ANACONDA MINING INC.
REPORT NUMBER: RPT-01_R1

GOLDBORO GOLD PROJECT

RESOURCE UPDATE PHASE 2
GUYSBOROUGH COUNTY, NOVA SCOTIA

DECEMBER 18, 2019

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ANACONDA MINING INC.

PROJECT NO.: 191-03382-00_RPT-01_R1
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ABBREVIATIONS

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in
cubic metre	m^3
cubic yard	yd^3
Coefficients of Variation	Cvs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	Ba
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	CAN$
dry metric ton	mt
foot	ft
gallon	gal
gallons per minute	gpm
Gigajoule	GJ
Gigapascal	GPA
Gigawatt	GW
Gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kiloton	kt
kilovolt	kV
kilovolt-ampere	kVa
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	Mpa
megavolt-ampere	Mva
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s

short ton (2,000 lb) .... st
short tons per day .... st/d
short tons per year .... st/y
specific gravity..SG
square centimetre ....cm^2
square foot ....ft^2
square inch...in^2
square kilometre...km^2
square metre ....m^2
three-dimensional.. 3D
tonne (1,000 kg) (metric ton)............................. t
tonnes per day .... t/d
tonnes per hour .... t/h
tonnes per year .... t/a
tonnes seconds per hour metre cubed ...... ts/hm^3
volt..V
week..wk
weight/weight .... w/w
wet metric ton... wmt

ACRONYMS

AA .... Atomic Absorption
ALS .... ALS Canada Ltd.
Anaconda .... Anaconda Mining Inc.
Ausenco .... Ausenco Engineering Canada Inc.
BMA .... Bulk Mineralogical Analysis
BR .... Boston-Richardson
BWI .... Bond Ball Mill Work Index
Cementation .... Cementation Canada Inc.
CGL .... Continuous-Grind-Leach
CIM .... Canadian Institute of Mining, Metallurgy and Petroleum
CLC .... the Goldboro Gold Project Community Liaison Committee
CND .... Cyanide Destruction
CRM .... Certified Reference Material
CWI .... Crushing Work Index
EG .... East Goldbrook
FA .... Fire Assay
Horvath Consulting .... A.S. Horvath Consulting
ID^2 .... Inverse Distance Squared
InnovExplo .... InnovExplo Inc.
IP .... Induced Polarization
KMKNO .... Kwilmu'kw Maw-klusuaqn Negotiation Office
LNG .... Liquefied Natural Gas
Mercator .... Mercator Geological Services Limited
MineTech.. MineTech International Limited
Minnova .... Minnova Exploration Inc.
MODG .... the Municipality of the District of Guysborough
MOU .... Memorandum of Understanding
NN .... Nearest Neighbour
Novagold .... Novagold Resources Inc.
NSDNR .... Nova Scotia Department of Natural Resources
OK .... Ordinary Kriging
Onitap .... Onitap Resources Inc.
Orex.. Orex Exploration Inc.

Osisko	Osisko Mining Corporation
P&E Mining	P & E Mining Consultants Inc.
PEA	Preliminary Economic Assessment
Placer	Placer Dome Inc.
Property (the)	Goldboro Property (the)
QP	Qualified Person
Rock Labs	Rock Labs Ltd.
Tech2Mine	Tech2Mine Inc.
Thibault	Thibault & Associates Inc.
VL	Vat Leaching
WG	West Goldbrook
WSP	WSP Canada Inc.

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1 SUMMARY

1.1 CURRENT TECHNICAL REPORT

Anaconda Mining Inc. (Anaconda) retained WSP Canada Inc. (WSP) in February of 2019 to prepare an updated, independent mineral resource estimate in accordance with National Instrument 43-101 (NI 43-101) for the company's Goldboro Gold Deposit, located in eastern Nova Scotia, Canada. The purpose of the new estimate was to reinterpret the geology and the mineralized belts based on updated diamond drilling and revised interpretation. The primary differences between the 2018 mineral resource model and the current mineral resource model is the increase in the number of boreholes, addition of new belts, and re-interpretation of the geology.

The Issue Date of this report is December 18, 2019. The effective date of the mineral resource estimation is August 21, 2019.

1.2 PROJECT DESCRIPTION, LOCATION, AND ACCESS

The Goldboro Property (the Property) is situated on the eastern shore of Nova Scotia, Canada, with the central point of the Property being approximately located at 45° 12' 2.6" N latitude and 61° 39' 2.0" W longitude. The Property consists of 37 contiguous claims covering a total area of approximately 592 hectares held under Exploration Licence No. 05888. This title is in its 40th year of issue in 2019.

The Property is located approximately 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro, on the eastern shore of Isaac's Harbour, in Guysborough County, Nova Scotia, Canada. The elevation is nominally 70 m above sea level. All-weather Highway 316 links the village of Goldboro to the town of Antigonish. A secondary gravel road (Goldbrook Road), accessed from Highway 316, crosses the Property and passes near the historic Boston-Richardson shaft and exploration decline. Smaller logging roads and trails provide good access to most areas of the Property.

Gold mineralization at the Goldboro Project is hosted within 3 key mineralized zones namely the West Goldbrook Gold System (WG Gold System), the Boston-Richardson Gold System (BR Gold System), and the East Goldbrook Gold System (EG Gold System).

1.3 HISTORY

Gold mineralization on the Property was first discovered in 1862 by Howard Richardson of the Geological Survey of Canada in quartz veins within the Isaac's Harbour anticline. The gold-bearing Boston-Richardson belt (slate and quartz) was subsequently discovered by Richardson in 1892. The Richardson Gold Mining Company began production from the belt in 1893 at an average reported grade of 0.38 oz. of gold per short ton (13.03 g/t Au;) milled. Milling recoveries were reported to be in the 50 to 60% range.

From 1901 to 1905, three gold-bearing belts were intersected in the Dolliver Mountain mine, located 2 km west of the Boston-Richardson mine. In 1904, 205 oz. (6,376 g) of gold were recovered from 8,059 short tons milled, producing an average gold grade of 0.87 g/t. Work at Dolliver ceased in 1905 due to unfavourable drilling results. In 1907, the East Goldbrook property that adjoins the Boston-Richardson property to the east was acquired by F.S. Andrews and others. A shaft was sunk 175 feet (53 m), and three promising gold-bearing belts were explored in 1908. One of these was reported as being well-mineralized but no other work was carried out on the property at that time. Operations were suspended on August 15, 1908 due to financial difficulties but were later resumed.

From 1909 to 1910, the West Goldbrook exploration shaft intersected five gold-bearing belts. Three of these were mill tested but results were unsatisfactory, and the mine was abandoned.

Government records show total gold recovery from 1893 to 1910 for the property to be 54,871 ounces (1,707 kg) from 414,887 short tons of material milled (376,303 t), with this producing an average recovered gold grade of 4.11 g/t. However, mill recovery is reported to be approximately 67 % (Roy, 1998). Intermittent activities on the property between 1910 and 1981 included metallurgical test work, reprocessing of mine tailings, shaft sinking, and cross-cutting.

In 1981, Patino Mines (Quebec) Ltd. completed a geophysical program covering the Upper Seal Harbour district. In 1984, Onitap Resources Inc. (Onitap) acquired 37 claims overlying the property. Between 1984 and 1988, Onitap conducted diamond drilling programs, airborne VLF-EM surveys and surface Induced Polarization (IP) surveys. During this period several new mineralized belts were discovered.

Orex Exploration Inc. (Orex) acquired the Goldboro Property from Onitap in 1988 and, with the exception of a period of inactivity from 1996 to 2004, since that time has actively pursued both surface and underground exploration programs, including large amounts of core drilling, metallurgical testing programs, resource estimation programs and economic assessments of the property. The most recent major exploration effort consisted of an extensive core drilling assessment of the property that was carried out by Osisko Mining Corporation (Osisko) under terms of agreement with Orex during the 2010 to 2012 period.

In March of 2017, Anaconda acquired control of the Goldboro Property under terms of a court-approved plan of Arrangement whereby Orex became a subsidiary of Anaconda. Work programs carried on in 2017, 2018 and 2019 by Anaconda are summarized in Section 1.5 Exploration.

1.4 GEOLOGICAL SETTING, MINERALIZATION, AND DEPOSIT TYPES

The Goldboro Property is underlain by folded sedimentary rocks of the Cambro-Ordovician Goldenville Formation of the Meguma Group. This formation consists of interbedded greywacke, arenite, and argillite (slate) that are affected by the east-west trending, upright, Upper Seal Harbour Anticline.

Quartz vein systems associated with the hinge zone of the shallowly east-plunging Upper Seal Harbour Anticline are the most important hosts for gold in this district, but gold values are also present in hosting argillite units in association with disseminated sulphides and adjacent to some vein contacts. Mineral resources reported below occur in three spatially contiguous zones along the Upper Seal Harbour Anticline. In combination, these comprise the total Goldboro Deposit for current reporting purposes and consist of the WG Gold System, the BR Gold System, and the EG Gold System. Each system is characterized by stacked, gold-bearing quartz-veined stratigraphic intervals that can be correlated both along strike and down dip.

Veins at Goldboro, which form during deformation, present three major geometries commonly referred to as 'reefs', these being saddle reefs, leg reefs, and spur reefs. Saddle reefs occur about the apex of the fold and are commonly the dominant vein types within some Nova Scotia gold districts. Leg reefs extend down the limbs of the fold, beyond the saddle reefs and are generally parallel with the argillite layers. They may also be identified as bedding parallel or 'BP' veins. Spur reefs are veins that cross between layers and may be in the apex of the fold or on its limbs.

The Goldboro Deposit contains all three types of quartz vein types outlined above but is also characterized by mineralization within the host argillite units. Because the Goldboro Deposit contains saddle, leg, and spur reefs and has gold within the argillite hosting the veins, it contains significantly more gold resources than deposits that contain gold only in the reefs and not in the host argillite.

The Goldboro Deposit contains at least 57, stacked, quartz veining-argillite belts that vary in thickness from less than 1 metre up to 20 metres. These are folded into a tight, gently east-plunging, anticline referred to as the Upper Seal Harbour Anticline. The EG and BR Gold Systems are separated by a thick greywacke sequence (the Marker Horizon) with the EG Gold System above the greywacke and the BR Gold System below. The WG Gold System is separated from the BR Gold System by a fault zone and represents the continuation of the BR Gold System on the west side of the fault. The trace of this Upper Seal Harbour Anticline crosses the Property and is found near the historical Dolliver Mountain mine, several kilometres to the west of the Goldboro Deposit demonstrating that the structure which hosts gold continues for several kilometres.

The turbidite-hosted gold deposits of Nova Scotia have been compared to similar-age turbidite-hosted quartz vein deposits elsewhere in the world, particularly those in the Bendigo and Ballarat areas of the Lower Paleozoic Lachlan Fold Belt in the state of Victoria, Australia, and have historically been similarly classified. This deposit class is identified as a member of the 'Turbidite-hosted, quartz-carbonate vein deposit (Bendigo Type)' category. Categorization within the USGS classification system of mineral deposits places the Goldboro Deposit in the broad 36A category of 'Low-Sulphide Gold-Quartz Vein Deposits'.

1.5 EXPLORATION

Anaconda acquired its interest in the Property early in 2017 under terms of a court-approved Plan of Arrangement whereby Orex became a subsidiary of Anaconda. On this basis, work completed by Orex and others prior to the acquisition is considered historic in terms of current NI 43-101 technical reporting. A summary of historic exploration was presented in Section 1.3 History.

Work completed by Anaconda on the Property since its acquisition in March of 2017 includes three drilling programs identified as Phase I, Phase II and Phase III, the completion of several mineral resource updates, a Preliminary Economic Assessment (PEA) of the Goldboro Project, and a 10,000 tonnes bulk sample program.

The Phase I core drilling program (holes BR17-01 to 05) was designed to obtain material for metallurgical testing and geotechnical analysis, and to initiate deposit exploration. The larger Phase II program during late 2017 and 2018 focused on infill and extension drilling (holes BR17-06 to 13 and BR18-14 to 42). Phase III program (BR18-43 to 99) was completed to delineate resource in the Boston-Richardson and EastGoldbrook domains.

In addition to the drilling and associated metallurgical programs, the company retained Mercator in 2017 to prepare an updated mineral resource estimate, Thibault to carry out metallurgical test work, WSP to prepare a PEA based on Mercator 2017 resource, and WSP was retained in 2018 to update the resource after Phase II drilling program.

1.6 DRILLING

A total of 65,968 m of surface and underground diamond drilling was completed between 1984 and 2011. Orex was corporately involved in all programs from 1988 through 2011, and earlier programs were carried out by Onitap, Petromet Resources Ltd., and Greenstrike Gold Corp.

In 2010, reverse circulation (RC) drilling equipment was used by Osisko to explore near-surface gold mineralized structures on the Property by recovering basal till and bedrock samples for gold assaying and whole-rock analysis. The program consisted of 64 RC drillholes completed in the East Goldbrook, Boston-Richardson Ramp, and West Goldbrook areas. Assay results from the RC drill program were not used for the resource estimate.

During the summer of 2017, Anaconda completed a five-hole (BR-17-01 to 05), 643 m, diamond drill program that tested the BR and EG Gold Systems. Drilling of the five holes was completed in order to collect samples for metallurgical test work on mineralization, with each of the completed holes twinning an historic drillhole.

In addition to the metallurgical drilling program described above, Anaconda completed thirty-nine additional core holes on the Property (BR-17-06 to BR-17-13 and BR-18-14 to BR-18-42) totaling 11,712.9 m of drilling.

The 2017-2018 drill program focused on infilling areas of Inferred resources as outlined in the 2018 PEA filed on March 2, 2018 and expanding the Goldboro Deposit along strike and down plunge, and at depth along the host fold structure. Drilling has focussed on testing the down-plunge, down-dip and along strike extension of the BR Gold System and the EG Gold System. Drilling also focussed on infilling under-drilled areas of the deposit in order to upgrade mineral resources from the Inferred to Indicated category with focus on sections 9000E, 9050E, 9100E, 9150E, 9250E, 9350E, 9450E, 9500E, 9550E and 9650E.

The 2018-2019 drill program focused on infilling areas of Inferred mineral resources and expanding the Goldboro Deposit along strike and at depth along the host fold structure. Infill drilling targeted under-drilled areas of the deposit in order to upgrade mineral resources from the Inferred to Indicated category within the WG and EG Gold Systems. Drilling also focussed on testing the down-plunge, down-dip and at depth expansion of the WG, BR and EG Gold Systems. From July 2018 to August 2019 Anaconda completed 63 drillholes (BR-18-43 to BR-18-71 and BR-19-72 to BR-19-104) on the Property totaling 15,851.5 m of drilling.

1.7 SAMPLING, ANALYSIS, AND DATA VERIFICATION

Sample preparation, analysis, and security discussions for all drilling programs carried out on the Property prior to 2010 were addressed in previous NI 43-101 mineral resource estimate technical reports completed by Gervais et al. (2009), Puritch et al. (2006), Bourgoin et al. (2004), Cullen and Yule (2013, 2017) and Robinson et al. (2018). Drillholes from programs completed between 1984 and 2011 are included in the database used for the current resource estimate. The sampling approaches in programs carried out prior to 2005 generally reflect sampling of visibly determined mineralized belts, respective of major geological units, plus varying amounts of adjacent material. Exceptions to this, which include continuous core sampling and/or total core rather than half core sampling, pertain to certain historic metallurgical programs. Continuous mineralized zone sampling, respective of major lithologic units, pertains to 2005 and later programs.

Drill core samples from surface drilling programs carried out in 2005 (HQ core) and 2008 (NQ core) were generated by Orex during this period. Samples were sent to ALS Canada Ltd. (ALS) facilities in either Val-d'Or, Québec (2005) or Timmins, Ontario (2008). Standard rock sample crushing and grinding procedures at ALS were followed by initial fire assay (FA) fusion Atomic Absorption (AA) finish analysis of 50 g pulp splits. If the initial result met or exceeded a 2.5 g/t gold threshold, analysis of a second coarse reject split was carried out using a gravimetric finish. Composite metallurgical samples were created from coarse reject materials selected by Orex consultants and these were submitted to SGS Lakefield for whole sample metallurgical testing. A quality assurance and quality control program that included analysis of Certified Reference Material (CRM), field duplicates, coarse reject duplicates, pulp split duplicates, and blank samples was carried out with respect to both the 2005 and 2008 programs, and results of these programs are presented in the report.

The 2010-2011 Osisko program was carried out under project supervision of Mr. J. Lafleur, P. Geo. and site supervision by consultant Mr. Bruce Mitchell, P. Geo. W.G. Shaw and Associates Ltd. provided most core logging, sample cutting, and field support staff for both programs and Mercator supplied one P. Geo. staff geologist to assist with the 2011 core logging. All the NQ-sized core was logged, photographed, sampled, bagged, tagged, and sealed at the Goldboro site by qualified personnel. Logging utilized Gemcom Gems™ Logger software, and project protocols included progressive, systematic, and secure offsite backup of digital drilling, logging, and sampling data. At ALS, each sample was crushed to 70% < 2 mm, split to 250 g using a riffle splitter, pulverized to 85% at < 0.075 mm, and made into a 50 g sample of the pulp. The 50 g pulp was fire assayed with atomic absorption spectrometry finish (ALS codes Au-AA24 and Au-AA26). Samples exceeding the atomic absorption spectrometry threshold were re-assayed using a gravimetric finish (ALS code Au-GRA22). All samples containing visible gold were directly assigned for processing using the total metallic screen method with FA and AA or gravimetric finish.

Review of assessment reporting related to the various drilling programs carried out during the 1984 to 2005 period showed that, with the exception of the metallurgical and check sampling program carried out by Placer in 1995, no structured programs designed to systematically monitor quality control and assurance issues for drill core were in place. Orex drilling programs in 2005 and 2008 and Orex-Osisko programs in 2010 and 2011 were subject to rigorous QA/QC programs, with some procedural changes incorporated during the period.

During the 2017, 2018 and 2019 Anaconda programs, drill core samples were collected systematically down the hole based on the occurrence of visual alteration, mineralization and quartz veining. Samples ranged in length from 0.3 to 1.0 m depending on the nature and width of veining and mineralization samples, while trying to best honour geological contacts. Samples were collected of half-sawn drill core and shipped to Eastern Analytical Limited in Springdale, Newfoundland and Labrador for analysis via standard 30 g FA-AA finish. Samples were also analyzed at Eastern Analytical Ltd. via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold, and samples up to drillhole BR-18-62 were submitted for 34-element ICP analysis. Check assays on all sample pulps assaying >0.5 g/t Au were completed at ALS for the 2017 and 2018 drill programs.

1.7.1 DATA VERIFICATION

Core sample records, lithologic logs, laboratory reports and associated drillhole information for all drill programs completed in the 1984 to 2011 period were digitally compiled for use in Gemcom-Surpac Version 6.2.1® (Surpac™) deposit modeling software. Historic and current drilling program information was reviewed and digital records of historic drilling were checked for both consistency and accuracy against original source documents available through NSDNR or received from Orex. All 2010, 2011, 2017, 2018 and 2019 drillhole coordination and orientation data inputs were checked, and validation of approximately 20% of the assay dataset for sample interval and assay value information against corresponding source documents was carried out.

After completion of all manual record checking procedures, the drilling and sampling database records were further assessed through digital error identification methods available through the Surpac™ modeling software. The digital review and import of the manually checked datasets through Surpac™ provided a validated Microsoft Access® database that Mercator and WSP considered to be acceptable with respect to support resource estimation programs.

In January of 2013, Mercator staff completed a site visit at Goldboro during preparation of the 2013 resource estimate. An independent check sampling program consisting of 22 quarter-cut core samples was completed during the visit. The check sample program results are interpreted as confirming the general mineralized character of the core intervals tested, with new data showing a low bias in most cases. This is considered a reflection of 'nugget-effect' that is a well-documented characteristic of gold mineralization on the Goldboro Property. A drillhole location check of 17 collar coordinates was also completed during the site visit with acceptable results.

Todd McCracken, Independent Qualified Person of WSP completed a site visit in October 2018 and in July 2019 to the Property and reviewed diamond drill core and sampling procedures for the ongoing Anaconda drilling program.

1.8 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing was completed to quantify the metallurgical response of samples from deposits in the Goldboro project. The testing focused on samples from Boston-Richardson and East Goldbrook deposits. The program was designed with the intent to develop the parameters for process design criteria for comminution, gravity concentration, leaching, carbon adsorption, cyanide destruction and carbon elution, and gold refining in the process plant. The metallurgical program was performed on the following composites:

— Boston-Richardson: low grade, medium grade, and high grade;
— East Goldbrook: marginal grade, low grade, medium grade, and high grade.

The composites were selected by Anaconda geologists with input from metallurgists to represent the spatial distribution, head grades, and mineralization types of the Goldboro project.

Boston-Richardson samples were tested through a suite of grinding characterization tests including Bond low energy, rod mill and ball work index, and Bond abrasion index tests. The Boston-Richardson samples were selected to complete SMC tests. The East Goldbrook samples were used for Bond ball mill work index variability tests.

The program included grind optimization, gravity concentration, leaching studies on the Boston-Richardson composites. The East Goldbrook composites were tested as variability samples to evaluate the optimum conditions determined through the Boston-Richardson tests. An overall Master Composite was assembled for bulk leach tests to provide feed material for cyanide destruction testing and carbon adsorption testing. Tailings solids samples were then used for environmental testing. An overall recovery model was derived from the test data for use in process design and mine scheduling.

The testing showed the samples to be amenable to standard crushing, grinding, gravity concentration and leaching for gold recovery. The testing showed that significant portions (>50%) of the contained gold could be recovered with gravity concentration. An overall recovery model was derived from the test data for use in process design and mine scheduling. Overall gold recovery will be approximately 95%, dependent of the head grade.

Cyanide detoxification and arsenic removal was tested using standard SO_2/air and ferric sulphate precipitation respectively. Both methods were found to be successful in reducing weak acid dissociable cyanide and arsenic concentrations that comply with environmental regulations. Tailings solids samples were then used for environmental testing.

1.9 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The current mineral resource estimate for the Goldboro Deposit is based on validated results of 492 surface and underground drillholes. Modeling was performed using Geovia Surpac™ 2019 modeling software with gold grades estimated using ordinary kriging (OK), inverse distance squared (ID^2) and nearest neighbor (NN) interpolation methodology and capped 1.0 m downhole assay composites. Block size is 1 m (x) by 1 m (y) by 1 m (z) with no sub-blocking allowed. The drilling-defined deposit is divided into three spatial domains for modeling purposes, these being (1) the WG Gold System, (2) the BR Gold System, and (3) the EG Gold System.

Sectional interpretations correlating folded 'belts' of argillite and quartz veining supporting a minimum gold grade of 0.50 g/t were first defined and then digitally wireframed to create three-dimensional solid model domains. A total of 16 belt domains were created for the Boston-Richardson System, 13 belt domains for the West Goldbrook System, and 25 belt domains for the East Goldbrook System. All mineralized belt domains are centered on the hinge area of the Upper Seal Harbour Anticline, which plunges 20 to 30° to the east over a strike length of 1,500 m in the deposit area. Belts have been defined to depths of up to 400 m below surface and vary in average thickness from a few metres or less in fold limb areas to tens of metres in hinge zone saddles. A digital terrain model of the top of bedrock surface was also developed to constrain the model, along with digital solid models for underground workings and an east trending fault zone that intersects the BR Gold System; the New Belt Fault.

Grade interpolation Mineral Resources was constrained within the various belt domain wireframes using four interpolation passes, utilizing a variable sized search ellipse, with contributing assay composites capped at 80 g/t gold. Multiple search ellipsoid orientations were applied in each pass to accommodate local variations in mineralization trends. These generally conform in strike and plunge of the fold axis.

Classification of the Indicated mineral resource was based in the following parameters:

— All material estimated in vario running 2
  (a) 75% search ellipse size
  (b) Minimum 3 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

  or

— Material estimated in vario running 3
  (a) 100% search ellipse size
  (b) Minimum 9 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

Classification of the Measured mineral resource was based in the following parameters:

— All material estimated in vario running 1
  (a) 40% search ellipse size
  (b) Minimum 3 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

  or

— Material estimated in vario running 2
  (a) 75% search ellipse size
  (b) Minimum 10 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

Upon completion of the classification macro, the model was reviewed and "orphan" blocks of Inferred resources were converted to Indicated resources, and "orphan" Indicated blocks converted to Inferred resource.

Block grade for the Goldboro Deposit were estimated using the methods described. Bulk Density was assigned to individual blocks based on a regression formula. Density and gold attributes for all resource blocks intersecting underground development and stoping solid models were defaulted to null values.

Subsequent application of resource category parameters set out above resulted in the mineral resource estimate statement presented in Table 1.1. Results are in accordance with Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines (the CIM Standards, 2014) as well as disclosure requirements of National Instrument 43-101.

Table 1.1 Goldboro Project Mineral Resource Statement – Effective August 21, 2019

Resource Type	Au Cut-off (g/t)	Category	Tonnes ('000)	Au (g/t)	Troy Ounces	% Change in Grade from Dec. 2018**	% Change in Ounces from Dec.2018***
Open Pit	0.5	Measured	844.00	2.40	65,200	-	-
		Indicated	111.00	2.63	9,400	-	-
		Measured + Indicated	955.00	2.43	74,600	-	-
		Inferred	22.00	2.79	2,000	-	-
Underground	2	Measured	967.00	6.08	189,200	-	-
		Indicated	2,174.00	6.22	434,800	-	-
		Measured + Indicated	3,141.00	6.18	624,000	-	-
		Inferred	2,985.00	7.12	683,200	-	-
Combined	0.50/2.00	Measured	1,811.00	4.37	254,400	3.3%	16.0%
		Indicated	2,285.00	6.05	444,200	10.0%	15.9%
		Measured + Indicated	4,096.00	5.30	698,600	6.9%	15.9%
		Inferred	3,007.00	7.09	685,100	6.9%	51.2%

Mineral Resource Estimate Notes:

1. *Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards (2014). Mineral resources that are not mineral reserves do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
2. *Open pit Mineral Resources are reported at a cut-off grade of 0.5 g/t gold that is based on a gold price of CAN$1,753/oz (~US$1,350/oz). and a gold processing recovery factor of 95%.*
3. *Underground Mineral Resource is reported at a cut-off grade of 2.0 g/t gold that is based on a gold price of CAN$1,753/oz (~US$1,350/oz). and a gold processing recovery factor of 95%.*
4. *Appropriate mining costs, processing costs, metal recoveries, and inter-ramp pit slope angles were used by WSP to generate the pit shell.*
5. *Appropriate mining costs, processing costs, metal recoveries and stope dimensions were used by WSP to generate the potential underground resource.*
6. *Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.*
7. *Tonnage and grade measurements are in metric units; contained gold ounces are in troy ounces.*
8. *Contributing assay composites were capped at 80 g/t Au.*
9. *A bulk density factor was calculated for each block based on a regression formula.*

1.10 RECOMMENDATIONS

Following up on the work completed and outlined in this report, the Phase I recommendations for the 2020 exploration program include the following.

- Phase I – Exploration:
  - Complete a hyperspectral imaging program on up to 7,500 m of core to better characterize alteration mineralogy with the goal of identifying more refined grade control techniques.

- Phase II – Upon completion of a positive feasibility study and successful project financing:
  - Complete a 300 m geotechnical drilling and site investigation around the proposal tailings storage facility and along the road re-alignment.
  - Complete 1,000 m condemnation drilling program in areas of planned mine infrastructure.
  - Complete a 10,000 m reverse circulation drill program within the volume of the conceptual open pit to upgrade all resource categorization to the Measured category.
  - Conduct a 5,000 m diamond drilling program to upgrade the first three years of underground mining outlined in the feasibility study to the Measured category.
  - Conduct a 20-line km, ground induced polarization survey, to identify the position of the alteration system along strike from the existing deposit and to identify high priority drill targets with open pit potential.

Tables 1.2 and 1.3 summarize the above recommendations and estimated costs, totaling CAN$2,575,000. Phase II work is not entirely contingent on successful completion of Phase I but could be modified in consideration of Phase I results.

Table 1.2 Phase I Summary of Recommended Work

Work Program	Estimated Cost (CAN$)
Hyperspectral Imaging	100,000
Phase I Total	**$100,000**

Table 1.3 Phase II Summary of Recommended Work

Work Program	Estimate Cost (CAN$)
Geotechnical program	200,000
Condemnation program	200,000
10,000 metre RC drill program	1,000,000
5,000 metre diamond drill program	1,000,000
Ground IP survey	75,000
Phase II Total	**$2,475,000**

2 INTRODUCTION

2.1 GENERAL

Anaconda Mining Inc. (Anaconda) retained WSP Canada Inc. (WSP) in March of 2019 to prepare an updated, independent mineral resource estimate in accordance with National Instrument 43-101 (NI 43-101) for the company's Goldboro Gold Deposit (the Deposit), located in eastern Nova Scotia, Canada. The purpose of the new estimate was to reinterpret the geology and the mineralized belts based on updated diamond drilling and revised interpretation. The updated mineral resource block model supersedes the March 2018 Preliminary Economic Assessment (PEA) study carried out by WSP and Thibault & Associates Inc. (Thibault). The updated mineral resource block model is being used for the ongoing feasibility study on the Project.

The Deposit contains at least 57 stacked mineralized zones (belts) that are folded into the tight, gently east-plunging, Upper Seal Harbour Anticline. The Deposit is divided into three broad zones or Gold Systems: the West Goldbrook (WG) Gold System, the Boston-Richardson (BR) Gold System, and the East Goldbrook (EG) Gold System. The EG and BR Gold Systems are separated by a 40 to 50 m thick greywacke sequence (the Boston-Richardson Marker Horizon or Marker Horizon), with the EG Gold System above the Marker Horizon and the BR Gold System below. The WG Gold System is separated from the BR Gold System by a N-S trending fault and represents the offset continuation of the BR Gold System on the west side of the fault. The mineral resource update on the Goldboro Project (the Project) has been prepared in accordance with National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines (*CIM, 2014*).

The Issue Date of this report is December 18, 2019. The Effective Date of the mineral resource estimation is August 21, 2019.

The Project is held by Orex Exploration Inc. (Orex), a wholly owned subsidiary of Anaconda. Anaconda's corporate offices are located at 150 York Street Suite 410, Toronto, Ontario, Canada and the company is listed on the Toronto Stock Exchange under the trading symbol ANX and the OTCQX under the trading symbol ANXGF.

2.2 QUALIFICATION OF CONSULTANTS

The consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource estimation and classification, metallurgical testing, mineral processing, and processing design.

None of the consultants or any associates employed in the preparation of this report has any beneficial interest in Anaconda. The consultants are not insiders, associates, or affiliates of Anaconda. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Anaconda and the consultants. The consultants are being paid a fee for their services in accordance with normal professional consulting practice.

2.3 QUALIFIED PERSONS

The individuals listed in Table 2.1, by virtue of their education, experience, and professional association, are considered Independent Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions.

Table 2.1 Qualified Persons

Qualified Person	Position/Title	Company	Responsibility
Todd McCracken, P. Geo.	Manager - Mining	WSP Canada Inc.	Section 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14,15, 16,18, 19, and 20 Portions of Section 1, and 17
Tommaso Roberto Raponi, P.Eng.	Senior Process Chemical Engineer	Ausenco Engineering Canada Inc.	Section 13, Portions of Section 1, and 17

2.4 DETAILS OF INSPECTION

The following Qualified Persons (QP) completed a site visit of the Project:

— Todd McCracken, P.Geo., of WSP has visited the site in October 2018, and July 2019.

— Tommaso Roberto Raponi, P. Eng., of Ausenco Engineering Canada Inc. has not visited the site.

2.5 SOURCES OF INFORMATION

The sources of information, including data and reports supplied by Anaconda personnel, as well as documents cited throughout the report, are referenced in Section 19.

The mineral resource estimate presented in this report is based on validated results of 485 surface and underground diamond drillhole totalling 93,916 metres completed between 1984 and August 21, 2019. Anaconda provided WSP with complete digital records of all exploration completed on the Project during this period. Historic exploration and mining information was assembled through a review of hard copy and digital records of historic data. This included previous NI 43-101 technical reports on the Project prepared in 2004, 2006, 2009, 2013 2017, and 2018, as well as assessment reports, drill logs, drill plans, assay records, and laboratory records for historic exploration and mining programs carried out in the area that are publicly available in hard copy or digital format. Additional information was acquired through published research and archived government files that pertain to the area and are available from Nova Scotia Department of Natural Resources (NSDNR).

2.6 UNITS OF MEASURE

The metric system has been used throughout this report. Tonnes are dry metric of 1,000 kg, or 2,204.6 lb. All currency is in Canadian dollars (CAN$), and referenced as '$', unless otherwise stated. Gold values for work performed by Anaconda and previous operators are reported as grams per tonne or parts per billion. A conversion factor of 31.1035 is used to convert grams to troy ounces. Map coordinates are given as MTM Zone 4 NAD83 coordinates (false easting of 304,800 m), although some information is given with respect to local field grid coordinates. Elevations on the projects are set using a historic mine grid of elevation above sea level + 4,940 m.

2.7 EFFECTIVE DATE

The Issue Date of this report is December 18, 2019. The Effective Date of the current mineral resource estimate is August 21, 2019.

3 RELIANCE ON OTHER EXPERTS

The QPs who prepared this technical report relied on information provided by experts who are not QPs:

— Todd McCracken, P.Geo., relied upon Anaconda for confirmation of mineral exploration title status, validity of exploration title and associated encumbrances, if any, status of land access agreements, an opinion with regard to site environmental liabilities and provision of royalty or other agreement information. This information was used in the preparation of Section 4 of this report and was confirmed for report purposes by Mr. Paul McNeill, Vice President of Exploration for Anaconda.

The relevant QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

4 PROPERTY DESCRIPTION AND LOCATION

The Goldboro Property (the Property) is situated on the eastern shore of Nova Scotia, Canada, with the central point of the Property being approximately located at 45° 12’ 2.6” N latitude and 61° 39’ 2.0” W longitude. The Property consists of 37 contiguous claims covering a total area of approximately 592 hectares held under Exploration Licence No. 05888 (Table 4.1; Figures 4.1 and 4.2). This title is in its 40th year of issue in 2019.

Table 4.1 Goldboro Property Claims held under Exploration License No. 05888

Exploration Licence No.	NTS Sheet	No. of Claims	Hectares	Expiry Date
05888	11-F-04-A	37	592	November 29, 2020

The surface rights are held by various private landowners and by the Nova Scotia government. Anaconda maintains a 100% interest in the exploration rights through its wholly-owned subsidiary Orex (with Orex having acquired the exploration rights from Onitap Resources Inc. (Onitap) in 1988), and at the effective date of this report, no lien, mortgage, royalty or other right in favour of third parties was registered with Nova Scotia Department of Natural Resources (NSDNR). NSDNR records also show that sufficient assessment work credits were available at the effective date to maintain the Property in good standing for several years. Exploration expenditures of $800 per claim per year (or the equivalent in banked expenditure credits) and renewal application fees of $320 per claim per year are required to keep Licence 05888 in good standing. WSP has not been advised of any liens, encumbrances, or royalties associated with this licence. At the report effective date, Licence 05888 was still registered to Orex and had not been transferred to Anaconda.

WSP consulted NSDNR records for purposes of this report and documented information set out above. However, reliance has been placed on Anaconda with respect to confirmation of title validity and for comment with respect to encumbrances, liens or royalties, if any, that apply to the title. A legal search of title was not carried out by WSP for current report purposes. However, WSP has no reason to question validity of the information and opinions presented above. There is no requirement to legally survey exploration licences in Nova Scotia.

Figure 4.1 General Location Map for the Goldboro Property, Nova Scotia, Canada

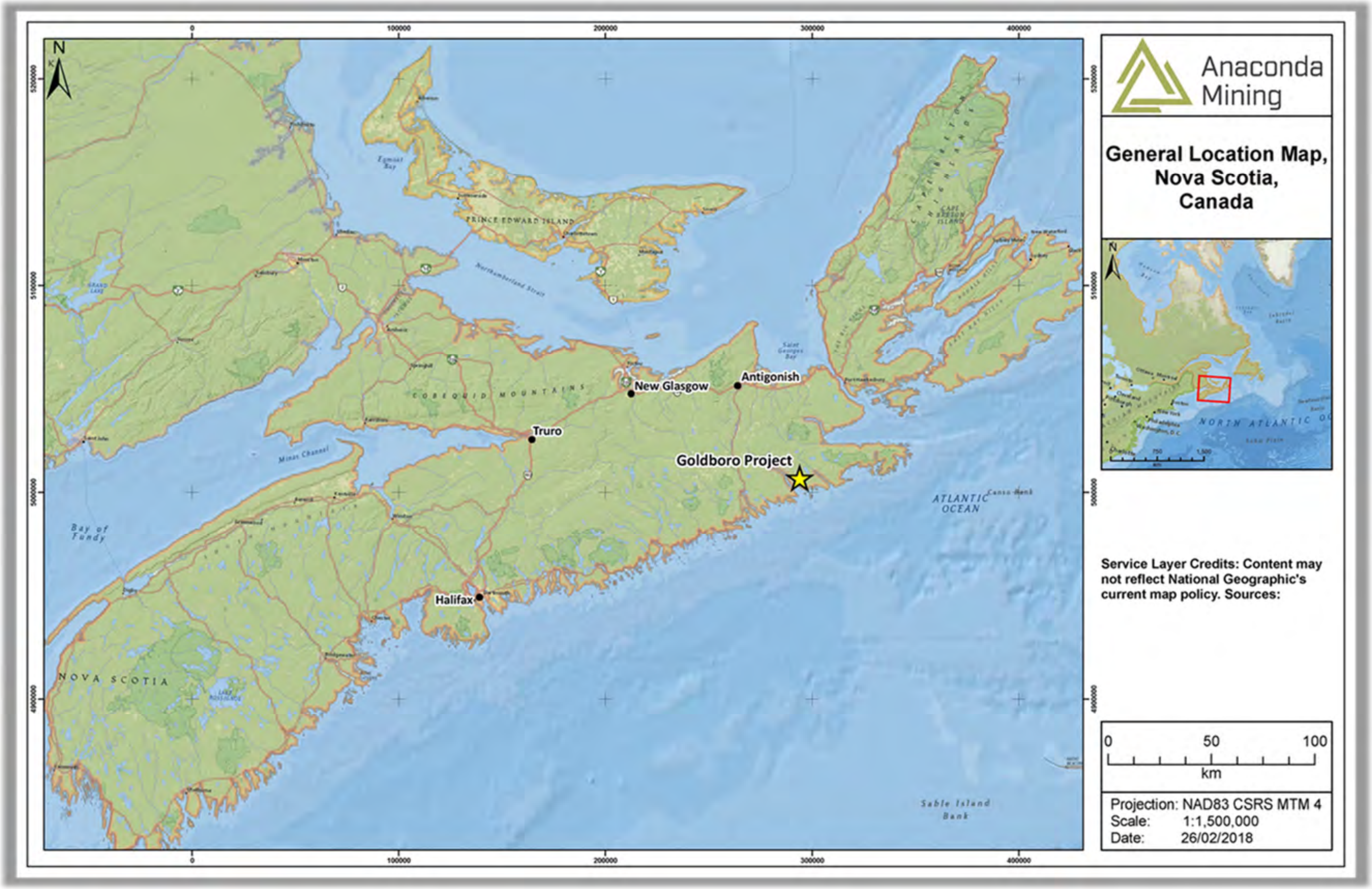
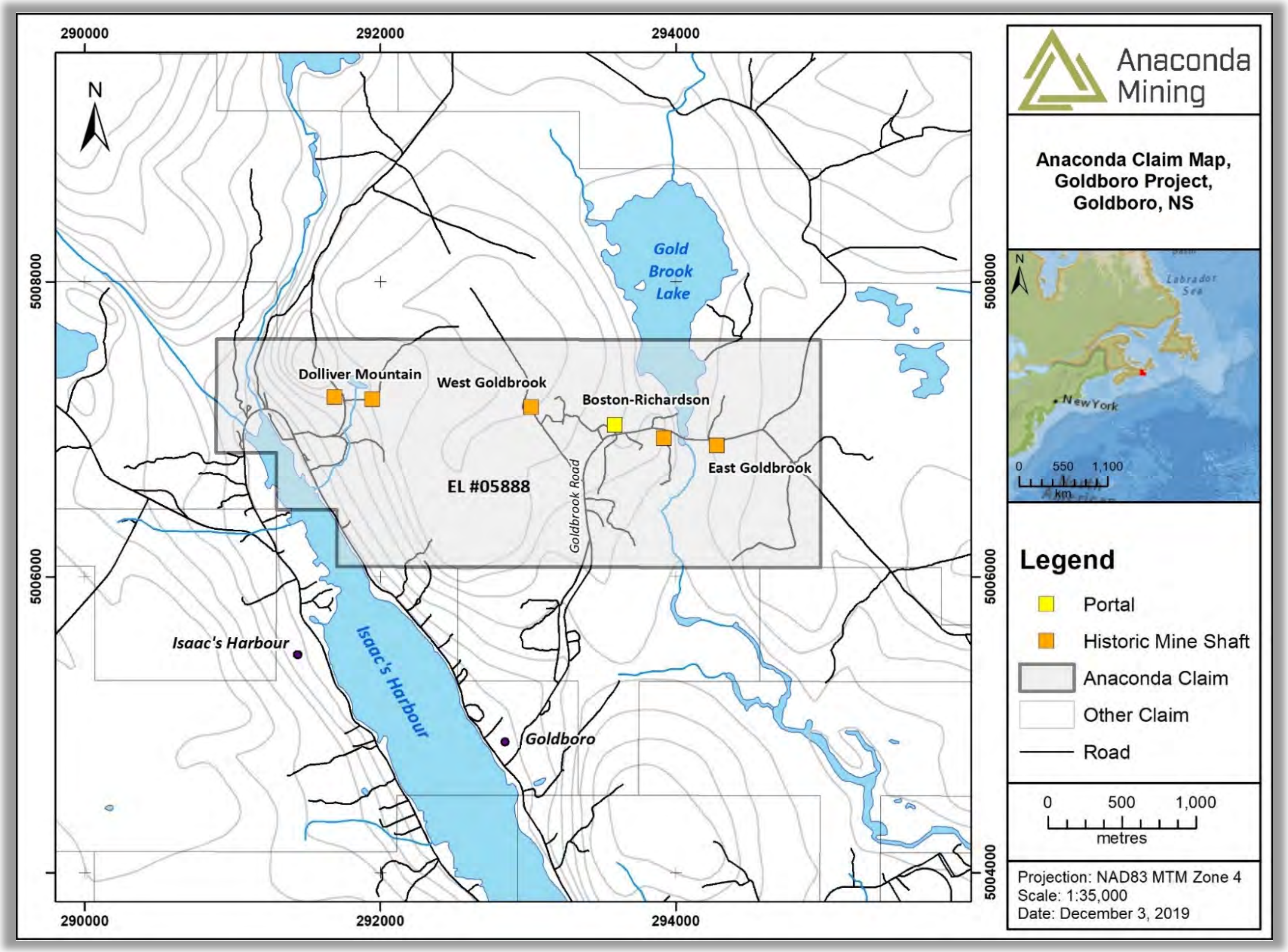


Figure 4.2 Claims Map of the Goldboro Property



4.1 ACCESS TO LANDS FOR FUTURE EXPLORATION AND DEVELOPMENT PURPOSES

To date, Anaconda has arranged access to the Property for the purpose of exploration through agreements with both private and Crown entities. Much of the Property, including all the Boston-Richardson, and East Goldbrook historic workings, is underlain by Crown Land and no difficulties have been encountered to date obtaining Crown Land permits. Similarly, access to private lands and securing agreements with landowners has not proven to be difficult. Anaconda has advised WSP that it knows of no reason that future access to the areas required for exploration would be withheld.

At the effective date of this report, Anaconda held access agreements that specifically apply to surface core drilling. Anaconda has the necessary Crown Land permits for additional drilling and trenching.

WSP is of the opinion that due to the location and size of the Property, its brownfield to undeveloped nature, and proximity to year-round habitations, it is reasonable to conclude that the site can support future mining operations and infrastructure.

4.2 ENVIRONMENTAL SITE CONDITIONS

Anaconda is not aware of any current site environmental issues resulting from the company's activities on the Property. However, the presence of past mining operation infrastructure, including several historic tailings sites associated with past operation of the historic Boston-Richardson Mine and location within the Gold Brook Lake-Seal Harbour Lake watershed are recognized as important environmental site factors. Provincial regulators indemnified Orex in 1995 from any environmental liabilities resulting from historical mining activities, assuming that old tailings storage areas are not impacted during exploration or mining activities.

No obvious environmental issues were identified during the 2018 and 2019 WSP site visits, and WSP is not aware of any subsequently developed site conditions that would materially alter the nature of this assertion.

4.3 ENVIRONMENTAL APPROVALS REQUIRED FOR FUTURE MINING

Anaconda has applied for environmental approval of a combined open pit and underground mining operation at the Goldboro site as a Class 1 registration under the Nova Scotia Environment Act in August 2018. Following the review of the Environmental Assessment application, the regulator has requested on October 15, 2018 that Anaconda provide more information in a form of a Focus Report. Based on subsequent resource growth, trade-off studies and initial results of the on-going feasibility study in late 2018 and early 2019, Anaconda decided to change the project scope and layout. Based on those changes Anaconda decided to withdraw 2018 EA registration and re-register the project with the new description early in 2020. It will also be necessary for Anaconda to make application for and receive an Industrial Approval under the same act, plus various additional permits associated with Mining and Crown Land access, mining and milling permits, water use, and sewage treatment to support authorization for future mining at this site. None of these approvals or permits had been received at the Effective Date of this report.

4.4 COMMUNITY AND STAKEHOLDER CONSULTATION

4.4.1 OVERVIEW

Productive and open relationships with all stakeholders and rightsholders is a key component of the Project. Anaconda has an active strategy for stakeholder consultation and Mi'kmaq engagement. These include, but not limited to, ongoing communications with The Assembly of Nova Scotia Mi'kmaw Chiefs, Kwilmu'kw Maw-klusuaqn Negotiation Office (KMKNO), the Municipality of the District of Guysborough (MODG), the Goldboro Gold Project Community Liaison Committee (CLC), and the general public.

4.4.2 MI'KMAQ ENGAGEMENT

The Company maintains an active information sharing relationship with officials of KMKNO and representatives of Patqnkek First Nations including more than a dozen meetings since June 2017. On July 2, 2019, Anaconda and the Assembly of Mi'kmaw Chiefs signed a Memorandum of Understanding (MOU) that will govern the process by which the parties shall negotiate a Mutual Benefits Agreement which reflects a desire to build a mutually beneficial relationship with respect to the Goldboro project. Baseline information for Indigenous Peoples was gathered through the ongoing engagement with the Mi'kmaq of Nova Scotia and completion of a Mi'kmaq Ecological Knowledge Study, with the purpose of identifying and documenting land and resource use (within 5 km of the project) which is recognized as holding great importance to the Mi'kmaq people.

4.4.3 REGIONAL PUBLIC ENGAGEMENT AND COMMUNICATION

Consultations have been ongoing with the MODG and people within the community. This includes annual meetings with MODG, quarterly newsletters sent to every household in the municipality as well as the creation of the CLC. The CLC was set up to foster environmental stewardship, and act as a vehicle for transparent and ongoing communications between community, stakeholders, and the Company on matters pertaining to potential development at the Goldboro Project and to ensure regular communication between the community and Anaconda. In addition, the Company has held public open houses within the Community. The company and municipality are working toward a Community Benefits Agreement.

5 ACCESSIBILITY, CLIMATE, LOCAL INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property is located 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro on the eastern shore of Isaac's Harbour, in Guysborough County, Nova Scotia, Canada. The central point of the Property is located at 45° 12' 2.6" N latitude and 61° 39' 2.0" W longitude. The elevation is approximately 70 m above sea level. All-weather Highway 316 links the village of Goldboro to the town of Antigonish. A secondary gravel road (Gold Brook Road), accessed from Highway 316, crosses the Property and passes near the historic Boston-Richardson shaft and portal. Smaller logging roads and trails provide good access to most areas of the Property.

5.2 CLIMATE AND PHYSIOGRAPHY

5.2.1 *CLIMATE*

The climate is moderated by the Atlantic Ocean and is typical of the eastern Canadian coast. Records for the Stillwater-Sherbrooke weather station, located 25 km west of the Property, recorded an average winter temperature of -3.8°C and an average summer temperature of 16.4°C for the 50-year period ending in 2012. The lowest recorded temperature was -39°C in February of 1985 and the highest recorded temperature was 35°C in June of 1976. The average winter monthly snowfall is29 cm and the average depth is 11 cm (Environment Canada). Field programs can be carried out throughout the year, with schedule allowances made for winter weather and wet site conditions during spring thaw.

5.2.2 *PHYSIOGRAPHY AND VEGETATION*

The topography of the Property is characterized by gently rolling hills, with elevations ranging from 65 m to 80 m above sea level. A portion of the Property in the northeast is covered by Gold Brook Lake which drains southward. The Property is moderately to heavily forested and covered with boulder-filled gravels, sandy clay, till, and muskeg. Swamp covers approximately one-fifth of the Property. The vegetation of the area includes secondary growth of tag alders, maple, birch, spruce, balsam, and tamarack. Outcrops of bedrock are rare on the Property.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

5.3.1 *LOCAL RESOURCES*

The villages of Goldboro and Isaac's Harbour offer minimal essential services. However, basic services are readily available in the village of Sherbrooke, located approximately 45 highway km west of the Property, and full services, including a hospital, mechanical and retail facilities are present in the town of Antigonish, located approximately 70 km to the northwest (population approximately 4,500). The nearest commercial airport is Halifax Stanfield International Airport, located 3.5 hours from the Property.

The Strait of Canso Superport is located in Port Hawkesbury-Mulgrave area, approximately 60 km northeast of the Property and provides access to ocean-going shipping. The Cape Breton and Central Nova Scotia Railway mainline between Sydney, Nova Scotia and Truro, Nova Scotia passes within 50 km of the site and ocean access for smaller vessels is possible from wharves in the local area along Isaac's Harbour and in nearby Country Harbour.

Population is sparse in this area and the current local workforce does not reflect an extensive history of mining industry exposure. A substantial component of the area's economy is related to forestry sector activities and some transfer of skills to potential future mining projects could be expected.

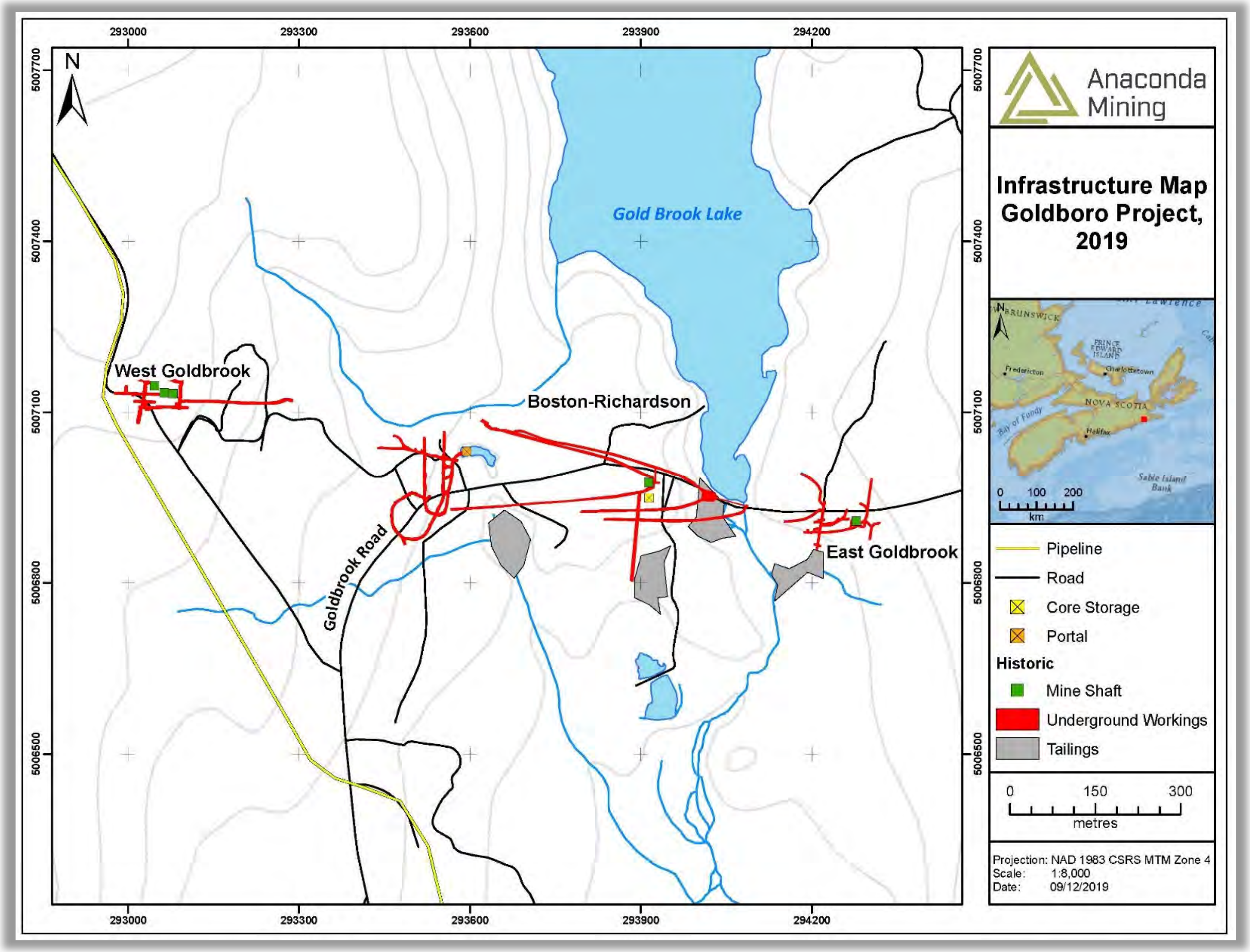
5.3.2 *SITE INFRASTRUCTURE*

Central to the Property is the historic Goldboro mine site, which includes the vertical, three-compartment Boston-Richardson shaft that was rehabilitated to a depth of 122 m in the late 1980s. At that time, a hoist capable of operating to depths of 600 m below surface was installed along with a new timber headframe. The headframe has since been dismantled. A service building that served as a warehouse, geology office, shaft house, and hoist room is also present on the Property and was in good repair at the time of the WSP 2018 and 2019 site visits. This building is currently being used as a core logging and covered core storage facility. External core racks are also present at the site (Figures 5.1, 5.2, and 5.3).

Figure 5.1 Core Logging Facility on Goldboro Property

Figure 5.2 Internal Core Storage Racks on Goldboro Property

Figure 5.3 Infrastructure Surface Map for Goldboro Property



A portal accesses a decline at the top of the BR Gold System located approximately 350 m west of the Boston-Richardson shaft. The decline measures approximately 5 x 4 m in cross-section and was developed during the 1988 exploration program. The ramp was driven with a grade of 15% and has a total inclined length of 416 m from surface to the 76 m level. It provides access to two 4 x 3 m cross-cuts, one at the 38 m level and another at the 76 m level. The decline is currently flooded to the portal collar.

Recently, site infrastructure has been improved with the completion of the underground bulk sample. Three-phase electricity has been restored to the site, the underground workings were temporarily dewatered, re-screened and re-bolted for stability, and underground ventilation was installed. The historic polishing ponds were used to treat water pumped from the re-established workings during the 2018/2019 bulk sample program. Following completion of the bulk sample the underground workings flooded. A small storage building remains near the portal.

There are four historic tailings sites on the Property (Figure 5.3). These are referred to as West, South, Southeast, and East sites, with material in the last comprising two separate areas along the Gold Brook River.

The Maritimes Northeast Limited natural gas pipeline crosses the western portion of the Property and a former producing gas fractionation plant is located 3.5 km to the south. This plant processed gas from ExxonMobil Canada Ltd.'s Sable Project offshore production facilities near Sable Island, approximately 160 km to the east. Production ceased in the summer of 2019. A second gas pipeline also makes landfall in this area and served Encana Ltd.'s formerly producing Panuke gas project, with production ceasing in May 2018 (Figure 5.3). Extensive work has also been completed in this area with respect to potential establishment of a liquefied natural gas (LNG) import facility, a petrochemical facility and an export LNG facility owned by Pieridae Energy.

The Gold Brook Lake and Goldbrook drainage system crosses the Property; other streams of lesser size are also present that could provide sufficient future plant and potable water requirements.

Sufficient undeveloped lands exist adjacent to the site to support potential tailings storage areas, potential waste disposal areas, heap leach pads, and potential processing plant sites.

6 HISTORY

6.1 INTRODUCTION

The summarized description of mining and exploration history prior to acquisition of the Property by Orex in 1988 is presented in report Section 6.2. It reflects compilation and modification of previous summaries by Faribault (1886), Malcolm (1976), and Gervais et al. (2009). The last source directly reflects earlier reporting by Rousseau (1990) and Roy (1998). Descriptions of exploration work carried out by Orex during the 1988 through 2017 period, up to the time of merger with Anaconda, are presented in report Sections 6.3 through 6.8, and have been taken directly from Cullen and Yule (2017), with minor modification as required to meet current report context regarding Anaconda. Section 6.9 summarizes exploration work conducted by Anaconda in 2017 and 2018 up to and including work disclosed in the NI 43-101 technical report, effective date July 19, 2018 (*McCracken et al., 2018*).

6.2 SUMMARIZED MINING AND EXPLORATION HISTORY

The first confirmed bedrock discovery of gold in Nova Scotia occurred in September 1858 when Lieutenant C. L'Estrange identified gold in a quartz outcrop along the Tangier River while moose hunting near Mooseland. Despite several years of prospecting in the area of Goldboro, it was not until 1862 that gold was discovered in quartz veins by Howard Richardson of the Geological Survey of Canada on the Isaac's Harbour anticline. The gold-bearing Boston-Richardson belt (slate and quartz) was subsequently discovered by Richardson in 1892. The Richardson Gold Mining Company began production from the belt in 1893 at an average reported grade of 0.38 oz. of gold per short ton (13.03 g/t gold) milled. Milling recoveries were reported to be in the 50% to 60% range.

By 1896 the Boston-Richardson mine was working at full capacity with a 40-stamp mill, and in 1897, three shafts were being worked from one shaft house. The main shaft was situated on the east side on the apex of the anticline and had an inclination of minus 21 degrees to the south. In 1898, 2,479 oz. (70 kg) of gold were recovered from 24,121 short tons (21,882 t) milled from this mine.

In 1899, 150 short tons (136 t) of concentrates worth $45 per ton were recovered from mill tailings and treated by a Wilfley concentrator. By 1900, substantial efforts were made to re-timber old workings, and large pillars were installed locally to stabilize the ground. In 1901, the mill was increased in size to sixty stamps, two more Wilfley concentrators were in operation, and an extensive cyanide plant was brought to the property from the Caribou gold district. In 1902, 29,000 short tons (26,308 t) of ore was milled and produced 3,459 oz. of gold (4.08 g/t). In 1903, mining was suspended after an extensive collapse, attributed to insufficient support of the hanging wall, destroyed the main shaft. The Boston-Richardson Mining Company took over ownership of the property in 1903.

In 1904, eight veins were cut and a seam of fault gouge at 386 ft. (118 m) depth, having a considerable quantity of associated quartz mixed with rock, was cut and proved to be part of the Boston-Richardson belt. In 1905, sampling and mill testing determined that all portions of the veins were not equally auriferous and that a large proportion of the gold was found in shoots.

From 1901 to 1905, three gold-bearing belts were intersected in the Dolliver Mountain mine, located 2 km west of the Boston-Richardson mine. In 1904, 205 oz. (6,376 g) of gold were recovered from 8,059 short tons milled, producing an average gold grade of 0.87 g/t. In 1905, several bodies of quartz and slate were intersected by a 152 m deep drillhole at the bottom of the main shaft along the anticlinal axis, but results were unsatisfactory and mining at Dolliver Mountain mine ceased.

In 1906, half the broken material mined was hoisted, and the remainder was held as a reserve in the mine. The bromo-cyanide plant brought from the Caribou gold district was in operation and recovered up to 80 % of the gold in concentrates in 1906 and 1907. Re-concentrated tailings from the cyanide plant grading approximately 40 % arsenic were shipped to Germany for further processing and gold recovery.

In 1907, the East Goldbrook property that adjoins the Boston-Richardson property to the east was acquired by F.S. Andrews and others. A shaft was sunk 175 feet (53 m), and three promising gold-bearing belts were explored in 1908. One of these was reported as being well-mineralized but no other work was carried out on the property at that time. Operations were suspended on August 15, 1908 due to financial difficulties but were later resumed.

In 1909, the New England Mining Company took over the Boston-Richardson mine and processed an additional 41,435 short tons (37,572 t) of material that produced 5,024 oz. of gold at an average grade of 4.14 g/t. Of this total, 82.6 % was recovered by crushing and amalgamation, and 17.4 % was recovered by bromo-cyanide processing. Of the 588 short tons (533 t) of arsenical concentrate produced, 446 short tons (405 t) were shipped to Swansea, Wales for final gold recovery. In 1910, 4,063 oz. (115 kg) of gold was recovered from 26,940 short tons (24,440 t) milled, producing an average gold grade of 5.17 g/t. Of this total, 715 oz. (20 kg) was recovered by the bromo-cyanide treatment of 965 short tons of concentrates and arsenical concentrates totaling 529 short tons (480 t) were shipped to Wales during the year for final gold recovery.

From 1909 to 1910, the West Goldbrook exploration shaft intersected five gold-bearing belts. Three of these were mill tested, but the result was unsatisfactory and the mine was abandoned.

The total gold recovery from 1893 to 1910 for the property has been estimated to be 54,871 oz. (1,707 kg) from 414,887 short tons of material milled (376,303 t), with this producing an average recovered gold grade of 4.11 g/t. However, mill recovery is reported to be approximately 67 % (*Roy, 1998*). Operations at the mine continued on a small scale in 1911 and 1912.

In 1926, the Metal Mining and Smelting Corporation of Canada Ltd. took over the property and treated tailings to recover auriferous arsenopyrite through 1927.

From 1929 to 1931, Locarno Copper Mines Ltd. sank a shaft on the West Goldbrook property, west of the earlier shaft. In 1931, a metallurgical test recovered 1.61 oz. (50 g) of gold from 1.1 short tons (1 t) of feed, producing an average gold grade of 50.1 g/t.

From 1931 to 1934, Renada Mines Ltd. dewatered and sampled the old workings at East Goldbrook, and obtained gold assay results ranging between 1.61 and 4.26 g/t.

In 1956, Canso Mining Corporation dewatered the shaft at West Goldbrook and carried out cross-cutting. This work ceased due to financial difficulties.

In 1981, Patino Mines (Quebec) Ltd. completed a detailed geophysical program which covered most of the Upper Seal Harbour district. The results of this program successfully defined an anticlinal axis extending from Dolliver Mountain mine in the west to the East Goldbrook shaft in the east and confirmed geological continuity of structures comprising the main target of exploration work planned by the company for the area.

In 1984, Onitap Resources Inc. (Onitap) acquired 37 claims covering the Boston-Richardson, East Goldbrook, West Goldbrook, and Dolliver Mountain prospects. Exploration commenced with drillhole BR-84-01 that tested the down-plunge extension of the Boston-Richardson belt. The hole intersected the belt at a depth of 360 m from its collar and also cut several new mineralized belts. The hole was terminated at a depth of 529 m after going beyond the extent of earlier underground workings.

In 1985, Onitap drilled five short holes, totalling 390 m, near the historic West Goldbrook mine (BR-85-01 to BR-85-04, and BR-85-01A). These holes intersected several belts of slate and quartz veining but were sampled on a selective basis only.

In 1987, two drilling programs focused exploration on the eastern portion of the property, in the Boston-Richardson and East Goldbrook areas. The first program consisted of five drillholes totalling 1,925 m (BR-87-01 to BR-87-05) and was conducted by Petromet Resources Ltd. and Greenstrike Corporation. These holes tested the lateral and depth extensions of the Boston-Richardson belt. The second drill program was conducted by Onitap and consisted of 33 holes (BR-87-06 to BR-87-38) totalling 11,862 m of drilling. Helicopter-borne geophysical surveys (VLF-EM and magnetics) plus ground Induced Polarization (IP) surveys were also carried out at this time. Four new mineralized belts were intersected stratigraphically beneath the Boston-Richardson belt by these 33 drillholes, and for project purposes were designated as the New, Third, Fourth, and Fifth Belts (top of the BR Gold System). Good gold grades were also intersected above the Boston-Richardson belt in the East Goldbrook area, and presence of visible gold was recorded for most holes. Mineral resources were estimated at the time to total 999,767 tonnes grading 6.6 g/t gold. The authors note that these resources are historic in nature, were not prepared in accordance with NI 43-101 or the CIM Standards, and should not be relied upon. Sufficient work has not been carried out by a Qualified Person, as defined in NI 43-101, to classify these as current resources, and Anaconda is not treating or considering these to be current resources.

Orex acquired the Property from Onitap in 1988 and, with the exception of a period of inactivity from 1996 to 2004, actively pursued exploration and economic assessment of the Property until amalgamation with Anaconda in 2017. Details of Orex programs carried out on the Property for the period beginning in 1988 and extending until the 2017 amalgamation with Anaconda are presented in report subsections 6.3 to 6.8.

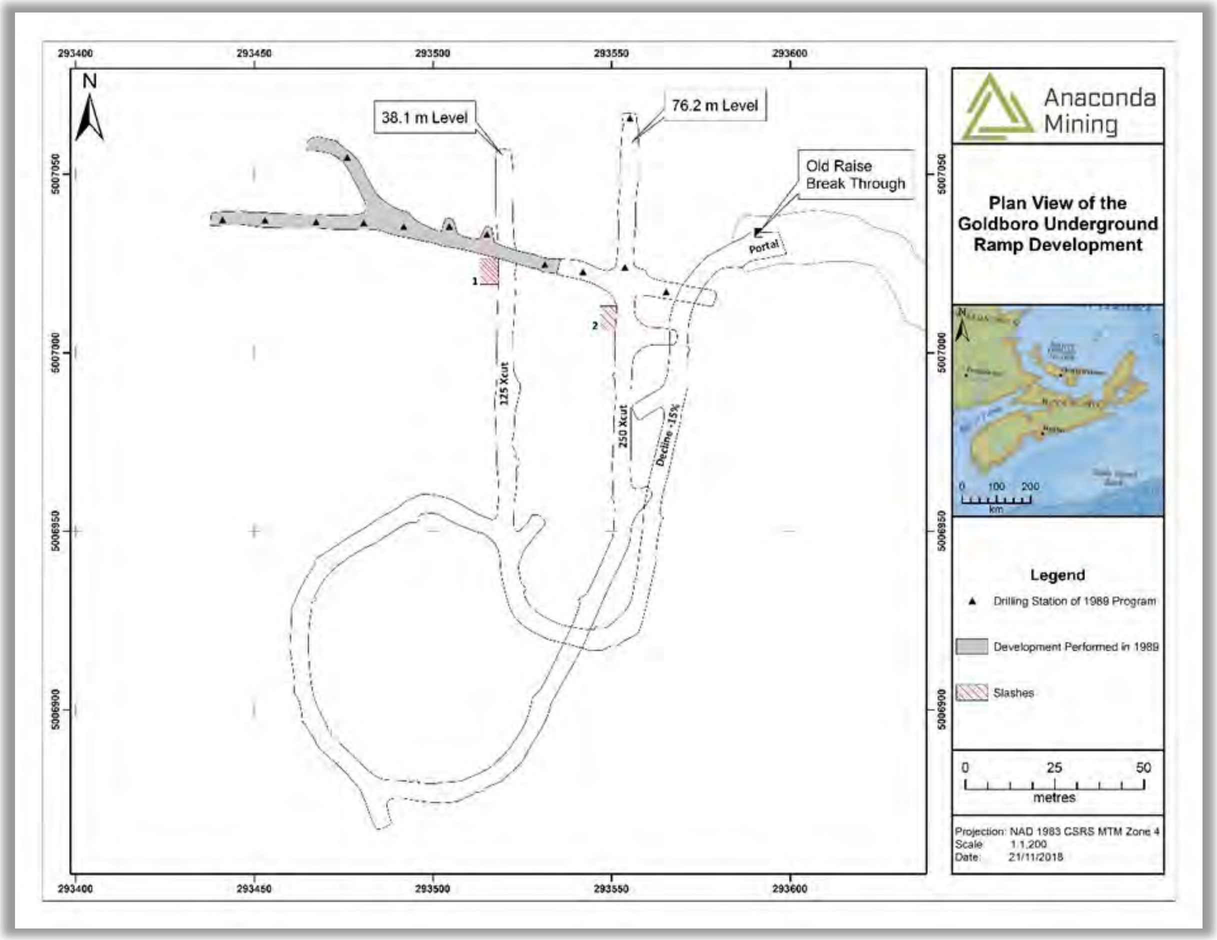
6.3 1988 TO 1991 PROGRAMS

After acquiring the Property in 1988, Orex conducted a surface and underground exploration program that focused on the central portion of the Property. This included the new belts previously discovered beneath the historic Boston-Richardson belt (the BR Gold System) and the area between the historic West Goldbrook and Boston-Richardson mines. The surface drilling component of the 1988 program consisted of 44 drillholes and is detailed in report Section 10. During the underground component of the 1988-1989 exploration programs, an access decline, measuring approximately 5 m by 4 m in cross-section, and two working levels consisting of a cross-cut and apex drift, measuring 4 m by 3 m in cross-section, were developed into the BR Gold System (Figure 6.1). The decline was driven for a total length of 416 m with a grade of 15 % from the surface to a depth of 76 m below surface. Orex spent a total of $8.2 million which also included rehabilitation of the historic Boston-Richardson shaft and other related technical services. The work was carried out under an option agreement between Orex, Onitap, and Narex Ore Search Consultants Inc.

As the cross-cut and apex drift were advanced, a muck sampling procedure was developed along with mapping and chip sampling programs. Coarse gold mineralization was found in the quartz veins associated with graphitic black shale, and free gold was also detected in the shale unit itself.

The Boston-Richardson vertical shaft was rehabilitated to provide direct access to the mineralized material in the eastern part of the BR Gold System. Dewatering started in May 1988, and the shaft collar was reinforced to support a new 28 m headframe. The shaft was rehabilitated to the 122 m level and contains three standard 1.8 x 1.8 m compartments.

Figure 6.1 Underground Workings in the Boston-Richardson Gold System Established by Orex



Source: Anaconda, 2018

In February of 1989, a milling test was completed on core material from six drillholes (BR88-48, BR88-60, BR88-61, BR88-62, BR88-85, and BR87-35A) that intersected the first five belts of the BR Gold System near the anticlinal apex. A comparison between original core sample assays and analyzed feed for the milling test material was carried out and core sample results were not reproduced. This variance was largely attributed to nugget effect. On the basis of these results, it was concluded that bulk sampling was the best way to determine gold grade without substantive discrepancy attributable to nugget effect being encountered (*Parent 1989, from Dionne and Vachon, 1989*).

A total of 134 holes were completed in two drilling campaigns during 1989. The first campaign included 108 short holes drilled from the 76 m level of the Boston-Richardson underground workings, and the second campaign consisted of 26 holes drilled from surface in the West Goldbrook area. Details of these programs are discussed in report Section 10.

Two underground bulk samples, each weighing approximately seven tonnes, were mined after establishment of the main Boston-Richardson underground accesses. The first was designated as 'Sample A' and consisted of a slash taken from the west wall of the cross-cut on the south limb of the one of the uppermost mineralized belts of the BR Gold System (historically referred to as the N1 belt) at the 38 m level. The slash area measured 4.5 m long by 1.5 m high and was 50 cm to 1 m deep. The second slash was designated as 'Sample B' and taken from the 76 m level, directly south of the apex drift on the west wall of the cross-cut, also on the south limb of the N1 belt. This slash measured 7 m long by 2 m high and was 30 to 50 cm deep. Materials from both samples were separately processed by Lakefield Research, a division of Falconbridge Limited, in a laboratory pilot plant using combined gravity and flotation methods. The purpose of such processing was to obtain metallurgical data for mill design with respect to grinding, gravity separation, and flotation processing. Cyanide leaching tests were also performed on the flotation concentrate products from each bulk sample. The rod mill feed gold grades for Samples A and B were 2.54 g/t and 4.07 g/t gold respectively. Comparatively, a gold grade range between 3.4 g/t (diluted) and 2.8 g/t (diluted) was previously established for in-situ material in this sample area that was classified as 'reserves' (*Labelle, 1990*). The authors caution that use of the term 'reserve' in this instance is historic in nature, is not compliant with meaning of the term under NI 43-101 and the CIM Standards, and should not be relied upon in this context.

In June 1990, Orex undertook legal proceedings against Onitap as a result of their conduct in relation to the project. A settlement of the dispute was negotiated in December 1990. Shortly after, the property was optioned to Minnova Exploration Inc. (Minnova) with the right to acquire 60% interest in the property by investing a total sum of $5 million in exploration work during the next three years (*Labelle, 1991*).

6.4 1991 TO 1993 PROGRAMS

Minnova completed a preliminary drill program in 1991 that consisted of five holes, four of which were completed as twins of earlier holes for which analytical data was available. The fifth hole tested a low-grade area defined by earlier drilling, and specifics of the program appear in report Section 10. The purpose of the program was to provide sample material for assessment of previously determined gold grades through application of crushing, grinding, and continuous-grind-leach (CGL) processing methods. Results showed that gold was not being fully recovered by such methods due to various factors, the most prominent being incomplete leaching by the cyanide solution. Analytical results comparable to those obtained for drill core samples processed by screen metallics methods were obtained but, as expected, these were substantially lower than results of previous core bulk sample processing. Minnova considered results to be problematic with regard to implications for future mine operations and grade control procedures and subsequently withdrew from the option agreement.

An agreement was reached with Novagold Resources Inc. (Novagold) in 1992, which at the time was operating a vat-leaching milling plant in the Bathurst Mining District of New Brunswick. An evaluation of vat leaching (VL) as a method for testing and processing the ore from the Property was initiated. Sample material for the VL evaluation came from the 4,000 t surface stockpile of underground development material from the BR Gold System, and this was sent to Halifax for processing at the Technical University of Nova Scotia. After four hours of leaching, recovery of 84% of the contained gold was achieved at grades in the 4 g/t to 6 g/t range. These results were considered positive but Novagold subsequently dropped the option due to financial difficulties (*Bourgoin et al., 2004*).

In the fall of 1993, Orex focused on better understanding the 'nugget effect' in gold distribution and drilled a total of six more holes to provide material for further metallurgical testing of the previously drilled and sampled test area. Processing of material took place at the *Centre de Recherches Minérales* in Québec City. Selected sample splits were ground and continuously leached in a ball mill for 24 hours in a high cyanide solution, and gold levels in the leach solution and in the solid residue were subsequently determined and a head grade for the material was calculated (*Bourgoin et al., 2004*).

Results of this test were similar to those reported from the earlier Minnova program, but core recovery issues were eliminated as a significant factor in assessment of gold grade comparisons. Investigation of low gold grades from this program compared to expected levels showed that further study of the cyanide dissolution systematics and retention of gold in the cyanide solution were required. Minnova subsequently withdrew from further assessment of the property (*Bourgoin et al., 2004*).

6.5 1995 PROGRAM

Placer Dome Inc. (Placer) contacted Orex in 1995 and entered into an agreement to explore the Property after completion of an initial lithology-specific sampling program focused on a 4,000-tonne surface stockpile generated during the underground program. This source had also been used in 1992 to provide material for the VL tests carried out by Novagold. The lithologies sampled were greywacke, slate (graphitic shale), quartz veins (bull quartz), and mineralized veins. Subsamples of each lithology were processed using metallurgical methods and carbon-in-leach gold extraction after crushing and grinding. Calculated head grades and assayed head grades from these tests were closer in agreement than in previous tests and are summarized in Table 6.1.

Table 6.1 Comparison of Placer Processing Results

Lithology	Calculated Gold Head Grade (g/t)	Assayed Gold Head grade (g/t)
Greywacke	1.00	1.00
Slate (graphitic shale)	2.64	2.82
Quartz veins (bull quartz):	16.18	6.16
Mineralized veins	33.62	36.37

Source: Gagnon et al, 1996

Placer subsequently completed seven core holes (BR95-119 to BR95-125) to assess open pit potential. Initially planned processing of resulting samples included conventional fire assay-metallics screen methods (CMS), metallurgical processing through gravity concentration followed by cyanide leaching of tailings (GCNL), and continuous grind cyanide leaching (CNL). Results of these programs were interpreted as indicating that GCNL processing provided the best assessment of total gold content in the test samples. CMS and GCNL results correlate well within the gold grade range of 0 to 4 g/t but at higher grade levels under-reporting characterizes CMS data relative to that from GCNL. CNL results show consistently poorer correlation and under-reporting relative to GCNL (*Gagnon et al., 1996*).

Based on results of its property assessment, Placer opted to cease involvement in the property after completion of the 1995 drilling program. Results of the property assessment indicated the following.

- The property did not meet corporate requirements with respect to large open pit mining opportunities.
- Best mineralization potential defined to that time on the property was within the New Belt zones.
- Fracture arrays in bedrock may require relatively low final open pit slope angles.

— Drill sludge samples provide a sensitive indication of bedrock mineralization.

— Characteristic wall rock alteration occurs in association with gold mineralization and is centred on the Upper Seal Harbour anticline.

— GCNL gold determinations are on average 1.5 times higher than those generated by CMS and CNL methods for samples in the 1 to 4 g/t gold range and are generally comparable at gold grade levels below 1 g/t.

6.6 2004 TO 2009 PROGRAMS

In February 2004, MRB & Associates Ltd. (MRB) of Val-d'Or, Québec was contracted by Orex to provide an independent NI 43-101 technical report on the Goldboro Property. The firm assembled a team of independent consultants consisting of InnovExplo Inc. (InnovExplo), A.S. Horvath Consulting (Horvath Consulting), Tech2Mine Inc. (Tech2Mine), and MRB to carry out this work, with main objectives being as follows.

— Completion of a review and comparison of Meguma-style deposits to those of the Bendigo and Ballarat districts in the Lachlan fold belt of Victoria, Australia.

— Review and assessment of historical sampling and analytical grade determination procedures at Goldboro.

— Assessment of the data quality and suitability of historical results for resource estimation application.

— Identification of gold mineralization characteristics.

— Provision of recommendations for future sampling, grade determination, and QA/QC protocols.

— Preparation of a mineral resource estimate by MRB in accordance with NI 43-101 and the CIM Standards for the deposit based on drilling data available to that time. Results of the work were reported by Bourgoin et al. (2004).

The Goldboro Mineral Resource Estimate developed during the MRB study is presented in Table 6.2, and was considered to be in accordance with NI 43-101 and the CIM standards at its August 31, 2004 Effective Date. These resources are now historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

Table 6.2 Historic MRB Resource Estimate - August 31, 2004

Zone	Cut-off (Au g/t)	Resource Category	Metric Tonnes	Au g/t	Ounces Au
Boston-Richardson Area	0.00	Measured Resource	755,000	1.21	29,300
	0.00	Indicated Resource	12,500,000	0.75	301,000
	0.00	Sub-Total	13,255,000	0.78	330,300
	0.00	Inferred Resource	7,000,000	0.78	175,500
West Goldbrook	0.00	Inferred Resource	8,600,000	0.50	138,200

Foremost among conclusions and recommendations of the 2004 technical report by MRB were recognition that historical sampling, processing, and analytical gold determination protocols have led to consistent under-estimation of gold grades above the 1g/t level due to extreme nugget effect; and that the most reliable method of obtaining accurate and precise grade determinations for Goldboro mineralized samples is by metallurgical testing/processing of adequately sized representative samples.

Orex carried out an exploration drilling program in 2005 that concentrated on a 1,500 m long by 225 m wide area centred on section 8675E of the BR Gold System. Details of this drilling program are described in report Section 10. A total of 23 holes (2,355 m) were completed in this program, and samples were prepared and analyzed using revised protocols. Results were used in combination with validated historic drilling results to support a new NI 43-101 resource estimate for the 'Ramp Area' in 2006 by P & E Mining Consultants Inc. (P&E Mining) (*Puritch et al., 2006*). This estimate was limited to the central deposit area where ramp development, bulk sampling, and metallurgical processing of composited drill core samples had been completed and estimation results are presented in Table 6.3. These resources are now historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

Table 6.3 Historic P&E Mining Resource Estimate - August 26, 2006

Zone	Cut-off (Au g/t)	Resource Category	Tonnes	Grade (Au g/t)	Contained Ounces (Au)
Ramp Area	0.5	Measured Resource	481,000	3.40	52,600
	0.5	Indicated Resource	2,624,000	2.17	183,400
	0.5	Measured Plus Indicated	3,105,000	2.36	236,000

A Conceptual Target as defined under NI 43-101 was also identified by P&E Mining at the time of resource estimation and this consisted of the potential gold inventory increase that would result if Ramp Area Indicated resources were increased in gold grade by a 1.23 g/t factor. This resulted in definition of potential for presence within the existing Indicated category tonnage for a gold inventory increase of 103,500 oz. Confirmation of such increase would require metallurgical processing of all available core materials from each contributing drillhole in the Indicated resource volume (*Puritch et al., 2006*). The P&E Mining resource estimate is now historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

Another drilling program was carried out in 2008 by Orex and this consisted of 45 holes (12,201.5 m). Focus was placed on a 975 m long area between the WG and EG Systems to infill on previous holes and to test for mineralization extensions east and west of the BR System. Details of the program appear in Section 10. InnovExplo subsequently provided a revised mineral resource estimate for the property having an effective date of July 29, 2009 (Table 6.4).

Table 6.4 Historic InnovExplo Resource Estimate - July 29, 2009

Cut-off (Au g/t)	Resource Category	Tonnes	Grade (Au g/t)	Contained Ounces (Au)
1.5	Measured Resource	270,000	4.99	43,000
1.5	Indicated Resource	2,441,000	4.51	353,900
1.5	Measured Plus Indicated	2,711,000	4.56	396,900
1.5	Inferred	3,438,000	3.67	405,900

The InnovExplo resource estimate is now historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

6.7 2009 TO 2017 PROGRAMS

In November of 2009, Orex signed an option to joint venture agreement with Osisko Mining Corporation (Osisko), under which Osisko became manager of the Goldboro Project (Orex Press Release No. 2009-15). If Osisko wished to acquire a 50% undivided interest in the Goldboro Property on or before September 29, 2013 (the initial option), the company needed to incur exploration and development expenditures in the amount of at least $1,500,000 on or before September 29, 2010, in the aggregate amount of at least $3,500,000 on or before September 29, 2011, and in the aggregate amount of at least $8,000,000 over the following two years, that is, on or before September 29, 2013. Osisko had the right to accelerate and apply exploration and development expenditures to future years' expenditures and, accordingly, the initial option could be exercised sooner. Osisko would have had to solely fund a pre-feasibility study to earn an aggregate 60% interest (this reflects an additional 10% interest) in the property on or before September 29, 2015. On September 29, 2011, it was announced in a news release by Orex that Osisko had terminated its option to acquire an interest in the Goldboro Property. Osisko had informed Orex earlier in the year that minimum expenditure threshold of $3,500,000 to be spent on exploration and development expenditures would not be met on or before the agreed date of September 29, 2011 (*Orex Press Release No. 2011-5*).

Osisko carried out two core drilling programs before terminating the option agreement with Orex. These consisted of 59 holes (12,992.7 m) drilled in 2010, and 10 holes drilled in 2011. Details of both programs are presented in report Section 10.0. Results from these holes were consistent with expectations based on past experience in the main deposit areas and were incorporated in the current resource estimate.

The 2010 holes were directed toward defining resources in the Dolliver Mountain and West Goldbrook areas as well as in the BR Gold System where historic drillholes were twinned for confirmation and sampling purposes. The 2011 program was completed east of the EG Gold System and west of West Goldbrook, near Dolliver Mountain. Local high gold grades over narrow widths were encountered in the program in areas not included in past resource estimates.

Orex considered results from the 2010 and 2011 drilling campaigns to be encouraging and, based on these, corporate interest for the project was directed toward assessment of narrow, higher-grade veins and belts showing underground mining potential or potential for development and mining by a combination of underground and shallow open pit methods. This contrasts with the earlier corporate focus on large open pit potential.

On March 30, 2012, Orex retained Mercator Geological Services Limited (Mercator) to manage work programs and complete an updated mineral resource estimate (*Cullen and Yule, 2013*). An updated geological model for the Deposit was created, with specific focus on definition of discrete, higher-grade mineralized belts with underground mining potential. The updated mineral resource estimate with Effective Date of February 11, 2013 included results from 69 drillholes completed in 2010 and 2011. (Table 6.5). This mineral resource estimate has been superseded and is now considered historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

In 2014, MineTech International Limited (MineTech) completed a PEA of the Project based on the 2013 mineral resource estimate prepared by Mercator. This analysis was focused on assessment of underground mining project viability and was presented in the NI 43-101 technical report prepared by MineTech for Orex *(Roy et al., 2014*).

In February of 2017, Mercator provided a revision to the 2013 mineral resource estimate which superseded the 2014 PEA technical report by MineTech. The revised mineral resource estimate, with effective date February 28, 2017, by Mercator was prepared to correct a non-material assay compositing error that affected certain incompletely sampled drill core intervals from historic drilling programs used in the 2013 estimate (Table 6.6; *Cullen and Yule, 2017*). The 2013 and 2017 mineral resource estimates as well as the 2014 PEA have been superseded by subsequent work, are historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

In May of 2017, Orex and Anaconda announced a plan of arrangement whereby Orex became a wholly-owned subsidiary of Anaconda.

During 2017 Anaconda contracted Mercator This estimate, used in the 2017 PEA, is historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

Table 6.5 Historic Mercator Resource Estimate - Effective February 11, 2013

Gold Cut-off g/t	Resource Category	Boston Richardson Zone		West Goldbrook Zone		East Goldbrook Zone		Total Goldboro Deposit		
		Tonnes*	Au g/t	Tonnes*	Au g/t	Tonnes*	Au g/t	Tonnes*	Au g/t	Ounces*
2.00	Measured	171,000	5.39					171,000	5.39	29,600
	Indicated	1,472,000	5.44	473,000	5.34	473,000	6.37	2,418,000	5.60	435,300
	Subtotal	*1,643,000*	*5.43*	*473,000*	*5.34*	*473,000*	*6.37*	*2,589,000*	*5.59*	*465,000*
	Inferred	953,000	5.04	345,000	4.40	1,245,000	5.45	2,543,000	5.15	421,100

Notes:

- *Tonnages have been rounded to the nearest 1,000 tonnes and ounces have been rounded to the nearest 100; average grades and contained ounces may not sum due to rounding.*
- *The resource statement gold cut-off grade is 2.0 g/t and is highlighted in the tabulation above.*
- *The 2.0 g/t gold resource statement cut-off grade reflects a reasonable expectation of economic development by underground mining methods based on a three-year trailing average gold price of US $1,492.*
- *Contributing 1.0 m assay composite populations were capped at gold grades of 80 g/t or 120 g/t and separately interpolated.*
- *A bulk density factor of 2.7 kg/m^3 was applied to all blocks.*
- *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, taxation, sociopolitical, marketing, or other relevant issues.*

Table 6.6 Historic Mercator Mineral Resource Estimate - Effective February 28, 2017

Gold Cut-off g/t	Resource Category	Boston Richardson Zone		West Goldbrook Zone		East Goldbrook Zone		Total Goldboro Deposit		
		Tonnes*	Au g/t	Tonnes*	Au g/t	Tonnes*	Au g/t	Tonnes*	Au g/t	Ounces*
2.00	Measured	171,000	5.39					171,000	5.39	29,600
	Indicated	1,507,000	5.27	464,000	5.39	414,000	6.91	2,385,000	5.58	427,800
	Subtotal	*1,678,000*	*5.28*	*464,000*	*5.39*	*414,000*	*6.91*	*2,556,000*	*5.57*	*457,400*
	Inferred	10,083,000	4.56	459,000	4.42	1,127,000	4.11	2,669,000	4.35	372,900

**** Notes:***

- *This revised resource estimate was prepared in accordance with National Instrument 43-101 and the CIM Standards.*
- *Tonnages have been rounded to the nearest 1,000 and ounces have been rounded to the nearest 100; average grades and contained ounces may not sum due to rounding,*
- *The 2.0 g/t gold resource statement threshold grade reflects a reasonable expectation of economic extraction by underground mining methods.*
- *Contributing 1.0 metre assay composite populations were capped at gold grades of 80 g/t or 120 g/t and separately interpolated.*
- *A bulk density factor of 2.7 kg/m^3 was applied to all blocks.*
- *Mineral resources that are not mineral reserves do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*

6.8 SUMMARY OF SELECTED OREX PROGRAMS BETWEEN 1988 AND 2017

Table 6.7 presents summarized exploration programs carried out by Orex on the Property between 1988 and 2017.

Table 6.7 Summarized History of Work Highlights on the Goldboro Property from 1988 to 2017

Year	Company	Location/Area	Work Description
1988	Orex and Onitap	Between West Goldbrook and Boston-Richardson	Surface drilling campaign completed 44 core holes (BR88-39 to -82) totalling 10,709 m; 14 new belts discovered; assays vary from 5.14 g/t over 0.2 m to 48.7 g/t over 7.6 m.
1988	Orex and Onitap	Boston-Richardson	Access decline and two levels (38 m and 76 m) established; underground drilling campaign completed; 4 core holes (BR88-80 to -82) totalling 234 m; rehabilitation of Boston-Richardson shaft to 122 m, hoist and headframe installed.
1989-1990	Orex	West Goldbrook	Surface drilling campaign of 26 core holes completed (BR89-83 to -108) totalling 2,822 m.
1989-1990	Orex	Ramp	Underground drilling campaign completed with 108 core holes totalling 4,741 m; 2 bulk samples (~7 t each) were mined out from South limb of the N1 belt.
1990-1993	Orex, Minnova, Novagold Resources Inc.	Ramp	Various methods used by several partners in an attempt to accurately estimate gold grades; core holes BR91-109 to BR93-177 were completed; Minnova dropped the property option.
1995-1996	Orex, Placer Dome Inc.	Ramp	Sampled surface stock pile; option to invest $30 million to acquire 65% in Goldboro property; drilling near ramp portal (BR95-119 to -125) to investigate open pit potential; Placer dropped the property option.
2004	Orex	Goldboro property	Review of geology and evaluation of characteristics of Meguma-style deposit as compared to Bendigo and Ballarat deposits (Australia); review of historical sampling and analytical grade determination procedures.
2005-2006	Orex	Ramp	Surface drilling campaign of 23 core holes completed (BR05-001 to BR05-023) totalling 2,355 m; twinned 4 historic holes for comparative analysis; metallurgical testing; received mineral resource estimate by P&E Mining Consultants Inc. in 2006.
2008-2009	Orex	1.5-km strike length from Ramp to East Goldbrook (section 8100E-9600E), 500 m vertical depth	Phase 2A and 2B programs consisting of 45 infill core holes (BR08-01 to -44, plus 20A) totalling 12,202 m; completion of mineral resource estimate by InnovExplo Inc. in 2009.
2010	Orex, Osisko	Ramp, West Goldbrook and Dolliver Mountain areas	Phase 2D, 2E, 2F drilling campaigns completed with 59 core holes totalling 12,989 m; un-sampled intervals from 97 historic holes were assayed.
2010	Orex, Osisko	East Goldbrook, Ramp and West Goldbrook areas	Identified future targets after completion of 64 reverse circulation holes (OSK10RC-130 to 194) totalling 505 m; positive results from East Goldbrook included 10.85 g/t Au and 25.65 g/t Au in bedrock chip samples.
2011	Orex, Osisko	East Goldbrook and area between West Goldbrook and Dolliver Mountain	Six drillholes (OSK11-01 to -06) were completed at East Goldbrook totaling 2,375 m, and 4 more (OSK11-07 to -10) were completed between West Goldbrook and Dolliver Mountain.
2012-2013	Orex	East and West Goldbrook areas, Boston-Richardson area and New Belts area	Revised geological model developed and used as basis for new resource estimate by Mercator Geological Services Limited.

(table continues on next page)

Year	Company	Location/Area	Work Description
2014	Orex	East and West Goldbrook areas, Boston-Richardson area and New Belt area	The 2013 resource estimate was used by MineTech International Limited as the basis for a Preliminary Economic Assessment of underground mining potential at the site.
2014-2017	Orex	All	The 2017 resource estimate was revised to address a non-material assay compositing error and management pursued opportunities for further property assessment and potential development. These efforts resulted in a Plan of Arrangement between Orex and Anaconda interests with respect to the Goldboro property.

6.9 ANACONDA 2017 – 2018 PROGRAMS

In 2017, Anaconda carried out a three-phase drill program from June to December. The first phase was designed to test grade repeatability of selected historic holes within the deposit. Anaconda completed five HQ-size drillholes (BR-17-01 to BR-17-05), totaling 649 m to test the BR and EG Gold Systems. Samples were collected for metallurgical test work on the Goldboro mineralization with each of the completed holes twinning a historic drillhole. During October 2017, Anaconda completed two holes (BR-17-06 and BR-17-07) totaling 1,173 m drilled to test continuity of the BR Gold System and lower belts at depth (Phase II). From November to December 2017, a total of 2,380.3 m was drilled in six holes (BR-17-08 to BR-17-13) targeting the BR and EG Gold Systems in historically under-drilled areas of the Property for the purpose of upgrading resource categorization of mineralized zones or expansion of mineralized zones down-dip or along strike (Phase 3). Further details of these drill campaigns are described in Sections 10.9 and 10.10.

Anaconda contracted Thibault to conduct a bench scale metallurgical test on a dry 324 kg sample of select drill core from the phase one drill program. Preliminary development of a process flowsheet defined overall gold recoveries between 92.7% to 95.3% by combination gravity separation, flotation and cyanide leaching of both gravity and flotation concentrates.

During 2017 an updated mineral resource estimate was completed by Mercator Geological Services Ltd. (Michael Cullen, P.Geo., Independent Qualified Person) with an effective date of December 31, 2017 *(Robinson et al., 2018)*. The mineral resource estimate comprises: an open pit resource including 1,059,00 tonnes of Measured and Indicated resource at a grade of 3.01 g/t gold (102,500 ounces), and 45,000 tonnes of Inferred mineral resource at a grade of 2.54 g/t gold (3,700 ounces) at a cut-off grade of 0.5 g/t gold; and an underground resource including 2,586,000 tonnes of Measured and Indicated mineral resources at a grade of 5.09 g/t gold (422,900 ounces) and 2,497,000 tonnes of Inferred resource at a grade of 4.28 g/t gold (343,600 ounces) at a cut-off grade of 2.0 g/t gold (Table 6.8). The 2017 updated mineral resource estimate has been superseded by subsequent work, are historic in nature and should not be relied upon. A Qualified Person, as defined in NI 43-101, has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves, and Anaconda is not treating or considering these as being current mineral resources or mineral reserves.

Anaconda retained WSP and Thibault to complete a PEA study. The PEA provided a base case economic assessment of developing the Goldboro mineral resource (effective date December 31, 2017), by open pit and underground mining, on site concentration through gravity and flotation circuits, and leaching of the concentrate and gold recovery at Anaconda's Pine Cove Mill in Newfoundland. The PEA, effective date March 2, 2018, was presented in the Robinson et al. (2018) NI 43-101 technical report.

In 2018, Anaconda completed 31 drillholes (BR-18-14 to BR-18-42) totaling 8,160 m designed as infill and exploration holes in the deposit area with focus on a 650 m strike-length overlying the BR and EG Gold Systems. Infill drill results provided confidence in the geological model and exploration drilling in EG delineated three new near-surface mineralized belts. Further details of this drill program are described in Section 10.10.

In 2018 Anaconda contracted WSP to provide an update to the mineral resource estimate to include the 2017-2018 drilling. The update was based on validated results of 316 surface drillholes and 119 underground drillholes, for a total of 79,104 m of diamond drilling that was completed between 1984 and June 2018. The updated mineral resource estimate, effective date July 19, 2018, prepared by McCracken et al. (2018) in a NI 43-101 technical report supersedes the 2017 mineral resource estimate presented in the 2018 PEA. Table 6.9 summarizes the 2018 mineral resource estimate.

Table 6.8 Historic Mercator Mineral Resource Estimate - Effective December 31, 2017

Classification		Cut-off (Au g/t)	Mercator 2017 Tonnes ('000)	Grade (Au g/t)	Ounces ('000)
Measured	Open Pit	0.5	397	2.88	36.8
	Underground	2.0	22	4.70	3.3
Indicated	Open Pit	0.5	662	3.09	65.8
	Underground	2.0	2,564	5.09	419.6
Measured and Indicated	Open Pit	**0.5**	**1,059**	**3.01**	**102.5**
	Underground	**2.0**	**2,586**	**5.09**	**422.9**
Inferred	Open Pit	0.5	45	2.54	3.7
	Underground	2.0	2,497	4.28	343.60

Notes:

- *Mineral resources were prepared in accordance with NI 43-101 and the CIM Standards (2014).*
- *Open pit mineral resources are reported at a cut-off grade of 0.5 g/t gold within the WSP base case pit shell and are based on a gold price of CAN$1,550/oz and a gold processing recovery factor of 95%. These include PEA base case open pit resources that have an estimated life of mine strip ratio of 7.3:1 (waste tonnes:PEA tonne).*
- *Appropriate mining costs, processing costs, metal recoveries and inter ramp pit slope angles were used by WSP to generate the base case pit design.*
- *Rounding may result in apparent summation differences between tonnes, grade and contained metal content. Tonnages have been rounded to the nearest 1,000 tonnes; ounces have been rounded to the nearest 100 ounces.*
- *Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.*
- *Contributing assay composites were capped at 80 g/t gold.*
- *A density factor of 2.70 g/cm^3 was applied to all blocks.*
- *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

Table 6.9 Historic WSP Mineral Resource Estimate - Effective July 19, 2018

Resource Type	Au Cut-off (g/t)	Category	Tonnes (rounded)	Au	Troy Ounces (rounded)
Open Pit	0.5	Measured	608,700	2.80	54,900
		Indicated	247,600	3.72	29,600
		Measured & Indicated	856,300	3.07	84,500
		Inferred	58,500	4.10	7,700
Underground	2.00	Measured	1,003,100	5.10	164,400
		Indicated	1,918,600	5.74	353,800
		Measured & Indicated	2,921,700	5.52	518,200
		Inferred	2,067,900	6.70	445,500
Combined	0.50/2.00	Measured	1,611,800	4.23	219,300
		Indicated	2,166,200	5.50	383,400
		Measured & Indicated	3,778,000	4.96	602,700
		Inferred	2,126,400	6.63	453,200

Notes:

- *Mineral resources were prepared in accordance with NI 43-101 and the CIM Standards (2014).*
- *Open pit mineral resources are reported at a cut-off grade of 0.5 g/t gold within the WSP base case pit shell and are based on a gold price of CAN$1,550/oz and a gold processing recovery factor of 95%.*
- *Appropriate mining costs, processing costs, metal recoveries and inter-ramp pit slope angles were used by WSP to generate the base case pit design.*
- *Rounding may result in apparent summation differences between tonnes, grade, and contained metal content. Tonnages have been rounded to the nearest 1,000 tonnes; ounces have been rounded to the nearest 100 ounces.*
- *Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.*
- *Contributing assay composites were capped at 80 g/t gold.*
- *A density factor of 2.70 g/cm^3 was applied to all blocks.*
- *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*

7 GEOLOGICAL SETTING AND MINERALIZATION

The following descriptions of geological setting and mineralization are modified after Carrier and Beausoleil (2004), as presented by Bourgoin et al. (2004), and others including observations and interpretation by Anaconda geologists.

7.1 REGIONAL GEOLOGY

There are five tectonostratigraphic zones within the Appalachian Belt in eastern Canada (Figure 7.1), these include the Humber, Dunnage, Gander, Avalon, and Meguma zones.

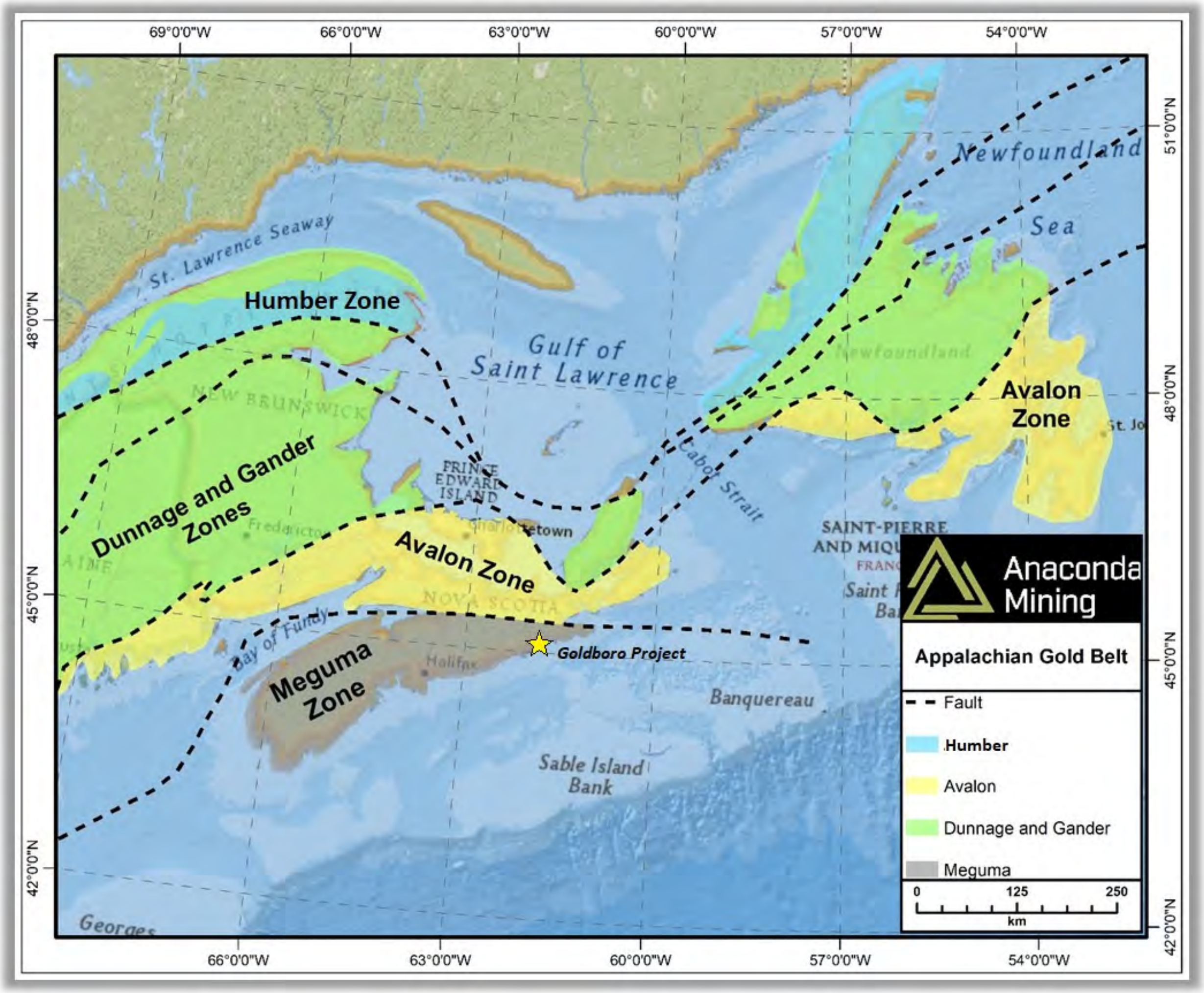
The Meguma zone underlies most of the southern mainland of Nova Scotia and is structurally juxtaposed against the Avalon zone to the north along the Cobequid-Chedabucto Fault System during the (Acadian) Orogeny (*Smith and Kontak, 1996)*. The Meguma zone is predominately made up of the Meguma Group consisting of Cambro-Ordovician sedimentary rocks deposited along the continental margin of the Gondwana paleo-continent during closure of the Iapetus and Rheic oceans (*Smith and Kontak, 1996*). The Meguma Group includes the Goldenville Formation, a basal sandy flysch sequence estimated to be 6.7 km thick, and the overlying Halifax Formation, a shaley flysch sequence approximately 11.8 km thick (*Sangster and Smith, 2007*).

The massive, thick-bedded greywacke sequence of the Goldenville Formation is dark grey (carbonaceous) to light grey in colour and contains thin argillite horizons that commonly separate the thick, coarser beds. The Goldenville Formation grades upwards through manganese-rich strata into a basal Halifax Formation unit that consists of sulphidic black slate. The manganese-rich section, along with Tremadocian fossils, marks the transition between the two formations. Black carbonaceous sulphidic slate and thinly bedded to cross-laminated siltstone comprises much of the Halifax Formation, but lithologies in the uppermost stratigraphy consist mostly of grey-green slate and siltstone (*Sangster and Smith, 2007*).

The Meguma Group is pervasively folded and characterized by kilometre-scale wavelengths and E-W to NE-SW axial trace directions. Folds are upright to slightly inclined, with plunges to both east and west. Doubly-plunging fold trends produce domal structural culminations that in many instances correspond with historic gold producing districts. Cleavages are also a predominant structural feature and include regional slaty cleavage, AC cleavage, and pressure-solution cleavage. The bedding-cleavage intersection lineation reflects local plunge variations and indicates a general non-cylindrical character (*Horne, 1996*).

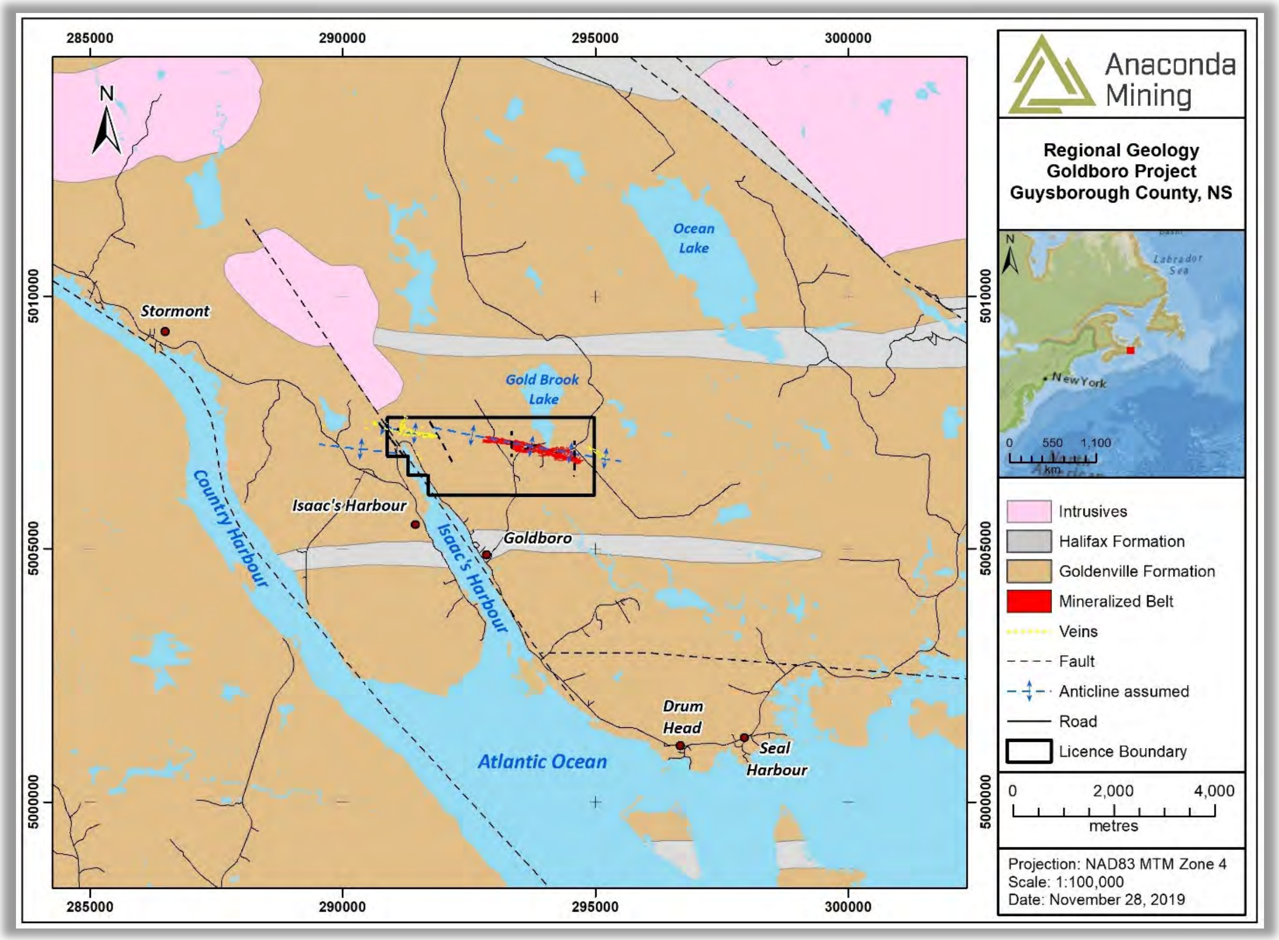
The Meguma Group in the eastern part of Nova Scotia (Figure 7.2) was metamorphosed to greenschist-amphibolite facies grade during the mid-Devonian Acadian Orogeny (ca. 400 Ma) and was subsequently intruded by peraluminous granite, granodiorite, and minor mafic intrusions of mid-Devonian to Carboniferous age (ca. 375 Ma) (*Sangster and Smith, 2007*).

Figure 7.1 Tectonostratigraphic Zones of the Northern Appalachian Orogen



Source: Anaconda 2019; after Pollock et al., 2012

Figure 7.2 Regional Geology of Eastern Nova Scotia



Source: Anaconda 2019

7.2 LOCAL GEOLOGY

7.2.1 GOLDBORO PROPERTY GENERAL GEOLOGICAL SETTING

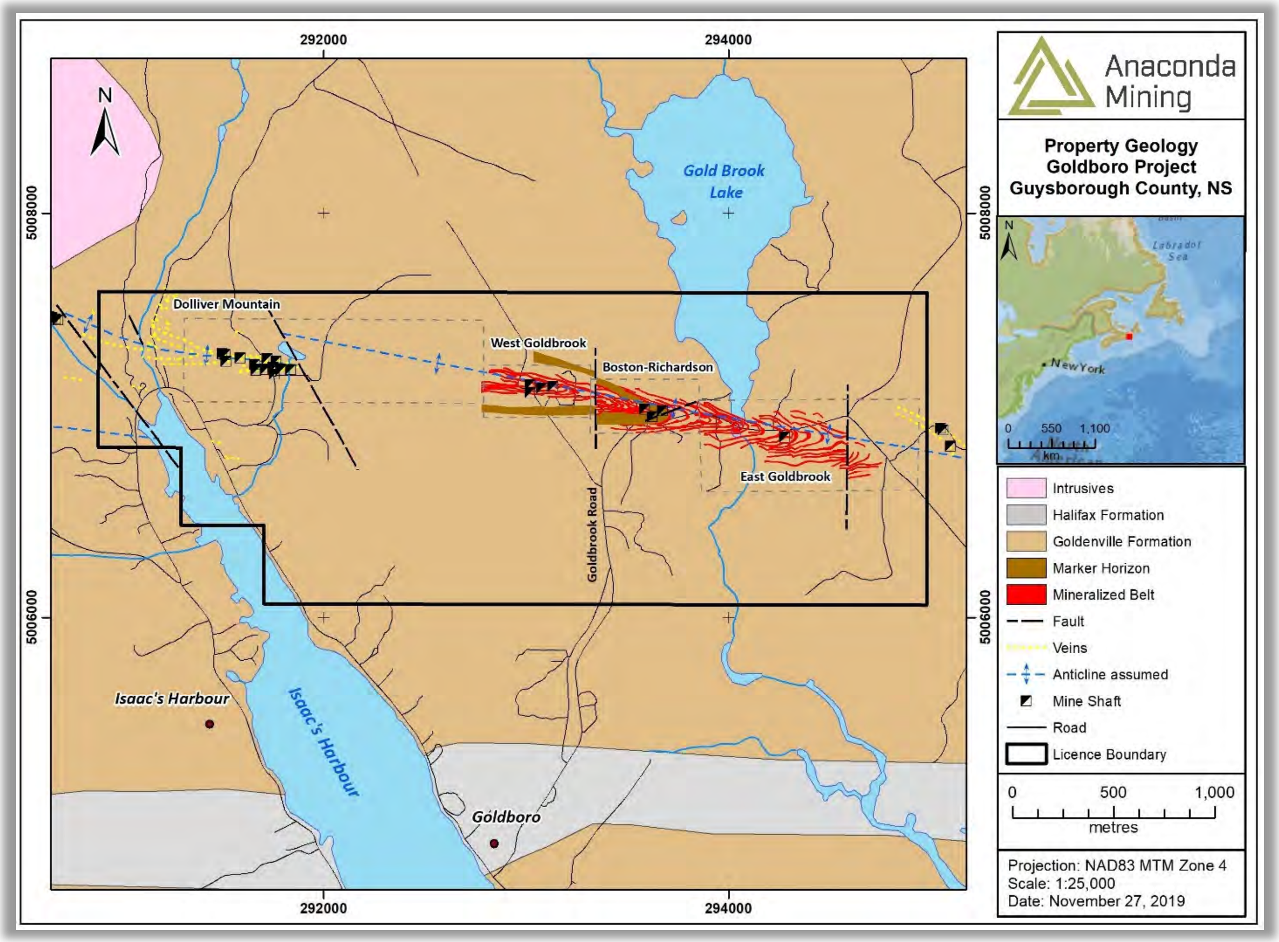
The Project is underlain by rocks within the Goldenville Formation dominated by greywacke and argillite. Rocks of the overlying Halifax Formation are located approximately 1.6 km south of the Project (Figure 7.3). The stratigraphy dips steeply to the south exposing a stratigraphic section of the Goldenville Formation from Goldboro to the Halifax Formation that spans approximately 1.5 km of true thickness.

At the Deposit, the Goldenville Formation consists of alternating greywacke and argillite beds with an approximate true thickness of 950 m. The base of the stratigraphic sequence intersected in the BR Gold System consists of roughly 325 m of alternating greywacke and argillite, with varying proportions of both rock types, ranging in thickness from less than 1 m up to 10 m. This is overlain by the Marker Horizon, which consists of a 40-50 m greywacke bed that is commonly intersected during drilling and in underground workings (Figure 7.4). The Marker Horizon appears to thin or get cut off by the New Belt Fault on the northern limb of the anticline towards the west. Above the Marker Horizon is the EG Gold System, approximately 560 m thick, consisting of alternating greywackes and argillites. Within the EG Gold System there is a second, thick, greywacke sequence varying in thickness from 20 – 60 m. This may represent a new marker unit within the stratigraphy.

The structure of the Goldboro area is dominated by the Upper Seal Harbour Anticline. The anticline folds all levels of stratigraphy observed in core and underground to form an upright, tight anticline that plunges shallowly to the east (10° to 30°). The enveloping surface to bedding dips moderately eastward at 20°. Younging is upward, orthogonal to the hinge and limbs of the fold. An axial planar cleavage is well developed at all levels of stratigraphy but pervasive within the hinge of the fold. The intersection of the axial planar cleavage forms an intersection lineation commonly observed on bedding surfaces that plunge parallel to the fold axis. All bedding and first-generation structures are refolded by open reclined folds that modify the axial plane and limbs of the Upper Seal Harbour Anticline. The axial plane of second-generation folds dips shallowly and an axial planar cleavage is observed in core and within underground workings.

All earlier structures are deformed by late brittle faults. One generation of these faults, which includes the New Belt Fault, are steeply dipping and occur both parallel and cross-cutting regionally folded stratigraphy. These faults also disrupt the stratigraphy on the northern limb of the fold structure in the WG and BR Gold Systems, although kinematics and displacement are not known. A second generation of faults strike northerly and are steeply dipping, these offset the axial trace of the anticline. The WG Fault forms the boundary between the WG and BR Gold Systems. Displacement along the WG Fault indicates roughly 50 m of normal, west side down movement and approximately 30 m of right lateral movement.

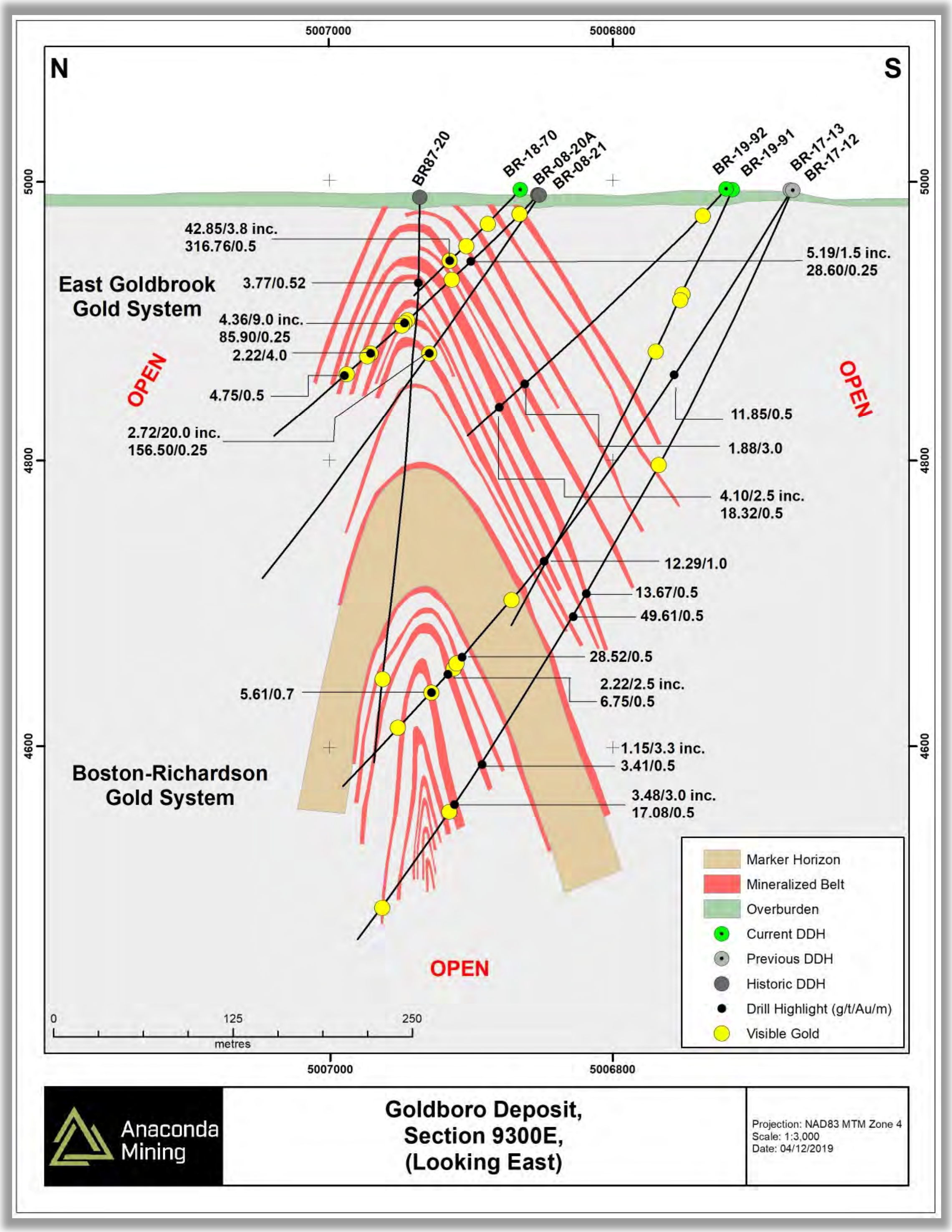
Figure 7.3 Location of Mineralized Belts at Goldboro Showing Outline of Eastward Plunging Anticlinal Fold



Source: Anaconda 2019

***Note**: All map area falls within Exploration Licence 05888*

Figure 7.4 Typical Cross-Section at 9300E through the Deposit Area



7.3 MINERALIZATION

7.3.1 GENERAL

Gold mineralization at the Deposit occurs in both quartz veins and within the argillite that hosts the veins, and to a lesser extent, greywacke. Disseminated, euhedral arsenopyrite is associated with gold mineralization, typically observed in the host wall rock and is usually present in mineralized veins. Other sulphides associated with mineralized quartz veins are pyrrhotite, chalcopyrite, pyrite and minor amounts of sphalerite and galena. Wall rock generally contains more pyrrhotite and arsenopyrite than directly associated quartz veins. The gold-bearing quartz veins are stratabound with lesser discordant quartz veins and vein arrays (Figure 7.5). Native gold is nuggety in nature, and grains range from microscopic up to several centimetres in size (e.g. Figure 7.6). Within quartz veins, gold is present as free gold in quartz, and within arsenopyrite grains, along grain boundaries and internal fractures, and is non-refractory in nature. Native gold also occurs as a disseminated phase in some altered argillite intervals, demonstrating is associated with both quartz veins and the altered wall rock.

Figure 7.5 Mineralized Quartz Veins in Graphitic Argillite: Hole BR-18-22

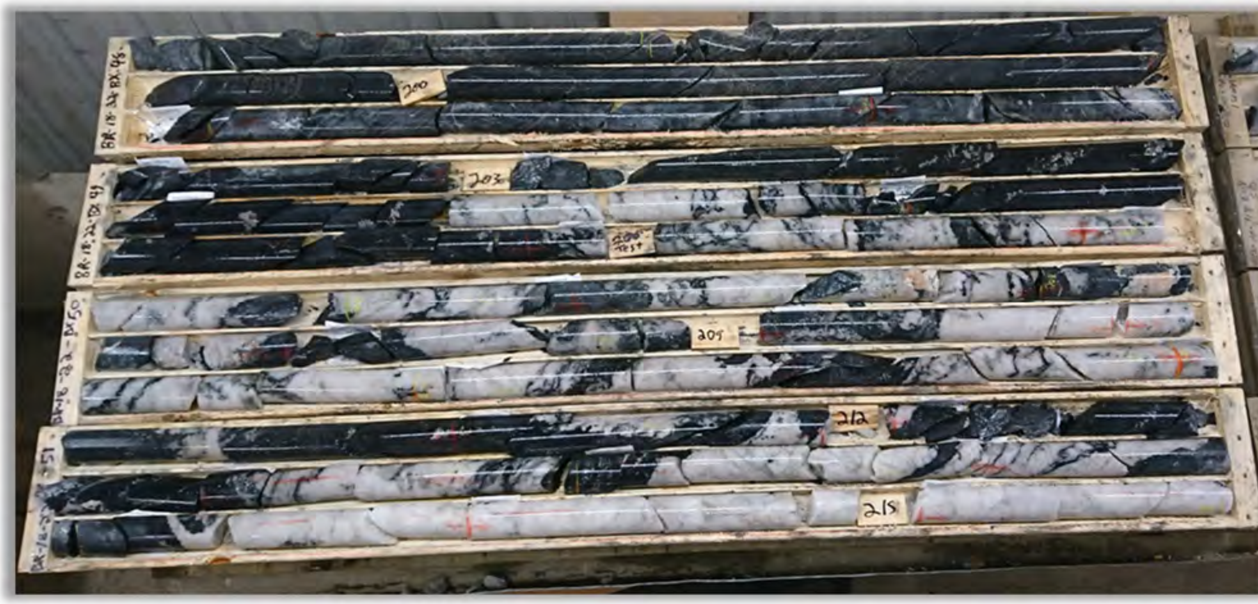


Figure 7.6 Visible Gold in Graphitic Argillite: Hole BR-17-03

A 3 mm grain of visible gold in a crack seal, bedding parallel white quartz vein.

7.3.2 VEIN AND WALL ROCK MINERALOGY

Gold is usually associated with sulphide-bearing quartz veins and altered, sulphidic wall rock. Arsenopyrite is the most common sulphide species present, although pyrite, pyrrhotite, chalcopyrite, galena, and sphalerite are also associated with gold-bearing quartz veins. Gold commonly occurs as a free-milling phase within quartz veins but is also present in direct association with vein-hosted arsenopyrite. In such cases, it commonly occurs as inclusions within arsenopyrite, as free particles associated with micro-fractures cutting arsenopyrite crystals, and as free particles attached to arsenopyrite crystal surfaces (*Ryan and Smith, 1998*). Pyrite also coats fracture and cleavage planes closest to vein contacts and occurs as fine-grained, disseminated subhedral crystals. Pyrite locally exhibits wispy and blebby textures and frequently shows association with late faults (*Gervais et al., 2009*).

Pyrrhotite is a commonly occurring sulphide phase in wall rock and typically is present as disseminated blebs, sometimes flattened in bands along foliation planes, or as irregular blebs at quartz vein contacts. Pyrrhotite also occurs in both wall rock and veins as a fracture coating phase and as very fine stringers. Chalcopyrite is almost exclusively confined to quartz veins and is present as fine-grained blebs concentrated along microfractures. Galena in small amounts is present in association with quartz vein hosted visible gold but is otherwise generally rare in occurrence. Sphalerite is rarely observed, but where present occurs as mm-scale blebs within or along fractures within quartz veins (*Gervais et al., 2009*).

7.4 STRUCTURE

7.4.1 STRUCTURAL GEOLOGY AND DEFORMATION PHASES

Five deformation phases (D_1 through D_5) are recognized in the Meguma Group and the descriptions presented below are modified slightly after those of Smith et al. (1985), Corey (1992), and Smith and Kontak (1988), unless otherwise indicated, as well as observations by Anaconda geologists.

The first deformation phase (D_1) is marked by an early bedding parallel foliation (S_1) and is not easily recognized on the Goldboro Property. The second deformation phase (D_2) resulted in development of northeast to east trending, upright regional folds with well-developed, steeply dipping axial planar pressure solution and slate cleavages. Pressure solution cleavage is well developed in thicker (1-50 m) greywacke beds on the Goldboro Property. Slate cleavage is ubiquitous in the finer clastic units and bedding transpositions along this fabric is commonly seen. The D_3 deformation phase consists of flat to shallowly-lying pressure solution cleavage developed locally and defined by the growth of oriented andalusite and biotite porphyroblasts in argillites. The D_2 and D_3 fabric elements have been recognized in outcrop on the Goldboro Property in the Dolliver Mountain area, approximately 2 km west of the Boston-Richardson mine, observed in the underground workings at Boston-Richardson, and in drill core throughout the Goldboro Gold Deposit. The D_4 deformation occurs as discrete brittle faults that both parallel and crosscut regionally folded stratigraphy and developed following peak metamorphic conditions and gold mineralization. The New Belt Fault is an example of D_4 deformation and strikes approximately 085° and ranges in dip between 90° and 85°. Displacement and kinematics along this structure has not been defined. The D_5 deformation phase was recognized in the form of late NW-trending brittle faults plus locally developed crenulation cleavage and kink bands that are seen in argillites.

The N-S trending, WG Fault which separates the WG Gold System from the BR Gold System is interpreted as a D_4 structure. Recent drilling by Anaconda in the vicinity has confirmed historic interpretations that the WG Gold System is the offset western extension of the BR Gold System. The displacement of the Marker Horizon across the WG Fault demonstrates it is an oblique normal fault with roughly 50 m of normal, west side down movement, and approximately 30 m of right lateral movement.

7.4.2 STRUCTURAL SETTING, VEIN STYLES, AND GENETIC MODEL

The following discussion of structural setting, vein styles, and genetic model were modified after descriptions presented in Gervais et al. (2009) and observations by Anaconda employees and consultants.

Gold mineralization at Goldboro occurs in quartz veins and wall rocks adjacent to the veins. At the deposit scale, the veins form a swarm and are clearly located in the flexure zone (hinge and adjacent limbs) of the Upper Seal Harbour Anticline. The gold-bearing veins are found in a 175 – 275 m wide envelope centred on the axial surface. The veins occur mostly on the limbs of the fold, but also in the hinge, and all are hosted by turbiditic metasedimentary rocks consisting of greywacke and argillite.

Gold-bearing quartz veins occur as both stratabound and cross-cutting entities and are most prevalent within bedding parallel stratigraphy occurring approximately 80 m north or south of the Upper Seal Harbour Anticline's axial surface. Deformation by compression and shearing associated with regional folding is greatest near this axial zone. A flexural slip model of fold evolution is considered to be most applicable in this area due to drilling defined thickening of individual slate units within the fold hinge zone relative to fold limb zone positions, and pervasive development of axial planar cleavage along which bedding transposition occurs in symmetry with that expected for a flexural slip process.

Gervais et al. (2009) and recent observations also indicate that progressive heterogeneous strain is localized within less competent argillites relative to greywacke. This is interpreted to be due to competency contrast between the argillites and greywackes. Contemporaneous fracturing and ductile deformation provides a mechanism for circulation of hydrothermal fluids from which gold-bearing quartz veins were developed (*Gervais et al., 2009*).

7.5 ALTERATION

Regionally metamorphosed turbidites in the hinge zone of the Upper Seal Harbour Anticline have been variably altered and are characterized by presence of carbonate, sericite, sulphide, tourmaline, and chlorite assemblages that post-date growth of regional metamorphic peak porphyroblasts (*Roy and Labelle, 1990*). Sericitic alteration and bleaching is common in greywacke units and preferentially developed along pressure solution cleavage planes and in association with irregular patches of silicification. Silicification and disseminated arsenopyrite are spatially centred on the axial plane, where dark green to black colouration associated with alteration typifies slate intervals. Chloritization of biotite produces dark grey to black host rock colours in gold-bearing zones. Whole-rock geochemistry results may indicate that potassium and carbon enrichment are often spatially associated with such zones and drill log records show that in some cases feldspar also occurs in related quartz veins (*Gervais et al., 2009*).

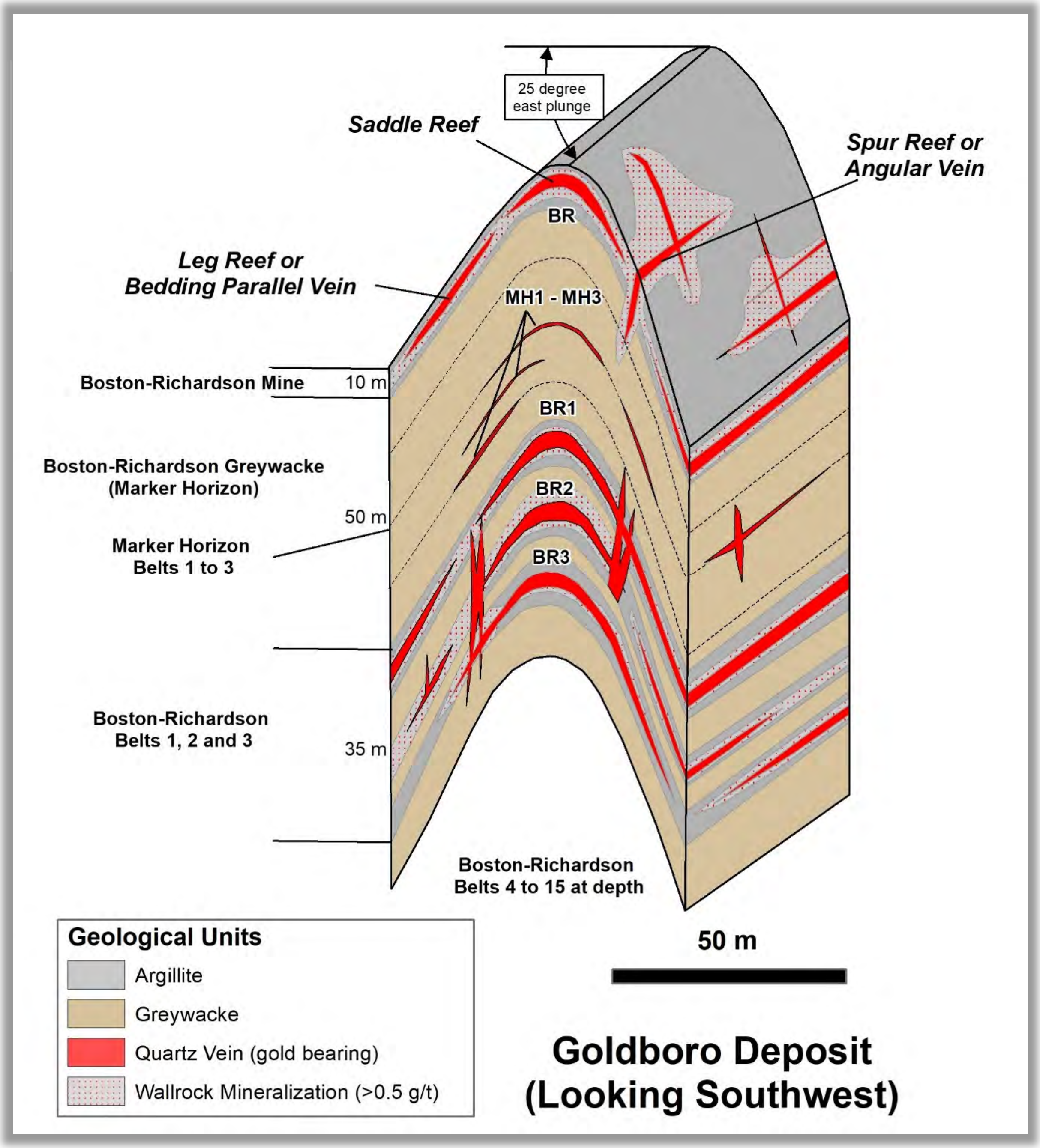
8 DEPOSIT TYPE

The turbidite-hosted gold deposits of Nova Scotia have been compared to similar-age turbidite-hosted quartz vein deposits elsewhere in the world, particularly those in the Bendigo and Ballarat areas of the Lower Paleozoic Lachlan Fold Belt in the state of Victoria, Australia, and have historically been similarly classified. Robert et al. (1997) recognized this deposit class and proposed that it be identified as a member of the 'Turbidite-hosted, quartz-carbonate vein deposit (Bendigo Type)' category. Ryan and Ramsay (1996) also addressed the similarity of Nova Scotia turbidite-hosted gold deposits with those in Victoria. As noted by Gervais et al. (2009), categorization within the USGS classification system of mineral deposits presented by Berger (1986) places the Deposit in the broad 36A category of 'Low-Sulphide Gold-Quartz Vein Deposits'.

The Deposit is a turbidite-hosted orogenic gold deposit hosted within a sequence of alternating argillites and greywacke metamorphosed to greenschist facies (Figure 8.1). These deposit types are typically characterized by the formation of gold-bearing quartz veins within the argillite units, commonly referred to as mineralized belts, that are interbedded with greywacke units. There are currently 57 stacked mineralized belts ranging in thickness from 1 m to 20 m in the Deposit. The metasedimentary units on the Property are folded into the tight, gently east-plunging Upper Seal Harbour Anticline and gold mineralization has typically been deposited at various positions and times during the fold formation process. Veins, which form during deformation, occur in three major geometries commonly referred to as reefs: saddle reefs, leg reefs and spur reefs. Saddle reefs occur about the apex of the fold and are the dominant vein types within some deposits. Leg reefs extend down the limbs of the fold, beyond the saddle reef and are generally parallel with the metasedimentary layers. These are also commonly termed bedding parallel or 'BP' veins in the Nova Scotia goldfields. Spur reefs are veins that cross between layers and may be in the apex of the fold or on its limbs. This style of vein is in part captured under the term "angular" in the Nova Scotia goldfields.

The Deposit contains all three types of reefs outlined above but is also characterized by mineralization within the argillite forming the belts. Because the Deposit contains saddle, leg and spur reefs and often has gold mineralization within the argillite hosting the veins, it has potential to contain significantly more gold resources than deposits of similar style that contain gold only within the quartz veins (reefs) themselves. The trace of the Upper Seal Harbour Anticline transects the Property and is found near the Dolliver Mountain prospect 2 km to the west of the Deposit demonstrating that the structure which hosts gold continues for several kilometres.

Figure 8.1 Generalized Model of Mineralization within the Goldboro Deposit



9 EXPLORATION

9.1 BOREHOLE GEOPHYSICAL PROGRAM

Anaconda completed borehole geophysical surveys on the Project in 2017 and 2018. The goal of the program was to identify geophysical signatures of the host rock to mineralization that might produce unique patterns of the various units and mineralized belts to aid in the geological interpretation and correlation.

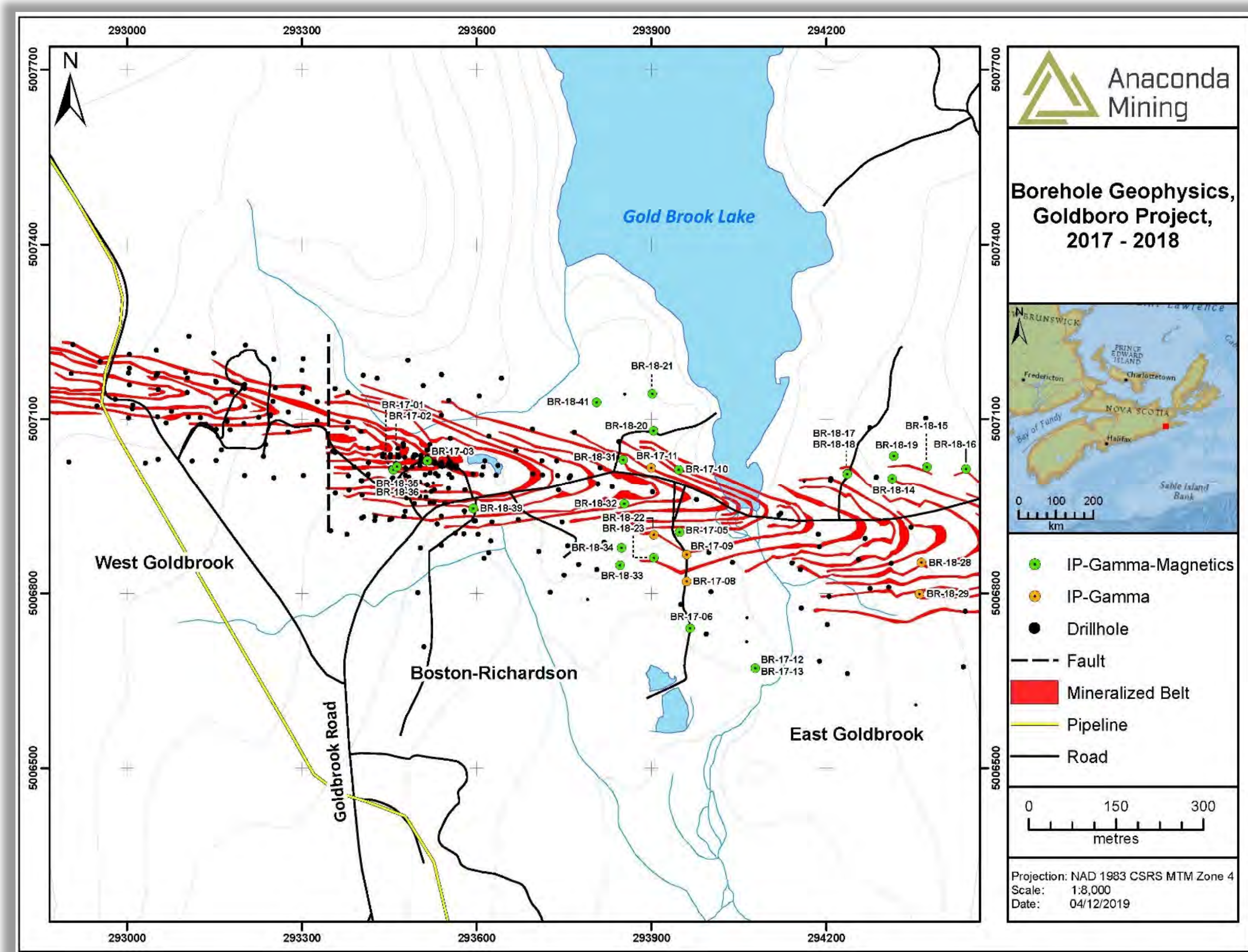
The geophysical equipment was rented from Terraplus Inc. of Richmond Hill, ON and included a 5MXA logging console, a 4MXC 1,000 m winch, a Q40GRA-1000 total count natural gamma probe, a Q40RES-1000 resistivity/IP probe, and a 2BSF-1000 Bartington magnetic susceptibility probe. Surveys were carried out by Anaconda employees Matt Walsh and Linsey Gale. The equipment was used to test 29 holes (8,843 m). Table 9.1 provides a summary of geophysical data collected for each drillhole. Figure 9.1 shows the collar location of the 29 drillholes colour coded to reflect the geophysical surveys completed. The raw data was sent to geophysical consultant Bob Lo, P.Eng., for processing and interpretation.

Results indicated that resistivity and gamma-ray logs were able to distinguish some units from each other. The resistivity and gamma-ray data showed characteristic signatures unique to the two main lithologies (argillite and greywacke) in the deposit. However, the data overall did not provide results which led to recognition of stratigraphic markers that could be used for correlation and geological modelling.

Table 9.1 Summary of the 2017-2018 Geophysical Logging Program

Hole ID	IP+Gamma and Mag	IP+Gamma
BR-17-08	X	X
BR-17-09	X	X
BR-17-11	X	X
BR-18-22	X	X
BR-18-28	X	X
BR-18-29	X	X
BR-17-01	X	X
BR-17-02	X	-
BR-17-03	X	-
BR-17-05	X	-
BR-17-06	X	-
BR-17-10	X	-
BR-17-12	X	-
BR-17-13	X	-
BR-18-14	X	-
BR-18-15	X	-
BR-18-16	X	-
BR-18-17	X	-
BR-18-18	X	-
BR-18-19	X	-
BR-18-20	X	-
BR-18-21	X	-
BR-18-23	X	-
BR-18-31	X	-
BR-18-32	X	-
BR-18-33	X	-
BR-18-34	X	-
BR-18-39	X	-
BR-18-41	X	-

Figure 9.1 Plan Map of Drill Collar Locations and Completed Geophysical Surveys



9.2 UNDERGROUND BULK SAMPLE

In the fall of 2018 and early winter of 2019 Anaconda completed an underground bulk sample (the Bulk Sample) within the BR Gold System consisting of approximately 10,000 tonnes. The objectives of the Bulk Sample were to provide a better understanding of the geology, test for spatial and grade continuity of the mineralized structures, validate the existing mineral resource model and explore various types of mining methods to best extract the mineralized material. The Bulk Sample has provided valuable geological, operational and processing information for design and optimization of the overall project in a feasibility study expected to be published in the first quarter of 2020.

The Bulk Sample was shipped to Anaconda's Pine Cove Mill at the Point Rousse Complex in Newfoundland during the summer of 2019 for processing. The mineralized material was processed using Anaconda's current Pine Cove mill configuration but with the addition of a gravity circuit to aid in gold recovery, in addition to the crushing, grinding, and flotation circuits. Anaconda is in the process of reconciling the Bulk Sample and expects to announce the results in Q1 of 2020.

9.2.1 OPERATIONS AND ENGINEERING

Anaconda engaged Cementation Canada Inc. (Cementation) as the mining contractor to assist in underground development and Bulk Sample extraction.

Site upgrades on surface were limited to those necessary to satisfy the site IA requirements for the Bulk Sample. These included securing the area around the historic Boston-Richardson shaft, prepping it for the installation of a pump, installing the necessary surface piping for the dewatering of the workings and supply of mine water, and replacing the decant construction in the settlement pond in order to control the discharge of mine water to the environment.

Additionally, all ground support in the portal area had to be replaced and upgraded with a reinforcement of concrete above the entrance to the underground workings. This included mechanical rock bolts, rebar, and screening of the portal walls. As much weathered material from the rim of the portal was removed as possible using an excavator to ensure that the ground support was securely fastened. The overburden around the rim of the portal was also benched back to ensure stable slopes around the entrance to the portal. A steel cable was positioned around the perimeter of the portal excavation which was attached to numerous concrete blocks to provide a visual barrier, six feet back from the crest. Survey control was established at the portal entrance to provide reference for underground surveying.

Extraction of the Bulk Sample utilized the existing decline, ramp, and cross-cuts developed in the late 1980s to minimize the development needed to access mineralized zones and planned mining stopes adjacent to existing workings. New development included 3 drifts off the existing cross-cuts in Level 1 (23 masl) and Level 2 (-13 masl) and the creation of a newly-developed ramp to Level 3 (-22 masl) with two additional drifts resulting in 213 m of new development. A total of 86 mining faces were completed. Additionally, a total of 13 raise faces were blasted throughout the workings. Mining was advanced using a single-boom jumbo, typically with one round blasted per 12-hour shift. Blasted rock was mucked using 1.5 and 2 cubic yard scooptrams, which were then loaded into a 7 cubic yard truck. The mineralized material was stockpiled on surface in a secure location immediately south of the portal. Site project management and engineering work was overseen and completed by Anaconda engineers Iain Smart, P.Eng., Bethany Lefort, Julian Vincent-Smith, and Brett Wing.

9.2.2 GEOLOGY, SURVEYING, AND SAMPLING

Mine faces were geologically mapped, photographed and sampled by Anaconda staff and contract geologists Garry Luffman, P.Geo., Michelle English, Even Stavre, P.Geo. (Consultant) and Morgan Silver (Terrane Geoscience Ltd.). Chip sampling was completed along the walls and across the face of each round by a geologist using a rock hammer, with an average of 14 samples collected per round. The lengths, locations and descriptions of each chip and grab sample were recorded by the geologist, compiled, and plotted in Surpac mining software. After each blast, ~0.5 kg to 1.0 kg muck samples were collected in a regular pattern using a five-dice method to get a good representation of material in the blast. Geologists collected 5 to 7 representative muck samples from various areas in the supported blast. In addition, samples were collected every second truck, resulting in one muck sample collected every ~12 tonnes of mined material. Each blast typically had 15 to 30 muck samples collected, and all samples from each round were averaged to determine if each round was considered to be waste (<1.0 g/t gold), low grade (1.0 to 3.0 g/t gold) or high grade (>3.0 g/t gold). Additionally, each mining face had a three-metre-long test hole drilled into each wall approximately one metre from the face, and two sets of longhole test holes drilled every third ring into the backs before stopes were completed. A total of 2,148 muck samples, 1,380 chip samples, 518 test holes, and 132 longhole test holes (excluding QA/QC samples, which were systematically inserted into each sample batch) were collected.

Samples were shipped daily to the assay lab at the Dufferin Mine ("Dufferin") (owned by Resource Capital Gold Corp.), near Sheet Harbour, Nova Scotia, where they were analyzed for gold by fire assay. Pulps and coarse rejects from the samples were routinely shipped back to and stored at the Goldboro Project site. A total of 769 samples (coarse reject material) from the initial assay at Dufferin assaying greater than 1.5 g/t gold were shipped to Eastern Analytical Limited in Springdale, NL for analysis via metallic screen fire assay.

The following provides a summary of the geology of the development and mining by level and heading during the Bulk Sample of the BR Gold System. The plan view map (Figure 9.2) outlines each of the areas of development that Anaconda Mining completed during the Bulk Sample. Five areas of development were completed on three levels at mine elevations of 4965 (Level 1), 4930 (Level 2 W, Level 2 E, Level 2 Ramp), and 4920 (Level 3 W and Level 3 E).

LEVEL 1 WEST AND EAST

Development in Level 1 consisted of a sill drift (3.0 x 3.5 m) comprising three development rounds taken from Level 1 West and one development round taken from Level 1 East. (Figure 9.3 and 9.4).

The geology of Level 1 consisted largely of weakly mineralized greywacke containing 0.5% to 1.0% medium- to coarse-grained arsenopyrite and 10% to 40% angular and en-echelon quartz veins that ranged from 0.1 to 0.3 m thick. Locally, the quartz veins contain 0.5% to 1.0% pyrite and arsenopyrite. A 0.5 to 0.8 m thick black argillite within the NW section of the sill contained the bulk of the mineralization.

The argillite unit hosts minor thin quartz veins (1 to 5 cm thick) and has sheared upper and lower contacts, with bedding/sheared fabric (S_0/S_1) dipping 70° to 75° to the south and striking approximately 100°. There are two pervasive fabrics within the greywacke, the predominant high angle S1 fabric represents the axial planar cleavage of the anticline (270°/80°) and a low angle S_2 cleavage (270°/40°) that overprints the S_1 fabric and quartz veins. The S_1 fabric proximal to the sheared argillite showed local warping/kinking with strike ranging from 270° to 310°.

Figure 9.2 Plan View Map

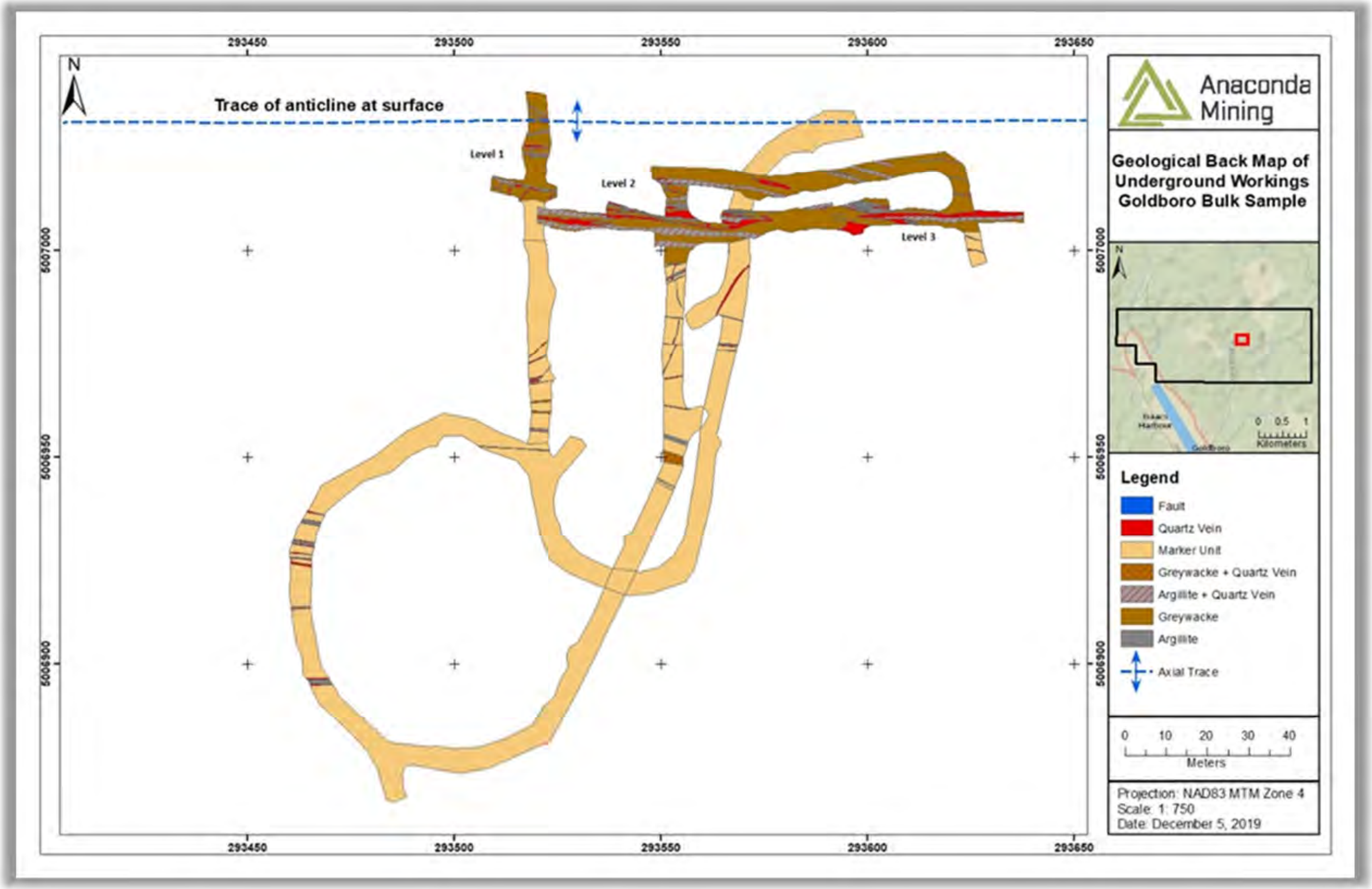


Figure 9.3 Level 1 Geological Back Plan Map

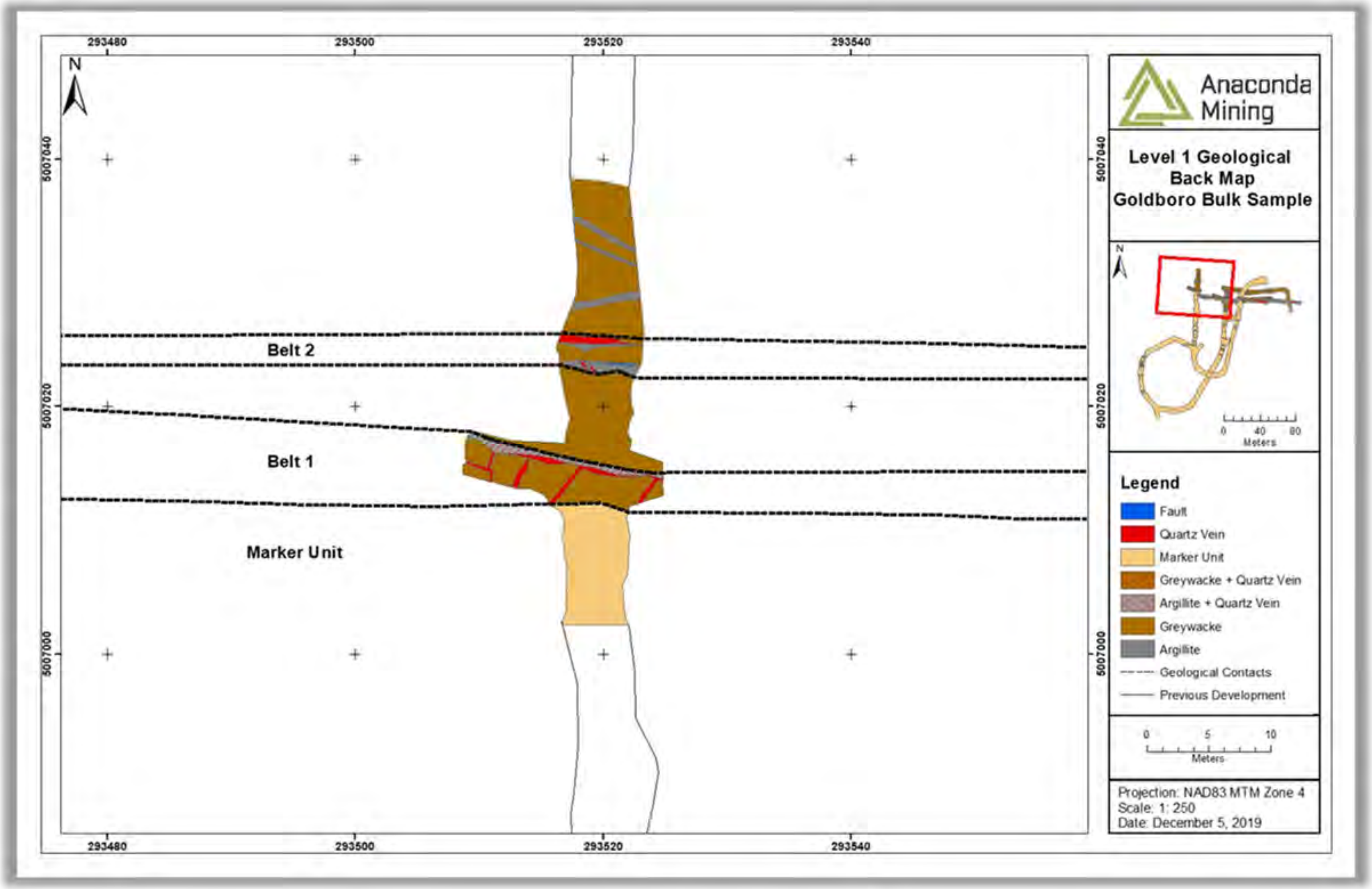


Figure 9.4 Level 1 West Round 4 - Thin Argillite-Quartz Veins Adjacent to Extensional Quartz Veins (looking west)

LEVEL 2 WEST

Development in Level 2 West consisted of a sill drift comprising 13 development rounds. A conventional raise on the northwest end of the sill including four 1.8 x 1.8 m rounds and one 20 m x 10 m x 3 m stope (Figure 9.5). Level 2 West was developed within Belt 2 in the BR Gold System. Geological modeling based on drill results show two to three distinct belts (Belts 1 to 3) within this area, however underground mapping show that Belt 1 and 2 are juxtaposed at this level without interbedded greywacke.

The geology of Level 2 West consists of two mineralized zones within Belt 2. Bedding parallel quartz breccia veins within the mineralized zones vary in thickness between 0.3 to 1.0 m and contain 5% to 30% wall rock clasts and hosted within argillite (Figure 9.6). The mineralized zones range between 0.7 to 1.5 m thick and include weakly mineralized argillite and lesser greywacke. Mineralized zones pinch and swell and locally merge.

Wall rock argillite within the mineralized zones contains 2 to 5% fine-grained arsenopyrite whereas angular greywacke clasts are characterized by 1 to 3% coarse-grained arsenopyrite. Mineralized quartz veins contain coarse- to very coarse-grained, trace to 1% arsenopyrite.

Contacts between argillites and greywacke units are typically sheared with bedding/sheared contacts dipping between 75° to 80° to the south and striking approximately 090º to 095º. Axial planar cleavage (S1) is expressed most clearly in the greywacke units and is defined by nearly vertically dipping (80° to 85°) striking east-west. There is a 1 to 2 cm strongly sheared/faulted zone on the southeast side of the drift.

Figure 9.5 Level 2 Geological Back Plan Map

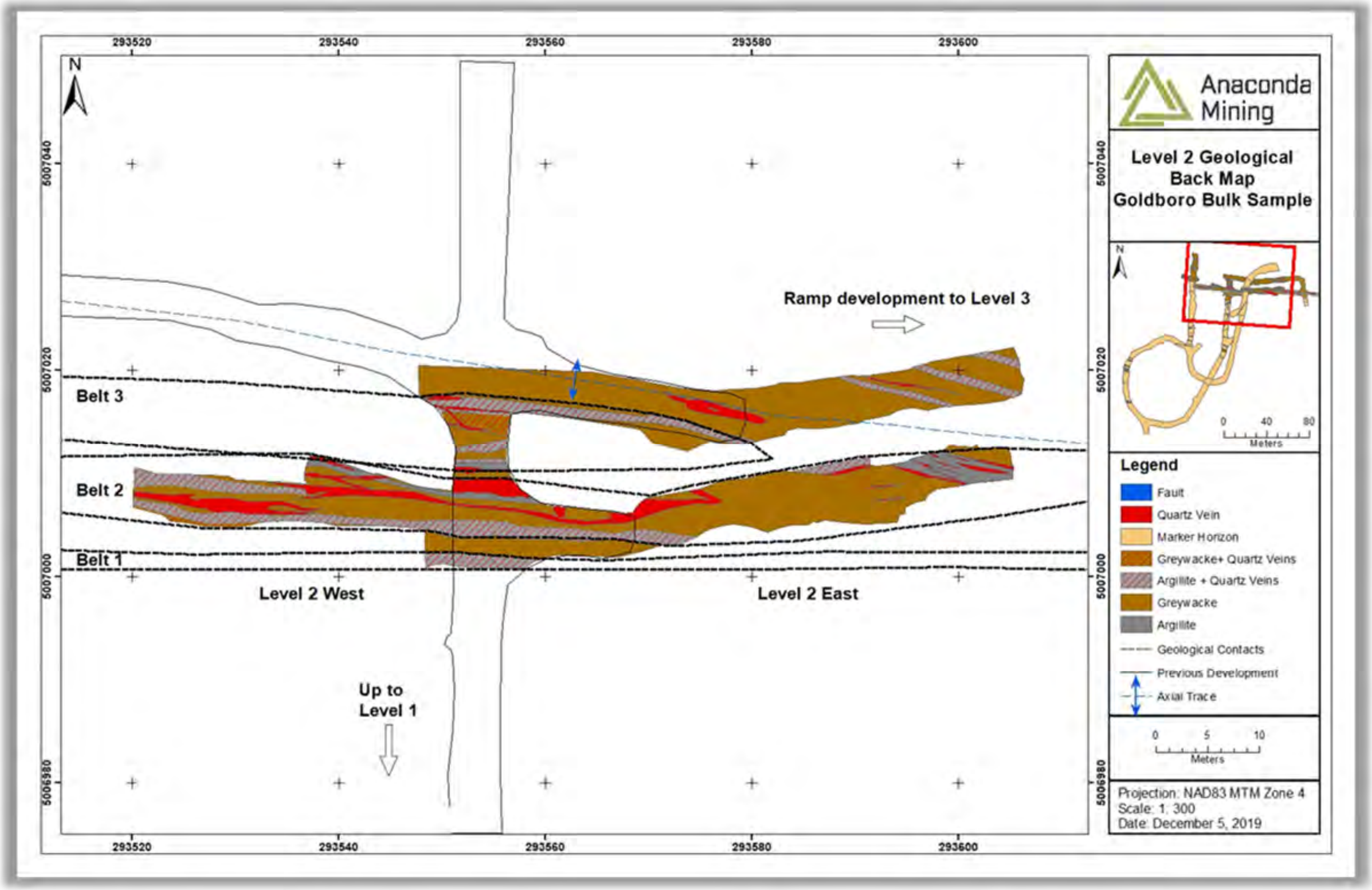
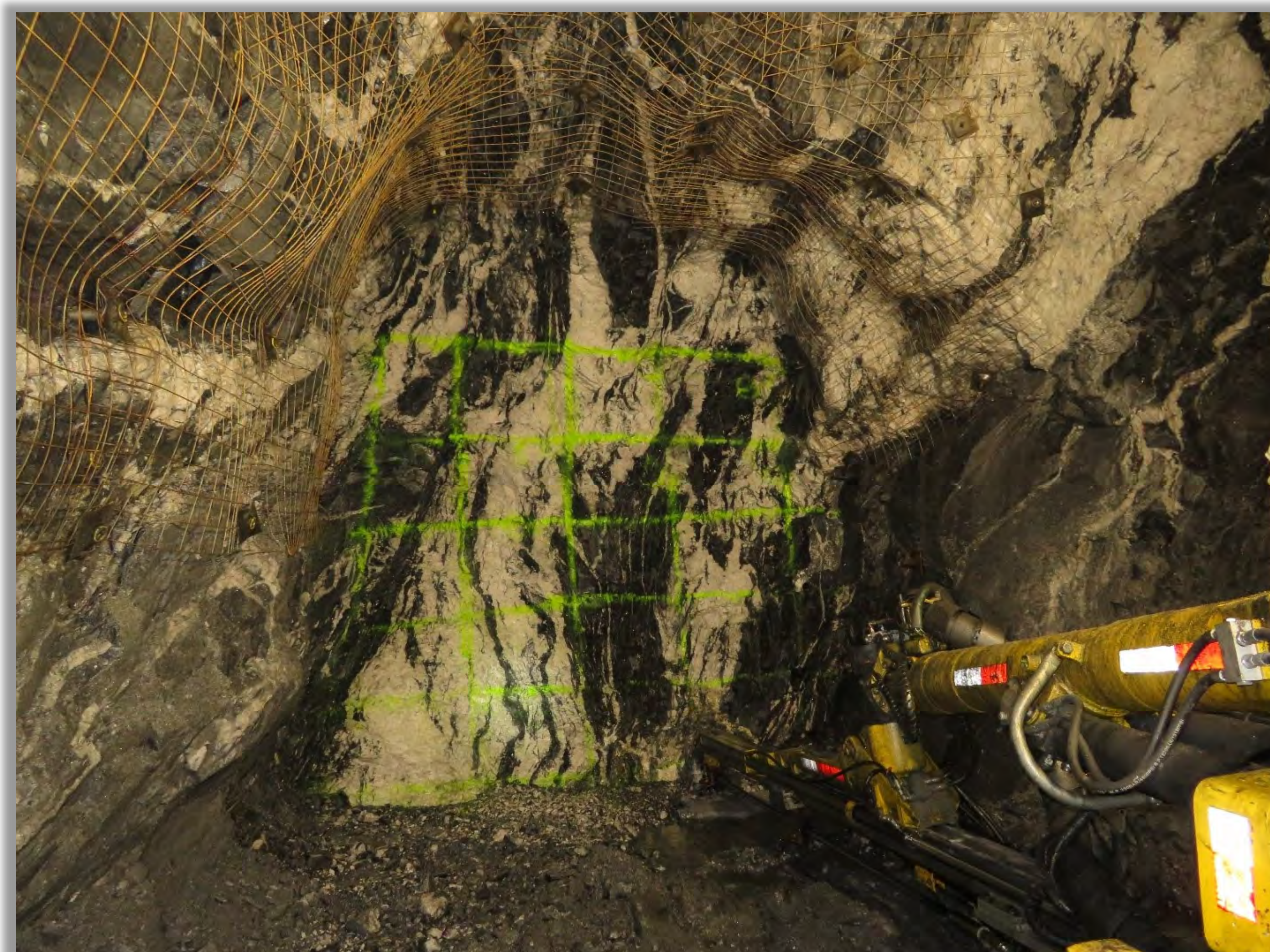


Figure 9.6 Level 2 West Round 8 - North and South Zone of Belt 1 of the BR Gold System (looking west)



LEVEL 2 EAST

Development in Level 2 East consisted of a sill drift including 16 development rounds and 4 slashes, a conventional raise on the northeast end of the sill including four 1.8 x 1.8 m rounds and a 40 m x 10 m x 3 m stope.

Level 2 East was developed within Belt 2. Level 2 East started in a thick folded quartz vein hosted in a 3.4 m-thick greywacke in fault contact (270°/80°) with a unit consisting of bedding parallel, weakly sheared argillite hosted quartz veins. Mineralization in the quartz veins consisted of 0.5 to 2.0% arsenopyrite, pyrite, and lesser chalcopyrite.

Proceeding eastward the quartz vein thinned to 1.1 m and outlined a fold hinge plunging shallowly eastward. The dip of the axial planar cleavage (~80°) within the greywacke fanned from north to south. Four slashes were taken along the north wall of the sill to drift into the mineralization of Belt 2. The mineralized belt consisted of black, graphitic argillite with 1.0% to 5.0% (up to 10.0% arsenopyrite) mineralization with 70% to 80% bedding-parallel, breccia quartz veins. Figure 9.7 shows 1 m thick mineralized argillite-quartz veins of Belt 2 on the left side of the photo. Right side of photo is weakly to non-mineralized greywacke with the top of a hinge on a parasitic fold in Belt 2.

Figure 9.7 Level 2 East Round 8. Looking East.

LEVEL 3 WEST

Development in Level 3 West was a bottom sill drift to Level 2 East consisting of 21 development rounds and a 40 m x 10 m x 2-3 m stope between Level 2 East and Level 3 W.

Level 3 West was developed within Belts 1 and 2. As in Level 2 West the belts are juxtaposed in this area making it difficult to determine exactly which belt was developed (Figure 9.8).

The start of Level 3 West consisted of a 2.5 m thick quartz vein within mineralized argillite and minor greywacke on the southern margin. The zone generally ranged from 1.2 to 3 m thick but did pinch and swell. The mineralization is associated with breccia quartz veins with 5% to 20% mineralized wall rock clasts (Figure 9.9). Locally there are massive quartz veins along the south face and wall. These veins contained trace coarse-grained arsenopyrite and thin laths of argillite within the folded veins.

Figure 9.8 Level 3 West Geological Back Plan Map

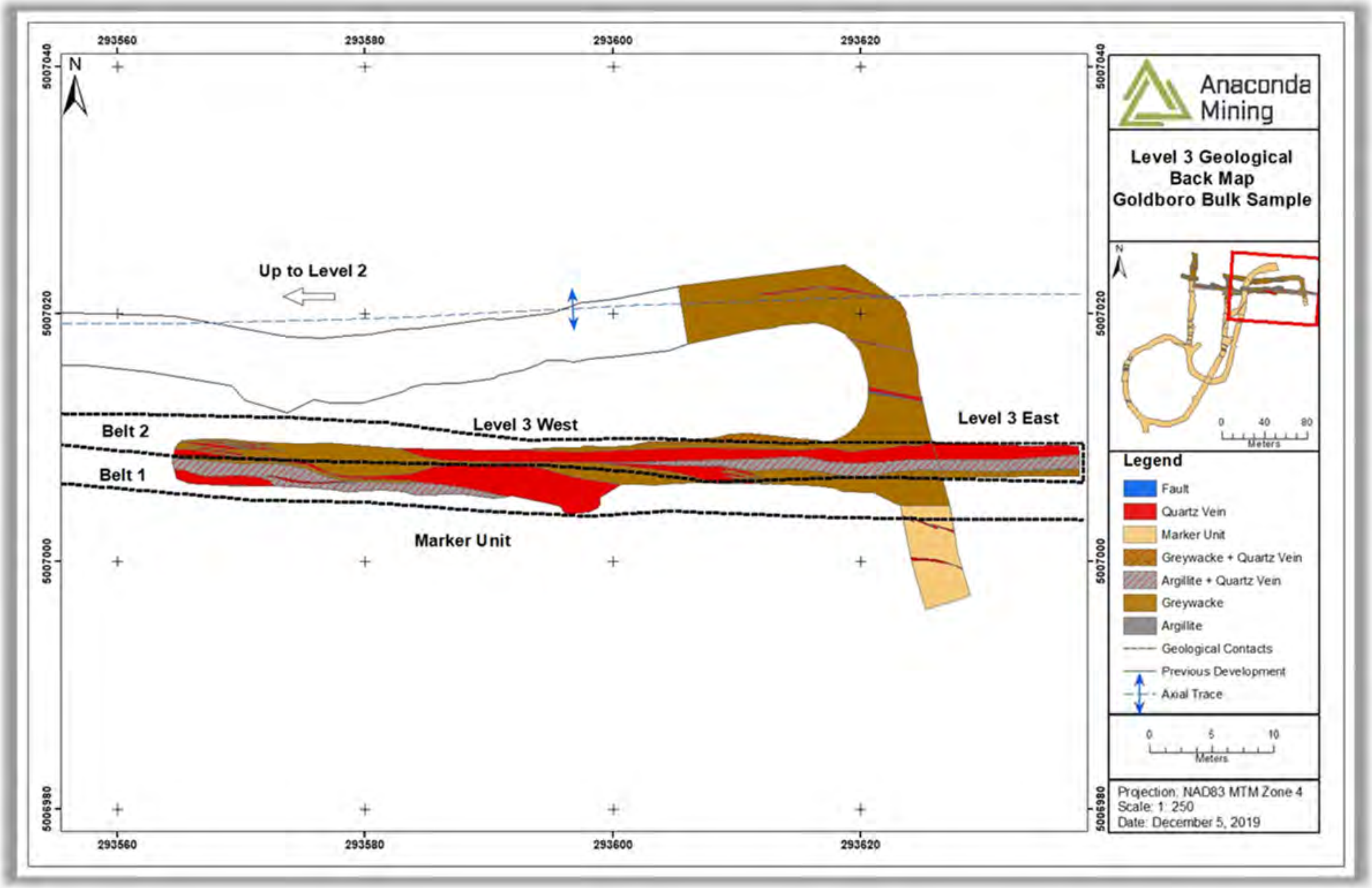


Figure 9.9 Level 3 West Round 30 - Steeply Dipping Breccia Quartz Veins Hosted in Argillite in Belt 2 (looking west)

LEVEL 3 EAST

Development in Level 3 East consisted of a sill drift including 4 development rounds and was developed within Belt 2 (Figure 9.10).

The start of Level 3 East consists of a bedding parallel, 2.5 m thick quartz-argillite unit (Figure 9.11). At the margins of this unit are weak to moderately mineralized greywacke with lesser bedding parallel quartz veins. Contacts between the mineralized unit and greywacke are sheared (270°/80°). Mineralization within the quartz veins consists of 0.5-2% medium-grained arsenopyrite, trace to 1.0% coarse-grained galena and trace amounts of visible gold (Figure 9.12). Mineralized argillite consists of 2 to 5% fine-grained arsenopyrite and medium- to coarse-grained arsenopyrite in greywacke.

Figure 9.10 Level 3 West Geological Back Plan Map

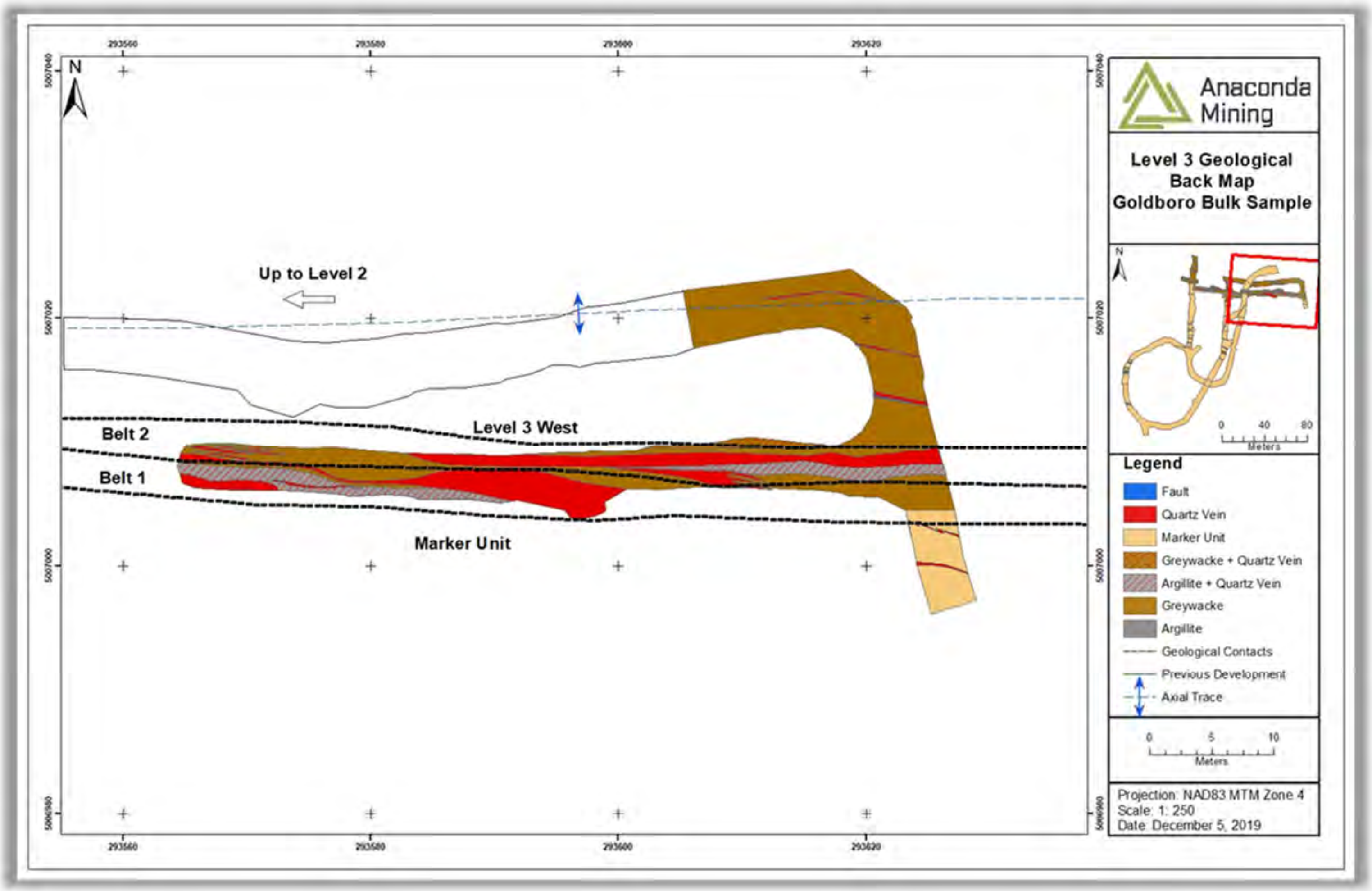


Figure 9.11 Level 3 East Round 2 (looking east)

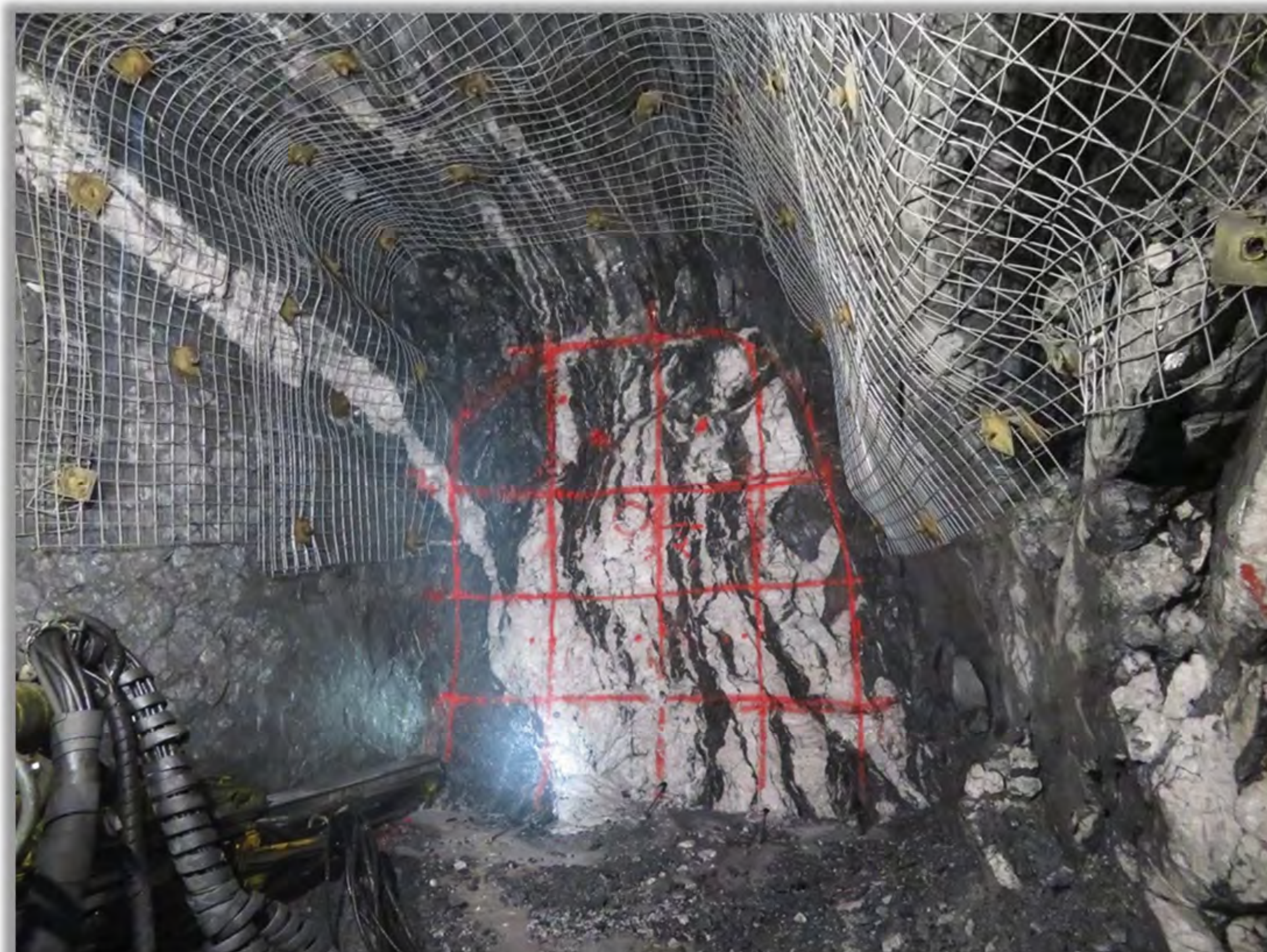


Figure 9.12 Visible Gold Along Margin of Quartz Vein- Argillite Contact in Level 3 East Round 3

10 DRILLING

10.1 INTRODUCTION

The current resource estimate includes drilling results obtained by several firms from 1984 to 2019. Table 10.1 provides a chronological summary of associated drillholes.

Representative geological and assay cross-sections based on the combined drilling database used in the current resource estimate are included in report Sections 7 and 14, and Figure 10.1 provides a summary view of drill collar distribution. A tabulation of drill collar coordinates and hole orientation data appears in Appendix A.

Gervais et al. (2009), Puritch et al. (2006) and Bourgoin et al. (2004) previously described details of drilling programs carried out in support of their respective mineral resource estimates and documented specific details of such programs. Review of these descriptions as well as underlying support documents by WSP showed that all programs were carried out to industry standards of their respective periods by competent technical and professional staff. All programs included detailed and systematic geological logging, sampling, and reporting procedures as well as systematic recording of downhole survey data which, with the exception for a few early holes, was captured using modern borehole survey instrumentation.

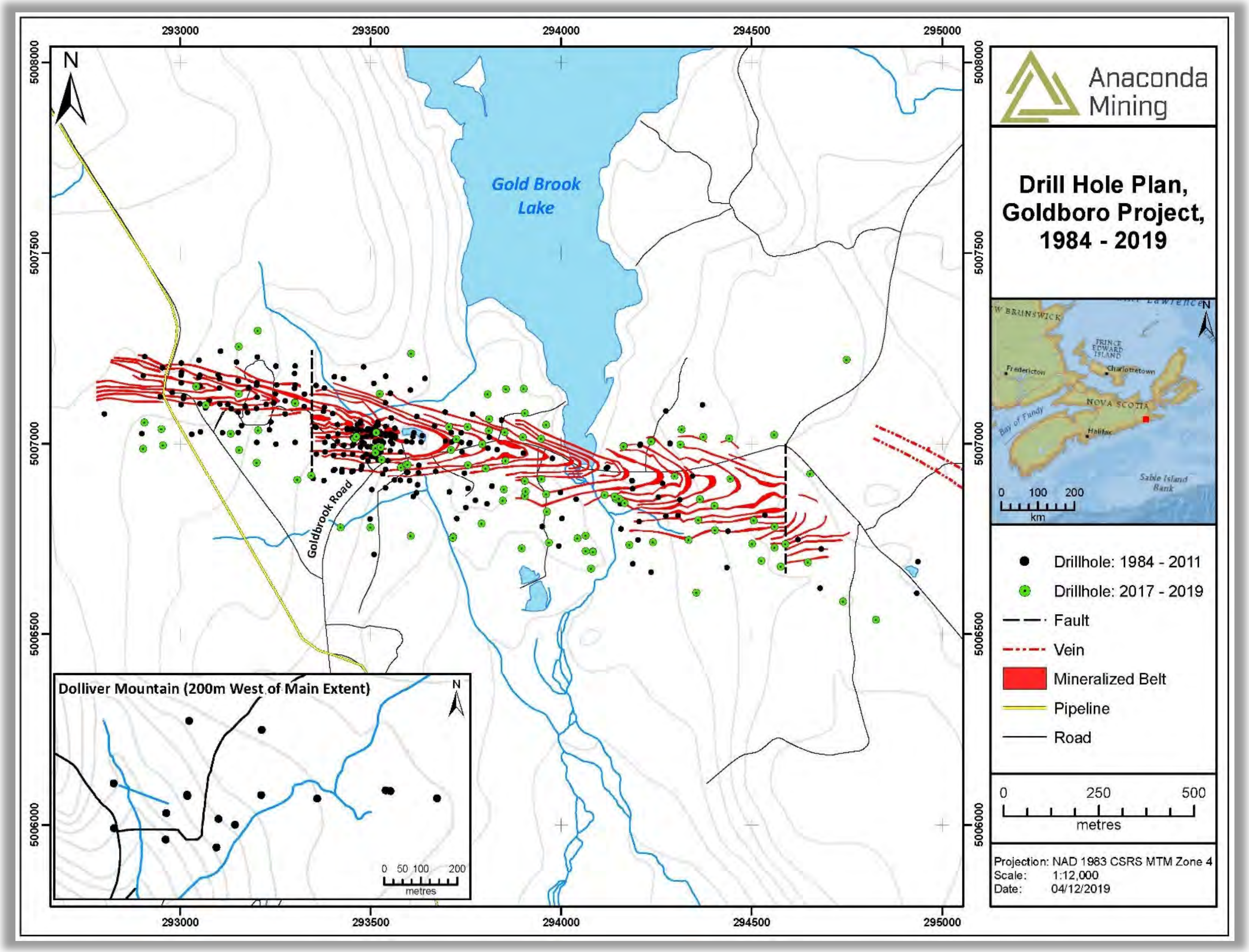
WSP considers all drilling programs carried out between 1984 and 2011, as outlined below, to meet requirements for use of associated data for mineral resource estimation purposes. The relationship between sample length and true thickness for the drilling programs described in this section vary due to the intersection angle of the drillholes and mineralization. In most cases, true thickness ranges from 60% to 85% of reported sample length. Drill program reporting does not highlight core loss as being a consistent and significant factor with respect to assessment of the Property, but triple-tube procedures were applied during the 1995 program to negate potential losses in check sampling carried out by Placer Dome.

In addition to the programs noted above, Anaconda completed a 107-hole (28,207.4 m) core drilling program consisting of 104 completed drillholes and three abandoned drillholes from 2017 to 2019 to expand and infill the Deposit as well as to obtain sample material for metallurgical testing.

Table 10.1 Diamond Drilling Program Summary for the 1984 to 2019 Period

Company	Year	Area	Metres	No. Holes	Series
Onitap	1984	Boston-Richardson	529	1	BR-84-01
Onitap	1985	West Goldbrook Mine	390	5	BR-85-01 to BR-87-04, incl. BR87-01A
Petromet Resources Ltd. & Greenstrike Gold Corp.	1987	Eastern part of the property	1,924	6	BR-87-01 to BR-87-05 and BR-87-05A
Onitap		Boston-Richardson belt and under & East Goldbrook property	11,621	33	BR-87-06 to BR87-38
Orex	1988	Upper Seal Harbour fold (8325E to 9100E) & West Goldbrook mine	10,822	41	BR-88-39 to BR-88-79
		Near decline	459	3	BR-88-80 to BR-88-82
	1988 to 1990	Underground 76-metre level (8637.5E to 8762.5E)	4,979	112	88U-01 to 88U-04 89U-05 to 88U-26 90U-27 to 90U-112
		West Goldbrook (8150E to 8600E)	2,811	26	BR-89-83 to BR-89-108
Minnova	1991	Twinned BR-88-48, BR-88-62, BR-88-60 and BR-87-35A	722	5	BR-91-109 to BR91-113
Orex	1993	Twinned BR-88-48/BR-91-109, BR-88-85, BR-88-62/BR-91-110 and BR-91-113	593	7	BR-93-114 to BR-93-117, BR-93-114B and BR-93-116B
Placer Dome	1995	Near ramp portal	1,263	7	BR-95-119 to BR-95-125
Orex	2005	Boston-Richardson Belt at 8675E	2,422	23	BR-05-001 to BR-05-023
	2008	West Goldbrook and East Goldbrook	12,065	45	BR-08-01 to BR-08-44 and BR-08-20A
Osisko	2010	Ramp, West Goldbrook and Dolliver Mountain	12,993	59	OSK10-01 to OSK10-59
Osisko	2011	East Goldbrook and West Goldbrook-Dolliver Mountain	2,375	10	OSK11-1 to OSK11-10
Anaconda	2017	Boston-Richardson and East Goldbrook	4,196.3	13	BR-17-01 to BR-17-13
Anaconda	2018	Boston-Richardson, East Goldbrook and West Goldbrook	18,277.3	61	BR-18-14 to BR-18-71
Anaconda	2019	East Goldbrook, Boston-Richardson, and West Goldbrook	5,733.8	33	BR-19-72 to BR-19-104

Figure 10.1 Drillhole Location for 1984-2019 Programs



10.2 1988 AND 1989 PROGRAMS

After acquiring the Property in 1988, Orex conducted a surface and underground exploration program that focused on the central portion of the Property. The surface drilling component of the program was comprised of 41 drillholes (BR88-39 to BR88-79, totalling 10,709 m) along the projected anticlinal axis of the Upper Seal Harbour fold, between local grid sections 8325E and 9100E. Ten of these holes were drilled in the former West Goldbrook mine area. At the end of the program, an additional three holes were drilled in the vicinity of the underground decline to investigate near-surface continuity of mineralized belts. NQ size core was recovered during the programs and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided by Narex Ore Search Consultants Inc. in 1988 and by Orex staff in 1989. Downhole orientation surveys were conducted under supervision of site technical staff.

For the 1989 exploration program, Orex was interested in developing a better understanding of the displacement and consequence of the east-trending fault that affects the axial zone of the anticline. One hundred and eight (108) underground drillholes (BR-89-U-05 to BR-89-U-112) for a total of 4,740 m were also drilled from the 76 m level on 11 local grid sections (8637.5E to 8762.5E) spaced 12.5 m apart. Each section drilled was comprised of a fan of 10 holes (when possible) spreading above the apex drift toward the 38 m level, with inclinations of 0°, +20°, +40°, +60°, and +75° to the north and south. This set-up allowed the holes to intersect the BR Gold System at right angles. In late 1989, another surface drilling campaign was completed that included 26 holes in the West Goldbrook area, between sections 8150E and 8600E, totalling 2,822 m of drilling (BR89-83 to BR89-108).

10.3 1991 AND 1993 PROGRAMS

In 1991, Orex optioned the Property to Minnova, whose initial objective was to twin drillholes BR88-48, BR88-62, BR88-60, and BR87-35A as BR91-109, BR91-110, BR91-111, and BR91-112, respectively. This program was designed to validate gold values returned for the earlier holes. CGL was used for initial analysis. Analytical results for the program were much lower than expected and this was attributed to poor core recovery of heavily broken mineralized zones resulting in the loss of free gold during the drilling process. A fifth hole (BR-91-113) was drilled using a 'triple-tube' system to increase core recovery and was collared on Section 8712.5E. Holes BR-88-62 and BR-91-110 were also drilled on this section. Core recovery was excellent for BR-91-113, and analytical results were within the range of gold grades encountered during the underground drilling campaign of 1989-1990 (*Labelle, 1991*).

In an attempt to resolve the question of whether high grades were present over large intervals in the central portion of the apex, Orex drilled four holes in the fall of 1993. Holes BR-93-114, BR-93-115, BR-93-116, and BR-93-117 were collared at the locations of Minnova test holes BR-88-48/BR-91-109, BR-88-85, BR-88-62/BR-91-110 and BR-91-113, respectively, and two more holes (BR-93-114A and BR-93-116A) were drilled as twins of BR-93-114 and BR-93-116 to ensure that high core recovery was attained over the total stratigraphic package. A total length of 593 m of drilling was carried out during this program. NQ sized core was collected with a triple-tube core barrel, high performance diamond bits were used, and controlled drilling conditions were established. Sludge samples (water return and drill cuttings) were also collected and assayed by the metallic screen method. Logan Drilling was contracted to provide drilling services, and core logging and sampling services were provided by Orex staff. Downhole orientation surveys were conducted under supervision of site technical staff.

The results from the sludge assays were much higher than the core assays and were considered unrepresentative. Six drillholes intersected similar rock formations to those observed in previous programs. The sample processing and gold determination methods applied were broadly similar to the CGL method used by Minnova in 1991. As described by Roy (1995) this began with crushing the 1 m samples to 100 % < 10 mesh, followed by splitting into four equal subsamples using a carrousel. One subsample was ground in a small ball mill for 24 hours in the presence of cyanide solution and gold levels in both liquid and solid materials were determined to provide a metallurgical balance and calculated head grade. The head grades were lower than expected, but similar to those obtained by Minnova. These results indicated that core recovery was not a significant contributing factor to gold grade discrepancies, however data showed that gold dissolution was less than 90% in more than three-quarters of the samples. In one instance where a large gold nugget was present, gold dissolution was less than 5%. The *Centre de Recherches Minérales* could not establish a methodology to ensure that greater than 85% gold was being taken into the cyanide solution and the program was therefore curtailed. Core from holes BR93-116, BR93-116B, and BR93-117 was not assayed completely due to cessation of the program (*Roy, 1995*).

10.4 1995 PROGRAMS

In 1995, after receiving the assay results from their sampling of the 4,000 t surface stockpile, Placer entered into an option to joint venture agreement with Orex under which a 65% interest in the Property could be earned. Placer became operator of the Project and subsequently completed a seven-hole core drilling program (holes BR-95-119 to BR-95-125) that totalled 1,263 m of drilling. Results of the program were reported by Gagnon et al. (1996). Triple-tube equipment recovering HQ size core was used and 1 m whole rock samples were split for comparison of gold grade determination processing techniques. Logan Drilling was contracted to provide drilling services, and core logging and sampling services were provided by Orex staff. Downhole orientation surveys were conducted under supervision of site technical staff.

Standard fire assay-metallics screen (CMS), GCNL, and CGL methods were used to determine gold grades for multiple sample splits generated from the 1995 core samples. Placer's analysis of results showed that the GCNL processing approach provided best representation of total contained gold levels. More specifically, screen metallics assaying of 45 samples from the BR Gold System returned gold grades between 0.02 g/t and 2.48 g/t, while GCNL results for corresponding splits returned gold grades between 0.07 g/t and 3.65 g/t. CGL results for additional splits of the same intervals returned gold grades between 0.03 g/t and 2.52 g/t, with best grades occurring in the south limb of the anticline Gagnon et al. (1996).

10.5 2005 PROGRAMS

The drilling program carried out by Orex in 2005 consisted of 23 diamond drillholes, designated BR05-001 to BR-05-023, from which HQ size core was recovered. A total of 2,355 m of drilling was completed in the program and this was concentrated in a 225 m wide zone centred on section 8675E of the BR Gold System. The drill pattern was designed to allow two separate shallow areas of mineralization to be similarly tested. Four holes were twinned from previous holes for comparative analysis and mineralized zones were assayed using fire assay and screen metallics methods with atomic absorption or gravimetric finish. At the end of the program, sample composites made by combining reject samples from multiple drillholes previously analyzed by screen metallics methods were sent for metallurgical testing.

HQ size core was recovered during the program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. William Shaw, P. Geo., of W. G. Shaw and Associates Ltd. of Antigonish, Nova Scotia (W. G. Shaw). Downhole orientation surveys were conducted under supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan and Associates Ltd. (C. J. MacLellan) using differential GPS methods after completion and were spotted using hand-held GPS units. Cement plugs were placed in holes immediately below the bedrock surface and casing was pulled. Holes were marked with a numbered wooden stake.

A mineral resource estimate partly supported by metallurgical sample results from the Ramp area of the deposit was carried out by P&E, after completion of the 2005 drilling program, results of which were presented earlier in report Section 6. P & E recommended completion of a two-phase diamond drill program for the purpose of:

1 Upgrading the Indicated resource within the Ramp area to the Measured category.
2 Upgrading a conceptual target of gold grade enhancement over a 1 km total strike length to the Indicated category.
3 Testing for extensions to the mineralized zones over a 2.5 km strike length.
4 Defining additional Inferred resources within the 2.5 km drilled deposit strike length.

10.6 2008 PROGRAM

Orex carried out an exploration drilling program in 2008 that consisted of 45 drillholes (BR-08-01 to BR-08-44, and BR-08-20A) comprising a total of 12,201.5 m and focused in the area between West Goldbrook and East Goldbrook. The purpose of this drill program was to infill as follow-up to the previous drill programs and to test potential extensions of gold mineralization to the east and west of the BR Gold System.

NQ size core was recovered during the 2008 program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. William Shaw, P. Geo., W.G. Shaw. Downhole orientation surveys were conducted under supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan using differential GPS methods after completion and were spotted using hand-held GPS units. Cement plugs were placed in holes immediately below the bedrock surface and casing was pulled. Holes were marked with a numbered wooden stake.

A mineral resource estimate, prepared in accordance with NI 43-101, was subsequently completed in 2009 by InnovExplo and results were reported by Gervais et al. (2009) with an effective date of July 29, 2009. Gervais et al. (2009) recommended a two-phase exploration program, the first phase of which consisted of a 35-drillhole program for a total of approximately 8,750 m to add additional Inferred resources. These drillholes were proposed for the area from local grid section 8100E for a distance of 1,500 m west along the strike of the anticlinal fold structure, including the West Goldbrook and Dolliver Mountain areas. The second phase of recommended exploration consisted of pilot milling and metallurgical testing of at least a 10,000 t bulk sample recovered from the resource model area. The design and execution of the second phase program was considered conditional on success of the first phase.

10.7 2010 PROGRAMS

10.7.1 CORE DRILLING

During the first half of 2010, Osisko completed 59 holes (OSK10-01 to OSK10-59) for a total of 12,989 m in the Ramp, West Goldbrook and Dolliver Mountain areas. NQ size core was recovered and drilling was carried out by Logan Drilling. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. Jean Lafleur, P. Geo., and geological staff from W.G. Shaw carried out field operations, including core logging and sampling activities. Mercator also assisted with core logging through provision of one geologist for part of the program. Downhole orientation surveys were conducted under supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan using differential GPS methods after completion and were spotted using hand-held GPS units. Cement plugs were placed in holes immediately below the bedrock surface and casing was pulled. Holes were marked with a numbered wooden stake. Drilling was carried out under separate programs identified as 2D, 2E, and 2F.

Angled holes were drilled in 2010 from both north and south sides of the approximately 100 m wide Boston-Richardson segment of the Upper Seal Harbour Anticline and terminated in the hinge area to prevent downdip drilling on the opposing limb. As such, each hole penetrated about 50% of the total width of the mineralized target corridor that occurs in the core of the anticline.

In January 2010, Osisko began the 2F program with the objective of replacing incomplete and/or non-compliant historic drill results with compliant data, primarily in the deeper zones of the Ramp area and extending westward towards West Goldbrook. Some of the historic drillholes in this area were not sampled in entirety, and others had been sampled on intervals considered too long for current interpretive purposes. The 2F program also included infill drilling which provides validation of the theory that a larger structural domain of gold mineralization is present on the Property, centered on the hinge zone of the Upper Seal Harbour Anticline. An important component of this assessment was to determine whether consistent and significant lower-grade gold mineralization was present in un-sampled intervals previously assumed to be barren of gold mineralization. Eighteen drillholes were completed for a total of 4,730 m of drilling in the 2F program. This was concentrated in a target area measuring approximately 250 m in east-west strike length along the anticlinal structure at elevations between 75 m and 250 m below surface.

The 2D drilling program was comprised of 25 holes for a total of 4,894 m and targeted a 350 m strike length along the anticline at depths between 25 m and 200 m below surface. These holes were drilled to replace historic holes having incomplete or otherwise deficient sampling and to assess possible downdip extensions to known mineralization. Resource extension drilling was also completed in the area between the west limit of past drilling and the Dolliver Mountain area.

The 2E program consisted of 16 holes for a total of 3,371.5 m that tested a 200 m strike length between West Goldbrook and the Dolliver Mountain area, as well as a 500 m strike length that included the Dolliver Mountain mine and potential westward extensions of associated geology. A 600 m gap in drilling between the two areas remained untested.

10.7.2 REVERSE CIRCULATION DRILLING

During the middle of 2010, reverse circulation (RC) drilling equipment was used to explore near-surface gold mineralized structures on the Property by recovering basal till and bedrock samples for gold assaying and whole-rock analysis. RC drilling was conducted by Archibald Drilling and Blasting of Truro, Nova Scotia at hole spacings of 25 m to 50 m along north-south fences oriented across the interpreted trend of bedrock gold mineralization. Mercator provided site supervision and reporting services for the program.

The program consisted of 64 RC drillholes (OSK10RC-130 to OSK10RC-194) for a total of 505 m. These were completed in the East Goldbrook, Boston-Richardson Ramp, and West Goldbrook areas. The program was successful in the identification of future diamond drilling target areas. Encouraging results from the under-explored East Goldbrook area included 10.85 g/t gold and 25.65 g/t gold in two bedrock samples, each of which measured 1 m in length. No estimates of sample true width were assigned to these samples. Selected highlights from the RC drilling campaign were disclosed in a November 18, 2010 news release by Orex and significant results are tabulated in Table 10.2.

Table 10.2 Significant 2010 Orex RC Drilling Results

Hole No.	Location	From (m)	To (m)	Width (m)	Type	Assay (Au ppm)
OSK10RC-135	East Goldbrook	7	8	1	Bedrock	1.38
OSK10RC-137	East Goldbrook	4	5	1	Bedrock	10.85
OSK10RC-147	East Goldbrook	5	6	1	Bedrock	25.65
OSK10RC-186	Ramp Area	3	4	1	Overburden	1.47
OSK10RC-194	West Goldbrook	4	5	1	Overburden	1.17
OSK10RC-194	West Goldbrook	6	7	1	Bedrock	4.23
OSK10RC-194	West Goldbrook	8	9	1	Bedrock	2.80

(Modified after November 18, 2010 Orex press release)

Overburden samples were collected at 1 m downhole intervals above the bedrock-overburden interface, and bedrock chip samples were collected at 1 m downhole intervals, beginning at the interface and generally extending to a depth of 1 m to 3 m into rock. Bedrock drilling was advanced to a depth of 12 m in selected holes. All samples collected during the program were delivered by Mercator or Armour Transport to the Minerals Engineering Centre (MEC) at Dalhousie University in Halifax, Nova Scotia, where processing and analysis were carried out. RC drilling program results are pertinent to future property exploration but are not used in the resource estimation program described in this report.

10.8 2011 PROGRAM

The 2011 core drilling campaign by Osisko focused on two areas within the Property, the first being a 750 m interval from East Goldbrook to the east property boundary, with drillholes on three local grid sections (9650E to 10150E) spaced 250 m apart; and the second being a 600 m interval between West Goldbrook and Dolliver Mountain, where drillholes on three sections were spaced 200 m apart in an area of no historic drilling. NQ-size core was recovered during the program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. Bruce Mitchell, P.Geo., and geological staff from W.G. Shaw carried out field operations, including core logging and sampling activities.

A total of six drillholes (OSK11-01 to OSK11-06) were completed at East Goldbrook for a total of 2,375.4 m. Results for these define narrow, high-grade quartz veins, often with visible gold, that are associated with wider zones of lower-grade gold values in argillite These occur along an interval approximately 500 m in strike length that extends to the eastern property boundary. The mineralization appears to be plunging shallowly to the east, and additional drilling in the East Goldbrook area is required to further delineate the mineralized zones.

A total of four drillholes (OSK11-07 to OSK11-10) were completed in the gap area between West Goldbrook and Dolliver Mountain for a total of 765.4 m. Three out of the four holes intersected occasional, narrow, high-grade quartz veins but wall rock was not well mineralized. One drillhole (OSK11-07) was abandoned before reaching its target depth due to drilling problems.

10.9 2017 SUMMER METALLURGICAL DRILLING PROGRAM

During the summer 2017, Anaconda completed a five-hole (BR-17-01 to 05), 643 m, diamond drill program that tested the BR and EG Gold Systems (Figure 10.2; Table 10.3). Drilling of the five holes was completed in order to collect samples for metallurgical test work on the Goldboro mineralization, with each of the completed holes twinning a historic drillhole. HQ-size core was recovered, and drilling was carried out by Logan Drilling using conventional wireline drilling equipment. Core logging and sampling were carried out by Anaconda employees under the project supervision of Mr. Steve Barrett, P.Geo. and Mr. David A. Copeland, P.Geo., both employees of Anaconda. Downhole orientation surveys were conducted under supervision of site technical staff using a Reflex downhole instrument at nominal 30 m intervals. Drill collars were surveyed using a differential GPS by Mr. Matt Walsh, also an employee of Anaconda.

Drill core samples were collected systematically down each hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 to 1.0 m depending on the nature and width of veining and mineralization, while trying to best honour geological contacts. Samples were collected of quarter-sawn drill core and shipped to Eastern Analytical Limited in Springdale, Newfoundland and Labrador (Eastern Analytical) for analysis via standard 30 g fire assay with Atomic Absorption (AA) finish. Samples were also analyzed at Eastern Analytical via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold and all samples for 34-element ICP analysis. Half of the sampled drill core assaying greater than 0.5 g/t gold and totalling approximately 25 kg in mass was shipped to Thibault in Fredericton, New Brunswick for metallurgical test work, while the remaining quarter of the core was retained as an archive split. Preliminary test work produced a total gold recovery of 91.9%. Metallurgical aspects of the program are discussed in report Section 13.

Results from the twinned metallurgical drilling confirmed the accuracy of historic drilling. Multiple occurrences of visible gold and assays with high gold grades are present in all five initial 2017 drillholes. Selected uncapped assay highlights for these holes were originally reported in an Anaconda press release date July 27, 2017 and subsequent screen metallics assay results for selected intervals were released in the PEA with effective date January 17, 2018.

Figure 10.2 2017 - 2018 Anaconda Drilling Plan

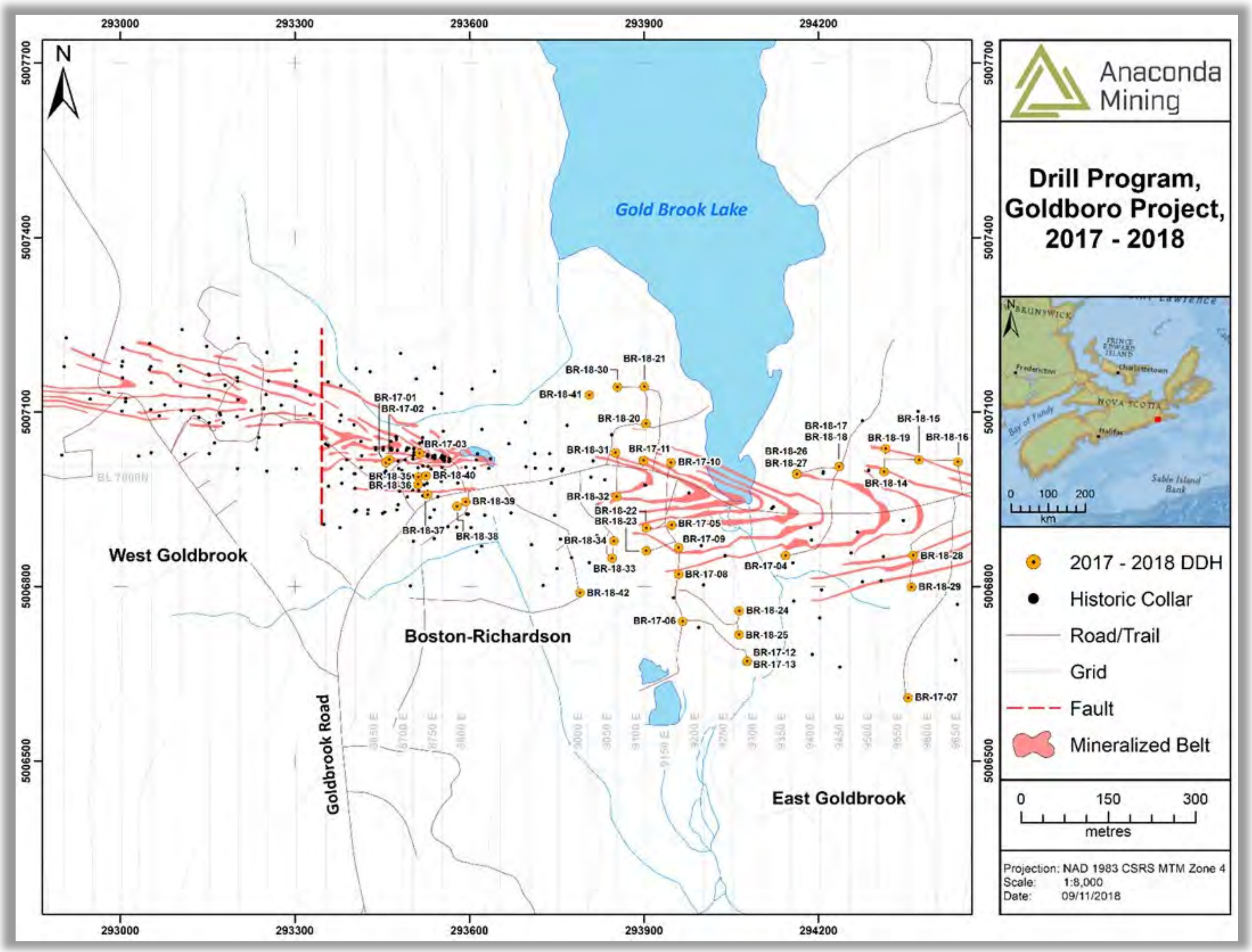


Table 10.3 Goldboro 2017 - 2018 Drill Collar Locations

Drillhole	Easting (m)	Northing (m)	Elevation RL (m)	Depth (m)	Azimuth deg)	Inclination (deg)
BR-17-01	293457.1	5007013.0	5004.5	73.0	360	-45.0
BR-17-02	293462.9	5007019.0	5004.2	197.0	360	-90.0
BR-17-03	293514.7	5007029.0	5003.5	72.0	360	-90.0
BR-17-04	294143.6	5006853.0	4992.0	151.0	360	-45.0
BR-17-05	293947.6	5006906.0	5000.4	150.0	360	-50.0
BR-17-06	293966.7	5006740.0	4994.4	549.0	352	-67.0
BR-17-07	294354.1	5006608.0	4992.7	624.0	355	-67.0
BR-17-08	293959.8	5006821.0	4995.5	401.7	360	-65.5
BR-17-09	293959.9	5006867.0	4996.1	350.3	360	-67.0
BR-17-10	293946.5	5007013.0	4998.6	149.0	180	-60.0
BR-17-11	293899.4	5007017.0	5000.9	149.0	180	-60.0
BR-17-12	294079.0	5006673.0	5000.0	677.0	360	-65.0
BR-17-13	294079.0	5006673.0	5000.0	653.3	360	-58.0
BR-18-14	294312.8	5006997.0	4999.3	151.0	180	-70.0
BR-18-15	294372.7	5007018.0	5002.6	225.0	180	-65.0
BR-18-16	294439.9	5007014.0	5003.0	325.0	180	-53.0
BR-18-17	294235.7	5007006.0	4997.0	175.0	180	-55.0
BR-18-18	294235.6	5007006.0	4997.0	273.0	180	-75.0
BR-18-19	294315.5	5007037.0	5000.3	242.0	180	-70.0
BR-18-20	293903.5	5007080.0	4999.5	287.0	180	-68.0
BR-18-21	293900.9	5007144.0	4996.5	472.6	180	-70.0
BR-18-22	293904.0	5006900.0	5003.3	278.0	356	-74.0
BR-18-23	293904.0	5006861.0	4995.6	434.0	358	-75.0
BR-18-23A	293904.0	5006861.3	4995.6	29.0	358	-74.0
BR-18-24	294063.9	5006758.0	4994.6	256.8	360	-50.0
BR-18-25	294063.5	5006717.0	4996.8	313.0	360	-55.0
BR-18-26	294162.7	5006994.0	4996.7	232.0	180	-68.0
BR-18-27	294162.8	5006993.0	4996.7	121.0	180	-50.0
BR-18-28	294363.0	5006853.0	4998.7	245.0	360	-78.5
BR-18-29	294359.7	5006799.0	4998.6	445.0	360	-74.0
BR-18-30	293854.5	5007143.0	4998.6	596.0	180	-72.0
BR-18-31	293851.2	5007030.0	5000.2	98.7	180	-72.0
BR-18-32	293853.4	5006954.0	5002.0	94.5	360	-75.0
BR-18-33	293845.6	5006848.0	4998.0	509.5	360	-75.0
BR-18-34	293848.6	5006878.0	4998.8	426.0	360	-75.0
BR-18-35	293512.2	5006988.0	5002.2	49.0	360	-48.0
BR-18-36	293511.9	5006976.0	5002.3	165.0	360	-65.0
BR-18-37	293528.4	5006958.0	5003.3	147.0	360	-53.0
BR-18-38	293579.0	5006938.0	5001.6	147.0	360	-50.0
BR-18-39	293593.7	5006946.0	5001.2	137.0	360	-50.0

(table continues on next page)

Drillhole	Easting (m)	Northing (m)	Elevation RL (m)	Depth (m)	Azimuth deg)	Inclination (deg)
BR-18-39A	293593.7	5006945.6	5001.2	96.3	360	-60.0
BR-18-40	293525.8	5006990.0	5002.2	50.0	360	-46.0
BR-18-41	293806.2	5007129.0	5000.5	603.2	180	-75.0
BR-18-42	293790.4	5006789.0	4994.2	536.0	354	-68.5

MTM Zone 4 NAD 83 Coordinate System; mine elevation datum (4,940 m=0 msl)

In 2017 the BR Gold System was host to seventeen tightly-stacked, high-grade, gold-bearing vein zones. Holes BR-17-01 to BR-17-03 are located within the core of the BR Gold System and intersected up to 7 of the 17 vein zones, which are characterized by thick gold-bearing quartz veins and thin vein arrays within highly altered argillite, separated from the neighbouring vein zones by un-mineralized greywacke as predicted by recent geological modelling. The high-grade assays returned from these drillholes are of a tenor consistent with previous drill programs on the Deposit as well as with recent geological and resource modelling. Program results are interpreted at this time as confirming the validity of the geological and resource models.

The EG Gold System was characterized by 11 stacked vein zones and is generally drilled at a broader spacing (~50 m) than Boston-Richardson. Hole BR-17-05 intersected 5 of the 11 East Goldbrook vein zones including vein Zones 3 and 4, which appear to extend 40 and 100 m, respectively, farther to the west beyond what has been currently modelled. Hole BR-17-04 within the EG Gold System contains the highest-grade intercepts returned from the 2017 program. A tabulation of drill collar, orientation, and length information for the 2017 program is included in previous Table 10.3.

Anaconda obtained a Crown Land access permit from the provincial government that covered completion of this drilling program. Terms and conditions of such permits cover various site considerations, including timber cutting, habitat disturbance, and requirements associated with reclamation of site impacts after completion of drilling activities. WSP understands that 2017 activities were carried out in compliance with terms of the associated access permit.

10.10 2017 – 2018 EXPLORATION AND INFILL DRILLING

In addition to the metallurgical drilling program described above, Anaconda completed 39 additional core holes on the property (BR-17-06 to BR-17-13 and BR-18-14 to BR-18-42) totaling 11,712.9 m of drilling. Hole locations were selected to provide local infill and mineralized zone extension information and drilling services were provided by Logan Drilling. NQ size core was recovered using conventional wireline drilling equipment. Core logging and sampling, downhole surveying, and collar location programs were completed in the same manner as used earlier in the year and the program under the project supervision of Mr. Steve Barrett, P.Geo., Ms. Tanya Tettelaar, P.Geo., and Mr. David A. Copeland, P.Geo., employees of Anaconda. Currently, casing remains for the 2017-2018 holes, however the drillholes have been plugged with concrete to a depth of 3 m below the bedrock surface, and casing cut flush to ground surface. Hole locations for this program appear on previous Figure 10.2. A tabulation of drill collar coordinates, orientation, and length information for the 2017 and 2018 exploration drillholes is included in previous Table 10.3.

Drill core samples were collected systematically down each hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 to 1.0 m depending on the nature and width of veining and mineralization while trying to best honour geological contacts. Samples were collected of half-sawn drill core and shipped to Eastern Analytical Limited in Springdale, Newfoundland and Labrador (Eastern Analytical) for analysis via standard 30 g fire assay with AA finish. Samples were also analyzed at Eastern Analytical via screen metallic using the entire sample for samples assaying greater than 0.5 g/t gold and all samples for 34-element ICP analysis. Check assays on the fine fraction from the metallic screen fire assay sample were analyzed for gold at ALS Minerals in North Vancouver, BC.

The 2017-2018 drill program focused on infilling areas of Inferred mineral resources and expanding the Deposit. Drilling has focussed on testing the down plunge, down-dip and along strike extension of the BR Gold System and the EG Gold System. Drilling also focussed on infilling under-drilled areas of the Deposit in order to upgrade mineral resources from the Inferred to Indicated category with focus on sections 9050E, 9100E, 9150E, 9250E, 9350E, 9450E, 9500E, 9550E and 9650E (see plan map Figure 10.2).

BR-18-35 to BR-18-40 on local grid sections 8700E to 8800E were designed to test the BR Gold System in an area of proposed underground bulk sampling. This drilling successfully intersected mineralization modelled from historic drilling and was used to refine selected locations for a 10,000 tonne bulk sampling program.

Infill drilling of the BR Gold System from 9000E to 9250E intersected high grade intervals in predicted previously modelled mineralized belts as well as intersecting new mineralized belts below historic drilling, extending mineralization at depth by up to 150 m over a strike length of 250 m. This drilling was used to refine the geological interpretation suggesting that the anticlinal fold axis was slightly inclined to the north.

Infill drilling of the EG Gold System from 9000E to 9500E typically intersected zones of gold mineralization, both low- and high-grade, but showing overall continuity between mineralized belts. Additionally, drilling from 9500E to 9650E intersected three new near-surface mineralized belts in the EG Gold System.

Drilling completed by Anaconda in 2017 and up to June 2018 was included as part of the previous NI 43-101 Mineral Resource Estimate, effective date July 19, 2018, prepared by WSP under supervision by Todd McCracken, P. Geo. Select assay highlights from this program are listed in Table 10.4.

Table 10.4 Goldboro 2017 - 2018 Assay Highlights

Drillhole	From (m)	To (m)	Interval (m)	Au (g/t)	Gold System	Visible Gold	Section
BR-18-36	106.0	121.0	15.0	2.73	BR	vg	8700E
including	106.5	107.5	1.0	9.30	BR		8700E
BR-18-37	77.5	89.0	11.5	21.05	BR		8725E
BR-18-37	104.0	105.5	1.5	23.74	BR		8725E
BR-18-40	38.0	47.0	9.0	3.69	BR	vg	8725E
BR-18-39	93.5	101.0	7.5	17.41	BR		8800E
BR-18-41	378.0	379.0	1.0	63.88	BR		9000E
BR-18-42	64.5	65.0	0.5	77.69	EG	vg	9000E
BR-18-42	472.0	475.7	3.7	6.05	BR	vg	9000E
including	475.0	475.7	0.7	28.12	BR	vg	9000E
BR-18-30	506.1	531.6	25.5	2.21	BR	vg	9050E
including	512.1	515.3	3.2	12.39	BR	vg	9050E
BR-18-34	177.0	178.0	1.0	24.49	EG	vg	9050E
BR-18-33	384.7	388.3	3.6	4.82	BR		9050E
including	385.7	386.8	1.1	9.90	BR		9050E
BR-18-34	270.5	278.0	7.5	3.00	BR		9050E
BR-18-22	201.0	214.5	13.5	11.27	BR		9100E
including	209.5	212.0	2.5	44.33	BR		9100E
BR-18-23	324.5	345.0	20.5	4.13	BR	vg	9100E
including	324.5	332.0	7.5	9.93	BR		9100E
BR-18-22	223.0	229.1	6.1	10.55	BR		9100E
including	224.5	227.6	3.1	18.78	BR	vg	9100E
BR-18-22	116.0	125.6	9.6	5.10	EG	vg	9100E
including	116.0	117.5	1.5	25.82	EG	vg	9100E
BR-18-23	310.5	317.0	6.5	7.22	BR		9100E
including	311.0	313.0	2.0	16.00	BR		9100E
BR-17-06	272.7	273.2	0.5	59.97	EG		9150E
BR-17-06	389.9	393.7	3.8	24.34	BR		9150E
including	391.5	392.5	1.0	86.48	BR		9150E
BR-17-08	293.8	297.0	3.2	9.12	BR		9150E
BR-17-09	82.0	85.5	3.5	34.70	EG		9150E
BR-17-10	69.6	70.1	0.5	17.68	EG	vg	9150E
BR-17-12	342.8	343.3	0.5	49.61	EG		9250E
BR-17-13	321.0	322.0	1.0	12.29	EG		9250E
BR-18-25	145.0	145.5	0.5	752.54	EG	vg	9250E
BR-18-25	39.0	40.0	1.0	17.00	EG		9250E
BR-18-25	132.5	133.5	1.0	56.67	EG		9250E
BR-18-24	84.5	87.0	2.5	6.55	EG		9250E
BR-18-26	108.5	110.0	1.5	62.01	EG	vg	9350E
BR-18-26	27.8	29.5	1.7	12.66	EG	vg	9350E
BR-17-04	33.1	35.7	2.6	151.42	EG	vg	9350E
BR-18-17	6.2	7.2	1.0	31.04	EG		9450E
BR-18-18	130.6	132.6	2.0	12.87	EG	vg	9450E
BR-18-18	62.0	63.0	1.0	25.31	EG		9450E
BR-18-28	21.5	24.0	2.5	23.24	EG	vg	9550E
BR-18-29	193.5	198.0	4.5	7.12	EG	vg	9550E
BR-18-15	76.6	77.0	0.4	252.76	EG	vg	9550E

10.11 2018 – 2019 EXPLORATION AND INFILL DRILLING

From July 2018 to August 2019 Anaconda completed 63 drillholes (BR-18-43 to BR-18-71 and BR-19-72 to BR-19-104) on the property totaling 15,851.5 m of drilling. Core size was typically NQ diameter except BR-18-68 to BR-18-71, which were drilled as HQ-size core to provide material for metallurgical studies. The drilling was performed by Logan Drilling and Geotechnical Limited of Stewiacke, NS, using a Duralite 1000 skid-mounted drill rig. Proposed collar locations were spotted by Anaconda employed geologists using a handheld Garmin 60 GPS. Downhole orientation surveys were conducted by Logan drilling staff using a Reflex downhole instrument at nominal 30 m intervals under the supervision of the site geologist. Drill collars were surveyed using a differential GPS by Anaconda employee Alison Cox or by consultant surveyor Carl Hovey of Terrane Geoscience Inc., Halifax, NS. Drill casing for drillholes BR-18-43 to BR-18-71, BR-19-72 to BR-19-76, BR-19-86 to BR-19-88, and BR-19-91 to BR-19-104 have remained in place, however the casing has been cut flush to ground surface. Drill casing was removed from drillholes BR-19-77 to BR-19-85 and BR-19-89 and -90. All above mentioned drillholes have been plugged with bentonite at 3 m below bedrock surface, filled with cement to ground surface, and any remaining casing was capped and buried. Drillhole collar locations are shown on the plan map, Figure 10.3. A tabulation of drill collar coordinates, orientation, and length information for the 2018 and 2019 exploration drillholes is included in Table 10.5.

Core logging and sampling was carried out by Anaconda employees under project supervision by Tanya Tettelaar, P.Geo. Core logging and geotechnical data were input into an Access database via Core Logger. Drill core sampling methods and analytical laboratories remain the same as the 2017-2018 program with the exception that ICP analyses ceased after drillhole BR-18-62.

The 2018-2019 drill program focused on infilling areas of Inferred mineral resources and expanding the Deposit along strike and at depth along the host fold structure. Infill drilling targeted under-drilled areas of the deposit in order to upgrade mineral resources from the Inferred to Indicated category within the WG and EG Gold Systems. Drilling also focussed on testing the down-plunge, down-dip and at depth expansion of the WG, BR and EG Gold Systems (see plan map Figure 10.3).

Drilling completed by Anaconda from July 2018 to June 2019 consisting of 57 drillholes totaling 15,111.5 m of drilling (BR-18-43 to BR-18-71, BR-19-72 to BR-19-97, and BR-19-99) have been included as part of the current NI 43-101 Mineral Resource Estimate. Results of this drill program are presented below summarized by key areas throughout the Deposit proceeding from west to east.

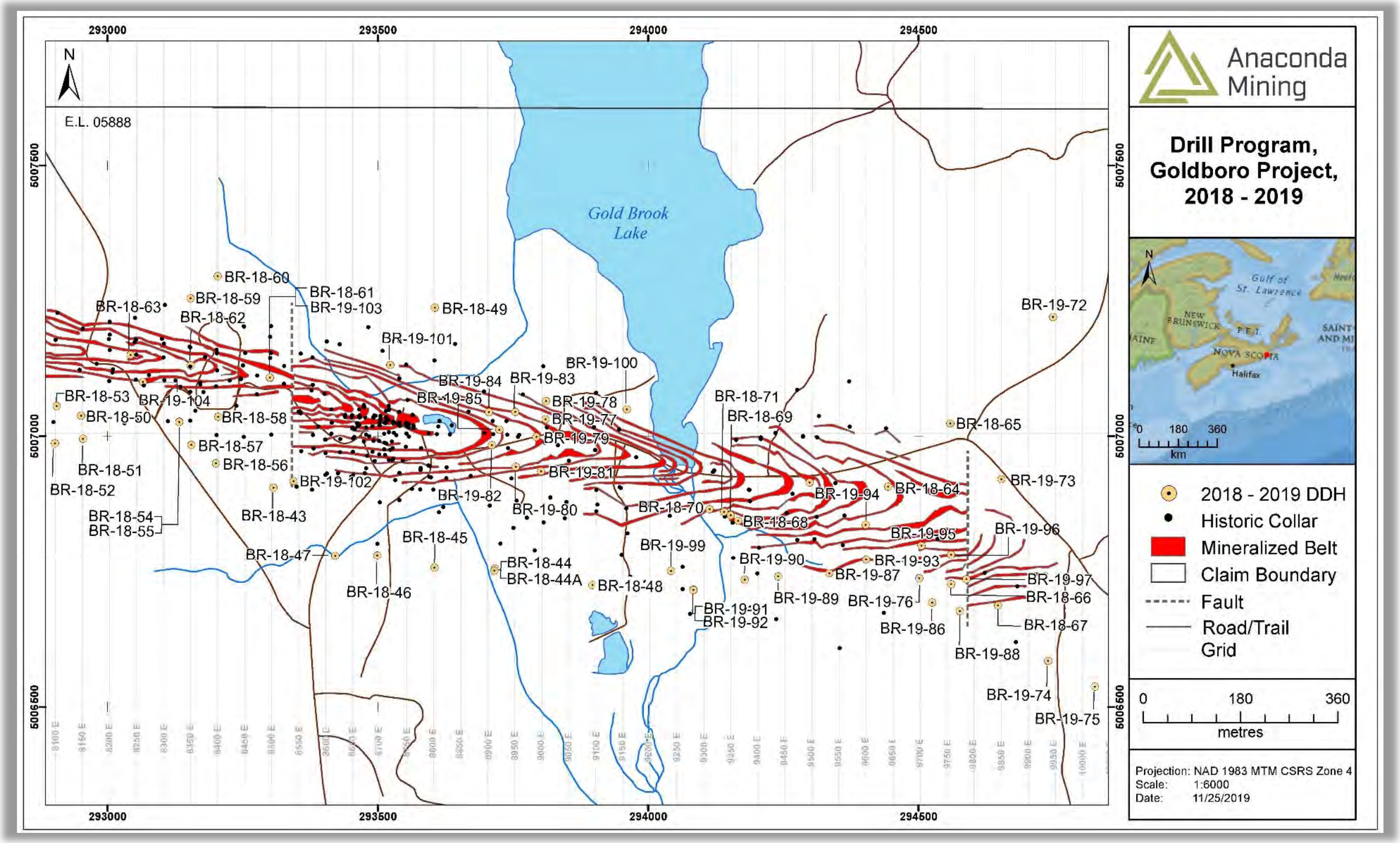
Table 10.5 Goldboro 2018 -2019 Drill Collar Locations

Drillhole	Easting (m)	Northing (m)	Mine Elevation (m)	Depth (m)	Azimuth (deg)	Inclination (deg)
BR-18-43	293306.4	5006904.8	5009.4	394.0	357	-55
BR-18-44	293716.4	5006755.0	4995.6	515.0	355	-60
BR-18-44A	293715.0	5006751.0	4998.0	14.0	355	-60
BR-18-45	293604.6	5006756.5	4999.6	515.0	357	-60
BR-18-46	293498.8	5006779.1	5002.0	494.0	360	-60
BR-18-47	293420.8	5006778.6	5002.8	506.0	357	-60
BR-18-48	293896.5	5006724.6	4992.6	596.0	357	-65
BR-18-49	293605.0	5007236.9	5005.9	611.0	175	-62
BR-18-50	292950.9	5007037.7	5016.9	260.0	360	-61
BR-18-51	292954.5	5006995.2	5016.5	332.0	360	-61
BR-18-52	292902.5	5006986.5	5017.1	368.0	360	-61
BR-18-53	292905.6	5007055.5	5017.4	246.0	360	-58
BR-18-54	293132.2	5007026.4	5015.3	149.0	360	-70
BR-18-55	293132.2	5007025.8	5015.2	143.0	360	-50
BR-18-56	293200.4	5006949.8	5012.1	347.0	360	-60
BR-18-57	293154.7	5006983.6	5013.6	362.0	360	-65
BR-18-58	293204.3	5007035.6	5013.3	182.0	360	-60
BR-18-59	293153.3	5007254.3	5016.1	350.0	180	-65
BR-18-60	293203.3	5007295.9	5013.8	449.0	177	-58
BR-18-61	293300.2	5007107.6	5011.5	410.0	180	-88
BR-18-62	293153.4	5007130.6	5017.7	407.0	180	-88
BR-18-63	293042.3	5007149.7	5020.2	467.0	180	-88
BR-18-64	294443.7	5006906.7	5001.5	485.0	180	-85
BR-18-65	294558.5	5007023.6	5006.6	338.0	180	-55
BR-18-66	294560.0	5006726.1	5004.3	304.4	360	-55
BR-18-67	294646.5	5006687.3	5006.0	328.3	360	-55
BR-18-68	294166.1	5006843.9	4991.7	137.0	360	-55
BR-18-69	294152.4	5006853.9	4992.0	167.0	360	-50
BR-18-70	294113.1	5006865.1	4992.2	107.0	360	-46
BR-18-71	294139.9	5006860.2	4992.2	134.0	360	-50
BR-19-72	294748.2	5007219.6	5006.1	168.0	180	-45
BR-19-73	294652.7	5006920.4	5010.6	225.1	180	-65
BR-19-74	294739.2	5006585.0	5011.9	329.0	360	-55
BR-19-75	294825.9	5006537.3	5006.8	359.0	360	-55
BR-19-76	294500.8	5006736.6	5001.3	296.0	360	-55
BR-19-77	293809.4	5007034.2	4999.3	83.0	180	-75
BR-19-78	293810.5	5007064.2	4998.8	120.0	180	-75
BR-19-79	293792.9	5006998.5	5000.4	62.0	270	-90
BR-19-80	293754.6	5006943.4	5001.4	75.5	360	-70
BR-19-81	293801.5	5006935.0	5002.2	90.4	360	-72
BR-19-82	293710.4	5006983.2	5001.8	37.0	270	-90
BR-19-83	293753.7	5007044.8	4999.9	66.0	180	-70
BR-19-84	293704.7	5007044.1	5000.3	38.0	180	-70
BR-19-85	293724.4	5007012.0	5003.3	30.8	270	-90
BR-19-86	294525.0	5006692.1	5003.1	176.0	360	-53
BR-19-87	294334.6	5006746.4	4994.9	332.0	360	-65
BR-19-88	294576.0	5006676.6	5004.5	194.0	360	-55
BR-19-89	294240.3	5006740.4	4992.7	326.0	360	-60
BR-19-90	294178.0	5006734.0	4991.7	341.0	360	-58
BR-19-91	294083.2	5006716.0	4995.7	347.0	18	-68
BR-19-92	294083.0	5006715.4	4995.6	392.0	18	-45

(table continues on next page)

Drillhole	Easting (m)	Northing (m)	Mine Elevation (m)	Depth (m)	Azimuth (deg)	Inclination (deg)
BR-19-93	294402.9	5006771.5	4998.1	125.0	360	-55
BR-19-94	294402.2	5006836.0	5002.1	86.0	360	-60
BR-19-95	294505.2	5006797.7	5006.2	95.0	360	-55
BR-19-96	294560.2	5006781.1	5007.1	128.0	360	-55
BR-19-97	294588.3	5006735.4	5004.3	149.0	360	-55
BR-19-98	294298.0	5006914.5	4997.5	101.0	270	-88
BR-19-99	294042.1	5006750.7	4994.7	323.0	360	-65
BR-19-100	293959.9	5007049.2	4996.2	170.0	145	-55
BR-19-101	293522.6	5007131.0	5004.9	140.0	180	-47
BR-19-102	293343.7	5006917.0	5007.7	143.0	360	-50
BR-19-103	293301.7	5007106.1	5011.9	101.0	90	-50
BR-19-104	293067.4	5007101.2	5022.6	85.0	360	-45

Figure 10.3 2018 - 2019 Anaconda Drilling Plan



10.11.1 WEST GOLDBROOK INFILL AND EXPANSION

Fifteen drillholes, BR-18-43 and BR-18-50 to BR-18-63 were designed to infill portions of the WG Gold System over 400 m of existing strike length. Vertically drilled holes BR-18-61 to BR-18-63 were also intended to test the system along the fold hinge below previously modelled mineralization. The infill drilling intersected mineralized belts in areas of known Inferred mineral resources and demonstrated continuity of mineralization, providing the requisite geological data to potentially convert those Inferred resources to the Indicated category. Expansion drilling intersected the host fold structure, alteration and mineralization characteristic of the Deposit to a depth of 450 m, demonstrating that the Deposit continues up to 175 m below the previously modeled WG Gold System. Additionally, drillholes BR-18-43, BR-18-51, BR-18-52, BR-18-56 and BR-18-57 intersected 45 to 50 m of massive greywacke. The atypical bedding thickness and lithological and alteration characteristics are comparable to the Marker Horizon, which stratigraphically overlies the BR Gold System. This evidence confirmed the historic hypothesis that the WG Gold System is the fault-offset western continuation of the BR Gold System. Select highlights from the WG portion of the drill program include:

— 78.07 g/t gold over 1.1 m (196.7 to 197.8 m) in hole BR-18-63;

— 32.42 g/t gold over 2.6 m (300.3 to 302.9 m) including 201.68 g/t gold over 0.4 m in hole BR-18-59;

— 24.06 g/t gold over 2.0 m (138.0 to 140.0 m) including 55.58 g/t gold over 0.5 m in hole BR-18-61;

— 20.02 g/t gold over 2.0 m (226.5 to 228.5 m) including 78.29 g/t gold over 0.5 m in hole BR-18-56;

— 25.45 g/t gold over 1.5 m (199.3 to 200.8 m) including 46.54 g/t gold over 0.8 m in hole BR-18-59; and

— 11.15 g/t gold over 1.0 m (179.0 to 180.0 m) in hole BR-18-51.

10.11.2 BOSTON-RICHARDSON EXPANSION AT DEPTH

Six drillholes, BR-18-44 to BR-18-49 between local grid sections 8600E to 9100E, targeted a previously untested deeper area of the BR Gold System over 350 m of strike and to depths of 525 m. Drilling expanded two mineralized belts an additional 200 m along strike and expanded five other belts over 350 m along strike. Nineteen occurrences of visible gold were observed in the six drillholes and the character of the mineralization in those holes is consistent with results seen throughout the BR Gold System to date. The BR Gold System remains open for further expansion at depth and down plunge. Select highlights from this drilling include:

— 8.79 g/t gold over 8.0 m (483.0 to 491.0 m) in hole BR-18-44, including 64.40 g/t gold over 0.8 m;

— 51.89 g/t gold over 1.0 m (224.5 to 225.5 m) in hole BR-18-46;

— 5.15 g/t gold over 4.0 m (390.9 to 394.9 m) including 10.08 g/t gold over 1.5 m in hole BR-18-47;

— 21.06 g/t gold over 1.0 m (200.1 to 201.1 m) in hole BR-18-48; and

— 6.39 g/t gold over 2.0 m (457.2 to 459.2 m) and 3.35 g/t gold over 4.5 m (539.0 to 543.5 m) in hole BR-18-49, including 25.68 g/t gold over 0.4 m.

10.11.3 EAST GOLDBROOK SHALLOW INFILL

Drilling of the near surface mineralization potential of the EG Gold System (BR-19-77 to BR-19-85) in proximity to the optimized open pit shell successfully intersected gold mineralization in all drill holes except BR-19-82 and BR-19-85. Geological interpretation of these zones demonstrates continuity of mineralization up-plunge in five mineralized belts, previously not modelled in the 2018 mineral resource west of drillholes BR-18-31 and BR-18-32, over a strike-length of 100 m and depths less than 70 m. Figure 10.4 illustrates the geological interpretation of these new belts in the EG Gold System on local grid section 9000E. Highlights from this drilling include:

— 1.89 g/t gold over 3.0 m (20.9 to 23.9 m) in hole BR-19- 83;

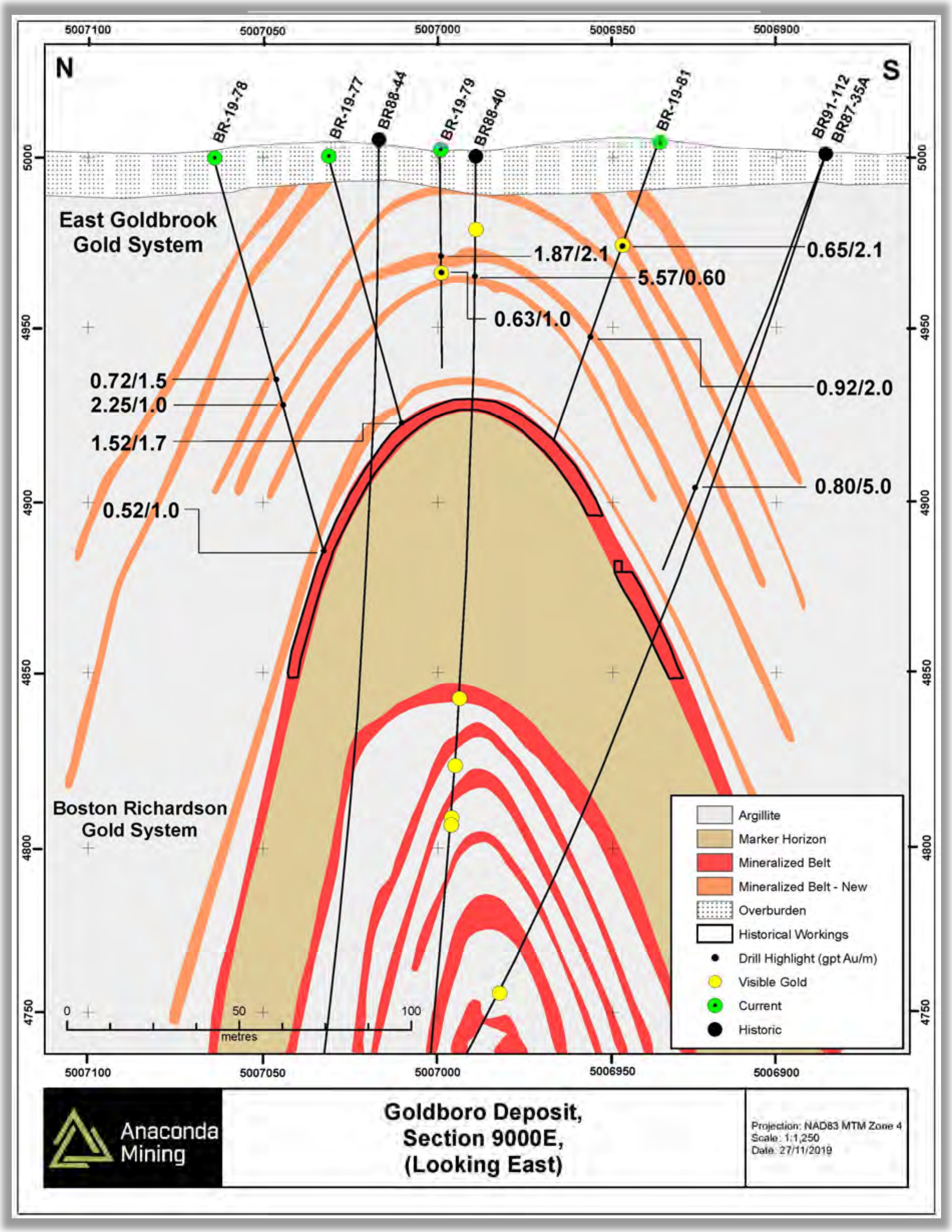
— 1.20 g/t gold over 2.2 m (13.4 to 15.6 m) in hole BR-19-84;

— 1.87 g/t gold over 2.1 m (28.9 to 31.0 m) in hole BR-19-79;

— 2.43 g/t gold over 2.0 m (46.0 to 48.0 m) in hole BR-19-80; and

— 1.52 g/t gold over 1.7 m (79.3 to 81.0 m) in hole BR-19-77.

Figure 10.4 Section 9000E - Goldboro Deposit



10.11.4 EAST GOLDBROOK METALLURGICAL DRILLING

Four drillholes, BR-18-68 to BR-18-71, intersected near-surface, previously modelled mineralization of the EG Gold System over a strike length of 50 m and 100 m vertical depth. Five targeted mineralized belts were intersected as predicted, providing confidence it the geological model. These holes were also used to provide representative mineralized material for a metallurgical test program to support an ongoing feasibility study. The remainder of the half-cut core from select core samples grading higher than 0.5 g/t gold from screen metallic results were combined to provide low-, medium-, high-, and very high-grade composites. This material was shipped to consultants Base Metallurgical Laboratories Ltd. in Kamloops, BC for test work. Highlights from this drilling include:

— 42.85 g/t gold over 3.8 m (66.6 to 70.4 m), including 316.76 g/t gold over 0.5 m in hole BR-18-70;

— 9.83 g/t gold over 3.5 m (81.1 to 84.6 m) including 58.31 g/t gold over 0.5 m in hole BR-18-69;

— 7.48 g/t gold over 4.1 m (125.0 to 129.1 m) including 35.45 g/t gold over 0.8 m in hole BR-18-68;

— 7.69 g/t gold over 2.9 m (34.0 to 36.9 m) including 43.87 g/t gold over 0.5 m in hole BR-18-69; and

— 16.54 g/t gold over 1.3 m (105.0 to 106.3 m) including 26.53 g/t gold over 0.8 m in hole BR-18-69.

10.11.5 EAST GOLDBROOK INFILL

The EG infill drilling (BR-19-87, BR-19-89 to BR-19-92 and BR-19-99) was designed to upgrade resource categorization in key areas that were identified for development in the 2018 PEA. These holes all targeted the south limb of the anticline within the EG Gold System between local grid sections 9250E to 9550E. All holes intersected gold mineralization in areas predicted by the geological model. Highlights from the infill drilling include:

— 50.60 g/t gold over 1.0 m (246.0 to 247.0 m) in hole BR-19-89;

— 12.23 g/t gold over 2.0 m (214.3 to 216.3 m) in hole BR-19-89;

— 6.03 g/t gold over 2.9 m (200.7 to 203.6 m) in hole BR-19-90;

— 72.40 g/t gold over 0.6 m (21.0 to 21.6 m) in hole BR-19-87; and

— 32.62 g/t gold over 0.9 m (290.7 to 291.6 m) in hole BR-19-87.

10.11.6 EAST GOLDBROOK EXPANSION

Exploration drilling of the EG Gold System (BR-18-64 to BR-18-67) focussed east of local grid section 9550E, which is the eastern limit of the previous 2018 resource model. This drilling intersected multiple high-grade gold zones mainly on the southern limb of the Upper Seal Harbour Anticline. These zones appeared to gently plunge toward the east-southeast, similar to the orientation of fold plunge, and correlated with mineralized zones in historic drillholes OSK11-03 and OSK11-04 50 m to the east. These results extended gold mineralization 150 m along strike to the east. In 2019, follow-up step-out drilling (BR-19-73 to BR-19-75) intersected high grade gold mineralization further down-plunge along the south limb, increasing the mineralized zone to over 375 m along strike.

Subsequently, the area of near-surface mineralization along the south limb was targeted (BR-19-76, BR-19-86, BR-19-88, and BR-19-93 to BR-19-97) to delineate mineralized belts and their continuity. This drilling intersected multiple intervals of high-grade gold mineralization, and added eight new belts in the EG Gold System over a 300 m strike-length and at least 50 m down-dip to a depth of at least 125 m. This area is currently open up- and down-dip as well as down plunge. Selected highlights include:

— 27.12 g/t gold over 2.5 m (51.3 to 53.8 m) in hole BR-19-97 including 133.11 g/t gold over 0.5 m;
— 102.43 g/t gold over 0.7 m (142.0 to 142.7 m) in hole BR-19-86;
— 25.70 g/t gold over 1.5 m (61.0 to 62.5 m) in hole BR-18-66 including 63.33 g/t gold over 0.6 m;
— 16.65 g/t gold over 2.0 m (167.5 to 169.5 m) in hole BR-19-88 including 65.49 g/t gold over 0.5 m;
— 8.00 g/t gold over 3.2 m (142.6 to 145.8 m) in hole BR-18-67 including 30.66 g/t gold over 0.8 m;
— 5.36 g/t gold over 2.2 m (321.1 to 323.3 m) in hole BR-19-75; and
— 17.79 g/t gold over 0.5 m (192.8 to 193.3 m) in hole BR-19-74.

Drillhole BR-19-72 testing an east-southeast trending IP chargeability anomaly 350 m north of the fold hinge did not intersect any significant gold mineralization. Intermittent quartz-carbonate+/-pyrite infilled fracturing in the host rock could possibly be responsible for the chargeability high.

10.12 QP’S OPINION

It is the QP’s opinion that the drilling and logging procedures put in place by Anaconda and the proceeding companies meet acceptable industry standards and that the information can be used for geological and resource modeling.

11 SAMPLE PREPARATION

11.1 INTRODUCTION

Sample preparation, analyses, and security discussions for all drilling programs carried out on the property prior to 2010 were addressed in previous NI 43-101 resource estimate technical reports completed by Gervais et al. (2009), Puritch et al. (2006) and Bourgoin et al. (2004). Cullen and Yule (2017) addressed subsequent programs completed by Orex and this report includes that information as well as new information pertaining to the 2017 to 2019 work completed by Anaconda. Drillholes from programs completed between 1984 and 2011 are included in the database used for the current resource estimate, and summary descriptions of associated procedures that have been developed from review of previous reporting are presented below for all programs carried out during that time period. Limited programs carried out by Minnova in 1991, Placer Dome in 1995, and Orex in 1993 in association with metallurgical and check sampling investigations are not included below but were briefly discussed in report Section 6.

The sampling approaches in programs carried out prior to 2005 generally reflect sampling of visibly determined mineralized belts, respective of major geological units, plus varying amounts of adjacent wall rock material. Exceptions to this include continuous core sampling and/or total core rather than half core sampling pertain to certain historic metallurgical programs. Continuous mineralized zone sampling, generally respective of major lithologic units, pertains to 2005 and later programs.

11.2 PROGRAM DESCRIPTIONS

11.2.1 1984 TO 1985: ONITAP RESOURCES INC.

Specific descriptions of core sample preparation, analysis and security protocols for drilling carried out during this period are not included in associated assessment reporting filed with the Nova Scotia government. Based on the assessment reporting, QA/QC procedures appear to have been restricted to those instituted by the laboratory that provided analytical services.

11.2.2 1987: PETROMET RESOURCES INC. – GREENSTRIKE GOLD CORP.

Specific descriptions of core sample preparation, analyses, and security protocols for drilling carried out during this period are not included in associated assessment reporting filed with the Nova Scotia government. However, as in the previous case, standard rock sample preparation followed by fire assay fusion and Atomic Absorption finish is indicated from laboratory certificates. Analytical services were provided by Chemlab Inc. of Saint John, New Brunswick, and a screen metallic processing protocol was used for samples returning an initial gold value of 1.0 g/t or more.

No specific comments on program security protocols are present in associated reporting, but industry standard field protocols of the era are assumed to have been in place. Based on assessment reporting, QA/QC procedures appear to have been restricted to those instituted by the independent commercial laboratories providing analytical services.

11.2.3 1987: ONITAP RESOURCES INC.

Drill core samples were shipped to Assayers (Ontario) Limited in Toronto, Ontario and processed by either a standard rock preparation protocol followed by Fire Assay pre-concentration and Atomic Absorption finish (FA-AA) or a screen metallic protocol. The FA-AA approach utilized a single 15 g sample split, and the screen metallics processing approach consisted of one FA-AA analysis of a 15 g split of the pulverized plus 50 mesh fraction, and two FA-AA analyses on 15 g splits of pulverised minus 50 mesh material. The three analyses were weight-averaged to produce a gold head grade for each sample. Criteria used to specify screen metallic processing were not cited in associated reporting.

No specific comments on program security protocols are present in associated reporting, but industry standard field protocols of the era are assumed to have been in place. Based on assessment reporting, QA/QC procedures appear to have been restricted to those instituted by the independent commercial laboratories providing analytical services.

11.2.4 1988 – 1990: OREX

Drill core samples from both surface and underground programs were generated during this period and a number of laboratories were used through time to meet sample preparation and analysis requirements. Samples generated prior to January1990 were processed using standard rock preparation followed by FA AA gold determination if visible gold was not noted for the sample. If visible gold was noted, or high values were returned from the initial analysis, the sample was processed using screen metallic methods. Samples generated in January 1990 and through to the end of the program were all processed using a double screen metallics processing protocol. This consisted of grinding the entire sample and screening at 28 mesh to create plus and minus fractions that were then separately pulverized and processed using screen metallic procedures. This processing stream generated six fire assay gold analyses per submitted sample and multiple laboratories were enlisted to carry out preparation and analytical services. These included Lakefield Research, CRM, Chimitec, Xral, CEGEP de l'Abitibi-Témiscamingue – St. Michel Géoconseil. Of these, Lakefield Research, Chimitec, and Xral provide analytical services.

No specific comments on program security protocols are present in associated reporting, but industry standard field protocols of the era are assumed to have been in place. Based on assessment reporting, QA/QC procedures appear to have been restricted to those instituted by the independent commercial laboratories providing analytical services.

11.2.5 2005 AND 2008: OREX

Drill core samples from surface drilling programs carried out in 2005 (HQ core) and 2008 (NQ core) were generated during this period. Samples were sent to ALS Chemex (ALS) facilities in either Val-d'Or, Québec (2005) or Timmins, Ontario (2008). ALS is an independent, internationally accredited laboratory with National Association of Testing Authorities (NATA) certification and also complies with standards of ISO 9001:2000 and ISO 17025:1999.

Standard rock sample crushing and grinding procedures at ALS were followed by initial fire assay fusion FA finish analysis of 50 g pulp splits. If the initial result met or exceeded a 2.5 ppm gold threshold, analysis of a second coarse reject split was carried out using a gravimetric finish. Composite metallurgical samples were also created from coarse reject materials selected by Orex consultants and these were submitted to SGS Lakefield in for whole sample metallurgical testing.

A quality assurance and quality control program that included analysis of Certified Reference Material (CRM), field duplicates, coarse reject duplicates, pulp split duplicates and blank samples was carried out with respect to both the 2005 and 2008 programs and results of these programs are presented in report Section 11.3.

Core from the 2005 program was photographed and quick-logged at the Goldboro site, and secured boxes were then wrapped in plastic and sent by commercial carrier to Val-d'Or, where detailed logging, sample mark-up, and half-core sawing was carried out under direction of MRB and Associates (MRB) geologists and project consultant, Mr. Alex Horvath, P. Geo. Samples cut by MRB were placed in labeled plastic bags along with a sample tag, then sealed and sent to ALS for analysis after insertion of quality control samples. ALS carried out an internal QC program that included analysis of duplicate splits, blanks, and reference materials. The first 17 holes of the 2008 drilling program were whole-core sampled (Holes BR-08-01 to BR 08-17), and all other holes of both the 2005 and 2008 programs were sawn and subjected to half-core sampling.

Both the 2005 and 2008 programs were supervised by Mr. J. Lafleur, P. Geo., and onsite logging, sampling, and field services were provided by W.G. Shaw. Reject core sample materials and pulps were returned to Orex after processing, and core not sent to Val-d'Or was securely stored at the Goldboro site. MRB provided detailed core logging and sampling services for the 2005 program after quick logging on site followed by shipment of core to Val-d'Or from Nova Scotia by secure commercial carrier. Security of site operations, core, samples and core storage were addressed on an ongoing basis by site staff employed by the two consulting firms noted above.

11.2.6 2010 – 2011: OSISKO

The following description of sample preparation and core handling protocols applies to all drilling carried out during the 2010 and 2011 programs by Osisko on the Goldboro property under terms of the option agreement with Orex. The 2010 program was carried out under project supervision of Mr. J. Lafleur, P. Geo., and that in 2011 was carried out under supervision of consultant Mr. Bruce Mitchell, P. Geo. W.G. Shaw provided most core logging, sample cutting, and field support staff for both programs, and Mercator supplied one P. Geo. staff geologist to assist with the 2011 core logging. All of the NQ-sized core was logged, photographed, sampled, bagged, tagged, and sealed at the Goldboro site by qualified personnel. Samples were placed in numbered pails, sealed, and shipped in batches by commercial carrier to the ALS laboratory. Logging utilized Gemcom Gems ™ Logger software and project protocols included progressive, systematic, and secure offsite backup of digital drilling, logging, and sampling data.

Drill core was oriented according to bedding and cleavage planes prior to logging and sampling. In 2010, drill cores were sampled continuously from top to bottom. In 2011, barren, non-mineralized greywacke was not sampled unless bounded by mineralized zones with 10 m or less of separation. The following sample interval length protocol was generally applied, but in some cases adjustments were made to respect major lithological boundaries and contacts of large quartz veins:

- Visually unmineralized, unaltered core - sample length of 1.5 m;
- Visually mineralized, altered core – sample length of 1.0 m;
- Core with visible gold – sample length of 0.5 m.

Core from both programs was split into two equal halves by sawing, and the top half of the oriented core was the portion placed in the sample bag for analysis. The bottom core half was retained in the core box for archival purposes. It was also noted whether visible gold was present in either the top half, bottom half, or both halves of the core.

At ALS, each sample was crushed to 70% < 2 mm, split to 250 g using a riffle splitter, pulverized to 85% at < 0.075 mm, and made into a 50 g sample of pulp. The 50 g pulp was fire assayed with atomic absorption spectrometry finish (ALS codes Au-AA24 and Au-AA26). Samples exceeding the atomic absorption spectrometry threshold were re-assayed using a gravimetric finish (ALS code Au-GRA22). Initial assays between 2.5 g/t Au and 0.5 g/t Au were re-assayed (ALS code Au-AA26D) and if the second results were ≥ 2.5 g/t Au, the laboratory was instructed to perform a total metallic screen (ALS code Au-SCR24) on the coarse reject of those samples. The reason for such is to assess the potential impact of coarse-grained Au in the sample. All samples containing visible gold were directly assigned for processing using the total metallic screen method with FA-AA or gravimetric finish.

Logged and sampled drill core from both programs was placed on core racks inside the core facility at Goldboro, and boxes were properly labeled with aluminum tape. Reject and pulp materials from both programs were recovered from the laboratory and placed in secure dry storage at the Goldboro site. A Quality Assurance and Quality Control program that included analysis of CRMs, field duplicates, coarse reject duplicates, pulp split duplicates, and blank samples was carried out with respect to both 2010 and 2011 programs. Results of these are presented in report Section 11.3.

Security of site operations, core, samples, and core storage were addressed on an ongoing basis by site staff employed by the two consulting firms noted above.

11.2.7 2017 – 2018 ANACONDA PROGRAM

Drill core samples were collected systematically down the hole based on the occurrence of visual alteration, mineralization and quartz veining. Samples ranged in length from 0.3 to 1.0 m depending on the nature and width of veining and mineralization samples, while trying to best honour geological contacts. Samples were collected of quarter-sawn drill core (BR-17-01 to -05) or half-sawn drill core (BR-17-06 to -13 and BR-18-14 to -42) and shipped to Eastern Analytical Limited in Springdale, Newfoundland and Labrador for analysis via standard 30 g fire assay with Atomic Absorption (AA) finish. Samples were also analyzed at Eastern Analytical via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold, and all samples were submitted for 34-element ICP analysis. For the 2017 Metallurgical drilling program (BR-17-01 to -05), half of the sampled drill core assaying greater than 0.5 g/t gold was shipped to Thibault for metallurgical test work while the remaining quarter of the core was retained for future record.

All pulp material of the fine fraction resulting from the Eastern metallic screen analysis were shipped to ALS in North Vancouver, BC for check analysis using method Au-AA23 with samples assaying greater than 10 g/t gold subsequently analyzed via method Au-GRA21.

Drill core from the 2008 Orex drilling, 2010 and 2011 Osisko drilling along with Anaconda's 2017 and 2018 drilling is currently stored at the company core storage building at the Goldboro site. The 2008 core, is currently stacked on pallets, but much of the core has been destroyed due to vandalism and inappropriate storage practices. As much core as could be saved has been re-assembled and stored at site.

11.2.8 2018 – 2019 ANACONDA PROGRAM

Drill core samples were collected systematically down the hole based on the occurrence of visual alteration, mineralization and quartz veining. Samples ranged in length from 0.2 to 2.0 m depending on the nature and width of veining and mineralization while trying to best honour geological contacts. Samples were collected of half-sawn drill core (BR-18-43 to BR-19-99) and shipped to Eastern Analytical Limited in Springdale, Newfoundland and Labrador for analysis via standard 30 g fire assay with AA finish. Samples were also analyzed at Eastern Analytical via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold, and all samples up to hole BR-18-59 were submitted for 34-element ICP analysis.

Pulps of the fine fraction resulting from the Eastern metallic screen analysis were shipped to ALS in Moncton, New Brunswick for check analysis using method Au-AA23 with samples assaying greater than 10 g/t gold subsequently analyzed via method Au-GRA21.

Drill core from the 2018 – 2019 drilling program is currently stored in racks at the company core storage building at the Goldboro site.

11.3 QUALITY CONTROL AND QUALITY ASSURANCE PROGRAMS

11.3.1 1984 TO 2005 PROGRAMS

Review of assessment reporting related to the various drilling programs carried out during the 1984 to 2005 period showed that, with the exception of the metallurgical and check sampling program carried out by Placer in 1995, no structured programs designed to systematically monitor quality control and assurance issues for drill core were in place. During this period, it is assumed that reliance was placed on internal laboratory protocols and their results. The Placer program carried out in support of 1995 metallurgical investigations included insertion of blank and certified reference material samples as well as analysis of duplicate analytical pulp splits. Results of the program are not specifically addressed in associated reporting by Gagnon et al. (1996).

11.3.2 2005 TO 2011 PROGRAMS

Orex drilling programs in 2005 and 2008 and Orex-Osisko programs in 2010 and 2011 were subject to rigorous QA/QC programs, with some procedural changes incorporated during the period.

In 2005, 2008, 2010, and 2011 programs, field, coarse reject, and pulp split duplicate samples were routinely prepared and analyzed. Coarse reject duplicate splits were analyzed for samples returning initial pulp split gold values between 0.25 g/t and 2.50 g/t, and those returning initial pulp split gold values of 2.5 g/t or more were submitted for screen metallics processing. Screen metallics processing was also requested for all samples having visible gold. In addition, blind blank and CRM samples were routinely inserted in the sample stream. Results were monitored by Orex and Osisko on an ongoing basis during the course of their respective programs. Figures 11.1 and 11.2 present flowsheet summaries that define separate processing and QA/QC procedures specified for Orex 2005 program core samples that are based on presence or absence of visible gold in the core materials.

The above detailed protocols were established to assess accuracy and precision of results and to better document the impact of coarse-grained gold (nugget effect) on the sample population. The following key components are included in the detailed 2005 flowsheets:

- Blind insertion of blank sample material at a frequency of 1 in 18 regular samples;
- Creation of core field duplicates at a frequency of 2 in 18 regular samples;
- Creation of coarse reject duplicate splits at a frequency of 2 in 21 regular samples;
- Creation of pulp duplicate splits at a frequency of 2 in every 21 regular samples;
- Blind insertion of a CRM at a frequency of 1 in 21 regular samples.

During the 2008 program the frequency of blind blank insertion was decreased to 1 in 50, while frequency of field duplicate, coarse reject duplicate, and pulp duplicate analysis was decreased to 1 in 25. The CRM insertion rate was decreased to 1 in 50 and check sample pulps were included at a rate of 1 in 25 regular samples.

The 2010 program was similar to that for 2008 but included insertion rates for blind blank sample and CRM materials that were generally in the range of 1 in 20 to 1 in 25.

Figure 11.1 QA/QC Routine Sample Flowsheet

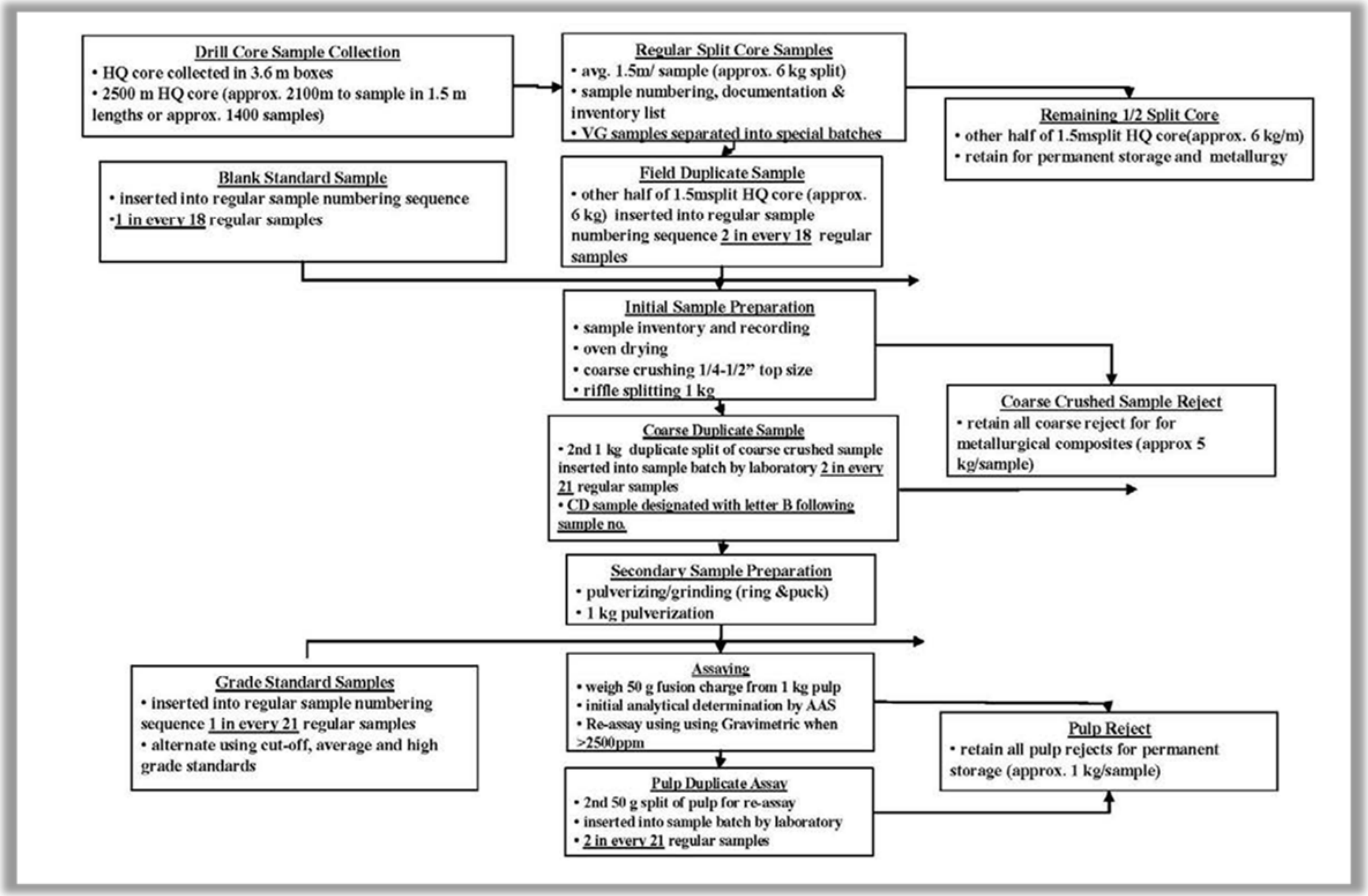
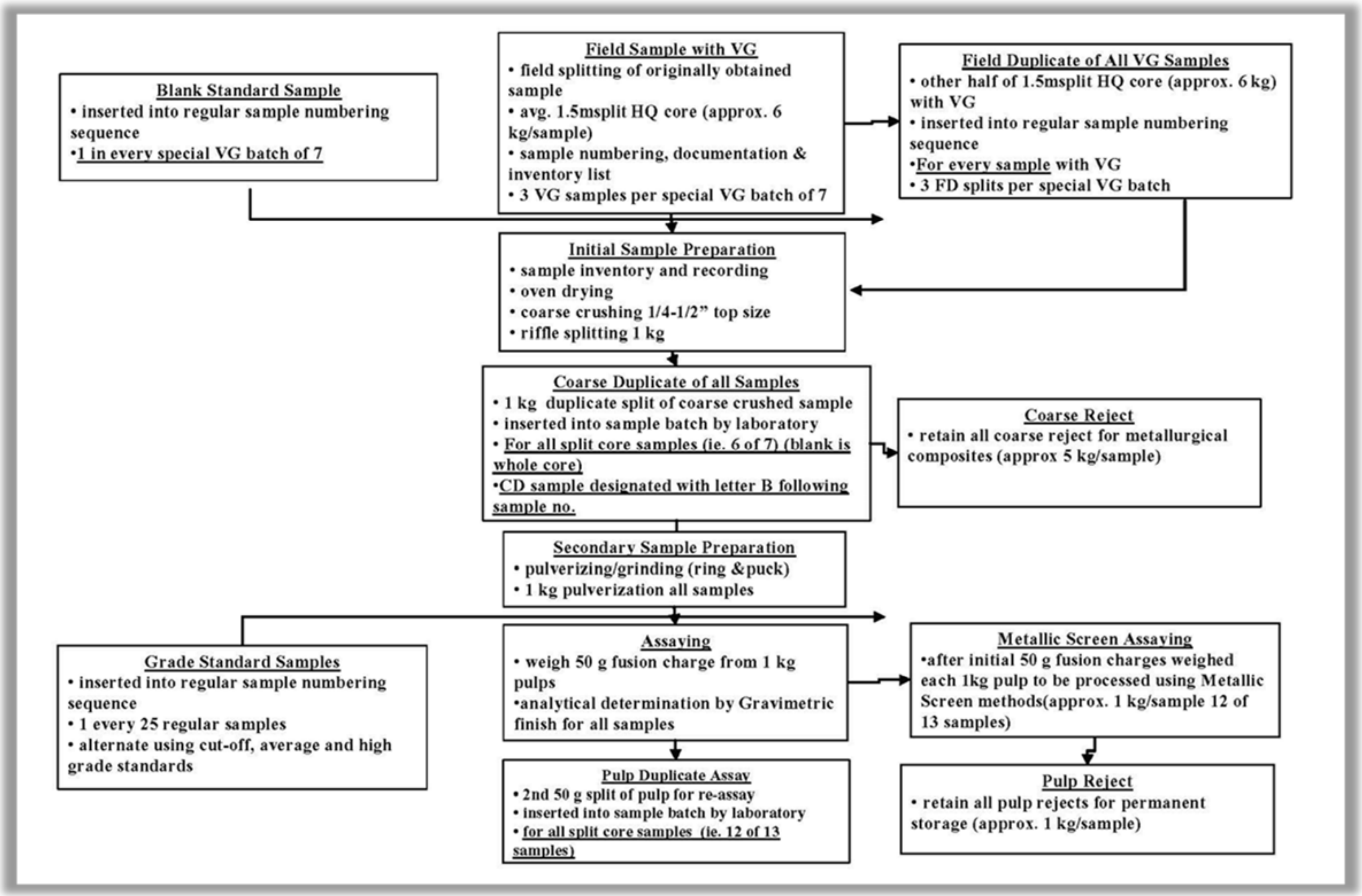


Figure 11.2 QA/QC Flowsheet for Samples with Visible Gold



11.3.3 CERTIFIED REFERENCE MATERIAL

2005 PROGRAM

Puritch et al. (2006) discussed the CRM results for the 2005 drilling campaigns and noted that this material was obtained from Rock Labs Ltd. (Rock Labs) of Auckland, New Zealand. Three different reference materials were used, these being SF-12, with a mean grade of 0.819 g/t gold; SK-21, with a mean grade of 4.048 g/t gold; and SN-16, with a mean grade of 8.367 g/t gold. The SF-12 and SN-16 standards returned satisfactory results, with 95% of the values falling within 2 standard deviations of the mean value. This result is in keeping with certified limits for the materials. The mid-grade SK-21 CRM produced poor results, with several consecutive samples falling between ±2 and 3 standard deviations and showing consecutive sample bias. Several consecutive samples exceeded ± 3 standard deviation limits. Investigation of these results indicated that the SK-21 material used on this project may not have been in compliance with stated confidence interval limits and associated grade ranges (*Puritch et al., 2006*).

2008 PROGRAM

Orex inserted one of two certified reference standards at a frequency rate of every 50th sample. The standards used and the results of the QC program were not reported within the 2009 technical report (*Gervais, et al, 2009*)

2010 PROGRAM

Osisko used four certified reference materials in 2010, all of which were provided by Rock Labs. These were designated OxJ64, OxK69, SF45, and SN50. Table 11.1 summarizes certified mean values of each.

Table 11.1 CRM Summary - 2010 Drilling Campaign

Reference Material	Certified Mean Value (Au g/t)	2 Standard Deviations (Au g/t)	Number Used
OxJ64	2.366	0.158	130
OxK69	3.583	0.172	23
SF45	0.848	0.056	153
SN50	8.685	0.360	142
TOTAL			**448**

Of the 448 samples of certified reference material that were analyzed, 9.15% of samples returned values exceeding mean ± 3 standard deviation control limits. Notably, results for SN50 material exceeded control limits 16.9% of the time, while other materials exceeded limits at rates of 4.62% (OxJ64), 8.70% (OxK69), and 5.88% (SF45). Figures 11.3 through 11.6 present the 2010 CRM program results and show that control limit exceedances define a low bias for OxJ64, SF45 and SN50. Results for SN50 are not consistent with those of the other materials. This may reflect inhomogeneity at the time of splitting or some other material quality issue.

Figure 11.3 Certified Reference Material OxJ64 - Gold Results (N=130)

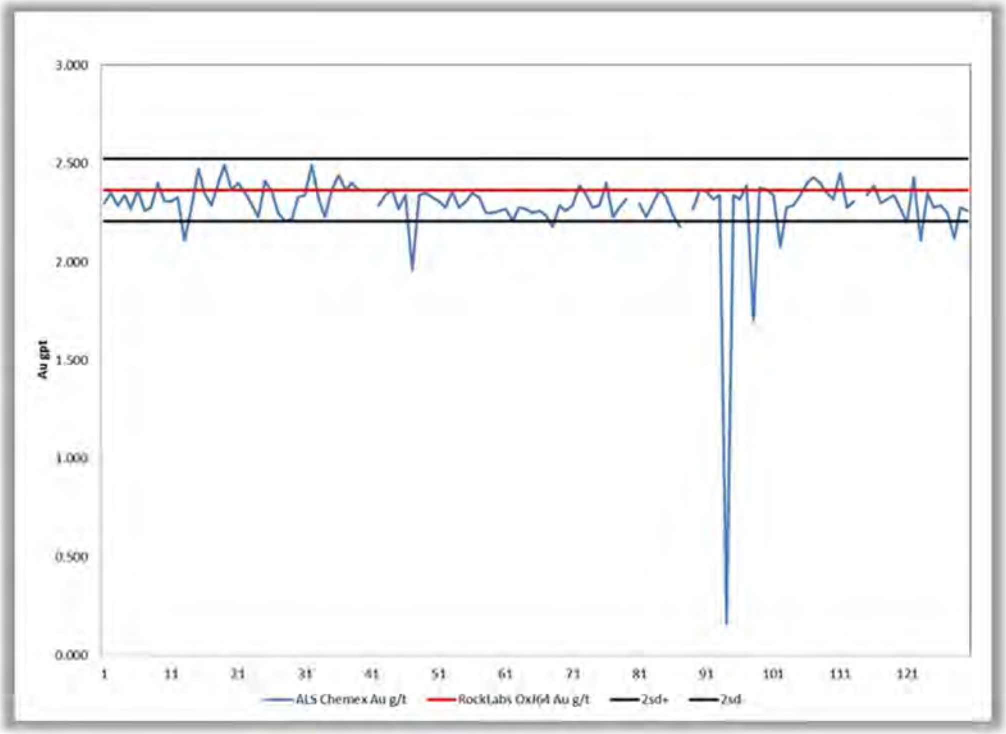


Figure 11.4 Certified Reference Material OxK69 - Gold Results (N=23)

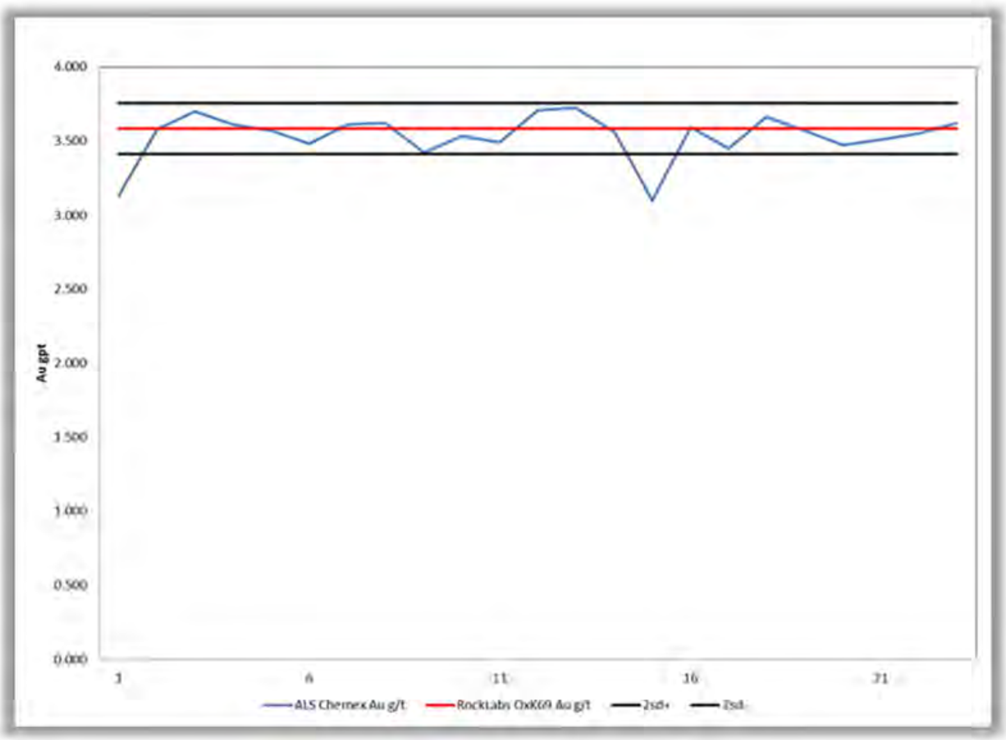


Figure 11.5 Certified Reference Material SF45 - Gold Results (N=153)

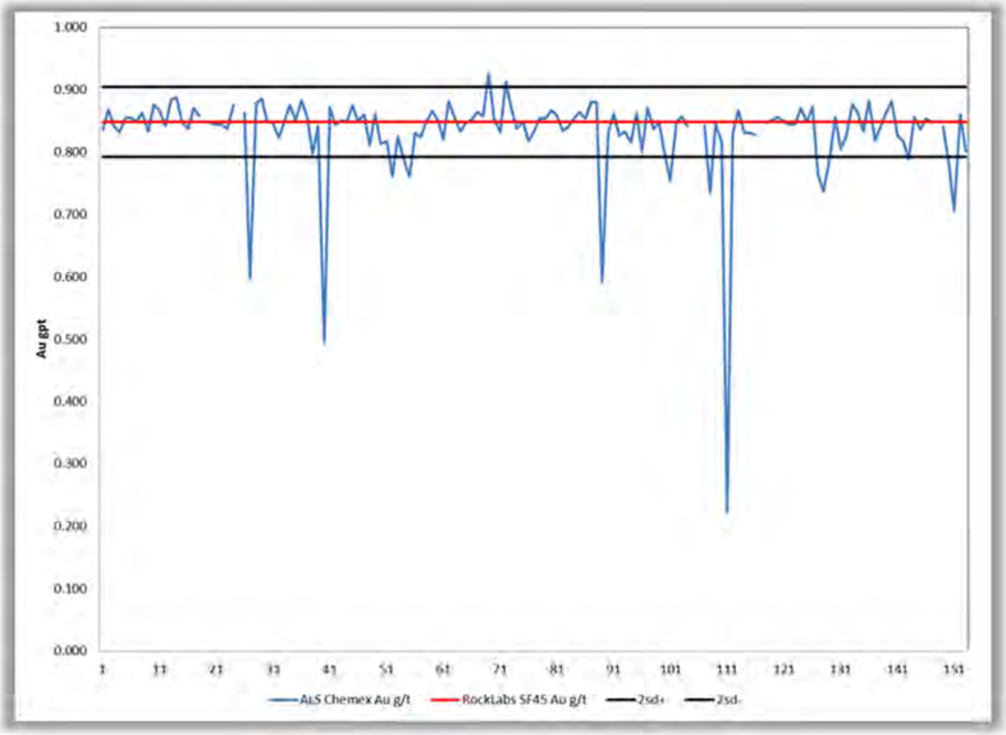
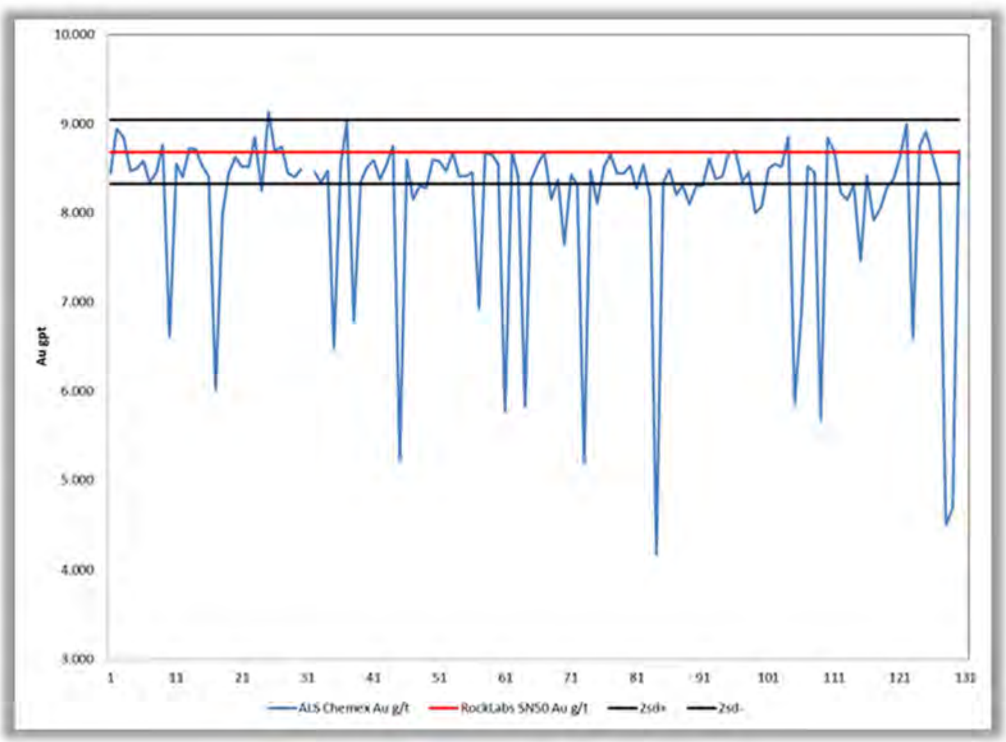


Figure 11.6 Certified Reference Material SN50 - Gold Results (N=142)



2011 PROGRAM

For the 2011 drilling campaign, Osisko used the SF45 and SK52 certified reference materials and both were provided by Rock Labs. Table 11.2 presents certified mean value and standard deviation data for each material.

Table 11.2 CRM Summary - 2011 Drilling Campaign

Reference Material	Certified Mean Value (Au g/t)	2 Standard Deviations (Au g/t)	Number Used
SF45	0.848	0.056	14
SK52	4.107	0.088	12
		TOTAL	**26**

Of the 26 samples of that were analyzed, 5 returned values that exceeded mean ± 3 standard deviation control limits and these represent a combined exceedance rate of 19.2%. The SK52 material produced a 16.67% exceedance rate, and the SF45 material produced a 21.43% exceedance rate. Figures 11.7 and 11.8 present results for the two materials used in 2011 and show that only low exceedances were present.

Figure 11.7 Certified Reference Material SF45 - Gold Results (N=14)

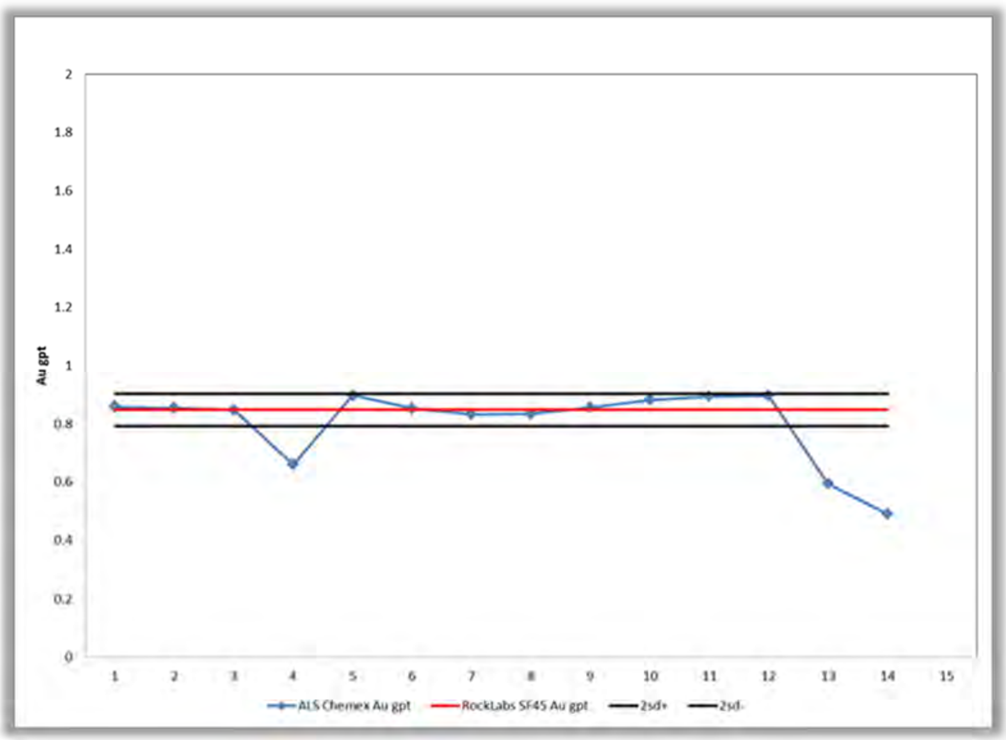
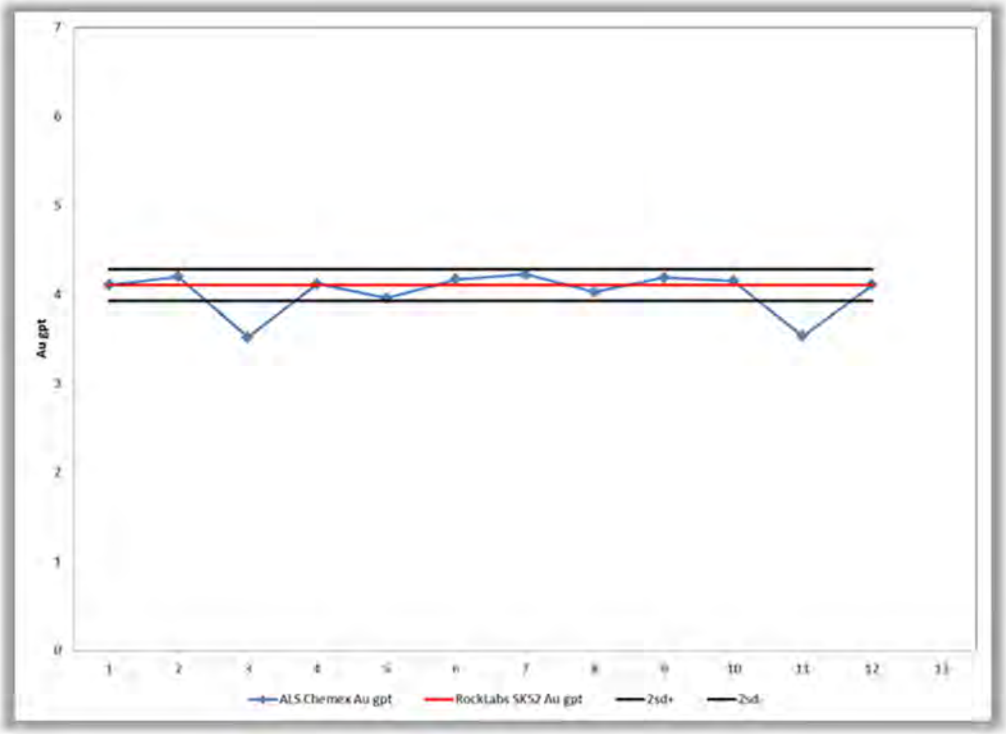


Figure 11.8 Certified Reference Material SK-52 - Gold Results (N=16)



2017 – 2018 AND 2018 – 2019 ANACONDA PROGRAM DRILL PROGRAMS

CRM samples were routinely inserted in the sample stream during the Phase II (2017 – 2018) and Phase III (2018 – 2019) programs, and results were monitored by Anaconda on an ongoing basis. Three pre-packaged, gold CRMs (CDN-GS-10E, CDN-GS-1M, CDN-GS-1U) from CDN were used and inserted into the sample stream every 25 samples. The three reference materials were similarly produced at CDN by blending high-grade gold mineralization with non-mineralized granitic material.

CDN-GS-10E has a certified mean value of 9.59 g/t gold, CDN-GS-1M has a certified mean value of 1.07 g/t gold, and CDN-GS-1U has a certified mean value of 0.968 g/t gold. The CRMs were inserted into the sample stream on a sequentially alternating basis to provide assessment of both lower grade (~1 g/t gold; CDN-GS-1M, 1U) and higher grade (~10 g/t gold; CDN-GS-10E) analytical ranges.

In total, 316 samples of CDN-GS-10E, 75 samples of CDN-GS-1M and 239 samples of CDN-GS-1U were analyzed, and sequential results are presented on Figures 11.9, 11.10, and 11.11. GS-10E has an accuracy and precision of -1.65% and 2.96% respectively relative to the certificate. Six blank samples mistakenly mislabelled were removed from the dataset. GS-1M has an accuracy and precision of -5.01% and 3.24% respectively relative to the certificate, with no samples outside the control limits. GS-1U has an accuracy and precision of 4.27% and 2.86% respectively relative to the certificate.

Figure 11.9 2018-2019 Results for CRM CDN-GS-10E (N=322)

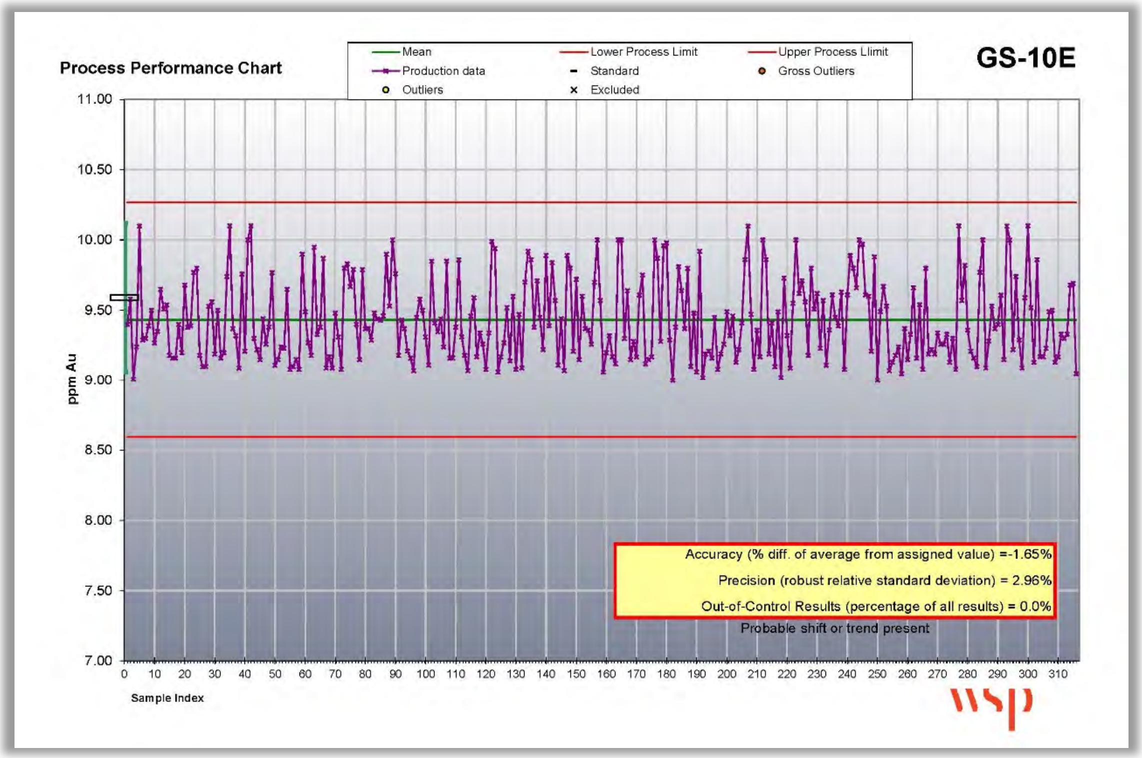


Figure 11.10 2018-2019 Results for CRM CDN-GS-1M (N=241)

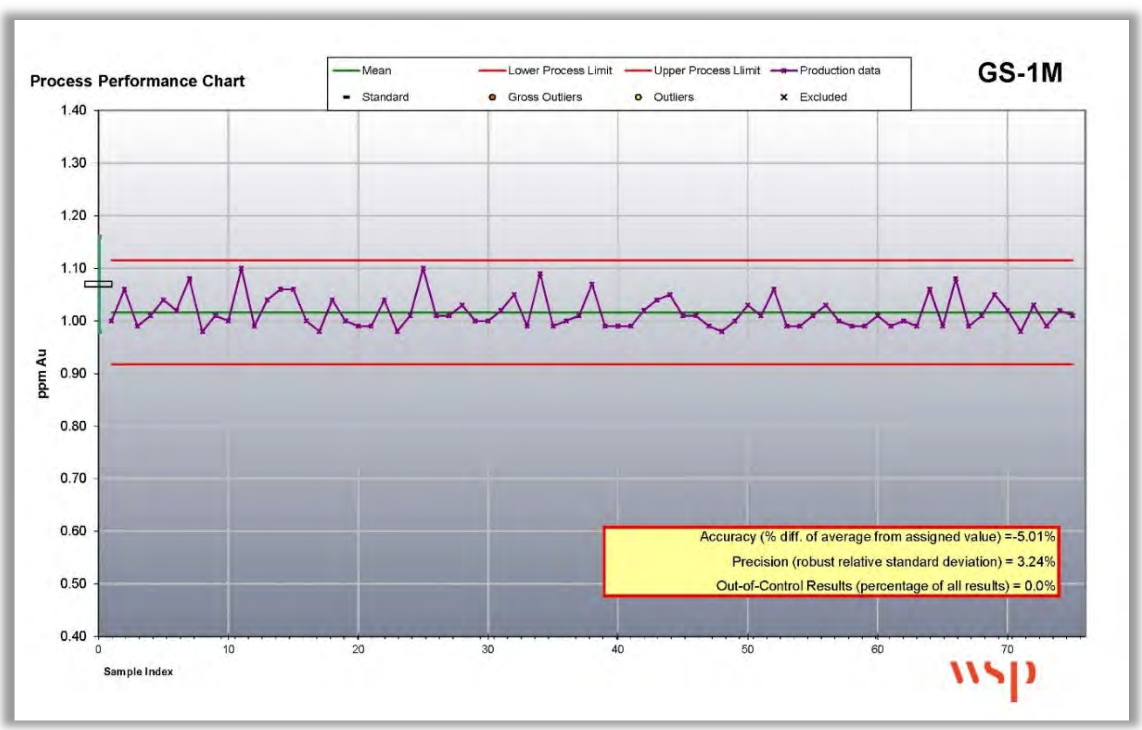
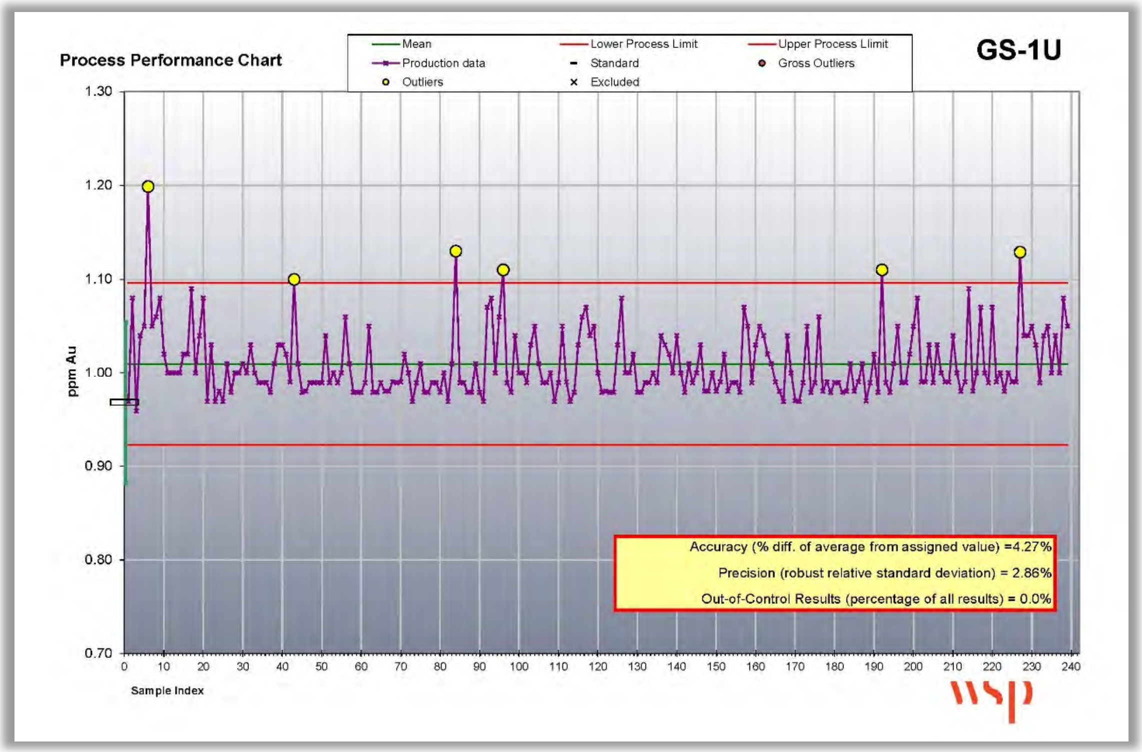


Figure 11.11 2018-2019 Results for CRM CDN-GS-1U (N=241)



2019 ANACONDA PROGRAM BULK SAMPLE

CRM samples were routinely inserted in the sample stream during the 2019 bulk sample program, and results were monitored by Anaconda on an ongoing basis. Two pre-packaged, gold CRMs (CDN-GS-10E and CDN-GS-1U) from CDN were used and inserted into the sample stream every 100 samples. The two reference materials were similarly produced at CDN by blending high-grade gold mineralization with non-mineralized granitic material.

CDN-GS-10E has a certified mean value of 9.59 g/t gold and CDN-GS-1U has a certified mean value of 0.968 g/t gold. The CRMs were inserted into the sample stream on a sequentially alternating basis to provide assessment of both lower grade (~1 g/t gold; CDN-GS-1U) and higher grade (~10 g/t gold; CDN-GS-10E) analytical ranges.

In total, 17 samples of CDN-GS-10E, and 16 samples of CDN-GS-1U were analyzed and sequential results are presented below on Figures 11.12 and 11.13. GS-10E has an accuracy and precision of -1.65% and 2.96% respectively relative to the certificate. Six samples (1.9%) were outside the control limits, returning zero grade. GS-1U has an accuracy and precision of 4.27% and 2.87% respectively relative to the certificate, with 0.8% samples outside the control limits.

Figure 11.12 2019 Bulk Sample Results for CRM CDN-GS-10E (N=14)

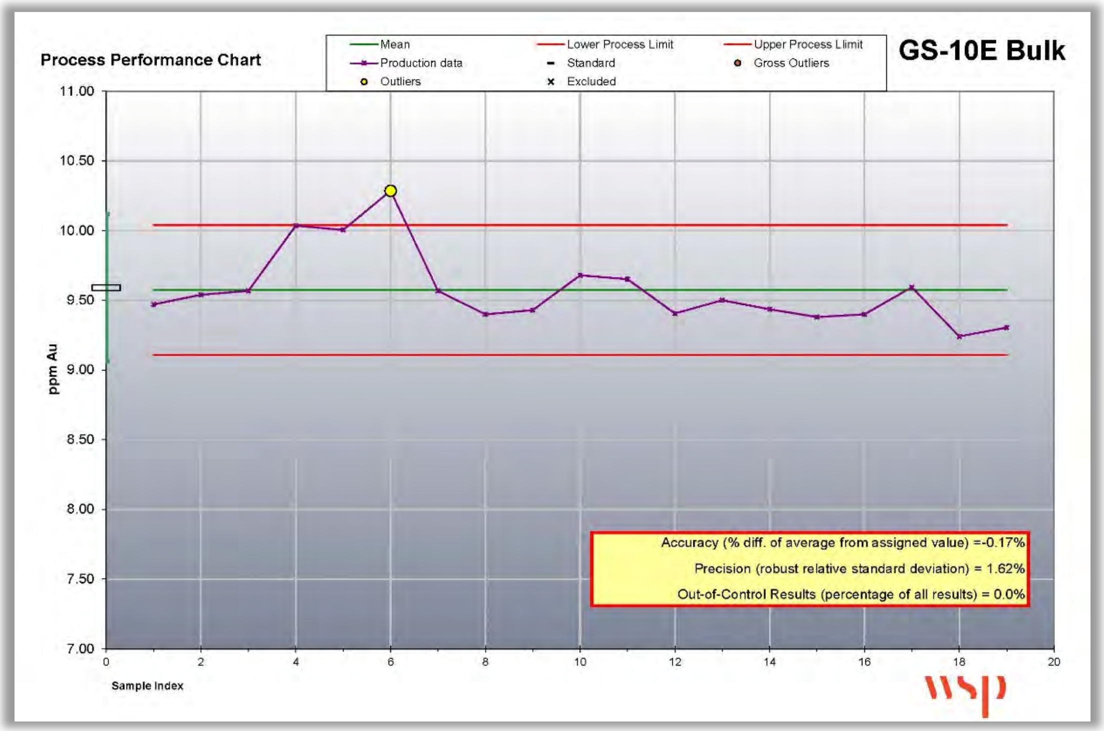
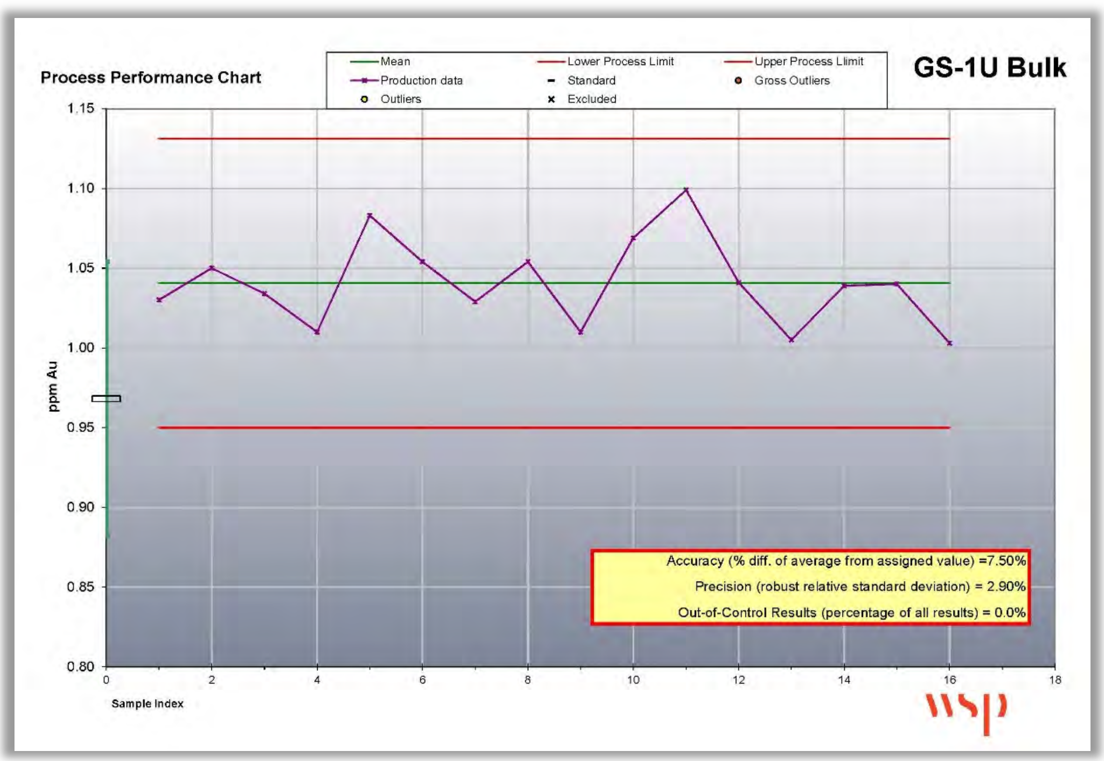


Figure 11.13 2019 Bulk Sample Results for CRM CDN-GS-1U (N=16)



11.3.4 BLANK SAMPLES

2010 AND 2011 PROGRAMS

The 2010 program by Osisko included analysis of 451 blank samples, and that in 2011 included analysis of 48 additional blank samples. All were comprised of sandstone from outcrops located near Antigonish, Nova Scotia. A single 1.52 g/t gold value spike characterizes the 2010 dataset, which has an average value of <0.005 g/t gold. The 2011 dataset shows no gold value spiking and has the same average value as the 2010 dataset. Checking of the database showed that a 44.1 g/t gold value was returned for the sample preceding the 2010 spike by 2 places in the analytical stream, and the sample preceding the blank spike returned a value of 3.26 g/t. This indicates that in this specific case, preparation stage contamination probably occurred in the sample stream immediately following processing of the high-grade sample. Figures 11.14 and 11.15 present 2010 and 2011 blank sample results. The combined results from the two programs demonstrate that cross-contamination of sample materials during these programs is not an issue.

Figure 11.14 2010 Blank Samples - Gold Results (N=451)

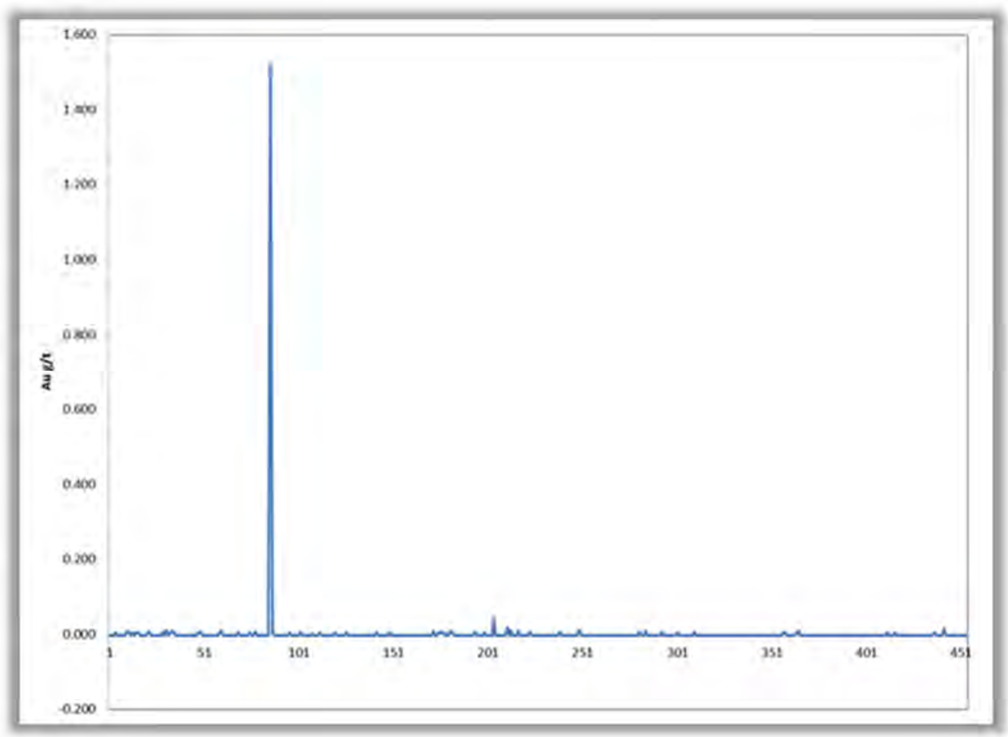
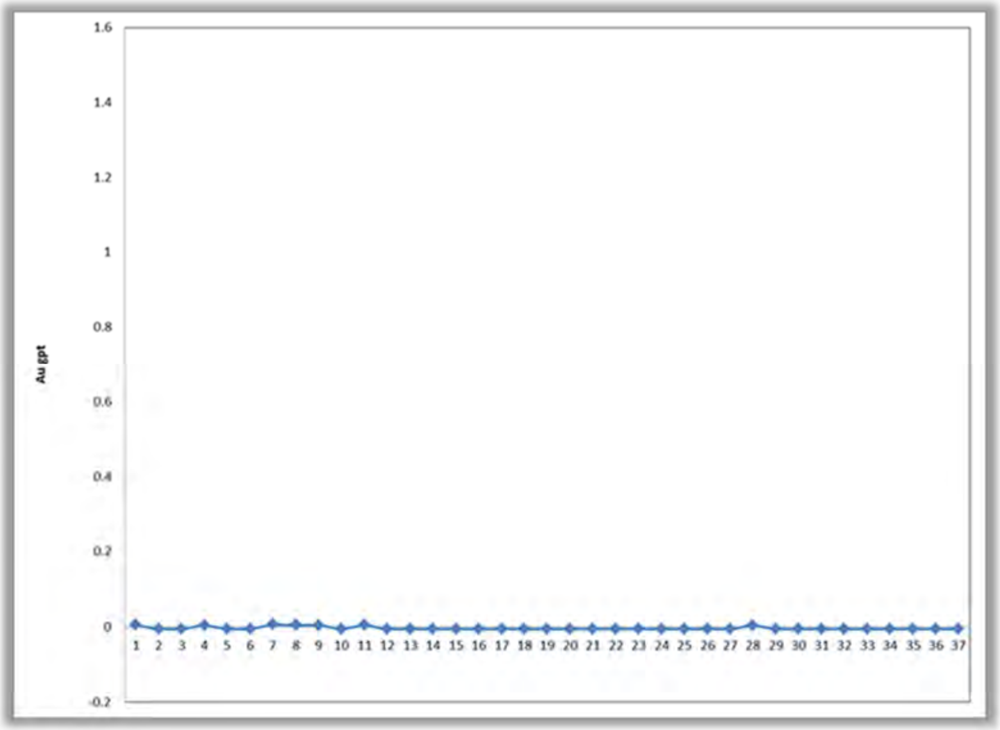


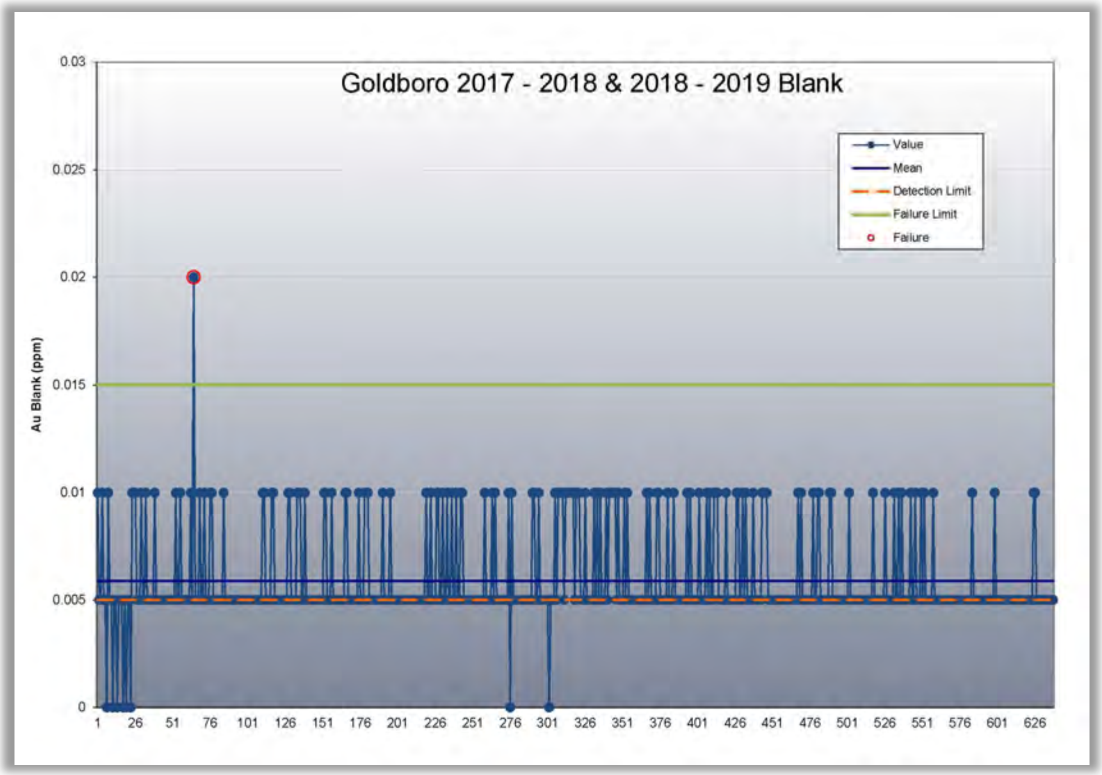
Figure 11.15 2011 Blank Samples - Gold Results (N=37)



2017 – 2018 AND 2018 – 2019 ANACONDA DRILL PROGRAMS

Blind blank samples consisting of non-mineralized granite obtained by Anaconda from Isaac's Harbour, Nova Scotia were routinely inserted in the 2017 – 2018 and 2018-2019 sample streams at a nominal frequency of 1 in 25. Results were monitored by Anaconda on an ongoing basis during the course of the drill program, and a total of 638 blank samples were analyzed. Only one sample returned above the failure limit of 0.015 g/t (Figure 11.16). This is interpreted as indicating that no systematic cross-contamination issue is present in the 2018 - 2019 results.

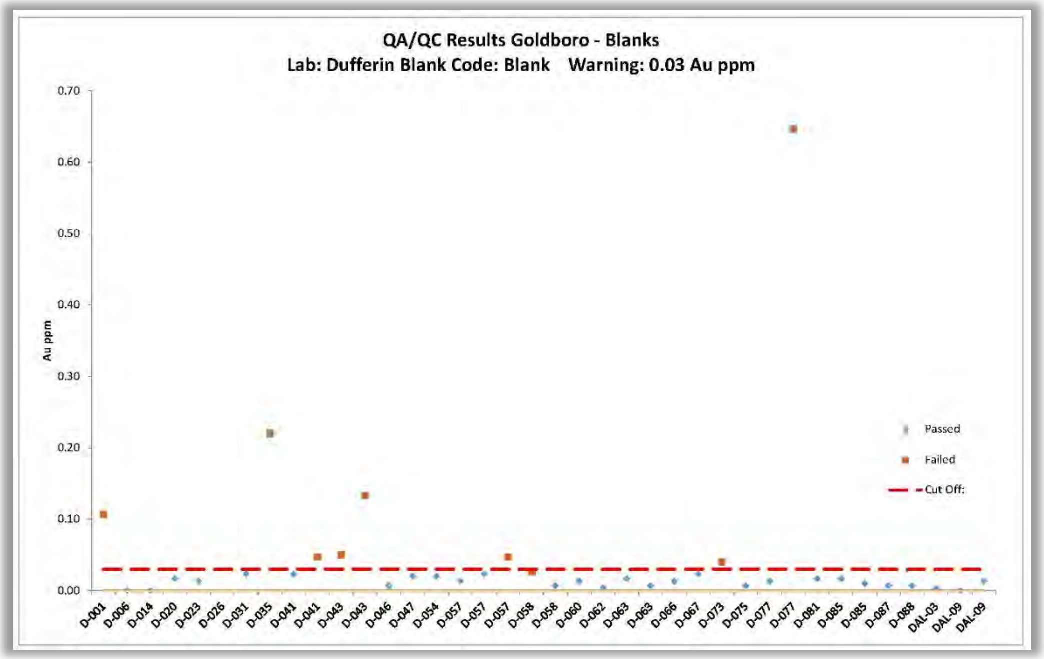
Figure 11.16 2018-2019 Anaconda Blank Samples - Gold Results (N=198)



2019 ANACONDA BULK SAMPLE PROGRAM

Blind blank samples consisting of non-mineralized granite obtained by Anaconda from Isaac's Harbour, Nova Scotia were routinely inserted in the 2019 bulk sample stream at a nominal frequency of 1 in 100. Results were monitored by Anaconda on an ongoing basis during the course of the bulk sample program, and a total of 38 blank samples were analyzed. Eight samples returned above the failure limit of 0.03 g/t (Figure 11.17). The failures may be due to the low detection limit used when operating in an "brown fields" environment.

Figure 11.17 2019 Anaconda Bulk Samples Blank Samples - Gold Results (N=38)

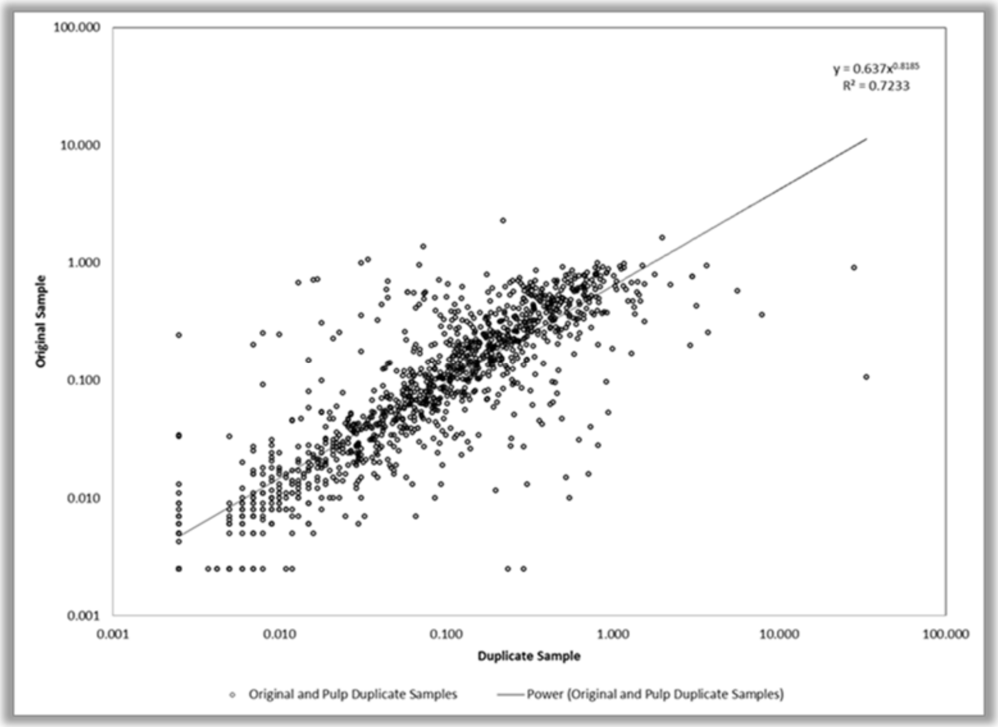


11.3.5 DUPLICATE COARSE REJECT SPLITS AND DUPLICATE PULP SPLITS

2010 AND 2011 PROGRAMS

Osisko analyzed duplicate coarse reject material splits for samples returning initial gold assays between 0.25 g/t and 2.5 g/t, and systematically analyzed pulp splits from the initial sample submission stream in both 2010 and 2011. Coarse reject split duplicate results for 2010 are presented on Figure 11.18, and the 2011 program included an additional 38 sample pairs. Results show that poor to moderate correlation (R^2 values of .7233 and .7359 for 2010 and 2011 respectively) exists between results for the coarse reject splits and corresponding original sample values. This trend is generally consistent with results reported earlier by Gervais et al. (2009) and Puritch et al. (2006) and is believed to be largely attributable to heterogeneity of gold distribution in the sample material (nugget effect) at the coarse reject grain size. In contrast, duplicate pulp split results for 2010 and 2011 show higher correlation and returned R^2 values of 0.94 and 0.99 for populations of 339 and 24 pairs, respectively.

Figure 11.18 2010 Coarse Reject Duplicate Samples - Gold Results (N=1203)



11.3.6 FIELD DUPLICATES

Osisko analyzed quarter core field duplicate samples on a nominal 1 in 25 basis in the 2010 program, and in 2011 continued such sampling. Results for 2010 and 2011 appear on Figures 11.19 and 11.20 respectively. The larger 2010 dataset shows that poor correlation exists between core splits (R^2 value of 0.586), but that for 2011 shows higher correlation (R^2 value of 0.7359).

Figure 11.19 2010 Core Field Duplicate Samples - Gold Results (N-455)

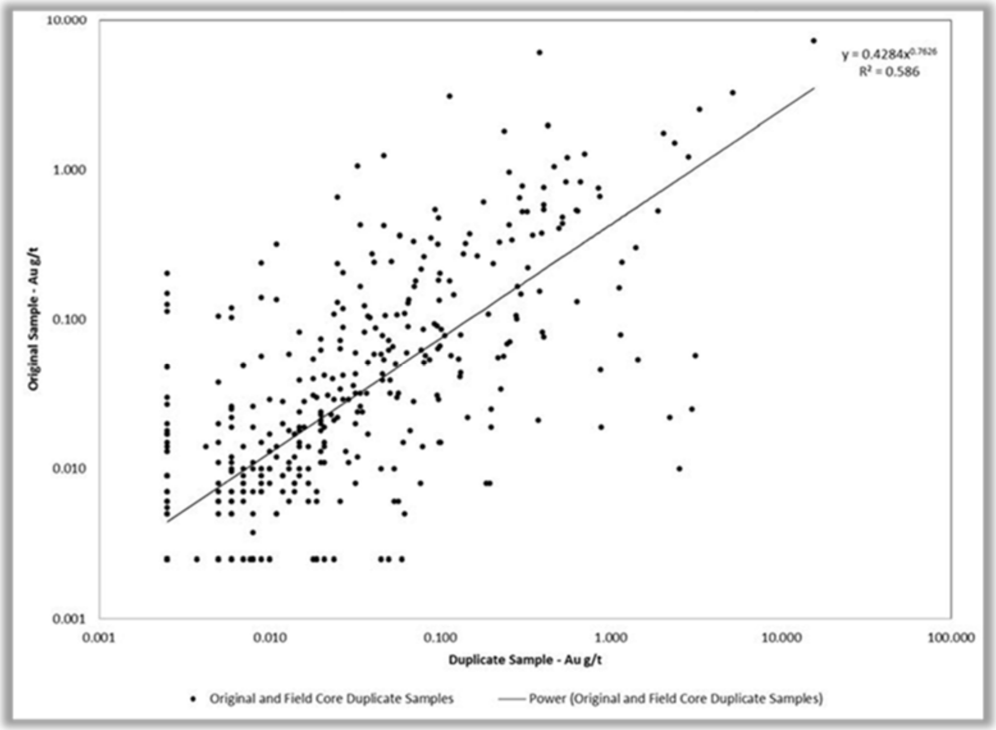
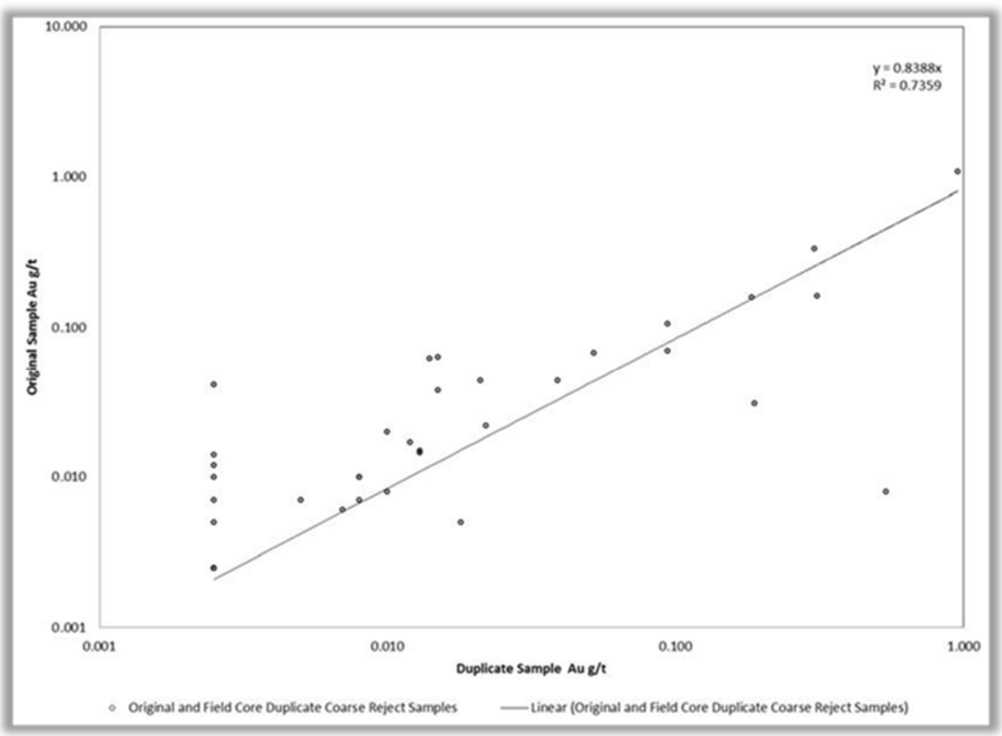


Figure 11.20 2011 Core Field Duplicate Samples - Gold Results (N=38)



11.3.7 2010 – 2011CHECK SAMPLE PROGRAM

In addition to scheduled analysis of duplicate splits of core sample pulps by ALS, Orex carried out a check sample program in 2010 and 2011 that was based on submission of prepared splits from coarse reject material from original samples and from several core field duplicate samples to Laboratoire Expert Inc., in Rouyn-Noranda, QC. This laboratory was not accredited according to ISO/IEC Guideline 17025 by the Standards Council of Canada at that time but it is a reputable, independent commercial firm providing analytical services to the exploration and mining industries. Results for a total of 349 pulps from original sample coarse reject material and 11 of core field duplicate materials for 2010 were reviewed for this report and are presented below on Figures 11.21 and 11.22.

The larger coarse reject population shows moderate correlation between analyses (R^2 value of 0.8419), while the very small core pulp split group shows poor correlation (R^2 value of 0.6247).

Figure 11.21 2010 Check Samples - Gold Results (N=349)

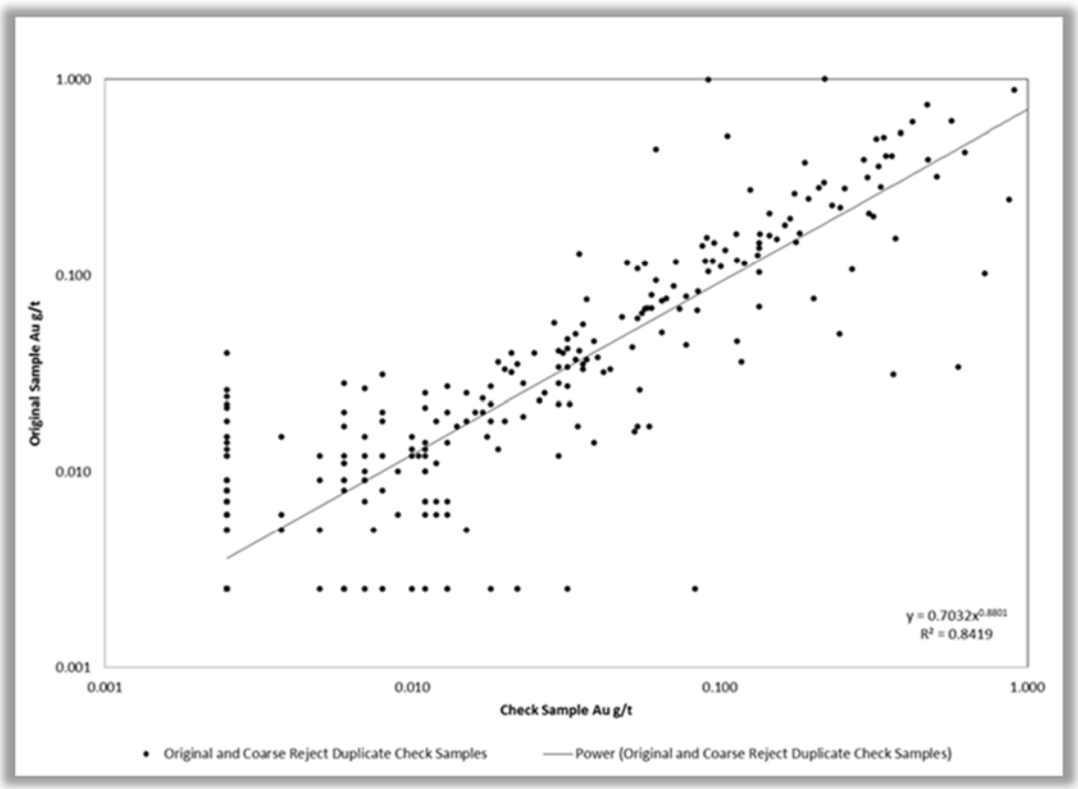
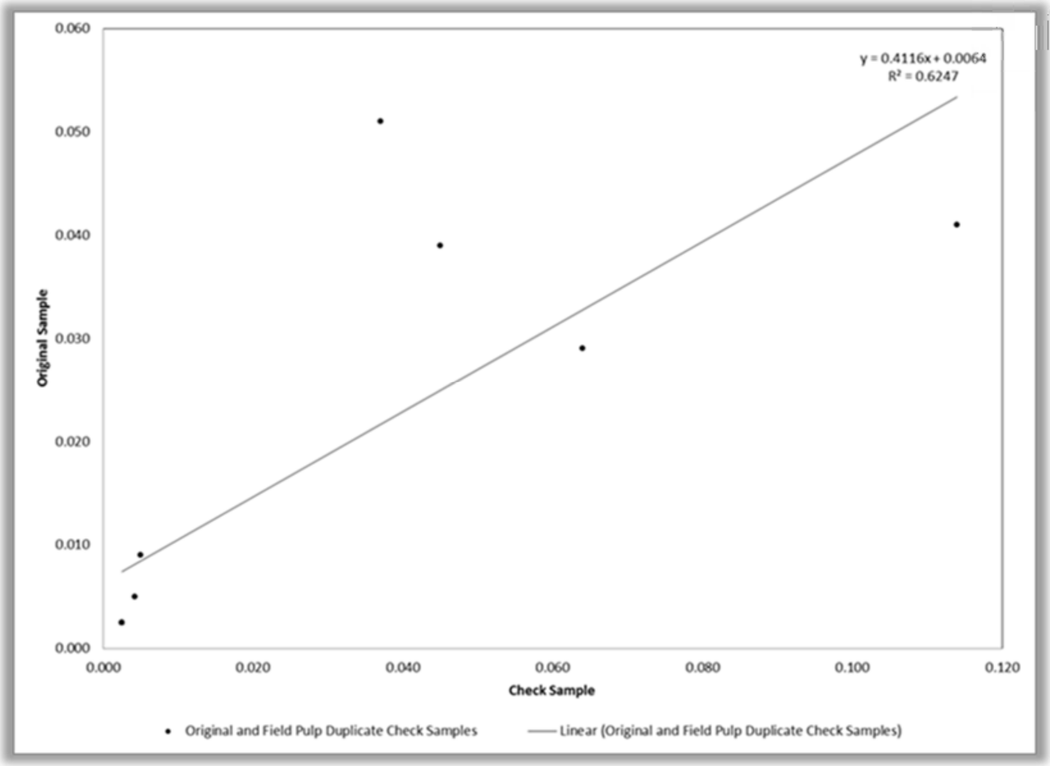


Figure 11.22 2010 Core Field Duplicate Check Samples - Gold Results (N=11)



11.3.8 2017 – 2018 ANACONDA CHECK PROGRAM

During the 2017 and 2018 drill programs a total of 738 samples were analyzed at ALS on pulps from initial fire assay and metallic screen analyses completed at Eastern Analytical (Figure 11.23). Check assays were completed on all samples assaying over 0.5 g/t gold. The pulp for the check assay was from the fine fraction of the metallic screen sample prepared at Eastern. The fine fraction split from the metallic screen assays shows a poor correlation with initial fire assay (R^2 value of 0.5643) likely owing to the nugget effect and natural erratic distribution of gold at the sample scale.

A total of 993 samples that had fire assays greater than 0.5 g/t gold were analyzed at Eastern Analytical via metallic screen fire assay. Overall the metallic screens show a poor correlation with initial fire assay (R^2 value of 0.6181) likely owing to the nugget effect and natural erratic distribution of gold at the sample scale (Figure 11.24).

Figure 11.23 2017-2018 Check Samples - Gold Results (N=738)

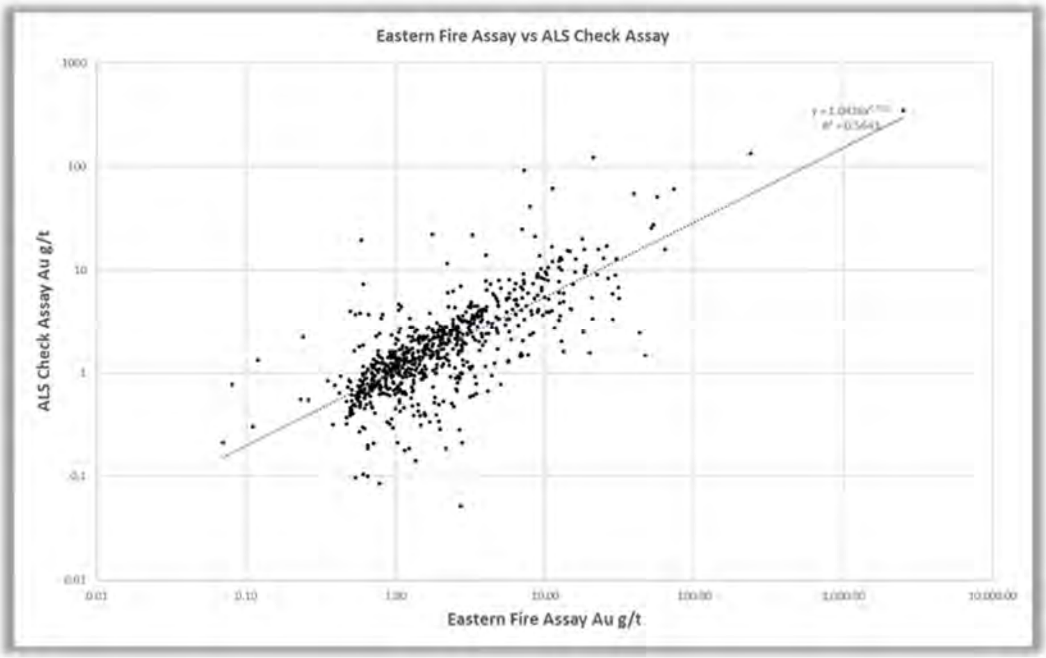
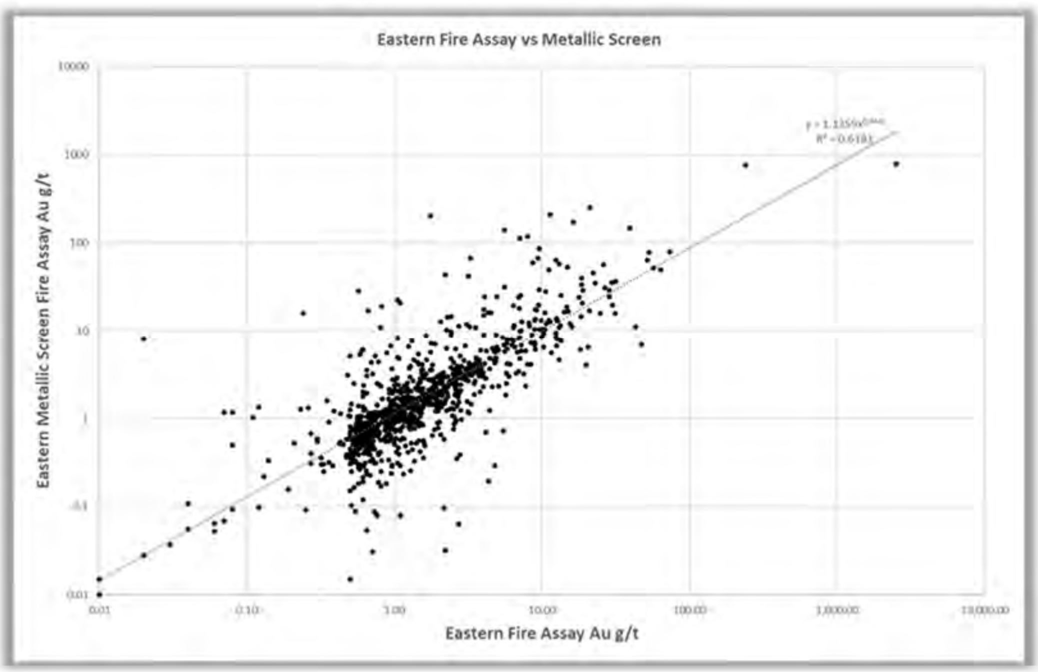


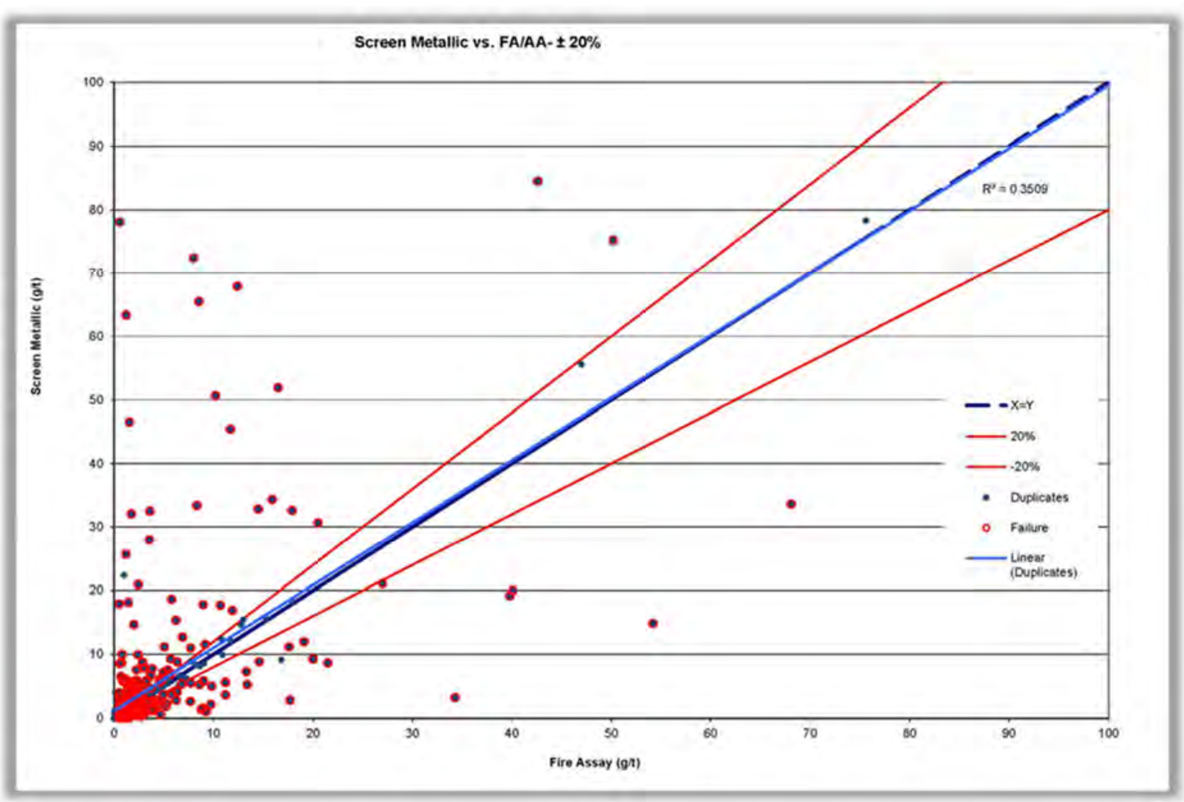
Figure 11.24 2017-2018 Eastern Metallic Screen Fire Assay versus Fire Assay (N=993)



11.3.9 2018 – 2019 SCREEN METALLIC VS. FA/AA

A total of 835 samples that had fire assays greater than 0.5 g/t gold were analyzed at Eastern Analytical via metallic screen fire assay. Overall the metallic screens show a poor correlation with initial fire assay (R^2 value of 3509) likely owing to the nugget effect and natural erratic distribution of gold at the sample scale (Figure 11.25).

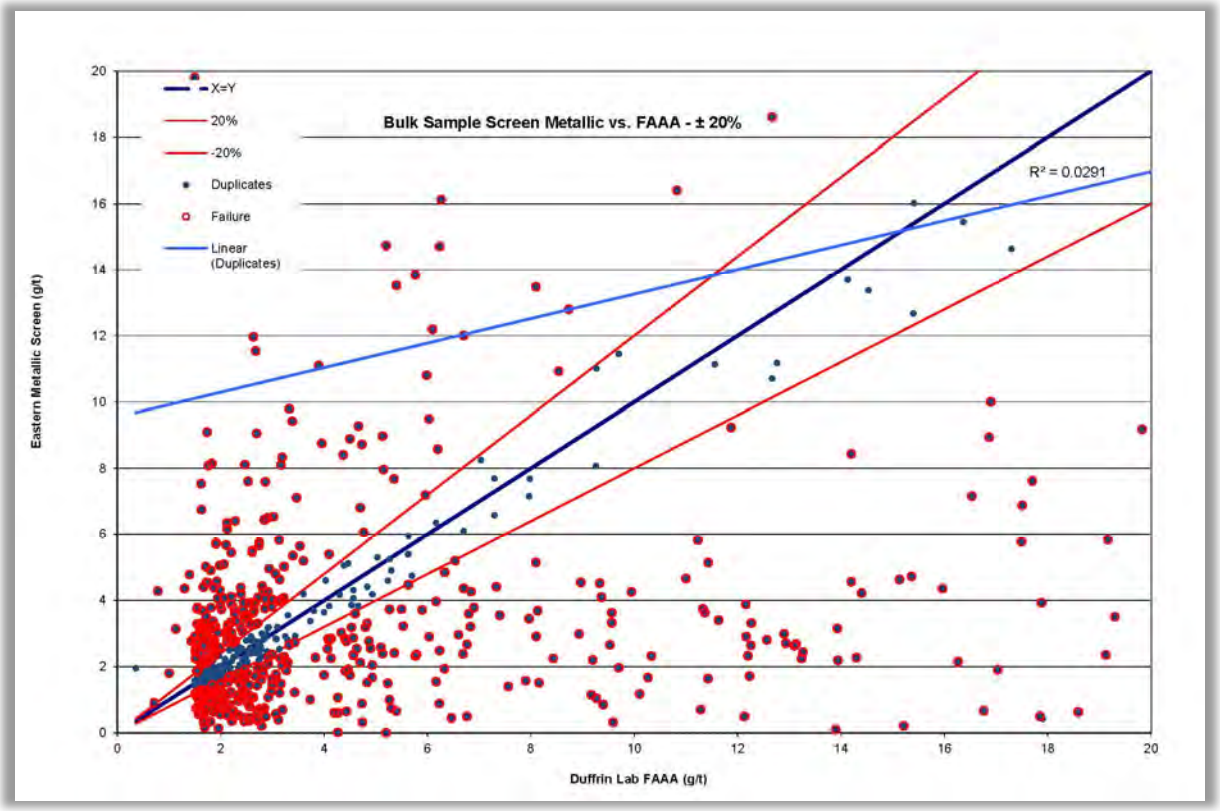
Figure 11.25 2018 – 2019 Drill Core Screen Metallic Assay vs. Fire Assay (N=835)



11.3.10 2018 – 2019 BULK SAMPLE SCREEN METALLIC VS. FA/AA

A total of 767 samples that had fire assays greater than 0.5 g/t gold assayed as the Dufferin Mine Lab were analyzed at Eastern Analytical via metallic screen fire assay. Overall the metallic screens show a poor correlation with initial fire assay (R^2 value of 0.029) likely owing to the nugget effect and natural erratic distribution of gold at the sample scale (Figure 11.26).

Figure 11.26 2018-2019 Bulk Sample Screen Metallic Assay vs. Fire Assay (N=769)



11.4 QP'S OPINION

The QP is of the opinion that sample preparation, analyses, and security methodologies employed at the Project by Anaconda, its predecessors, and related parties during the period 1984 through 2011 reflect evolving industry standards and are generally acceptable and sufficient for such a project.

With respect to results from the 2017 - 2019 Anaconda drilling program, the QP is of the opinion that associated QA/QC protocols reflect current industry standards and that data associated with such drilling would be acceptable for use in future resource estimation programs.

12 DATA VERIFICATION

12.1 REVIEW AND VALIDATION OF PROJECT DATASETS

WSP reviewed the validation conducted by Mercator and agrees with the findings.

Core sample records, lithologic logs, laboratory reports, and associated drillhole information for all drill programs completed in the 1984 to 2011 period were digitally compiled for use in Geovia-Surpac™ deposit modeling software. This information was sourced in Microsoft® Excel spreadsheet (.xls) files provided by Orex that had been created from Project records. Orex also provided access to the Project's drillhole logging database that contained data from the most recent drilling campaigns. Information pertaining to the exploration history of the Property area was also made available to Mercator, primarily as electronic document copies, but also as hard copy bound reports and files that constitute part of the Project archives.

Historic and current drilling program information was reviewed and digital records of historic drilling were checked for both consistency and accuracy against original source documents available through NSDNR or received from Orex. Parameters reviewed in detail include drillhole collar coordinates, downhole survey values, hole depths, sample intervals, assay values, and litho-codes. This included records of underground and surface drilling. All 2010 and 2011 drillhole coordination and orientation data inputs were checked, and validation of approximately 20% of the assay dataset for sample interval and assay value information against corresponding source documents was carried out.

Manual review and checking of logging and sample records showed generally good agreement between original records and digital database values for all datasets. No historic or current drillholes were excluded from the resource drilling database due to lack of support records but several items attributed to data entry errors were identified, and necessary changes were made in the database. These were identified in a validation comments field of the review drilling database.

WSP validated the data from the 2017 – 2018 and 2018 – 2019 programs.

After completion of all manual record checking procedures, the drilling and sampling database records were further assessed through digital error identification methods available through the Surpac™ modeling software. This provided a check on items such as sample record duplications, end-of-hole errors, survey and collar file inconsistencies, and certain potential litho-code file errors. The digital review and import of the manually checked datasets through Surpac™ provided a validated Microsoft® Access database that WSP considers to be acceptable with respect to support of the resource estimation program described in this report.

12.2 SITE VISITS BY WSP

12.2.1 *SCOPE OF VISIT ACTIVITIES*

Ms. Darlene Nelson, P.Eng., Senior Mining Engineer with WSP carried out a site visit to the Goldboro Property from June 20 to 24, 2018 inclusive and again from July 8 to 10, 2019 inclusive. Mr. Todd McCracken, P. Geo., Manager – Mining with WSP visited the Goldboro site on October 29, 2018 and again from July 8 to 10, 2019 inclusive.

General observations regarding the character of the landscape, surface drainage, road and drill pad features, drill sites, site reclamation, historic shaft reclamation features, and physical facilities of the Property were made during the site visit and these were photographically documented. During this time, it was observed that while most drill collars are marked by wooden posts (Figure 12.1), these are commonly not well labelled, or written labels have faded from exposure. The drill sites from the 2017-2018 program have been reclaimed as required by provincial permits. No casing or hole marker exists on the drill sites. The drill sites from the 2018 – 2019 program were available to review during the 2019 site visit (Figure 12.2). The collars from the 2018 – 2019 program would be pulled by year end as required by the provincial permit.

On the east side of the Property, several open historic mine shafts have been backfilled, and access roads associated with this work were visible. These sites are well marked with flagging tape and new Nova Scotia Department of Natural Resources warning signs (Figure 12.3).

Figure 12.1 Drill Collar Marker – prior to 2018 Program

Figure 12.2 Drill Collar Marker – 2018 – 2019 Program

Figure 12.3 Historic Shaft Backfilled by NSDNR

12.2.2 DRILL CORE CHECK SAMPLING PROGRAM

WSP did not collect quarter core from Goldboro. In a high nugget gold project, quarter core can provide deceptive results. Since most of the Anaconda Goldboro samples are analyzed by screen metallic, there are not enough course rejects to sample and the pulps are only of the fine fraction.

WSP has reviewed the results completed by Mercator in 2013 and agrees with the findings.

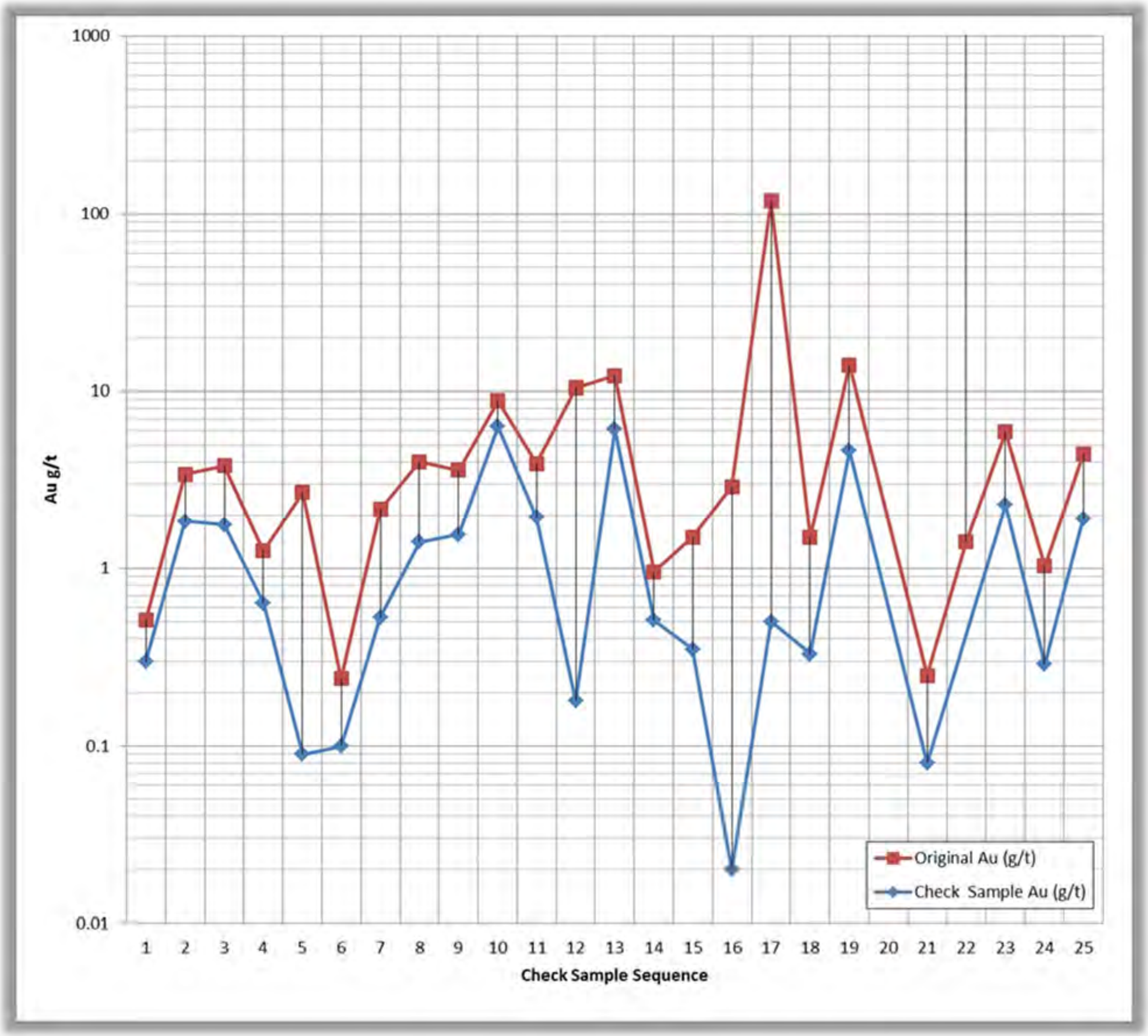
Mercator collected 22 quarter-core check samples of archived drill core from the 2010 and 2011 drilling campaigns and these were submitted to SGS for gold analysis using fire assay fusion-atomic absorption and screen metallics processing techniques. One blind blank sample and two certified reference material samples were also submitted for quality control purposes.

Mercator core check sample gold results are compared to original drill program results shown on Figure 12.4, and a correlation coefficient of 0.46 applies to the dataset after removal of an outlier sample having an original value of 117.5 g/t gold and a check sample gold result of 0.5 g/t (the associated scatter plot is not included). Inspection of the data pairs shows that correlation is best for gold grades below 5 g/t, with deterioration of the correlation occurring above that level.

Check sample program results for the 22 quarter core samples and 2 QA/QC samples are interpreted as confirming the general mineralized character of the core intervals tested, with new data showing a low bias in most cases. Results also indicate that gold distribution within drill core at the scale of quarter core subsamples is heterogeneous. This is considered a reflection of 'nugget-effect' that is a well-documented characteristic of gold mineralization on the Goldboro Property. Another contributing factor to poor correlation between gold results for the original core samples and check samples is relative sample size. Original samples were twice the size of the check samples and therefore had a greater opportunity to contain coarse-grained gold particles.

Results of the January 2013 independent QA/QC program to be as expected, recognizing that a coarse gold component is present on the Property and that metallurgical test results have consistently shown drill core sample assays to understate global gold grades.

Figure 12.4 Mercator Drill Core Check Sample Program Results N=24



12.2.3 DRILL COLLAR COORDINATE CHECKING

WSP was not able to check the specific collar coordinated for the 2017-2018 or program as the drill casing have been pulled. WSP did inspect several of the drill sites using a Garmin GPSmap 62 hand-held GPS unit. The sites inspected in the East Goldbrook area match the coordinated in the resource database.

WSP has reviewed the results completed by Mercator in 2013 and agree with the findings.

Collar coordinates for 17 drillholes completed on the Goldboro Property were checked by Mercator during the January 2013 site visit. A Garmin Map 60Cx hand-held GPS unit was used to collect collar coordinate check values and these were then compared to validated resource database collar values. Excellent correlation exists between the two datasets with respect to UTM Easting and Northing values, with the total range in Easting variation being -1.5 m and +2.35 m and the total range for Northing being -2.19 m and +4.16 m. Values in the collar check elevation dataset range between -6.41 m and +9.67 m of corresponding database collar values and, as expected, provide only an order of magnitude check on project database values. The differential GPS methods used for drillhole pickup are considered more accurate. Tabulated results of the drill collar checking program are presented in Table 12.1.

Table 12.1 Mercator Drill Collar Coordinate Checking Results *

Hole ID	GPS Easting (m)	GPS Northing (m)	GPS Elevation (m)	Database Easting (m)	Database Northing (m)	Database Elevation (m)
BR08-42	606390	5006360	67.40	606389.91	5006357.54	66.65
BR08-44	606369	5006352	73.90	606367.03	5006351.97	69.18
OSK10-12	606486	5006325	65.30	606487.50	5006322.51	67.40
OSK10-14	606465	5006312	66.00	606464.34	5006309.40	68.25
OSK10-16	606435	5006314	62.40	606435.88	5006312.83	66.40
OSK10-19	606412	5006311	65.50	606412.74	5006309.24	66.48
OSK10-26	606312	5006382	70.10	606311.23	5006381.06	70.12
OSK10-29	606263	5006374	74.10	606260.65	5006376.19	71.57
OSK10-32	606212	5006377	76.80	606210.39	5006379.17	73.08
OSK10-35	606163	5006403	82.10	606161.20	5006404.38	75.58
OSK10-39	606120	5006401	86.40	606119.93	5006402.55	76.73
OSK11-01	607452	5006173	58.80	607451.15	5006168.84	59.14
OSK11-02	607451	5006074	52.30	607449.81	5006073.05	58.71
OSK11-03	607698	5006130	63.60	607696.22	5006126.65	67.50
OSK11-04	607695	5006028	66.20	607695.44	5006023.89	65.22
OSK11-05	607952	5006101	66.90	607950.63	5006098.13	70.60
OSK11-06	607951	5006019	70.80	607949.13	5006015.58	69.30

**UTM Zone 20 NAD 83 Coordination; sea level elevation datum*

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 SUMMARY

The objective of the 2019 feasibility level metallurgical study was to quantify the metallurgical response of ores from deposits in the Goldboro project. This study focused on Boston-Richardson and East Goldbrook deposits. The program was designed with the intent to develop the parameters for process design criteria for comminution, gravity concentration, leaching, carbon adsorption, cyanide destruction and carbon elution, and gold refining in the process plant. The metallurgical program was conducted at Base Metallurgical Laboratories Ltd., Kamloops, BC. The results and data is contained within "BL0395 Report #1 July12, 2019" (*Lang, J., Angove, B., 2019)*. Interpretation of the SMC comminution tests are found in "SMC Test Report for JKTech Job No. 19008/P3" dated February 2019 (*Weier, 2019*). The metallurgical program was performed on the following composites:

— Boston-Richardson: low grade, medium grade and high grade;

— East Goldbrook: marginal grade, low grade, medium grade and high grade.

The composites were selected by Anaconda geologists with input from Ausenco to represent the spatial distribution, head grades, and mineralization types of the Goldboro Project.

Boston-Richardson samples were tested through a suite of grinding characterization tests including Bond low energy, rod mill and ball work index, and Bond abrasion index tests. The Boston-Richardson samples were selected to complete SMC tests. The East Goldbrook samples were used for Bond ball mill work index variability tests.

The program included grind optimization, gravity concentration, leaching studies on the Boston-Richardson composites. The East Goldbrook composites were tested as variability samples to evaluate the optimum conditions determined through the Boston-Richardson tests. An overall Master Composite was assembled for bulk leach tests to provide feed material for cyanide destruction testing and carbon adsorption testing. Tailings solids samples were then used for environmental testing. An overall recovery model was derived from the test data for use in process design and mine scheduling.

13.2 SAMPLE DESCRIPTION

13.2.1 PEA SAMPLES

Anaconda collected diamond drill core samples for metallurgical testing in June 2017 from BR-17-01 to BR-17-05. Half core was provided for the test program. Based on drill core assay results, Anaconda provided a selection of samples from these drillholes that graded above 0.5 g/t gold. The samples were selected to represent the Boston-Richardson and East Goldbrook Zones, and were assembled into one overall composite for the testing program. The sample had a calculated average head grade of 3.11 g/t Au from all tests in the program. This compares to direct assays of screened metallics average assay of 2.35 g/t Au and 2.51 g/t Au from fire assaying. The samples and test results are described in the Thibault and Associates Inc. Report, (Project Number 6429 Phase I) March 8, 2018.

13.2.2 FEASIBILITY STUDY SAMPLES

Samples for the feasibility study test program were obtained from a combination of face samples from underground workings for Boston-Richardson and from available diamond drill core samples for East Goldbrook. Sample selection was by Anaconda geologists with input from Ausenco. The Boston-Richardson samples were taken from Levels 2 and 3 in Belt 1. East Goldbrook samples were from Belts 1 to 7. Samples from each zone were assembled into grade composites for the testing program (Anaconda memo to Base Metallurgical Laboratories, January 22, 2019). Information on the samples from the referenced document is provided in Table 13.1.

Boston-Richardson (BR) composites were used for:

- Comminution tests (SMC tests, Bond crusher, rod mill and ball mill work indices, abrasion index);
- Extended gravity recoverable gold (E-GRG) tests;
- Grind – leach optimization tests;
- Cyanide destruction tests as part of the Master Composite sample.

East Goldbrook (EG) composites were used for:

- Comminution variability tests (Bond ball mill work index);
- Gravity – leach variability tests.
- Cyanide destruction tests as part of the Master Composite sample.

For the bulk leach tests required for cyanide destruction testing and carbon loading tests, a large composite (Master Composite) sample was assembled, based on Anaconda instructions. The proportion of each composite was based on the resource model to provide a sample with containing 6.15 g/t gold, close to the average grade in the mine schedule of 5.13 g/t Au. The composition is shown in Table 13.2.

Table 13.1 Feasibility Study Metallurgical Test Program Samples

Zone	Bucket #	Location	Grade		Lithology	Zone	Composite
Boston-Richardson	3	Level 2 West R11	Low Grade	1-2 g/t	4WQ, 4AQ, QV, 1-3% asp	BR, Belt 1	Low Grade (LG)
	4	Level 2 West R11	Low Grade	1-2 g/t	QV, 4A, 4AQ, 4W,2-5% asp, VG	BR, Belt 1	
	1	Level 2 West R13	Medium Grade	3-5 g/t	4A, 4W, QV, 4AQ, 1-5% asp, 2% py	BR, Belt 1	Medium Grade (MG)
	2	Level 2 West R13	Medium Grade	3-5 g/t	4A, QV, 4WQ, 1-5% asp	BR, Belt 1	
	5	Level 2 West R9	Medium Grade	~ 3 g/t	QV, 4A, 4AQ, 4W,2-5% asp, py	BR, Belt 1	
	6	Level 2 West R9	Medium Grade	~ 3 g/t	QV, 4A, 4W, 2-5% asp, py	BR, Belt 1	
	7	Level 3 East R00	High Grade	> 7 g/t	QV, 4A, 4AQ, 4QW, 2-5% asp, VG	BR, Belt 1	High Grade (HG)
	8	Level 3 East R00	High Grade	> 7 g/t	QV, 4A, 4AQ, 4QW, 2-5% asp, VG	BR, Belt 1	
East Goldbrook	n/a	Composited from 42 samples from drillholes BR-17-04, 05 and BR-18-68 to 71	Marginal	0.73 g/t	QV, 4A, 4AQ, 4W, 2-5% asp, py	EG Belts 1 to 7	Marginal Grade (EG-1)
	n/a	Composited from 39 samples from drillholes BR-17-04, 05 and BR-18-68 to 71	Low Grade	1.59 g/t	QV, 4A, 4AQ, 4W, 2-5% asp, py	EG Belts 1 to 7	Low Grade (EG-2)
	n/a	Composited from 19 samples from drillholes BR-17-04, 05 and BR-18-68 to 71	Medium Grade	4.06 g/t	QV, 4A, 4AQ, 4W, 2-5% asp, py	EG Belts 1 to 7	Medium Grade (EG-3)
	n/a	Composited from 20 samples from drillholes BR-17-04, 05 and BR-18-68 to 71	High Grade	24.5 g/t	QV, 4A, 4AQ, 4W, 2-5% asp, py	EG Belts 1 to 7	High Grade (EG-4)

Table 13.2 Composition of Master Composite Sample

Composite		Ratio (% of Overall)
Boston Richardson	LG	13
	MG	31
	HG	13
East Richardson	EG-1	20
	EG-2	13
	EG-3	10
Total		100

Calculated head grades from four bulk gravity/leach tests was 5.05 g/t gold. A fifth bulk leach test returned a calculated head grade of 142.9 g/t gold due to a large piece of gold in the gravity concentrate and was not included in this average.

Screened metallics gold assays were done on the composites due to the occurrence of coarse free gold. Large samples (1 kg) of each composite were pulverized and then screened at 105 µm with the oversize and undersize fractions assayed separately. The head grade was calculated from the weighted assays from the two fractions; the results are shown in Table 13.3. The results clearly show the presence of coarse free gold, particularly with the BR samples. Most of the composites (with the exception of EG-1 and EG-4) showed far more gold in the +105 µm than the mass contained in this fraction, which was between 2.9% and 3.5%. BR composite LG had 77.5% contained in the +105 µm fraction with the fraction assaying 248 g/t Au.

Table 13.3 Results of Screened Metallics Head Grades Gold Assays

Composite	Calc. Head Grade (g/t Au)	+105 µm Fraction		-105 µm Fraction		Distribution (% Au)	
		Mass (%)	Assay (g/t Au)	Mass (%)	Assay (g/t Au)	+105 µm	-105 µm
LG	11.1	3.5	248	96.5	2.58	77.5	22.5
MG	3.07	3.5	34.9	96.5	1.91	40.1	59.9
HG	12.0	3.1	230	96.9	5.03	59.4	40.6
EG-1	2.62	3.2	1.85	96.8	2.65	2.2	97.8
EG-2	2.34	2.9	23.3	97.1	1.71	29.2	70.8
EG-3	4.84	3.4	85.3	96.6	1.99	60.3	39.7
EG-4	0.62	3.3	0.73	96.7	0.62	3.9	96.1

Head grades for relevant elements and compounds are shown in Table 13.4. Silver assays are not economically significant and are not modelled in the resource model. Sulphur occurs as sulphide sulphur and is associated predominantly with arsenopyrite. Copper is at very low levels (typically < 0.005%) such that the impact of cyanide-soluble copper on consumption is not significant Carbon as total organic carbon (TOC) and graphite (Cg) is visible in the samples and occurs at concentrations that may cause preg-robbing issues in leaching. However, the graphite is not active and showed no interference with leaching.

Table 13.4 Head Assays

Composite	Assay							
	Ag (g/t)	Cu (%)	Fe (%)	As (%)	S= (%)	C (%)	TOC (%)	Cg (%)
LG	1	0.003	2.91	0.40	0.46	0.18	0.09	0.06
MG	<1	0.001	2.98	1.37	0.84	0.39	0.26	0.17
HG	1	<0.001	2.08	1.22	0.66	0.37	0.15	0.10
EG-1	2	0.005	3.76	0.62	0.63	0.35	0.27	0.18
EG-2	2	0.003	3.63	0.63	0.56	0.18	0.12	0.07
EG-3	2	0.003	2.84	0.39	0.41	0.21	0.18	0.12
EG-4	2	0.003	3.02	0.30	0.36	0.14	0.09	0.05

13.3 MINERALOGY

The PEA testing program did not include mineralogical studies.

Representative samples of each composite were submitted for a feed mineralogy study to determine major mineral species. The study was completed by QEMSCAN (quantitative mineralogy) with the primary intention to collect mineral abundance data. QEMSCAN data was collected by Bulk Mineralogical Analysis (BMA). Mineral abundance information is provided in Table 13.5.

Key observations related to mineral abundance include:

— Sulphides are present as arsenopyrite, pyrrhotite and pyrite;

— Non-sulphide gangue is predominately comprised of quartz, plagioclase, biotite and muscovite;

— Clay content is low in the ~ 0.3% range.

Table 13.5 Mineral Abundance Summary

Modal	Composite						
	LG	MG	HG	EG1	EG2	EG3	EG4
Arsenopyrite	0.65	3.26	3.01	1.29	1.36	0.86	0.37
Pyrrhotite	0.48	0.14	0.07	0.15	0.44	0.12	0.57
Pyrite	0.09	0.13	0.09	0.43	0.06	0.15	<0.1
Chalcopyrite	<0.1	<0.1	0.15	<0.1	<0.1	<0.1	<0.1
Galena	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Sphalerite	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
FeNi(Co)-Sulpharsenide	<0.1	<0.1	<0.1	0.00	<0.1	<0.1	<0.1
Iron Metal	0.19	0.12	0.32	0.06	0.15	<0.1	0.05
Goethite	<0.1	<0.1	0.21	<0.1	<0.1	<0.1	<0.1
Ilmenite	0.24	0.06	<0.1	0.42	0.31	0.21	0.32
Quartz	45.8	55.4	66.4	39.8	49.1	43.4	41.8
Muscovite/Illite	16.8	16.5	10.6	17.1	14.7	21.6	15.0
Plagioclase Feldspar	14.6	6.19	4.95	12.8	11.2	13.0	18.7
Biotite/Phlogopite	14.0	10.2	5.12	14.7	13.6	9.95	12.5

(table continues on next page)

Modal	Composite						
	LG	MG	HG	EG1	EG2	EG3	EG4
K-Feldspars	3.15	3.06	3.22	6.68	5.18	6.35	5.75
Chlorite	2.60	2.37	3.69	4.40	2.94	3.01	3.80
'Kaolinite' (clay)	0.33	0.34	0.27	0.12	0.27	0.31	0.29
Calcite	0.36	0.58	0.95	1.00	0.14	0.12	0.12
Sphene	0.10	0.84	0.25	0.30	0.09	0.20	0.13
Apatite	0.27	0.21	0.21	0.45	0.29	0.27	0.31
Amphibole (Actinolite)	0.14	<0.1	<0.1	<0.1	0.06	0.06	0.07
Zircon	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	0.07
Others	0.05	0.13	0.26	<0.1	<0.1	0.08	<0.1
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**

The predominant sulphide mineral is arsenopyrite.

13.4 COMMINUTION TESTING

13.4.1 PEA TESTING

PEA testing was completed on the PEA sample used for metallurgical testing and included the Bond rod and ball mill testing at standard closing screen apertures; the results are shown in Table 13.6. The Bond ball mill work index test was closed at 150 µm and the product size distribution was 80% passing 111 µm.

Table 13.6 PEA Comminution Testing Results

Test	Units	Result
Abrasion Index	g	0.267
Bond Rod Mill Work Index	kWh/t	15.8
Bond Ball Mill Work Index	kWh/t	14.8

13.4.2 FEASIBILITY STUDY TESTING

Samples were subjected to Bond Low Energy Impact Crushing Work Index (CWI), SMC, Bond Ball Mill Work Index (BWI) and Abrasion (Ai) testing according to the following scope:

— Integrated JK drop-weight and SMC tests on the three BR composites;
— Bond rod mill grindability and low-energy impact test on the three BR composites;
— Bond ball mill grindability test on the BR and EG composites;
— Bond abrasion test on the three BR composites.

The SMC tests were prepared to -31.5 +26.5 mm prior to rock selection. The BWI grindability tests were completed with a closing screen size of 150 µm. Bond Abrasion (Ai) tests were completed as per the standard Bond Ai procedure. The summary of the comminution test results is shown in Table 13.7.

Table 13.7 Summary of Comminution Test Results (from February 2019 JKTech Report)

Sample ID	Relative Density	JK Data SMC					CWI Parameters			RWI parameters					BWI parameters				Bond Ai
							Work Index	Density	Mesh of Grind	F80	P80	Gram/rev	Work Index	Mesh of Grind	F80	P80	Gram/rev	Work Index	Ai
	SMC	A	b	A x b	t_a	DWI (kWh/m³)	kWh/t	g/cm³		µm	µm		kWh/t		µm	µm		kWh/t	(g)
LG Comp	2.72	61.3	0.60	36.8	0.35	7.41	-	-	16	9 366	937	8.50	15.8	100	1 743	114	1.58	15.2	0.248
MG Comp	2.77	61.6	0.81	49.9	0.47	5.56	9.86	2.78	16	6 916	951	10.5	15.2	100	1 953	115	1.50	15.7	0.219
HG Comp	2.59	52.6	1.51	79.4	0.79	3.26	7.03	2.64	16	7 000	933	11.8	13.8	100	1 907	120	1.37	17.5	0.313
EG1	-	-	-	-	-	-	-	-	-	-	-	-	-	100	1 910	110	1.45	15.7	-
EG2	-	-	-	-	-	-	-	-	-	-	-	-	-	100	1 979	111	1.47	15.7	-
EG3	-	-	-	-	-	-	-	-	-	-	-	-	-	100	1 903	114	1.40	16.6	-
EG4	-	-	-	-	-	-	-	-	-	-	-	-	-	100	1 982	110	1.40	16.1	-

Variability: Overall Statistics

	SMC	A	b	A x b	t_a	DWI	CWI Work Index	Density	Mesh of Grind	RWI F80	RWI P80	Gram/rev	RWI Work Index	Mesh of Grind	BWI F80	BWI P80	Gram/rev	BWI Work Index	Ai
Average	**2.69**	**58.5**	**0.97**	**55.4**	**0.54**	**5.41**	**8.45**	**2.71**		**7 761**	**940**	**10.3**	**14.9**		**1 911**	**113**	**1.45**	**16.1**	**0.260**
Std. Dev.	0.09	5.1	0.48	21.8	0.23	2.08	2.00	0.10		1 391	9.45	1.66	1.03		81	4	0.07	0.8	0.048
Rel. Std. Dev.	3.45	8.7	49.0	39.5	42.4	38.4	23.7	3.65		17.9	1.01	16.1	6.87		4	3	4.96	4.7	18.5
Minimum	2.59	52.6	0.60	36.8	0.35	3.26	7.03	2.64		6 916	933	8.50	13.8		1 743	110	1.37	15.2	0.219
10th Percentile	2.62	54.3	0.64	39.4	0.37	3.72	7.31	2.65		6 933	934	8.90	14.1		1 839	110	1.39	15.5	0.225
25th Percentile	2.66	57.0	0.71	43.3	0.41	4.41	7.74	2.68		6 958	935	9.49	14.5		1 905	111	1.40	15.7	0.233
Median	**2.72**	**61.3**	**0.81**	**49.9**	**0.47**	**5.56**	**8.45**	**2.71**		**7 000**	**937**	**10.5**	**15.2**		**1 910**	**114**	**1.45**	**15.7**	**0.248**
75th Percentile	2.75	61.5	1.16	64.7	0.63	6.49	9.15	2.75		8 183	944	11.1	15.5		1 966	115	1.49	16.4	0.280
90th Percentile	2.76	61.5	1.37	73.5	0.73	7.04	9.58	2.77		8 893	948	11.5	15.7		1 980	117	1.53	17.0	0.300
Maximum	2.77	61.6	1.51	79.4	0.79	7.41	9.86	2.78		9 366	951	11.8	15.8		1 982	120	1.58	17.5	0.313

The composites tested fell in the range of 37 to 79 for Axb with an average of 55 and a 90th percentile of 39 (Axb increases in hardness as the value decreases so the value numerically is the 10th percentile). The Bond ball mill work index averaged 16.1 W/t and ranged from 15.2 to 17.5 kWh/t. The values span the medium to medium hard range of hardness. The Bond ball mill work values are higher than the single test from the PEA testing. The measured rock relative densities averaged 2.69 and ranged from 2.59 to 2.77. The Bond abrasion index averaged 0.26, with a maximum of 0.31. This is considered as moderately abrasive.

For comminution design, the values shown in Table 13.8 have been derived from the available test results. The design Axb, Bond rod and ball mill work indices are the 75th percentile values.

Table 13.8 Design Comminution Design Values

Parameter	Units	Value
SMC test (Axb)	-	43.3
Bond rod mill work index (RWi)	kWh/t	15.5
Bond ball mill work index (BWi)	kWh/t	16.4
Abrasion index (Ai)	g	0.26

13.5 GRAVITY GOLD RECOVERY

13.5.1 PEA TESTING

The PEA testing included a series of non-standard gravity recovery tests to assess the quantity of gravity recoverable gold in the composite sample. One series of tests using a laboratory scale Knelson concentrator showed gravity recoveries ranging from 43% to 55% at grinds ranging from 80% passing 80 µm to 150 µm. The results did not include the corresponding concentrate mass recoveries. The results are not considered for process design.

13.5.2 FEASIBILITY STUDY TESTING

The test program included two phases of gravity recovery testing: the first phase tested the BR composites with the extended gravity recoverable gold (E-GRG) protocol to determine theoretical maximum gravity recoverable gold content for each sample. The second phase included batch gravity testing of each composite to provide gravity tailings samples for leach tests.

E-GRG TESTING

As part of sample preparation, a 20 kg portion from each of the LG, MG and HG composites was stage-crushed to 100% passing 1.7 mm (80% passing or K_{80} ~ 850 µm). This material was utilized for E-GRG testing. The E-GRG test is conducted by passing the entire crushed sample through a Knelson MD-3 concentrator operating at a force of 60-Gs, the concentrate is retained and sized for assay, the tailings are sub-sampled for sizing. The tailings are ground in a laboratory rod mill and repassed (Pass 2) at a grind target K_{80} 250 µm, the concentrate and tailings are sampled as per the initial pass before regrinding (K_{80} 75 µm) the tailings and repassing (Pass 3). The final tailings are sampled, sized and each size fraction assayed. The concentrate size fractions assayed (Au fire assays) to extinction.

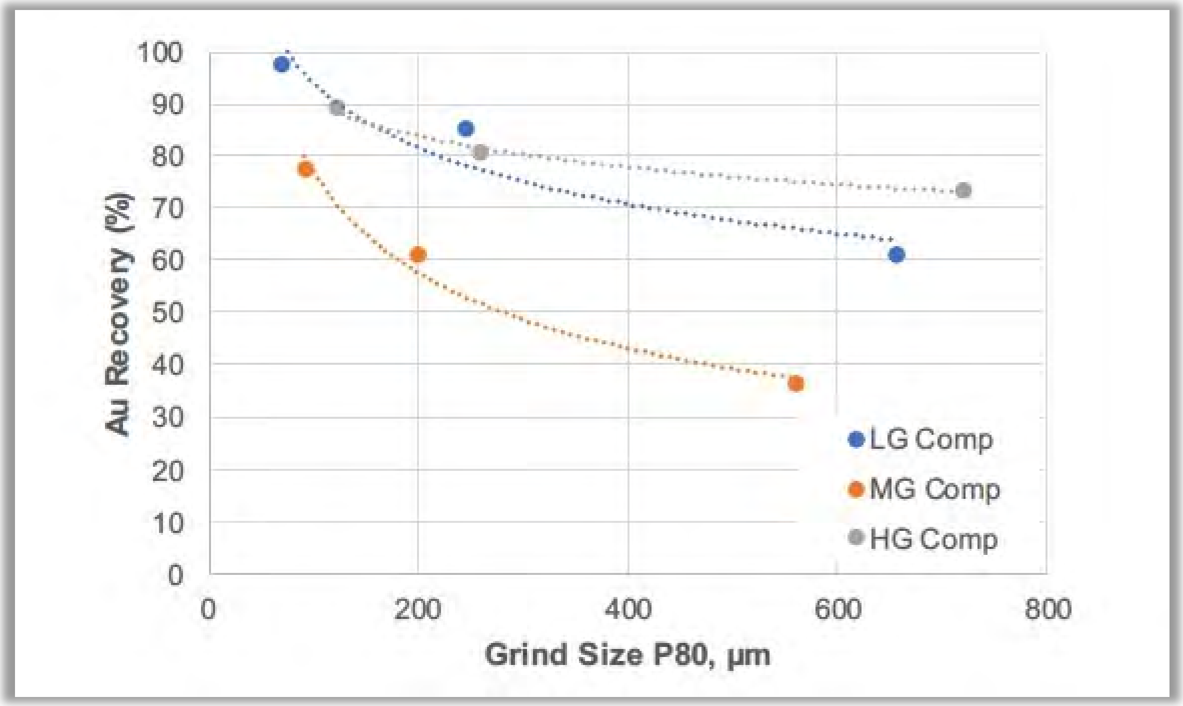
Due to sample mass limitations there was not sufficient mass available to generate a complete grind calibration for each composite that would form the basis for the 2nd and 3rd pass of the E-GRG targeting K_{80} 250 µm and 75 µm. As a substitute, grind times were estimated by stopping the grind, sizing and estimating time required to attain target grinds.

All composites generated very high GRG values with a range of 78% (MG) to 98% (LG); an overall summary is provided in Table 13.9 and on Figure 13.1. The GRG value does not directly predict or correlate gold recovery results from a closed-circuit milling operation. It is indicative of gravity gold amenability, and in this scenario, all three samples would benefit from the inclusion of a gravity circuit.

Table 13.9 E-GRG Summary, Boston-Richardson

Composite	Product	Feed Size per Stage (K_{80}, µm)	Mass (%)	Assay (g/t Au)	Distribution (% Au)
LG	Stage 1 Concentrate	656	0.43	606	61.0
	Stage 2 Concentrate	246	0.47	219	24.1
	Stage 3 Concentrate	70	0.50	109	12.6
	Tailing	-	98.6	0.10	2.3
	Combined Conc.	-	1.40	299	97.7
	Calculated Feed	-	-	4.30	-
MG	Stage 1 Concentrate	560	0.46	409	36.6
	Stage 2 Concentrate	200	0.54	231	24.2
	Stage 3 Concentrate	70	0.66	132	16.7
	Tailing	-	98.3	1.18	22.5
	Combined Conc.	-	1.66	242	77.5
	Calculated Feed	-	-	5.17	-
HG	Stage 1 Concentrate	719	0.46	3 562	73.5
	Stage 2 Concentrate	258	0.49	319	7.0
	Stage 3 Concentrate	122	0.79	245	8.7
	Tailing	-	98.3	2.45	10.8
	Combined Conc.	-	1.74	1 143	89.2
	Calculated Feed	-	-	22.3	-

Figure 13.1 E-GRG Tests Gold Recovery by Grind Size



VENDOR MODELING

The E-GRG results were provided to FLS Knelson and Sepro to model the data to provide scale-up projections to plant performance. Knelson has the largest installed number of centrifugal gravity concentrators and has an extensive data base of laboratory results with corresponding plant actual operating results. Knelson recalibrated the E-GRG results for the design grind of 80% passing 110 µm. The recalibrated results are shown in Table 13.10. The results show that a very high proportion of the gold can be recovered to a gravity concentrate with a typical plant scale mass recovery of approximately 0.05%.

Table 13.10 Recalibrated E-GRG Results

Composite	Head Grade (g/t Au)	As-Tested GRG Content (% Au)	Recalibrated GRG Content (% Au)
LG	4.50	92.4	84
MG	5.20	77.5	73
HG	22.3	89.2	74

Knelson evaluated several scenarios treating either 42% of cyclone underflow or 91% of ball mill discharge. These values correspond to capacities of selected models for the duty. A summary of the evaluations and resulting plant scale gravity gold recoveries is shown in Table 13.11. Knelson modeling shows higher recovery from treating almost the entire ball mill discharge. This has become a recent innovation in some new installations (e.g. B2Gold Otjikoto).

Table 13.11 FLS Knelson Plant Scale Modeling Results

Composite	Stream Treated	Amount Treated (% of Feed Stream)	Recovery (% Au)
LG	Cyclone Underflow	42	70 – 72
	Ball Mill Discharge	91	77
MG	Cyclone Underflow	42	56 – 58
	Ball Mill Discharge	91	65
HG	Cyclone Underflow	42	62 – 63
	Ball Mill Discharge	91	68

Sepro provided plant scale modeling results for the same two flowsheet options as Knelson. The Sepro results are shown in Table 13.12. Unlike FLS Knelson, Sepro estimates the similar recoveries whether cyclone underflow or ball mill discharge is treated at the same ratio. Sepro's predicted plant scale recoveries are also higher than the Knelson modeling.

Table 13.12 Sepro Plant Scale Modeling Results

Composite	Stream Treated	Amount Treated (% of Feed Stream)	Gravity Recovery (% Au)
LG	Cyclone Underflow	40	80
	Ball Mill Discharge	40	80
MG	Cyclone Underflow	40	66
	Ball Mill Discharge	40	66
HG	Cyclone Underflow	40	80
	Ball Mill Discharge	40	80

For design purposes, the process design criteria nominates gravity gold recovery of 50% treating ball mill discharge. Leach and carbon-in-pulp (CIP) are designed on the basis of 30% Au recovery in gravity (70% Au reporting to leach and CIP).

13.5.3 BATCH GRAVITY TESTING

A single pass gravity pre-concentration was included ahead of each leach test. For each test 2 kg test, charges were ground to the target grind size, then slurried to 10% to 20% solids by weight and fed through the Knelson concentrator. The concentrate was removed and cleaned to a low-weight final gravity concentrate using a Mozley Table (C-800) Laboratory Mineral Separator. The target Mozley concentrate mass recovery was 0.05% to simulate typical plant conditions. The final concentrate was assayed to extinction for gold, the tailings were combined and split in half for leaching. The leach calculated assay was used to balance the gravity stage and assess gravity recovery. A summary of the results is presented in Table 13.13.

Table 13.13 Batch Gravity Test Results

Test	Composite	Grind (K_{80}, µm)	Calc. Head Grade (Au g/t)	Mass Recovery (%)		Mozley Conc. Grade (Au g/t)	Gravity Test Reconciliation		
				Mozley Conc.	Gravity Tailings		Leach Feed Grade (g/t Au)	Mozley Concentrate Dist. (% Au)	Gravity Tailings Dist. (% Au)
G4	LG	254	4.37	0.047	99.95	2 279	3.29	24.7	75.3
G7	LG	152	3.17	0.062	99.94	1 892	2.00	36.9	63.1
G1	LG	123	2.24	0.049	99.95	2 433	1.04	53.6	46.4
G8	LG	108	3.34	0.113	99.89	1 112	2.08	37.8	62.2
G9	LG	72	3.71	0.100	99.90	2 466	1.25	66.3	33.7
G10	LG	63	2.24	0.087	99.91	1 154	1.24	44.8	55.2
G2	MG	152	4.47	0.115	99.88	1 877	2.31	48.5	51.5
G11	MG	141	3.96	0.081	99.92	429	3.61	8.77	91.2
G5	MG	118	4.97	0.063	99.94	1 442	4.06	18.3	81.7
G12	MG	101	3.66	0.150	99.85	350	3.14	14.3	85.7
G13	MG	83	2.48	0.314	99.69	175	1.94	22.2	77.8
G14	MG	65	2.61	0.244	99.76	431	1.56	40.3	59.7
G3	HG	215	7.23	0.060	99.94	1 507	6.33	12.5	87.5
G6	HG	159	6.66	0.127	99.87	433	6.12	8.26	91.7
G15	HG	134	21.9	0.137	99.86	618	21.1	3.88	96.1
G16	HG	121	12.9	0.121	99.88	1 966	10.5	18.4	81.6
G17	HG	99	17.1	0.227	99.77	4 596	6.72	60.9	39.1
G18	HG	65	9.38	0.320	99.68	1 491	4.62	50.9	49.1
G37	HG	120	23.9	0.060	99.94	17 932	13.1	45.2	54.8
G41	EG-1	104	2.13	0.042	99.96	803	1.79	16.0	84.0
G42	EG-2	105	3.01	0.083	99.92	1 610	1.67	44.6	55.4
G43	EG-3	80	21.0	0.034	99.97	53 202	3.15	85.0	15.0
G44	EG-4	103	0.90	0.031	99.97	226	0.83	7.68	92.3
G53A	Mast. Comp.	100	7.03	0.100	99.90	4 808	2.22	68.4	31.6
G53B	Mast. Comp	112	5.49	0.115	99.88	3 500	1.46	73.4	26.6
G55	Mast. Comp	75	4.55	0.039	99.96	6 713	1.96	57.0	43.0
G56	Mast. Comp	100	3.12	0.043	99.96	2 353	2.11	32.4	67.6
G57	Mast. Comp	150	142.9	0.057	99.94	242 233	4.24	97.0	2.97

The Goldboro composites contain high levels of free gold. No clear trend related to gold recovery by grind size is observed. In the case of test G57 which treated 2 kg of the Master Composite, a very high-grade nugget (a portion of the concentrate) was recovered assaying > 940,000 g/t Au. Future test work considerations should utilize larger starting mass for gravity of at least 10 kg. Images of large gold flakes from the HG Composite Mozley concentrate are provided on Figures 13.2 and 13.3. The large flakes shown on Figures 13.2 and 13.3 are approximately 10 mm on their longest side.

Figure 13.2 HG Composite Mozley Concentrate

Figure 13.3 HG Composite Mozley Concentrate

13.6 FLOTATION TESTING

The PEA testing included flotation testing to support flowsheet design of gravity recovery followed by flotation of gravity tailings with leaching of reground rougher flotation concentrate. Flotation of the gravity tailings produced a concentrate mass yield of 5.7% to 6.7%, grading 19.4 to 24.3 g/t gold. The combined gravity and flotation recovery of gold was 95.2% to 97.8% (flotation recovery of 94.2% to 97.8% Au).

A trade-off study completed in January 2019 resulted in the selection of the gravity recovery followed by leach flowsheet. The gravity/whole ore leach flowsheet has slightly higher operating costs, however it has slightly higher gold recovery, and therefore over the life-of-mine provides higher revenue. No additional flotation testing was included in the FS test program.

13.7 LEACH TESTING

13.7.1 PEA LEACH TESTING

The PEA leach testing included tests on flotation concentrate and gravity tailings. Cyanide leaching of flotation concentrate, reground to 80% passing18 µm or finer, resulted in 96.4% to 97.3% extraction of the contained gold within 48 hours. The gold recovery for the overall flowsheet including gravity, flotation, reground flotation concentrate, cyanide leach of the flotation concentrate, and intensive cyanide leaching of the gravity concentrate was 95.1% to 95.3% for a sample grading 4.11 g/t Au. A sample with 2.75 g/t Au achieved an overall gold recovery of 93.5% using the same process.

13.7.2 FEASIBILITY STUDY LEACH TESTING

Optimized leaching conditions were developed for the three Boston Richardson sub-composites (BR-: LG, MG, HG). These composites represented varying levels of gold grades described as Low, Mid and High Gold. The East Goldbrook (EG-comps) sub-composites were benchmarked with optimized leach conditions from the BR test work. All test work included gravity concentration prior to leaching. Cyanide leach test work evaluated the following main areas of optimization:

- Effect of grind;
- Cyanide (NaCN) concentration / Pre-aeration.

LEACHING BASELINE AND EFFECT OF GRIND

Cyanide leach testing was completed using 1 kg of gravity tailing in bottles on rolls measuring leach kinetics at specified increments of 2, 6, 24 and 48 hours, at which point the leach was terminated. The initial baseline kinetic tests were completed on each of the three BR sub-composites with standard conditions summarized below:

- Pulp Density = 50% Solids (by weight);
- Pulp pH: 11.0 (maintained);
- Dissolved oxygen concentration: sparged with gaseous oxygen;
- NaCN concentration = 1.0 g/L (maintained to 24 hours); for Master Composite tests maintained at 0.5 g/L NaCN;

— Retention Time = 48 hours (kinetic samples taken at 2, 6, 24, 48 hours);

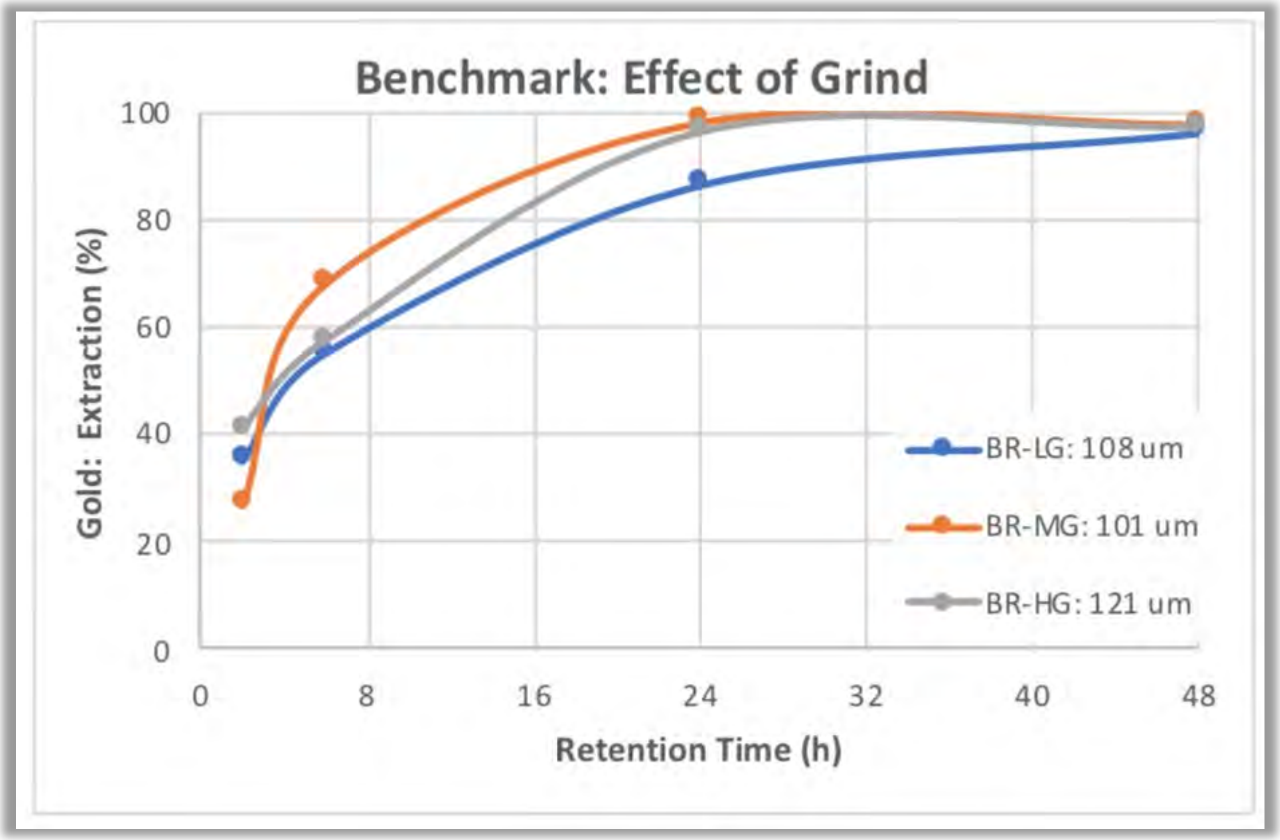
— Grind = nominal target grinds K_{80} 50 to 150 µm (5 points per composite)

Baseline and effect of grind consolidated results are shown in Table 13.14 for all grind sizes evaluated. The baseline staged leach results for each composite benchmarked around the optimal grind are shown on Figure 13.4. All samples demonstrated high levels of gold extraction, reaching a plateau near 24 hours, with the exception of the LG composite, which continued to incrementally leach up to 48 hours. The optimal grind appears to be in the range of 80% passing 100 µm to 110 µm.

Table 13.14 Summary of Baseline Kinetic Leach Tests

Test ID	Sample ID	Gravity Test ID	Grind (µm)	Leach Time (h)	NaCN	Addition (g)			Consumption (kg/t)		Head Analysis Cu	S	Cg	Au Grade Feed Stage (calc.)		Hd (dir)	Residue	Recovery (%) Grav	Leach Kinetics (hours)				Comb
					g/L	NaCN	CaO	Carb (g/L)	NaCN	CaO	ppm	%	%	Grav	Leach	g/t	g/t	Au	2	6	24	48	Au
CN22	LG Comp	G4	254	48	1.00	1.27	0.62	0.00	0.33	0.47	30	0.46	0.00	4.37	3.29	0.00	0.19	24.7	28.3	49.5	90.4	94.1	95.6
CN25	LG Comp	G7	152	48	1.00	1.28	0.61	0.00	0.38	0.45	30	0.46	0.00	3.17	2.00	0.00	0.09	36.9	36.0	59.1	86.3	95.6	97.2
CN19	LG Comp	G1	121	48	1.00	1.27	0.48	0.00	0.41	0.36	30	0.46	0.00	2.24	1.04	0.00	0.09	53.6	46.1	74.1	91.4	91.4	96.0
CN26	LG Comp	G8	108	48	1.00	1.42	0.60	0.00	0.54	0.45	30	0.46	0.00	3.34	2.08	0.00	0.07	37.8	35.2	55.4	86.6	96.4	97.8
CN27	LG Comp	G9	72	48	1.00	1.62	0.58	0.00	0.74	0.44	30	0.46	0.00	3.71	1.25	0.00	0.06	66.3	47.0	75.9	102.7	95.3	98.4
CN28	LG Comp	G10	63	48	1.00	1.74	0.51	0.00	0.93	0.37	30	0.46	0.00	2.24	1.24	0.00	0.04	44.8	35.7	72.5	95.5	96.4	98.0
CN20	MG Comp	G2	152	48	1.00	1.39	0.45	0.00	0.58	0.34	10	0.84	0.00	4.47	2.31	0.00	0.16	48.5	37.7	65.9	89.2	92.9	96.3
CN29	MG Comp	G11	141	48	1.00	1.24	0.60	0.00	0.35	0.45	10	0.84	0.00	3.96	3.61	0.00	0.12	8.8	26.6	68.0	79.0	96.7	97.0
CN23	MG Comp	G5	118	48	1.00	1.52	0.63	0.00	0.58	0.47	10	0.84	0.00	4.97	4.06	0.00	0.13	18.3	28.3	46.6	82.6	96.7	97.3
CN30	MG Comp	G12	101	48	1.00	1.31	0.78	0.00	0.47	0.59	10	0.84	0.00	3.66	3.14	0.00	0.06	14.3	27.0	68.3	98.4	98.1	98.3
CN31	MG Comp	G13	83	48	1.00	1.54	0.74	0.00	0.68	0.56	10	0.84	0.00	2.48	1.94	0.00	0.01	22.2	27.2	76.7	100.9	99.2	99.4
CN32	MG Comp	G14	65	48	1.00	1.60	0.84	0.00	0.80	0.62	10	0.84	0.00	2.61	1.56	0.00	0.04	40.3	30.4	65.6	96.3	97.2	98.3
CN21	HG Comp	G3	215	48	1.00	1.26	0.37	0.00	0.42	0.27	<10	0.66	0.00	7.23	6.33	0.00	0.80	12.5	41.0	66.3	81.5	87.3	88.9
CN24	HG Comp	G6	159	48	1.00	1.53	0.54	0.00	0.72	0.40	<10	0.66	0.00	6.66	6.12	0.00	0.30	8.3	43.1	65.6	89.4	95.1	95.5
CN33	HG Comp	G15	134	48	1.00	1.70	0.72	0.00	0.76	0.54	<10	0.66	0.00	21.9	21.1	0.00	0.46	3.9	25.7	45.2	86.1	97.8	97.9
CN34	HG Comp	G16	121	48	1.00	1.76	0.62	0.00	0.86	0.46	<10	0.66	0.00	12.9	10.5	0.00	0.27	18.4	40.9	57.5	96.5	97.4	97.9
CN38	HG Comp	G37	121	48	1.00	1.40	0.54	0.00	0.55	0.40	<10	0.66	0.00	23.9	13.1	0.00	0.46	45.2	33.6	49.4	78.4	96.5	98.1
CN35	HG Comp	G17	99	48	1.00	1.91	0.62	0.00	0.99	0.46	<10	0.66	0.00	17.1	6.72	0.00	0.25	60.9	42.9	60.8	96.5	96.3	98.5
CN36	HG Comp	G18	65	48	1.00	2.08	0.69	0.00	1.17	0.51	<10	0.66	0.00	9.38	4.62	0.00	0.24	50.9	41.5	70.3	93.5	94.8	97.5
CN39	MG Comp	G5	118	48	1.00	1.31	0.70	0.00	0.45	0.52	10	0.84	0.00	4.97	1.98	0.00	0.15	18.3	60.4	78.0	94.1	92.4	93.8
CN40	HG Comp	G17	99	48	1.00	1.31	0.68	0.00	0.52	0.49	<10	0.66	0.00	17.1	7.97	0.00	0.29	60.9	43.4	59.9	96.5	96.4	98.6
CN45	EG1 Comp	G41	104	48	1.00	1.18	0.79	0.00	0.26	0.58	0	0.00	0.00	2.13	2.49	0.00	0.14	16.0	37.1	50.9	92.1	94.6	95.4
CN49	EG1 Comp	G41	104	48	0.50	0.64	0.70	0.00	0.12	0.53		0.00	0.00	2.13	1.09	0.00	0.07	16.0	54.0	64.7	91.4	93.2	94.3
CN46	EG2 Comp	G42	105	48	1.00	1.19	0.77	0.00	0.24	0.58		0.00	0.00	3.01	1.93	0.00	0.20	44.6	58.0	75.5	89.6	89.9	94.4
CN50	EG2 Comp	G42	105	48	0.50	0.61	0.71	0.00	0.15	0.52		0.00	0.00	3.01	1.41	0.00	0.10	44.6	61.8	74.5	89.4	93.2	96.2
CN47	EG3 Comp	G43	80	48	1.00	1.22	0.74	0.00	0.27	0.56		0.00	0.00	21.0	3.14	0.00	0.15	85.0	66.0	80.7	95.0	95.2	99.3
CN51	EG3 Comp	G43	80	48	0.50	0.65	0.81	0.00	0.18	0.60		0.00	0.00	21.0	3.17	0.00	0.09	85.0	57.8	72.7	97.2	97.2	99.6
CN48	EG4 Comp	G44	103	48	1.00	1.16	0.58	0.00	0.23	0.44		0.00	0.00	0.90	0.88	0.00	0.12	7.7	68.5	81.1	85.5	86.3	87.4
CN52	EG4 Comp	G44	103	48	0.50	0.61	0.61	0.00	0.10	0.46		0.00	0.00	0.90	0.78	0.00	0.04	7.7	59.8	76.3	94.3	95.3	95.6
CN54A	Bulk Comp	G53	100	48	0.50	33.12	40.8	0.00	0.28	0.60				7.03	2.22		0.12	68.4	58.1	83.4	89.7	94.7	98.3
CN54B	Bulk Comp	G53	112	48	0.50	34.81	47.8	0.00	0.41	0.71				5.49	1.46		0.16	73.4	73.3	90.4	88.3	89.0	97.1
CN60	Bulk Comp	G57	150	48	0.50	0.89	1.05	0.00	0.42	0.84				143	4.24		0.22	97.0	63.1	97.8	103	94.7	99.8
CN59	Bulk Comp	G56	100	48	0.50	0.94	1.09	0.00	0.53	0.86				3.12	2.11		0.08	32.4	52.6	81.3	88.4	96.1	97.4
CN58	Bulk Comp	G55	75	48	0.50	0.93	0.97	0.00	0.57	0.78				4.55	1.96		0.08	57.0	67.2	97.9	104	95.8	98.2

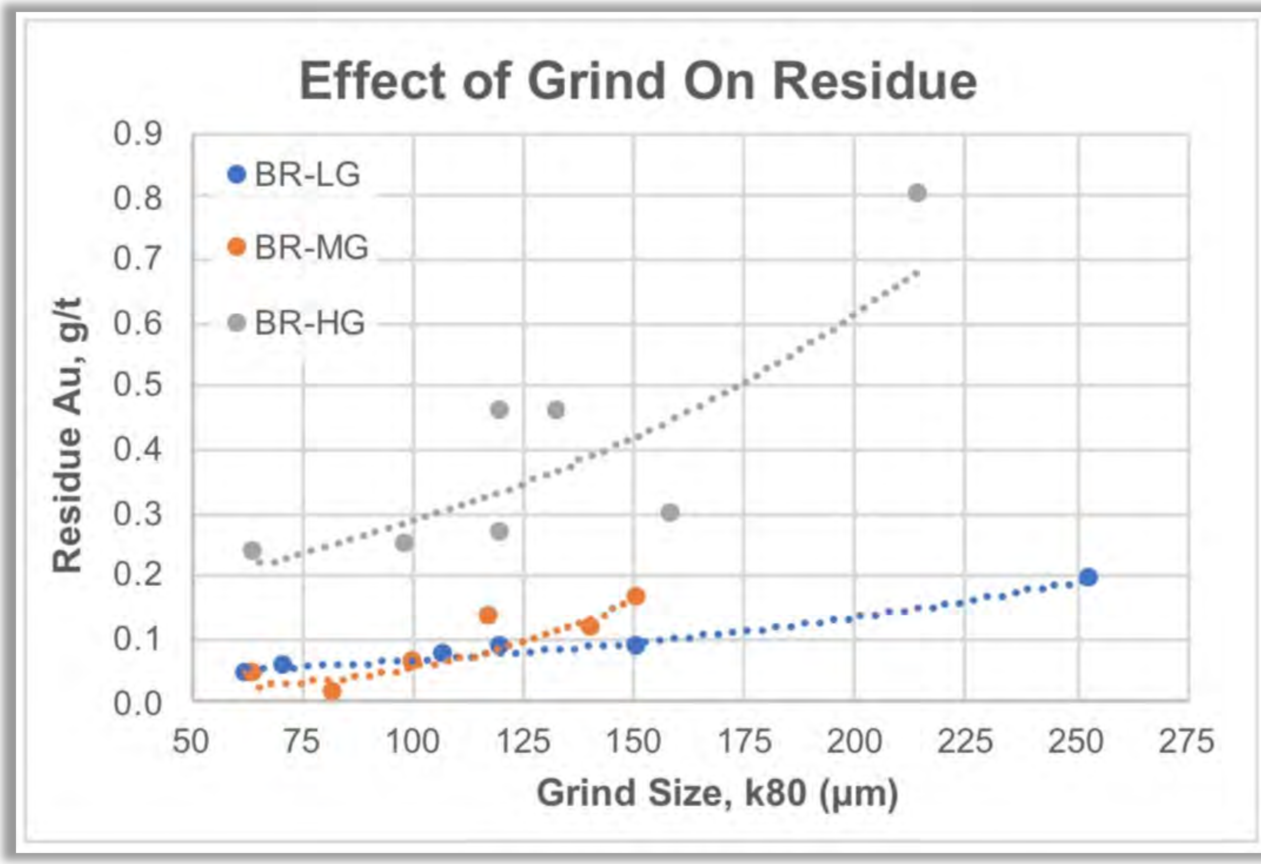
Figure 13.4 Gold Extraction as a Function of Retention Time at Three Different Grind Sizes



Source: BaseMet Report

Results showing the effect of grind on leach residue for the BR composites is provided on Figure 13.5. Grind sizes finer than a nominal K_{80} of 100 µm did not show significant increases in gold extraction when comparing final residue assays however they did show improved leach kinetic performance. Cyanide consumption increased as the primary grind size decreased, which is likely due to increase exposure of sulphide mineral surfaces. Pre-oxidation was evaluated to determine its impact on cyanide consumption.

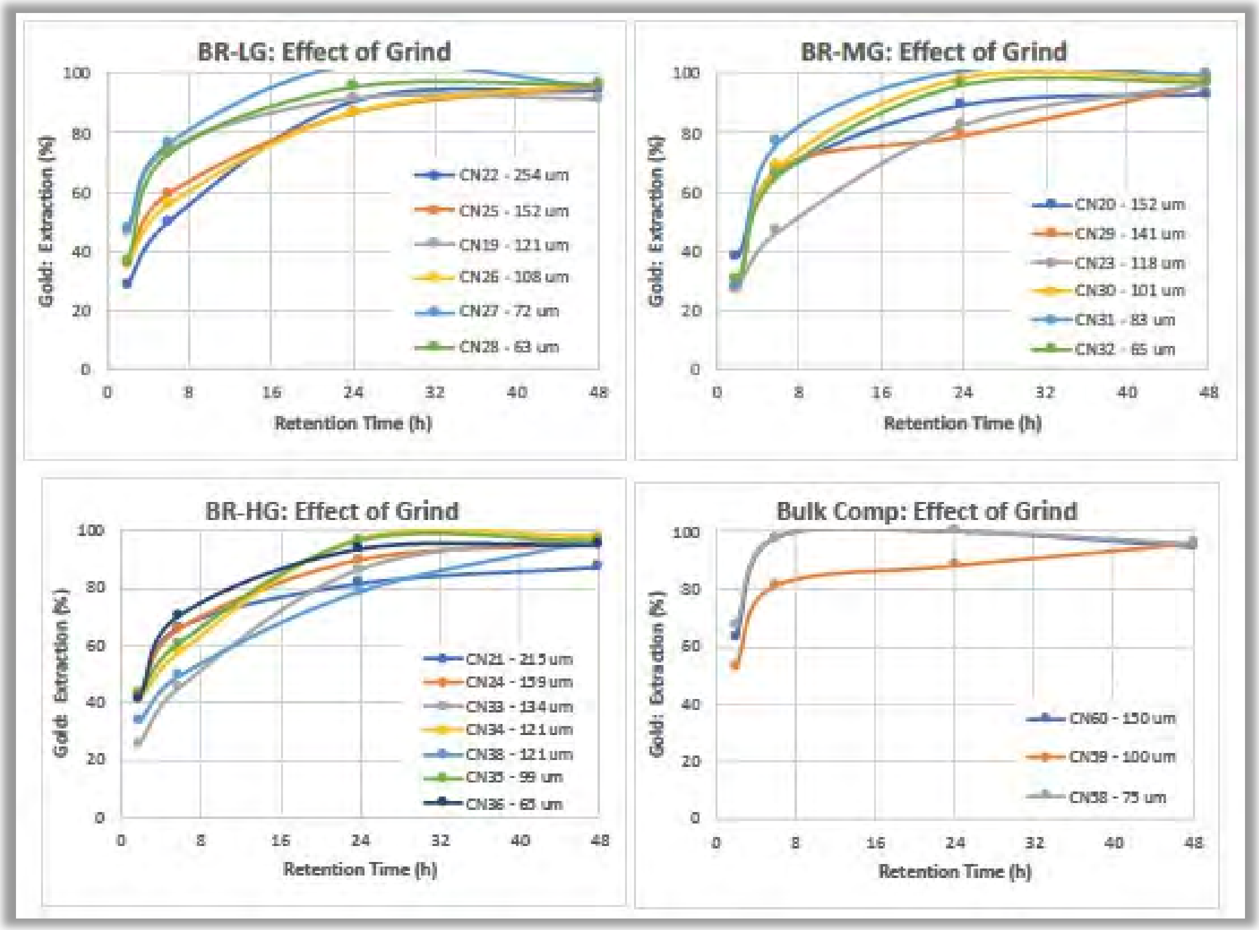
Figure 13.5 Effect of Grind on Residue Grade for the Boston Richardson Composites



Source: BaseMet Report

Results of all leach tests (extraction as a function of retention time) for each BR composite and the Master Composite at all grind sizes evaluated are shown on Figure 13.6. Extractions up to 48 hours are based on solution samples taken from the leach test and due to variations within the leach solutions, calculated extractions exceed 100% Au for some samples. The Master Composite bulk leach test showed an anomalous result with the finest grind (80% passing 75 µm) producing the lowest extraction with the slowest kinetics.

Figure 13.6 Extraction as a Function of Retention Time for Boston Richardson and Master Composites at All Grinds



Source: BaseMet Report

EFFECT OF CYANIDE CONCENTRATION

The EG composites were used to evaluate cyanide leach solution concentrations of 0.5 and 1.0 g/L as NaCN. A summary of results showing kinetics is provided in Table 13.15. Testing included 1 hour of pre-aeration with oxygen sparging to maintain a dissolved oxygen concentration of 15 mg/L; elevated dissolved oxygen levels were also maintained during the leach. The test results show there is no impact from pre-aeration on overall leaching kinetics or ultimate recovery. However, maintaining NaCN at 0.5 g/L vs. 1.0 g/L reduces cyanide consumption by approximately half. Leach residue grades were lower at the decreased NaCN concentrations. This is not an expected outcome with lowering the cyanide concentration. For the bulk leach tests, the cyanide concentration was set to 0.5 g/L NaCN with no pre-aeration.

Table 13.15 Summary of Leach Test Results at 0.5 g/L and 1.0 g/L NaCN

Test	Sample	Gravity Test	Grind K_{80} (,µm)	Leach Time (h)	NaCN g/L	Consumption (kg/t)		Grade (g/t Au)			Recovery (% Au)					
								Feed Stage (calc.)		Residue	Grav	Leach Kinetics (hours)				Comb Au
						NaCN	CaO	Grav	Leach	g/t	Au	2	6	24	48	
CN39	MG	G5	118	48	1.00	0.45	0.52	4.97	1.98	0.15	18.3	60.4	78.0	94.1	92.4	93.8
CN40	HG	G17	99	48	1.00	0.52	0.49	17.1	7.97	0.29	60.9	43.4	59.9	96.5	96.4	98.6
CN45	EG1	G41	104	48	1.00	0.26	0.58	2.13	2.49	0.14	16.0	37.1	50.9	92.1	94.6	95.4
CN49	EG1	G41	104	48	0.50	0.12	0.53	2.13	1.09	0.07	16.0	54.0	64.7	91.4	93.2	94.3
CN46	EG2	G42	105	48	1.00	0.24	0.58	3.01	1.93	0.20	44.6	58.0	75.5	89.6	89.9	94.4
CN50	EG2	G42	105	48	0.50	0.15	0.52	3.01	1.41	0.10	44.6	61.8	74.5	89.4	93.2	96.2
CN47	EG3	G43	80	48	1.00	0.27	0.56	21.0	3.14	0.15	85.0	66.0	80.7	95.0	95.2	99.3
CN51	EG3	G43	80	48	0.50	0.18	0.60	21.0	3.17	0.09	85.0	57.8	72.7	97.2	97.2	99.6
CN48	EG4	G44	103	48	1.00	0.23	0.44	0.90	0.88	0.12	7.7	68.5	81.1	85.5	86.3	87.4
CN52	EG4	G44	103	48	0.50	0.10	0.46	0.90	0.78	0.04	7.7	59.8	76.3	94.3	95.3	95.6

13.8 CARBON LOADING TESTING

Carbon loading test work generates carbon modeling data which supports a carbon management strategy. This can be used to optimize plant design and carbon gold inventory management. Test data generated is specific to the carbon tested. The carbon used for all tests was pre-attritioned carbon, manufactured by 'Calgon'. Material represented by EG3 gravity tailings produced by test G43 (8 kg) were leached in a stirred cyanide leach (test CN61). At termination, the pulp was split; one half retained as slurry for sequential triple contact carbon loading and a second half filtered to obtain pregnant leach solution (PLS) for carbon contacting to produce isotherm curves. The leach conditions used included:

— Pulp density = 50% Solids (wt. % basis);
— Pulp pH: 11.0 (maintained);
— Dissolved oxygen concentration: sparged with gaseous oxygen;
— NaCN concentration = 0.5 g/L NaCN maintained to 24 h;
— Retention Time = 48 hours;
— Grind = K_{80} 80 µm.

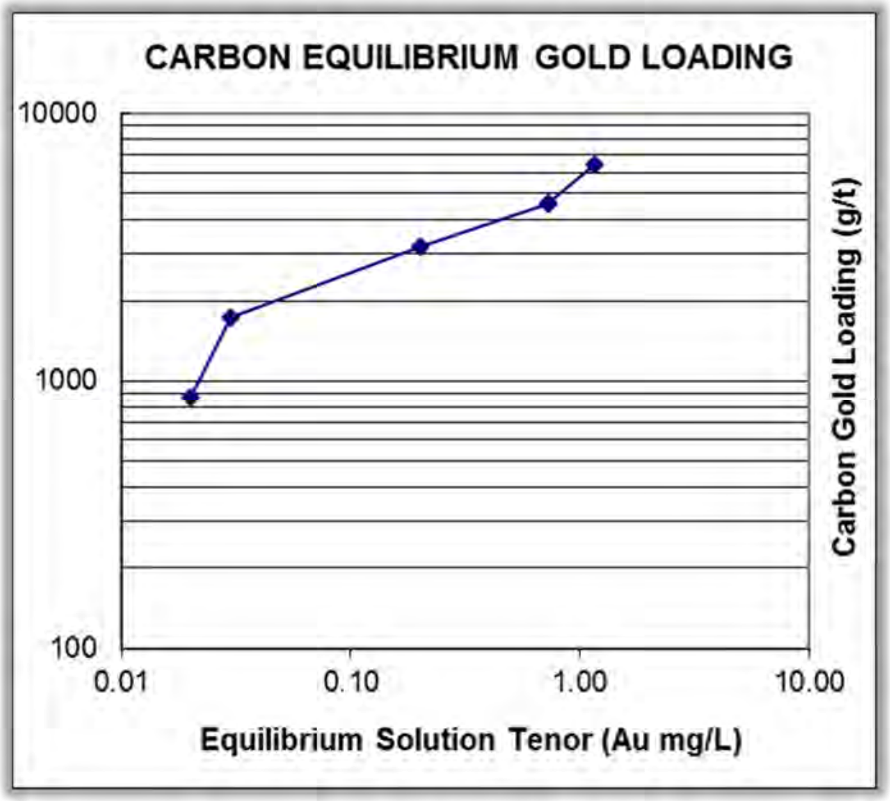
13.8.1 CARBON EQUILIBRIUM LOADING (LOADING ISOTHERMS)

Carbon equilibrium loading data was generated by contacting known volumes of pregnant leach solution with varying concentrations of activated carbon. Data is provided in Table 13.16 and graphically on Figure 13.7. The results show no issues impacting carbon loading. For design purposes a CIP carbon loading of 4,700 g/t Au has been used on the basis of 30% Au recovery to gravity and 95% Au recovery overall.

Table 13.16 Carbon Equilibrium Loading

Test No.	Carbon Concentration (g/L)	Carbon Added (g)	Solution mL	Solution Assay (mg/L Au)		Carbon Loading (g/t Au)
				Initial	Final	
CN61-N1	2.45	2.00	815	2.15	0.020	868
CN61-N2	1.23	1.00	815	2.15	0.030	1728
CN61-N3	0.61	0.50	815	2.15	0.200	3179
CN61-N4	0.31	0.25	815	2.15	0.730	4629
CN61-N5	0.15	0.13	815	2.15	1.160	6455

Figure 13.7 Carbon Equilibrium Gold Loading



Source: BaseMet Report

13.8.2 SEQUENTIAL CARBON-IN-PULP (TRIPLE CONTACT)

The Fleming Kinetic Constants are determined from sequential carbon triple contact testing, in which a known mass of fresh activated carbon is contacted with a known pulp volume, kinetic solution samples are removed over a 2-hour period at which point the carbon is transferred to fresh leached slurry and contacted for 4 hours. A third and final carbon transfer to fresh pulp which is contacted for an additional 16 hours (24 total) before the carbon is assayed. The Fleming constant 'k' was determined to be 156 h-1 and constant 'n' 0.527. These values are both considered to be within the typical range, based on Base Metallurgy's experience.

13.9 CYANIDE DESTRUCTION

Cyanide destruction testing was completed using the proven SO_2/air process, which is a robust and cost-effective method of treating slurry streams to achieve weak acid dissociable cyanide concentrations (CN_{WAD}) of less than 5 mg/L. For the FS testing program, a target concentration of 3 mg/L CN_{WAD} was selected. The site water balance was not developed at the time that the test program was completed that would have provided CN_{WAD} concentrations in the tailings facility required to meet environmental discharge requirements. This value was considered to be conservative based on experience for the Moose River operation in Nova Scotia.

13.9.1 PEA TESTING

Cyanide destruction testing in the PEA metallurgical testing program included treatment of:

— Solution from the tailings from flotation concentrate;

— Flotation concentrate leach tailing;

— Leach tailings from whole ore leaching.

Conditions for both are considered conservative in terms of retention time (120 minutes compared to typical 45 to 60 minutes) but fairly typical reagent additions. Cyanide destruction testing on the tailings solution and whole ore leach tailings were able to achieve very low CN_{WAD} concentrations, < 0.5 mg/L, while on the flotation concentration leach tailings CN_{WAD} concentrations < 1.5 mg/L were achieved.

13.9.2 FEASIBILITY STUDY TESTING

Cyanide Destruction (CND) testing was completed on bulk leach slurry using the Master Composite produced in test CN54. The test leached two barrels of 25 kg solids each.

13.9.3 CYANIDE FEED CHARACTERISTICS

Feed CND was taken from the leached pulp and filtered; the filtrate was submitted for analysis of cyanide species and metals. The cyanide solution from the pulp from each barrel contained 180/369 mg/L CNT, 150/335 mg/L CN_{WAD}, 10/12 mg/L Fe, and 4.6/6.3 mg/L Cu from each barrel reactor respectively. Detoxification test work was completed to produce a treated product using the SO_2/air process targeting below 3 mg/L weak acid dissociable cyanide (CN_{WAD}).

13.9.4 THE SO_2 / AIR PROCESS

The chemical reaction for the oxidation of weak-acid dissociable cyanide (CN_{WAD}) using sodium metabisulphite ($Na_2S_2O_5$ as a source of SO_2) is as follows:

$$2CN^- + Na_2S_2O_5 + 2O_2 + 2OH^- \longrightarrow 2CNO^- + Na_2SO_4 + SO_4^{2-} + H_2O$$

Copper acts as a catalyst for the reaction and any solution containing copper (copper cyano-complexes) will contribute to overall copper. As required, additional copper is added as copper sulphate. Hydrated lime is added to the reactor to provide hydroxide ion to complete the reaction.

Base metals such as copper, zinc and nickel, if present in the leached solution as cyanide complexes, precipitate as metal hydroxides during the cyanide destruction process. Ferrocyanide is not destroyed using the SO_2/Air process but precipitates with other base metals as a mixed metal ferrocyanide solid:

$$Fe(CN)_6^{4-} + 2Cu^{2+} \longrightarrow Cu_2Fe(CN)_6$$

$$Fe(CN)_6^{4-} + 2Zn^{2+} \longrightarrow Zn_2Fe(CN)_6$$

Thiocyanate (SCN^-), if present is partially oxidized to cyanate (CNO^-) and sulphate (SO_4^{2-}). Typically, less than 10% of the SCN^- in solution will be destroyed during SO_2/air cyanide destruction. Thiocyanate ion is unstable and slowly hydrolyzes to ammonium and carbonate:

$$CNS^- + Na_2S_2O_5 + 3O_2 + 4OH^- \longrightarrow CNO^- + Na_2SO_4 + 2SO_4^{2-} + 2H_2O$$

$$CNO^- + 2H_2O \longrightarrow CO_3^{2-} + NH_4^+$$

Process development testing for the SO_2/air process is completed in two stages. The first stage is batch testing, followed by second stage continuous testing. The batch reactor is first filled with feed slurry and the required copper sulphate is added. The reactor content is then treated in batch mode with sodium metabisulphite ($Na_2S_2O_5$ or SMBS) as the SO_2 source and air to reduce the concentration of CN_{WAD} in solution to target less than 5 mg/L. The oxidation reduction potential (ORP) of the pulp is monitored with a Pt/Ag/AgCl combination electrode, while the residual CN_{WAD} concentration in the solution phase is analyzed during the test determined using the Modified Potentiometric Titration method. Target batch retention times are between 45 and 60 minutes. The batch test serves to produce treated material with low residual CN_{WAD}, the product is used as starting feed material for the initial continuous test. Final solutions are submitted for analysis at the completion of each test or run.

A 2-L reactor was used for both batch and continuous tests. For the continuous tests, an overflow nozzle on the reactor transferred treated slurry to a storage tank.

When scaling laboratory data to plant design applications, a correction for SO_2 and lime consumptions must be accounted for. During laboratory SMBS is used in place of SO_2 for laboratory tests, while commercial plants can use either SMBS or SO_2 gas. The lime requirement for pH control will likely be slightly higher by approximately 0.5 mole lime per mole SO_2 or 0.58 g lime per g SO_2 as suggested by the following reactions:

$$2SO_2 + 2NaOH \rightarrow Na_2S_2O_5 + H_2O$$

$$Na_2S_2O_5 + Ca(OH)_2 \rightarrow CaSO_3 + Na_2SO_3 + H_2O$$

$$2SO_2 + 2Ca(OH)_2 \rightarrow 2CaSO_3 + 2H_2O$$

13.9.5 CYANIDE DESTRUCTION OPTIMIZATION

A batch test (CND-B1) was conducted on leached slurry to produce a charged reactor for starting continuous CND testing with a low residual CN_{WAD}. A series of continuous CND tests were completed to establish design criteria and understand the effect of reagent dosages on the oxidation of cyanide. The test conditions and results are summarized by previous Table 13.13. The treated slurry produced during the test program responded well to cyanide destruction by SO_2/air producing a discharge with < 2.0 mg/L CN_{WAD} using the following suggested optimum treatment conditions (reference test CND-C7):

— Pulp density = 45.5% solids (by weight);

— Pulp pH = 8.4 (maintained);

— Retention time = 45 minutes;

— Dissolved oxygen concentration = 4.0 mg/L minimum;

— SO_2:CN_{WAD} addition rate = 5.0:1 equivalent;

— Hydrated lime addition rate = 3.5 to 5.0 CaO:CN_{WAD} equivalent

— Copper addition = 40 mg/L Cu^{2+}:CN_{WAD}

A summary of the batch and continuous tests completed is shown in Table 13.17. The batch tests all achieved the CN_{WAD} < 3.0 mg/L target. Retention time was reduced to 45 minutes and copper addition reduced without impacting the cyanide discharge concentration. During the continuous runs, changes to the reagent additions and retention times were made to optimize both. The last four runs were completed at 45.5% solids, close to the expected operating conditions of the plant. At times, air sparging limitations required supplemental oxygen addition to maintain the minimum 4.0 mg/L of dissolved oxygen required with the lower density. At the lower density, the target CN_{WAD} concentration of 3.0 mg/L was achieved with a 45 minutes retention time, SO_2:CN_{WAD} ratio of 4.0:1 w/w and copper addition of 42 mg/L Cu^{2+}:CN_{WAD}.

Table 13.17 Summary of Cyanide Destruction Tests

Test	Mode	Objective	Slurry Density (% solids)	Retention Time (min.)	Solution Composition (pH, mg/L)					Slurry Treated (L)	Reagent Addition (g/g CN_{WAD})		
					pH	CNT	CN_{WAD}	Cu	Fe		SO_2 Equiv.	Lime	Cu mg/L.
Master Comp: CN54B - Residue Slurry					10.4	369	335	6.3	12	2	-	-	-
CND-C1.1	Continuous	Baseline	50	54	8.4	1.38	1.24	0.48	<1	4.95	5.0	7.2	66
CND-C1.2	Continuous	Lower Retention	50	43	8.4	1.91	1.77	1.02	<1	8.66	5.0	5.2	66
CND-C2.1	Continuous	Cu 45 mg/L	50	43	8.4	6.84	1.25	0.31	2	13.5	5.0	6.1	45
CND-C2.2	Continuous	Cu 30 mg/L	50	43	8.4	6.79	1.20	0.31	2	4.10	5.0	6.1	30
Master Comp: CN54A - Residue Slurry					10.4	180	152	4.6	10	-	-	-	-
CND-C3.1	Continuous	As per CND-C2.2	50	45	8.32		8.53			8.61	5.0	9.0	30
CND-C3.2	Continuous	Up SO_2 ratio, Cu	50	44	8.37	8.53	2.94	4.36	2	3.08	11.0	9.0	45
CND-C4.1	Continuous	As per CND-C3.2	50	43	8.35	1.86	1.73	<0.01	<1	8.66	11.0	11.5	45
CND-C4.2	Continuous	Ext. retention O_2	50	53	8.25	2.15	2.01	<0.01	<1	6.44	11.0	14.3	46
CND-C5.1	Continuous	As per CND-C4.2	50	53	8.29	1.34	1.20	<0.01	<1	9.41	8.5	14.7	46
CND-C5.2	Continuous	45% sol, low SO_2	45.5	45	8.30	1.64	1.50	0.68	<1	6.56	4.0	4.9	42
CND-C6.1	Continuous	As per CND-C5.2	45.5	46	8.38	5.76	2.96	0.45	1	11.7	3.9	3.5	41
CND-C6.2	Continuous	Increase. SO_2 ratio	45.5	46	8.31	5.35	2.55	0.45	1	6.56	5.0	3.5	37
CND-C7	Continuous	As per CND-C6.2	45.5	45	8.34	4.37	1.58	0.25	1	16.6	5.0	5.0	41

13.10 ARSENIC PRECIPITATION

The elevated arsenic concentrations in the samples tested from arsenopyrite (and in the PEA testing) result in soluble arsenic concentrations in the tailings solution that require treatment. Iron compounds, such as ferric sulphate, are commonly used for the removal of soluble metals such as arsenic. The precipitation of arsenic with iron results in stable ferric compounds (arsenates and hydroxides) that are suitable for long term disposal when the Fe:As molar ratio is at least 4:1 (3:1 weight basis) with a pH > 5. The two common reactions that characterize the process are:

$$Fe_2(SO_4)_3 + 2H_3AsO_4 + 4H_2O \rightarrow 2FeAsO_4 \cdot 2H_2O + 3H_2SO_4 \text{ or}$$

$$H_3AsO_4 + Fe^{3+} \rightarrow FeAsO_4 + 3H^+$$

The reactions require the arsenic to be in the As^{+5} state. The treatment is commonly done on wastewater streams but can also be employed on tailings slurry streams. The strong oxidizing conditions in the SO_2/air process and pH 8.5 provide good conditions for the precipitation of arsenic with iron.

13.10.1 PEA TESTING

The PEA testing program included testing of flotation concentrate leach tailings solution and slurry and whole ore leach tailings slurry. The tailings leach solution and the whole ore leach tailings slurry had low levels of arsenic, 0.7 mg/L and 0.3 mg/L respectively. Whole ore leach tailings slurry did not require treatment and the flotation concentrate leach tailings solution was easily treated. The arsenic concentration in the flotation concentrate leach tailings increased in cyanide destruction testing from 19 mg/L to 37 mg/L and as a result, insufficient ferric sulphate was added to remove the arsenic (feed assay not available until testing completed).

13.10.2 FEASIBILITY STUDY TESTING

Arsenic was precipitated from the slurry produced from the continuous cyanide destruction testing. Ferric Sulphate was tested at four ratios of Ferric to Arsenic 10:1, 19:1, 60:1 and 119:1. The ratios are quite conservative compared to the published information for this process. The tests were operated at pH 8.5 with 45 minutes of retention, kinetic solutions were removed at 10, 30 and 45 minutes. Results are summarized by Table 13.18. Arsenic removal efficiency increases with increased Fe:As ratio; As in solution below 0.1 mg/L is achieved with 45 minutes and 50 mg/L of Fe as Ferric.

Test ASP-4 was designed for an Fe:As ratio of 8:1 but the lower than expected arsenic grade in the feed solution increased the ratio to 10:1. The process design criteria includes 45 minutes retention time with an addition ratio of 8:1 Fe:As on a weight basis.

Table 13.18 Results of Arsenic Precipitation Tests

Test	Fe:As Ratio (weight)	As Assay (mg/L)			
		Feed	10 minutes	30 minutes	45 minutes
ASP-1	19	6.3	0.601	0.534	0.220
ASP-2	60	6.3	0.540	0.234	0.234
ASP-3	119	6.3	0.231	0.139	<0.1
ASP-4	10	5.0	0.295	0.225	0.225

13.11 ENVIRONMENTAL TESTING

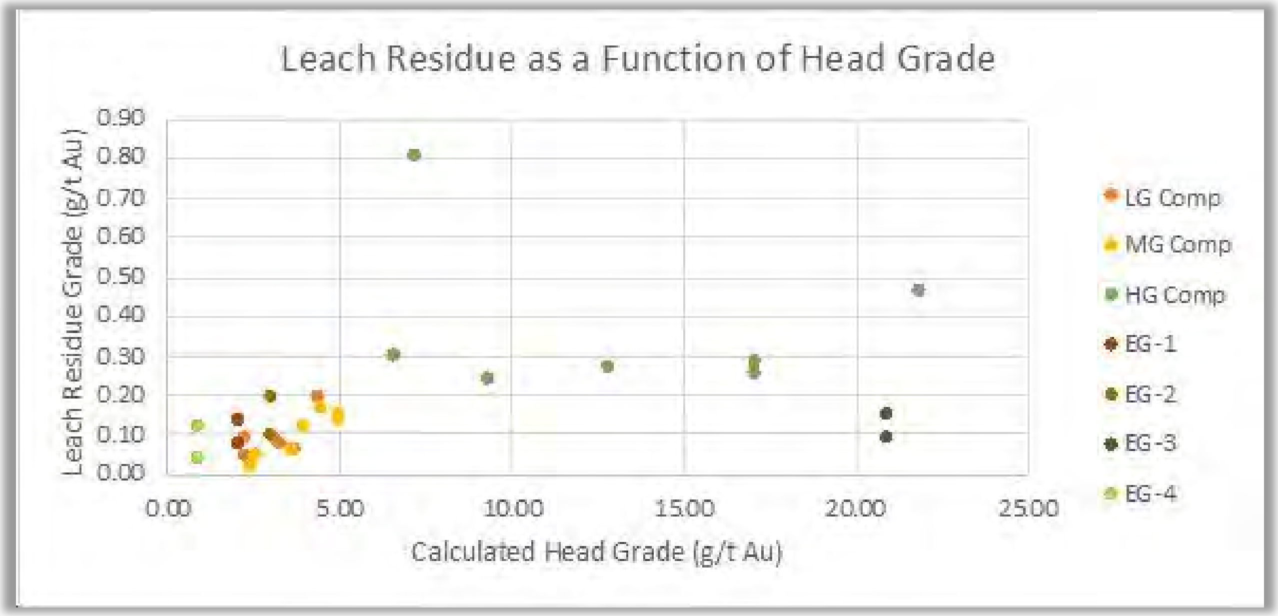
Slurry produced during cyanide destruction were shipped to SGS-CEMI in Burnaby, BC (an environmental lab) for further testing and analysis which included the following.

- Environmental Characterization Studies including:
  - Feed Analysis (Metals, Mineralogy, Solution Aging);
  - Static Testing (Shake Flask, TCLP, ABA/NAG);
  - Kinetic Testing (Humidity Cell Tests; 20-week period).
- Geotechnical Studies including:
  - Standard Proctor, Atterberg Limit, Settling Density, Drained Settling Density, Sieve Analysis;
  - Hydrometer, Hydraulic Conductivity, Consolidation Testing, Air Drying, Soil Water Characterization Curve;
  - Full environmental details are provided by BL395 Report No. 2, pending completion of the test program.

13.12 PREDICTED METALLURGICAL RESULTS

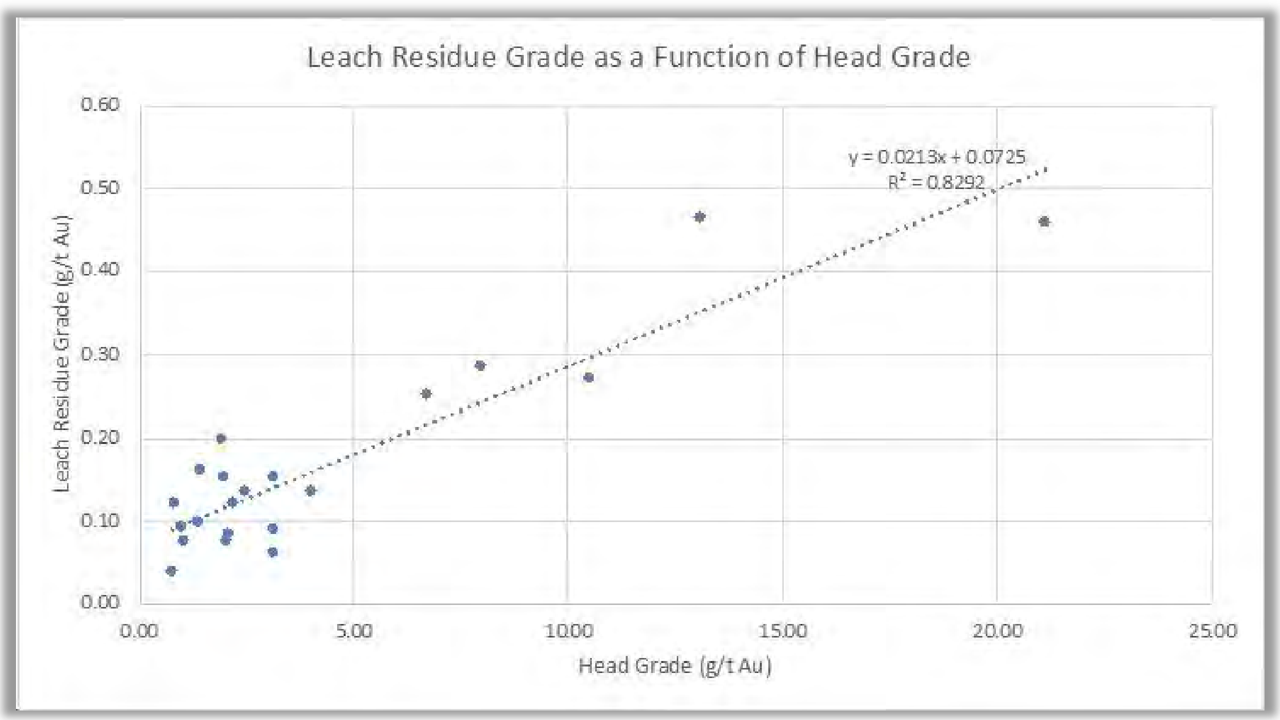
The leach test results were analyzed to provide a recovery model for use with the mine production schedule to provide gold recovery and production data. Test results for tests at the target grind of 80% passing 110 µm were included in the modeling exercise for all composite samples. Leach residue assays as a function of calculated head grades formed the basis for the model. As described in the Leaching Baseline and Effect of Grind section, recovery did not increase beyond 24 hours leaching. Using the residue assay at 48 hours provides the most accurate data for the model. Kinetic samples with calculated extraction over 100% were not used. The results are shown on Figure 13.8. The results show that the residue grade is somewhat insensitive to head grade. There is also no clear difference in results between the BR and EG composites; both produce similar residue grades for the same head grades. The high-grade EG composite had slightly lower residue grades than the BR HG composite but these grades are not representative of ore grades.

Figure 13.8 Leach Residue as a Function of Head Grade



Test results for BR and EG composites with calculated head grades ranging from 0.90 g/t Au to 23.9 g/t Au were selected that had grinds ranging from K_{80} = 80 µm to K_{80} = 134 µm. Results from BR and EG composites were combined as the leach test results from both are similar. The data and fitted model are shown on Figure 13.9. The model predicts the leach residue grade based on the head grade.

Figure 13.9 Leach Residue Grade as a Function of Head Grade



In addition to the predicted extraction, plant losses including:

- Soluble losses of 0.010 g/t Au for head grades > 2.0 g/t Au;
- Carbon losses of 40 g/t;
- Fine carbon assays of 80 g/t Au for carbon losses;
- Other plant losses of 0.2% Au.

Using the model shown on Figure 13.9 is along with plant losses of 0.83% Au the following equation is used to predict plant gold recovery:

$$\frac{0.9729\ x\ head\ grade - 0.0725}{head\ grade} \text{ x } 100\% - 0.83\%.$$

14 MINERAL RESOURCE ESTIMATES

14.1 GENERAL

WSP completed an update to the mineral resource estimation of the Goldboro Property. The effective date of the updated mineral resource is August 21, 2019.

The definition of mineral resource and associated mineral resource categories used in this report are those recognized under National Instrument 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines (2014) (the CIM Standards). Assumptions, metal threshold parameters, and deposit modeling methodologies associated with the current Goldboro Deposit resource estimate are discussed in Sections 14.2 through 14.4.

The 2018 PEA has not been updated in light of the Mineral Resource Update for the Goldboro Gold Project. The 2019 resource update is being used as the basis for the on-going Feasibility Study.

14.2 DATABASE

WSP compiled all the data used in completing the Mineral Resource from original source drillhole documents, and from plan and section originals and copies. The Project has 492 drillholes, however, only drillholes within the areas of interest and with exploration potential were included in the database.

All mineral resource estimations were conducted using Surpac™ 2019. Table 14.1 summarizes the drillholes entered into the database.

Table 14.1 Drillhole Summary

Year	Number of Drillhole	Length (m)
Total	435	81,871
1984	1	529
1985	5	390
1987	32	10,710
1988	46	10,640
1989	77	11,813
1990	86	3,981
1991	5	722
1993	6	593
1995	7	1,263
2005	23	2,422
2008	43	11,687
2010	49	10,738
2011	5	1,000
2017-2018	44	12
2018-2019	57	15,111
Year not identified	6	260

14.3 BULK DENSITY

A bulk density value of 2.70 kg/m^3 was applied in the previous resource estimate for the Goldboro Deposit on the basis of nine verification samples measured by ALS Chemex in 2006. This value corresponds closely with density values applied for other Meguma hosted gold deposits and therefore has been retained for the current mineral resource estimate.

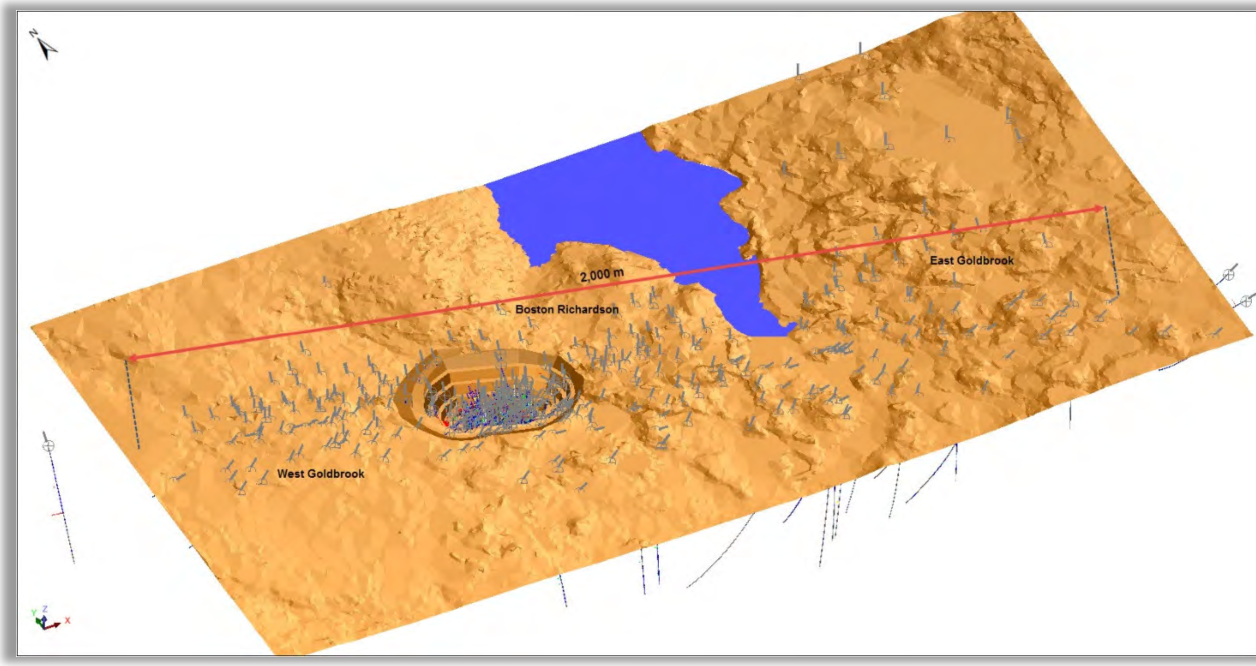
Anaconda collected bulk density data during the 2017- 2018 drill program using the water immersion method. A total of 1,935 measurements have been taken by Anaconda from drillholes resulting in an average specific gravity of 2.72 kg/m^3.

WSP has reviewed the results of the bulk density data and applied a regression formula on the data. Only gold samples above 0.06 g/t were considered in the review resulting. The remaining data set was review and the outliers (top cut and bottom cut) were removed. This review lead to a dataset of 649 samples. The regression formula used was Density=2.7578+(0.006*au_19_ppm_ok).

14.4 TOPOGRAPHIC DATA

The topographic data was provided by Anaconda and is based on a high-resolution LiDAR survey. The collars were compared to the topographic survey. Figure 14.1 shows the resolution of the topography with the drill collars.

Figure 14.1 Goldboro Topographic Data



14.5 GEOLOGICAL INTERPRETATION

The stratigraphic succession of the Goldboro area is repeated across the axial zone of the Upper Seal Harbour Anticline, which is an upright, ENE-trending fold that in the immediate property area plunges to the east at an inclination of 20° to 30°. Argillite intervals occurring in the core of the anticline are thicker than their laterally equivalent intervals on the north and south fold limbs and this is interpreted as indicating substantive influence of flexural slip folding processes.

With respect to the digital geological model developed for the mineral resource area by WSP, the tightly appressed, upright, east plunging, and symmetrical aspects of the Upper Seal Harbour Anticline were consistently respected during section-based wireframing of the multiple mineralized belts present. Similarly, modeling of the New Belt Fault considered all available information that could be used to identify the fault location on successive drilling sections and this was wireframed to create the fault solid.

Three-dimensional wireframe models of mineralization were developed for the belts based on geology and a gold cut-off of greater than 0.3 g/t near surface to accommodate open pit development, and 1.0 g/t deeper for underground development. A minimum 1 m horizontal width was the general design guideline, yet there maybe occurrences where the width is less than 1 m.

Areas of drift development in mineralization were also created.

Sectional interpretations were digitized in Surpac™ software, and these interpretations were linked and triangulated to build the three-dimensional solids. WSP provided Anaconda copies of all the solids for review and approval.

The mineralization belts are generally contiguous; however, there are occurrences where the grade is below cut-off, yet still captured in the solids. These zones display all the characteristics of a mineralized belt, without the gold grade.

The wireframes extend at depth, well below the deepest diamond drillholes. This is to provide the exploration group with target areas for future exploration. The resource model did not estimate grades into the full volume of the wireframes due to sheer size of the wireframes.

Each domain was modeled using the similar principal assumptions and methodology.

Table 14.2 tabulates the solids and associated volumes. The solids were validated in Surpac™ and no errors were found.

Figures 14.2 to 14.4 display the belts for the Boston-Richardson Zone, West Goldbrook and East Goldbrook.

Table 14.2 Goldboro Solids Summary

Domain	Minimum X	Maximum X	Minimum Y	Maximum Y	Minimum Z	Maximum Z	Surface Area (m^2)	Volume (m^3)
BR-01-LG	293,347.09	294,190.67	5,006,820.09	5,007,141.66	4,492.75	5,059.94	874,711	1,352,014
BR-01a-LG	293,561.37	293,691.91	5,007,015.87	5,007,080.48	4,799.86	4,941.88	43,212	31,637
BR-02-LG	293,347.40	294,191.75	5,006,833.86	5,007,123.62	4,492.63	5,033.33	694,535	926,920
BR-02a-LG	293,670.45	293,715.53	5,006,967.19	5,007,042.15	4,735.03	4,887.96	18,638	16,150
BR-03-LG	293,347.50	294,171.16	5,006,855.38	5,007,125.48	4,505.05	5,015.60	696,275	834,165
BR-03a-LG	293,347.50	294,171.16	5,006,855.38	5,007,125.48	4,505.05	5,015.60	825,701	986,741
BR-04-LG	293,347.67	294,193.39	5,006,864.32	5,007,115.10	4,474.09	4,985.16	663,981	912,193
BR-04a-LG	293,394.34	293,587.13	5,006,978.28	5,007,074.77	4,780.58	4,944.18	40,130	33,995
BR-05-LG	293,347.75	294,191.02	5,006,869.31	5,007,106.02	4,488.45	4,955.95	551,902	693,887
BR-06-LG	293,347.81	294,191.11	5,006,882.46	5,007,094.33	4,480.74	4,927.30	539,149	552,826
BR-07-LG	293,347.87	294,188.13	5,006,884.88	5,007,088.45	4,466.43	4,907.01	439,736	443,686
BR-08-LG	293,347.96	294,180.25	5,006,883.70	5,007,084.91	4,453.16	4,887.52	365,804	326,166
BR-09-LG	293,348.04	294,175.65	5,006,888.60	5,007,081.57	4,431.82	4,857.41	328,340	291,226
BR-10-LG	293,349.19	293,822.48	5,006,948.22	5,007,077.50	4,443.19	4,838.97	224,046	175,714
BR-11-LG	293,368.26	293,821.47	5,006,952.37	5,007,071.83	4,439.55	4,803.52	141,819	151,850
BR-12-LG	293,413.17	293,524.93	5,006,995.06	5,007,049.36	4,614.94	4,755.53	37,574	36,774
Marker Unit	293,340.01	294,429.22	5,006,772.17	5,007,170.55	4,381.95	5,110.62	1,394,144	21,702,968
Marker Unit 01	293,354.78	293,662.51	5,006,937.51	5,007,034.25	4,837.46	5,005.61	115,994	103,662
Marker Unit 02	293,400.26	293,544.82	5,006,919.68	5,007,018.73	4,760.19	5,003.44	45,417	31,898
Marker Unit 03	293,600.61	293,712.74	5,006,945.33	5,006,987.77	4,828.15	4,923.59	15,415	8,116
EG-00-LG	293,337.92	294,466.55	5,006,779.13	5,007,146.87	4,419.14	5,102.78	1,063,989	1,529,960
EG-01-LG	293,880.00	294,590.00	5,006,760.91	5,006,990.20	4,751.08	5,036.65	405,312	598,670
EG-02-LG	293,880.00	294,453.09	5,006,730.60	5,007,103.04	4,699.06	5,061.72	429,370	649,946
EG-02a-LG	294,100.00	294,570.00	5,006,797.83	5,007,000.24	4,786.66	4,979.35	190,950	167,333
EG-03-LG	293,880.00	294,590.00	5,006,728.50	5,007,103.41	4,660.42	5,050.96	450,664	772,712
EG-03a-LG	294,020.00	294,460.00	5,006,788.12	5,006,973.35	4,699.97	4,983.53	167,568	168,601
EG-04-LG	293,840.00	294,370.00	5,006,738.90	5,007,119.85	4,649.36	5,031.19	403,019	591,721
EG-04a-LG	294,020.00	294,152.46	5,006,849.57	5,006,967.93	4,799.65	4,984.74	43,664	43,589
EG-05-LG	293,760.00	294,350.00	5,006,755.66	5,007,106.63	4,640.15	5,030.73	440,706	604,028

(table continues on next page)

Domain	Minimum X	Maximum X	Minimum Y	Maximum Y	Minimum Z	Maximum Z	Surface Area (m^2)	Volume (m^3)
EG-06-HG	293,540.00	294,350.00	5,006,759.11	5,007,142.12	4,639.74	5,089.29	725,945	1,006,914
EG-06a-LG	293,400.00	294,200.00	5,006,849.76	5,007,169.69	4,760.09	5,141.22	388,306	507,826
EG-07-LG	293,400.00	294,464.90	5,006,766.76	5,007,148.63	4,607.51	5,123.33	672,814	1,019,050
EG-07a-LG	293,790.00	294,464.16	5,006,778.14	5,007,052.09	4,463.91	4,967.87	385,589	435,567
EG-07b-LG	293,355.83	293,852.39	5,006,883.40	5,007,152.62	4,720.03	5,116.00	424,896	597,129
EG-08-LG	294,060.00	294,360.00	5,006,756.41	5,006,998.96	4,780.43	5,040.21	206,928	277,293
EG-09-LG	294,140.00	294,750.00	5,006,713.27	5,007,015.66	4,722.65	5,051.70	284,264	404,822
EG-09a-LG	294,200.00	294,750.00	5,006,736.97	5,006,970.81	4,782.84	5,057.22	212,867	222,133
EG-10-LG	294,150.00	294,750.00	5,006,693.53	5,007,042.36	4,779.41	5,067.44	346,974	446,723
EG-10a-LG	294,200.00	294,750.00	5,006,726.63	5,006,972.80	4,789.05	5,070.78	257,239	364,678
EG-11a-LG	294,359.72	294,830.00	5,006,699.52	5,006,995.17	4,720.64	5,031.24	204,695	253,071
EG-11b-LG	294,500.00	294,830.00	5,006,692.42	5,006,859.22	4,743.04	5,024.40	122,399	133,979
EG-12-LG	294,180.00	294,824.20	5,006,677.79	5,007,061.07	4,755.62	5,097.38	412,062	603,472
EG-13-LG	294,510.00	294,740.00	5,006,667.88	5,006,812.64	4,845.72	5,050.15	85,283	89,335
EG-14-LG	294,550.00	294,700.00	5,006,655.50	5,006,788.63	4,850.66	5,031.45	51,578	48,339
EG-15-LG	294,550.00	294,700.00	5,006,638.98	5,006,772.83	4,850.90	5,035.54	48,930	58,160
WG-01-LG	293,145.31	293,316.61	5,007,018.72	5,007,225.22	4,824.24	5,069.20	101,438	128,198
WG-02-LG	292,781.43	293,348.44	5,007,046.14	5,007,259.02	4,850.69	5,130.28	307,387	461,200
WG-03-A-LG	292,781.44	293,344.52	5,007,040.46	5,007,231.20	4,798.00	5,107.61	340,157	347,749
WG-03-B-LG	293,200.66	293,301.48	5,007,096.48	5,007,131.35	4,929.00	4,996.67	8,343	6,738
WG-04-LG	292,781.49	293,215.88	5,007,062.27	5,007,225.09	4,819.02	5,079.83	186,743	198,347
WG-05-A-LG	292,781.50	293,344.70	5,007,006.40	5,007,216.24	4,601.14	5,068.24	391,426	391,252
WG-05-B-LG	293,037.54	293,046.99	5,007,138.52	5,007,157.74	4,958.16	4,985.53	1,036	836
WG-06-A-LG	292,781.53	293,331.52	5,007,019.65	5,007,195.10	4,604.05	5,050.24	350,503	357,627
WG-06-B-LG	293,194.81	293,299.60	5,007,048.59	5,007,142.04	4,669.78	4,900.24	70,697	37,659
WG-07-LG	292,786.59	293,324.28	5,007,051.78	5,007,202.12	4,660.00	4,965.21	264,980	237,837
WG-08-LG	292,875.76	293,310.81	5,007,074.47	5,007,166.93	4,682.26	4,876.52	169,281	144,543
WG-09-LG	292,947.53	293,300.04	5,007,080.20	5,007,154.29	4,642.66	4,811.86	67,456	75,359
WG-10-LG	293,027.55	293,048.03	5,007,125.30	5,007,146.06	4,703.51	4,739.69	2,161	959

Figure 14.2 Boston-Richardson Belt

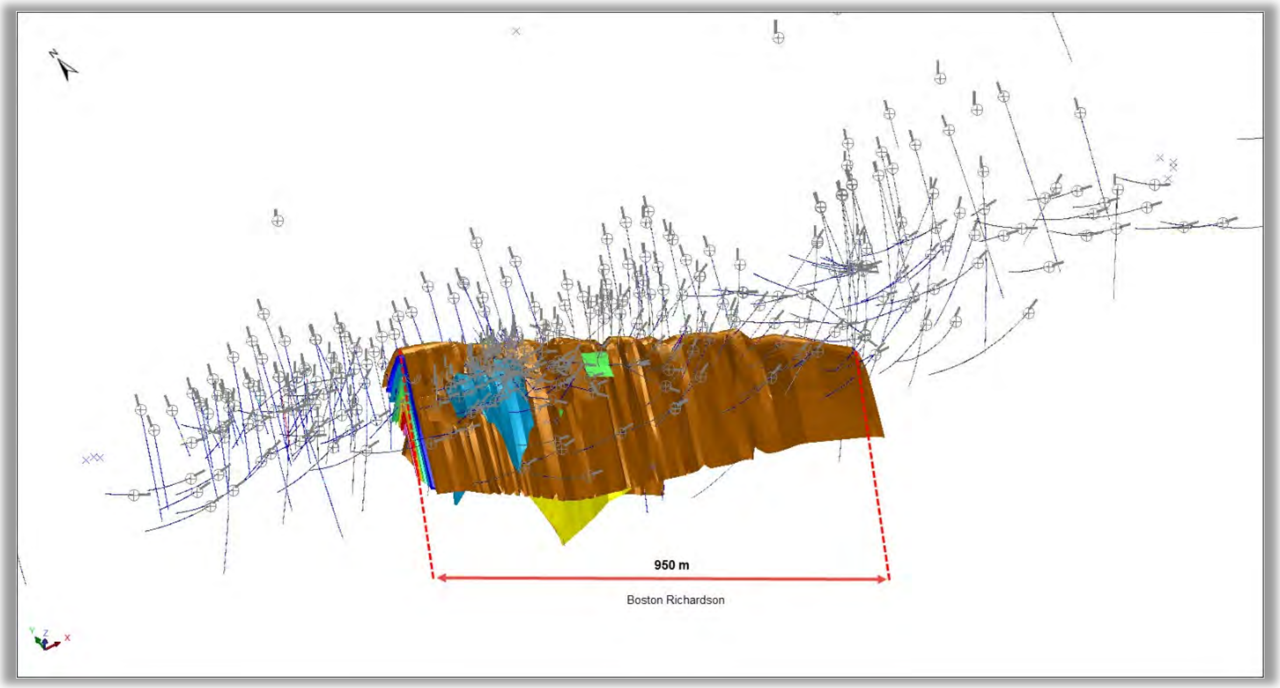


Figure 14.3 West Goldbrook Belt

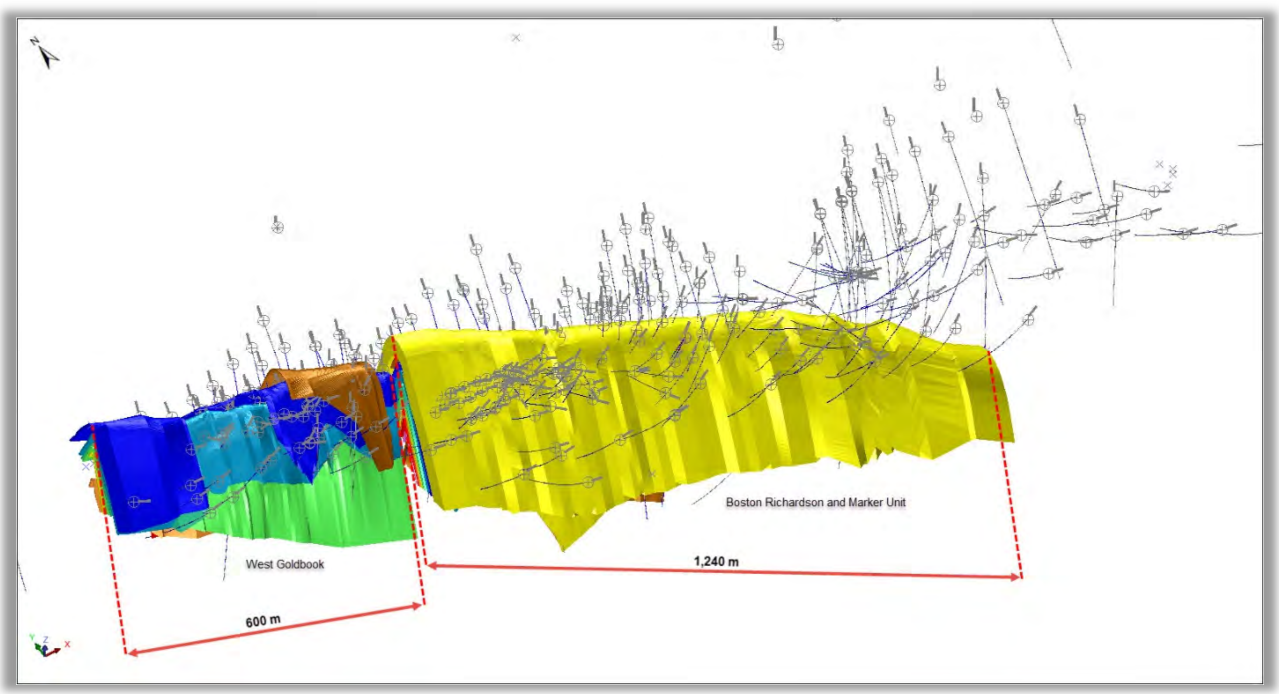
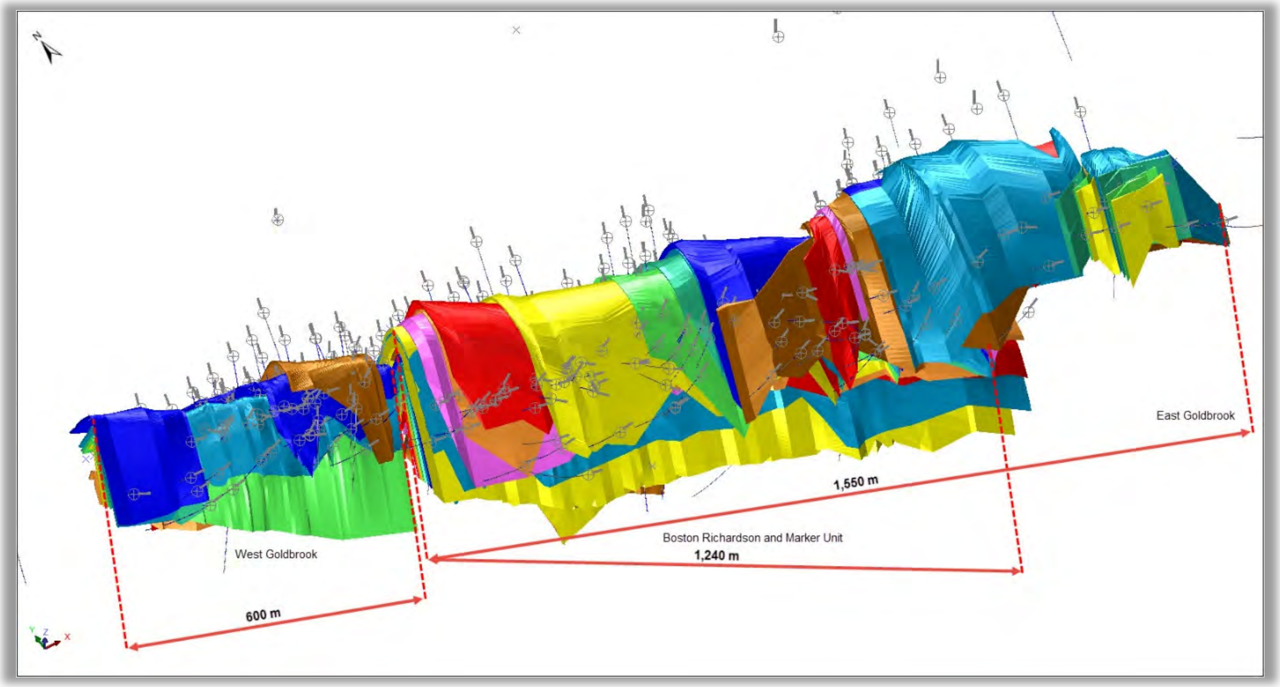


Figure 14.4 East Goldbrook Belt



14.6 EXPLORATORY DATA ANALYSIS

14.6.1 ASSAYS

A Surpac routine was run to flag the sample intervals within each of the belts. WSP visually inspected the flagged intervals to ensure the correct intervals were flagged. Several boreholes were removed from the flagging process due to inappropriate interval lengths with elevated grades. Table 14.3 lists the holes removed from the dataset.

Table 14.3 Boreholes Removed from Dataset

Borehole Removed				
BR89-83	BR89-87	BR89-92	BR89-96	BR89-105
BR89-84	BR89-90	BR89-93	BR89-97	-
BR89-85	BR89-91	BR89-94	BR89-104	-

The 57 belts that form the Mineral Resource were sampled by a total of 13,896 sample intervals. The statistics for this data are summarized in Table 14.4.

Table 14.4 Goldboro Borehole Statistics by Belt

Domain	Field	No of Records	Minimum	Maximum	Mean	Standard Deviation
BR_01_LG	au_ppm	1076	0.003	298.088	2.213	13.888
BR_01A_LG	au_ppm	60	0.02	6.860	0.839	1.116
BR_02_LG	au_ppm	1511	0.003	876.000	2.703	23.624
BR_02A_LG	au_ppm	74	0.01	4.250	0.643	0.845
BR_03_LG	au_ppm	1242	0.003	483.000	2.918	17.940
BR_03A_LG	au_ppm	457	0.003	84.600	2.032	6.671
BR_04_LG	au_ppm	1181	0.003	425.000	2.334	14.730
BR_04A_LG	au_ppm	119	0.003	26.540	2.116	4.799
BR_05_LG	au_ppm	604	0.003	190.970	2.176	11.836
BR_06_LG	au_ppm	360	0.003	78.170	1.764	6.114
BR_07_LG	au_ppm	495	0.003	79.334	1.871	6.004
BR_08_LG	au_ppm	220	0.003	116.500	1.780	8.219
BR_09_LG	au_ppm	149	0.003	67.167	2.019	6.206
BR_10_LG	au_ppm	81	0.003	4.980	0.923	1.095
BR_11_LG	au_ppm	71	0.003	14.647	1.061	2.109
BR_12_LG	au_ppm	31	0.02	11.518	2.158	3.197
MARKER UNIT 1	au_ppm	177	0.003	13.850	0.650	1.280
MARKER UNIT 2	au_ppm	51	0.01	122.500	3.900	17.530
MARKER UNIT 3	au_ppm	16	0.01	1.410	0.370	0.400
EG_00_LG	au_ppm	442	0.003	117.500	1.332	6.060
EG_01_LG	au_ppm	292	0.005	779.966	4.213	46.497
EG_02_LG	au_ppm	250	0.005	63.300	1.622	6.841
EG_02A_LG	au_ppm	103	0.005	5.336	0.390	0.783
EG_03_LG	au_ppm	377	0.005	316.757	2.443	20.024
EG_03A_LG	au_ppm	131	0.005	709.000	7.852	62.019
EG_04_LG	au_ppm	287	0.005	800.000	5.379	48.705
EG_04A_LG	au_ppm	34	0.005	18.318	1.676	4.282
EG_05_LG	au_ppm	389	0.005	76.000	1.493	6.454
EG_06_LG	au_ppm	383	0.003	74.900	1.247	5.856
EG_06A_LG	au_ppm	200	0.003	262.000	2.840	19.616
EG_07_LG	au_ppm	517	0.003	85.900	1.554	6.225
EG_07A_LG	au_ppm	237	0.005	156.500	1.934	11.172
EG_07B_LG	au_ppm	280	0.003	51.886	0.416	3.192
EG_08_LG	au_ppm	150	0.005	56.671	1.251	5.572
EG_09_LG	au_ppm	142	0.005	48.690	0.768	4.225
EG_09A_LG	au_ppm	73	0.005	33.631	0.936	4.043
EG_10_LG	au_ppm	108	0.005	252.759	5.120	27.448
EG_10A_LG	au_ppm	77	0.003	102.429	2.421	12.017
EG_11_LG	au_ppm	96	0.005	1,570.000	17.821	160.141
EG_11A_LG	au_ppm	40	0.003	412.000	11.730	64.991
EG_11B_LG	au_ppm	32	0.01	6.009	0.833	1.226
EG_12_LG	au_ppm	66	0.005	72.403	2.378	9.339

(table continues on next page)

Domain	Field	No of Records	Minimum	Maximum	Mean	Standard Deviation
EG_13_LG	au_ppm	23	0.003	133.108	10.664	30.426
EG_14_LG	au_ppm	15	0.005	5.937	1.038	1.717
EG_15_LG	au_ppm	10	0.01	6.469	0.977	1.959
WG-01-LG	Au (g/t)	51	0.003	33.383	1.812	5.218
WG-02-LG	Au (g/t)	146	0.003	154.500	2.566	13.604
WG-03-A-LG	Au (g/t)	129	0.003	84.489	2.164	8.679
WG-03-B-LG	Au (g/t)	17	0.06	3.722	0.830	0.905
WG-04-LG	Au (g/t)	98	0.003	11.146	1.312	2.398
WG-05-A-LG	Au (g/t)	179	0.003	1,005.000	7.523	75.308
WG-05-B-LG	Au (g/t)	10	0.01	3.005	0.714	1.190
WG-06-LG	Au (g/t)	209	0.003	180.000	3.577	15.012
WG-07-LG	Au (g/t)	169	0.003	44.100	0.956	3.849
WG-08-LG	Au (g/t)	132	0.003	78.069	2.041	7.653
WG-09-LG	Au (g/t)	24	0.005	15.329	2.227	4.046
WG-10-LG	Au (g/t)	3	0.16	2.806	1.671	1.362

Due to the historical sampling practices, some non-assayed or non-mineralized intervals were incorporated into the interpreted solids. It is WSP’s option that non-assayed material should not be assigned a zero value. Non-assayed intervals were assigned a grade of half the detection limit value across the entire missing interval (0.0025 g/t).

14.6.2 COMPOSITING

Gold assay data were composited into one-metre downhole intervals honouring the interpreted geological solids. A 1 m composite length was selected as most of the assays are in the 1 m range for length. A composite length should correspond to approximately one-half to one-third the cell size to be used in the modeling process. In the case of Goldboro, the composite lengths match the block size.

The result is individual boreholes have composites that vary in length, yet have a mean composite length of 1 m. Composite intervals less than 0.75 m in length are flagged as type 2 by Surpac. It is important to retain the “tails” in a gold system, as it is documented that gold grades can be elevated at the contacts. These “tails are retained in the estimation process using a length weighting algorithm.

Table 14.5 summarizes the statics of the composite samples for each of the belts.

Table 14.5 Goldboro Composite Sample Statistics

Domain	Field	No. of Records	Minimum	Maximum	Mean	Standard Deviation
BR_01	Au (g/t)	1545	0.003	153.519	1.747	8.417
BR_01A	Au (g/t)	49	0.02	6.86	0.891	1.127
BR_02	Au (g/t)	2114	0.003	876	2.705	28.312
BR_02A	Au (g/t)	80	0.003	6.981	0.501	1.163
BR_03	Au (g/t)	1820	0.003	145.506	2.013	7.614
BR_03A	Au (g/t)	842	0.003	42.43	1.7	4.052
BR_04	Au (g/t)	1530	0.003	213.055	1.889	8.934

(table continues on next page)

Domain	Field	No. of Records	Minimum	Maximum	Mean	Standard Deviation
BR_04A	Au (g/t)	184	0.003	25.841	2.142	3.939
BR_05	Au (g/t)	788	0.003	190.97	1.88	11.871
BR_06	Au (g/t)	442	0.003	78.17	1.618	6.523
BR_07	Au (g/t)	500	0.003	55.7	1.435	4.346
BR_08	Au (g/t)	284	0.003	29.563	1.814	4.764
BR_09	Au (g/t)	253	0.003	37.087	0.824	2.751
BR_10	Au (g/t)	104	0.003	4.98	0.727	0.948
BR_11	Au (g/t)	97	0.003	7.677	0.622	1.231
BR_12	Au (g/t)	23	0.02	10.163	1.807	2.422
MARKER UNIT 1	Au (g/t)	369	0.003	13.85	0.4	0.957
MARKER UNIT 2	Au (g/t)	83	0.003	98.026	2.516	11.405
MARKER UNIT 3	Au (g/t)	19	0.003	1.41	0.327	0.399
EG_00	Au (g/t)	639	0.003	117.5	0.876	5.018
EG_01	Au (g/t)	276	0.003	390.367	2.203	23.635
EG_02	Au (g/t)	242	0.003	31.885	0.98	3.756
EG_02A	Au (g/t)	119	0.003	3.322	0.245	0.495
EG_03	Au (g/t)	391	0.003	158.68	1.262	9.823
EG_03A	Au (g/t)	158	0.003	181.563	2.365	14.71
EG_04	Au (g/t)	298	0.003	200.728	2.11	13.193
EG_04A	Au (g/t)	42	0.003	9.369	0.808	2.189
EG_05	Au (g/t)	371	0.003	61.5	1.044	4.576
EG_06	Au (g/t)	433	0.003	23.539	0.523	2.149
EG_06A	Au (g/t)	267	0.003	65.808	1.048	5.703
EG_07	Au (g/t)	604	0.003	50.603	0.834	3.636
EG_07A	Au (g/t)	241	0.003	59.97	1.177	5.111
EG_07B	Au (g/t)	394	0.003	51.886	0.28	2.677
EG_08	Au (g/t)	148	0.003	56.671	1.057	5.3
EG_09	Au (g/t)	156	0.003	15.09	0.443	1.51
EG_09A	Au (g/t)	75	0.003	20.207	0.647	2.486
EG_10	Au (g/t)	125	0.003	101.146	2.355	10.614
EG_10A	Au (g/t)	95	0.003	102.429	2.057	11.291
EG_11	Au (g/t)	118	0.003	785.345	7.436	72.247
EG_11A	Au (g/t)	53	0.003	206.33	4.853	28.29
EG_11B	Au (g/t)	44	0.003	4.296	0.526	0.811
EG_12	Au (g/t)	83	0.003	43.51	1.227	4.978
EG_13	Au (g/t)	33	0.003	66.794	4.649	14.209
EG_14	Au (g/t)	16	0.005	5.937	0.962	1.535
EG_15	Au (g/t)	14	0.003	6.469	0.964	1.793
WG-01-LG	Au (g/t)	71	0.003	23.374	1.760	3.543
WG-02-LG	Au (g/t)	218	0.003	58.722	1.729	5.696
WG-03-A-LG	Au (g/t)	222	0.003	67.661	1.576	5.037
WG-03-B-LG	Au (g/t)	13	0.166	2.712	0.703	0.692
WG-04-LG	Au (g/t)	180	0.003	11.146	1.173	2.067

(table continues on next page)

Domain	Field	No. of Records	Minimum	Maximum	Mean	Standard Deviation
WG-05-A-LG	Au (g/t)	282	0.003	502.640	3.426	30.017
WG-05-B-LG	Au (g/t)	7	0.03	3.005	0.875	1.078
WG-06-LG	Au (g/t)	253	0.003	171.001	3.436	12.613
WG-07-LG	Au (g/t)	179	0.003	44.100	0.779	3.439
WG-08-LG	Au (g/t)	116	0.003	78.069	2.210	7.661
WG-09-LG	Au (g/t)	23	0.005	15.329	2.283	3.928
WG-10-LG	Au (g/t)	3	0.16	2.806	1.482	1.323

14.6.3 GRADE CAPPING

The composite data was examined to assess the amount of metal that is at risk from high-grade assays. Several processes were used to determine the capping level, including cumulative frequency plots, grade histograms, and Q-Q plots. WSP elected to apply a global top cut to each zone based on the statistics the entire composite data set. A value of 80 g/t gold was applied to the data set. Thirty-three composites had the cap applied or 0.2% of the data set.

In addition to the capping strategy, the distribution of high grade in the block model is controlled in the estimation process by using restricted search ellipse sizes and controlling the minimum and maximum number of composites per estimation. Table 14.6 summarizes the impact of the grade capping on each belt.

Table 14.6 Goldboro Capped Composite Statistics

Domain	Field	No of Records	Uncapped				Capped				No. of Records Capped
			Minimum	Maximum	Mean	Standard Deviation	Minimum	Maximum	Mean	Standard Deviation	
BR_01	Au (g/t)	1545	0.003	153.519	1.747	8.417	0.003	80.000	1.568	5.582	5
BR_01a	Au (g/t)	49	0.02	6.860	0.891	1.127	0.02	6.860	0.891	1.127	0
BR_02	Au (g/t)	2114	0.003	876.000	2.705	28.312	0.003	80.000	1.810	5.271	5
BR_02a	Au (g/t)	80	0.003	6.981	0.501	1.163	0.003	6.981	0.501	1.163	0
BR_03	Au (g/t)	1820	0.003	145.506	2.013	7.614	0.003	80.000	1.917	6.186	4
BR_03a	Au (g/t)	842	0.003	42.430	1.700	4.052	0.003	42.430	1.700	4.052	0
BR_04	Au (g/t)	1530	0.003	213.055	1.889	8.934	0.003	80.000	1.704	5.435	3
BR_04a	Au (g/t)	184	0.003	25.841	2.142	3.939	0.003	25.841	2.142	3.939	0
BR_05	Au (g/t)	788	0.003	190.970	1.880	11.871	0.003	80.000	1.481	6.714	4
BR_06	Au (g/t)	442	0.003	78.170	1.618	6.523	0.003	78.170	1.618	6.523	0
BR_07	Au (g/t)	500	0.003	55.700	1.435	4.346	0.003	55.700	1.435	4.346	0
BR_08	Au (g/t)	284	0.003	29.563	1.814	4.764	0.003	29.563	1.814	4.764	0
BR_09	Au (g/t)	253	0.003	37.087	0.824	2.751	0.003	37.087	0.824	2.751	0
BR_10	Au (g/t)	104	0.003	4.980	0.727	0.948	0.003	4.980	0.727	0.948	0
BR_11	Au (g/t)	97	0.003	7.677	0.622	1.231	0.003	7.677	0.622	1.231	0
BR_12	Au (g/t)	23	0.02	10.163	1.807	2.422	0.02	10.163	1.807	2.422	0
MARKER UNIT 1	Au (g/t)	369	0.003	13.850	0.400	0.957	0.003	13.850	0.400	0.957	0
MARKER UNIT 2	Au (g/t)	83	0.003	98.026	2.516	11.405	0.003	80.000	2.299	9.591	1
MARKER UNIT 3	Au (g/t)	19	0.003	1.410	0.327	0.399	0.003	1.410	0.327	0.399	0
EG_00	Au (g/t)	639	0.003	117.500	0.876	5.018	0.003	80.000	0.818	3.698	1
EG_01	Au (g/t)	276	0.003	390.367	2.203	23.635	0.003	80.000	1.079	5.610	1
EG_02	Au (g/t)	242	0.003	31.885	0.980	3.756	0.003	31.885	0.980	3.756	0
EG_02A	Au (g/t)	119	0.003	3.322	0.245	0.495	0.003	3.322	0.245	0.495	0
EG_03	Au (g/t)	391	0.003	158.680	1.262	9.823	0.003	80.000	0.993	6.034	2
EG_03A	Au (g/t)	158	0.003	181.563	2.365	14.710	0.003	80.000	1.722	7.058	1
EG_04	Au (g/t)	298	0.003	200.728	2.110	13.193	0.003	80.000	1.705	7.842	1
EG_04A	Au (g/t)	42	0.003	9.369	0.808	2.189	0.003	9.369	0.808	2.189	0
EG_05	Au (g/t)	371	0.003	61.500	1.044	4.576	0.003	61.500	1.044	4.576	0
EG_06	Au (g/t)	433	0.003	23.539	0.523	2.149	0.003	23.539	0.523	2.149	0
EG_06A	Au (g/t)	267	0.003	65.808	1.048	5.703	0.003	65.808	1.048	5.703	0
EG_07	Au (g/t)	604	0.003	50.603	0.834	3.636	0.003	50.603	0.834	3.636	0
EG_07A	Au (g/t)	241	0.003	59.970	1.177	5.111	0.003	59.970	1.177	5.111	0

(table continues on next page)

Domain	Field	No of Records	Uncapped				Capped				No. of Records Capped
			Minimum	Maximum	Mean	Standard Deviation	Minimum	Maximum	Mean	Standard Deviation	
EG_07B	Au (g/t)	394	0.003	51.886	0.280	2.677	0.003	51.886	0.280	2.677	0
EG_08	Au (g/t)	148	0.003	56.671	1.057	5.300	0.003	56.671	1.057	5.300	0
EG_09	Au (g/t)	156	0.003	15.090	0.443	1.510	0.003	15.090	0.443	1.510	0
EG_09A	Au (g/t)	75	0.003	20.207	0.647	2.486	0.003	20.207	0.647	2.486	0
EG_10	Au (g/t)	125	0.003	101.146	2.355	10.614	0.003	80.000	2.185	9.085	1
EG_10A	Au (g/t)	95	0.003	102.429	2.057	11.291	0.003	80.000	1.821	9.214	1
EG_11	Au (g/t)	118	0.003	785.345	7.436	72.247	0.003	80.000	1.458	7.513	1
EG_11A	Au (g/t)	53	0.003	206.330	4.853	28.290	0.003	80.000	2.469	11.068	1
EG_11B	Au (g/t)	44	0.003	4.296	0.526	0.811	0.003	4.296	0.526	0.811	0
EG_12	Au (g/t)	83	0.003	43.510	1.227	4.978	0.003	43.510	1.227	4.978	0
EG_13	Au (g/t)	33	0.003	66.794	4.649	14.209	0.003	66.794	4.649	14.209	0
EG_14	Au (g/t)	16	0.005	5.937	0.962	1.535	0.005	5.937	0.962	1.535	0
EG_15	Au (g/t)	14	0.003	6.469	0.964	1.793	0.003	6.469	0.964	1.793	0
WG-01-LG	Au (g/t)	71	0.003	23.374	1.760	3.543	0.003	23.374	1.760	3.543	0
WG-02-LG	Au (g/t)	218	0.003	58.722	1.729	5.696	0.003	58.722	1.729	5.696	0
WG-03-A-LG	Au (g/t)	222	0.003	67.661	1.576	5.037	0.003	67.661	1.576	5.037	0
WG-03-B-LG	Au (g/t)	13	0.166	2.712	0.703	0.692	0.166	2.712	0.703	0.692	0
WG-04-LG	Au (g/t)	180	0.003	11.146	1.173	2.067	0.003	11.146	1.173	2.067	0
WG-05-A-LG	Au (g/t)	282	0.003	502.640	3.426	30.017	0.003	80.000	1.928	5.720	1
WG-05-B-LG	Au (g/t)	7	0.03	3.005	0.875	1.078	0.03	3.005	0.875	1.078	0
WG-06-LG	Au (g/t)	253	0.003	171.001	3.436	12.613	0.003	80.000	3.077	8.415	1
WG-07-LG	Au (g/t)	179	0.003	44.100	0.779	3.439	0.003	44.100	0.779	3.439	0
WG-08-LG	Au (g/t)	116	0.003	78.069	2.210	7.661	0.003	78.069	2.210	7.661	0
WG-09-LG	Au (g/t)	23	0.005	15.329	2.283	3.928	0.005	15.329	2.283	3.928	0
WG-10-LG	Au (g/t)	3	0.16	2.806	1.482	1.323	0.16	2.806	1.482	1.323	0

14.7 SPATIAL ANALYSIS

Variography, using Surpac™ software, was completed for gold in all seven zones individually. Downhole variograms were used to determine nugget effect and then variograms were modelled to determine spatial continuity in the zones. Table 14.7 summarizes results of the variography. The initial variograms were reviewed in an "unfolded" space. This allows drillholes from either limb of the fold nose influence the variography.

From the "unfolded" space, the preferred orientation is in the down plunge direction of the fold axis. This preferred orientation was then used to evaluate the variograms for each limb and the fold nose.

The search ellipses are based on the geometry of the solids and the results of the variogram models. Table 14.8 summarizes the search ellipse dimensions and rotation angles for each limb and the fold nose.

Table 14.7 Variogram Summary

Domain	Element	Nugget	Sill		Range	
			1st. S	2nd. S	1st. S	2nd. S
North Limb	Au (g/t)	0.074	0.236	0.687	30.643	92.522
South Limb	Au (g/t)	0.439	0.424	0.137	30.643	92.522
Center Axis	Au (g/t)	0.142	0.697	0.160	71.844	101.076

Table 14.8 Search Ellipse Summary

Domain	Element	Bearing	Plunge	Dip	Major axis	Semi-major axis	Minor Axis	Anisotropy Ratio	
								Major / Semi-major	Major / Minor
North Limb	Au (g/t)	288.96	24.18	-74.96	92.52	25.30	19.07	3.66	4.85
South Limb	Au (g/t)	97.59	-27.10	-48.01	92.52	32.20	24.44	2.87	3.79
Center Axis	Au (g/t)	101.40	-28.52	0.00	101.08	29.28	18.34	3.45	5.51

14.8 MINERAL RESOURCE BLOCK MODEL

A single block model was established in Surpac™ for all 57 belts using one parent model as the origin.

Drillhole spacing varies with most the drilling tightly spaced from 15 m. A block size of 1 x 1 x 1 m was selected in order to accommodate the more closely spaced drilling and the narrow nature of the mineralization. Sub-celling of the block model was not used. Table 14.9 summarizes the parent block model.

Table 14.9 Goldboro Parent Model Summary

Parameter	
Minimum X Coordinate	292,755
Minimum Y Coordinate	5,006,555
Minimum Z Coordinate	4,350
Maximum X Coordinate	294,895
Maximum Y Coordinate	5,007,395
Maximum Z Coordinate	5,140
Block Size (m)	1 x 1 x 1
Rotation	0
Sub-block	none
Total No. Blocks	1,420,104,000

14.8.1 ESTIMATION PARAMETERS

The interpolations of the 57 belts were completed using the estimation methods: nearest neighbour (NN), inverse distance squared (ID^2) and ordinary kriging (OK). The estimations were designed for four passes. In each pass, a minimum and maximum number of samples were required as well as a maximum number of samples from a borehole in order to satisfy the estimation criteria. Only the composite samples within a belt were used to estimate the blocks with that belt.

Table 14.10 summarizes the interpolation criteria for all 57 belts within the three domains.

Table 14.10 Goldboro Estimation Parameter Summary

Domains		Pass No. Estimation	Search Ellipse Factor	Minimum No. of Composites	Composites	Maximum No. of Composites per BH
Boston-Richardson	South Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	North Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	Center Axis	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
East Goldbrook	South Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	North Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	Center Axis	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
West Goldbrook	South Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	North Limb	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2
	Center Axis	1	0.4	3	12	2
		2	0.75	3	12	2
		3	1	3	12	2
		4	1.08	2	12	2

14.9 MINERAL RESOURCE CLASSIFICATION

Several factors are considered in the definition of a resource classification.

— Canadian Institute of Mining, Metallurgy and Petroleum (CIM - 2014) requirements and guidelines.

— The author's experience with the Proterozoic gold deposits.

— Spatial continuity based on variography of the assays.

— Classification of the Inferred mineral resources was based in the following parameters:

— All blocks are classified as Inferred at the start.

Classification of the Indicated mineral resource was based in the following parameters:

— All material estimated in vario running 2
  (a) 75% search ellipse size
  (b) Minimum 3 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

  or

— Material estimated in vario running 3
  (a) 100% search ellipse size
  (b) Minimum 9 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

Classification of the Measured mineral resource was based in the following parameters:

— All material estimated in vario running 1
  (a) 40% search ellipse size
  (b) Minimum 3 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

  or

— Material estimated in vario running 2
  (a) 75% search ellipse size
  (b) Minimum 10 composites
  (c) Maximum 12 composites
  (d) Maximum 2 composites from per drillhole

Upon completion of the classification macro, the model was reviewed and "orphan" blocks of Inferred resources were converted to Indicated resources, and "orphan" Indicated blocks converted to Inferred resource.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, or other relevant issues are known to the authors that may affect the estimate of mineral resources. Mineral reserves can only be estimated on the basis of an economic evaluation that is used in a preliminary feasibility study or a feasibility study of a mineral project; thus, no reserves have been estimated. As per NI 43-101, mineral resources, which are not mineral reserves, do not have demonstrated economic viability.

14.10 MINERAL RESOURCE TABULATION

The mineral resource has an effective date of August 21, 2019 has been tabulated in terms of a gold cut-off grade created from a pit constrained Whittle pit shell and mineable blocks for underground.

A pit shell was generated using the Lerchs-Grossman (LG) algorithm in Whittle. Table 14.11 is a summary of the parameters used in the pit optimization process and underground block. The optimized pit shell suggests a resource using a 0.5 g/t gold cut-off and 2.00 g/t gold for underground using a CAN$1,753 gold price.

Table 14.11 Potential Mining Parameters

Parameter	Units	Base Case Onsite Processing
Process Production Rate	tpd	575
Mining Dilution	%	15
Mining Recovery	%	95
Overall Slope Angle	Degrees	42
Mining Cost	$/tonne mined	5
UG Mining Cost	$/tonne	125
Processing Cost (including additional costs for G&A, shipping, etc.)	$/tonne processed	30
Metallurgical Recovery	%	93
Payable Factor	%	98
Metal Prices - Gold	$/oz.	1753
Selling Cost	$/oz.	20.94
Mineral Resource Classifications Used in Optimization		Measured
		Indicated
		Inferred

Table 14.12 summarizes the mineral resources within the constrained pit and underground block by mineral resource classification. Table 14.13 provide the breakdown of the mineral resource by resource classification and domain (BR Gold System, EG Gold System and WG Gold System). Appendix B contains the detailed mineral resource breakdown by resource classification and belt.

Table 14.12 Goldboro Mineral Resource Summary

Resource Type	Au Cut-off (g/t)	Category	Tonnes ('000)	Au (g/t)	Troy Ounces	% Change in Grade from Dec. 2018**	% Change in Ounces from Dec.2018***
Open Pit	0.5	Measured	844.00	2.40	65,200	-	-
		Indicated	111.00	2.63	9,400	-	-
		Measured + Indicated	955.00	2.43	74,600	-	-
		Inferred	22.00	2.79	2,000	-	-
Underground	2	Measured	967.00	6.08	189,200	-	-
		Indicated	2,174.00	6.22	434,800	-	-
		Measured + Indicated	3,141.00	6.18	624,000	-	-
		Inferred	2,985.00	7.12	683,200	-	-
Combined	0.50/2.00	Measured	1,811.00	4.37	254,400	3.3%	16.0%
		Indicated	2,285.00	6.05	444,200	10.0%	15.9%
		Measured + Indicated	4,096.00	5.30	698,600	6.9%	15.9%
		Inferred	3,007.00	7.09	685,100	6.9%	51.2%

Table 14.13 Goldboro Mineral Resource Summary by Domain

Classification	Rock Type	Cut-off	Tonnes	Grade (Au g/t)	Ounces
Measured	BR	0.5	741,200	2.45	58,289
		2.0	628,200	6.09	122,962
		Total 0.5+2.0	1,369,400	4.12	181,251
	EG	0.5	58,200	2.32	4,349
		2.0	231,100	6.12	45,443
		Total 0.5+2.0	289,300	5.35	49,791
	WG	0.5	44,500	1.80	2,570
		2.0	108,000	6.00	20,817
		Total 0.5+2.0	152,500	4.77	23,386
	Total	0.5	843,900	2.40	65,208
		2.0	967,300	6.08	189,221
		Total 0.5+2.0	1,811,200	4.37	254,429
Indicated	BR	0.5	32,000	1.61	1,654
		2.0	806,700	5.76	149,316
		Total 0.5+2.0	838,700	5.60	150,970
	EG	0.5	7,100	1.70	388
		2.0	964,600	6.90	213,897
		Total 0.5+2.0	971,700	6.86	214,286
	WG	0.5	72,000	3.18	7,364
		2.0	402,800	5.53	71,552
		Total 0.5+2.0	474,800	5.17	78,915
	Total	0.5	111,100	2.63	9,406
		2.0	2,174,100	6.22	434,765
		Total 0.5+2.0	2,285,200	6.05	444,171

(table continues on next page)

Classification	Rock Type	Cut-off	Tonnes	Grade (Au g/t)	Ounces
M&I	BR	0.5	773,200	2.41	59,944
		2.0	1,434,900	5.90	272,278
		Total 0.5+2.0	2,208,100	4.68	332,221
	EG	0.5	65,300	2.26	4,737
		2.0	1,195,700	6.75	259,340
		Total 0.5+2.0	1,261,000	6.51	264,077
	WG	0.5	116,500	2.65	9,933
		2.0	510,800	5.62	92,368
		Total 0.5+2.0	627,300	5.07	102,302
	Total	0.5	955,000	2.43	74,614
		2.0	3,141,400	6.18	623,986
		Total 0.5+2.0	4,096,400	5.30	698,600
Inferred	BR	0.5	3,900	0.98	123
		2.0	1,071,500	5.44	187,546
		Total 0.5+2.0	1,075,400	5.43	187,669
	EG	0.5	4,800	2.99	461
		2.0	1,292,000	8.76	363,719
		Total 0.5+2.0	1,296,800	8.73	364,180
	WG	0.5	13,200	3.26	1,384
		2.0	621,300	6.60	131,899
		Total 0.5+2.0	634,500	6.53	133,283
	Total	0.5	21,900	2.79	1,967
		2.0	2,984,800	7.12	683,164
		Total 0.5+2.0	3,006,700	7.09	685,131

Mineral Resource Estimate Notes

1. *Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards (2014). Mineral resources that are not mineral reserves do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
2. *Open pit Mineral Resources are reported at a cut-off grade of 0.5 g/t gold that is based on a gold price of CAN$1,753/oz (~US$1,350/oz). and a gold processing recovery factor of 95%.*
3. *Underground Mineral Resource is reported at a cut-off grade of 2.0 g/t gold that is based on a gold price of CAN$1,753/oz (~US$1,350/oz). and a gold processing recovery factor of 95%.*
4. *Appropriate mining costs, processing costs, metal recoveries, and inter ramp pit slope angles were used by WSP to generate the pit shell.*
5. *Appropriate mining costs, processing costs, metal recoveries and stope dimensions were used by WSP to generate the potential underground resource.*
6. *Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.*
7. *Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.*
8. *Contributing assay composites were capped at 80 g/t Au.*
9. *A bulk density factor was calculated for each block based on a regression formula.*

Figure 14.5 displays the pit constrained resource by grade attributed, while Figure 14.6 displays the pit constrained resource by resource classification. Figure 14.7 displays the underground resource by grade attributed, while Figure 14.8 displays the underground resource by resource classification.

Figure 14.5 Goldboro Pit Constrained Mineral Resource with Grade

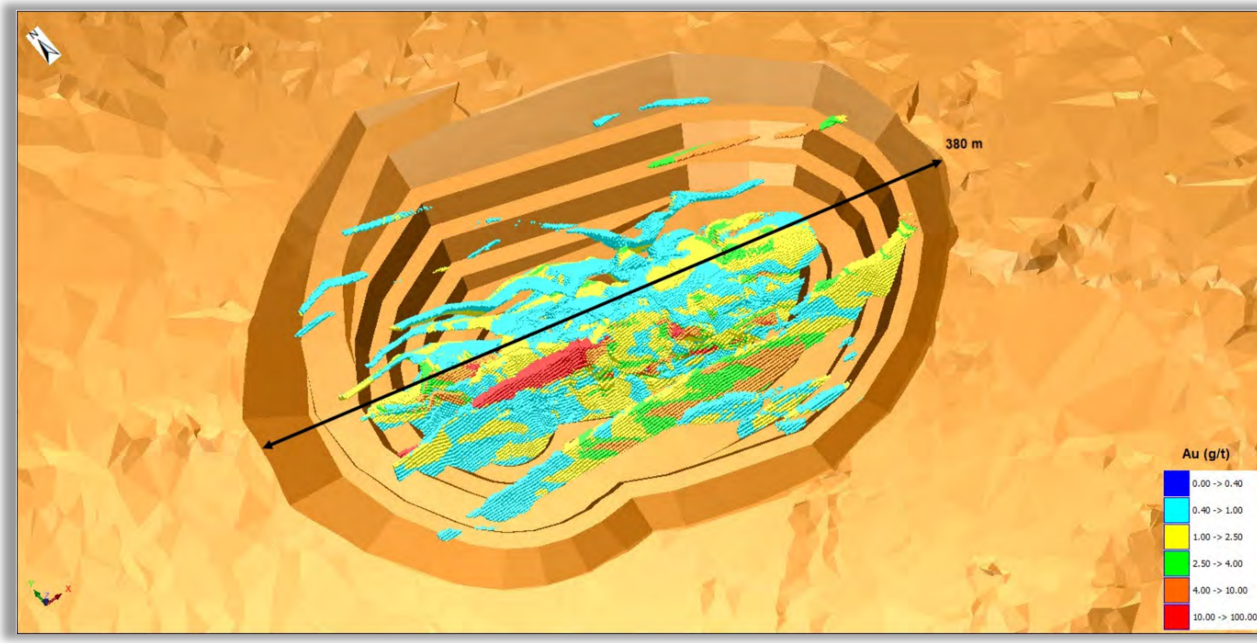


Figure 14.6 Goldboro Pit Constrained Mineral Resource with Mineral Resource Classification

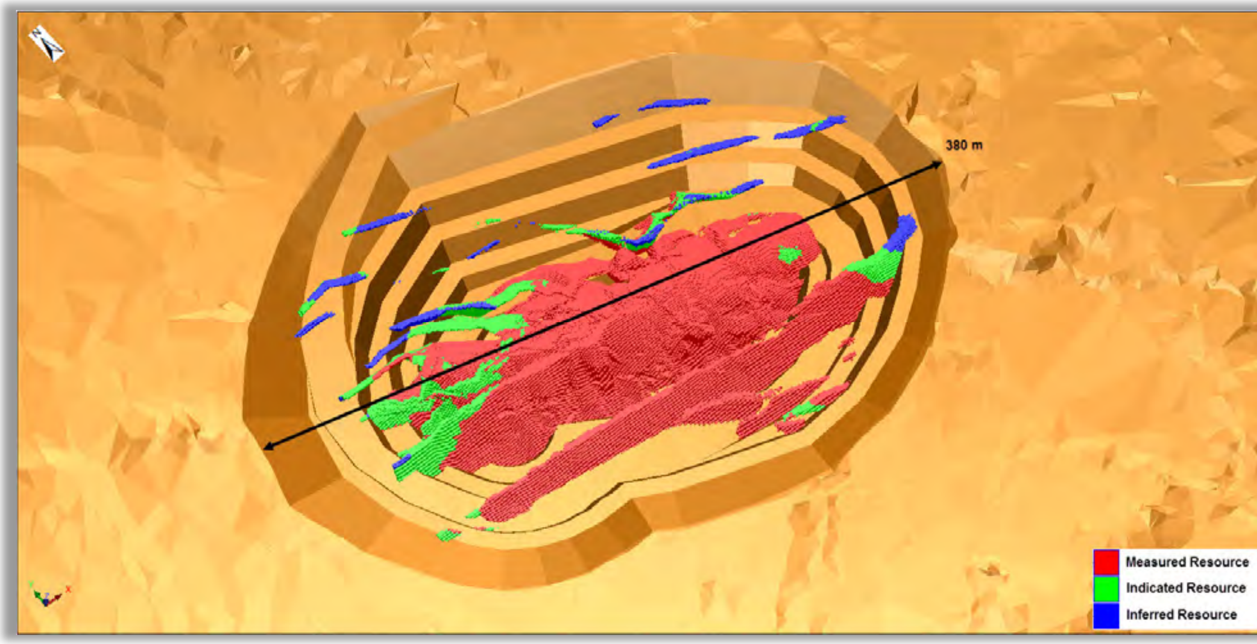


Figure 14.7 Goldboro Underground Mineral Resource with Grade

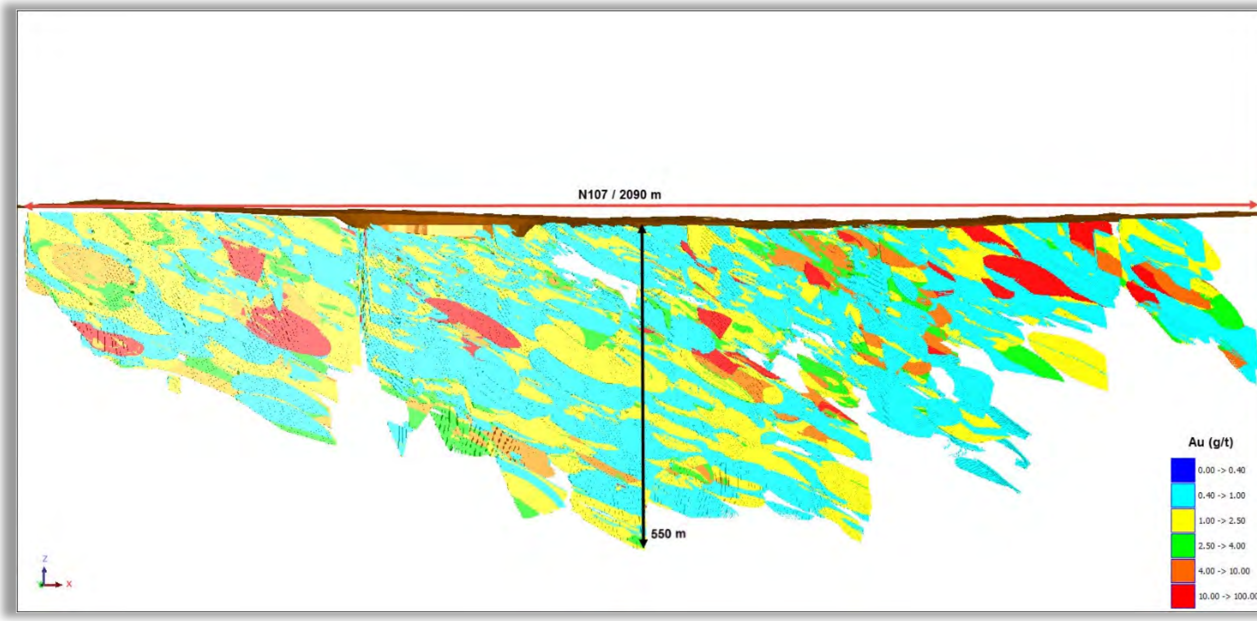
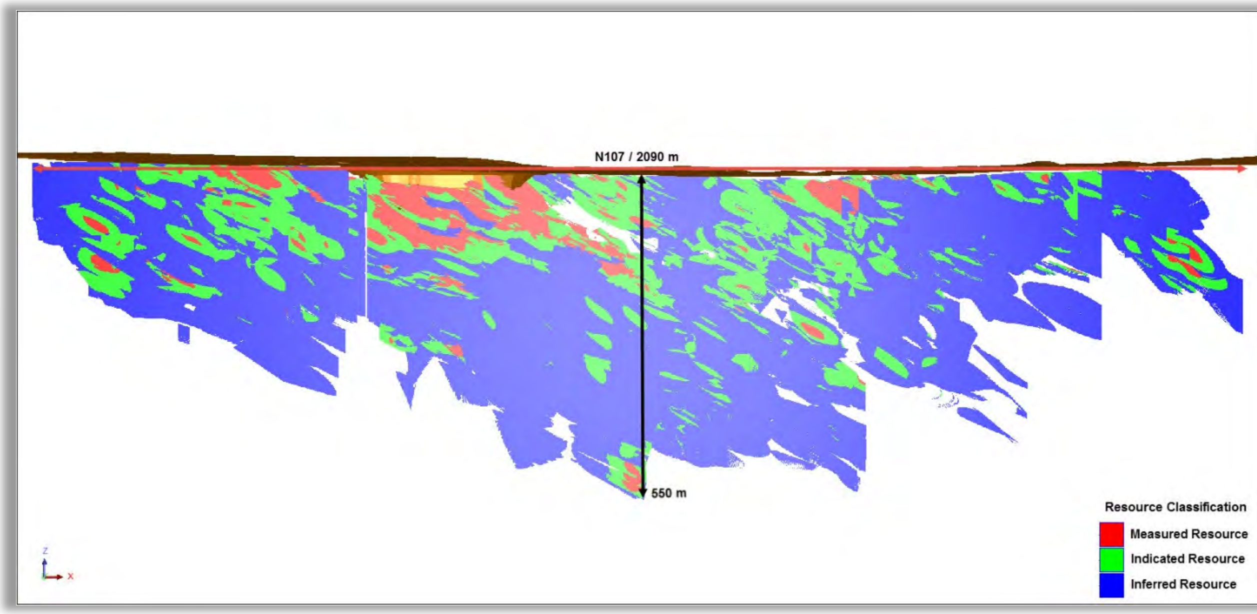


Figure 14.8 Goldboro Underground Mineral Resource with Mineral Resource Classification



14.11 VALIDATION

The Goldboro Deposit gold grade models were validated by three methods.

— Visual comparison of colour-coded block model grades with composite grades on section and plan.

— Comparison of the global mean block grades for OK, ID^2, NN and composites.

— Swath plots in the section and elevation orientation.

14.11.1 VISUAL VALIDATION

The visual comparisons of block model grades with composite grades for each of the zones show a reasonable correlation between the values. No significant discrepancies were apparent from the sections and plans reviewed, yet grade smoothing is apparent.

Figure 14.9 is a section on 293200 E through the West Goldbrook domain. Figure 14.10 is a section on 293400 E through the Boston-Richardson domain. Figure 14.11 is a section on 294150 E through the East Goldbrook domain.

Figure 14.9 Section on 293200 E through the West Goldbrook Domain

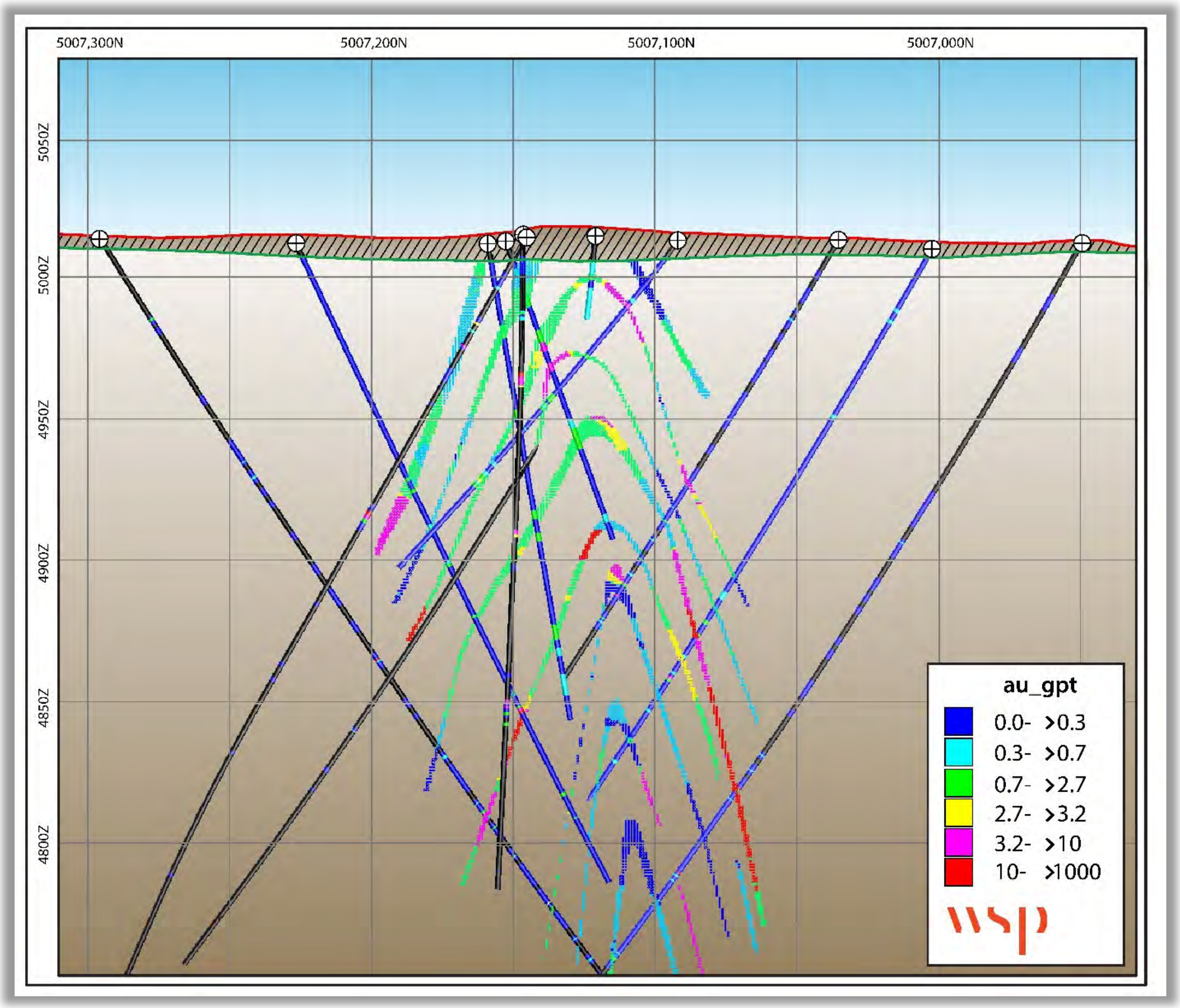


Figure 14.10 Section on 293400 E through the Boston-Richardson Domain

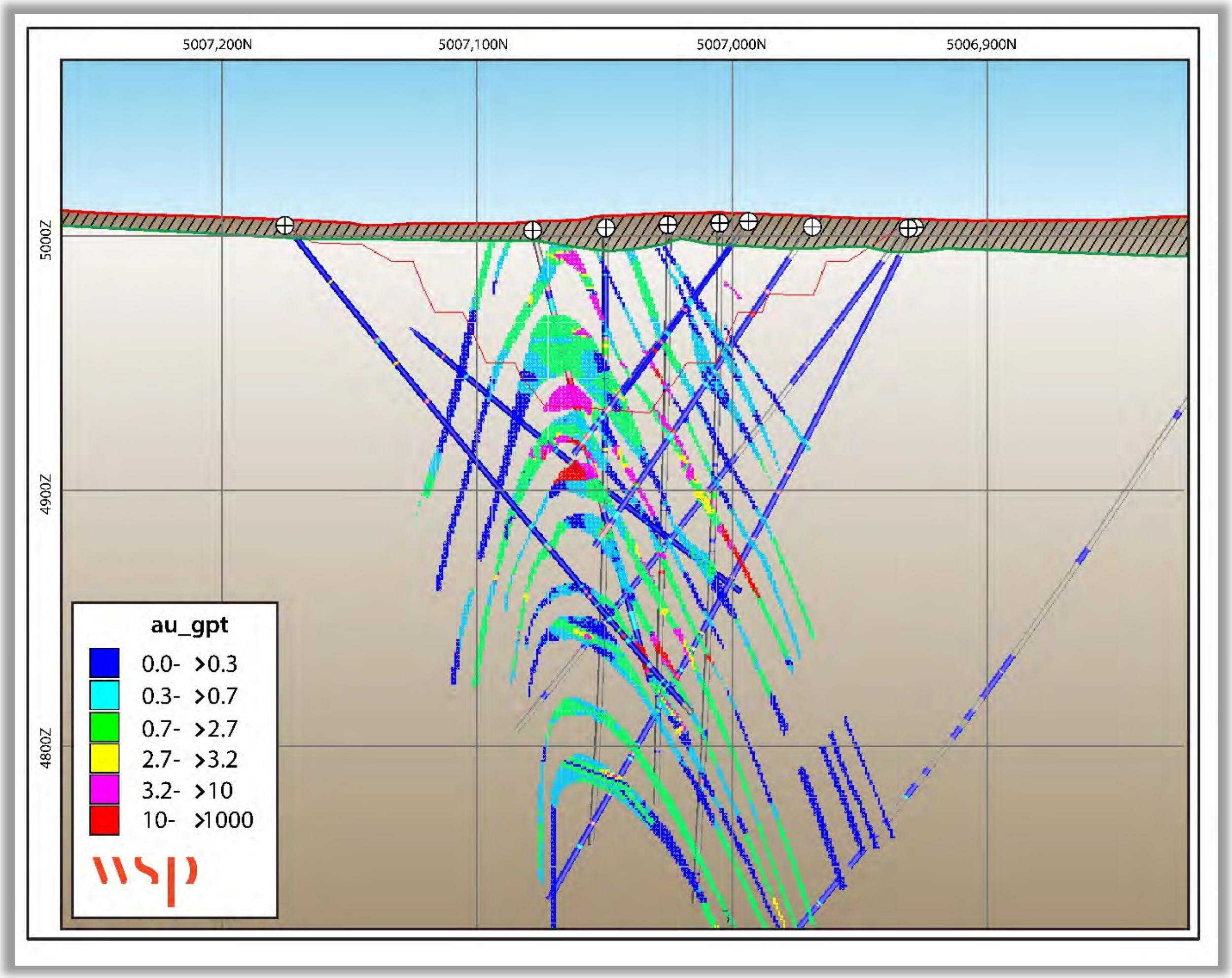
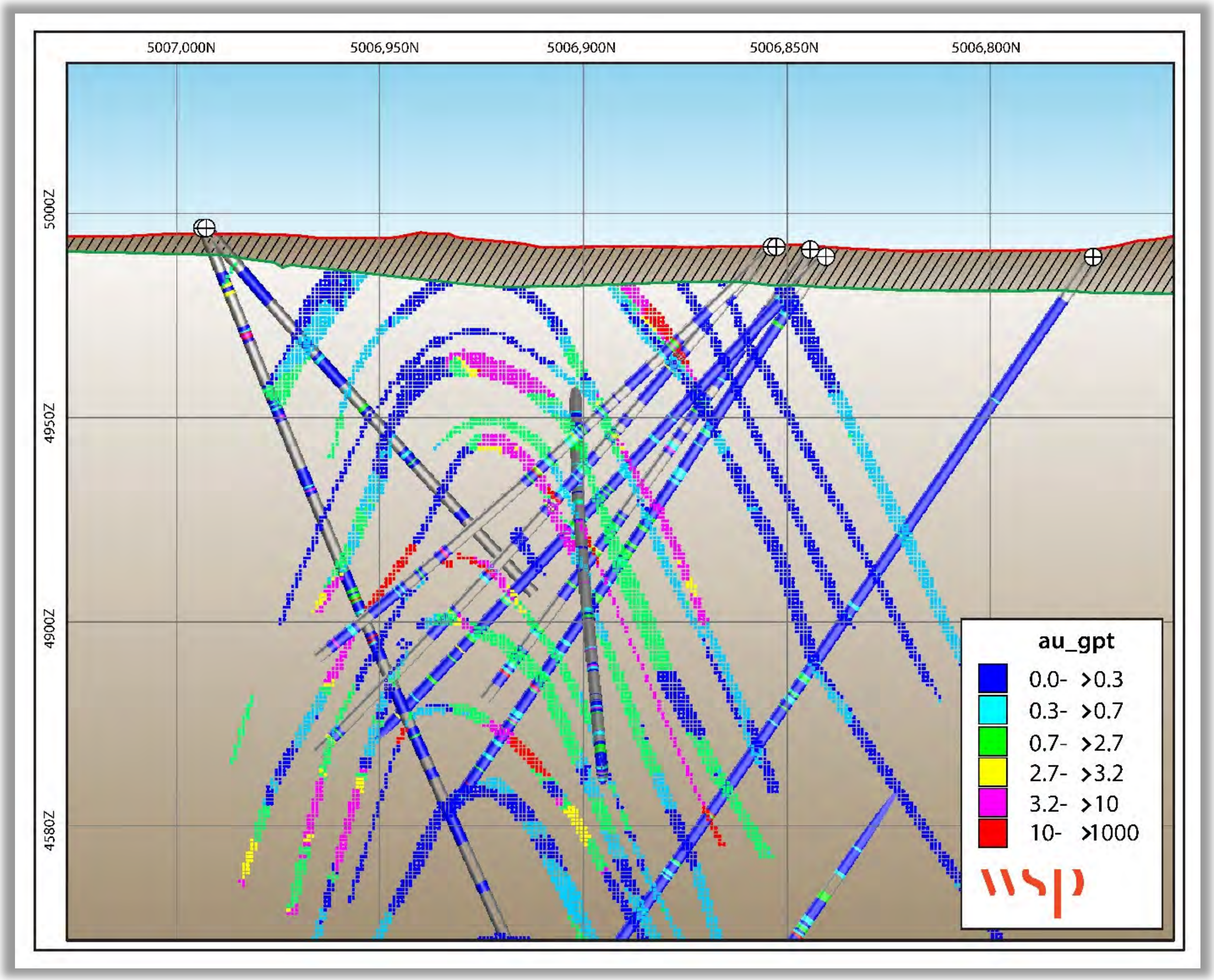


Figure 14.11   Section on 294150 E through the East Goldbrook Domain



14.11.2 GLOBAL COMPARISON

The global block model statistics for the ordinary kriging model were compared to the global ID^2 and NN model values as well as the composite drill data. Table 14.14 shows the comparisons of the global estimates for the three estimation method calculations of NN, ID^2, and OK with the composite samples for each belt. In general, there is agreement between the OK model and ID^2 model and NN model. Larger discrepancies are reflected as a result of lower drill density in some portions of the model. There is a degree of smoothing apparent from the ordinary kriging, which reflects the data density to a great extent. Comparisons were made using all blocks above a 0.0 g/t cut-off.

Table 14.14 Goldboro Global Comparison

Domain Cap/Composite	Element Cap/Composite	DDH cap/composite	NN Grade	ID^2 Grade	OK Grade
BR_01_LG	au_ppm	1.568	1.432	1.533	1.580
BR_01A_LG	au_ppm	0.891	1.322	1.022	1.050
BR_02_LG	au_ppm	1.810	1.773	1.549	1.539
BR_02A_LG	au_ppm	0.501	0.853	0.920	0.962
BR_03_LG	au_ppm	1.917	1.860	1.899	1.940
BR_03A_LG	au_ppm	1.700	1.425	1.400	1.367
BR_04_LG	au_ppm	1.704	1.753	1.498	1.535
BR_04A_LG	au_ppm	2.142	1.971	1.747	1.756
BR_05_LG	au_ppm	1.481	1.623	1.597	1.393
BR_06_LG	au_ppm	1.618	2.473	2.035	1.809
BR_07_LG	au_ppm	1.435	1.603	1.487	1.542
BR_08_LG	au_ppm	1.814	1.548	1.413	1.330
BR_09_LG	au_ppm	0.824	0.880	0.924	0.896
BR_10_LG	au_ppm	0.727	0.739	0.708	0.732
BR_11_LG	au_ppm	0.622	0.606	0.656	0.672
BR_12_LG	au_ppm	1.807	1.414	1.695	1.674
MARKER UNIT 1	au_ppm	0.400	0.494	0.477	0.512
MARKER UNIT 2	au_ppm	2.299	1.865	1.500	1.499
MARKER UNIT 3	au_ppm	0.327	0.308	0.328	0.331
EG_00_LG	au_ppm	0.818	0.822	0.796	0.696
EG_01_LG	au_ppm	1.079	0.779	0.813	0.797
EG_02_LG	au_ppm	0.980	1.452	1.052	0.952
EG_02A_LG	au_ppm	0.245	0.181	0.217	0.226
EG_03_LG	au_ppm	0.993	1.216	1.204	1.315
EG_03A_LG	au_ppm	1.722	2.534	1.783	1.805
EG_04_LG	au_ppm	1.705	1.765	1.568	1.709
EG_04A_LG	au_ppm	0.808	1.281	1.336	1.372
EG_05_LG	au_ppm	1.044	0.743	0.780	0.703
EG_06_LG	au_ppm	0.523	0.809	0.619	0.607
EG_06A_LG	au_ppm	1.048	1.639	1.170	1.196
EG_07_LG	au_ppm	0.834	0.822	0.760	0.770
EG_07A_LG	au_ppm	1.177	1.806	1.365	1.044
EG_07B_LG	au_ppm	0.280	1.375	0.794	0.828
EG_08_LG	au_ppm	1.057	0.837	0.647	0.711
EG_09_LG	au_ppm	0.443	0.601	0.460	0.421
EG_09A_LG	au_ppm	0.647	0.964	0.807	1.035
EG_10_LG	au_ppm	2.185	4.013	3.941	3.975
EG_10A_LG	au_ppm	1.821	5.325	4.157	3.173
EG_11_LG	au_ppm	1.458	1.689	1.252	1.299
EG_11A_LG	au_ppm	2.469	3.314	6.035	7.726
EG_11B_LG	au_ppm	0.526	0.459	0.469	0.469

(table continues on next page)

Domain Cap/Composite	Element Cap/Composite	DDH cap/composite	NN Grade	ID^2 Grade	OK Grade
EG_12_LG	au_ppm	1.227	1.258	1.084	1.084
EG_13_LG	au_ppm	4.649	7.466	5.551	5.555
EG_14_LG	au_ppm	0.962	1.317	0.938	0.772
EG_15_LG	au_ppm	0.964	0.987	0.833	0.756
WG-01-LG	Au (g/t)	1.760	1.615	1.549	1.431
WG-02-LG	Au (g/t)	1.729	1.441	1.815	2.018
WG-03-A-LG	Au (g/t)	1.576	1.267	1.364	1.510
WG-03-B-LG	Au (g/t)	0.703	0.518	0.487	0.478
WG-04-LG	Au (g/t)	1.173	1.409	1.324	1.338
WG-05-A-LG	Au (g/t)	1.928	2.259	2.514	2.363
WG-05-B-LG	Au (g/t)	0.875	2.022	1.496	1.278
WG-06-A-LG	Au (g/t)	3.077	2.441	2.530	2.580
WG-07-LG	Au (g/t)	0.779	1.025	1.178	1.352
WG-08-LG	Au (g/t)	2.210	2.095	2.354	2.473
WG-09-LG	Au (g/t)	2.283	5.758	4.819	4.584
WG-10-LG	Au (g/t)	1.482	0.820	1.145	1.310

14.11.3 SWATH PLOTS

Swath plots of eastings and elevations were generated for each mineralized zone respectively. These plots are comparing the OK estimates with the NN, ID^2 estimates and the drillhole composites. Figures are found in Appendix C.

The plots display a good correlation between the three estimation methodologies. There is evidence of grade smoothing compared to the borehole date, which is a typical artifact in the grade estimation process.

14.12 PREVIOUS ESTIMATES

The previous Goldboro mineral resource estimation was completed by WSP in 2018 (*McCracken et al., 2018*).

Table 14.15 compares the basic parameters used in the previous 2018 estimate with the current 2019 NI 43-101 mineral resource, which would explain some of the differences in the results. Table 14.16 illustrates the differences in the 2018 resource estimate with the current NI 43-101 mineral resource from 2019.

Table 14.15 Comparison of Model Parameters

Description	2018 WSP Model	2019 WSP Model
Number of drillholes	435 drillholes	492 drillholes
Grade capping	80 g/t	80 g/t
Composite lengths	1 m	1 m
Cut-off grade	0.5 g/t pit constrained	0.5 g/t pit constrained
	2.0 g/t underground	2.0 g/t underground
Number of mineral zones	3 domains - 38 belts	3 domains - 57 belts
Block size	1 x 1 x 1 (1 m^3)	1 x 1 x 1 (1 m^3)
Estimation passes	4	4
Search ellipse size - Pass 1	38 m x 11 m x 8 m	38 m x 11 m x 8 m
Search ellipse size - Pass 2	71 m x 21 m x 15 m	71 m x 21 m x 15 m
Search ellipse size - Pass 3	95 m x 28 m x 20 m	95 m x 28 m x 20 m
Search ellipse size - Pass 4	102 m x 30 m x 21 m	102 m x 30 m x 21 m
Minimum Composites	3	3
Maximum Composites	12	12
Max Composites/borehole	2	2
Estimation method	OK with ID^2 and NN validation	OK with ID^2 and NN validation

Table 14.16 Comparison of Previous Resource

Classification	Cut-off (Au g/t)	WSP 2018			WSP 2019		
		Tonnes ('000)	Grade (Au g/t)	Ounces ('000)	Tonnes ('000)	Grade (Au g/t)	Ounces ('000)
Measured	0.5	608.7	2.80	54.9	844.0	2.40	65.2
	2.0	1,003.1	5.10	164.4	967.0	6.08	189.2
Indicated	0.5	247.6	3.72	29.6	111.0	2.63	9.4
	2.0	1,918.6	5.74	353.8	2174.0	6.22	434.8
Measured and Indicated	**0.5**	**856.3**	**3.07**	**84.5**	**955.0**	**2.43**	**74.6**
	2.0	**2,921.7**	**5.52**	**518.2**	**3141.0**	**6.18**	**624.0**
Inferred	0.5	58.5	4.10	7.7	22.0	2.79	2.0
	2.0	2,067.9	6.70	445.50	2985.0	7.12	683.2

The primary differences between the 2018 mineral resource model and the current mineral resource model is the increase in the number of boreholes and the addition of new belts based on a re-interpretation of the geology and the fold geometry.

15 ADJACENT PROPERTIES

On July 17, 2019 Anaconda announced they had the option to acquire from a private vendor a 100% interest in the Country Harbour Property, which comprises seven mineral licences totaling over 858 hectares. The Country Harbour Property is located 15 km northwest of Anaconda's Goldboro Gold Project (Figure 15.1). Anaconda may exercise the option by making total cash payments of$100,000 and issuing of $27,500 of Anaconda common shares over a 4-year period. A 2.0% Gross Metal Royalty ("GMR") has been granted to the optionor, with 1.0% of the GMR purchasable for $750,000 with a right of first refusal on the remaining 1.0%.

The Country Harbour Mines area was discovered in 1861 and produced a total of approximately 9,960 ounces of gold after mining 26,301 tonnes, mostly prior to 1900. Gold is hosted by north-south trending quartz veins in the folded sedimentary rocks of the Goldenville Formation, with disseminated pyrite, pyrrhotite, chalcopyrite and galena. Located five kilometres along strike from the past producing Forest Hill Mine (*Parsons et al., 2012*).

On July 17, 2019 Anaconda also announced the option to acquire a 100% interest in the Lower Seal Harbour Property from Crosby Gold Ltd. (a 100%-owned subsidiary of Osprey Gold Development Ltd. The property comprises two mineral licences totaling over 291 hectares and is located five kilometres south of Anaconda's Goldboro Project (Figure 15.2). Anaconda may exercise the option by making total cash payments of $85,000 and issuing of $85,000 worth of Anaconda common shares over a 3-year period. Anaconda has assumed a 2.0% GMR granted to a private vendor under the terms of a preceding underlying agreement, with right to buy-back 75% of the GMR for $375,000. Total work commitments on the Lower Seal Harbour property under the agreement are $150,000 over four years.

The Lower Seal Harbour Mine produced 34,295 ounces of gold, after mining 394,905 tonnes between 1894 and 1949. Gold is hosted in both the quartz veining and anticlinal host greywacke and argillite of the Goldenville Formation, with accessory arsenopyrite, pyrite and galena. Previously recognized mineralization is hosted solely on the limb of the fold with no past exploration undertaken in the hinge of the fold, one of the key mineralized environments at Goldboro (*Parsons et al., 2012*).

Immediately east of the Goldboro Project, St. Barbara Ltd. (through subsidiary Atlantic Mining NS Corp.) holds 34 claims and south east and additional 14 claims (Nova Scotia Geoscience Atlas) (Figure 15.2). There is no public disclosure of activity on these claims in 2019.

Immediately to the west, south and north of the Goldboro Project, MegumaGold Corp. (through subsidiary 1156219 B.C. Ltd.) holds a total of 174 claims (Nova Scotia Geoscience Atlas) (Figure 15.2). There is no public disclosure of activity on these claims in 2019.

Figure 15.1 Anaconda Property Locations, Eastern Shore, Nova Scotia

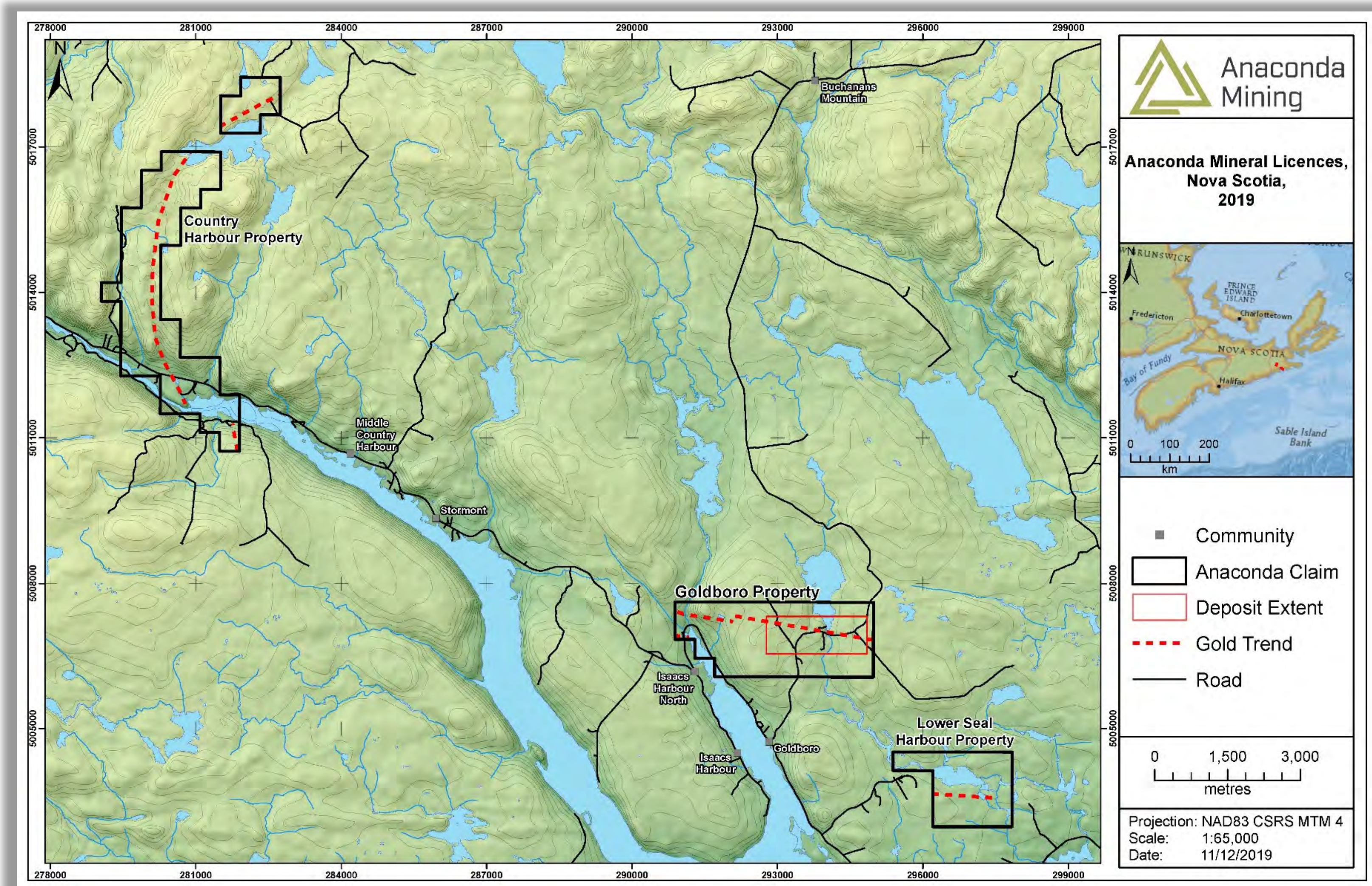
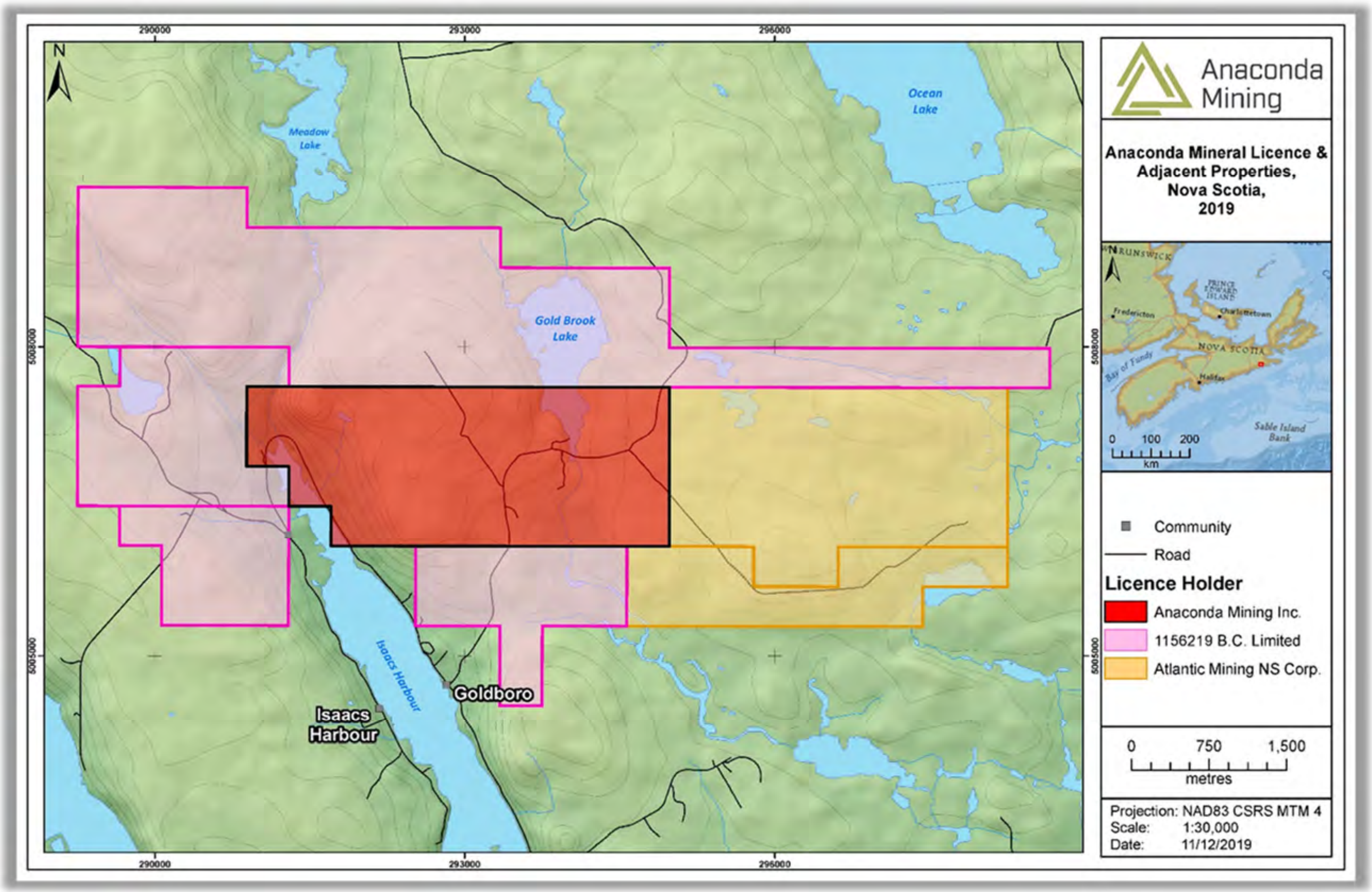


Figure 15.2 Adjacent Properties



16 OTHER RELEVANT DATA AND INFORMATION

In support of future engineering studies, Anaconda has undertaken the following studies:

— Hydrogeology;

— Hydrology;

— Geotechnical site investigation;

— Historic tailings review;

— Acid rock drainage and metals leaching;

— Rock mechanics;

— Pollutant dispersion model;

— Metallurgical studies;

— Environmental baseline studies including water, fauna, flora, air and noise;

— Archeological studies;

— MEKS or Mi'Kmaq Ecological Knowledge Studies.

17 INTERPRETATION AND CONCLUSIONS

17.1 GEOLOGY, EXPLORATION, AND DRILLING

Anaconda has completed 15,112 m of diamond drilling in 57 drillholes since the previous mineral resource estimate. Since acquiring the project in 2017 the Company has completed a total of 28,207.4 m in 107 diamond drillholes. All these drillholes were designed for infill and expansion purposes throughout the Goldboro Deposit area, and as sample material for metallurgical studies and are included in the current mineral resource estimate.

This work has resulted in the definition of 27 new mineralized belts in the Goldboro Gold Deposit – 21 new mineralized zones since the previous mineral resource estimate. The Deposit has expanded 375 m along strike to the east in the EG Gold System, up to 175 m at depth in the WG Gold System, up to 150 m at depth in the BR Gold System, and up to 100 m down-dip on the south limb of the BR Gold System. Infill drilling intersected multiple targeted mineralized belts and infilled a 100 m gap of near surface EG mineralized belts adjacent to the conceptual open pit, all of which has demonstrated the robust nature of the geological interpretation.

The Goldboro Deposit is hosted within the Goldenville Formation, which consists of alternating greywacke and argillite beds with an approximate true thickness of 950 m. The base of the stratigraphic sequence intersected in the BR Gold System consists of roughly 325 m of alternating greywacke and argillite, with varying proportions of both rock types, ranging in thickness from less than 1 m up to 10 m. This is overlain by the Marker Horizon, which consists of a 50 – 80 m greywacke bed that is commonly intersected during drilling and in underground workings. The Marker Horizon appears to thin or get cut off by the New Belt Fault on the northern limb of the anticline towards the west. Above the Marker Horizon is the EG Gold System, approximately 560 m thick, consisting of alternating greywackes and argillites.

The serigraphy of the Goldboro Project is deformed and dominated by the Upper Seal Harbour Anticline. The anticline folds all levels of stratigraphy observed in core and underground to form an upright, tight anticline that plunges shallowly to the east (10° to 30°). The enveloping surface to bedding at Goldboro dips moderately eastward at 20°. Younging is orthogonal to the hinge and limbs of the fold. All bedding and first-generation structures are refolded by open reclined folds that modify the axial plane and limbs of the Upper Seal Harbour Anticline. The axial plane of second-generation folds dips shallowly and an axial planar cleavage is observed in core, within underground workings and can be seen on the scale of the resource model. All earlier structures are deformed by two generations of steeply dipping, late brittle faults: One striking easterly and the other striking northerly. Analysis of the WG Fault indicates there is less than 50 metres of oblique normal slip on this fault demonstrating the WG and BR Gold Systems are extensions of the same mineralized system.

Previous mapping and drilling indicates that the Upper Seal Harbour Anticline transects the entire Goldboro Property, signifying an additional 2,000 m strike-length of exploration potential to the west of the current resource and further potential to the east. The Deposit also remains open down-dip of the axial plane of the fold and on the limbs of the fold. Since the main controlling structure to mineralization is the plunge of the fold, the deposit remains open along the plunge of the fold at depth indicating there is potential to further expand the existing Deposit and grow the resource

17.2 METALLURGICAL STUDIES

Metallurgical testing was completed to quantify the metallurgical response of samples from deposits in the Goldboro project. The testing focused on samples from Boston-Richardson and East Goldbrook deposits. The program was designed with the intent to develop the parameters for process design criteria for comminution, gravity concentration, leaching, carbon adsorption, cyanide destruction and carbon elution, and gold refining in the process plant. The metallurgical program was performed on the following composites:

- Boston-Richardson: low grade, medium grade and high grade;
- East Goldbrook: marginal grade, low grade, medium grade and high grade.

The composites were selected by Anaconda geologists with input from metallurgists to represent the spatial distribution, head grades, and mineralization types of the Goldboro project.

Boston-Richardson samples were tested through a suite of grinding characterization tests including Bond low energy, rod mill and ball work index, and Bond abrasion index tests. The Boston-Richardson samples were selected to complete SMC tests. The East Goldbrook samples were used for Bond ball mill work index variability tests.

The program included grind optimization, gravity concentration, leaching studies on the Boston-Richardson composites. The East Goldbrook composites were tested as variability samples to evaluate the optimum conditions determined through the Boston-Richardson tests. An overall Master Composite was assembled for bulk leach tests to provide feed material for cyanide destruction testing and carbon adsorption testing. Tailings solids samples were then used for environmental testing. An overall recovery model was derived from the test data for use in process design and mine scheduling.

The testing showed the samples to be amenable to standard crushing, grinding, gravity concentration and leaching for gold recovery. The testing showed that significant portions (>50%) of the contained gold could be recovered with gravity concentration. An overall recovery model was derived from the test data for use in process design and mine scheduling. Overall gold recovery will be approximately 95%, dependent of the head grade.

Cyanide detoxification and arsenic removal was tested using standard SO2/air and ferric sulphate precipitation respectively. Both methods were found to be successful in reducing weak acid dissociable cyanide and arsenic concentrations that comply with environmental regulations. Tailings solids samples were then used for environmental testing.

17.3 BULK SAMPLE

The objectives of the Bulk Sample were to provide a better understanding of the geology, test for spatial and grade continuity of the mineralized structures, validate the existing mineral resource model and explore various types of mining methods to best extract mineralization. Reconciliation of the Bulk Sample is expected in the first quarter of 2020.

Geological mapping underground led to a better understanding of the geometry of the top of the BR Gold System. Differentiation of the host rocks and mapping of gold bearing structures was accomplished by visual inspection by geologists resulting in geological maps of mineralized zones. Continuity of the zones was observed at the scale of the bulk sample with internal variability in character noted on a round-by-round basis.

Extraction of the Bulk Sample utilized the existing decline, ramp, and cross-cuts developed in the late 1980s to minimize the development needed to access mineralized zones and planned mining stopes adjacent to existing workings. New development included 3 drifts off the existing cross-cuts in Level 1 (23 masl) and Level 2 (-13 masl) and the creation of a newly-developed ramp to Level 3 (-22 masl) with two additional drifts resulting in 213 m of new development. A total of 86 mining faces were completed. Additionally, a total of 13 raise faces were blasted throughout the workings. Mining was advanced using a single-boom jumbo, typically with one round blasted per 12-hour shift. Blasted rock was mucked using 1.5 and 2 cubic yard scooptrams, which were then loaded into a 7 cubic yard truck. Resource Estimate

Of the 107 diamond drillholes carried out by Anaconda since 2017, 101 drillholes totaling 27,467 m have been used toward updating the updated mineral resource estimate and includes 15,112 m of diamond drilling in 57 drillholes since the Previous Mineral Resource Estimate, effective date July 19, 2018. Historic drilling results from 1984 to 2011 were also incorporated into this Updated Mineral Resource Estimate which is based on validated results of 485 surface and underground drillholes, for a total of 93,916 m of diamond drilling.

The updated mineral resource estimate has a combined open pit and underground Measured and Indicated mineral resource of 698,600 ounces of gold (4,095,000 tonnes at 5.30 g/t gold), and open pit and underground Inferred mineral resource of 685,100 ounces of gold (3,007,000 tonnes at 7.09 g/t gold). Recent drilling has increased the combined open pit and underground Measured and Indicated ounces by 15.9% and increased the combined open pit and underground Inferred ounces by 51.2%. Additionally, the grade of both categories increased by 6.9%.

Mineral resources presented in this report have been estimated in accordance with NI 43-101 and CIM Standards. As defined under these standards, mineral resources that are not mineral reserves do not have demonstrated economic viability.

18 RECOMMENDATIONS

18.1 EXPLORATION RECOMMENDATIONS

In parallel with the work outlined in this technical report, Anaconda has been advancing various aspects of development of the Goldboro Projects. These activities include:

- Environmental assessment work;
- Community consultations with municipal, regional and provincial governments;
- Community consultation with the KMKNO and representatives of Patqnkek First Nations communities;
- Geotechnical studies;
- Site investigations;
- Hydrology and hydrogeology studies;
- Detailed metallurgical studies and feasibility level plant design;
- Mine, site and infrastructure design and planning.

This work will culminate in the production of a feasibility study that will be based on the resource estimate outlined in this technical report.

Following up on the work completed and outlined in this report, The Phase I recommendations for the 2020 exploration program include the following.

- Phase I – Exploration:
  - Complete a hyperspectral imaging program on up to 7,500 m of core to better characterize alteration mineralogy with the goal of identifying more refined grade control techniques.
- Phase II – Upon completion of a positive feasibility study and successful project financing:
  - Complete a 300 m geotechnical drilling and site investigation around the proposal tailings storage facility and along the road re-alignment.
  - Complete 1,000 m condemnation drilling program in areas of planned mine infrastructure.
  - Complete a 10,000 m reverse circulation drill program within the volume of the conceptual open pit to upgrade all resource categorization to the Measured category.
  - Conduct a 5,000 m diamond drilling program to upgrade the first three years of underground mining outlined in the feasibility study to the Measured category.
  - Conduct a 20-line km, ground induced polarization survey, to identify the position of the alteration system along strike from the existing deposit and to identify high priority drill targets with open pit potential.
- Tables 18.1 and 18.2 summarize the above recommendations and estimated costs, totaling CAN$2,575,000. Phase II work is not entirely contingent on successful completion of Phase I but could be modified in consideration of Phase I results.

Table 18.1 Phase I – Exploration

Work Program	Estimated Cost (CAN$)
Hyperspectral Imaging	100,000
Phase I Total	**$100,000**

Table 18.2 Phase II – Detailed Engineering

Work Program	Estimate Cost (CAN$)
Geotechnical program	200,000
Condemnation program	200,000
10,000 metre RC drill program	1,000,000
5,000 metre diamond drill program	1,000,000
Ground IP survey	75,000
Phase II Total	**$2,475,000**

18.2 OTHER RECOMMENDATIONS

The following are procedural recommendations to improve future technical projects.

- Continue to collect bulk density (specific gravity) samples during all phases of drilling:
  - Samples should be collected in all rock types;
  - Samples should be a consistent length (minimum 0.30 m).
- Use a downhole optical televiewer during exploration programs to determine bedding - vein orientations.
- Maintain a record of the overburden for all drill programs:
  - Organic thickness;
  - Till thickness;
  - Till description.
- Sample continuously during exploration programs.

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20 CERTIFICATES OF QUALIFIED PERSONS

TODD MCCRACKEN, P.GEO.

I, Todd McCracken, P. Geo., of Sudbury, Ontario do hereby certify:

— I am a Manager with WSP Canada Inc. with a business address at 93 Cedar Street, Suite 300, Sudbury, Ontario P3E 1A7.

— This certificate applies to the technical report titled *Goldboro Gold Project Resource Update Phase 2 Guysborough County, Nova Scotia*, with an effective date of August 21, 2019 (the "Technical Report").

— I am a graduate of the University of Waterloo, with a Bachelor of Science (Honours) in Applied Earth Science in 1992.

— I am a member of the Association of Professional Geoscientists of Ontario and License 0631. My relevant experience includes 27 years of experience in exploration and operations, including Archean hosted gold deposits.

— I have read the definition of "Qualified Person" as set out in National Instrument 43-101 *Standards of Disclosure for Mineral Properties* ("the Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of the Instrument.

— My most recent personal inspection of the Goldboro Project was between July 9 and 11 inclusive, as well as a previous visit in October 2018.

— I am responsible for Sections 1 to 20, excluding 13 of the Technical Report.

— I am independent of Anaconda Mining Inc. as defined by Section 1.5 of the Instrument.

— I have prior involvement with the Goldboro Project that is the subject of the Technical Report. I authored previous technical reports on the Goldboro Project in December 2018 and March 2018.

— I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument.

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

Signed and stamped this 18th day of December 2019 at Sudbury, Ontario.

Original signed and stamped by
Todd McCracken, P.Geo.

Todd McCracken, P.Geo.
Manager - Mining
WSP Canada Inc.

TOMMASO ROBERTO RAPONI, P. ENG.

I, Tommaso Roberto Raponi, P. Eng., of Oakville, Ontario do hereby certify:

— I am a Senior Mineral Processing Specialist at Ausenco Engineering Canada Inc., 11 King St West, Suite 1550, Toronto, ON, M5H 4C7.

— This certificate applies to the technical report titled *Goldboro Gold Project Resource Update Phase 2 Guysborough County, Nova Scotia*, with an effective date of August 21, 2019 (the "Technical Report").

— I am a graduate of the University of Toronto, with a Bachelor of Applied Science (Honours) in Geological Engineering in 1984.

— I am registered as a Professional Engineer in Ontario and British Columbia. I have worked for more than 35 years in the mining industry in various positions continuously since my graduation from university. I have worked as an independent consultant since 2016.

— I have read the definition of "Qualified Person" as set out in National Instrument 43-101 *Standards of Disclosure for Mineral Properties* ("the Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of the Instrument.

— I have not visited site.

— I am responsible for Sections 13, parts of sections 1 of the Technical Report.

— I am independent of Anaconda Mining Inc. as defined by Section 1.5 of the Instrument.

— I have not had prior involvement with the property that is the subject of the Technical Report.

— I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument.

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

Signed and stamped this 18th day of December 2019 at Toronto, Ontario.

Original signed and stamped by
Tommaso Roberto Raponi, P.Eng.

Tommaso Roberto Raponi, P.Eng.
Senior Mineral Processing Specialist
Ausenco Engineering Canada Inc.

APPENDIX

A

DRILLING PROGRAM

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
68-01	137.8	607547.00	5006212.00	60	9732.23	6805.71	5000.00	271.06	270	-62
68-02	337.5	607635.00	5006150.00	67	9819.08	6742.08	5007.00	1.06	0	-90
88-U-01	50.91	606527.06	5006438.53	23.13	8716.52	7051.17	4963.13	0	358.94	2
88-U-02	82.93	606527.78	5006430.77	21.42	8717.10	7043.39	4961.42	180	178.94	-30
88-U-03	52.44	606527.80	5006432.43	21.46	8717.15	7045.05	4961.46	180	178.94	-50
88-U-04	47.87	606531.44	5006357.54	20.1	8719.40	6970.10	4960.10	171	169.94	-41
89-U-05	17.68	606573.62	5006398.10	-12.88	8762.33	7009.88	4927.12	180	178.94	20
89-U-06	21.94	606548.56	5006403.17	-12.87	8737.36	7015.41	4927.13	180	178.94	0
89-U-07	26.52	606573.68	5006398.24	-12.06	8762.40	7010.01	4927.94	180	178.94	40
89-U-08	20.73	606548.63	5006403.36	-12.23	8737.45	7015.59	4927.77	180	178.94	20
89-U-09	25.3	606573.58	5006398.90	-9.84	8762.30	7010.67	4930.16	180	178.94	60
89-U-10	36.88	606562.18	5006450.12	-13	8751.86	7062.11	4927.00	360	358.94	0
89-U-11	31.09	606573.63	5006399.44	-9.93	8762.37	7011.21	4930.07	180	178.94	75
89-U-12	27.43	606565.35	5006399.15	-13.34	8754.08	7011.07	4926.66	180	178.94	20
89-U-13	72.25	606573.72	5006403.13	-12.93	8762.53	7014.90	4927.07	360	358.94	0
89-U-14	28.35	606565.40	5006399.58	-12.34	8754.14	7011.51	4927.66	180	178.94	40
89-U-15	70.1	606573.72	5006403.36	-12.5	8762.54	7015.13	4927.50	360	358.94	20
89-U-16	32.01	606565.28	5006400.14	-10.38	8754.04	7012.08	4929.62	180	178.94	60
89-U-17	38.1	606573.73	5006403.56	-11.6	8762.54	7015.34	4928.40	360	358.94	40
89-U-18	42.47	606573.61	5006402.68	-9.25	8762.40	7014.45	4930.75	360	358.94	60
89-U-19	39.02	606565.25	5006400.47	-10.19	8754.01	7012.40	4929.81	180	178.94	75
89-U-20	51.51	606573.60	5006402.24	-9.1	8762.40	7014.01	4930.90	360	358.94	75
89-U-21	29.57	606565.24	5006400.28	-10.26	8754.00	7012.21	4929.74	180	178.94	75
89-U-22	32.93	606499.52	5006416.51	-11.18	8688.58	7029.66	4928.82	180	178.94	0
89-U-23	31.1	606512.70	5006411.44	-11.08	8701.67	7024.34	4928.92	180	178.94	0
89-U-24	28.05	606499.47	5006415.79	-9.36	8688.51	7028.94	4930.64	180	178.94	20
89-U-25	31.1	606512.70	5006411.32	-10.52	8701.66	7024.23	4929.48	180	178.94	20
89-U-26	28.96	606499.47	5006416.82	-9.08	8688.54	7029.97	4930.92	180	178.94	40
90-U-100	32.92	606499.52	5006417.12	-8.88	8688.58	7030.27	4931.12	180	178.94	60
90-U-101	28.96	606523.06	5006409.40	-11.02	8711.98	7022.11	4928.98	180	178.94	40
90-U-102	42.98	606499.50	5006417.52	-8.8	8688.57	7030.67	4931.20	180	178.94	75
90-U-103	40.84	606523.06	5006409.79	-10.33	8712.00	7022.50	4929.67	180	178.94	60
90-U-104	50.6	606523.05	5006410.12	-10.34	8711.99	7022.84	4929.66	180	178.94	75
90-U-105	30.78	606548.71	5006403.30	-10.01	8737.53	7015.54	4929.99	180	178.94	40
90-U-106	27.13	606501.14	5006420.40	-9.92	8690.27	7033.52	4930.08	360	358.94	0
90-U-107	28.65	606548.61	5006403.63	-9.14	8737.42	7015.87	4930.86	180	178.94	60
90-U-108	22.86	606501.07	5006420.14	-9.61	8690.20	7033.27	4930.39	360	358.94	20
90-U-109	24.99	606501.08	5006419.79	-9.16	8690.19	7032.91	4930.84	360	358.94	40
90-U-110	41.76	606548.74	5006404.23	-9.07	8737.57	7016.47	4930.93	180	178.94	75
90-U-111	35.05	606500.98	5006419.33	-8.76	8690.08	7032.46	4931.24	360	358.94	60
90-U-112	28.65	606500.98	5006419.03	-8.76	8690.09	7032.16	4931.24	360	358.94	73
90-U-27	54.86	606512.32	5006419.72	-11.01	8701.44	7032.63	4928.99	360	358.94	0
90-U-28	68	606564.93	5006406.59	-13.63	8753.80	7018.52	4926.37	360	358.94	20
90-U-29	57.91	606512.31	5006419.59	-10.42	8701.42	7032.50	4929.58	360	358.94	20
90-U-30	57.91	606564.90	5006406.46	-12.84	8753.77	7018.39	4927.16	360	358.94	40
90-U-31	45.42	606512.34	5006419.37	-9.45	8701.45	7032.28	4930.55	360	358.94	40
90-U-32	45.12	606564.80	5006405.60	-11.19	8753.66	7017.54	4928.81	360	358.94	60
90-U-33	52.43	606512.23	5006419.31	-8.37	8701.34	7032.22	4931.63	360	358.94	60
90-U-34	56.08	606564.99	5006405.24	-10.96	8753.83	7017.18	4929.04	360	358.94	78
90-U-35	61.87	606512.34	5006418.66	-8.33	8701.44	7031.58	4931.67	360	358.94	75
90-U-36	62.48	606523.02	5006417.63	-11.59	8712.10	7030.35	4928.41	360	358.94	0
90-U-37	64.62	606523.04	5006417.53	-11.46	8712.12	7030.24	4928.54	360	358.94	20
90-U-38	64.31	606481.40	5006439.29	-9.77	8670.88	7052.77	4930.23	360	358.94	0
90-U-39	62.48	606481.47	5006439.33	-9.28	8670.95	7052.82	4930.72	360	358.94	20
90-U-40	53.64	606481.41	5006439.47	-8.25	8670.90	7052.96	4931.75	360	358.94	40
90-U-41	57.61	606523.06	5006417.56	-11.07	8712.13	7030.28	4928.93	360	358.94	40
90-U-42	57.61	606481.44	5006438.80	-8	8670.91	7052.29	4932.00	360	358.94	60
90-U-43	62.48	606523.03	5006416.92	-9.62	8712.10	7029.63	4930.38	360	358.94	60
90-U-44	58.52	606481.38	5006438.61	-7.84	8670.85	7052.10	4932.16	360	358.94	75
90-U-45	53.95	606523.03	5006416.60	-9.62	8712.09	7029.31	4930.38	360	358.94	75
90-U-46	60.05	606482.00	5006434.80	-9.16	8671.39	7048.27	4930.84	180	178.94	20
90-U-47	40.84	606447.09	5006416.48	-9.63	8636.15	7030.60	4930.37	180	178.94	0
90-U-48	47.24	606481.91	5006435.34	-8.23	8671.32	7048.82	4931.77	180	178.94	40
90-U-49	40.54	606447.10	5006416.52	-9.19	8636.16	7030.64	4930.81	180	178.94	20
90-U-50	72.54	606473.46	5006419.86	-9.53	8662.59	7033.49	4930.47	360	358.94	20
90-U-51	46.02	606447.09	5006416.64	-8.66	8636.15	7030.77	4931.34	180	178.94	40
90-U-52	56.99	606482.03	5006435.66	-8.04	8671.44	7049.14	4931.96	180	178.94	60
90-U-53	35.05	606446.98	5006417.20	-7.98	8636.05	7031.32	4932.02	180	178.94	60
90-U-54	79.25	606473.46	5006419.57	-8.93	8662.58	7033.21	4931.07	360	358.94	40
90-U-55	47.85	606481.95	5006436.10	-7.79	8671.37	7049.58	4932.21	180	178.94	75
90-U-56	43.28	606447.08	5006417.51	-7.6	8636.15	7031.64	4932.40	180	178.94	73

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
90-U-57	59.13	606473.51	5006419.21	-8.52	8662.62	7032.84	4931.48	360	358.94	62
90-U-58	74.37	606446.94	5006420.20	-9.2	8636.06	7034.32	4930.80	360	358.94	0
90-U-59	25.91	606536.27	5006406.42	-13.1	8725.14	7018.88	4926.90	180	178.94	0
90-U-60	27.13	606536.29	5006406.43	-12.57	8725.15	7018.89	4927.43	180	178.94	20
90-U-61	29.87	606536.36	5006406.63	-11.64	8725.23	7019.09	4928.36	180	178.94	40
90-U-62	45.72	606536.27	5006407.23	-10.85	8725.16	7019.69	4929.15	180	178.94	75
90-U-63	45.11	606447.00	5006420.18	-8.8	8636.13	7034.30	4931.20	360	358.94	20
90-U-64	57	606473.48	5006418.86	-8.52	8662.58	7032.50	4931.48	360	358.94	75
90-U-65	40.23	606536.21	5006406.78	-10.97	8725.09	7019.25	4929.03	180	178.94	60
90-U-66	46.94	606446.99	5006419.87	-8.51	8636.11	7033.99	4931.49	360	358.94	40
90-U-67	54.25	606446.98	5006419.50	-8.04	8636.09	7033.62	4931.96	360	358.94	60
90-U-68	39.93	606473.91	5006416.10	-10.21	8662.96	7029.73	4929.79	180	178.94	0
90-U-69	57	606446.86	5006419.20	-7.64	8635.97	7033.33	4932.36	360	358.94	75
90-U-70	69.49	606536.13	5006409.89	-12.92	8725.06	7022.37	4927.08	360	358.94	0
90-U-71	38.71	606474.61	5006416.40	-9.49	8663.67	7030.01	4930.51	180	178.94	20
90-U-72	63.7	606536.14	5006409.86	-12.43	8725.07	7022.34	4927.57	360	358.94	20
90-U-73	75.29	606459.94	5006420.09	-10.48	8649.06	7033.97	4929.52	360	358.94	0
90-U-74	39.93	606474.65	5006416.31	-8.83	8663.70	7029.93	4931.17	180	178.94	40
90-U-75	57.91	606536.13	5006409.71	-11.63	8725.06	7022.19	4928.37	360	358.94	40
90-U-76	40.54	606474.67	5006416.90	-8.46	8663.74	7030.51	4931.54	180	178.94	60
90-U-77	51.51	606459.95	5006420.21	-9.99	8649.08	7034.09	4930.01	360	358.94	20
90-U-78	39.93	606459.93	5006419.94	-9.44	8649.05	7033.82	4930.56	360	358.94	40
90-U-79	42.06	606474.59	5006417.39	-8.39	8663.67	7031.00	4931.61	180	178.94	74
90-U-80	54.25	606459.97	5006419.44	-9.13	8649.09	7033.33	4930.87	360	358.94	60
90-U-81	54.86	606537.14	5006408.76	-11.11	8726.06	7021.22	4928.89	360	358.94	60
90-U-82	35.97	606460.07	5006419.06	-8.81	8649.17	7032.95	4931.19	360	358.94	75
90-U-83	57.91	606537.07	5006408.43	-10.99	8725.97	7020.89	4929.01	360	358.94	75
90-U-84	30.48	606484.49	5006415.97	-10.52	8673.54	7029.40	4929.48	180	178.94	0
90-U-85	29.87	606459.48	5006415.99	-10.41	8648.52	7029.88	4929.59	180	178.94	0
90-U-86	28.04	606512.77	5006411.88	-9.71	8701.74	7024.78	4930.29	180	178.94	40
90-U-87	29.57	606459.55	5006416.08	-9.97	8648.60	7029.98	4930.03	180	178.94	20
90-U-88	31.39	606548.29	5006408.33	-13.43	8737.20	7020.58	4926.57	360	358.94	0
90-U-89	45.11	606512.82	5006412.18	-9.17	8701.79	7025.08	4930.83	180	178.94	60
90-U-90	31.7	606548.32	5006408.35	-12.9	8737.22	7020.60	4927.10	360	358.94	20
90-U-91	29.87	606459.53	5006416.28	-9.31	8648.59	7030.17	4930.69	180	178.94	40
90-U-92	34.14	606459.61	5006416.61	-8.76	8648.67	7030.51	4931.24	180	178.94	60
90-U-93	31.09	606548.35	5006408.23	-11.82	8737.25	7020.48	4928.18	360	358.94	40
90-U-94	44.81	606512.55	5006413.01	-9.34	8701.55	7025.93	4930.66	180	178.94	76
90-U-95	39.01	606459.51	5006417.25	-8.81	8648.59	7031.14	4931.19	180	178.94	75
90-U-96	39.93	606548.31	5006407.65	-9.66	8737.20	7019.89	4930.34	360	358.94	60
90-U-97	25.91	606523.03	5006409.11	-11.97	8711.94	7021.82	4928.03	180	178.94	0
90-U-98	29.87	606523.02	5006409.25	-11.46	8711.94	7021.97	4928.54	180	178.94	20
90-U-99	52.73	606548.21	5006405.72	-9.26	8737.06	7017.97	4930.74	360	358.94	75
BR05-001	133.5	606462.53	5006352.54	66.35	8650.40	6966.38	5003.35	0	358.94	-55
BR05-002	96.4	606462.22	5006376.85	65.66	8650.54	6990.69	5002.66	0	358.94	-50
BR05-003	67	606460.92	5006396.67	65.42	8649.61	7010.54	5002.42	0	358.94	-45
BR05-004	145.5	606489.30	5006345.65	65.84	8677.04	6959.00	5002.84	0	358.94	-55
BR05-005	107	606482.57	5006372.04	64.91	8670.80	6985.51	5001.91	0	358.94	-53
BR05-006	65	606486.15	5006397.54	65.09	8674.86	7010.94	5002.09	0	358.94	-60
BR05-007	135.5	606438.58	5006351.49	66.69	8626.43	6965.78	5003.69	0	358.94	-55
BR05-008	86	606439.06	5006375.90	67.09	8627.37	6990.18	5004.09	0	358.94	-55
BR05-009	122.3	606511.54	5006348.34	64.52	8699.33	6961.26	5001.52	0	358.94	-53
BR05-010	81.7	606511.07	5006373.08	63.42	8699.32	6986.01	5000.42	0	358.94	-60
BR05-011	58.8	606511.10	5006401.72	63.91	8699.88	7014.65	5000.91	0	358.94	-75
BR05-012	152	606412.51	5006349.24	66.88	8600.31	6964.01	5003.88	0	358.94	-55
BR05-013	114	606412.15	5006374.12	68.48	8600.43	6988.89	5005.48	0	358.94	-55
BR05-014	125	606536.07	5006338.67	64.23	8723.69	6951.15	5001.23	0	358.94	-55
BR05-015	83	606536.32	5006360.34	62.83	8724.34	6972.80	4999.83	0	358.94	-60
BR05-016	68	606537.70	5006389.23	63.81	8726.26	7001.67	5000.81	0	358.94	-74
BR05-017	118	606498.62	5006353.35	64.4	8686.51	6966.51	5001.40	0	358.94	-55
BR05-018	123	606500.51	5006373.76	63.66	8688.78	6986.90	5000.66	0	358.94	-45
BR05-019	115.6	606555.18	5006337.54	64.14	8742.76	6949.66	5001.14	0	358.94	-57
BR05-020	142	606555.18	5006337.54	64.14	8742.76	6949.66	5001.14	0	358.94	-63
BR05-021	74	606560.73	5006363.77	62.96	8748.80	6975.78	4999.96	0	358.94	-60
BR05-022	59	606562.15	5006401.73	64.48	8750.93	7013.72	5001.48	0	358.94	-90
BR05-023	150	606410.79	5006374.07	68.43	8599.07	6988.87	5005.43	330	328.94	-50
BR08-01	171	606588.89	5006361.39	62.87	8776.93	6972.89	4999.87	0	358.94	-50
BR08-02	220	606588.89	5006361.39	62.87	8776.93	6972.89	4999.87	0	358.94	-75
BR08-03	217	606588.90	5006325.09	63.52	8776.25	6936.59	5000.52	350	348.94	-60
BR08-04	221	606588.90	5006325.09	63.52	8776.25	6936.59	5000.52	15	13.94	-45
BR08-05	233	606608.00	5006308.40	61.35	8795.04	6919.53	4998.35	0	358.94	-65

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
BR08-06	268	606636.40	5006326.79	59.12	8823.78	6937.41	4996.12	0	358.94	-70
BR08-07	281	606636.30	5006307.19	58.58	8823.32	6917.81	4995.58	10	8.94	-50
BR08-08	290	606634.79	5006274.69	57.59	8821.22	6885.34	4994.59	15	13.94	-55
BR08-09	269	606634.79	5006274.69	57.59	8821.22	6885.34	4994.59	25	23.94	-45
BR08-10	332	606631.90	5006254.00	57.69	8817.93	6864.69	4994.69	25	23.94	-55
BR08-11	300	606717.70	5006257.89	57.8	8903.80	6867.00	4994.80	0	358.94	-50
BR08-12	314	606717.70	5006257.89	57.8	8903.80	6867.00	4994.80	10	8.94	-55
BR08-13	350	606767.30	5006268.19	62.75	8953.59	6876.38	4999.75	5	3.94	-65
BR08-14	372	606761.99	5006217.80	59.51	8947.36	6826.08	4996.51	5	3.94	-50
BR08-15	377	606786.60	5006236.89	62.57	8972.31	6844.72	4999.57	5	3.94	-65
BR08-16	395	606619.30	5006402.20	61.51	8808.80	7013.12	4998.51	100	98.94	-45
BR08-17	251	606959.39	5006294.99	61.88	9146.19	6899.61	4998.88	0	358.94	-50
BR08-18	251	607009.59	5006261.70	54.22	9195.77	6865.38	4991.22	0	358.94	-50
BR08-19	251	607014.40	5006194.19	54.46	9199.32	6797.79	4991.46	0	358.94	-50
BR08-20A	249.6	607050.70	5006245.30	53.83	9236.56	6848.21	4990.83	10	8.94	-45
BR08-21	329	607050.70	5006245.30	53.83	9236.56	6848.21	4990.83	10	8.94	-55
BR08-22	251	607152.50	5006246.00	53.8	9338.38	6847.02	4990.80	345	343.94	-45
BR08-23	295	607152.50	5006246.00	53.8	9338.38	6847.02	4990.80	345	343.94	-60
BR08-24	251	607167.20	5006235.09	52.69	9352.87	6835.85	4989.69	355	353.94	-45
BR08-25	285	607167.20	5006235.09	52.69	9352.87	6835.85	4989.69	0	358.94	-55
BR08-26	302	607169.89	5006169.29	52.53	9354.35	6770.00	4989.53	0	358.94	-55
BR08-27	240	607217.19	5006189.60	53.1	9402.03	6789.42	4990.10	0	358.94	-65
BR08-28	308	607214.99	5006141.30	52.36	9398.93	6741.16	4989.37	0	358.94	-70
BR08-29	299	607287.29	5006204.89	52.9	9472.41	6803.42	4989.90	0	358.94	-65
BR08-30	299	607266.90	5006254.19	53.35	9452.93	6853.10	4990.35	0	358.94	-60
BR08-31	302	607319.30	5006207.20	55.07	9504.45	6805.12	4992.07	0	358.94	-65
BR08-32	284	607322.70	5006248.69	55.71	9508.62	6846.56	4992.71	0	358.94	-60
BR08-33	271	606502.90	5006336.19	67.04	8690.46	6949.29	5004.04	0	358.94	-60
BR08-34	247.3	606514.29	5006260.09	64.57	8700.45	6872.98	5001.57	0	358.94	-55
BR08-35	201.6	606511.00	5006183.39	63.06	8695.72	6796.34	5000.06	0	358.94	-55
BR08-36	268	606523.50	5006090.39	60.23	8706.49	6703.11	4997.23	0	358.94	-55
BR08-37	250	606485.29	5006313.40	68.28	8672.44	6926.81	5005.28	0	358.94	-55
BR08-38	259	606460.60	5006307.10	68.29	8647.62	6920.97	5005.29	0	358.94	-55
BR08-39	250	606435.30	5006305.70	66.23	8622.30	6920.04	5003.23	0	358.94	-55
BR08-40	251	606415.59	5006311.69	66.86	8602.71	6926.41	5003.86	0	358.94	-55
BR08-41	251	606388.09	5006281.10	67.4	8574.64	6896.32	5004.40	0	358.94	-55
BR08-42	250	606389.89	5006357.50	68.65	8577.86	6972.69	5005.65	0	358.94	-55
BR08-43	251	606360.00	5006286.50	68.2	8546.64	6902.24	5005.20	0	358.94	-55
BR08-44	257	606366.99	5006352.00	69.18	8554.86	6967.61	5006.18	0	358.94	-55
BR84-01	529.57	607216.68	5006392.91	57.16	9405.29	6992.74	4997.16	187	185.94	-76
BR85-01	31.4	606015.56	5006490.66	74.5	8206.00	7112.80	5014.50	7	5.94	-60
BR85-01A	90.53	606016.16	5006496.45	77.23	8206.70	7118.58	5017.23	7	5.94	-60
BR85-02	78.03	605954.80	5006494.24	75	8145.30	7117.50	5015.00	7	5.94	-55
BR85-03	102.72	606074.00	5006467.45	74	8264.00	7088.50	5014.00	7	5.94	-60
BR85-04	87.18	606141.98	5006466.44	74.98	8331.97	7086.23	5014.98	7	5.94	-55
BR87-01	73.76	607377.59	5006499.41	58.88	9568.18	7096.24	4998.88	180	178.94	-65
BR87-02	501.4	607250.90	5006057.66	50.07	9433.27	6656.87	4990.08	7	5.94	-70
BR87-03	422.45	606963.16	5006171.44	52.56	9147.66	6775.99	4992.57	7	5.94	-65
BR87-04	54.25	607007.46	5006121.13	52.56	9191.03	6724.86	4992.57	7	5.94	-65
BR87-04A	413.92	607007.46	5006121.13	52.56	9191.03	6724.86	4992.57	7	5.94	-65
BR87-05	50.62	607203.13	5006078.28	49.27	9385.89	6678.37	4989.27	7	5.94	-70
BR87-05A	407.82	607203.13	5006078.28	49.27	9385.89	6678.37	4989.27	7	5.94	-70
BR87-06	673	607196.36	5006296.10	49.99	9383.17	6896.31	4989.99	279	277.94	-65
BR87-07	396.54	607197.07	5006295.93	49.77	9383.87	6896.13	4989.77	7	5.94	-90
BR87-08	471.22	607355.69	5006311.33	60	9542.78	6908.58	5000.00	279	277.94	-75
BR87-09	611.73	607355.69	5006311.33	60	9542.78	6908.58	5000.00	327	325.94	-90
BR87-10	153.31	607216.19	5006390.16	57.02	9404.74	6989.99	4997.02	187	185.94	-50
BR87-11	230.73	607216.29	5006390.83	56.92	9404.86	6990.67	4996.92	187	185.94	-70
BR87-12	541.93	607277.17	5006290.26	52.27	9463.87	6888.97	4992.27	307	305.94	-88
BR87-13	282.87	607294.52	5006397.10	55.15	9483.20	6995.48	4995.15	187	185.94	-70
BR87-14	238.67	607294.33	5006396.00	54.99	9482.99	6994.38	4994.99	187	185.94	-50
BR87-15	539.52	607198.65	5006275.04	49.31	9385.06	6875.20	4989.31	279	277.94	-90
BR87-16	380.71	606738.76	5006188.16	57.63	8923.57	6796.87	4997.63	7	5.94	-45
BR87-17	639.19	607281.98	5006481.80	58.88	9472.23	7080.41	4998.88	211	209.94	-70
BR87-18	351.45	606817.75	5006228.79	37.29	9003.31	6836.03	4977.29	7	5.94	-70
BR87-19	198.13	606617.42	5006387.77	58.84	8805.94	6998.73	4998.84	279	277.94	-90
BR87-20	395.34	607131.72	5006330.73	50.23	9319.17	6932.14	4990.23	7	5.94	-90
BR87-21A	226.48	606602.47	5006389.16	60.5	8791.01	7000.40	5000.50	279	277.94	-60
BR87-22	337.43	607129.01	5006327.72	50.29	9316.40	6929.18	4990.29	289	287.94	-62
BR87-23	297.8	606769.37	5006389.30	58.91	8957.91	6997.44	4998.91	277	275.94	-90
BR87-24	383.13	606911.26	5006363.92	60.25	9099.33	6969.42	5000.25	334	332.94	-90

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
BR87-25A	290.79	606767.47	5006385.91	59.05	8955.95	6994.08	4999.05	307	305.94	-60
BR87-26	337.43	606910.44	5006364.09	60.34	9098.51	6969.61	5000.34	292	290.94	-61
BR87-27	267.32	606774.94	5006464.15	56.75	8964.87	7072.18	4996.75	187	185.94	-70
BR87-28	346.59	606986.54	5006351.98	56.89	9174.39	6956.08	4996.89	307	305.94	-90
BR87-29	276.46	606703.06	5006468.13	57.63	8893.07	7077.50	4997.63	187	185.94	-70
BR87-30	270.37	606756.67	5006309.17	58.11	8943.72	6917.54	4998.11	7	5.94	-65
BR87-31	370.96	606705.46	5006391.88	59.7	8894.05	7001.20	4999.70	287	285.94	-90
BR87-32	331.33	606841.39	5006371.92	58.06	9029.61	6978.72	4998.06	357	355.94	-90
BR87-33	297.8	606628.63	5006454.85	58.8	8818.39	7065.60	4998.80	187	185.94	-75
BR87-34	361.81	606545.53	5006407.14	62.05	8734.41	7019.43	5002.05	347	345.94	-90
BR87-35A	320.66	606829.85	5006274.69	58.78	9016.26	6881.70	4998.78	7	5.94	-72
BR87-36	215.5	606544.68	5006405.36	62.12	8733.52	7017.67	5002.12	277	275.94	-65
BR87-37	343.52	606851.77	5006449.87	57.51	9041.44	7056.47	4997.51	187	185.94	-70
BR87-38	241.72	606605.22	5006308.68	58.3	8792.27	6919.87	4998.30	7	5.94	-65
BR88-39	215.5	606681.36	5006311.97	54.81	8868.47	6921.74	4994.81	7	5.94	-65
BR88-40	337.43	606817.23	5006375.81	57.68	9005.52	6983.06	4997.68	7	5.94	-90
BR88-41	306.95	606747.72	5006389.26	60.7	8936.26	6997.80	5000.70	277	275.94	-90
BR88-42	338.65	606851.71	5006449.58	57.26	9041.37	7056.18	4997.26	187	185.94	-81
BR88-43	337.43	606721.68	5006389.88	60.75	8910.23	6998.90	5000.75	7	5.94	-90
BR88-44	337.43	606820.55	5006404.14	37.26	9009.37	7011.32	4977.26	7	5.94	-90
BR88-45	337.43	606720.35	5006414.50	58.2	8909.36	7023.55	4998.20	7	5.94	-90
BR88-46	337.43	606749.68	5006364.45	59.19	8937.76	6972.95	4999.19	337	335.94	-90
BR88-47	329.81	606745.87	5006414.88	58.81	8934.89	7023.45	4998.81	7	5.94	-90
BR88-48	331.33	606470.58	5006404.69	62.09	8659.41	7018.38	5002.09	7	5.94	-90
BR88-49	221.29	606622.05	5006412.58	59.58	8811.03	7023.45	4999.58	7	5.94	-90
BR88-50	197.21	606547.33	5006427.73	62.49	8736.59	7039.99	5002.49	7	5.94	-90
BR88-51	215.5	606497.83	5006400.21	61.77	8686.58	7013.39	5001.77	337	335.94	-90
BR88-52	215.5	606542.46	5006384.82	60.54	8730.92	6997.17	5000.54	7	5.94	-90
BR88-53	245.98	606465.04	5006380.14	62.42	8653.42	6993.93	5002.42	337	335.94	-90
BR88-54	227.69	606422.76	5006405.63	64.61	8611.61	7020.20	5004.61	7	5.94	-90
BR88-55	274.33	606503.83	5006419.73	61	8692.94	7032.80	5001.00	7	5.94	-90
BR88-56	319.14	606641.81	5006387.87	57.95	8830.33	6998.37	4997.95	7	5.94	-90
BR88-57	266.71	606422.44	5006385.53	65.11	8610.92	7000.11	5005.11	7	5.94	-90
BR88-58	258.18	606644.62	5006404.70	57.73	8833.45	7015.15	4997.73	7	5.94	-90
BR88-59	273.42	606397.74	5006398.06	65.84	8586.45	7013.10	5005.84	307	305.94	-90
BR88-60	215.5	606572.31	5006422.24	61.67	8761.47	7034.03	5001.67	7	5.94	-90
BR88-61	242.33	606421.58	5006430.18	63.4	8610.89	7044.77	5003.40	277	275.94	-90
BR88-62	297.8	606522.38	5006409.45	61.49	8711.30	7022.17	5001.49	277	275.94	-90
BR88-63	319.14	606388.36	5006382.40	64.87	8576.78	6997.61	5004.87	307	305.94	-90
BR88-64	228.91	606565.92	5006382.46	60.14	8754.34	6994.37	5000.14	277	275.94	-90
BR88-65	230.74	606558.92	5006404.63	61.63	8747.75	7016.67	5001.63	337	335.94	-90
BR88-66	306.95	606472.89	5006431.13	62.27	8662.21	7044.77	5002.27	337	335.94	-90
BR88-67	306.95	606519.99	5006386.62	60.54	8708.49	6999.39	5000.54	277	275.94	-90
BR88-68	306.95	606522.22	5006428.96	61.61	8711.50	7041.69	5001.61	227	225.94	-90
BR88-69	306.95	606492.33	5006381.84	61.67	8680.74	6995.12	5001.67	337	335.94	-90
BR88-70	252.08	606242.45	5006413.09	69.2	8431.44	7031.01	5009.20	277	275.94	-90
BR88-71	151.49	606245.65	5006433.45	69.03	8435.02	7051.31	5009.03	246	244.94	-90
BR88-72	145.4	606170.05	5006422.80	73.33	8359.22	7042.07	5013.33	157	155.94	-90
BR88-73	96.63	606252.86	5006482.92	71.78	8443.15	7100.65	5011.78	7	5.94	-90
BR88-74	72.24	606178.67	5006471.57	74.16	8368.75	7090.68	5014.16	7	5.94	-88
BR88-75	303.89	606252.98	5006484.44	71.58	8443.31	7102.17	5011.58	7	5.94	-55
BR88-76	307.13	606178.16	5006474.46	74.12	8368.30	7093.59	5014.12	7	5.94	-60
BR88-77	258.94	606256.82	5006507.80	71.53	8447.58	7125.46	5011.53	7	5.94	-90
BR88-78	230.88	606186.44	5006523.13	75.24	8377.49	7142.09	5015.24	7	5.94	-90
BR88-79	316.28	606260.47	5006531.08	70.9	8451.66	7148.68	5010.90	267	265.94	-90
BR88-80	307.13	606186.12	5006522.01	73.91	8377.15	7140.99	5013.91	7	5.94	-60
BR88-81	43.45	606675.40	5006430.81	58.64	8864.72	7040.69	4998.64	180	178.94	-50
BR88-82	108.84	606486.80	5006384.56	61.78	8675.26	6997.94	5001.78	0	358.94	-50
BR89-100	90.22	606009.36	5006560.17	78.27	8201.10	7182.43	5018.27	180	178.94	-45
BR89-101	90.22	605986.19	5006506.08	77.26	8176.92	7128.76	5017.26	360	358.94	-44
BR89-102	90.22	605959.01	5006570.59	78.48	8150.94	7193.79	5018.48	180	178.94	-45
BR89-103	29.87	606009.77	5006531.79	78.17	8200.98	7154.03	5018.17	180	178.94	-85
BR89-104	29.87	606059.31	5006524.28	77.18	8250.38	7145.60	5017.18	360	358.94	-85
BR89-105	29.87	606109.38	5006520.14	75.84	8300.37	7140.54	5015.84	180	178.94	-85
BR89-106	29.87	606208.78	5006497.89	74.94	8399.35	7116.45	5014.94	360	358.94	-85
BR89-107	29.87	606283.24	5006489.70	70.66	8473.66	7106.87	5010.66	360	358.94	-85
BR89-108	29.87	606333.09	5006475.77	68.29	8523.25	7092.01	5008.29	360	358.94	-85
BR89-83	81.68	606158.84	5006504.01	74.94	8349.52	7123.49	5014.94	180	178.94	-65
BR89-84	121.92	606384.19	5006464.50	64.69	8574.15	7079.79	5004.69	180	178.94	-65
BR89-85	178.92	606157.46	5006540.63	74.72	8348.82	7160.13	5014.72	180	178.94	-70
BR89-86	111.86	606207.69	5006529.41	72.53	8398.84	7147.99	5012.53	180	178.94	-70

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
BR89-87	138.99	606364.97	5006422.03	67.91	8554.13	7037.68	5007.91	360	358.94	-70
BR89-88	170.69	606207.41	5006535.87	72.08	8398.68	7154.45	5012.08	180	178.94	-80
BR89-89	187.76	606184.24	5006457.57	72.91	8374.07	7076.57	5012.91	360	358.94	-70
BR89-90	199.95	606305.76	5006561.93	67.86	8497.52	7178.67	5007.87	180	178.94	-70
BR89-91	203	606306.75	5006522.81	67.04	8497.78	7139.55	5007.04	330	178.94	-78
BR89-92	142.34	606232.65	5006479.88	70.52	8422.88	7097.98	5010.52	360	358.94	-80
BR89-93	190.8	606333.01	5006508.52	65.82	8523.78	7124.76	5005.83	180	178.94	-78
BR89-94	55.17	606132.62	5006477.72	75.44	8322.82	7097.68	5015.44	360	358.94	-45
BR89-95	124.05	606283.62	5006455.41	69.46	8473.40	7072.58	5009.46	360	358.94	-70
BR89-96	182.88	606408.55	5006458.52	62.11	8598.38	7073.36	5002.12	184	182.94	-75
BR89-97	90.22	606109.45	5006545.38	75.4	8300.91	7165.77	5015.40	180	178.94	-45
BR89-98	90.22	606083.06	5006487.03	76.45	8273.43	7107.92	5016.45	360	358.94	-45
BR89-99	90.22	606055.48	5006547.44	77.66	8246.97	7168.84	5017.66	180	178.94	-45
BR91-109	149.35	606469.66	5006405.10	62.7	8658.50	7018.80	5002.70	0	358.94	-90
BR91-110	150.87	606522.14	5006406.37	61.5	8711.00	7019.10	5001.50	0	358.94	-90
BR91-111	165.2	606571.74	5006422.59	62	8760.90	7034.40	5002.00	0	358.94	-90
BR91-112	127.1	606829.57	5006275.48	57.5	9016.00	6882.50	4997.50	360	358.94	-72
BR91-113	129.84	606522.23	5006347.47	61	8710.00	6960.20	5001.00	0	358.94	-45
BR93-114	114.9	606511.12	5006407.07	61	8700.00	7020.00	5001.00	0	358.94	-90
BR93-114B	41.4	606520.12	5006407.24	60	8709.00	7020.00	5000.00	0	358.94	-90
BR93-115	163.5	606520.12	5006407.24	60	8709.00	7020.00	5000.00	177	175.94	-70
BR93-116	110	606473.71	5006402.37	63	8662.50	7016.00	5003.00	0	358.94	-90
BR93-116B	58	606473.71	5006402.37	63	8662.50	7016.00	5003.00	0	358.94	-90
BR93-117	105.25	606524.33	5006368.81	62	8712.50	6981.50	5002.00	0	358.94	-45
BR95-119	138	606549.37	5006315.09	61.08	8736.54	6927.31	5001.08	360	358.94	-45
BR95-120	207	606550.19	5006265.19	58.16	8736.43	6877.40	4998.16	360	358.94	-45
BR95-121	132.5	606546.13	5006489.29	61.11	8736.53	7101.57	5001.11	180	178.94	-45
BR95-122	22	606545.31	5006558.71	60	8737.00	7171.00	5000.00	180	178.94	-45
BR95-123	224.43	606545.02	5006559.35	65.27	8736.73	7171.65	5005.27	180	178.94	-45
BR95-124	315	606648.22	5006554.89	60.71	8839.84	7165.27	5000.71	180	178.94	-57
BR95-125	224	606623.70	5006244.07	54.8	8809.55	6854.91	4994.80	4	2.94	-45
GA1-125	21	606528.44	5006389.89	21.5	8717.00	7002.50	4961.50	0	358.94	0
GA1-250	17	606561.48	5006388.00	-13.5	8750.00	7000.00	4926.50	0	358.94	0
GA2-125	17.6	606528.01	5006412.98	22	8717.00	7025.60	4962.00	0	358.94	0
GA2-250	33.5	606561.10	5006408.50	-14	8750.00	7020.50	4926.00	0	358.94	1
GAL125	100.3	606529.39	5006338.71	21.5	8717.00	6951.30	4961.50	0	358.94	1
GAL1-250	70.4	606561.20	5006403.00	-14	8750.00	7015.00	4926.00	180	178.94	0
GAL2-250	40	606561.10	5006408.20	-14	8750.00	7020.20	4926.00	0	358.94	1
OSK10-01	300	606613.15	5006285.34	60.03161	8799.76	6896.38	4997.03	1.06	0	-70
OSK10-02	300	606610.59	5006523.68	66.31356	8801.64	7134.77	5003.31	181.06	180	-65
OSK10-03	304	606588.38	5006285.66	61.36039	8775.01	6897.16	4998.36	1.06	0	-65
OSK10-04	233	605908.08	5006396.99	78.19261	8096.78	7021.12	5015.19	1.06	0	-60
OSK10-05	225	606573.55	5006286.38	61.56217	8760.18	6898.15	4998.56	1.06	0	-55
OSK10-06	150	605909.27	5006548.92	81.48938	8100.80	7173.03	5018.49	181.06	180	-55
OSK10-07	275	606558.25	5006515.04	67.93572	8749.14	7127.10	5004.94	181.06	180	-60
OSK10-08	250	606542.32	5006301.65	63.12842	8729.24	6914.00	5000.13	1.06	0	-65
OSK10-09	227	605911.88	5006598.75	81.79477	8104.33	7222.81	5018.80	181.06	180	-65
OSK10-10	275	606514.38	5006539.65	66.38746	8705.72	7152.52	5003.39	181.06	180	-55
OSK10-11	176	605809.08	5006446.75	79.7571	7998.71	7072.73	5016.76	1.06	0	-50
OSK10-12	303	606487.50	5006322.51	67.40306	8674.81	6935.88	5004.40	1.06	0	-70
OSK10-13	275	606486.76	5006582.72	69.35147	8678.90	7196.10	5006.35	181.06	180	-50
OSK10-14	302	606464.34	5006309.40	68.25099	8651.40	6923.21	5005.25	1.06	0	-65
OSK10-15	251.3	604750.81	5006550.68	51.60592	6942.38	7196.31	4988.61	1.06	0	-45
OSK10-16	300	606435.88	5006312.83	66.39504	8623.01	6927.16	5003.40	1.06	0	-65
OSK10-17	175.6	604747.77	5006672.76	57.99575	6941.60	7318.45	4995.00	1.06	0	-50
OSK10-18	226.5	606434.80	5006550.48	66.8856	8626.34	7164.83	5003.89	181.06	180	-45
OSK10-19	300	606412.74	5006309.24	66.48303	8599.81	6924.00	5003.48	1.06	0	-65
OSK10-20	173	604949.95	5006642.24	64.10039	7143.21	7284.17	5001.10	181.06	180	-55
OSK10-21	245	606410.69	5006555.14	67.39326	8602.32	7169.94	5004.39	181.06	180	-55
OSK10-22	200	606386.28	5006474.68	67.53633	8576.41	7089.93	5004.54	161.06	160	-70
OSK10-23	251	606385.08	5006525.38	66.14888	8576.16	7140.65	5003.15	181.06	180	-70
OSK10-24	150	606361.04	5006424.52	71.48818	8550.25	7040.24	5008.49	1.06	0	-70
OSK10-25	250	606362.98	5006531.79	67.51552	8554.18	7147.47	5004.52	181.06	180	-70
OSK10-26	254	606311.23	5006381.06	70.1188	8499.63	6997.71	5007.12	1.06	0	-65
OSK10-27	151	606307.96	5006531.85	70.43982	8499.16	7148.55	5007.44	181.06	180	-55
OSK10-28	251	606307.10	5006581.83	71.97155	8499.23	7198.55	5008.97	181.06	180	-70
OSK10-29	260	606260.65	5006376.19	71.57029	8448.96	6993.78	5008.57	1.06	0	-60
OSK10-30	182.5	606257.15	5006579.80	73.33872	8449.24	7197.44	5010.34	181.06	180	-50
OSK10-31	250	606257.01	5006580.51	73.44624	8449.11	7198.16	5010.45	181.06	180	-70
OSK10-32	250	606210.39	5006379.17	73.08386	8398.75	6997.69	5010.08	1.06	0	-60
OSK10-33	152	606207.76	5006468.72	76.13621	8397.79	7087.29	5013.14	1.06	0	-55

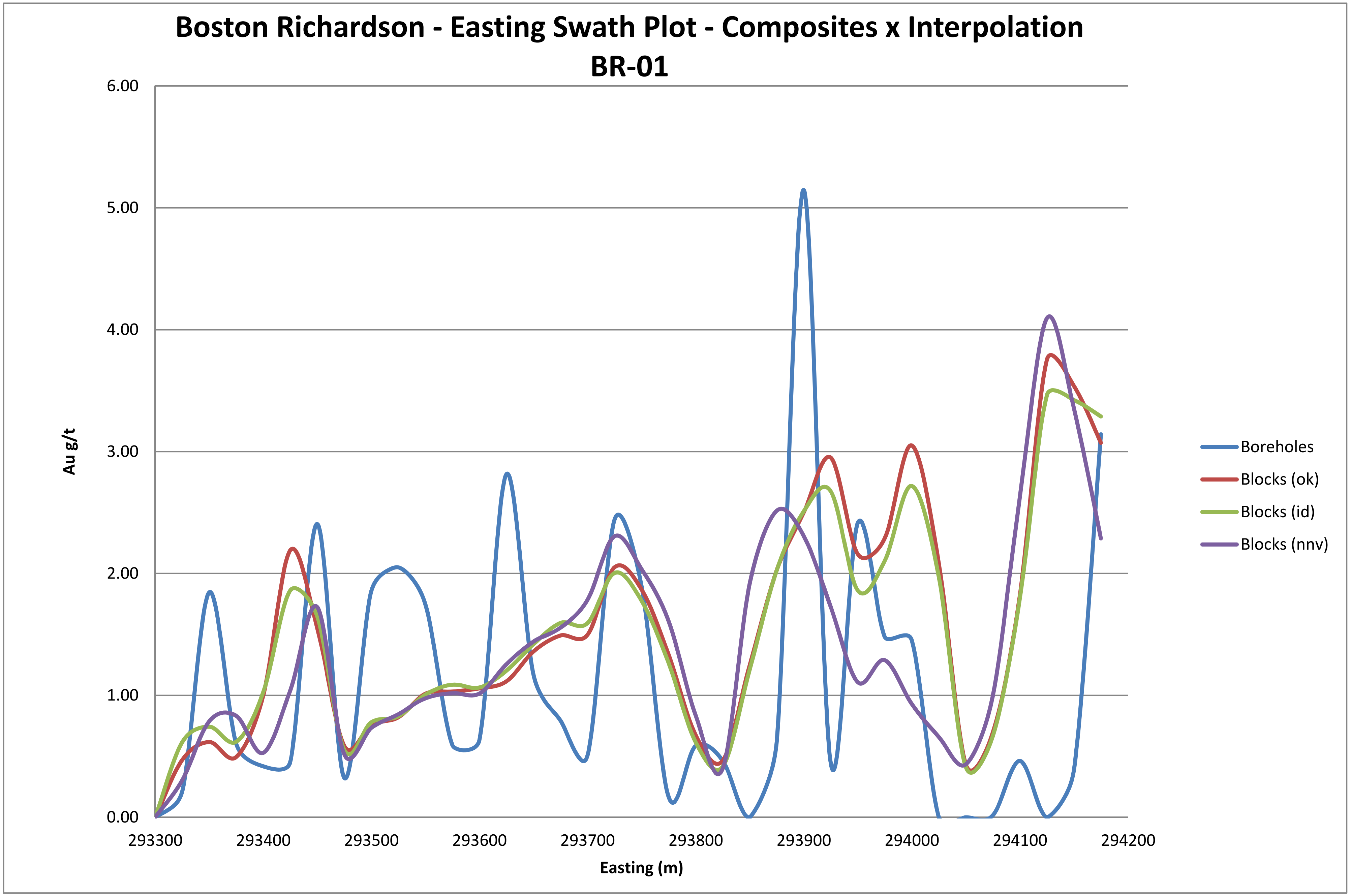
Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
OSK10-34	251	606207.50	5006603.57	75.15294	8400.04	7222.14	5012.15	181.06	180	-65
OSK10-35	191	606161.20	5006404.38	75.58188	8350.03	7023.81	5012.58	1.06	0	-65
OSK10-36	152	606160.01	5006457.61	77.2683	8349.83	7077.06	5014.27	1.06	0	-70
OSK10-37	176	606158.45	5006539.17	77.89784	8349.79	7158.65	5014.90	181.06	180	-70
OSK10-38	225.2	606153.45	5006588.70	76.57309	8345.71	7208.27	5013.57	181.06	180	-65
OSK10-39	200	606119.93	5006402.55	76.72908	8308.73	7022.74	5013.73	1.06	0	-60
OSK10-40	75	606111.97	5006477.88	79.32594	8302.17	7098.23	5016.33	1.06	0	-50
OSK10-41	156.2	606105.79	5006554.48	78.83569	8297.41	7174.94	5015.84	181.06	180	-65
OSK10-42	227	606110.38	5006616.83	78.23842	8303.16	7237.20	5015.24	181.06	180	-60
OSK10-43	254	606061.70	5006404.35	77.50739	8250.53	7025.63	5014.51	1.06	0	-65
OSK10-44	150	606059.07	5006477.88	79.80635	8249.27	7099.21	5016.81	1.06	0	-70
OSK10-45	230	606055.15	5006591.88	80.85986	8247.47	7213.28	5017.86	181.06	180	-65
OSK10-46	114	606057.88	5006552.29	80.98583	8249.46	7173.64	5017.99	181.06	180	-70
OSK10-47	225	606039.62	5006397.68	76.99155	8228.33	7019.37	5013.99	1.06	0	-65
OSK10-48	150.1	606009.65	5006474.55	79.4009	8199.79	7096.80	5016.40	1.06	0	-70
OSK10-49	131	606007.98	5006551.96	81.77402	8199.55	7174.24	5018.77	181.06	180	-70
OSK10-50	224	606008.70	5006583.88	81.74077	8200.87	7206.15	5018.74	181.06	180	-65
OSK10-51	249.5	605151.60	5006648.67	76.41309	7344.98	7286.86	5013.41	181.06	180	-60
OSK10-52	301.5	605149.29	5006825.99	80.5908	7345.97	7464.21	5017.59	181.06	180	-60
OSK10-53	253	604950.51	5006846.77	76.27897	7147.58	7488.69	5013.28	181.06	180	-55
OSK10-54	175.5	604949.17	5006645.24	64.20734	7142.49	7287.19	5001.21	1.06	0	-55
OSK10-55	200.1	605032.07	5006502.98	66.21046	7222.74	7143.39	5003.21	1.06	0	-70
OSK10-56	134	604892.30	5006522.40	52.66431	7083.34	7165.40	4989.66	1.06	0	-60
OSK10-57	236	605081.29	5006566.25	67.36056	7273.15	7205.74	5004.36	271.06	270	-65
OSK10-58	243.7	605035.89	5006581.05	69.7649	7228.02	7221.39	5006.77	181.06	180	-83
OSK10-59	200	604892.72	5006594.71	57.11633	7085.10	7237.71	4994.12	181.06	180	-80
OSK11-01	247	607451.15	5006168.84	59.138	9635.58	6764.33	4996.14	1.06	0	-50
OSK11-02	286	607449.81	5006073.05	58.706	9632.46	6668.57	4995.71	1.06	0	-55
OSK11-03	251	607696.22	5006126.65	67.498	9879.86	6717.59	5004.50	1.06	0	-50
OSK11-04	299	607695.44	5006023.89	65.218	9877.17	6614.85	5002.22	1.06	0	-55
OSK11-05	293	607950.63	5006098.13	70.604	10133.74	6684.35	5007.60	1.06	0	-55
OSK11-06	234	607949.13	5006015.58	69.304	10130.71	6601.83	5006.30	1.06	0	-50
OSK11-07	42.4	605504.01	5006665.74	81.05	7697.70	7297.36	5018.05	181.06	180	-45
OSK11-08	245	605491.22	5006667.16	81.002	7684.94	7299.02	5018.00	181.06	180	-45
OSK11-09	240	605632.73	5006648.70	82.625	7826.10	7277.94	5019.63	181.06	180	-45
OSK11-10	238	605304.63	5006641.98	77.544	7497.89	7277.31	5014.54	181.06	180	-45
BR-17-01	73	606465.22	5006394.39	64.54	5007012.97	293457.11	5004.54	360		-45
BR-17-02	197	606470.91	5006400.07	64.24	5007018.54	293462.91	5004.24	360		-90
BR-17-03	72	606522.46	5006411.12	63.50	5007028.63	293514.67	5003.50	360		-90
BR-17-04	151	607154.44	5006247.48	51.99	5006853.25	294143.60	4991.99	360		-45
BR-17-05	150	606957.58	5006296.25	60.44	5006905.68	293947.65	5000.44	360		-50
BR-17-06	549	606979.71	5006131.18	54.36	5006740.19	293966.71	4994.36	352		-67
BR-17-07	624	607369.38	5006006.50	52.73	5006608.27	294354.05	4992.73	355		-67
BR-17-08	401.7	606971.27	5006211.85	55.48	5006821.02	293959.77	4995.48	360		-65.5
BR-17-09	350.3	606970.58	5006257.41	56.06	5006866.59	293959.93	4996.06	360		-67
BR-17-10	149	606954.44	5006403.44	58.57	5007012.92	293946.49	4998.57	180		-60
BR-17-11	149	606907.29	5006406.20	60.88	5007016.56	293899.40	5000.88	180		-60
BR-17-12	677			51.68	5006671.35	294077.56	4991.68	360		-65
BR-17-13	653.3			51.68	5006671.53	294077.58	4991.68	360		-58
BR-18-14	151			59.25	5006997.42	294312.81	4999.25	180		-70
BR-18-15	225			62.60	5007017.81	294372.73	5002.60	180		-65
BR-18-16	325			63.03	5007014.45	294439.90	5003.03	180		-53
BR-18-17	175			56.96	5007006.48	294235.65	4996.96	180		-55
BR-18-18	273			57.02	5007005.91	294235.59	4997.02	180		-75
BR-18-19	242			60.33	5007036.89	294315.52	5000.33	180		-70
BR-18-20	287			59.45	5007079.99	293903.53	4999.45	180		-68
BR-18-21	472.6			56.49	5007143.70	293900.93	4996.49	180		-70
BR-18-22	278			63.26	5006900.49	293903.96	5003.26	356		-74
BR-18-23	434			55.56	5006861.32	293904.03	4995.56	358		-75
BR-18-23A	29			0.00	5006874.00	293906.00	5000.00	358		-74
BR-18-24	256.8			54.56	5006758.14	294063.85	4994.56	360		-50
BR-18-25	313			56.79	5006717.00	294063.48	4996.79	360		-55
BR-18-26	232			56.70	5006993.56	294162.70	4996.70	180		-68
BR-18-27	121			56.73	5006993.03	294162.79	4996.73	180		-50
BR-18-28	245			58.65	5006853.47	294363.02	4998.65	360		-78.5
BR-18-29	445			58.60	5006798.90	294359.71	4998.60	360		-74
BR-18-30	596			58.59	5007142.97	293854.49	4998.59	180		-72
BR-18-31	98.7			60.24	5007029.92	293851.22	5000.24	180		-72
BR-18-32	94.5			62.02	5006954.47	293853.38	5002.02	360		-75
BR-18-33	509.5			58.04	5006848.45	293845.58	4998.04	360		-75

Hole Number	Depth (m)	East (m) UTM Zone 20 NAD 83	East (m) UTM Zone 20 NAD 83	Elevation (m) ASL	Local Grid East (m)	Local Grid North (m)	Local Grid Elevation (m)	Grid Azimuth	UTM Azimuth	Inclination (Deg.)
BR-18-34	426			58.82	5006878.19	293848.63	4998.82	360		-75
BR-18-35	49			62.19	5006988.14	293512.21	5002.19	360		-48
BR-18-36	165			62.32	5006975.92	293511.85	5002.32	360		-65
BR-18-37	147			63.28	5006957.79	293528.39	5003.28	360		-53
BR-18-38	147			61.62	5006937.83	293579.04	5001.62	360		-50
BR-18-39	137			61.24	5006945.60	293593.73	5001.24	360		-50
BR-18-39A	96.3			0.00	5006945.00	293596.00	5000.00	360		-60
BR-18-40	50			62.17	5006990.21	293525.78	5002.17	360		-46
BR-18-41	603.2			60.54	5007129.09	293806.15	5000.54	180		-75
BR-18-42	536			54.21	5006789.24	293790.37	4994.21	354		-68.5
BR-18-43	394			69.41	5006904.81	293306.40	5009.41	357		-55
BR-18-44	515			55.56	5006755.04	293716.38	4995.56	355		-60
BR-18-44A	14			58.00	5006751.00	293715.00	4998.00	355		-60
BR-18-45	515			59.60	5006756.52	293604.61	4999.60	357		-60
BR-18-46	494			61.97	5006779.10	293498.84	5001.97	360		-60
BR-18-47	506			62.78	5006778.58	293420.79	5002.78	357		-60
BR-18-48	596			52.62	5006724.56	293896.51	4992.62	357		-65
BR-18-49	611			65.91	5007236.95	293605.04	5005.91	175		-62
BR-18-50	260			76.88	5007037.74	292950.86	5016.88	360		-61
BR-18-51	332			76.55	5006995.20	292954.50	5016.55	360		-61
BR-18-52	368			77.09	5006986.49	292902.53	5017.09	360		-61
BR-18-53	246			77.37	5007055.53	292905.64	5017.37	360		-58
BR-18-54	149			75.26	5007026.36	293132.23	5015.26	360		-70
BR-18-55	143			75.16	5007025.82	293132.17	5015.16	360		-50
BR-18-56	347			72.07	5006949.81	293200.44	5012.07	360		-60
BR-18-57	362			73.55	5006983.64	293154.68	5013.55	360		-65
BR-18-58	182			73.29	5007035.60	293204.26	5013.29	360		-60
BR-18-59	350			76.12	5007254.35	293153.28	5016.12	180		-65
BR-18-60	449			73.90	5007295.95	293203.26	5013.80	177		-58
BR-18-61	410			71.54	5007107.61	293300.17	5011.54	180		-88
BR-18-62	407			77.66	5007130.59	293153.36	5017.66	180		-88
BR-18-63	467			80.17	5007149.74	293042.27	5020.17	180		-88
BR-18-64	485			61.51	5006906.67	294443.71	5001.51	180		-85
BR-18-65	338			66.62	5007023.58	294558.47	5006.62	180		-55
BR-18-66	304.4			64.33	5006726.10	294559.97	5004.33	360		-55
BR-18-67	328.3			66.04	5006687.25	294646.51	5006.04	360		-55
BR-18-68	137			51.69	5006843.86	294166.15	4991.69	360		-55
BR-18-69	167			52.00	5006853.86	294152.44	4992.00	360		-50
BR-18-70	107			52.21	5006865.13	294113.09	4992.21	0		-46
BR-18-71	134			52.24	5006860.21	294139.85	4992.24	0		-50
BR-19-72	168			66.15	5007219.58	294748.23	5006.15	180		-45
BR-19-73	225.1			70.60	5006920.38	294652.74	5010.60	180		-65
BR-19-74	329			71.85	5006584.99	294739.23	5011.85	360		-55
BR-19-75	359			66.80	5006537.26	294825.85	5006.80	360		-55
BR-19-76	296			71.85	5006584.99	294739.23	5011.85	360		-55
BR-19-77	83			59.09	5007031.00	293809.51	4999.09	180		-75
BR-19-78	120			58.84	5007064.20	293810.47	4998.84	180		-75
BR-19-79	62			60.37	5006998.50	293792.87	5000.37	270		-90
BR-19-80	75.5			61.42	5006943.42	293754.61	5001.42	360		-70
BR-19-81	90.4			62.22	5006935.01	293801.45	5002.22	360		-72
BR-19-82	37			61.78	5006983.17	293710.36	5001.78	270		-90
BR-19-83	66			59.91	5007044.85	293753.67	4999.91	180		-70
BR-19-84	38			60.29	5007044.09	293704.75	5000.29	180		-70
BR-19-85	30.8			63.31	5007011.96	293724.38	5003.31	270		-90
BR-19-86	176			63.07	5006692.15	294524.97	5003.07	360		-53
BR-19-87	332			54.89	5006746.40	294334.55	4994.89	360		-65
BR-19-88	194			64.47	5006676.62	294575.96	5004.47	360		-55
BR-19-89	326			52.70	5006740.40	294240.35	4992.70	360		-60
BR-19-90	341			51.65	5006734.04	294178.00	4991.65	360		-58
BR-19-91	347			55.66	5006716.02	294083.21	4995.66	18		-68
BR-19-92	392			55.58	5006715.38	294083.01	4995.58	18		-45
BR-19-93	125			58.13	5006771.50	294402.95	4998.13	360		-55
BR-19-94	86			62.10	5006836.02	294402.16	5002.10	360		-60
BR-19-95	95			66.15	5006797.69	294505.24	5006.15	360		-55
BR-19-96	128			67.10	5006781.07	294560.20	5007.10	360		-55
BR-19-97	149			64.27	5006735.41	294588.29	5004.27	360		-55
BR-19-99	323			54.66	5006750.72	294042.09	4994.66	360		-65

END OF DOCUMENT

APPENDIX

B SWATH PLOTS



Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-01
Au g/t
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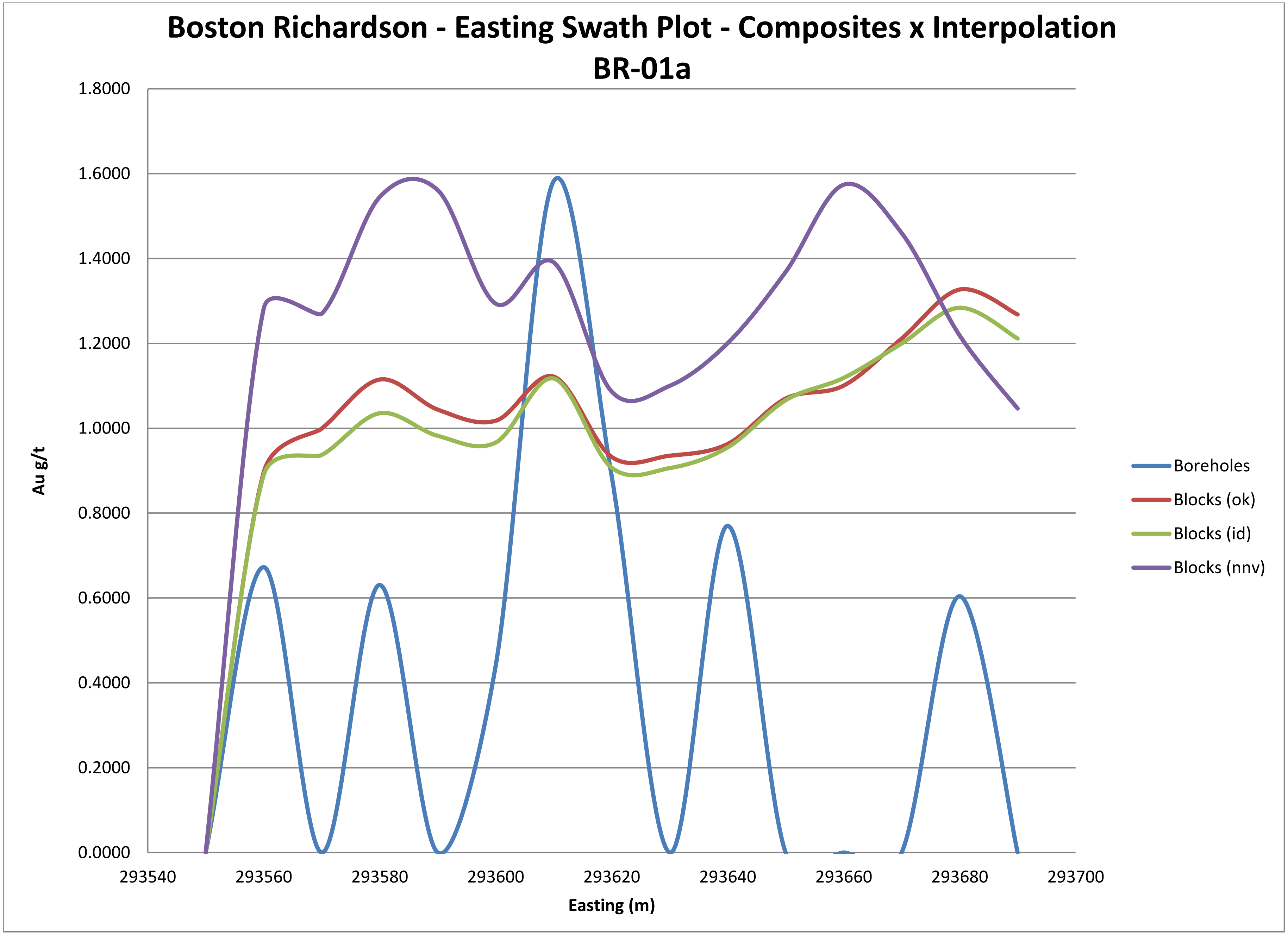
4.00
3.00
2.00
1.00
0.00
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-01a
Au g/t
Easting (m)
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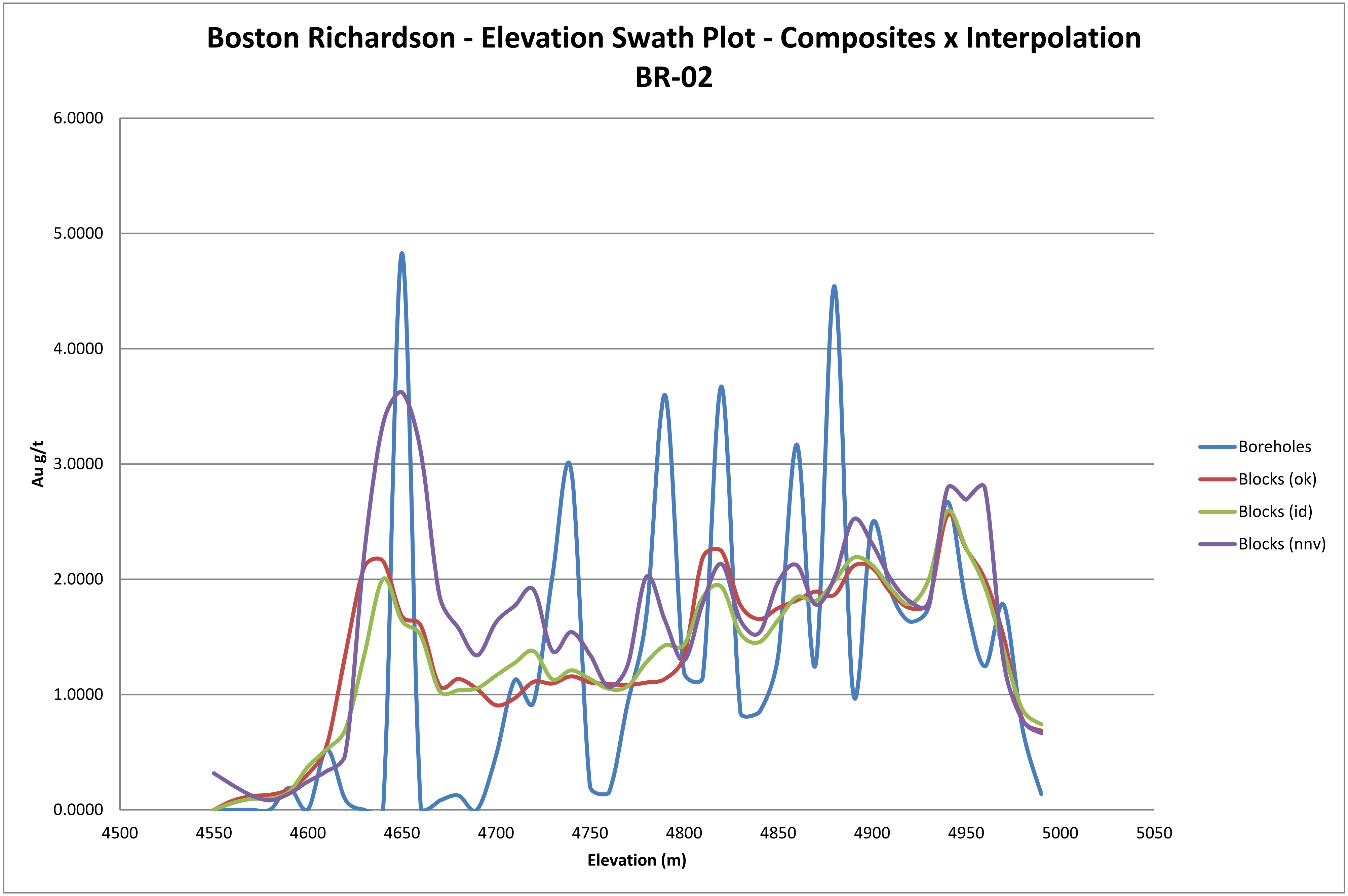
1.4000
1.2000
1.0000
0.8000
0.6000
0.4000
0.2000
0.0000
293540
293560
293580
293600
293620
293640
293660
293680
293700
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)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Au g/t
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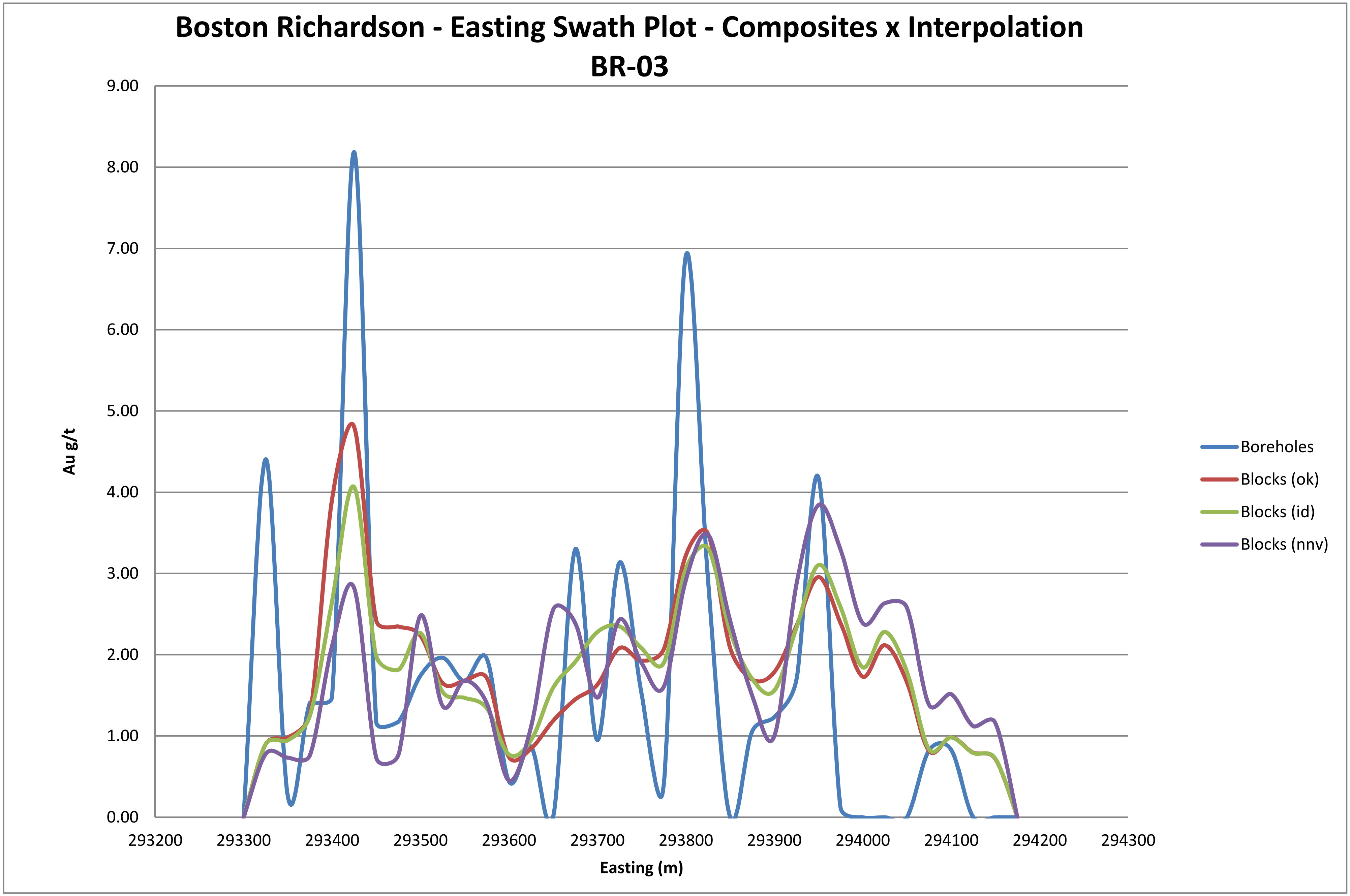
5.0000
4.0000
3.0000
2.0000
1.0000
0.0000
4500
4550
4600
4650
4700
4750
4800
4850
4900
4950
5000
5050
Elevation (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-02a
Au g/t
1.20
1.00
0.80
0.60
0.40
0.20
0.00
293580
293600
293620
293640
293660
293680
293700
293720
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)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BR-03
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7.00
6.00
5.00
4.00
3.00
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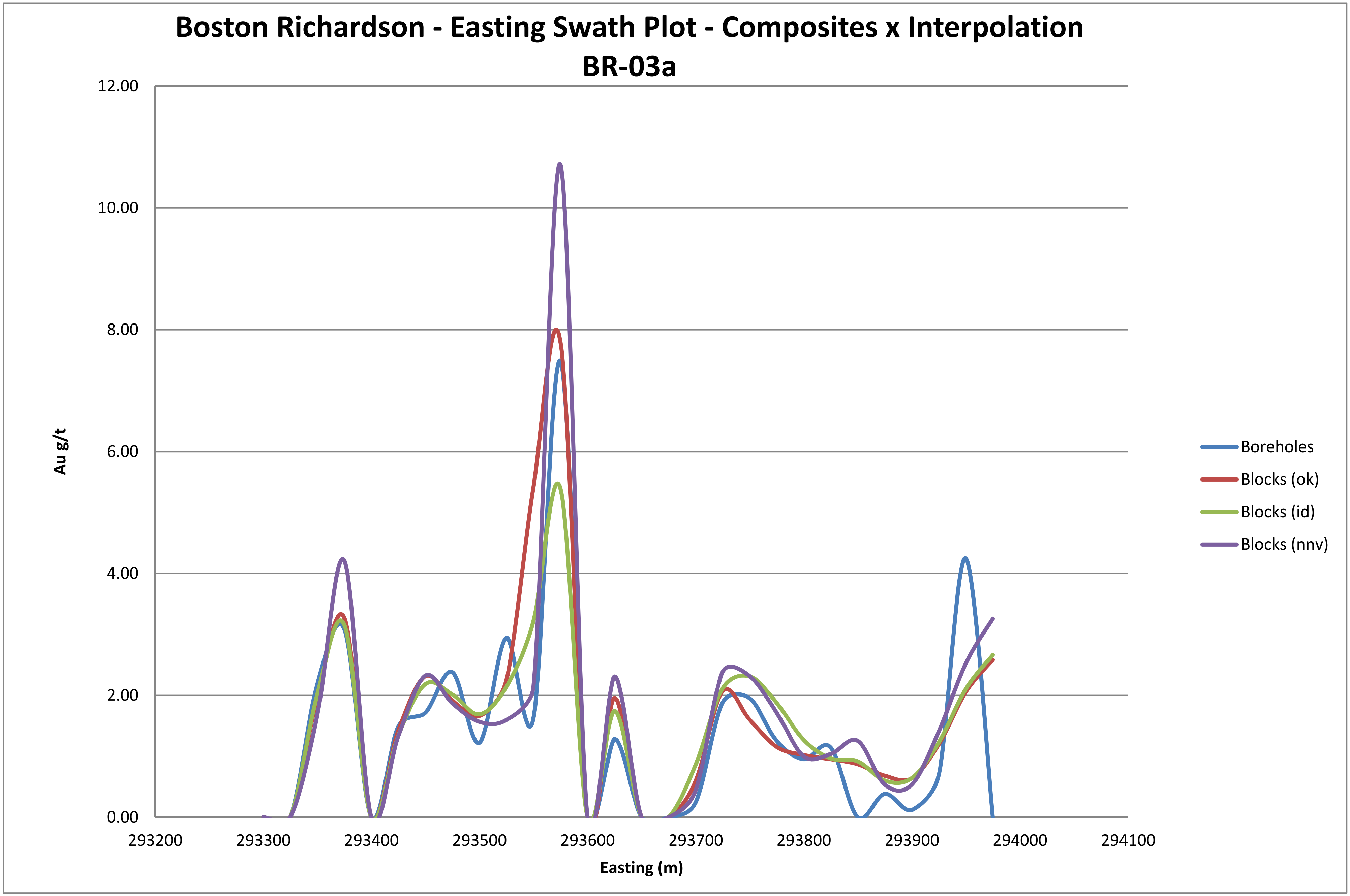
1.00
0.00
Au g/t
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-03a
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8.00
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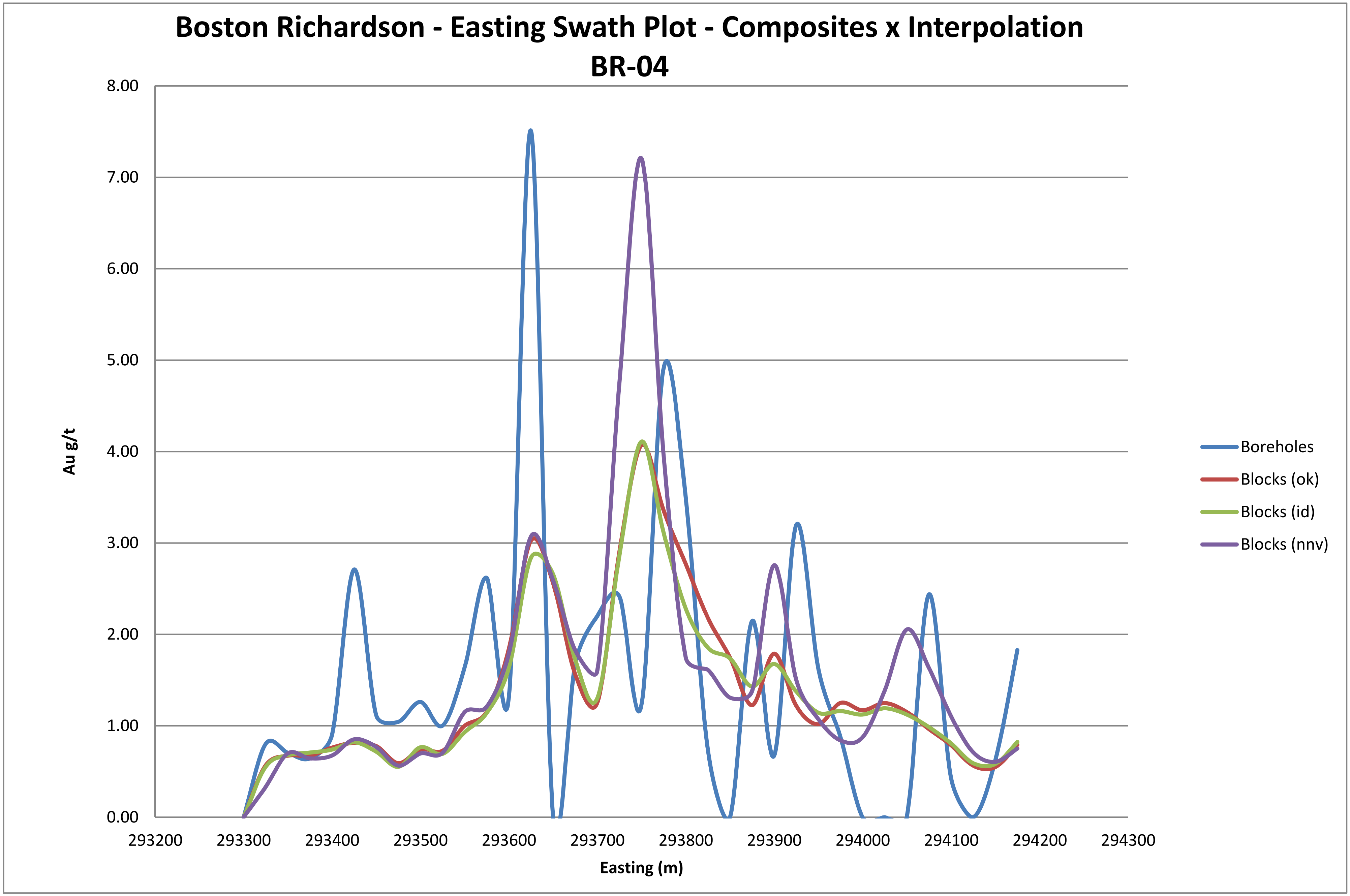
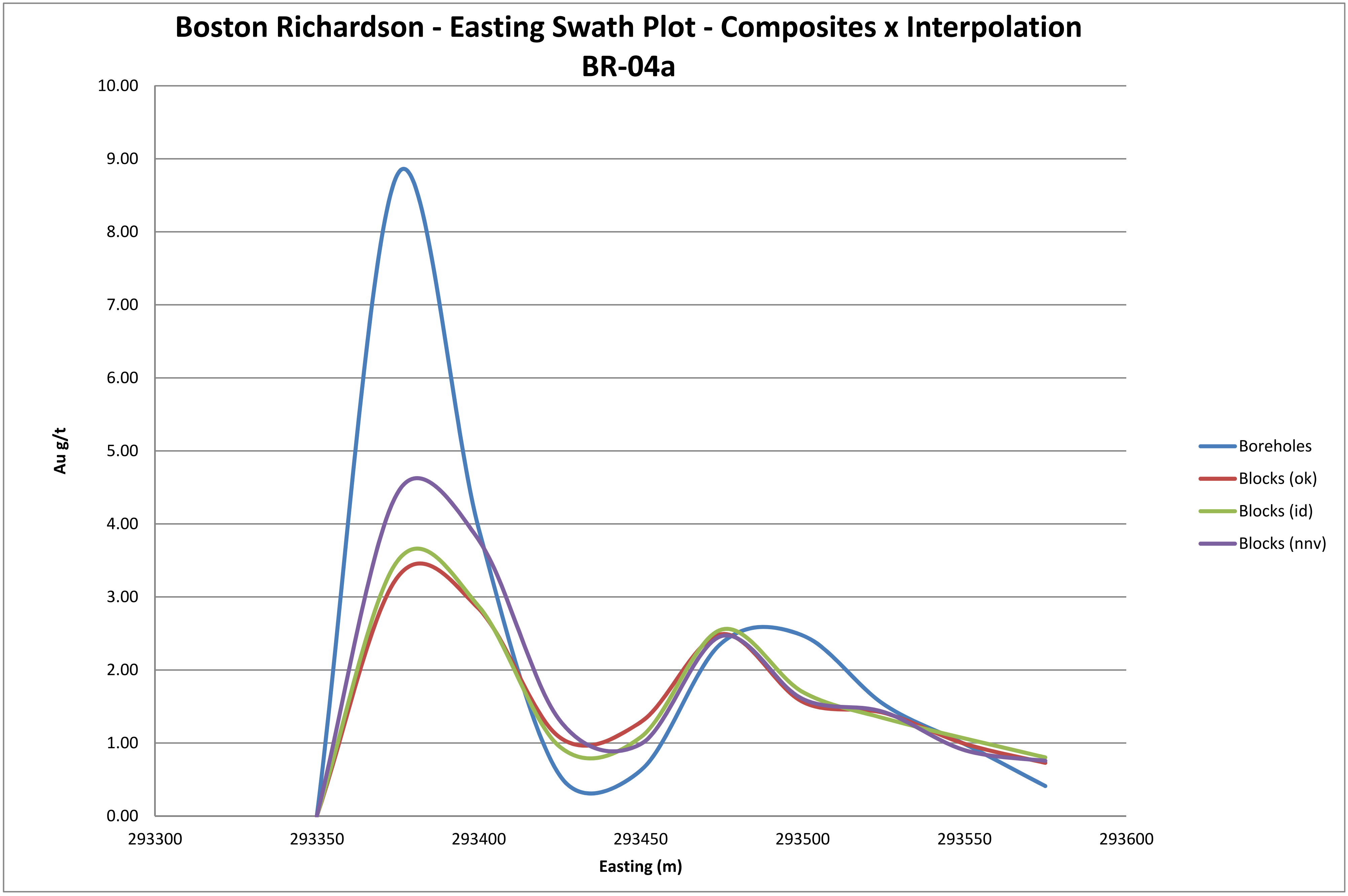
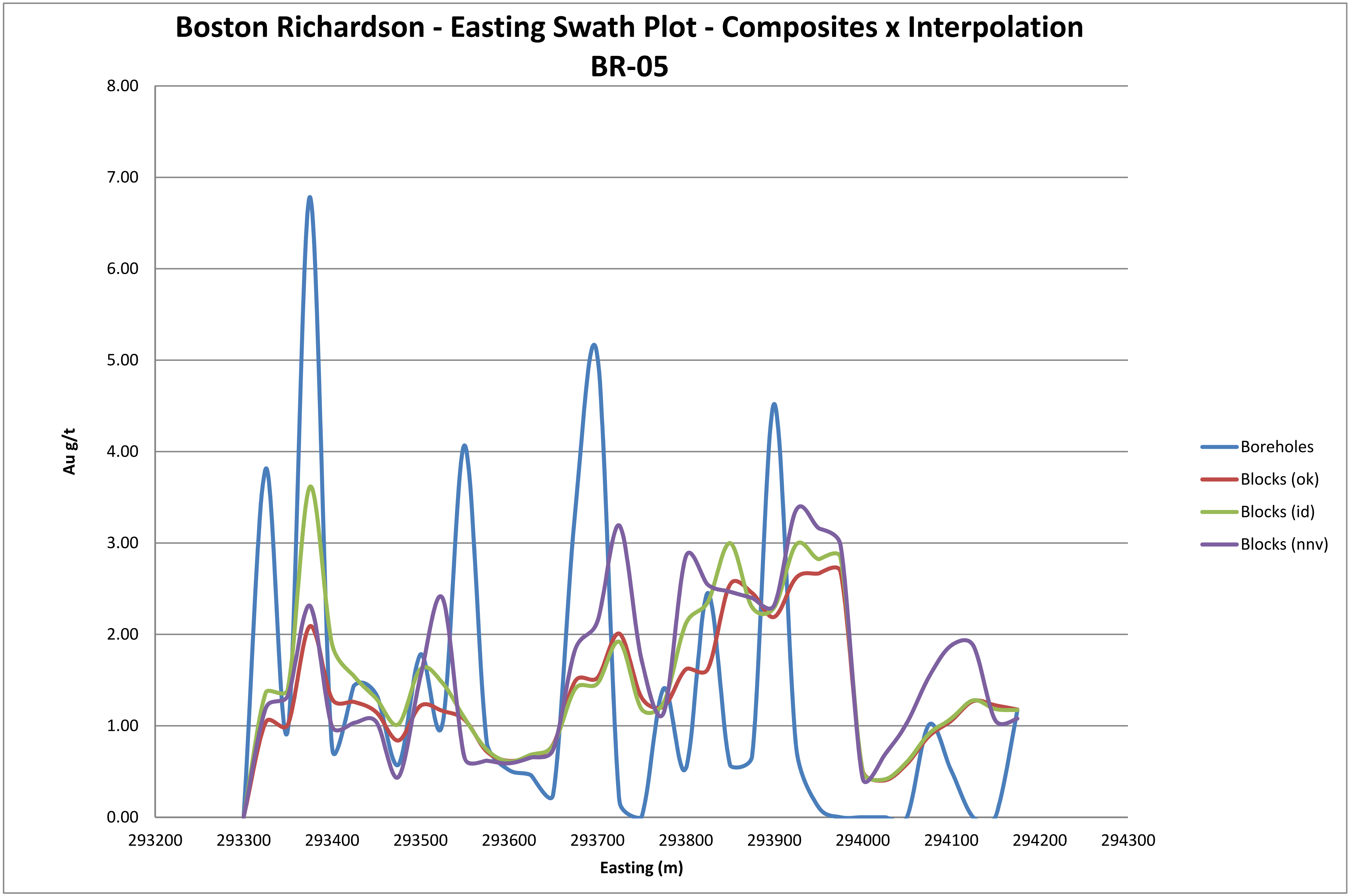
4.00
2.00
0.00
Au g/t
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)



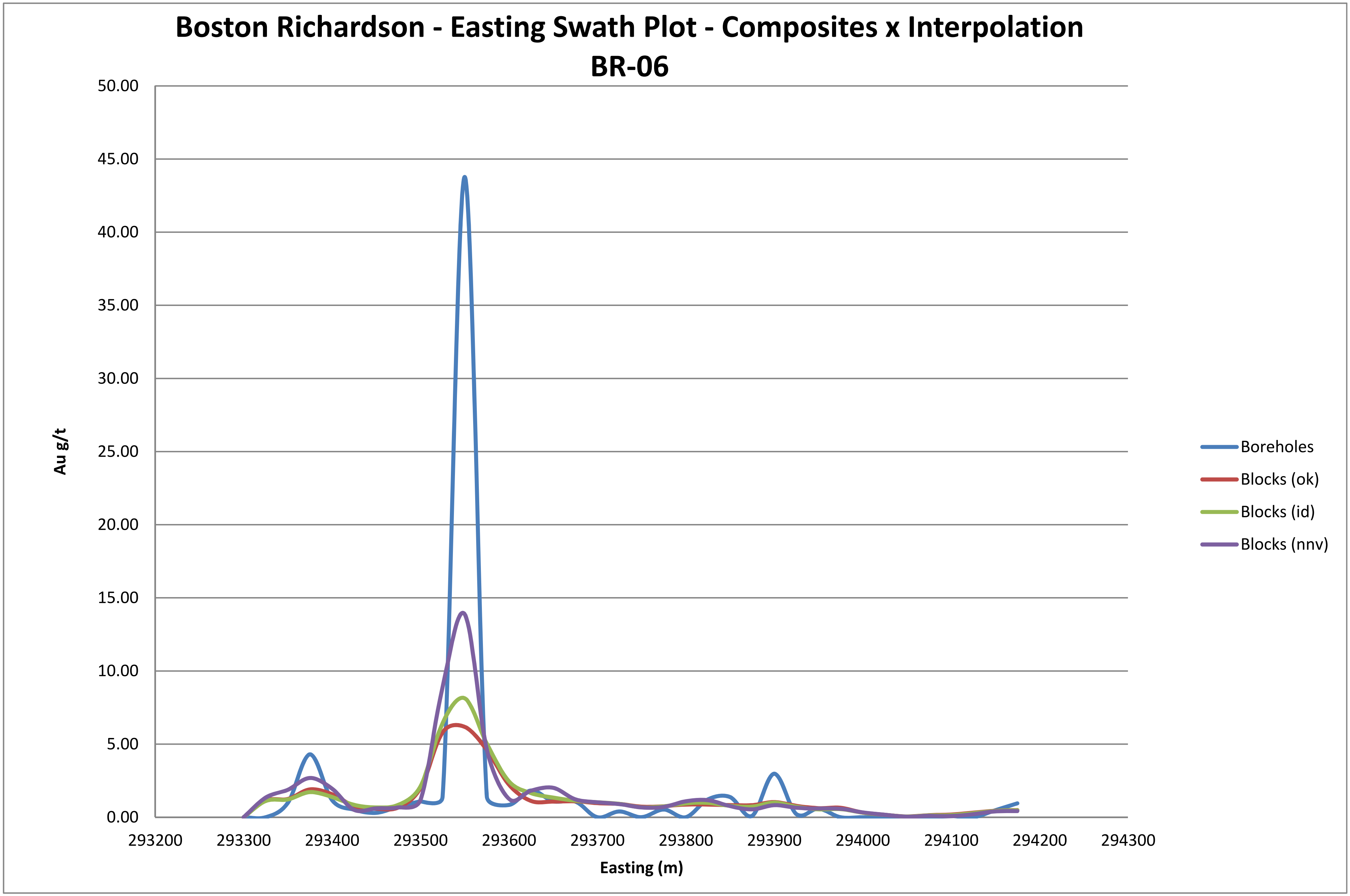
Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-04
Au g/t
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)



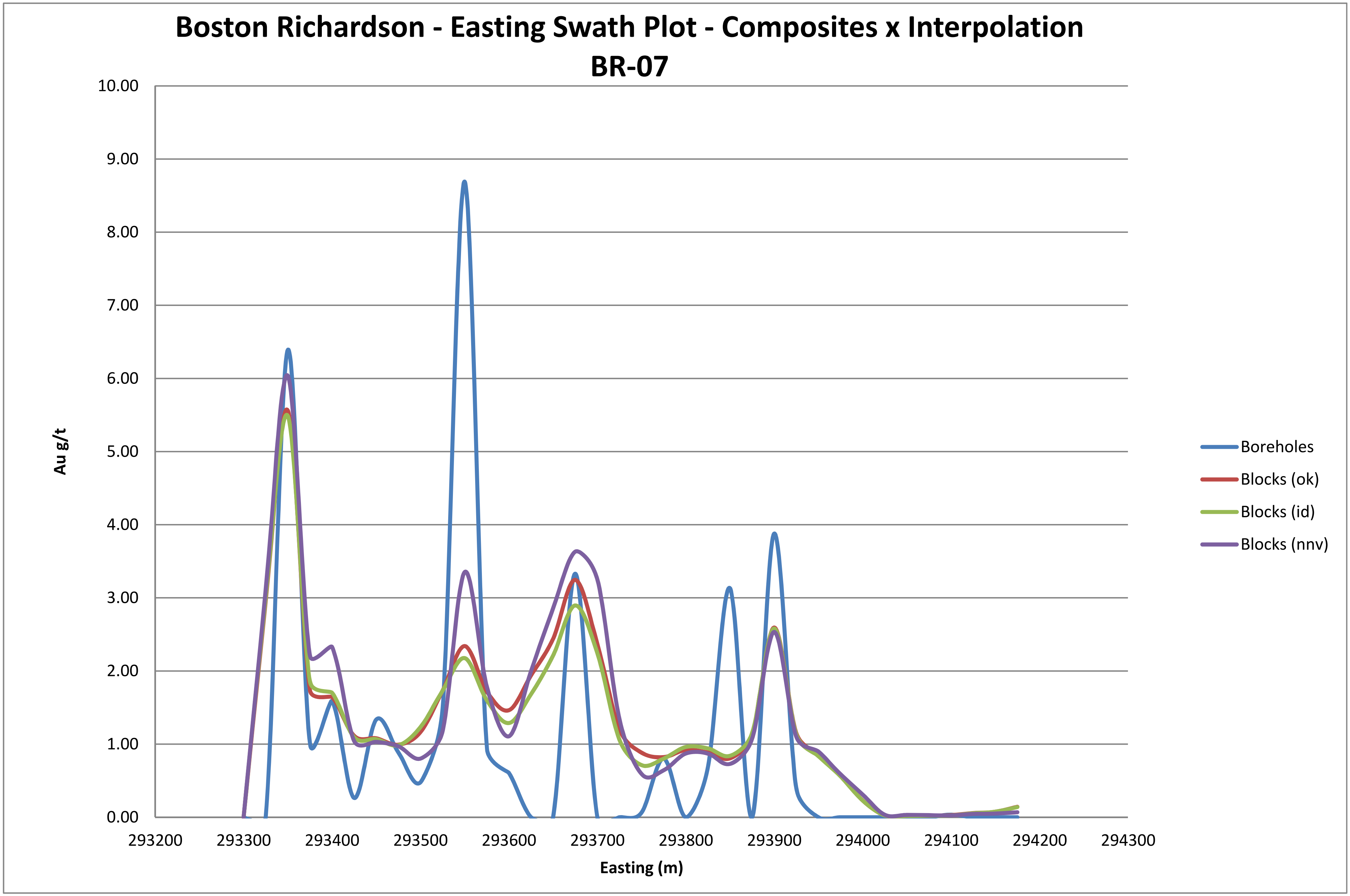
Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-04a
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Au g/t
293300
293350
293400
293450
293500
293550
293600
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

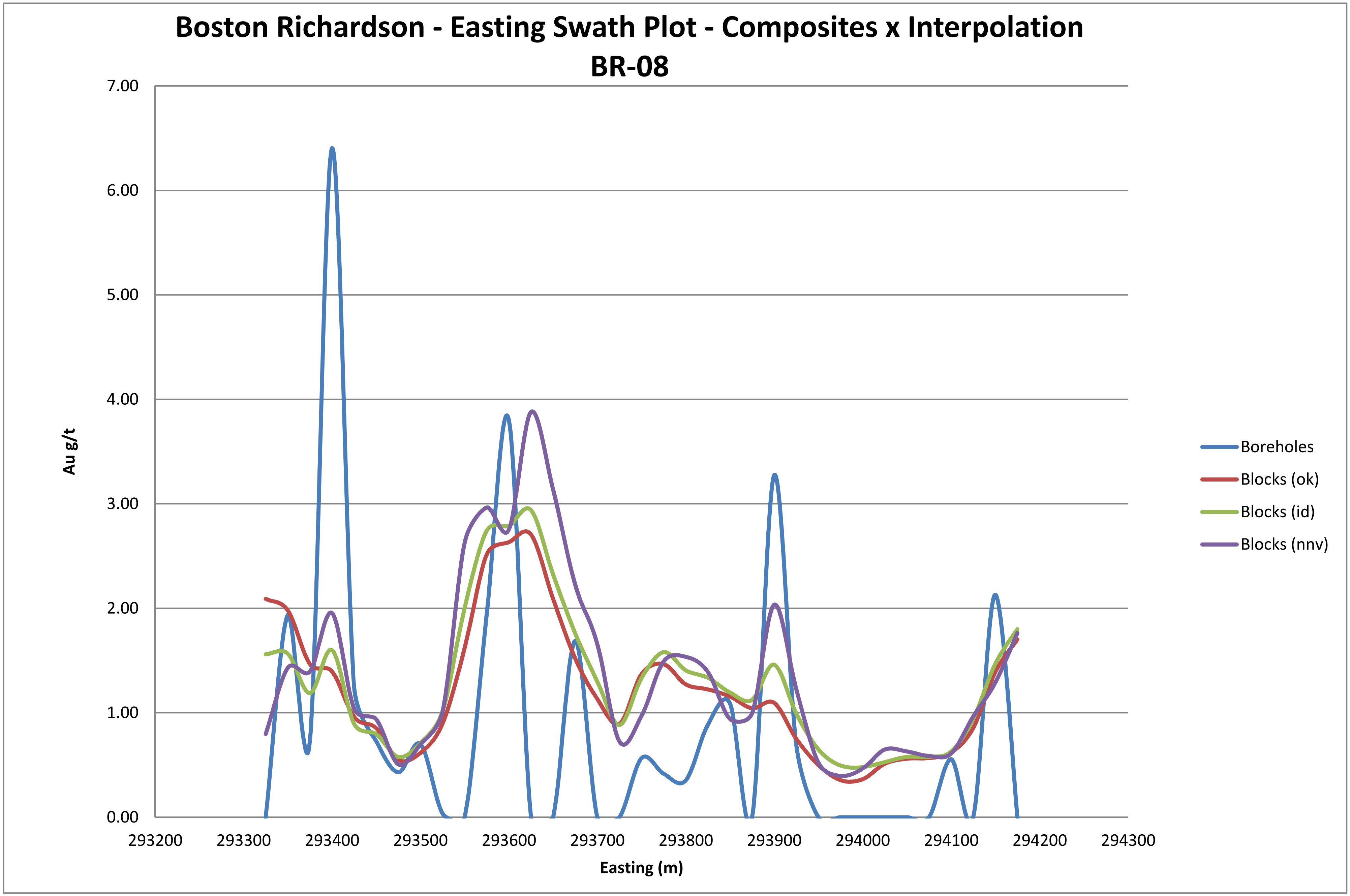


Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-05
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Au g/t
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)



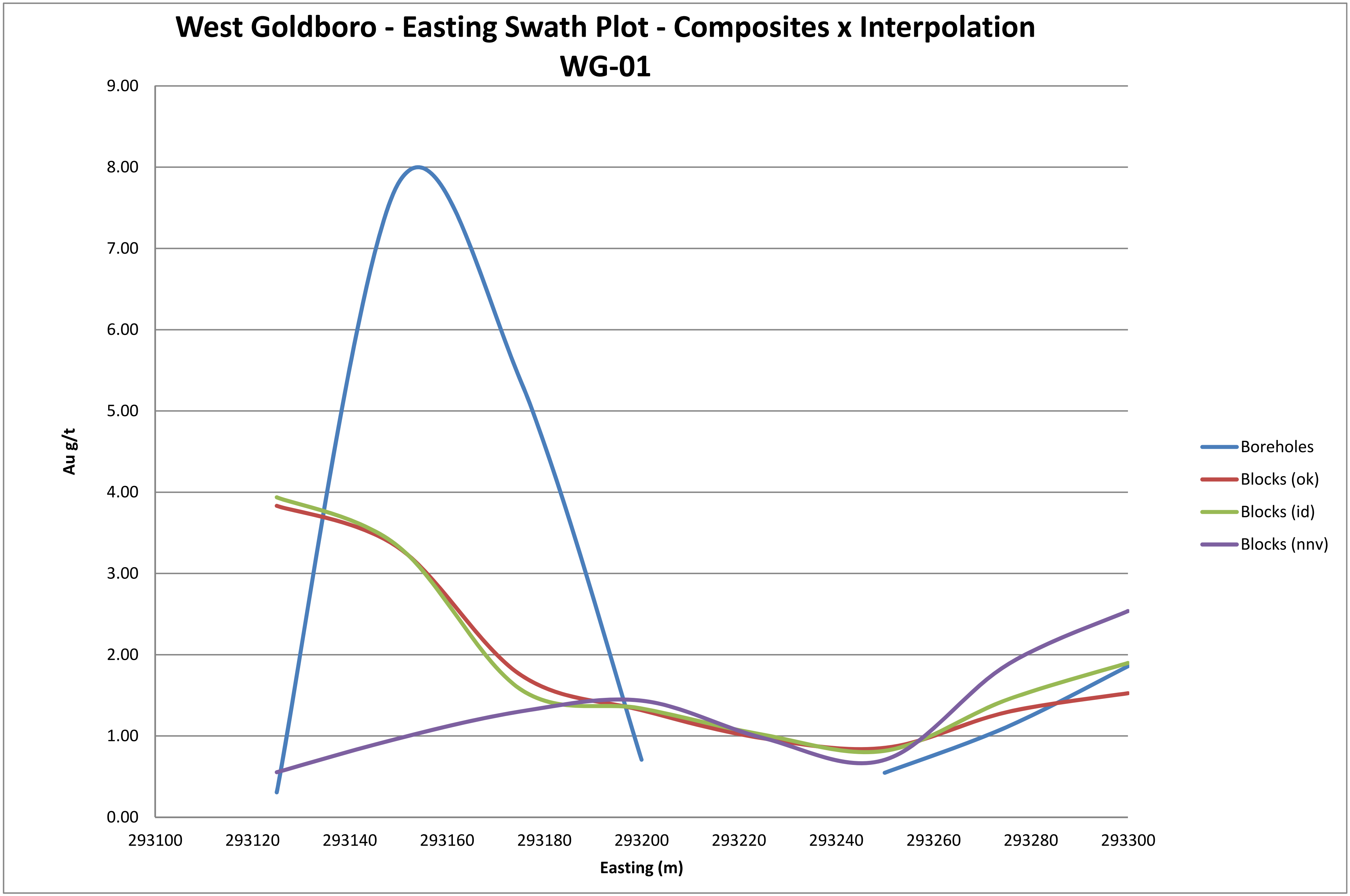
Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-06
50.00
45.00
40.00
35.00
30.00
25.00
20.00
15.00
10.00
5.00
0.00
Au g/t
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-07
Au g/t
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

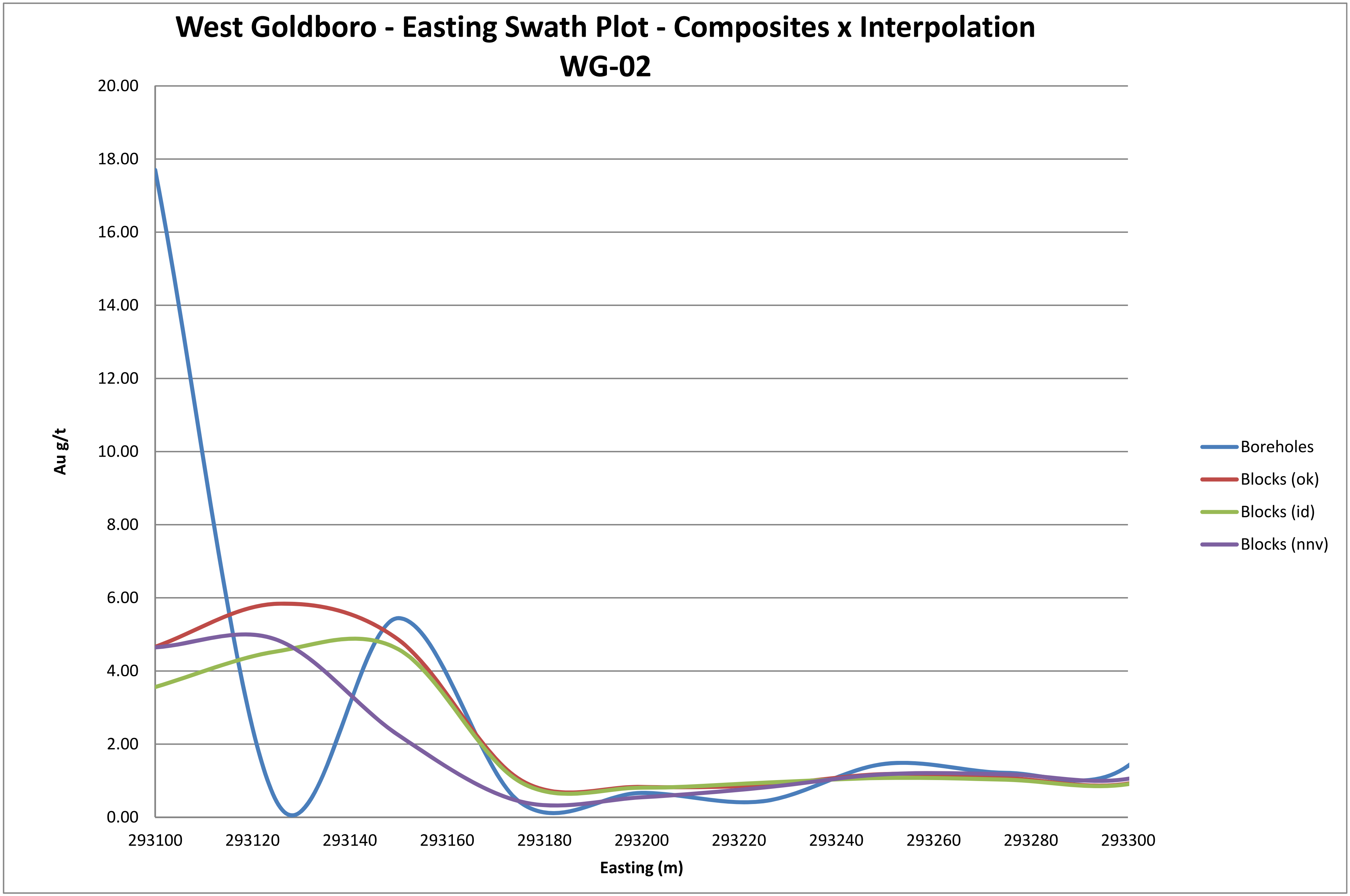


Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-08
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
Au g/t
293200
293300
293400
293500
293600
293700
293800
293900
294000
294100
294200
294300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

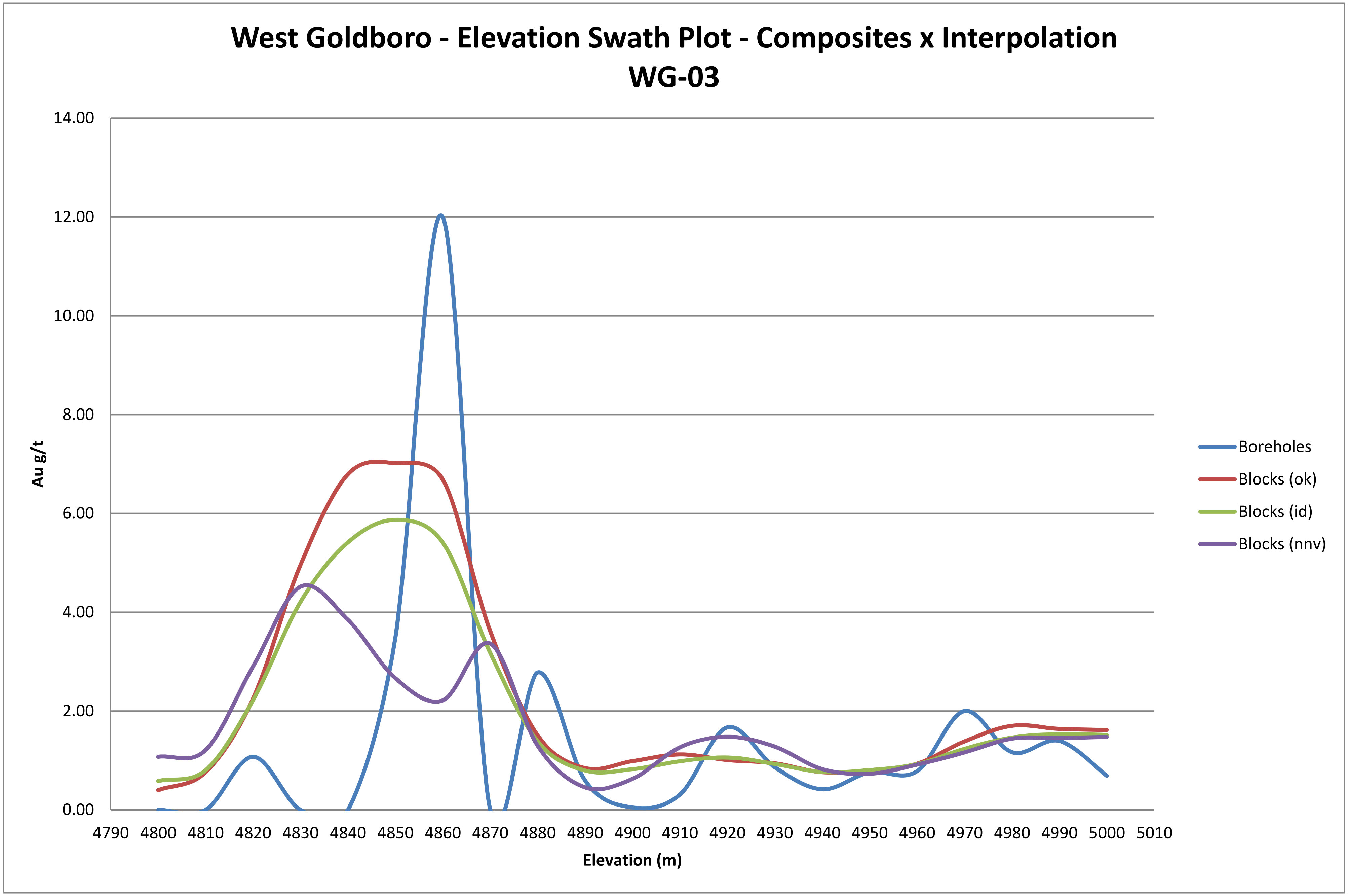
Boston Richardson - Easting Swath Plot - Composites x Interpolation
BR-08
Au g/t
12.00
10.00
8.00
6.00
4.00
2.00
0.00
293950
294000
294050
294100
294150
294200
294250
294300
294350
294400
Easting (m)
Boreholes
Blocks (ok)
Blocks (nnv)



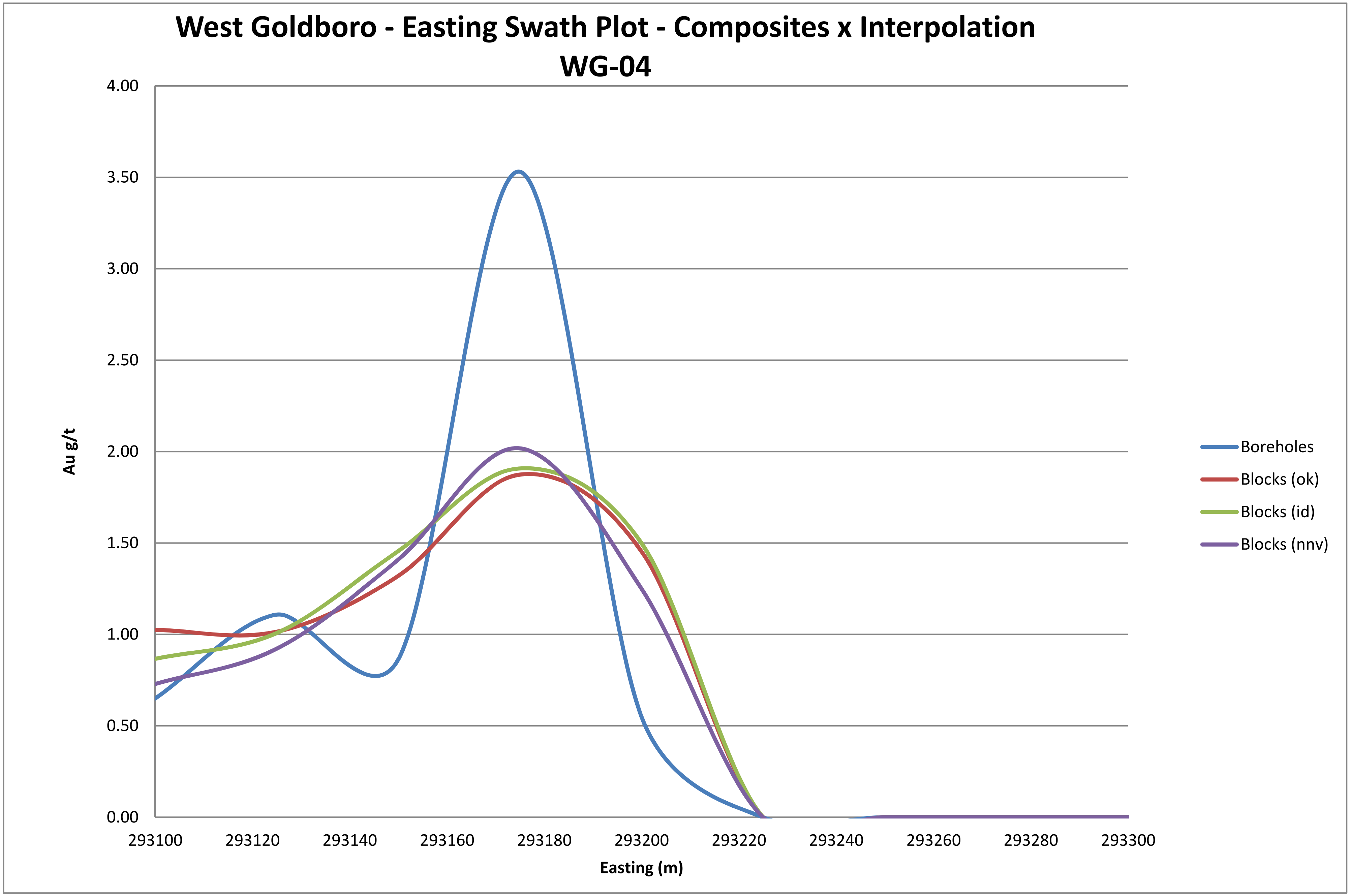
West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-01
Au g/t
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
293100
293120
293140
293160
293180
293200
293220
293240
293260
293280
293300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

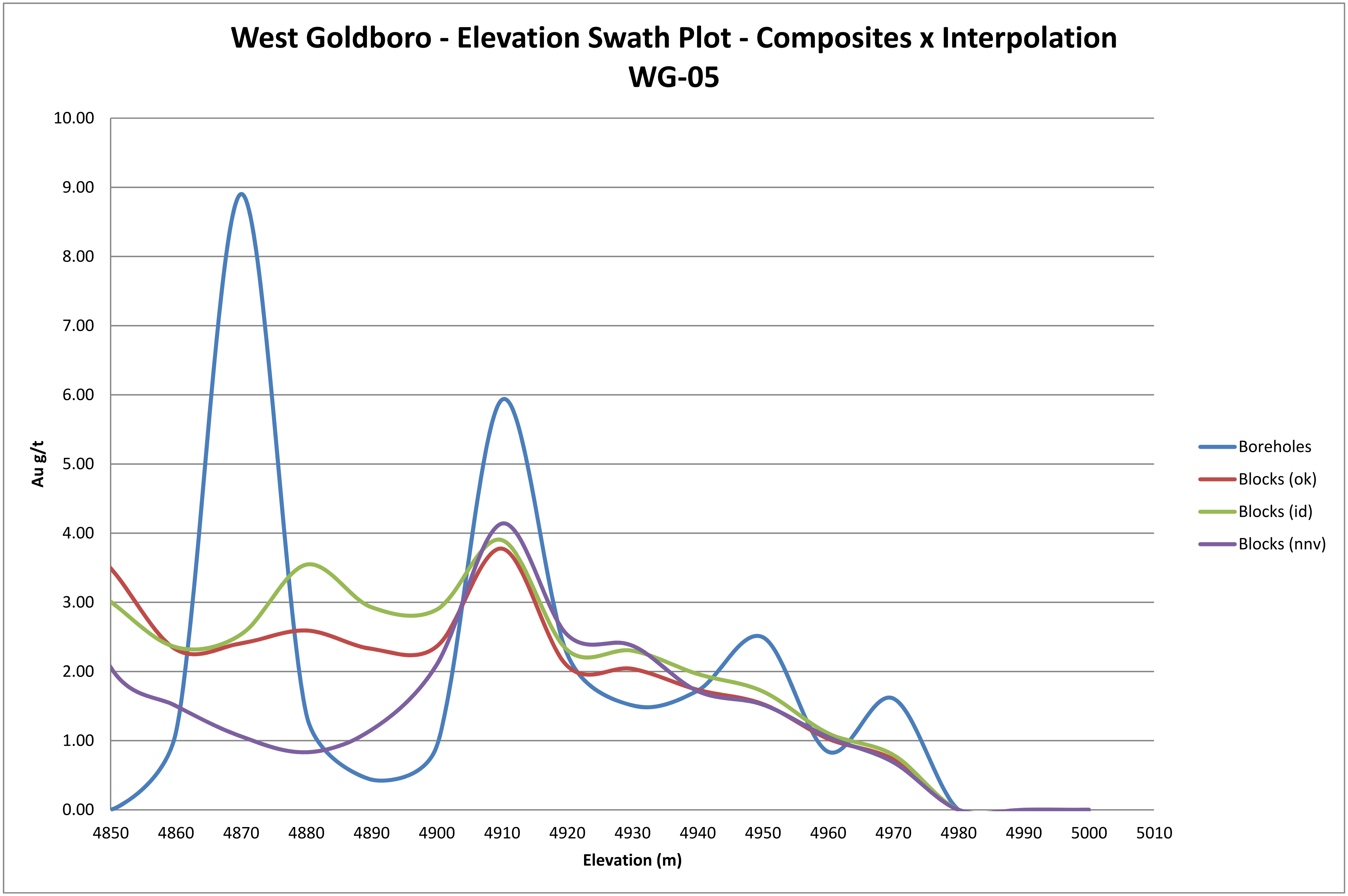


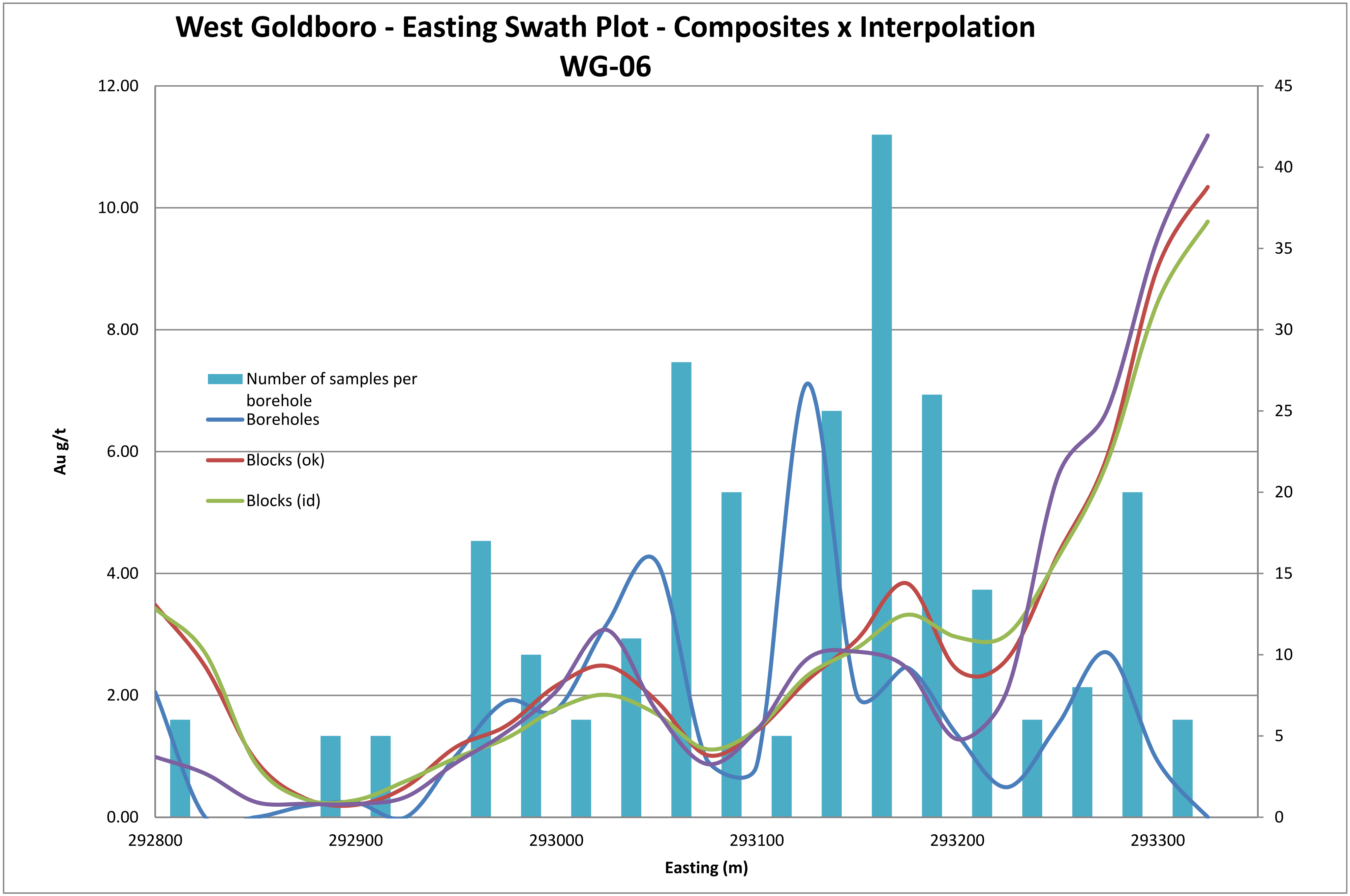
West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-02
20.00
18.00
16.00
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0.00
Au g/t
293100
293120
293140
293160
293180
293200
293220
293240
293260
293280
293300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

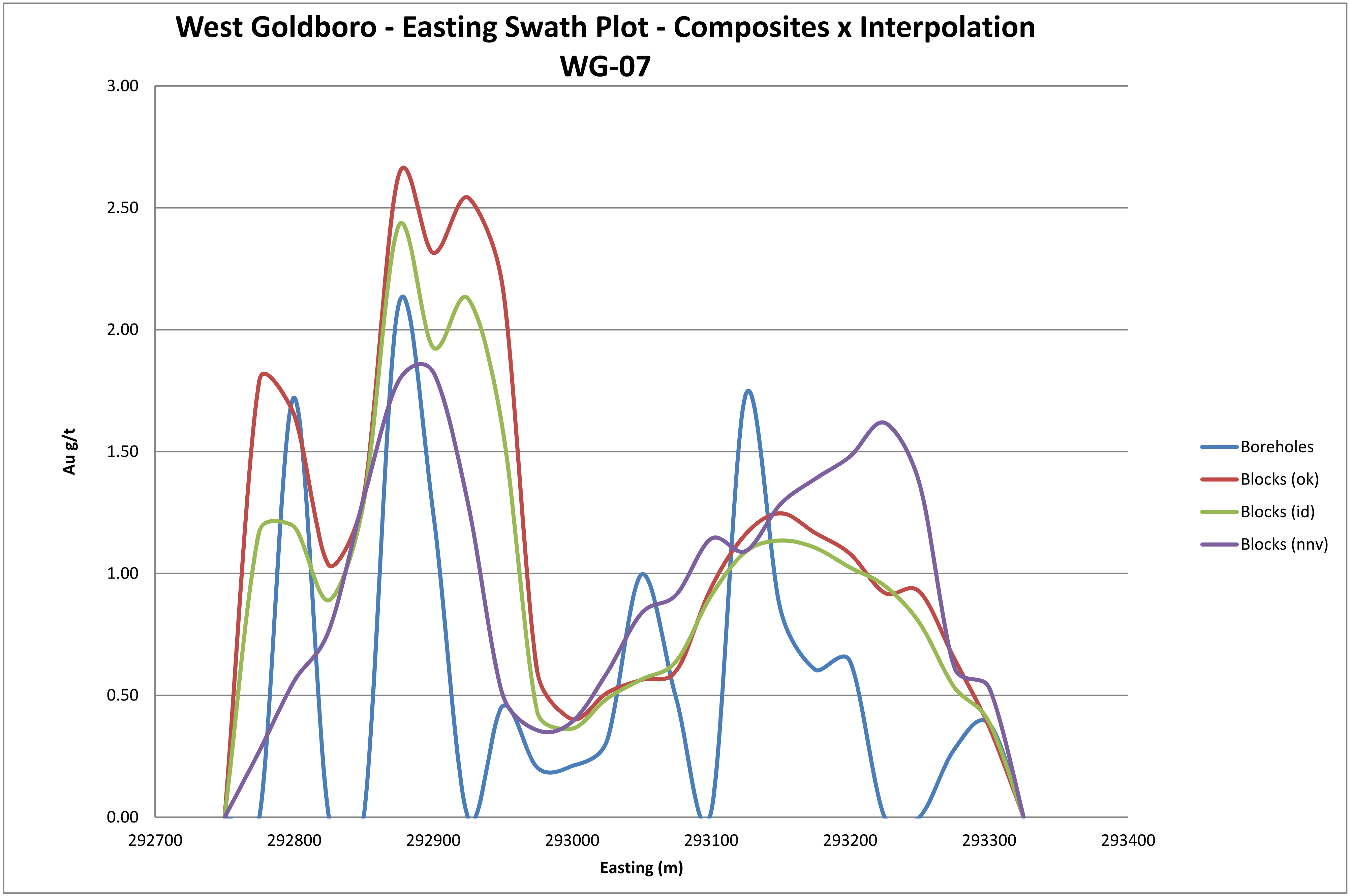


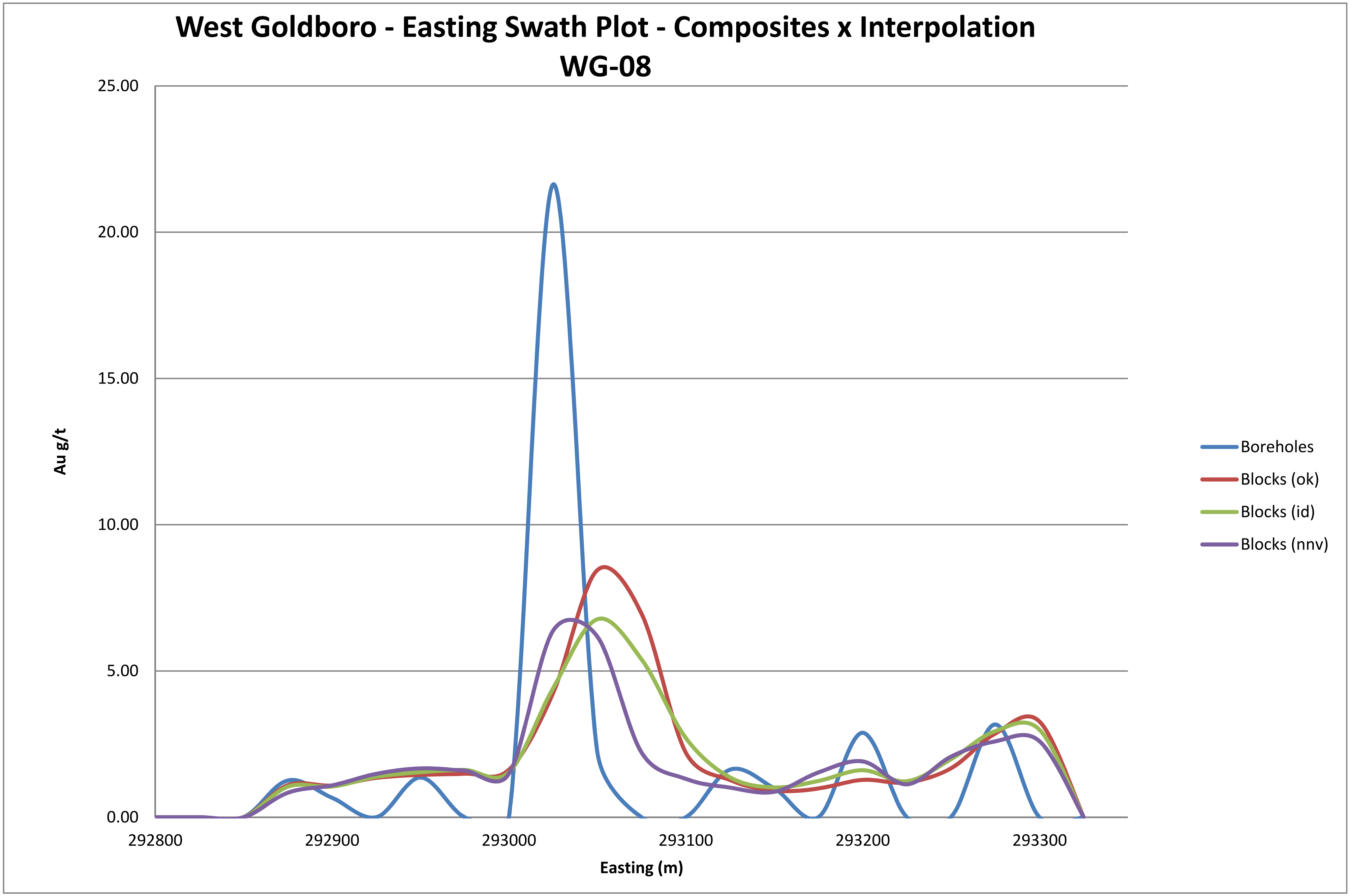
West Goldboro - Elevation Swath Plot - Composites x Interpolation
WG-03
Au g/t
14.00
12.00
10.00
8.00
6.00
4.00
2.00
0.00
4790 4800 4810 4820 4830 4840 4850 4860 4870 4880 4890 4900 4910 4920 4930 4940 4950 4960 4970 4980 4990 5000 5010
Elevation (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

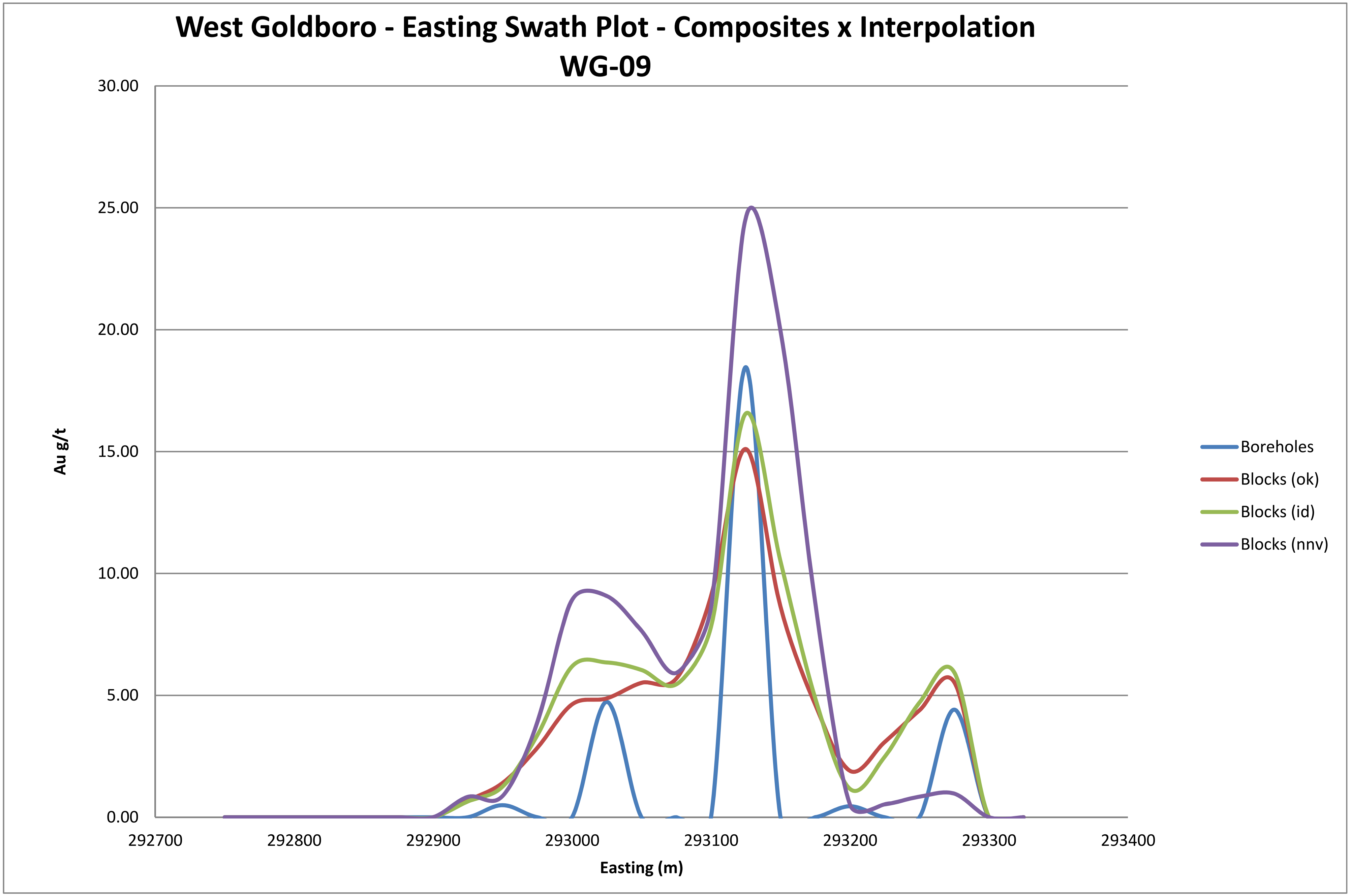

West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-04
4.00
3.50
3.00
2.50
2.00
1.50
1.00
0.50
0.00
Au g/t
293100
293120
293140
293160
293180
293200
293220
293240
293260
293280
293300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


West Goldboro - Elevation Swath Plot - Composites x Interpolation
WG-05
Au g/t
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
4850
4860
4870
4880
4890
4900
4910
4920
4930
4940
4950
4960
4970
4980
4990
5000
5010
Elevation (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-06
Au g/t
12.00
10.00
8.00
6.00
4.00
2.00
0.00
45
40
35
30
25
20
15
10
5
0
292800
292900
293000
293100
293200
293300
Easting (m)
Number of samples per borehole
Boreholes
Blocks (ok)
Blocks (id)


West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-07
3.00
2.50
2.00
1.50
1.00
0.50
0.00
Au g/t
292700
292800
292900
293000
293100
293200
293300
293400
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-08
25.00
20.00
15.00
10.00
5.00
0.00
Au g/t
292800
292900
293000
293100
293200
293300
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)


West Goldboro - Easting Swath Plot - Composites x Interpolation
WG-09
Au g/t
30.00
25.00
20.00
15.00
10.00
5.00
0.00
292700
292800
292900
293000
293100
293200
293300
293400
Easting (m)
Boreholes
Blocks (ok)
Blocks (id)
Blocks (nnv)

ABOUT US

WSP is one of the world's leading professional services consulting firms. We are dedicated to our local communities and propelled by international brainpower. We are technical experts and strategic advisors including engineers, technicians, scientists, planners, surveyors and environmental specialists, as well as other design, program and construction management professionals. We design lasting solutions in the Buildings, Transportation, Infrastructure, Oil & Gas, Environment, Geomatics, Mining, Power and Industrial sectors as well as project delivery and strategic consulting services. With over 8,000 talented people across Canada and 49,000 people globally we engineer projects that will help societies grow for generations to come.

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