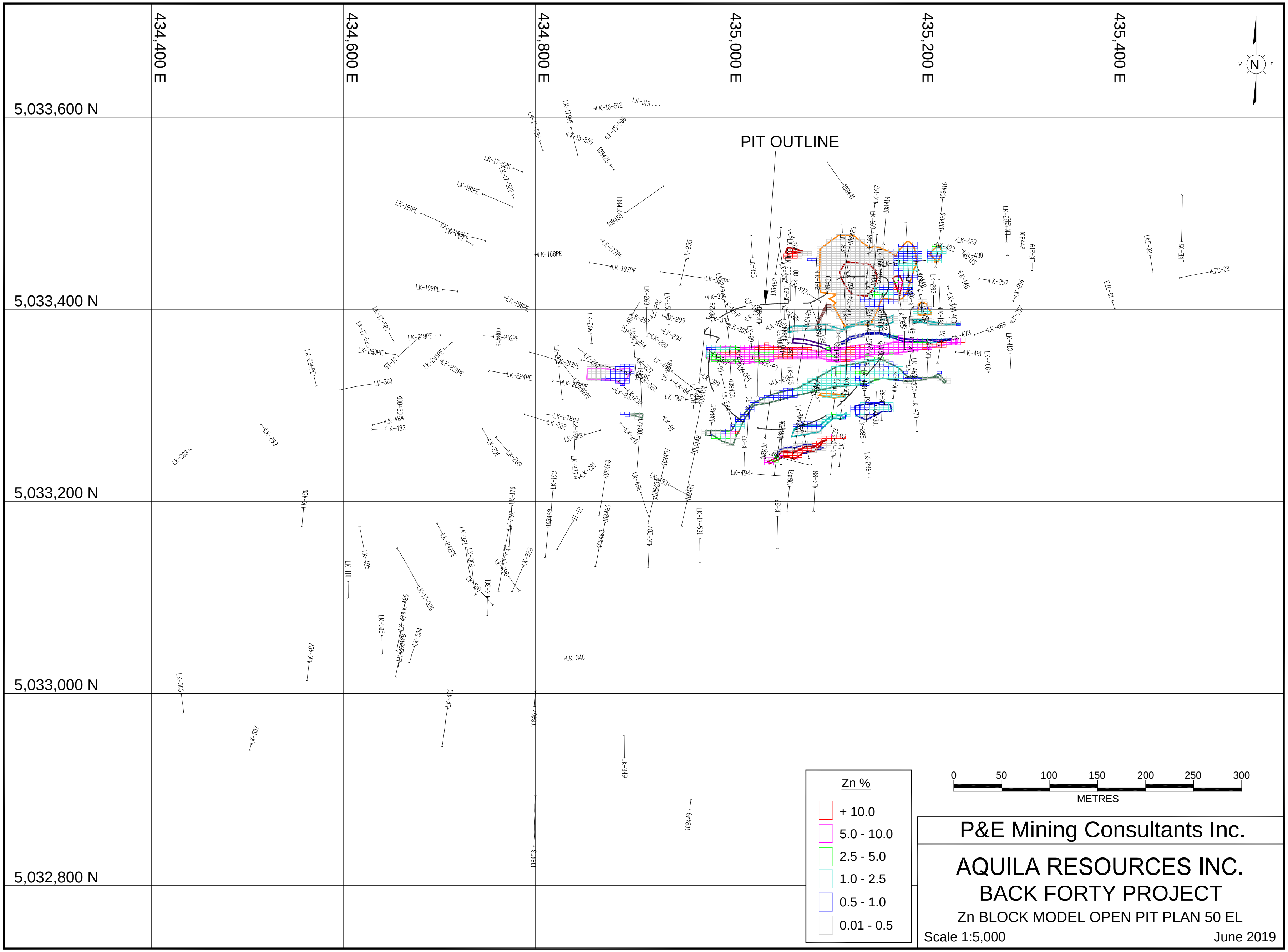

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT SECTION 435,190 W
Scale 1:3,000
June 2019

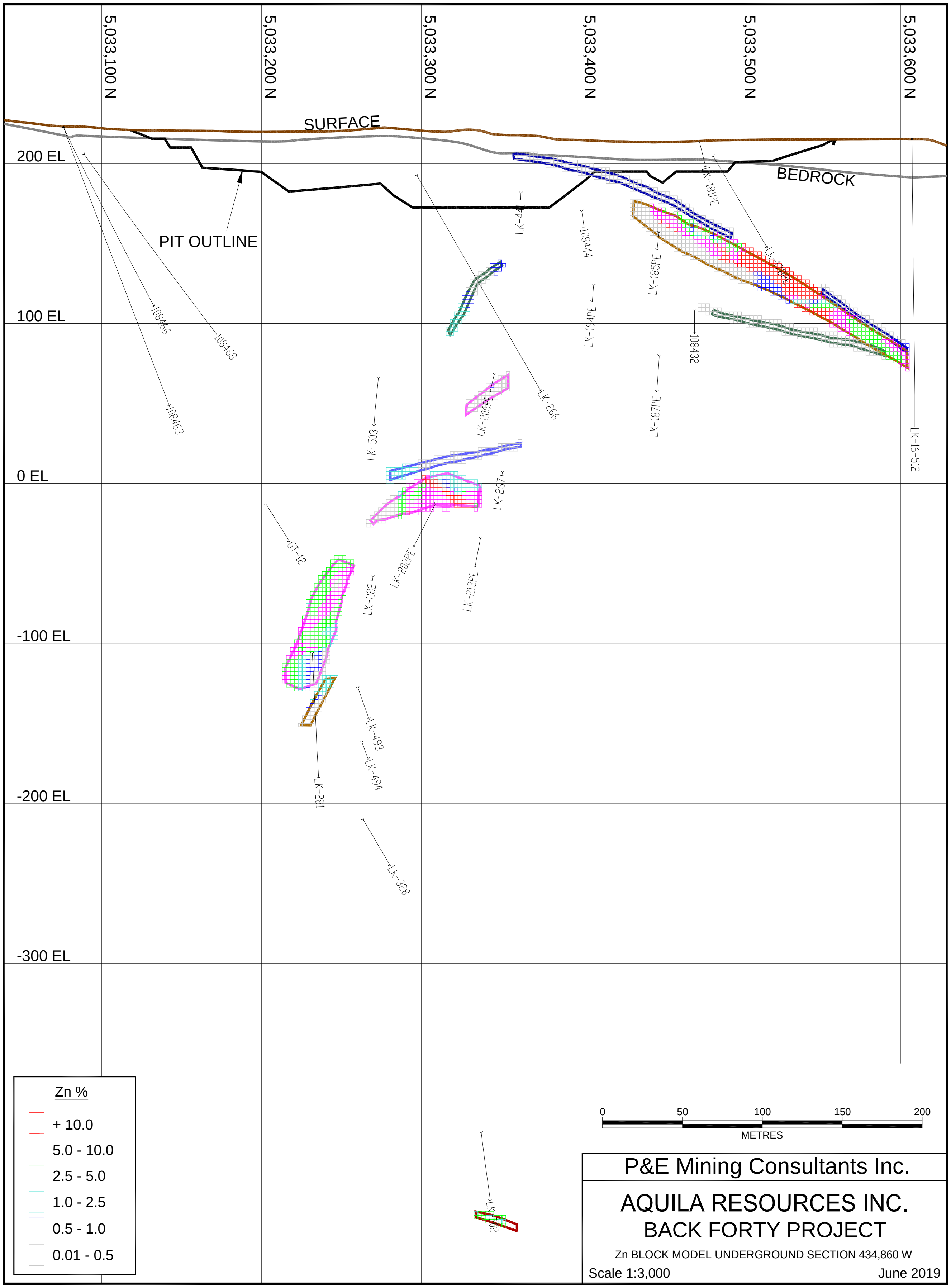

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-464
LK-78
LK-161
LK-495
LK-491
LK-501
LK-347
LK-142
LK-253
108416
108420
LK-115
LK-430
LK-372
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT SECTION 435,240 W
Scale 1:3,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT PLAN 200 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT PLAN 150 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT PLAN 100 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
N
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL OPEN PIT PLAN 50 EL
Scale 1:5,000
June 2019

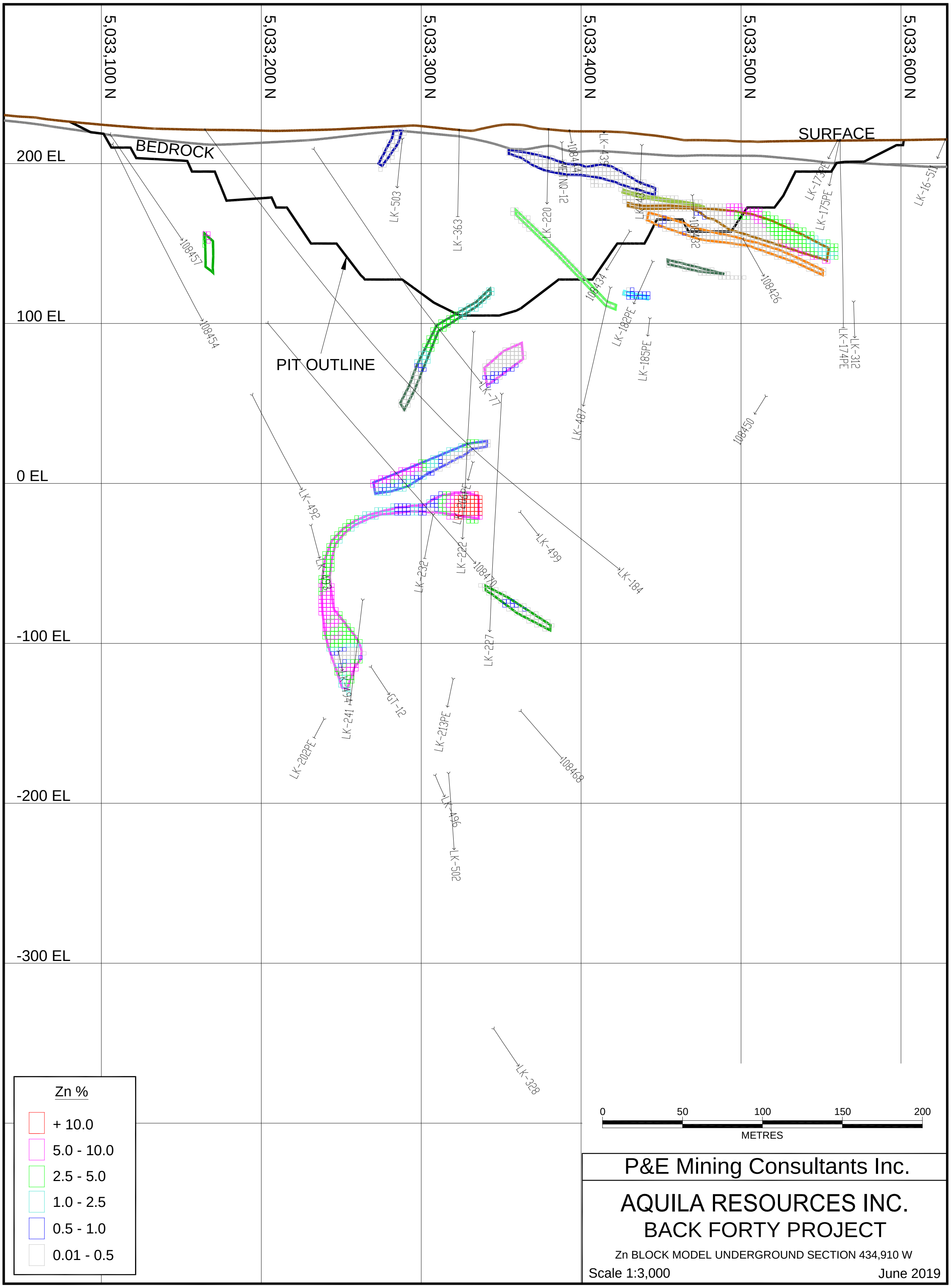


P&E Mining Consultants Inc.

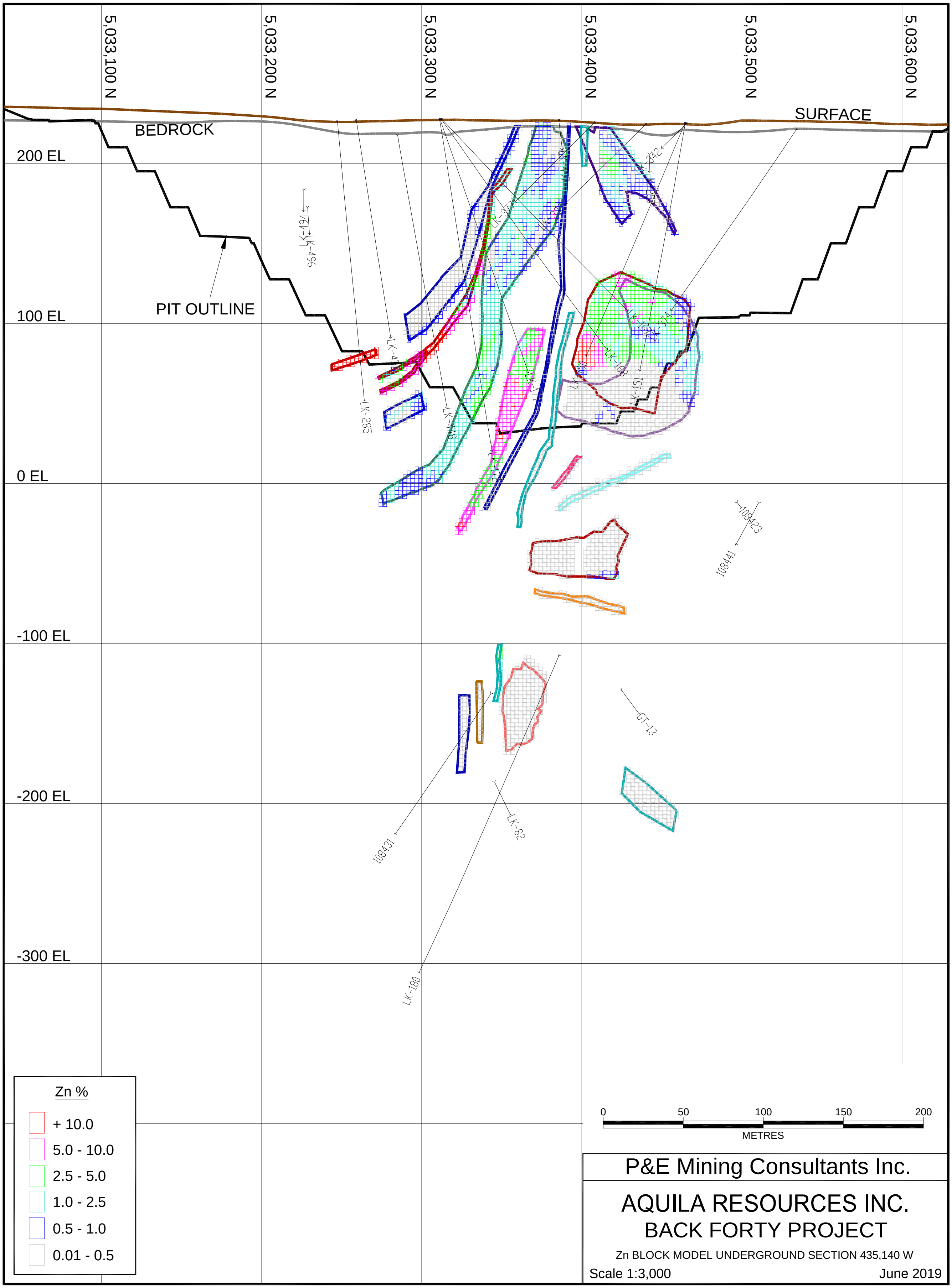
AQUILA RESOURCES INC.
BACK FORTY PROJECT

Zn BLOCK MODEL UNDERGROUND SECTION 434,860 W

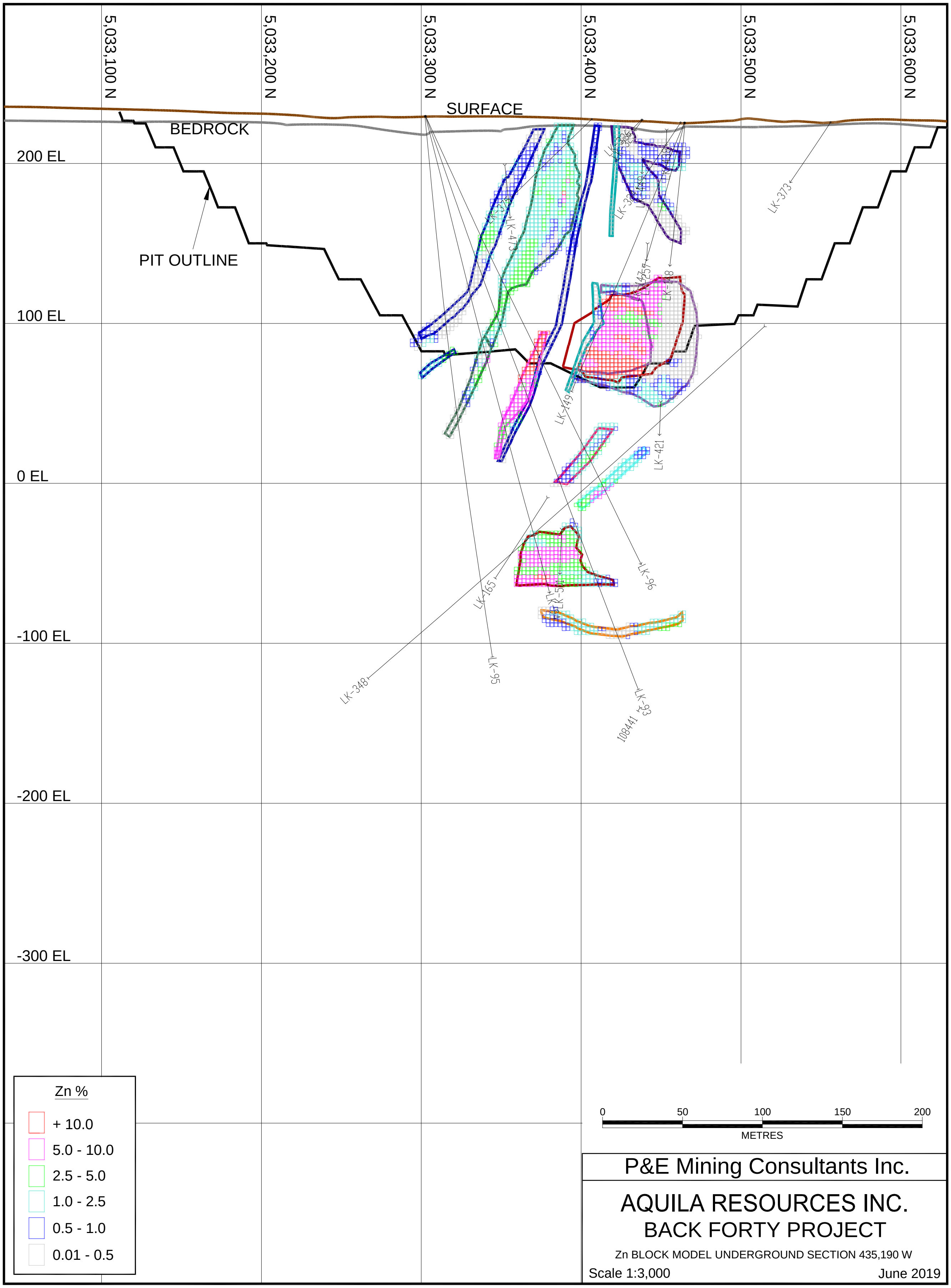
Scale 1:3,000 June 2019

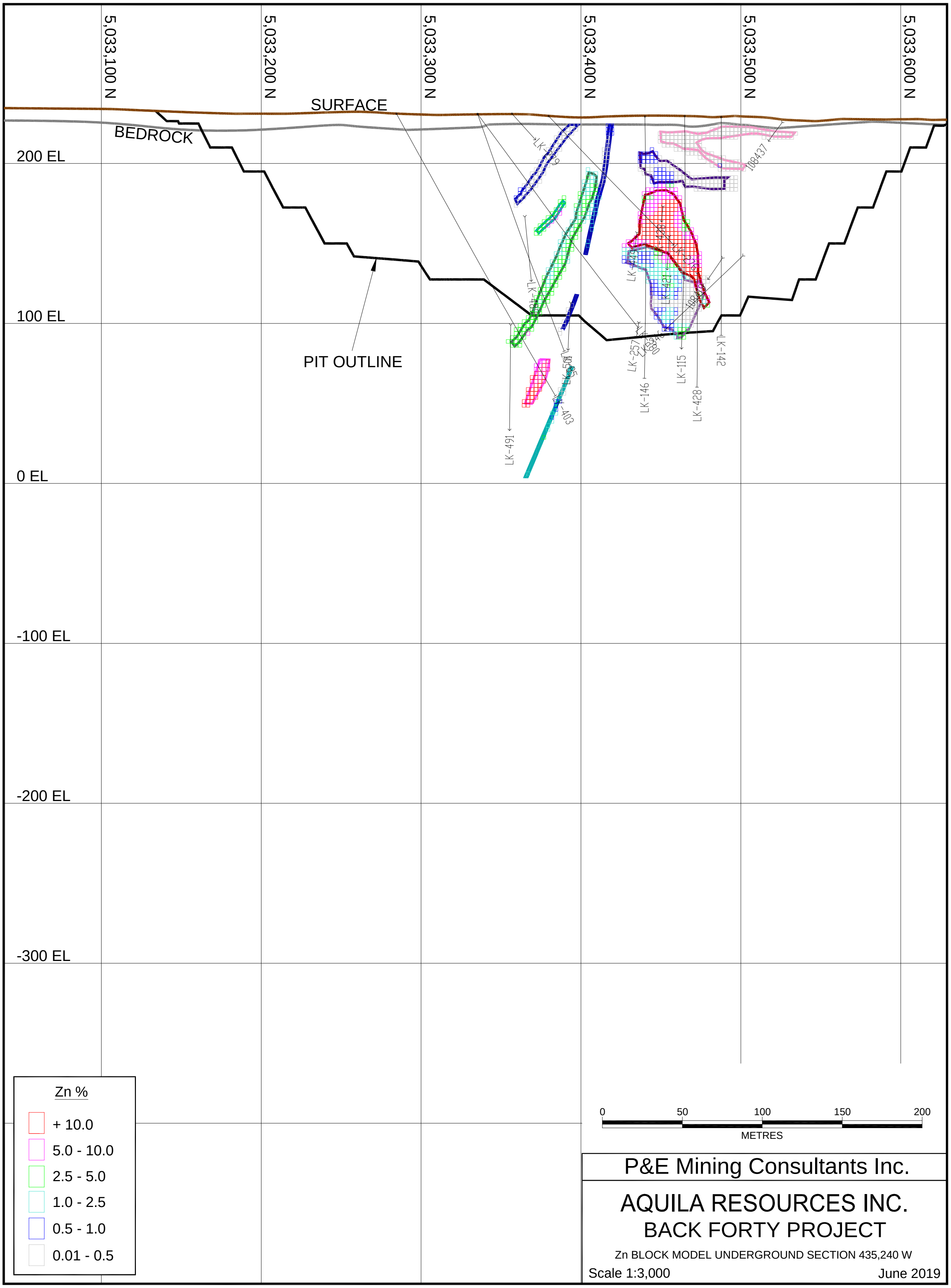


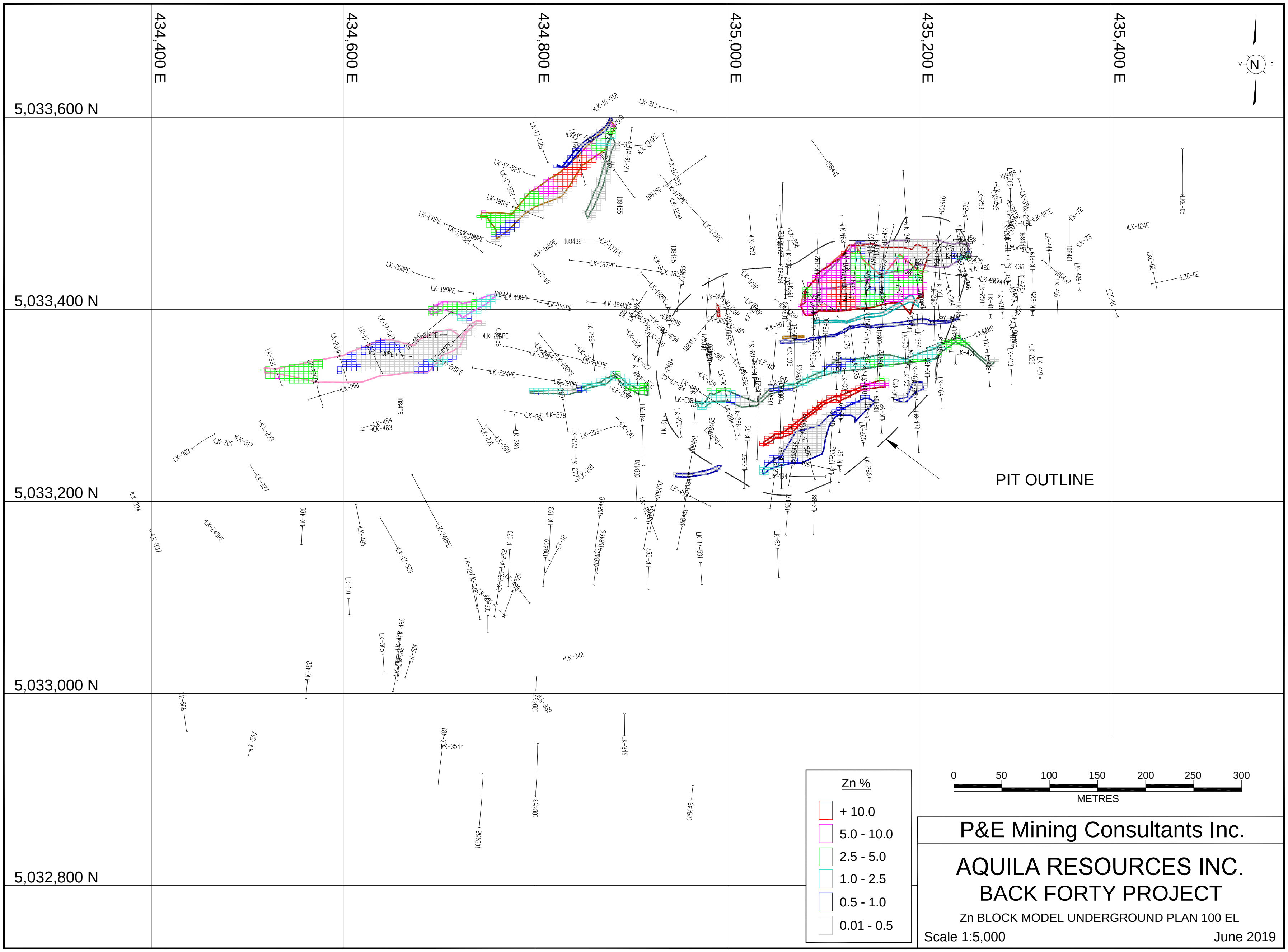
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND SECTION 434,910 W
Scale 1:3,000
June 2019

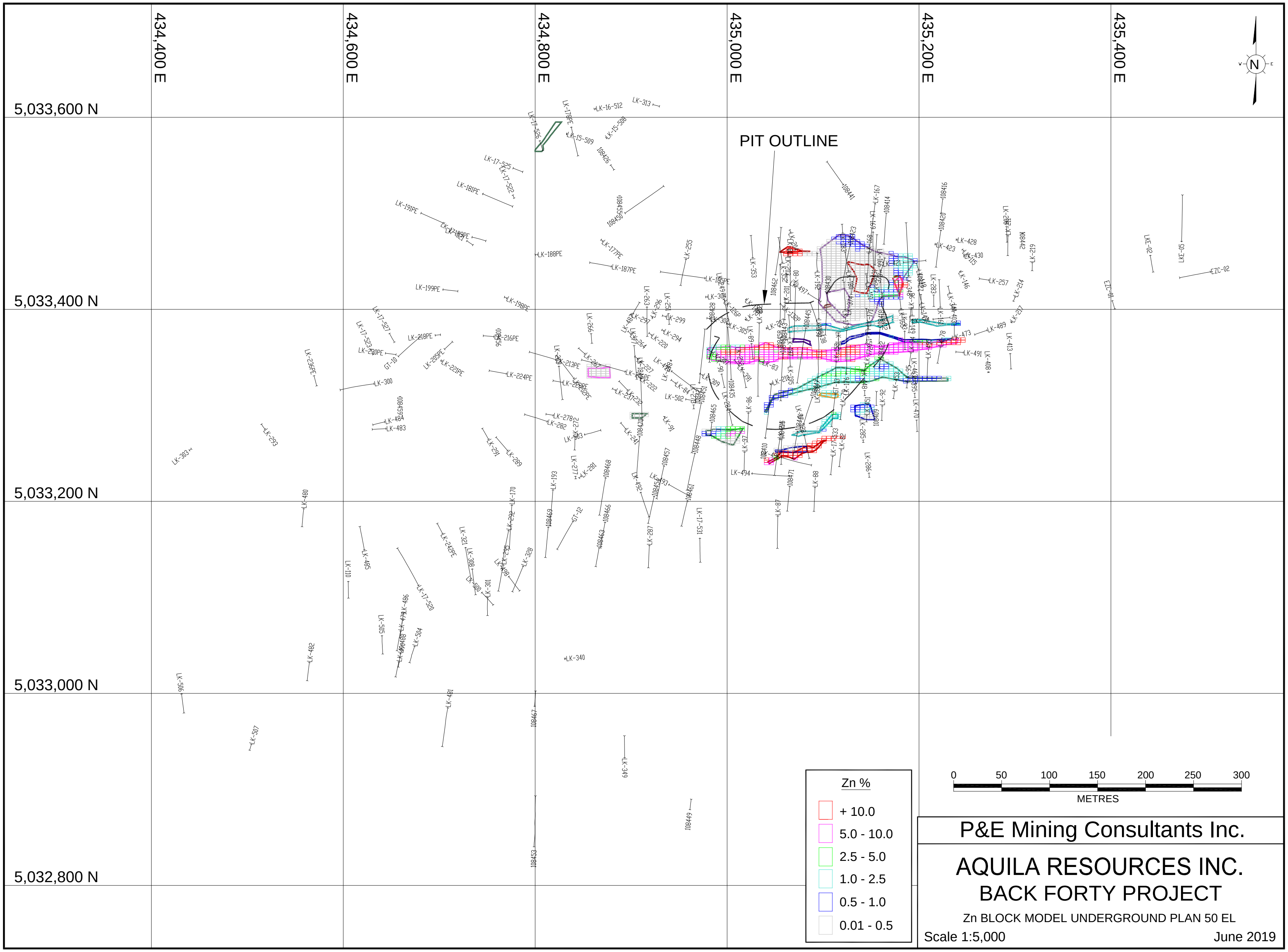


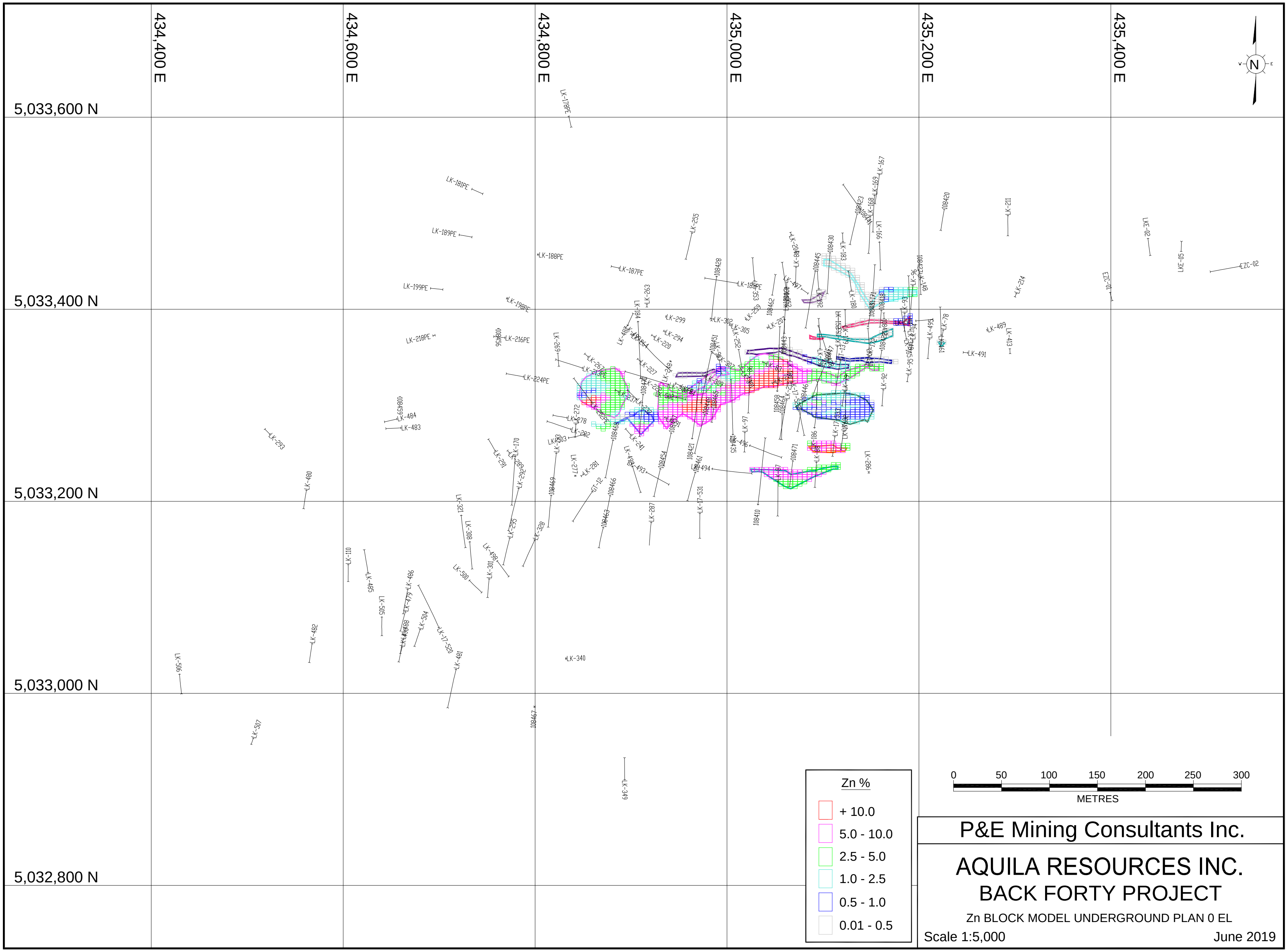
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
BEDROCK
SURFACE
200 EL
100 EL
PIT OUTLINE
0 EL
-100 EL
-200 EL
-300 EL
LK-494
LK-496
LK-285
LK-82
LK-180
LK-151
GT-13
108423
108441
108431
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND SECTION 435,140 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-373
LK-421
LK-96
LK-165
LK-348
LK-95
LK-93
108441
LK-149
LK-419
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND SECTION 435,190 W
Scale 1:3,000
June 2019

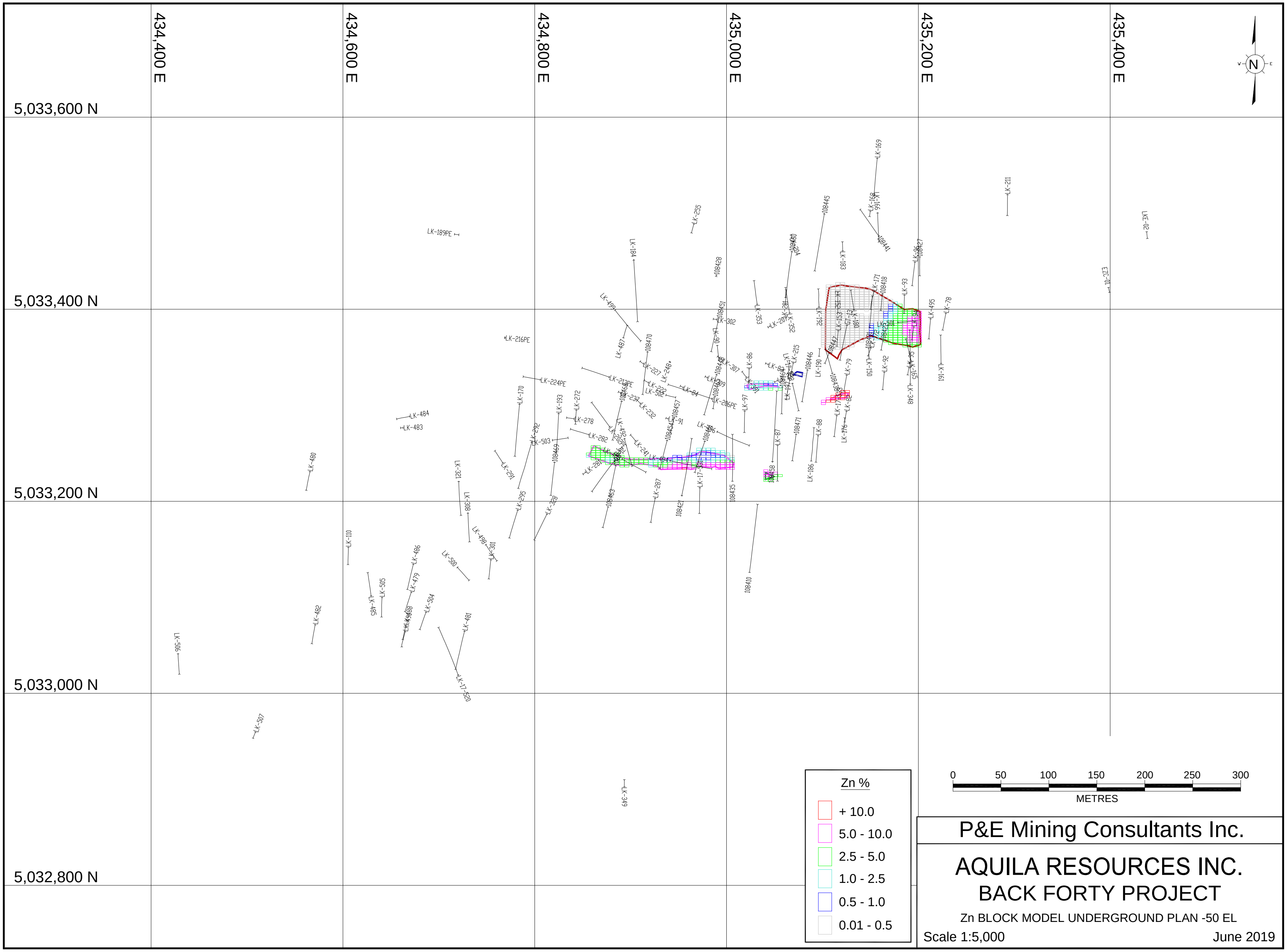

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-491
LK-403
LK-257
LK-146
LK-115
LK-428
LK-142
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND SECTION 435,240 W
Scale 1:3,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND PLAN 100 EL
Scale 1:5,000
June 2019

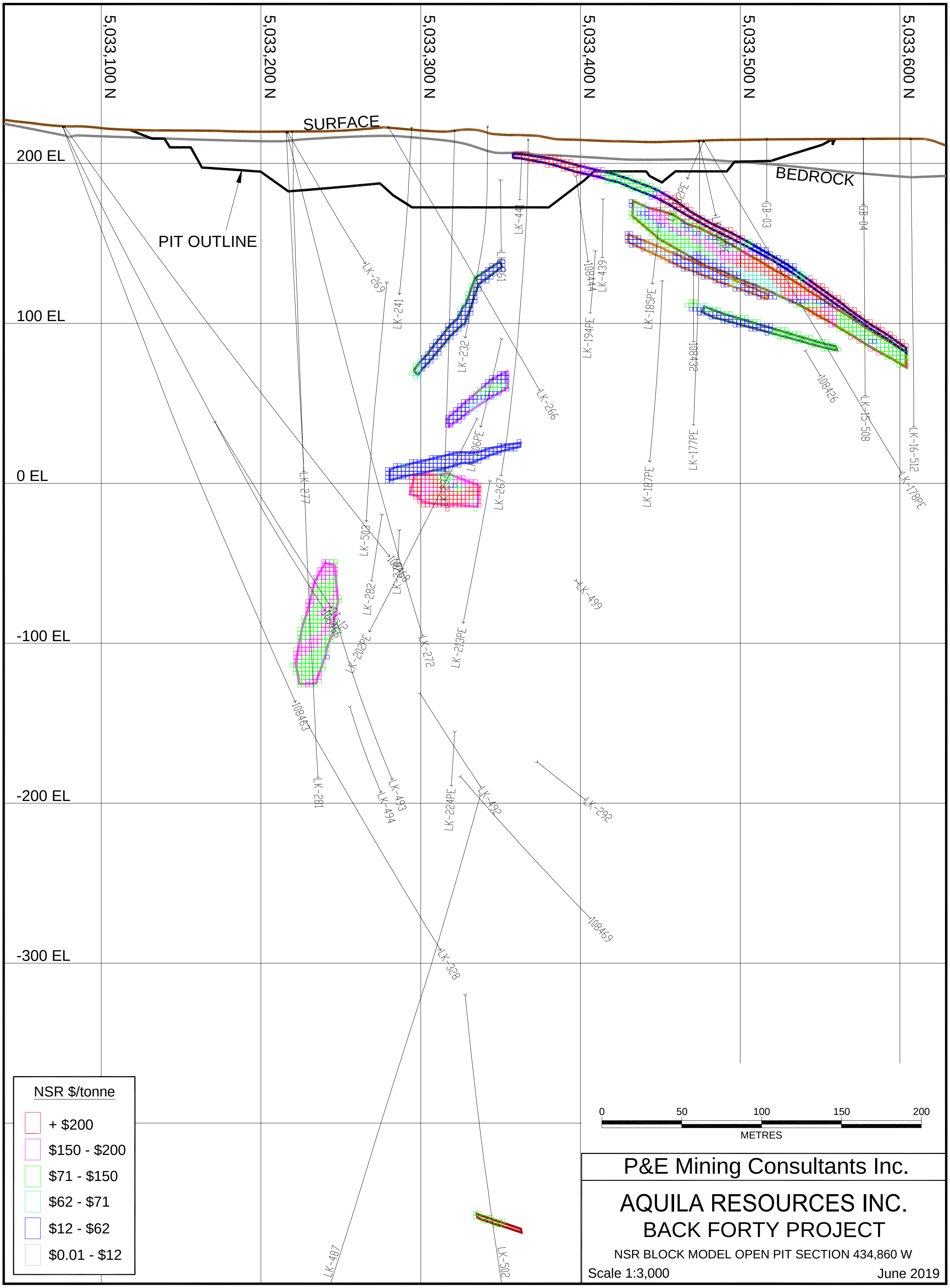

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
N
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND PLAN 50 EL
Scale 1:5,000
June 2019



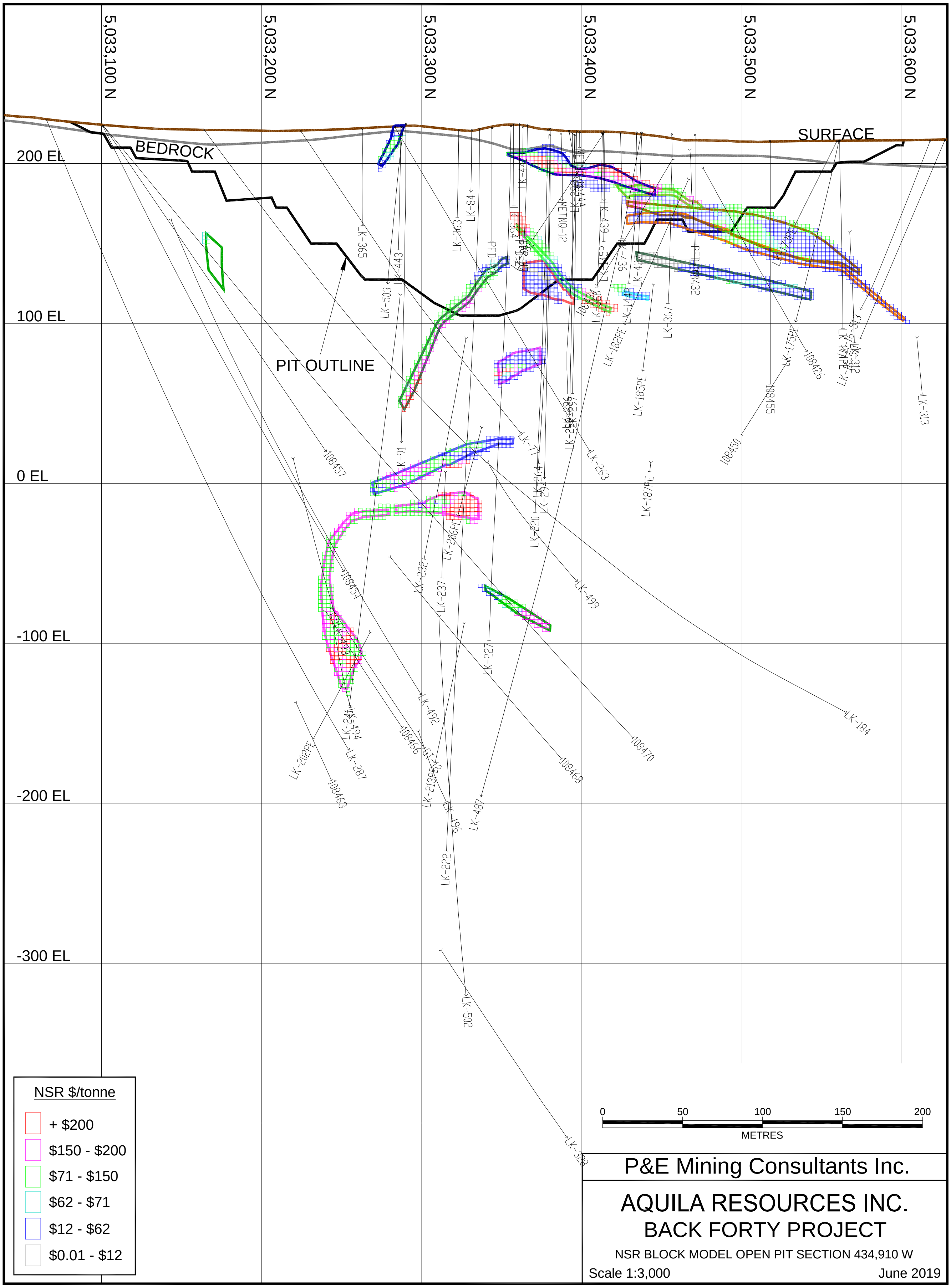
434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND PLAN 0 EL
Scale 1:5,000
June 2019

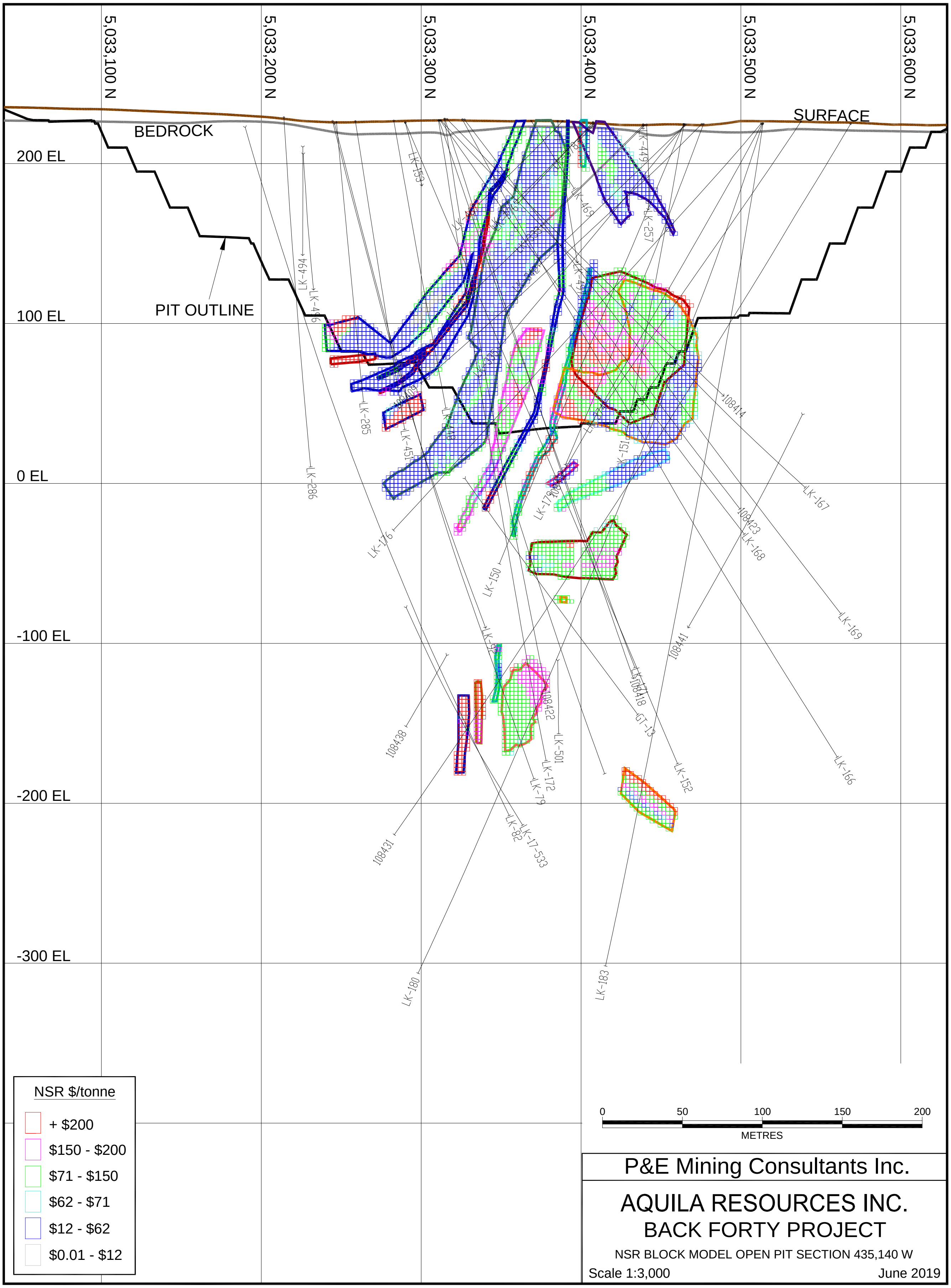

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
Zn %
+ 10.0
5.0 - 10.0
2.5 - 5.0
1.0 - 2.5
0.5 - 1.0
0.01 - 0.5
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
Zn BLOCK MODEL UNDERGROUND PLAN -50 EL
Scale 1:5,000
June 2019

APPENDIX I NSR BLOCK MODEL CROSS SECTIONS AND PLANS

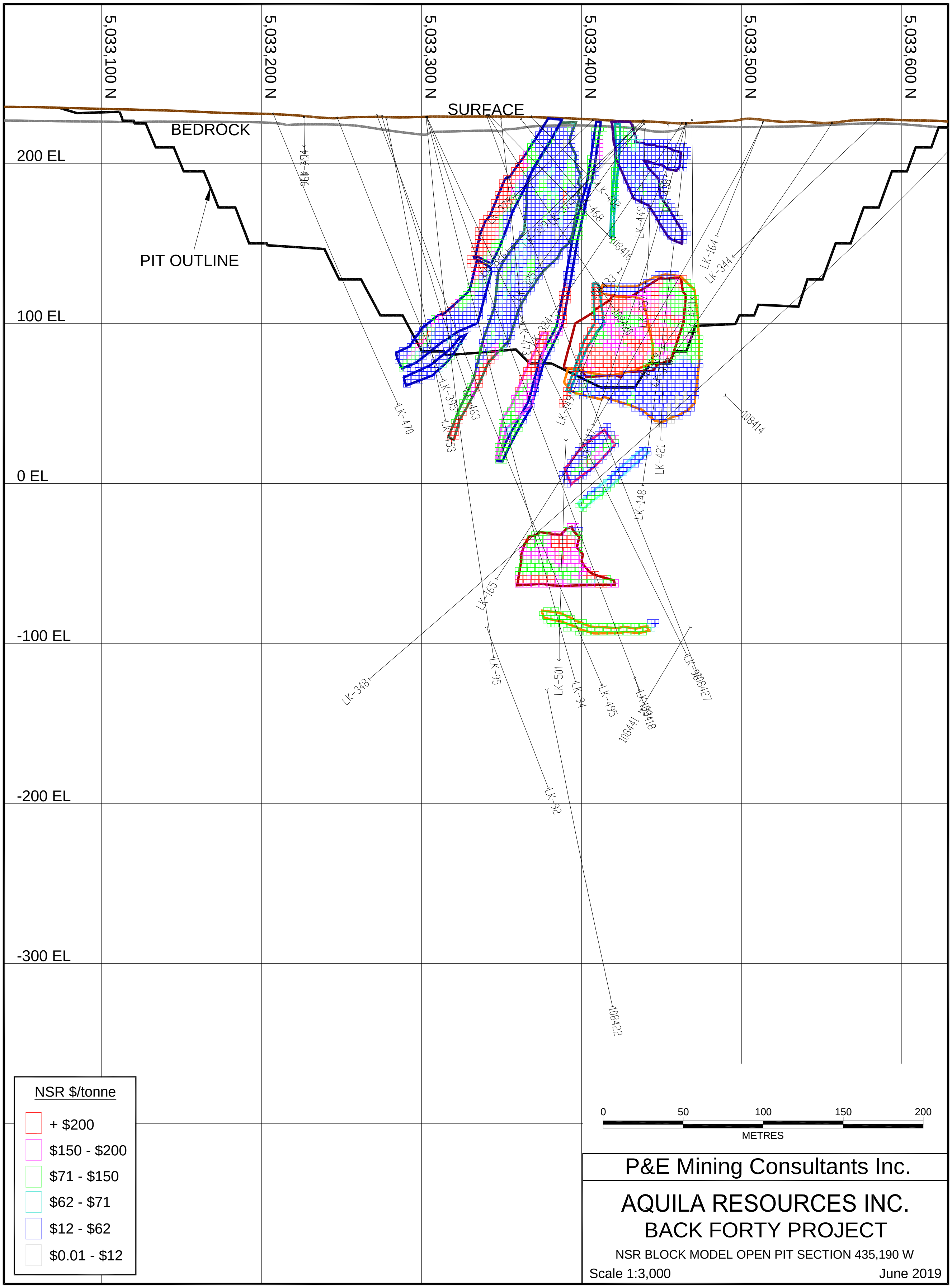


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT SECTION 434,860 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
BEDROCK
SURFACE
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT SECTION 434,910 W
Scale 1:3,000
June 2019



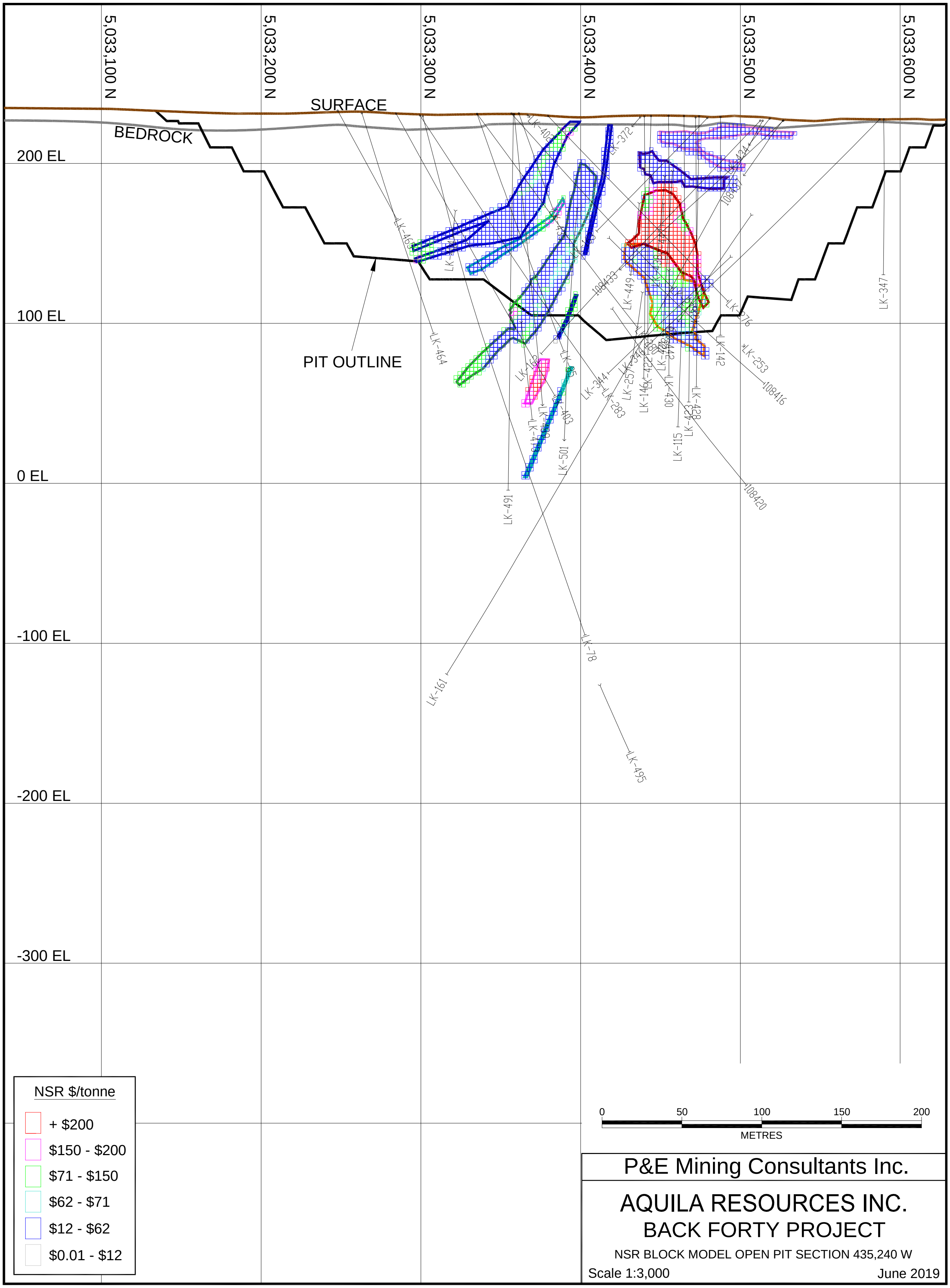
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
PIT OUTLINE
0 EL
-100 EL
-200 EL
-300 EL
LK-153
LK-449
LK-469
LK-257
LK-494
LK-496
LK-285
LK-286
LK-176
LK-150
108414
LK-167
108423
LK-168
LK-169
108441
LK-166
LK-152
GT-13
LK-501
LK-172
LK-79
LK-82
LK-17-533
108431
108438
108422
LK-180
LK-183
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT SECTION 435,140 W
Scale 1:3,000
June 2019

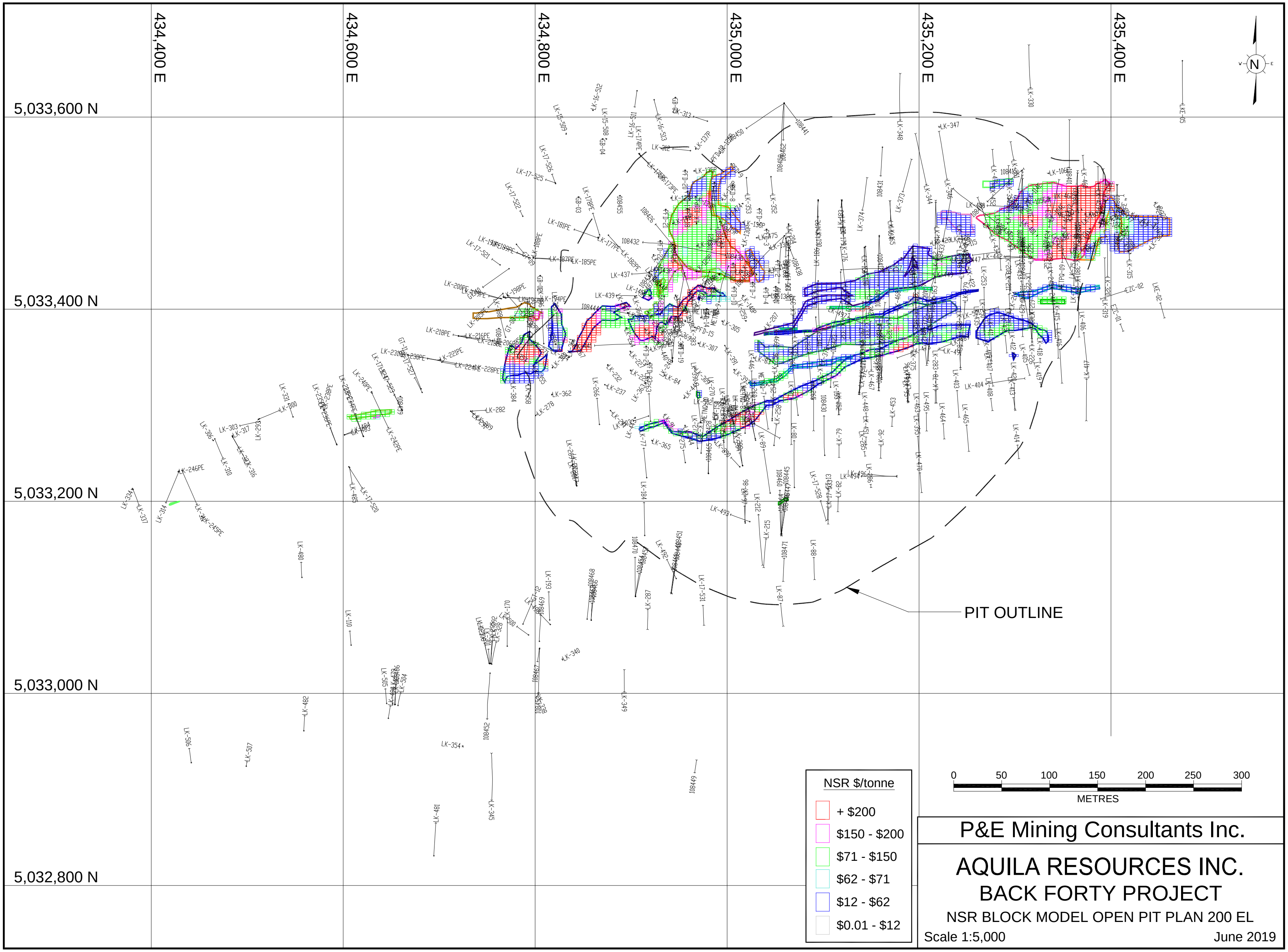


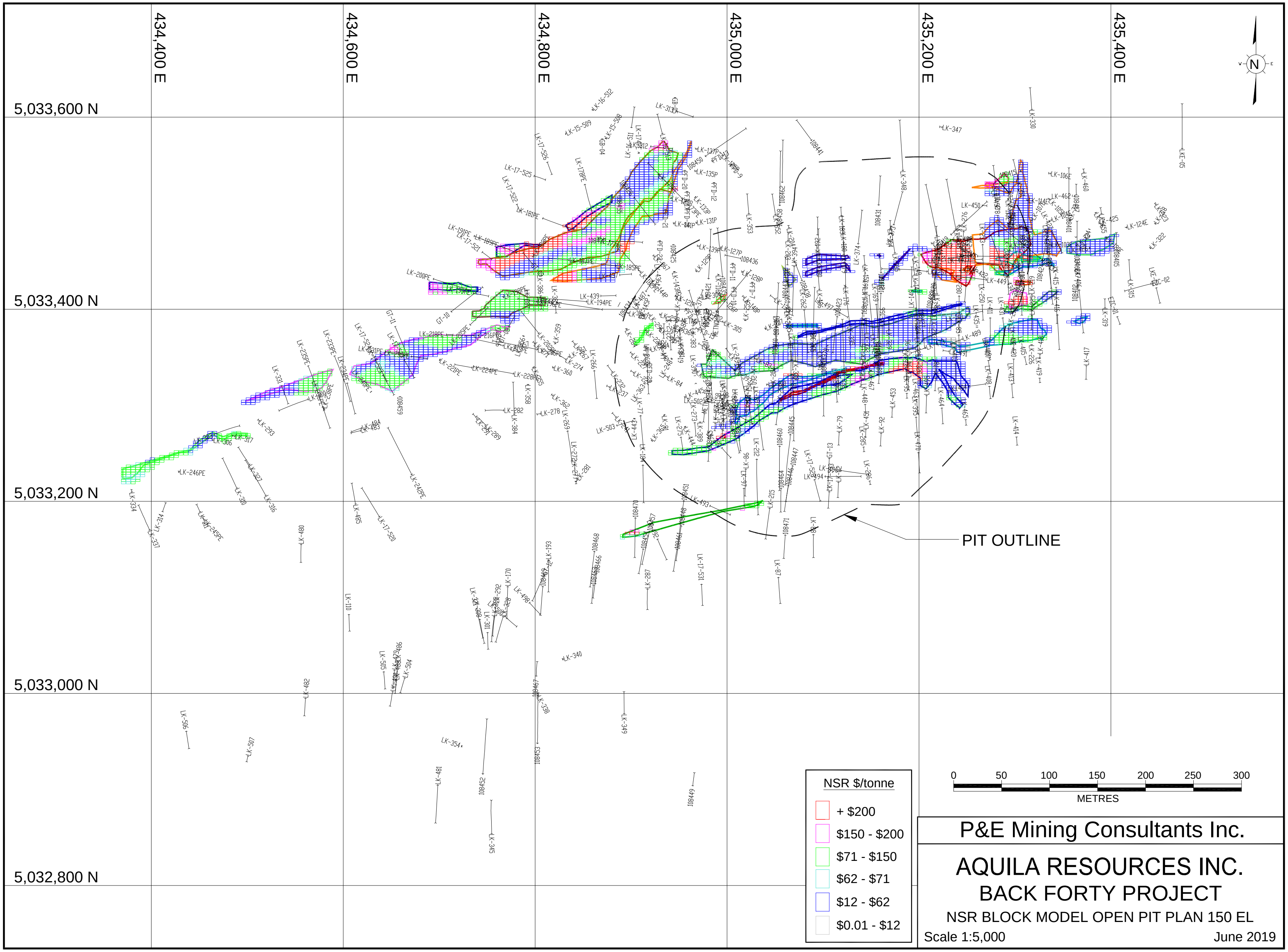
P&E Mining Consultants Inc.

AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT SECTION 435,190 W

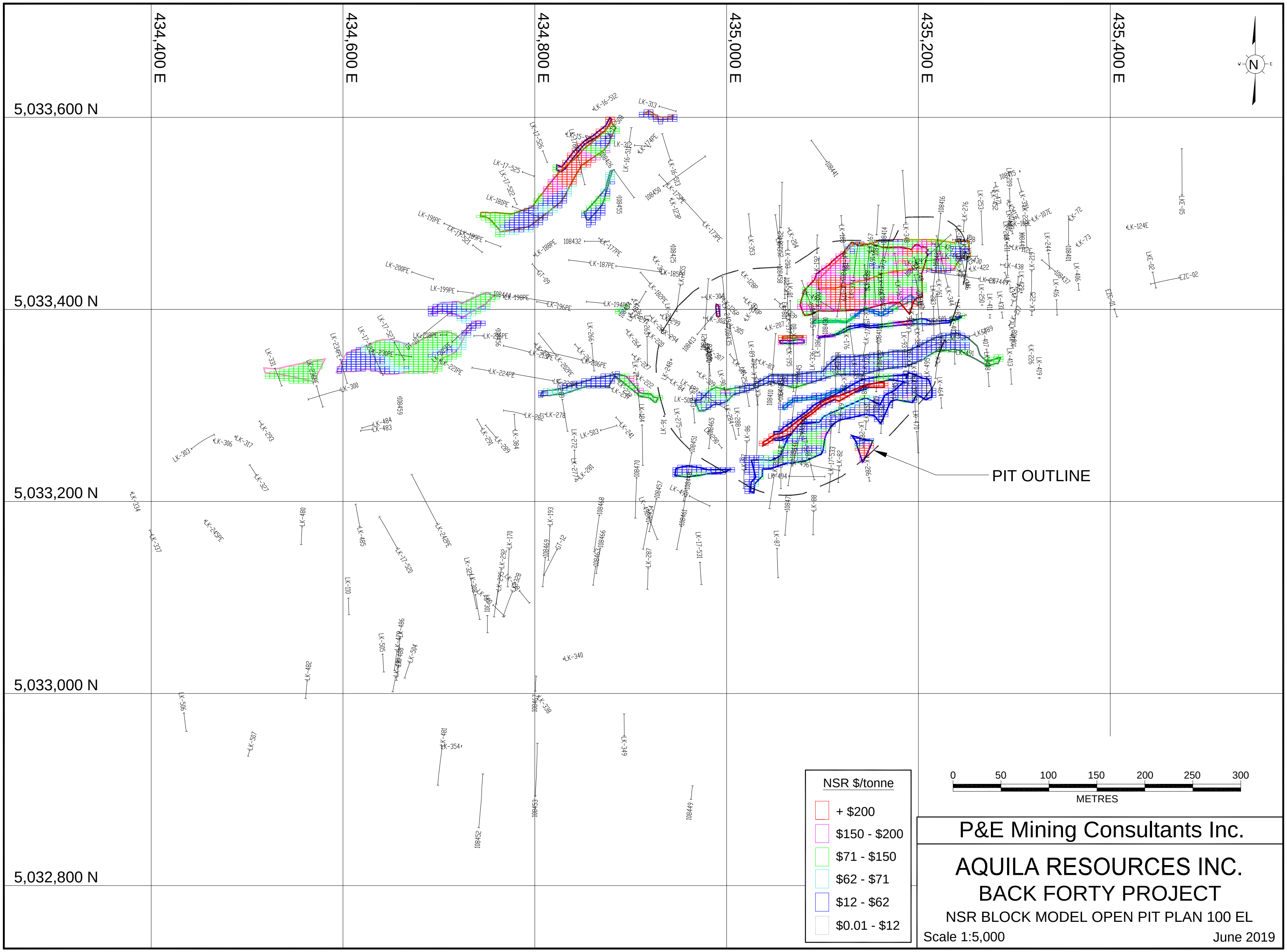
Scale 1:3,000 June 2019

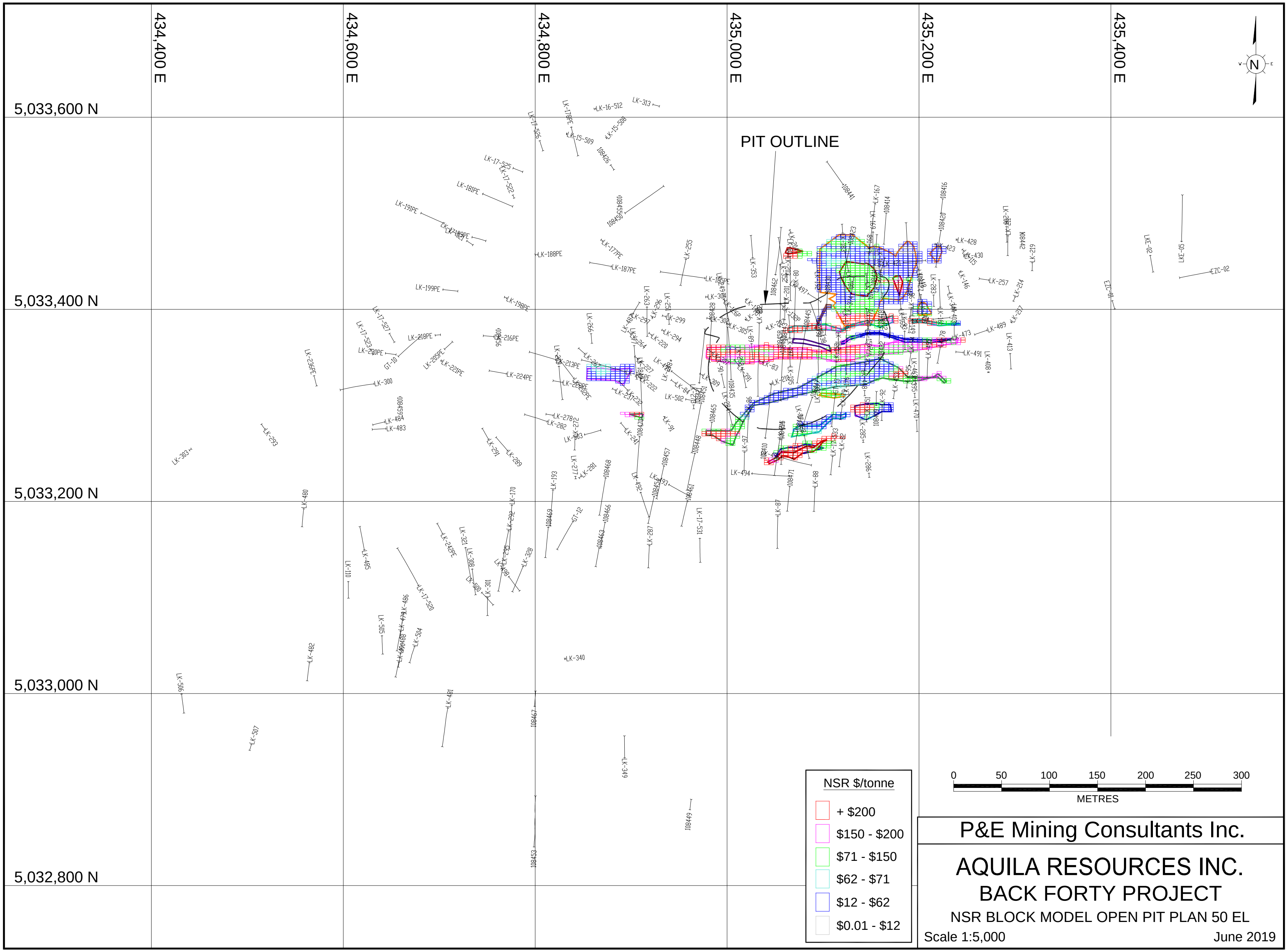

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT SECTION 435,240 W
Scale 1:3,000
June 2019

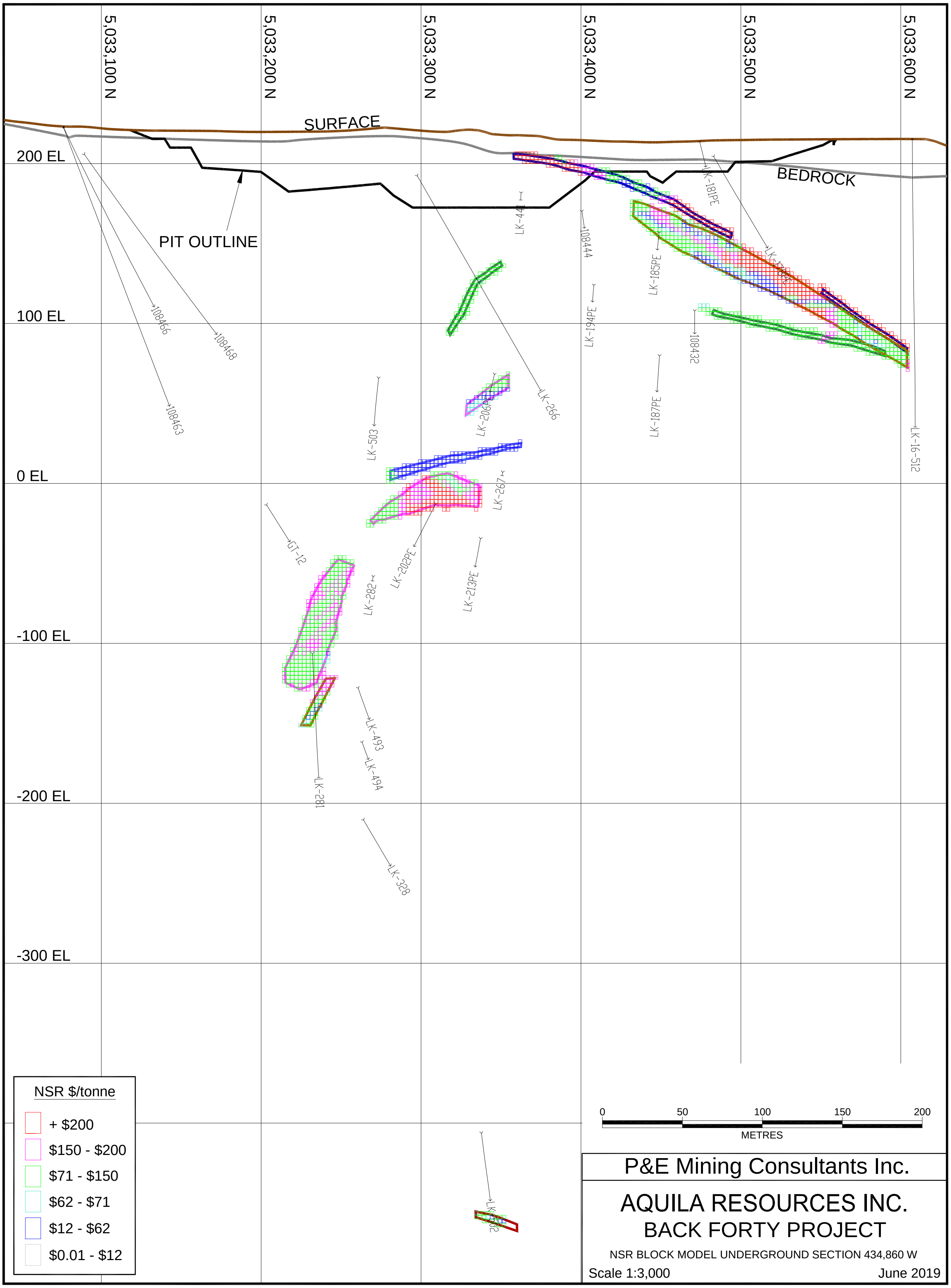

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT PLAN 200 EL
Scale 1:5,000
June 2019



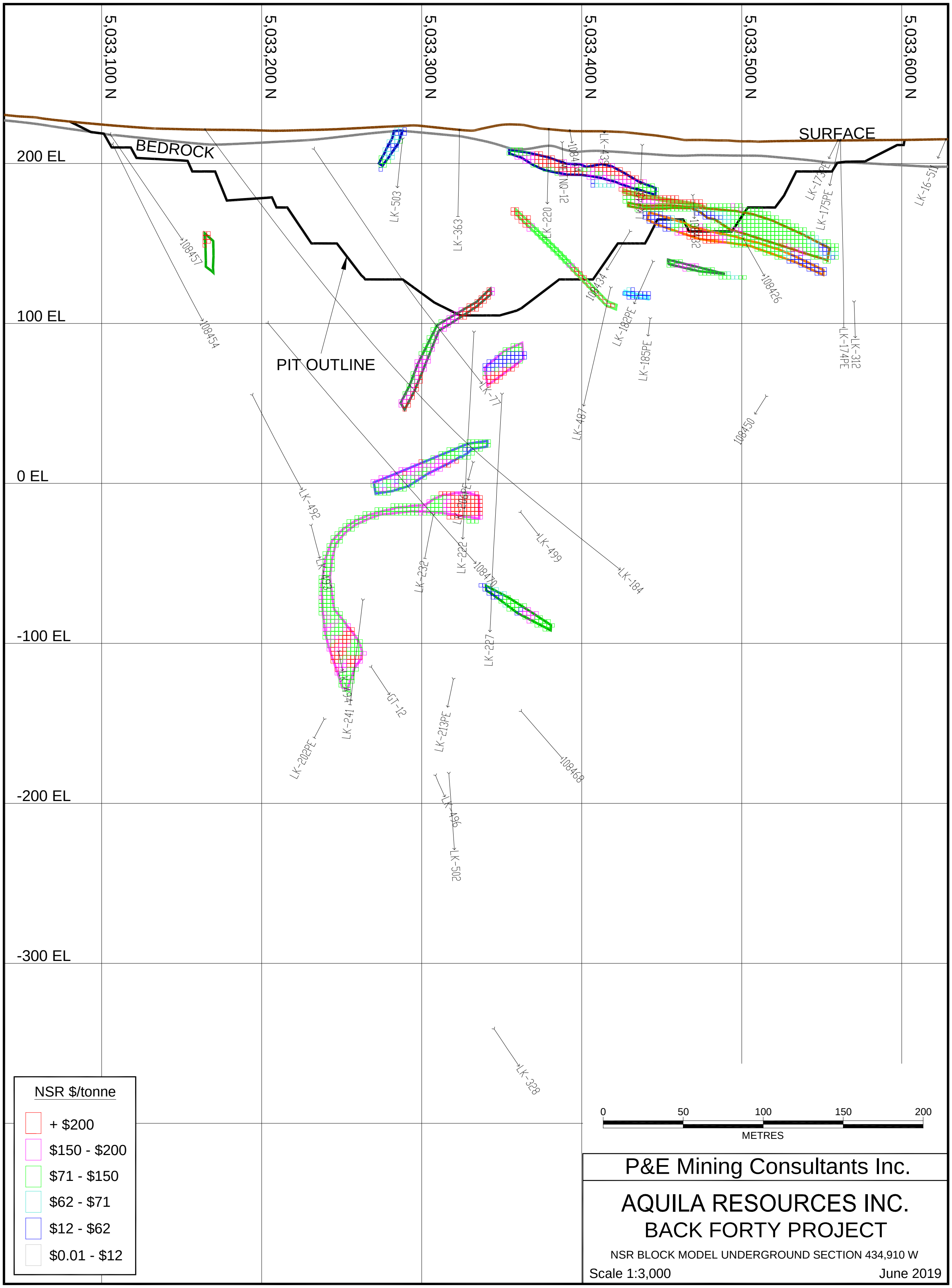
434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT PLAN 150 EL
Scale 1:5,000
June 2019

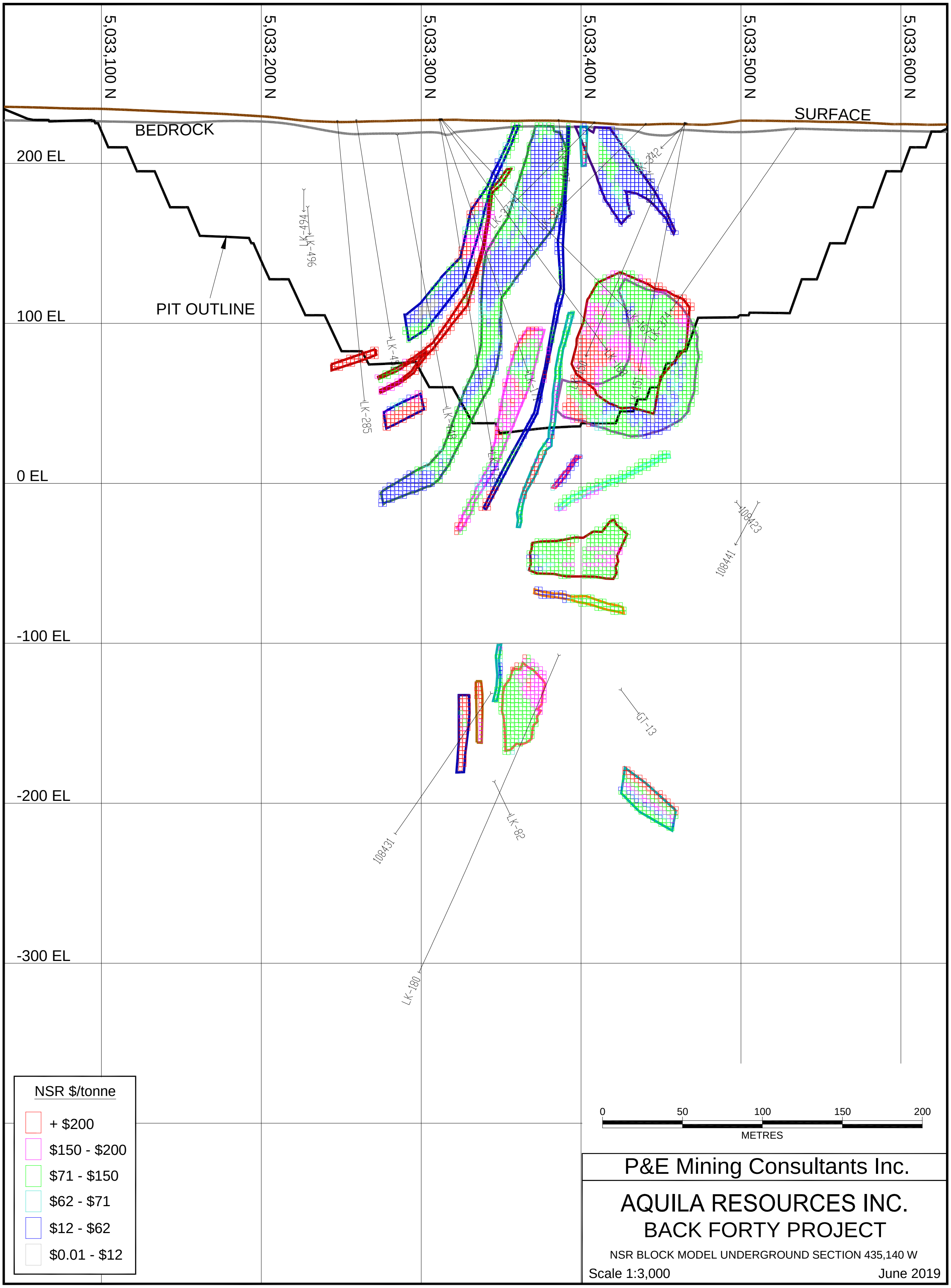

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
N
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT PLAN 100 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL OPEN PIT PLAN 50 EL
Scale 1:5,000
June 2019

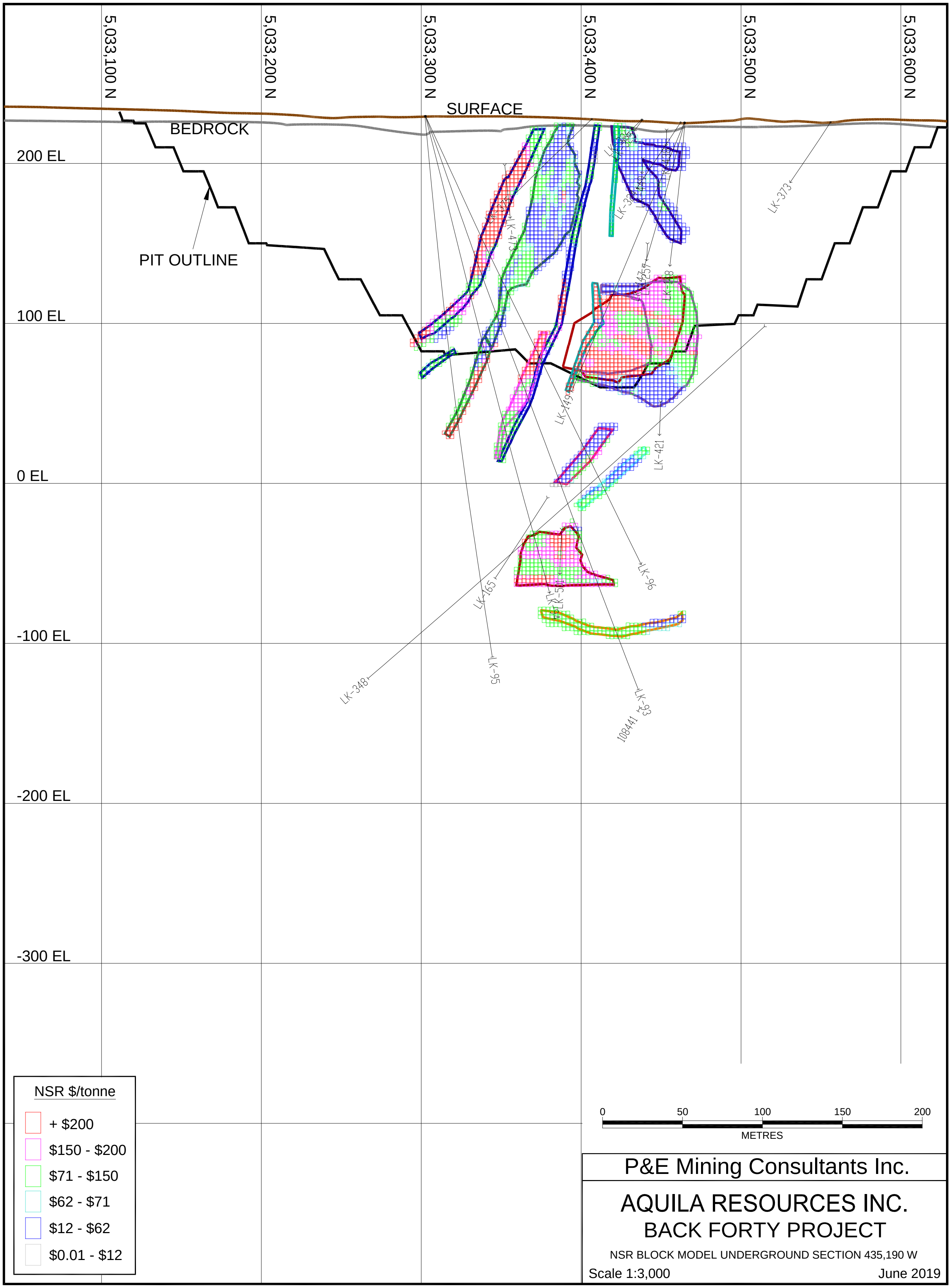


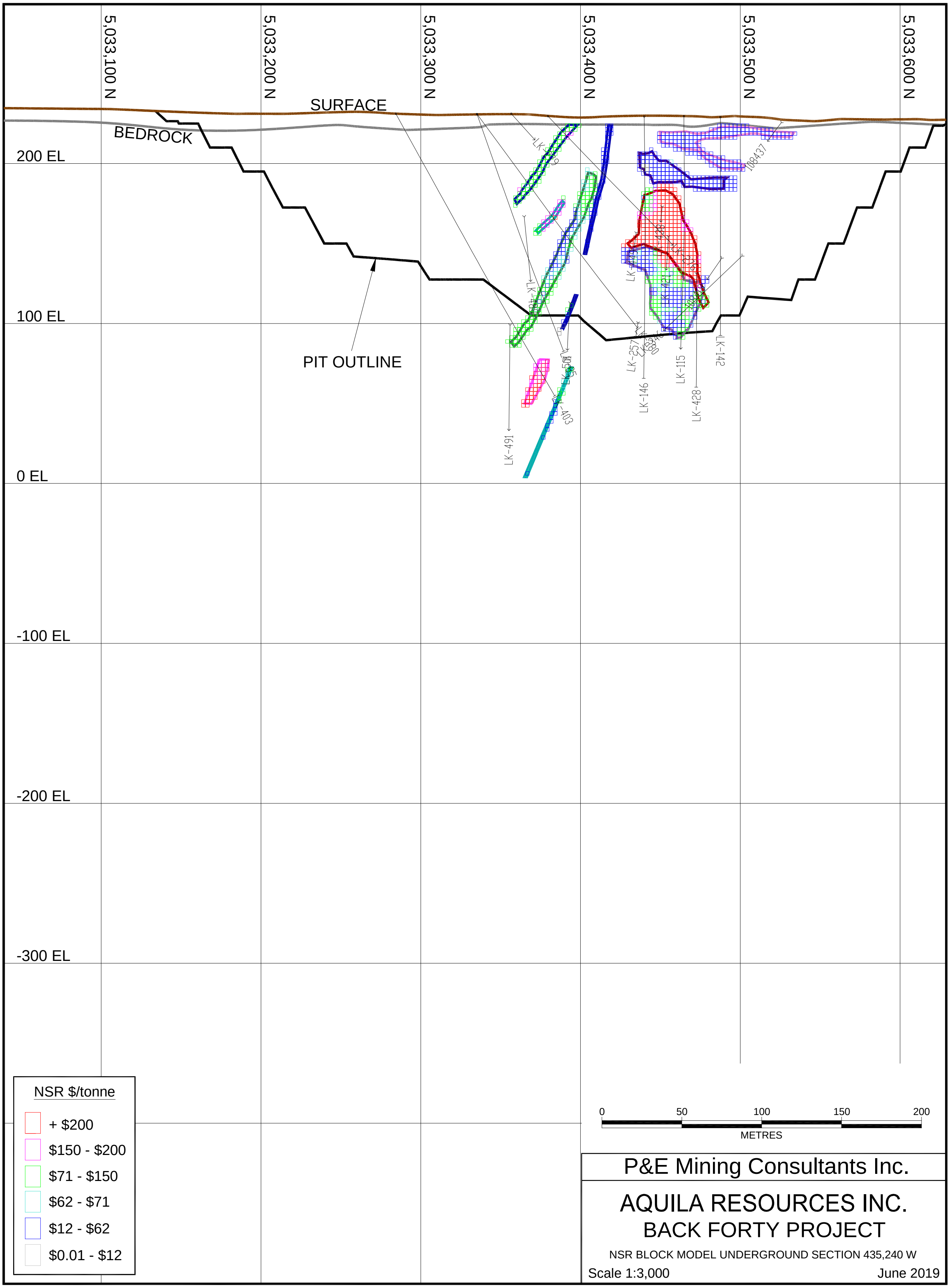
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
108466
108468
108463
LK-503
LK-266
LK-441
108444
LK-194PE
LK-181PE
LK-185PE
108432
LK-187PE
LK-16-512
LK-206PE
LK-267
LK-202PE
LK-213PE
LK-282
GT-12
LK-493
LK-494
LK-281
LK-328
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND SECTION 434,860 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
BEDROCK
SURFACE
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND SECTION 434,910 W
Scale 1:3,000
June 2019



5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-494
LK-496
LK-285
LK-180
LK-82
GT-13
108431
108423
108441
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND SECTION 435,140 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
PIT OUTLINE
LK-373
LK-421
LK-96
LK-165
LK-348
LK-95
LK-93
108441
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND SECTION 435,190 W
Scale 1:3,000
June 2019

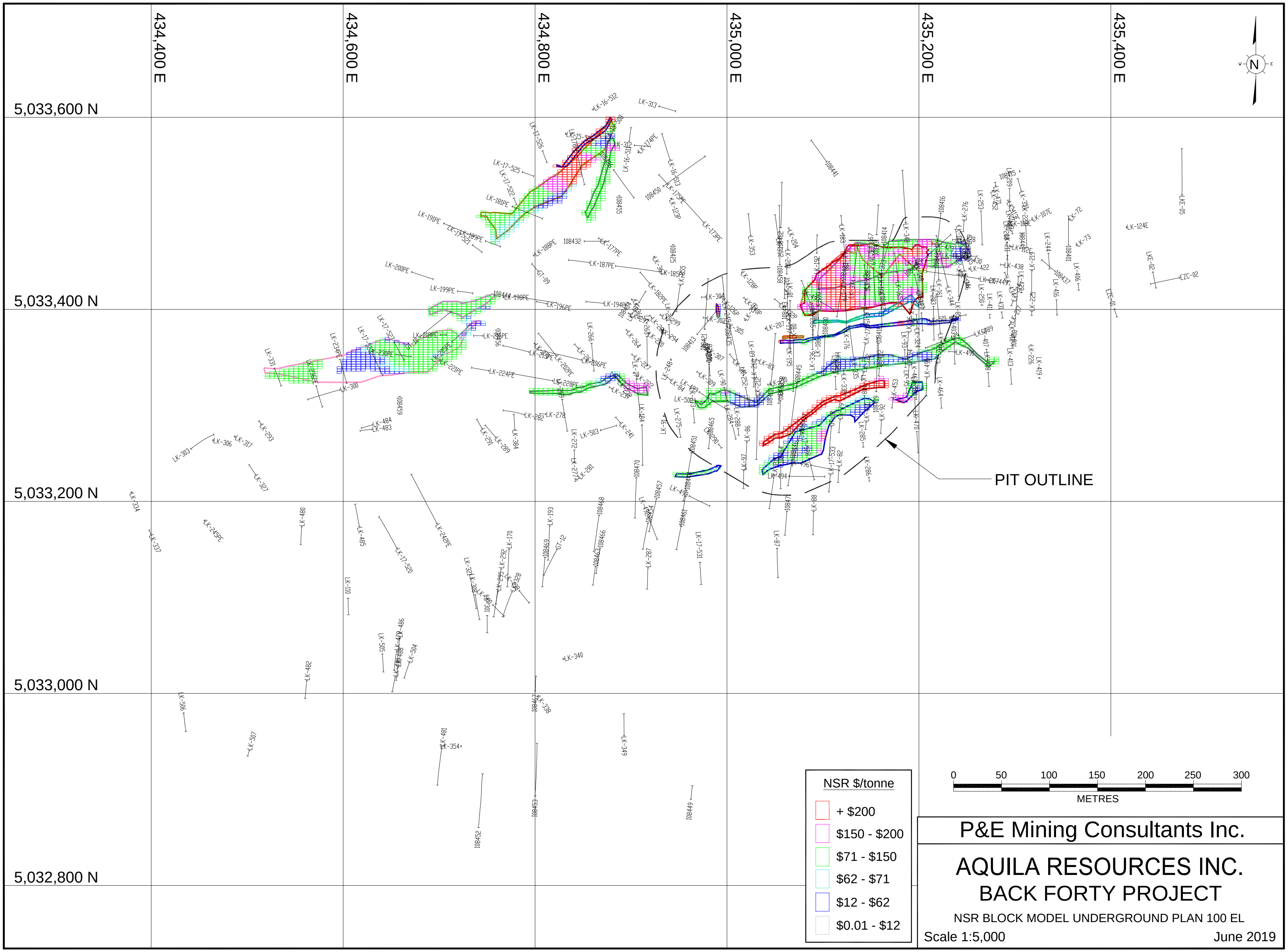


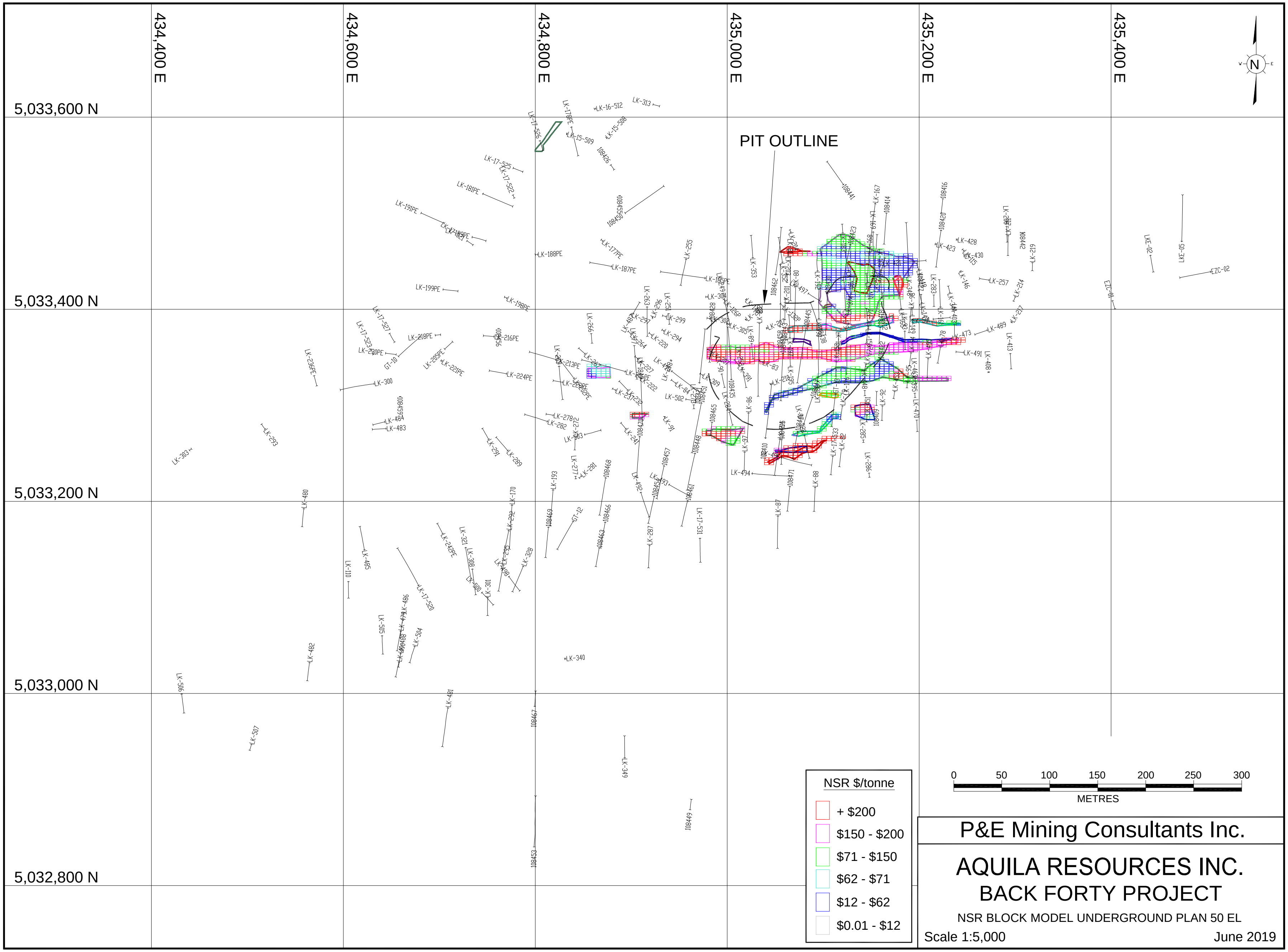
P&E Mining Consultants Inc.

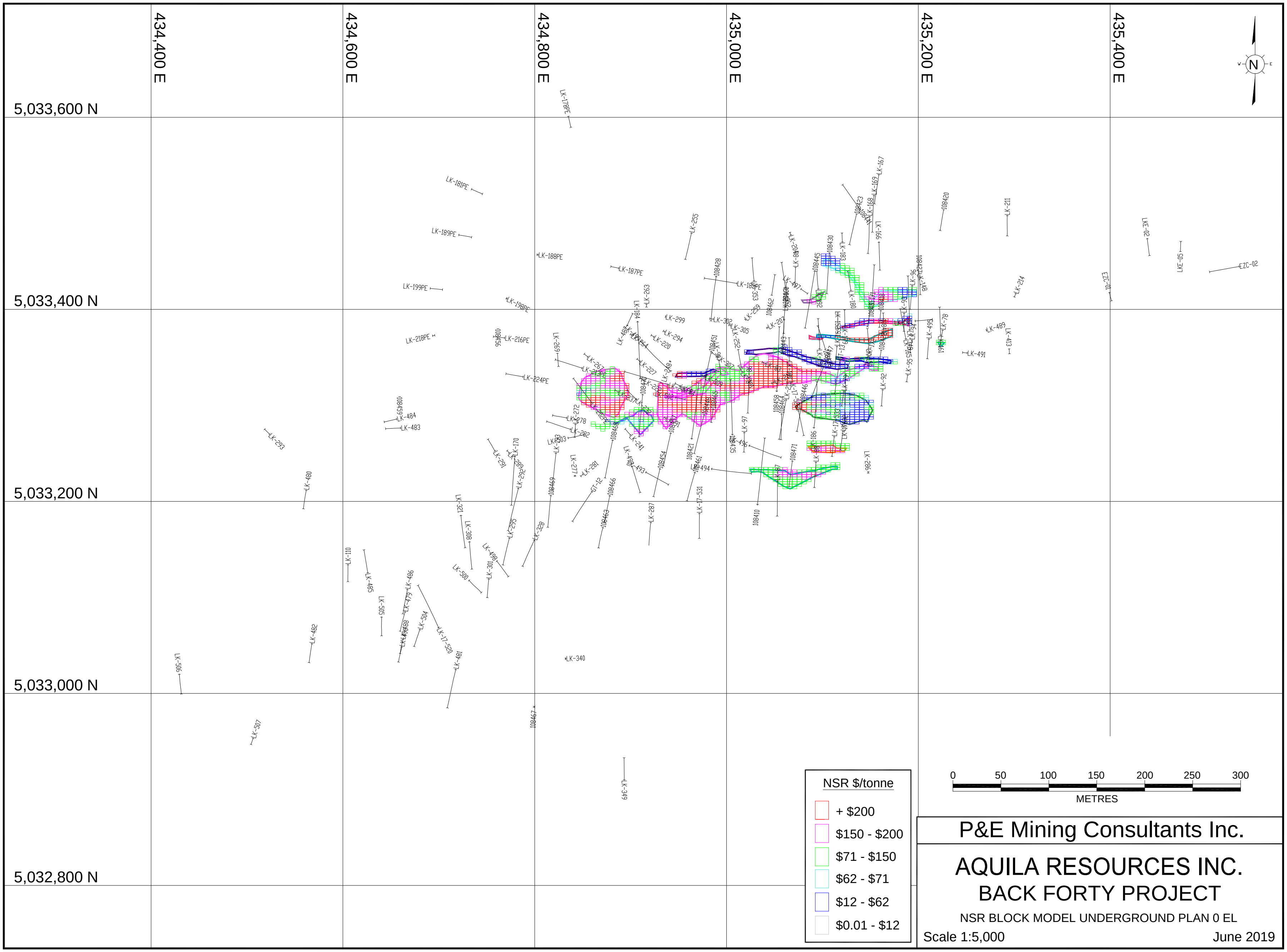
AQUILA RESOURCES INC.
BACK FORTY PROJECT

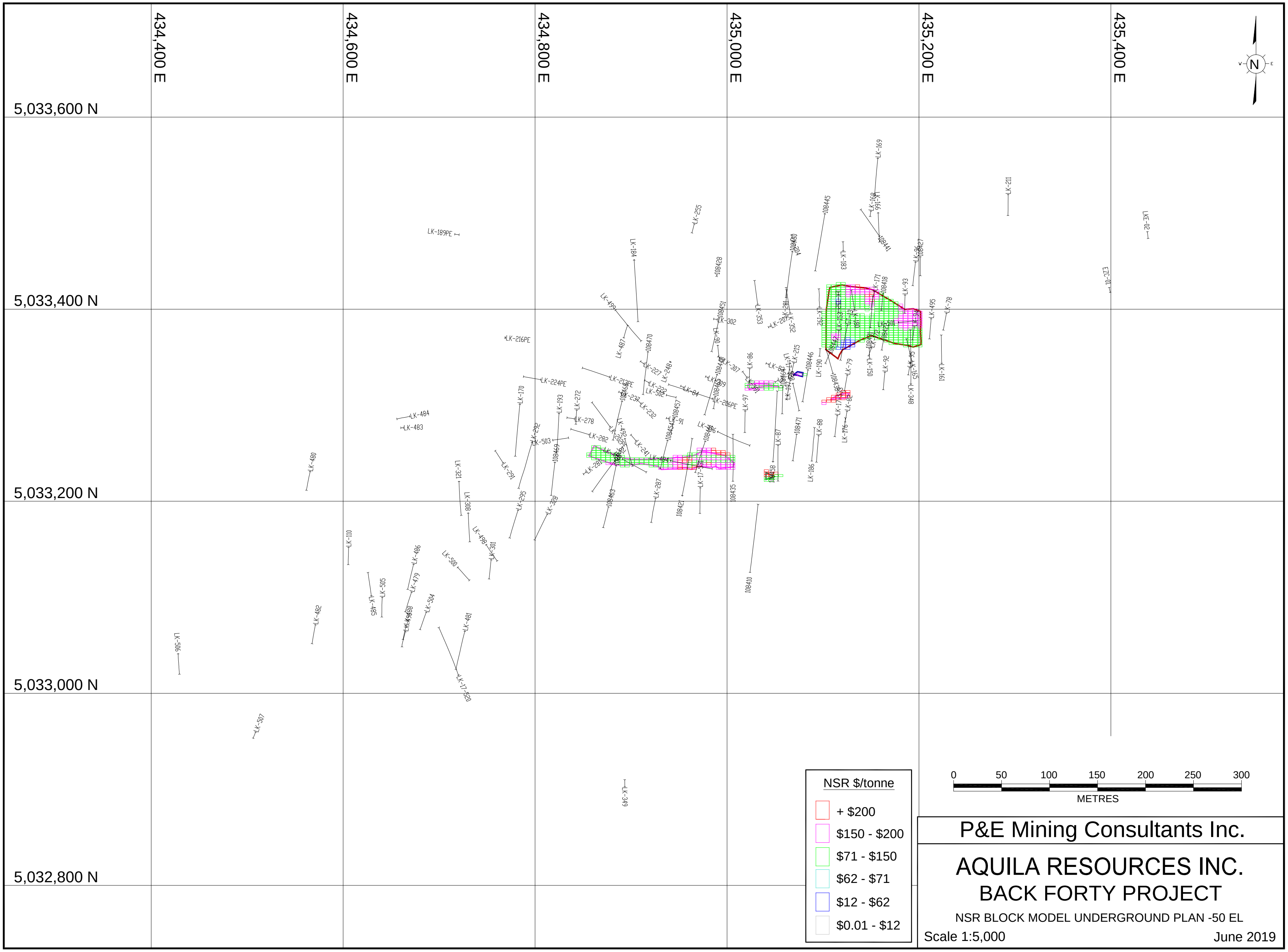
NSR BLOCK MODEL UNDERGROUND SECTION 435,240 W

Scale 1:3,000 June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0 50 100 150 200 250 300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND PLAN 100 EL
Scale 1:5,000
June 2019

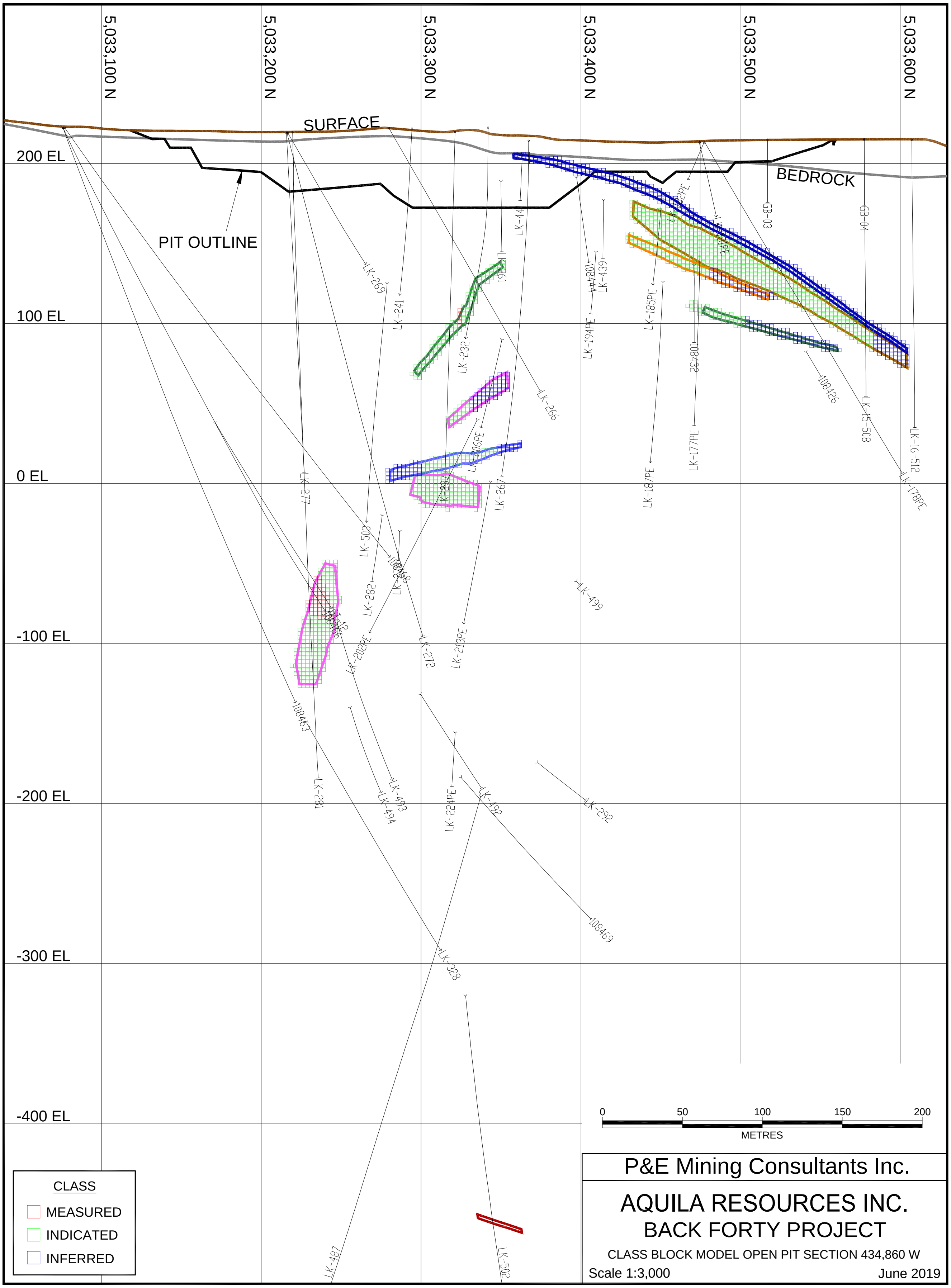

434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND PLAN 50 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND PLAN 0 EL
Scale 1:5,000
June 2019

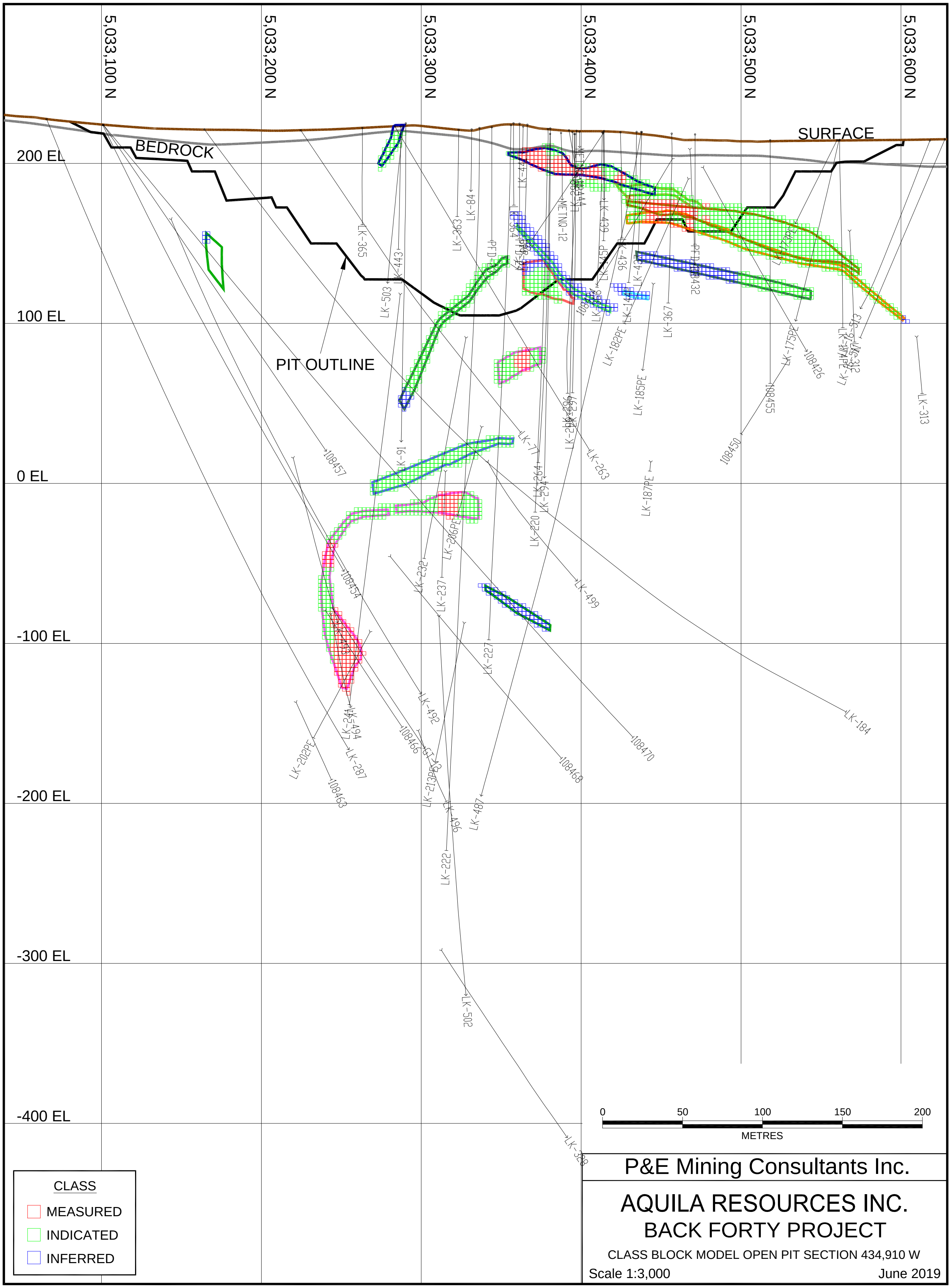


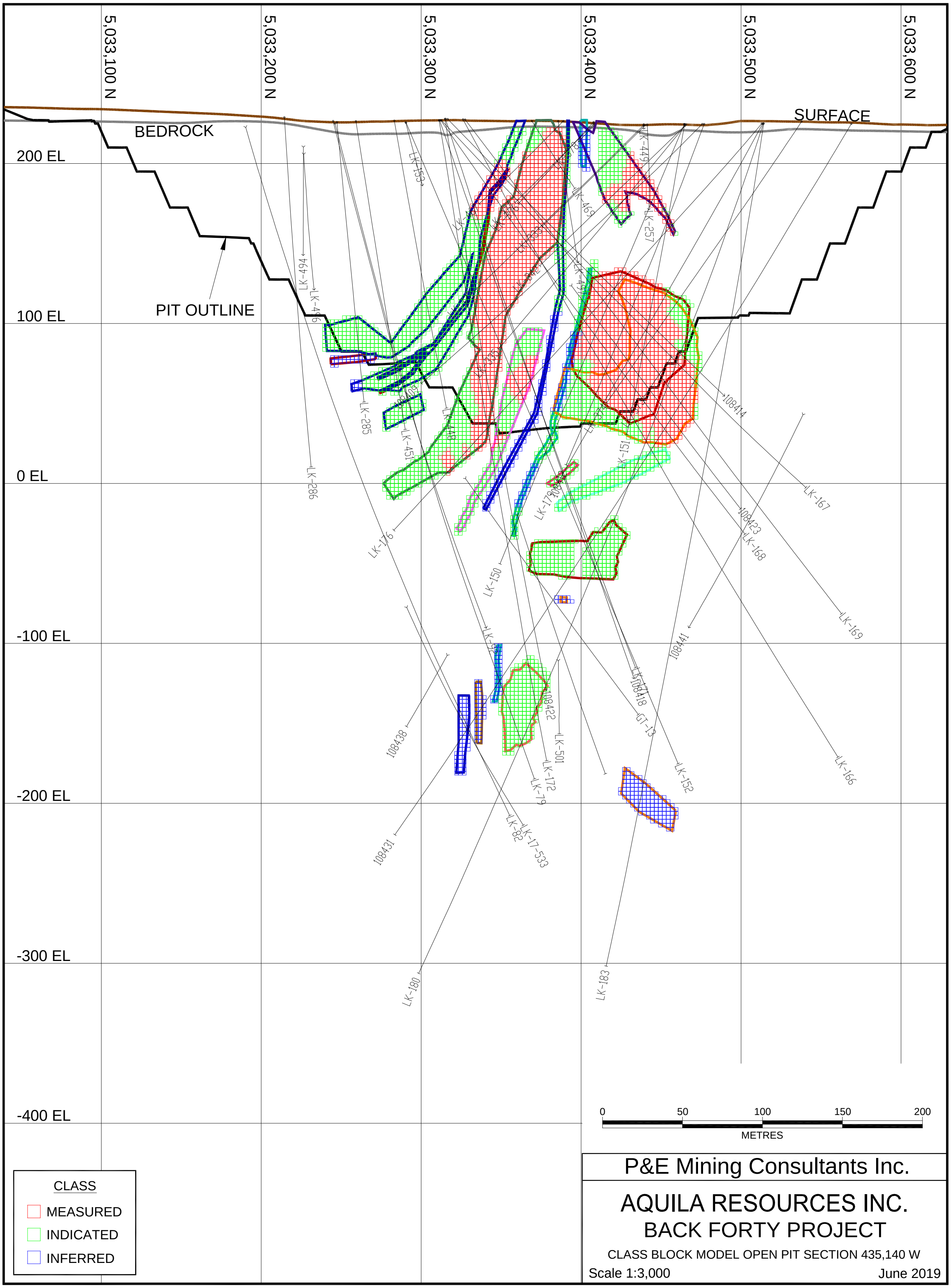
434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
NSR $/tonne
+ $200
$150 - $200
$71 - $150
$62 - $71
$12 - $62
$0.01 - $12
0
50
100
150
200
250
300
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
NSR BLOCK MODEL UNDERGROUND PLAN -50 EL
Scale 1:5,000
June 2019

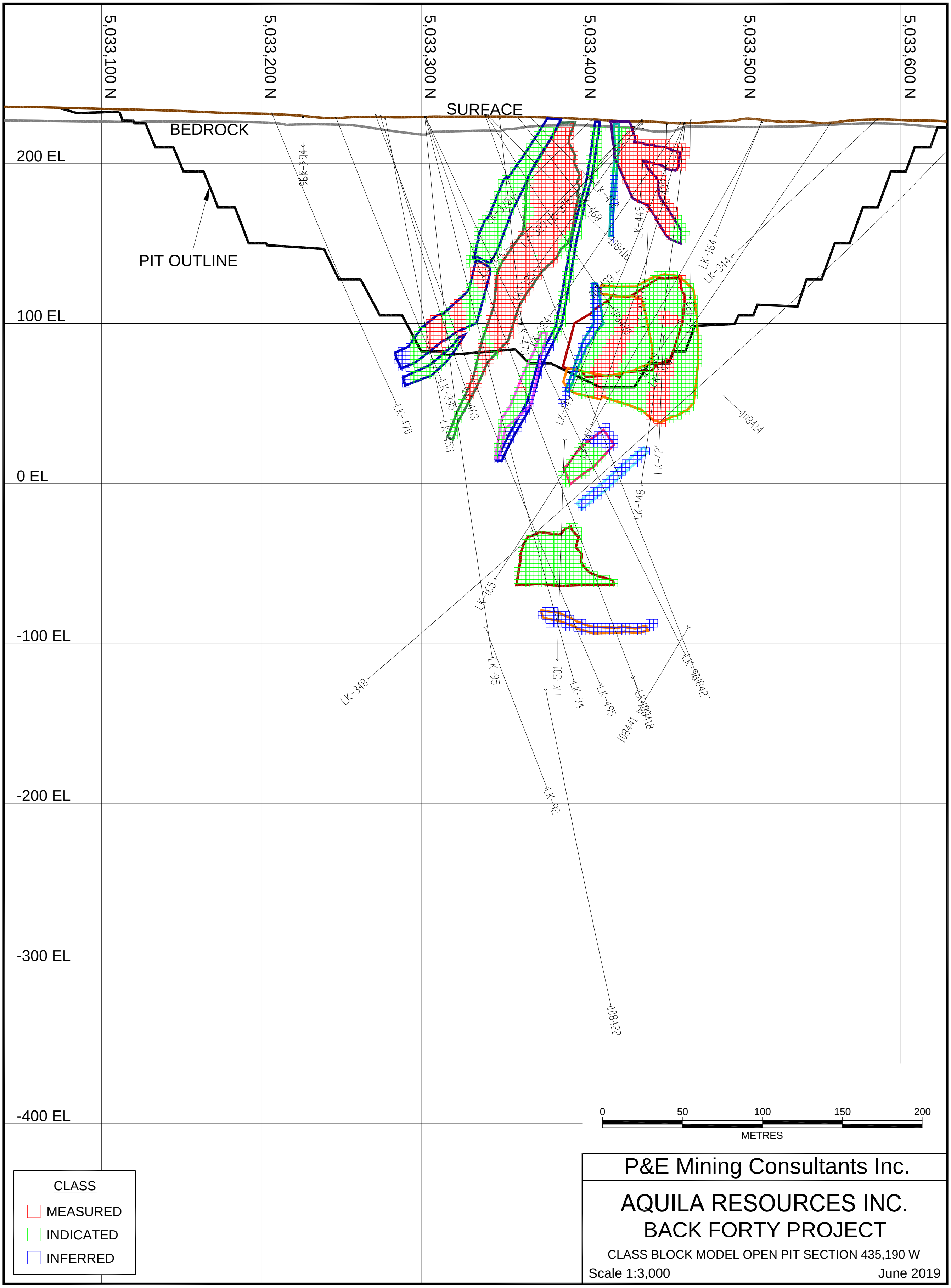
APPENDIX J CLASSIFICATION BLOCK MODEL CROSS SECTIONS AND PLANS

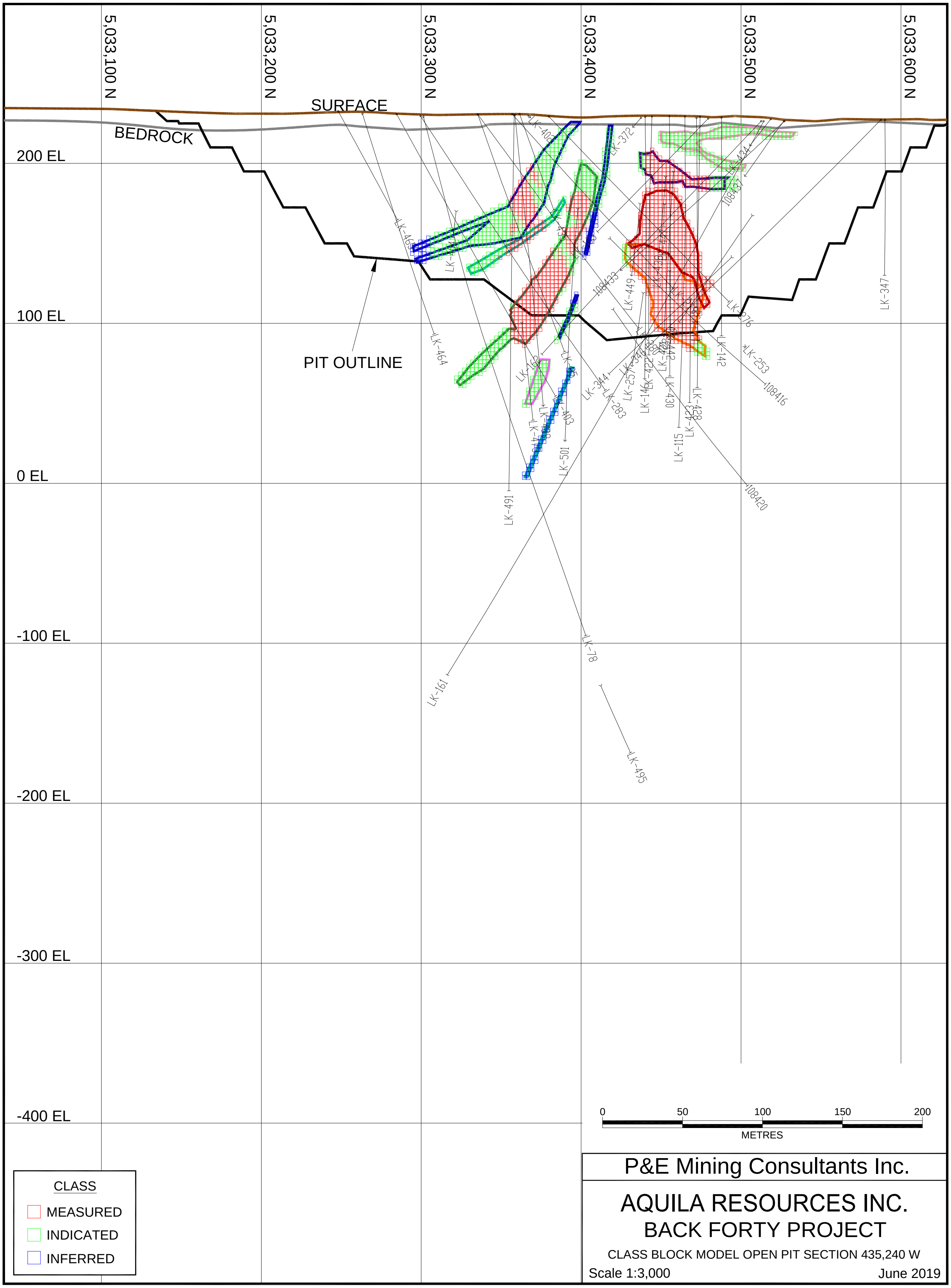


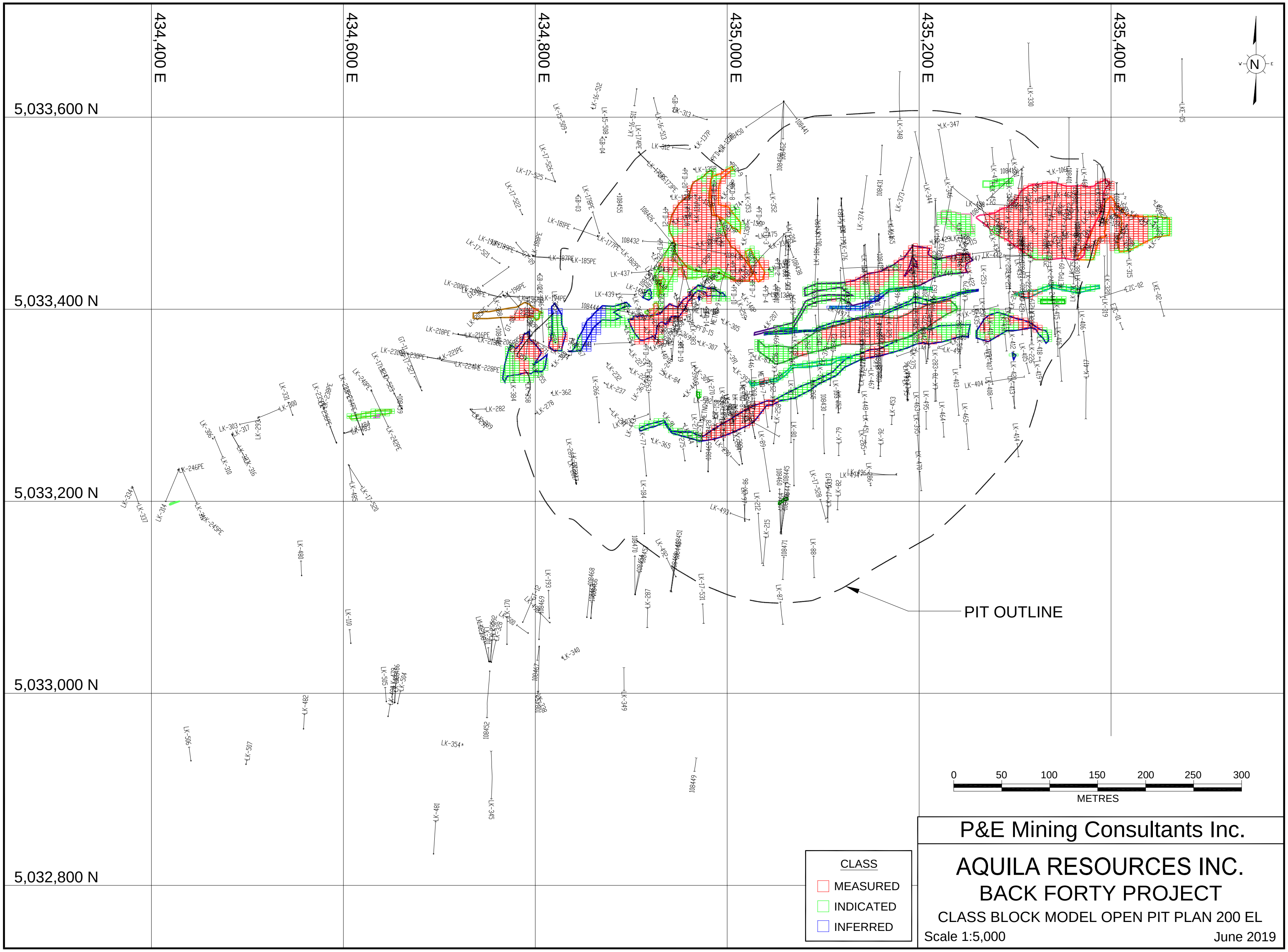
5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
CLASS
MEASURED
INDICATED
INFERRED
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT SECTION 434,860 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
SURFACE
BEDROCK
PIT OUTLINE
0
50
100
150
200
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT SECTION 434,910 W
Scale 1:3,000
June 2019

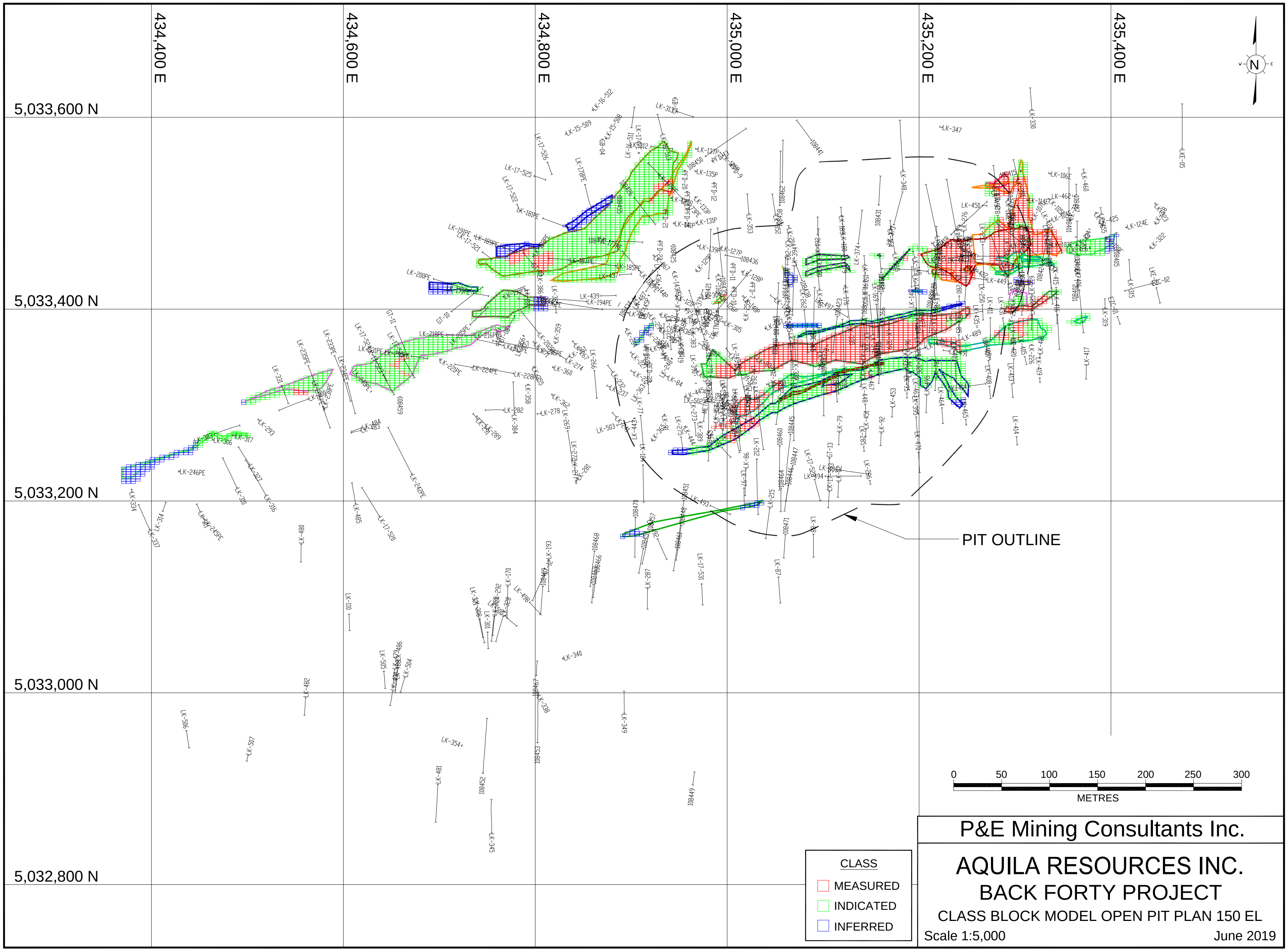

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
CLASS
MEASURED
INDICATED
INFERRED
0 50 100 150 200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT SECTION 435,140 W
Scale 1:3,000
June 2019

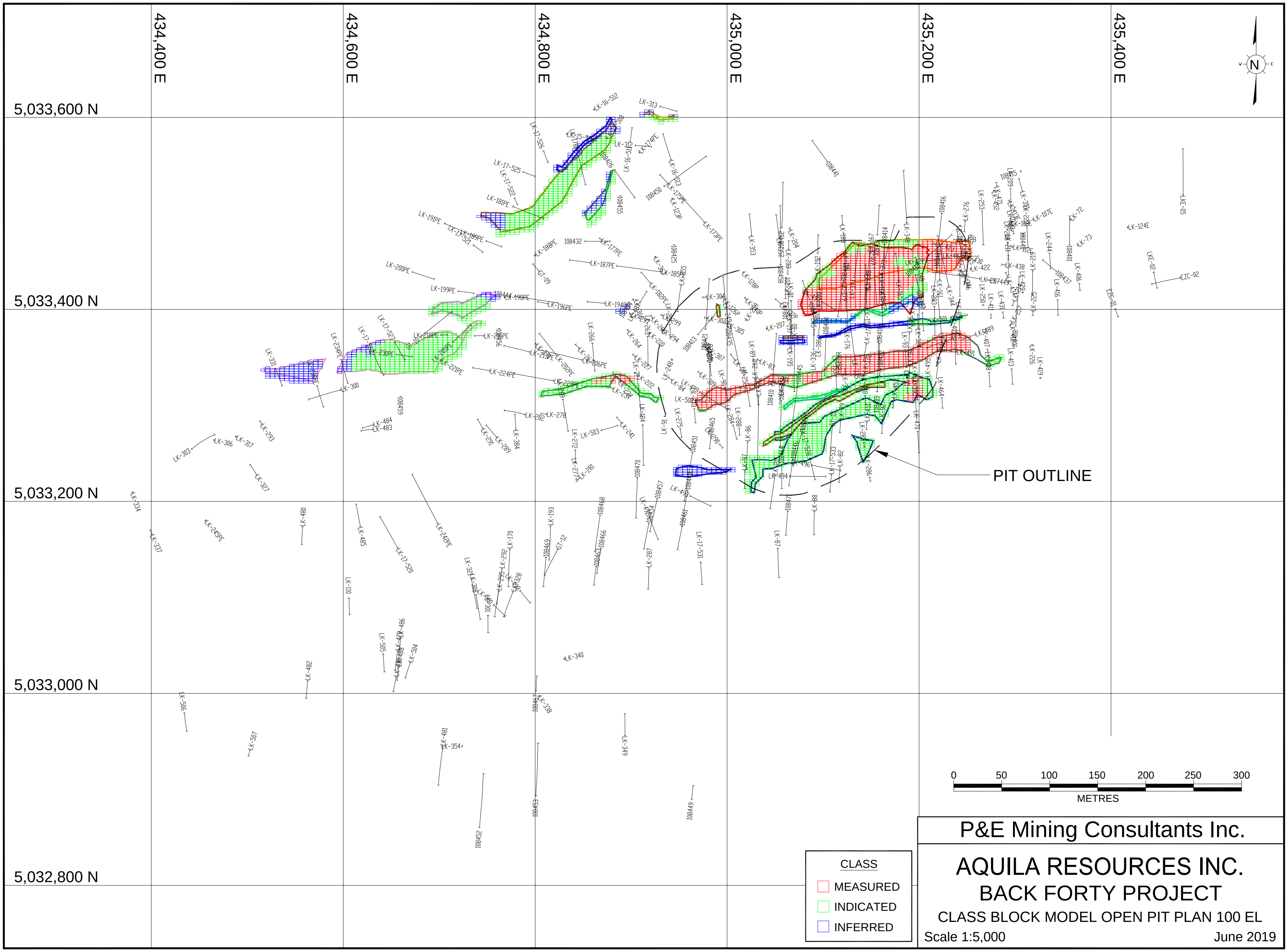

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
CLASS
MEASURED
INDICATED
INFERRED
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT SECTION 435,190 W
Scale 1:3,000
June 2019

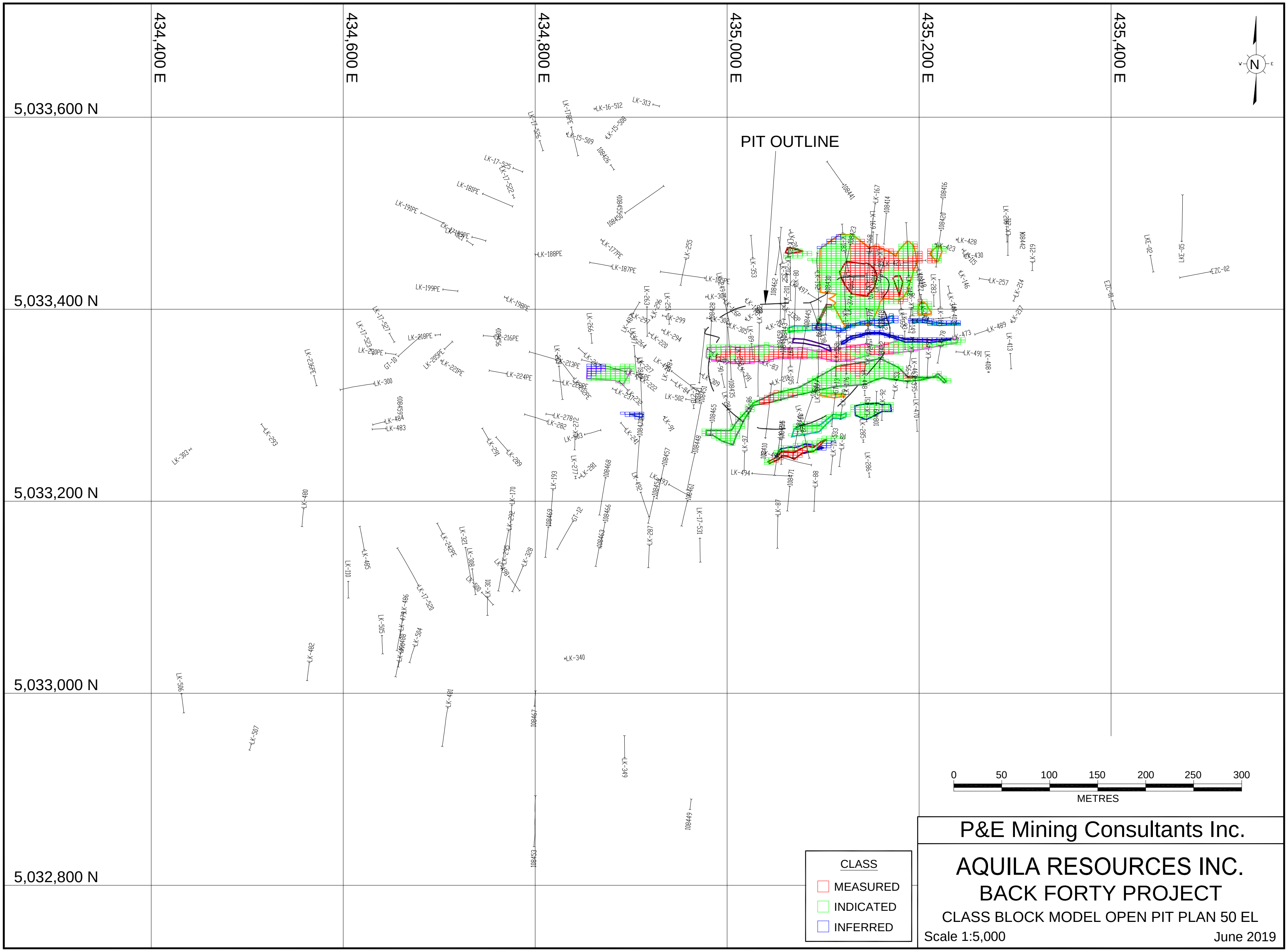

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
PIT OUTLINE
LK-464
LK-491
LK-501
LK-78
LK-161
LK-495
LK-347
LK-142
LK-253
108416
108420
LK-115
LK-430
LK-428
LK-423
LK-344
LK-283
LK-449
108433
LK-372
0
50
100
150
200
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT SECTION 435,240 W
Scale 1:3,000
June 2019

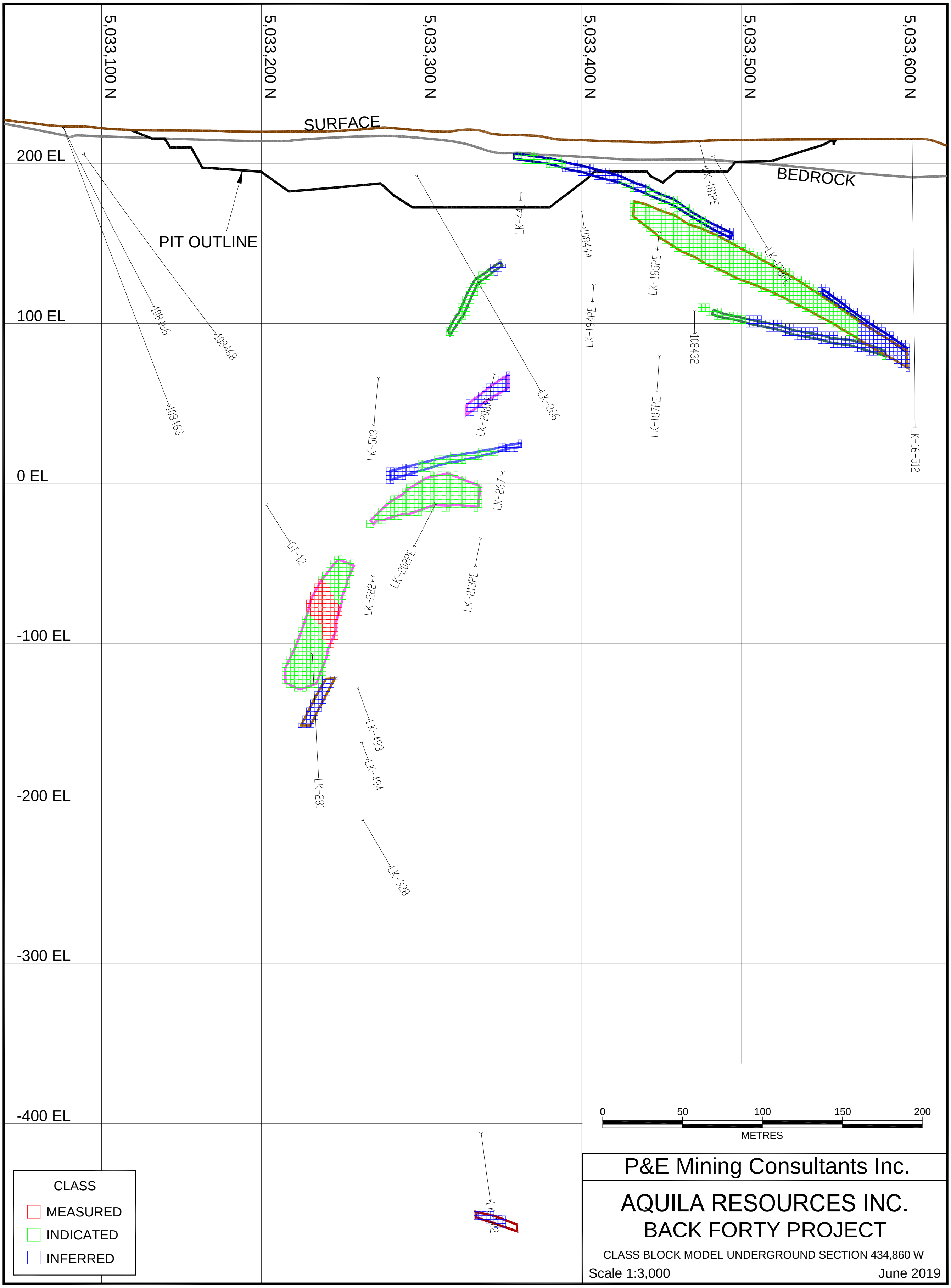


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
0 50 100 150 200 250 300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT PLAN 200 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
0 50 100 150 200 250 300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT PLAN 150 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
0
50
100
150
200
250
300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT PLAN 100 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
N
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
PIT OUTLINE
0
50
100
150
200
250
300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL OPEN PIT PLAN 50 EL
Scale 1:5,000
June 2019

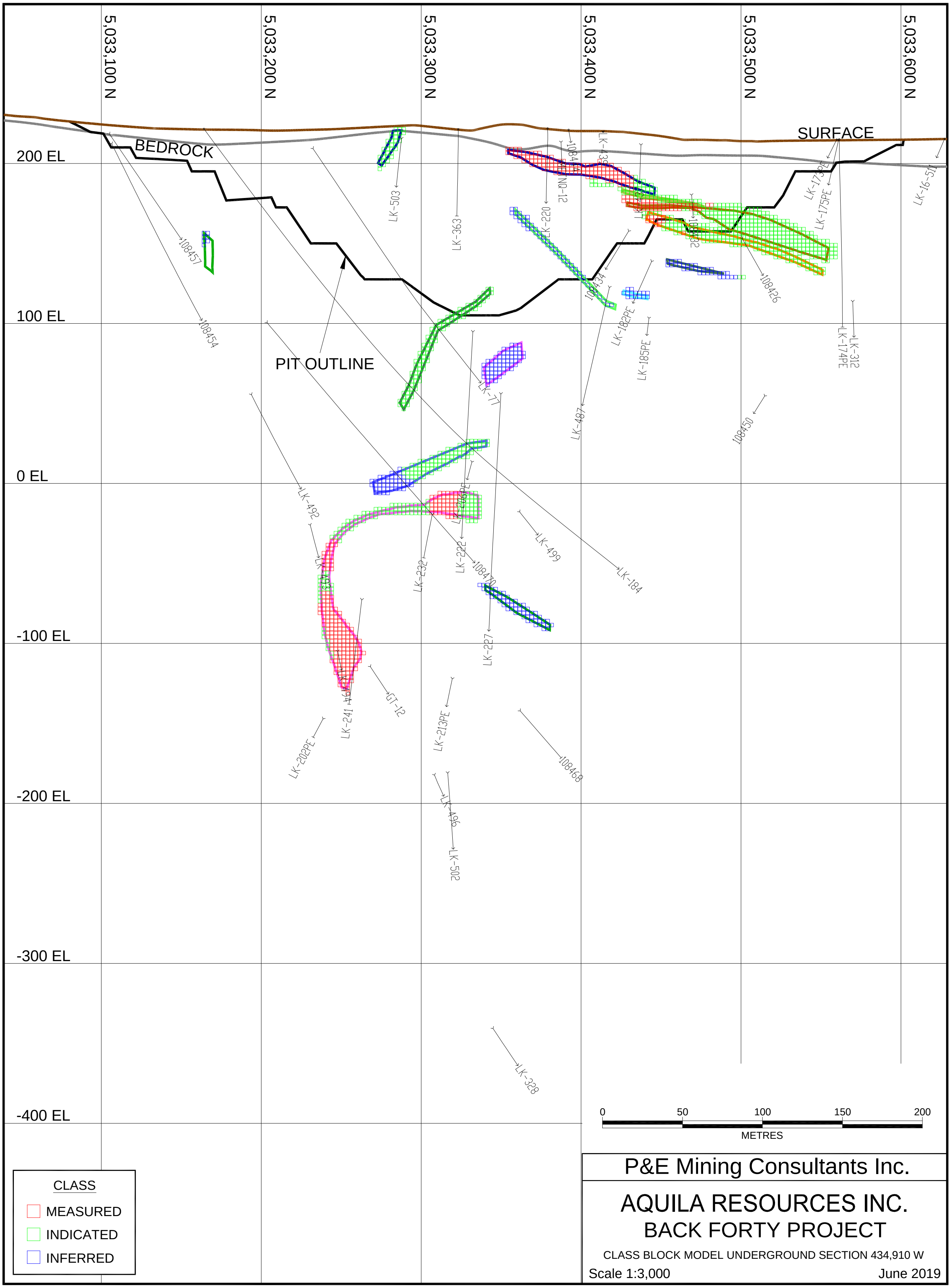


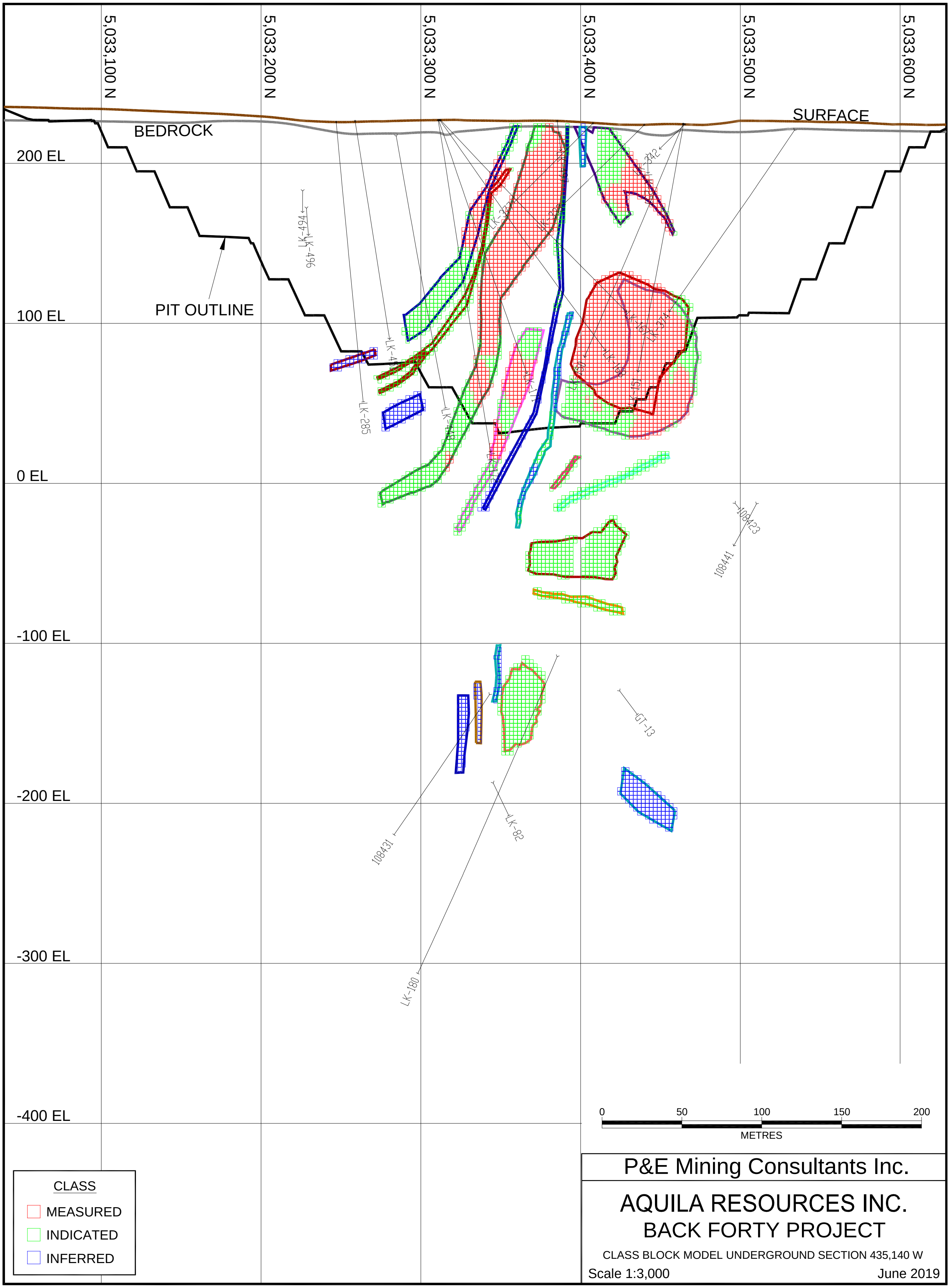
P&E Mining Consultants Inc.

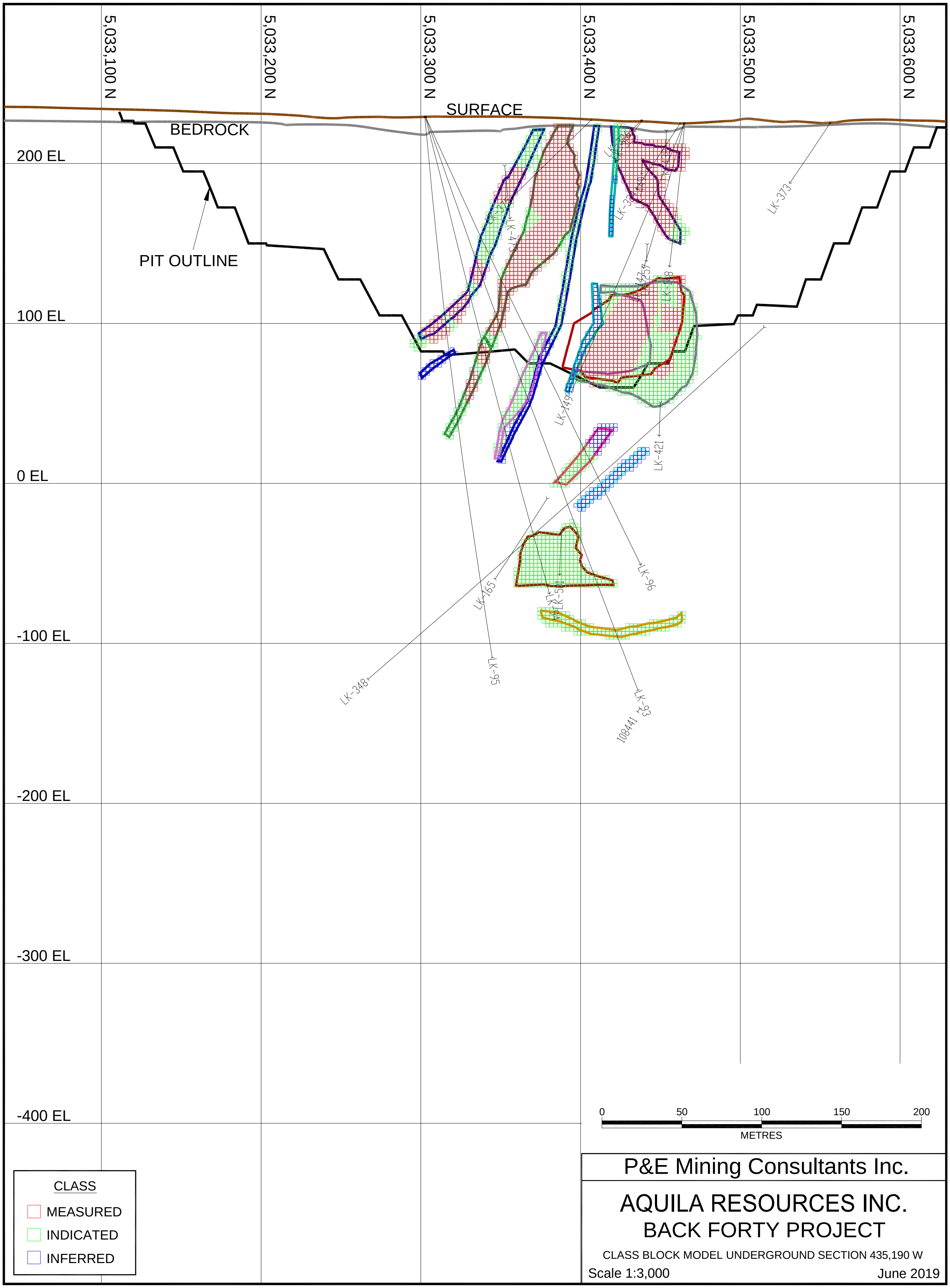
AQUILA RESOURCES INC.
BACK FORTY PROJECT

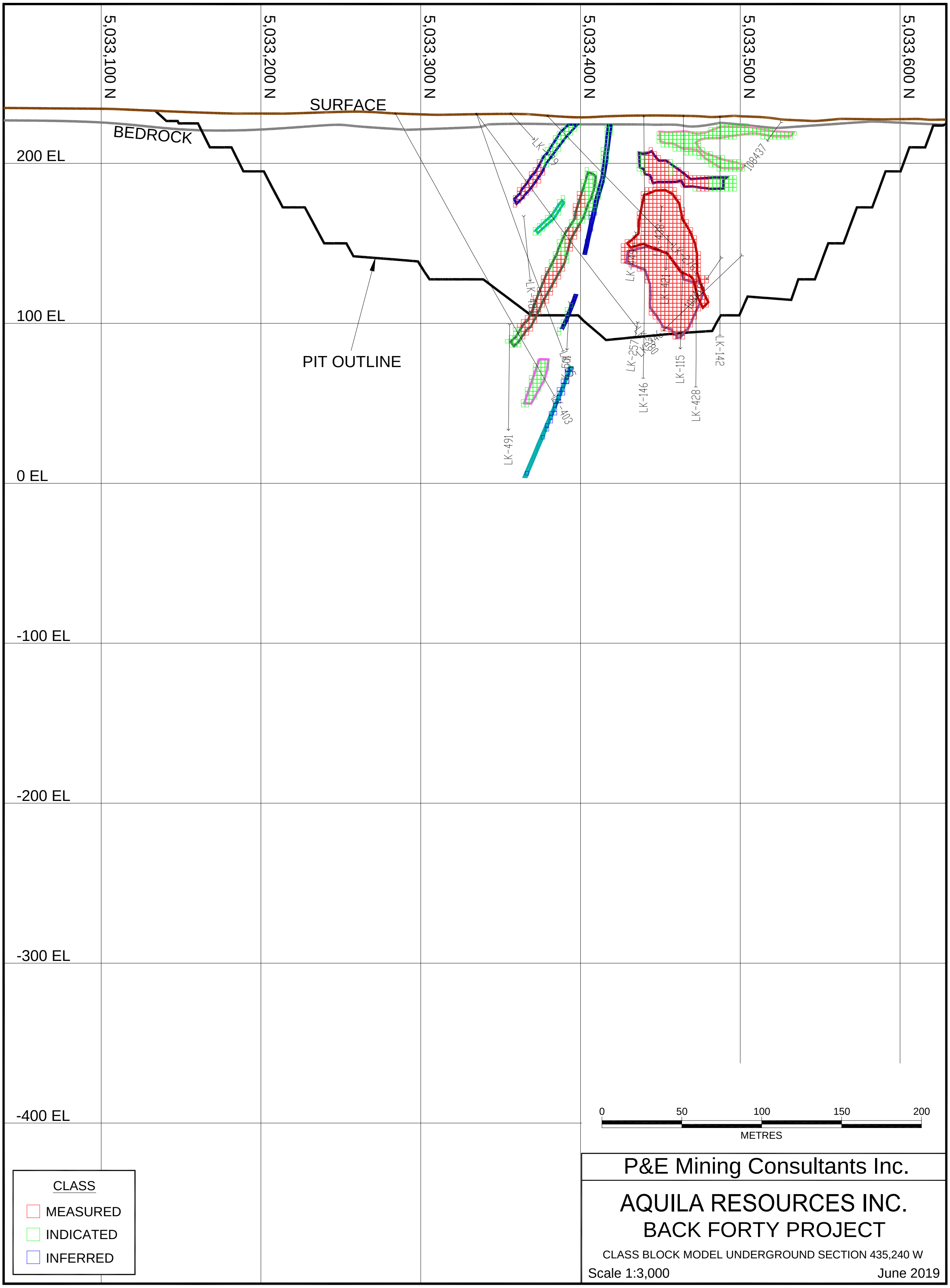
CLASS BLOCK MODEL UNDERGROUND SECTION 434,860 W

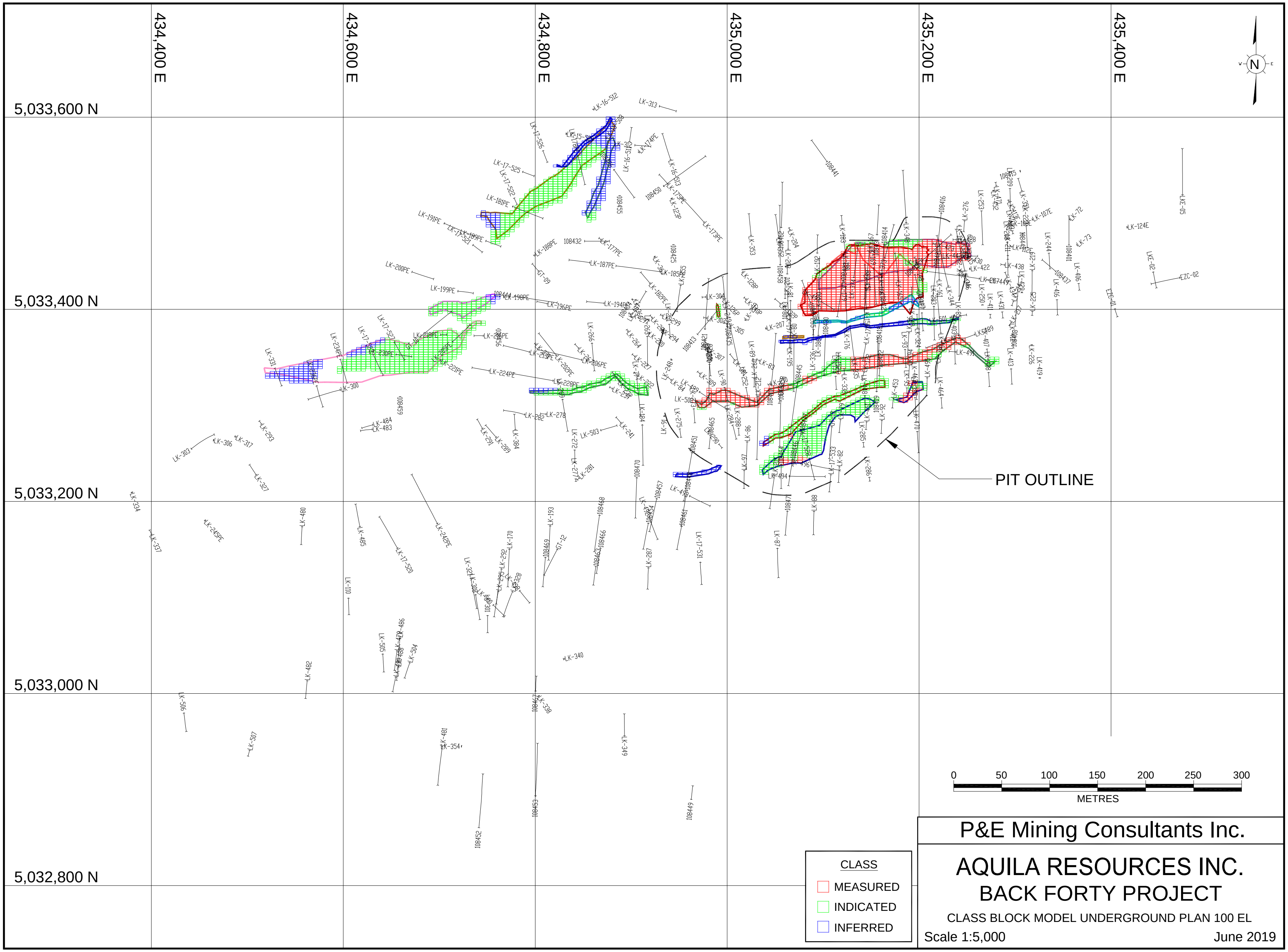
Scale 1:3,000 June 2019

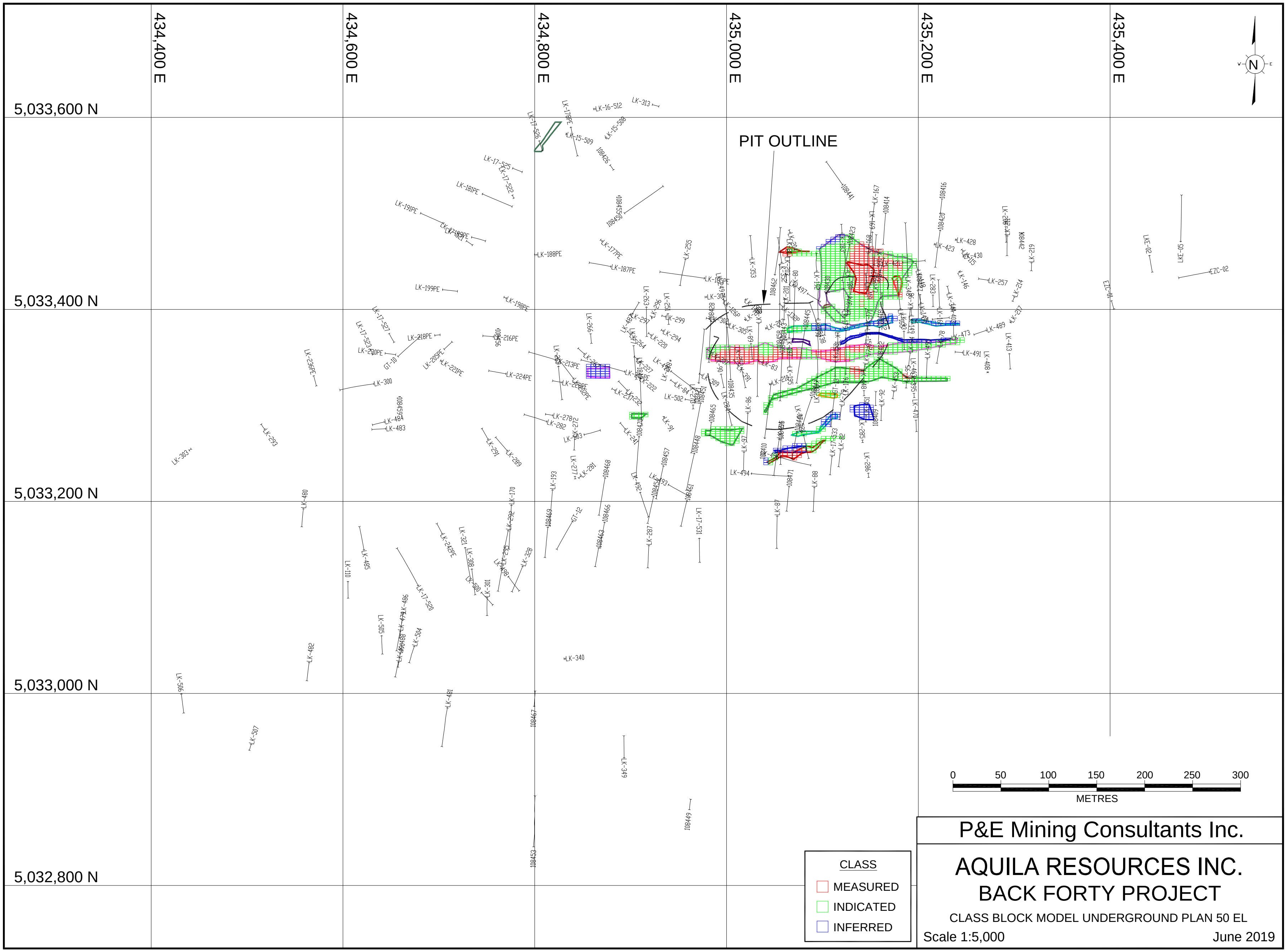

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
CLASS
MEASURED
INDICATED
INFERRED
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND SECTION 434,910 W
Scale 1:3,000
June 2019


5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
PIT OUTLINE
LK-494
LK-496
LK-285
LK-82
LK-180
108431
108441
108423
GT-13
0 50 100 150 200
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND SECTION 435,140 W
Scale 1:3,000
June 2019

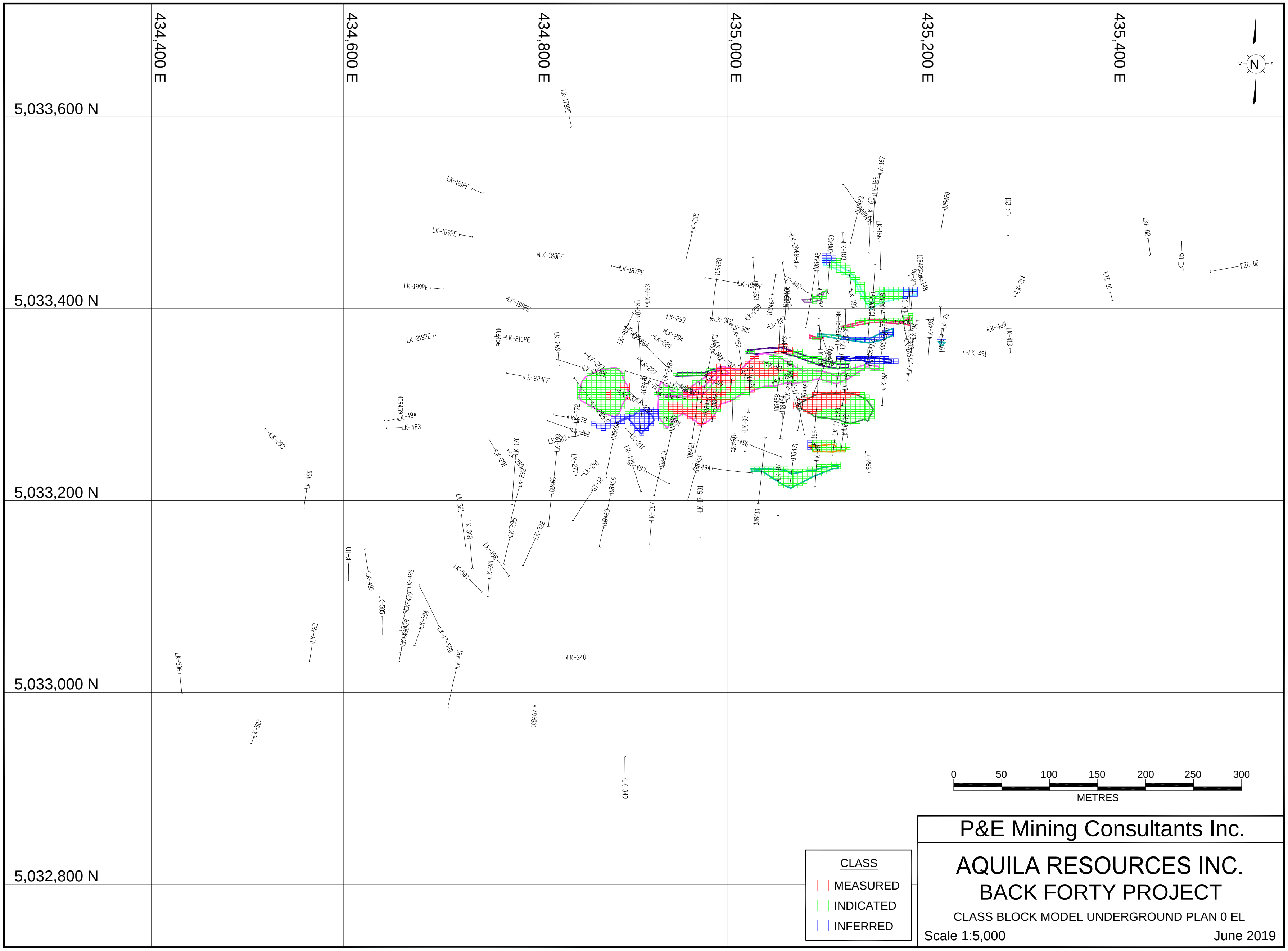

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
PIT OUTLINE
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
LK-373
LK-421
LK-96
LK-95
LK-93
LK-348
LK-165
108441
0
50
100
150
200
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND SECTION 435,190 W
Scale 1:3,000
June 2019

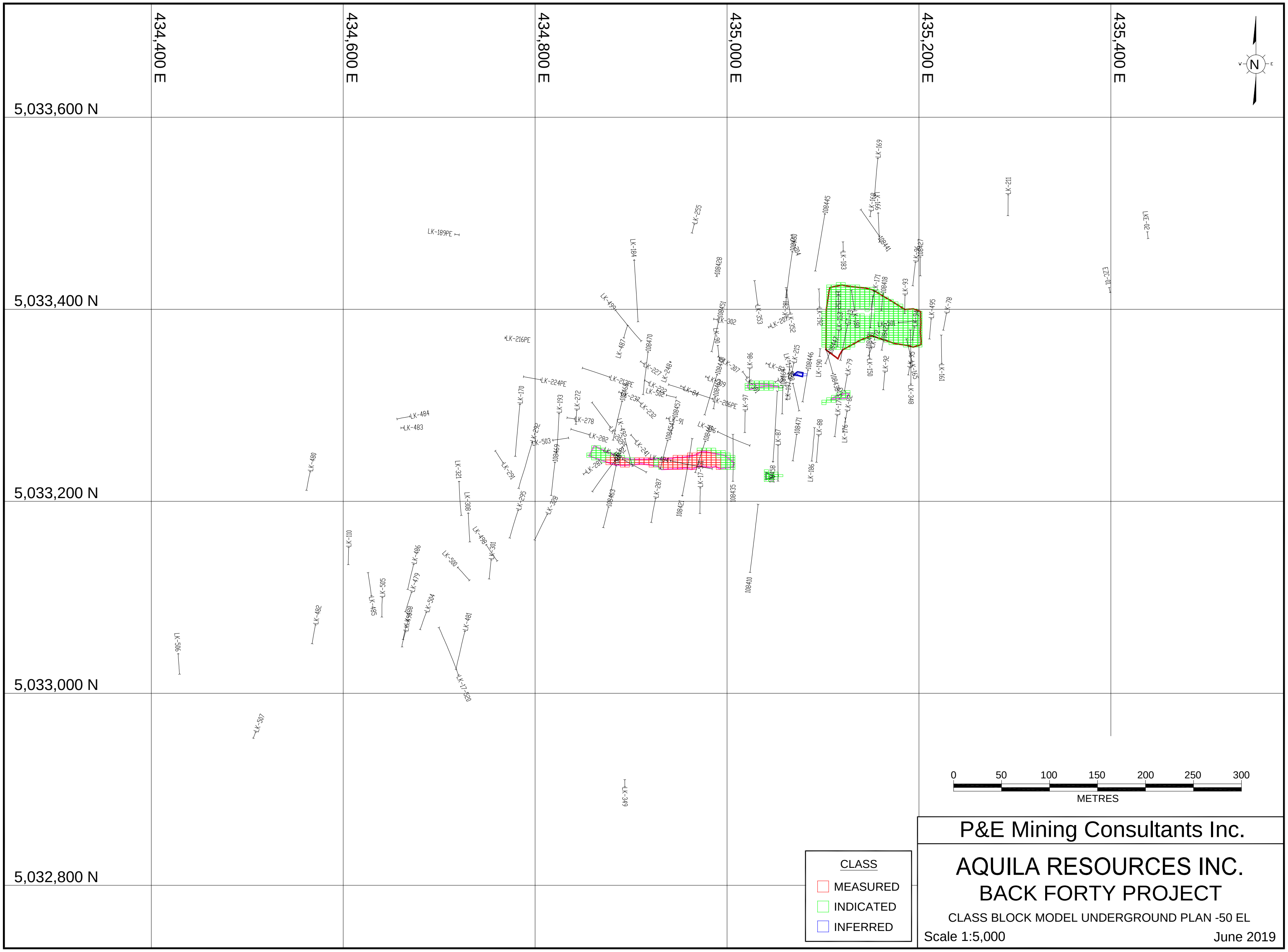

5,033,100 N
5,033,200 N
5,033,300 N
5,033,400 N
5,033,500 N
5,033,600 N
SURFACE
BEDROCK
200 EL
100 EL
0 EL
-100 EL
-200 EL
-300 EL
-400 EL
PIT OUTLINE
LK-491
LK-257
LK-146
LK-115
LK-428
LK-142
LK-403
108437
0
50
100
150
200
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND SECTION 435,240 W
Scale 1:3,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
0 50 100 150 200 250 300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND PLAN 100 EL
Scale 1:5,000
June 2019



434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
PIT OUTLINE
0
50
100
150
200
250
300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND PLAN 50 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
0
50
100
150
200
250
300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND PLAN 0 EL
Scale 1:5,000
June 2019


434,400 E
434,600 E
434,800 E
435,000 E
435,200 E
435,400 E
5,033,600 N
5,033,400 N
5,033,200 N
5,033,000 N
5,032,800 N
N
0
50
100
150
200
250
300
METRES
CLASS
MEASURED
INDICATED
INFERRED
P&E Mining Consultants Inc.
AQUILA RESOURCES INC.
BACK FORTY PROJECT
CLASS BLOCK MODEL UNDERGROUND PLAN -50 EL
Scale 1:5,000
June 2019