4.1.2 Regional Structure

The Bowen Basin has undergone NE-SW oriented, extensional and compressional geological events.

Structurally, the deposit lies on the western boundary of the deformed Nebo Synclinorium immediately west of a regional thrust fault system- the Burton Range Thrust. To the east (of the thrust) seams are repeated in the IPE deposit.

Further to the west is the structurally benign Collinsville Shelf.

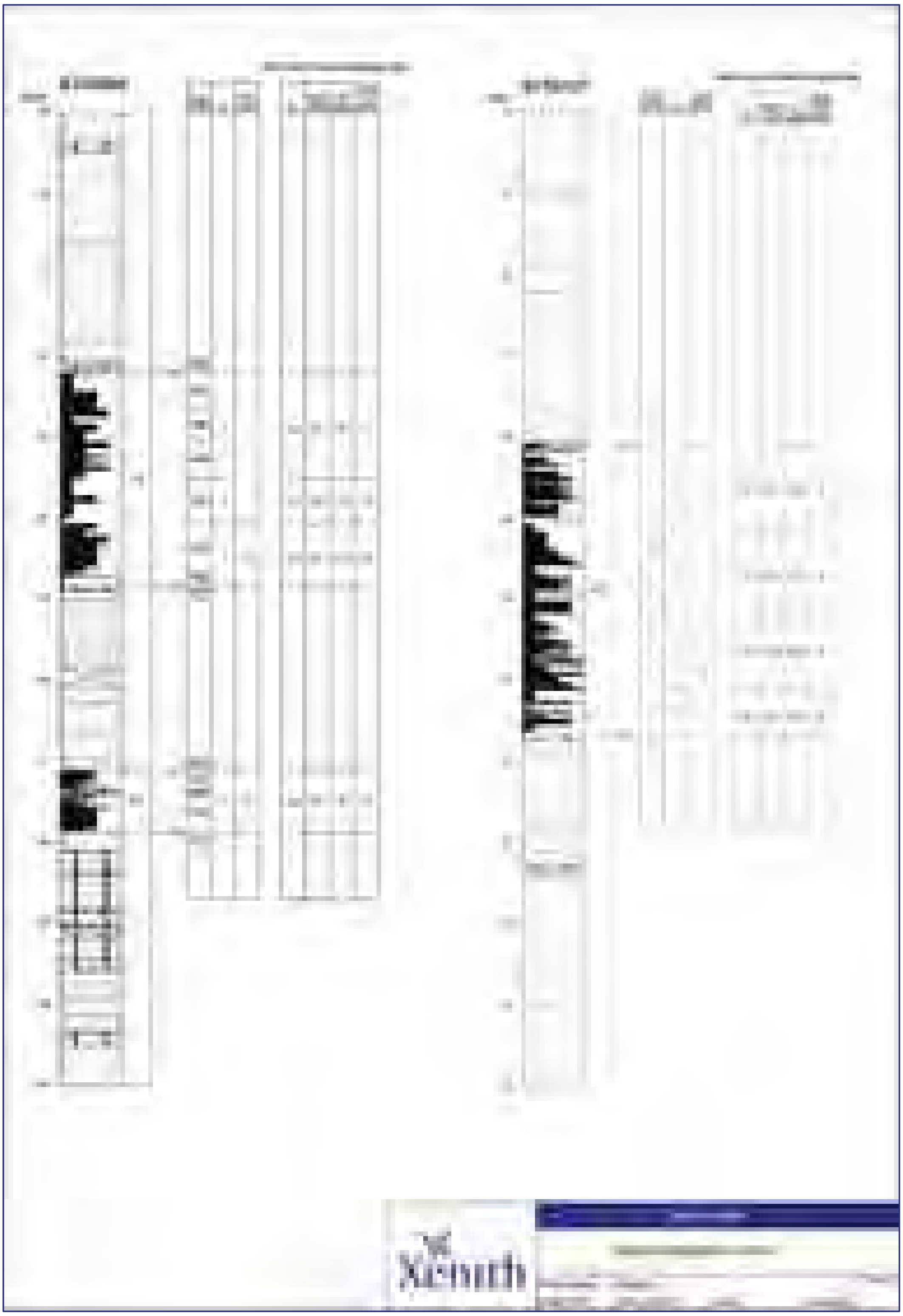
The economic coal seams are contained in the Late Permian Rangal Coal Measures which is an approximately 100m thick regional geological formation. The Rangal Coal Measures are underlain by the Fort Cooper Coal Measures and overlain by the Late Permian to Early Triassic Rewan Group.

4.1.3 Regional Stratigraphy

The Rangal Coal Measures comprise light grey, cross bedded, fine to medium grained labile sandstones, grey siltstones, mudstones and coal seams. Cemented sections are common in the sandstones. The transition from the Rangal Coal Measures to the Rewan Formation is generally difficult to define and is often based on the change from the green-grey colour of the Rewan sandstones to the blue-grey colour of the Rangal sandstones. The transition between the formations is 15 to 60m above the first major seam in the Rangal Coal Measures, the Leichhardt Seam.

The Fort Cooper Coal Measures comprise typically tuffaceous sandstones, siltstones, mudstones and coal seams. The transition between the Rangal Coal Measures and the Fort Cooper Coal Measures is generally clearly marked by the Yarrabee Tuff - a basin-wide marker bed comprised of weak, brown tuffaceous claystone. A regional stratigraphic column is provided as Figure 4.1, previous page.

Figure 4.2 – Local Stratigraphy of the Isaac Plains Deposit



4.2 Local Geology

4.2.1 Stratigraphy

There are no significant Tertiary or Quaternary sediments in the mine area. Soil and sub soil derived from Permian sediments are 2-5m thick. A small amount of Quaternary alluvium exists in Smoky Creek area in the centre of the deposit. Refer to Figure 4.1 for a stratigraphy column of the area.

The Leichhardt (LHD) and, when split, the Leichhardt Upper (LHU) seams form the principal economic coal resources in the Isaac Plains mine area. The LHU seam exists to the north of where the principal LHD seam splits into the LHU and the thinner LHL (Lower). Below the Leichhardt seam(s) there is a coalescence of the Vermont and Girrah seams in the north of the mine area, and separate Vermont and Girrah seams for the southern section of the mine area.

The Leichhardt Seam (LHD) is typically 3.5m thick and splits in the north to form the Leichhardt Upper and Leichhardt Lower (LHL) seams. The Leichhardt Upper seam is typically 2.3m thick. The coalesced LHD seam can include some stone bands that are consistent over relatively short distances. Where possible these stone bands have been identified from the geophysical density log and included into the dataset for consistency. Close to the LHD/LHL split line the presence of the “split” stone band in the LHD seam is consistent up to 500 metres from the split line. Typical seam brightness profiles are presented in Figure 4.2.

Faulting has been observed to cause both positive and negative localised variation in seam thickness.

The Lower Leichhardt Seam (LHL) splits off the Leichhardt Seam floor in the north. The seam is typically 0.7 to 0.8m thick. The LHL occurs only in a small zone in the north of IPC and exhibits some thickness variability due to both sedimentological and faulting reasons.

Two thin coal occurrences have been noted between the LHD and Vermont Seams. These have been named the L2 and L3 seams. The seams are typically not thick enough (<~1m) to consider for underground resource but are noted occasionally in the down hole geophysics as points of reference.

The Vermont (VER) seam lies approximately 25-30 m below the Leichhardt and varies in thickness between 5-7m and is of poor quality, exhibiting high raw ash values. The Vermont seam can coalesce (northern area) with the Girrah seam of the Fort Cooper Coal Measures to form a 20m thick stony coal seam. Regionally the Girrah seam is typically high ash with plentiful tuffaceous bands and due to the high inherent ash the seam generally does not wash well.

4.2.2 Structural Framework

4.2.2.1 Faults

Immediately east of the deposit is the major Burton Range thrust system which delimits the down dip extent of the deposit. The area has suffered significant deformation with east over west thrust faulting and orthogonal tear faulting. A major NS oriented thrust splits the deposit in the north. The maximum throw of this fault is in the order of 60m. The throw decreases considerably to the south.

Normal faults also occur. A NNE trending normal fault cuts across the middle of the mine. The fault has throws up to 30m which appears to diminish significantly to the north.

2D Mini-Sosie seismic surveys have provided clear evidence of the degree and complexity of faulting across the mine area. As discussed above, they have been used to interpret the “major” fault zones that can have an impact on resources estimates, and identify the zones of “lesser” faulting that have an impact on potential mining operations.

In 2017, a 3D seismic survey has been carried out over most of the underground area (Figure 5.2). The interpretation of the seismic data delineated a block of repeat seam caused by a scissor thrust fault in the south-east of the underground area. The north-south trending fault is sub-parallel to the Burton Fault. The seismic interpretation shows a displacement of up to 100m. The area where the seismic survey interpreted the upper repeat is shown in Figure 5.2. The seismic survey was not able to interpret the repeat further south due to poor data quality, probably as the seam comes close to the weathering horizon.

4.2.2.2 Base of Weathering

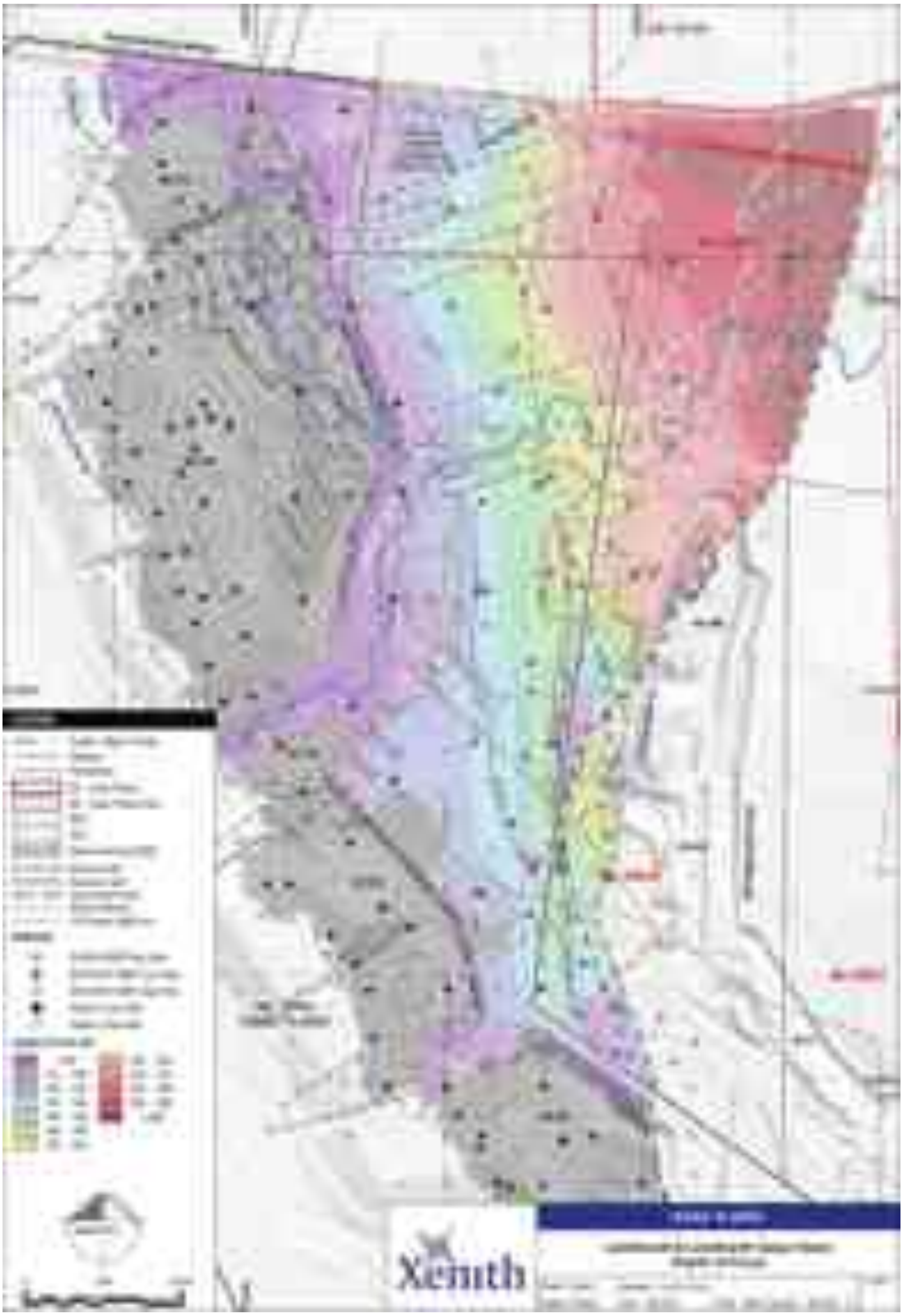
Depth of weathering over the whole deposit ranges from 7 to 47.8m averaging 20m. In the seam subcrop zone the depth of weathering averages 16.9m. Deeper weathering zones are generally related to local faulting.

4.2.2.3 Overburden Material

Non-coal Permian sediments consist of moderately weak to strong sandstones, siltstones and minor mudstones. Sandstones and siltstones appear to be co-dominant. It is likely that weathering does not overtly diminish the strength of the Permian rocks. The strength characteristics of the non-coal units was summarised in a Geotechnical evaluation completed prior to mining commencement and was summarised in previous resource reports.

Figure 4.3 shows depth of cover to the primary LHD or LHU seam. Where fault repeats occur, the depth to the upper repeat is shown.

Figure 4.3 – Depth of Cover – Leichhardt Seam



5 EXPLORATION AND DRILLING

5.1 Drilling History

The complete discussion of historic exploration activities prior to the sale of Isaac Plains in 2015 to Stanmore Coal, is included in the report *"Isaac Plains Mine (ML 70342) - Resource Statement as at December 2009,* JB Mining, January 2010.

Prior to Aquila being granted the current EPC 755 to the north, the area had been explored in a number of phases of activity as summarised below:

- The earliest recorded exploration in the area was carried out by the Utah Development Company Pty Ltd in the 1960's. Although coal was intersected at shallow depth, it was not investigated further. Some six (6) holes were located in the Isaac Plains mine area, but "missed" the Leichhardt seam due to maximum drill depth of 60 metres and spacing between holes.
- Thiess Peabody Mitsui Pty Ltd conducted drill traverses in the area from the mid-1960's into the 1970's.
- Queensland Mines Department in the 1970's drilled some regional exploration holes in the south of EPC755, outside the Isaac Plains mine area.
- Iscor Australia Pty Ltd as the holder of EPC602 drilled six holes in the southern part of the area, all of which targeted the deeper Moranbah Measures. The potential of the Rangal and Fort Cooper Measures was not investigated although coal was intersected at very shallow depths in one of these holes. These holes were south of the Isaac Plains mine area.
- MGC Resources Australia Pty Ltd conducted 2D dynamite seismic surveys across the area and followed this up with some gas/oil exploration holes. Moranbah 1S was completed in November 1993 to a depth of 636 metres. This hole is not within the Isaac Plains mine area but was spudded in the Fort Cooper Measures some 1.5 kilometres south west of the mine area.
- Nebo Coal Pty Ltd drilled some 16 holes in EPC667, in the north of the ML area in early 2002/2003. These intersected the LHD seam in the south (N1 pit) and the LHU/LHL seam split in the north of the area (N2 pit).

BCCM drilled some 35,754 metres in 559 holes to prove up the initial 48.8Mt resource within ML 70342. This work started in April 2004 and was completed by early April 2006 just prior to the commencement of mining in July 2006. Other work involved in the mine development program included (discussed following):

- Coal quality work on some 89 X 100mm cores, 7 X 63mm cores and 5 sites for 200mm cores (17 X 200mm cores holes),
- Line of oxidation (LOX) drilling was completed in 149 holes on drill line spaced approximately every 60 metres (north – south),
- Geotechnical work from 7 HQ fully cored holes,
- 18 km of 2D MiniSosie seismic survey in two phases,
- Ground magnetics over some 8km^2 to determine the likely intrusives in the area,
- Air photo interpretations,
- Structural interpretation from sectional analysis,
- Geostatistical analysis of the initial dataset,
- Geochemical review of the overburden to determine spoil waste nature,
- Soil chemistry and soil distribution to allow planning for rehabilitation, and

- Cultural heritage review of the mine site prior to mine activities.

From 2008 to September 2009, BCCM on behalf of IPCM drilled a further 19,206m in 278 holes for gas analysis, fault delineation and in-pit coal quality reasons. The majority of the 2008 / 2009 drilling was confined to the working open-cut areas which have since been mined.

Location and distribution of the drill holes in the database are shown Figure 5.1. The drilling series outlined previously is not entirely visible at this plan resolution; that is certain holes may be hidden by other holes in close proximity.

5.2 2015 / 16 Drilling

The 2015/16 drilling programme commenced at Stanmore's acquisition of Isaac Plains and was completed in March 2016.

A total of 76 holes were completed, 66 open holes and 10 cored holes. The open holes were mainly aimed at improving the understanding of reverse faulting in the northern part of the lease, in the area where open-cut mining operations is presently ongoing.

The cored holes were drilled in order to assess coal quality in areas of limited coverage and in turn upgrade the resource classification confidence. The eastern part of the ML was targeted, in order to confirm an indicated resource in an area of the mine with future underground potential. The core drilling included holes drilled to the east of ML 70342, within the IPE leases.

5.3 2016 / 17 Drilling

The 2016 / 2017 exploration activities predominately comprised of drilling with a total of 19 holes completed, being 14 open holes and 5 cored holes. The open holes have again targeted the areas around identified reverse faulting ahead of the present open-cut operations.

The cored holes were drilled in order to assess coal quality and geotechnical conditions in areas ahead of the working mining face.

5.4 2017 Underground Drilling Program and Opencut Drill holes

An underground drilling program of 19 drill holes was completed in the second half of 2017. 13 drill holes were drilled on ML 70342, 5 on MDL135 and 1 on MDL137. 2 holes were re-drills as the first attempts showed poor core recovery due to faulting.

The drill holes were all partially cored and drilled for coal quality, gas, permeability and geotechnical purposes.

14 chip holes were drilled ahead of the existing N1S pit operations.

5.5 2018/2019 IPU Drilling Program

A program of 23 drill holes was completed in May-June 2018 and April 2019 to provide additional information for the Isaac Plains Underground (IPU) BFS mine plan design. The drill holes were targeting seam limits adjacent to the major Isaac Thrust and smaller structures identified from the 3D seismic data that were likely to constrain underground conveyor designs.

All the drill holes were chip holes. All the holes were geophysically logged except for three holes close to the Isaac Thrust that didn't intersect the LHD. These holes were unable to be cleaned out due to persistent blockages in the weathered tuffs of the overthrust FCC.

All drill holes to date can be seen below (Figure 5.1).

Figure 5.1 – Drill Hole Locations

5.6 Geotechnical Investigations

Historically seven fully cored (diamond) geotechnical holes were drilled as part of the 2004 to 2006 exploration programs, to assess the overburden, coal, and floor sediments for relevant geotechnical parameters and issues, which would impact and inform open-cut mining activities at Isaac Plains.

Since Stanmore's involvement at IPC commenced, the geotechnical database has been further enhanced with thirty (30) of the HQ size partially cored (diamond) holes, drilled as part of the exploration campaigns undertaken between 2015 and 2017, having been subjected to geotechnical sampling and testing.

All core holes completed by Stanmore are geo-technically logged according to CoalLog Standard.

A standard geotechnical testing procedure was designed for Stanmore by Geotechnical Engineer Rob Thomas, of Golder Associates, with the view of gathering information on the overburden and floor sediment material for rock strength testing and assessment of geotechnical issues associated with mining activities both in the present open-cut and the possible future underground areas.

All samples taken and analysed for laboratory testing had to meet certain criteria which were:

- All samples were taken within 10m of the roof and 6m of the floor from the target LHD seam.
- Samples had to be > 200mm in length, with recording of fracture frequency and RQD and general geological observations noted.
- Sampled core to be photographed both at site and prior to testing.

Laboratory testing of selected core sampled for drill holes within the IPC was carried out by Cardno, Ullman & Nolan Geotechnic Pty Ltd, Mackay (NATA Accreditation #910).

All samples were wrapped in foil, plastic shrink wrap and bagged prior to courier despatch to the laboratory.

Presently the Stanmore dataset consists of laboratory tests conducted on selected samples as follows:

- 202 Unconfined Compressive Strength (UCS) sample tests; including modulus, moisture density and Poisson's Ratio
- 44 Lab Sonic Velocity sample tests
- 32 slake durability sample measurements
- 1 three-stage tri-axial test on coal core

Additionally, 3779 field point load tests (diametral and axial) have been conducted and recorded on selected roof and floor samples and these also form part of the Stanmore IPC geotechnical dataset.

Results of this sampling and work has been retained for use and help to inform ongoing and future mine planning activities but are not included as part of this report.

5.7 Seismic Surveys

5.7.1 2D Seismic Lines

Historically 2D Mini-Sosie surveys have been undertaken, primarily to inform the nature of the faulting within the lease area. An initial 8.7 km survey was completed by Velseis Pty Ltd in March/April 2004 on three (3) lines to determine the ability to successfully use the technique in the area to determine faulting. Following the successful use in the initial survey and the need to get more seismic coverage a further 9.3 km survey was completed by Velseis Pty Ltd in July/August 2005. The second survey covered 3 additional lines and extended one of the original lines (Figure 5.2).

The seismic data in conjunction with 250m grid spaced drill hole data provided the key to interpretation of the major faults within the mine area. The major faults have been modelled and used to delineate the resource domain areas. Minor faulting (where possible) has also been identified. This minor faulting was considered not to impact resource estimates but will have some operational issues during mining.

In 2015 / 2016, Stanmore commissioned Velseis to undertake a Mini-Sosie survey to complement the historic seismic data. The work included 15 lines within the IPC and IPE leases. During this work the 2004 and 2005 data was reprocessed, utilising any new drill holes that coincided with these lines.

The additional 15 lines covered approximately 32 km encompassing both the Isaac Plains and Isaac Plains East deposits. Of the additional 32km of 2D coverage, 10 km occurred within ML 70342 and a further 7.7 km extended into the western limit of the Isaac Plains East resource, allowing for the interpretation of the eastern extension to the Isaac Plains deposit.

Interpretations were supplied for the purpose of establishing fault locations and seam structure and thickness continuity between the drilling data points and historic seismic interpretation. The major faults identified in the seismic interpretation have been truthed against the modelled faulting and used for validation. Minor faulting (where possible) has also been identified. This minor faulting was considered not to impact resource

The seismic interpretation was reviewed and utilised to assist with the model and resource update. Figure 5.2 shows locations of relevant historic and recent 2D seismic lines.

5.7.2 3D Seismic Survey

A 3D seismic survey (Vibroseis) was undertaken in 2017. The area covered was approximately 6km2 and is shown in Figure 5.2. Acquisition and processing/interpretation were carried out by Velseis. Fault delineations (location and throw) and LHD roof elevation grids were generated from the seismic data. The elevation grids were calibrated against existing borehole data, including the 2017 drilling.

The 3D seismic interpretation delineated the major faults of the underground area and revealed a – previously unknown - repeat fault in the south-east of the underground area. The southern extent of this repeat block could not be defined as the seismic data were of too poor quality for an interpretation.

Figure 5.2 – Seismic Survey and Drill Hole Locations

5.8 Magnetic Surveys

Following a review of the publicly available Department of Natural Resources aero-magnetic data, and the intersection of some minor intrusive material in five (5) drill holes in the mine area, a small ground magnetic survey was undertaken initially in two zones within the mine area. The northerly survey was on a zone surrounding the four holes that had intersected basaltic material at depth. The southern survey was over a small intrusive pod surrounding hole 043R.

An initial magnetic survey was conducted by Resolve Geological in October 2004 using a Geometrix G858 Cesium based Magnetometer, an Omistar differential GPS unit and a Processor Magnetometer as a Base station. The survey was over 38-line kms (19 line km per area) with 20 metre line spacing.

The magnetics data was later processed and reviewed by Geological Resources & Services Pty Ltd (GRS). GRS concluded that there is a sill (of basaltic nature) probably limited by fault structures in Area A (northern area), and that in the southern area (One Tree Hill – south of S2 pit) there is evidence of a small vertical plug. Later drill campaigns better defined the small plug structure in S2 pit. These reports were presented in previous resource statements.

Following the GRS review a more thorough ground magnetic survey was undertaken by UltraMag Geophysics in 2006 over the intrusive plug area in S2 pit. A further short magnetic survey was undertaken in the subcrop zone of the N1 pit (in January 2006) to see if the technique identified any other unknown intrusives in the initial mining area. A small (<2-5m) dyke was identified in this zone from this survey. This was later identified from inpit mapping and photography to be real but of limited extent. The Ultramag data has been the better of the magnetic datasets and was interpreted thoroughly for lineaments and possible faults.

The Ultramag interpretation of the intrusive plug has turned out to be quite "accurate" for identifying small scale faults (S3 pit, approximately 2m fault identified) and dykes/sills emanating from the intrusive source (S2 pit).

5.9 Geophysical Logging

For the initial mine development work program, all holes were geophysically logged soon after drilling, using slim-line logging equipment. All downhole geophysical data was captured electronically and stored in LAS and PDF format. Three (3) holes were not logged due to hole collapse or difficult drilling conditions. All holes were corrected to geophysical depths for top and base of coal from the BRD log. The suite of tools used was Natural Gamma, Density (LSD, HRD, BRD), calliper, verticality, and sonic (m per second). In some holes a resistivity log was also run if there was a "suspicion" of any intrusive rock types. Sonic logs were only able to be run to the standing water level in the drill holes. For LOX holes only gamma, density and calliper logs were run as they were quite shallow (usually less than 30m) and generally above the water table.

For the Stanmore exploration drilling campaigns (2015 onwards), all holes were geophysically logged soon after drilling using slim-line logging equipment. All downhole geophysical data was captured electronically and stored in at least LAS and PDF formats. All holes were logged with the exception of one fault delineation hole where mining activities prevented access to the site. All Stanmore drilled holes, including the 2018/2019 holes were corrected to geophysical depths for top and base of coal from the coal detail and lithology logs. The minimum suite of tools used was Natural Gamma (G), long and short spaced density (D), calliper (C), and verticality (V). Selected holes also had a combination Full Waveform Sonic, Acoustic Scanner or Optical Scanner tools run.

Geophysical logging was undertaken by Weatherford who conduct regular audits on their equipment in accordance with Australian Standards.

6 SAMPLING AND ANALYSIS

No new coal quality holes were drilled since the previous resource report (2018). Sampling and analysis of earlier drill holes is given below.

6.1 Historic Drilling

6.1.1 Sampling Procedure

The complete details of the historic coal quality sampling procedure are outlined in the 2010 resource report, *Isaac Plains Mine (ML 70342) - Resource Statement as at December 2009,* JB Mining, January 2010 (Chapter 6.1.1).

The main details of this chapter are outlined below:

Any cored sections in coal where there was not greater than 95% core recovery were re-drilled. Coal cores were sampled into plastic bags and sent to Casco laboratory in Mackay (within days of completing the drilling) for the test program. Casco Australia Pty Ltd is a NATA registered and a well-recognized coal analytical organization which has been in business for close to 50 years in Australia. Casco was taken over by SGS in 2007, but still retains its NATA registration. Samples were stored in a freezer until instructions were available to conduct the analytical program.

In the historic BCCM drilling (100mm core and 63mm core), coal sampling was determined by the brightness profile and then sampled to "relevant" plies up to 25 to 50 cm thick. These samples were subsequently composited to four (4) major ply divisions. Ply coal analysis was initially sized by conventional size fractions (+32mm sized and then passed 32mm, -32mm, -16mm, -8mm, -4mm, - 2mm, -0.125mm), then analysed for raw analyses, and a "quick" F1.50 density separation (haematite wash). Subsequently each size fraction in each ply was detailed analysed by various float sink densities (F1.30, F1.35, F1.40, F1.45, F1.50, F1.60, F1.70, F1.80, F2.00, F2.20, S2.20). From that data there were clean coal composite samples compiled and analysed for a Coking and "basic" Thermal coal composite. Where possible a Haematite composite was also analysed for coking properties to compare to the coking composite data. Coal petrography was then undertaken on the coking and haematite composites by ply. Plant simulation work was undertaken on these holes to focus the second phase of coal quality work, as detailed below.

The second phase of 100mm core assay work consisted of; compositing into 4 plies, raw Ultimate coal analysis by ply, quick float F1.375 by ply, and sizing of all ply data (-32mm, -16mm, -8mm, - 4mm, -2mm). Then detailed float sink analysis (F1.30, F1.35, F1.40, F1.45, F1.50, F1.60, F1.70, F1.80, F2.00, F2.20, S2.20) was completed on -16mm to 2mm, -2mm to -0.25mm, and -0.25mm. From these fractions clean coal composites were derived and analysed for the "typical" product types; i.e., Coking, PCI and Thermal.

The various product types were identified for each hole (from the float sink dataset) and clean coal composite samples were derived and assayed for the various representative properties.

For the Vermont seam samples, coal samples were composited, where possible, into sub-plies (V1, V2, V3) and then analysed for raw and a quick 1.375 float sample. Given the poor results from the F1.375 data there were no clean coal composites analysis work undertaken.

A series of 200mm cores were drilled from late 2004 through to early 2006. The initial three (3) sites were in N1 pit (one shallow and then one down dip) and then S3 pit in the south of the mining area. After the coking test data was reviewed an additional two sites close to the early mining areas were drilled to take a bulk sample for detailed sizing, washability and then coke oven test work (from the bottoms ply coking product).

6.1.2 LOX Drilling and Analyses

LOX holes were drilled by a reverse circulation hammer drill rig in order to obtain a "relatively" uncontaminated chip sample of coal for subsequent analysis. In LOX holes, 0.5 m samples were taken in coal. These were bagged on site and sent to CCI Australia laboratory in Moranbah for quick analysis. The samples were analysed for raw quality on ash%, CSN, Volatile Matter%, and Inherent Moisture% and some had SE. There was no float/sink work done on these samples as this was considered unnecessary given the raw analysis program. From the coal analysis, the physical log of the cuttings in the field, and the density log, the boundary of weathered and fresh coal was determined in all LOX holes. CCI was taken over by Bureau Veritas in late 2007 or early 2008.

The LOX program has defined the full seam fresh line at approximately 62m line spacing intervals between lines. Along LOX drill lines holes were spaced approximately 5-10 metres apart. All LOX drilling was used as part of the structural interpretation.

6.2 Stanmore Exploration Programs Sampling and Analysis

6.2.1 Drilling Programs

All Structural and Coal Quality drilling works undertaken in ML 70342 by Stanmore Coal have been conducted by Wizard Drilling of Bundaberg.

Works were conducted either under the supervision of IMC Mining Pty Ltd, operating on behalf of Stanmore, or directly by Stanmore itself.

Three phases of drilling have been undertaken as follows:

An initial late 2015 / early 2016 exploration campaign which was spread between near mine and future underground areas of the Mining Lease, which consisted of:

- 9 x part-cored holes of HQ3 size (61mm diameter) for coal quality, geotechnical and gas content testing.
- 1 x part core 4C (100mm diameter) hole for coal quality
- 66 x Chip holes for structural interpretations (thickness, continuity and fault delineation) using either poly-crystalline diamond or blade bits. Hole size varied from a minimum of 99mm to a maximum of 229mm, depending on the and diameter of bit used and purpose of the drilling.

A second smaller exploration campaign, conducted in August-September 2016, carried out in the area proximal to the operational N1N and N1S pits, which consisted of:

- 5 x part core 4C (100mm diameter) holes for coal quality

- 14 x 99mm diameter chip holes for structural interpretations (thickness, continuity and fault delineation) using either poly-crystalline diamond bits.

A third exploration campaign was undertaken from September to December 2017, post completion of the August 2017 resource report, it consisted of:

- A total of 19 part-cored holes of HQ3 size (inclusive of 2 re-drills) for purposes which included coal quality, geotechnical and gas content, permeability and spontaneous combustion assessments, primarily in areas of potential future underground development.
- 14 chip holes ahead of operational pit N1S with the purpose of informing short-term control of seam structure in the present open-cut mining area.

In all cases, drill holes were extended at least 6m below the base of the last intercepted coal seam to allow for geophysical logging of the entire seam.

All core was photographed in 0.5m intervals against a blackboard with depth markings, lithology and sample numbers added. Chips were laid out on bare ground in lines of 30, one metre samples further subdivided into 6m runs. Chips were photographed in 6m runs with a whiteboard showing hole number, date and depth range. In all photographs, depth increases from left to right.

6.2.2 Sampling Procedure

General details sampling procedures undertaken for all Stanmore drilling programs are outlined below:

- All cored intervals were sampled where coal was present at thickness of 0.1m or more, with a maximum sample thickness of 0.5 m. Coal plies were sampled discretely on the basis of lithological characteristics and quality. All non-coal material and partings less than 0.1 m were included with the coal ply and noted in the lithological description. Non-coal interburden material greater than 0.1 m and up to a maximum of 0.3 m was sampled separately. Approximately 0.2m to 0.3 m of immediate roof and floor were also collected as dilution samples.
- Geotechnical samples were collected from roof (up to 10 m above seam) and floor sections (up to 6 m below seam). Selected samples were analysed, with testing including UCS, Young's Modulus, Poisson's Ratio, Slake Durability and Tri-axial testing.
- All remaining un-cored material has been retained in marked core boxes for future reference.
- All coal quality samples were double plastic bagged at site and marked with sample number, hole and project. The samples were then kept in cold storage on site before dispatch to the laboratory via a tracked freight service. Chain of custody and sample documentation was sent to the laboratory by email ahead of the samples. Coal was held in cold storage on site for periods of no more than two weeks prior to dispatch. Geophysical corrections were undertaken as soon as practicable following sample collection and these were used to confirm representative core recovery.
- Any cored sections in coal with less than 95% core recovery, were reviewed by the CP to assess whether the possibility of a re-drilling was required. The assessment involved a review of the core photography, down hole geophysics and lithology logged and also historic knowledge of similar stratigraphy from the neighbouring leases.
- Coal quality samples were sent to Bureau Veritas Laboratories in Brendale, Queensland. Bureau Veritas Minerals Pty Ltd ("BV") is a NATA registered and a well-recognized coal analytical organization conducting coal quality sampling for many years. Bureau Veritas are accredited for compliance with ISOMEC 17025, corporate accreditation number 1805. Site accreditation number 18415.

- Samples are stored in cold storage at BV until instructions are available to conduct the analytical program.

6.2.3 Coal Analysis

- From all cored holes, coal was ply sampled in the field by the field geologist, according to lithology and brightness profiles. Coal samples typically were between, 20 to 40 cm in length. These ply samples were initially tested by BV for Apparent Relative Density (ARD), which is a non – destructive water immersion density test. The results were provided and analysed prior to creation of float – sink (wash) composite sections.
- Wash composites were created per each LHD seam intersection, consisting of either:
  - A single full-seam section, being the total intersected coal thickness at a core hole location, with composited full-seam thickness for the LHD seam ranging from 2.85 to 4.01m *or*
  - Two composites per seam being:
    - a Top of seam composite (approx. 2.0m to 2.3m thickness)
    - a Bottom of seam (remainder of seam, generally 1.3m to 1.8m thickness)
- The decision to create either a one or two composites was based on several factors, primary among which were the core holes' physical location and seam thickness.
- To simulate mine transport conditions each composite sample was then drop shattered 20 times from a height of 2 metres, any sample mass remaining of > 50 mm was hand knapped to 50 mm, dry tumbled and dry sized at 31.5 mm, 25 mm, 16 mm, 8 mm, 4 mm and 2 mm.

Composite samples were then split and further analysed as follows:

- 1/8 for quick coke: Crush to 11.2mm, float sink at 1.425 density, crush to 4mm and mill sample to test for Proximate, CSN, Gieseler & Dilatation
- 1/8 for raw analysis: Crush to 4mm, mill sample to test for RD, MHC, Proximate, TS, CSN, Calorific Value & Chlorine
- ¾ for float sink: Wet tumble and wet size at 31.5, 25, 16, 8, 4, 2, 1, 0.5, 0.25, 01.25 & 0.063mm. Re-combine samples in following fractions: -50+16mm, -16+8mm, -8+2mm and -2+0.25mm. Float sink each size fraction at densities (F1.30, F1.35, F1.375, F1.40, F1.45, F1.50, F1.55, F1.60, F1.70, F1.80, F2.00). -0.25+0mm fraction subject to tree froth flotation. All fractions analysed for ash and CSN.
- Washability simulations are performed on the float sink results of individual wash composites, which target following coal products:
  - Primary SSCC Product of 9.5% ash
  - Secondary Export Thermal Product of 16.0% ash
- Clean coal composite samples are then compiled, for each equivalent wash composite, and analysed for: Primary Coking (-16+0mm), Coarse Coking (-50+16mm) and Secondary Thermal Coal Composites.
- Clean coal composite samples and assayed for the various representative quality properties.

The detailed dataset of coal quality analysis is retained and maintained by Stanmore Coal and its independent coal quality consultants on an ongoing basis.

6.2.4 Dilution

- Eleven (11) dilution composite samples (roof and floor) were tested in a similar manner to the coal up to and including wet tumbling and sizing.

Wet tumbled and fully sized dilution sample composite results are then able to be included in wash simulations at a level of dilution appropriate with the employed mining practices. For the current open-cut yield simulations 0.10m of roof and 0.10m of floor was included in washability simulations.

7 DATA VERIFICATION

7.1 Database Integrity

Historical data was reviewed by Xenith prior to completing the 2015 resource estimate. The database used for the review of the supplied geological model was compiled from outputs of historic models that were located in the data room. As part of Xenith standard practice, a review of the geological model was conducted in comparison to the supplied drilling database to test its status as “fit for purpose” for conducting a resource estimate.

The data from the Stanmore exploration programs were supplied progressively as results became available. The data is stored on Xenith servers, not in a formal geological database, but as individual files organised in folders per borehole.

7.2 Data Spacing and Distribution

There is a consistent coverage of drilling within the ML area. In general structure holes occur at a regular 250m x 250m grid spacing to assist with determining the stratigraphic continuity the main Leichhardt Seams. Cored holes were generally located at sites of 500m x 500m spacing, in some areas they exceed this 500m spacing but are on average less than 600m apart.

The 2013 drilling was designed and conducted around the interpreted major fault locations to assist with the “firming up” of the fault interpretation.

The 2015/16 program has achieved improved data coverage down-dip. This has decreased the core hole spacing and extended the range of structural data points to the East, particularly in the northern part of IPC.

The 2016 / 2017 program and the second half 2017 underground and open-cut drilling programs has added to the knowledge of the conditions and seam structure ahead of the advancing mining face and for the proposed underground mine.

The 2018/2019 drilling program further improved understanding of the faulting in the underground area. Particularly concerning has been the upper repeat of the LHD seam that was proven to be covering a smaller area than initially believed from only interpreting the 3D seismic survey results.

Considering the continuity of the target seam in the deposit, the drill spacing generally has proven to be sufficient to give adequate control to the model and give the required confidence in the geological interpretation.

8 GEOLOGICAL MODELLING

8.1 Software

Both the structure and coal quality models have been constructed using the Minescape Software Suite, version 5.12.

8.1.1 Lithological and Structural Data

Historically data was captured in the field and then entered into the LogCheck software. Various checks were reported to have been undertaken prior to loading into the modelling system.

The lithological database, on which the historic modelling is based, was reported in 2010 Resource Estimate to contain industry standard data as well as seam names and working section flags. All interpretation of seam thickness from the supplied holes was based on geophysical interpretation.

Details of the historic data compilation and validation are included in Xenith's "Isaac Plains Coal Mine 2015 JORC Resource Estimate", report

Data captured from Stanmore exploration campaigns in the field are entered into LogCheck for validation and then corrected for geophysics prior to modelling.

The current geological model contains all exploration data completed within IPC up to May 2020. There are a total of 1115 drill holes (285 excluding mined out areas and IPE holes) including:

- 20 - 2018/19 chip holes in the IPU area. 3 of these were blocked, but the rest are validated with geophysics.
- 14 - 2017 chip holes ahead of N1S pit validated with geophysics
- 5 – 2017 blastholes ahead of N1S pit validated with geophysics
- 17 – 2017 cored holes in the underground area validated with geophysics and core photography
- 4 - 2016 / 2017 cored holes validated with geophysics and core photography
- 10 - 2015 / 2016 cored holes validated with geophysics and core photography (2 from the IPE program).
- 14 - 2016 / 2017 structural chip holes validated with geophysics
- 61 - 2015 / 2016 structural chip holes validated with geophysics
- 887 - Historic core holes (only verified coal quality data used).
- 674- Historic structural chip holes.
- 209 holes from IPE

The Leichhardt seams and the Base of Weathering are well represented in the available data, see Table 8.1.

Table 8.1 – Intersects Per Stratigraphic Unit

Stratigraphy	No. Intersects
Base of Tertiary (BUTE)	545
Base of Weathering (BW)	994
Leichhardt Seam (LHD)	781
Upper Leichhardt Seam (LHU)	221
Lower Leichhardt Seam (LHL)	218
Vermont Seam (V1)	175

Interpretation of seam thickness from the Stanmore supplied holes was based on geophysical interpretation.

8.1.2 Coal Quality Data

The historic coal quality database was exported from the Vulcan format in the form of Microsoft Excel spread sheets and uploaded into the Minescape software. Details for the construction and validation of the historic coal quality data are located in the "Isaac Plains Coal Mine 2015 JORC Resource Estimate", report.

The coal quality database has since been updated with the analytical results of the Stanmore drilling programs. The raw coal analysis results from the 2017 underground exploration program were included in the coal quality model.

No new coal quality data was acquired since the previous resource estimate. Minor changes were made to the coal quality model including:

- 2 LHL intersections with a large inclusion of stone were excluded. This resulted in lower, more representative ash values for the LHL.
- Renaming of one seam to LHD from LHU+LHL. This resulted in a reclassification of measured resource in that area.

8.2 Topography

The topographic surface has been generated from LiDAR, which was flown by Aerometrex, February 2020. Vertical Accuracy: +/- 0.15m.

The 2015/16 drill holes were surveyed by MSS and JTH Surveys, Moranbah, using site base station (RTCM0000) and Trimble R10 GPS. Previous drilling was surveyed by Shield Surveying Pty Ltd (Mackay) and Mackay Surveys Pty Ltd. The datum used is AGD84 and the projection used AMG 84 Z55.

All holes from the 2017/2018/2019 campaigns were professionally surveyed by MSS (Golding) surveyors that currently undertake all survey control at the Isaac Plains Mine Site. The origin of the survey was based on the calculated site base station coordinates and level of the site survey station from the AUSPOS static data listed below. All values are in AMG84 Zone55 coordinates as is the site base station RTCM0000 coordinates.

The supplied drilling locations were validated against the Pre-mining CCS AMG84 Z55 survey. It was noted that all but 18 drill locations had a difference in RL of less than 1m.

8.3 Base of Weathering

Base of weathering was modelled to the visual base of weathering and to LOX hole analytical results where available. The Base of Weathering has been modelled as a non-conformable continuous surface in Minescape.

8.4 Modelling Technique

8.4.1 General

Modelling was completed using Minescape software version 5.12.1.325, using the Stratmodel module. The modelling parameters for the resource estimate are all contained and controlled within the Minescape Software Schema, named "IPC_0620". The details of this schema are outlined in Table 8.2.

The faulting regime was updated to more accurately reflect the latest information derived from recent mining activity, the latest drilling information and the fault interpretation from the 2017 3D seismic survey.

Where seams did not occur within the drill hole or location was not controlled by the constraint file the modelled seam was pinched out. The model grid size is 20 x 20 metres to allow for the necessary detail around faults. The interpolation method used to construct all seams was the Finite Element Method (FEM) with minimum interval thickness of 0.3m.

The modelled area was sufficient to cover the entire project area.

Table 8.2 – Modelling Parameters

Modelling Element	Name / Description
Schema	IP_0620
Thickness Interpolator	Finite element method (FEM)
Trend Interpolator	FEM
Surface Interpolator	FEM (First Order)
Minimum Interval thickness	0.3 m
Seams Modelled	Leichhardt Upper (LHU), Leichhardt Lower (LHL), Leichhardt (LHD), Lower Leichhardt seams (L2 and L3), Vermont Seams (V1, V2, V31, V32), and Girrah Seams(G1-G6)
Additional Surfaces Modelled	Base of Weathering (BW) and Base of Tertiary (TES)
Seam Relationship	Conformable
Seam Continuity	Pinch
Compound Seams	LHD (LHU and LHL), V3 (V31 and V32), V12 (V1 and V2) and Ver (V12 and V3)
Additional Survey	LHD_FLOOR, LHD_ROOF, LHU_FLOOR, LHU_ROOF, LHL_FLOOR, LHL_ROOF (In-pit survey points to control the model) and LHD_FL_DUM (dummy points interpreted).
Faults Modelled	58 faults, 22 Reverse, 36Normal.
Constraint File	Constr
Grid Spec	IP_IPE_EXT20
Grid Spacing	20 m
Grid Origin	614855.42m East, 7566372.52m North
Number of Row and Columns in Grid	364 Rows and 225 Columns
Grid Dimensions	7260 m length, 4480 m width
Quality Model	IP_RAW_T20
Raw Quality Table	raw_load_0620
Composite Quality Table	raw_comp_0620

Table 8.3 – Geological Model Spatial Extents

Minimum Easting (m)	Maximum Easting (m)	Minimum Northing (m)	Maximum Northing (m)	Grid Size (m)
614,855	619,325	7,566,372	7,573,622	20 x 20

8.4.2 Stratigraphic Model

1116 drill holes were utilised to construct the stratigraphic model. This includes 182 holes from IPE (open-cut) which are included to control the Burton Range Thrust fault. A total of 125 holes from the database were not included. These holes were either too close to a preferred neighbouring hole (e.g. pilot holes), too shallow to intersect coal or deemed unreliable based on unverifiable depths or thicknesses.

A total of 58 faults were included in the model, based on drill hole intersections, seismic interpretations or inferred from abnormal changes in surface contours. In particular, the 2017 3D seismic survey provided more detail on faulting in the underground area.

The Leichhardt seam splits into an upper and a lower ply in the northern part of the deposit, and the model allows for this distinction. The elements (plies) are called LHU and LHL, while the compound (seam) is named LHD. The ply separation distance is 0.3m, which means that the LHD will be modelled until the parting between the LHU and LHD becomes more than 0.3m. At this point the LHD ceases to exist in the model and only the plies exist.

8.4.3 Coal Quality Modelling

The raw coal quality model from 2018 was reviewed and an updated model was created which included some minor changes/improvements compared to the previous model.

A coal quality load file was created and loaded into Minescape using the “Import quality, dh load – user defined” form. This module matched the sample intervals with seam intervals. The import table was checked for mismatches between sample and seams by checking the “waste” column for values between 0 and 1. Any mismatches were checked and adjusted if needed.

The import table with all the samples were then composited by using the “Composite, Drillhole” form. Key qualities were contoured and checked for inconsistencies and the raw coal quality model was updated.

8.5 Model Results

8.5.1 Structure

The IPC resource area is faulted and reasonably structurally complex. It is located to the west of the regional Burton Range thrust fault whose throw is in the order of 200m.

The resource area shows predominantly north-south trending reverse faulting and east-west trending normal faulting. High density, complex faulting can be found in the IPC area. The frequency of faults appears to reduce down-dip in the underground area. Fault displacement is generally from a few metres up to 20m or 30m.

The 2017 3D seismic survey interpretation shows a scissor thrust fault in the south-east of the underground area with a displacement of up to 100m. The fault caused a block of repeat Leichhardt seam (Figure 5.2). The definition of this scissor fault was further improved with the 2018 /19 drilling campaign. The Upthrown area was proven to be smaller than what was initially thought, and the intersection of the two major faults causes the uplifted area to pinch out towards the south of the lease.

The general dip of the deposit is to the east at between 4 to 14 degrees (Figure 8.1 and Figure 8.2).

Figure 8.1 – Leichhardt Seam Structure Floor

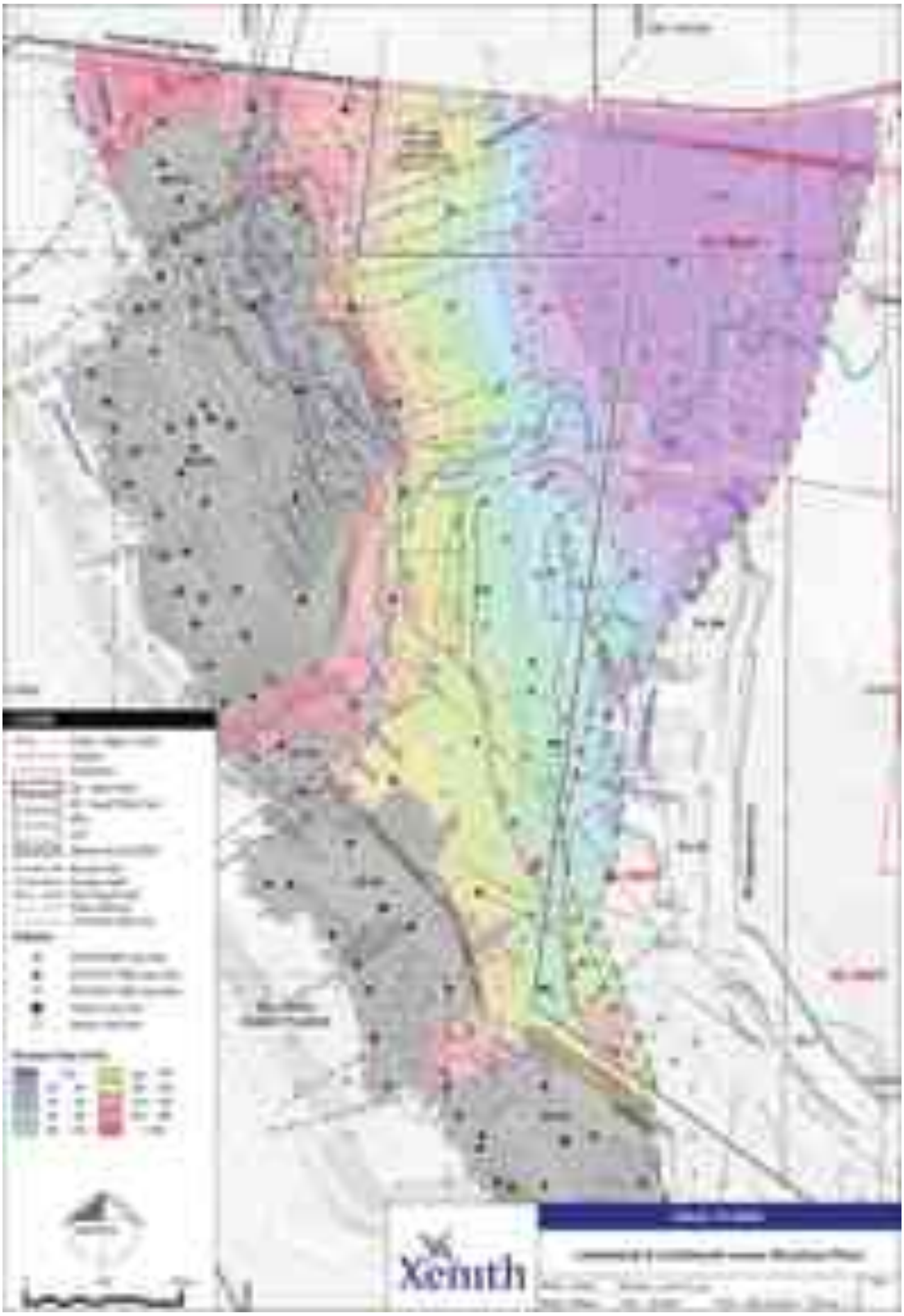


Figure 8.2 – Leichhardt Upper Seam Structure Floor

8.5.2 Stratigraphy

8.5.2.1 Leichhardt Seam

The depth of the cover for the main Leichhardt Seam within the un-mined areas is shown Figure 8.3. The depth of cover above the seam ranges between 64m in the west to 300m in the east. Some sections of deeper overburden occur in the western area of the remaining resource and are associated with the toe (western side) of the main thrust. As shown in the plans, depth of coal is highly variable around the interpreted faults.

Seam thickness is reasonably consistent across the resource area averaging approximately 3.5m and ranging from approximately 3.0m in the central area and to 4.1m in the south. Thickness for the main Leichhardt seam is in part controlled by the faulting but is also inclusive of a small stone band that gradually thickens in the north to split the seam into it upper and lower plies.

The LHD thickness is shown in Figure 8.4. In areas of seam repeats, the upper repeat thickness is shown. In general, lower fault repeats shows similar thickness as the upper repeats.

8.5.2.2 Leichhardt Upper Seam

Depth of cover for the Leichhardt Upper Seam is gently increasing from less than 20m in the west to 300m in the east, north of the split line (Figure 8.3). The Leichhardt Upper seam is intersected by the interpreted base of weathering in the area immediately to the east of the main central thrust fault.

Seam thickness for the Upper ply is reasonably consistent across the resource area, typically 2.3m thick, ranging from just under 2m thick in the west and increasing to 2.6 m in the east (Figure 8.5).

8.5.2.3 Leichhardt Lower Seam

Depth of cover for the Leichhardt Lower Seam is gently increasing from approximately 25m in the west to 310m in the east, north of the split line. The Leichhardt Lower seam is not interpreted to be intersected by the interpreted base of weathering in the area immediately to the east of the main central thrust fault but is in the vicinity.

Seam thickness for the Lower ply is also reasonably consistent across the resource area, typically 0.7m thick. Ranging from less than 0.4m thick in the north proximal to the main thrust fault and increasing to 1.1 m in the east (Figure 8.6).

Interburden thickness between the Leichhardt Upper and Lower plies is at its greatest approximately 13.5m (hole BC407) in proximity to the main thrust fault Figure 8.7). To the west of the fault the interburden between plies is within the range of 8 to 10m, to the south and east it thins and coalesces to form the main Leichhardt Seam.

Figure 8.3 – Leichhardt Seam Depth of Cover

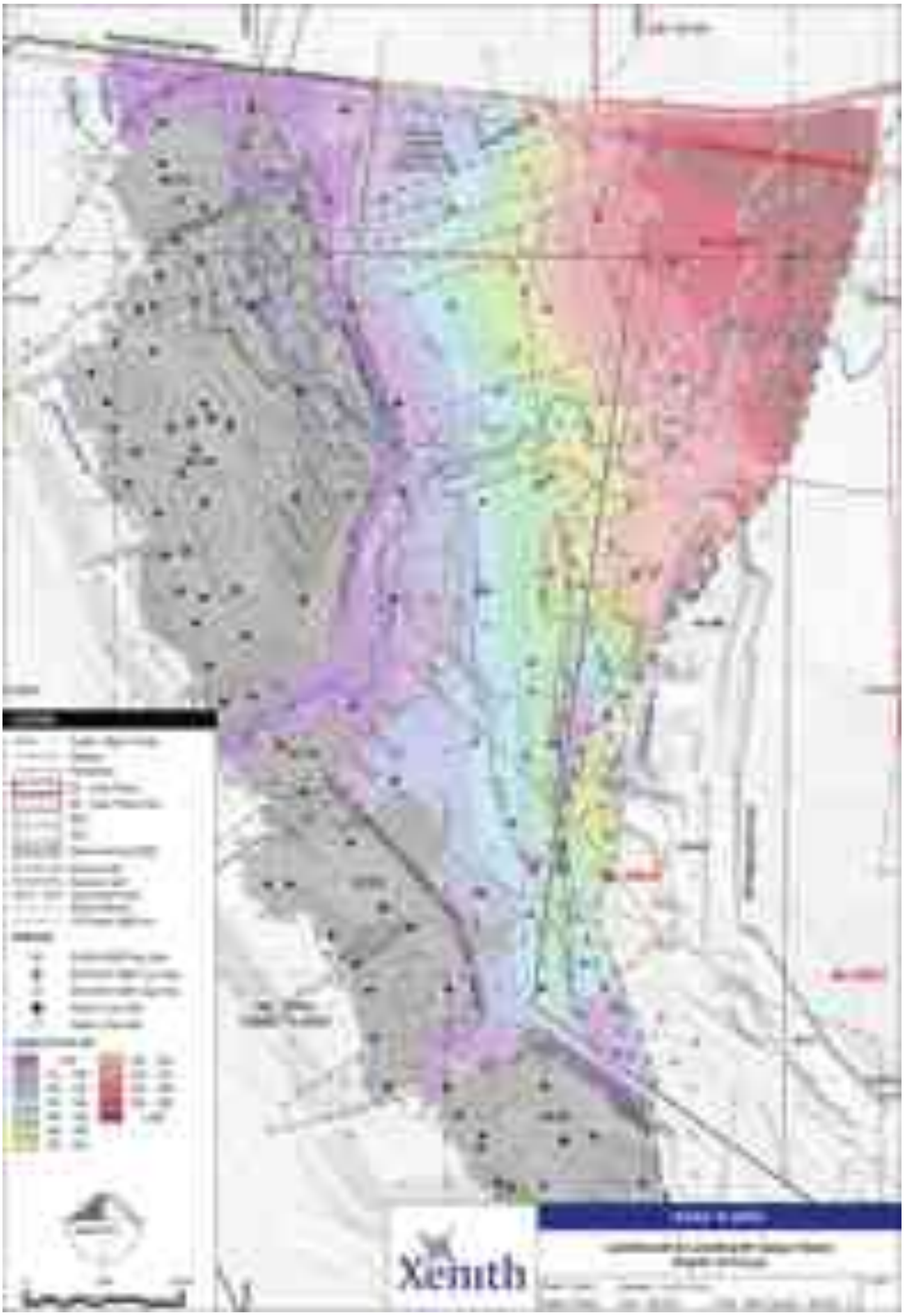


Figure 8.4 – Leichhardt Seam Thickness

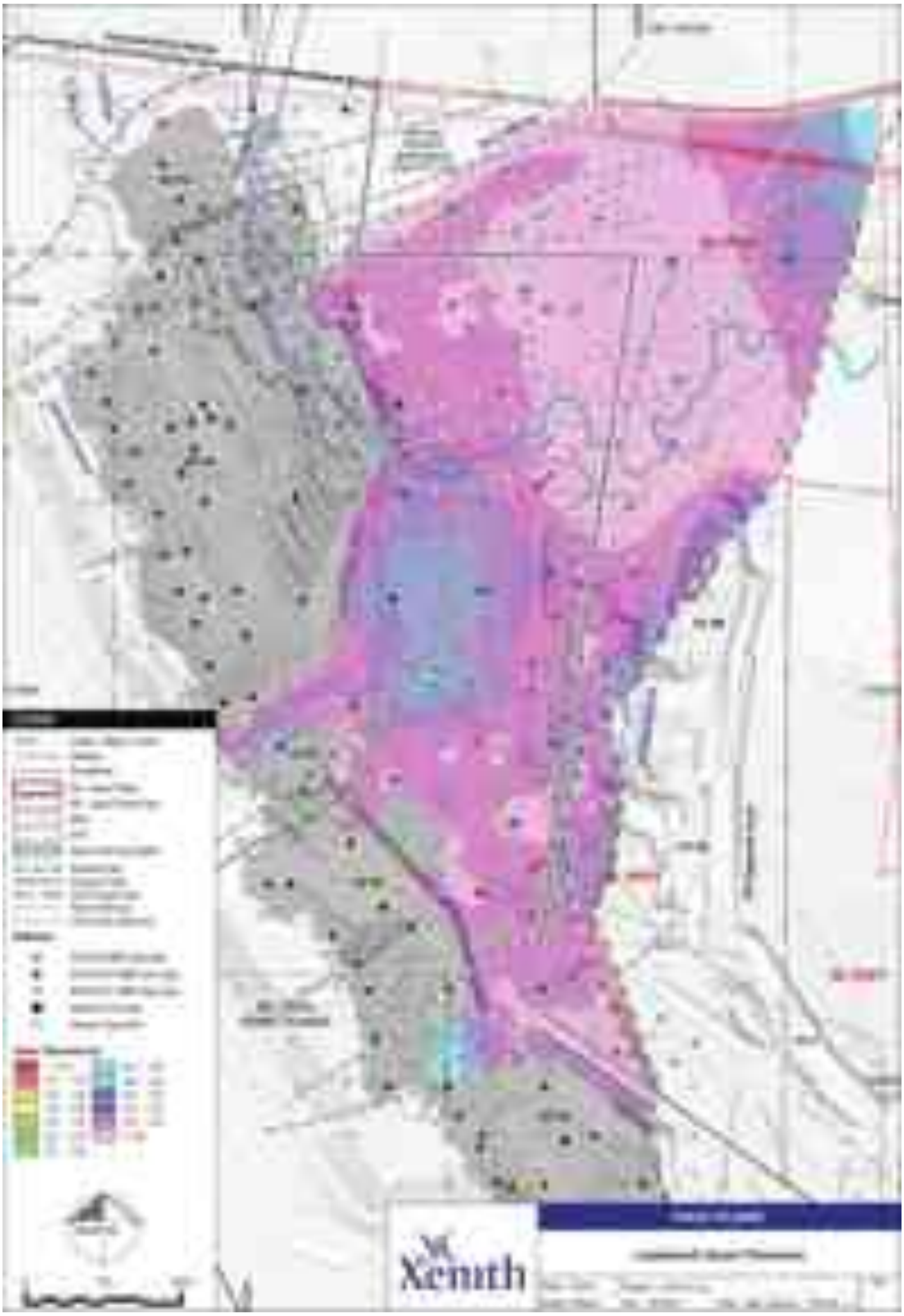


Figure 8.5 – Leichhardt Upper Seam Thickness

Figure 8.6 – Leichhardt Lower Seam Thickness

Figure 8.7 – Leichhardt Upper and Lower Interburden Thickness

8.5.3 Raw Coal Quality

The coal quality model for IPC is based solely on core samples; no chip samples have been included in the creation of the coal quality grids. The Leichhardt seam coal at Isaac Plains can be classified as a medium volatile, bituminous coal. The seam is considered low in raw ash, exhibiting good washability characteristics.

As with seam thickness for the Leichhardt Seam, raw ash values remain reasonably consistent throughout the ML area. For the entire resource the Leichhardt Seam averages 16.0% raw ash (adb), ranging from approximately 15.2% in the central western section of the pit to 17.4% in the far North East (Figure 8.8).

The Leichhardt Upper (LHU) seam raw ash values averages 15.6% (adb), increasing to the North; from 14.7% raw ash (adb) near the split line to 16.6% raw ash (adb) at the northern limit of the ML 70342 boundary (Figure 8.9).

The Leichhardt Lower seam (LHL) has typically higher average raw ash values (24.4% (adb) within the resource area) than the overlying Leichhardt Upper Seam and the main Leichhardt seam. Values of 24.0% raw ash (adb) occur at the split line in the South -west increasing to the North of the Resource area (Figure 8.10).

The average raw coal qualities for resources of the main LHD seam are shown in Table 8.4 below.

Table 8.4 – Leichhardt Full Seam (LHD) Resources - Weighted Average Qualities

Quality	Measured	Indicated	Inferred	Total
RD % (adb)	1.43	1.43	1.44	1.43
IM% (adb)	2.4	2.8	2.7	2.6
Ash% (adb)	15.9	16.0	16.6	16.0
Total Sulphur % (adb)	0.40	0.40	0.38	0.40
VM % (adb)	24.5	23.9	23.8	24.2

Figure 8.8 – Leichhardt Seam Raw Ash % (adb)

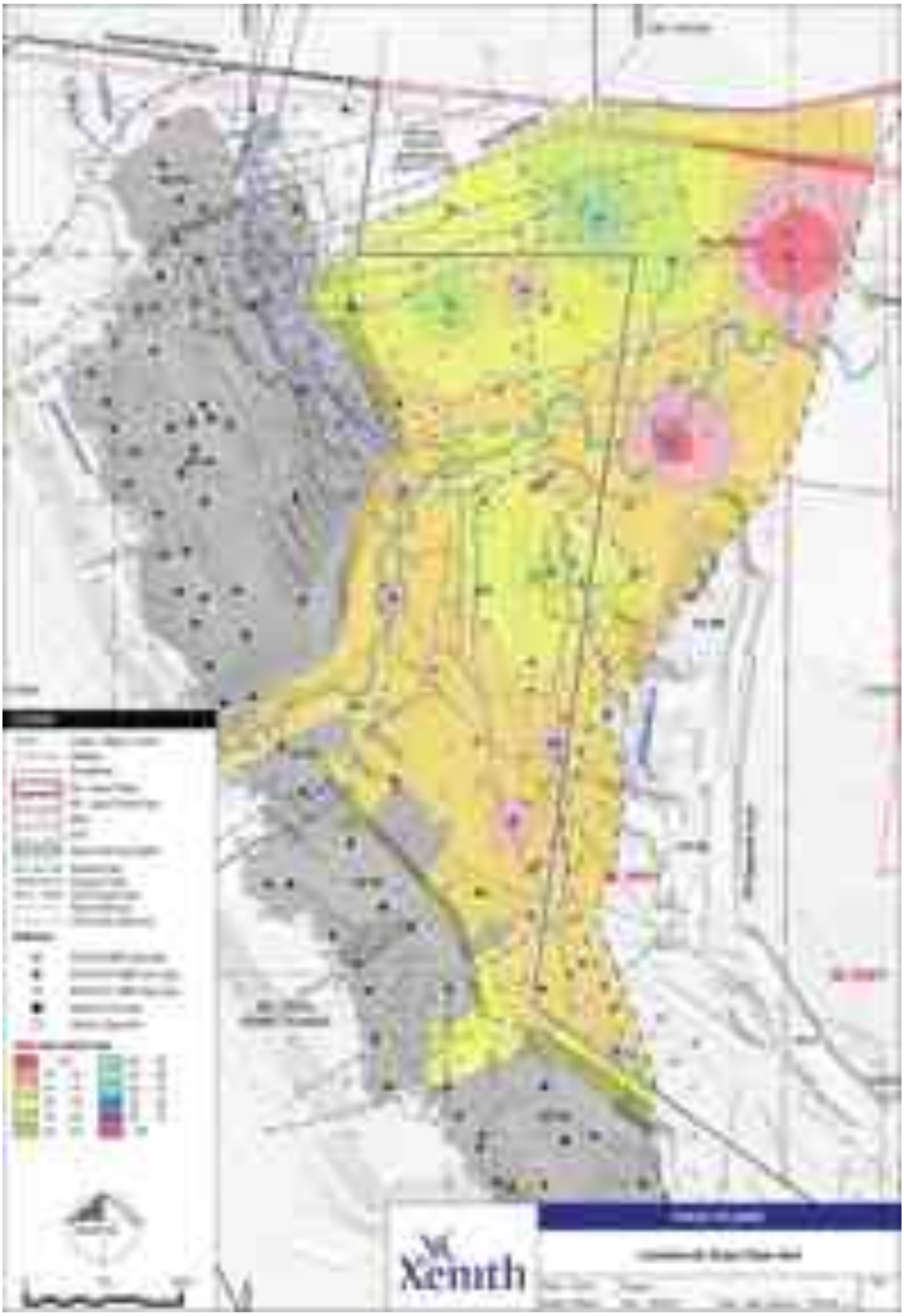


Figure 8.9 – Leichhardt Upper Seam Raw Ash % (adb)

Figure 8.10 – Leichhardt Lower Seam Raw Ash % (adb)

9 RESOURCE CLASSIFICATION

9.1 JORC Requirements

The JORC Code outlines minimum standards and guidelines for the reporting of a Mineral Resource. The JORC Code defines three resource classification levels dependent on the confidence in the mineral being estimated.

In this 2018 resource estimate, for a drill hole to be classified as a Point of Observation ("PoO") for a particular seam, it must be a cored hole and have:

- A geophysical log including density and gamma-ray data,
- Greater than 95% recovery across a seam or accepted by CP as being representative of the seam through analysis of the coal quality results, core photography and geophysical signature, and
- Have raw coal quality data.
- Values for raw ash % (adb) at less than 50%.

There are currently 96 holes in the model that meet the requirements as PoOs for this estimate. 36 of these PoOs in unmined areas.

This includes the 4 fully cored gas holes of the 2017 underground drilling program[2]. These holes did not have the full seam analysed. Selected horizons (4-6) - spread across the seams - were tested for raw coal qualities. The results confirm continuation the LHD coal qualities. In connection with the geological log, core photography and density measurements of the geophysical log (and its good correlation to raw ash), the holes have been deemed valid Points of Observation.

9.2 Classification Definition

Of the 96 PoOs, 79 had valid intersects of main Leichhardt Seam, 17 for the Leichhardt Upper ply and 10 PoOs had valid intersects for the Leichhardt Lower seam in the area north of the split line.

Three resource categories have been identified in the IPC, depending on the level of confidence in the seam structure and continuity plus the level of variability in the coal quality data, in accordance with the JORC Code 2012 Edition.

The following definitions from the JORC Code related to the classification of Mineral Resources are provided for information:

- A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

[2] The gas holes PoOs have no effect on Measured resources and only one of them adds to the Indicated resources (approximately 2 Mt in the north-western corner of the deposit).

- An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.
- An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation where data and samples are gathered.

The term 'Mineral' is equivalent to Coal when considering Coal Resources.

9.3 Resource Category Classification

9.3.1 Points of Observations (PoO) Distance

The requirements of drill hole spacing are not prescribed for the 2014 Coal Guidelines so long as consistent seam stratigraphy and coal quality can be established. This is generally the case for the Isaac Plains resource as is summarised in earlier chapters of this report and supported by drilled chip holes with down-hole geophysics at regular grid intervals.

For this resource estimate, as for the estimates previously, the maximum radius of influence around Points of Observation for measured resource was set as ~250m, for indicated resources at ~500m and for inferred resources at ~2,000m.

9.3.2 PoO Orientation

A minimum of 3 PoOs were required to generate resource estimates for a single resource category. Where these PoOs formed a linear relationship relative to each other, the continuity of the deposit could not be established. Therefore, as a minimum 3 PoOs needed to form a 'triangle shape' spatially, allowing the continuity between these points to be established.

9.3.3 Maximum Raw Ash Percentage

A maximum raw ash percentage of 50% (adb) has been applied to the resource estimate.

9.4 Coal Quality

9.4.1 Preston and Sanders Relative Density Adjustment

The modelled insitu relative density was calculated using the Preston Sanders method.

$$RD(in\ situ) = \frac{RDad \times (100 - Mad)}{\{100 + RDad \times (ISM - Mad) - ISM\}}$$

Where:

RD (in situ) = Relative density (in situ moisture basis)

RDad = Relative density (air-dried basis)

Mad = Air dried moisture

ISM = in situ moisture (4.5%)

The insitu density derived from the resultant ash values and true RD was applied to the estimated coal volume to generate the coal tonnes.

9.5 Areal Constraints

The resource polygons are limited by the following parameters:

- The up-dip limit is the fresh coal line or the current (as at May 2020) mined face position.
- The down dip limit is set by the Burton Range thrust fault.
- The northern limit is set by the Goonyella to DBCT railway line or boundary of ML70342.
- The southern limit is set where the southern pit boundary meets the Isaac Thrust. (Figure 10.1).

Further constraints were applied to the LHL seam which was only estimated in an area of proposed open-cut extraction (<150 depth of cover). The thickness of the seam would not be viable based on current underground extraction methods.

The Leichhardt Seam in its entirety was reported within the constraints listed above but the northern limit for full seam reporting was restricted to the split line. The split line is located at the point where the stone parting between the Leichhardt Upper and Leichhardt Lower plies is greater than 0.3m. North of this line the resource is restricted to the Leichhardt Upper Seam and Leichhardt Lower seam, where they meet their restrictions. The Leichhardt Upper Seam is reported for the entire resource area north of the split line as seam thickness is considered viable for proposed open-cut and underground extraction by current methods (Figure 10.1).

Additional drilling would be recommended to improve the structural interpretation and confidence towards the north eastern extremities of the ML boundary.

The following areas have been excluded from resource estimation:

- A small area between the S2 and S3 pits where the LHD seam is intruded.
- 2 areas down-dip of the S2 pit where the LHD seam has been Highwall mined (HWM)

9.6 Resource Estimate Methodology

Resources were estimated using the Minescape Reserves Polygons samples module.

Average raw coal quality parameters were estimated per resource area and reported on an air-dried basis:

- Raw RD,
- In Situ RD,
- Raw ash,
- Inherent moisture,
- Volatile matter,
- Chlorine,
- Total sulphur, and
- Specific Energy

10 RESOURCE ESTIMATE

10.1 Resource Areas

Resources are reported over several different tenements:

- ML 70342,
- The portion of ML 700019 located between the boundary of ML 70342, and the Goonyella railway.
- The portion of ML 700018 and ML 700019 where the Leichhardt seam is on the western (downthrown) side of the Burton Range thrust.

In the main area south of the split line the Leichhardt Seam is reported as full seam. In the main area north of the split line, the Leichhardt Upper and Leichhardt Lower seams are reported only where they meet their individual criteria.

The resources have been sub-divided according to their depth of cover, i.e. to 100m, between 100 and 150m and beyond 150m. Depending on expected coal prices and foreign exchange rates, these ranges are considered to represent possible borders between open-cut and underground areas.

At this stage, as in previous resource estimates, the area under the Smoky Creek has not been excluded or discounted. The CP has regarded this coal as having reasonable prospects for eventual economic extraction due to its shallow nature and seam thickness results.

Due to the uncertainty of mining conditions near the major Burton Fault, the resources within 100m of the fault have been classified as Inferred Resources.

The 2017 3D seismic survey interpreted a fault repeat in the south-west of the underground area. Due to poor seismic data quality in this area, the seismic interpretation could not interpret the southern extent of this repeat. Drilling from 2018/19 has provided enough information to confidently interpret the extent of this uplifted area. The upper repeat contains approximately 0.9 Mt of resources, mostly in the less than 100m (35%) and 100-150m (50%) depth ranges. This area is isolated from the current open-cut mining areas. These resources have been classified as Inferred.

Figure 10.1 outlines the resource classification areas for the Leichhardt Seam, including the Inferred resources area along the Burton Fault, but not including the Inferred resources of abovementioned upper repeat.

Figure 10.2 outlines the resource classification areas for the Leichhardt Lower, to a depth of cover of 150m.

Figure 10.1 – Leichhardt and Leichhardt Upper Seam Resource Classification

Figure 10.2 – Leichhardt Lower Seam Resource Classification

10.2 Tonnage Estimate

The total coal resource estimate is 46 Mt, of which 25.6 Mt is classified as Measured resource, 16.4 Mt is classified as Indicated resource and 4 Mt is classified as Inferred resource. A summary of the coal resource estimate for the complete project area can be found in Table 10.1, and by seam/ply in Table 10.2, below.

Table 10.1 – Total Resource Estimate June 2020

Seam	Measured Tonnes x 10^6	Indicated Tonnes x 10^6	Measured and Indicated Tonnes x 10^6	Inferred Tonnes x 10^6	Total Tonnes x 10^6
ML 70342 Total	21.5	1.9	23.3	0.5	23.8
ML 700018 & ML 700019 Total	3.7	14.1	17.8	4.7	22.5
Total Resource	**25.2**	**16.0**	**41.2**	**5**	**46**

Note – Rounding to the nearest significant figure is applied to Total Resource Tonnes in the Inferred Category. This is deemed conservative and reflective of the Inferred Resource category confidence level and accounts for the minor differences in the overall reported resource

Table 10.2 – Total Resource Tonnes by Seam

Seam	Measured Tonnes x 10^6	Indicated Tonnes x 10^6	Measured and Indicated Tonnes x 10^6	Inferred Tonnes x 10^6	Total Tonnes x 10^6
LHD	23.8	14.7	38.6	4.2	42.8
LHU	1.2	1.1	2.3	0.2	2.6
Total LHD/LHU	25.1	15.9	40.9	4.5	45.4
LHL	0.1	0.1	0.3	0.7	1.0
Total Resource	**25.2**	**16.0**	**41.2**	**5**	**46**

Note – Rounding to the nearest significant figure is applied to Total Resource Tonnes in the Inferred Category. This is deemed conservative and reflective of the Inferred Resource category confidence level and accounts for the minor differences in the overall reported resource

10.2.1 Comparison to 2018 Resource Estimate

The total resource decreased by 6.0 Mt or 12% since the previous resource estimate. The main changes are as follows:

- Depletion by mining since May 2018 (>1.8 Mt).
- Re-interpretation of the Burton Thrust fault from 2018 drilling combined with the 3D seismic survey has resulted in the fault moving to the west in the central part of the deposit and conversely to the east in the northern part. Additionally, the uplifted area between the two main faults that intersect has become smaller after re-interpretation. This change accounts for most of the difference in the total resource.

A part of the northern resource has been converted from Indicated to Measured after re-evaluation of PoOs and confidence levels, which has caused an increase of the Measured resource of 3.8 Mt (17%)

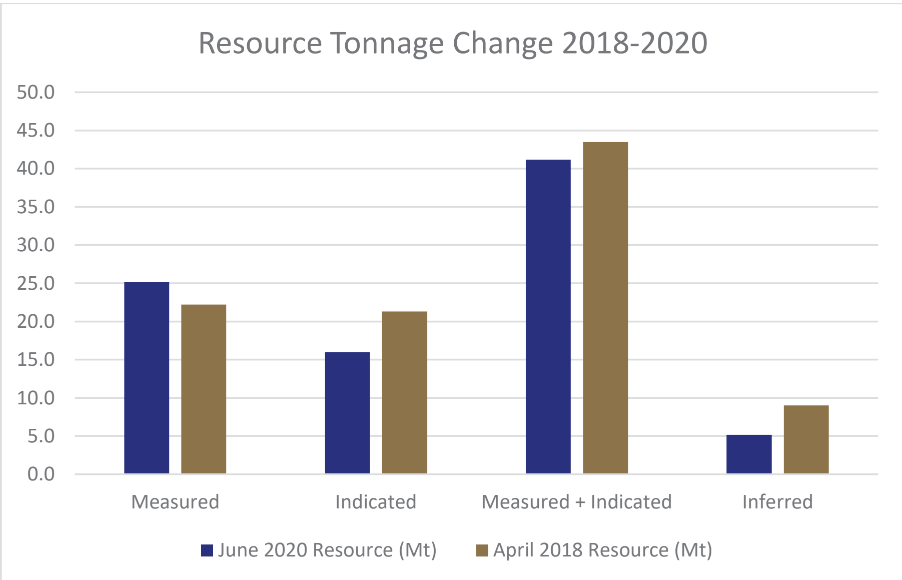
The tonnage differences can be seen in Table 10.3 and Figure 10.3.

Table 10.3 – Tonnage Comparison per Resource Category

	June 2020 Resource (Mt)	April 2018 Resource (Mt)	Difference (Mt)	Difference (%)
Measured	25.2	22.2	3.0	13%
Indicated	16.0	21.3	-5.3	-25%
Measured + Indicated	41.2	43.5	-2.3	-5%
Inferred	5	9	-4	-43%
Total	**46**	**52.5**	**-6**	**-12%**

Note – Rounding to the nearest significant figure is applied to Total Resource Tonnes in the Inferred Category. This is deemed conservative and reflective of the Inferred Resource category confidence level and accounts for the minor differences in the overall reported resource

Figure 10.3 – Resource Comparison 2018 - 2020



10.3 Coal Quality

Table 10.4 details the average raw qualities broken down by resource classification and seam. The 46 Mt of resource has an average raw ash (adb) of 16.0%, at an insitu RD of 1.42. The LHL seam exhibits on average higher raw ash values, but the overall averages are influenced by results from the main Leichhardt seam, as expected.

Table 10.4 – Resource & Tonnages with Quality by Depth and Resource Category

Depth Interval	Ply	Resource Classification				Thickness	RD In Situ	Inherent. Moisture	Ash	Fixed Carbon	Volatile Matter	Total Sulphur	Specific Energy
		Measured	Indicated	Inferred	Total (Mt)	(m)	(T/M³)	% (Adb)	% (Adb)	% (Adb)	% (Adb)	% (Adb)	kcal/Kg (Adb)
≤100m	LHD	7.0	0.1	0.0	7.1	3.5	1.41	2.2	15.6	57.5	24.7	0.43	6755
	LHU	1.2	0.3	0.0	1.5	2.2	1.42	2.5	15.8	57.3	24.4	0.43	6918
	LHL	0.1	0.1	0.3	0.6	0.8	1.51	2.7	24.4	50.8	22.1	0.44	5848
	TOTAL	8.2	0.6	0.3	9.1	3.1	1.42	2.3	16.2	57.1	24.5	0.43	6726
≥100m & ≤150m	LHD	10.3	1.1	0.0	11.5	3.5	1.42	2.5	16.0	57.0	24.6	0.41	6666
	LHU	0.1	0.8	0.1	0.9	2.4	1.41	2.5	15.0	58.0	24.5	0.40	6959
	LHL	0.0	0.0	0.4	0.4	1.0	1.51	2.7	24.9	50.3	22.2	0.45	5810
	TOTAL	10.4	1.9	0.5	12.8	3.3	1.42	2.5	16.2	56.8	24.5	0.41	6661
≥150m	LHD	7.0	13.9	3.3	24.2	3.5	1.42	2.7	15.9	57.4	24.0	0.38	6590
	LHU	0.0	0.0	0.1	0.2	2.6	1.42	2.6	15.0	58.0	24.5	0.39	6956
	LHL	0.0	0.0	0.0	0.0	0.9	1.51	2.9	24.3	51.0	21.8	0.46	5795
	TOTAL	7.0	13.9	3.5	24.4	3.5	1.42	2.7	15.9	57.4	24.0	0.38	6592
Total	LHD	24.3	15.1	3.4	42.8	3.5	1.42	2.6	15.8	57.3	24.3	0.40	6638
	LHU	1.2	1.1	0.2	2.6	2.3	1.42	2.5	15.5	57.6	24.5	0.41	6935
	LHL	0.1	0.1	0.7	1.0	0.9	1.51	2.7	24.6	50.6	22.1	0.44	5832
	TOTAL	25.6	16.4	4	46	3.4	1.42	2.6	16.0	57.2	24.2	0.40	6638

Note – Rounding to the nearest significant figure is applied to Total Resource Tonnes in the Inferred Category. This is deemed conservative and reflective of the Inferred Resource category confidence level and accounts for the minor differences in the overall reported resource

10.4 Recommendations

The open-cut area is well explored by drilling for resource estimation.

Due to the 3D seismic survey, the structure for the underground area is generally well defined. Recent drilling in this area has improved the interpretation of faulting within this area. Further production drilling will be necessary to firm up the definition of smaller faults if underground mining is approved.

11 JORC STATEMENT

The information in this report relating to exploration results and coal resources is based on information compiled by Mr Troy Turner who is a member of the Australasian Institute of Mining and Metallurgy and is a full time employee of Xenith Consulting Pty Ltd.

Mr Turner is a qualified geologist and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking, to qualify as Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves."

Mr Turner consents to the inclusion in the report of the matters based on the information, in the form and context in which it appears.

Troy Turner
M AusIMM
227689

12 REFERENCES

Isaac Plains Mine (ML 70342) - Resource Statement as at December 2009, JB Mining, January 2010.

Isaac Plains North-2013 Fault Drilling Results and Updated Structural Model Details, JB Mining 2013.

Stanmore Coal Ltd, Isaac Plains Coal Mine, 2015 Coal Resource Estimate, Xenith Consulting 2015.

Stanmore Coal Ltd, Isaac Plains Coal Mine, April 2016 Coal Resource Estimate, Xenith Consulting 2016.

Stanmore Coal Ltd, Isaac Plains Coal Mine, August 2017 Coal Resource Estimate, Xenith Consulting 2017.

Stanmore Coal Ltd, Isaac Plains Coal Mine, May 2018 Coal Resource Estimate, Xenith Consulting 2018.

APPENDIX A. JORC CODE, 2012 EDITION – TABLE 1

This Appendix details sections 1, 2 and 3 of the JORC Code 2012 Edition Table 1. Sections 4 'Estimation and Reporting of Ore Reserves' and 5 Estimation and Report of Diamonds and Other Gemstones' have been excluded as they are not applicable to this deposit and estimation.

SECTION 1 SAMPLING TECHNIQUES AND DATA

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	CP Comments
Sampling Techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	Exploration April 2018 – present: • 23 open holes were drilled in the IPU area , mainly for the purpose of fault delineation. Exploration 2015 – April 2018: • 94 open holes were drilled, mainly for the purpose of fault delineation. • 29 cored coal quality holes were completed within the ML. An additional 5 holes were drilled within Isaac Plains East where the LHD seam has been intersected on the western side of the Burton Range thrust and is consequently included in the IPC area. 19 holes were drilled in the potential underground mining area in the second half of 2017. Four (4) of these were for the purpose of gas testing. • For the Stanmore 2015/2016 and 2016 / 2017 program, all cored intervals were sampled where coal was present at thickness of 0.1m or more, with a maximum sample thickness of 0.5m. Coal plies were sample discretely on the basis of lithological characteristics and quality. All non-coal material and partings less

Criteria	JORC Code Explanation	CP Comments
		than 0.1m were included with the coal ply and noted in the lithological description. Non-coal interburden material greater than 0.1m and up to a maximum of 0.3m were sampled separately. Approximately 0.30m of immediate roof and floor were also collected as dilution samples. • Geotechnical samples were collected from roof (up to 10m above seam) and floor sections (up to 6 metres below seam). Selected samples were analysed with testing including UCS, Young's Modulus, Poisson's Ratio, Slake Durability or Tri-axial testing. • All remaining un-sampled cored material has been retained in marked core boxes for future reference. • All coal quality samples were double bagged at site and marked with sample number, hole and project. The samples were then kept in cold storage on site before dispatch to the laboratory via a tracked freight service. Chain of Custody and sample documentation were sent to the laboratory by email ahead of the samples. Coal was stored on site for periods of no more than two weeks prior to dispatch. Geophysical corrections were undertaken as soon as practicable following sample collection and these were used to confirm representative core recovery. • Line of Oxidation chip samples were collected from the shallowest coal seam in each hole where coal was intersected, regardless of whether it appeared weathered or not. If deeper seams also appeared weathered, these too were sampled. Samples were collected in 1m intervals in sealed plastic bags and marked with sample number, hole number and project. These sample bags were then grouped into larger plastic bags. These samples were stored and shipped in the same manner as the coal quality core samples.

Criteria	JORC Code Explanation	CP Comments
		• Coal quality samples were sent to Bureau Veritas Laboratories in Brendale, Queensland. Bureau Veritas Minerals Pty Ltd is a NATA registered and a well-recognized coal analytical organization conducting coal quality sampling for many years. Bureau Veritas are accredited for compliance with ISOMEC 17025, corporate accreditation number 1805. Site accreditation number 18415. • Samples were stored in cold storage at Bureau Veritas until instruction are available to conduct the analytical program. Exploration 2009 to 2014: • Xenith is not aware of any Coal quality drilling undertaken within in this period. • Exploration drilling in 2013 involving 36 holes of structural fault definition. Exploration 2008 to 2009: • In July 2008 to September 2009 BCCM drilled a further 287 drill holes to assist with determining gas content, improving fault definition. • For the 2008 program, samples were taken at approximately 30cm intervals (2010 JORC Resource report) • All cored holes were photographed in the field (digital camera), sampled, boxed into core trays, where depths were recorded for subsequent reference. • No detail of interburden thickness sampling rules was presented.

Criteria	JORC Code Explanation	CP Comments
		• The immediate roof and floor have been sampled of lengths >than 0.1m in general. At the minimum Ash and RD analysis has been conducted. • All coal samples were collected into plastic bags and then transported to the laboratory via courier and were accompanied by a sample advice sheet. • Coal Quality samples were sent to ALS / Actest Laboratory in Maitland NSW, or Bureau Veritas (previously CCI) Laboratory in Newcastle. • All coal quality samples were prepared and analysed using ALS/ Actest or Bureau Veritas testing parameters. Both laboratories are NATA registered and have been operating in Australia for over 50 years. Exploration 2004 to 2006: • For the 2004 program, samples were taken on approximately 25-30cm intervals (2010 JORC Resource report) • For cored holes, coal seams were sampled discretely on the basis of lithological characteristics such as the brightness profile, and where reasonable were sampled on a ply basis into approximately 0.5m plies • No detail of interburden thickness sampling rules was presented. • The immediate roof and floor have been sampled of lengths >than 0.1m in general. At the minimum Ash and RD analysis has been conducted. • All coal samples were collected into plastic bags and then transported to the laboratory via courier and were accompanied by a sample advice sheet.

Criteria	JORC Code Explanation	CP Comments
		• Coal Quality samples were sent to Casco Australia Pty Ltd (Casco) laboratory in Mackay. • All coal quality samples were prepared and analysed using Casco testing methodologies. Casco is a National Association of Testing Authorities (NATA) registered organisation. • Line of oxidation (lox) samples were collected in 0.5m samples. • Lox samples were bagged on site and sent to CCI Australia Laboratory in Moranbah for analysis. • Gas sampling was conducted at three sites, located in pits N1, N2 and S3. The full seam was sampled into gas canisters. • Q1 gas testing was undertaken by the field Geologist in the field. The process of analysis involved Geogas standard procedures. • Gas samples were sent to Geogas laboratory in Mackay for gas analysis (Q2 and Q3). • Seven fully cored (diamond) holes were drilled to analyse the overburden, coal and floor sediments for rock strength and other geotechnical issues. Samples were stored in core trays, with representative 30cm length samples wrapped in plastic and sealed from moisture. • Geotechnical samples were reviewed from 7 HQ fully cored drill holes by Insite Geology and sent samples for destructive geotechnical test work with Ullman and Nolan laboratories I Mackay. • Multiple mini-Sosie seismic work undertaken by Velseis Pty Ltd in March/April 2004 and July/August 2005 (8.7km and 9.3km surveys respectively) to better delineate structure within the deposit.

Criteria	JORC Code Explanation	CP Comments
		• Ground magnetic survey undertaken by Resolve Geological in October 2004 to delineate extent of intrusive material within the area. • 15 lines of Mini-Sosie seismic survey were completed by Velseis in 2015 / 2016 covering 32 km. These traverses both the IPC and the IPE . Historic exploration: • Details for the sampling of historic drilling information Pre -2004 are not available. • A review of suitable historic holes was reported to have been conducted as part of the 2010 resource estimate.
Drilling Techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	2015/16 and 2016 / 2017 exploration: • The 2018/2019 open holes were 100m diameter drilled with either PCD or Blade bit. • For the Stanmore 2015/2016 and 2016 / 2017 exploration program, part-cored holes for coal quality were drilled in HQ3 diameter (61.1mm diameter core). Holes were extended at least 4m below the base of the last intercepted coal seam to allow for geophysical logging of the entire seam. • Chip holes were drilled using either poly-crystalline diamond or blade bits. Hole size varied between a minimum of 99 mm and a maximum of 229mm, depending on the type and diameter of bit used. • All core was photographed in 0.5m intervals against a blackboard with depth markings, lithology and sample numbers added. Chips were laid out on bare ground in lines of 30 one metre samples

Criteria	JORC Code Explanation	CP Comments
		further subdivided into 6m runs. Chips were photographed in 6m runs with a whiteboard showing hole number, date and depth range. In all photographs, depth increases from left to right. Historic exploration: • All coal quality holes were cored (partially or fully) using core barrel, producing a 63.5 mm and 100mm core diameter (also a series of 200mm cores were drilled late 2004). • Structural holes were drilled as part of a fault delineation program. As part of this work, these holes were fully open (chipped). • Lines of Oxidation ("LOX") holes were drilled by a reverse circulation hammer drill rig. • Non-cored holes were used in the model to define structure and stratigraphy but were not used as Points of Observation ("POO"). • A full list of drill holes and drilling types is available at the end of Table 1 in Appendix C
Drill Sample Recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	2015/16 and 2016 / 2017 program: • Only cores were sampled for analysis • Adequate recovery was assessed on a length basis • A 95% linear seam recovery was required; otherwise the seam would be redrilled. • The CP is adequately satisfied no sample bias has occurred. Pre 2015: • No details of the process followed for determining % recovery were viewed for the purpose of producing this resource report.

Criteria	JORC Code Explanation	CP Comments
		• If there was less than 95% core recovery, it appears the seam was required to be redrilled. • No details were available on the relationship between sample recovery and quality or sample bias.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.* • *The total length and percentage of the relevant intersections logged.*	• All drill core was geologically logged, marked and photographed prior to sampling. Geological and geotechnical features were identified and logged as part of this process. • All chip holes had chips collected every metre, which were then geologically logged and photographed. • All drill holes have been geophysically logged (except where blocked) with the minimum suite of tools run including: Density, Calliper, Verticality/Deviation and Gamma. • A full list of the suite of geophysical logs that have been run on each drill hole can be found in Chapter 6.7 of the Resource estimate report. • The calibration of the geophysical tools was conducted by the geophysical logging company engaged in the project at the time.
Sub-Sampling Techniques and Sample Preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*	2015/16 and 2016/17 program: • All core coal samples were double bagged on site and were transported by tracked freight courier to the laboratory for testing. • Ply samples were initially tested by Bureau Veritas for Apparent Relative Density (ARD), which is a non-destructive water immersion density test. The results were provided and analysed prior to creation of float-sink (wash) composite sections. • Wash composites were created per each LHD seam intersection, consisting of either:

Criteria	JORC Code Explanation	CP Comments

	• *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.*
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*
		– A single full-seam section, being the total intersected coal thickness at a core hole location, with composited full-seam thickness for the LHD seam ranging from 2.85 to 4.01m *or*
		– Two composites per seam being:
		  – **Top** of seam composite (approx. 2.0m to 2.3m thickness)
		  – **Bottom** of seam (remainder of seam, generally 1.3m to 1.8m thickness)
		• The decision to create either a one or two composites was based on several factors, primary among which were the core holes' physical location and seam thickness.
		• To simulate mine transport conditions each composite sample was then drop shattered 20 times from a height of 2 metres, any sample mass remaining of >50mm was hand knapped to 50mm, dry tumbled and dry sized at 31.5, 25, 16, 8, 4 and 2mm.
		• Composite samples were then split and further analysed as follows:
		• 1/8 for quick coke: Crush to 11.2mm, float sink at 1.425 density, crush to 4mm and mill sample to test for Proximate, CSN, Gieseler & Dilatation
		• 1/8 for raw analysis: Crush to 4mm, mill sample to test for RD, MHC, Proximate, TS, CSN, Calorific Value & Cl
		• ¾ for float sink: Wet tumble and wet size at 31.5, 25, 16, 8, 4, 2, 1, 0.5, 0.25, 01.25 & 0.063mm. Re-combine samples in following fractions: -50+16mm, -16+8mm, -8+2mm and -2+0.25mm. Float sink each size fraction at densities (F1.30, F1.35, F1.375, F1.40, F1.45, F1.50, F1.55, F1.60, F1.70, F1.80, F2.00). -0.25+0mm fraction subject to tree froth flotation. All fractions analysed for ash and CSN.

Criteria	JORC Code Explanation	CP Comments
		• Washability simulations were performed on the float sink results and from that data clean coal composite samples were compiled and analysed for: Primary Coking (-16+0mm), Coarse Coking (-50+16mm) and Secondary Thermal Coal Composites. • The various product types were identified for each hole (from the float sink dataset) and clean coal composite samples were derived and assayed for the various representative properties • Gas holes: Selected coal core sequences from the 4 designated gas-holes were placed in canisters on site and tested for gas content (Q1 test). Subsequent laboratory testing completed (Q2 and Q3) the testing for gas content. Pre 2015: • Casco complies with the Australian Standards for sample preparation and sub-sampling. • All coal samples were crushed to a top size of 32mm before analysis, for HQ and PQ core (63.5 mm and 85 mm core diameter) and for 100mm core. • Two, 200mm cores were drilled to take a bulk sample for detailed sizing, washability and coke oven testing.
Quality of Assay Data and Laboratory Tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.*	• Bureau Veritas Minerals Pty Ltd is a NATA registered and a well-recognized coal analytical organization conducting coal quality sampling for many years. Bureau Veritas are accredited for compliance with ISOMEC 17025, corporate accreditation number 1805. Site accreditation number 18415. • Casco in Mackay, QLD comply with the Australian Standards for coal quality testing and are certified by the NATA.

Criteria	JORC Code Explanation	CP Comments
	• *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	• Geophysical tools were calibrated by the logging company engaged in the project at the time.
Verification of Sampling and Assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Bureau Veritas in Brendale, QLD comply with the Australian Standards for coal quality testing, and as such conduct the verifications for coal quality analysis outlined in the standards. Casco in Mackay, QLD comply with the Australian Standards for coal quality testing, and as such conduct the verifications for coal quality analysis outlined in the standards. • Coal quality results were verified by Stanmore and Xenith Consulting Pty Ltd (“Xenith”) personnel before inclusion into the geological model and resource estimate. • Coal quality procedure design, data validations, washability simulations and product coal assessment and analysis was undertaken by Chris McMahon of McMahon Coal Quality Resources (MCQR).
Location of Data Points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• The topographic surface has been generated from LiDAR, which was flown by Aerometrix, February 2020. Vertical Accuracy: +/- 0.15m. • All holes from the 2016, 2017, 2018 and 2019 campaigns were professionally surveyed by MSS (Golding) surveyors that currently undertake all survey control at the nearby Stanmore owned Isaac Plains Mine Site. The origin of the survey was based on the calculated site base station coordinates and level of the site survey station from the AUSPOS static data listed below. All values are in AMG84 Zone55 coordinates as is the site base station RTCM0000 coordinates

Criteria	JORC Code Explanation	CP Comments
		• The 2015/16 drill holes were surveyed by MSS and JTH Surveys, Moranbah, using site base station (RTCM0000) and Trimble R10 GPS. • Previous drilling was surveyed by Shield Surveying Pty Ltd (Mackay) and Mackay Surveys Pty Ltd. • The datum used AGD 84 and the projection used AMG 84 Z55.
Data Spacing and Distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Drill hole spacing has been dictated by the characteristics and consistency of the target seams within the deposit. • Exploration drilling has been conducted on different drilling patterns depending on the nature of the program. For instance, the fault delineation drill holes were spaced between 10 to 20m apart along a pre-determined targeted line. • Structural drilling is in general on 250m centres and coal quality drilling is located on approximately 500m centres. • The inclusion of holes from neighbouring areas has given the model a reasonable amount of lateral continuity in the north of the ML area. • Samples were reported to have been taken on approximately 20 - 40 cm interval and compositing into top and bottom plies. As such, where appropriate, sample compositing has been completed. • Considering the continuity of the target seam(s) in the deposit, this spacing has proven to be sufficient to give adequate control to the model and give the required confidence in the geological interpretation.
Orientation of Data in Relation	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	• The orientation and spacing of the drilling grid are deemed to be suitable to detect geological structures and coal seam continuity within the resource area.

Criteria	JORC Code Explanation	CP Comments
to Geological Structure	• *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• 2D seismic sections complement the distribution of drill holes. • Comprehensive 3D seismic data was acquired in late 2017. Data points and fault interpretations were included in the geological model to compliment the 2D seismic and drill hole intersections.
Sample Security	• *The measures taken to ensure sample security.*	• All coal quality cored samples were double bagged in plastic bags on site and the dispatched to Bureau Veritas in Brendale Queensland via tracked freight service. Chain of custody and sample information was emailed to the laboratory ahead of the sample. • All samples were held in cold storage prior to leaving site and at laboratory prior to analysis. • The same procedure was used for all geotechnical samples derived from the cored holes. • Previous programs provide no details on sample security from the provided literature.
Audits or Reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Cross plots for raw Rd and raw ash% have been produced to validate the results of the coal quality data. The variability of the data is within the expected range. • Bureau Veritas undertake internal audits and checks in line with the Australian Standards and their NATA certification. Corporate Accreditation no. 1805 and site no. 18415 • Casco undertake internal audits and checks in line with the Australian Standards and their NATA certification. • Vale reported to have performed a high level technical review of the geological data system during the sale process in 2007

SECTION 2 REPORTING OF EXPLORATION RESULTS

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code Explanation	CP Comments
Mineral Tenement and Land Tenure Status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• Isaac Plains Mine consists of Mining Lease 70342, held by Stanmore IP Coal Pty Ltd, and fully owned subsidiary of Stanmore Coal Limited. • Isaac Plains East (IPE) is covered by four (4) Mining Leases, ML 700016, ML 700017, ML 700018, and ML 700019, each of which was granted to Stanmore IP Coal Pty Ltd on 1st March 2018. • Stanmore Wotonga Pty Ltd is contractual holder of MDL 137 (north) although this portion of the MDL, which is north of the Peak Downs Highway, continues to be formally held by Millennium Coal Pty Ltd. Tenure title of MDL 137 (Wotonga) must remain with Millennium due to the fact that this MDL also continues to the south of the highway; however, the full underlying contractual rights are held by Stanmore. • The eastern part of the underground resource estimated herein is now covered under ML700018 & ML700019. ML 700018 and ML 70019 also cover Stanmore Coal's Isaac Plain East Mine(IPE). IPE targets the Leichhardt (LHD) seam on the up-thrown side of the Burton Thrust Fault. The eastern portion of the IPE resources is a fault repeat and overlies the Isaac Plains underground resource. • EPC 677 is located to the North of the ML and is currently held by Fitzroy (CQ) Pty Ltd. Stanmore have a signed Designated Area Agreement (DAA) with Fitzroy. The DAA allows Stanmore to explore and apply for a Mining Lease over the area of the DAA within EPC 667 between ML 70342 & MDL135 to the South of the Goonyella to DBCT Rail line. Stanmore subsequently, explored and applied for a Mining Lease (ML 700019) over this area, which was granted on 1 March 2018.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>CP Comments</th></tr>
<tr><td></td><td></td><td>

- Stanmore has the relevant licences to operate in the Isaac Plains area.

<table>
<tr><th>Tenure</th><th>Tenement Holder</th><th>Grant Date</th><th>Expiry Date</th><th>Area (Ha)</th></tr>
<tr><td>ML 70342</td><td>Stanmore IP Coal Pty Ltd</td><td>1/12/2005</td><td>31/12/2025</td><td>2141.9</td></tr>
<tr><td>EPC 667</td><td>Fitzroy Australia (CQ) Pty Ltd</td><td>17/10/1997</td><td>30/05/2021</td><td>10807,
(34 Sub-blocks)</td></tr>
<tr><td>ML700018</td><td>Stanmore IP Coal Pty Ltd</td><td>01/03/2018</td><td>31/03/2030</td><td>369.1</td></tr>
<tr><td>ML700019</td><td>Stanmore IP Coal Pty Ltd</td><td>01/03/2018</td><td>31/03/2030</td><td>353.8</td></tr>
</table>

*MDL135 was extinguished on 1st March 2018 upon grant of MLA700018 and 70019 which fully overlie its area

</td></tr>
<tr><td>Exploration Done by Other Parties</td><td>

- *Acknowledgment and appraisal of exploration by other parties.*

</td><td>

- Historically (since the early 1970's), there have been 6 EPC's (EPC 6, 3, 292, 755, 602, 1454) held over the Isaac Plains area.
- A total of 7 parties have undertaken exploration activities within IPC.
- Exploration drilling and geophysical surveys that have been completed within and in close proximity to the Isaac Plains area have been reviewed as part of this report.
- Within the lease boundary and EPC 677 resource zone, a total of 37 drill holes with publicly available information drilled by other parties were reviewed, including drilling for coal Among them, 36 historic holes were considered suitable for use in the geological model.
- An additional 3 drill holes located outside of the lease boundary and EPC resource zone were included to ensure adequate structural control of the resource deposit.
- MGC Resources Australia Pty Ltd conducted 2D dynamite seismic surveys within the area during the early 1990's.

</td></tr>
</table>

Criteria	JORC Code Explanation	CP Comments
Geology	• *Deposit type, geological setting and style of mineralisation.*	• IPC lies within the Permo-Triassic Bowen Basin. The Bowen Basin consists of 10 kilometre (km) thick sequences of volcanic, shallow marine and terrestrial sediments and is categorised back-arc to foreland basin. • The general stratigraphy of IPC includes (oldest to youngest) – • Lower-Permian Reids Dome Beds, Lower-Upper Permian Back Creek Group, Upper Permian Blackwater Group, and Rewan group. • Coal seams occur within the Rangal Coal Measures which are Late Permian in age. These seams dip gently to the east at approximately 5 degrees. • The coal seams found within the Rangal Coal Measures are the Leichhardt, Leichhardt Upper and Leichhardt Lower, and Vermont. • The seams have a cumulative thickness of approximately 7-10 m across the deposit. • The Vermont seam was not included in the resource estimate due to the lack of geological information. The results at hand indicate the coal to be of poorer quality.
Drill Hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* – easting and northing of the drill hole collar – elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar – dip and azimuth of the hole – down hole length and interception depth	• A detailed list of the drill holes used to define the coal quality of the resource in IPC can be found in Appendix C. • Geophysical deviation logs (verticality) are available for all holes. • Shallow holes (open-cut area) have been modelled as vertical holes, i.e. deviation has not been modelled. • The verticality data for the deeper underground holes has been loaded and the holes were modelled with account of any inclination.

Criteria	JORC Code Explanation	CP Comments
	– hole length. • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	
Data Aggregation Methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• It is reported that all seams where multiple coal quality samples were taken were given composite coal quality values based on the length and relative density weighted sum of the raw ply results. • Seams with a raw ash (adb) above 50% are not classified as coal and has not been included as a resource.
Relationship Between Mineralisation Widths and Intercept Lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	• All holes were drilled vertical. • Constraints were applied in thickness modelling to exclude over thickened and under thickened working sections in the model. The variations in the thickness were attributable to faulting.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include,*	• All appropriate diagrams are contained within the main body of the report

Criteria	JORC Code Explanation	CP Comments
	but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	
Balanced Reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• All available exploration data for the Isaac Plains area has been collated and reported.
Other Substantive Exploration Data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• All exploration data was gathered and or utilised in the resource estimation. • Geotechnical logging, sampling and testing from the overburden, interburden, seam roof/floor and coal (such as defect logging, field point load testing and laboratory testing) has been undertaken. • A geostatistical assessment of the Isaac Plains deposit was reported to have been undertaken by Snowden Mining Industry Consultants (Snowdens) in 2010. The original report and date for which were not sited. This study concluded that a drill hole spacing of 250m is "suitable for to confirm the thickness continuity as indicated by the JORC Code of 1999 for the definition of Measured Resources". • Velseis conducted a 2D seismic survey featuring 15 lines to further define faults in the IPC and IPE areas. Historical seismic data as described above was re-evaluated. This work resulted in updated fault interpretations which were used in the creation of the geological model.
Further Work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations*	• Production drilling will be planned based on the mine reserves and mining schedule. • Further resource drilling may be planned for the area of potential underground area, including structural drilling in the fault repeat block in the south-west.

Criteria	JORC Code Explanation	CP Comments
	and future drilling areas, provided this information is not commercially sensitive.	

SECTION 3 ESTIMATION AND REPORTING OF MINERAL RESOURCES

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	*JORC Code Explanation*	CP Comments
Database Integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	• Data was entered in the field by the field Geologist into LogCheck software. • All lithological logs, and coal intersection depths have been reconciled and corrected to the geophysical log. • A review of the historical geophysical logs was conducted as part of the 2015 resource estimate. • All new data was validated by Xenith post correction by exploration geologists. • All bore hole collars were checked against the natural topographic surface and with the exception of approximately 18 drill holes the difference in RL was less than 1m. • Coal Quality data has been checked against lab reports and cross referenced with lithology and ply logs. • As part of the 2015 resource estimate seam picks and sample thicknesses for historical holes were validated and raw qualities were compared to results from the historic resource reports.
Site Visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• Mr T. Turner as Competent Person conducted a site visit in late November 2015. Drilling, logging and sampling procedures and techniques were evaluated. All works sighted during the site visit were found to be of a satisfactory standard. • The Competent Person's familiarity with IPC and stratigraphy is thorough and sufficient. Review of the previous exploration data indicates that the geology is typical of the area.

Criteria	*JORC Code Explanation*	CP Comments
Geological Interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The drill hole density (core and chip) in IPC allows good level of confidence in the nature of seam splitting, seam thickness, coal quality, the location of sub-crops and general location of faults.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The Leichhardt target seam(s) extends approximately 5 km along strike and from 3km (max) in the North to less than 100m (min) in the South, perpendicular to strike with an approximate average cumulative thickness of 3.5m. • The depth of first coal ranges from between 15m in the proximal to the main central thrust fault (uplifted), and 300m in the Northeast. • The current resource extent covers approximately 9.2km^2 • Variability in the coal seam parameters, such as seam thickness and raw coal quality, is reflected in the resource classifications assigned to each seam.
Estimation and Modelling Techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	• The geological model was constructed in ABB Minescape version 5.12 using different modelling algorithms for structure and coal quality parameters. The Finite Element Method (FEM) interpolator with Order: 0 for thickness, 1 for surface and 0 for trend. • The inverse distance squared interpolator was used for raw coal quality modelling. • A maximum extrapolation distance of 3000m from the last data point has been used.

Criteria	*JORC Code Explanation*	CP Comments
	• *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	• Limits were placed on the Resource Estimate with cut-offs at 0.3m thickness for all coal seams within the proposed open-cut region and 1.5m for the remainder of the resource, with the minimum parting thickness of 0.3m to be considered within the seam. Stone bands greater than 0.3m are not included within the seam, so modelling of the seam split occurs.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Coal resource tonnages were estimated using a calculated Preston and Sanders in situ relative density. • Based on the results from coal quality testing, the in-situ moisture has been estimated to be 4.5%. The 4.5% was assumed based on similar Rangal Coal Measure seams located within the area, as well as MHC data. • Coal qualities relating to the resource tonnages are reported on an air-dried basis.

Criteria	JORC Code Explanation	CP Comments
Cut-Off Parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• A maximum raw ash percentage has been applied, where a maximum raw ash of 50%, air-dried basis, has been applied to the resource estimate.
Mining Factors or Assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• Xenith have applied a minimum thickness appropriate to the potential mining method, see '*Modelling technique*' and deem the coal resource have reasonable prospects of economic extraction. • The depth limit of potential open-cut mining varies based on multiple and variable inputs. Presently the limit of open-cut mining is likely to occur between 100 to 150m (depth from surface). If underground mining were to take place, a minimum mining thickness of 1.5m would be required. As such a minimum seam mining thickness was applied to depths >150m, thereby excluding any seams <1.5m thickness from the resource estimate. • Absolute depth of resource was a maximum of 330m from topography.
Metallurgical Factors or Assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• It is Xenith's opinion that at this stage of the project that there are no limiting metallurgical factors. • Isaac Plains has been an operating open-cut mine since 2006. • Some historically reported higher than average Rangal Coal Measures phosphorous percentages may potentially require blending before shipping.

Criteria	*JORC Code Explanation*	CP Comments
Environmental Factors or Assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• It is Xenith's opinion that at this stage of the project that there are no limiting environmental factors. • The coal below "Smoky Creek" has been included in the resource estimate. The CP has regarded this coal as having reasonable prospects for eventual economic extraction due to its shallow nature and seam thickness results. The necessary approvals will need to be obtained to divert this creek, for this coal to be extracted within the open-cut mine.
Bulk Density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• Preston and Sanders In situ Relative Density Estimation – The in situ density of the coal seams has been estimated using the Preston and Sanders in situ relative density estimation equation: $RD(in\ situ) = \frac{RDad \times (100 - Mad)}{\{100 + RDad \times (ISM - Mad) - ISM\}}$ • Inherent (air dried) moisture values have been derived from sampled core intervals. • In situ Moisture was assumed to be 4.5% for the purpose of the resource estimation.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in*	• Three resource categories have been identified within the Isaac Plains area, depending on the level of confidence in the seam structure and continuity plus the level of variability in the coal quality data.

Criteria	*JORC Code Explanation*	CP Comments
	continuity of geology and metal values, quality, quantity and distribution of the data). • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• Drill holes, mined out areas, and seismic sections provide the basis for structural/thickness continuity. • Points of Observation have been used to establish coal quality continuity. • The level of drilling information and presence of an operating mine also assist with the classification of resource categories.
Audits or Reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• No external audits have been performed on the Mineral Resource estimate, but internal QAQC protocols have been followed. • A review of the geological model was undertaken by Palaris in February 2017. The results of which are included in "*Report – Isaac Plains Reconciliation Process*"
Discussion of Relative Accuracy/ Confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.*	• Xenith have assigned three level(s) of confidence to the coal resource estimate, depending on the seam and drill hole spacing, as described in the Chapter 10 of the 2020 JORC Resource report. • A geostatistical review of the coal seam thickness data for the IPC was conducted in 2010 by Snowden. • Factors that could affect accuracy include unknown structures between completed drill holes, seam washouts in roof or inseam stone bands developing. No evidence exists at this point in time for these, apart from what has currently been geologically modelled or exists within the models design database. The inclusion/exclusion of these features was discussed in the report.

Xenith

Criteria	*JORC Code Explanation*	CP Comments
	• *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

APPENDIX B. DRILLING DATA

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
001R	615998.813	7572000.5	238.5	155	Chip	96		2004-2006		22
002R	615999.313	7571499.5	234.2	149	Chip	96		2004-2006		23
003R	615436.688	7571499.5	236.4	113	Chip	96		2004-2006		21
004R	614930.75	7571494	237.52	87	Chip	96		2004-2006		19
005R	615999.313	7570998.5	230.4	95	Chip	96		2004-2006		26
006R	616000.375	7570501	226.6	89	Chip	96		2004-2006		17
007R	615999.688	7570001	224.3	95	Chip	96		2004-2006		18
008R	615999.813	7569499	222.7	72	Chip	96		2004-2006		13
009R	616498.5	7569499	227.2	143	Chip	96		2004-2006		18.5
010R	616498	7568999	228.2	124	Chip	96		2004-2006		21
011R	616500.813	7568498.5	228.2	100	Chip	96		2004-2006		14.5
012R	617003.625	7567504	224.78	89	Chip	96		2004-2006		21
013R	616748.875	7568505.5	231.8	101	Chip	96		2004-2006		25
014R	616994.625	7568505	234.8	149	Chip	96		2004-2006		24
015R	617259.625	7568504.5	236.8	125	Chip	96		2004-2006		35
016R	617493.5	7568504	236.8	146	Chip	96		2004-2006		26
017R	616991.688	7567997	231.69	99	Chip	96		2004-2006		21
018R	617497.188	7568000.5	231.71	83	Chip	96		2004-2006		27.5
019R	617238.563	7567993.5	231.1	65	Chip	96		2004-2006		25
020R	617492.875	7567499	227.86	53	Chip	96		2004-2006		23.5
021R	617743.313	7567505	229.78	71	Chip	96		2004-2006		25
022R	617502	7566898.5	224.54	83	Chip	96		2004-2006		22.5
023R	618002.063	7567002	226.77	29	Chip	96		2004-2006		17.8
024R	618502.5	7567000	229.83	77	Chip	96		2004-2006		24
025R	618405.063	7567499	232.93	60	Chip	96		2004-2006		24
026R	617152.813	7568997	234.02	169	Chip	96		2004-2006		26
027R	615762	7572002	238.8	78	Chip	96		2004-2006		24.5
028R	615497.125	7572001	240.6	48	Chip	96		2004-2006		19
029R	615245.875	7571998	241.8	78	Chip	96		2004-2006		17.15
030R	615249.5	7571513.5	237.2	48	Chip	96		2004-2006		14.5

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
031R	615744.5	7571501	234.7	72	Chip	96		2004-2006		20
032R	615755.813	7570997	229.6	54	Chip	96		2004-2006		17
033R	615489.375	7570999	231.5	78	Chip	96		2004-2006		15
034R	615747.313	7570505	224.4	45	Chip	96		2004-2006		13.5
035R	615506.188	7570507.5	228.1	72	Chip	96		2004-2006		11.84
036R	615750.938	7569998	223.35	78	Chip	96		2004-2006		17.8
037R	616498.125	7571001.5	231.8	96	Chip	96		2004-2006		21
038R	616496	7570503.5	225.9	78	Chip	96		2004-2006		13
039R	616255.313	7570497.5	228	66	Chip	96		2004-2006		17
040R	616512.5	7569956	222	54	Chip	96		2004-2006		12.8
041R	616254.875	7569512.5	225.3	60	Chip	96		2004-2006		19
042R	616249.688	7569003.5	227.6	42	Chip	96		2004-2006		19
043R	616995	7568246	235.76	90	Chip	96		2004-2006		25
044R	616749.813	7568004.5	229	114	Chip	96		2004-2006		19
045R	617798.75	7567500	230.32	72	Chip	96		2004-2006		24
046R	616248.313	7571998.5	238	102	Chip	96		2004-2006		25
047R	616998.563	7571996	234.39	156	Chip	96		2004-2006		20
048C	615997.5	7569999.5	224.2	42	Core		100	2004-2006	No	14
049C	615254.75	7571510.5	237.26	42	Core		100	2004-2006	No	15
050R	617015.188	7571501	232.04	126	Chip	96		2004-2006		21
051R	616968	7570008	230.1	132	Chip	96		2004-2006		19
052R	617014.938	7570480	225.97	126	Chip	96		2004-2006		17
053R	616482.625	7568018.5	225.6	72	Chip	96		2004-2006		24
054C	616990.688	7567997	231.7	40	Core		100	2004-2006	No	22.03
055C	617492.563	7567496.5	227.86	48	Core		100	2004-2006	No	22
056C	616501.688	7568993	228.77	48.03	Core		100	2004-2006	No	22
056CR	616499.438	7568990.5	228.71	60	Core		100	2004-2006	No	
057C	615755.625	7570993.5	229.61	54	Core		100	2004-2006	No	17
058C	615498.313	7572001	240.59	54	Core		100	2004-2006	No	19
059R	614878.875	7571560	237.8	199	Chip	96		2004-2006		15
060C	614884.688	7571562.5	238	77	Core		100	2004-2006	No	15
061C	615495.625	7571773.5	238.76	72.17	Core		63	2004-2006	yes	25.27

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
062C	615705	7571269	232.57	69.17	Core		63	2004-2006	yes	21.54
063C	615659	7570742.5	228.27	42.17	Core		63	2004-2006	yes	14.74
064R	617014.75	7571043.5	227.26	129	Chip	96		2004-2006		17
065R	617032	7570236.5	229.26	66	Chip	96		2004-2006		18
066R	617032.25	7570236.5	229.29	114	Chip	96		2004-2006		19.5
067R	616777.25	7569966	227.8	102	Chip	96		2004-2006		24
068R	615850.625	7570249	223.91	90	Chip	96		2004-2006		17
069R	615991.25	7570242	225.03	54	Chip	96		2004-2006		18
070R	616769.563	7569283.5	230.3	102	Chip	96		2004-2006		21
071R	616280.063	7569283	227.93	96	Chip	96		2004-2006		20
072R	616520.875	7572213	237.91	120	Chip	96		2004-2006		23
073R	616316.688	7572204	239.12	90	Chip	96		2004-2006		24.5
074R	616005	7572220.5	240.81	48	Chip	96		2004-2006		21
075R	615734	7572235.5	241.23	117	Chip	96		2004-2006		20
076R	615495.813	7572246	242.41	102	Chip	96		2004-2006		11
077R	616320.875	7571736.5	235.71	66	Chip	96		2004-2006		20
078R	616008.5	7571757	236.46	120	Chip	96		2004-2006		25
079R	615750.563	7571766	237.08	96	Chip	96		2004-2006		23
080R	615237.625	7571793.5	240.16	90	Chip	96		2004-2006		25
081R	615080.938	7571789.5	240.01	78	Chip	96		2004-2006		18
082R	616277.625	7571249.5	232.56	75	Chip	96		2004-2006		20
083R	616008.688	7571252.5	232.19	84	Chip	96		2004-2006		16
084R	615758.75	7570741	227.29	54	Chip	96		2004-2006		17.5
085R	615499.563	7570747	229.42	84	Chip	96		2004-2006		15
086R	615981	7570759.5	229.1	72	Chip	96		2004-2006		14
087R	615331.938	7572250	243.63	90	Chip	96		2004-2006		15.85
088R	615501.563	7570250.5	223.48	66	Chip	96		2004-2006		5
089R	616282.625	7570225.5	225.11	54	Chip	96		2004-2006		16
090R	616542.25	7570222	224.03	72	Chip	96		2004-2006		19
091R	616545.188	7571256.5	232.95	90	Chip	96		2004-2006		20.5
092R	615514.563	7571268	233.61	108	Chip	96		2004-2006		17.5
093R	615250.625	7571263	234.59	78	Chip	96		2004-2006		11.94

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
094R	616750.938	7572286.5	237.43	132	Chip	96		2004-2006		23
095R	617002.313	7572271	235.05	163	Chip	96		2004-2006		30
096R	616774.375	7572067	235.49	133	Chip	96		2004-2006		22
097R	616748.25	7571503	233.29	108	Chip	96		2004-2006		31
098R	616239.875	7571564	234.44	84	Chip	96		2004-2006		21
099C	616242.75	7571567	234.46	78	Core		100	2004-2006	yes	21
1001R	615952.125	7572108	239.59	90	Chip	96		2013		24
1002R	616007.438	7572126	239.47	96	Chip	96		2013		24
1003R	616053.25	7572139	239.77	84	Chip	96		2013		18
1004R	616033.188	7572132.5	239.66	96	Chip	96		2013		20
1005R	616078.75	7572147.5	239.71	90	Chip	96		2013		20
1006R	615956.75	7572307	241.74	78	Chip	96		2013		24
1007R	616052.125	7572314	241.18	132	Chip	96		2013		13
1008R	616004.188	7572312	241.5	84	Chip	96		2013		10
1009R	615979.563	7572310.5	241.69	72	Chip	96		2013		10
100R	616187.625	7569249.5	227.63	71	Chip	96		2004-2006		19.5
1010R	615992.125	7572311	241.53	66	Chip	96		2013		10
1011R	616021.5	7572313	241.39	54	Chip	96		2013		10
1012R	616036.875	7572316	241.33	48	Chip	96		2013		10
1013R	615940.438	7572505.5	243.68	54	Chip	96		2013		21
1014R	615866.563	7572505	244.01	54	Chip	96		2013		21
1015R	615987.75	7572501.5	243.43	54	Chip	96		2013		24
1016R	616054.938	7572503.5	243.1	48	Chip	96		2013		18.16
1017R	616011.188	7572503	243.26	48	Chip	96		2013		18.1
1018R	616160.375	7571748.5	235.68	120	Chip	96		2013		18
1019R	616198.063	7571744	235.78	96	Chip	96		2013		23
101R	616500.188	7569252	228.87	83	Chip	96		2004-2006		24
1020R	616234.375	7571739.5	235.66	96	Chip	96		2013		21
1021R	616278.813	7571742.5	235.6	84	Chip	96		2013		20
1022R	616217.125	7572209.5	239.51	96	Chip	96		2013		23.5
1023R	616290.938	7571994.5	237.59	102	Chip	96		2013		26
1024R	616332.625	7571994.5	237.37	102	Chip	96		2013		21

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
1025R	616370.438	7571993.5	237.18	102	Chip	96		2013		27
1026R	616409.375	7571996	237.05	102	Chip	96		2013		26
1027R	616462.375	7571990.5	236.92	102	Chip	96		2013		27
1028R	615791.063	7572755	246.46	72	Chip	96		2013		24
1029R	615854.75	7572747.5	246.09	78	Chip	96		2013		27
102R	616085.438	7569248	226.78	83	Chip	96		2004-2006		23
1030R	615915	7572740.5	245.88	84	Chip	96		2013		29
1031R	616034.75	7572731	245.2	72	Chip	96		2013		26
1032R	616095.75	7572724	245.08	72	Chip	96		2013		25
1033R	615942.063	7572734.5	245.86	96	Chip	96		2013		22.38
1034R	615888.625	7572747.5	246.16	102	Chip	96		2013		26
1035R	616598.375	7571126	231.68	102	Chip	96		2013		23
1036R	616636.563	7570910	229.97	102	Chip	96		2013		23
103R	616199.688	7569029.5	227.5	42	Chip	96		2004-2006		17
104R	616098.313	7569018.5	226.91	42	Chip	96		2004-2006		14.5
105R	616000.063	7568997.5	225.87	66	Chip	96		2004-2006		17.25
106R	615922.125	7569255.5	224.73	36	Chip	96		2004-2006		13.5
107R	615964.188	7569261.5	225.2	60	Chip	96		2004-2006		20
108R	616515.25	7571771	235.27	87	Chip	96		2004-2006		21
109R	616724.438	7571768	234.55	108	Chip	96		2004-2006		22
110R	616968.438	7571761	231.8	132	Chip	96		2004-2006		20
111R	616474.125	7571498	233.67	95	Chip	96		2004-2006		24
112R	617495.75	7572025	228.9	213	Chip	96		2004-2006		21
113R	617476.688	7571525	227.57	110	Chip	96		2004-2006		27.5
114R	617469.75	7571532	227.85	93	Chip	96		2004-2006		27.5
115R	616479.25	7569751.5	224.09	51	Chip	96		2004-2006		20.3
116R	616765.188	7569735	228.41	120	Chip	96		2004-2006		16.5
117R	617001.625	7569728.5	231.59	126	Chip	96		2004-2006		19
118R	617262.75	7569738.5	234.92	132	Chip	96		2004-2006		31
119R	617241.563	7569521	234.58	144	Chip	96		2004-2006		25
120C	615746.563	7571498	234.84	72	Core		100	2004-2006	yes	20
121R	616212.938	7571019	231.76	90	Chip	96		2004-2006		24

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
122C	616217.625	7571019.5	231.66	90	Core		100	2004-2006	yes	24
123R	616768.5	7571048	228.64	108	Chip	96		2004-2006		18
124C	616765.25	7571048.5	228.59	102	Core		100	2004-2006	yes	19
125C	616505.5	7571995.5	237.01	108	Core		100	2004-2006	yes	21
126C	615999.188	7571996.5	238.75	126	Core		100	2004-2006	yes	28
127C	615753.125	7570510	225.54	42	Core		100	2004-2006	yes	13.5
128C	616256.563	7570493.5	228.07	66	Core		100	2004-2006	yes	17
129C	616514.938	7569955.5	222.27	54	Core		100	2004-2006	yes	12.8
130R	616716.125	7569565.5	228.74	114	Chip	96		2004-2006		21
131R	616969.313	7569505	231.49	120	Chip	96		2004-2006		21
132R	617523.25	7569559.5	238.65	174	Chip	96		2004-2006		15
133R	617299.75	7570007	234.49	141	Chip	96		2004-2006		24
134R	617496.688	7570001.5	237.99	168	Chip	96		2004-2006		23
135R	616202.688	7570028	222.78	48	Chip	96		2004-2006		17
136R	616512.188	7570746	229.24	90	Chip	96		2004-2006		23
137R	616998.813	7571266	228.6	126	Chip	96		2004-2006		16
138R	616741.188	7571245	231.78	96	Chip	96		2004-2006		24
139R	616244.375	7570760	230.1	78	Chip	96		2004-2006		24
140R	616714.25	7570503	223.55	90	Chip	96		2004-2006		19
141C	616717.75	7570504	223.57	90	Core		100	2004-2006	yes	19
142C	616256.5	7569515.5	225.51	60	Core		100	2004-2006	yes	20
143R	616764.375	7570250	223.33	108	Chip	96		2004-2006		16
144R	616757.188	7570735	225.68	102	Chip	96		2004-2006		15
145R	616992	7570753	226.45	123	Chip	96		2004-2006		22.5
146R	616010.438	7569742	222.18	90	Chip	96		2004-2006		10
147R	616302	7569799.5	223.4	96	Chip	96		2004-2006		13
148R	616962.625	7569252.5	231.46	120	Chip	96		2004-2006		24
149R	617238.625	7569219	234.86	168	Chip	96		2004-2006		19
150R	617294.688	7569014	235.2	132	Chip	96		2004-2006		21
151R	616547.75	7568793	229.07	90	Chip	96		2004-2006		21
152R	616742.875	7569006	230.13	84	Chip	96		2004-2006		18
153R	617045.438	7568807	233.89	96	Chip	96		2004-2006		22

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
154R	617272.438	7568725	236.27	114	Chip	96		2004-2006		17
155R	617525.875	7568805.5	237.32	156	Chip	96		2004-2006		36
156R	618038.625	7568265	239.81	73	Chip	96		2004-2006		
157R	618249.25	7568000.5	238.64	132	Chip	96		2004-2006		49
158R	617746.875	7568006.5	234.74	109	Chip	96		2004-2006		29.5
159R	618030.25	7568005	237.42	123	Chip	96		2004-2006		21
160R	617725.625	7568260.5	236.17	99	Chip	96		2004-2006		27
161R	617498.313	7568252.5	235.1	111	Chip	96		2004-2006		29
162C	616144.25	7569747.5	223.25	32.45	Core		63	2004-2006	yes	12.84
163C	616807.438	7568818	231.52	62.57	Core		63	2004-2006	yes	
164C	616404.875	7568778	227.88	29.99	Core		63	2004-2006	yes	17.3
165R	617017.688	7567746	227.83	102	Chip	96		2004-2006		17.22
166R	617500.313	7567751	229.97	66	Chip	96		2004-2006		27
167C	617174.25	7567751	228.13	38.48	Core		63	2004-2006	yes	15.34
168R	616742	7568243	231.67	120	Chip	96		2004-2006		22
169R	617252.75	7568249.5	235.4	108	Chip	96		2004-2006		24
170R	617749.813	7567750.5	232.41	90	Chip	96		2004-2006		32
171R	617994.75	7567746.5	234.83	96	Chip	96		2004-2006		25
172R	618252.313	7567748.5	235.46	111	Chip	96		2004-2006		32.24
173R	617253.688	7567493.5	225.06	108	Chip	96		2004-2006		13
174R	617982.75	7567508	232.01	78	Chip	96		2004-2006		24
175R	618259.625	7567552.5	233.13	93	Chip	96		2004-2006		24
176R	617250.75	7567250	223.6	60	Chip	96		2004-2006		21
177R	617494.813	7567231.5	225.18	114	Chip	96		2004-2006		17
178R	617746.938	7567249.5	226.92	120	Chip	96		2004-2006		24
179R	618029.688	7567318	229.9	54	Chip	96		2004-2006		29
180R	618250	7567250	229.95	28	Chip	96		2004-2006		24
181C	617163.625	7568997.5	234.15	113	Core		100	2004-2006	yes	23
182C	616190.25	7569040	227.46	42	Core		100	2004-2006	yes	18
183R	616570	7568470	229.14	23	Chip	96		2004-2006		13
184C	616595.875	7568496.5	229.65	36	Core		100	2004-2006	yes	16
185C	617002.563	7568504.5	235.14	84	Core		100	2004-2006	yes	28

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
186R	616616.5	7568013	227.48	102	Chip	96		2004-2006		16
187C	616692.875	7568002	228.54	36	Core		100	2004-2006	yes	17
188C	617494.125	7568006	231.88	78	Core		100	2004-2006	yes	25
189C	617323.125	7567485	225.59	36	Core		100	2004-2006	yes	14
190C	617996.25	7567504.5	232.06	72	Core		100	2004-2006	yes	25
191C	617737.063	7567251	227.2	42	Core		100	2004-2006	yes	24
192C	617482.813	7568505	237.18	144	Core		100	2004-2006	yes	27
193R	616994.625	7569007.5	232.28	102	Chip	96		2004-2006		29
194C	616724.875	7569576.5	228.8	112	Core		100	2004-2006	yes	22
195R	620100	7563200	100	129	Chip	96		2004-2006		23
196C	616747.938	7571506.5	233.27	102	Core		100	2004-2006	yes	31
197R	614854.313	7571805	240.55	60	Chip	96		2004-2006		16
198R	615127.875	7572263	244.41	84	Chip	96		2004-2006		19
199R	614757.938	7572276.5	246.44	60	Chip	96		2004-2006		18
200L	615283.75	7571260.5	234.6	27	Chip	96		2004-2006		14
201L	615309.438	7571260.5	234.67	29	Chip	96		2004-2006		15
202L	615329.438	7571020.5	232.06	26	Chip	96		2004-2006		15
203L	615375.313	7571019	232.03	29	Chip	96		2004-2006		17
204L	615469.438	7570752	229.54	29	Chip	96		2004-2006		15
205L	615499.938	7570752.5	229.49	29	Chip	96		2004-2006		13.5
206L	615563.875	7570497	228.03	26	Chip	96		2004-2006		13.8
207L	615605.188	7570508.5	227.89	29	Chip	96		2004-2006		16
208L	615698.875	7570251	223.66	29	Chip	96		2004-2006		15
209L	615196.375	7571262.5	234.21	17	Chip	96		2004-2006		15
210L	615228.5	7571263.5	234.12	20	Chip	96		2004-2006		14
211L	615292.813	7571023.5	231.85	26	Chip	96		2004-2006		17.68
212L	615275.875	7571026.5	232	26	Chip	96		2004-2006		17.44
213L	615234.5	7571121.5	232.76	20	Chip	96		2004-2006		10.7
214L	615374.813	7570757.5	229.54	29	Chip	96		2004-2006		15
215L	615417.125	7570753	229.54	20	Chip	96		2004-2006		15.5
216L	615443.625	7570752.5	229.6	20	Chip	96		2004-2006		15.8
217L	615487.75	7570517	228.29	26	Chip	96		2004-2006		11.2

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
218L	615454.938	7570518.5	228.11	20	Chip	96		2004-2006		10.1
219L	615445.25	7570630.5	228.83	23	Chip	96		2004-2006		13.3
220L	615603.563	7570253.5	223.35	23	Chip	96		2004-2006		11.76
221L	615697.25	7570007.5	223.36	23	Chip	96		2004-2006		20
222L	615722.375	7570001.5	223.35	7	Chip	96		2004-2006		
223L	615854.438	7569984	223.76	29	Chip	96		2004-2006		13.5
224L	615798.938	7569985.5	223.67	29	Chip	96		2004-2006		19.2
225L	615725.125	7570136	223.33	23	Chip	96		2004-2006		14.5
226L	615690.438	7570134.5	223.23	23	Chip	96		2004-2006		12.2
227L	615547.438	7570419.5	226.63	29	Chip	96		2004-2006		12.5
228L	615471.813	7570418	226.81	23	Chip	96		2004-2006		13.5
229L	615435.75	7570418.5	226.83	20	Chip	96		2004-2006		18
230L	615379.75	7570875.5	230.52	17	Chip	96		2004-2006		16
231R	621002.375	7561002	203.31	90	Chip	96		2004-2006		20
232R	623798.375	7559001.5	200.27	204	Chip	96		2004-2006		30
233R	622990.313	7560020.5	205.29	186	Chip	96		2004-2006		20
234R	623970.313	7560007	205.39	198	Chip	96		2004-2006		22
235R	620999.5	7562002	205.18	60	Chip	96		2004-2006		24
236R	619540	7564050	0	42	Chip	96		2004-2006		19
237R	619203.438	7564901	243.08	60	Chip	96		2004-2006		22
238R	620002	7566015	247.29	139	Chip	96		2004-2006		57
239P	616544.813	7571999.5	236.52	105	Piezo	96		2004-2006		23
240P	616572.813	7571996.5	236.32	79	Piezo	96		2004-2006		22.5
241L	615428.188	7570875	230.44	17	Chip	96		2004-2006		11.44
242L	615497.5	7570876	230.5	23	Chip	96		2004-2006		15.4
243L	615871.75	7569823	223.05	32	Chip	96		2004-2006		11.5
244L	615789.563	7569814	222.91	23	Chip	96		2004-2006		13.7
245L	616089	7569498.5	224.66	23	Chip	96		2004-2006		12.8
246L	616089.375	7569355	226.14	8	Chip	96		2004-2006		
247L	616132.188	7569354	226.83	29	Chip	96		2004-2006		18
248L	615961.813	7569003	225.51	29	Chip	96		2004-2006		15.5
249L	616330.188	7568783.5	226.9	26	Chip	96		2004-2006		17.4

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
250L	616438.375	7568510.5	227.7	23	Chip	96		2004-2006		21
251L	616498.875	7568496	228.3	23	Chip	96		2004-2006		18
252L	616570.375	7568498	229.29	29	Chip	96		2004-2006		18.48
253L	616467.563	7568684	228.32	32	Chip	96		2004-2006		19
254L	616569.438	7568011	226.82	26	Chip	96		2004-2006		18
255L	616553.188	7568139.5	227.78	23	Chip	96		2004-2006		22
256L	616606	7568138.5	228.69	32	Chip	96		2004-2006		18.5
257L	616567.375	7568264	228.91	26	Chip	96		2004-2006		16.55
258L	616556.063	7568372	229.07	26	Chip	96		2004-2006		16
259L	616996.938	7567748	227.68	23	Chip	96		2004-2006		15.69
260L	616859.875	7567842	227.98	20	Chip	96		2004-2006		
261L	616963.375	7567843.5	228.89	26	Chip	96		2004-2006		18
262L	617188.938	7567498	226.09	32	Chip	96		2004-2006		17.93
263L	617087	7567596	226.36	29	Chip	96		2004-2006		20.2
264L	617118.938	7567594.5	226.49	32	Chip	96		2004-2006		18.08
265L	617408.938	7567242	224.02	26	Chip	96		2004-2006		17.2
266L	617285.875	7567365	224.07	20	Chip	96		2004-2006		12.04
267L	617359.438	7567363	225.21	29	Chip	96		2004-2006		16.53
268L	617627.688	7567091.5	224.56	23	Chip	96		2004-2006		17
269L	617597.438	7567089.5	224.43	29	Chip	96		2004-2006		19
270L	617996.313	7566993	226.68	26	Chip	96		2004-2006		18.74
271P	616563.188	7569972.5	221.54	74				2004-2006		15
272P	616545.438	7569948.5	221.8	42				2004-2006		14
273P	617256.75	7569503.5	234.78	126.5				2004-2006		26
274P	616471.563	7568045	225.98	50.8				2004-2006		27
275P	617980.125	7567535	232.24	100				2004-2006		22
276P	618005.813	7567550	232.59	66.8				2004-2006		28
277P	618012.875	7567518	232.08	57.8				2004-2006		27
278P	617265.313	7569474.5	234.99	109.7				2004-2006		18
279R	614744.625	7571253	234.27	54	Chip	96		2004-2006		18
280P	615223.438	7571306	234.85	45				2004-2006		17
281L	617781.125	7566998.5	225.58	35	Chip	96		2004-2006		24.1

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
282L	617722.188	7566955.5	223.75	26	Chip	96		2004-2006		20.4
283L	617524.938	7567059	223.36	26	Chip	96		2004-2006		18.32
284L	616393.813	7568679.5	227.74	26	Chip	96		2004-2006		16.69
285L	616371.313	7568679.5	227.38	26	Chip	96		2004-2006		16.6
286L	616100.25	7568901	226.66	29	Chip	96		2004-2006		17.57
287L	616044.75	7568880	226.33	26	Chip	96		2004-2006		17.5
288L	616007.75	7569181	226.12	41	Chip	96		2004-2006		18
289L	615960.375	7569176.5	225.41	26	Chip	96		2004-2006		19.8
290L	616065	7569354	225.76	23	Chip	96		2004-2006		17
291L	616092.188	7569353	226.23	29	Chip	96		2004-2006		20.7
292L	615796.75	7569905	223.63	26	Chip	96		2004-2006		16.5
293L	615756.563	7569895.5	223.39	20	Chip	96		2004-2006		15.3
294L	615093.938	7571529	237.58	29	Chip	96		2004-2006		19.3
295L	615158.75	7571789	239.97	32	Chip	96		2004-2006		9
296L	615046.438	7571537.5	237.8	20	Chip	96		2004-2006		16
297L	615096.813	7571631.5	238.78	32	Chip	96		2004-2006		17.5
298L	615045.5	7571627.5	238.97	29	Chip	96		2004-2006		19.48
299L	615239.938	7571899	240.97	23	Chip	96		2004-2006		14.04
300P	616767.5	7568237.5	232.06	80.65				2004-2006		22.5
301R	613964.313	7571687	240.41	209	Chip	96		2004-2006		21
302R	613061.938	7571799	235.88	200	Chip	96		2004-2006		17.8
303R	612408.438	7571281	230.14	153	Chip	96		2004-2006		21
304R	614272	7567122.5	214.65	177	Chip	96		2004-2006		17
305R	615005.688	7567911	219.73	129	Chip	96		2004-2006		16
306R	619000	7566000	0	51	Chip	96		2004-2006		18
310L	615283.25	7571895.5	241.04	26	Chip	96		2004-2006		19.1
311L	615312.375	7572001.5	242.21	29	Chip	96		2004-2006		16.84
312L	615478.875	7572244	242.52	20	Chip	96		2004-2006		13.8
313L	615390.625	7572146.5	242.6	41	Chip	96		2004-2006		14.24
314L	615133.25	7571787	239.73	29	Chip	96		2004-2006		13
315L	615173.875	7571393.5	235.8	29	Chip	96		2004-2006		21.3
316L	615113.875	7571382	235.79	23	Chip	96		2004-2006		18.72

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
322R	615257	7570247	224.4	45	Chip	96		2004-2006		17.54
324R	615498.188	7569745.5	222.47	57	Chip	96		2004-2006		16.9
325C	615500.125	7569752	222.54	33	Core		100	2004-2006	no	16
326R	615962.813	7569641.5	222.4	60	Chip	96		2004-2006		12
327R	615404.375	7571260	234.45	42	Chip	96		2004-2006		23
328C	615403.25	7571255	234.41	39	Core		100	2004-2006	yes	23
329R	615471.25	7571264	233.95	60	Chip	96		2004-2006		22
330C	615735.125	7572233.5	241.27	63	Core		100	2004-2006	yes	20
331R	615725	7570499.5	225.65	31	Chip	96		2004-2006		14
332C	615719.625	7570504	225.73	39	LD		200	2004-2006		14
333C	615719.5	7570500	225.67	41.77	LD		200	2004-2006		14
334C	616253.125	7570494.5	228.16	64.69	LD		200	2004-2006		17
335C	616253.75	7570490	228.16	64	LD		200	2004-2006		17
336C	617740.375	7567246	226.81	45	LD		200	2004-2006		24
337C	617740.25	7567251.5	226.86	45	LD		200	2004-2006		24
338R	618024.563	7567124.5	227.5	36	Chip	96		2004-2006		17.5
339R	615739.313	7568987.5	223.61	78	Chip	96		2004-2006		24
340R	615528.063	7569003	222.41	28	Chip	96		2004-2006		17.7
341R	615616.625	7568994.5	222.81	28	Chip	96		2004-2006		18
342C	615739.063	7568984	223.65	75	Core		100	2004-2006	no	24
343R	616235.563	7567998	225.09	52	Chip	96		2004-2006		23
344C	616235.5	7568001.5	225.1	48	Core		100	2004-2006	no	23
345R	616247.875	7568492.5	226.63	64	Chip	96		2004-2006		24
346R	615929.5	7568497.5	225.68	40	Chip	96		2004-2006		20.1
347R	616071.313	7568508	226.26	64	Chip	96		2004-2006		21
348C	616071.688	7568511	226.3	57.58	Core		100	2004-2006	no	19
349R	616756.313	7567503	223.48	46	Chip	96		2004-2006		18.5
350C	618025.313	7567123	227.42	36	Core		100	2004-2006	yes	15
351C	617749.688	7567749	232.39	90	Core		100	2004-2006	yes	33
352C	616499.563	7569250.5	228.98	78	Core		100	2004-2006	yes	24
353C	616756.563	7567500.5	223.47	40.77	Core		100	2004-2006	no	18.5
354R	616998.5	7567259.5	223.47	40	Chip	96		2004-2006		16

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
355C	616998.25	7567261.5	223.44	46	Core		100	2004-2006	no	16
356R	617271.688	7566966	223.81	40	Chip	96		2004-2006		20
357C	617271.75	7566963.5	223.91	40	Core		100	2004-2006	no	20
358R	614682.125	7571580.5	237.56	46	Chip	96		2004-2006		21
359R	614989.375	7572247	245.21	63	Chip	96		2004-2006		14.5
360C	614988.625	7572249.5	245.34	70	Core		100	2004-2006	no	14.5
361R	615072.063	7571990.5	242.05	64	Chip	96		2004-2006		19
362R	614947	7571257	234.91	52	Chip	96		2004-2006		18
363C	614947.25	7571258.5	235	51	Core		100	2004-2006	no	18
364R	615212.313	7570609.5	228.41	64	Chip	96		2004-2006		12.3
365R	615386.5	7570271	224.48	58	Chip	96		2004-2006		13
366C	615384.063	7570270.5	224.53	58	Core		100	2004-2006	no	13
367R	615399.813	7570006.5	222.44	58	Chip	96		2004-2006		16
368R	615609.063	7569504.5	222.07	52	Chip	96		2004-2006		14.51
369R	615725.438	7569263.5	223.36	46	Chip	96		2004-2006		14.21
370R	615818.375	7568723.5	224.42	64	Chip	96		2004-2006		21.5
371R	616203.75	7568240	225.72	82	Chip	96		2004-2006		20
372R	616501.5	7567757.5	224.53	46	Chip	96		2004-2006		19
373R	614914.813	7571008	231.97	52	Chip	96		2004-2006		18
374R	615096.5	7570813.5	230.11	52	Chip	96		2004-2006		18.2
375C	615097.75	7570811.5	230.1	52	Core		100	2004-2006	no	17.5
376C	614682.5	7571582	237.53	41	Core		100	2004-2006	no	21
377C	614883	7571558.5	237.94	76	Core		100	2004-2006	no	15
400R	615364.438	7571090	232.99	35	Chip	96		2004-2006		10.5
401C	615364.688	7571092	232.98	35	Core		100	2004-2006	yes	10.5
402C	615553.438	7570722	229.32	35	Core		100	2004-2006	yes	13.5
403R	615712.5	7570374	224.22	41	Chip	96		2004-2006		18
404C	615709.313	7570374.5	224.27	40	Core		100	2004-2006	yes	18
405C	615777.938	7570161.5	223.6	35	Core		100	2004-2006	yes	15
406C	615851.313	7569968.5	223.76	39	Core		100	2004-2006	yes	14.5
407C	616074.5	7569043.5	226.76	35	Core		100	2004-2006	No	20.39
408C	615284.813	7571375	235.84	38	Core		100	2004-2006	No	15

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
409C	615519.875	7570941	231	32	Core		100	2004-2006	No	15.7
410C	615627.125	7570542	228.02	35	Core		100	2004-2006	yes	14
411C	615174.875	7571660	238.69	41	Core		100	2004-2006	yes	16.5
412C	615300.063	7571816	240.42	58	Core		100	2004-2006	yes	19
413C	615419.125	7571986	241.25	41	Core		100	2004-2006	yes	16
414R	615524.875	7572155	241.46	47	Chip	96		2004-2006		19
415C	615524.375	7572153.5	241.45	49	Core		100	2004-2006	yes	19
416R	615611.813	7572340.5	242.57	41	Chip	96		2004-2006		22
417C	615611.625	7572339	242.55	44	Core		100	2004-2006	yes	22.4
418R	615626.563	7572547	245.08	47	Chip	96		2004-2006		18.5
419C	615627.125	7572546.5	245.08	51	Core		100	2004-2006	yes	18.5
420R	615539.688	7572730.5	248.01	50	Chip	96		2004-2006		15
421C	615539.75	7572730	248.01	47.5	Core		100	2004-2006	yes	15
422R	615535.813	7572944	250.33	53	Chip	96		2004-2006		21
423C	615535.875	7572943.5	250.3	56	Core		100	2004-2006	yes	21
424R	616225.563	7568908.5	227.27	33	Chip	96		2004-2006		21.8
425C	616274.875	7568937	227.53	35	Core		100	2004-2006	yes	22.18
426R	616719.125	7568407	231.72	41	Chip	96		2004-2006		16.2
427R	616867.063	7568397.5	233.75	60	Chip	96		2004-2006		26
428R	617018.188	7568399	235.84	81	Chip	96		2004-2006		29
429R	616795.188	7568504.5	232.59	60	Chip	96		2004-2006		24
430R	616888.188	7568506	233.71	108	Chip	96		2004-2006		19
431R	616947.063	7568254	235.02	72	Chip	96		2004-2006		20
432R	617040.625	7568253.5	236.11	84	Chip	96		2004-2006		29
433R	616895.75	7568263.5	234.21	67	Chip	96		2004-2006		28
434R	616852.625	7568256.5	233.47	65	Chip	96		2004-2006		21
435R	616972.063	7568194.5	234.73	74	Chip	96		2004-2006		27
436R	616984.313	7568098.5	233.51	68	Chip	96		2004-2006		25
437R	616993.125	7568048.5	232.75	65	Chip	96		2004-2006		27
438R	616627	7568254	229.85	29	Chip	96		2004-2006		18.38
439R	616535.125	7568629.5	229.04	35	Chip	96		2004-2006		18
440R	617054.25	7567855	229.52	38	Chip	96		2004-2006		17

Xenith

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
455C	616626.438	7568252.5	229.8	35	Core		100	2004-2006	yes	15.5
456C	616532.563	7568630.5	229.04	41	Core		100	2004-2006	yes	16
457R	616995.625	7568287.5	236.21	76	Chip	96		2004-2006		22
458R	617098.625	7568258	236.25	86	Chip	96		2004-2006		25.5
459R	616802.313	7568651.5	231.84	59	Chip	96		2004-2006		22
460R	616726.125	7568512.5	231.62	89	Chip	96		2004-2006		27
461R	617163.313	7567689	227.24	38	Chip	96		2004-2006		14
462R	617255	7567510	225.44	41	Chip	96		2004-2006		14
463R	616841.313	7568025	230.78	52	Chip	96		2004-2006		22
464R	616838.188	7568122	232.25	59	Chip	96		2004-2006		21
465R	617399.563	7567359.5	225.37	35	Chip	96		2004-2006		16
466R	617544.375	7567235	225.15	41	Chip	96		2004-2006		18
467R	617710.188	7567145	225.29	41	Chip	96		2004-2006		21
468R	617904.125	7567129.5	226.8	41	Chip	96		2004-2006		19
469C	617053.5	7567852.5	229.5	41	Core		100	2004-2006	yes	15
470C	617163.688	7567690.5	227.26	41	Core		100	2004-2006	yes	15.3
471C	617256.625	7567509	225.4	35	Core		100	2004-2006	yes	14
472C	617399.938	7567362	225.39	35	Core		100	2004-2006	yes	15
473C	617543.813	7567233.5	225.11	41	Core		100	2004-2006	yes	17.5
474C	617710.063	7567143.5	225.23	34	Core		100	2004-2006	yes	20.5
475C	617904.375	7567127.5	226.62	41	Core		100	2004-2006	yes	19
476C	615286.875	7571374.5	235.81	40	Core		100	2004-2006	yes	14
477C	615518.313	7570940.5	230.96	35	Core		100	2004-2006	yes	12
478C	615420.313	7571985.5	241.27	41	Core		100	2004-2006	yes	17
479R	616102.188	7571398.5	233.36	98	Chip	96		2004-2006		23
480R	616159.375	7571392.5	233.29	101	Chip	96		2004-2006		24.3
481R	616209.25	7571387	233.26	101	Chip	96		2004-2006		20
482R	616257.438	7571382	233.41	149	Chip	96		2004-2006		27
483R	615498.313	7571479	235.81	101	Chip	96		2004-2006		19
484R	615288.563	7571507	237.03	47	Chip	96		2004-2006		15
485R	615896.75	7571428.5	233.78	89	Chip	96		2004-2006		21
486R	616313.875	7572107	238.5	161	Chip	96		2004-2006		26

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
487R	616200.063	7572112	239.13	80	Chip	96		2004-2006		23.8
488R	616125.813	7572013.5	238.48	77	Chip	96		2004-2006		17.2
489R	615799.875	7571906	237.99	107	Chip	96		2004-2006		22
490R	615687	7571904	238.4	95	Chip	96		2004-2006		22.8
491R	615646.063	7571788	237.8	95	Chip	96		2004-2006		19.7
492R	616801.5	7568118	231.58	53	Chip	96		2004-2006		26
493R	617055	7568140.5	234.56	77	Chip	96		2004-2006		21.7
501L	615351.313	7572997.5	252.85	32	Chip	96		2004-2006		17.2
502L	615028.188	7572997.5	254.83	17	Chip	96		2004-2006		16.3
503L	615162.813	7572875.5	252.72	21	Chip	96		2004-2006		9.9
504L	615329	7572875	251.29	30	Chip	96		2004-2006		15
505L	615170.375	7572751.5	250.59	30	Chip	96		2004-2006		12
506L	615296.938	7572755	249.9	33	Chip	96		2004-2006		12
507L	615454.25	7572625.5	246.69	42	Chip	96		2004-2006		10
508L	615283.25	7572623.5	248.08	33	Chip	96		2004-2006		12
509L	615373.938	7572500.5	245.67	33	Chip	96		2004-2006		10
510L	615569.75	7572500	244.44	33	Chip	96		2004-2006		13
511L	615374.938	7572373	244.25	33	Chip	96		2004-2006		15.2
512L	615499.688	7572377.5	243.49	39	Chip	96		2004-2006		18
513L	615344	7572249	243.51	27	Chip	96		2004-2006		16.26
514L	615462.813	7572244.5	242.53	30	Chip	96		2004-2006		12.44
515L	615417.375	7572124	242.19	39	Chip	96		2004-2006		19
516L	615273.625	7572124.5	243.16	39	Chip	96		2004-2006		15
517L	615199.938	7571998.5	241.67	24	Chip	96		2004-2006		14.7
518L	615296.875	7572001	242	30	Chip	96		2004-2006		15.04
519L	615256	7571874.5	240.69	27	Chip	96		2004-2006		16.26
520L	615337.375	7571875	241.07	36	Chip	96		2004-2006		19
521L	615100.25	7571715.5	239.26	39	Chip	96		2004-2006		18
522L	615081.188	7571450	237.02	27	Chip	96		2004-2006		17.32
523L	615166.438	7571326	234.45	30	Chip	96		2004-2006		9.2
524L	615238.5	7571198.5	233.52	24	Chip	96		2004-2006		13.1
525C	615402.125	7571245.5	234.34	40	LD		200	2004-2006		19

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
526C	615399.813	7571260	234.48	40	LD		200	2004-2006		20
527C	615400.813	7571276	234.58	34.44	LD		200	2004-2006		21
528C	615391.25	7571246.5	234.34	40	LD		200	2004-2006		9
529C	615409.25	7571253.5	234.36	40	LD		200	2004-2006		19
530L	615245.375	7571077	232.3	33	Chip	96		2004-2006		15.4
531L	615367	7570948.5	231.13	27	Chip	96		2004-2006		6
532L	615444.125	7570816	229.93	30	Chip	96		2004-2006		10.48
533L	615521.5	7570696.5	229.4	33	Chip	96		2004-2006		12
534L	615442.125	7570695	229.22	30	Chip	96		2004-2006		12
535L	615442.438	7570585.5	228.52	27	Chip	96		2004-2006		9
536L	615459.375	7570478.5	227.7	30	Chip	96		2004-2006		15
537L	615414.438	7570470	227.46	27	Chip	96		2004-2006		15.04
538L	615537.438	7570366	225.49	27	Chip	96		2004-2006		9
539L	615483.25	7570355	225.43	21	Chip	96		2004-2006		10.98
540L	615464.125	7570367	225.79	21	Chip	96		2004-2006		10.32
541L	615697.938	7570180.5	223.16	27	Chip	96		2004-2006		13
542R	615401	7571278.5	234.6	87	Chip	96		2004-2006		15
543R	616310	7571375	233	148	Chip	96		2004-2006		27
544R	616360	7571375	232.9	148	Chip	96		2004-2006		26
545L	615675.375	7570049.5	223.27	24	Chip	96		2004-2006		13.1
546L	615728.813	7570049.5	223.27	18	Chip	96		2004-2006		16.5
547L	615770.688	7570050.5	223.41	30	Chip	96		2004-2006		16.5
548L	615510.875	7570360.5	225.46	24	Chip	96		2004-2006		11.62
549L	615774.25	7569758.5	222.84	27	Chip	96		2004-2006		16
550L	615806.063	7569770	222.35	27	Chip	96		2004-2006		13.8
551L	615819.125	7569682	220.4	21	Chip	96		2004-2006		11
552L	615842.063	7569684	220.37	24	Chip	96		2004-2006		10.7
553L	615862.688	7569577	219.92	24	Chip	96		2004-2006		6
554L	615900.75	7569568	220.9	27	Chip	96		2004-2006		9
555L	615929.188	7569560.5	221.23	27	Chip	96		2004-2006		8
556L	616057.563	7569546.5	222.95	24	Chip	96		2004-2006		5
557L	616011.5	7569528	222.51	21	Chip	96		2004-2006		10.4

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
558L	616910.063	7567920	229.66	30	Chip	96		2004-2006		16
559L	616881.875	7567932	229.6	21	Chip	96		2004-2006		17.24
560L	616930.125	7567815.5	228.09	27	Chip	96		2004-2006		15
561L	617070.25	7567674.5	227.11	27	Chip	96		2004-2006		17
562L	617121.563	7567545.5	226.03	21	Chip	96		2004-2006		19
563L	617137.5	7567551	226.25	27	Chip	96		2004-2006		16.98
564L	617224.188	7567429	225.09	27	Chip	96		2004-2006		15.68
565L	617346.188	7567303	224.47	24	Chip	96		2004-2006		14.56
566L	617360.688	7567325	224.7	27	Chip	96		2004-2006		17.56
567L	617410.375	7567174.5	223.41	27	Chip	96		2004-2006		16.82
568L	617429.313	7567182.5	223.62	30	Chip	96		2004-2006		16
569L	617681.313	7567011.5	225.23	26	Chip	96		2004-2006		18
570L	617644.75	7566970.5	223.69	27	Chip	96		2004-2006		17.5
571P	617269.563	7566976	223.72	61				2004-2006		19
586LD	615520	7570943	230.75	29.73	LD		200	2004-2006		12
587LD	615522	7570943	230.74	30.39	LD		200	2004-2006		12
588LD	615518	7570941	230.76	30.39	LD		200	2004-2006		12
589LD	615519	7570946	230.76	29.75	LD		200	2004-2006		12
590LD	615523	7570947	230.74	30.5	LD		200	2004-2006		12
591LD	615524	7570944	230.72	31	LD		200	2004-2006		16.91
602C	615230	7571575	237.93	41	Core		100	2004-2006	no	15.5
603R	615360	7571235	234.27	31	Chip	96		2004-2006		13
604C	615445	7571035	231.66	37	Core		100	2004-2006	no	14
605C	615446	7571032	231.65	41	Core		100	2004-2006	no	14.87
606C	615592	7570640	228.8	37	Core		100	2004-2006	no	13
BC041	615549.625	7573241	253.71	149	Chip	96		Historic		17
BC042	615032.813	7573307.5	261.65	95	Chip	96		Historic		31
BC043	616495.5	7573113.5	244.25	173	Chip	96		Historic		25
BC045	616967.438	7573058	238.92	125	Chip	96		Historic		22
BC048	617518.063	7573014	232.84	209	Chip	96		Historic		20
BC049	615995.75	7573183	250.42	59	Chip	96		Historic		29
BC050	616499.375	7572385	239.59	101	Chip	96		Historic		24

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
BC051	615993.75	7572388.5	242.23	16	Chip	96		Historic		
BC059	616498.5	7572699.5	241.98	89	Chip	96		Historic		20
BC082	616997.688	7572701.5	237.4	129	Chip	96		Historic		29
BC089	617245	7573100.5	235.25	153	Chip	96		Historic		26
BC095	616503.313	7571996.5	236.73	114	Chip	96		Historic		21
BC096	616496.5	7571350	232.29	150	Chip	96		Historic		24
BC401	616216.75	7572713.5	244.83	84	Chip	96		Historic		17
BC402	615985.938	7572738	245.88	96	Chip	96		Historic		18
BC403	615734.5	7572760.5	246.98	78	Chip	96		Historic		20.5
BC404	615506.625	7572768	248.74	120	Chip	96		Historic		18
BC405	616500	7573000	243.6	105	Chip	96		Historic		22
BC406	616219.563	7573009.5	247.07	180	Chip	96		Historic		27
BC407	615995.563	7573018.5	249.27	70	Chip	96		Historic		26
BC408	615731.625	7573005.5	249.78	78	Chip	96		Historic		24
BC409	615500	7573000	251.3	114	Chip	96		Historic		24.5
BC410	615238.438	7573005.5	254.21	108	Chip	96		Historic		21.5
BC411	615241.375	7572754.5	250.45	108	Chip	96		Historic		
BC412	615250.063	7572501.5	247.08	102	Chip	96		Historic		23
BC413	615498.813	7572494.5	245	108	Chip	96		Historic		12
BC414	615748.688	7572503.5	244.44	60	Chip	96		Historic		23.5
BC415	616001.75	7572500.5	243.47	84	Chip	96		Historic		19
BC416	616747.375	7572498	238.57	120	Chip	96		Historic		24
BC417	617000.5	7572501.5	236.31	132	Chip	96		Historic		24
BC418	616751.438	7572750	240.04	108	Chip	96		Historic		26
BC419	615499.438	7573000	251.29	36	Core		100	Historic	No	25.12
BC420	615992.875	7573017	249.34	48	Chip	96		Historic		20
BC421	615996.375	7573014.5	249.25	48.7	Core		100	Historic	yes	21
BC422	616469.813	7573006	243.66	97	Core		100	Historic	yes	22
BC423	616748.063	7572970.5	241.59	118	Chip	96		Historic		28
BC424	615747.5	7572502.5	244.45	49	Core		100	Historic	yes	
BC425	616217.75	7572498	242.2	72	Chip	96		Historic		23
BC441	616218.563	7572500.5	242.26	66	Core		100	Historic	yes	23

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
BC442	615497.875	7573000	251.19	42	Core		100	Historic	No	24.63
IPC620	615682.75	7570025.5	223.69	58.9	Chip	96		2008		16
IPC621	615635.813	7570005.5	222.9	58.2	Chip	96		2008		11
IPC622	615694.75	7569979.5	223.72	64.2	Chip	96		2008		14
IPC623	615704.75	7569945.5	223.69	58.17	Chip	96		2008		14
IPC624	615723.688	7569900	223.62	64.2	Chip	96		2008		16
IPC625	615768.625	7569958.5	223.8	64.2	Chip	96		2008		18
IPC626	615753.625	7569867	223.59	21.5	Chip	96		2008		14
IPC627	615770.625	7569836	223.37	23.5	Chip	96		2008		14.7
IPC628	615841.5	7569848	223.79	28.3	Chip	96		2008		22.78
IPC629	615844.063	7569780.5	221.73	29.5	Chip	96		2008		19.7
IPC630	615919.375	7569867	224.01	34.2	Chip	96		2008		8
IPC631	615905.438	7569832	223.47	34.5	Chip	96		2008		10
IPC632	615942.5	7569814	220.37	28.2	Chip	96		2008		7
IPC633	615997.25	7569911	223.74	40.4	Chip	96		2008		11
IPC634	616025.938	7569870.5	220.92	34.3	Chip	96		2008		7
IPC635	615974.688	7569744.5	220.59	34.3	Chip	96		2008		8
IPC636	615992.5	7569772	220.38	34.3	Chip	96		2008		8
IPC637	615842	7569684	220.14	28.3	Chip	96		2008		7
IPC638	615970.313	7570142.5	224.66	35.6	Chip	96		2008		17
IPC639	615950	7570090	224.59	34.2	Chip	96		2008		13
IPC640	615921.375	7570008.5	224.19	34.2	Chip	96		2008		10
IPC641	615951.313	7570302	224.75	38	Chip	96		2008		16
IPC642C	615964.313	7570314	224.83	41	Core		100	2008	no	18
IPC643	615919.25	7570549	226.58	48	Chip	96		2008		15
IPC644C	615919.25	7570549	226.02	47.36	Core		100	2008	no	18
IPC645	615899.438	7571414.5	233.83	80	Chip	96		2008		18
IPC646C	615881.125	7571388.5	233.88	78.88	Core		100	2008	no	20
IPC647	615694.625	7571438.5	234.21	73	Chip	96		2008		23
IPC648C	615688.75	7571407.5	234.02	68.52	Core		100	2008	no	16
IPC649	615594.5	7571381.5	234.23	66	Chip	96		2008		17
IPC650C	615578.938	7571379	234.23	60.92	Core		100	2008	no	16

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC651	615635.438	7571146.5	231.61	38	Chip	96		2008		16
IPC652C	615634.5	7571143.5	231.57	37.38	Core		100	2008	no	15
IPC653	615729.125	7570873.5	228.49	60	Chip	96		2008		17
IPC654C	615720.688	7570877.5	228.45	58	Core		100	2008	no	16
IPC657	615540.875	7572538.5	244.59	42.3	Chip	96		2008		12
IPC658	615610.5	7572420	242.89	48	Chip	96		2008		19
IPC659	615694.688	7572226	240.83	54.71	Chip	96		2008		20
IPC660	615677.25	7572059	239.64	63.3	Chip	96		2008		22
IPC661	615704.188	7572036.5	239.25	66	Chip	96		2008		21.5
IPC662	615705.375	7572026.5	239.19	66.3	Chip	96		2008		24
IPC663	615705.938	7572017	239.05	66	Chip	96		2008		28
IPC664	615706.313	7572007	238.9	66	Chip	96		2008		24
IPC665	615707	7571997	238.88	66	Chip	96		2008		24
IPC666	615707.5	7571987.5	238.77	66	Chip	96		2008		24
IPC667	615707.875	7571977.5	238.71	67	Chip	96		2008		24
IPC668	615713.938	7571895.5	237.95	96	Chip	96		2008		24.75
IPC669	615713.188	7571905.5	238.2	96.3	Chip	96		2008		24.5
IPC670	615711.313	7571926	238.36	90	Chip	96		2008		26.5
IPC671	615712.25	7571916.5	238.18	96	Chip	96		2008		25
IPC672	615710.625	7571937	238.36	68	Chip	96		2008		24
IPC673	615800.375	7572060	239.19	78	Chip	96		2008		26
IPC674	615808.375	7572005	238.62	84	Chip	96		2008		21
IPC675	615812	7571979.5	238.41	78	Chip	96		2008		23
IPC676	615813.625	7571970	238.29	78	Chip	96		2008		22
IPC677	615814.875	7571960	238.22	138	Chip	96		2008		27
IPC678	615816.188	7571950	238.06	108	Chip	96		2008		23
IPC679	615817.688	7571939.5	238	108	Chip	96		2008		21
IPC680	615804.875	7572029.5	238.89	72	Chip	96		2008		24
IPC681	615907.75	7572091.5	239.43	84	Chip	96		2008		20
IPC682	615911.813	7572081	239.41	87	Chip	96		2008		23
IPC683	615915.375	7572072	239.31	90	Chip	96		2008		25
IPC684	615927.063	7572040.5	239.02	90	Chip	96		2008		24

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC685	615930.688	7572030.5	238.93	90	Chip	96		2008		26
IPC686	615934.125	7572021	238.83	90	Chip	96		2008		29
IPC687	615937.563	7572012	238.68	90	Chip	96		2008		24
IPC688	615940.875	7572003	238.62	132	Chip	96		2008		26
IPC689	615944.438	7571994	238.51	115	Chip	96		2008		25
IPC690	615947.688	7571985	238.45	114	Chip	96		2008		25
IPC691	615778.438	7571428.5	233.86	78	Chip	96		2008		18
IPC692G	615791.125	7571427.5	233.86	80.35	Gas		63	2008		19
IPC693C	615785.125	7571428.5	233.84	83.02	Core		100	2008	no	20
IPC694	615703.5	7572049.5	239.56	66	Chip	96		2008		18
IPC695	615702.688	7572060	239.62	66	Chip	96		2008		18
IPC696	615702	7572070	239.75	66	Chip	96		2008		18.5
IPC697	615798.625	7572071	239.42	72	Chip	96		2008		25
IPC698	615796.813	7572081	239.5	72	Chip	96		2008		25
IPC699	615794.813	7572091	239.6	72	Chip	96		2008		25
IPC700	615794.75	7572100.5	239.7	72	Chip	96		2008		25
IPC701	615793.563	7572110.5	239.81	72	Chip	96		2008		24.5
IPC702	615902.625	7572106.5	239.77	84	Chip	96		2008		19
IPC703	615898.313	7572117.5	240.01	84	Chip	96		2008		20
IPC704	615893.25	7572128	240	84	Chip	96		2008		18.5
IPC705	615889.313	7572138.5	240.11	84	Chip	96		2008		21.5
IPC706	615884.625	7572149	240.33	84	Chip	96		2008		22.5
IPC707	615880	7572159.5	240.38	84	Chip	96		2008		21
IPC708	616128.563	7569386	226.18	30	Chip	96		2008		17
IPC709	616178.188	7569334	227.28	36	Chip	96		2008		16
IPC710	616227.875	7569280.5	227.58	60	Chip	96		2008		25
IPC711	616219.438	7569289	227.56	60	Chip	96		2008		24
IPC712	616211.625	7569297	227.48	60	Chip	96		2008		22
IPC713	616203.875	7569305	227.4	54	Chip	96		2008		19.5
IPC714	616195.688	7569314	227.41	60	Chip	96		2008		20
IPC715	616187.688	7569322.5	227.34	36	Chip	96		2008		20.5
IPC716	616218.313	7569448.5	226.12	42	Chip	96		2008		18.5

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC717	616267.938	7569397.5	227.22	48	Chip	96		2008		22
IPC718	616275	7569390.5	227.33	48	Chip	96		2008		22
IPC719	616282.938	7569382.5	227.41	48	Chip	96		2008		23.5
IPC720	616290.875	7569374	227.51	84	Chip	96		2008		27
IPC721	616298.625	7569366	227.57	72	Chip	96		2008		22
IPC722	616306.063	7569358	227.56	72	Chip	96		2008		21
IPC723	616182.5	7569486.5	225.42	42	Chip	96		2008		20.5
IPC724	616190.438	7569478.5	225.55	42	Chip	96		2008		20.8
IPC725	616198.188	7569470.5	225.77	42	Chip	96		2008		21.2
IPC726	616205.938	7569462	225.93	42	Chip	96		2008		21.8
IPC727	616235.938	7569431	226.63	42.7	Chip	96		2008		23.9
IPC728	616253.375	7569413.5	227.03	48	Chip	96		2008		20
IPC729	616090.25	7569426.5	224.6	36	Chip	96		2008		19
IPC730	616098.125	7569418.5	224.94	36.3	Chip	96		2008		19
IPC731	616127.313	7569387.5	226.16	8	Chip	96		2008		
IPC732	616145.813	7569368	226.71	36	Chip	96		2008		20
IPC733	616109.313	7569407	225.27	36	Chip	96		2008		22.24
IPC734	616163.188	7569350	227.05	36	Chip	96		2008		22
IPC735	616247.125	7569260	227.65	60	Chip	96		2008		21
IPC736	616265.875	7569239.5	227.72	60	Chip	96		2008		21.5
IPC737	616285	7569218	227.82	60	Chip	96		2008		24
IPC738	616305.063	7569196	227.9	60	Chip	96		2008		25
IPC739	616324.813	7569174	228.05	60	Chip	96		2008		24
IPC740	616387.5	7569592	225.02	84	Chip	96		2008		16
IPC741	616395.25	7569584	225.18	84	Chip	96		2008		22
IPC742	616403.063	7569576	225.42	84	Chip	96		2008		19
IPC743	616410.313	7569568	225.5	84.3	Chip	96		2008		16
IPC744	616417.813	7569560	225.63	84.3	Chip	96		2008		24
IPC745	616432.875	7569544	225.93	84.3	Chip	96		2008		22
IPC746	616447.125	7569529	226.25	84.3	Chip	96		2008		24
IPC747	616461.25	7569514	226.63	84.3	Chip	96		2008		25
IPC748	616475.5	7569499	226.9	96.3	Chip	96		2008		28

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC749	616468.5	7569506.5	226.82	96.3	Chip	96		2008		22
IPC750	616482.25	7569492	227.05	96.3	Chip	96		2008		21
IPC751	616496.125	7569476	227.38	96.3	Chip	96		2008		20
IPC752	616510.25	7569461	227.68	96.3	Chip	96		2008		18
IPC753	616622.375	7569584.5	227.13	108.3	Chip	96		2008		17
IPC754	616607.125	7569598.5	226.58	108.3	Chip	96		2008		18
IPC755	616591.688	7569613	226.39	108.3	Chip	96		2008		20
IPC756	616576.875	7569627.5	226.02	108.3	Chip	96		2008		18
IPC757	616561.938	7569641.5	225.86	108.3	Chip	96		2008		19
IPC758	616547	7569656	225.56	108.3	Chip	96		2008		19
IPC759	616531.75	7569671	225.07	108.3	Chip	96		2008		19
IPC760	616516.625	7569685	224.73	96.3	Chip	96		2008		20
IPC761	616523.75	7569678	224.93	102.3	Chip	96		2008		19
IPC762	616746.75	7569689.5	228.1	114.3	Chip	96		2008		16
IPC763	616732.313	7569704	227.97	114.3	Chip	96		2008		16
IPC764	616718.125	7569719	227.5	114.3	Chip	96		2008		16
IPC765	616703.813	7569734.5	227.09	114.3	Chip	96		2008		14.5
IPC766	616689.313	7569749.5	226.64	114.3	Chip	96		2008		12
IPC767	616674.438	7569764	226.24	111.15	Chip	96		2008		14
IPC768	616659.563	7569778.5	225.81	114.3	Chip	96		2008		15
IPC769	616644.625	7569794	225.43	114.3	Chip	96		2008		18
IPC770	616629.563	7569808.5	225.28	114.3	Chip	96		2008		15
IPC771	616614.438	7569823	224.84	90.3	Chip	96		2008		23.5
IPC772	616606.813	7569830.5	224.48	84.76	Chip	96		2008		26.5
IPC773	616621.688	7569816.5	225.13	114.3	Chip	96		2008		23
IPC774	616591.625	7569839	224.15	108.5	Chip	96		2008		20
IPC775	617243.938	7568040	231.12	54.3	Chip	96		2008		22
IPC776	617240.063	7568060.5	231.56	60.6	Chip	96		2008		28
IPC777	617235.938	7568081	231.86	60.3	Chip	96		2008		27.5
IPC778	617232.125	7568101.5	232.06	66.3	Chip	96		2008		28
IPC779	617228.313	7568122	232.34	90.3	Chip	96		2008		26
IPC780	616666.5	7568918.5	229.64	66.3	Chip	96		2008		19

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC781G	616663.25	7568922.5	229.49	66	Gas		63	2008		18.5
IPC782	617588.313	7567729.5	230.51	72	Chip	96		2008		25.5
IPC783G	617588.188	7567734	230.67	66.3	Gas		63	2008		26
IPC784C	617596	7567721	231.4	70.87	Core		100	2008	no	25
IPC785C	617596	7567728	231.44	68.12	Core		100	2008	no	26
IPC786C	616673	7568911	229.98	56	Core		100	2008	no	19
IPC787C	616673	7568922	229.96	65.02	Core		100	2008	no	19.5
IPC788C	616662.188	7568918	229.53	65.02	Core		100	2008	no	19
IPC789C	615791	7571426	233.75	83.02	Core		100	2008	no	19
IPC790	616196.375	7569308	227.49	54.54	Chip	96		2008		18.5
IPC791	617372.375	7567661	227.53	48.54	Chip	96		2008		20.5
IPC792C	617373.875	7567656	227.58	53.02	Core		100	2008	no	20.5
IPC793C	617376.938	7567663	227.64	53.02	Core		100	2008	no	24
IPC794	617589.5	7567326.5	226.7	42.54	Chip	96		2008		18
IPC795C	617586.813	7567330	226.77	41.02	Core		100	2008	no	19.5
IPC796C	617584.25	7567322.5	226.59	41.02	Core		100	2008	no	19
IPC797	617939.188	7567124.5	226.54	36.54	Chip	96		2008		15
IPC798C	617936.063	7567126	226.53	34.34	Core		100	2008	no	16.5
IPC799C	617931.75	7567127.5	226.42	35.02	Core		100	2008	no	17
IPC800	617142.5	7568013	231.55	45.54	Chip	96		2008		21
IPC801C	617140.438	7568016	231.62	47.02	Core		100	2008	no	22
IPC802C	617138.063	7568020	231.75	47.02	Core		100	2008	no	22
IPC803C	616198.5	7569306	227.26	56	Core		100	2008	no	20
IPC804C	616201.188	7569304.5	227.41	56	Core		100	2008	no	19.5
IPC805	616426	7569069	229.07	56	Chip	96		2008		18
IPC806C	616427.438	7569066	228.18	53	Core		100	2008	no	19
IPC807C	616430.813	7569062.5	228.14	53	Core		100	2008	no	19.5
IPC808	616702.063	7568775	230.23	54.54	Chip	96		2008		21
IPC809C	616699.625	7568777.5	230.33	54.54	Core		100	2008	no	21.5
IPC810C	616696.938	7568780.5	230.31	54.54	Core		100	2008	no	22.5
IPC811	616877.75	7568419.5	233.76	54.54	Chip	96		2008		27
IPC812C	616876.563	7568422.5	233.65	53.02	Core		100	2008	no	24.5

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC813	616879.063	7568417	233.72	59.02	Chip	96		2008		25.5
IPC814C	616881.625	7568418.5	233.78	59.02	Core		100	2008	no	26
IPC815C	616876.313	7568415.5	233.78	59.02	Core		100	2008	no	28
IPC816	617266.438	7567869	229.33	54.54	Chip	96		2008		28.5
IPC817C	617264.188	7567871	229.42	53.02	Core		100	2008	no	27.5
IPC818C	617261.5	7567873.5	229.47	53.09	Core		100	2008	no	27.5
IPC819	617461.625	7567562	228.07	51.55	Chip	96		2008		27
IPC820C	617463.688	7567559	228.18	53	Core		100	2008	no	27.5
IPC821C	617466.188	7567555.5	228.29	53	Core		100	2008	no	26
IPC822	618140.125	7567081.5	227.53	36	Chip	96		2008		17
IPC823C	618138.625	7567084	227.57	35	Core		100	2008	no	17.5
IPC824C	618137.063	7567087.5	227.56	35	Core		100	2008	no	17
IPC825C	617249.938	7568043	231.59	53	Core		100	2008	no	22
IPC826C	617246.125	7568042.5	231.51	53	Core		100	2008	no	23
IPC827	617423	7567754	229.4	54.55	Chip	96		2008		20.5
IPC828C	617419.188	7567751	229.04	53	Core		100	2008	no	22
IPC829C	617423.875	7567753.5	229.01	53	Core		100	2008	no	22.5
IPC830	617635.063	7567419.5	228.24	48	Chip	96		2008		23
IPC831C	617638.5	7567421.5	228.32	53	Core		100	2008	no	24
IPC832C	617641.313	7567423	228.38	53.2	Core		100	2008	no	24
IPC833	617977.688	7567215.5	227.97	42	Chip	96		2008		20.5
IPC834C	617980	7567218	228.12	44	Core		100	2008	no	20.5
IPC835C	617982.188	7567220	228.14	44	Core		100	2008	no	21
IPC836	617753.625	7567285.5	227.23	45	Chip	96		2008		27
IPC837C	617755.875	7567287	227.26	47	Core		100	2008	no	27
IPC838C	617758.25	7567288.5	227.45	47	Core		100	2008	no	27
IPC839	617224.063	7568141.5	233.09	81	Chip	96		2008		28
IPC840	617219.125	7568160.5	233.42	84	Chip	96		2008		32
IPC841	617213.688	7568179	233.67	84.5	Chip	96		2008		27
IPC842	617230.25	7568113	232.65	61	Chip	96		2008		23
IPC843	617525.25	7568072.5	232.9	84	Chip	96		2008		23.5
IPC844	617514.188	7568090	232.94	84	Chip	96		2008		23.5

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC845	617503.25	7568108	233.26	84	Chip	96		2008		24
IPC846	617492.438	7568124.5	233.21	84.5	Chip	96		2008		29
IPC847	617481.563	7568142.5	233.52	84.5	Chip	96		2008		28
IPC848	617471.313	7568159.5	233.71	87	Chip	96		2008		31
IPC849	617460.75	7568177	233.88	96	Chip	96		2008		30.5
IPC850	617455.25	7568186.5	233.93	90.5	Chip	96		2008		22
IPC851	617465.438	7568169	233.84	87	Chip	96		2008		25
IPC852	617445.125	7568203	234.3	90.5	Chip	96		2008		24.5
IPC853	617434.5	7568221	234.61	93	Chip	96		2008		24
IPC854	617423.5	7568239	234.9	90.6	Chip	96		2008		33
IPC855	617418.25	7568247.5	235.03	111	Chip	96		2008		31.5
IPC856	617412.563	7568256.5	235.21	111	Chip	96		2008		31
IPC857	617401.313	7568274	235.52	114.5	Chip	96		2008		31.5
IPC858	617429.063	7568229.5	234.73	90.5	Chip	96		2008		31
IPC859	617390.313	7568290.5	235.6	117	Chip	96		2008		30
IPC860	617919.813	7567096	226.46	36	Chip	96		2008		20
IPC861	617901.313	7567106	226.22	36.5	Chip	96		2008		19
IPC862	617882.5	7567114.5	226.22	36	Chip	96		2008		19
IPC863	617863.813	7567124	226.13	34	Chip	96		2008		16
IPC864	617844.938	7567134	226.14	36	Chip	96		2008		19
IPC865	617826.438	7567143.5	226.1	36	Chip	96		2008		20
IPC866	617807.5	7567153.5	226	36	Chip	96		2008		17
IPC867	617788.813	7567163.5	225.9	36	Chip	96		2008		17
IPC868	617770.063	7567173	225.93	36.5	Chip	96		2008		16.5
IPC869	617753.75	7567182	225.91	36.5	Chip	96		2008		17
IPC870	618207.75	7567196	228.84	66.5	Chip	96		2008		26
IPC871	618198.313	7567192	228.73	66.5	Chip	96		2008		25.5
IPC872	618189.125	7567188	228.76	66.5	Chip	96		2008		25
IPC873	618179.625	7567184	228.59	66.5	Chip	96		2008		26
IPC874	618170.438	7567180	228.55	66.5	Chip	96		2008		26
IPC875	618161.063	7567176	228.41	66.5	Chip	96		2008		26
IPC876	618151.813	7567172	228.34	64.14	Chip	96		2008		28

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
IPC877	618142.688	7567168.5	228.29	42.5	Chip	96		2008		26
IPC878	618133.5	7567166	228.29	42.5	Chip	96		2008		23.5
IPC879	618123.75	7567162	228.12	42.5	Chip	96		2008		20
IPC880	618114.5	7567158.5	228.12	42.5	Chip	96		2008		23
IPC881	618105.063	7567154.5	227.99	42.5	Chip	96		2008		18
IPC882	618095.688	7567150.5	227.87	42.5	Chip	96		2008		18.5
IPC883	618086.563	7567147	227.87	39	Chip	96		2008		19.5
IPC884	618077.25	7567144	227.78	38	Chip	96		2008		21
IPC885	617737.5	7567420	228.71	54.5	Chip	96		2008		22.5
IPC886	617746.188	7567415	228.76	54.5	Chip	96		2008		21
IPC887	617764.25	7567405	228.76	56	Chip	96		2008		21
IPC888	617782.188	7567395	228.7	56	Chip	96		2008		21
IPC889	617800.125	7567385	228.62	56	Chip	96		2008		21
IPC890	617817.563	7567374.5	228.51	56	Chip	96		2008		21
IPC891	617835.688	7567365	228.36	54.5	Chip	96		2008		20.5
IPC892	617852.438	7567355.5	228.46	54.5	Chip	96		2008		21
IPC893	617924.5	7567486.5	231.03	72	Chip	96		2008		35
IPC894	617906.563	7567495.5	231.06	72	Chip	96		2008		34
IPC895	617888.375	7567505.5	231.01	76	Chip	96		2008		36
IPC896	617870.125	7567516	230.97	78	Chip	96		2008		34
IPC897	617,852	7567525	231.01	72.5	Chip	96		2008		35
IPC898	617833.375	7567535	230.9	78	Chip	96		2008		33
IPC899	617815.375	7567544.5	230.75	78	Chip	96		2008		35
IPC900	617797.375	7567554.5	230.85	74	Chip	96		2008		37
IPC901	617910.125	7567661.5	233.36	84.5	Chip	96		2008		34
IPC902	617881.938	7567703.5	233.47	90	Chip	96		2008		35.5
IPC903	617853.438	7567745	233.58	94	Chip	96		2008		35
IPC904	618,061.5	7567619.5	233.26	66.85	Chip	96		2008		20
IPC905	618043.063	7567619.5	233.41	78.5	Chip	96		2008		24.5
IPC906	618,081.5	7567619	233.25	126	Chip	96		2008		23
IPC907	618,052.25	7567619.5	233.37	126	Chip	96		2008		21
IPC908	618055.375	7567479	231.29	60.76	Chip	96		2008		25

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
CAIE0034	618140.523	7572226.9	235.8	273.8	Core	96	HQ3	2015 / 2016	no	
CAIE0035	618,144.74	7572228.63	235.8	273.8	Core	96	HQ3	2015 / 2016	Yes	30
CAIN0002	617332.142	7569353.27	235.96	138.4	Core	96	HQ3	2015 / 2016	Yes	
CAIN0006	617460.501	7571080.06	228.84	165.14	Core	96	HQ3	2015 / 2016	Yes	
CGIE0025	617953.348	7570624.59	237.15	190.43	Core	96	HQ3	2015 / 2016	Yes	
CGIE0026	617688.066	7568633.17	238.24	131	Core	96	HQ3	2015 / 2016	Yes	
CGIE0027	617688.972	7568634.74	238.25	135.59	Core	96	HQ3	2015 / 2016	yes	
CGIN0007	617427.688	7570176.23	235.07	156.22	Core	96	HQ3	2015 / 2016	yes	
CGIN0012	617504.692	7571987.57	228.92	150	Core	96	HQ3	2015 / 2016	no	
CGIN0042	617504.69	7571987.57	228.92	204.88	Core	96	HQ3	2015 / 2016	yes	14
CGIN0067	617023.175	7572498.59	236.48	125.9	Core	96	HQ3	2015 / 2016	yes	26
CGIN0073	617423.270	7568883.44	236.82	137.3	Core	96	HQ3	2016	Yes	20
CGIN0090	616503.558	7571040.31	207.81	57.22	Core	96	HQ3	2016	Yes	
CQIN0026	615961.21	7572811.79	246.92	77	Core	96	HQ3	2015 / 2016	Yes	16
CQIN0074	616183.02	7572252.51	240.29	89	Core	96	HQ3	2016	Yes	14
CQIN0075	616084.69	7572356.32	241.35	36	Core	96	HQ3	2016	Yes	12
CQIN0091	616448.31	7571399.63	225.3	89.3	Core	96	HQ3	2016	Yes	17
RSIN0001	617831.417	7572125.83	231.32	24	Chip	100	-	2015 / 2016		
RSIN0003	617840.943	7572126.74	231.46	250	Chip	100	-	2015 / 2016		
RSIN0004	617349.244	7571786.88	229.79	179.1	Chip	100	-	2015 / 2016		
RSIN0005	617237	7571350.17	227.69	160	Chip	100	-	2015 / 2016		
RSIN0008	617460.69	7570626.61	230.87	167	Chip	100	-	2015 / 2016		
RSIN0009	617192.608	7570365.75	228.96	140	Chip	100	-	2015 / 2016		
RSIN0010	617353.781	7572372.66	232.51	198.5	Chip	100	-	2015 / 2016		15
RSIN0011	616000	7572568	243.4	92	Chip	100	-	2015 / 2016		16
RSIN0012	616035	7572572	243.3	89	Chip	100	-	2015 / 2016		21
RSIN0013	615967	7572641	243.4	137	Chip	100	-	2015 / 2016		17
RSIN0014	616008	7572643	244.1	143	Chip	100	-	2015 / 2016		19.95
RSIN0015	616043	7572641	244.4	89	Chip	100	-	2015 / 2016		15
RSIN0016	615935	7572882	247.1	113	Chip	100	-	2015 / 2016		23
RSIN0017	615894	7572881	247.9	101	Chip	100	-	2015 / 2016		21
RSIN0018	615854	7572876	247.7	89	Chip	100	-	2015 / 2016		19

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
RSIN0019	615919	7573018	249.6	95	Chip	100	-	2015 / 2016		18
RSIN0020	615836	7573010	250.4	89	Chip	100	-	2015 / 2016		22
RSIN0021	615943	7572683	244.7	89	Chip	100	-	2015 / 2016		21.09
RSIN0022	616004	7572608	243.7	119	Chip	100	-	2015 / 2016		19
RSIN0023	616083.21	7572642.25	244.3	93	Chip	100	-	2015 / 2016		20
RSIN0024	615896	7572964	249.3	88	Chip	100	-	2015 / 2016		30
RSIN0025	615961	7572812	246.9	88	Chip	100	-	2015 / 2016		16
RSIN0027	616077.62	7572572.69	243.84	88	Chip	100	-	2015 / 2016		22
RSIN0028	616005.99	7572965.3	248.7	100	Chip	100	-	2015 / 2016		16
RSIN0029	616286.53	7571479.07	233.27	136	Chip	100	-	2015 / 2016		24
RSIN0030	616263.16	7571487.1	234.01	118	Chip	100	-	2015 / 2016		22
RSIN0031	616270.96	7571647.4	234.54	118	Chip	100	-	2015 / 2016		17
RSIN0032	616233.1	7571650.84	234.75	118	Chip	100	-	2015 / 2016		25
RSIN0033	616232.12	7571578.08	234.03	112	Chip	100	-	2015 / 2016		28
RSIN0034	616217.86	7571612.98	234.39	112	Chip	100	-	2015 / 2016		25
RSIN0035	616199.67	7571650.35	234.89	112	Chip	100	-	2015 / 2016		24
RSIN0036	616273.91	7571782	235.67	148	Chip	100	-	2015 / 2016		22
RSIN0037	616226.12	7571792.91	235.87	130	Chip	100	-	2015 / 2016		18
RSIN0038	616195.15	7571761.45	235.75	117	Chip	100	-	2015 / 2016		20
RSIN0039	616246.53	7571870.2	236.31	142	Chip	100	-	2015 / 2016		
RSIN0040	616164.45	7571877.53	236.83	135	Chip	100	-	2015 / 2016		
RSIN0041	616124.15	7571882.66	237.05	135	Chip	100	-	2015 / 2016		
RSIN0043	616204.76	7571873.86	236.6	141	Chip	100	-	2015 / 2016		
RSIN0044	616211.09	7572001.58	237.77	135	Chip	100	-	2015 / 2016		
RSIN0045	616171	7572007	237.9	129	Chip	100	-	2015 / 2016		20
RSIN0046	616131	7572014	238.22	135	Chip	100	-	2015 / 2016		17
RSIN0047	616091	7572019	238.66	135	Chip	100	-	2015 / 2016		19
RSIN0048	616125	7572048	239	111	Chip	100	-	2015 / 2016		20
RSIN0049	616033	7572459	243	69	Chip	100	-	2015 / 2016		15.21
RSIN0050	615993	7572458	242.38	63	Chip	100	-	2015 / 2016		19.21
RSIN0051	615988	7572403	242.15	75	Chip	100	-	2015 / 2016		20.27
RSIN0052	616025	7572403	242	75	Chip	100	-	2015 / 2016		15.67

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
RSIN0053	617392	7572701	240	178	Chip	100	-	2015 / 2016		21
RSIN0054	616058	7572460	243	75	Chip	100	-	2015 / 2016		22.92
RSIN0055	616045	7572220	240.82	105	Chip	100	-	2015 / 2016		6
RSIN0056	616008	7572218	240.23	105	Chip	100	-	2015 / 2016		6
RSIN0057	615968.33	7572219.31	237.34	93	Chip	100	-	2015 / 2016		
RSIN0058	616103.9	7571828.32	237.38	27	Chip	100	-	2015 / 2016		
RSIN0059	616139.98	7571965.56	237.82	129	Chip	100	-	2015 / 2016		
RSIN0060	616102.78	7571966.96	237.96	123	Chip	100	-	2015 / 2016		
RSIN0061	616067.45	7571968.14	237.79	123	Chip	100	-	2015 / 2016		18
RSIN0062	616093.61	7571915.56	237.29	123	Chip	100	-	2015 / 2016		29
RSIN0063	616304.62	7571612.01	234.25	105	Chip	100	-	2015 / 2016		23
RSIN0064	616300.68	7571692.85	234.85	105	Chip	100	-	2015 / 2016		17
RSIN0065	616101.55	7571847.33	237.29	123	Chip	100	-	2015 / 2016		24
RSIN0066	616152.35	7572409.49	241.57	93	Chip	100	-	2015 / 2016		27
RSIN0068	616269.22	7571247.37	208.04	93	Chip	100	-	2015 / 2016		10
RSIN0069	616303.51	7571252.43	209.34	93	Chip	100	-	2015 / 2016		8
RSIN0070	616184	7571580.6	204.5	111	Chip	100	-	2015 / 2016		
RSIN0071	616213.07	7571474.27	202.76	81	Chip	100	-	2015 / 2016		
RSIN0072	616099.36	7572079.53	238.67	105	Chip	100	-	2015 / 2016		10
RSIN0076	616109.68	7572971.54	247.76	93	Chip	100	-	2016		13
RSIN0077	616097.95	7572900.78	246.98	87	Chip	100	-	2016		25
RSIN0078	616086.81	7572825.25	246.14	75	Chip	100	-	2016		18
RSIN0079	616180.16	7572091.87	238.56	73	Chip	100	-	2016		18
RSIN0080	616189.56	7572077.31	238.41	101	Chip	100	-	2016		18
RSIN0081	616324.32	7571782.04	235.45	89	Chip	100	-	2016		22
RSIN0082	616314.65	7571814.04	235.78	77	Chip	100	-	2016		20
RSIN0083	616311.32	7571824.84	235.83	89	Chip	100	-	2016		19
RSIN0084	61098.53	7569560.33	227.32	32	Chip	100	-	2016		
RSIN0085	616125.22	7569581.05	227.46	35	Chip	100	-	2016		
RSIN0086	616618.83	7569948.64	226.43	59	Chip	100	-	2016		17
RSIN0087	616708.27	7569939.35	226.36	89	Chip	100	-	2016		6
RSIN0088	616127.50	7572868.47	246.57	83	Chip	100	-	2016		14

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
RSIN0089	616095.12	7572876.23	246.72	65	Chip	100	-	2016		13
CAIE0308	618735.62	7572252.74	236.83	294.65	Core	96	HQ3	2017	Yes	19
CAIE0309	618139.48	7571298.16	226.74	228.07	Core	96	HQ3	2017	Yes	17
CAIN0097	617421.14	7570029.93	236.26	153.4	Core	96	HQ3	2017	Yes	25
CAIN0098	617388.72	7572078.17	230.04	198.6	Core	96	HQ3	2017	Yes	21
CGIE0303	617861.29	7568177.8	236.96	104.27	Core	96	HQ3	2017	Yes	28
CGIE0304	617799.62	7570177.11	241.03	159.34	Core	96	HQ3	2017	Yes	21.62
CGIE0305	617923.37	7570974.88	233.28	207.52	Core	96	HQ3	2017	Yes	24
CGIE0310	618165.91	7571628.96	228.24	249.51	Core	96	HQ3	2017	Yes	19
CGIN0092	617529.78	7570639.12	231.83	180.54	Core	96	HQ3	2017	Yes	21
CGIN0093	617180.92	7570999.83	227.64	140.95	Core	96	HQ3	2017	Yes	21.5
CGIN0094	617148.19	7570543.9	226.83	132.44	Core	96	HQ3	2017	No	18
CGIN0095	617569.63	7569759.99	239.86	171.47	Core	96	HQ3	2017	No	22
CGIN0096	617143.89	7569767.68	232.86	135.56	Core	96	HQ3	2017	Yes	24
CGIN0099	616919.76	7570141.28	228.48	120.21	Core	96	HQ3	2017	Yes	20
CGIN0100	617257.46	7571566.18	227.18	147.3	Core	96	HQ3	2017	Yes	19
CGIN0101	617522.94	7569760.85	239.25	166.21	Core	96	HQ3	2017	Yes	22
CGIN0102	617023.99	7572006.04	234.08	153.48	Core	96	HQ3	2017	Yes	22
CGIN0103	617766.77	7572446.88	232.15	250	Core	96	HQ3	2017	Yes	14
CGIN0104	617199.22	7570546.25	227.59	144.35	Core	96	HQ3	2017	Yes	17
RSIE0342	617678.49	7570192.48	239.59	180	Chip	100		2018/2019	No	30
RSIE0343	617808.26	7569895.12	242.62	225	Chip	100		2018/2019	No	23
RSIE0344	617630.92	7569593.31	240.06	171	Chip	100		2018/2019	No	24
RSIE0346	617623.67	7568852.87	238.14	153	Chip	100		2018/2019	No	24
RSIE0347	617804.42	7569154.95	242.34	195	Chip	100		2018/2019	No	38
RSIE0348	617628.77	7568574.64	237.33	162	Chip	100		2018/2019	No	36
RSIE0349	617869.9	7568413.94	239.35	189	Chip	100		2018/2019	No	28
RSIE0358	617643.44	7569534.29	239.71	177	Chip	100		2018/2019	No	17
RSIE0444	618266.4	7568233.34	241.92	194	Chip	100		2018/2019	No	16
RSIE0445	618091.29	7568355.53	241.24	206	Chip	100		2018/2019	No	28
RSIE0446	618156.08	7568129.05	239.46	194	Chip	100		2018/2019	No	20
RSIN0125	617597.87	7569887.97	239.78	165	Chip	100		2018/2019	No	32

Hole	East	North	RL	Total Depth	Hole Type	Hole Diameter	Core Size	Exploration Program	POO	Depth to BW
RSIN0126	617392.58	7569692.36	237.01	147	Chip	100		2018/2019	No	48
RSIN0127	617529.11	7569239.55	237.23	178	Chip	100		2018/2019	No	25
RSIN0128	617452.79	7569137.91	237.23	144	Chip	100		2018/2019	No	34
RSIN0129	617440.28	7569125.52	237.22	153	Chip	100		2018/2019	No	30
RSIN0130	617422.77	7569108.59	237.1	147	Chip	100		2018/2019	No	23.5
RSIN0131	617471.05	7569153.75	237.22	142	Chip	100		2018/2019	No	27
RSIN0132	617545.11	7571160.14	229.74	22	Chip	100		2018/2019	No	33
RSIN0133	617546.4	7571158.99	229.76	183	Chip	100		2018/2019	No	33
RSIN0134	617530.04	7571142.99	229.56	183	Chip	100		2018/2019	No	26
RSIN0135	617506.23	7571120.06	229.38	169	Chip	100		2018/2019	No	36
RSIN0136	617490.89	7571104.14	229.2	165	Chip	100		2018/2019	No	29

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
001R	LHL	111.12	112.32	1.20	1
001R	LHL	111.12	112.32	1.20	1
001R	LHU	108.48	110.92	2.44	1
002R	LHD	73.36	77.20	3.84	1
003R	LHD	39.80	43.64	3.84	1
005R	LHD	53.86	57.98	4.12	1
006R	LHD	51.55	55.40	3.85	1
007R	LHD	30.32	34.05	3.73	1
008R	LHD	7.90	9.16	1.26	1
009R	LHD	86.64	90.12	3.48	1
011R	LHD	14.84	16.42	1.58	1
013R	LHD	34.70	34.71	0.01	1
015R	LHD	105.06	108.94	3.88	1
017R	LHD	24.46	27.92	3.46	1
018R	LHD	64.32	68.00	3.68	1
019R	LHD	39.90	46.86	6.96	1
021R	LHD	52.74	56.68	3.94	1
023R	LHD	15.68	19.84	4.16	1
026R	LHD	100.68	104.52	3.84	1
027R	LHL	62.92	64.05	1.13	1
027R	LHU	59.84	62.62	2.78	1
028R	LHL	37.16	37.97	0.81	1
028R	LHU	32.40	35.13	2.73	1
029R	LHL	17.15	17.60	0.45	1
029R	LHU	10.38	12.00	1.62	1
030R	LHL	27.80	28.80	1.00	1
030R	LHU	25.12	27.76	2.64	1
032R	LHD	36.46	40.48	4.02	1
033R	LHD	22.80	26.88	4.08	1
034R	LHD	27.40	31.04	3.64	1
035R	LHD	11.84	15.64	3.80	1
036R	LHD	15.03	17.80	2.77	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
037R	LHD	76.76	80.36	3.60	1
038R	LHD	61.59	64.63	3.04	1
039R	LHD	52.25	55.72	3.47	1
041R	LHD	42.60	45.93	3.33	1
042R	LHD	26.22	28.92	2.70	1
044R	LHD	31.20	31.30	0.10	1
045R	LHD	56.92	60.77	3.85	1
046R	LHL	85.58	86.65	1.07	1
046R	LHU	83.56	85.44	1.88	1
047R	LHD	136.15	139.76	3.61	1
049C	LHL	27.99	29.01	1.02	1
049C	LHU	25.34	27.99	2.65	1
051R	LHD	112.63	115.60	2.97	1
052R	LHD	111.50	114.50	3.00	1
055C	LHD	36.60	39.88	3.28	1
056C	LHD	42.45	46.42	3.97	1
058C	LHL	37.10	37.86	0.76	1
058C	LHU	32.20	34.84	2.64	1
061C	LHD	60.65	64.50	3.85	1
062C	LHD	56.82	60.80	3.98	1
063C	LHD	30.68	34.31	3.63	1
064R	LHD	115.22	117.26	2.04	1
066R	LHD	96.19	99.41	3.22	1
067R	LHD	87.00	90.00	3.00	1
068R	LHD	26.09	29.68	3.59	1
069R	LHD	38.18	41.96	3.78	1
070R	LHD	88.89	92.66	3.77	1
071R	LHD	54.68	57.94	3.26	1
072R	LHL	104.20	105.20	1.00	1
072R	LHU	101.60	104.02	2.42	1
073R	LHL	77.28	78.30	1.02	1
073R	LHU	74.58	77.09	2.51	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
075R	LHL	53.08	53.80	0.72	1
075R	LHU	46.96	49.38	2.42	1
076R	LHL	22.75	23.15	0.40	1
076R	LHU	11.72	14.27	2.55	1
077R	LHL	55.52	56.50	0.98	1
077R	LHU	53.28	55.47	2.19	1
078R	LHD	103.63	107.10	3.47	1
079R	LHL	83.82	84.94	1.12	1
079R	LHU	81.08	83.72	2.64	1
080R	LHL	38.78	39.78	1.00	1
080R	LHU	35.85	38.38	2.53	1
083R	LHD	66.84	70.72	3.88	1
084R	LHD	38.34	42.25	3.91	1
085R	LHD	17.94	22.07	4.13	1
086R	LHD	58.73	62.38	3.65	1
087R	LHL	15.30	15.85	0.55	1
087R	LHU	8.58	9.50	0.92	1
088R	LHD	5.70	6.60	0.90	1
089R	LHD	40.68	44.38	3.70	1
090R	LHD	51.78	55.39	3.61	1
091R	LHD	73.91	77.33	3.42	1
092R	LHD	43.63	47.46	3.83	1
093R	LHD	10.84	14.68	3.84	1
094R	LHD	117.79	121.32	3.53	1
095R	LHD	145.83	149.16	3.33	1
096R	LHD	117.37	120.90	3.53	1
097R	LHD	89.56	92.96	3.40	1
099C	LHD	54.82	68.77	13.95	1
1001R	LHL	80.48	81.42	0.94	1
1001R	LHU	77.36	80.08	2.72	1
1002R	LHL	77.60	78.78	1.18	1
1002R	LHL	84.78	85.94	1.16	2

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
1002R	LHU	74.48	77.36	2.88	1
1002R	LHU	81.32	84.46	3.14	2
1003R	LHL	37.20	38.24	1.04	1
1003R	LHL	55.80	58.38	2.58	2
1003R	LHL	66.82	68.22	1.40	3
1003R	LHL	74.34	75.82	1.48	4
1003R	LHU	33.40	37.00	3.60	1
1003R	LHU	49.20	52.80	3.60	2
1003R	LHU	61.94	66.54	4.60	3
1003R	LHU	69.18	72.78	3.60	4
1004R	LHL	76.36	77.60	1.24	1
1004R	LHL	84.35	85.35	1.00	2
1004R	LHU	73.26	76.12	2.86	1
1004R	LHU	83.20	84.35	1.15	2
1005R	LHL	40.08	41.20	1.12	1
1005R	LHU	36.86	39.84	2.98	1
1006R	LHL	65.00	65.88	0.88	1
1006R	LHU	59.70	62.37	2.67	1
1007R	LHL	26.40	27.38	0.98	1
1007R	LHU	23.00	24.97	1.97	1
1008R	LHL	24.40	25.02	0.62	1
1008R	LHL	40.12	45.84	5.72	2
1008R	LHU	20.24	23.30	3.06	1
1008R	LHU	35.02	38.72	3.70	2
1009R	LHL	62.32	63.20	0.88	1
1009R	LHU	56.66	59.44	2.78	1
100R	LHD	42.42	45.46	3.04	1
1010R	LHL	53.44	54.48	1.04	1
1010R	LHU	19.32	50.16	30.84	1
1011R	LHL	24.86	25.78	0.92	1
1011R	LHL	40.00	40.94	0.94	2
1011R	LHU	20.42	22.94	2.52	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
1011R	LHU	35.00	37.06	2.06	2
1012R	LHL	25.12	26.00	0.88	1
1012R	LHU	20.46	22.32	1.86	1
1013R	LHL	44.10	44.70	0.60	1
1013R	LHU	33.22	35.66	2.44	1
1014R	LHU	34.40	46.98	12.58	1
1015R	LHL	41.28	41.98	0.70	1
1015R	LHU	8.60	33.14	24.54	1
1016R	LHL	26.94	27.64	0.70	1
1016R	LHU	15.68	18.16	2.48	1
1018R	LHL	68.56	69.80	1.24	1
1018R	LHL	108.86	109.84	0.98	2
1018R	LHU	65.60	68.40	2.80	1
1018R	LHU	106.02	108.68	2.66	2
1019R	LHL	54.04	54.86	0.82	1
1019R	LHL	83.04	84.16	1.12	2
1019R	LHU	51.24	53.82	2.58	1
1019R	LHU	79.82	82.80	2.98	2
1020R	LHL	53.48	85.96	32.48	1
1020R	LHU	51.00	53.26	2.26	1
1021R	LHL	52.02	53.40	1.38	1
1021R	LHU	49.56	51.88	2.32	1
1022R	LHL	76.28	77.26	0.98	1
1022R	LHU	73.46	76.06	2.60	1
1023R	LHD	86.62	90.16	3.54	1
1024R	LHD	86.62	89.82	3.20	1
1025R	LHD	87.08	90.55	3.47	1
1026R	LHD	87.88	91.34	3.46	1
1027R	LHD	91.38	94.98	3.60	1
1028R	LHL	63.65	64.24	0.59	1
1028R	LHU	51.28	53.56	2.28	1
1029R	LHL	69.18	69.66	0.48	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
1029R	LHU	55.80	58.02	2.22	1
102R	LHD	40.50	42.20	1.70	1
1030R	LHL	27.00	27.42	0.42	1
1030R	LHL	78.93	79.37	0.44	2
1030R	LHU	18.70	19.95	1.25	1
1030R	LHU	67.56	69.50	1.94	2
1031R	LHL	64.15	65.10	0.95	1
1031R	LHU	54.36	56.38	2.02	1
1032R	LHL	64.16	65.04	0.88	1
1032R	LHU	55.00	57.06	2.06	1
1033R	LHL	33.58	34.22	0.64	1
1033R	LHL	84.13	84.68	0.55	2
1033R	LHU	22.38	24.32	1.94	1
1033R	LHU	53.45	72.98	19.53	2
1034R	LHL	73.28	73.86	0.58	1
1034R	LHU	60.46	62.80	2.34	1
1035R	LHD	82.80	86.38	3.58	1
1036R	LHD	79.56	84.18	4.62	1
103R	LHD	25.86	28.58	2.72	1
104R	LHD	22.83	25.52	2.69	1
105R	LHD	15.30	17.25	1.95	1
108R	LHD	70.71	74.19	3.48	1
109R	LHD	92.52	96.06	3.54	1
110R	LHD	116.16	119.58	3.42	1
111R	LHD	79.04	82.59	3.55	1
112R	LHD	196.16	199.92	3.76	1
115R	LHD	36.02	39.78	3.76	1
116R	LHD	102.78	106.40	3.62	1
117R	LHD	110.41	114.32	3.91	1
118R	LHD	118.08	122.20	4.12	1
119R	LHD	130.52	133.93	3.41	1
120C	LHD	56.16	60.00	3.84	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
122C	LHD	58.28	80.00	21.72	1
123R	LHD	81.67	95.32	13.65	1
125C	LHD	94.55	98.06	3.51	1
127C	LHD	28.08	31.68	3.60	1
128C	LHD	52.13	55.72	3.59	1
129C	LHD	37.27	40.80	3.53	1
130R	LHD	99.48	103.18	3.70	1
131R	LHD	105.64	108.78	3.14	1
132R	LHD	149.00	152.50	3.50	1
133R	LHD	125.58	129.04	3.46	1
134R	LHD	147.74	151.41	3.67	1
135R	LHD	26.14	29.78	3.64	1
136R	LHD	72.52	76.00	3.48	1
137R	LHD	110.60	114.02	3.42	1
138R	LHD	76.18	79.44	3.26	1
139R	LHD	61.24	65.26	4.02	1
141C	LHD	74.97	78.10	3.13	1
142C	LHD	43.24	46.51	3.27	1
143R	LHD	91.94	95.20	3.26	1
144R	LHD	84.12	87.22	3.10	1
145R	LHD	108.74	111.80	3.06	1
146R	LHD	20.74	24.22	3.48	1
147R	LHD	23.50	26.94	3.44	1
148R	LHD	101.57	105.44	3.87	1
149R	LHD	123.04	126.40	3.36	1
150R	LHD	114.48	117.76	3.28	1
151R	LHD	33.06	35.88	2.82	1
152R	LHD	65.80	68.84	3.04	1
153R	LHD	79.38	82.88	3.50	1
154R	LHD	99.64	103.08	3.44	1
155R	LHD	143.05	146.54	3.49	1
158R	LHD	93.44	97.18	3.74	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
159R	LHD	103.37	108.88	5.51	1
160R	LHD	84.49	88.55	4.06	1
161R	LHD	91.58	95.45	3.87	1
162C	LHD	15.06	24.76	9.70	1
163C	LHD	52.10	55.03	2.93	1
164C	LHD	19.12	22.05	2.93	1
165R	LHD	14.28	17.22	2.94	1
166R	LHD	49.90	53.48	3.58	1
167C	LHD	27.56	30.85	3.29	1
168R	LHD	33.77	36.68	2.91	1
169R	LHD	89.81	93.08	3.27	1
170R	LHD	76.54	80.76	4.22	1
171R	LHD	76.72	80.98	4.26	1
173R	LHD	18.70	22.01	3.31	1
174R	LHD	61.18	65.38	4.20	1
177R	LHD	21.04	24.82	3.78	1
178R	LHD	30.97	34.98	4.01	1
179R	LHD	42.01	46.38	4.37	1
181C	LHD	101.53	105.26	3.73	1
182C	LHD	25.92	29.36	3.44	1
184C	LHD	19.58	22.29	2.71	1
185C	LHD	69.93	72.95	3.02	1
186R	LHD	18.51	21.28	2.77	1
187C	LHD	24.04	26.85	2.81	1
188C	LHD	63.52	67.31	3.79	1
189C	LHD	23.10	26.64	3.54	1
190C	LHD	60.66	64.82	4.16	1
192C	LHD	128.24	131.68	3.44	1
193R	LHD	86.60	90.29	3.69	1
194C	LHD	99.92	103.58	3.66	1
196C	LHD	89.38	92.80	3.42	1
200L	LHD	14.44	18.74	4.30	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
201L	LHD	17.48	21.44	3.96	1
202L	LHD	15.53	19.88	4.35	1
203L	LHD	18.07	22.34	4.27	1
204L	LHD	16.68	20.69	4.01	1
205L	LHD	18.16	22.08	3.92	1
206L	LHD	13.80	17.52	3.72	1
207L	LHD	17.42	21.16	3.74	1
208L	LHD	17.72	21.22	3.50	1
209L	LHD	8.20	10.34	2.14	1
210L	LHD	9.40	13.04	3.64	1
211L	LHD	13.68	18.04	4.36	1
212L	LHD	13.14	17.44	4.30	1
213L	LHD	9.20	13.08	3.88	1
215L	LHD	13.40	13.90	0.50	1
216L	LHD	8.10	8.90	0.80	1
217L	LHD	11.20	15.20	4.00	1
218L	LHD	9.60	13.30	3.70	1
219L	LHD	12.80	16.72	3.92	1
220L	LHD	10.76	14.48	3.72	1
223L	LHD	23.40	27.20	3.80	1
224L	LHD	19.20	22.90	3.70	1
225L	LHD	14.00	17.60	3.60	1
226L	LHD	11.20	15.30	4.10	1
227L	LHD	17.64	21.10	3.46	1
228L	LHD	11.50	15.30	3.80	1
229L	LHD	10.64	11.78	1.14	1
230L	LHD	7.36	8.48	1.12	1
231R	LHD	27.30	31.10	3.80	1
233R	LHD	124.00	125.00	1.00	1
237R	LHD	19.35	20.50	1.15	1
239P	LHD	97.42	101.03	3.61	1
241L	LHD	9.24	11.44	2.20	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
242L	LHD	12.44	16.76	4.32	1
243L	LHD	19.42	23.16	3.74	1
244L	LHD	11.75	15.32	3.57	1
245L	LHD	11.72	15.09	3.37	1
247L	LHD	19.42	23.02	3.60	1
249L	LHD	14.75	17.40	2.65	1
250L	LHD	13.80	14.00	0.20	1
251L	LHD	15.28	16.80	1.52	1
252L	LHD	18.48	21.28	2.80	1
253L	LHD	21.09	23.75	2.66	1
254L	LHD	14.80	18.00	3.20	1
255L	LHD	12.20	13.22	1.02	1
256L	LHD	22.24	25.63	3.39	1
257L	LHD	15.55	18.75	3.20	1
258L	LHD	15.54	18.38	2.84	1
259L	LHD	13.20	15.69	2.49	1
260L	LHD	10.87	11.64	0.77	1
261L	LHD	17.20	20.12	2.92	1
262L	LHD	17.43	20.78	3.35	1
263L	LHD	13.45	15.15	1.70	1
264L	LHD	14.81	18.08	3.27	1
265L	LHD	15.10	18.85	3.75	1
266L	LHD	10.76	12.04	1.28	1
267L	LHD	16.53	20.16	3.63	1
269L	LHD	21.40	23.37	1.97	1
270L	LHD	16.04	18.74	2.70	1
271P	LHD	40.08	43.58	3.50	1
272P	LHD	37.88	41.29	3.41	1
273P	LHD	120.72	124.33	3.61	1
275P	LHD	63.70	69.00	5.30	1
280P	LHD	11.00	14.00	3.00	1
281L	LHD	23.25	27.48	4.23	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
282L	LHD	16.36	20.40	4.04	1
283L	LHD	14.53	18.32	3.79	1
284L	LHD	16.19	18.81	2.62	1
285L	LHD	15.18	17.84	2.66	1
286L	LHD	17.57	20.40	2.83	1
288L	LHD	27.40	30.30	2.90	1
289L	LHD	15.12	18.30	3.18	1
290L	LHD	15.10	15.80	0.70	1
291L	LHD	17.50	20.70	3.20	1
292L	LHD	14.95	18.96	4.01	1
293L	LHD	13.36	15.90	2.54	1
294L	LHD	17.44	21.28	3.84	1
296L	LHL	13.24	13.92	0.68	1
296L	LHU	12.15	13.12	0.97	1
297L	LHL	22.86	23.84	0.98	1
297L	LHU	20.15	22.68	2.53	1
298L	LHL	19.48	20.91	1.43	1
298L	LHU	16.70	19.36	2.66	1
299L	LHL	13.63	14.04	0.41	1
299L	LHU	9.92	10.92	1.00	1
300P	LHD	40.24	43.43	3.19	1
310L	LHL	18.32	19.10	0.78	1
310L	LHU	14.45	16.20	1.75	1
311L	LHL	21.84	22.28	0.44	1
311L	LHU	14.39	16.84	2.45	1
312L	LHU	10.72	13.28	2.56	1
313L	LHL	13.95	14.24	0.29	1
315L	LHD	19.04	23.82	4.78	1
316L	LHD	14.96	18.72	3.76	1
326R	LHD	14.56	18.00	3.44	1
327R	LHD	29.08	33.12	4.04	1
328C	LHD	28.86	32.74	3.88	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
329R	LHD	39.87	43.66	3.79	1
331R	LHD	24.30	28.00	3.70	1
332C	LHD	24.71	28.40	3.69	1
333C	LHD	24.63	28.34	3.71	1
334C	LHD	52.05	55.52	3.47	1
335C	LHD	51.76	55.23	3.47	1
338R	LHD	25.28	29.45	4.17	1
350C	LHD	23.82	27.98	4.16	1
351C	LHL	78.79	80.62	1.83	1
351C	LHU	76.19	78.79	2.60	1
352C	LHD	63.58	67.14	3.56	1
400R	LHD	20.98	25.40	4.42	1
401C	LHD	21.12	25.50	4.38	1
402C	LHD	21.96	25.84	3.88	1
403R	LHD	24.82	28.51	3.69	1
404C	LHD	24.70	28.39	3.69	1
405C	LHD	20.02	23.53	3.51	1
406C	LHD	22.70	26.48	3.78	1
407C	LHD	22.54	25.28	2.74	1
408C	LHD	23.28	27.11	3.83	1
410C	LHD	20.46	24.15	3.69	1
4110	LHD	41.50	44.20	2.70	1
411C	LHL	29.84	30.78	0.94	1
411C	LHU	27.14	29.63	2.49	1
412C	LHL	44.57	46.28	1.71	1
412C	LHU	42.38	44.57	2.19	1
413C	LHL	30.31	31.07	0.76	1
413C	LHU	25.04	27.70	2.66	1
414R	LHL	37.51	38.12	0.61	1
414R	LHU	29.39	32.15	2.76	1
415C	LHL	37.14	37.69	0.55	1
415C	LHU	29.05	31.73	2.68	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
417C	LHL	33.88	34.24	0.36	1
417C	LHU	23.22	25.75	2.53	1
418R	LHL	39.81	40.47	0.66	1
418R	LHU	28.36	30.58	2.22	1
419C	LHL	39.94	40.56	0.62	1
419C	LHU	28.41	30.61	2.20	1
420R	LHL	41.72	42.52	0.80	1
420R	LHU	31.36	33.69	2.33	1
421C	LHL	41.77	42.54	0.77	1
421C	LHU	31.37	33.66	2.29	1
422R	LHL	45.24	45.92	0.68	1
422R	LHU	31.98	34.08	2.10	1
423C	LHL	45.30	45.60	0.30	1
423C	LHU	31.83	33.90	2.07	1
424R	LHD	19.80	22.82	3.02	1
4257	LHD	17.98	20.42	2.44	1
4258	LHD	46.94	49.38	2.44	1
4259	LHD	35.05	37.31	2.26	1
425C	LHD	23.74	25.12	1.38	1
426R	LHD	26.78	29.53	2.75	1
428R	LHD	67.52	70.93	3.41	1
429R	LHD	43.42	46.20	2.78	1
430R	LHD	53.64	56.40	2.76	1
432R	LHD	65.96	69.06	3.10	1
433R	LHD	52.24	54.16	1.92	1
434R	LHD	48.13	50.30	2.17	1
435R	LHD	59.85	60.06	0.21	1
436R	LHD	52.57	55.52	2.95	1
437R	LHD	50.24	53.10	2.86	1
438R	LHD	18.38	21.42	3.04	1
440R	LHD	24.98	26.88	1.90	1
455C	LHD	18.43	21.43	3.00	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
456C	LHD	24.68	27.42	2.74	1
458R	LHD	70.76	73.78	3.02	1
459R	LHD	46.09	49.26	3.17	1
460R	LHD	29.90	32.24	2.34	1
461R	LHD	22.80	26.13	3.33	1
462R	LHD	20.57	24.18	3.61	1
463R	LHD	36.12	39.40	3.28	1
464R	LHD	41.30	44.56	3.26	1
465R	LHD	20.31	24.08	3.77	1
466R	LHD	24.15	27.97	3.82	1
467R	LHD	22.25	26.29	4.04	1
469C	LHD	24.11	27.00	2.89	1
470C	LHD	23.18	26.24	3.06	1
471C	LHD	20.67	24.24	3.57	1
472C	LHD	20.43	24.31	3.88	1
473C	LHD	24.02	27.86	3.84	1
474C	LHD	22.18	26.20	4.02	1
475C	LHD	24.47	28.71	4.24	1
476C	LHD	23.55	27.72	4.17	1
477C	LHD	17.37	21.82	4.45	1
478C	LHL	30.40	31.24	0.84	1
478C	LHU	25.17	27.89	2.72	1
479R	LHD	81.52	85.68	4.16	1
480R	LHD	84.34	88.53	4.19	1
481R	LHD	86.18	90.19	4.01	1
482R	LHD	52.81	86.84	34.03	1
483R	LHD	47.99	47.99	0.00	1
484R	LHD	26.53	30.25	3.72	1
485R	LHD	71.92	76.02	4.10	1
486R	LHL	89.76	90.75	0.99	1
486R	LHU	87.24	89.64	2.40	1
487R	LHL	65.54	66.47	0.93	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
487R	LHU	47.83	65.41	17.58	1
489R	LHD	90.67	94.66	3.99	1
490R	LHL	83.26	84.36	1.10	1
490R	LHU	80.34	83.09	2.75	1
491R	LHD	75.79	79.79	4.00	1
492R	LHD	39.04	42.26	3.22	1
493R	LHD	60.39	63.36	2.97	1
501L	LHU	17.94	20.34	2.40	1
502L	LHL	7.28	7.96	0.68	1
503L	LHL	9.10	9.76	0.66	1
504L	LHL	26.88	27.85	0.97	1
504L	LHU	16.24	18.40	2.16	1
505L	LHU	17.70	20.26	2.56	1
506L	LHL	31.00	32.00	1.00	1
506L	LHU	20.30	22.80	2.50	1
507L	LHL	32.29	32.99	0.70	1
507L	LHU	22.26	24.90	2.64	1
508L	LHL	23.15	24.22	1.07	1
508L	LHU	12.96	15.50	2.54	1
509L	LHL	23.42	24.22	0.80	1
509L	LHU	12.08	14.60	2.52	1
510L	LHU	22.20	24.52	2.32	1
511L	LHL	21.33	22.06	0.73	1
511L	LHU	13.62	15.20	1.58	1
512L	LHL	28.30	29.32	1.02	1
512L	LHU	19.95	22.48	2.53	1
513L	LHL	15.54	16.26	0.72	1
513L	LHU	9.28	10.08	0.80	1
514L	LHL	20.29	20.72	0.43	1
514L	LHU	9.85	12.44	2.59	1
515L	LHL	30.12	30.68	0.56	1
515L	LHU	20.76	23.44	2.68	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
517L	LHL	14.34	14.70	0.36	1
517L	LHU	7.20	8.01	0.81	1
518L	LHL	20.38	20.86	0.48	1
518L	LHU	13.56	15.04	1.48	1
519L	LHL	15.66	16.26	0.60	1
519L	LHU	13.12	14.24	1.12	1
520L	LHL	24.00	24.98	0.98	1
520L	LHU	20.85	23.45	2.60	1
521L	LHL	28.28	29.24	0.96	1
521L	LHU	25.60	28.13	2.53	1
522L	LHD	14.30	17.32	3.02	1
523L	LHD	7.69	9.20	1.51	1
524L	LHD	9.32	13.10	3.78	1
525C	LHD	28.30	32.14	3.84	1
526C	LHD	28.32	32.41	4.09	1
528C	LHD	26.68	30.55	3.87	1
529C	LHD	29.68	33.52	3.84	1
530L	LHD	11.14	15.40	4.26	1
531L	LHD	15.63	15.92	0.29	1
532L	LHD	8.32	10.48	2.16	1
533L	LHD	19.05	22.96	3.91	1
534L	LHD	14.16	18.29	4.13	1
535L	LHD	12.18	16.32	4.14	1
537L	LHD	11.29	15.04	3.75	1
538L	LHD	12.09	15.76	3.67	1
539L	LHD	9.44	10.98	1.54	1
540L	LHD	9.12	10.32	1.20	1
541L	LHD	13.92	17.66	3.74	1
542R	LHD	35.40	39.08	3.68	1
543R	LHD	54.76	58.52	3.76	1
544R	LHD	61.41	65.20	3.79	1
545L	LHL	12.00	13.10	1.10	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
546L	LHD	14.52	15.56	1.04	1
547L	LHD	16.11	19.88	3.77	1
548L	LHD	9.62	13.54	3.92	1
550L	LHD	12.05	15.56	3.51	1
551L	LHD	7.84	9.00	1.16	1
552L	LHD	8.36	12.32	3.96	1
556L	LHD	12.99	16.52	3.53	1
557L	LHD	7.98	12.02	4.04	1
558L	LHD	18.44	21.48	3.04	1
559L	LHD	16.78	17.24	0.46	1
560L	LHD	12.98	15.88	2.90	1
561L	LHD	14.50	17.30	2.80	1
5620	LHD	34.70	38.30	3.60	1
562L	LHD	13.40	15.08	1.68	1
563L	LHD	13.80	16.98	3.18	1
564L	LHD	12.44	15.68	3.24	1
565L	LHD	11.96	14.56	2.60	1
566L	LHD	13.64	17.56	3.92	1
567L	LHD	13.72	16.82	3.10	1
568L	LHD	14.32	17.20	2.88	1
569L	LHD	21.26	25.20	3.94	1
570L	LHD	14.89	17.88	2.99	1
602C	LHD	26.59	30.28	3.69	1
603R	LHD	21.76	26.26	4.50	1
605C	LHD	25.43	29.65	4.22	1
606C	LHD	23.79	27.61	3.82	1
BC041	LHL	56.70	57.80	1.10	1
BC041	LHU	44.09	46.36	2.27	1
BC042	LHL	22.40	23.70	1.30	1
BC042	LHU	6.00	7.70	1.70	1
BC043	LHL	91.00	91.75	0.75	1
BC043	LHU	84.17	86.20	2.03	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
BC045	LHL	120.32	121.46	1.14	1
BC045	LHU	116.40	118.71	2.31	1
BC048	LHL	198.92	199.92	1.00	1
BC048	LHU	195.71	198.34	2.63	1
BC049	LHU	47.60	49.75	2.15	1
BC050	LHL	90.23	91.25	1.02	1
BC050	LHU	87.51	90.04	2.53	1
BC051	LHU	13.50	15.00	1.50	1
BC059	LHL	79.70	80.70	1.00	1
BC059	LHU	76.45	78.71	2.26	1
BC082	LHL	115.34	116.36	1.02	1
BC082	LHU	112.46	114.97	2.51	1
BC089	LHL	144.47	145.45	0.98	1
BC089	LHU	141.00	143.60	2.60	1
BC096	LHD	79.73	83.31	3.58	1
BC401	LHL	68.77	69.43	0.66	1
BC401	LHU	61.12	63.22	2.10	1
BC402	LHL	37.62	38.01	0.39	1
BC402	LHU	26.50	28.54	2.04	1
BC403	LHL	57.53	57.92	0.39	1
BC403	LHU	46.33	48.57	2.24	1
BC404	LHL	42.56	43.42	0.86	1
BC404	LHU	32.13	34.43	2.30	1
BC405	LHL	93.10	93.84	0.74	1
BC405	LHU	87.36	89.68	2.32	1
BC406	LHL	83.68	84.36	0.68	1
BC406	LHU	74.92	76.86	1.94	1
BC408	LHL	64.24	64.61	0.37	1
BC408	LHU	53.00	55.15	2.15	1
BC409	LHL	42.32	42.64	0.32	1
BC409	LHU	28.72	30.76	2.04	1
BC410	LHL	29.00	30.04	1.04	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
BC410	LHU	15.14	17.68	2.54	1
BC411	LHL	29.36	30.30	0.94	1
BC411	LHU	18.92	21.32	2.40	1
BC412	LHL	21.28	21.94	0.66	1
BC412	LHU	12.06	13.05	0.99	1
BC413	LHL	30.00	30.76	0.76	1
BC413	LHU	18.29	20.70	2.41	1
BC414	LHL	50.58	50.92	0.34	1
BC414	LHU	37.32	39.56	2.24	1
BC415	LHL	17.60	18.32	0.72	1
BC415	LHL	37.68	38.81	1.13	2
BC415	LHU	9.08	10.04	0.96	1
BC415	LHU	30.31	33.00	2.69	2
BC416	LHL	107.92	108.96	1.04	1
BC416	LHU	105.16	107.69	2.53	1
BC417	LHL	118.08	119.00	0.92	1
BC417	LHU	115.44	117.90	2.46	1
BC418	LHL	98.61	99.54	0.93	1
BC418	LHU	95.65	98.00	2.35	1
BC421	LHU	26.71	41.29	14.58	1
BC422	LHL	92.98	93.67	0.69	1
BC422	LHU	87.24	89.49	2.25	1
BC423	LHL	105.44	106.24	0.80	1
BC423	LHU	100.53	102.72	2.19	1
BC441	LHL	58.16	59.04	0.88	1
BC441	LHU	53.22	55.54	2.32	1
CAIE0035	LHD	263.56	267.90	4.34	1
CAIE0308	LHD	60.21	285.51	225.30	1
CAIE0309	LHD	216.45	220.45	4.00	1
CAIN0002	LHD	125.58	129.48	3.90	1
CAIN0006	LHD	154.20	157.80	3.60	1
CAIN0097	LHD	141.72	145.26	3.54	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
CAIN0098	LHD	187.68	191.37	3.69	1
CGIE0025	LHD	180.65	184.13	3.48	1
CGIE0026	LHD	121.43	124.90	3.47	1
CGIE0027	LHD	121.68	125.37	3.69	1
CGIE0303	LHD	89.31	92.94	3.63	1
CGIE0304	LHD	21.62	148.05	126.43	1
CGIE0305	LHD	194.05	198.06	4.01	1
CGIE0310	LHD	235.64	240.07	4.43	1
CGIN0007	LHD	146.04	149.64	3.60	1
CGIN0042	LHD	195.19	199.18	3.99	1
CGIN0067	LHL	119.85	120.63	0.78	1
CGIN0067	LHU	117.16	119.56	2.40	1
CGIN0073	LHD	125.15	128.62	3.47	1
CGIN0090	LHD	49.14	52.71	3.57	1
CGIN0092	LHD	161.38	164.73	3.35	1
CGIN0093	LHD	126.79	130.13	3.34	1
CGIN0096	LHD	119.82	123.07	3.25	1
CGIN0099	LHD	106.53	109.38	2.85	1
CGIN0100	LHD	133.18	136.59	3.41	1
CGIN0101	LHD	150.10	153.64	3.54	1
CGIN0102	LHD	140.09	143.80	3.71	1
CGIN0103	LHD	238.16	242.29	4.13	1
CGIN0104	LHD	133.33	136.52	3.19	1
CQIE0011	LHD	29.48	32.16	2.68	1
CQIE0012	LHD	50.97	53.82	2.85	1
CQIE0013	LHD	18.25	21.25	3.00	1
CQIE0014	LHD	21.98	24.96	2.98	1
CQIE0015	LHD	55.51	58.30	2.79	1
CQIE0016	LHD	23.95	26.69	2.74	1
CQIE0017	LHD	91.13	94.28	3.15	1
CQIE0018	LHD	16.80	20.07	3.27	1
CQIE0019	LHD	30.31	33.55	3.24	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
CQIE0031	LHD	62.88	66.56	3.68	1
CQIE0032	LHD	34.51	38.46	3.95	1
CQIN0026	LHL	32.73	33.15	0.42	1
CQIN0026	LHU	21.36	23.35	1.99	1
CQIN0026	LHU	67.80	69.97	2.17	2
CQIN0074	LHL	77.45	78.53	1.08	1
CQIN0074	LHU	74.16	77.17	3.01	1
CQIN0075	LHL	22.79	23.66	0.87	1
CQIN0075	LHU	18.67	21.05	2.38	1
CQIN0091	LHD	73.07	76.61	3.54	1
IPC623	LHD	16.30	18.00	1.70	1
IPC625	LHD	25.40	27.80	2.40	1
IPC626	LHD	11.50	14.00	2.50	1
IPC627	LHD	12.08	14.70	2.62	1
IPC628	LHD	19.00	22.78	3.78	1
IPC629	LHD	16.15	19.70	3.55	1
IPC630	LHD	22.28	25.98	3.70	1
IPC631	LHD	19.84	23.44	3.60	1
IPC632	LHD	17.17	20.79	3.62	1
IPC633	LHD	25.80	29.50	3.70	1
IPC634	LHD	21.48	25.04	3.56	1
IPC635	LHD	18.28	21.74	3.46	1
IPC636	LHD	18.48	22.03	3.55	1
IPC638	LHD	23.00	26.55	3.55	1
IPC639	LHD	21.80	25.48	3.68	1
IPC640	LHD	20.81	24.60	3.79	1
IPC641	LHD	32.60	36.10	3.50	1
IPC642C	LHD	32.60	36.10	3.50	1
IPC643	LHD	39.41	43.18	3.77	1
IPC644C	LHD	39.41	43.18	3.77	1
IPC645	LHD	71.68	75.84	4.16	1
IPC646C	LHD	71.30	75.46	4.16	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC647	LHD	61.74	65.80	4.06	1
IPC648C	LHD	62.82	66.75	3.93	1
IPC649	LHD	52.74	56.68	3.94	1
IPC650C	LHD	53.14	57.02	3.88	1
IPC652C	LHD	30.32	34.46	4.14	1
IPC654C	LHD	45.51	49.50	3.99	1
IPC657	LHL	32.71	33.26	0.55	1
IPC657	LHU	20.69	22.98	2.29	1
IPC658	LHL	38.10	38.47	0.37	1
IPC658	LHU	26.74	28.82	2.08	1
IPC659	LHL	47.44	48.18	0.74	1
IPC659	LHU	40.86	43.48	2.62	1
IPC660	LHL	55.00	56.04	1.04	1
IPC660	LHU	51.03	53.80	2.77	1
IPC661	LHL	57.52	58.63	1.11	1
IPC661	LHU	54.24	56.99	2.75	1
IPC662	LHL	57.92	58.97	1.05	1
IPC662	LHU	54.66	57.42	2.76	1
IPC663	LHL	58.05	59.12	1.07	1
IPC663	LHU	54.87	57.60	2.73	1
IPC664	LHL	58.28	59.34	1.06	1
IPC664	LHU	55.09	57.88	2.79	1
IPC665	LHL	58.58	59.65	1.07	1
IPC665	LHU	55.44	58.20	2.76	1
IPC666	LHL	58.66	59.68	1.02	1
IPC666	LHU	55.56	58.33	2.77	1
IPC667	LHL	58.72	59.72	1.00	1
IPC667	LHU	55.68	58.40	2.72	1
IPC668	LHD	83.26	87.27	4.01	1
IPC669	LHD	82.79	86.94	4.15	1
IPC670	LHL	80.58	81.92	1.34	1
IPC670	LHU	77.52	80.39	2.87	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC671	LHL	85.32	86.44	1.12	1
IPC671	LHU	82.74	85.10	2.36	1
IPC672	LHL	60.28	61.34	1.06	1
IPC672	LHU	57.26	60.04	2.78	1
IPC673	LHL	65.05	66.09	1.04	1
IPC673	LHU	61.89	64.62	2.73	1
IPC674	LHD	64.25	68.15	3.90	1
IPC675	LHD	64.90	68.88	3.98	1
IPC676	LHD	65.31	69.39	4.08	1
IPC678	LHD	92.18	96.18	4.00	1
IPC679	LHD	92.22	96.27	4.05	1
IPC680	LHL	65.94	67.00	1.06	1
IPC680	LHU	62.85	65.62	2.77	1
IPC681	LHL	76.56	77.56	1.00	1
IPC681	LHU	73.64	76.30	2.66	1
IPC682	LHD	74.35	78.25	3.90	1
IPC683	LHD	75.01	78.85	3.84	1
IPC684	LHD	75.67	79.58	3.91	1
IPC685	LHL	79.14	80.18	1.04	1
IPC685	LHU	76.26	78.90	2.64	1
IPC686	LHL	78.66	79.66	1.00	1
IPC686	LHU	75.78	78.43	2.65	1
IPC687	LHL	78.85	79.88	1.03	1
IPC687	LHU	76.00	78.62	2.62	1
IPC689	LHD	103.49	107.22	3.73	1
IPC690	LHD	103.56	107.33	3.77	1
IPC694	LHL	57.38	58.40	1.02	1
IPC694	LHU	53.90	56.65	2.75	1
IPC695	LHL	57.07	58.06	0.99	1
IPC695	LHU	53.34	56.11	2.77	1
IPC696	LHL	57.03	58.04	1.01	1
IPC696	LHU	53.08	55.83	2.75	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC697	LHL	64.84	65.90	1.06	1
IPC697	LHU	61.63	64.38	2.75	1
IPC698	LHL	64.90	65.95	1.05	1
IPC698	LHU	61.54	64.28	2.74	1
IPC699	LHL	64.66	65.66	1.00	1
IPC699	LHU	61.14	63.83	2.69	1
IPC700	LHL	64.75	65.74	0.99	1
IPC700	LHU	61.07	63.82	2.75	1
IPC701	LHL	64.68	65.70	1.02	1
IPC701	LHU	60.84	63.58	2.74	1
IPC702	LHL	75.91	76.91	1.00	1
IPC702	LHU	72.86	75.48	2.62	1
IPC703	LHL	75.64	76.61	0.97	1
IPC703	LHU	72.50	75.14	2.64	1
IPC704	LHL	75.06	76.01	0.95	1
IPC704	LHU	71.90	74.48	2.58	1
IPC705	LHL	74.71	75.66	0.95	1
IPC705	LHU	71.40	74.02	2.62	1
IPC706	LHL	74.38	75.31	0.93	1
IPC706	LHU	70.82	73.38	2.56	1
IPC707	LHL	74.11	75.00	0.89	1
IPC707	LHU	70.54	72.96	2.42	1
IPC708	LHD	20.82	24.26	3.44	1
IPC709	LHD	21.93	25.64	3.71	1
IPC710	LHD	50.00	53.30	3.30	1
IPC711	LHD	48.65	52.02	3.37	1
IPC712	LHD	48.30	52.58	4.28	1
IPC713	LHD	44.37	47.77	3.40	1
IPC714	LHD	43.94	47.35	3.41	1
IPC715	LHD	21.95	25.52	3.57	1
IPC716	LHD	32.21	35.56	3.35	1
IPC717	LHD	36.54	39.78	3.24	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC718	LHD	37.03	40.30	3.27	1
IPC719	LHD	37.73	40.94	3.21	1
IPC720	LHD	38.82	40.10	1.28	1
IPC721	LHD	58.32	61.52	3.20	1
IPC722	LHD	58.58	61.76	3.18	1
IPC723	LHD	30.50	33.78	3.28	1
IPC724	LHD	30.98	34.40	3.42	1
IPC725	LHD	31.60	34.98	3.38	1
IPC726	LHD	31.80	35.10	3.30	1
IPC727	LHD	33.58	36.92	3.34	1
IPC728	LHD	35.18	38.44	3.26	1
IPC729	LHD	21.22	24.73	3.51	1
IPC730	LHD	21.83	25.44	3.61	1
IPC732	LHD	22.90	26.40	3.50	1
IPC733	LHD	22.24	26.16	3.92	1
IPC734	LHD	24.06	27.52	3.46	1
IPC735	LHD	49.10	52.30	3.20	1
IPC736	LHD	48.30	51.36	3.06	1
IPC737	LHD	47.56	50.56	3.00	1
IPC738	LHD	46.44	49.38	2.94	1
IPC739	LHD	46.90	49.88	2.98	1
IPC740	LHD	69.01	72.79	3.78	1
IPC741	LHD	69.74	73.33	3.59	1
IPC742	LHD	70.11	73.74	3.63	1
IPC743	LHD	70.27	73.92	3.65	1
IPC744	LHD	70.75	74.26	3.51	1
IPC745	LHD	71.48	75.09	3.61	1
IPC746	LHD	72.16	75.77	3.61	1
IPC747	LHD	72.25	75.86	3.61	1
IPC748	LHD	82.97	86.81	3.84	1
IPC749	LHD	74.80	75.76	0.96	1
IPC750	LHD	84.02	87.70	3.68	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC751	LHD	85.89	89.32	3.43	1
IPC752	LHD	84.54	87.81	3.27	1
IPC753	LHD	96.24	99.88	3.64	1
IPC754	LHD	95.05	98.56	3.51	1
IPC755	LHD	93.48	96.98	3.50	1
IPC756	LHD	92.77	96.07	3.30	1
IPC757	LHD	93.19	96.61	3.42	1
IPC758	LHD	94.85	98.58	3.73	1
IPC759	LHD	96.25	100.15	3.90	1
IPC760	LHD	84.70	86.04	1.34	1
IPC762	LHD	102.43	106.05	3.62	1
IPC763	LHD	101.91	105.54	3.63	1
IPC764	LHD	101.06	104.58	3.52	1
IPC765	LHD	100.28	103.94	3.66	1
IPC766	LHD	99.62	103.15	3.53	1
IPC767	LHD	99.20	102.62	3.42	1
IPC768	LHD	100.15	103.46	3.31	1
IPC769	LHD	101.50	104.76	3.26	1
IPC770	LHD	102.51	105.87	3.36	1
IPC771	LHD	80.06	81.72	1.66	1
IPC772	LHD	74.36	78.20	3.84	1
IPC774	LHD	53.26	53.75	0.49	1
IPC776	LHD	42.87	52.29	9.42	1
IPC777	LHD	44.44	54.21	9.77	1
IPC778	LHD	47.05	54.10	7.05	1
IPC779	LHD	71.20	72.56	1.36	1
IPC780	LHD	53.81	56.68	2.87	1
IPC782	LHD	57.09	61.09	4.00	1
IPC783G	LHD	57.30	61.36	4.06	1
IPC784C	LHD	57.30	61.30	4.00	1
IPC785C	LHD	57.69	61.71	4.02	1
IPC789C	LHD	68.40	72.44	4.04	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC790	LHD	43.55	47.05	3.50	1
IPC793C	LHD	40.14	43.71	3.57	1
IPC796C	LHD	31.50	35.45	3.95	1
IPC799C	LHD	24.81	28.89	4.08	1
IPC802C	LHD	35.57	38.58	3.01	1
IPC803C	LHD	44.54	47.50	2.96	1
IPC804C	LHD	44.31	47.61	3.30	1
IPC806C	LHD	43.10	46.06	2.96	1
IPC809C	LHD	41.44	44.32	2.88	1
IPC812C	LHD	42.34	45.21	2.87	1
IPC818C	LHD	42.38	45.40	3.02	1
IPC821C	LHD	40.57	43.77	3.20	1
IPC822	LHD	23.75	28.08	4.33	1
IPC823C	LHD	23.82	28.17	4.35	1
IPC824C	LHD	23.85	28.26	4.41	1
IPC826C	LHD	42.09	44.91	2.82	1
IPC829C	LHD	42.94	45.90	2.96	1
IPC831C	LHD	38.72	42.21	3.49	1
IPC834C	LHD	31.52	35.85	4.33	1
IPC837C	LHD	34.54	38.40	3.86	1
IPC839	LHD	70.08	73.53	3.45	1
IPC840	LHD	72.14	75.73	3.59	1
IPC841	LHD	73.94	77.24	3.30	1
IPC843	LHD	73.96	77.77	3.81	1
IPC844	LHD	73.59	77.46	3.87	1
IPC845	LHD	74.10	78.08	3.98	1
IPC846	LHD	74.26	78.22	3.96	1
IPC847	LHD	74.78	78.64	3.86	1
IPC848	LHD	75.37	79.60	4.23	1
IPC849	LHD	82.32	84.15	1.83	1
IPC850	LHD	81.31	85.05	3.74	1
IPC851	LHD	76.05	80.47	4.42	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC852	LHD	81.58	85.37	3.79	1
IPC853	LHD	81.74	85.58	3.84	1
IPC855	LHD	100.63	104.58	3.95	1
IPC856	LHD	101.11	104.83	3.72	1
IPC857	LHD	105.42	109.23	3.81	1
IPC858	LHD	81.61	85.41	3.80	1
IPC859	LHD	105.84	109.74	3.90	1
IPC860	LHD	22.41	26.64	4.23	1
IPC861	LHD	22.86	27.03	4.17	1
IPC862	LHD	23.32	27.44	4.12	1
IPC863	LHD	23.55	27.63	4.08	1
IPC864	LHD	24.32	28.38	4.06	1
IPC865	LHD	25.04	28.96	3.92	1
IPC866	LHD	25.50	29.52	4.02	1
IPC867	LHD	25.72	29.68	3.96	1
IPC868	LHD	26.24	30.18	3.94	1
IPC869	LHD	26.40	30.42	4.02	1
IPC877	LHD	33.64	35.14	1.50	1
IPC878	LHD	27.44	34.76	7.32	1
IPC879	LHD	29.80	34.48	4.68	1
IPC880	LHD	29.40	33.82	4.42	1
IPC881	LHD	28.48	33.27	4.79	1
IPC882	LHD	27.28	31.96	4.68	1
IPC883	LHD	25.98	31.93	5.95	1
IPC884	LHD	26.49	31.09	4.60	1
IPC885	LHD	45.09	49.01	3.92	1
IPC886	LHD	45.37	49.32	3.95	1
IPC887	LHD	45.65	49.72	4.07	1
IPC888	LHD	45.40	49.18	3.78	1
IPC889	LHD	45.32	49.17	3.85	1
IPC890	LHD	45.11	48.93	3.82	1
IPC891	LHD	44.44	48.33	3.89	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
IPC892	LHD	43.65	47.63	3.98	1
IPC893	LHD	60.44	64.69	4.25	1
IPC894	LHD	61.95	66.16	4.21	1
IPC895	LHD	63.88	67.98	4.10	1
IPC896	LHD	66.03	69.88	3.85	1
IPC897	LHD	61.06	67.00	5.94	1
IPC898	LHD	63.30	68.01	4.71	1
IPC899	LHD	63.69	68.36	4.67	1
IPC900	LHD	63.52	67.57	4.05	1
IPC901	LHD	74.75	79.15	4.40	1
IPC902	LHD	78.68	82.91	4.23	1
IPC903	LHD	82.19	86.47	4.28	1
IPC905	LHD	66.31	70.81	4.50	1
IPC907	LHD	65.92	70.61	4.69	1
IPC908	LHD	57.12	60.76	3.64	1
MW017	LHD	80.00	113.00	33.00	1
MW018	LHD	140.82	141.91	1.09	1
MW019	LHD	143.82	144.87	1.05	1
MW020	LHD	134.84	135.72	0.88	1
N1SPRB06	LHD	28.65	56.50	27.85	1
N1SPRB07	LHD	31.97	58.00	26.03	1
N1SPRB08	LHD	30.00	59.00	29.00	1
N1SPRB09	LHD	33.25	36.95	3.70	1
N1SPRB10	LHD	37.60	41.15	3.55	1
RLIE0020	LHD	16.89	17.09	0.20	1
RLIE0021	LHD	12.99	13.89	0.90	1
RLIE0022	LHD	14.19	16.08	1.89	1
RLIE0023	LHD	25.16	25.84	0.68	1
RSIE0004	LHD	36.74	39.54	2.80	1
RSIE0005	LHD	49.58	52.28	2.70	1
RSIE0006	LHD	47.96	51.00	3.04	1
RSIE0007	LHD	41.21	44.24	3.03	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIE0008	LHD	36.16	39.48	3.32	1
RSIE0009	LHD	37.49	40.20	2.71	1
RSIE0010	LHD	47.72	50.34	2.62	1
RSIE0028	LHD	48.46	51.36	2.90	1
RSIE0030	LHD	43.69	47.19	3.50	1
RSIE0033	LHD	54.00	57.15	3.15	1
RSIN0003	LHD	240.05	244.05	4.00	1
RSIN0004	LHD	167.44	170.91	3.47	1
RSIN0005	LHD	146.60	150.04	3.44	1
RSIN0009	LHD	128.82	132.02	3.20	1
RSIN0010	LHD	187.96	191.79	3.83	1
RSIN0011	LHL	16.00	16.68	0.68	1
RSIN0011	LHU	7.23	7.88	0.65	1
RSIN0012	LHL	27.56	28.30	0.74	1
RSIN0012	LHU	16.57	18.59	2.02	1
RSIN0013	LHL	15.41	15.63	0.22	1
RSIN0014	LHL	19.83	20.46	0.63	1
RSIN0014	LHL	78.00	79.00	1.00	2
RSIN0014	LHU	9.00	11.00	2.00	1
RSIN0014	LHU	67.55	71.00	3.45	2
RSIN0015	LHL	25.47	26.09	0.62	1
RSIN0015	LHL	56.24	57.00	0.76	2
RSIN0015	LHU	14.51	16.76	2.25	1
RSIN0015	LHU	47.16	49.36	2.20	2
RSIN0016	LHL	72.08	72.73	0.65	1
RSIN0016	LHU	60.34	62.52	2.18	1
RSIN0017	LHL	70.09	70.81	0.72	1
RSIN0017	LHU	58.97	61.08	2.11	1
RSIN0018	LHL	67.75	68.59	0.84	1
RSIN0018	LHU	57.04	59.18	2.14	1
RSIN0019	LHL	74.42	75.28	0.86	1
RSIN0019	LHU	64.09	66.13	2.04	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIN0020	LHL	71.36	71.69	0.33	1
RSIN0020	LHU	58.00	60.03	2.03	1
RSIN0021	LHL	20.43	59.39	38.96	1
RSIN0022	LHL	18.06	18.42	0.36	1
RSIN0023	LHL	58.70	59.38	0.68	1
RSIN0023	LHU	49.92	52.11	2.19	1
RSIN0024	LHL	72.18	73.04	0.86	1
RSIN0024	LHU	62.15	64.17	2.02	1
RSIN0027	LHL	54.32	54.86	0.54	1
RSIN0027	LHU	45.19	47.33	2.14	1
RSIN0028	LHL	76.00	77.09	1.09	1
RSIN0028	LHU	23.29	70.07	46.78	1
RSIN0029	LHL	58.06	58.93	0.87	1
RSIN0029	LHU	55.47	57.93	2.46	1
RSIN0030	LHL	56.93	57.87	0.94	1
RSIN0030	LHU	54.29	56.80	2.51	1
RSIN0031	LHL	53.91	55.51	1.60	1
RSIN0031	LHL	68.72	69.68	0.96	2
RSIN0031	LHL	86.10	87.26	1.16	3
RSIN0031	LHU	50.78	53.75	2.97	1
RSIN0031	LHU	66.13	68.59	2.46	2
RSIN0031	LHU	83.59	86.01	2.42	3
RSIN0032	LHL	67.76	68.72	0.96	1
RSIN0032	LHL	87.04	87.99	0.95	2
RSIN0032	LHU	65.11	67.59	2.48	1
RSIN0032	LHU	84.40	86.91	2.51	2
RSIN0034	LHL	87.36	88.38	1.02	1
RSIN0034	LHU	60.75	87.21	26.46	1
RSIN0035	LHL	87.04	88.02	0.98	1
RSIN0035	LHU	84.41	86.87	2.46	1
RSIN0036	LHD	51.12	54.34	3.22	1
RSIN0037	LHL	60.75	61.73	0.98	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIN0037	LHU	57.84	60.60	2.76	1
RSIN0039	LHL	66.32	67.21	0.89	1
RSIN0039	LHU	62.68	66.14	3.46	1
RSIN0040	LHL	61.79	62.77	0.98	1
RSIN0040	LHU	59.19	61.64	2.45	1
RSIN0041	LHL	63.22	64.19	0.97	1
RSIN0041	LHL	105.20	106.21	1.01	2
RSIN0041	LHU	60.65	63.05	2.40	1
RSIN0041	LHU	101.70	105.05	3.35	2
RSIN0043	LHL	61.82	62.72	0.90	1
RSIN0043	LHU	59.37	61.64	2.27	1
RSIN0044	LHL	83.13	84.10	0.97	1
RSIN0044	LHU	80.60	82.97	2.37	1
RSIN0045	LHL	64.35	65.27	0.92	1
RSIN0045	LHU	61.85	64.15	2.30	1
RSIN0046	LHL	60.30	61.38	1.08	1
RSIN0046	LHU	57.70	60.12	2.42	1
RSIN0047	LHL	79.67	81.22	1.55	1
RSIN0047	LHU	58.60	79.48	20.88	1
RSIN0048	LHL	59.19	60.36	1.17	1
RSIN0048	LHU	56.53	58.98	2.45	1
RSIN0049	LHL	19.43	20.15	0.72	1
RSIN0049	LHU	11.83	14.18	2.35	1
RSIN0050	LHL	18.27	42.24	23.97	1
RSIN0050	LHU	11.00	12.07	1.07	1
RSIN0051	LHL	19.41	20.27	0.86	1
RSIN0051	LHL	53.99	54.75	0.76	2
RSIN0051	LHU	12.66	13.70	1.04	1
RSIN0051	LHU	45.96	49.18	3.22	2
RSIN0052	LHL	20.28	21.03	0.75	1
RSIN0052	LHU	13.15	15.67	2.52	1
RSIN0053	LHL	169.25	169.50	0.25	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIN0053	LHU	165.83	168.50	2.67	1
RSIN0054	LHL	22.67	23.50	0.83	1
RSIN0054	LHU	15.02	17.30	2.28	1
RSIN0055	LHL	41.72	42.78	1.06	1
RSIN0055	LHU	38.52	41.15	2.63	1
RSIN0056	LHL	76.55	83.55	7.00	1
RSIN0056	LHU	35.18	75.60	40.42	1
RSIN0068	LHD	34.19	39.00	4.81	1
RSIN0069	LHD	34.91	39.50	4.59	1
RSIN0070	LHL	63.90	64.92	1.02	1
RSIN0070	LHU	60.24	63.80	3.56	1
RSIN0071	LHD	56.00	59.77	3.77	1
RSIN0072	LHL	35.94	37.15	1.21	1
RSIN0072	LHU	32.35	35.65	3.30	1
RSIN0076	LHL	50.48	50.87	0.39	1
RSIN0076	LHL	78.83	79.13	0.30	2
RSIN0076	LHU	40.37	42.25	1.88	1
RSIN0076	LHU	68.60	70.70	2.10	2
RSIN0077	LHL	49.45	50.37	0.92	1
RSIN0077	LHL	77.35	77.60	0.25	2
RSIN0077	LHU	37.10	39.05	1.95	1
RSIN0077	LHU	67.10	68.80	1.70	2
RSIN0078	LHL	61.38	61.69	0.31	1
RSIN0078	LHU	52.60	54.65	2.05	1
RSIN0079	LHL	56.85	57.90	1.05	1
RSIN0079	LHU	54.20	56.60	2.40	1
RSIN0080	LHL	69.40	70.45	1.05	1
RSIN0080	LHU	66.65	69.20	2.55	1
RSIN0081	LHD	58.10	61.55	3.45	1
RSIN0082	LHD	57.81	61.20	3.39	1
RSIN0083	LHD	58.30	73.98	15.68	1
RSIN0084	LHD	18.98	22.35	3.37	1

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIN0085	LHD	16.21	17.55	1.34	1
RSIN0086	LHD	50.00	53.50	3.50	1
RSIN0087	LHD	75.42	78.75	3.33	1
RSIN0088	LHL	74.52	74.80	0.28	1
RSIN0088	LHU	65.54	67.49	1.95	1
RSIN0089	LHU	55.15	58.25	3.10	1
RSIN0105	LHD	58.05	61.85	3.80	1
RSIN0106	LHD	50.40	53.97	3.57	1
RSIN0107	LHD	48.55	51.70	3.15	1
RSIN0108	LHD	50.60	54.10	3.50	1
RSIN0109	LHD	46.60	47.65	1.05	1
RSIN0111	LHD	49.40	52.95	3.55	1
RSIN0112	LHD	54.10	57.45	3.35	1
RSIN0113	LHD	51.15	54.72	3.57	1
RSIN0114	LHD	78.00	81.65	3.65	1
RSIN0115	LHD	82.20	85.80	3.60	1
RSIN0116	LHD	85.90	86.50	0.60	1
RSIN0117	LHD	83.00	86.55	3.55	1
RSIN0118	LHD	61.00	67.30	6.30	1
RSIE0342	LHD	123.96	132.2	8.24	
RSIE0342	LHD	162.95	166.55	3.60	Seam Repeat
RSIE0343	LHD	162.86	166.53	3.67	
RSIE0346	LHD	124.18	128	3.82	
RSIE0346	LHD	143.89	146.95	3.06	Seam Repeat
RSIE0347	LHD	130	135	5.00	
RSIE0347	LHD	136	141	5.00	Seam Repeat
RSIE0348	LHD	117.83	119.44	1.61	
RSIE0348	LHD	152.1	155.16	3.06	Seam Repeat
RSIN0125	LHD	67.82	70.56	2.74	
RSIN0125	LHD	149	152.79	3.79	Seam Repeat
RSIN0126	LHD	133.5	136.53	3.03	
RSIN0127	LHD	145.06	148.76	3.70	

Hole	Seam	DOR	DOF	Thick.	Seam Occurrence
RSIN0128	LHD	120.85	123.86	3.01	
RSIN0128	LHD	129.42	132.95	3.53	Seam Repeat
RSIN0129	LHD	138.37	142.09	3.72	
RSIN0130	LHD	136.32	139.79	3.47	
RSIN0131	LHD	126.27	129.81	3.54	
RSIN0133	LHD	169.4	174.75	5.35	
RSIN0134	LHD	171	174.85	3.85	
RSIN0136	LHD	156	159.66	3.66	

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memo

To: Jon Romcke

From: Bronwyn Leonard

CC: Leandro Pires

Date: 12/01/2020

Re: **Isaac Plains East Resource Estimate Update to 31st Dec 2020**

Stanmore Coal have requested an update to the coal resource estimate for the Isaac Plains East deposit to the 31st Dec 2020. There has been no change to the geological model or the resource confidence categories since the previous resource estimate in June 2020. The updated resource estimate is therefore simply the previous resource estimate depleted by mining from 1st July 2020 to 31st Dec 2020. This update should be read in conjunction with the previous resource report "IPE Resource Estimate June 2020".

Background

The Isaac Plains East (IPE) deposit is in the northern central Bowen Basin, Queensland, Australia. It lies approximately 10km east of Moranbah and is part of the Isaac Plains Mining Complex. IPE is covered by four (4) Mining Leases, ML 700016, ML 700017, ML 700018, and ML 700019, each of which was granted to Stanmore IP Coal Pty Ltd on 1st March 2018.

The Leichhardt (LHD) Seam of the Rangal Coal Measures forms the principal economic coal resource in the IPE area. The seam is typically 2.8 m thick. The IPE deposit is hosted within a synclinal structure that plunges generally to the east north east. Immediately west of the IPE deposit is the major Burton Range Thrust Fault which delimits the down dip extent of the LHD seam from the western Isaac Plains Mine resource.

The current geological model was updated in May 2020 and no additional exploration work has been carried out since then. The previous resource report ("IPE Resource Estimate June 2020") provides details on modelling method and the data used in the model.

Coal Resources have been estimated for the LHD at IPE in accordance with the JORC Code, 2012. The previous resource report ("IPE Resource Estimate June 2020') provides details on the Points of Observation used to define the resource categories, the limitations applied to the resource polygons and the method used to determine in-situ coal tonnages.

Previous Resource Estimate

The previous resource estimate from June 2020 showed a total coal resource of 22 Mt (as at 30th June 2020).

Table 1 Insitu Coal Resources by Seam (30th June 2020)

Seam	Resource Category			Total (Mt)
	Measured (Mt)	Indicated (Mt)	Inferred (Mt)	
LHD	9.8	8.0	4	22

The resource estimate was based on the May 31st survey face positions and forecast mining to 30th June 2020. The forecast was for mining to progress to the end of Pit 3 Strip 7 and consume 213,480t of the resource. In reality mining only progressed to half way through Pit 3 Strip 7 by the 30th June and coal was also mined from the southern end of Pit 4 (previous Pit 3/ 4 land bridge). However, the total coal resource consumed (211,524t) was not materially different to the forecast.

Resource Depletion

From the 1st July to the 31st Dec mining continued in Pit 2 (Strip 8, 9 and 10), Pit 3 (completed remainder of Strip 7 and also Strip 8) and Pit 4 (completed Strip 9 and half of Strip 10). The mined areas are shown in Figure 1. Face positions from 30th June and 31st Dec were used to calculate the resource consumed in this 6-month period (using the same geological model and assumptions as the previous resource estimate). The total resource depletion was 1.40Mt. All the consumed resource was Leichhardt Seam (LHD) with a depth of cover of 0-100m. It was split between ML00017 (0.67Mt) and ML700018 (0.73Mt). The average raw ash of the depleted area was 13.0% (air dried basis), which is lower than the average raw ash of the total measured resource. This means the depletion has effectively raised the ash of remaining resource from 14.2% to 14.7%.

Updated Resource Estimate

The updated resource estimate as at 31st Dec 2020 is given in the tables below.

Table 2 In-situ Coal Resources by Seam (31st Dec 2020)

Seam	Resource Category			Total (Mt)
	Measured (Mt)	Indicated (Mt)	Inferred (Mt)	
LHD	8.4	8.0	4	20

Table 3 In-situ Coal Resources by Lease (31st Dec 2020)

Lease	Resource Category			Total (Mt)
	Measured (Mt)	Indicated (Mt)	Inferred (Mt)	
ML700016		0.3	1	1
ML700017	3.9	5.4	1	11
ML700018	2.5			3
ML700019	2.0	2.5	2	6

Table 4 Coal Resources and Seam Thickness by Depth of Cover (31st Dec 2020)

Depth*		Resource Category			Total
		Measured	Indicated	Inferred	
0-50m	In-situ Resource (Mt)	1.4	0.6	1	3
	Average Thickness (m)	3.2	2.8	2.5	2.9
50-100m	In-situ Resource (Mt)	5.2	2.5	2	10
	Average Thickness (m)	2.9	3.1	2.5	3.0
100-150m	In-situ Resource (Mt)	1.7	3.9	1	7
	Average Thickness (m)	3.0	2.9	2.6	2.8
150-200m	In-situ Resource (Mt)	0.1	1.1	0	1
	Average Thickness (m)	3.1	2.9	2.6	2.8

*Depth is from the pre-mining topography and does not account for any pre-strip already undertaken in advance of mining

Table 5 Coal Resource Raw Quality (31st Dec 2020)

Raw Coal Quality	Resource Category			Total
	Measured	Indicated	Inferred	
RD (ad)	1.43	1.43	1.43	1.43
Ash (ad%)	14.7	14.9	15.5	14.7
Volatile Matter (ad%)	24.3	24.3	24.1	24.2
Moisture (ad%)	1.9	1.8	2.0	1.9
Total Sulphur (ad%)	0.50	0.47	0.41	0.48

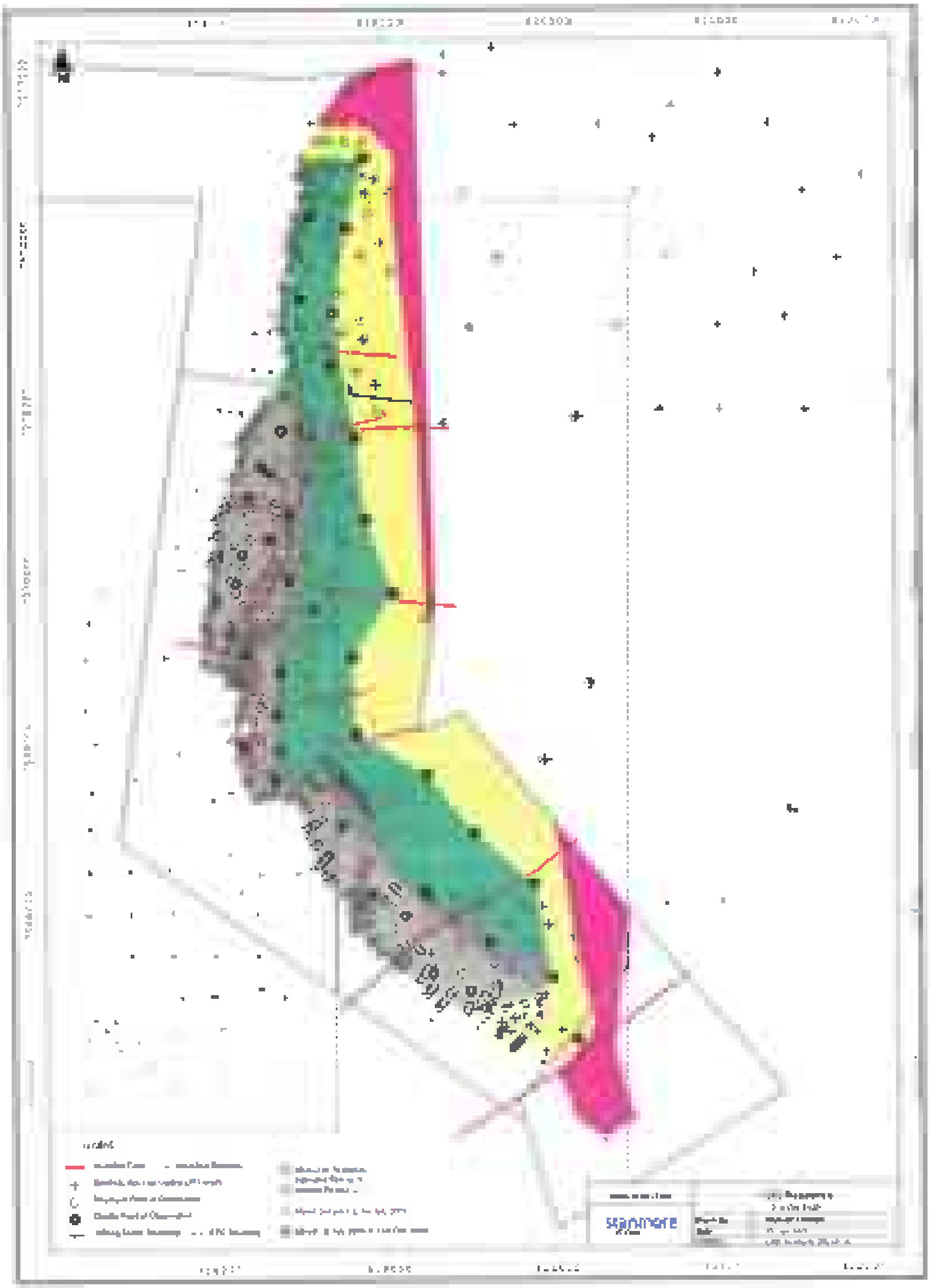

Figure 1 Resource Areas 31st Dec 2020

Competent Person Statement

I, Bronwyn Leonard, confirm that I am the Competent Person for this report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code 2012 Edition, having at least five years of experience that is relevant to the style of mineralisation and type of deposit described in this Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy (AusIMM).
- I have reviewed the Report to which this Consent Statement applies.

I am a full time employee of Stanmore IP Coal Pty Ltd and have been engaged by Stanmore Coal Ltd to prepare the documentation for Isaac Plains East, on which the Report is based, for the period ended 30th June 2020.

I have more than 15 years' experience in exploration and coal resource modelling in both Queensland and New Zealand. I have been involved with the estimation of coal resources since 2010 and have acted as a Competent Person for resource reporting since 2013. I have previously been employed by Solid Energy New Zealand, Xstrata Coal Queensland and Glenore Newlands Opencut. I have been employed by Stanmore IP Coal and based at the Isaac Plains Mine as the Superintendent Mine Geology since October 2017.

Dr Bronwyn Leonard, PhD

MAusIMM 315295

Stanmore IP Coal

Isaac Plains East Resource Estimate

June 2020

Bronwyn Leonard
10/06/2020

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Executive Summary

This report provides an estimate of the Coal Resources contained within the Isaac Plains East (IPE) deposit as at the end of June 2020. The Coal Resource estimate and report have been prepared in accordance with the principles of the Joint Ore Reserves Committee's Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code) 2012 Edition.

The IPE deposit lies approximately 10km east of Moranbah and is part of the Isaac Plains Mining Complex. The coal resources are within mining leases (ML 700016, ML 700017, ML 700018, and ML 700019) that are held by Stanmore IP Coal Pty Ltd. The coal resources are all contained within the Leichhardt Seam of the Late Permian Rangal Coal Measures. The resources have been estimated using Maptek's Vulcan modelling software and are based on the May 2020 geological model.

The total IPE Coal Resource estimate at as 30th June 2020 is 22 million tonnes (Mt), of which 10Mt is classified as Measured Resources, 8 Mt is classified as Indicated Resources and 4Mt is classified as Inferred Resources.

1. Introduction

1.1. Location

The Isaac Plains East (IPE) deposit is in the northern central Bowen Basin, Queensland, Australia. It lies approximately 10km east of Moranbah and is part of the Isaac Plains Mining Complex. The deposit is immediately south of the Goonyella – Hay Point Railway and north of the Peak Downs Highway. The Hay Point / Dalyrmple Bay Coal export facility is some away 170 kilometres by rail (Figure 1-1).

1.2. Tenure

IPE is covered by four (4) Mining Leases, ML 700016, ML 700017, ML 700018, and ML 700019, each of which was granted to Stanmore IP Coal Pty Ltd on 1st March 2018 (Figure 1-2). MDLs 135 & 137 were both pre-cursor permits to the IPE Mining Leases. MDL 135 is now entirely covered by Mining Leases 700018 and 700019 but MDL 137 is still current, given that portions of the permit continue to exist outside of the newly granted Mining Lease areas.

A granted Petroleum Lease ("PL") 191 is held by CH4 Pty Ltd and covers the western half of the IPE area and also overlies the neighbouring Isaac Plains ML. The eastern half and northern portion of the IPE area are overlain by a Petroleum Lease Application ("PLa") 1034 and Authority to Prospect for petroleum ("ATP") 814 under the ownership of Eureka Petroleum Pty Ltd.

A majority of the tenure area falls on the Wotonga Station property, with a smaller section in the northern area falling on the Broadlea property.

TABLE 1-1 TENURE DETAILS

Tenure	Tenement Holder	Grant Date	Expiry Date	Area (Ha)
ML700016	Stanmore IP Coal Pty Ltd	01-Apr-2018	31-Mar-2030	138.50
ML700017	Stanmore IP Coal Pty Ltd	01-Apr-2018	31-Mar-2030	387.60
ML700018	Stanmore IP Coal Pty Ltd	01-Apr-2018	31-Mar-2030	369.10
ML700019	Stanmore IP Coal Pty Ltd	01-Apr-2018	31-Mar-2030	353.80
MDL135	Stanmore Wotonga Pty Ltd	10–Feb-1993	1–Mar-2018*	589.41
MDL137	Stanmore IP Sth Pty Ltd	10–Feb-1993	30-June-2023	652.00

*MDL135 was extinguished upon grant of ML700018 and 70019 which fully overlie its area

FIGURE 1-1 PROJECT LOCATION MAP

FIGURE 1-2 PROJECT TENURE

stanmore IP Coal

2. Geological Setting

2.1. Regional Geology

The Isaac Plains Mine is located in the northern part of the Permo-Triassic Bowen Basin containing principally fluvial and some marine sediments. The Bowen Basin is part of a connected group of Permo-Triassic basins in eastern Australia which includes the Sydney and Gunnedah Basins. The Basins axis orientation is NNW-SSE, roughly parallel to the Palaeozoic continental margin. Structurally, the deposit lies on the western boundary of the deformed Nebo Synclinorium immediately east of a regional thrust fault system- the Burton Range Thrust. The economic coal seams are contained in the Late Permian Rangal Coal Measures which is an approximately 100 m thick regional geological formation. The Rangal Coal Measures are underlain by the Fort Cooper Coal Measures and overlain by the Late Permian to Early Triassic Rewan Group.

Table 2-1 Regional Stratigraphy

Age	Group	Formation
Tertiary		
Triassic	Rewan Group	Rewan Formation
Permian	Blackwater Group	Rangal Coal Measures
		Fort Cooper Coal Measures
		Moranbah Coal Measures
		Exmoor Formation
	Back Creek Group	Blenheim Formation
		Gebbie Formation
		Tiverton Formation

2.2. Local Geology

2.2.1. Stratigraphy

The Quaternary sediments in the IPE area are comprised of soil and sub soil derived from the underlying Permian sediments with limited alluvials along the creeks. The Tertiary unit ranges from 2 to 30 m thick and averages <10m thickness. Tertiary basalt infills paleochannels that cut roughly east west across the deposit at the boundary between ML700018 and ML700016, at the boundary between ML700018 and ML700019 and at the northern limit of ML700019.

The Leichhardt (LHD) Seam of the Rangal Coal Measures forms the principal economic coal resource in the IPE area (Figure 2-1). The seam is typically 2.8 m thick and rarely includes some stone bands that are consistent over relatively short distances. Anomalously thinning and thickening of the seam occurs around faulting and the weathering horizon. Beneath the LHD seam there are a number of smaller coal occurrences, which develop at approximately 8m to 30m below and range between 0.3m to 1.2m thickness. The L2 seam split has been identified and modelled across the IPE deposit but it is not considered to be resource. The L2 is sometimes referred to as the Leichhardt Lower

Seam but it should be noted that is not equivalent to the Leichhardt Lower ply (LHL) mined in the Isaac Plain deposit.

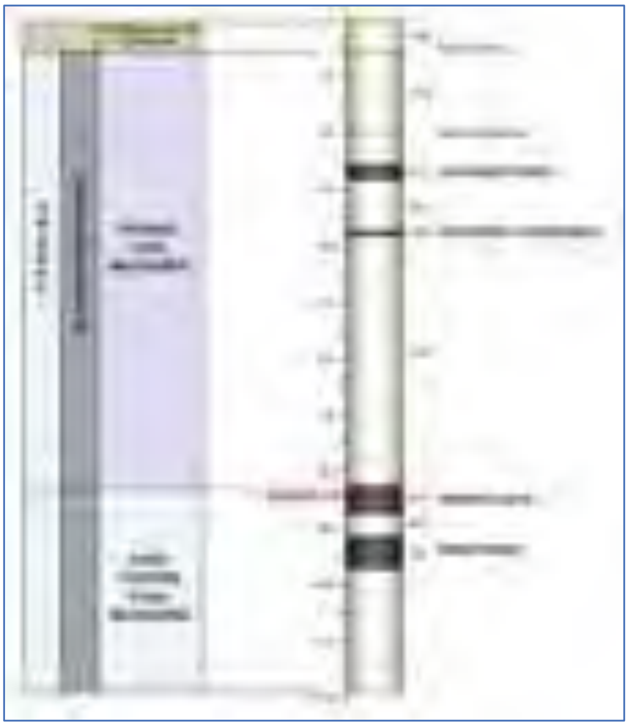


FIGURE 2-1 LOCAL STRATIGRAPHY

2.2.2. Structure

The IPE deposit is hosted within a synclinal structure that plunges generally to the east north east. Immediately west of the IPE deposit is the major Burton Range Thrust Fault which delimits the down dip extent of the LHD seam from the western Isaac Plains Mine resource. Locally the Burton Range Thrust fault is interpreted to have a throw in the order of 180m or slightly greater.

Within the IPE deposit the most significant fault system is a north south trending pair of thrust faults which run parallel to Pit 4. The throw on each of these faults is ~12m and between the faults is a zone of intense deformation. To the north of Pit 4 the faults appear to merge to a single structure with greater displacement (~30m) and the fault partially defines the western limit of the deposit in Pit 5.
There is a major east-west trending normal fault present between Pit 2 and Pit 3 with ~10m displacement. Minor east-west and northeast-southwest trending faults have also been observed as mining has progressed in Pits 2, 3 and 4. These structures typically transition from distinct faults to rolls over distances of 50-100m and are not easily predicted from exploration data. They are mostly normal faults and have been observed in boreholes where the seam is absent or anomalously thinned. Mini-sosie seismic surveys have also provided evidence of the faulting across the mine area but as the pit has progressed some of the fault interpretations from the seismic data have been found to be anomalous, with either no fault observed in the pit or a smaller structure than inferred from the seismic data. The seismic surveys have therefore been used to assist fault modelling but not all seismic faults have been included in the geological model.

2.2.3. Quality

The LHD full seam raw ashes at IPE range from 10% to 21% (adb). The majority of the LHD seam mined from IPE has been processed to produce a single semi-soft coking coal product at 9.5% ash (adb). Minor amounts of secondary thermal product has been produced, most notably along the limit of oxidation in the initial boxcuts where weathering had led to a degradation of the inherent coking properties of the seam.

David Hornsby of Minserve Group reviewed and assessed the coal quality (and dilution) dataset and produced simulated yields for each of IPE borecores. Various processing options were investigated but the current set-up used to process the IPE coal is to target a 9.5% Primary product and a 16% secondary product with the coarse (+16mm) primary DMC fraction going to the primary product. Typically there is very little secondary product and where the total product ash of the combined primary and secondary products is less than 9.5% ash, the secondary product is diverted to the primary product belt and the coal is stockpiled as a single semi-soft coking coal product. Yield results are not directly assessed as part of the resource estimate, apart from providing evidence that the seam is amendable to beneficiation through the Isaac Plains CHPP and that a saleable product can be achieved.

3. Geological Data

3.1. Exploration Drilling

The exploration drilling data from IPE has historically been held in a series of modelling databases, first in Minescape and then in Vulcan. In late 2018 Stanmore Coal invested in a dedicated geological database (Geobank). This database is now fully populated and validated. Exporters have been created to allow the modelling databases to be generated directly from Geobank to ensure the data used for modelling is complete and consistent. The exporters also include the use of downhole survey data, where it is available.

There are 738 boreholes in the IPE geological model, of which 641 intercepted the LHD seam. The boreholes are a mixture of chipped and cored holes and were drilled in campaigns across several phases of exploration activity.

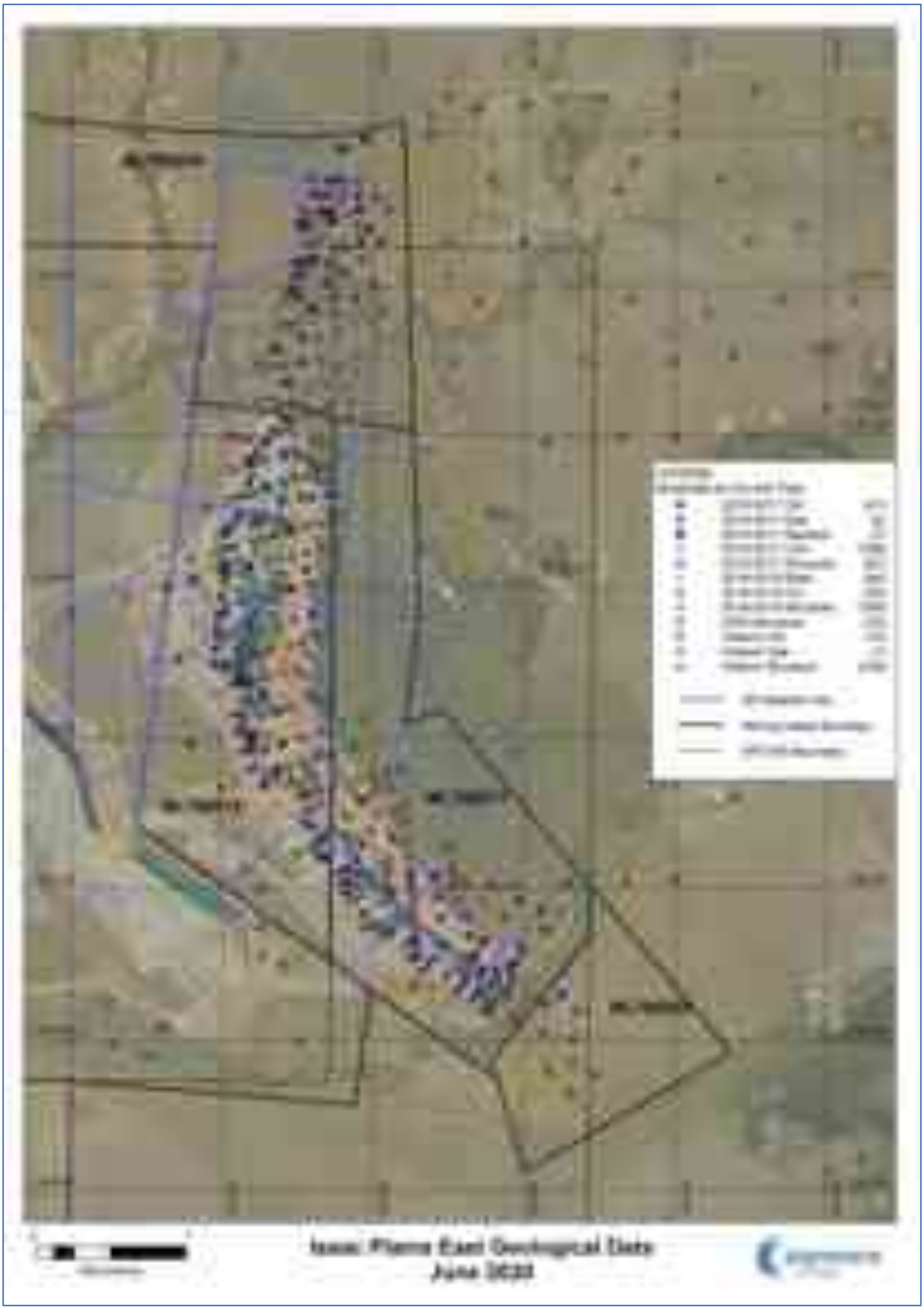


FIGURE 3-1 EXPLORATION DRILLING DATA

TABLE 3-1 DRILLING CAMPAIGNS

Era	Purpose	Number of holes	Number of holes with downhole geophysics
Historic (pre 2015)	CQ	13	1
	Structure	216	14
	Gas	1	1
2015-2017	CQ	21	21
	LOX	199	190
	Structure	67	65
	Gas	2	2
	Geotech	7	7
2018-2019	Blast	54	54
	CQ	20	20
	Structure	106	105
2020	Structure	32	30
Total		**738**	**510**

3.1.1. Historic Drilling

Prior to Stanmore acquiring the IPE leases in 2015, a significant amount of exploration had been conducted by previous tenure holders. The IPE geological model includes 230 historic holes which were provided to Stanmore by the previous tenure holder as part of the sale process. The majority of these (214) do not have geophysical logs available. The detail in the seam picks suggests that geophysical logs were run for at least some of the holes, but the data is not available to confirm this or allow validation of seam picks. In 2015 Xenith performed a review of the historic data on behalf of Stanmore and in the 2016 and 2017 drilling programs holes were 'twinned' with historic holes to test their reliability. Based on the results of this analysis, historic boreholes were included in the model, even without supporting geophysics, but these boreholes are not considered as 'Structural Points of Observation' for the resource estimate. Thirteen of the historic holes were cored holes drilled by TDM with washability data available. Theses hole were twinned with Stanmore chip holes with geophysical logs to increase the confidence in the seam picks and then used as 'Coal Quality Points of Observation' for the resource estimate.

Blue Energy Limited drilled several CSG wells within and around the area under ATP 814P in 2011. One hole, Sapphire_4 was drilled within the IPE area. Data supplied for this hole included detailed lithology and geophysical logs and this borehole was also included in the model and used as a 'Structural Point of Observation' for the resource estimate.

3.1.2. Stanmore Drilling

2015-2017 Drilling

Stanmore undertook a series of exploration campaigns from late 2015 through to late 2017 to collect data for the IPE pre-Feasibility and Feasibility studies and the mining lease application process. These included:

- Chip holes to confirm seam thicknesses and investigate possible faults
- Chip holes with samples taken to confirm limits of oxidation (LOX)
- Partially cored holes for coal quality data
- Partially cored holes for gas analysis
- Fully cored holes for geotechnical analysis

296 holes from this era are included in the geological model. There are eleven holes in the model without geophysical logs. These were shallow LOX holes and/or holes that did not intersect the coal seam and they are not considered as 'Structural Points of Observation' for the resource estimate.

2018-2019 Drilling

Mining commenced at IPE in 2018 and a program of intensive drilling was undertaken in the area of the planned pits to define faults and to collect additional structural data as the pit progressed down dip. This program also included geophysically logging 54 holes drilled as part of the drill and blast program. Ten cored holes were drilled within the planned Pit 2, 3 and 4 extents to improve forecasting of coal washability and eleven partially cored holes were drilled down dip of the current pits to provide additional coal quality data to support resource estimates. One hole was drilled as a water monitoring bore and was not geophysically logged but was included in the resource model to provide control on the base of weathering model. The hole intercepted the LHD but is not considered as a 'Structural Point of Observation' for the resource estimate.

2020 Drilling

A small drilling program was undertaken in early 2020 to investigate Pit 5. The program particularly targeted the northern area and the basalt extent. Two holes from this program could not be geophysically logged due to hole collapse. Neither hole intersected the LHD seam but they were included in the model to provide base of weathering and basalt thickness control.

3.2. Geophysical Surveys

3.2.1. Seismic

2D Mini-sosie surveys were undertaken as part of the 2016 exploration campaign to better understand the nature of the faulting and structure at IPE. Six survey lines had been completed at Isaac Plains in 2004/2005 which identified the location of the Burton Range Thrust but did not extend to the Isaac Plains East deposit. In 2016 Velseis Pty Ltd conducted a further survey of 15 additional lines covering a total of approximately 32 km. Of these additional lines, nine transect the IPE resource, seven lines across strike and down dip and two lines running in approximate parallel to the syncline covering and approximate total distance of 22km.

The seismic data interpretation was used to establish fault locations and seam structure and thickness continuity between the drilling data points and down dip from the last line of drilling. The major faults identified in the initial seismic interpretation have been truthed against the modelled faulting and used for validation. Minor faulting (where possible) has also been identified..

A major 3D Seismic survey (Vibroseis) was undertaken in 2017 to investigate the deep coal to the east of the Burton Range Thrust at Isaac Plains. This survey extended across the IPE mining leases but it was targeted on reflections from the deeper coal and therefore does not inform the IPE model or resource estimate.

3.2.2. Magnetics

In October / November 2017 a ground magnetic data survey was conducted by Atlas Geophysics. The survey was conducted over the entire IPE area at 50m line spacing running east west. The resultant data was reviewed by Geo Discovery Pty Ltd and an interpretation of the surface basalt coverage was produced (Figure 3-2). This data was used to inform the IPE model and to define the limits of the resource estimate where basalt was interpreted to intersect the coal seam.

3.2.3. Deep Ground Penetrating Radar

In May 2018 a Deep Ground Penetrating Radar (DGPR) survey was conducted by Ultramag Geophysics across the Pit 3 and Pit 4 area prior to mining commencing. The survey response from Pit 3 was poor, due to high ground water salinity, but the results from Pit 4 were used to assist in fault interpretation and provided targets for the 2018 drilling campaign.

FIGURE 3-2 GROUND MAGNETIC SURVEY DATA

3.3. Sampling and Analysis

3.3.1. Core sampling

For the Stanmore drilling programs all cored intervals were sampled where coal was present at thickness of 0.1m or more, with a maximum sample thickness of 0.5 m. Boreholes used for washability analysis were drilled at 4C or PQ size. Coal plies were sampled discretely on the basis of lithological characteristics and quality. All non-coal material and partings less than 0.1 m were included with the coal ply and noted in the lithological description. Non-coal interburden material greater than 0.1 m and up to a maximum of 0.3 m was sampled separately. Approximately 0.3 m of immediate roof and floor were also collected as dilution samples from the cores drilled in the 2015-2017 drilling programs.

3.3.2. Analysis

All coal quality samples were tested at NATA approved laboratory. Samples from the 2015-2017 drilling programs were sent to Bureau Veritas Laboratories in Brendale, Queensland. Samples from the 2018 program went to SGS Laboratories in Mackay, Queensland. Samples from the 2019 program went to Mitra PTS (formerly Preplab) in Gladstone, Queensland.

All samples followed the same analysis process. Ply samples were initially tested for Apparent Relative Density (ARD) before being composited into washability sections. This data was used to check that there was adequate (>90%) sample recovery for each of the washability sections.

In the earlier programs (2015-2017) two wash composites were typically created per seam. In the 2018 and 2019 programs a single full seam composite was produced for infill boreholes adjacent to the active mining area and two composites were created per seam for boreholes further downdip, away from any existing data.

To simulate mine transport conditions each composite sample was drop shattered 20 times from a height of 2 metres, any sample mass remaining of > 50 mm was hand knapped to 50 mm, dry tumbled and dry sized at 31.5 mm, 25 mm, 16 mm, 8 mm, 4 mm and 2 mm.
Composite samples were then split and further analysed as follows:

- 1/8 for quick coke: Crush to 11.2mm, float sink at 1.425 density, crush to 4mm and mill sample to test for Proximate, CSN and Gieseler & Dilatation
- 1/8 for raw analysis: Crush to 4mm, mill sample to test for RD, Proximate, TS, CSN. Selected samples were also tested for MHC, Calorific Value & Chlorine
- ¾ for float sink: Wet tumble and wet size at 31.5, 25, 16, 8, 4, 2, 1, 0.5, 0.25, 01.25 & 0.063mm. Re-combine samples in following fractions: -50+16mm, -16+8mm, -8+2mm and -2+0.25mm. Float sink each size fraction at densities (F1.30, F1.35, F1.375, F1.40, F1.45, F1.50, F1.55, F1.60, F1.70, F1.80, F2.00). -0.25+0mm fraction subject to tree froth flotation. All fractions analysed for ash and CSN.

Washability simulations were performed on the float sink results and from that data clean coal composite samples were compiled and analysed.

3.3.3. Historic TDM Data– Large Wash Simile

The historic washability data collected from the Thiess Dampier Mitsui (TDM) drilling in the mid-2000's was from smaller diameter cores that were not pre-treated and were crushed to a reduced top size such as an -11.2mm size fraction. This generates an imprecise size distribution which is not representative of mining and washing reality due to lesser fines production and inducement of excessive coarse coal breakage.

Stanmore engaged Chris McMahon (MCQR) to conduct a "large wash simile" process on the historic TDM small wash or crushed samples to better represent mining and washing reality. MCQR validated and produced large wash simile data from the TDM borecores by employing steps of density standardisation, pre-treatment alignment and size splitting of the crushed coal. This data was then used to produce yield simulations comparable to the Stanmore large washability data.

4. Geological Model

4.1. Modelling Method

4.1.1. Software

Modelling was done in Maptek's Vulcan 12.0.4 modelling software using the Integrated Stratigraphic Modelling package to produce grids and triangulations. FixDHD was used to interpolate drillhole data prior to structure modelling. The structural model was updated in May 2020 and the coal quality model was updated in Jan 2020. Both models were produced and validated by Bronwyn Leonard, Stanmore IP Coal.

4.1.2. Structural model

- Seam surfaces and thicknesses were modelled using triangulation
- Seams were stacked using the LHD roof as the reference surface
- Modelled grid size is 5m
- Seam grids were cropped to the Permian base of weathering
- Faults are treated as vertical and modelled using throw
- Dummy points were used to control the LHD roof to the west beyond the subcrop line and adjacent to some faults where data is sparse.
- The model extends beyond the IPE leases but the resource estimate is confined to the Stanmore MLs
- The topographic model is based on the 2015 aerial LiDAR survey which covered the full area of the mining leases. The survey accuracy was +/-0.25m.

TABLE 4-1 MODEL EXTENT

	Min	Max	Range
Easting	617310	622300	4990
Northing	7566030	7573340	7310

4.1.3. Coal Quality Model

Raw Quality

Raw coal quality data for each sample was composited using the Vulcan coal compositor. Samples were mass weighted to produce a single LHD value for each location. Modelled variables were: Relative Density (ad), Raw Ash (adb%), Inherent Moisture (adb %), Volatile Matter (adb%) and Total Sulphur (adb%). Raw coal quality models were generated using the inverse distance algorithm with a power factor of 2, using the same extents as the structural model and a grid size of 20m.

Yield

The yields modelled are 'in-seam' yields based on the borecore simulations performed by David Hornsby (Minserve). 'In-seam' yields are produced by running diluted simulated yields and then backing out the bulk dilution from the diluted yield [in-seam yield = diluted yield / (1 - % dilution)]. They differ from yields obtained by running the simulations on coal-only washability data because they contain a "process dilution" effect, meaning that some of the high ash dilution in the plant feed is misplaced to product due to process inefficiencies, particularly in the fines circuits, where it raises the circuit product ash. Yield grids were produced for the full seam Primary In-seam yield at 9.5% target ash and the secondary in-seam yield at 16% target, using the same algorithm and grid size as the raw coal grids.

4.2. Model Results

4.2.1. Structural Model

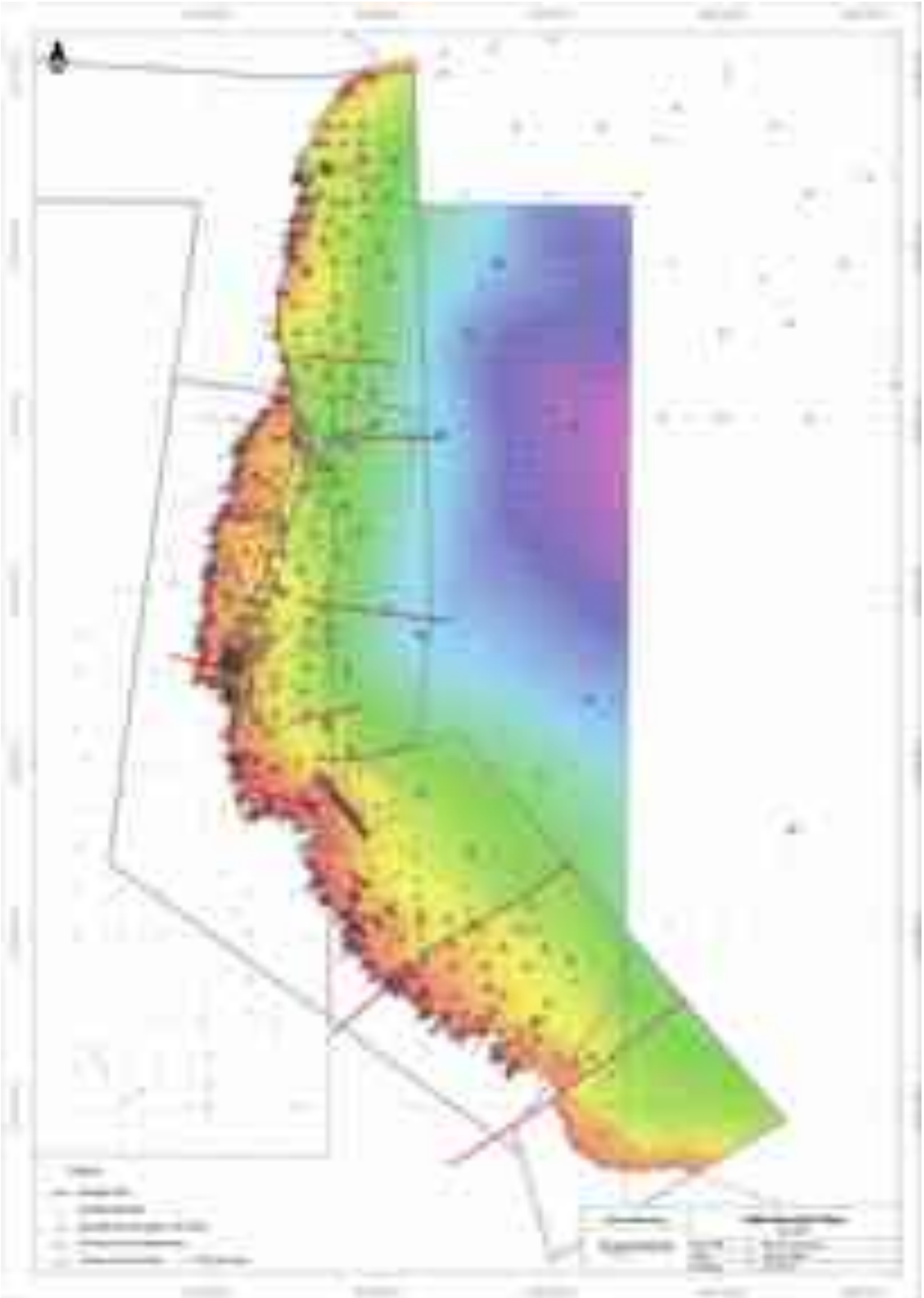


FIGURE 4-1 LHD STRUCTURE FLOOR CONTOURS

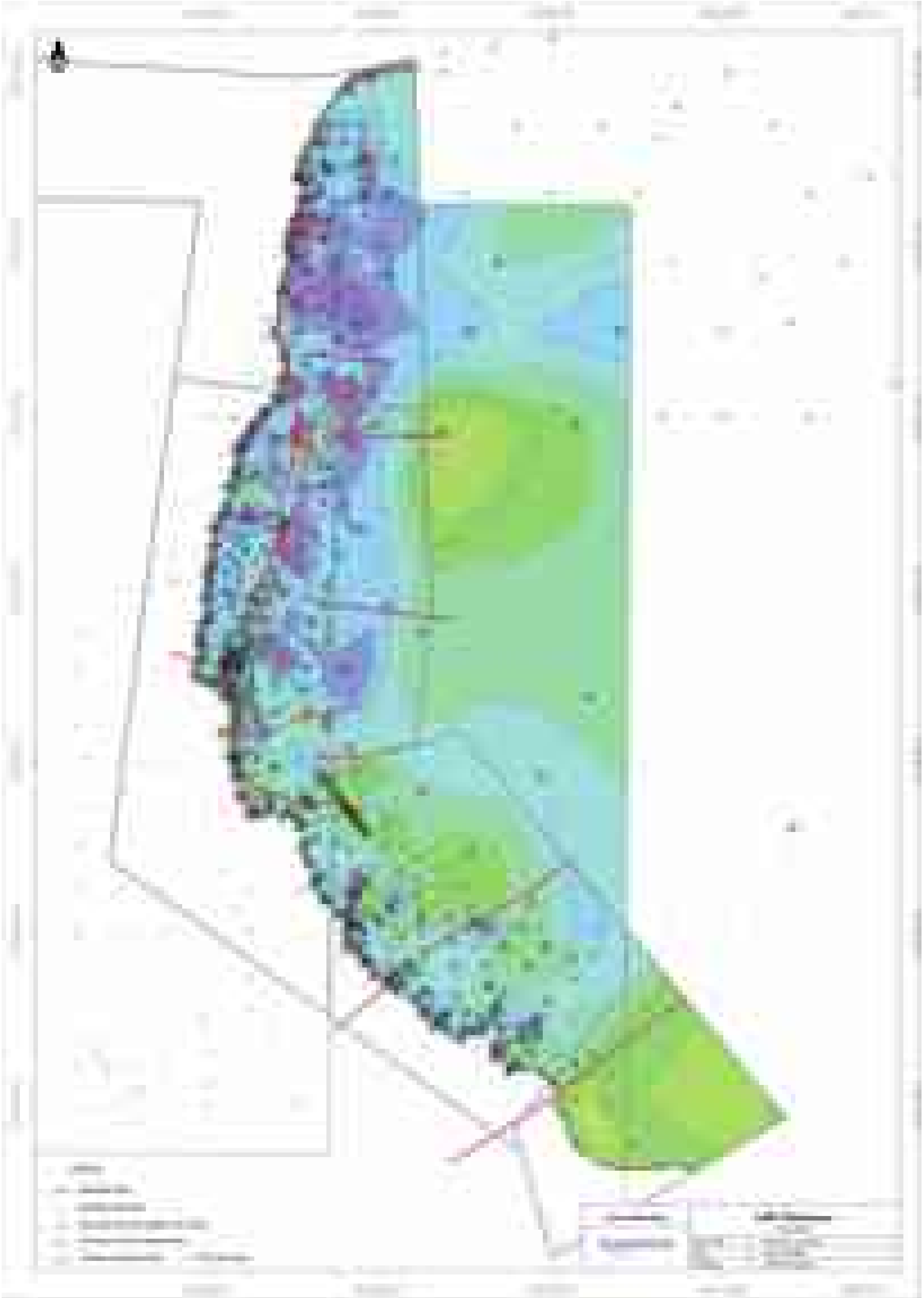


FIGURE 4-2 LHD SEAM THICKNESS

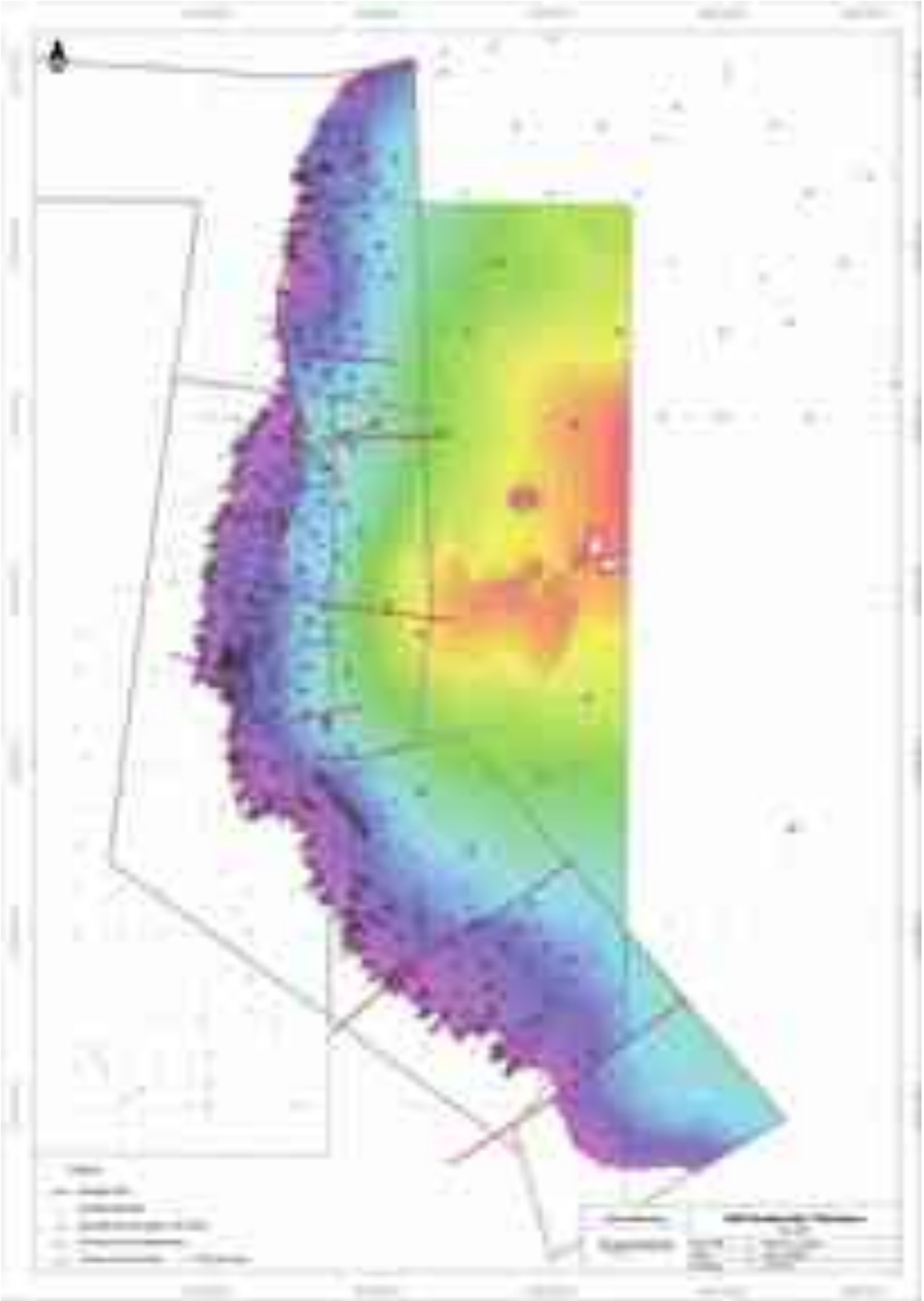


FIGURE 4-3 LHD OVERBURDEN THICKNESS

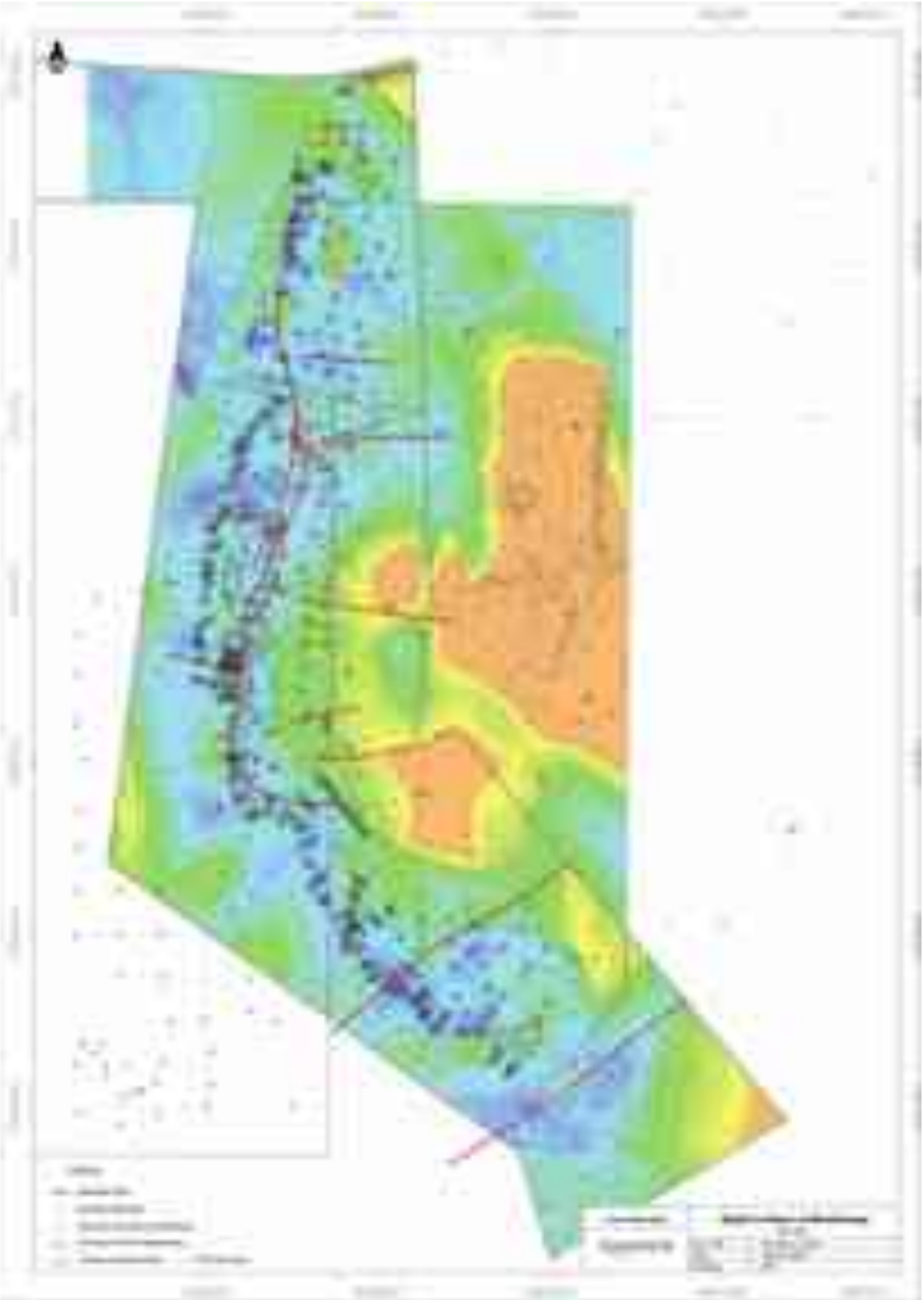


FIGURE 4-4 DEPTH TO BASE OF WEATHERING

4.2.2. Coal Quality Model

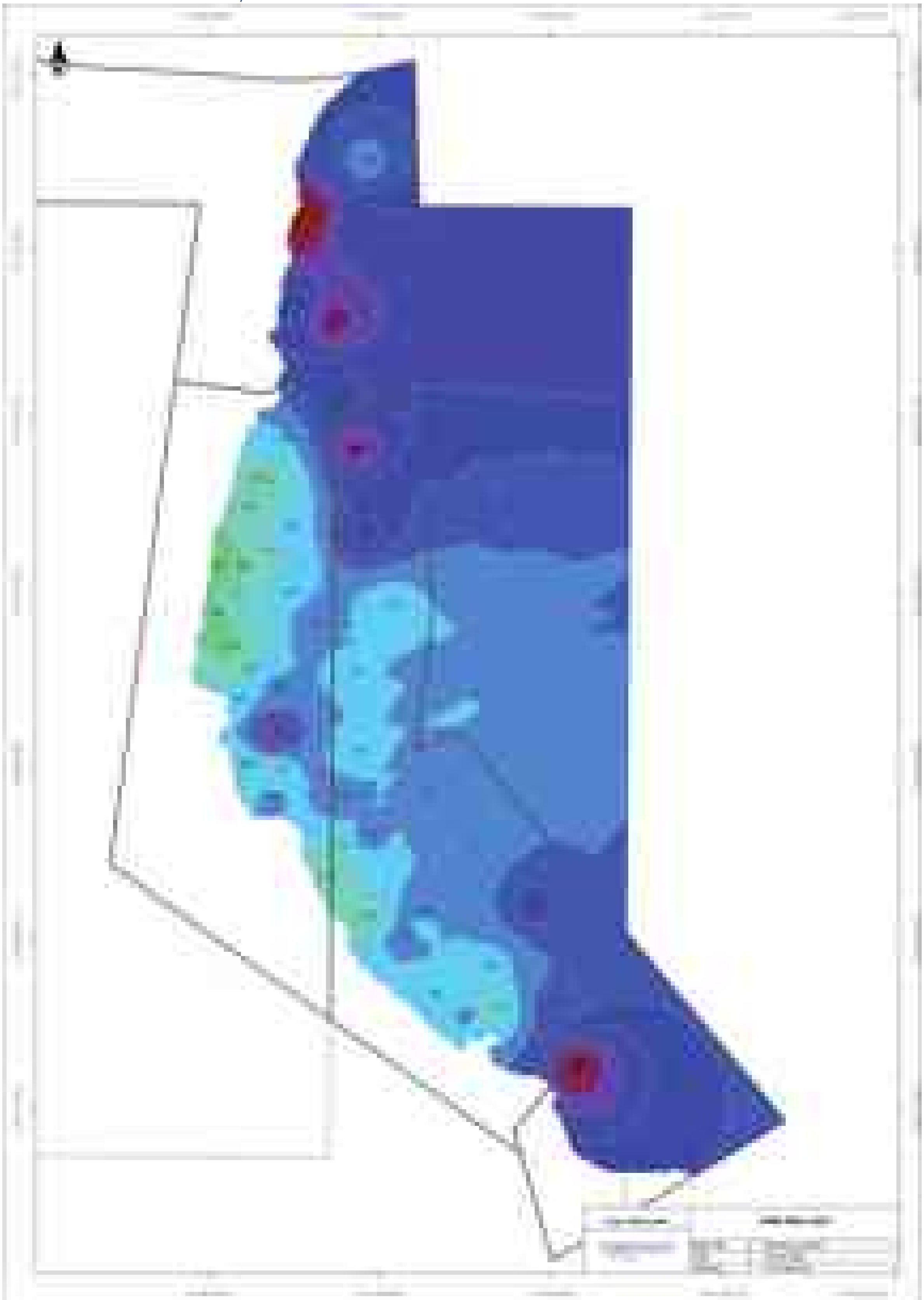


FIGURE 4-5 LHD FULL SEAM RAW ASH

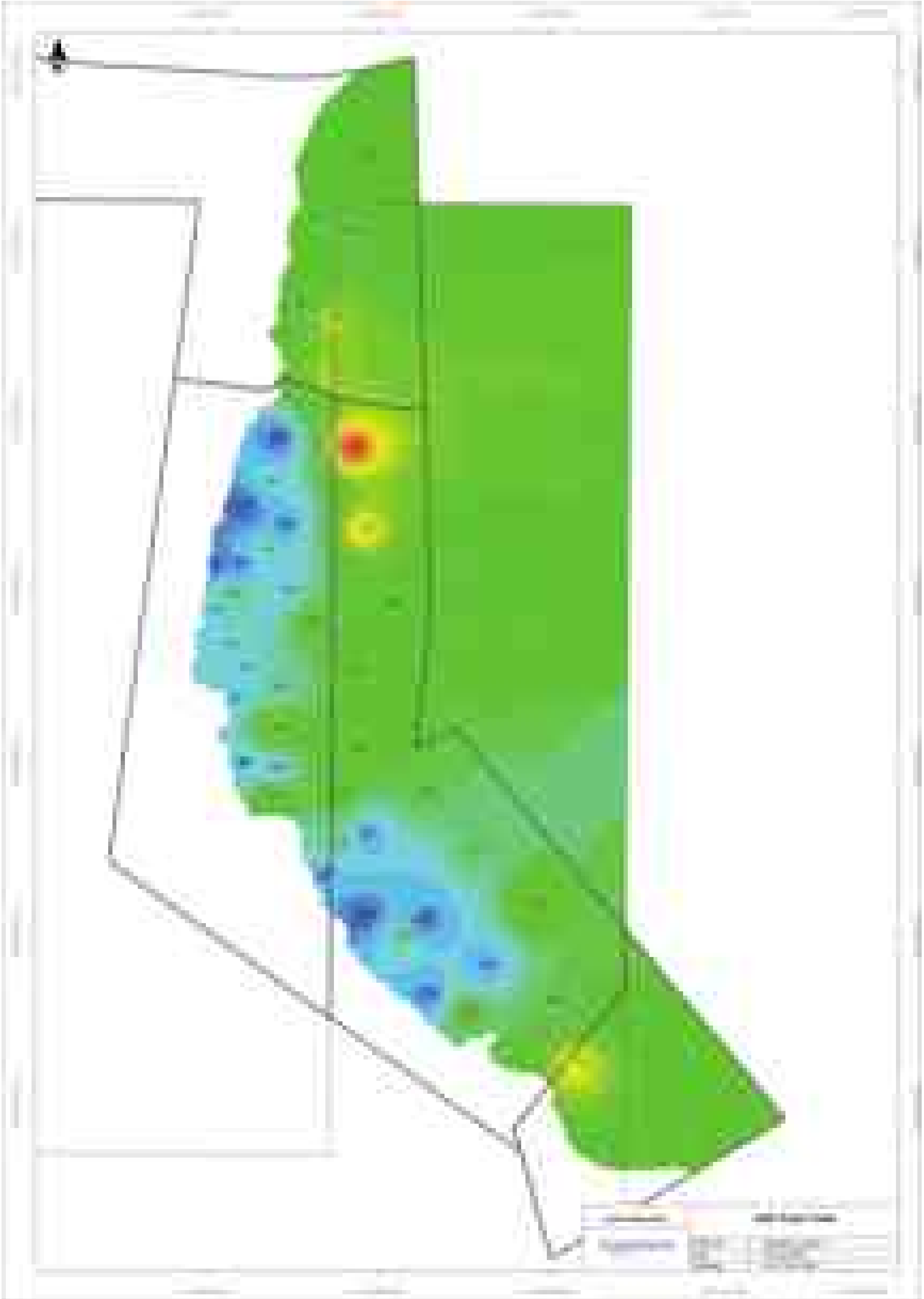


FIGURE 4-6 LHD TOTAL YIELD

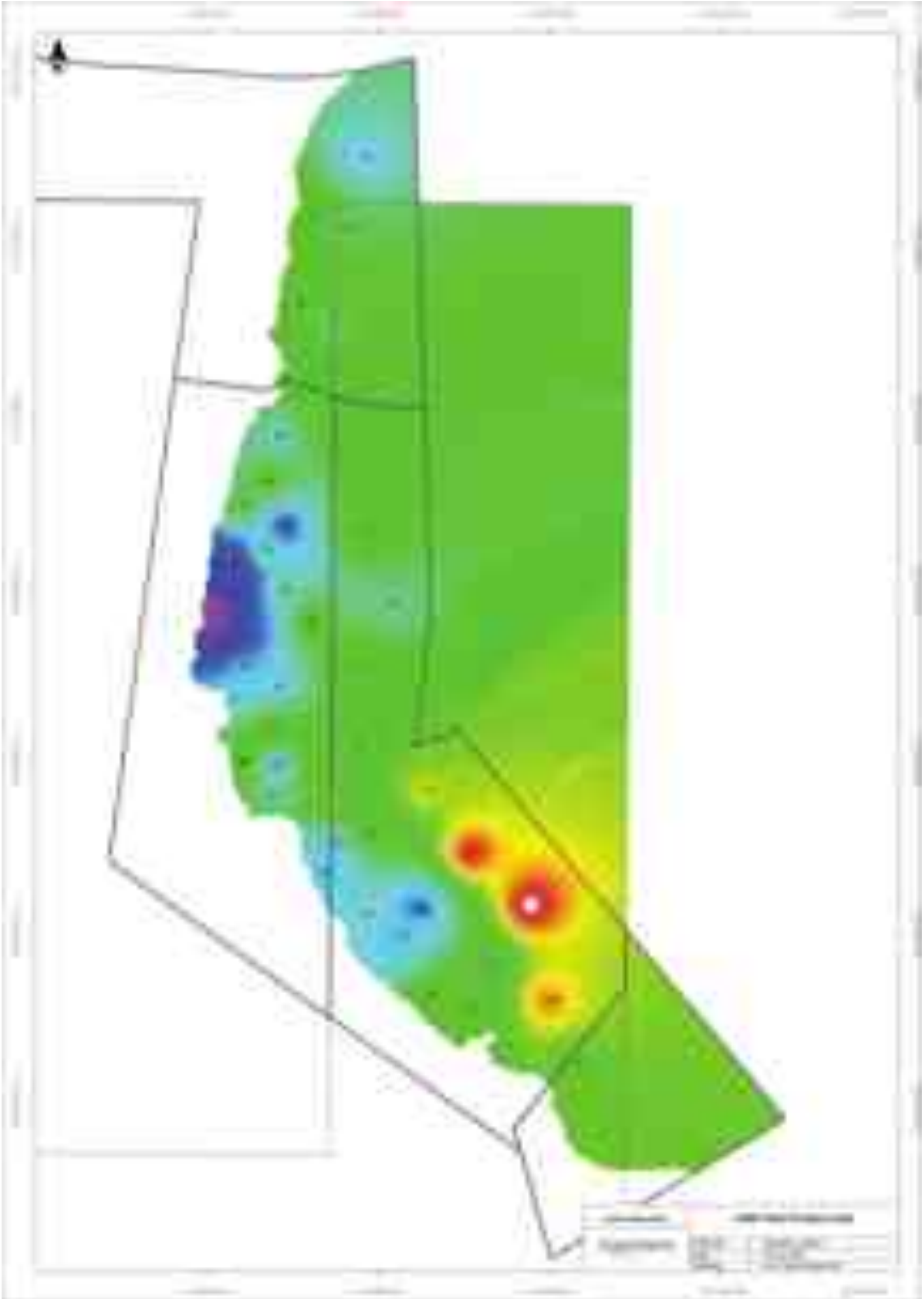


FIGURE 4-7 LHD TOTAL PRODUCT ASH

5. Resource Classification

Coal Resources have been estimated for the Leichhardt Seam (LHD) at IPE in accordance with the JORC Code, 2012. A Coal Resource as defined in the Code is not simply a summation of all the coal drilled or sampled, regardless of coal quality, mining dimensions, location or continuity. It is a realistic estimate of the coal that, under assumed and justifiable technical, economic and development conditions, is more like that not to become economically extractable.

The resource categories recognised under the JORC Code (2012) are:

Measured Mineral Resource

"When the nature, quality, amount and distribution of data are such as to leave ***no reasonable doubt****, in the opinion of the Competent Person determining the Mineral Resource, that the tonnage and grade of the mineralisation can be estimated to* ***within close limits****, and that any variation from the estimate would be unlikely to significantly affect potential economic viability."*

Indicated Mineral Resource

"When the nature, quality, amount and distribution of data are such as to allow ***confident*** *interpretation of the geological framework and to* ***assume*** *continuity of mineralisation."*

Inferred Mineral Resource

"That part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of ***limited geological evidence*** *and sampling. Geological evidence is sufficient to* ***imply*** *but not verify geological and grade (or quality) continuity."*

5.1. Points of Observation

According to the Australian Guidelines for the Estimation and Classification of Coal Resources (2014) 'Points of Observation' are:

"Sections of coal-bearing strata, at known locations, which provide information about the coal by observation, measurement and/or testing. They allow the presence of coal to be unambiguously determined."

Points of observation may be classed by quantity (structure) or coal quality.

For the IPE resource estimate coal quality points of observation are defined as:

- Cored boreholes with greater than 90% recovery across the seam or accepted by the Competent Person as being representative of the seam through analysis of the coal quality results, core photography and geophysical signature, and
- Raw and Washability coal quality data

Quantity (structure) points of observation are defined as:

- Boreholes with downhole geophysical gamma and density logs through the coal seam

Pre-Stanmore boreholes without geophysical logs are treated as supporting data points and not as points of observation.

5.2. Geostatistics

Variography was undertaken on the seam thickness and raw ash for the IPE deposit by Peter Handley of Measured Group (Appendix 2). Seam thickness is very consistent across the deposit and the structure points of observation were found to have a long range (5000m). Coal quality (as represented by raw ash) was more variable. The histogram for ash is positively skewed with a tail of high ash points. The range for the raw ash was determined to be 2000m. The coal quality of point of observation spacing was therefore set at 2000m for inferred, 1000m (1/2 of the range) for indicated and 500m (1/4 range) for measured.

5.3. Resource Polygons

Rather than constructing circles of influence around each point, triangulations were constructed with maximum side lengths based on the determined point of observation spacing. This created polygons which encompassed areas where the points of observation spacing was equal or less than the criteria but did not extend past the last point of observation. Separate triangulations were constructed for the quality points and the structure points.

5.3.1. Measured

Coal quality points of observation were considered most important for the determination of the Measure Resource polygon and the Measured polygon was primarily based on the limit of triangulation constructed at 500m spacing of quality points. The Measured polygon was extrapolated up dip toward the subcrop (or current pit exposure) but it was not extrapolated down dip any distance from the points of observation. The raw ash increases in holes to the north and south and this results in a significant drop in yield. Assumptions about yield and coking quality (and therefore price) would be considered key in any reserve evaluation, particularly as the seam gets deeper and extrapolation beyond the coal quality points of observations could not be justified.

5.3.2. Indicated

Indicated resources assume continuity of quality and thickness. Existing knowledge of the deposit, including the consistent coal quality observed in the areas mined to date, allows the assumption to be made that that any borehole with a clean geophysical signature and close to an existing coal quality hole could produce a similar semi-soft coal product to that currently produced from Isaac Plains East at a reasonable yield. The Indicated polygon was primarily based on the limit of the triangulation for the quality points of observation constructed at 1000m spacing but it was extrapolated to any structure points less than one third the observation spacing (330m) away from a coal quality point.

5.3.3. Inferred

Inferred resources imply continuity of quality and thickness and can rely on much less geological data. Existing knowledge of the deposit, including the consistent coal quality observed in the areas mined to date, implies that any borehole with a clean geophysical signature in the resource area could produce a similar semi-soft coal product to that currently produced from Isaac Plains East at a reasonable yield. The inferred polygon was therefore primarily based on the limit of the triangulation for the structure points of observation at 5000m spacing and does not directly rely on quality points. The value of 5000m is a nominal spacing as the maximum possible triangle side length is only 3060m. The Inferred polygon was limited at the mining lease boundary (as coal beyond this cannot be considered to have reasonable prospects of economic extraction) and therefore there was generally no need to extrapolate beyond the points of observation. The exception to this in the south of the deposit in ML700016 where there are no Stanmore boreholes with geophysical logs. In this area the polygon was extended 600m beyond the structure points of observation as far as supporting data points (historic boreholes with no geophysical logs and a 2D seismic line). The 2017 magnetic survey shows the presence of basalt paleochannels in this area, which have the potential to impact the seam, so no extrapolation is taken beyond the supporting data points.

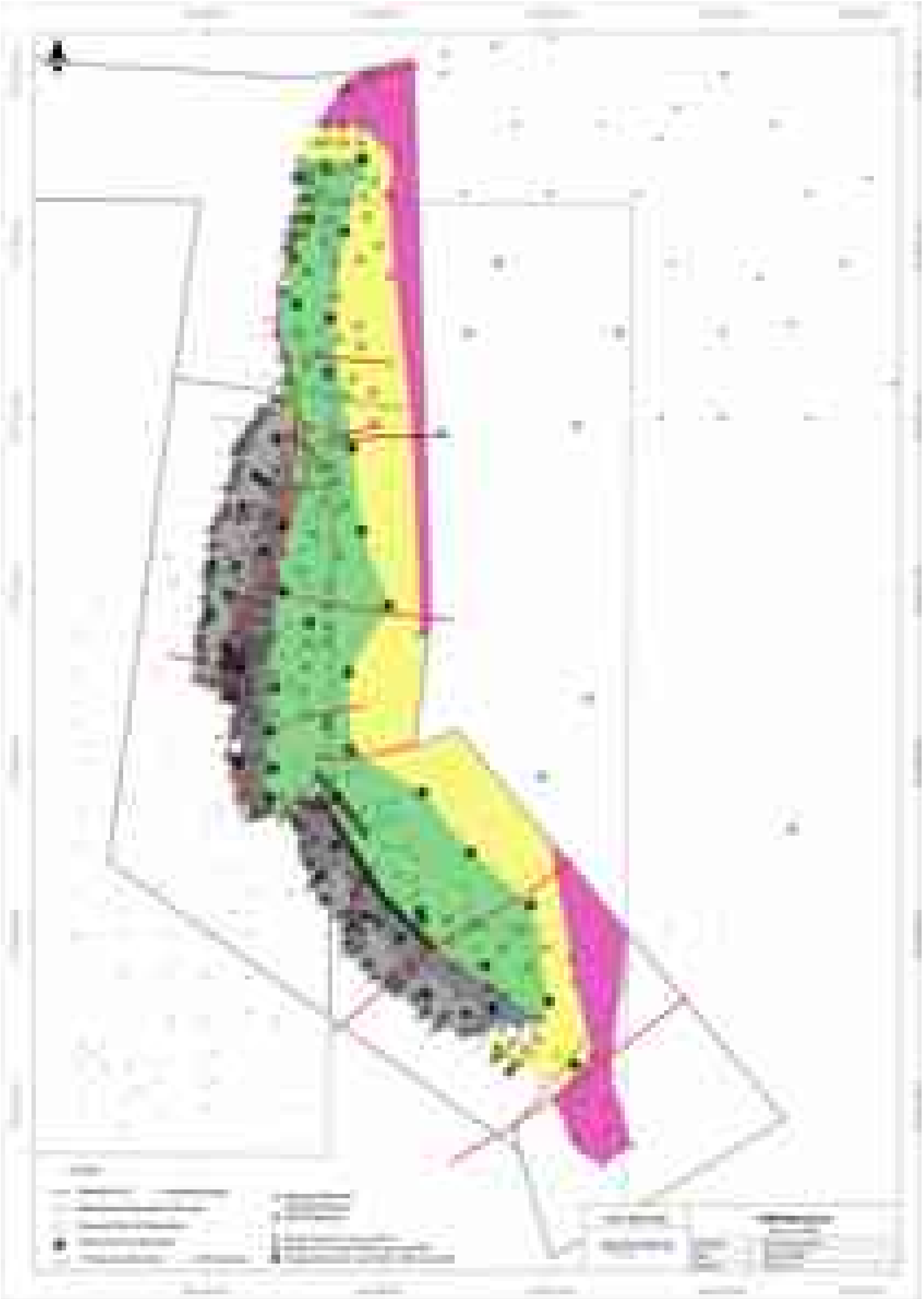


FIGURE 5-1 RESOURCE CLASSIFICATION POLYGONS

6. Resource Estimate

6.1. In-situ density

The resource estimate is an in-situ tonnage. The modelled relative densities (RD ad) were used to calculate an in-situ density using the Preston and Sanders formula:

$$RD(in\ situ) = \frac{RDad \times (100 - Mad)}{\{100 + RDad \times (ISM - Mad) - ISM\}}$$

Where:

RD (in situ) = Relative density (in situ moisture basis)

RDad = Relative density (air-dried basis)

Mad = Air dried moisture

ISM = in situ moisture (4.34%)

The average in-situ moisture value of 4.34% was calculated to assist with the Preston Sanders calculation based on the ACARP formula and values derived from Moisture Holding Capacity values for each sample (see formula below).

ISM = 0.348 + 1.1431 x MHC

MHC = Moisture Holding Capacity *(Formula derived from ACARP report C10041)*

The historic core data did not include values for RD. An Ash regression formula was derived based on the Stanmore analyses (see chart below) and used to calculate the RDad for the historic cores prior to modelling.

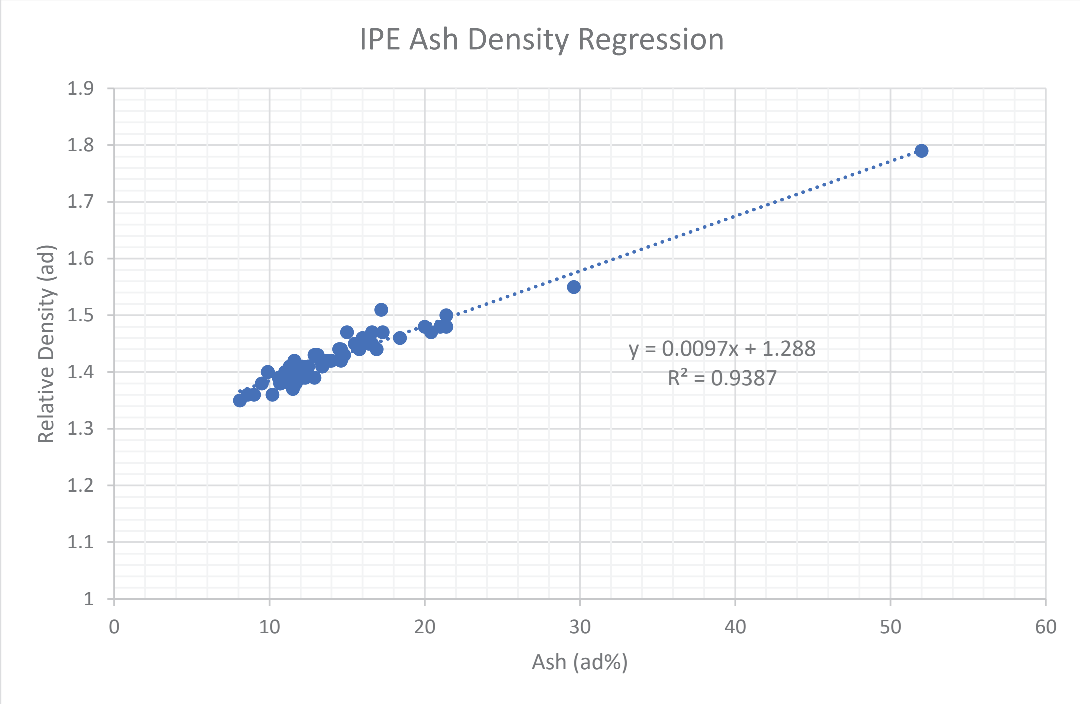


FIGURE 6-1 ASH DENSITY REGRESSION

6.2. 2020 IPE Coal Resources

The total Coal Resource estimate at as 30th June 2020 is 22 million tonnes (Mt), of which 10Mt is classified as Measured Resources, 8 Mt is classified as Indicated Resources and 4Mt is classified as Inferred Resources. The resources were estimated from the May 2020 structural model grids and the Jan 2020 quality model grids, using Maptek's Vulcan Reserve Utility (Rsvute).

TABLE 6-1 IN-SITU COAL RESOURCES BY SEAM

Seam	Resource Category			Total (Mt)
	Measured (Mt)	Indicated (Mt)	Inferred (Mt)	
LHD	9.8	8.0	4	22

TABLE 6-2 IN-SITU COAL RESOURCES BY LEASE

Lease	Resource Category			Total (Mt)
	Measured (Mt)	Indicated (Mt)	Inferred (Mt)	
ML700016		0.1	1	1
ML700017	4.5	5.4	1	11
ML700018	3.3			3
ML700019	2.0	2.5	2	6

TABLE 6-3 COAL RESOURCES AND SEAM THICKNESS BY DEPTH OF COVER

Depth*		Resource Category			Total
		Measured	Indicated	Inferred	
0-50m	In-situ Resource (Mt)	2.0	0.6	1	3
	Average Thickness (m)	3.0	2.8	2.5	2.9
50-100m	In-situ Resource (Mt)	6.0	2.5	2	10
	Average Thickness (m)	2.9	3.1	2.5	3.0
100-150m	In-situ Resource (Mt)	1.7	3.9	1	7
	Average Thickness (m)	3.0	2.9	2.6	2.8
150-200m	In-situ Resource (Mt)	0.1	1.1	0	1
	Average Thickness (m)	3.1	2.9	2.6	2.8

*Depth is from the pre-mining topography and does not account for any pre-strip already undertaken in advance of mining

TABLE 6-4 COAL RESOURCE RAW QUALITY

Raw Coal Quality	Resource Category			Total
	Measured	Indicated	Inferred	
RD (ad)	1.42	1.43	1.43	1.43
Ash (ad%)	14.2	14.9	15.5	14.7

Volatile Matter (ad%)	24.2	24.3	24.1	24.2
Moisture (ad%)	1.9	1.8	2.0	1.9
Total Sulphur (ad%)	0.51	0.47	0.41	0.48

TABLE 6-5 COAL RESOURCE SIMULATED WASHABILITY

In-seam Yield Simulations	Resource Category			Total
	Measured	Indicated	Inferred	
Primary In-seam Yield (%)	75.7	73.8	73.9	75.7
Primary Product Ash (ad%)	9.3	9.3	9.3	9.3
Secondary In-seam Yield (%)	3.3	4.0	4.0	3.3
Secondary Product Ash (ad%)	18.0	18.0	17.9	18.0
Total In-seam Yield (%)	78.9	77.8	77.9	78.9
Total Product Ash (ad%)	9.6	9.7	9.6	9.6

6.3. Reconciliation to previous resource estimate

The previous resource estimate was conducted by Troy Turner of Xenith in May 2018. The geological model used for the resource estimate was based on the exploration data available up to the end of 2017. At the time of the 2018 resource estimate no coal had been mined from IPE.

Comparison to the previous estimate shows that there is a decrease in the total Coal Resource of approximately 8Mt. A majority of the this (5.3Mt) is due to mining. Despite the increased number of boreholes in the 2020 geological model there is very little difference in the tonnage estimate from the two models. Using the 2018 resource classification polygons and the 2020 geological model gives a decrease in the tonnage estimate of only 0.1Mt. The remaining 2.7Mt decrease in resources is due to a reduction in the extent of the 2020 resource polygons compared to the 2018 polygons in the south of the deposit. As discussed above, due to the lack boreholes and the evidence of a basalt paleochannel, the 2020 resource polygons are not extrapolated to the lease boundaries in ML700016. In contrast the 2018 resource estimate included the full area of ML700016.

TABLE 6-6 COMPARISON OF 2018 AND 2020 RESOURCE ESTIMATES

	2018 Resource Estimate (Mt)	2020 Resource Estimate (Mt)	Difference (Mt)
Measured	12.9	9.8	-3.1
Indicated	8.8	8.0	-0.8
Inferred	8	4	-4
Total	**29.7**	**21.6**	**-8.1**
Mined to 30th June 2020		5.3	
Total incl mined	**29.7**	**26.8**	**-2.8**

New exploration drilling, particularly additional coal quality holes downdip of the current pits, has allowed to an upgrade in resource classification between 2018 and 2020. The 5.3Mt of coal mined since the 2018 resource estimate came from the 2018 measured resource. 2.2Mt tonnes of this depletion has been offset by upgrades of indicated and inferred resources to measured status and therefore the majority of the decrease in resources is in the lowest confidence inferred category at depths greater than 100m.

TABLE 6-7 COMPARISON OF 2018 AND 2020 RESOURCE ESTIMATES BY DEPTH

	2018 Resource Estimate (Mt)		2020 Resource Estimate (Mt)	
	<100m depth	**>100m depth**	**<100m depth**	**>100m depth**
Measured	12.6	0.3	8.0	1.8
Indicated	5.7	3.1	3.1	5.0
Inferred	2.6	5.6	2.5	1.2
Total	**20.9**	**9.0**	**13.6**	**8.0**

The new coal quality data has also led to an increase in the average ash estimated for the deposit from 13.8% in 2018 to 14.7% in 2020. This is because the new data came from areas away from the centre of the deposit and raw ash increases moving down dip and to the north and south of the current mining areas.

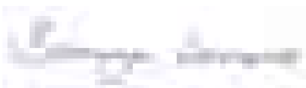
7. Competent Person Statement

I, Bronwyn Leonard, confirm that I am the Competent Person for this report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code 2012 Edition, having at least five years of experience that is relevant to the style of mineralisation and type of deposit described in this Report, and to the activity for which I am accepting responsibility.
- I am a Member of The Australasian Institute of Mining and Metallurgy (AusIMM).
- I have reviewed the Report to which this Consent Statement applies.

I am a full time employee of Stanmore IP Coal Pty Ltd and have been engaged by Stanmore Coal Ltd to prepare the documentation for Isaac Plains East, on which the Report is based, for the period ended 30th June 2020.

I have more than 15 years' experience in exploration and coal resource modelling in both Queensland and New Zealand. I have been involved with the estimation of coal resources since 2010 and have acted as a Competent Person for resource reporting since 2013. I have previously been employed by Solid Energy New Zealand, Xstrata Coal Queensland and Glenore Newlands Opencut. I have been employed by Stanmore IP Coal and based at the Isaac Plains Mine as the Superintendent Mine Geology since October 2017.



Dr Bronwyn Leonard, PhD

MAusIMM 315295

8. References

Coalfield Geology Council of NSW and Queensland Resources Council (2014) Australian Guidelines for the Estimation and Classification of Coal Resources

Xenith (2018) JORC Resource Estimate Update May 2018

Appendix 1: JORC CODE, 2012 Edition Table 1

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	CP Comments
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• Vertical drillholes were used to obtain core samples of the coal seam and associated stone partings. • Cored intervals were sampled where coal was present at thickness of 0.1m or more, with a maximum sample thickness of 0.5 m. Holes used for washability analysis were drilled at 4C or PQ size. Coal plies were sampled discretely on the basis of lithological characteristics and quality. All non-coal material and partings less than 0.1 m were included with the coal ply and noted in the lithological description. • Cored holes were geophysically logged with down-hole wireline gamma/density/calliper tools to confirm sample recovery and ply representation. • Open hole rotary drilling for structure holes and non-cored intervals of quality holes provided chip samples for the description of geological units. Downhole geophysical logs were acquired to supplement the geological description of the drillholes, to assist with correlation of the various seams and to demonstrate continuity of seam character. • Geophysical logging was carried out by external contractors and subject to their internal calibration, quality assurance and quality control procedures.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• All Stanmore coal quality holes were cored (partially or fully) using a conventional 4" core barrel, producing a 101mm core diameter. • Structural holes were drilled as openholes using a polycrystalline diamond hammer or blade bit depending on the lithology. • Lines of Oxidation ("LOX") holes were drilled by a reverse circulation hammer drill rig. • Details of the drill type is not available for all historic (pre-Stanmore) holes
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• Linear core recovery was calculated by dividing the measured length of the core by the drilled length. Geophysical density logs were used to confirm seam thicknesses and adjust seam depths if required. • Laboratory ARD (Apparent Relative Density) were used to calculate the expected mass of each sample based on the recorded length and this was compared to the laboratory weight to ensure that the seam recoveries were satisfactory (> 90%)
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• All Stanmore drill core was geologically logged, marked and photographed prior to sampling. Geological and geotechnical features were identified and logged as part of this process. • All Stanmore open holes had chips collected every metre, which were then geologically logged and photographed. • Geological and geotechnical logging was undertaken in accordance with the CoalLog industry standard. • Details of the logging is not available for historic (pre-Stanmore) holes
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Sampling of core was in accordance with the CoalLog industry standard. • Cored intervals were sampled where coal was present at thickness of 0.1m or more, with a maximum sample thickness of 0.5 m. Holes used for washability analysis were drilled at 4C or PQ size. Coal plies were sampled discretely on the basis of lithological characteristics and quality. All non-coal material and partings less than 0.1 m were included with the coal ply and noted in the lithological description. • All core coal samples were double bagged on site and were transported to a NATA accredited laboratory for testing. • Coal samples were initially tested for Apparent Relative Density (ARD). Samples were then composite to form washability sections.

Criteria	JORC Code explanation	CP Comments

		• To simulate mine transport conditions each composite sample was then drop shattered 20 times from a height of 2 metres, any sample mass remaining of > 50 mm was hand knapped to 50 mm, dry tumbled and dry sized at 31.5 mm, 25 mm, 16 mm, 8 mm, 4 mm and 2 mm.
		• After the dry pre-treatment each composite sample was divided into three parts:
		  • 1/8 for quick coke: Crush to 11.2mm, float sink at 1.425 density, crush to 4mm and mill sample to test for Proximate, CSN, Gieseler & Dilatation
		  • 1/8 for raw analysis: Crush to 4mm, mill sample to test for RD, Proximate, TS and CSN. Selected samples were also test for Calorific Value, Moisture Holding Capacity & Chlorine
		  • ¾ for float sink: Wet tumble and wet size at 31.5, 25, 16, 8, 4, 2, 1, 0.5, 0.25, 01.25 & 0.063mm. Re-combine samples in following fractions: -50+16mm, -16+8mm, -8+2mm and -2+0.25mm. Float sink each size fraction at densities (F1.30, F1.35, F1.375, F1.40, F1.45, F1.50, F1.55, F1.60, F1.70, F1.80, F2.00). -0.25+0mm fraction subject to tree froth flotation. All fractions analysed for ash and CSN.
		• Washability simulations were performed on the float sink results and from that data clean coal composite samples were compiled
		• The historic washability data collected from the Thiess Dampier Mitsui (TDM) drilling in the mid-2000's was from smaller diameter cores that were not pre-treated and were crushed to a reduced top size such as an -11.2mm size fraction. Chris Mcmahon (MCQR) validated and produced large wash simile data from the TDM borecores by employing steps of density standardisation, pre-treatment alignment and size splitting of the crushed coal. This data was then used to produce yield simulations comparable to the Stanmore large washability data.

Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	• All coal quality analysis techniques are per Australian Standards and completed at NATA accredited laboratories. • All coal quality results were checked by cross plots and comparison to original geological logging for accuracy. • David Hornsby of Minserve Group reviewed and assessed the coal quality (and dilution) dataset. • Geophysical logging was carried out by external contractors (Weatherford and Kinetic) and subject to their internal calibration, quality assurance and quality control procedures. • No geophysical logging was conducted on the historic drilling.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Coal quality sample intervals and results were checked and correlated against lithological and geophysical logs. • Raw coal quality data was checked for internal consistency and consistency with the existing data set by checking cumulative totals and cross correlations. • Validation processes by a NATA registered laboratory were conducted for all samples as well as an internal statistical check for anomalies within the laboratory dataset. • Data is stored within Stanmore Geobank database and copies of lab reports are also stored digitally on a separate server
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• Survey of drill collars was conducted using high precision differential GPS • Survey was undertaken by the Isaac Plains mine surveyor or a qualified contract surveyor • The coordinate system used was AGD 84 Z55 which is the system used at the Isaac Plains Mine. • The aerial topographic survey was conducted in September 2015 by Atlass (Aerometrex). The survey accuracy is determined to be +-0.25m.

Criteria	JORC Code explanation	CP Comments
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Borehole spacing has been dictated by the characteristics and consistency of the target seams within the deposit. • Geostatistical and classical statistical analysis of coal ply and working section parameters (thickness and ash) were used to assist in determining the variability of the deposit. • Cored holes are generally spaced between 300m and 600m apart • Structural holes are generally spaced ~100m apart in areas where a pit is planned and up to 800m apart at the limits of the resources. • Structural holes may be very closely spaced (~25m) to define areas of rapid change (e.g. along the Limit of Oxidation, across a fault, along the edge of a basalt channel). • Considering the continuity of the target seam(s) in the deposit, this spacing has proven to be sufficient to give adequate control to the model and give the required confidence in the geological interpretation.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Samples distributed along known coal seam strike and down dip to ensure unbiased sampling. • All drillholes used as points of observation were drilled as vertical holes, which is appropriate given the flat lying and stratiform nature of the coal deposits.
Sample security	• *The measures taken to ensure sample security.*	• All coal quality cored samples were double bagged in plastic bags on site and the dispatched via tracked freight service. Chain of custody and sample information was emailed to the laboratory ahead of the sample
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• No audits or data reviews have been undertaken as part of this resource update • The testing laboratories undertake internal audits and checks in line with the Australian Standards and their NATA certification • The IPE data was fully reviewed as part of the Bankable Feasibility Study (BFS) in 2017 prior to commencement of mining • Prior to this resource update the previous resources estimates were reviewed and any variances between the current model and the model used for the last resource estimate were investigated. • Since mining commenced in 2018 reconciliations have been conducted for both coal quality and coal quantity on each IPE strip and these have shown very good agreement with the geological model

Section 2 - Reporting of Exploration Results

(Criteria	JORC Code explanation	CP Comments
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The IPE resource is covered by four Mining Leases, ML 700016, ML 700017, ML 700018, and ML 700019, each of which was granted to Stanmore IP Coal Pty Ltd on 1st March 2018.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Prior to Stanmore acquiring the IPE tenure, Thiess Dampier Mitsui, Peabody Energy and Blue Energy had all undertaken exploration activities within the project area • Xenith reviewed the historic data prior to Stanmore undertaking their own exploration program
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The IPE deposit occurs in the northern Bowen Basin • The economic coal is contained in the Leichhardt (LHD) Seam of the late Permian Rangal Coal Measures (RCM) • The RCM are unconformably overlain by Tertiary sediments and basalt flows • The LHD has an average thickness of 2.8m and is able to produce a primary semi-soft coking coal +/- a secondary low ash thermal

Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:*
	  • *easting and northing of the drill hole collar*
	  • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar*
	  • *dip and azimuth of the hole*
	  • *down hole length and interception depth*
	  • *hole length.*
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*
		• Detailed drillhole data has not been included as it is deemed commercially sensitive. This information may be supplied if requested.
		• Given that coal is bulk commodity and that there are a large number of drillholes (738) in the deposit individual drillhole details are not considered Material to understanding the resource report

Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Resources have been estimated and reported on a full seam basis. • Where multiple coal quality samples were taken from the seam results have been composited within the modelling software. • Individual samples have been weighted by thickness and density (mass weighting). Laboratory determined relative density (RD ad) has been used for the density weighting.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• Seam thicknesses have been reconciled to geophysics to ensure accuracy. • Coal thicknesses shown are for downhole thickness. Coal resource modelling and estimation adjusts for seam thickness versus the apparent thickness modelled. • Seam thickness was contoured, and any bullseyes were investigated. • The variations in the thickness was largely attributable to faulting and LOX thinning
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• All appropriate diagrams are contained within the main body of the report
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• All available exploration data for the Isaac Plains area has been collated and reported.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• 2D Mini-sosie surveys were undertaken as part of the 2016 exploration campaign to better understand the nature of the faulting and structure at IPE. • Ground Magnetic Survey was carried out in October / November 2017 by Atlas Geophysics across the entire area on east west lines spaced every 50m. The resultant data was reviewed by Geo Discovery Pty Ltd and an interpretation of the surface basalt coverage was produced
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions,*	• No future work has been planned for the IPE area. • Recommendations for future work have been proposed for the southern limit of the deposit but no detailed planning has been

(Criteria	JORC Code explanation	CP Comments
	including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	undertaken.

Section 3 - Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	CP Comments

Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*
	• *Data validation procedures used.*
		• The Isaac Plains geological database (Geobank) contains all hole surveys, drilling details, lithological data, and coal quality results and is the primary source for all such information.
		• Original geological field logs (scanned), down hole geophysics (LAS) files and hard copy logs, hole collar survey files, digital laboratory data and reports and other similar source data are maintained on the Stanmore servers and available for reference at any time
		• A number of validations were undertaken on the database that help ensure consistency and integrity of data including, but not limited to:
		  • relational link between geological, down hole geophysical and coal quality data;
		  • exclusion of overlapping geological intervals;
		  • restriction of data entry to the interval of the defined hole depth;
		  • use only of defined rock type and stratigraphic codes; and
		  • basic coal quality integrity checks such ensuring data is within normal range limits, that proximate analyses add to 100 percent.
		• Lithological logs, geophysical wireline logs, assay results and coal intersection depths were adjusted to geophysics before modelling and resource estimation.
		• Coal quality data checked against NATA laboratory reports where available prior to resource estimation.

Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• The competent person works at the Isaac Plains Complex and frequently visits the active mining areas at IPE. She also oversees any exploration activity undertaken on the IPE mining leases.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The borehole density (core and chip) in the IPE area allows for a good level of confidence in the nature of seam splitting, seam thickness, coal quality, the location of sub-crops and general location of faults. • Interpretation of Basalt affected areas is from the drilling and ground magnetic Survey. Interpretation is predominately reliant on the results of the drilling program.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The Leichhardt target seam(s) extends approximately 7 km along strike and approximately 1.2km perpendicular to strike with an approximate average cumulative thickness of 2.8m. • The depth of first coal ranges from between 15 to 20 m in the west at the fresh coal interface, and 195m in the east under the central topographical high. • Variability for the LHD seam is very minimal; the thickness generally increases to the central north and raw ash increase slightly to the south, north and down dip.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding by-products recovery.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using (or not) grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	• The structural model was updated in May 2020 and the coal quality model was updated in Jan 2020. • Modelling was done in Maptek's Vulcan 12.0.4 modelling software using the Integrated Stratigraphic Modelling package to produce grids and triangulations. FixDHD was used to interpolate drillhole data prior to structure modelling. • Seam surfaces and thicknesses were modelled using triangulation and coal quality was modelled using inverse distance squared • Seams were stacked using the LHD roof as the reference surface • Modelled grid size is 5m for the structure model and 20m for the coal quality model • Seam grids were cropped to the Permian base of weathering • Faults are treated as vertical and modelled using throw • Dummy points were used to control the LHD roof to the west beyond the subcrop line and adjacent to some faults where data is sparse.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Coal resource tonnages were estimated using a calculated Preston and Sanders in situ relative density, using air-dried moisture, total moisture and moisture holding capacities from coal samples (where available). • Based on the results from coal quality testing, the in situ moisture has been estimated to be 4.3%. The 4.3% was derived from the analysed Moisture Holding Capacity values.

Criteria	JORC Code explanation	CP Comments
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• A raw ash % (ad) cut-off grade of 50% was used to distinguish between coal and rock material. • No weathered or oxidised coal was included in the Coal Resource estimate.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• It is assumed that the mining methods currently used at IPE (a combination of dragline and CDX (cast doze excavate)) will continue down dip as long as it economic to do so. No depth cut off has been applied but resources have been reported by overburden depth and a depth of 100m to the top of the LHD seam is considered a nominal limit for opencut mining. • The LHD seam thickness and depth is deemed suitable for highwall or underground development and therefore resources have been classified below the nominal limit for opencut mining.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• The coal from IPE has been successfully processed through the Isaac Plains CHPP since 2018. • Washability simulations from exploration cores show that the remainder of the IPE deposit is similar in character and is therefore very unlikely to have any processing limitations
Environmenta l factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• Two drainage channels lie across the IPE area one in the north, Smokey Creek and one in the south, Billy's Gully. • Neither channel is a permanent water course but should be considered for future evaluation.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• The in situ density of the coal seams has been estimated using the Preston and Sanders in situ relative density estimation equation. • Inherent moisture values have been derived from the coal quality grids which are based on analysis of the exploration cores. • In situ Moisture ("ISM") was assumed to be 4.3% for the purpose of the resource estimation. The average ISM was calculated from the analysed moisture holding capacity values derived from the cored holes. Formula for calculation was based on the ACARP report C10041 and is: ISM= 0.348 + 1.1431 x MHC. • Air dried RD values have been derived from the coal quality grids which are based on the analysis of exploration cores

Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.*
	• *Whether appropriate account has been taken of all relevant factors (ie. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).*
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.*
		• The classification of resources is based on the spacing and distribution of "points of observation" for coal quality and structure.
		• Coal quality points of observation are defined as cored boreholes with greater than 90% recovery across the seam (or accepted by the Competent Person as being representative of the seam through analysis of the coal quality results, core photography and geophysical signature), and Raw and Washability coal quality data
		• Quantity (structure) points of observation are defined as boreholes with downhole geophysical gamma and density logs through the coal seam
		• Statistical analysis was conducted to determine optimal ranges for each resource category, consisting of general statistics and variography based on seam thickness and raw ash (ad%).
		• **Measured Resources:**
		  • 500m spacing of coal quality points of observation
		  • Extrapolated up dip or towards the current pit exposure
		  • No extrapolation down dip
		• **Indicated Resources:**
		  • 1000m spacing of coal quality points of observation
		  • Extrapolation out a structure point of observation if no more than 333m (1/3 of the observation spacing) away from the coal quality point of observation
		• **Inferred Resources:**
		  • 5000m spacing of structure points of observation
		  • Extrapolation 600m to supporting data points (historic drillholes with no geophysical logs) in the south of the deposit

Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• John Bamberry of Palaris Australia audited the Xenith modelling procedures and dataset in May 2017. • No audits or reviews were conducted for the current resource

Criteria	JORC Code explanation	CP Comments
		estimate
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The coal seam resource polygons are limited by the modelled coal seam sub crops and by the drillhole distribution. This ensures no weathered coal can be counted within the estimate. • The thickness grids of each of the seams are based on actual drill intersections. These intersections are checked and adjusted against geophysics in both cored and chip holes. • A geostatistical review of the coal seam thickness data for the Isaac Plains East Project area has been conducted. • Overlying basalt altered areas have been recognised at site and interpreted for the resource estimate. • The geological model in-situ coal estimate has been reconciled against production on a strip by strip basis and these have shown very good agreement with the geological model. The main variance was in the initial boxcuts where production included weathered coal, which had been excluded from the resource estimate.

OPTIMAL MINING

JORC Reserves Statement for

ISAAC PLAINS COMPLEX

as at 31st December 2020

Stanmore Coal Limited

February 2021

Document Information

PROJECT:	**ISAAC PLAINS COMPLEX**
DOCUMENT NUMBER:	**IPC JORC DEC_2020**
TITLE:	**2020 COAL RESERVES ESTIMATE REPORT IPC**
CLIENT:	**STANMORE COAL LIMITED**
DATE:	**25TH FEBRUARY 2021**

Contributors

	NAME	**POSITION**	**SIGNATURE**
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PURPOSE OF RESERVE STATEMENT

Optimal Mining Solutions Pty Ltd (**OMS**) have prepared a report on the Coal Reserves of the Isaac Plains Complex (**IPC**) for Stanmore Coal Limited (**Stanmore**). The Coal Reserves are estimated as at **31st December 2020**.

The purpose of the report is to provide for the company an objective estimate of the Coal Reserves that is compliant with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 edition (**The JORC Code**).

Limitations and Assumptions

Optimal Mining Solutions, after due enquiry and subject to the limitations of the Report hereunder, confirms that:

The conclusions presented in this report are professional opinions based solely upon Optimal Mining Solutions' interpretations of the documentation received, interviews and conversations with personnel knowledgeable about the site and other available information, as referenced in this report. Due to COVID-19 restrictions, the competent person has not visited the site in the past 12 months but has visited the site numerous times in the preceding 4 years. These conclusions are intended exclusively for the purposes stated herein.

For these reasons, prospective estimators must make their own assumptions and their own assessments of the subject matter of this report.

Opinions presented in this report apply to the site's conditions and features as they existed at the time of Optimal Mining's investigations, and those reasonably foreseeable. These opinions do not necessarily apply to conditions and features that may arise after the date of this report, about which Optimal Mining Solutions have had no prior knowledge nor had the opportunity to evaluate.

Limited Liability

Optimal Mining Solutions will not be liable for any loss or damage suffered by a third party relying on this report regardless of the cause of action, whether breach of contract, tort (including negligence) or otherwise unless and to the extent that that third party has signed a reliance letter in the form required by Optimal Mining Solutions (in its sole discretion). Optimal Mining Solutions' liability in respect of this report (if any) will be specified in that reliance letter.

Responsibility and context of this report

The contents of this report have been created using data and/or information provided by or on behalf of the client. Optimal Mining Solutions accepts no liability for the accuracy or completeness of data and information provided to it by, or obtained by it from, the Client or any third parties, even if that data and information has been incorporated into or relied upon in creating this report. The report has been produced by Optimal Mining Solutions using information that is available as at the date stated on the cover page. This report cannot be relied upon in any way if the information provided to Optimal Mining Solutions changes. Optimal Mining Solutions is under no obligation to update the information contained in the report at any time.

COMPETENT PERSON STATEMENT

The information in this report which relates to Coal Reserves, is based on information compiled by a team of suitably qualified Principal Mining Consultants under the management of Mr. Tony O'Connell, who is a Member of the Australasian Institute of Mining and Metallurgy (AusIMM) and is a Principal Mining Consultant at Optimal Mining Solutions Pty Ltd.

Mr Tony O'Connell has more than 22 years' experience in the estimation of coal and mineral reserves relevant to the style of mineralization and type of deposit under consideration. This experience is more than adequate to qualify him as a Competent Person as defined in The JORC Code.

……………………………………………………..

Tony O'Connell BE (Mining), MAusIMM

25th February 2021

The estimates of Coal Reserves for the Isaac Plain Complex presented in this report have been carried out in accordance with the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (2012 Edition) prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia.

Executive Summary

This document forms the supporting documentation for the Coal Reserve Estimate, prepared according to *The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, December 2012*, as at 31st December 2020 for the Isaac Plains Complex (**IPC**). The IPC consists of:

- Isaac Plains Mine (**IPM**); and
- Isaac Plains East (**IPE**).

Stanmore Coal Ltd (**Stanmore**) has commissioned Optimal Mining Solutions (**OMS**) to prepare a Coal Reserve Estimate in accordance with the 2012 JORC Code for the Isaac Plains Complex, which is covered by Mining Leases 70342, 700016, 700017, 700018 and 700019 (ML70342, ML700016, ML700017, ML700018, ML700019 respectively).

This JORC Coal Reserve Estimate is for open cut reserves and highwall augering only.

The IPC is located in the northern part of the Bowen Basin in Central Queensland and targets the productive Leichhardt (**LHD**) coal seam.

The Isaac Plains Mine originally commenced operation in 2006 and has been in production since January 2016 under Stanmore's ownership. The Isaac Plains Mine has historically produced approximately 2.8 Mt per annum of coal for export, comprising coking, PCI and thermal coal products. Isaac Plains East mining leases were granted on 1st March 2018 with first coal being mined in August 2018.

The mining method is a strip mining technique with waste removed by a combination of cast blasting, dozing and dragline spoiling or truck and excavator removal. Maximising dragline usage is targeted in order to minimise costs.

The productive coal seam, the LHD, exists largely as one composite seam across the IPC, however it does split into an upper (**LHU**) and a lower (**LHL**) ply in the north of IPM.

Two separate geological models have been used for two separate Coal Resource Estimates within the Isaac Plains Complex.

The Resource Estimate reports, geological models and data were provided to Optimal Mining Solutions and form the basis of the Coal Reserve Estimate. The following reports should be read in conjunction with this Coal Reserve Estimate:

- ***Isaac Plains Coal Resource Estimate, June 2020*** (for IPM) completed by Xenith Consulting Pty Ltd, and signed off by Mr Troy Turner, as the Competent Person; and
- ***Isaac Plains East Coal Resource Estimate, June 2020*** (for IPE) completed by Stanmore IP Coal Pty Ltd, and signed off by Dr Bronwyn Leonard, as the Competent Person.

Both Mr Turner and Dr Leonard qualify as the Competent Persons in accordance with the requirements of the 2012 JORC Code.

These Reserves are a sub-set of the underlying Resource Estimates; therefore, the Resources are inclusive of the Reserves. The total open cut ROM Coal Reserves for the IPC are presented in Table 1.

Table 1 – Isaac Plains Complex - Open Cut and Auger Mining ROM Reserves Estimate

Coal Reserve (million ROM tonnes)*		
Budget Plan	Proved	1.53
	Probable	0.11
	Total	1.64
Final Cut	Proved	0.22
	Probable	0.03
	Total	0.26
Pit 5	Proved	0.82
	Probable	0.42
	Total	1.25
Auger Mining	Proved	0.47
	Probable	0.10
	Total	0.58
Total	**Proved**	**3.05**
	Probable	**0.67**
	Total	**3.72**

** Tonnages and qualities in the above table are expressed on a ROM basis, incorporating the effects of mining loss, dilution and aggregation, and on a 7.0% ROM moisture basis.*

The marketable coal consists of two products; coking and thermal coal. Estimates have been made for the product split of the two product types. This has formed the basis of an estimate of Marketable Reserves that are derived from the ROM Reserve Estimates. Therefore, Marketable Coal Reserves are a sub-set of ROM Coal Reserves. Total open cut Marketable Coal Reserves for IPE are presented in Table 2.

Table 2 - Isaac Plains Complex - Open Cut and Auger Mining Marketable Reserves Estimate

Marketable Reserves (Product tonnes)		Stanmore Coking Coal (Mt)	Thermal Coal (Mt)	Total (Mt)
Budget Plan	Proved	1.09	0.06	**1.15**
	Probable	0.07	0.01	**0.08**
	Total	1.16	0.08	**1.24**
Final Cut	Proved	0.15	0.01	**0.17**
	Probable	0.02	0.01	**0.03**
	Total	0.17	0.02	**0.19**
Pit 5	Proved	0.59	0.01	**0.59**
	Probable	0.31	0.00	**0.31**
	Total	0.89	0.01	**0.90**
Auger Mining	Proved	0.37	0.03	**0.40**
	Probable	0.08	0.01	**0.09**
	Total	0.45	0.04	**0.49**
Total	**Proved**	**2.20**	**0.11**	**2.31**
	Probable	**0.47**	**0.03**	**0.50**
	Total	**2.67**	**0.15**	**2.82**

All Marketable Reserves tonnages have been expressed on an as-received product moisture basis, which is 11.0% for semi-soft and 9.0% for thermal. The product coal ash levels vary according to the product type in a range from 8% to 16% on an air-dried basis.

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1 INTRODUCTION

1.1 Process

The process adopted for completing the 2020 Isaac Plains Complex JORC Reserve Estimate is described below:

- Geological models and Coal Resource Estimates have been prepared by Xenith Consulting Pty Ltd and Stanmore IP Coal Pty Ltd for IPM and IPE respectively and published in June 2020.
- For IPM, only auger reserves are being calculated as Stanmore has decided it will not progress with any further open cut mining operations at this stage.
- For IPE, Stanmore has decided it will complete open cut mining of the current active pits at the end of 2021, and transition the dragline to the new Isaac Downs Mine. The IPE pit and strip designs are based on the current budget planning.
- Additional identified economical areas were then also designed for IPE and 3-dimensional solids generated in Deswik software. The mine designs included pit wall batters, berm offsets and stratigraphical subdivisions. The solids were also subdivided into the appropriate pits, strips and blocks.
- The in situ coal solids were interrogated against the latest geological model, including qualities for all coal solids for further processing which included minimum mining thicknesses, coal losses and dilution factors being applied to the coal solids. Moisture adjustments, ROM ash cut off and coal recovery assumptions were also applied to convert the in situ values to ROM values.
- The Coal Resource geological confidence limit polygons have been overlaid on the strip solids, and any Inferred or unclassified tonnes were excluded from the Reserve Estimate.
- Product tonnes, for both semi-soft and thermal, were calculated for all coal solids based on the modelled in-seam yield and ROM moisture values with offset factors applied from recent field reconciliation studies.
- Unit cost values were applied to all mining, processing, railing and shipping processes to calculate the total cost for each solid.
- Forecast sale prices were applied to the product tonnage to calculate the overall revenue generated by each coal solid. The total margin for each mining block and strip was calculated. The margin was then used to determine the economic limits for each pit.
- Designs for recovering additional coal beyond the open cut economic limits using highwall augering mining was completed. 3-dimensional solids were generated and interrogated against the geological model with a similar process to the open cut solids being applied to convert insitu coal values to ROM and product coal values. An economic analysis of the auger reserves was also undertaken to ensure auger mining generated a positive margin.
- The Coal Reserve has been categorised as Proved or Probable based on the Coal Resource confidence, the level of detail in the mine planning and considering all the modifying factors to quantify the risks surrounding the project.

- Checks of all quantities and qualities quoted in this report have been undertaken and all work peer reviewed internally by Optimal Mining Solutions.

1.2 Location

Isaac Plains Mine (**IPM**) is located in Central Queensland in the Northern Bowen Basin approximately 7 km directly east of Moranbah.

Isaac Plains East (**IPE**) is adjacent to and east of Isaac Plains Mine, approximately 10 km to the east of Moranbah.

Together the two areas are referred to Isaac Plains Complex (**IPC**) and their location is shown in Figure 1-1.

1.3 Tenure

The coal deposit at Isaac Plains commenced open cut operation in late 2006, ceasing production in January 2015 before recommencing operations with Stanmore as owner in mid-2016.

Isaac Plains Complex consists of Mining Leases ML70342, ML700016, ML700017, ML700018 and ML700019, held by Stanmore IP Coal Pty Ltd (Figure 1-2).

Table 1-1 below provides a summary of each mining tenement.

Table 1-1 - Summary of Isaac Plains Mining Tenure

Tenure	Tenement Holder	Grant / Lodge Date	Expiry Date	Area (Ha)
ML70342	Stanmore IP Coal Pty Ltd	1-Dec-2005	31-Dec-2025	2,143
ML700016	Stanmore IP Coal Pty Ltd	1-Mar-2018	31-Mar-2030	138
ML700017	Stanmore IP Coal Pty Ltd	1-Mar-2018	31-Mar-2030	387
ML700018	Stanmore IP Coal Pty Ltd	1-Mar-2018	31-Mar-2030	369
ML700019	Stanmore IP Coal Pty Ltd	1-Mar-2018	31-Mar-2030	353

There are several overlapping Petroleum Leases and Petroleum Authority to Prospect (**ATP**) that impact on the IPC area. They include the following:

- Petroleum Lease (**PL**) 191 which is currently held by CH4 Pty Ltd and overlaps the western half of the IPE area and into the neighbouring IPM.
- PL223 which is held by CH4 Pty Ltd and overlaps ML700016.
- PL196 which is held by CH4 Pty Ltd and overlaps a small section at the southern extent of ML70342.
- Application for PL1034 which is held by Eureka Petroleum Pty Ltd and overlaps the eastern half and northern portion of the IPE area.

Table 1-2 below provides a summary of each overlapping tenement.

Table 1-2 - Summary of Overlapping Tenure

Tenure	Tenement Holder	Grant / Lodge Date	Expiry Date	Area (Ha)
PL191	CH4 Pty Ltd	21-Mar-2002	20-Mar-2032	20,700
PL223	CH4 Pty Ltd	16-Dec-2004	15-Dec-2024	15,600
PL196	CH4 Pty Ltd	16-Dec-2004	15-Dec-2024	3,600
PLA1034	Eureka Petroleum Pty Ltd	9-Jun-2017	Application	7,628

Co-ordination Agreements are in place with CH4 Pty Ltd and Stanmore IP Coal Pty Ltd, as well as a Joint Interaction Management Plan that deals with the gas and mining operations in relation to the overlapping gas and mining tenures.

Figure 1-1 - Regional Location Map

Figure 1-2 - Regional Tenement Location Map

1.4 Infrastructure

Isaac Plains Complex is accessed by existing road and rail infrastructure. A haul road has been constructed to allow coal mined from IPE to use the current infrastructure at IPM for coal handling, washing and train loadout.

The Queensland Rail (**QR**) Isaac Plains railway branch is part of the Goonyella Coal railway line that connects areas of Central Queensland to the Dalrymple Bay Coal Terminal (**DBCT**) used to export coal from Isaac Plains.

A gravel road to the south of IPM connects the IPC to the Peak Downs Highway (~6 km) which connects the mine to Moranbah (~25 km) to the west and Mackay (~170 km) to the east.

The main infrastructure at the IPC includes the main rail line, rail loop, Coal Handling and Preparation Plant (**CHPP**), ROM dump station, Mining (and maintenance) Industrial Area (**MIA**) and office buildings.

1.5 Environmental Approvals

On 24th January 2018, Stanmore received approval to operate both IPM and IPE under an amended version of the original Isaac Plain's Mine Environmental Authority EPML009327813.

The environmental authority allows ROM coal from IPE to be transported to and processed at the existing Isaac Plains Coal Handling and Preparation facility located on ML70342.

The environmental authority was again amended and approved on 26th February 2020 to expand the mining limits at IPE. Commonwealth Government approvals (EPBC2019/8548) for this expansion was also granted on 4th December 2020 under the EPBC Act.

1.6 Topography and Land Use

The topography of the IPC area and surrounds is gently sloping to the west and north. Two shallow creeks cut across the deposit, draining to the west before discharging into the Isaac River. Smoky Creek in the north and Billy's Gully cuts across in the south.

IPC was originally a grazing property and is located in relatively flat lying terrain. The area has been extensively cleared for cattle grazing, with minor remnant vegetation areas of poplar box woodlands.

A typical elevation for Isaac Plains Complex is approximately 235 m above sea level.

2 Geology

2.1 Regional Geology

IPC is located in the northern part of the Permian-Triassic Bowen Basin, and contains principally fluvial and some marine sediments. The Bowen Basin is part of a connected group of Permian-Triassic basins in eastern Australia which include the Sydney and Gunnedah Basins.

The Bowen Basin axis orientation is north-northwest (**NNW**) - south-southeast (**SSE**) and roughly parallel to the Palaeozoic continental margin. Tectonically, the basin can be divided into NNW-SSE trending platforms or shelves separated by sedimentary troughs. The units from west to east are the Springsure Shelf, Denison Trough, Collinsville Shelf/Comet Platform, Taroom Trough, Connors and Auburn Arches (interrupted by the Gogango Over-folded Zone) and the Marlborough Trough.

Development of the basin in the Early Permian was in the form of a half graben which subsequently became areas of regional crustal sag. Variations in depositional patterns and deformation styles occur along strike suggesting the possibility of northeast (**NE**) trending deep seated crustal transfer faults, evidence for such occurs in the current IPM pits.

2.2 Local Geology

There are no significant Tertiary or Quaternary sediments in the mine area. Soil and sub-soil derived from the Permian sediments are 2-5m thick. A small amount of Quaternary alluvium exists in Smoky Creek area in the centre of the IPC.

The main seam targeted within IPC is the Leichhardt Seam (**LHD**) of the Rangal Coal Measures. It exists largely as one composite seam across IPC but does split into the Leichhardt Upper (**LHU**) and Leichhardt Lower (**LHL**) seams in the north of IPM.

The Burton Range Thrust Fault, with a throw of approximately 180m, is a large scale regional thrust fault that forms the geological boundary between the IPM and IPE resources. The thrust fault has resulted in the coal being displaced east over west, resulting in the sub-cropping of the LHD seam within the IPE area.

The Vermont and Girrah seams which occur up to 40m below the LHD seam are currently not mined as part of the open-cut operation. Exploration drilling has targeted these seams in some areas, but the results have shown them to be uneconomic. The Vermont seam has limited structural and coal quality information at present, and in some areas is considered too thin or of poor quality. For these reasons, a coal resource has not been estimated for the Vermont Seam within IPE.

2.3 Stratigraphy

2.3.1 Leichhardt Seam

In IPM, the LHD Seam is typically 3.5m thick and splits in the north to form the LHU and LHL Seams. The LHU Seam is typically 2.3m thick, while the LHL Seam is typically less than 1.4m thick.

In IPE, the LHD Seam is typically 2.8m thick and dips to the east between 4 and 10 degrees. The seam appears to thicken marginally to the central and northern areas, while dip also steepens towards the central area and to the north. There are several smaller coal occurrences beneath the LHD Seam, which develop at approximately 8m to 30m below and range between 0.3 and 1.2m thick. The minor coal seams are called the Lower Leichhardt Seam (**L2**) and Lower - Lower Leichhardt Seam (**L3**). The L2 should not be confused with the Leichhardt Lower (LHL) ply that occurs in the northwest, within the Isaac Plains Mine area. The L2 seam typically contains a stone band ranging between 0.1 to 0.3m thick.

The Vermont Seam occurs between 30 and 60m below the target LHD seam and is generally distinguished by the increase in tuffaceous material. The Vermont Seam typically splits into several plies and is considered of poorer quality from studies conducted at IPM. The boundary between the Rangal Coal Measures and the underlying Fort Cooper Coal Measures is a tuffaceous claystone band immediately below or within the Vermont seam (locally named the Yarrabee Tuff)

Faulting has been observed to thicken and/or thin all seams, to varying degrees.

Significant splitting (greater than 0.3m) of the main LHD seam is not observed within the IPE area.

The Girrah seam of the Fort Cooper Coal Measures can form a 20m thick stony coal seam and has been noted in some drilling in the west of IPE where the main thrust fault has brought the seam to shallow depth. Regionally the Girrah seam is typically high ash with plentiful tuffaceous bands and due to the high inherent ash, the seam does not wash well.

A stratigraphic column of the representative coal seam for IPC is displayed in Figure 2-1.

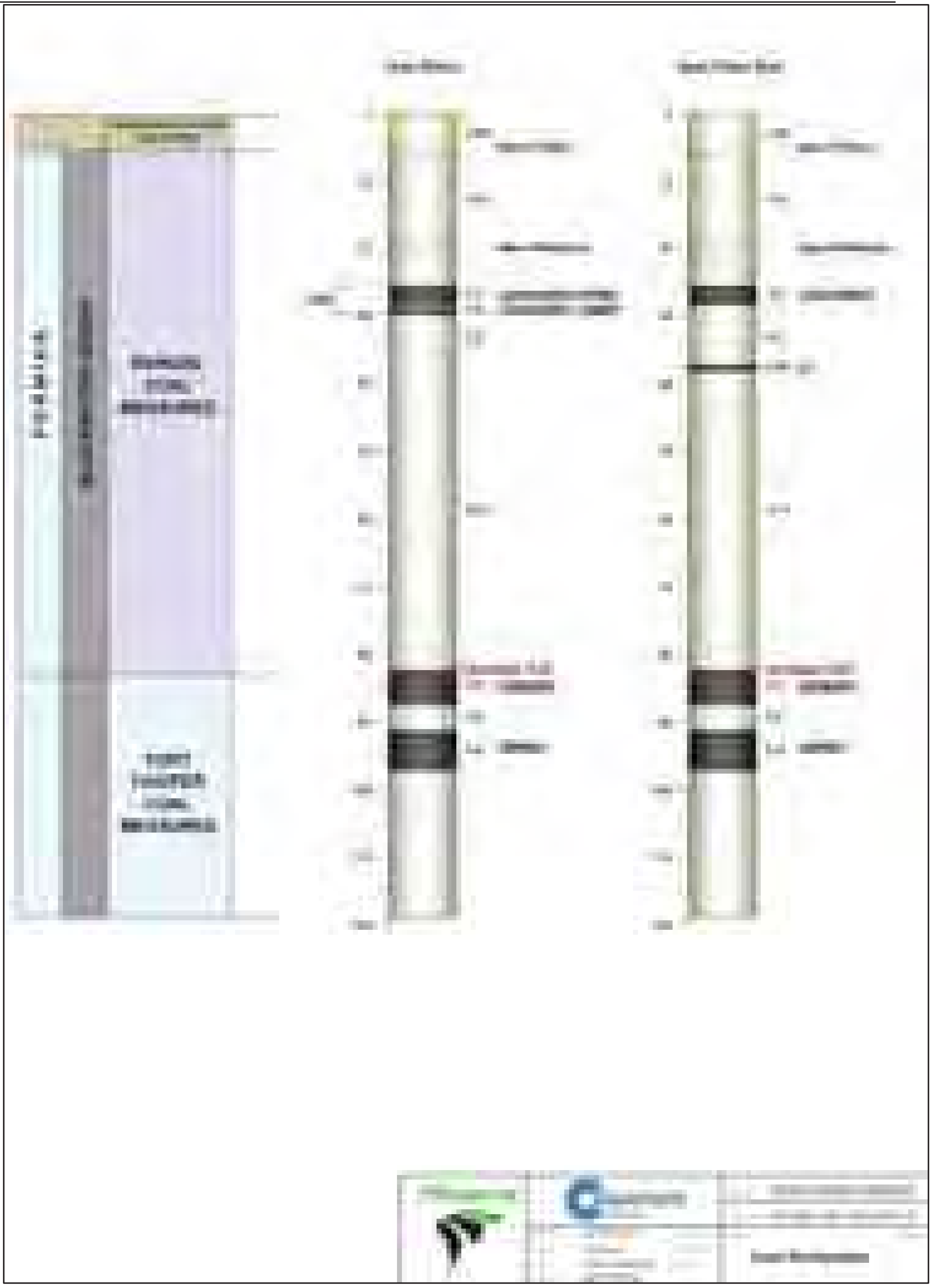

Figure 2-1 - Stratigraphic Column – Isaac Plains Complex

2.4 Historical Exploration Programs

A summary of exploration activities within the IPC area is provided below.

The earliest recorded exploration in the area was carried out by the Utah Development Company Pty Ltd in the 1960's. Although coal was intersected at shallow depth, it was not investigated further. Some six (6) holes were located in the Isaac Plains mine area, but "missed" the Leichhardt seam due to maximum drill depth of 60 metres and spacing between holes.

Queensland Mines Department in the 1970's drilled some regional exploration holes to the near south of the present Isaac Plains mine area.

Thiess Peabody Mitsui Pty Ltd and Thiess Dampier Mitsui Pty Ltd conducted drill traverses in the area from the mid-1960's into the early 1980's, primarily under EPCs 6 and 292. Several drilling campaigns were undertaken, resulting in 227 holes relevant to IPE that were included in the database. Hard copies of the drilling details for these 227 holes were not reviewed for this report but were utilised to compile the 2002 Resource Statement. No downhole geophysics was conducted on this historic drilling.

In the mid 1990's Iscor Australia Pty Ltd, as the holder of EPC602, drilled six holes which targeted the deeper Moranbah Measures. The potential of the Rangal and Fort Cooper Measures was not investigated although coal was intersected at very shallow depths in one of these holes. These holes were south of the Isaac Plains mine area.

In the early 1990's the area coincident with Isaac Plains East became subject of Mineral Development Licences MDL135 (**Morambah**) and MDL 137 (**Wotonga**), held and managed by BHP Coal Pty Ltd.

MGC Resources Australia Pty Ltd conducted 2D dynamite seismic surveys from 1992 to 1995. Three lines, MGS93-BA, MGC92-5 and MGC94-2 are in the south of the present IPM and IPE lease areas.

Historic drilling data for Morambah and Wotonga was reviewed in 2002 and culminated in the Geological model and resource statement undertaken by JB Mining on behalf of then owners, BHP Mitsui Coal Pty Ltd.

MDLs 135 and 137 remained in their current state when purchased by Millennium Coal Pty Ltd in 2003.

In early 2002 and 2003, Nebo Coal Pty Ltd drilled some 16 holes in EPC667, in the north of the present ML70342 area. These intersected the LHD seam in the south (present N1 pit) and the LHU/LHL seam split in the north of the area (present N2 pit).

Bowen Central Coal Management (**BCCM**) drilled some 559 holes for a total 35,754 metres in order to prove an initial 48.8Mt resource within the area of ML 70342. This work started in April 2004 and was completed by early April 2006 just prior to the commencement of mining in July 2006. The drilling was inclusive of the following:

- Coal quality work on some 89 X 100mm cores, 7 X 63mm cores and 5 sites for 200mm cores (17 X 200mm cores holes).
- Line of oxidation (**LOX**) drilling was completed in 149 holes on drill line spaced approximately every 60 metres (north – south).
- Geotechnical work from 7 HQ fully cored holes.

BCCM also undertook 18 km of 2D seismic survey in two phases, 2003 and 2004. Ground magnetics were also conducted over some 8km^2 to determine the influence of the likely intrusives in the area.

From 2008 to September 2009, BCCM on behalf of IPCM drilled a further 19,206m in 278 holes for gas analysis, fault delineation and in-pit coal quality reasons. The majority of the 2008 / 2009 drilling was confined to the working opencut areas of ML70342, which have since been mined.

Blue Energy Limited drilled several CSG wells within and around the area under ATP 814P in 2011. One hole, Sapphire_4 was drilled within the east of the IPE area. Data supplied for this hole was sufficient enough to be incorporated into the resource model.

Stanmore IP Coal Pty Ltd then purchased the IPE leases (MDL135 and MDL137 north) in July 2015 from Peabody Australia Pty Ltd.

Stanmore IP Coal Pty Ltd took over ML70342, EPC 755 and a small portion of EPC 667 in December 2015 following purchase from joint owners Vale Australia Pty Ltd and Sumitomo Corporation.

2.5 Stanmore Coal Exploration Programs

Stanmore has continued to explore the IPC area to build confidence in the coal geometry and coal quality of the deposit, and to safeguard future development.

Since 2015, Stanmore has undertaken several phases of drilling within IPC the drill hole locations are shown in Figure 2-2 and Figure 2-3.

Drilling activity within IPC has chiefly been undertaken to inform and assess the following:

- Resource definition;
- Structural interpretation and clarification;
- Geotechnical assessment;
- Gas content and permeability;
- LOX line and pit boundary definition; and
- Coal quality assessment.

In addition to drilling, Stanmore has undertaken the following supplementary exploration and/or data gathering programs:

- 2016: 32.5km of 2D seismic was captured, with the resulting interpretation being utilised to inform and augment structural interpretations for both IPM and IPE.
- 2017: A 3D seismic survey across an area of 6km2 was undertaken to provide clear structural definition in the area to be assessed for future potential underground extraction.
- October / November 2017: A ground magnetic data survey was conducted over the entire IPE area at 50m line spacing running east west. This data was used to inform the IPE model and to define the limits of the Resource Estimate where basalt was interpreted to intersect the coal seam.

- May 2018: A Deep Ground Penetrating Radar (DGPR) survey was conducted across the Pit 3 and Pit 4 area prior to mining commencing. The survey response from Pit 3 was poor, due to high ground water salinity, but the results from Pit 4 were used to assist in fault interpretation and provided targets for the 2018 drilling campaign.

Figure 2-2 - IPM Drilling and Seismic Data Locations

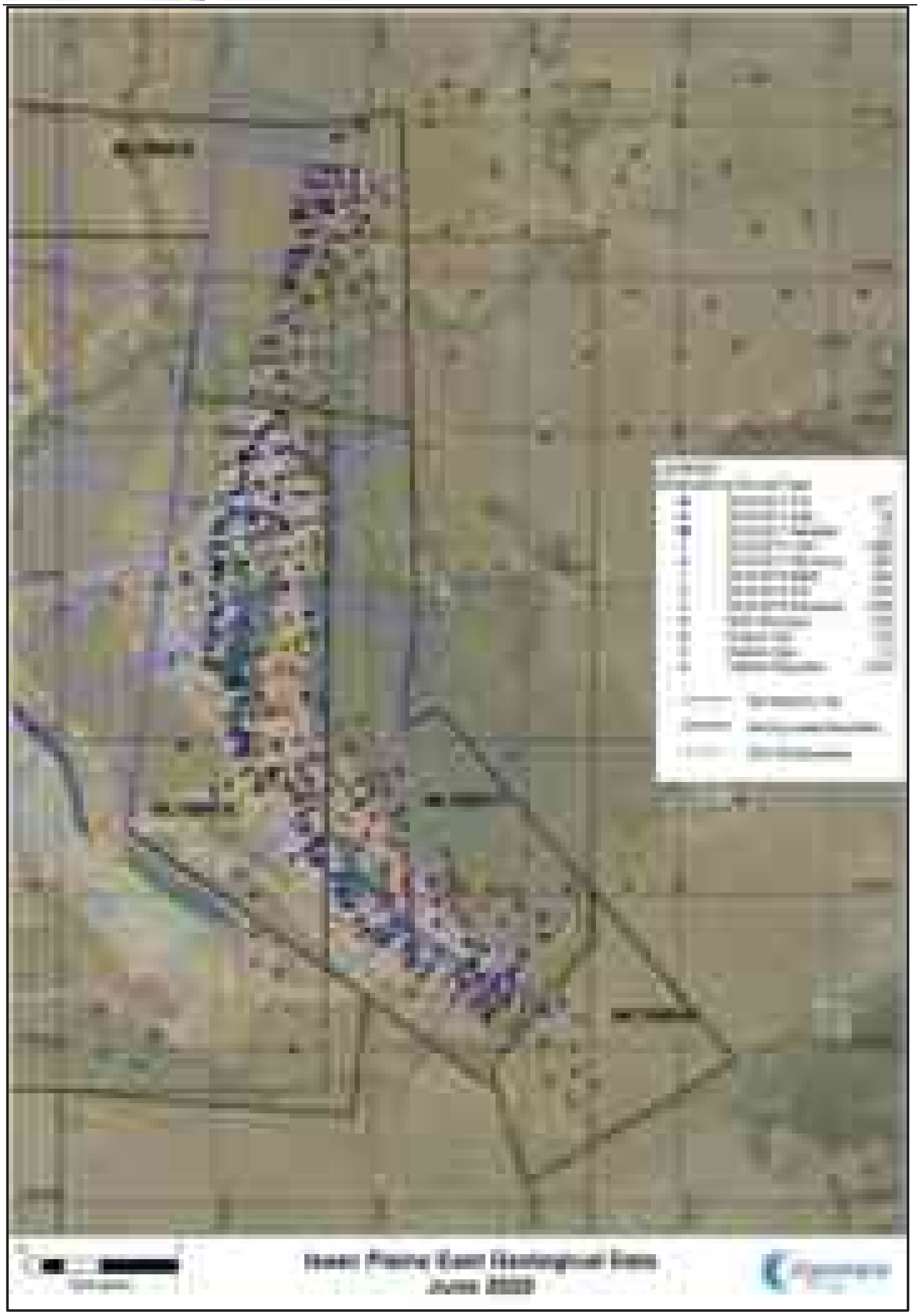

Figure 2-3 - IPE Drilling and Seismic Data Locations

2.6 Data Supporting Coal Resource and Reserve Estimates

Structural models were derived from historic and recent drilling information as well as recent seismic data.

The historic drilling data and historic seam structural interpretation was tested in part by the Stanmore Exploration drilling programmes. Predictions of coal intersections were derived from the historic structural interpretation and the drilling of control holes near historic drilling was undertaken in IPE. The results of these control drilling holes were considered reasonable and gave confidence to the relative location of the historic drilling data points.

Structural and fault interpretations were derived from historic information with adjustments based on the most recent drilling programmes and interpretation derived from the 2D seismic information completed in 2016. A considerable effort has been made to remodel the structural interpretation within the IPC. The original models were created in versions of software that were unable to adequately reflect the nature of the underlying geology, particularly relating to fault geometry and seam dip in down dip areas of both IPM and IPE where limited information exists.

Coal quality information has been derived from both historical sources and from the most recent Stanmore exploration programmes. The IPM historic data was assumed to be reasonable and fit for use, as it was utilised for historic mining processes and had downhole geophysics to support seam thickness and location. The historic coal quality data within IPE, however, was not well supported with down hole geophysics or associated data, and as a result, was not used prior to the 2016/2017 exploration programme.

The 2016/2017 drilling programme undertaken within the IPE area was designed in part to test the seam thickness predictions and locations of the LHD seam within those historic cored holes, with relevant coal quality data. This method was deemed acceptable by the CP, providing there was minimal variance in predicted thickness and location. A total of 11 historic core holes were added to the coal quality model for the IPE.

3 Coal Quality

3.1 Coal Sampling Procedures

General details sampling procedures undertaken for all Stanmore drilling programs are provided in the Isaac Plains and Isaac Plains East Coal Resource Estimate Reports of June 2020.

3.2 Raw Coal

The LHD Seam at IPM and IPE can be classified as a medium volatile, bituminous coal. The seam is considered low in raw ash, exhibiting good washability characteristics.

For the entire resource area within IPM, the tonnage weighted modelled raw ash values for the LHD plies averages 17.9% raw ash (ad).

Within IPE, the LHD seam raw coal quality is quite consistent across the area, with only marginal increases in raw ash % (ad) noted towards the south and north of the deposit. The thickness weighted modelled raw ash value average is lower than within IPM, being 14.6% (ad).

The combined raw ash across the entire IPC area is 15.1% (ad).

3.3 Clean Coal

3.3.1 Coal Testing Procedure

All coal samples from core drilling undertaken by Stanmore within IPC have been analysed according to a thorough coal analysis and testing procedure, as prepared by McMahon Coal Quality Resources (**MCQR**). The complete procedure is explained in the 2020 Resource Estimate reports, for IPM and IPE.

Following pre-treatment and sizing, coal samples were float-sunk. Typically, each core had two (2) composite washability sections, a “Top” section (generally 1.8 to 2.0m) and “Bottom” section, where the balance of the seam was designated.

A portion of each composite section was also taken for raw and “quick coke” analysis. The quick coke analysis involves a small subsample of raw coal which is crushed to 11.2mm top size and floated at density 1.425, prior to milling and analysis for indicative coking properties such as Gieseler Fluidity, Crucible Swell Number (**CSN**) and Dilatation.

Following receipt of full laboratory washability, the resultant size-by-size float and sink data was entered into MCQR’s proprietary CHPP simulation software before advising the laboratory on the makeup of the clean coal (product) composites for testing. All product composites targeted a 9.5% primary product with a secondary 16% product.

Generally, clean coal composites were made up and tested according to MCQR instructions for four different product types:

- The coarse fraction (nominally -50 +16 mm) from the primary dense medium cyclone (**DMC**) product;
- The primary high quality (**PHQ**) coking product (nominally -16 +0 mm). In this mode the coarse -50 +16 mm fraction is directed to the secondary product;

- The primary high yield (**PMY**) coking or pulverised coal injection (**PCI**) product (nominally -50 + 0mm). In this mode the coarse -50 +16mm fraction is directed to the primary product; and
- The secondary DMC thermal product (nominally -50 +2 mm) generated under the PHY processing mode (that is, the coarse fraction does not go to thermal product).

The test regime on the product composites is comprehensive, including:

- For coking coal: Proximate, Calorific Value (**CV**), Hardgrove Grindability Index (**HGI**), ash analysis, total sulphur, CSN, Gieseler fluidity, Gray King, trace elements, dilatation, petrographics, Roga, G index, ultimate analysis, HGI and occasional Sapozhnikov plastometer test.
- For thermal coal: Proximate, CSN, CV, total sulphur, HGI, ultimate analysis, trace elements, AFT reducing and ash analysis.

The procedure recognises the capability and flexibility of the Isaac Plains CHPP, including its ability to produce a primary coking coal, as well as a range of PCI and secondary thermal coals.

3.4 Product Quality

Historically, the Isaac Plains CHPP has primarily produced Semi-Soft Coking Coal (**SSCC**), with lesser quantities of Semi-Hard Coking Coal, PCI and thermal coal. The ability to produce various products has been enabled by either:

- mining the tops and bottoms separately or combined;
- the ability of the Isaac Plains CHPP to change plant settings to include or re-direct the +16mm fraction from the primary to the secondary product stream;
- the move to Isaac Plains East has improved the coal quality with the associated increase in Rank, and this has resulted in a higher proportion of coking coal (of improved quality) and a lower proportion of secondary (thermal) when operating the PHY mode.

Inclusion of the +16mm fraction in the primary product stream produces the current coking coal (known as the high yield product). Re-direction of the +16mm to the secondary product stream (high quality product) reduces primary yield to improve coking coal quality.

At present, the full seam is washed together in the CHPP to produce a primary coking product (targeting 9.5% ash) and secondary thermal product. This approach is applied for the purpose of this IPC JORC Reserve Estimate, for the estimation of marketable reserves and revenue.

3.4.1 Semi-Soft Coking Coal

The SSCC product (high yield product) as presently produced at IPM, can be described as a mid-volatile, weak coking coal (very similar to Blackwater weak coking coal) labelled SSCC for convenience. The coke oven yield is substantially higher than the Newcastle SSCC coals, due to the product's lower volatile matter % relativity. The product is low in sulphur and displays moderate phosphorus content. Plastic properties are moderate and alkali content is low.

The IPE coal is of a slightly higher rank and therefore lower volatile matter (**VM**) content (Isaac Plains Rv-max ~0.98 compared to IPE 1.05). The % phosphorous is also lower (IPM 0.10% compared to IPE

0.06%), and these improved specifications have made this coal very attractive to the key Japanese customers.

The IPE coarse coal product could also be used for PCI purposes for mills using mid-volatile coal in addition to as the standard ULV PCI coal. The CV is high and mill capacity would be generally enhanced with an HGI of ~65.

For this JORC Reserve Estimate the PHY semi-hard coking product option (targeting 9.5% ash), with a secondary thermal product has been modelled. Other product options would be subject to further review depending on market conditions and pricing.

The Isaac Plains CHPP product specifications for the Stanmore Coking Coal and Semi-Hard Coking Coal are presented in Figure 3-1 and Figure 3-2..

3.4.2 Thermal Coal

A secondary thermal coal is produced across the IPC, which varies in volume between IPM and IPE and has the following indicative saleable properties.

This thermal coal continues to find a ready market. The coal is low in nitrogen content and has a high calorific value (~104 % of NEWC6000 nar index CV). Figure 3-3 presents the thermal coal product specifications.

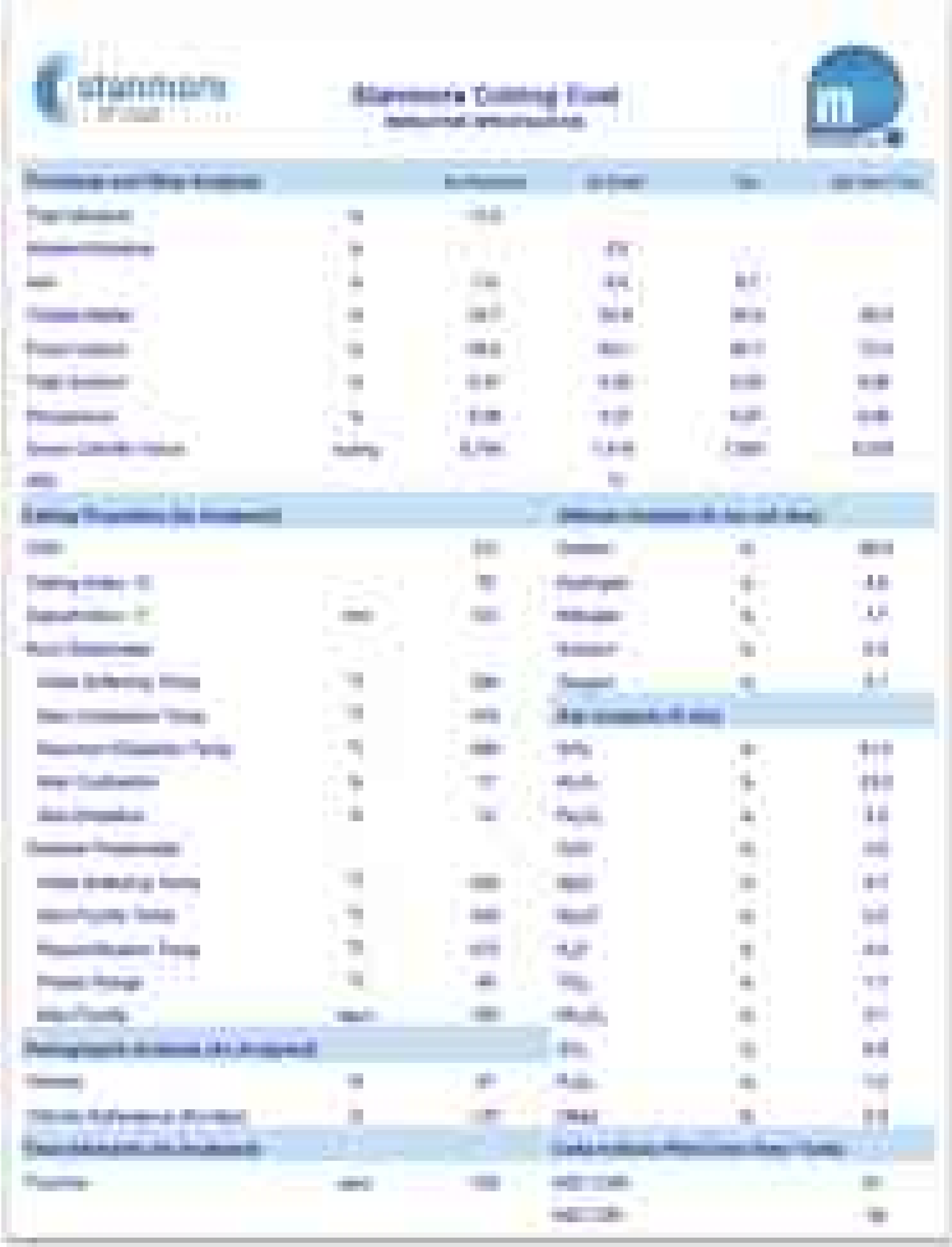

Figure 3-1 – Isaac Plains Complex - Stanmore Coking Coal

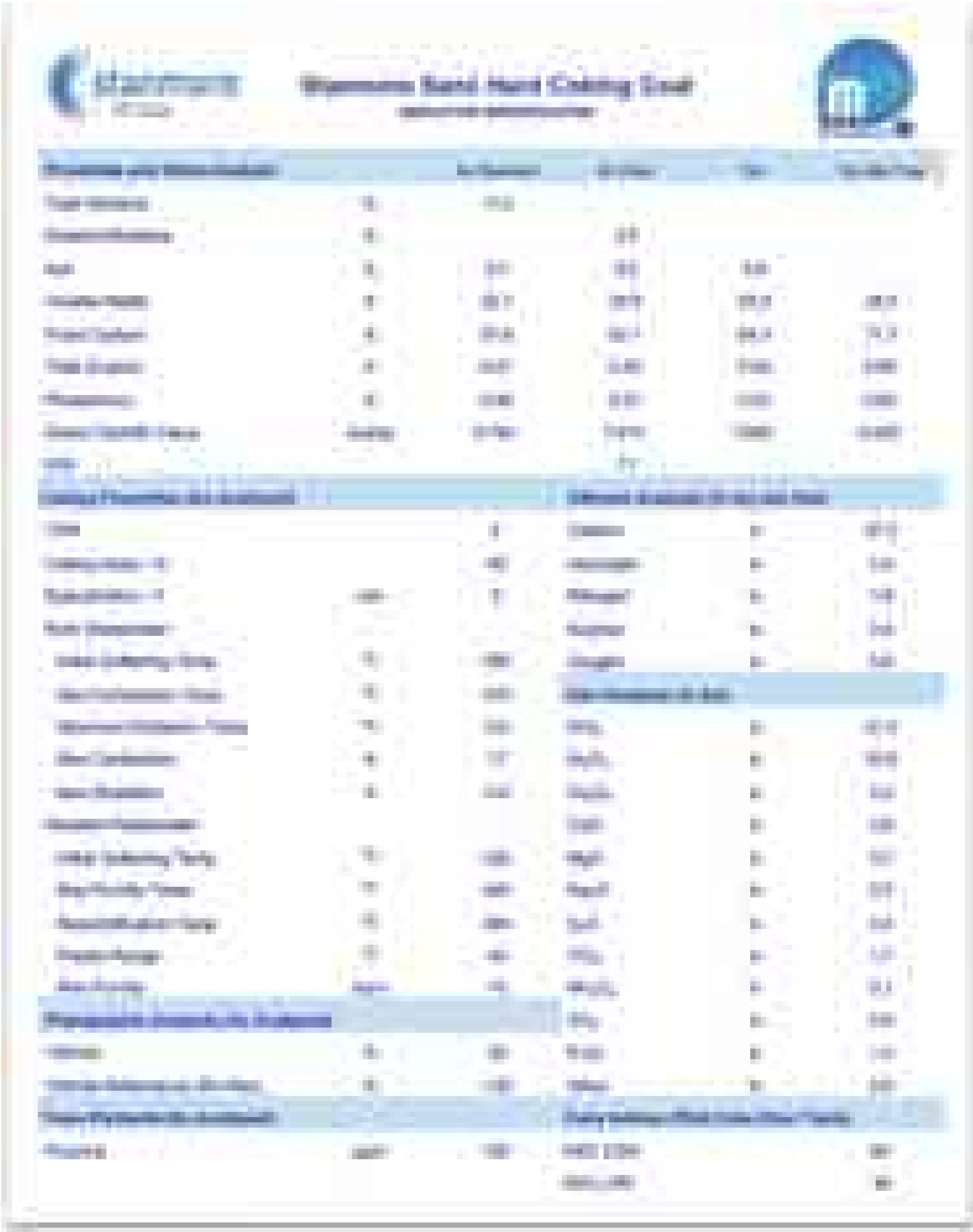

Figure 3-2 – Isaac Plains Complex - Semi-Hard Coking Coal

Figure 3-3 – Isaac Plains Complex - Thermal Coal

3.5 Resource Estimate

The June 2020 Coal Resource Estimates for IPC is presented in Table 3-1.

Table 3-1 - June 2020 JORC Resource Estimates

	IPM Coal Resource* (Mt)	IPE Coal Resource (Mt)
Measured	25.2	**9.8**
Indicated	16.0	**8.0**
Inferred	5.0	**4.0**
Total	**46.2**	**21.8**

* IPM Coal Resource Estimate includes potential underground mining resource.

4 Mine Planning

4.1 Reserve Areas

It has been determined that there will be four key areas that will contribute to the IPC Coal Reserves. Each area's economic mining limits and associated Reserves have been estimated using separate methods due to their different characteristics. The four areas are:

1. IPE Budget Plan (Pits 2, 3 and 4)
2. IPE Final Cut (Pits 2 and 3)
3. IPE Pit 5
4. IPC Auger Mining

Stanmore has decided on a strategic direction to finish open cut mining in IPE Pits 2, 3 and 4 at the end of CY2021 and relocate the dragline to their nearby Isaac Downs Mine pending final statutory approvals. At the completion of the IPE Budget Plan, if the dragline cannot relocate to the Isaac Downs Mine due to a delay in approvals or a change in the mine schedule, a final cut along the highwall of Pits 2 and 3 may be undertaken where economically feasible and/or the dragline could relocate to the unmined Pit 5 in the north.

Auger mining along the exposed highwalls and endwalls of the IPE Pits as well as the N1 Pit at IPM will be employed to access some coal beyond the economic reach of the open cut mining operations.

4.2 Mine Setting

The key features of the IPC that have an influence on the mine plans for all four areas are:

- Topography is relatively flat with minor undulation across the deposit. The topography rises to the east with a 60m high hill located in the central region. The crest of the hill is located approximately 150m to the east of ML700017.
- One main coal seam (Leichhardt) exists across IPC. The weighted average total in situ seam thickness in IPM is 3.08m, whilst it is marginally thinner in IPE at 2.87m. The average coal thickness across IPC is 2.93m. Coal thickness is displayed in Figure 4-1.
- The stripping ratio (waste volume/coal tonnage) gradually increases to the east as the coal dips and the topography rises. The depth to top of coal is shown in Figure 4-2.
- Faulting is present in both IPM and IPC, however faulting is more complex and extensive at IPM.
- Dips are generally quite moderate in the geological model with some steeper dips in the northern area. Dips (excluding the fault zones) vary generally between approximately 4° to 10°.
- Smoky Creek and Billy's Gully form the major water flows across the IPC with water shedding off the hill and through these water courses. It was assumed that no mining would occur through these corridors.
- Powerlines cross IPC in the north. It was assumed that these would not be relocated for this Reserves estimate.
- There are no endangered ecosystems in the mining area as displayed in Figure 4-3. The pit area includes areas in the regional ecosystem map labelled "of least concern". Offsets may be required for some areas of disturbance and an Offset Management Plan has been agreed with regulators.

Figure 4-1 - Total Coal Thickness – LHU and LHD Seams

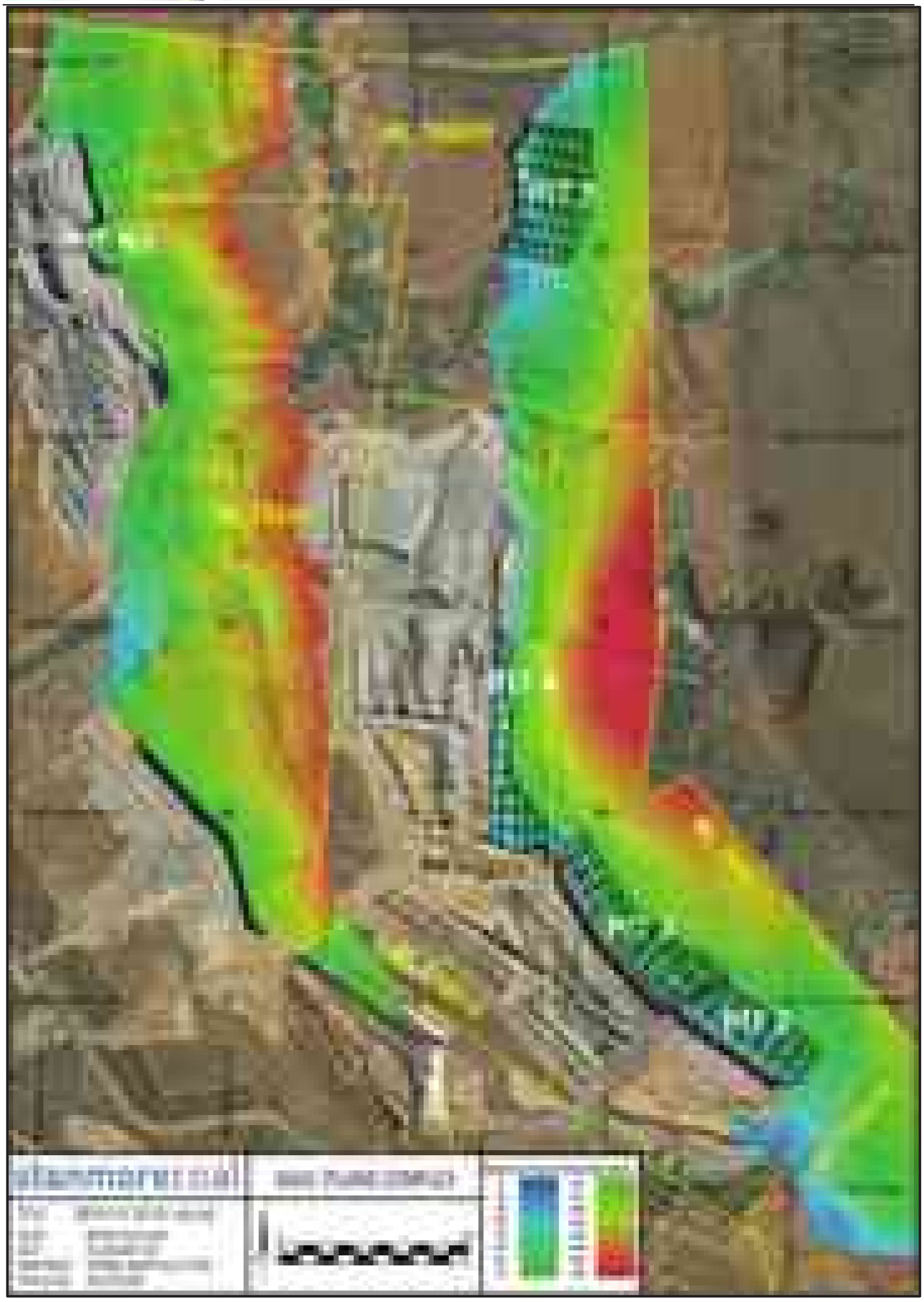

Figure 4-2 - Depth to Top of Coal – LHU and LHD Seams

Figure 4-3 – Regional Ecosystems

4.3 IPE Budget Plan

Optimal Mining has reviewed Stanmore's current LOM plan and budget for IPE Pits 2, 3 and 4 and determined that it is a mine plan and production schedule that is technically achievable and economically viable and from which the Ore Reserves can be derived.

4.3.1 Mining Limits

A number of constraints which have an effect on the mine layout at IPE were:

- All pits are constrained by economic margin down dip. The final highwall position will be determined by the economics and coal sales price at the time of mining.
- The pits are limited to the west by the LHD subcrop or extents of previous mining.
- Pit 2 is constrained by the 1:1000 year flood lines for Billy's Gully in the south.

4.3.2 Mining Method

The current mining method employed at IPE is as follows:

- Topsoil is removed and stockpiled for later spoil rehabilitation.
- The overburden is drilled and blasted in one or two passes depending upon total depth.
- Overburden is then removed using a combination of truck and excavator, cast blasting, dozing and dragline.
- Coal mining is then undertaken.

To minimise waste removal costs, the emphasis will be on maximising the proportion of waste allocated to the dragline system (dragline, dozing and cast blasting). Waste exceeding the dragline horizon will be removed by excavator and trucked to the appropriate waste dump.

Coal will be loaded by excavators into rear dump trucks and hauled to the Isaac Plains ROM stockpile area where it will be crushed and conveyed to the coal preparation plant for processing. Product coal will be stockpiled separately by product type then loaded onto trains at the coal loadout and railed to Dalrymple Bay Coal Terminal.

Progressive rehabilitation of the spoil dumps would be undertaken as soon as practicable to meet approval conditions as required.

4.3.3 Strip Designs

The strips were designed primarily upon the constraints discussed in Section 4.3.1. The delineation between adjoining pits was determined by the location of the faults running perpendicular to the subcrop. The final highwall was ultimately set at the economic margin for the budget unit cost rates, yields and revenue assumptions. Figure 4-4 shows the Budget plan mining limits. The pit design parameters are shown in Table 4-1. Strip widths are nominally 50m, however variations occur around subcrop and faulted areas to ensure resource recovery is safely maximised. The strip orientations and dimensions are suitable for strip mining with a dragline.

Figure 4-4 – Budget Plan Mining Areas

Table 4-1 – Budget Plan Strip Design Parameters

Item	Units	IPE (Pits 2,3,4)
Strip Width	m	50
Highwall Batter Angle in Hard	degs	70
Highwall Batter Angle in Soft	degs	45
Endwall Batter Angle	degs	45
Dragline Horizon Depth	m	40
Offset Bench Width	m	25

4.3.4 Block Solids Generation and Interrogation

The strips designs were converted to three dimensional solids limited by the surveyed prime topography surface as at 28th December 2020 and cut into blocks nominally 100m wide along strike. The block solids were cut by the geological structure model to generate coal and overburden solids.

The solids were then interrogated by the geological model to generate in situ quantities (volume, area, thickness, dip etc) and coal qualities (relative density, raw ash, product yield, inherent moisture etc).

4.3.5 Mining Assumptions and Modifying Factors

Modifying factors were applied to the in situ quantities and qualities to convert from an in situ basis to a ROM / product basis.

4.3.5.1 Loss and Dilution

To convert the composited coal tonnes to ROM coal tonnes, mining floor, roof loss and dilution thicknesses were applied. The loss and dilution parameters based on reconciliations of current operations are shown in Table 4-2.

Table 4-2 – Budget Plan Loss and Dilution Parameters

Item	Units	Unfaulted	Faulted
Coal Roof Loss	m	0.17	0.245
Coal Floor Loss	m	0.17	0.245
Coal Strip Edge Loss	m	0.250	0.250
Coal Roof Dilution	m	0.08	0.155
Coal Floor Dilution	m	0.08	0.155
Coal Strip Edge Dilution	m	0.250	0.250
Other Loss	%	1%	1%
Other Dilution	%	1%	1%
Dilution Ash	%	85	85
Dilution Density	t/bcm	2.39	2.39

4.3.5.2 Moisture Adjustments

Coal densities were adjusted in the database to convert from the geological model air-dried values to an in situ moisture basis, using the Preston Sanders method. ROM and product tonnes were then determined using additional moisture assumptions, as detailed in Table 4-3.

Table 4-3 – Moisture Assumptions

Item	Units	Value
Air-dried Moisture	%	As modelled*
Insitu Moisture	%	5.0%
ROM Moisture	%	7.0%
Semi-soft Product Moisture	%	9%
Thermal Product Moisture	%	11%

** - as modelled air-dried moisture values are indicatively 2.3%*

4.3.5.3 Process Simulations

Independent coal quality consultants, MCQR, produced a laboratory test program for core samples based on "pre-treatment" washability procedures to accurately represent the likely size distribution and coal types that will be encountered in mining and processing via the Isaac Plains CHPP.

The Isaac Plains CHPP provides either a "High Yielding" or "High Quality" primary product processing option.

1. The "High Yielding" option keeps the primary dense medium cyclone (DMC) coarse coal size fraction (nominally -50mm +16mm) within the Primary Product. This processing method generally yields a semi-soft coking coal.
2. The "High Quality" option directs the primary coarse size coarse fraction (+16mm) to the secondary thermal product, allowing the generation of a higher quality coking coal at a lower primary yield.

Prior to laboratory flotation analysis, the LHD seam samples are combined into two (2) wash composites, on a borehole-by-borehole basis, as follows:

1. "Top" seam section (generally 1.8 to 2.0m) *and*
2. "Bottom" seam section, where the balance of the seam is designated.

Wash simulations at IPM and IPE target the following:

A. Primary coking product of 9.5% ash *and*
B. A secondary thermal product of 16% ash

4.3.5.4 Product Yield

Reconciliations of recent CHPP results have indicated that there is a divergence between actual yields and the geological grid model yields, mainly in Pits 2 and 3.

It was found that in general the following adjustments to the primary and secondary grid values are to be applied:

- Pit 2 – Primary Yield = Grid Value – 6.5%
- Pit 2 – Secondary Yield = Grid Value + 8.0%
- Pit 3 – Primary Yield = Grid Value – 0.5%
- Pit 3 – Secondary Yield = Grid Value – 0.3%
- For all other pits, no adjustments are required.

Table 4-4 lists the yield applied for each strip in the Budget plan.

Table 4-4 Yields by Strip

Pit	Strip	Description	Primary	Secondary	Total
Pit2	**10**	All	71.40	13.60	85.00
Pit2	**11**	All	70.49	14.21	84.70
Pit2	**12**	All	69.57	14.75	84.32
Pit2	**13**	All	67.76	15.89	83.65
Pit3	**9**	All	82.49	1.46	83.95
Pit3	**10**	All	82.08	1.68	83.76
Pit3	**11**	All	81.35	2.11	83.46
Pit3	**12**	All	81.35	2.74	84.09
Pit4	**10**	Central	85.34	-	85.34
Pit4	**10**	South	81.36	0.87	82.23
Pit4	**11**	All	79.84	1.26	81.10
Pit4	**12**	All	78.99	1.73	80.72
Pit4	**13**	All	78.41	2.18	80.59
Pit4	**14**	All	77.45	2.85	80.30
Pit4	**15**	All	76.36	3.45	79.81
Pit4	**16**	All	74.89	4.63	79.52

The yields are applied to the undiluted ROM coal then converted from the ROM moisture basis to their respective product moisture basis.

4.3.6 Equipment Waste Allocation

Due to the cost effectiveness of draglines for waste removal, where possible, all waste up to 50m thick was assigned to the dragline system, with any waste above this assigned to truck shovel. The dragline system includes waste cast blast to final, waste pushed by dozers and waste removed by the dragline.

The only exception to the allocation of up to 50m of waste to the dragline was where the seam dip exceeded 8 degrees. In these instances, the dragline waste thickness was reduced to 40m due to dig depth constraints.

The dragline system thickness was also reduced in the scheduling progress if the machine progressed too slowly and minimum coal targets were not achieved.

4.3.7 Mine Schedule

The mining budget opencut schedule was completed to ensure that coal can be delivered to the CHPP until the end of CY2021 and to generate physical values for the financial evaluation. Figure 4-5 charts the Budget scheduled prime waste and ROM coal movement by month.

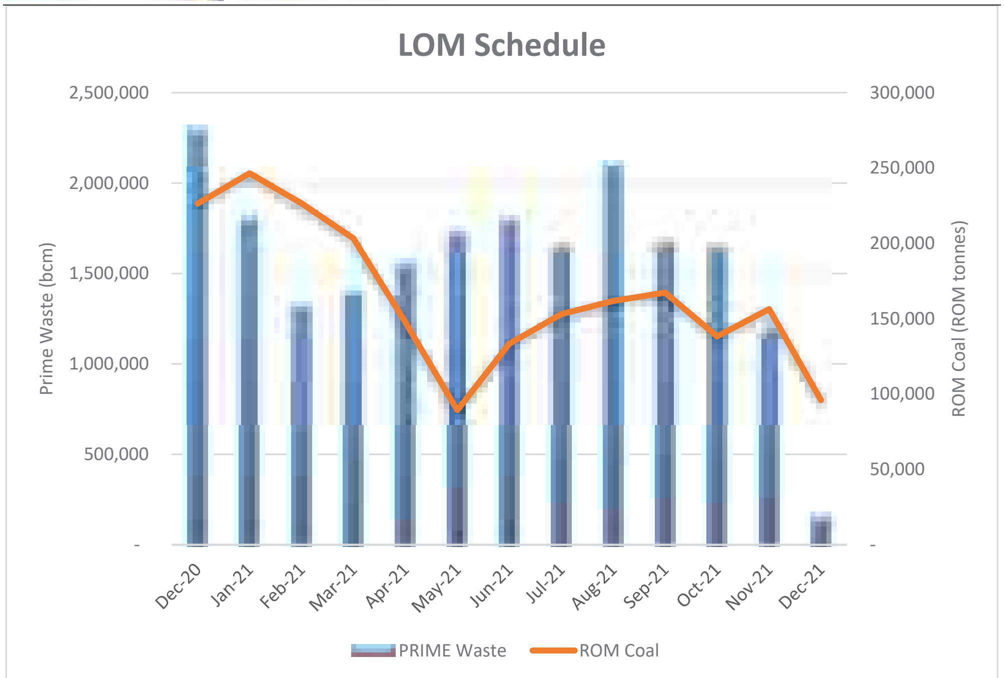

Figure 4-5 – Budget Schedule Waste and Coal Quantities

4.3.8 Financial Evaluation

The Budget plan physicals were imported into a financial model for economic evaluation and to confirm the economic viability of these Coal Reserves utilising up-to-date economic assumptions.

The competent person has reviewed the results of the financial modelling and is satisfied that the JORC Reserves can be economically extracted with the current mining practices, unit cost rates and sale prices.

4.3.8.1 Cost Assumptions

Unit costs have been supplied by Stanmore and are based on the historical and/or budget forecast costs of the operations at IPE. The average calculated unit cost for the reserves in the Budget plan is AU$80.72/ROM t or AU$109.07/Product t.

4.3.8.2 Revenue Assumptions

Revenue assumptions are based on the historical relative price Stanmore receives for IPE's semi hard coking coal and thermal coal products compared to the benchmark Hard Coking Coal (**HCC**) price.

Based on KPMG's 'Coal Price and FX Market Forecast' report dated September/October 2020, Stanmore has taken the following position on the HCC forecast:

	2021	2022	2023	2024	Long Term
Stanmore Benchmark HCC Forecast (USD/tonne FOB)	$142	$143	$145	$147	$149

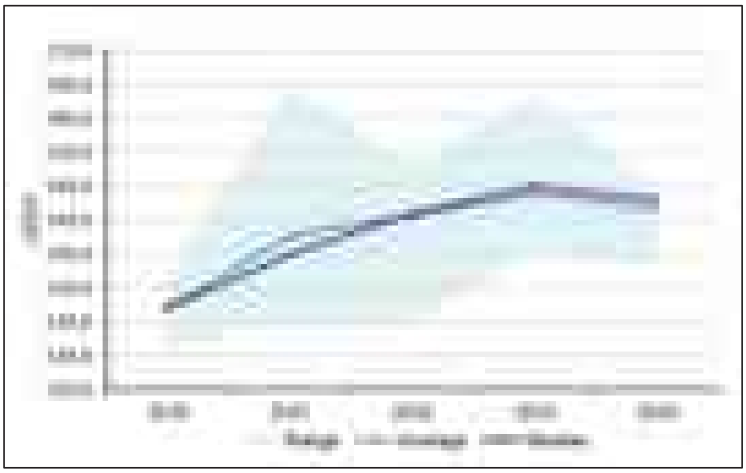

Figure 4-6 – KPMG's Benchmark HCC Forecast Summary

Therefore, for the 2021 budget, the HCC benchmark price used is US$142/tonne which equates to US$98.94/tonne (AU$133.70/tonne) for the IPE coking product and US$67.07/tonne (AU$90.04/tonne) for the thermal product.

The foreign exchange rate used to convert USD to AUD is 0.74.

The average sales price achieved for the Budget plan reserves after product splits are applied is AU$130.90.

4.4 IPE Final Cut (Pits 2 and 3)

A final narrow strip was designed at the back of the IPE Budget Plan limits in Pits 2 and 3 to analyse whether a final cut undertaking a two-pass dragline operation would be economically feasible.

4.4.1 Mining Limits

The mining limits followed the endwall limits of the Budget highwall for Pits 2 and 3.

4.4.2 Mining Method

To generate a final cut that would be economically feasible beyond the Budget plan highwall it is planned that the dragline would mine all the overburden in two passes to substitute the more expensive pre-strip excavator and truck mining with the dragline.

4.4.3 Strip Designs

To allow for the dragline spoil to fit when undertaking a two-pass operation the strip width was reduced to 40m, the dragline offset back bench was also reduced to 15m to reduce the prime volume being taken.

The pit design parameters are shown in Table 4-5 and location of the Final Cut strips in Figure 4-7.

Table 4-5 – Final Cut Strip Design Parameters

Item	Units	IPE Pit 2 & 3 Final Cut
Strip Width	m	40
Highwall Batter Angle in Hard	degs	70
Highwall Batter Angle in Weathered	degs	45
Endwall Batter Angle	degs	45
Dragline Horizon Depth – bottom pass	m	50
Offset Bench Width	m	15

4.4.4 Block Solids Generation and Interrogation

The strips designs were converted to three dimensional solids limited by the surveyed prime topography surface as at 28th December 2020 and cut into blocks nominally 100m wide along strike. The block solids were cut by the geological structure model to generate coal and overburden solids.

The solids were then interrogated by the geological model to generate physical quantities (volume, area, thickness, dip etc) and coal qualities (relative density, raw ash, product yield, inherent moisture etc)

4.4.5 Mining Assumptions and Modifying Factors

Modifying factors were applied to the in situ quantities and qualities to convert from an in situ basis to a ROM / product basis and were similar to the Budget plan.

4.4.5.1 Loss and Dilution

To convert the composited coal tonnes to ROM coal tonnes, mining floor, roof loss and dilution thicknesses were applied. The loss and dilution parameters based on reconciliations of current operations but on the expectations that operations will not be as urgent to turn the strips around so coal uncovery and recovery will be more deliberate are shown in Table 4-6.

Figure 4-7 – Final Cut Strips

Table 4-6 – Final Cut Loss and Dilution Parameters

Item	Units	Un-faulted	Faulted
Coal Roof Loss	m	0.08	0.245
Coal Floor Loss	m	0.08	0.245
Coal Strip Edge Loss	m	0.250	0.250
Coal Roof Dilution	m	0.08	0.155
Coal Floor Dilution	m	0.08	0.155
Coal Strip Edge Dilution	m	0.250	0.250
Other Loss	%	1%	1%
Other Dilution	%	1%	1%
Dilution Ash	%	85	85
Dilution Density	t/bcm	2.39	2.39

4.4.5.2 Moisture Adjustments

The moisture assumptions used are the same as the Budget plan moistures listed in Table 4-3.

4.4.5.3 Product Yield

As explained in Section 4.3.5.4, reconciliations of current operations recommended the following adjustments to the primary and secondary grid values be applied:

- Pit 2 – Primary Yield = Grid Value – 6.5%
- Pit 2 – Secondary Yield = Grid Value + 8.0%
- Pit 3 – Primary Yield = Grid Value – 0.5%
- Pit 3 – Secondary Yield = Grid Value – 0.3%

The yields are applied to the undiluted ROM coal then converted from the ROM moisture basis to their respective product moisture basis.

4.4.6 Financial Evaluation

The quantities and qualities of the Final Cut solids were inputted into a financial model to evaluate the economic margin of the cuts utilising the up-to-date economic assumptions.

4.4.6.1 Cost Assumptions

Unit costs have been supplied by Stanmore and are based on the historical and forecast costs of the operations at IPE. The average calculated unit cost for the reserves in the Final Cut is AU$99.52/ROM t or AU$127.95/Product t.

4.4.6.2 Revenue Assumptions

The revenue assumptions are based on the forecast prices as described in Section 4.3.8.2, using the 2022 HCC price of US$143 which equates to US$99.36/tonne (AU$134.270/tonne) for the IPE coking product and US$67.54/tonne (AU$91.27/tonne) for thermal product.

The average sales price achieved for the budget plan reserves after product splits are applied is AU$129.68.

4.5 Pit 5

As the Pit 5 area is an unmined area and not in the budget plan it has been flagged as potential future mining area for Stanmore, therefore a pit optimisation analysis was undertaken to determine the economical limits.

4.5.1 Pit Shell Optimisation

The Pit 5 area was assessed using Deswik Pseudoflow software which applies an algorithm to the unit costs, revenues and geological structure and quality models to determine a maximum economic pit shell.

The strip designs were then limited to this economic pit shell in addition to the mine layout constraints (see Figure 4-8).

4.5.1.1 Cost Assumptions

Unit costs have been supplied by Stanmore and are based on the historical and forecast costs of the operations at IPE.

4.5.1.2 Revenue Assumptions

The revenue assumptions are based on the forecast prices as described in Section 4.3.8.2, using the long term HCC price of US$149 which equates to US$103.19/tonne (AU$139.45/tonne) for the IPE coking product and US$70.38/tonne (AU$95.11/tonne) for the thermal product.

Figure 4-8 – Pit Limits for Pit 5

4.5.2 Mining Method

The mining method at Pit 5 will be similar to the existing operations employed at IPE as follows:

- Topsoil is removed and stockpiled for later spoil rehabilitation.
- The overburden is drilled and blasted in one or two passes depending upon total depth.
- Overburden is then removed using a combination of truck and excavator, cast blasting, dozing and dragline.
- Coal mining is then undertaken.

An initial boxcut will be required at the subcrop that will be excavated fully by truck and excavator methods.

To minimise waste removal costs, the emphasis will be on maximising the proportion of waste allocated to the dragline system (dragline, dozing and cast blasting). Waste exceeding the dragline horizon will be removed by excavator and trucked to the appropriate waste dump.

Coal will be loaded by excavators into rear dump trucks and hauled to the Isaac Plains ROM stockpile area where it will be crushed and conveyed to the coal preparation plant for processing. Product coal will be stockpiled separately by product type then loaded onto trains at the coal loadout and railed to Dalrymple Bay Coal Terminal.

Progressive rehabilitation of the spoil dumps would be undertaken as soon as practicable to meet approval conditions as required.

4.5.3 Pit Layout and Strip Designs

The following constraints were taken into consideration in the design of the pit layout:

- The pit is limited to the west by the LHD subcrop.
- The final highwall position is limited by the economic limits from the Pit Optimisation Shell.
- The pit is constrained to the north by the rail line, power lines, road reserve as well as environmental disturbance limits.
- The pit is constrained by the 1:1000 year flood lines and associated environmental disturbance limits for Smoky Creek in the south.

The pit design parameters are the same as the Budget plan design parameters shown in Table 4-1. Strip widths are nominally 50m, however variations occur around subcrop to remove the steeper dipping areas prior to the dragline commencing.

The strip orientations and dimensions are suitable for strip mining with a dragline.

4.5.4 Block Solids Generation and Interrogation

The strips designs were converted to three dimensional solids limited by the topography surface and cut into blocks nominally 100m wide along strike. The block solids were cut by the geological structure model to generate coal and overburden solids.

The solids were then interrogated by the geological model to generate in situ quantities (volume, area, thickness, dip etc) and coal qualities (relative density, raw ash, product yield, inherent moisture etc).

4.5.5 Mining Assumptions and Modifying Factors

Modifying factors were applied to the in situ quantities and qualities to convert from an in situ basis to a ROM / product basis and were similar to the plan.

4.5.5.1 Loss and Dilution

To convert the composited coal tonnes to ROM coal tonnes, mining floor, roof loss and dilution thicknesses were applied. Best practice loss and dilution parameters have been applied (Table 4-7) on the expectations that a lower production rate will be targeted compared to current operations so it will not be as urgent to turn the strips around.

Table 4-7 – Pit 5 Loss and Dilution Parameters

Item	Units	Un-faulted	Faulted
Coal Roof Loss	m	0.08	0.245
Coal Floor Loss	m	0.08	0.245
Coal Strip Edge Loss	m	0.250	0.250
Coal Roof Dilution	m	0.08	0.155
Coal Floor Dilution	m	0.08	0.155
Coal Strip Edge Dilution	m	0.250	0.250
Other Loss	%	1%	1%
Other Dilution	%	1%	1%
Dilution Ash	%	85	85
Dilution Density	t/bcm	2.39	2.39

4.5.5.2 Moisture Adjustments

The moisture assumptions used are the same as the Budget plan moistures listed in Table 4-3.

4.5.5.3 Product Yield

The product yields applied are from the geological grid model applied to the undiluted ROM coal then converted from the ROM moisture basis to their respective product moisture basis.

4.5.6 Financial Evaluation

A block margin rank was undertaken to confirm the detailed pit design blocks generated within the optimised pit shell were still generating a positive margin. Any continuous areas of blocks that had negative margins (excluding endwall blocks) were then removed and not included as Coal Reserves.

4.5.6.1 Financial Assumptions

The cost and revenue assumptions used for the block margin rank are the same assumptions used in the Pit Shell Optimisation as mentioned in Section 4.5.1. The average calculated unit cost for the reserves in Pit 5 is AU$90.79/ROM t or AU$125.26/Product t.

The average sales price achieved for the Pit 5 reserves after product splits are applied is AU$139.04.

4.5.6.2 Margin Rank Results

The margin analysis calculated the profitability of each block, with all blocks that provided a positive cashflow included in the Reserve Estimate. Some negative cashflow blocks which were surrounded by economical blocks in the same strip were included in the Reserve Estimate.

The overall cashflow for each strip was calculated to ensure that all strips provided a positive cash flow.

4.6 Auger Mining

At the completion of mining in each pit it is planned to undertake auger mining to recover some coal beyond the economic reach of the open cut mining operations. Where the LHD seam is exposed in the highwalls and endwalls and it is continuous beyond the walls (i.e. no faulting) auger mining will be employed.

The following areas have been identified as having the suitable conditions for auger mining:

- IPM N1 Pit
- IPE Pits 2, 3, 4 and 5

4.6.1 Auger Mining Parameters

Geotechnical evaluations at IPC have recommended specific web widths (pillars) be applied to the design of the auger mining layouts using a 1.9m diameter auger hole depending on the penetrating depth into the coal seam and the overburden thickness. Figure 4-9 provides a schematic of the key elements of the auger design.

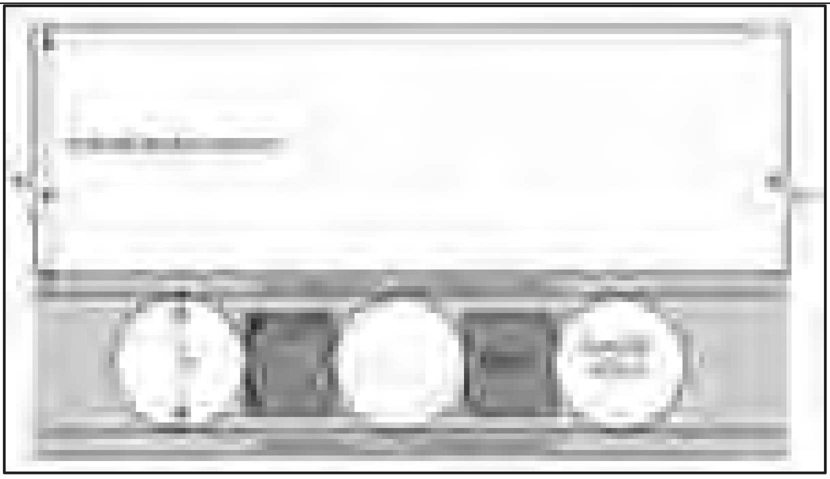

Figure 4-9 – Key Elements of Auger Design

Table 4-8 – Auger Design Parameters by Pit/Block

Pit	Block	Ave OB Thickness (m)	Ave Penetration Depth (m)	Web Width (m)
Pit N1	1	80	129	2.4
	2	95	128	2.8
	3	95	150	2.8
	4	85	142	2.4
Pit 4	A	85	138	2.7
	B1	90	150	2.65
	B2	90	150	2.9
	B3	85	150	2.5
	C	80	148	2.65
	E	80	150	2.65
	1	70	112	2.3
	2	80	132	2.2
	3	80	116	2.1
Pit 3	3	96	150	2.35
	4	85	40	2.10
	5	100	150	2.47
	6	100	150	2.47
	7	96	135	2.35
	8	98	139	2.47
Pit 2	2	80	150	1.97
Pit 5	10	30	60	0.7
	11	50	85	1.2
	12	75	150	1.8
	13	88	150	2.2
	14	93	150	2.2

4.6.2 Auger Hole Solids Generation and Interrogation

Each auger hole was designed as a 1.9m diameter three dimensional solid (cylinder) and laid out along the auger areas applying the web width and penetration depth.

The solids were then interrogated by the geological model to generate in situ quantities (volume) and coal qualities (relative density, raw ash, product yield, inherent moisture etc).

The layout of the auger holes is shown in Figure 4-10.

4.6.3 Mining Assumptions and Modifying Factors

Modifying factors were applied to the in situ quantities and qualities to convert from an in situ basis to a ROM / product basis and were similar to the Budget plan.

4.6.3.1 Loss and Dilution

No loss or dilution was applied as this mining method does not incur any.

4.6.3.2 Moisture Adjustments

The moisture assumptions used are the same as the Budget plan moistures listed in Table 4-3.

4.6.3.3 Product Yield

As explained in Section 4.3.5.4, reconciliations of current operations recommended the following adjustments to the primary and secondary grid values be applied:

- Pit 2 – Primary Yield = Grid Value – 6.5%
- Pit 2 – Secondary Yield = Grid Value + 8.0%
- Pit 3 – Primary Yield = Grid Value – 0.5%
- Pit 3 – Secondary Yield = Grid Value – 0.3%
- For all other pits, no adjustments are required.

The yields are applied to the undiluted ROM coal then converted from the ROM moisture basis to their respective product moisture basis.

4.6.4 Auger Scheduling

Stanmore plans to commence auger mining in Pit N1 at IPM then relocate to IPE Pit 4, Pit 2, Pit 3 and Pit 5 in that order. The auger can achieve an average production rate of 1,000 tonnes/day working 5 days a week. Figure 4-11 shows the quantities of ROM coal recovered by calendar year by pit.

Figure 4-10 – Auger Mining Locations

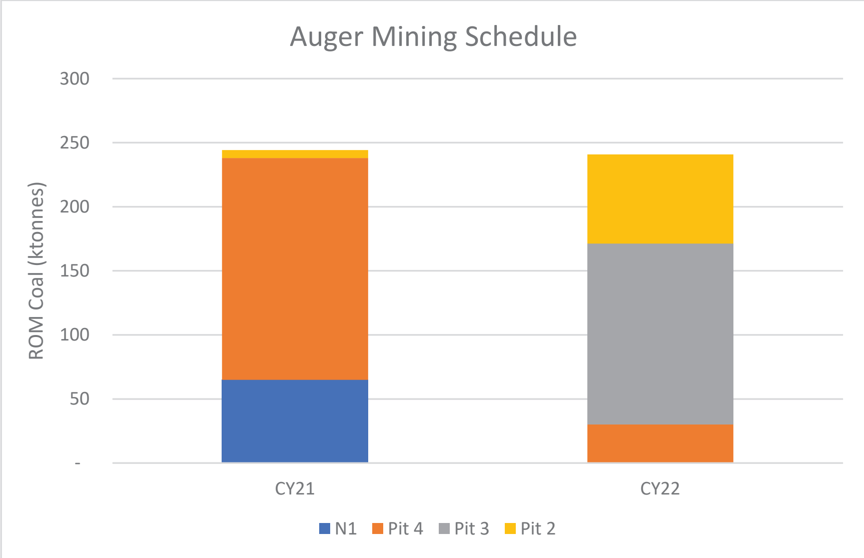

Figure 4-11 - Auger Mining ROM Coal Schedule

4.6.5 Financial Evaluation

The quantities and qualities of the auger hole solids were inputted into a financial model to evaluate the economic margin of the auger mining areas utilising. Stanmore has signed a services agreement with Coal Augering Services Pty Ltd (CAS) to undertake auger mining on site which forms the basis of the input costs.

4.6.5.1 Cost Assumptions

Stanmore's budgeted costs provided by CAS and site contractors (including mobilisation/ demobilisation, coal haulage and pit services) results in an average unit cost of AU$44.87/ROM t delivered to the CHPP. This results in a total FOB cost including CHPP, rail, port, and royalties of AU$91.10/product tonnes.

4.6.5.2 Revenue Assumptions

The revenue assumptions are based on the forecast prices as described in Section 4.3.8.2.

Table 4-9 lists the revenue assumptions used for each pit based on the auger mine schedule and the time that the coal will be recovered.

Table 4-9 – Revenue Assumptions by Pit

Auger Pit	HCC Forecast Price (USD)	SSCC Equivalent Price (USD)	Thermal Equivalent Price (USD)	Average Sale Price (AUD) after Product Split
Pit N1	$142	$98.94	$67.07	**$124.72**
Pit 4	$142	$98.94	$67.07	**$133.03**
Pit 3	$143	$99.36	$67.54	**$130.51**
Pit 2	$143	$99.36	$67.54	**$123.79**
Pit 5	$149	$103.19	$70.38	**$139.01**

5 Reserve Estimate

5.1 Reserve-Resource Clarification

- Proved Reserves are subsets of areas of Measured Resources category
- Probable Reserves are subsets of areas of Indicated Resources category

5.2 Reserve Locations

The Reserves are wholly contained within MLs 70342, 700017, 700018 and 700019 with Figure 5-1 showing the location of the open cut mining Coal Reserves and Figure 5-2 showing the location of the Auger Mining Coal Reserves.

5.3 ROM Coal Reserves

The total open cut ROM Coal Reserves are presented in Table 5-1 and Table 5-2.

Table 5-1 – Isaac Plains Complex - Open Cut and Auger ROM Reserves Estimate

		Coal Reserve (million ROM tonnes)*
Budget Plan	Proved	1.53
	Probable	0.11
	Total	1.64
Final Cut	Proved	0.22
	Probable	0.03
	Total	0.26
Pit 5	Proved	0.82
	Probable	0.42
	Total	1.25
Auger Mining	Proved	0.47
	Probable	0.10
	Total	0.58
Total	**Proved**	**3.05**
	Probable	**0.67**
	Total	**3.72**

** Tonnages in the above table are expressed on a ROM basis, incorporating the effects of mining losses and dilution, and on a 7.0% ROM moisture basis.*

Table 5-2 – Coal Reserves by Mine

Coal Reserve (million ROM tonnes)	JORC Category	Open-cut Mining (Mt)	Auger Mining (Mt)	Total (Mt)
Isaac Plains	Proved	0	0.06	0.06
	Probable	0	0.00	0.00
Isaac Plains East	Proved	2.6	0.41	2.99
	Probable	0.6	0.10	0.67
Isaac Plains Complex	**Proved**	**2.6**	**0.47**	**3.05**
	Probable	**0.6**	**0.10**	**0.67**
	Total	**3.1**	**0.58**	**3.72**

5.4 Marketable Coal Reserves

The total Open Cut Marketable Coal Reserves are shown in Table 5-3.

Table 5-3 – Isaac Plains Complex - Marketable Open Cut Coal Reserve Estimate

Marketable Reserves (Product tonnes)		Stanmore Coking Coal (Mt)	Thermal Coal (Mt)	Total (Mt)
Budget Plan	Proved	1.09	0.06	**1.15**
	Probable	0.07	0.01	**0.08**
	Total	1.16	0.08	**1.24**
Final Cut	Proved	0.15	0.01	**0.17**
	Probable	0.02	0.01	**0.03**
	Total	0.17	0.02	**0.19**
Pit 5	Proved	0.59	0.01	**0.59**
	Probable	0.31	0.00	**0.31**
	Total	0.89	0.01	**0.90**
Auger Mining	Proved	0.37	0.03	**0.40**
	Probable	0.08	0.01	**0.09**
	Total	0.45	0.04	**0.49**
Total	**Proved**	**2.20**	**0.11**	**2.31**
	Probable	**0.47**	**0.03**	**0.50**
	Total	**2.67**	**0.15**	**2.82**

Note: Tonnages have been expressed on an as-received product moisture basis, which is 11.0% for semi-soft coking coal and 9.5% for thermal coal and account for product yield.

Figure 5-1 – Open Cut Mining JORC Reserve Areas

Figure 5-2 – Auger Mining JORC Reserve Areas

5.5 Accuracy of Estimate

Small differences may be present in the totals due to tonnage information being rounded so as to reflect the usual uncertainty associated with the estimate.

5.6 Previous Reserve Estimates

The most recent Coal Reserves Estimate for the Isaac Plains Complex was completed by Measured Group in July 2020.

During this time 1.19 million ROM tonnes and 0.9 million product tonnes have been depleted at IPC by mining. The remaining difference in the Coal Reserves is due to the updated economic assumptions reducing the economic pit limits.

The tonnages and differences are shown in Table 5-4.

Table 5-4 – Previous Reserve Estimate

Reserves Type		July 2020 (Mt)	December 2020 (Mt)	Difference (Mt)
Isaac Plains Complex	ROM (Proved + Probable)	11.26	3.72	-7.54
	Marketable (Proved + Probable)	8.48	2.82	-5.66

The proportion of the December 2020 ROM Coal Reserve classified as Proved is 83% which is an increase of 1% from the July 2020 Estimate. This is due to the increase in the Measured Resource category being within the economic pit limits.

Appendix A. JORC CODE 2012 EDITION – TABLE 1 FOR ISAAC PLAINS COMPLEX COAL RESERVE AS AT 31ST DECEMBER 2020

This Appendix details section 4 of the JORC Code 2012 Edition Table 1. Section 5 Estimation and Report of Diamonds and Other Gemstones has been excluded as they are not applicable to this deposit and estimation.

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in Section 1, and where relevant in Sections 2 and 3, also apply to Section 4)

Criteria	JORC Code Explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. • Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	• The JORC Coal Resource for Isaac Plains Mine (**IPM**) (June 2020) was estimated by Troy Turner, a full-time employee of Xenith Consulting Pty Ltd. Mr Turner is a qualified geologist and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking, to qualify as Competent Person as defined in the 2012 Edition of the “Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.” • The Coal Resource Estimate for the Isaac Plains Mine is:  • The JORC Coal Resource for Isaac Plains East (IPE) (June 2020) was estimated by Bronwyn Leonard, a full-time employee of Stanmore IP Coal Pty Ltd. Dr Leonard is a qualified geologist and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking, to qualify as

Criteria	JORC Code Explanation	Commentary
		Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves." • The Coal Resource Estimate for Isaac Plains East is: Resource Category / IPE: Measured (Mt) 9.8; Indicated (Mt) 8; Inferred (Mt) 4; **Total (Mt) 22** • Both estimates have been used as the basis for the estimate of Coal Reserves for the Isaac Plains Complex. • Coal Resource estimates are inclusive of Coal Reserve estimates.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The Competent Person, Mr Tony O'Connell, has visited the site on multiple occasions in the past 3 years. • The site visits, reports and a review of mining, production and reconciliation data confirms the mining methods used at IPM and IPE are suitable for current and planned open-cut mining operation; and are being well managed by the IPC operations teams.
Study status	• The type and level of study undertaken to enable Mineral Resources to be converted to Ore • Reserves. • The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically	• Mine planning for IPC has been undertaken to a high level of detail to support current open-cut mining operations. Stanmore maintains an in-house mine planning function for mid to long term planning, and the current mining contractor (Golding) maintains a mine planning function to manage the open-cut mining operation. • The mining parameters and modifying factors are based on the experience of the current operations.

Resource Category	IPE
Measured (Mt)	9.8
Indicated (Mt)	8
Inferred (Mt)	4
Total (Mt)	**22**

Criteria	JORC Code Explanation	Commentary
	viable, and that material Modifying Factors have been considered.	
Cut-off parameters	• The basis of the cut-off grade(s) or quality parameters applied.	• The pit designs for the IPC were developed to cover all coal production that is expected to be economical. • At Isaac Plains Mine, open cut mining is complete, whilst at Isaac Plains East, the economic limits of the active pits was based on Stanmore's 2021 LOM plan financial model whilst for the undisturbed Pit 5 area Deswik (Pseudoflow) was utilised to determine the economic pit shell backed up by a block margin rank to confirm the limits.
Mining factors or assumptions	• The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). • The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. • The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling. • The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate). • The mining dilution factors used. • The mining recovery factors used. • Any minimum mining widths used.	• The open cut mining methodology considered for this estimate is: ○ a combination of cast, doze, dragline or truck & excavator to move waste into the adjacent strip or dump. The strip width selected is nominally 50m at IPE. ○ Drilling and blasting (D&B) of the in situ waste. ○ A maximum horizon of 50m of waste is allocated to the dragline. ○ Remaining waste is removed by truck and excavator. ○ Coal mining using excavators and rear dump trucks haul the coal to the Isaac Plains Complex Coal Preparation Plant (IPC CHPP) for washing. ○ Parting > 0.3m thick is stripped separately. • Batter allowances that have been considered are: ○ Highwall (hard): 70° ○ Highwall (soft): 45° ○ Spoil Lowwall & Angle of Repose: 37° Loss & Dilution factors used are:

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- The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.
- The infrastructure requirements of the selected mining methods.

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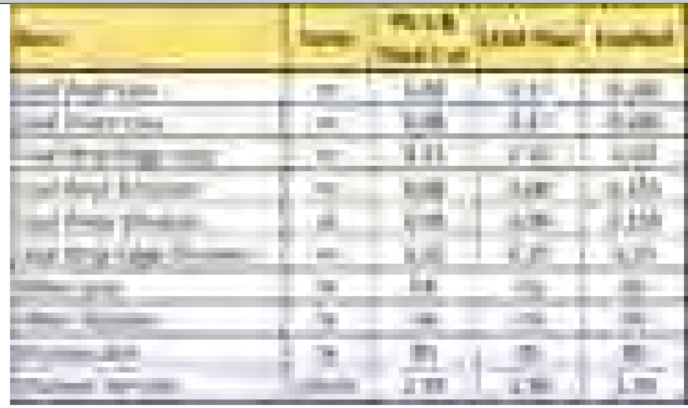

- Auger mining will also be employed to recover coal exposed in the highwall and endwall beyond the economic limits of open cut mining:
  - 1.9m diameter auger holes
  - Maximum penetration depth 150m
  - Web Widths (pillars) based on overburden thickness

Moisture Assumptions used:

Item	Units	Value
Air-dried Moisture	%	As modelled*
Insitu Moisture	%	5%
ROM Moisture	%	7%
Semi-soft Product Moisture	%	9%
Thermal Product Moisture	%	11%

- The existing infrastructure at IPC is suitable for the methodology described.

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Criteria	JORC Code Explanation	Commentary
Metallurgical factors or assumptions	• The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. • Whether the metallurgical process is well-tested technology or novel in nature. • The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. • Any assumptions or allowances made for deleterious elements. • The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. • For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	• The existing IPC CHPP is suitable to process the target seams. • Two products are planned, a primary product semi-soft coking coal and a secondary product thermal coal. • The CHPP yield predictions are based on modelled theoretical laboratory yield data with reconciliation adjustments applied to predict plant performance.

Environmen-tal	• The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	• All Mining Leases within the IPC are subject to environmental authority (**EA**) EPML00932713.
		• Stanmore's onsite activities are managed in accordance with the following:
		  ○ Environmental Management Strategy;
		  ○ Environmental management procedures for complaints, stakeholder interaction, water management, dams, air quality/dust, land (including permit to disturb, weed and pest control, and spills management), waste, blasting and safety;
		  ○ IPM Mine environmental management plan; and
		  ○ contractor's environment management plans.
		• These strategies, procedures and plans will be amended as required.

Criteria	JORC Code Explanation	Commentary
		• Environmental risk assessments of the following aspects have been undertaken, in conjunction with relevant specialists: ○ Groundwater ○ Flood modelling ○ Water management ○ Air quality ○ Noise ○ Terrestrial ecology ○ Aquatic ecology. • Stanmore assesses and monitors environmental and approvals risks on an ongoing basis.
Infrastructure	• The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.	• Existing Infrastructure supporting IPC operations includes: ○ Mine infrastructure Area; ○ Heavy vehicle haul roads connecting IPE to IPM CHPP; ○ Workshop including surrounding laydown areas; ○ Light vehicle maintenance igloo; ○ Boiler makers area; ○ Fuel storage and distribution; ○ Administration Office (including parking areas); ○ Warehouse; ○ Emergency Response Facilities Equipment; ○ Fuel and Lubrication Facilities; ○ Electrical and communications; and ○ Water Infrastructure (Raw, Potable & Process) • The original design criteria for the Isaac Plains mine was 3.5 Mtpa ROM and the existing infrastructure capacity is currently surplus to requirements.

Criteria	JORC Code Explanation	Commentary
Costs	• The derivation of, or assumptions made, regarding projected capital costs in the study. • The methodology used to estimate operating costs. • Allowances made for the content of deleterious elements. • The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products. • The source of exchange rates used in the study. • Derivation of transportation charges. • The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. • The allowances made for royalties payable, both Government and private.	• The utilised costs have been sourced from current contractor rates or built up from first principles where required. • All unit cost rates are in Australian Dollars. • Royalty charges were applied as follows: ○ up to and including $100 per tonne: 7.0% ○ over $100 up to including $150 per tonne: 12.5% ○ above $150 per tonne: 15.0% • Private royalties are also included.
Revenue factors	• The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. • he derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	• Revenue assumptions are based on the historical relative price Stanmore receives for IPE's coking coal (SSCC) and thermal coal products compared to the benchmark Hard Coking Coal (HCC) price. • Using KPMG's 'Coal Price and FX Market Forecast' report dated September/October 2020, Stanmore has taken the following position on the HCC forecast:

	2021	2022	2023	2024	Long Term
Stanmore Benchmark HCC Forecast (USD/tonne FOB)	$142	$143	$145	$147	$149

Criteria	JORC Code Explanation	Commentary
		• A USD:AUD exchange rate of 0.74 has been used.
Market assessment	• The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. • A customer and competitor analysis along with the identification of likely market windows for the product. • Price and volume forecasts and the basis for these forecasts. • For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	• Two product coal types are produced by IPC, these coal products have been successfully marketed by Stanmore and sold into export markets for the past 10 years (approximately). • It would be reasonable to expect that the IPC will have no difficulty in successfully marketing future coal tonnes produced (coking and Thermal).
Economic	• The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. • NPV ranges and sensitivity to variations in the significant assumptions and inputs.	• The Competent Person has assessed the latest net present value analysis and is confident that the analysis provides accurate forecasts of the economic viability of the Coal Reserves. The details of the internally generated economic evaluation is commercially sensitive and is not disclosed.
Social	• The status of agreements with key stakeholders and matters leading to social licence to operate.	• The mining tenure for Isaac Plains is Mining Lease (**ML**) 70342. Isaac Plains East is covered by Mining Leases 700016, 700017, 700018, and 700019 which are all held by Stanmore IP Coal Pty Ltd. • All Mining Leases for IPC are current and are subject to environmental authority (**EA**) EPML00932713. • Stanmore will continue to manage the IPC mining operations, which they have successfully done so to date, whilst developing and maintaining good

Criteria	JORC Code Explanation	Commentary
		relationships with key stakeholders and maintaining their social licence to operate.
Other	• To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: • Any identified material naturally occurring risks. • The status of material legal agreements and marketing arrangements. • The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	• There are no known issues that impact might impact on the Coal Reserve Estimate and classifications of the Coal Reserves. • Stanmore commenced mining operations at IPE in mid-2018.
Classification	• The basis for the classification of the Ore Reserves into varying confidence categories. • Whether the result appropriately reflects the Competent Person's view of the deposit. • The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	• Measured, Indicated and Inferred Coal Resources are estimated for IPC. All of the Measured Coal Resources contained within the economic limit of the open-cut pit have been classified as Proved Coal Reserves, while all Indicated Coal Resources contained within the economic limit of the open cut pit have been classified as Probable Coal Reserves. • The Coal Reserve Estimate and classification of Coal Reserves reflect the Competent Person's view and assessment of the deposit.

Criteria	JORC Code Explanation	Commentary

Coal Reserve (million ROM tonnes)		
Budget Plan	Proved	1.53
	Probable	0.11
	Total	1.64
Final Cut	Proved	0.22
	Probable	0.03
	Total	0.26
Pit 5	Proved	0.82
	Probable	0.42
	Total	1.25
Auger Mining	Proved	0.47
	Probable	0.10
	Total	0.58
Total	**Proved**	**3.05**
	Probable	**0.67**
	Total	**3.72**

Marketable Reserves (Product tonnes)		Stanmore Coking Coal (Mt)	Thermal Coal (Mt)	Total (Mt)
Budget Plan	Proved	1.09	0.06	**1.15**
	Probable	0.07	0.01	**0.08**
	Total	1.16	0.08	**1.24**
Final Cut	Proved	0.15	0.01	**0.17**
	Probable	0.02	0.01	**0.03**
	Total	0.17	0.02	**0.19**
Pit 5	Proved	0.59	0.01	**0.59**
	Probable	0.31	0.00	**0.31**
	Total	0.89	0.01	**0.90**
Auger Mining	Proved	0.37	0.03	**0.40**
	Probable	0.08	0.01	**0.09**
	Total	0.45	0.04	**0.49**
Total	**Proved**	**2.20**	**0.11**	**2.31**
	Probable	**0.47**	**0.03**	**0.50**
	Total	**2.67**	**0.15**	**2.82**

Criteria	JORC Code Explanation	Commentary
Audits or reviews	• The results of any audits or reviews of Ore Reserve estimates.	• Coal Reserve Estimates were reconciled back to previous estimates to ensure consistency.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• No statistical or geostatistical procedures have been used in the estimation of Coal Reserves themselves. • The most significant areas of uncertainty in the Isaac Plains Complex open-cut reserve estimate relates to the coal pricing and foreign exchange rate. However, the present forecasts are based on highly regarded industry experts in this field. • Small differences may be present in the totals due to the tonnage information being rounded to reflect the usual uncertainty associated with the estimate. • The in-seam yields for IPM and IPE have been adjusted by factors calculated via a robust reconciliation process.

APPENDIX C — RAVENSWOOD JORC REPORTS

See next page.

Ravenswood Gold

Sarsfield-Nolans Mineral Resource Estimate July 2020

FINAL

Project Code:	**RAV002007**
Report Date:	**27 November 2020**
Effective Date	**30 September 2020**
Authors:	**Scott Dunham**

Release Date Addendum

Sarsfield-Nolans Mineral Resource Report -27 November 2020

SD2 Pty Ltd (SD2) completed a Mineral Resource Estimate for the Sarsfield-Noland gold deposit in July 2020. This addendum is an addition to the then published Mineral Resource technical report outlining SD2's analysis of changes at Sarsfield-Nolans between 10 July 2020 and 30 September 2020 (the 'effective date') and SD2's opinion on the materiality of any changes identified during that period.

Changes Potentially Effecting the Mineral Resource

The Sarsfield-Nolans deposit is part of the greater Ravenswood Gold Pty Ltd (Ravenswood) Mineral Resource base. Ravenswood are currently in the process of developing an open pit mining operation to extract the near-by Buck Reef West Mineral Resource. Mining of the Sarsfield-Nolans mineralisation has not yet commenced.

In the period between 10 July 2020 and 30 September 2020 the following resource estimation related activities occurred:

1. Development activities focused on establishing and refurbishing the pre-existing mine camp, ore treatment plant and on procurement and delivery of the new mining fleet for the nearby Buck Reef West open pit;
2. Production was restricted to processing of historical sub-grade stockpiles (not included in this report);
3. No surface mining took place. The topographic surface for 10 July 2020 and 30 September 2020 is identical;
4. Sixteen (16) new drill holes were completed into the mineralisation for a total of 3,414m. This represents a 1% increase in the number of holes and a 2% increase in the drilled metres compared to the data available at 10 July 2020.

Of these four activities, only the additional drilling has the potential for material impact on the quality and quantity of the estimated Mineral Resource.

Materiality Checks

SD2 reviewed the 16 drill holes completed between 10 July 2020 and 30 September 2020 (Figure 1).

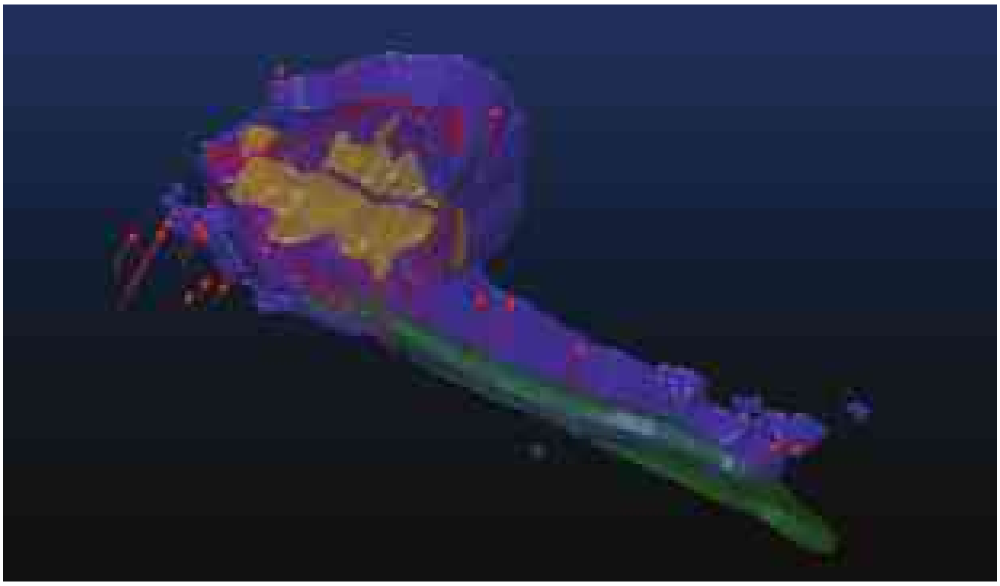

Figure 1. Location of holes drilled between 10 July 2020 and 30 September 2020.

Eight (8) of the new holes were targeted at the zone between the Sarsfield-Nolans resource and the Buck Reef West resource, south of the reported Sarsfield-Nolans resource estimate. These eight holes all lie beyond the reporting limit of the Sarsfield mineralisation. The holes indicate sporadic, narrow medium-to-high mineralisation in the region; however, the level of understanding of this region was incomplete as at 30 September 2020. Therefore, these holes, while promising, are not considere3d material to the global resource estimate.

Two (2) new holes were drilled at the far eastern end of the Sarsfield-Nolans resource, testing the continuity of the mineralisation around a number of existing drill holes to check the potential of an extended eastern zone. These two holes (NLRC001 and NLRC002) did not intersect any mineralisation, a result that is aligned with the 10 July 2020 estimate. Neither hole has the potential to change the mineral resource.

Three (3) new holes were drilled into the Nolans region of the mineral resource. Hole NLRC004 intersected 27m @ 0.8 g/t Au and was stopped before reaching the end of the mineralised zone due to drilling constraints. This intersection compares favourably with the grade and location of the estimated mineralisation. Hole NLRD005 intersected weak mineralisation (<0.2 g/t Au) in the interpreted mineralised zone. This is lower than the predicted grade which is supported by a cluster of higher grade holes approximately 50m up-dip. Hole NLRC003 did not reach the mineralised zone due to drilling constraints. SD2's analysis of these three holes indicates they will not materially impact on the global estimated tonnes and grade. While NLRD005 intersected lower than predicted grades, the maximum zone of influence of this single hole is constrained by adjacent pre-existing holes. In SD2's opinion the maximum impact expect by including NLRD005 in an updated estimate would be a change of less than 1% in both the reported tonnes and reported ounces. This is not considered material.

The remaining three new holes were drilled along the northern edge of the Sarsfield portion of the mineral resource, testing the continuity of the mineralisation at the limits of the reporting boundary beyond the interpreted domains. Hole SFRD007 intersected several (7) narrow veins across a 300m down hole length including a zone of 12m @ 0.9 g/t Au. Hole SFRD008 intersected the Bucks Reef Fault at the interpreted location. The 10 July 2020 estimate predicted the intersected zone to be less than 0.3 g/t Au. The actual intersection was 17m @ 0.6g/t Au indicating the potential for a slight increase in the estimated grade adjacent to the new hole; however the intersection is outside the limits of the 'reasonable prospects' test (AUD3800 shell). Hole SFRC005 intersected 7m @ 1.3 g/t Au west of the limit of the estimated mineral resource. While this hole is promising, the level of geological understanding is insufficient for the intersection to be included in the estimate. Based on SD2's analysis, none of these three holes are considered material.

The impact of the holes drilled between July 2020 and September 2020 is summarised in Table 1. The 8 new holes with potential to impact on the Mineral Resource estimate are consistent with the July 2020 estimate. These 8 holes represent a less than 1% increase in the number of drill holes in the mineralisation.

Table 1. Materiality of new drilling.

Hole category	**Number of Holes**	**Material to Estimate**	**Comment**
Greater than 200m from resource	8	No	Drilled to test exploration targets.
Far eastern Nolans region	2	No	Drill results match resource prediction.
Mid-Nolans region	3	No	Mineralisation not tested in two holes due to drilling constraints. Remaining hole intersected lower than anticipated grades; however influence is limited by adjacent holes and is likely to be less than 1% of the reported tonnes and ounces
Northern Sarsfield region	3	No	Holes are all beyond the reported limits of the mineralisation. Results look promising however additional geological knowledge is required before they can be successfully incorporated into the resource estimate.
Total Number of new holes	**16**		

Conclusions

In SD2's opinion the Sarsfield-Nolans 10 july 2020 Mineral Resource estimate is suitable for reporting as at 30 September 2020. No mining activities have occurred at Sarsfield-Nolans and therefore the resource estimate does not need to be depleted to allow for extraction. Similarly, in the absence of mining there are no reconciliation data available or geological observations to justify altering the interpretation of the resource.

While Ravenswood drilled an additional 16 holes between July 2020 and September 2020, only 8 of these holes targeted areas within or adjacent to the reported resource. Seven (7) of these 8 additional holes confirmed or improved the grade tenor, width and location of the interpreted mineralisation. One (1) hole was of lower than predicted grade. In SD2's opinion, the new drilling has had minimal impact on the global resource estimate (<1%) and they do not materially alter the quality or quantity of the estimate.

Tabulated Mineral Resource as at 30 September 2020

Sarsfield-Nolans Mineral Resource estimate as at 30 September 2020

Classification	Tonnes (kt)	Au (g/t)	Ounces (koz)
Measured	32,213.0	0.71	739.9
Indicated	71,354.2	0.65	1,498.2
Inferred	29,394.2	0.63	597.8
Grand Total	**132,961.4**	**0.66**	**2,835.9**

Reported above a 0.3 g/t Au cut-off above AUD3800 shell assuming material above 0.63g/t Au is direct feed to the Ravenswood ore treatment plant and material between 0.30g/t and 0.63g/t is beneficiated at the Ravenswood beneficiation plant.

This resource statement is based on an estimate of the Sarfield-Nolans mineralisation completed 10 July 2020 and an assessment of materiality of changes between July 2020 and September 2020. The complete technical report for the estimate is included in the following documentation.

Executive Summary

The Sarsfield-Nolans gold deposit is part of the Ravenswood gold mine in north Queensland. An updated mineral resource estimate has been developed incorporating the most recent drilling and geological information. This estimate supersedes previous estimates for the Sarsfield-Nolans mineralisation.

The July 2020 resource model is based on a combination of ordinary kriging and multiple indicator kriging in domains developed from manually controlled implicit modelling.

The resource is classified under the JORC Code (2012) as Measured, Indicated and Inferred. Classification was on the basis of sample spacing, geological confidence and a range of estimation quality metrics including the block-to-sample distance and configuration.

The Sarsfield-Nolans mineral resource estimate is reported above a cut-off of 0.3 g/t Au within an AUD3,800[1] optimised pit shell as at 10 July 2020.

Classification	Tonnes (kt)	Au (g/t)	Ounces (koz)
Measured	32,213.0	0.71	739.9
Indicated	71,354.2	0.65	1,498.2
Inferred	29,394.2	0.63	597.8
Grand Total	**132,961.4**	**0.66**	**2,835.9**

[1] AUD3,800 selected as 1½ the gold price as at the time of reporting.
AUD3,800 pit shell assuming material above 0.63g/t Au is direct feed to the Ravenswood ore treatment plant and material between 0.30g/t and 0.63g/t is beneficiated at the Ravenswood beneficiation plant.

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1. Introduction and Scope

Ravenswood Gold Pty Ltd (Ravenswood, or RAV) recently acquired the Ravenswood gold mine located 130km by road from Townsville and 90km from Charters Towers. Ravenswood mine is not currently operating; however, a feasibility study outlining the mining opportunity and steps required to start production formed a central part of the sale and acquisition process. As part of the acquisition process SD2 Pty Ltd (SD2) was engaged to review the resource model and estimate for Ravenswood and consider the suitability of the model for future mine planning. This investigation identified risks with the resource estimate, mainly around the grade-tonnage distribution. Consequently, RAV engaged SD2 to re-estimate the mineral resources at Ravenswood.

This report outlines the resource estimation approach adopted for the Sarsfield-Nolans (Sars) deposit, the second of two major resources in the area.

1.1 Location and History

The Ravenswood gold mine is one of a number of gold deposits in the Ravenswood-Lolworth Province of northeast Queensland. Alluvial gold was discovered at Ravenswood in 1868 followed by the discovery of oxidised gold-bearing quartz reefs. By 1872 most of the near-surface oxide mineralisation had been depleted (McIntosh et al. 1995) and only the refractory sulphide-associated mineralisation remained. A second phase of production started with the formation of the New Ravenswood Company in 1896 and focused on extracting this sulphide-associated gold from lodes and veins including the Duke of Edinburgh, General Grant, Sunset, London, Mellaneur, Shelmallier (MSA) and Black Jack systems. The majority of gold was from the Sunset lode which produced 208,949 oz from a 45° dipping vein to a depth of 200m below surface (Collett et al., 1998). Production decreased rapidly after 1912 due to exhaustion of the Sunset Lode, an extended miner's strike and the impact of World War 1.

There was limited activity at Ravenswood from 1917 to 1980. Silver was produced from the nearby (1.6km north) Totley mine in the 1950s; otherwise production was limited to minor underground extensions and few drill holes. In the early 1980's The North Queensland company reprocessed several old mullock dumps and tails dams. In 1985, MIM Exploration Pty Ltd (MIM) began exploring the Ravenswood district and, following early success MIM's subsidiary Carpentaria Gold (CG) began open pit production at Bucks Reef West (BRW) , Slaughter Yard Creek (SYC) and OCA in 1987. The operation commenced as a heap leach (250 Ktpa) and small (100 Ktpa) CIL operation before the construction of a 2.4 Mtpa CIL plant in 1993. This plant was expanded to 5.5 Mtpa in 2000 to enable treatment of production from the Sarsfield and Nolan's open pits (Lisoweic, 2009).

Production at the Sarsfield open pit was completed in 2009 and the ore treatment plant was de-rated to 1.5 Mtpa in 2011 while focus switched to the nearby Mt Wright underground operation. There was a hiatus at Ravenswood until 2016 when the Nolan's East open pit commenced. As of 2020 production is limited to treating old stockpiles and dumps until the plant is refurbished and approval given to recommence operations at Buck Reef West.

Historically the Ravenswood area has produced approximately 2.4Moz at an overall average grade of 1.7g/t. (Table 2). Excluding production prior to 1987 the area produced 1.5Mz at a grade of 1.1g/t.

Year	Operation	Recorded Production	Ounces	Source
Pre 1987	Lode mineralisation across entire field.	No tonnes and grade reported. Estimated grades reported as 30 g/t (Lisowiec, 2009).	900-950,000	Collett et al. 1998. Lisowiec, 2009.
1987 – 1990	SYC (pit)	526,000 @ 2.7 g/t	45,700	Collett et al. 1998
1987 – 1989	OCA (pit)	290,000 @ 3.4 g/t	31,700	Collett et al. 1998
1988 – 1991	BRW (pit)	160,000 @ 2.8 g/t	14,400	Collett et al. 1998
1991	OCA (ug)	149,000 @ 4.1 g/t	19,600	Collett et al. 1998
1990	Area 4 (pit)	50,000 @ 2.4 g/t	3,900	Collett et al. 1998
1988 -1991	Area 5 (pit)	260,000 @ 2.4 g/t	20,000	Collett et al. 1998
1990 – 1991	MSA (pit)	48,000 @ 3.5 g/t	5,400	Collett et al. 1998
1992 – 1993	Area 2 (ug)	174,000 @ 10.1 g/t	56,500	Collett et al. 1998
1993 – 1996	Nolans (pit)	4,100,000 @ 1.25 g/t	164,800	Collett et al. 1998
2003 – 2005	BRW (ug)	376,000 @ 4.0 g/t	48,400	Lim et al., 2018
2000 – 2003	Sarsfield (pit)	3,900,000 @ 1.24 g/t	155,500	Haoma Mining Annual Report 2003
2004 – 2009	Sarsfield (pit) Note, introduction of MIK for resource estimation	33,490,000 @ 0.91 g/t	980,000	Lim et al., 2018
Total Recorded Production		**44.3Mt @ 1.7 g/t**	**2,400– 2,450,000**	
Open Pit Only		**40.0Mt @ 1.7 g/t**	**2,150-2,200,000**	
Pits After 2000		**37.4 Mt @ 0.9 g/t**	**1,100,000**	

Table 2. Historic Production (Ravenswood).

SD2 note the differences in average mined grade between production at Sarsfield/Nolans in 1993-1996 (1.25 g/t), 200-2003 (1.24 g/t) and 2004-2009 (0.91 g/t). While it is not possible to directly relate the decrease in grade to a single cause, it is notable that the 2004-2009 production was carried out by the new operator (Resolute). At the time of Resolute's acquisition of Ravenswood from Xstrata, there was a stated plan to improve operational performance by reducing strip ratio and changing grade control practices (Resolute 2004 Annual Report; Figure 2).

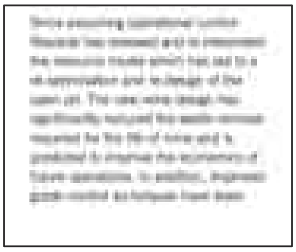


Figure 2. Extract form Resolute 2004 Annual Report.

It is possible this change in strategy was linked to the adoption of unconstrained multiple indicator kriging (MIK) as the resource estimation and grade control estimation methodology and the decrease in grade may reflect a corresponding decrease in selectivity. Much of the detail of estimation and operational practices in and around the 2003-2004 period have been lost following a legal dispute between Haoma Mining and MIM Holding Limited and the subsequent sale of Haoma's interest in the Nolans open pit to MIM which was shortly followed by Xstrata's acquisition of MIM itself.

1.2 Work Completed

The July 2020 Sarsfield mineral resource estimate was a complete revision of the previous modelling reported by Resolute Mining Limited. Consequently, SD2 completed a comprehensive review of past practices, the data quality and previous estimates as part of developing the new estimation strategy. In addition to the activities completed in the Due Diligence study, SD2's work included the following:

- Review of the geology database and request for an extract covering the Sarsfield mineralisation;
- Review of surfaces required for the estimate including topography and voids;
- Review of the operational grid system and the history of grid transformations;
- Review of drill hole collars against topography (where appropriate);
- Review of geological logging systems and results. Consideration of ways to best incorporate logging into the resource estimate;
- Review of bulk density data and consideration of its suitability for resource estimation;
- Review of sampling and assay data with a particular focus on high-grade samples occurring adjacent to unsampled intervals;

- Review of quality control performance data collected at the time of drilling and sampling;
- Collation of all files and metadata used for previous estimates where possible;
- Review of the structural geology of the Ravenswood district and Sarsfield-Nolans specifically. Consideration of how the geological structure influences the resource model and estimate;
- Review and update of the geological interpretation. Development of domains suitable for resource estimation;
- Estimate the mineral resource and document the estimation process and results (this document);
- Prepare the July 2020 estimate for use in mine planning; and
- Prepare a geology/resource risk assessment and report.

All of the work completed for the mineral resource estimate was carried out under the guidelines of the JORC Code[2] (2012 edition) reporting framework.

1.3 Previous Estimates

There are two recent estimates for the Sarsfield-Nolans deposit, one completed by MPR Geological Consultants Pty Ltd (MPR) in August 2019 as part of Resolute Mining Limited's REP200 project and a second 'sensitivity estimate' completed by SD2 in September 2019 as part of the Due Diligence review completed during Ravenswood Gold's purchase of the operation. These two estimates were similar, differing only in the approach adopted for the main stockwork mineralisation zone.

Both models were multiple indicator kriging estimates. The SD2 sensitivity estimate applied a volume constraint to the stockwork zone whereas the MPR estimate was unconstrained. By constraining the stockwork, the SD2 sensitivity model limited the estimation distance away from the available data, restricting the estimate to a volume with a higher likelihood of containing mineralisation. This also altered the grade-tonnage curve, removing a portion of low-grade estimates found in the MPR model.

The results of this new (July 2020) estimate are compared to both the MPR and sensitivity models using identical reporting parameters. The results of these comparisons are shown in Appendix H. In broad terms the estimates are similar; however, the spatial distribution of the remaining mineralisation is slightly different reflecting the impact of SD2's incorporation of new structural orientations as part of the controls on the mineralisation.

[2] JORC Code – The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.

1.4 Changes in Methodology

The July 2020 uses a combination of ordinary kriging (OK) and multiple indicator kriging (MIK). Domains with a strong geological basis were estimated using OK whereas the stockwork zone was estimated using MIK. The estimation strategy uses the following steps:

1. Implicit modelling geology interpretation for:
    a. Buck Reef Fault (BRF);
    b. A4 Fault (A4);
    c. Keel Zone; and
    d. Nolan.
2. Grade shell based implicit modelling for:
    a. West of BRF;
    b. BRF to A4;
    c. South of A4; and
    d. Nolans background.
3. Estimation of BRF, A4, Keel Zone, Nolans, Nolans background and West of BRF by ordinary kriging;
4. Estimation of BRF to A4 and South of A4 by multiple indicator kriging;
5. Estimation of global background grades by ordinary kriging;
6. Assignment of a low-grade default (0.005 g/t Au) to all unestimated blocks; and
7. Post-processing to assign bulk density, fill and void values.

1.5 Critical Risks

The Sarsfield-Nolans mineralisation comprises a zone of weak stockwork development with relatively low fluid flow. The mineralisation, while associated with the pervasive stockwork veins, is discontinuous over short distances. This impact on the quality of the estimate. The MIK approach is suited to the mineralisation style; however, it potentially overstates the underlying grade continuity. In particular the estimate is sensitive to the composite search strategy. This is typical of some MIK estimates. In this case SD2 has ameliorated the risk by requiring a relatively high number of composites. This provides a better estimate of the cumulative distribution function (cdf) resulting in a more realistic picture of the grade distribution.

SD2 reviewed multiple alternate geological interpretations while completing the Sarsfield estimate. While some structures (BRF and A4 Fault) are easily defined, recognising other spatial controls on the extent of mineralisation proved difficult. A preliminary manual interpretation was provided by Ravenswood; however, both statistical analysis and a review of the spatial extent of the interpreted zones failed to support any meaningful differentiation.

Furthermore, the provided interpreted volumes were not well aligned with the main structural trends (or variography). SD2 also attempted to develop robust domains using an implicit modelling approach. Models were developed directly from the grade data and by using various proxies for mineralisation including a 'grade-density' indicator derived by calculating the number of composites above a series of grade thresholds within a 10mx10m block. While the 'grade-density' approach showed some initial promise, the modelled volumes and shapes were discontinuous and centred around individual drill holes rather than reflecting the extent of the mineralisation.

The difficult in developing geologically based domains for the stockwork mineralisation is a risk. There is a low degree of differentiation in the host tonalite and the apparent correlation between structural location and absolute grade is poor. The quality of the estimate is therefore dominated by the drill hole density. The final grade control drilling method and sample spacing will be a fundamental driver of estimation quality and ultimately ore recovery.

As well as the risks associated with the style of mineralisation, it is worth noting that the reporting pit shells are also data constrained. This is also the case for the likely ore reserve pit shell.

Additional drilling at depth and to the peripheries of the mineralised zone is recommended.

2. Project Description

2.1 Site and Existing Infrastructure

The Ravenswood gold mine is accessed by sealed road from a turn-off on the Burdekin Falls Dam Road. The site sits adjacent to the historic Ravenswood township (population 200) and there are several heritage-listed structures in and around the district. Mining operations at Ravenswood ceased in 2009; however, the ore treatment plant was used to treat production from the nearby Mt Wright underground mine and therefore is still operational. At present (July 2020) low grade stockpiles produced during earlier mining of the Sarsfield-Nolans pit are being treated through the plant.

The site is well equipped with the requisite infrastructure for mining and ore treatment operations (Resolute, 2018). Power is supplied through existing connections to the state-wide grid (PowerLink) via the Ergon Energy distribution network. Water is supplied via a 20km pipeline from the Burdekin River and the site operates two surge dams to manage seasonal flow variations. Telecommunications are provided by Telstra and the site operates a dedicated frame-relay data link provided by Optus.

Other existing site infrastructure includes workshops and warehouses to service the ore treatment plant and mining fleet, offices, sewage treatment plant, on-site accommodation and messing. While much of this infrastructure will require upgrading over the life of the combined BRW and Sarsfield production, there are no known impediments preventing the provision.

The existing tailing storage facilities (TSF) are insufficient for full the currently planned production from BRW and the Sarsfield/Nolan pits. RAV are developing a tailings management strategy and have had several options developed for evaluation. Tails from processing of Sarsfield/Nolans low grade stockpiles are being dewatered and dry-stacked back in the Sarsfield open pit. Discussions with the Queensland Department of Environment and Science (DES) are on-going and SD2 is unaware of any impediment likely to prevent resolution of the tails storage requirements for Sarsfield-Nolans.

In SD2's opinion there are no infrastructure-related issues that would prevent production from the Sarsfield-Nolans deposit. Further information on the existing and planned infrastructure requirements is contained in the REP 200 feasibility study (Lim et al., 2018).

2.2 Tenements and Tenure

Ravenswood Gold took possession of a number of Mining Leases (ML), Mining Lease Applications (MLA) and Exploration Permits (EPM) as part of the acquisition of the Ravenswood operation in 2020. During the acquisition process (December 2019) Ravenswood engaged Hetherington Exploration and Mining Title Services Pty Ltd (Hetherington) to review the status of the acquired leases. Hetherington prepared a report (Martin, 2019) on the status of the leases based on information obtained from the Department of Natural Resources, Mines and Energy ("DNRME") My-Mines-Online ("MMOL") database and other information as supplied by DNRME, the tenement holder/s (obtained via the digital data room) and the Department of Environment & Science ("DES").

The status of these MLs and MLAs is summarised in Table 3 (after Martin, 2019).

Tenement	Native Title	Holder/s	Status	Granted	Expiry	Minerals	Area (Ha)	Security Deposit	Financial Assurance
ML 1380	Section 31	CG	Granted	28-11-74	30-11-34	Gold, copper, lead, molybdenum, silver, zinc	60.79 (total) 58.59 (surface)	Nil	(note 2)
ML 1412	Pre NTA	CG	Granted	15-01-81	31-01-23	Gold, bismuth, cobalt, copper, silver, tungsten, zinc	2.024 (whole)	Nil	(note 2)
ML 1532	Pre NTA	CG	Granted	24-10-85	31-10-27	Antimony, arsenic, bismuth, copper, gold, lead, silver, zinc	0.2023 (whole)	Nil	(note 2)
ML 1722	N/A	CG	Terminated (note 1)	05-09-91	14-05-19	N/A	N/A	Nil	N/A
MLA 100145	Section 31	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	1.03 (total) 0.34 (surface)	Nil	(note 2)
MLA 100147	Exclusive	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	0.2023 (whole)	Nil	(note 2)
MLA 100149	Exclusive	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	1.3 (whole)	Nil	(note 2)
MLA 100172	Section 31	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	58.46 (whole)	Nil	(note 2)

Note 1 - ML 1722 is not current. This ML was conditionally surrendered in favour of ML 10172 which was granted on 13-5-2019

Note 2 - All of these ML's are currently included in EPML00979013. Carpentaria have advised that they have recently paid an amount of $280,000 into the financial provisioning scheme. This is assumed to be an annual payment.

Table 3. Status of BRW tenements (After Martin, 2019).

At the time of Martin's report, the tenement holder was Carpentaria Gold Pty Ltd (CG). The transfer of the tenements to RAV is in progress and is expected to be finalised in due course. The MLs and MLAs include gold in the list of exploitable minerals and metals. The rent for all leases has been paid to 31-8-2020. The leases are all covered by a site-specific Environmental Agreement (EPML00979013).

Martin (2019) concluded that the MLs and MLAs appear to be in good standing with two caveats:

1. Local government authority (council) rates for some recently granted leases had not been paid.
2. Hetherington has relied on information provided by CA as the lease holder and therefore recommended a direct search application to DNRME to verify lease status and conditions.

In SD2's opinion there are no material issues related to tenement status and ownership. Ravenswood Gold own 100% of the listed titles.

2.3 Grid System

Multiple grid systems have been used at Ravenswood, reflecting the long production history and variable lode orientations. The mine grid (known as the A45 grid) has local north oriented to bearing 030° magnetic. Coordinates are truncated and lie between 12,000 and 14,000 in both northing and easting. Complicating matters further there is a 32.813 translation in elevation between the mine grid and other grids in the field. A list of the different grids and their translations was compiled by Kelly & Partners Consulting Surveyors in 1993. This list further validated in 2004. Figure 3 illustrates the differences in orientation between a selection of the known grids and Table 4 shows the 2-point rotation and translation data for conversions between local, AGD84_Z55 and MGA94 mapping grids.

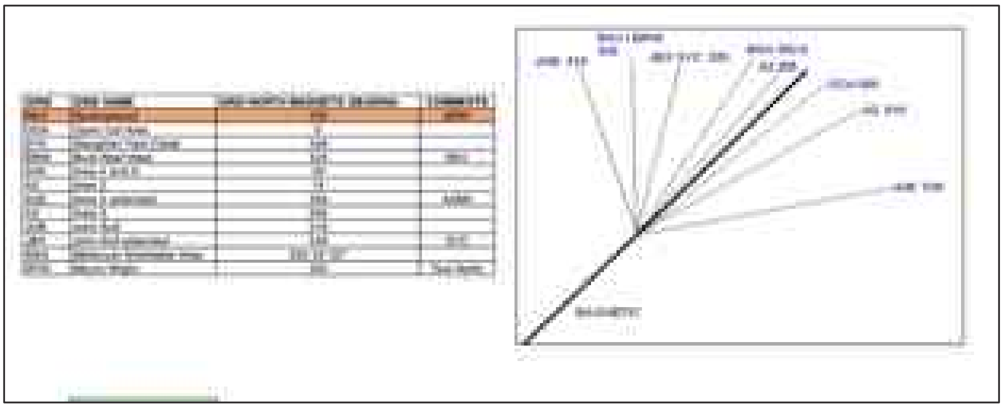


Figure 3. Ravenswood Local Grids.

Table 4. Ravenswood grid twin points.

The mine grid is reasonable well aligned with the axes of the mineralised zones in Sarsfield-Nolans. The Nolans domain runs east-west on the mine grid and the Sarsfield domain shows reasonable alignment north-south.

2.4 Site Visit

Scott Dunham completed three site visits to Ravenswood. The first during the due diligence study as part of the acquisition assessment team (August 2019) a second in January 2020 and a third visit in June 2020 specifically to discuss the Sarsfield-Nolans mineralisation.

3. Geology and Mineralisation

3.1 Regional Geology

The geology of the Ravenswood district has been described by several authors including Lim et al. (2018), Derham (2014), Berry et al. (1992), and Switzer (2000). The Ravenswood gold deposits lie within the Lolworth-Ravenswood block of the Charters Towers Province, a poorly exposed part of the regional Thomson Fold Belt (Figure 4). The Lolworth-Ravenswood Block comprises remnant amphibolite-grade metamorphic rocks intruded by an elongate east-west Ordovician-Silurian batholith (the Ravenswood Batholith) with an outcrop of 150km by 220km. The batholith is bound to the south by the Cambrian-Ordovician Seventy Mile Range Group of the Thalanga Province and the Devonian-Carboniferous Drummond Basin. The Devonian Burdekin Basin forms a northern boundary and to the east by the Carboniferous-Permian Coastal Range Igneous Complex, Permian-Triassic Bowen Basin and Quaternary sediments, and to the west by Permian-Jurassic basins such as the Galilee, and Tertiary and younger cover sequences.

The Ravenswood Batholith intruded the basement Cape River Province and Seventy Mile Range Group in three phases:

1. Hornblende and/or biotite bearing I-type granitoids ranging from granite to lesser extent gabbro intruded during the early-to-mid Ordovician contemporaneously with the formation of elements of both the Cape River Province and Seventy Mile Range Group. Minor S-type, peraluminous granites of a similar age have also been identified in the Ravenswood Batholith;
2. The bulk of the batholith (>60%) formed during the development of the Mid-Silurian to mid-Devonian Pama Igneous Complex consisting of undeformed I-type hornblende-biotite bearing granites and granodiorite with lesser s type granitoids. These intrusions were coeval with a regional northeast-southwest compression (D4) and gold mineralisation at both Charters Towers and Hadleigh's Castle, west of Ravenswood; and
3. The late Carboniferous to early Permian Kennedy Igneous Association, a group of high K calc-alkaline intrusions with a diverse range of I, S and lesser A type magmas. Rocks of the Kennedy Igneous Association increase in abundance to the south of the Ravenswood Batholith and typically form localised, ring-fracture controlled stocks and/or trachytic plugs with little preserved deformation. *This intrusive phase is likely associated with gold mineralisation at Ravenswood.*

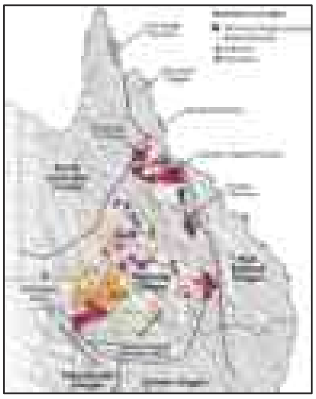


Figure 4. Location of Charters Towers Province

The region is characterised by east-west structures such as the Alex Hill Shear Zone, a 2-5km wide east-west shear zone extending over 100km across the northern edge of the Ravenswood Batholith (Figure 5) and the Mosgardies Shear Zone, a less continuous east-west mylonite zone extending from Ravenswood some 30km west to the Rochford area. The regional structural geology is considered to have formed in seven recognisable events defined as D1 to D7 (Kruezer 2005). Across the district, gold mineralisation is associated with D5 (Charters Towers) and D7 (Ravenswood). The seven deformation events include:

D1: Development of poorly preserved SE striking foliations in the Cape River and Charters Towers Metamorphics as a result of NE-SW compression.

D2: NW striking platy foliations formed during crustal extension and deposition of the Seventy Mile Range Group, synchronous with intrusion of some Ordovician Granitoids.

D3: E-W trending transcurrent shear zones developed as transfer faults or lateral ramps related to eastward progressing accretion (e.g. Alex Hill Shear Zone). Localised N-S compression related to the intrusion of Ordovician – Silurian granitoids into E-W shear zones.

D4: Development of NW-striking structures with both steep-pitching lineations and transcurrent fabrics (e.g. Burdekin River Lineament) as a result of NE-SW compression. Synchronous intrusion of Silurian-Devonian plutons into active transcurrent faults.

D5: Middle Devonian NE-SW compression concurrent with hydrothermal alteration and gold mineralisation at Charters Towers and Hadleigh's Castle.

D6: NW-SE compression producing sinistral movement on the Jessop Ck Fault and dextral movement on the Plumwood-Connolly Fault.

D7: Carboniferous E-W to NW-SE compression concurrent with rhyolitic magmatism, and alteration-gold mineralisation at Ravenswood and Mt Wright.

Figure 5. Lithostratigraphy of the Ravenswood District. (AUSGIN Geoscience Portal).

In summary, the regional geology suggests that the Ravenswood gold mineralisation formed during D7 deformation associated with the late Carboniferous to early Permian Kennedy Igneous Association. The regional structural setting at the time of mineralisation included east-west to northwest-southeast compression with a likely corresponding north-south to northeast-southwest dilation.

3.2 Deposit Geology and Structure

The Sarsfield-Nolans mineralisation is hosted by the Jessop Creek Tonalite[3], a variable light grey phaneritic to weakly hornblende-phyric medium to coarse grained tonalite. In the Sarsfield area the Jessop Creek Tonalite comprises diorite, quartz diorite, microdiorite and minor gabbro. Boundaries between these units vary from sharp to indistinct and often show complex relationships, including stoping, xenoltihs and irregular dykes. The Jessop Creek Tonalite displays variable degrees of alteration with primary biotite weakly to moderately altered to chlorite and epidote while hornblende is only weakly altered to chlorite in most cases. Alteration is concentrated along grain margins and particularly cleavage plans of biotite. No association between the host lithology and gold mineralisation has been established other than it is a competent host that was amenable to the development of several styles of quartz-sulphide-veins.

[3] Tonalite – A granitoid (a coarse grained igneous rock with <90% mafics; felsic minerals are composed mostly of quartz (20-60%), Kspar (alkali-feldspar) and plagioclase), where plagioclase is >90% of the total feldspar on the QAPF diagram (quartz - alkali feldspar – plagioclase feldspar – feldspathoids or foids)

The local structural geology is complex. The dominant structure is the Buck Reef Fault (BRF), a northeast trending, vertical zone within the Jessops Creek Tonalite with a strike extent of greater than 3km. The BRF has strong sub-horizontal lineations suggesting a dominantly strike-slip movement. Several authors (e.g., Switzer 2000, Laing 2005, Cowan and Davis 2017) note that the BRF pre-dates gold mineralisation at Ravenswood and has acted as a partial locus for mineralisation; in particular where it is intersected by cross-cutting low angle structures. Two other structures, the A4 Fault and the Keel, and a dominant mineralised trend are recognised in the Sarsfield region (Figure 6).

Figure 6. BRF (red) A4 (orange) and Keel (yellow) Structures.

Historical records, interpretations from ~1998 and descriptions in several reports (e.g., Collett et al., 1998, Lisowiec and Morrison, 2017 and Switzer, 2000) describe the Sarsfield-Nolans mineralisation as a complex vein network loosely controlled by reactivation of earlier structures (e.g., BRF, A4 Fault). Switzer provides a good summary of the styles of mineralisation and vein orientations, noting that the Sarsfield mineralisation is 'identical' to the Nolans region but more dispersed and lower grade. The major mineralised components in the Sarsfield area in Switzer's report include:

1. The Keel structure, a southwest dipping shear zone up to 36m thick;
2. Northeast dipping veins orthogonal to the Keel; and
3. An along-strike continuation of the Nolans mineralisation.

The Nolans mineralisation is confined to a narrow northwest trending zone hosting a conjugate vein set. Intersections of the two vein directions plunge to the northwest in a northly dipping package.

The historical descriptions and diagrams from Collett et al., (1998), Lisowiec and Morrison, (2017) and some hand-drawn sections dated 1998 were used to guide the geological interpretation at Sarsfield. Orientations of the major/minor variogram and search axes were aligned with the described mineralisation trends where practical. Figure 7 illustrates an overlay of the 2020 estimate interpretation with a diagrammatic section from Collett et al (1998). The orientations of the search ellipses used for the stockwork mineralisation are shown to demonstrate their alignment with the conjugate vein set. A second section (13450m E) comparing the hand-draw section to the current interpretation and modelled grade trends is presented in Figure 8.

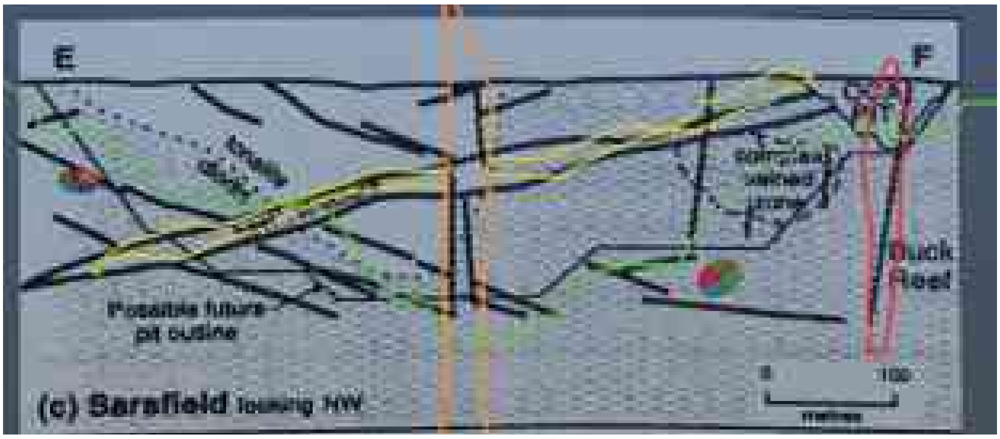


Figure 7. 2020 Interpretation superimposed on 1998 diagrammatic section (Collett et al).

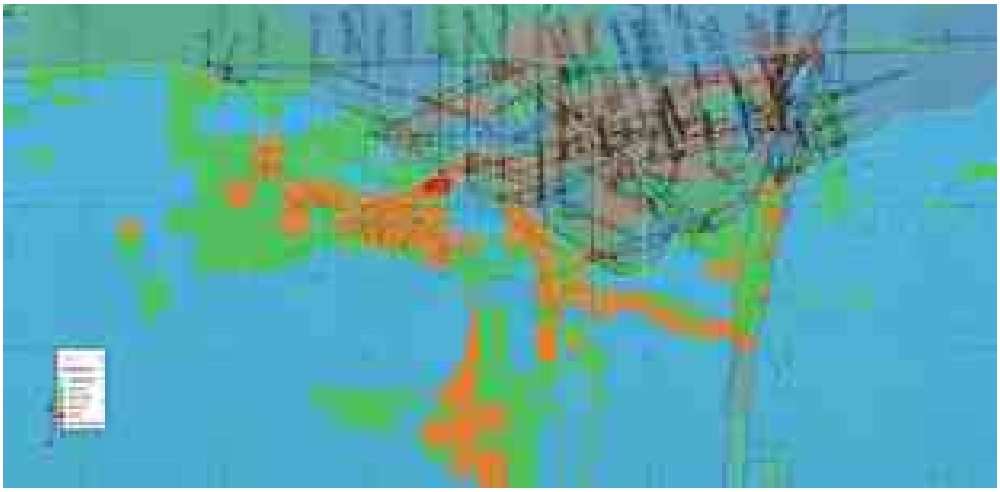


Figure 8. 2020 interpretation and estimated grades superimposed on hand-draw section circa 1998.

3.3 Resource Estimation Implications

The geology of the Sarsfield-Nolans mineralisation has direct implications for resource estimation. The host tonalite is relatively featureless and, while there are some recognised zones with higher density veining, much of the mineralisation is sporadic and discontinuous. Adding to the complexity, gold occurs in both vein directions of a conjugate set in an overlapping pattern. In some cases, it appears the intersections are preferentially mineralised whereas in other cases elevated gold grades occur in isolated portions of vein. This is described in Davis and Cowan (2017;Figure 9).

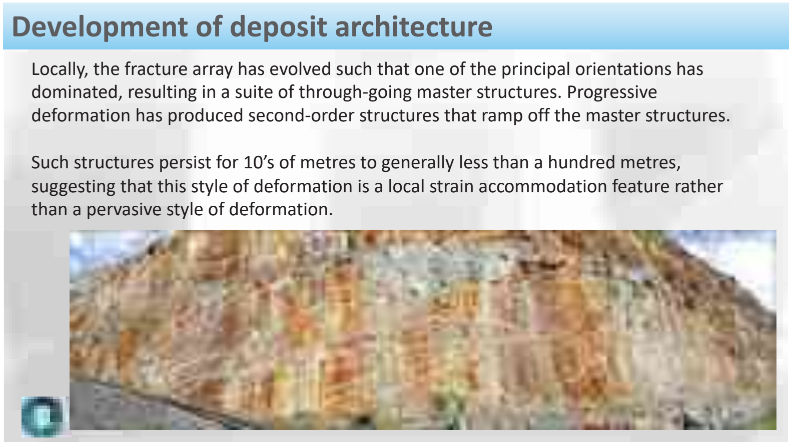


Figure 9. Fracture array architecture I (After Davis and Cowan 2017).

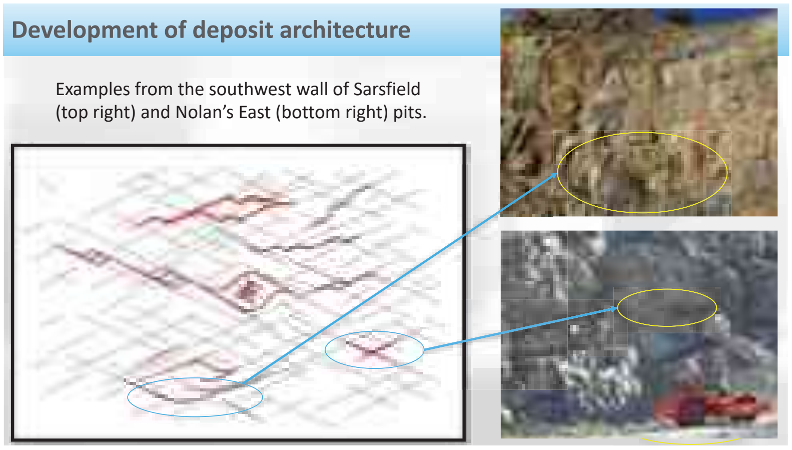


Figure 10. Fracture array architecture II (After Davis and Cowan 2017).

The complex vein geometry is attributed to a 3D orthorhombic system (Davis and Cowan, 2017, Switzer, 2000). In an orthorhombic system, there are four discontinuity directions to accommodate the 3D strain. Polymodal shearing results in fractures oblique to the principal stress axes with multiple fluid pathways (Figure 11). This pattern precludes a simple intersection lineation as the dominant control on mineralisation orientation.

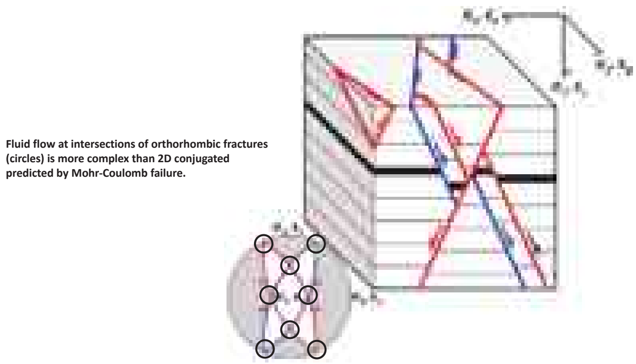


Figure 11. Orthorhombic faulting (After McCormack and McClay, 2018).

Observations by Davis and Cowan illustrate the fluid flow complexity in the Sarsfield-Nolans system. Mineralisation is not developed consistently on similarly oriented structures (Figure 12) and adjacent fractures can equally mineralised or unmineralised. The fracture system is permeable but the highly variable interconnections and orientations are a poor focus for deposition. Consequently, mineralisation is sporadic and the observed continuity of mineralised veins is low.

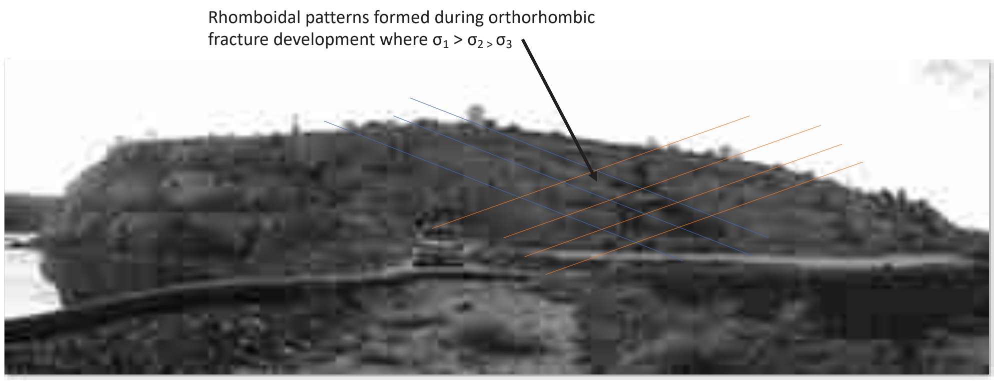


Figure 12. Rhomboidal patterns at Sarsfield pit. (After Davis and Cowan (2017)).

During past phases of mining at Sarsfield, workers have made loose correlations between vein density and average gold grade (e.g. Collett, 1998 as reported in Switzer, 2000). Based on the described sporadic local grade distribution, this relationship is logical if not necessarily useful. Where there is a high vein density there is a highly probability that one or more of the fluid paths could be mineralised. Raw vein density may, however, also display false negatives and false positives. This was apparent in the 'grade density' modelling attempted by SD2 (Section

1.5). While the average number of composites greater than 0.2g/t Au increased and showed a population break when plotted against increasing grade (Figure 13) the range of composite numbers varied widely and did not show a similar pattern.

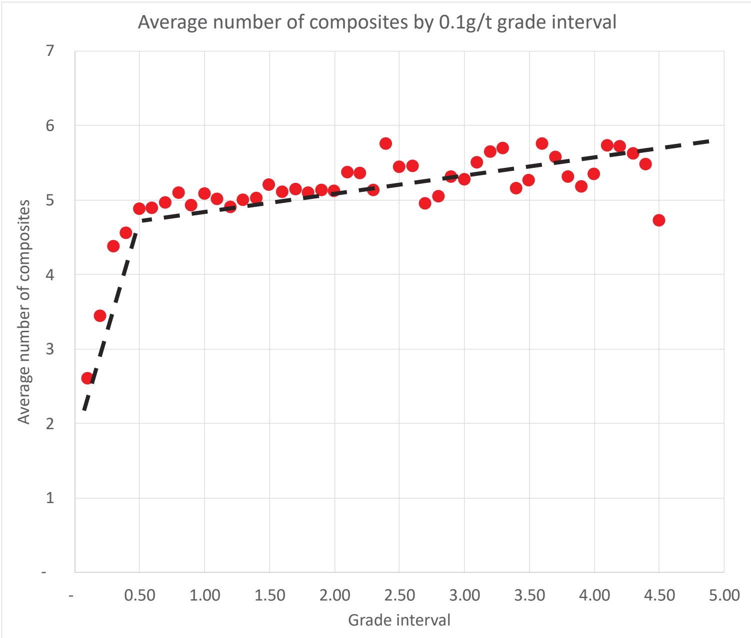


Figure 13. Grade-Density analysis (12.5 x 12.5 x 6.25 panels).

The anecdotal correlation between grade and vein intensity (or count) is promising and may be useful in future models. In the absence of quality vein intensity logging data however it was not possible to prove a meaningful relationship.

SD2's analysis and consideration of the geology and mineralisation was used to develop the Sarsfield-Nolans estimation strategy. For the major identifiable zones of mineralisation, estimation domains were developed using strongly controlled implicit modelling. These domains (Table 5) were estimated using ordinary kriging. The remaining mineralisation (domains 1006, 1007) were estimated using multiple indicator kriging (MIK) with a e-type grade evaluation. The variogram and search anisotropies were determined through comprehensive 3D analysis in 10° increments for a complete 3D fan. The resulting orientations are well aligned with the reported mineralisation trends.

The relationships and locations of the estimation domains are shown in Figure 14. All domains are separate volumes with lower domain numbers taking precedent over higher numbers.

Object	Domain Number	Estimation Method
Buck Reef Fault	1001	Ordinary Kriging

A4 Fault	1002	Ordinary Kriging
Keel Structure	1003	Ordinary Kriging
Nolans Main Zone	1004	Ordinary Kriging
Nolans Background	1005	Ordinary Kriging
A4 – to – Buck Reef Fault	1006	Multiple Indicator Kriging
Southeast of A4 Fault	1007	Multiple Indicator Kriging
West of Buck Reef Fault	1008	Ordinary Kriging
Background / Waste	9999	Ordinary Kriging

Table 5. Domains and estimation method.

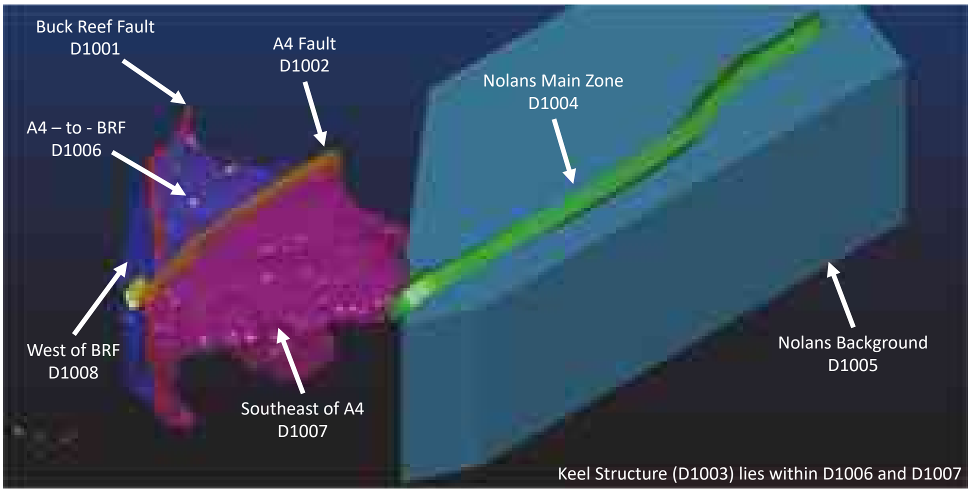


Figure 14. Sarsfield-Nolans estimation domains.

4. Resource Estimation Data

4.1 Data Provided

SD2 was provided with a comprehensive data set including:

- The geology drill hole database in MS Access format dated 02 June 2020. This database includes tables for:
  - Hole collars;
  - Down hole surveys;
  - Gold assays;
  - Multielement assays;
  - Quality control sample results;
  - Structural logs;
  - Lithological logs; and
  - Geotechnical logs.

- The ‘as-mined’ topographic for the Ravenswood area including a lidar survey combined with historical end-of-pit mining surfaces;
- The back-filled open pit surface for Sarsfield (as at 16 June 2020);
- Wireframe solids for historical underground mining for all lodes at Nolans (stopes and development);
- A variety of reports including general geology descriptions, structural geology analyses and past mineral resource estimation reports;
- Miscellaneous data stored on archived CDs including partial grade control records; and
- Miscellaneous plans and sections from 1998 developed as part of the then active mining operations.

4.2 Database Assessment

The mineralisation is relatively well drilled in the first 100m below the as-mined topography and in the centre of the Sarsfield region corresponding to the A4 Fault. At depth, drill hole spacing decreases. The database includes 1,423 holes, of which 951 have some proportion remaining below the as-mined topography (Figure 15 and Figure 16). Approximately 53% of the 2m composites are below the surface.

The database includes a combination of blast holes (BH), reverse circulation (RC), diamond drill holes (DD) and diamond drill holes with a RC pre-collar (RCD) (Table 6). All data types were used in the 2020 estimate including the 43 blast holes (Figure 15) which have limited influence and affect the upper regions of the Keel structure and the background estimates only.

Figure 15. Sarsfield drill hole coverage (all assayed intervals).

Figure 16. Sarsfield drill hole coverage (hole remaining below topographic surface).

Hole Type	Number of Holes
BH	43
DD	159
RC	1120
RCD	101

Table 6. Drill hole types.

The database is dominated by holes drilled prior to 2001 (Figure 17), reflecting the date of the last major mining campaign. After 2001, drilling focused on deeper zones, below the existing open pit and these more recent holes inform the majority of the remaining estimated volume.

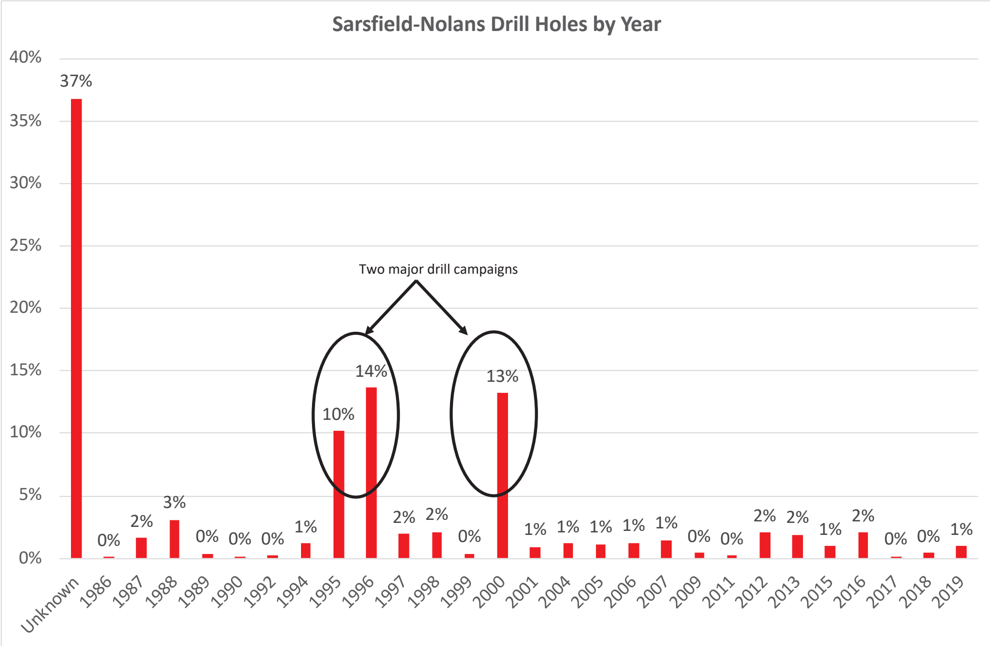


Figure 17. Percentage of drill holes by year.

SD2 reviewed the provided MS Access database and discussed data management practices with the on-site team. Both the digital and hard-copy records were examined, and it was clear that significant time and effort had been spent on data quality.

A suite of routine checks was completed to identify data errors. Checks included:

- Missing data (collar, survey, assay, lithology);
- Duplicate holes, collars, surveys and samples;
- Sample from/to values beyond the recorded length of the hole;
- Invalid data including out-of-range coordinates, negative grades; and
- Spurious survey deviations based on angular rate of change tolerances.

No errors were found by these checks and visual examination of the desurveyed drill hole data supported SD2's opinion of the high quality of the geology database.

4.2.1 Treatment of absent data

Approximately 20% of the drill hole data in the provided Access database have negative gold grade values. These values represent intersections that were not sent for analysis (<1%) and samples that were below detection limit. For the purpose of this estimate SD2 assigned a very low grade (0.005 g/t) to these intervals.

4.3 Quality Management

4.3.1 Pre-2004 Drilling

Information regarding sample quality for holes drilled before 2004 is limited. SD2 discovered some archived data from 1996, 1997 and 1998 (Appendix E); however, the information is incomplete and there is no formal performance reporting from the results. The data appears to be a combination of 'standards' submitted to the ALS laboratory in Townsville and to the on-site lab. While the data is incomplete and there is some uncertainty, it appears to indicate:

- A negative grade bias of between 5% and 10% from the on-site lab. This may be due to the analytical process (Leachwell);
- A wide dispersion (low precision) of results from the on-site lab;
- Good performance from ALS with no apparent bias;
- Duplicate performance is reasonable (Figure 18) for all size fractions presented (pulp, coarse, chips). The original-to-repeat correlation for all duplicates is 0.93. For samples less than 2g/t this reduces to 0.91. Eighteen percent of all duplicates have a precision outside of a +/-10% relative difference from the original assay;
- The original-repeat misallocation at a 0.4g/t threshold is 6% for pulp duplicates (Figure 19). All other duplicate types exhibit a lower misallocation percentage.

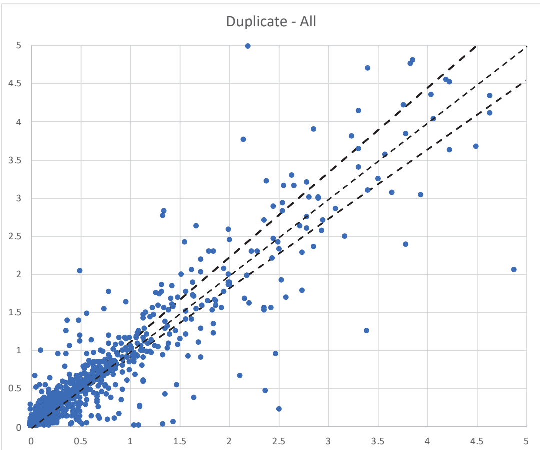


Figure 18. All duplicates (~1997-98 Data).

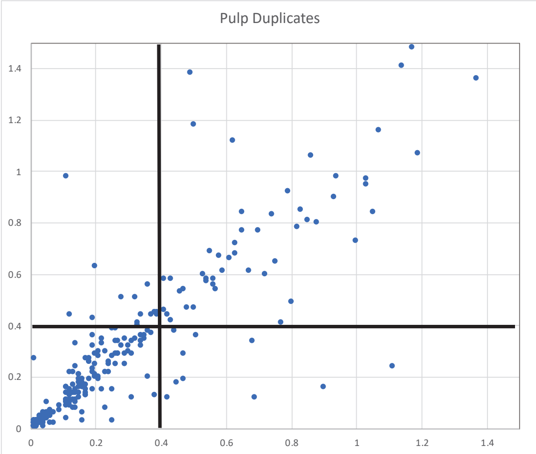


Figure 19. Misallocations for original-repeat samples (pulp duplicates).

SD2 reviewed and compared the pre-2004 and post-2004 drill hole data as part of the assessment of the quality of the early drilling programs (Figure 20). The domain-by-domain statistics (Table 7), and QQ-plots (Figure 21) are inconclusive; however for the main MIK domains (1006 and 1007) the two data sets are reasonable comparable. Differences across all domains may reflect the spatial positioning of the drilling (pre-2004 is shallower).

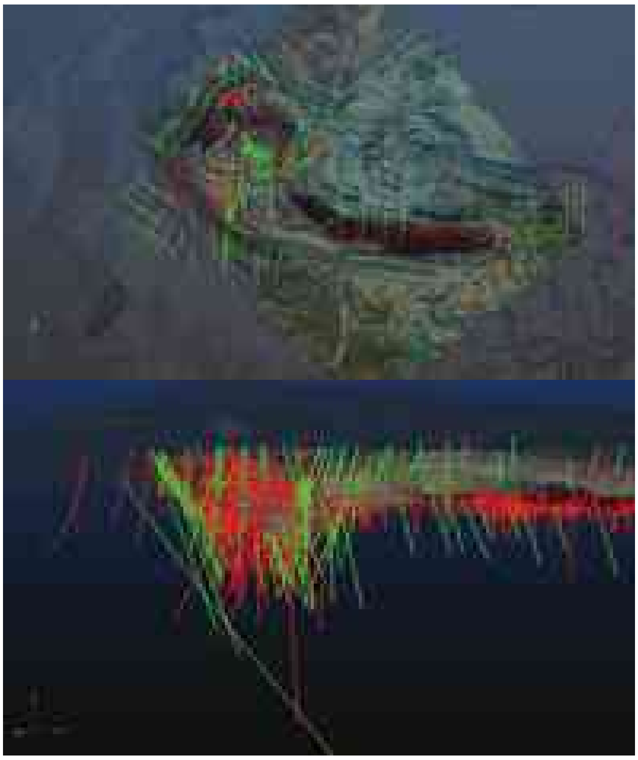


Figure 20. Location of pre-2004 (red) and post-2004 (green) drill holes.

Domain	Average		CV		25th Percentile		50th Percentile		75th Percentile		95th Percentile	
	Post-2004	Pre-2004	Post-2004	Pre-2004	Post-2004	Pre-2004	Post-2004	Pre-2004	Post-2004	Pre-2004	Post-2004	Pre-2004
1001	0.78	1.54	2.27	3.16	0.06	0.09	0.24	0.29	0.65	1.03	2.87	6.58
1002	0.99	0.91	2.54	2.81	0.05	0.06	0.18	0.23	0.76	0.71	4.69	3.72
1003	1.30	1.43	3.21	2.73	0.02	0.16	0.20	0.48	0.96	1.18	4.56	5.61
1004	0.40	0.86	3.20	2.58	0.01	0.05	0.05	0.18	0.28	0.74	1.85	3.82
1005	0.07	0.10	5.39	5.91	0.01	0.01	0.01	0.01	0.03	0.03	0.26	0.34
1006	0.59	0.57	4.07	3.65	0.04	0.04	0.11	0.13	0.37	0.43	2.34	2.27
1007	0.44	0.54	4.38	10.01	0.01	0.02	0.05	0.08	0.21	0.33	1.81	2.15
1008	0.25	0.53	3.08	3.08	0.01	0.03	0.04	0.09	0.17	0.34	1.06	2.22

Table 7. Domain-by-domain sample statistics pre and post 2004 (remaining samples only).

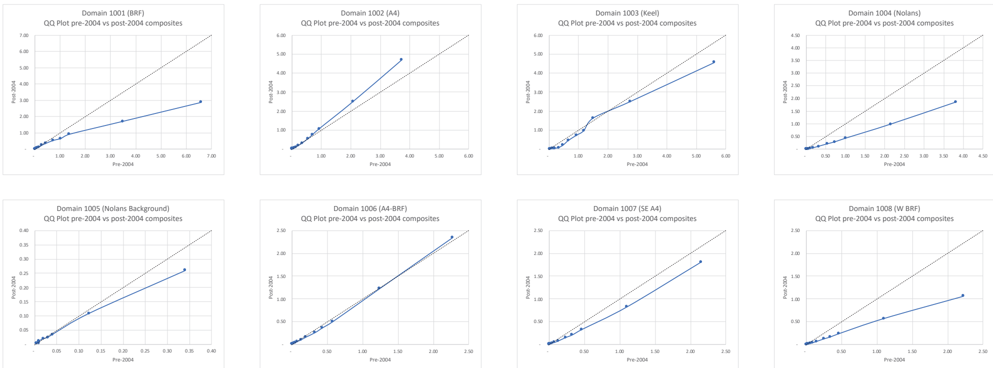


Figure 21. Domain QQ Plots. Pre and Post 2004 drill programs.

SD2 attempted to compare pre- and post-2004 drilling where samples lie within close proximity (5m). The drill patterns severely restrict the numbers of closely spaced samples (280 pre- and 302 post-2004; Figure 22). The data is limited to two locations; one in the centre of the Sarsfield pit and one on the southern edge of the Sarsfield pit. The relatively low number of samples and data clustering reduces the value of this spatial analysis; however, a QQ-Plot of the two data sets shows good correlation up to the 90th percentile (Figure 23).

Figure 22. Location of pre- and post-2004 samples within a 5m radius (red and green).

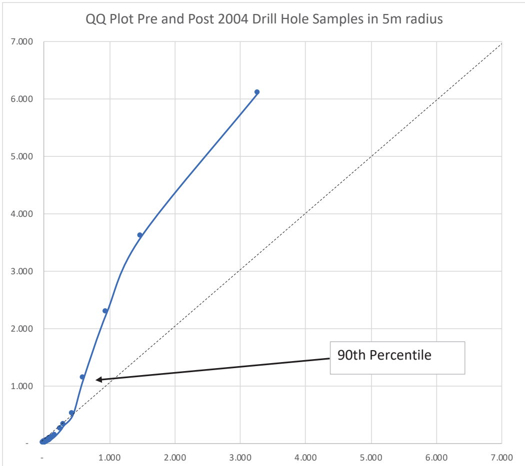


Figure 23. QQ-Plot pre- and post-2004 data within 5m radius.

In SD2's opinion the pre-2004 data is reasonable for resource estimation. The available quality data and comparisons of the pre- and post-2004 drilling do not indicate a bias, and the vast majority of the pre-2004 drilling lies above the as-mined topography, limiting its influence on the remaining resource. As a check on this assumption, a sensitivity estimate was completed using only post-2004 drill hole date. In volumes estimated in the first search pass in both the complete data set and the post-2004 data only, there was a 5% increase in metal above a zero cut-off and also above a 0.4 g/t Au cut-off (the post-2004 estimate higher).

The biggest difference between estimating with and without the drill holes from before 2004 is in the volume estimated. This is in line with the differing spatial coverage of the two data sets. Removing the early data reduces the volume estimated as fewer blocks meet the minimum number of sample criteria. Globally, the sensitivity estimate contains 13% fewer ounces than the final estimate using the complete data set.

Based on the impact of including the pre-2004 data and on the historical evidence indicating the potential for this data to be biased low (if any bias exists), the data has been included in the final estimate. The quality of the drill hole data was considered in the resource classification (Section 7) and the risk associated with the relatively low amount of quality control data for the pre-2004 data is captured in the classification.

4.3.2 Post-2004 Drilling

The quality control results for the post-2004 data are well documented in a report prepared by Resolute Mining Limited in 2013 (Resolute, 2013; Appendix E). This report presents the results from submissions of 11 certified reference materials (CRM), blanks, quartz flushes, pulverisation checks, duplicates collected from field, coarse crush and lab and umpire assaying comparing ALS Townsville to SGS Townsville.

The quality control samples consist of results from 647 batches testing the on-site lab (43) and ALS Townsville (604). The data was analysed using conventional time-series control chart techniques, box-and-whisker plots., scatterplots and HARD[4] precision plots.

The results indicate a well-managed sampling and assaying system. Resolute noted that most CRMs were within expected limits despite a high number of failures that were attributed to data entry errors. The total 'fail' rate for all CRMs in the Sarsfield-Nolans volume was 3.7% with the majority relating to pre-2004 data with unknown provenance. The more recent sampling showed much better performance with a 'fail' rate of 2.2%.

All nine of the CRMs submitted to ALS Townsville demonstrate a slight low bias (Table 8). The grade ranges from 0.05g/t Au to 4.36g/t Au was tested and the bias seems consistent across the spectrum. This minor bias is not considered material.

CRM	Number Submitted	Expected Value	Average Assay Grade	Variance
G300-9	120	1.53	1.51	99%
G302-6	39	0.99	0.98	99%
G310-7	70	1.01	0.99	98%
G910-8	79	0.63	0.61	97%
G998-4	120	4.36	4.36	100%
G999-2	40	0.63	0.60	95%
ST06/3317	43	1.10	1.06	96%

Table 8. CRM Summary.

Duplicate samples from field, coarse duplicate, pulp and lab tests indicate good sampling precision with the expected pattern of reducing precision at very low grades (<0.1 g/t Au). Precision improves as the sample top-size decreases. QQ Plots generally lie close to the X=Y

[4] HARD – half absolute pair difference

line. Analysis of the pair-by-pair relative difference shows that 69% of the data has a precision better than 25%. The greatest pair-by-pair differences occur above 1.2 g/t Au.

Umpire lab checks between ALDS Townsville and SGS Townsville show good repeatability with 87% of paired data having a relative difference less than 25%.

In SD'2 opinion the analysis completed by Resolute demonstrates that the drill hole samples are good quality and acceptable for use in resource estimation.

4.4 Collar and Down Hole Surveying

All data used for the Sarsfield-Nolans estimate was reported in the local A45 grid, a rotation of 30° clockwise from magnetic north. This grid also includes a datum height adjustment of -32.813m to the Australian Hight Datum.

There is limited information on the collar survey methods used for holes drilled prior to 2004. More recent hole collars were surveyed by the Ravenswood gold mine in-house survey team using Leica TPS1100 total station and optical techniques. SD2 reviewed hole collars against the survey topography and while most holes were situated on the topographic surface, 24 holes (Appendix L) were found to be approximately 32m above the surface. This corresponds to the vertical offset applied to the local mine grid and it appears the correct translation was not applied to this data. These holes are not within the boundaries of potentially economic mineralisation and have no impact on the 2020 estimate. Ravenswood were advised of this data error so it can be rectified for future work.

Where available, the down hole survey method is captured in the site's geology database. A large proportion of survey methods (for pre-2004 holes) were not recorded. The number of holes using each survey method are shown in Table 9. The survey technology shows incremental improvement over time from mechanical single-shot cameras to more modern gyro-based approaches.

Method	Number of Holes[5]
DeviFlex	3
Compass	3
NSGyro	4
PFMS	6
MEMSGyro	14
RTKGPS	35
ElectronicSS	87
MechanicalSS	170
ElectronicMS	243

[5] Note, some holes have more than one recorded survey method. RTKGPS refers to collar only.

NR	1280

Table 9. Down hole survey methods.

The down hole survey data as reviewed by Ravenswood and flagged as valid or invalid. SD2 completed further reviews based on rates of hole deviation and no errors were identified in the validated data.

Sample locations within drill holes were based on the Datamine Studio RM standard desurvey method. This approach calculates the XYZ centre point, bearing and dip for each interval based on spherical arcs. Survey measurements are treated as 3D unit vectors (i.e., they are *not* independent) and therefore sample intervals lie tangential to the unique arc defined by the survey data. After desurveying, SD2 examined the hole traces for data artefacts. Only one hole (SFD506) showed any measurable deviation in 3D. This is the deepest hole (1,371m) at Sarsfield. The hole commences at -53 -> 174 and deviates to -40 -> 183 over its length. Surveys were by north-seeking gyro at 5m intervals.

All drill hole traces were deemed valid.

4.5 Data Distribution and Spacing

Drilling at Sarsfield-Nolans is dominated by north-south holes dipping at 50-60° (Figure 24). There is a much smaller west-northwest population, primarily testing the BRF at depth. While these hole orientations are well placed to intersect the major stockwork and structural orientation, the dominance of north-south holes represents a strong orientation bias. If there are additional vein sets aligned semi-parallel to the drill direction, they will be under-represented in the estimate. SD2 note that there is no indication of north-south veins or structures in the pit walls and no record in the historical descriptions of Sarsfield.

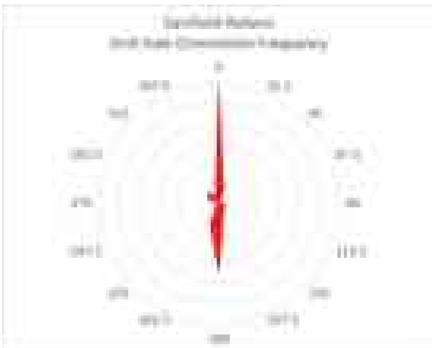


Figure 24. Drill hole orientation frequency.

The drill hole coverage is moderate to good. Approximately 47% of the data lies above the previously mined open pit. In the first 100m below the as-mined surface, the hole spacing is approximately 25m x 25m. As the depth below surface increases the drill hole spacing decreases. This decrease is particularly noticeable in the Nolans region (Figure 25 and Figure 26).

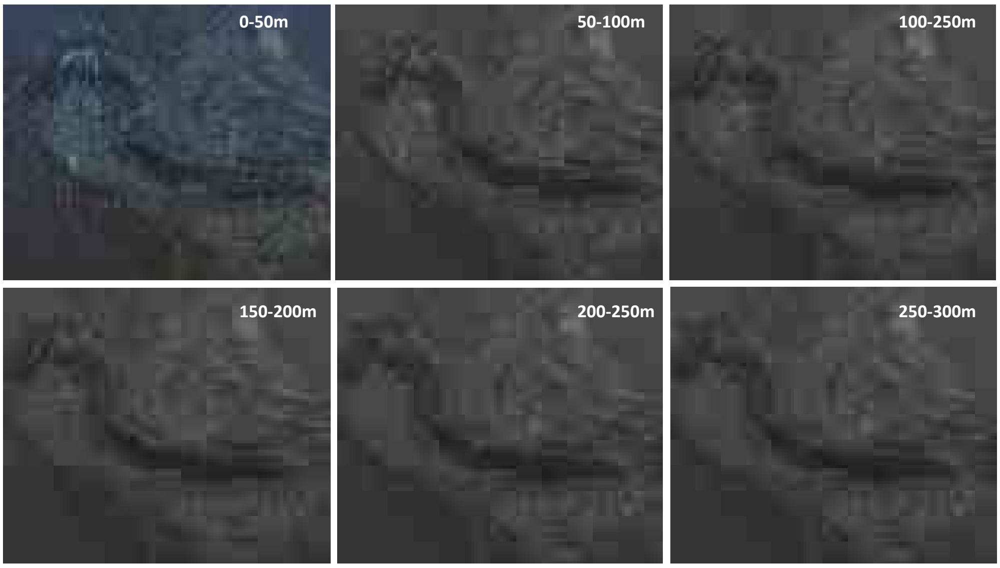


Figure 25. 50m slices below pit as mined surface.

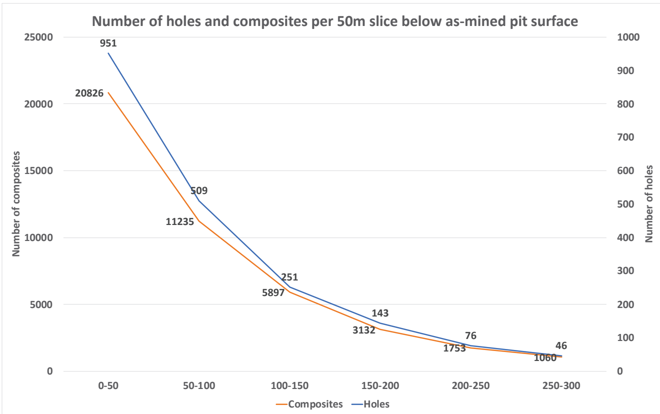


Figure 26. Number of holes and composites in 50m steps below as mined surface.

4.6 Bulk Density

The bulk density used for Sarsfield is based on 1,957 Archimedes measurements collected from drilling supervised by Resolute Mining Limited between 2014 and 2018. While this drill program focused on the nearby Buck Reef West deposit, the host lithology and alteration at Sarsfield-Nolans is the same. SD2 examined both fresh and oxide densities (Table 10). The measured bulk density values are supported by tonnage reconciliations from production at BRW and Sarsfield.

	Number	Average	Median	Minimum	Maximum	CV
Oxide	20	2.75	2.73	2.63	2.84	0.020
Fresh	1937	2.80	2.78	1.79	4.13	0.038

Table 10. Basic statistics for bulk density measurements.

Based on the sample data and reconciliation, SD2 assigned a bulk density of 2.78 g/cm^3 to fresh material. There are only a limited number of oxide samples in the data set; therefore, SD2 assigned the historical bulk density to the remaining oxide material (2.4 g/cm^3.).

5. Resource Estimation

5.1 Interpretation and Domaining

As described in Section 3.2, the Sarsfield-Nolans mineralisation a structurally controlled zone of variable developed stockwork veins. There are three major fault zones/corridors in the Sarsfield region and a distinct vein corridor at Nolans. The remaining mineralisation (Sarsfield) is separated into three regions, each bound by the Sarsfield structures.

The stockwork mineralisation is controlled by an orthorhombic fracture system and fluid pathways during mineralisation were permeable but ineffective traps. This resulted in discontinuous and patchy mineralisation. Anecdotally, gold grades are correlated with vein density; however, the historical logging does not adequately capture vein density suitable for resource estimation.

Multiple interpretations approaches were trialled for Sarsfield-Nolans. The Nolans mineralisation is relatively simple and well constrained in an east-west corridor, therefore the alternative interpretations showed little variability. At Sarsfield, however, the mineralisation is less focused. While the interpretation of the three major structures is well constrained, the mineralisation between these structures shows less geometric coherence. There were three different interpretation scenarios developed:

1. A manual interpretation completed by M. Lindsay from Ravenswood. This was a traditional cross-sectional interpretation that defined the fault structures (BRF and A4) but not the Keel zone. Between the major faults, Lindsay interpreted multiple flat-lying zones as the focus of mineralisation. Statistical investigations of these zones failed to identify any material difference in the populations within and outside the wireframes and the interpretation was discarded;

2. SD2 attempted to develop a sensible domain interpretation using a proxy for vein density. As described in Section 3.3, the number of composites above 0.2 g/t Au within a specified volume (12.5x12.5x6.25 and 25x25x12.5 were tested) was determined. This 'grade-density' variable was then modelled using implicit modelling tools (Figure 27). The results initially looked reasonable and statistical and variogram analysis was attempted using these volumes. This highlighted that the developed volumes and geometries were strongly controlled by the drill hole locations – effectively the model 'hugged' the hole traces where those traces had a high number of grades above 0.2 g/t.

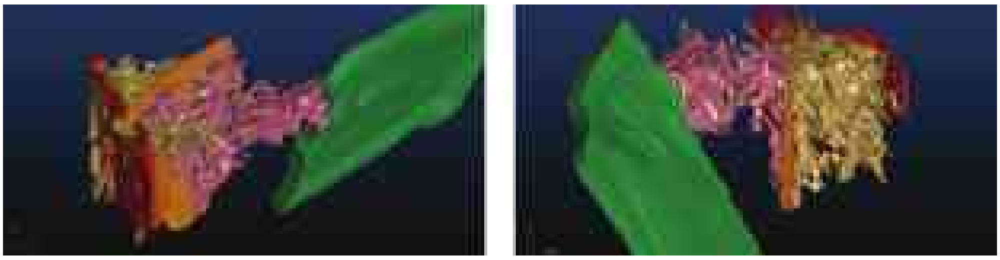


Figure 27. Example of 'grade-density' domain development.

This grade-density (or preferably vein-density) approach still holds some merit and it is worthwhile continuing investigations into developing domains based on some geological feature. This will likely require re-logging (or checking photos), additional pit mapping and additional drilling; and

3. The final (and accepted) domain interpretation adopted a hybrid approach. Domains for Nolans, BRF, A4 and Keel were developed using structurally controlled implicit modelling. These domains closely match those of Lindsay (where applicable) and are similar to domains found in historical reports (e.g. Keel). Domains for the remainder of Sarsfield stockwork mineralisation were developed as follows:
    a. All drill holes were flagged with an indicator based on the grade-density variable. The zone of mineralisation was flagged between the first occurrence and the last occurrence of a grade-density above 4. This created a broadly continuous zone where veining occurs and excluded zones with little or no veining;
    b. The indicator was estimated; and
    c. The estimate was iso-surfaced at a 0.2 probability threshold. This corresponded to a distinct boundary in the indictor estimate, partitioning the volume with some stockwork mineralisation from volumes where veining is totally absent.
    d. The iso-surface was post-processed to remove small isolated volumes.

The final domain model includes eight wireframe solids (Table 11 and Figure 28, Figure 29) and a background/waste zone.

Object	Domain Number	Estimation Method
Buck Reef Fault	1001	Ordinary Kriging
A4 Fault	1002	Ordinary Kriging
Keel Structure	1003	Ordinary Kriging
Nolans Main Zone	1004	Ordinary Kriging
Nolans Background	1005	Ordinary Kriging
A4 – to – Buck Reef Fault	1006	Multiple Indicator Kriging
Southeast of A4 Fault	1007	Multiple Indicator Kriging
West of Buck Reef Fault	1008	Ordinary Kriging
Background / Waste	9999	Ordinary Kriging

Table 11. List of domains and estimation method.

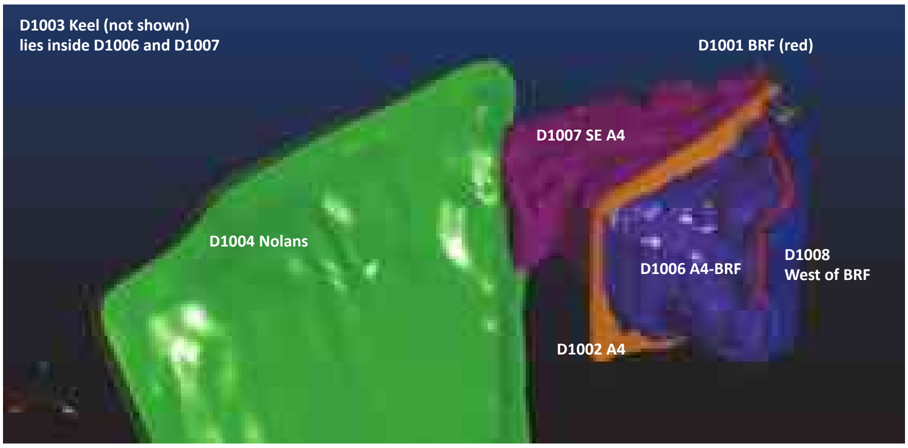


Figure 28. Sarsfield-Nolans domains.

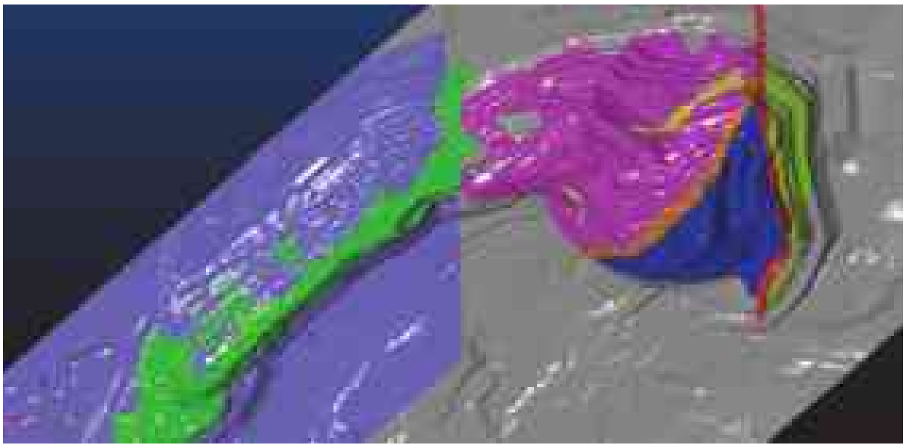


Figure 29. Sarsfield-Nolans domains (exposed on as-mined surface).

5.2 Compositing

There are two distinct sample length populations at Sarsfield-Nolans (Figure 30). Drilling completed prior to 2004 was largely sampled on 2m intervals, whereas drilling after 2004 was sampled on 1m intervals. There is no relationship between sample length and grade (Figure 31). Based on this analysis and consideration of the style of mineralisation, a composite length of 2.0m was selected.

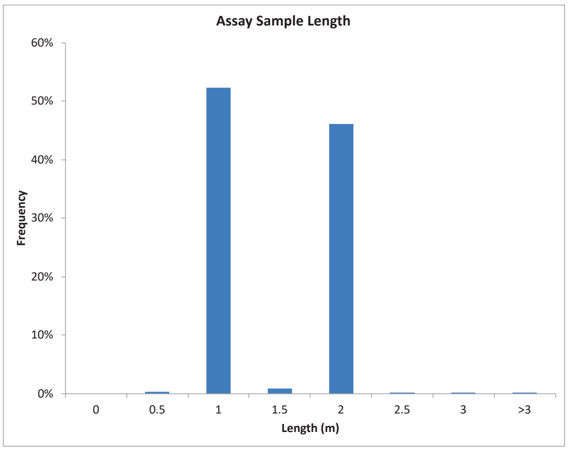


Figure 30. Assay sample length frequency distribution.

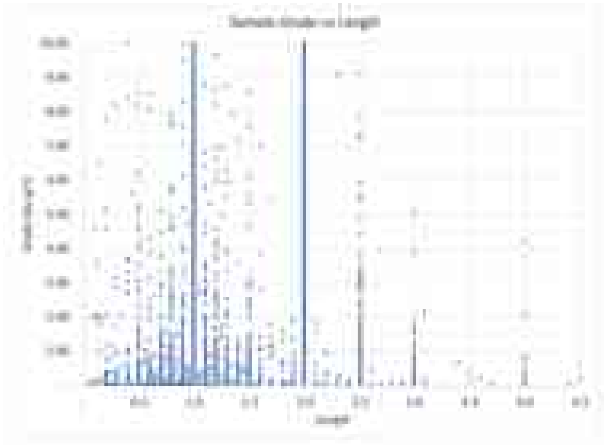


Figure 31. Sample length vs. gold grade.

Samples were flagged by domain prior to compositing. The minimum composite length allowed was 0.1m. Compositing used Datamine's @mode=1 option which retains all sample data by adjusting the composite length to values approaching the designated metreage. In practice this approach resulted in 70% of the composites equalling exactly 2.0m and 95% of the composites having a length of 2.0m _/- 0.05m.

5.3 Grade Caps

Examination of the univariate statistics of the composited data shows that the grade distribution in every domain has a strong positive skew. The distributions include some samples that appeared to be outliers or inconsistent with the distribution of the majority of composites. SD2 examined the rate of change of the CV as the highest-grade samples were removed from the domain data sets. Where the rate of change accelerates rapidly it is likely that it is affected by outlier samples.

Based on the rate-of-change analyses, grade caps[6] were selected for each domain. The grade cap applied, equivalent percentile and impact of the cap on the mean grade and CV for each domain is presented in Table 12. A total of 899 composite grades were capped, equating to 1.21% of the data. On a domain-by-domain basis the grade cap ranged from the 97.7th percentile (Domain 1002; 8.0 g/t Au) to the 99.41th percentile (Domain 1007; 9.5 g/t Au). The average cap percentile across all domains was the 98.7th percentile. The grade caps reduced the CVs of all domains estimated using ordinary kriging to less than 3.0 except for the Nolans background and global background. Grade caps presented for Domains 1006 and 1007 are presented for information only. These domains were estimated using MIK and grade capping is not relevant.

[6] Grade cap refers to capping the grade for composites within a domain to a maximum value. The composites are kept as members of the domain during estimation.

Domain	Number of Composites	Minimum	Maximum	Average	Std Dev	CV	Cap Value	Number Capped	Cap Percentile	Average After Cap	CV After Cap	Avg Grade Change	CV Change
1001	1629	-	93.00	1.43	4.647	3.249	15.00	27	98.34%	1.20	2.18	83.60%	67.20%
1002	3275	-	41.40	0.95	2.672	2.814	8.00	75	97.71%	0.79	2.47	83.30%	87.90%
1003	1219	-	57.00	1.42	3.939	2.778	17.00	11	99.12%	1.25	1.77	88.30%	63.80%
1004	10085	-	58.40	0.79	2.137	2.696	6.50	194	98.07%	0.69	2.57	86.40%	95.30%
1005	5841	-	21.31	0.08	0.462	5.975	1.10	72	98.78%	0.06	4.79	71.50%	80.10%
1006	17850	-	107.00	0.55	2.130	3.838	8.00	147	99.17%	0.48	4.36	87.00%	113.50%
1007	19248	-	681.00	0.52	5.267	10.194	9.50	113	99.41%	0.42	5.75	82.10%	56.40%
1008	2130	-	23.61	0.40	1.315	3.257	4.00	42	98.05%	0.33	2.48	80.80%	76.30%
9999	13130	-	23.00	0.10	0.590	6.018	1.00	218	99.65%	0.06	4.68	66.10%	77.70%

Table 12. Grade cap values and statistics.

The impact of the grade caps applied was examined by running an uncapped grade estimate in parallel (Table 13). The difference between capped and uncapped grade estimates is 16% which is in line with the change in average capped composite grade. As the cut-off grade increases the grade cap affects both tonnes above cut-off and grade above cut-off. The cumulative effect is approximately 20% decrease in ounces for cut-offs around the economic range.

Cut-Off	Tonnes	Ounces
0.00	0%	-16%
0.10	-7%	-17%
0.20	-14%	-20%
0.30	-17%	-21%
0.40	-14%	-20%
0.50	-13%	-20%

Table 13. Change in tonnes and ounces with grade caps (OK domains only).

No spatial restriction was applied. All capped composites were allowed to inform all blocks during estimation.

5.4 Indicator Thresholds and Grades

Multiple indicator kriging involves setting a series of binary indicators above increasing grade thresholds. These grade thresholds characterise the frequency distribution of the sample population. Selection of the grade thresholds can impact on the quality of the estimate, particularly in strongly skewed distributions. Traditionally MIK grade thresholds are set at each decile with increase discretisation in the higher-grade ranges. This approach often results in over representation of the structural influence of very low grades in the presence of skewed distributions. For example, at Sarsfield, the 70th percentile for the two MIK domains is less than 0.3 g/t Au. Under a decile-based threshold approach the variograms for the first seven indicators would be identical and estimating these grade ranges would be of little value when defining the grade-tonnage curve.

As an alternative SD2 adopted indicator grade thresholds based on evenly distributing the metal (as defined by domain composites) in the high-grade ranges and then evenly distributing the number of composites in the lower grade ranges. This approach places equal importance across the domain grade range. Twelve indicator grade thresholds were defined for each MIK domain (Table 14). The highest-grade thresholds each contain 15% of the metal. The lower-grade thresholds each contain 15% of the remaining composites. The grade for each indicator bin was set to the average of the composites within the bin. The grade for the top bin was set as the average above the last threshold for Domain 1006 and the median above that last threshold for Domain 1007[7].

The indicator thresholds, equivalent population percentile and the grade for the indicator bins are shown in Table 14.

Domain 1006 (A4-to-BRF)			Domain 1007 (SE of A4)		
Indicator Threshold	**Percentile**	**Bin Grade**	**Indicator Threshold**	**Percentile**	**Bin Grade**
0.00		0.03	0.00		0.02
0.08	43.4%	0.08	0.05	47.3%	0.06
0.09	46.2%	0.10	0.07	52.9%	0.08
0.11	50.9%	0.13	0.09	57.0%	0.10
0.14	55.7%	0.16	0.12	61.9%	0.14
0.18	60.2%	0.20	0.16	66.8%	0.19
0.22	65.2%	0.25	0.23	72.4%	0.27
0.28	69.9%	0.49	0.33	77.4%	0.58
0.84	86.6%	1.20	1.03	90.5%	1.50
1.74	93.4%	2.40	2.25	95.5%	3.00
3.34	96.9%	4.50	4.38	98.1%	6.20
6.5	98.8%	8.80	8.80	99.3%	14.00
13.27	99.7%	27.00	24.20	99.9%	30.00

Table 14. Indicator thresholds and bin grades.

5.5 Statistical and Geostatistical Analysis

In conjunction with the grade cap analysis, basic statistics were calculated for the domained and composited data (Table 12). This was followed by spatial statistical analysis and modelling.

[7] The median was selected to reduce the impact of seven extremely high-grade composites that are considered true outliers.

5.5.1 Ordinary Kriged Domains

Experimental variograms were calculated for raw and Gaussian transformed composites. This included both downhole and directional variograms. For Domains 1001, 1002, 1003 and 1004 the variogram was first aligned to the plan of the structure. For Domains 1005 and 1008 the maximum direction of continuity was determined from a full 3D analysis.

Variograms were modelled in Gaussian space and then back-transformed. The back-transformed models were compared to the experimental variogram in true space and minor adjustments were made to the nugget based on downhole variography.

The variograms ranged from excellent to poorly structured. A full set of the Gaussian (Normal Scores) variogram models is given in Appendix B. Each variogram is presented with a corresponding set of 3-dimensional images showing the domain and the variogram model overlaid as an ellipse. This approach ensures the axial rotations defined in the model are logical with respect to the orientation of the domain.

All variograms were modelled using spherical models. Models have been normalised with the total modelled variance equal to 1.0. The variogram models are presented in Table 15. Nugget effects are generally moderate; however, the majority of the models exhibit a steep slope near the origin, commonly reaching >65% of the total variance within 10m. This is in line with expectations based on the geology of the mineralisation and the sporadic gold distribution.

		Rotations						Variogram Structures			Ranges - Structure 1			Ranges - Structure 2		
Domain	Description	Axis 1	Axis 2	Axis 3	Angle 1	Angle 2	Angle 3	C0(Nugget)	C1 (sph)	C2 (sph)	X	Y	Z	X	Y	Z
1001	Buck Reef Fault	3	1	3	-45.0	-70.0	25.0	0.389	0.313	0.298	9.6	3.5	3.0	23.5	17.9	9.7
1002	A4 Fault	3	1	3	-60.0	-170.0	40.0	0.210	0.428	0.362	6.1	6.4	8.6	21.5	36.4	24.9
1003	Keel Structure	3	1	3	10.0	-20.0	-25.0	0.499	0.276	0.225	2.3	2.5	2.8	10.8	12.2	6.7
1004	Nolans	3	1	3	0.0	-50.0	90.0	0.347	0.414	0.239	2.4	13.4	8.7	24.7	40.2	66.0
1005	Nolans Background	3	1	3	80.0	-80.0	-5.0	0.313	0.502	0.184	2.3	6.6	8.5	12.0	45.3	27.0
1008	West of BRF	3	1	3	70.0	-35.0	-40.0	0.354	0.170	0.476	6.0	26.0	16.4	16.0	9.0	44.1
9999	Background	3	1	3	-80.0	-100.0	5.0	0.621	0.151	0.228	10.6	4.2	24.2	26.9	13.5	58.9

Axis Convention: 1 = X, 2 = Y, 3=Z

Table 15. Sarsfield-Nolans variogram models for OK domains.

5.5.2 Indicator Kriged Domains

Directional and down hole experimental variograms were calculated from the indicator transformed composites for each indicator threshold. The maximum direction of continuity was determined from a full 3D analysis of the ~0.2 g/t indicator on 10° intervals in both strike and dip.

Variograms for lower-grade indicators show better structure and lower nugget effects compared to higher-grade indicators. Variogram models were developed in-line with this observation ensuring that the nugget effect increased and range decreased as the indicator grade increased. This practice reduces the number of order-relation problems during estimation.

The indicator variograms for both D1006 and D1007 show a decreasing anisotropy as the threshold increases. The indicator nugget effects increase from moderate (~30-35%) to high (~65%) at higher grades. The slope near the origin is steep with 70-80% of the total variance occurring in the first 15m or less. The total range for low grade indicators shows a 0.7:1.0 aspect ratio with a maximum continuity of 115-120m. As the indicator threshold increases the total range decreases dramatically and above grades of 1.0 g/t Au the maximum range is less than 30m.

These variogram features are consistent with the geology observations in the mined open pit. Several authors (e.g. Davis and Cowan) have noted that the mineralisation is discontinuous and higher-grade zones lack a measurable spatial orientation. Naïve implicit modelling of the drill hole data (Figure 32) also reflects these variogram structures.

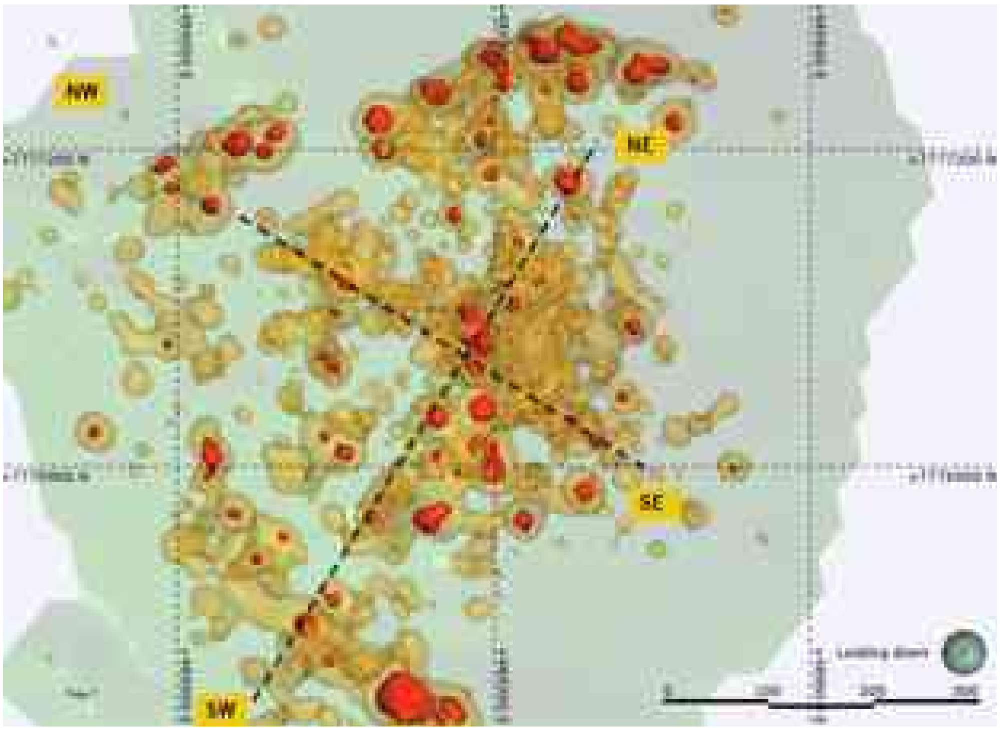


Figure 32. Implicit grade model (After Davis and Cowan, 2017).

All variograms were modelled using spherical models. Models have been normalised with the total modelled variance equal to 1.0. The variogram models are presented in Table 16.

			Rotations						Variogram Structures			Ranges - Structure 1			Ranges - Structure 2		
Domain	Indicator Number	Threshold	Axis 1	Axis 2	Axis 3	Angle 1	Angle 2	Angle 3	C0(Nugget)	C1 (sph)	C2 (sph)	X	Y	Z	X	Y	Z
1006	1	0.08							0.361	0.493	0.148	7.0	15.0	13.5	83.0	114.8	93.0
	2	0.09							0.361	0.493	0.148	7.0	15.0	13.5	83.0	114.8	93.0
	3	0.11							0.398	0.458	0.146	7.0	12.2	9.7	83.0	112.2	82.0
	4	0.14							0.398	0.458	0.146	7.0	12.2	9.7	83.0	112.2	82.0
	5	0.18							0.435	0.435	0.132	7.0	9.0	8.2	47.3	98.7	75.2
	6	0.22	3	2	1	24.3	27.0	-37.5	0.435	0.435	0.132	7.0	9.0	8.2	39.9	91.0	66.4
	7	0.28							0.517	0.359	0.126	7.0	5.4	6.5	26.6	72.8	55.4
	8	0.84							0.517	0.359	0.126	4.6	3.8	6.5	14.5	41.0	29.9
	9	1.74							0.567	0.309	0.126	4.6	3.8	4.8	9.3	21.7	17.7
	10	3.34							0.642	0.234	0.126	4.6	3.8	4.8	9.3	17.1	14.3
	11	6.50							0.642	0.234	0.126	5.0	5.0	5.0	10.0	10.0	10.0
	12	13.27							0.642	0.234	0.126	5.0	5.0	5.0	10.0	10.0	10.0
1007	1	0.05							0.292	0.516	0.192	16.2	11.4	15.2	83.9	122.4	79.3
	2	0.07							0.292	0.516	0.192	16.2	11.4	15.2	83.9	122.4	79.3
	3	0.09							0.292	0.516	0.192	16.2	8.4	10.7	79.4	114.3	57.5
	4	0.12							0.292	0.516	0.192	16.2	8.4	10.7	72.0	106.4	44.6
	5	0.16							0.350	0.461	0.189	16.2	8.4	8.0	61.2	83.5	33.4
	6	0.23	3	2	1	70.0	0.0	15.0	0.464	0.347	0.189	6.8	8.4	5.5	45.5	69.3	23.3
	7	0.33							0.464	0.347	0.189	6.8	8.4	5.5	34.5	45.0	12.5
	8	1.03							0.573	0.255	0.172	6.8	5.4	5.5	21.8	30.6	12.5
	9	2.25							0.573	0.255	0.172	6.8	5.4	5.5	17.5	16.3	12.5
	10	4.38							0.646	0.182	0.172	3.2	3.2	3.8	12.5	11.3	11.8
	11	8.80							0.650	0.180	0.170	3.2	3.2	3.8	12.5	11.3	11.8
	12	24.20							0.650	0.180	0.170	3.2	3.2	3.8	12.5	11.3	11.8

Table 16. Sarsfield indicator variogram models.

5.6 Block Model Framework

The block model covers a volume of 2,600m x 1,300m x 550m (XYZ) enclosing the full interpreted extent of the deposit with an additional margin. The model is based on 20m x 20m x 10m parent blocks with sub-blocking down to 5m x 5m x 1.25m.

Block size was selected based on a combination of drill hole spacing and consideration of likely mining selectivity. The Sarsfield-Nolans mineralisation is discontinuous with limited visual or other controls to allow highly selective mining. While it may be possible to improve selectivity with more data and better geological understanding, in SD2’s opinion the current block size represents the smallest selection unit (or SMU).

The block model framework was used to create a volume model flagged with the interpreted domain codes. Due to the angular difference between the local mine grid and the Buck Reef Fault, Domain 1001 has a relatively high proportion of sub-blocks. The other domains are relatively orthogonal to the grid and sub-blocks are due to edge/boundary effects.

The wireframe vs. block model volume was compared for all domains. No material differences were identified.

The model was coded for the following binary variables:

- Air (1 = above topo, 0 = below topo);
- Oxide (1 = oxide, 0 = fresh);
- Rock (1 = in situ, 0 = air);

- Void (1 = mined void, 0 = unmined rock);
- Fill (1 = in-pit fill or tails, 0 = no fill);
- Design (1 = in design pit, 0 = outside of design pit); and
- E-Method (OK or MIK or Background).

The model is fully depleted. That is, blocks above the as-mined topography have a density and grade of zero. Similarly, the model is depleted for the underground workings at Nolans.

5.7 Estimation

Estimations was by a combination of ordinary kriging (OK) and multiple indicator kriging (MIK) into parent blocks. Block discretisation was set to 4 x 4 x 2

5.7.1 Kriging Neighbourhood and Search Strategy

The search strategy determines what composites are used to estimate each block in the model and, after domaining, selecting of a well-designed search is one of the most critical factors in developing a robust resource estimate. The kriging weights assigned by the kriging equations are a function of the block size, the variogram and the sample-to-block vectors for all samples in the search neighbourhood. The weights themselves are independent of the grades of the samples. An overly restrictive kriging neighbourhood restricts the number of composites that can inform a block estimate. This can result in conditional bias and poor estimation quality depending on the sample spacing and distribution. Similarly, a loose kriging neighbourhood potentially allows too many samples to be included in the weighting assignment. This can lead to broad grade smoothing or averaging and, in some instances the generation of negative weights and potentially negative grades.

There are several levers that can be used when designing a kriging neighbourhood. The search is typically defined using a combination of distances in three orthogonal axes (rotated to align with the variogram) forming an ellipse, plus a requirement for a certain minimum and/or maximum number of composites within the ellipse.

This can be further modified by applying a variety of declustering constraints such as octant/sector limits and specifying the maximum number of samples allowable from an individual drill hole. These declustering approaches have the effect of increasing the average sample-to-block distance compared to undeclustered searches. Thus, declustering is a trade-off between sample-block distance (a direct driver of estimation quality through the kriging matrix) and the potential for spatial bias generated by clustered data. While the kriging equations do, to some extent, result in declustering of the kriging weights, some block-to-sample arrangements can adversely impact on estimation performance.

Further complicating the selection of a kriging neighbourhood, it is rare for composites to be regularly arranged (on a grid pattern) for a resource estimate. This regular arrangement is much more commonly associated with grade control drill patterns. The inconsistency of block-to-sample geometry across any given domain means that any single neighbourhood definition will be sub-optimal in some regions.

In SD2's experience the most practical approach to optimising the kriging neighbourhood is to focus on the minimum and maximum numbers of composites used to inform a block estimate. The search distances are secondary as long as they are sufficiently large to capture the specified number of composites. Effectively, a wide search range is applied and when the maximum number of allowed composites is reached within that search range, no more composites are added. The practical range is therefore a function of sample spacing. In areas of widely spaced drilling the average sample-to-block distance will be greater when compared to areas of more closely spaced drilling. Likewise, the estimation performance (as measured by metrics such as kriging efficiency and slope of regression) will vary as a function of sample spacing.

There are additional considerations in the case of MIK estimation, including:

- The requirement for sufficient composites to allow sufficient probability for the estimated block grade distribution function; and
- The need for caution in changing the search neighbourhood at different indicator thresholds as this may cause an increase in order relationship errors and, in some extreme cases, result in some indicators not being estimated (when there are too few composites in the neighbourhood).

These considerations tend to drive MIK estimation to use a higher number of composites during estimation and to standardise the search strategy across all indicators.

The search neighbourhoods at Sarsfield-Nolans were developed after examining the key estimation quality metrics[8] for a series of sensitivity models. The primary control was based on defining the minimum/maximum numbers of composites required to inform a block estimate. The search ranges were then superimposed on the primary control, maintaining the orientation and anisotropy defined by the variogram model (i.e. the search is aligned with the variogram model). Stepwise increases in the minimum and maximum number of composites defined a target range of between 20 and 40 composites for all OK domains. For MIK domains the maximum increased to 60 composites.

All domains were estimated with hard boundaries (i.e., estimated only with samples lying within the interpreted domain volume). Search ellipse orientations were aligned to the variogram model (no dynamic anisotropy imposed).

[8] Slope of regression, sum of positive weights, kriging efficiency, weight of the mean in a Simple Kriging estimate.

A limit was set on the maximum number of composites per hole. For Domain 1004 (Nolans main zone) the maximum was 5. For all other domains the maximum was 10. The full search strategy is outlined in Table 17 and Appendix C.

			Rotations						Search Range (Pass 1)			Minimum	Maximum	Expansion	Minimum	Maximum	Expansion	Minimum	Maximum
Domain	Description	Type	Angle 1	Angle 2	Angle 3	Axis 1	Axis 2	Axis 3	X	Y	Z	Composites	Composites	Factor	Composites	Composites	Factor	Composites	Composites
1001	BRF	OK	(45)	(70)	25	3	1	3	40	40	15	20	40	1.5	20	40	2	12	40
1002	A4	OK	(60)	(170)	40	3	1	3	20	30	20	20	40	1.5	20	40	2	12	40
1003	Keel	OK	10	(20)	(25)	3	1	3	20	25	15	20	40	1.5	20	40	2	12	40
1004	Nolans	OK	-	(50)	90	3	1	3	35	55	90	20	40	1.5	20	40	2	12	40
1005	Nolans Background	OK	80	(80)	(5)	3	1	3	15	60	35	20	40	1.5	20	40	2	12	40
1006	A4-BRF	MIK	(165)	60	(120)	3	1	3	40	90	60	20	60	1.5	20	60	2	16	60
1007	SE A4	MIK	160	165	(90)	3	2	1	45	70	25	20	60	1.5	20	60	2	16	60
1008	W BRF	OK	70	(35)	(40)	3	1	3	20	15	60	20	40	1.5	20	40	2	12	40
9999	Waste	OK	(80)	(100)	5	3	1	3	70	35	150	20	40	1.5	20	40	2	12	40

Table 17. Search neighbourhood definitions.

5.7.2 Order Relationship

MIK estimates are prone to order relationship errors. This error occurs when the estimated proportion of a higher-grade indicator is greater than the estimated proportion of a lower-grade indicator for the same block. These errors are commonly caused by changes in the variogram models and/or search neighbourhoods at different indicator thresholds.

The MIK domains were examined for order relationship errors and the errors were rectified. Approximately 4% of the full array of indicator estimates were found to have an order relation problem. The highest percentage occurred between the 4th and 5th indicator and 5th and 6th indicators in both domains. This corresponds to the grade range between 0.1 g/t and 0.2 g/t Au.

There are several methods for correcting order relationship errors. The most common are:

- Top-down where the proportions of lower threshold indicators are increased to equal a higher indicator;
- Bottom-up where the proportions of higher threshold indictors are decreased to equal a lower indicator; and
- Averaging the top-down and bottom-up approach.

SD2 applied the bottom-up approach; this is the more conservative approach. The impact of top-down vs. bottom-up approach was less than 0.5%, affecting the third decimal place in the grade estimate.

5.7.3 Post-Processing

After estimation the model was checked for common estimation artefacts including negative grades and blocks that were un-estimated after applying all search options.

There were no negative grade (or indicator proportion) estimates. Un-estimated blocks were assigned a default grade of 0.001 g/t Au. The majority of un-estimated blocks occur at depth, below the Inferred resource limit. The remainder are located on the edges of the interpreted domains in positions where there is low drill coverage (Figure 33) placing them in extrapolation positions.

Figure 33. Location of un-estimated blocks.

6. Validation

A range of validation and comparisons were used to assess the quality of the resource estimate. The estimated domain grades were compared to the declustered composite grades (Figure 34) and swath plots were created in plan and section (Appendix I).

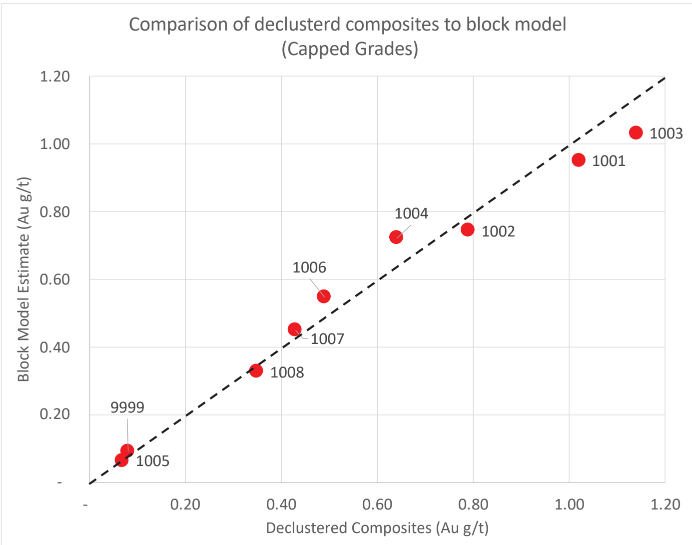


Figure 34. Scatterplot - composites vs model estimate.

Comparing block model estimates to declustered composite grades shows that there are some minor biases. Domains 1001, 1002 and 1003 are biased low compared to the composite data and domains 1004 and 1006 are biased high. The differences in each case are within reasonable precision tolerance limits. The apparent bias in both domain 1004 and 1006 appears to be caused by the number of unsampled intervals (~6%) that have been assigned a default grade of 0.005. If these data are excluded from the composite, the bias is negligible.

The swath plots (Appendix I) show a reasonable agreement between the composite grade and estimated grade, although there are some artifacts associated with clustering and isolated data. The apparent biases in domain 1001 are due to clustering in the upper regions of the domain (now mined out). This has affected bench, easting and northing plots in some sections. The apparent bias in domain 1004 reflects grade extrapolation below the near-surface drilling as can be seen in the bench swath (Figure 35). Both IK domains show good correlation with the composite except when the number of composites is low.

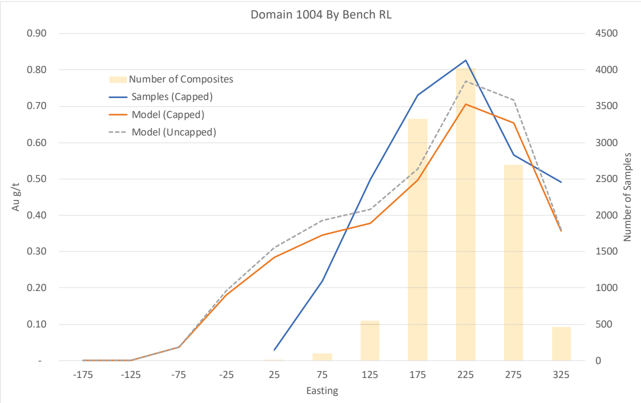


Figure 35. Domain 1004 Bench Swath Plot.

The features observed in the swath plots contributed to resource classification.

7. Mineral Resource Classification

7.1 Jurisdiction and Competent Person

This mineral resource estimate is classified and reported under the guidelines of the JORC Code[9] (2012). The estimate has been prepared by Mr Scott Dunham, a Fellow of the AusIMM (membership number 112857). Mr Dunham has more than 30 years of experience in the resource industry including more than the requisite five (5) years relevant experience in the estimation of mineral resources for the commodity and style of mineralisation at Buck Reef West. A brief summary of Mr Dunham's experience is provided in Appendix M. His expertise

[9] The Australasian Code for reporting of exploration results, mineral resources and ore reserves. 2012 Edition.

covers the complete range of resource estimation practices including geological sampling, interpretation and domaining, geostatistical analysis, estimation and reporting.

7.2 Reasonable Prospects Assessment

The JORC Code requires reported mineral resources to have 'reasonable prospects of eventual economic extraction'. In Mr Dunham's opinion this expectation has been demonstrated for Sarsfield-Nolans as follows:

- A positive NPV generated by Resolute Mining Limited in 2018;
- A positive NPV generated by EMR Capital during the due diligence study for the acquisition of the Ravenswood Gold Mine;
- The recently completed acquisition transaction.

Mr Dunham is aware that RAV are currently negotiating social, heritage and environmental licensing conditions. The negotiations are well advanced and no material impediments are likely. Ownership transfer is expected to be finalised in August/September 2020.

The reported resource lies above an AUD3800 optimised pit shell.

7.3 Classification Definitions

The Sarsfield-Nolans resource is classified as Measured, Indicated and Inferred. The classification is based on a combination of:

- Drill hole spacing and orientations;
- Estimation performance metrics including the slope of regression, kriging efficiency, sum of positive weights and weight of the mean; and
- The search pass and number of composites used during block estimation.

The classification limits were developed through a combination is implicit modelling and manual interpretation of the above metrics. In practice the classes strongly reflect drill hole spacing and the decrease in the number of drill holes at depth. This forms a set of nested shells beneath the current as-mined pit surface (Figure 36).

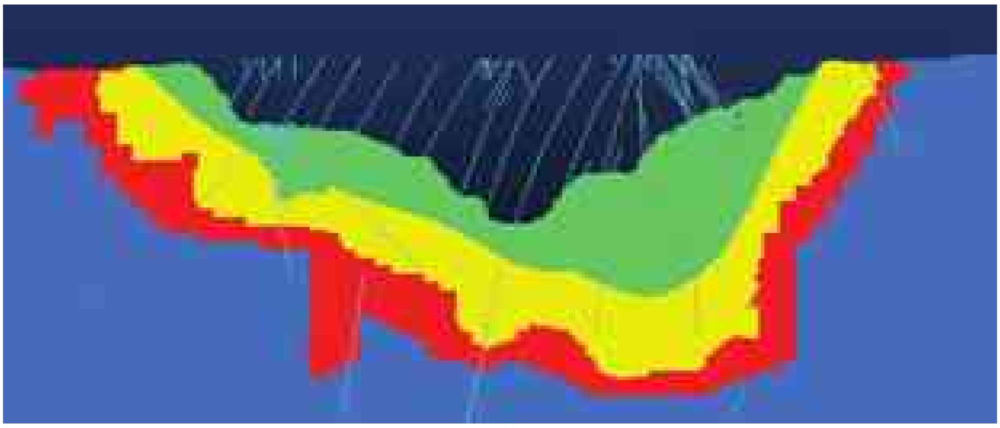


Figure 36. Cross section showing nested classification shells.

7.4 Risk and Range Assessment

During the course of preparing the Sarsfield-Nolans mineral resource estimate SD2 investigated:

- Alternate domaining options based on manual interpretation, pseudo-vein density and grade-based implicit modelling;
- The impact of Ordinary Kriging vs Multiple Indicator Kriging;
- The impact of grade capping at different values; and
- The impact of changes in the search strategy.

These investigations highlight that the primary risk is associated with data density and domain interpretation. While regions directly underneath the as-mine topography are reasonably well informed, the drill density decreases rapidly (see Figure 25 and Figure 26).

The July 2020 estimate was also compared to the previous model released by Resolute Mining Limited (an MIK model by MPR Geological Consultants) and the due diligence 'sensitivity' model prepared by SD2 in September 2019 (Table 18). There is a close agreement between this estimate and the Resolute estimate. The capped estimate and the due diligence estimate bracket the current model. This analysis and the results of the various investigations support a likely precision of +/-10% on the global contained ounces.

Model	Difference in Ounces
No capping	109%
MPR MIK Estimate	101%
SD2 Due Diligence	93%

Table 18. Comparison to previous estimates (above 0.3g/t).

While the Resolute estimate and the July 2020 estimate show good correlation on the global contained ounces, the two models differ in the spatial distribution of those ounces. The Resolute model did not have a separate domain for the A4 structure and other domains varied. The differences in the two models can be seen in the ounces by bench comparison (Figure 37).

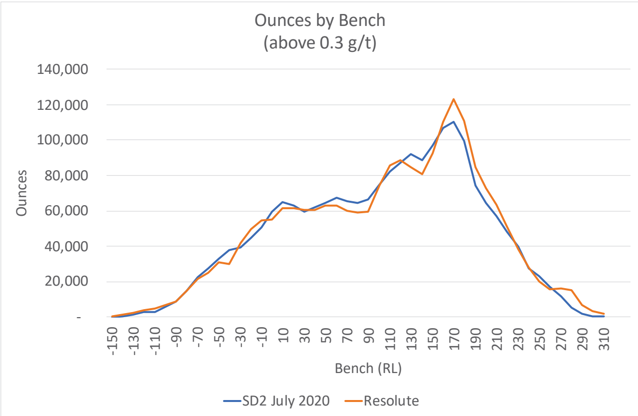


Figure 37. Ounces by Bench.

7.5 Upside Potential

The current estimate is limited by drill hole spacing at depth. Additional drilling focused on depth extensions in both the Nolans and Sarsfield regions should be investigated.

There is also potential to better understand the controls on the stockwork vein mineralisation. This may allow improved domaining which in turn would allow better estimation of the grade-tonnage curve.

8. Resource Statement

The Sarsfield-Nolans mineral resource estimate is reported above a cut-off of 0.3 g/t Au within an AUD3,800[10] optimised pit shell as at 10 July 2020.

Classification	Tonnes (kt)	Au (g/t)	Ounces (koz)
Measured	32,213.0	0.71	739.9
Indicated	71,354.2	0.65	1,498.2
Inferred	29,394.2	0.63	597.8
Grand Total	**132,961.4**	**0.66**	**2,835.9**

[10] AUD3,800 pit shell assuming material above 0.63g/t Au is direct feed to the Ravenswood ore treatment plant and material between 0.30g/t and 0.63g/t is beneficiated at the Ravenswood beneficiation plant.

9. References

McCormack, K., and McClay, K., 2018. Orthorhombic faulting in the Beagle Sub-basin, North West Shelf, Australia. In, McClay, K. R. & Hammerstein, J. A. (eds) Passive Margins: Tectonics, Sedimentation and Magmatism. Geological Society, London, Special Publications

Appendix A Competent Persons Consent Form

Competent Person's Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

Report name

Sarsfield-Nolans Mineral Resource Estimate July 2020

(*Insert name or heading of Report to be publicly released*) ('Report')

Ravenswood Gold Pty Ltd

(*Insert name of company releasing the Report)*

Sarsfield-Nolans

(Insert name of the deposit to which the Report refers)

If there is insufficient space, complete the following sheet and sign it in the same manner as this original sheet.

27 November 2020

(Date of Report)

Statement

I/We,

Scott Dunham

(*Insert full name(s)*)

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member or Fellow of *The Australasian Institute of Mining and Metallurgy* or the *Australian Institute of Geoscientists* or a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I am a full-time employee of

(Insert company name)

Or

I/We am a consultant working for

SD2 Pty Ltd

(Insert company name)

and have been engaged by

Ravenswood Gold Pty Ltd

(Insert company name)

to prepare the documentation for

Sarsfield-Nolans

(Insert deposit name)

on which the Report is based, for the period ended

30 September 2020

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Exploration Targets, Exploration Results, Mineral Resources and/or Ore Reserves *(select as appropriate).*

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

Ravenswood Gold Pty Ltd

(Insert reporting company name)

	27 November 2020
Signature of Competent Person:	Date:
Australasian Institute of Mining and Metallurgy	112857
Professional Membership: *(insert organisation name)*	Membership Number:
	Sherrill Dunham - Nanango Queensland
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

Additional deposits covered by the Report for which the Competent Person signing this form is accepting responsibility:

None

Additional Reports related to the deposit for which the Competent Person signing this form is accepting responsibility:

None

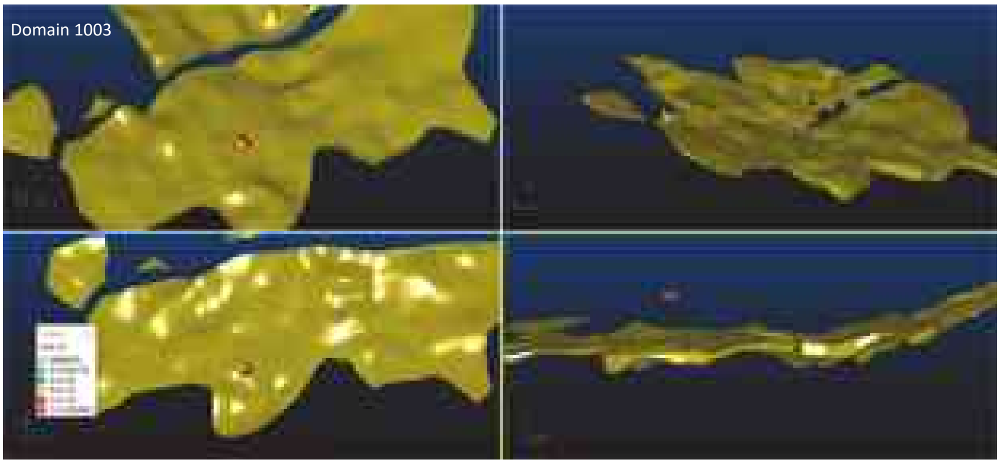
Signature of Competent Person:	Date:
Professional Membership: *(insert organisation name)*	Membership Number:
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

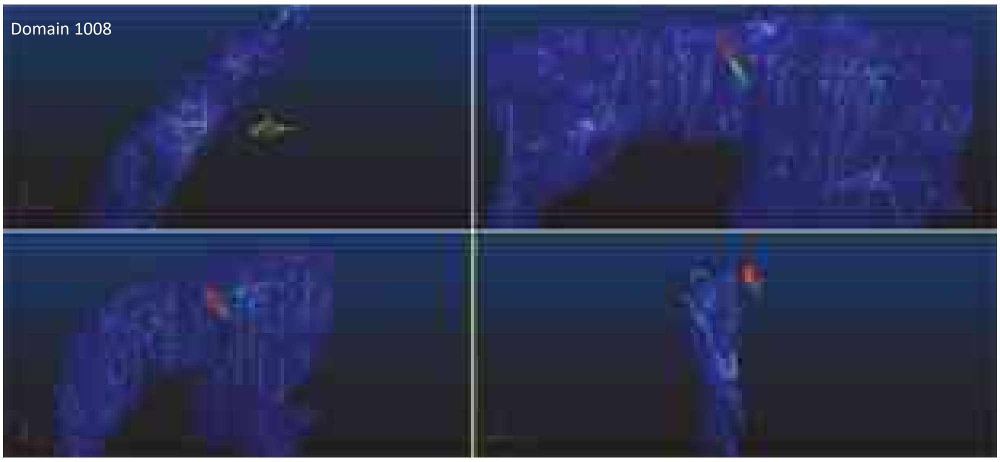
Appendix B Variogram Models

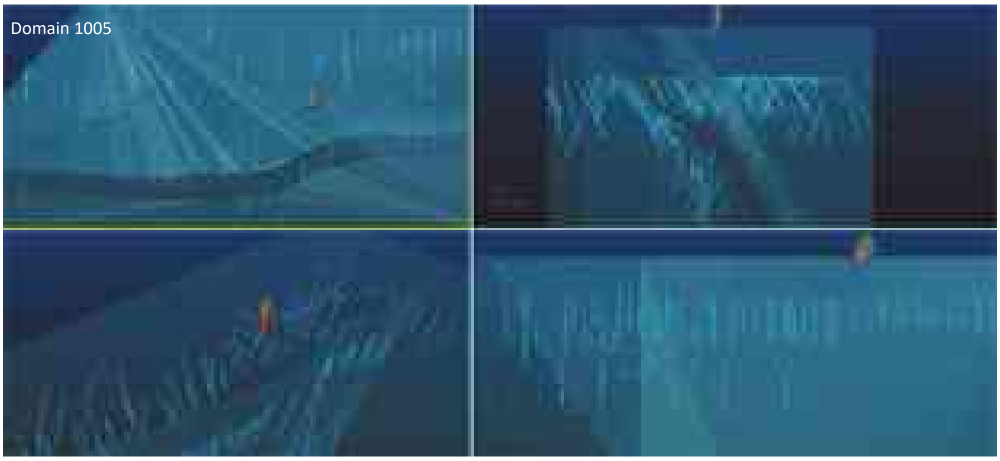
Appendix C Search Ellipses

	Rotations						Distances (Search 1)											
Domain	Axis 1	Axis 2	Axis 3	Angle 1	Angle 2	Angle 3	X	Y	Z	Minimum Composites	Maximum Composites	Hole Restriction	Expansion Factor	Minimum Composites	Maximum Composites	Expansion Factor	Minimum Composites	Maximum Composites
1001	3	1	3	-45	-70	25	40	40	15	20	40	10	1.5	20	40	2	12	40
1002	3	1	3	-59.98	-170	40.02	20	30	20	20	40	10	1.5	20	40	2	12	40
1003	3	1	3	10	-20	-25	20	25	15	20	40	10	1.5	20	40	2	12	40
1004	3	1	3	0	-50	90	35	55	90	20	40	5	1.5	20	40	2	12	40
1005	3	1	3	80	-80	-5	15	60	35	20	40	10	1.5	20	40	2	12	40
1008	3	1	3	70	-35	-40	20	15	60	20	40	10	1.5	20	40	2	12	40
9999	3	1	3	-80	-100	5	70	35	150	20	40	10	1.5	20	40	2	12	40


Domain 1002


Domain 1003


Domain 1008


Domain 1005

Appendix D Drill Hole Coverage by Data Type

Figure 38. RQD and Geotechnical Drilling.

Figure 39. Structural logging coverage.

Figure 40. Lithology logging coverage.

Appendix E QC Performance Data

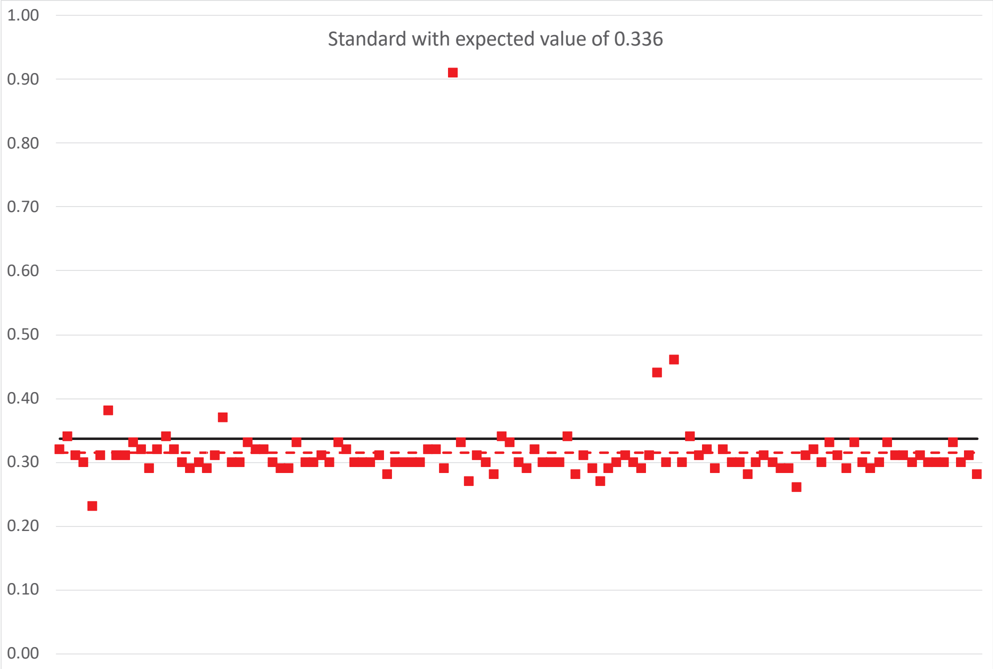


Figure 41. QC Standard from ~1997-98.

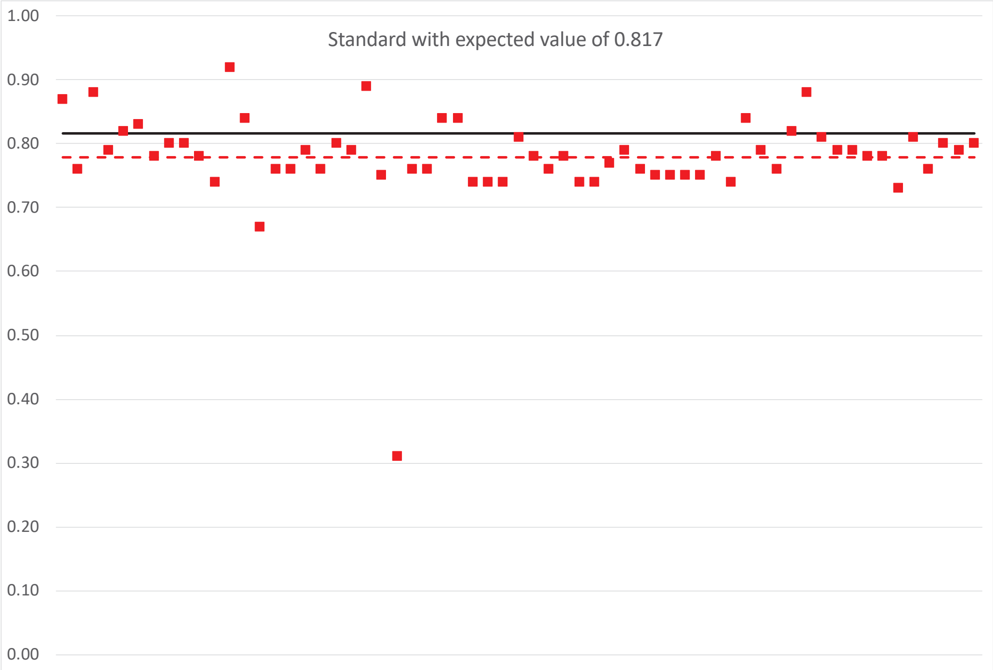


Figure 42. QC Standard from ~ 1997-98 (2).

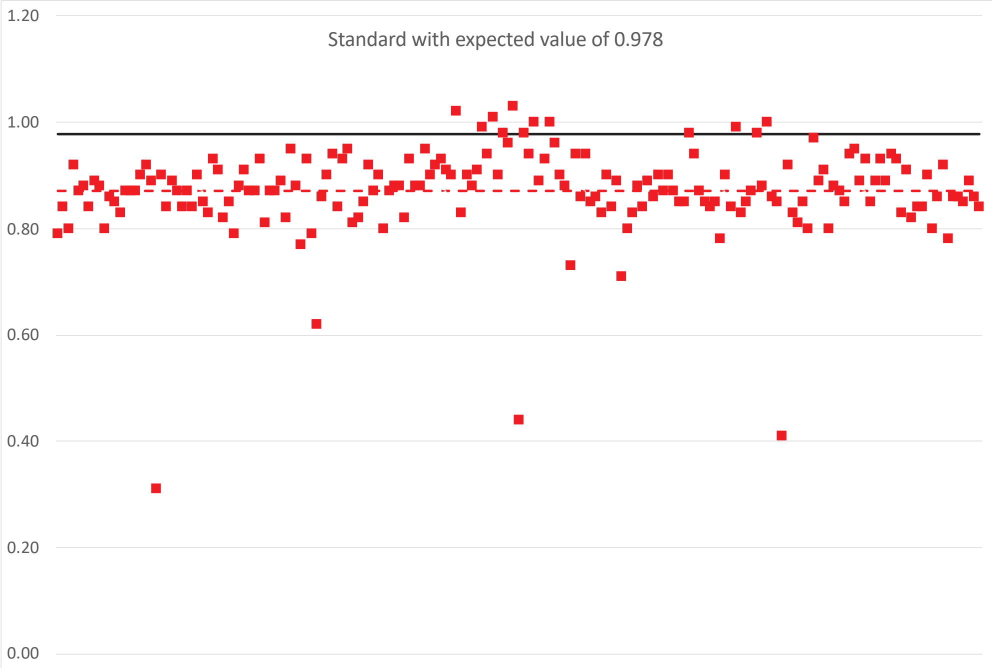


Figure 43. QC Standard from ~ 1997-98 (3).

Figure 44. QC Standard from ~1997-98 (4).

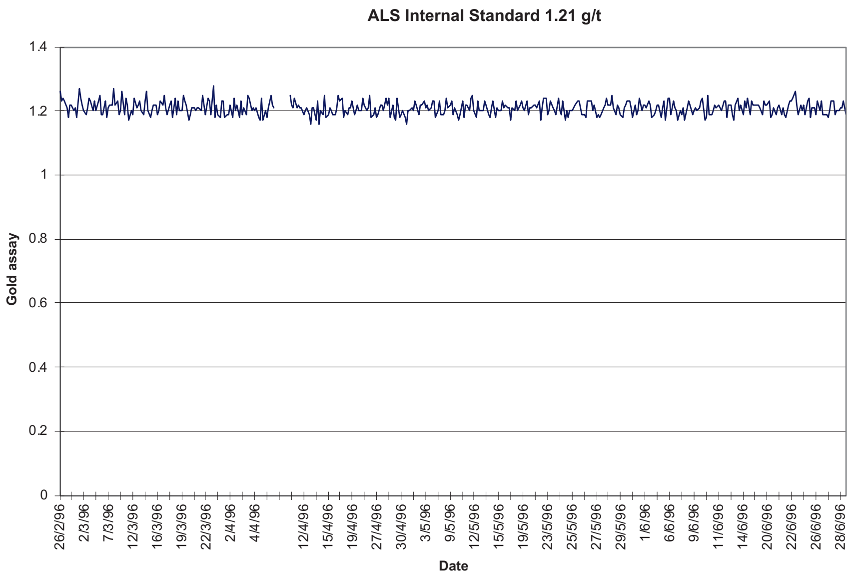


Figure 45. ALS Internal Standard 1996.

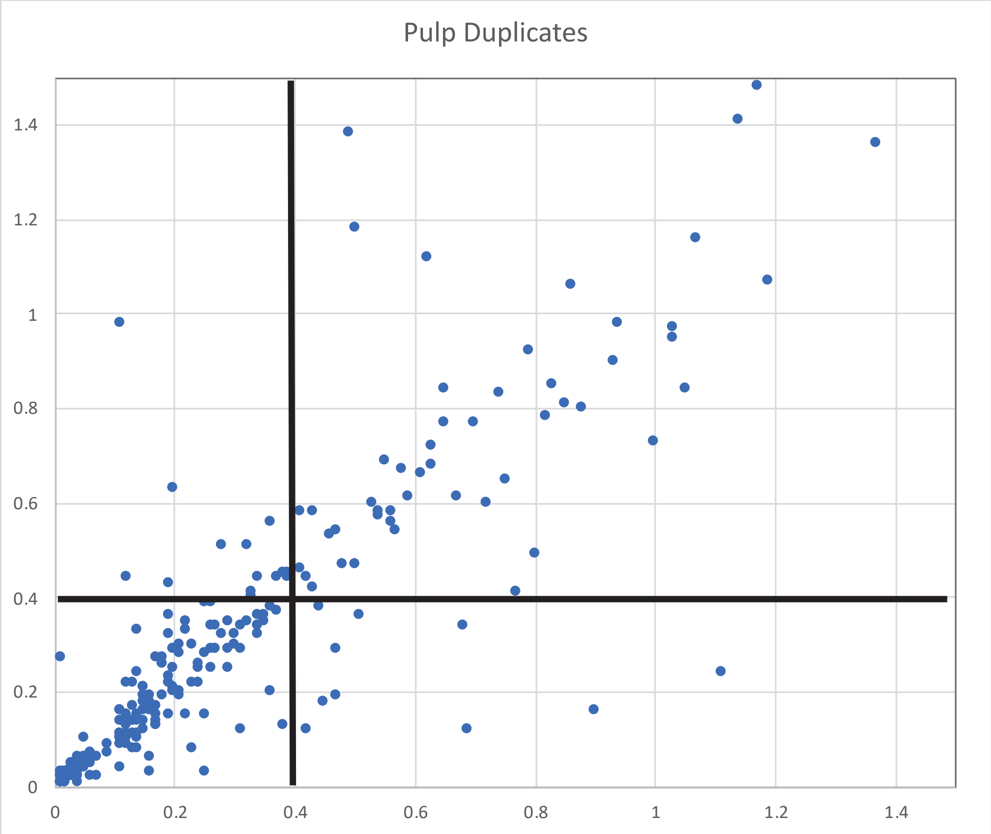


Figure 46. Pulp Duplicates ~ 1997-98.

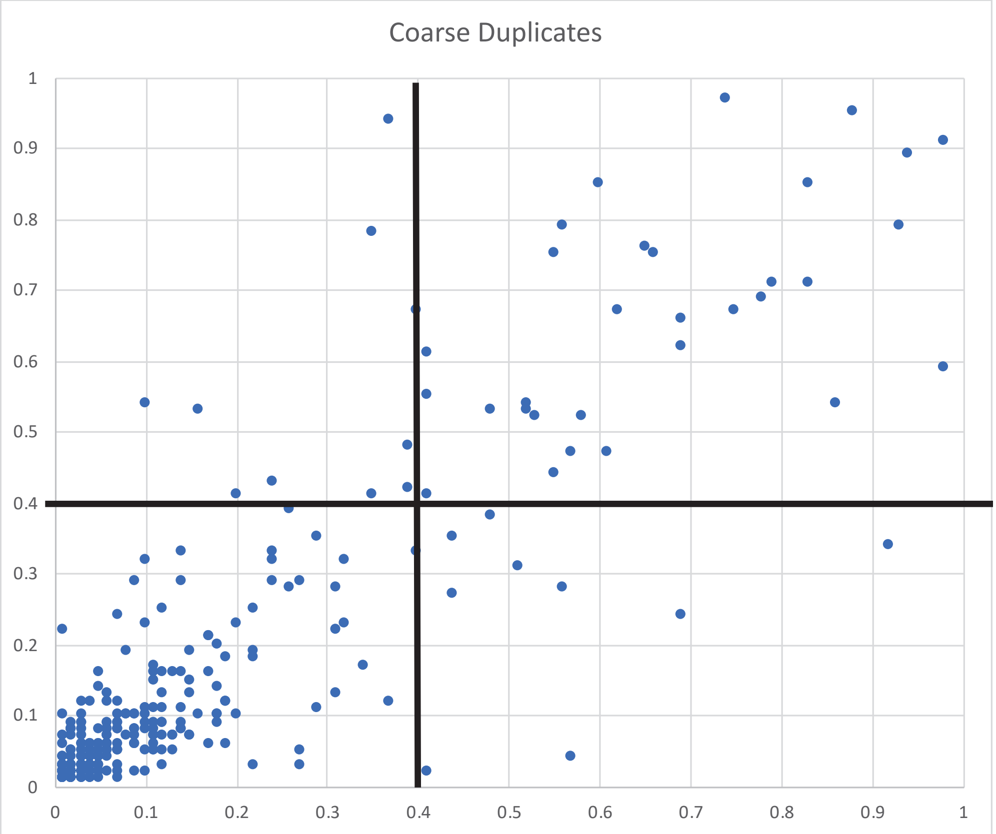


Figure 47. Coarse Duplicates ~ 1997-98.

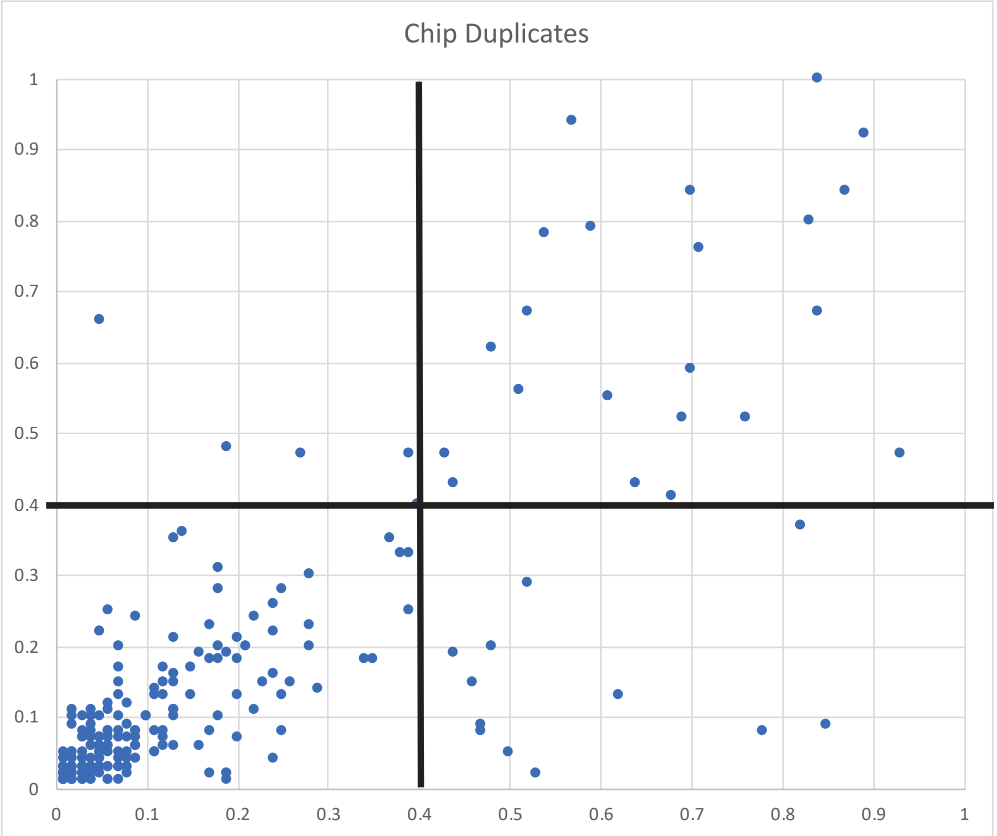


Figure 48. Chip Duplicates ~ 1997-98.

Appendix F Data Listing

Appendix G Model Field Names and Definitions

Field Name	Type	Description
DOMAIN	Numeric	1001 – Buck Reef Fault 1002 – A4 Fault 1003 – Keel Structure 1004 – Nolans 1005 – Nolans region background 1006 – A4-to-BRF 1007 – Southeast of A4 1008 – West of BRF 9999 - Background
VOID	Binary	1 – void (ug stope or development) 0 – solid rock
ROCK	Binary	1 – solid rock 0 – air or void
OXIDE	Binary	1 – above top of fresh rock (i.e., oxide and partially oxidised) 0 – fresh rock
REPORT	Binary	Used to limit size of reporting only. No meaning for JORC Class
Density	Numeric	In situ bulk density. Set to zero for air and voids.
Au_Capped	Numeric	Estimated gold grade – reportable grade for JORC Code reporting
Au_No_Cap	Numeric	Sensitivity only – estimate without capped grades. Not for public reporting
CLASS	Numeric	JORC Code reporting classification 0 - Unclassified 1 – Measured 2 – Indicated 3 – Inferred
AMDAD	Binary	1 – Inside AMDAD due diligence Stage 2 pit 0 – not in AMDAD due diligence Stage 2 pit
XMORIG	Numeric	Model origin Easting (bottom left)
YMORIG	Numeric	Model origin Northing (bottom left)
ZMORIG	Numeric	Model origin RL (bottom left)
NX	Numeric	Number of parent blocks in Easting (X)
NY	Numeric	Number of parent blocks in Northing (Y)
NZ	Numeric	Number of parent blocks in RL (Z)
XINC	Numeric	Sub-block Easting dimension
YINC	Numeric	Sub-block Northing dimension
ZINC	Numeric	Sub-block RL dimension
XC	Numeric	Block centre Easting
YC	Numeric	Block centre Northing
ZC	Numeric	Block centre RL
IJK	Numeric	Datamine Studio RM unique parent block index

Appendix H Grade Tonnage Curves

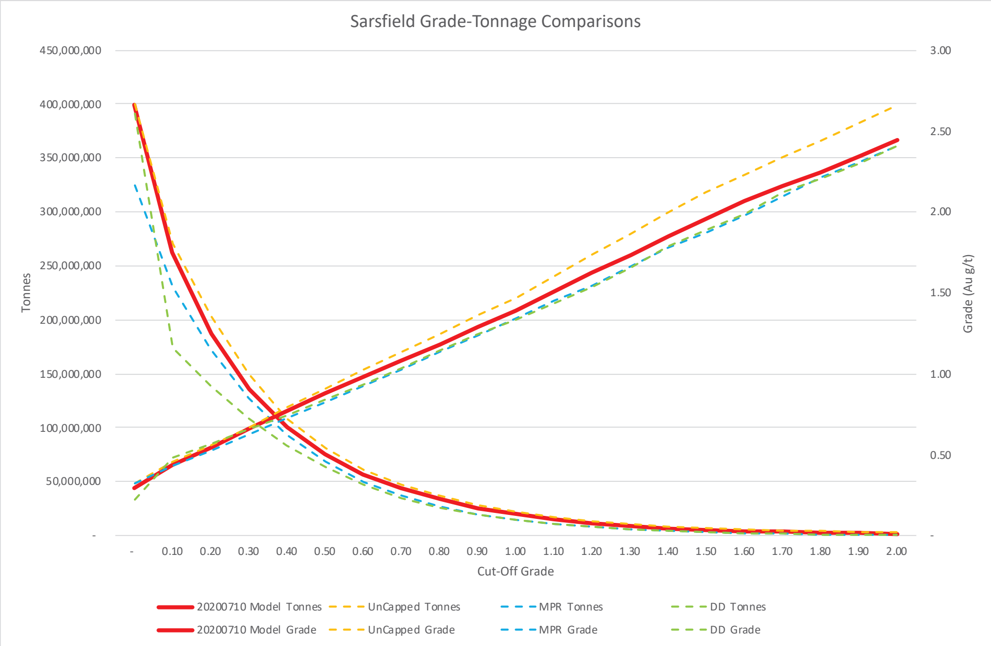
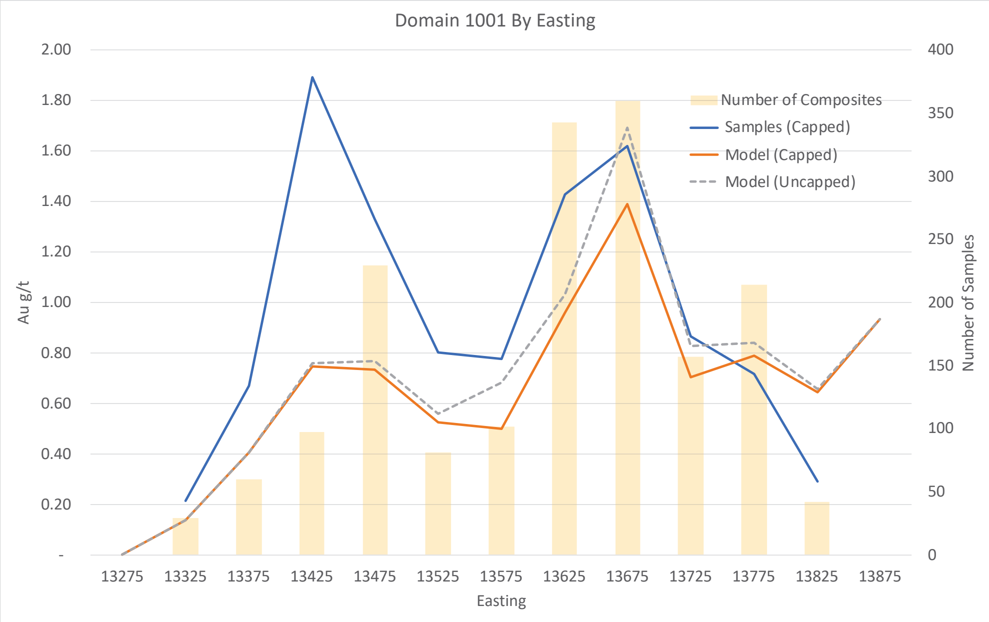
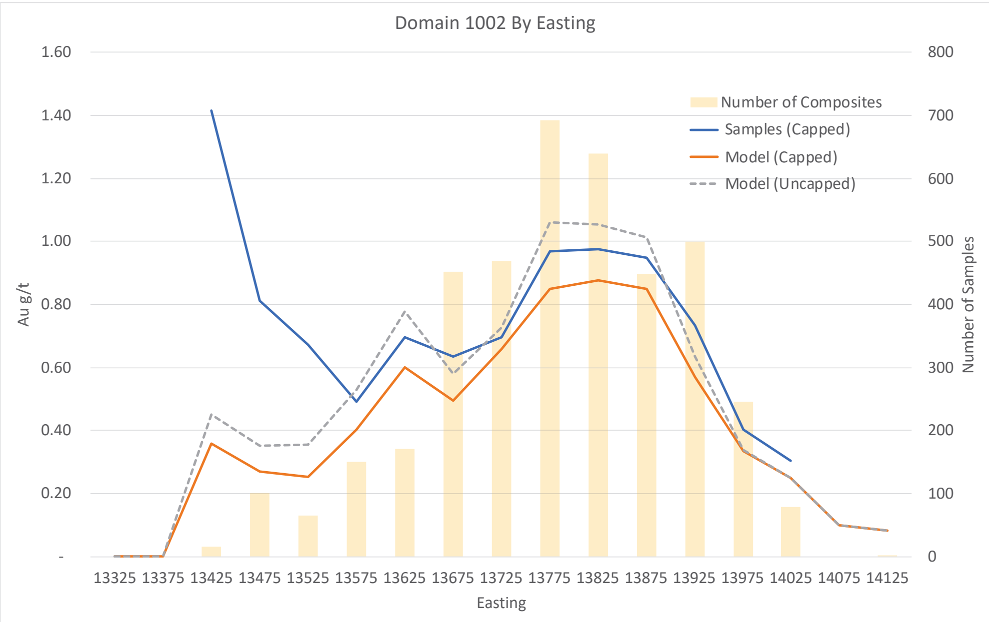
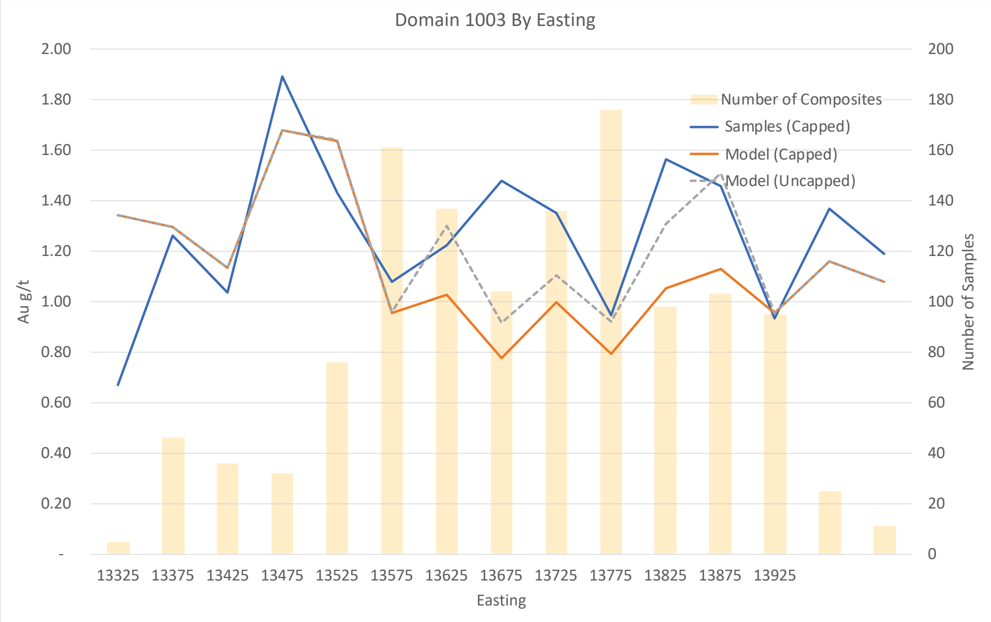


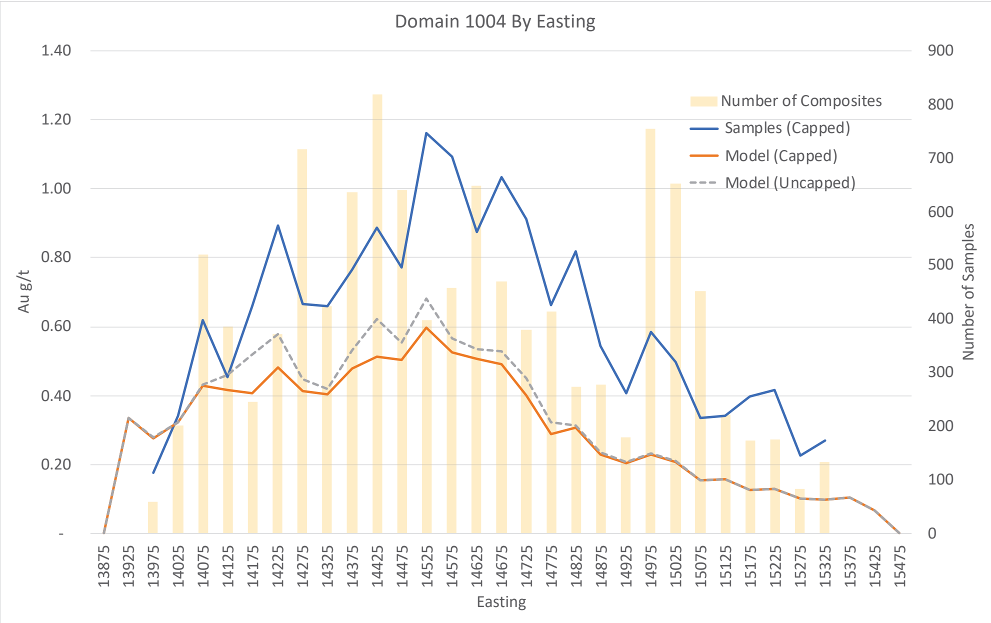
Figure 49. Grade-tonnage curve in AMDAD AUD3800 pit shell. All resource classes.

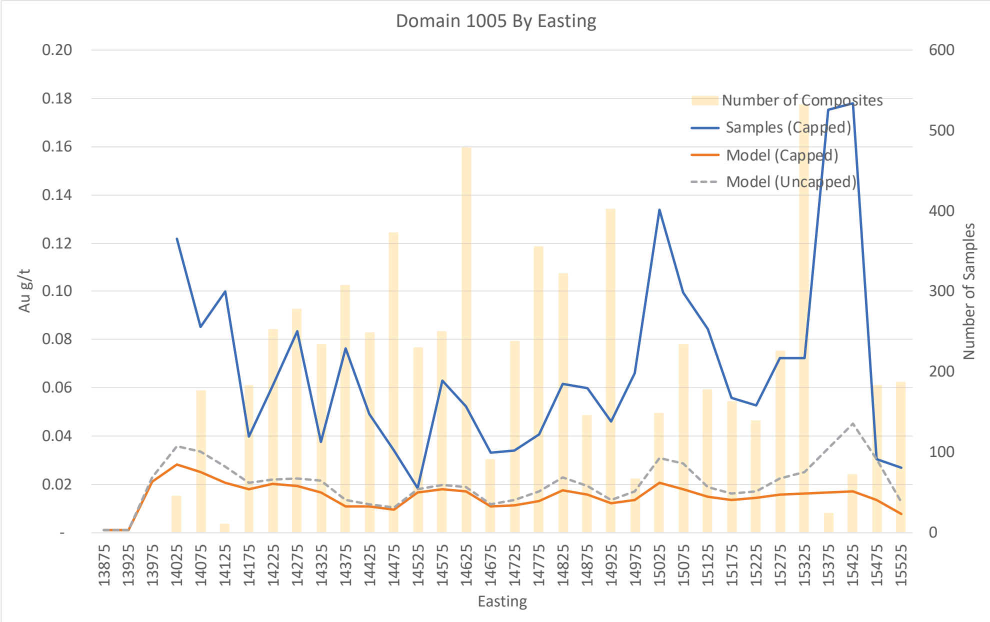
Appendix I Swath Plots (Eastings)






Domain 1003 By Easting
Au g/t
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
2.00
1.80
1.60
1.40
1.20
1.00
0.80
0.60
0.40
0.20
-
200
180
160
140
120
100
80
60
40
20
0
13325 13375 13425 13475 13525 13575 13625 13675 13725 13775 13825 13875 13925
Easting


Domain 1004 By Easting
Au g/t
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
1.40
1.20
1.00
0.80
0.60
0.40
0.20
-
900
800
700
600
500
400
300
200
100
0
13875 13925 13975 14025 14075 14125 14175 14225 14275 14325 14375 14425 14475 14525 14575 14625 14675 14725 14775 14825 14875 14925 14975 15025 15075 15125 15175 15225 15275 15325 15375 15425 15475
Easting


Domain 1005 By Easting
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
Number of Samples
Easting
0.20
0.18
0.16
0.14
0.12
0.10
0.08
0.06
0.04
0.02
-
600
500
400
300
200
100
0
13875
13925
13975
14025
14075
14125
14175
14225
14275
14325
14375
14425
14475
14525
14575
14625
14675
14725
14775
14825
14875
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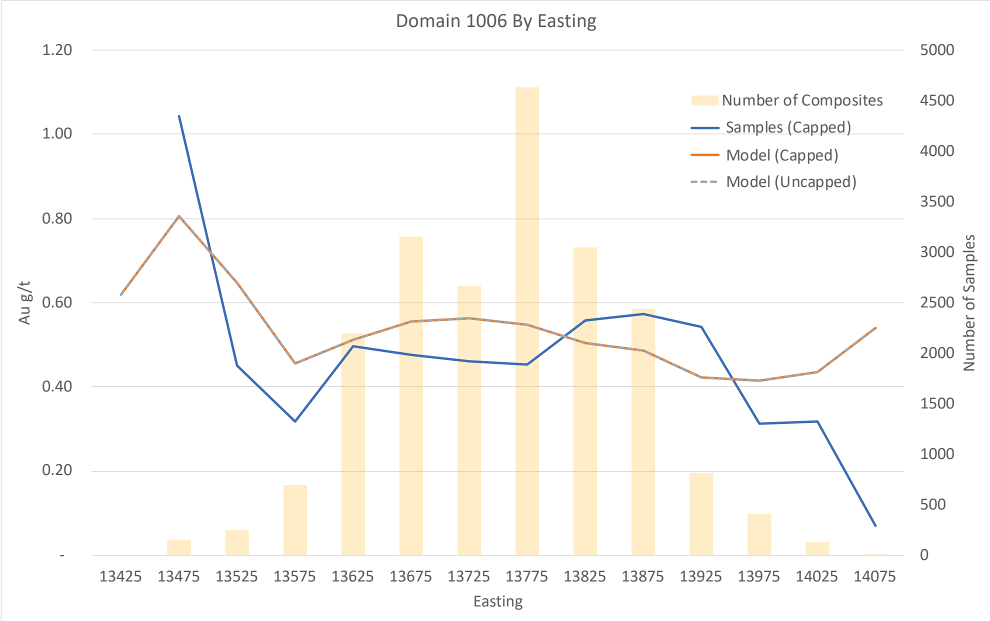
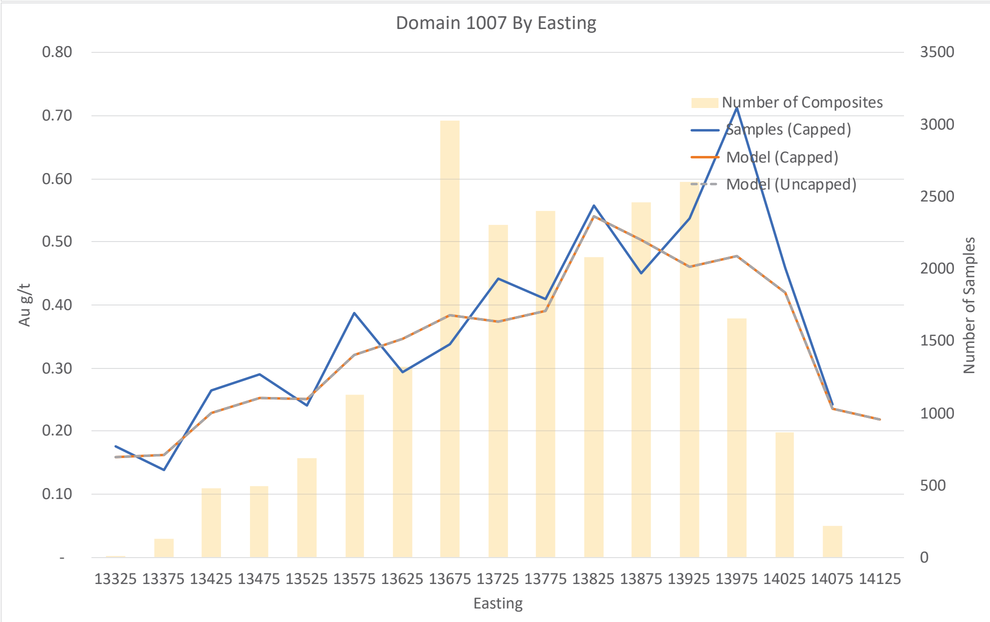
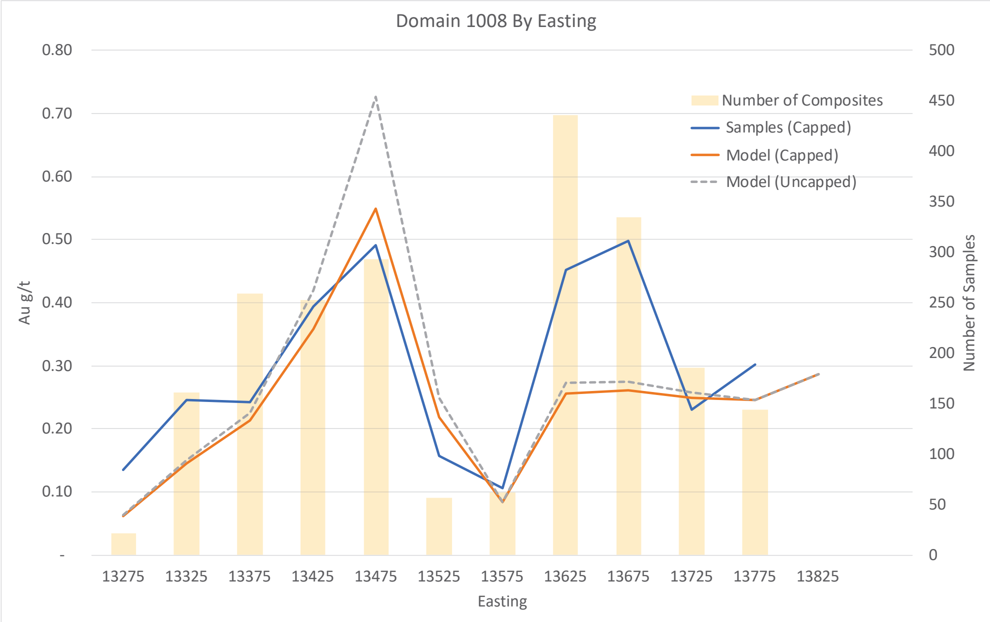
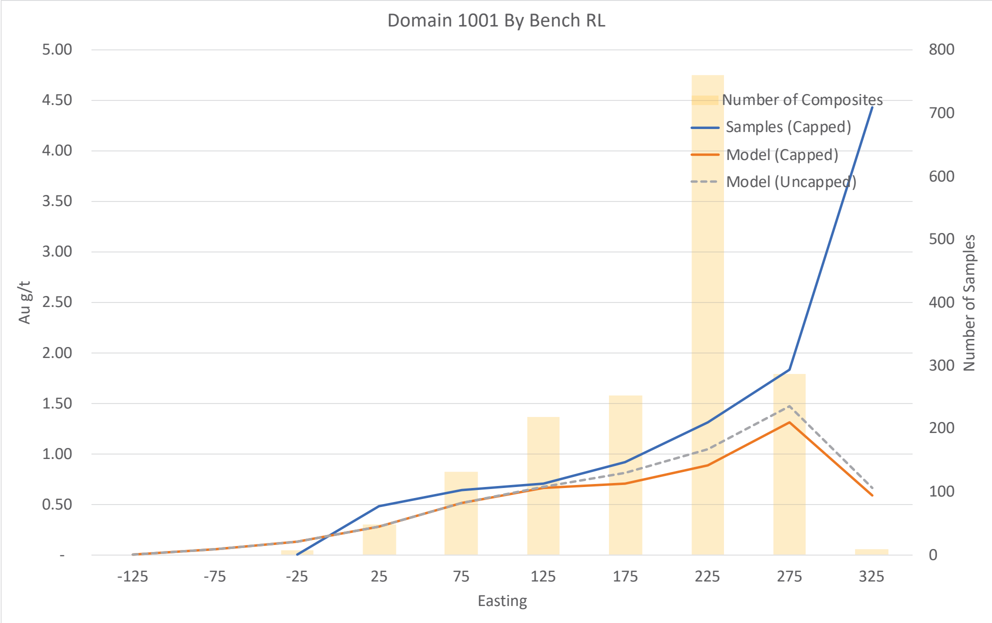
14975
15025
15075
15125
15175
15225
15275
15325
15375
15425
15475
15525

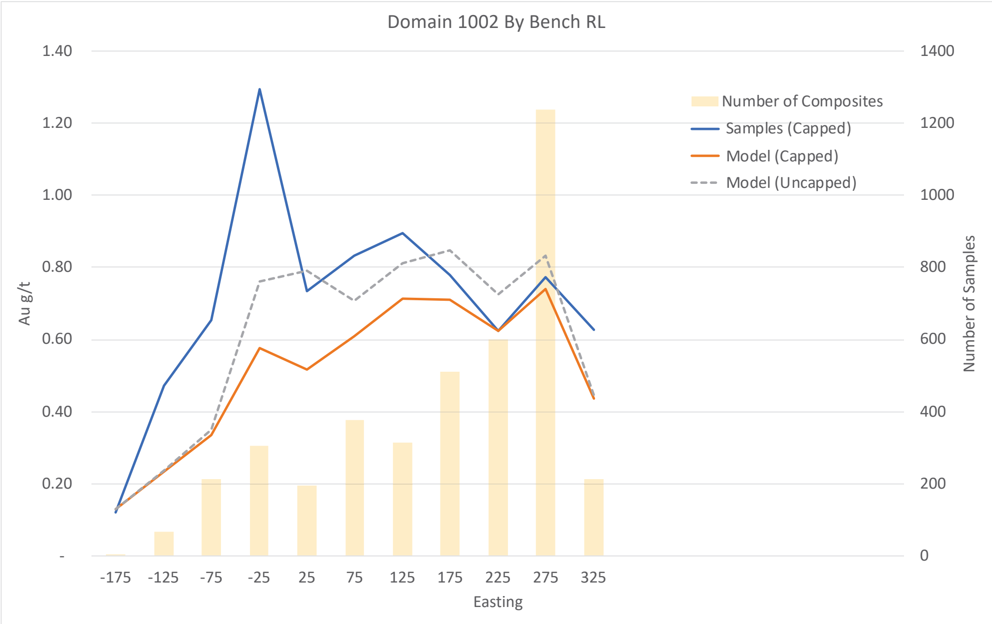

Domain 1006 By Easting
Au g/t
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
1.20
1.00
0.80
0.60
0.40
0.20
-
5000
4500
4000
3500
3000
2500
2000
1500
1000
500
0
13425 13475 13525 13575 13625 13675 13725 13775 13825 13875 13925 13975 14025 14075
Easting

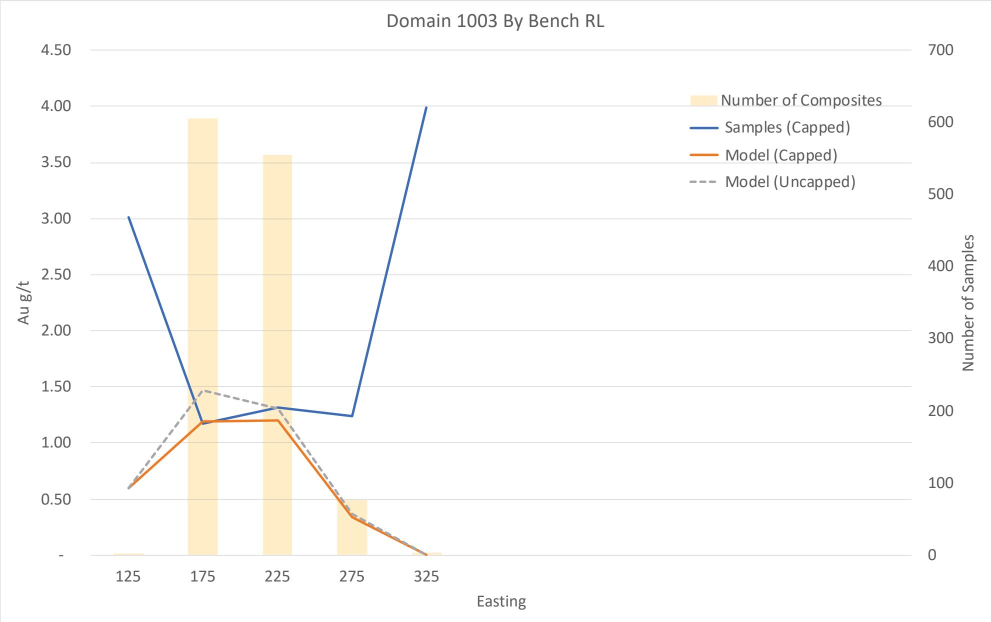

Domain 1007 By Easting
Au g/t
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
3500
3000
2500
2000
1500
1000
500
0
13325 13375 13425 13475 13525 13575 13625 13675 13725 13775 13825 13875 13925 13975 14025 14075 14125
Easting

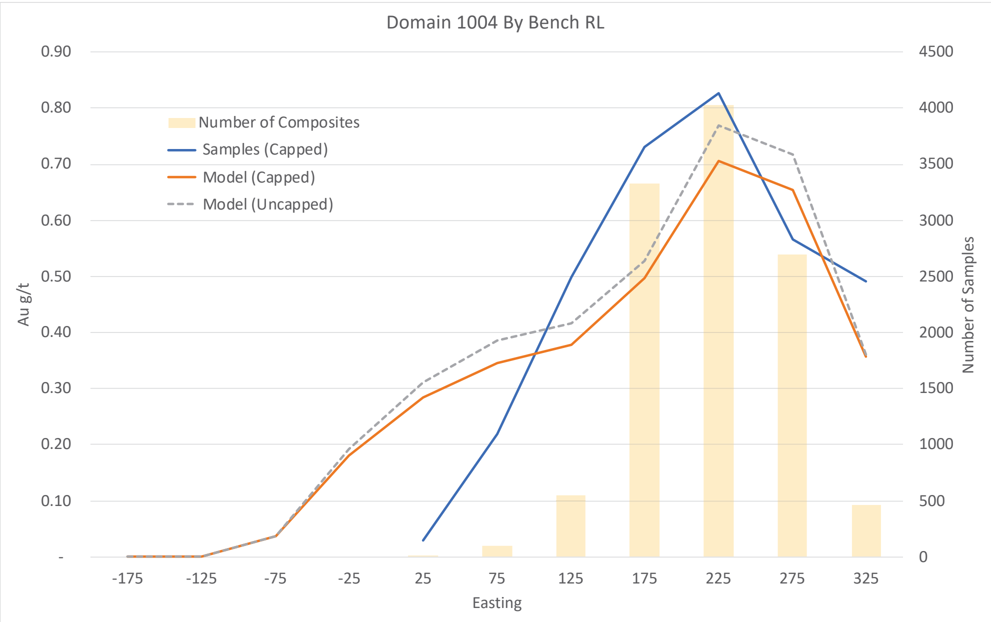

Domain 1008 By Easting
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
Au g/t
500
450
400
350
300
250
200
150
100
50
0
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
13275
13325
13375
13425
13475
13525
13575
13625
13675
13725
13775
13825
Easting

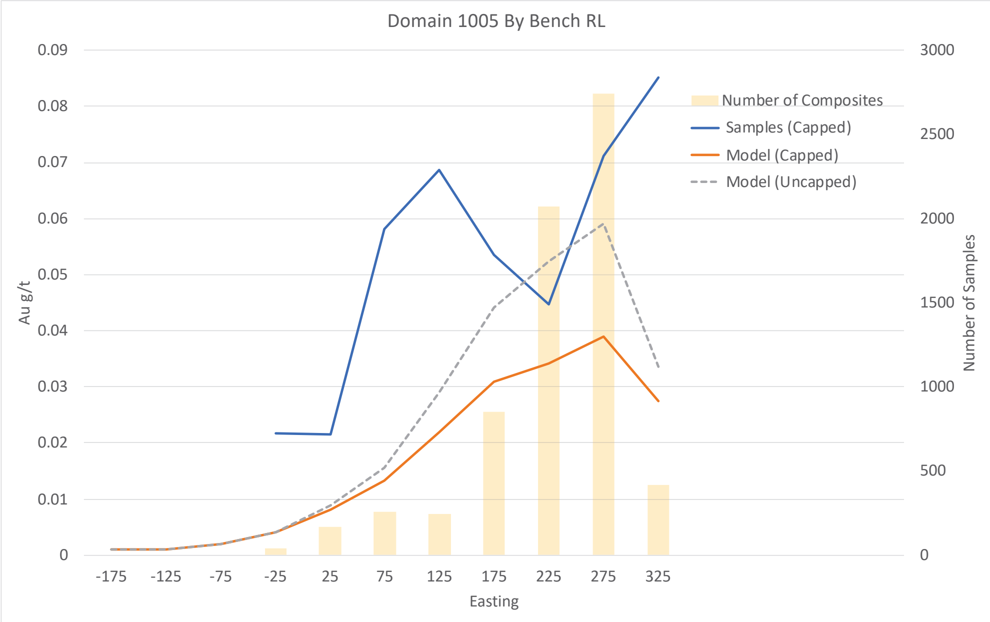
Appendix J Swath Plots (Plans)



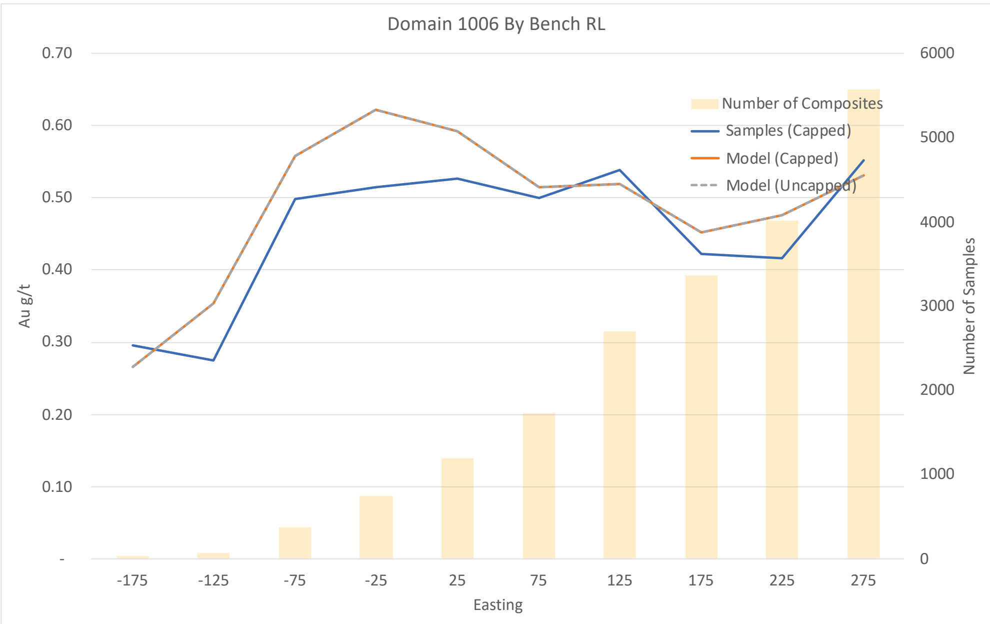



Domain 1003 By Bench RL
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
Number of Samples
Easting
4.50
4.00
3.50
3.00
2.50
2.00
1.50
1.00
0.50
-
700
600
500
400
300
200
100
0
125
175
225
275
325

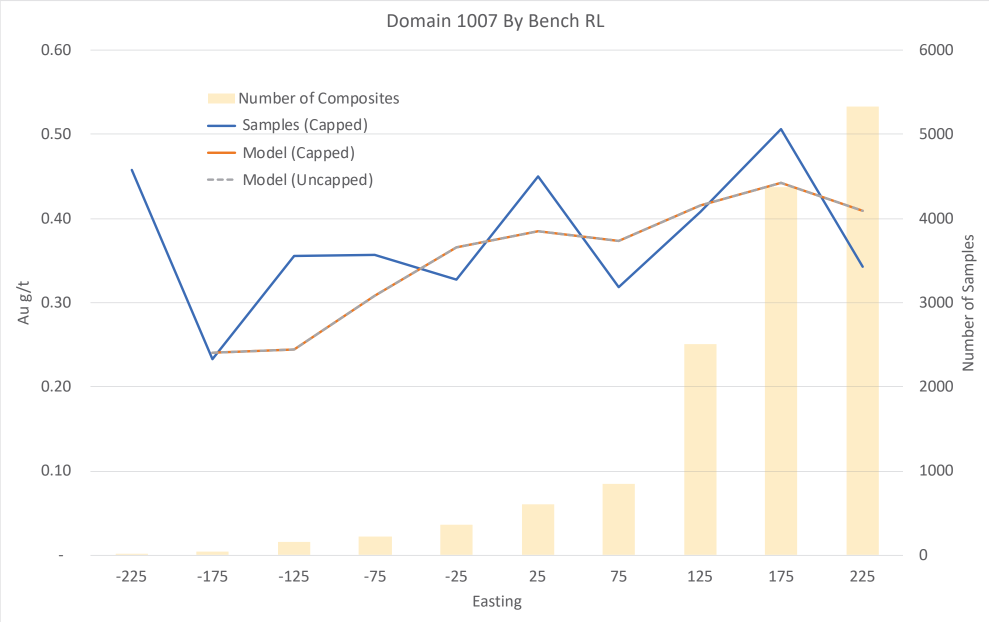

Domain 1004 By Bench RL
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
Number of Samples
Easting
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
4500
4000
3500
3000
2500
2000
1500
1000
500
0
-175
-125
-75
-25
25
75
125
175
225
275
325



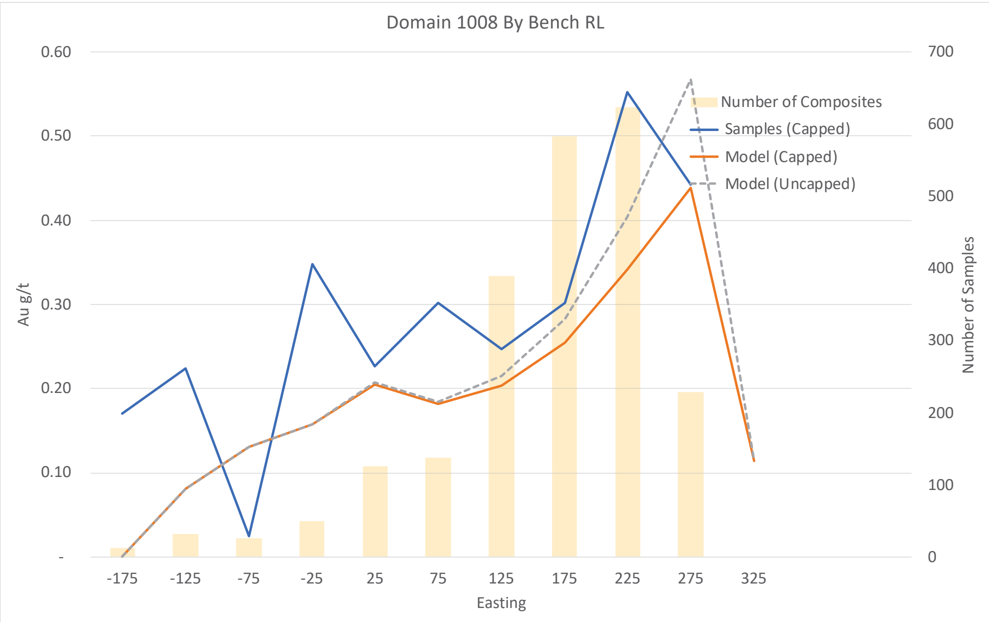
Domain 1005 By Bench RL
Au g/t
0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Number of Samples
3000
2500
2000
1500
1000
500
0
-175
-125
-75
-25
25
75
125
175
225
275
325
Easting
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)



Domain 1006 By Bench RL
Au g/t
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
Number of Samples
6000
5000
4000
3000
2000
1000
0
-175
-125
-75
-25
25
75
125
175
225
275
Easting
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)

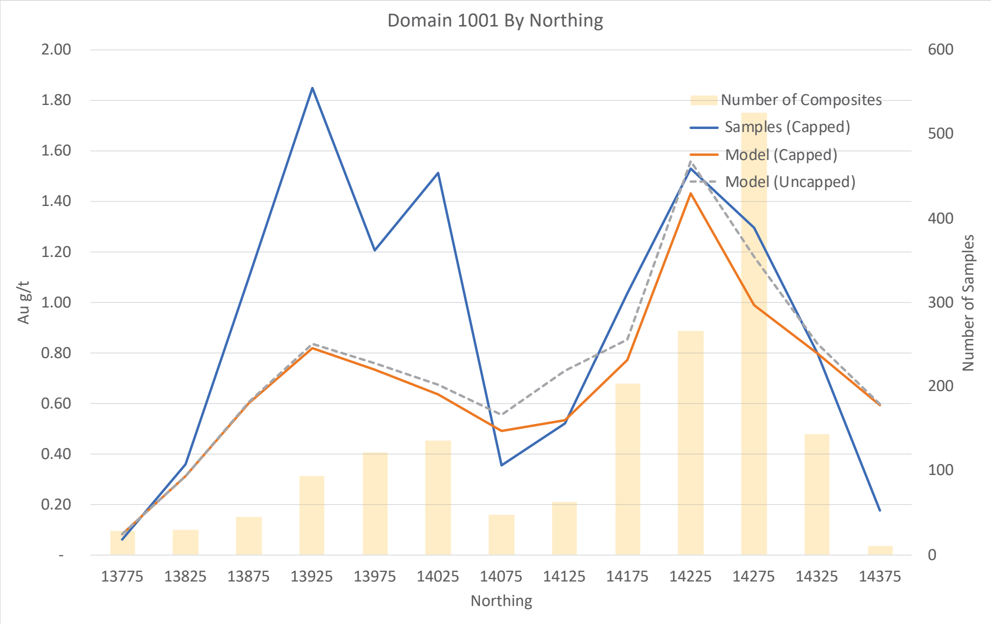


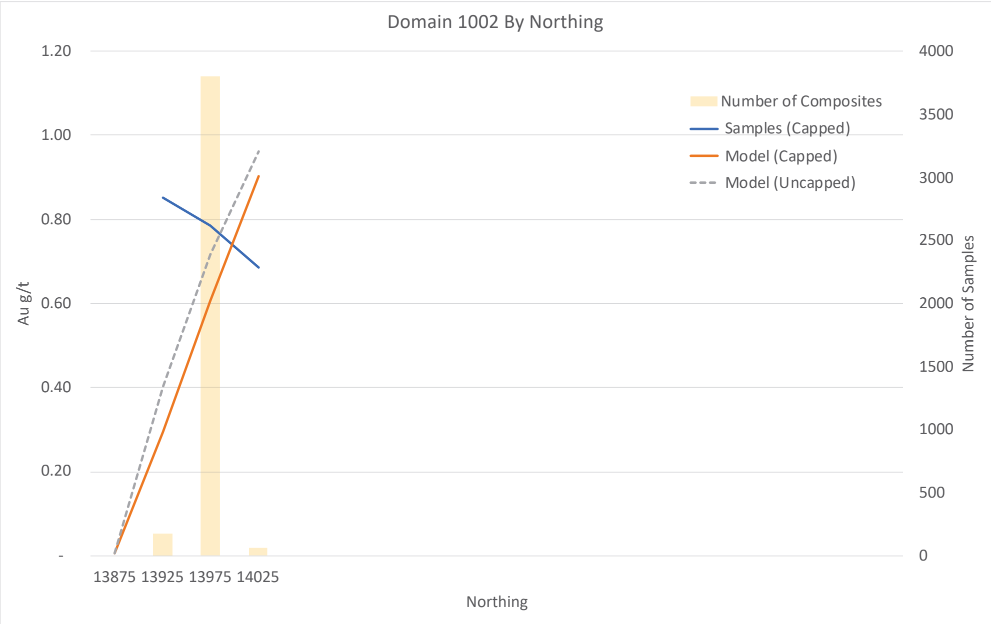
Domain 1007 By Bench RL
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
0.60
0.50
0.40
0.30
0.20
0.10
-
Number of Samples
6000
5000
4000
3000
2000
1000
0
-225
-175
-125
-75
-25
25
75
125
175
225
Easting

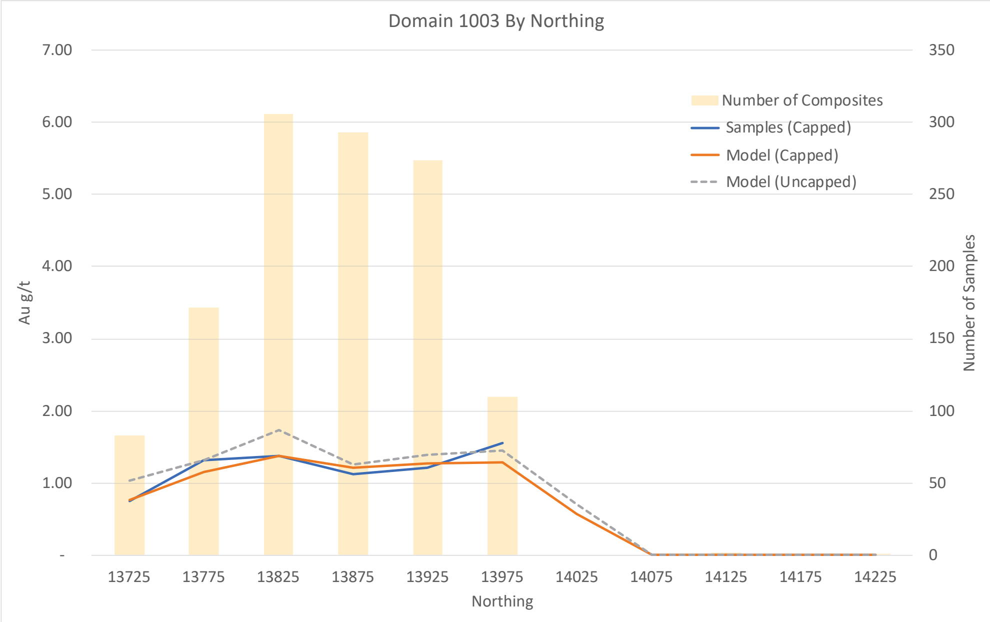


Domain 1008 By Bench RL
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
0.60
0.50
0.40
0.30
0.20
0.10
-
Number of Samples
700
600
500
400
300
200
100
0
-175
-125
-75
-25
25
75
125
175
225
275
325
Easting

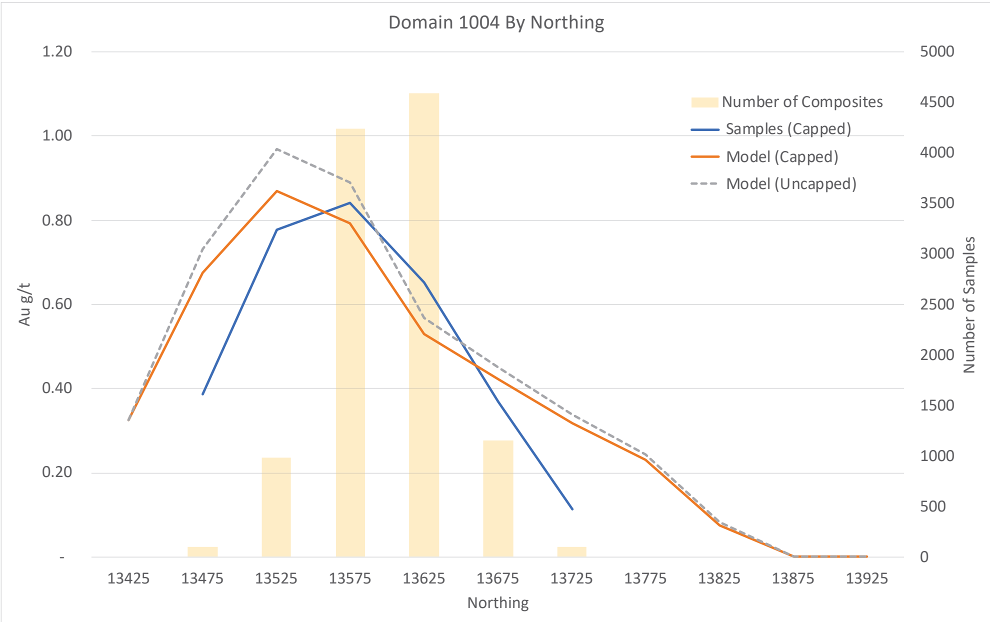
Appendix K Swath Plot – Northings



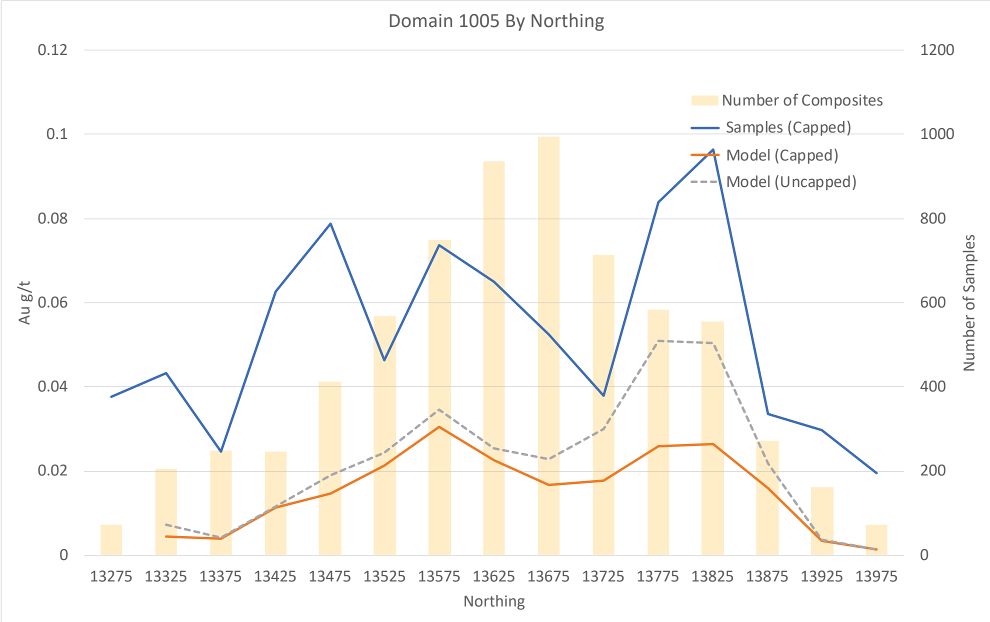


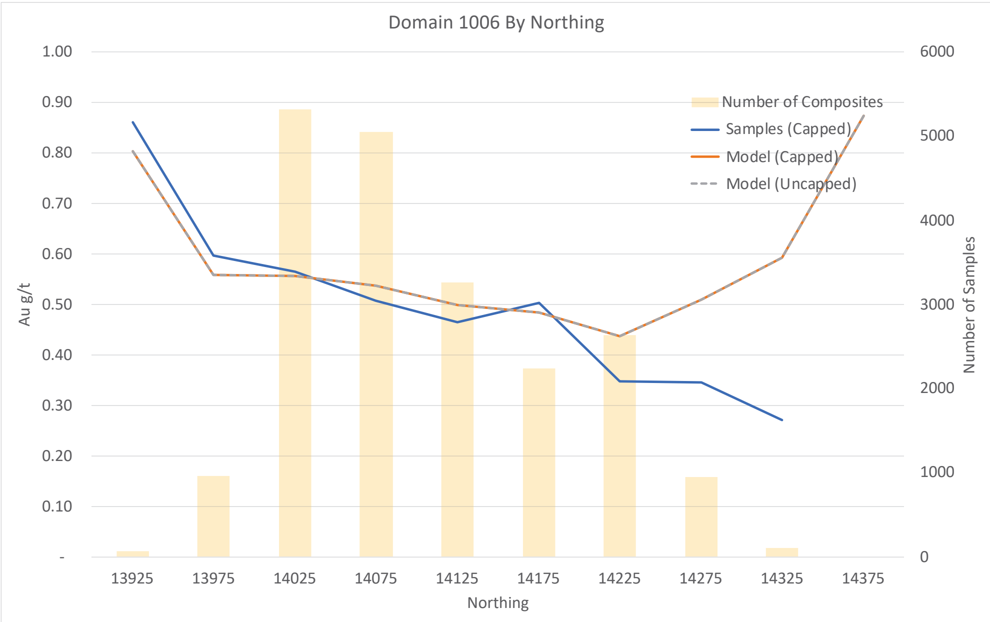


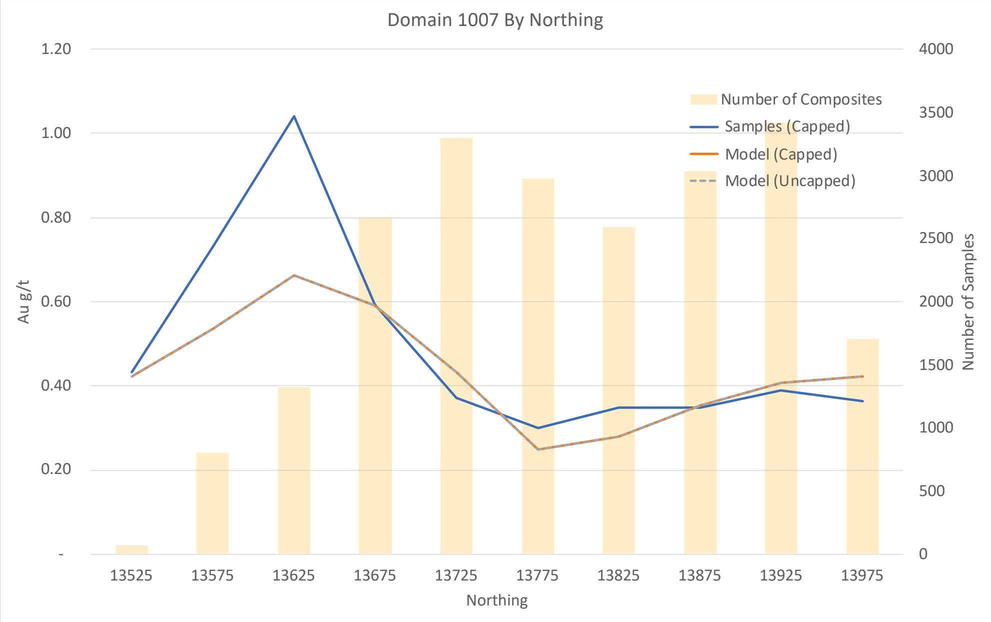
Domain 1003 By Northing
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
Number of Samples
7.00
6.00
5.00
4.00
3.00
2.00
1.00
-
350
300
250
200
150
100
50
0
13725
13775
13825
13875
13925
13975
14025
14075
14125
14175
14225
Northing

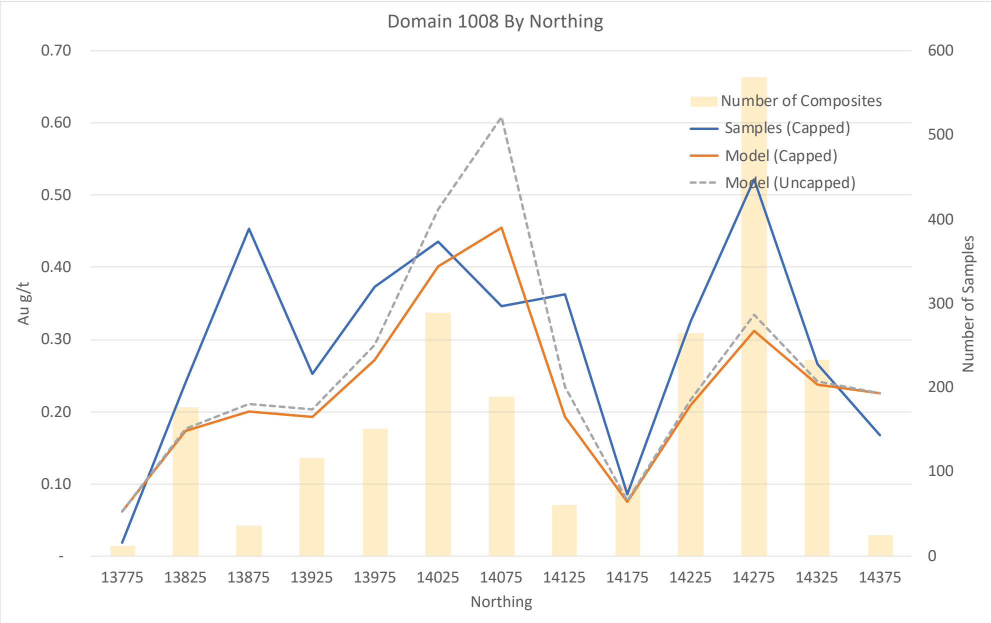


Domain 1004 By Northing
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
Au g/t
Number of Samples
1.20
1.00
0.80
0.60
0.40
0.20
-
5000
4500
4000
3500
3000
2500
2000
1500
1000
500
0
13425
13475
13525
13575
13625
13675
13725
13775
13825
13875
13925
Northing


Domain 1005 By Northing
Au g/t
Number of Samples
0.12
0.1
0.08
0.06
0.04
0.02
0
1200
1000
800
600
400
200
0
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
13275 13325 13375 13425 13475 13525 13575 13625 13675 13725 13775 13825 13875 13925 13975
Northing

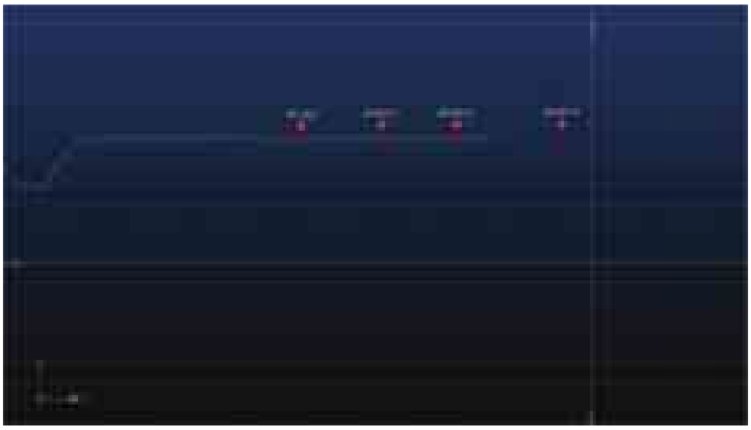

Domain 1006 By Northing
Au g/t
Number of Samples
1.00
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
6000
5000
4000
3000
2000
1000
0
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
13925 13975 14025 14075 14125 14175 14225 14275 14325 14375
Northing


Domain 1007 By Northing
1.20
1.00
0.80
0.60
0.40
0.20
-
Au g/t
4000
3500
3000
2500
2000
1500
1000
500
0
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
13525
13575
13625
13675
13725
13775
13825
13875
13925
13975
Northing


Domain 1008 By Northing
0.70
0.60
0.50
0.40
0.30
0.20
0.10
-
Au g/t
600
500
400
300
200
100
0
Number of Samples
Number of Composites
Samples (Capped)
Model (Capped)
Model (Uncapped)
13775
13825
13875
13925
13975
14025
14075
14125
14175
14225
14275
14325
14375
Northing

Appendix L Holes with suspect collars

BODD15	BODP7
BODD16	BODP8
BODD17	BODP9
BODP1	TRC052
BODP10	TRC074
BODP11	TRC075
BODP18	TRC076
BODP2	TRC077
BODP3	TRC079
BODP4	TRC080
BODP5	TRC081
BODP6	TRC082



Appendix M Scott Dunham – Brief CV

May 2017 – Present

SD2 Pty Ltd Principal Consultant and Director

- Resource estimation
- Resource audits and reviews
- Due Diligence investigations
- Reconciliation and grade control
- Variability and uncertainty studies
- Operational performance assessments
- Geometallurgical studies
- Training and professional development

March 2016 – October 2016

CRC ORE Ltd - Program Coordinator

- Research program coordination
- Foster collaboration between miners, METS and researchers
- Heterogeneity modelling and research
- Sensor, sampling and material evaluation adoption methodologies

August 2006 – February 2016

QG Australia Pty Ltd - Managing Director and Senior Principal Consultant

- Resource consulting including estimation, review/audit, advisory services
- Reconciliation and grade control
- Geometallurgical consulting
- Training and professional development

2004 – 2006

Newcrest Mining Limited – Technical Services Manager

2001 – 2004

WMC Resources Limited – Planning and Development Manager, Geology Manager

1998 – 2001

AMC Consultants – Senior Geologist

1994 – 1998

RGC Tasmania – Geology Manager Henty Gold Mine

1989 – 1994

Renison Goldfields Consolidated - Senior Geologist Renison Tin Mine

1987 – 1989

Mt Isa Mines Limited – Mine Geologist.

Ravenswood Gold

Buck Reef West Mineral Resource Estimate
April 2020

FINAL

Project Code:	**RAV002003**
Report Date:	**27 November 2020**
Effective Date:	**30 September 2020**
Author:	**Scott Dunham**

The conclusions and recommendations expressed in this report represent the opinions of the author(s) based on the data available to them. The opinions and recommendations provided from this information are in response to a request from the client and no liability is accepted for commercial decisions or actions resulting from them.

Release Date Addendum

Buck Reef West Mineral Resource Report – 27 November 2020

SD2 Pty Ltd (SD2) completed a Mineral Resource Estimate for the Buck Reef West gold deposit in April 2020. This addendum is an addition to the then published Mineral Resource technical report outlining SD2's analysis of changes at Buck Reef West between 20 April 2020 and 30 September 2020 (the 'effective date') and SD2's opinion on the materiality of any changes identified during that period.

Changes Potentially Effecting the Mineral Resource

The Buck Reef West deposit is part of the greater Ravenswood Gold Pty Ltd (Ravenswood) Mineral Resource base. Ravenswood are currently in the process of developing an open pit mining operation to extract the Buck Reef West Mineral Resource. In the period between 20 April 2020 and 30 September 2020 the following resource estimation related activities occurred:

1. Development activities focused on establishing and refurbishing the pre-existing mine camp, ore treatment plant and on procurement and delivery of the new mining fleet;
2. Production was restricted to processing of historical sub-grade stockpiles (not included in this report);
3. No surface mining took place. The topographic surface for 20 April 2020 and 30 September 2020 is identical;
4. Forty-two (42) new drill holes were completed into the mineralisation for a total of 8,827m. This represents a 6% increase in the number of holes and a 7% increase in the drilled metres compared to the data available at 20 April 2020.

Of these four activities, only the additional drilling has the potential for material impact on the quality and quantity of the estimated Mineral Resource.

Materiality Checks

SD2 reviewed the additional 42 drill holes completed between 20 April 2020 and 30 September 2020 (Figure 1).

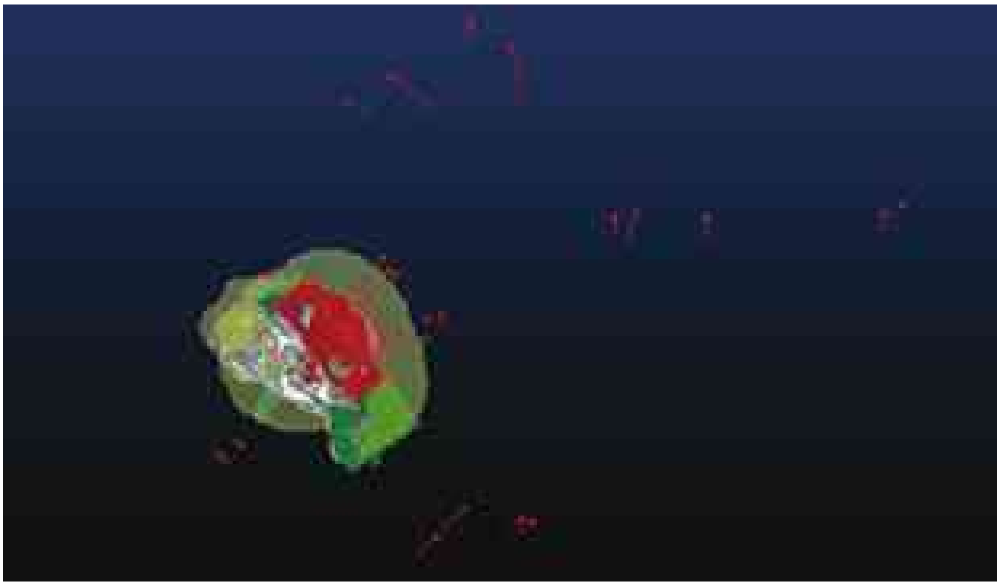


Figure 1. Location of holes drilled between April 2020 and September 2020.

Thirteen (13) of the new drill holes were targeted at areas more than 300m from the crest of the optimised pit shell used to report the Buck Reef West resource. These holes are not considered material to the April 2020 estimate and their inclusion in the available data will not impact on the local or global resource estimate. The remaining 29 holes were focused on the northwest portion of the reported Mineral Resource, targeting potential depth extensions and in-filling regions of low drill density to improve the Mineral Resource classification confidence.

Fifteen (15) of the 29 holes drilled in the footprint of the reported Mineral Resource estimate did not have all sample assays available as at 30 September 2020. Inspection of the remaining 14 holes with complete data sets indicated no material difference in the grade tenor, width of mineralisation or location of the mineralised zones. Some holes (e.g., BRRD469, BRRD470, BRRD472, BRRD474, BRRD483) indicate sporadic medium to high grade mineralisation to the hangingwall of the Duke lodes; however the level of geological understanding of this potential new line of mineralisation was incomplete as at 30 September. Therefore, these holes, while promising, are not considered material to the global resource estimate.

The impact of the holes drilled between April 2020 and September 2020 is summarised in Table 1. The 14 new holes with potential to impact on the Mineral Resource estimate are consistent with the April 2020 estimate. These 14 holes represent a 2% increase in the number of drill holes in the mineralisation.

Table 1. Materiality of new drilling.

Hole category	Number of Holes	Material to Estimate	Comment
Greater than 300m from resource	13	No	Drilled to test exploration targets.
Information incomplete (e.g. awaiting assays)	15	No	Geology is consistent with interpretation. No grade data available to inform resource estimate
Intersected reported resource	14	No	Intersections are consistent with April 2020 estimate (grade, width, location)
Total Number of new holes	**42**		

Conclusions

In SD2's opinion the Buck Reef West April 2020 Mineral Resource estimate is suitable for reporting as at 30 September 2020. No mining activities have occurred at Buck Reef West and therefore the resource estimate does not need to be depleted to allow for extraction. Similarly, in the absence of mining there are no reconciliation data available or geological observations to justify altering the interpretation of the resource.

While Ravenswood drilled an additional 42 holes between April 2020 and September 2020, only 14 of these holes were completed with full assay data and targeted areas within the reported resource. These 14 additional holes confirmed the grade tenor, width and location of the interpreted mineralisation and therefore, in SD2's opinion, they do not materially alter the quality or quantity of the estimate.

Tabulated Mineral Resources as at 30 September 2020.

Buck Reef West Mineral Resource Statement as at 30 September 2020.

Open Pit Above 0.3 g/t	Tonnes	Grade (Au g/t)	Ounces
Measured	-	-	-
Indicated	25,050,000	1.03	833,000
Inferred	1,170,000	1.11	42,000
Total Open Pit Resource	**26,220,000**	**1.04**	**875,000**

Underground Above 3.5 g/t	Tonnes	Grade (Au g/t)	Ounces
Measured	-	-	-
Indicated	91,000	4.97	14,600
Inferred	65,000	4.71	9,800
Total Underground Resource	**156,000**	**4.86**	**24,400**

Total Measured and Indicated	**25,141,000**	**1.11**	**899,400**

Open put resources above AUD3800 shell.
Underground resources within continuous zones >2.0m wide and > 1,000m3
Model: BRW200410.bm
Rounding errors may occur

This resource statement is based on an estimate of the Buck Reef West mineralisation completed 20 April 2020 and an assessment of materiality of changes between April 2020 and September 2020. The complete technical report for the estimate is included in the following documentation.

Executive Summary

The Buck Reef West gold deposit is part of the Ravenswood gold mine in north Queensland. An updated mineral resource estimate was developed incorporating the most recent drilling and geological information as at 18 March 2020. This estimate supersedes previous estimates for the Buck Reef West mineralisation.

The April 2020 Buck Reef West resource model[1] is based on similar geology interpretation and methodology to the previous Resolute Mining Limited 2018 estimate. This a reversion to ordinary kriging as the preferred estimation methodology. This 2020 estimate has simplified the estimation approach and it has been developed to be readily updateable and to be suitable as a basis for future grade control modelling methods.

The resource is classified under the JORC Code (2012) as Indicated and Inferred. There is no Measured resource at Buck Reef West. Classification was on the basis of sample spacing, geological confidence and a range of estimation quality metrics including the block-to-sample distance and configuration.

The Buck Reef West mineral resource estimate includes both open pit and underground potential. Open pit resources are reported at a 0.3 g/t Au cut-off above an AUD4000 optimised pit shell. Underground resources are reported within continuous zones greater than 2m wide and more than 1,000m3 at a cut-off of 3.5 g/t in close proximity to the pit shell. The resource is reported below the topographic surface as at 12 March 2020 and excludes known mined voids from historic workings.

[1] Brwestimate200410.bm.dm completed and released 20 April 2020.

Buck Reef West April 2020 Mineral Resource Statement

Open Pit Above 0.3 g/t	Tonnes	Grade (Au g/t)	Ounces
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1. Introduction and Scope

Ravenswood Gold Pty Ltd (Ravenswood, or RAV) recently acquired the Ravenswood gold mine located 130km by road from Townsville and 90km from Charters Towers. Ravenswood mine is not currently operating; however a feasibility study outlining the mining opportunity and steps required to start production formed a central part of the sale and acquisition process. As part of the acquisition process SD2 Pty Ltd (SD2) was engaged to review the resource model and estimate for Ravenswood and consider the suitability of the model for future mine planning. This investigation identified risks with the resource estimate, mainly around the grade-tonnage distribution. Consequently, RAV engaged SD2 to re-estimate the mineral resources at Ravenswood.

This report outlines the resource estimation approach adopted for the Buck Reef West (BRW) deposit, the first of two major resources in the area.

1.1 Location and History

The Ravenswood gold mine is one of a number of gold deposits in the Ravenswood-Lolworth Province of northeast Queensland. Alluvial gold was discovered at Ravenswood in 1868 followed by the discovery of oxidised gold-bearing quartz reefs. By 1872 most of the near-surface oxide mineralisation had been depleted (McIntosh et al. 1995) and only the refractory sulphide-associated mineralisation remained. A second phase of production started with the formation of the New Ravenswood Company in 1896 and focused on extracting this sulphide-associated gold from lodes and veins including the Duke of Edinburgh, General Grant, Sunset, London, Mellaneur, Shelmallier (MSA) and Black Jack systems. The majority of gold was from the Sunset lode which produced 208,949 oz from a 45° dipping vein to a depth of 200m below surface (Collett et al., 1998). Production decreased rapidly after 1912 due to exhaustion of the Sunset Lode, an extended miner's strike and the impact of World War 1.

There was limited activity at Ravenswood from 1917 to 1980. Silver was produced from the nearby (1.6km north) Totley mine in the 1950s; otherwise production was limited to minor underground extensions and few drill holes. In the early 1980's The North Queensland company reprocessed several old mullock dumps and tails dams. In 1985, MIM Exploration Pty Ltd (MIM) began exploring the Ravenswood district and, following early success MIM's subsidiary Carpentaria Gold (CG) began open pit production at Bucks Reef West (BRW) , Slaughter Yard Creek (SYC) and OCA in 1987. The operation commenced as a heap leach (250 Ktpa) and small (100 Ktpa) CIL operation before the construction of a 2.4 Mtpa CIL plant in 1993. This plant was expanded to 5.5 Mtpa in 2000 to enable treatment of production from the Sarsfield and Nolan's open pits (Lisoweic, 2009).

Production at the Sarsfield open pit was completed in 2009 and the ore treatment plant was de-rated to 1.5 Mtpa in 2011 while focus switched to the nearby Mt Wright underground operation. There was a hiatus at Ravenswood until 2016 when the Nolan's East open pit commenced. As of 2020 production is limited to treating old stockpiles and dumps until the plant is refurbished and approval given to recommence operations at Buck Reef West.

Historically the Ravenswood area has produced approximately 2.4Moz at an overall average grade of 1.7g/t. (Table 2). Excluding production prior to 1987 the area produced 1.5Mz at a grade of 1.1g/t.

Year	Operation	Recorded Production	Ounces	Source
Pre 1987	Lode mineralisation across entire field.	No tonnes and grade reported. Estimated grades reported as 30 g/t (Lisowiec, 2009).	900-950,000	Collett et al. 1998. Lisowiec, 2009.
1987 – 1990	SYC (pit)	526,000 @ 2.7 g/t	45,700	Collett et al. 1998
1987 – 1989	OCA (pit)	290,000 @ 3.4 g/t	31,700	Collett et al. 1998
1988 – 1991	BRW (pit)	160,000 @ 2.8 g/t	14,400	Collett et al. 1998
1991	OCA (ug)	149,000 @ 4.1 g/t	19,600	Collett et al. 1998
1990	Area 4 (pit)	50,000 @ 2.4 g/t	3,900	Collett et al. 1998
1988 -1991	Area 5 (pit)	260,000 @ 2.4 g/t	20,000	Collett et al. 1998
1990 – 1991	MSA (pit)	48,000 @ 3.5 g/t	5,400	Collett et al. 1998
1992 – 1993	Area 2 (ug)	174,000 @ 10.1 g/t	56,500	Collett et al. 1998
1993 – 1996	Nolans (pit)	4,100,000 @ 1.25 g/t	164,800	Collett et al. 1998
2003 – 2005	BRW (ug)	376,000 @ 4.0 g/t	48,400	Lim et al., 2018
2000 – 2003	Sarsfield (pit)	3,900,000 @ 1.24 g/t	155,500	Haoma Mining Annual Report 2003
2004 – 2009	Sarsfield (pit) Note, introduction of MIK for resource estimation	33,490,000 @ 0.91 g/t	980,000	Lim et al., 2018
Total Recorded Production		**44.3Mt @ 1.7 g/t**	**2,400– 2,450,000**	
Open Pit Only		**40.0Mt @ 1.7 g/t**	**2,150-2,200,000**	
Pits After 2000		**37.4 Mt @ 0.9 g/t**	**1,100,000**	

Table 2. Historic Production (Ravenswood).

SD2 note the differences in average mined grade between production at Sarsfield/Nolans in 1993-1996 (1.25 g/t), 200-2003 (1.24 g/t) and 2004-2009 (0.91 g/t). While it is not possible to directly relate the decrease in grade to a single cause, it is notable that the 2004-2009 production was carried out by the new operator (Resolute). At the time of Resolute's acquisition of Ravenswood from Xstrata, there was a stated plan to improve operational performance by reducing strip ratio and changing grade control practices (Resolute 2004 Annual Report; Figure 2).

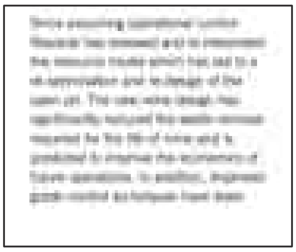

Figure 2. Extract form Resolute 2004 Annual Report.

It is possible this change in strategy was linked to the adoption of multiple indicator kriging (MIK) as the resource estimation and grade control estimation methodology and the decrease in grade may reflect a corresponding decrease in selectivity. Much of the detail of estimation and operational practices in and around the 2003-2004 period have been lost following a legal dispute between Haoma Mining and MIM Holding Limited and the subsequent sale of Haoma's interest in the Nolans open pit to MIM which was shortly followed by Xstrata's acquisition of MIM itself.

1.2 Work Completed

The April 2020 Buck Reef West mineral resource estimate was a complete revision of the previous modelling reported by Resolute Mining Limited. Consequently SD2 completed a comprehensive review of past practices, the data quality and previous estimates as part of developing the new estimation strategy. In addition to the activities completed in the Due Diligence study, SD2's work included the following:

- Review of the geology database and request for an extract covering the Buck Reef West mineralisation;
- Review of surfaces required for the estimate including topography and voids;
- Review of the operational grid system and the history of grid transformations;
- Review of drill hole collars against topography (where appropriate);
- Review of geological logging systems and results. Consideration of ways to best incorporate logging into the resource estimate;
- Review of bulk density data and consideration of its suitability for resource estimation;
- Review of sampling and assay data with a particular focus on high-grade samples occurring adjacent to unsampled intervals;
- Review of quality control performance data collected at the time of drilling and sampling;
- Collation of all files and metadata used for previous estimates where possible;
- Review of the structural geology of the Ravenswood district and BRW specifically. Consideration of how the geological structure influences the resource model and estimate;
- Review and update of the geological interpretation. Development of domains suitable for resource estimation;
- Estimate the BRW mineral resource and document the estimation process and results (this document);
- Prepare the April 2020 estimate for use in mine planning; and
- Prepare a geology/resource risk assessment and report.

All of the work completed for the BRW mineral resource estimate was carried out under the guidelines of the JORC Code[2] (2012 edition) reporting framework.

1.3 Previous Estimates

There have been three recent estimates of the Buck Reef West mineral resource, one developed in 2018 and two developed in 2019. These three estimates all adopted different estimation strategies (Table 3).

Name	Date	Practitioner	Estimation Method	Comment
BRW_2018	June 2018	Resolute (A. Pedersen)	Ordinary and simple kriging in 2D and 3D rotated space	Complex estimate based on interpreted mineralised lodes and indicator proportions for mineralisation outside of lodes
MPR_2019	August 2019	MPR Geological Consultants (J. Abbott)	Multiple indicator kriging (MIK) with broad domains defined by mineralisation trends	MIK estimate exhibits a left-shifted grade-tonnage distribution compared to other estimates reflecting the influence of higher-grade samples spreading into lower grade zones (based on the geological interpretation).
SD2_Sensitivity	September 2019	SD2 Pty Ltd	Ordinary kriging within domains developed from the MPR MIK estimate	Estimate designed to shift the MIK grade-tonnage curve to the right and reduce high-grade to low-grade interference patterns. Estimate also designed as a 'down-side' prediction for evaluation of the Ravenswood acquisition.
SD2_2020	April 2020	SD2 Pty Ltd	Ordinary kriging in 3D using dynamic anisotropy. Based on BRW_2018 domaining logic.	Simplified modelling approach applied to the same geological interpretation used in the BRW_2018 estimate. Higher-grades lying outside of interpreted lodes constrained by a 25% probability iso-surface of the 0.5 g/t Au grade indicator. Soft boundaries applied one-way from this iso-surface domain to background samples (i.e., the domain restrict tonnage but uses background samples as well as samples within the domain for estimation)

Table 3. Previous BRW Estimates.

As part of this report on the 2020 BRW mineral resource estimate, SD2 has compared these three previous estimates to the current estimate using identical reporting parameters as far as possible. The results of these comparisons are shown in Appendix G. In broad terms the estimates are similar (except for the SD2 Sensitivity model); however they differ in the grade-tonnage distribution above a range of cut-offs. This reflects the differences in estimation approaches.

[2] JORC Code – The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.

1.4 Changes in Methodology

The April 2020 Buck Reef West estimate is most similar to the BRW_2018 estimate by Pedersen. The estimation strategy uses the following steps:

5. Manual interpretation of lodes and structures for:
    a. Buck Reef Fault;
    b. Duke of Edinburgh (Duke) lodes;
    c. General Grant (Grant) lodes;
    d. Sunset lodes;
6. Development of a buffer zone around each of these lodes to encapsulate near-lode high grade samples;
7. Development of a domain representing 'proto-lodes' or zones of discontinuous mineralisation containing isolated high-grades outside of the lodes and buffer zones;
8. Estimation by ordinary kriging using dynamic anisotropy[3] and parameter developed for each domain;
9. Estimation using a combination of hard and one-way-soft contacts (section 5.6.2) based on domain statistics and estimation performance;
10. Assignment of the estimated domain average grade to blocks that remain unestimated after all permissible sample searches have been applied; and
11. For the background model, apply a broad search with a high number of samples required to provide a grade trend with high degree of smoothing.

The estimate was post-processed to remove any negative grades, and set appropriate flags to indicate rock/air interface, underground voids and maximum reporting volumes (section 5.7.)

1.5 Critical Risks

The Buck Reef West deposit is characterised by a high proportion of isolated, relatively high-grade samples that lie outside of the interpreted lode domains. These samples reflect the sporadic grade distribution at Buck Reef West outside of the main lode and Buck Reef Fault structures (refer section on geology). While these isolated high grades are 'real', modelling and estimating their spatial distribution is challenging and can have a large impact on the contained metal. At the current drill hole spacing these samples often appear discontinuous; however indicator modelling at a range of cut-offs implies that there may be additional 'proto-lodes' at Buck Reef West, zones or corridors of preferential mineralisation (Figure 6). This is consistent with the structural framework model developed by OreFind (2018) which suggests

[3] An approach that dynamically changes the search and (optionally) variogram model orientation on a block-by-block basis to align with the interpreted strike/dip/plunge of the mineralisation.

that Buck Reef West (and the wider Ravenswood district) form an orthorhombic system of vein and faults (Figure 9 and Figure 10).

2. Project Description

2.1 Site and Existing Infrastructure

The Ravenswood gold mine is accessed by sealed road from a turn-off on the Burdekin Falls Dam Road. The site sits adjacent to the historic Ravenswood township (population 200) and there are several heritage-listed structures in and around the district. Mining operations at Ravenswood ceased in 2009; however the ore treatment plant was used to treat production from the nearby Mt Wright underground mine and therefore is still operational. At present (April 2020) low grade stockpiles produced during mining of the Sarsfield-Nolans pit are being treated through the plant.

The site is well equipped with the requisite infrastructure for mining and ore treatment operations (Resolute, 2018). Power is supplied through existing connections to the state-wide grid (PowerLink) via the Ergon Energy distribution network. Water is supplied via a 20km pipeline from the Burdekin River and the site operates two surge dams to manage seasonal flow variations. Telecommunications are provided by Telstra and the site operates a dedicated frame-relay data link provided by Optus.

Other existing site infrastructure includes workshops and warehouses to service the ore treatment plant and mining fleet, offices, sewage treatment plant, on-site accommodation and messing. While much of this infrastructure will require upgrading over the life of the combined BRW and Sarsfield production, there are no known impediments preventing the provision.

The existing tailing storage facilities (TSF) are insufficient for full the currently planned production from BRW and the Sarsfield/Nolan pits. RAV are developing a tailings management strategy and have had several options developed for evaluation. Tails from processing of Sarsfield/Nolans low grade stockpiles are being dewatered and dry-stacked back in the Sarsfield open pit. Discussions with the Queensland Department of Environment and Science (DES) are on-going and SD2 is unaware of any impediment likely to prevent resolution of the tails storage requirements for Buck Reef West.

In SD2's opinion there are no infrastructure-related issues that would prevent production from the Buck Reef West deposit. Further information on the existing and planned infrastructure requirements is contained in the REP 200 feasibility study (Lim et al., 2018).

2.2 Tenements and Tenure

Ravenswood Gold took possession of a number of Mining Leases (ML), Mining Lease Applications (MLA) and Exploration Permits (EPM) as part of the acquisition of the Ravenswood operation in 2020. During the acquisition process (December 2019) Ravenswood engaged Hetherington Exploration and Mining Title Services Pty Ltd (Hetherington) to review the status of the acquired leases. Hetherington prepared a report (Martin, 2019) on the status of the leases based on information obtained from the Department of Natural Resources, Mines and Energy ("DNRME") My-Mines-0nline ("MMOL") database and other information as supplied by DNRME, the tenement holder/s (obtained via the digital data room) and the Department of Environment & Science ("DES").

The titles relating to Buck Reef West include:

- ML 1380 (expires 30 November 2034)
- ML 1412 (expires 31 January 2023)
- ML 1532 (expires 31 October 2027)
- ML 1722 (currently an infrastructure only ML that will be conditionally surrendered on the grant of MLA100172)
- MLA100145
- MLA100149
- MLA100147
- MLA100172

The status of these MLs and MLAs is summarised in Table 4(after Martin, 2019).

Tenement	Native Title	Holder/s	Status	Granted	Expiry	Minerals	Area (Ha)	Security Deposit	Financial Assurance
ML 1380	Section 31	CG	Granted	28-11-74	30-11-34	Gold, copper, lead, molybdenum, silver, zinc	60.79 (total) 58.59 (surface)	Nil	(note 2)
ML 1412	Pre NTA	CG	Granted	15-01-81	31-01-23	Gold, bismuth, cobalt, copper, silver, tungsten, zinc	2.024 (whole)	Nil	(note 2)
ML 1532	Pre NTA	CG	Granted	24-10-85	31-10-27	Antimony, arsenic, bismuth, copper, gold, lead, silver, zinc	0.2023 (whole)	Nil	(note 2)
ML 1722	N/A	CG	Terminated (note 1)	05-09-91	14-05-19	N/A	N/A	Nil	N/A
MLA 100145	Section 31	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	1.03 (total) 0.34 (surface)	Nil	(note 2)
MLA 100147	Exclusive	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	0.2023 (whole)	Nil	(note 2)
MLA 100149	Exclusive	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	1.3 (whole)	Nil	(note 2)
MLA 100172	Section 31	CG	Granted	13-05-19	31-05-39	Gold, copper, lead, molybdenum, silver, zinc	58.46 (whole)	Nil	(note 2)

Note 1 - ML 1722 is not current. This ML was conditionally surrendered in favour of ML 10172 which was granted on 13-5-2019
Note 2 - All of these ML's are currently included in EPML00979013. Carpentaria have advised that they have recently paid an amount of $280,000 into the financial provisioning scheme. This is assumed to be an annual payment.

Table 4. Status of BRW tenements (After Martin, 2019).

At the time of Martin's report the tenement holder was Carpentaria Gold Pty Ltd (CG). SD2 understand that these leases were transferred to RAV at the settlement of the acquisition transaction (1 April 2020). The BRW MLs and MLAs include gold in the list of exploitable minerals and metals. The rent for all BRW leases has been paid to 31-8-2020. The leases are all covered by a site specific Environmental Agreement (EPML00979013).

Martin (2019) concluded that the BRW MLs and MLAs appear to be in good standing with two caveats:

1. Local government authority (council) rates for some recently granted leases had not been paid.
2. Hetherington has relied on information provided by CA as the lease holder and therefore recommended a direct search application to DNRME to verify lease status and conditions.

In SD2's opinion there are no material issues related to tenement status and ownership. Ravenswood Gold own 100% of the listed titles.

2.3 Grid System

Multiple grid systems have been used at Ravenswood, reflecting the long production history and variable lode orientations. The BRW grid (known as the A45 grid) has local north oriented to bearing 030° magnetic. Coordinates are truncated and lie between 12,000 and 14,000 in both northing and easting. Complicating matters further there is a 32.813 translation in elevation between the local Nolans grid and other grids in the field. A list of the different grids and their translations was compiled by Kelly & Partners Consulting Surveyors in 1993. This list further validated in 2004. Figure 3 illustrates the differences in orientation between a selection of the known grids and Table 5 shows the 2-point rotation and translation data for conversions between local, AGD84_Z55 and MGA94 mapping grids.

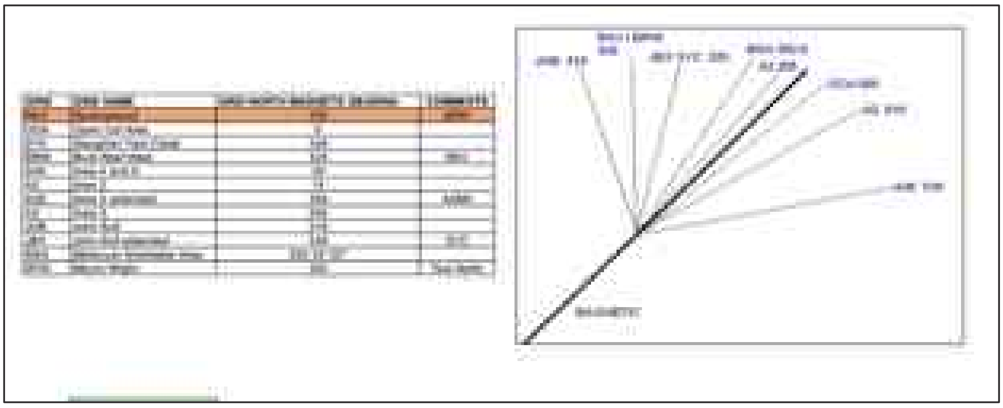

Figure 3. Ravenswood Local Grids.

Table 5. Ravenswood grid twin points.

SD2 note that the BRW local grid is not well aligned with either the Buck Reef Fault or the main lode structures. The lodes dip approximately towards 050 on the local grid and the Buck Reef Fault strikes along 025 local grid. The angular difference between the main structures and the grid introduces an unnecessary complication in the resource modelling requiring either rotated models[4] or acceptance of partial blocks within the interpreted geology. SD2 has chosen the later strategy (section 5.5). Importantly, the orientations of BRW structures are better aligned with MGA84, the Australian national mapping grid system. At the risk of introducing further grid-related complications, SD2 strongly recommend RAV investigate adopting MGA84 (or an equivalent with truncated coordinated) across the Ravenswood operation.

2.4 Site Visit

Scott Dunham completed two site visits to Ravenswood. The first during the due diligence study as part of the acquisition assessment team (August 2019) and the second in January 2020.

[4] Rotated models typically allow blocks to be aligned at any angle compared to the grid system. While this is useful in reducing volumetric errors and block numbers, there is no consistency between different software packages and rotated models in one general mining package are generally incompatible with another. This limits their usefulness.

3. Geology and Mineralisation

3.1 Regional Geology

The geology of the Ravenswood district has been described by several authors including Lim et al. (2018), Derham (2014), Berry et al. (1992), and Switzer (2000). The Ravenswood gold deposits lie within the Lolworth-Ravenswood block of the Charters Towers Province, a poorly exposed part of the regional Thomson Fold Belt (Figure 4). The Lolworth-Ravenswood Block comprises remnant amphibolite-grade metamorphic rocks intruded by an elongate east-west Ordovician-Silurian batholith (the Ravenswood Batholith) with an outcrop of 150km by 220km. The batholith is bound to the south by the Cambrian-Ordovician Seventy Mile Range Group of the Thalanga Province and the Devonian-Carboniferous Drummond Basin. The Devonian Burdekin Basin forms a northern boundary and to the east by the Carboniferous-Permian Coastal Range Igneous Complex, Permian-Triassic Bowen Basin and Quaternary sediments, and to the west by Permian-Jurassic basins such as the Galilee, and Tertiary and younger cover sequences.

The Ravenswood Batholith intruded the basement Cape River Province and Seventy Mile Range Group in three phases:

1. Hornblende and/or biotite bearing I-type granitoids ranging from granite to lesser extent gabbro intruded during the early-to-mid Ordovician contemporaneously with the formation of elements of both the Cape River Province and Seventy Mile Range Group. Minor S-type, peraluminous granites of a similar age have also been identified in the Ravenswood Batholith;
2. The bulk of the batholith (>60%) formed during the development of the Mid-Silurian to mid-Devonian Pama Igneous Complex consisting of undeformed I-type hornblende-biotite bearing granites and granodiorite with lesser s type granitoids. These intrusions were coeval with a regional northeast-southwest compression (D4) and gold mineralisation at both Charters Towers and Hadleigh's Castle, west of Ravenswood; and
3. The late Carboniferous to early Permian Kennedy Igneous Association, a group of high K calc-alkaline intrusions with a diverse range of I, S and lesser A type magmas. Rocks of the Kennedy Igneous Association increase in abundance to the south of the Ravenswood Batholith and typically form localised, ring-fracture controlled stocks and/or trachytic plugs with little preserved deformation. *This intrusive phase is likely associated with gold mineralisation at Ravenswood.*

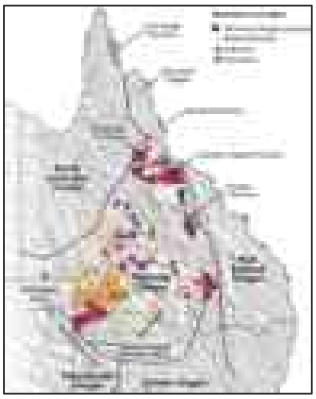

Figure 4. Location of Charters Towers Province

The region is characterised by east-west structures such as the Alex Hill Shear Zone, a 2-5km wide east-west shear zone extending over 100km across the northern edge of the Ravenswood Batholith (Figure 5) and the Mosgardies Shear Zone, a less continuous east-west mylonite zone extending from Ravenswood some 30km west to the Rochford area. The regional structural geology is considered to have formed in seven recognisable events defined as D1 to D7 (Kruezer 2005). Across the district, gold mineralisation is associated with D5 (Charters Towers) and D7 (Ravenswood). The seven deformation events include:

D1: Development of poorly preserved SE striking foliations in the Cape River and Charters Towers Metamorphics as a result of NE-SW compression.

D2: NW striking platy foliations formed during crustal extension and deposition of the Seventy Mile Range Group, synchronous with intrusion of some Ordovician Granitoids.

D3: E-W trending transcurrent shear zones developed as transfer faults or lateral ramps related to eastward progressing accretion (e.g. Alex Hill Shear Zone). Localised N-S compression related to the intrusion of Ordovician – Silurian granitoids into E-W shear zones.

D4: Development of NW-striking structures with both steep-pitching lineations and transcurrent fabrics (e.g. Burdekin River Lineament) as a result of NE-SW compression. Synchronous intrusion of Silurian-Devonian plutons into active transcurrent faults.

D5: Middle Devonian NE-SW compression concurrent with hydrothermal alteration and gold mineralisation at Charters Towers and Hadleigh's Castle.

D6: NW-SE compression producing sinistral movement on the Jessop Ck Fault and dextral movement on the Plumwood-Connolly Fault.

D7: Carboniferous E-W to NW-SE compression concurrent with rhyolitic magmatism, and alteration-gold mineralisation at Ravenswood and Mt Wright.

Figure 5. Lithostratigraphy of the Ravenswood District. (AUSGIN Geoscience Portal).

In summary, the regional geology suggests that the Ravenswood gold mineralisation formed during D7 deformation associated with the late Carboniferous to early Permian Kennedy Igneous Association. The regional structural setting at the time of mineralisation included east-west to northwest-southeast compression with a likely corresponding north-south to northeast-southwest dilation.

3.2 Deposit Geology and Structure

The Buck Reef West mineralisation is hosted by the Jessop Creek Tonalite[5], a variable light grey phaneritic to weakly hornblende-phyric medium to coarse grained tonalite. In the BRW area the Jessop Creek Tonalite comprises diorite, quartz diorite and minor gabbro. Boundaries between these units vary from sharp to indistinct and often show complex relationships, including stoping, xenoltihs and irregular dykes. The Jessop Creek Tonalite displays variable degrees of alteration with primary biotite weakly to moderately altered to chlorite and epidote while hornblende is only weakly altered to chlorite in most cases. Alteration in concentrated along grain margins and particularly cleavage plans of biotite. No association

[5] Tonalite – A granitoid (a coarse grained igneous rock with <90% mafics; felsic minerals are composed mostly of quartz (20-60%), Kspar (alkali-feldspar) and plagioclase), where plagioclase is >90% of the total feldspar on the QAPF diagram (quartz - alkali feldspar – plagioclase feldspar – feldspathoids or foids)

between the host lithology and gold mineralisation has been established other than it is a competent host that was amenable to the development of several styles of quartz-sulphide-veins.

The local structural geology is complex. The dominant structure is the Buck Reef Fault (BRF), a northeast trending, vertical zone within the Jessops Creek Tonalite with a strike extent of greater than 3km. The BRF has strong sub-horizontal lineations suggesting a dominantly strike-slip movement. Several authors (e.g., Switzer 2000, Laing 2005, Cowan and Davis 2017) note that the BRF pre-dates gold mineralisation at Ravenswood and has acted as a partial locus for mineralisation; in particular where it is intersected by cross-cutting low angle structures. This pattern can be seen in Figure 7 which shows the grade control ore outlines generated during mining of the BRW open pit draped with the interpreted positions of the major cross-cutting lodes. The grade control outlines have a good correlation to the interpreted lodes and expected higher-grade zones.

Other large-scale structures observed at BRW include sets of moderately dipping quartz-sulphide filled tension veins and joints locally know as ‘lodes’. There are three named lode structures; the General Grant (Grant), Duke of Edinburgh (Duke) and Sunset lodes. These lodes developed in preferred structural corridors and the intensity of veining focused the gold mineralisation. Evidence from the drill hole data set indicates that there are potentially other ‘proto-lode’ structures (Figure 6) lying between the named lodes at BRW. The proto-lodes have similar orientations; however the structural preparation was insufficient to develop more continuous mineralised corridors. The lodes and proto-lodes cut across BRF (Figure 8).

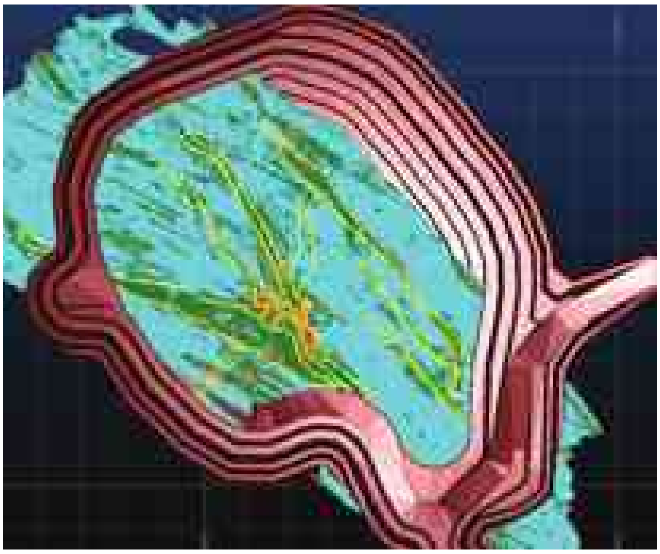

Figure 6. Indicator probability map at 0.25g/t Au showing interpreted lodes (yellow) and proto-lodes lying intermediate to the lode positions.

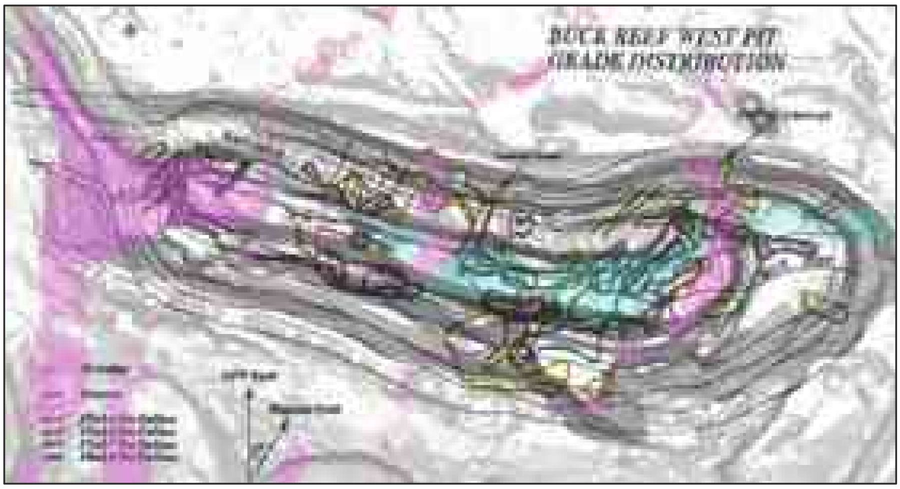

Figure 7. BRW grade distribution (modified from Switzer 2000).

Figure 8. Local structural deformation history and mineralisation timing (After Cowan and Davis, 2017).

Gold mineralisation at BRW is structurally controlled. The tonalite and associated igneous units were subjected to complex brittle deformation resulting in multiple vein orientations. A conjugate orthorhombic structural model is proposed by Cowan and Davis (2017) (Figure 10) with three major classes of veins (Categories A, B and C) each exhibiting a different gold endowment. This simplified structural model predicts widely variable vein patterns and gold distribution. The proposed orthorhombic (non-planar) strain pattern is based on north-south shortening with a maximum extension plunging 50 -> 270[6] (AMG). The system gives rise to a subordinate (intermediate strain axis) plunging 40 -> 090 (AMG).

The three structural categories identified by Cowan and Davis are:

1. Category A characterised by the Sunset-Grant-Duke lodes. These lodes are the most continuous zones of gold mineralisation with an intersection plunge to the northeast). The proto-loads are also Category A features;
2. Category B observed primarily in the Nolans area with an intersection plunge to the south-east. These veins have lower continuity and gold endowment compared to Category A; and
3. Category C sets of sub-horizontal, north-south trending veins with little gold mineralisation.

These lode and vein geometries generate widely variable dilation features with equally variable connectivity and continuity. Mineralisation (gold and sulphides) was inconsistently deposited along the conjugate features (Figure 9). The number of geometries and variety of orientations (in 3D) increases the complexity of modelling the BRW deposit. Locally, some fracture arrays have developed as dominant features persisting 10's of metres to less than 100m as a result of local strain accommodation. Where the distance between the dominant features (i.e. the lodes) increases, weaker stockwork and proto-lode mineralisation developed.

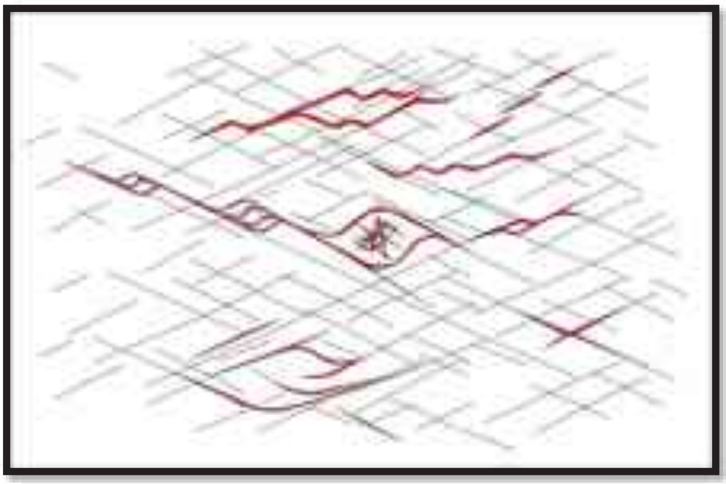

Figure 9. Diagrammatic examples of BRW fracture arrays (After Cowan and Davis, 2017).

[6] Equivalent to 50->240 in the local mine grid (A45)

The number and orientation of mineralised positions in a 3D orthorhombic system like BRW is materially more complex than a simpler 2D conjugate system (Figure 10). The orthorhombic model predicts decreasing strength of mineralisation from

1. Grant-Duke-Sunset and Buck Reef Fault intersection; to
2. Grant/Sunset and Duke intersection.

Other orientations exist at BRW; however they are more sporadic.

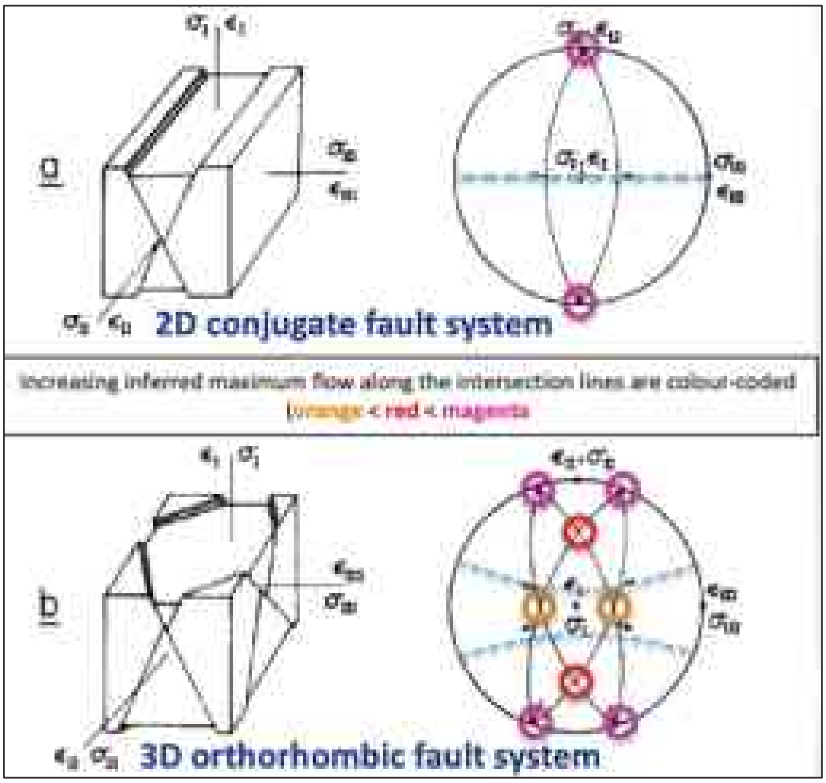

Figure 10. Generalised mineralisation sites 2D conjugate vs. 3D orthorhombic system (After Cowan and Davis, 2017).

This structural model matches well with the observed grade distribution at BRW. A significant proportion (5%) of the sample composites above 0.5 g/t sit outside the interpreted BRF and lode domains (Figure 11). These isolated higher-grade composites reflect samples close to the 'lode' domains, proto-lodes and general background scatter. The 'lode' domains are preferred corridors where veins of differing orientations concentrate. They are not classic lodes in the sense of massive quartz/sulphide mineralised zones.

The vein corridor lodes have inconsistent and variable contacts. Interpretation of the corridors leaves some adjacent grades outside of the interpreted zones. Adding a buffer zone around the lodes captures a further 2% of the composites above 0.5 g/t Au. The remaining

3% are indicative of short range and highly variable mineralisation in veins corresponding to the less dominant locations indicated in Figure 9 and Figure 10.

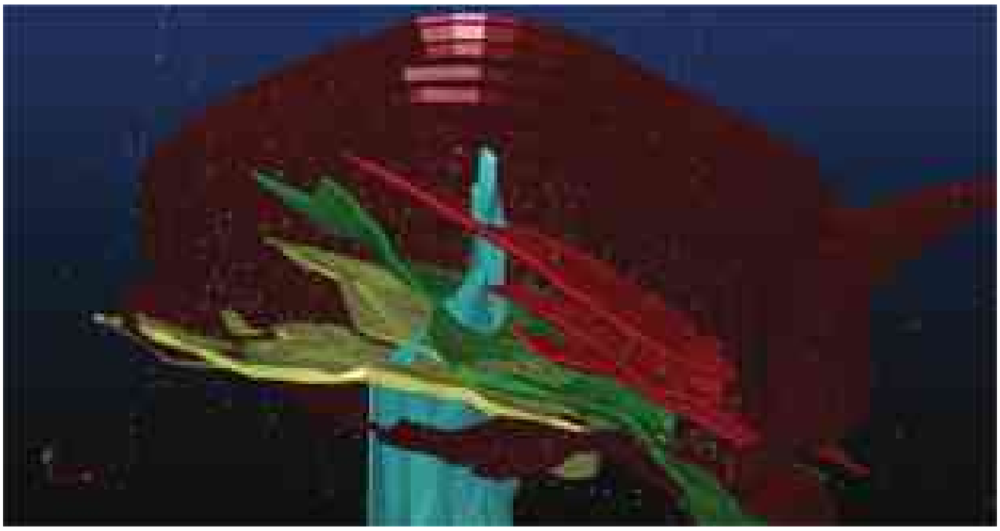

Figure 11. Scattered and isolated composites above 0.5g/t Au and outside interpreted lodes.

3.3 Resource Estimation Implications

The complex and discontinuous nature of the BRW mineralisation directly impacts on the resource model and estimate. As described in section 3.2, while three lodes and the BRF can be defined geologically a relatively large percentage of higher-grade composites are not directly captured within the interpretation. While these data are real and reflect the sporadic nature of some of the mineralisation, there is a risk that they may adversely affect the estimated tonnes and grade if they are not well managed during estimation.

Previous resource models have taken different approaches to managing these background composites (Table 6). For this estimate SD2 has applied three constraints:

Estimate	Practitioner	Approach	Discussion
BRW_2018	Resolute (A. Pedersen)	Indictor krige a variable based on gold grade and sulphide mineralogy. Estimate grades for proportion above/below indictor threshold and assign block grade as average grade of each proportion.	Pro: Incorporates sulphide mineralogy known to have an association with gold mineralisation. Con: Mixes mineralised and unmineralised material to determine average block grade. Block proportions from indicator are data-dependent and method will have vastly different performance in areas of close-spaced data compared to more widely-spaced samples.

Estimate	Practitioner	Approach	Discussion
MPR_2019	MPR Geological Consultants (J. Abbott)	Use broad domains to envelop all mineralised composites. Domain on dominant orientation alone and let the indicator kriging manage both in-lode and out-of-lode composites.	Pro: Simple and fast Con: Does not account for geology. Mixes grade populations from different geological features. Results in 'patchwork' grade distributions that bear little resemblance to the underlying geological framework.
SD2_2020 (this estimate)	SD2 (Scott Dunham)	Develop 'buffer' domains surrounding the main lode interpretations to capture high-grade samples in proximity to the lodes (**Figure 12**). Additionally, create grade and sulphide mineralogy based domain using indicator kriging (as per BRW_2018) and select a probability iso-surface from the indicator to represent the spatial domain. Apply one-way soft boundaries for buffer domains. Require high number of samples for background mineralisation. Coupled with steep short-range variogram model (>70% variance within 10m) this acts as a moving window average, dampening the impact of isolated extreme grades	Pro: Buffer captures 'vein leakage' from region around lodes and applies a spatial restriction the high-grades in the buffer to reduce smearing. Indicator capture some proto-lode mineralisation. Applying high number of composites, smoothing and spatial restriction in background model acts to reduce grade smearing for samples not inside a domain. Con: Background stationarity is low, so solution relies on quasi-stationarity of the search ellipse. This is a reasonable assumption in regions of dense data but less reasonable where the samples are sparse. Size of buffer domain is not fully informed by geology. In the next iteration of this estimate the buffer should be more data-driven and incorporate geological observations / knowledge.

- Buffer Zones (Figure 12) were created around the interpreted lode positions. These buffers were estimated as separate domains using a one-way soft contact into the background mineralisation. The impact of these buffers is to capture higher-grades adjacent to the lodes while simultaneously limiting the spatial extent these grade can exert on the estimate. This is particularly effective where lodes intersect and bifurcate. In these cases the buffer zones capture the likely increased veining and fracturing in these complex areas;
- An additional proto-lode domain was developed based on the 20% probability iso-surface of the 0.5g/t Au indicator (Figure 13). This domain captured some of the higher-grade composites lying outside the lode and buffer domains, effectively restricting the spatial extent of these samples; and
- Use of a wide search with high numbers of composites required for the background mineralisation (section 5.6.1). A minimum of 20 composites from at least six drill holes were required for the initial background estimate. If this target was not met, the search ranges were expanded; however the minimum number of composites were also increased which further smoothed the estimated grade, dampening the influence of any isolated high grades. In well drilled areas where there are several high grade samples, this approach allows for the background grade to increase while at the same time reducing the likelihood of high grades smearing into unsampled areas. This approach relies on the concept of quasi-stationarity of the search

neighbourhood and, given the very steep slope of the variogram model (>70% of total variance within 10m) tends towards an estimate based on a moving window average.

Table 6. Difference in modelling approaches for BRW isolated grades.

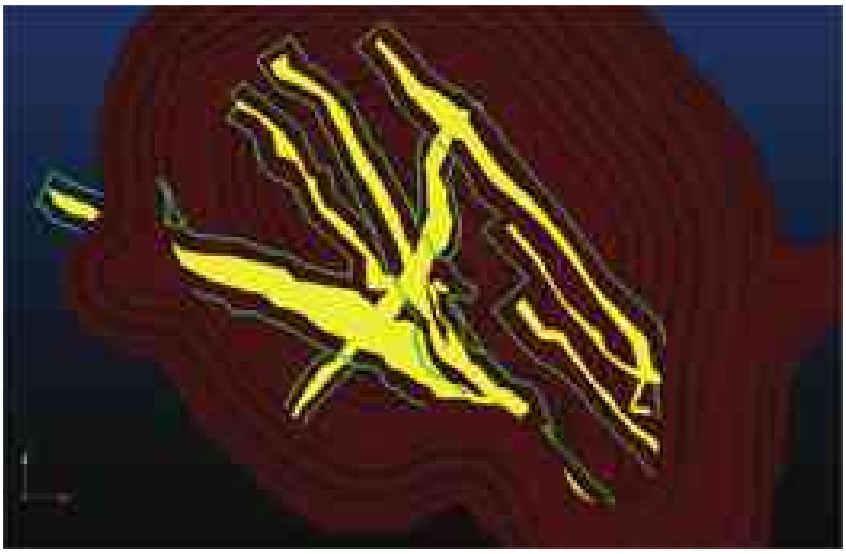

Figure 12. 'Buffer' Domains around interpreted lodes at BRW.

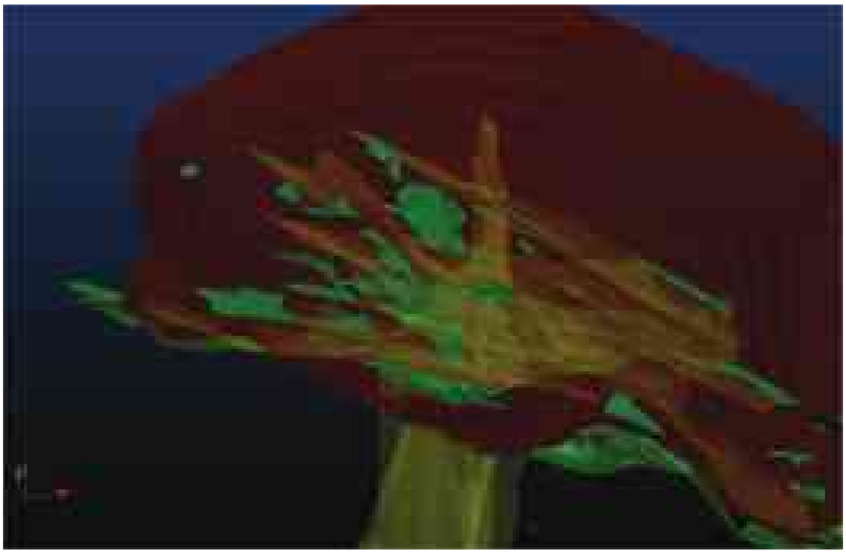

Figure 13. Proto-Lode domain (green).

4. Resource Estimation Data

4.1 Data Provided

SD2 was provided with a comprehensive data set including:

- The geology drill hole database in MS Access format dated 17 February 2020. This database included tables for:
  - Assays
  - Collars
  - Down hole surveys
  - Lithology logs
  - Structural measurements and
  - Magnetic susceptibility.
- The topographic surface for the Ravenswood area including a lidar survey combined with historical end-of-pit mining surfaces;
- Wireframe solids for historical underground mining for all lodes at BRW (stopes and development);
- Wireframe surface for a range of conceptual open pits based on analysis of past resource estimates;
- A variety of reports including general geology descriptions, structural geology analyses and past mineral resource estimation reports; and
- A set of interpreted wireframe solids for the Duke, Grant, Sunset Lodes and the Buck Reef Fault.

4.2 Database Assessment

The geology database contains both recent (post 2006) and historical data (1986-2006) collected by multiple companies in multiple campaigns. Documentation of drilling approach and management methods for the historical data is sparse. The more recent drilling was supervised by Resolute Mining and procedures were well documented.

The data consists of observations from drilling using multiple methods including open hole, aircore, reverse circulation and diamond drilling (Table 7). Aircore, blast holes, RAB and water bore drilling was removed from the data set before estimation. Open hole percussion samples were retained in keeping with previous work by MPR Geological Consultants (Abbott, 2019). MPR compared 266 open hole 2.0m composites against nearby diamond and RC composites (maximum separation 4.0m) and concluded there was a favourable correlation between the different sample types. MPR noted that some very low grade and very high grade data had poorer correlation; however, MPR considered the data did not exhibit a material bias. MPR's pair comparison statistics and data scatter charts are replicated in Table 8 and Figure 14 for reference. SD2 note that the distribution of the open hole data is limited (Figure 15). Many of the holes lie above the current topographic surface or outside the margins of the deposit. The exception to this is some close-spaced, near-surface open hole percussion drilling at the southern end of the General Grant lode. This area is also supported by several RC and diamond drill holes and the potential impact of the open hole percussion samples is limited.

Hole Type	Total Number of Holes	Number Used for BRW Estimate
Air Core	28	0
Air Core – Diamond Tail	16	15
Blast Hole	2	0
Diamond	149	140
Open Hole Percussion	96	96
Open Hole – Diamond Tail	146	146
Rotary Air Blast	34	0
Reverse Circulation	281	273
Reverse Circulation – Diamond Tail	50	48
Sludge	34	0
Water Bore	4	0
Water Bore RAB	4	0
Total Number of Holes	**844**	**718**

Table 7. Number of drill holes at Buck Reef West.

	All Pairs < 4m		Pairs < 10g/t		Pairs 0.05 - 5.0 g/t	
	OHP	Other	OHP	Other	OHP	Other
Number	266		252		96	
Average	1.48	1.22	0.62	0.60	0.87	0.88
Avg. Diff		-18%		-3%		2%
CV	4.15	4.47	2.31	2.12	1.16	1.22
Minimum	0.01	0.01	0.01	0.01	0.05	0.05
1st Quartile	0.01	0.01	0.01	0.01	0.16	0.17
Median	0.09	0.09	0.08	0.09	0.44	0.44
3rd Quartile	0.58	0.62	0.46	0.57	1.23	1.23
Maximum	75.80	77.50	8.68	8.57	4.35	4.97

Table 8. OHP vs RC+DD pair statistics (After MPR, 2018).

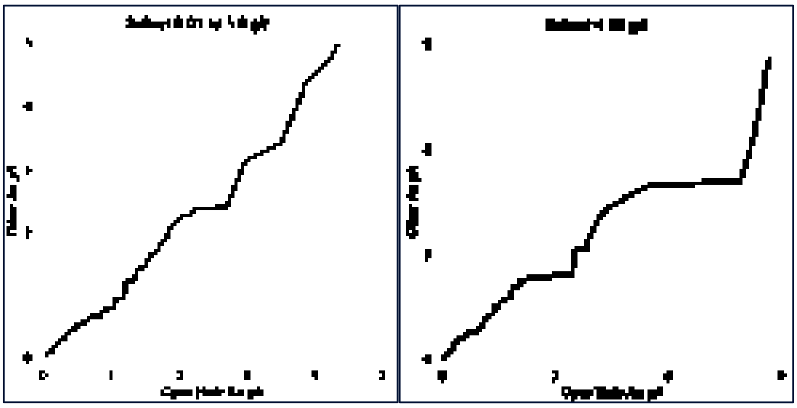

Figure 14. OHP vs RC + DD pair statistics (After MPR, 2018).

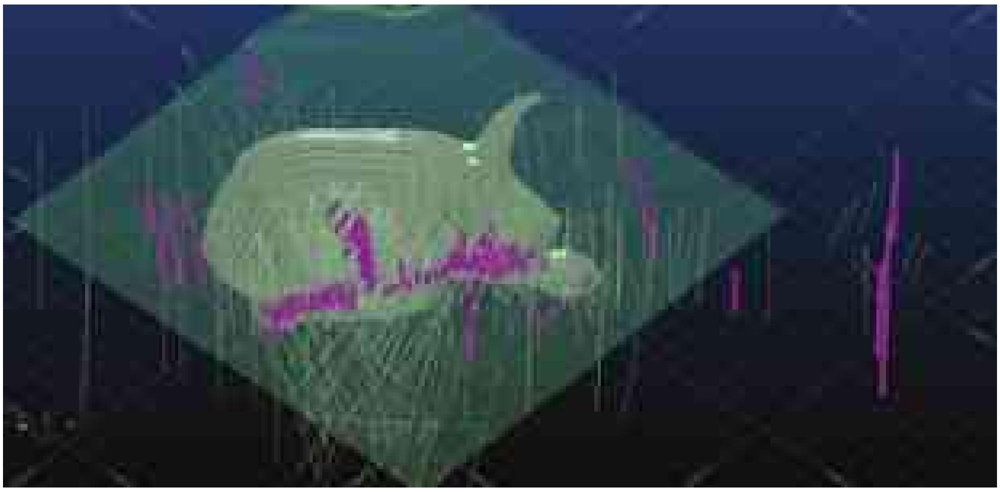

Figure 15. Location of OHP drill holes.

SD2 reviewed the provided MS Access database and discussed data management practices with the on-site team. Both the digital and hard-copy records were examined, and it was clear that significant time and effort had been spent on data quality.

A ran a suite of routine checks were completed identify data errors. Checks included:

- Missing data (collar, survey, assay, lithology);
- Duplicate holes, collars, surveys and samples;
- Sample from/to values beyond the recorded length of the hole;
- Invalid data including out-of-range coordinates, negative grades;
- Spurious survey deviations based on angular rate of change tolerances;

No errors were found by these checks and visual examination of the desurveyed drill hole data supported SD2's opinion of the high quality of the geology database.

4.3 Drilling and Sampling

The drilling and sampling procedures used for recent drilling at BRW are well described in Lim et al. (2018). Drilling at BRW can be divided into roughly 4 periods (Table 9, Figure 16 and Figure 17).

Period	Number of Holes
1900 – 1986	41 (5%)
1986 – 2004	469 (56%)
2004 – 2012	86 (10%)
2012 – Present	248 (29%)
Total	**844**

Table 9. Drill holes by date.

Figure 16. Drill holes by date (plan).

Figure 17. Drill holes by date looking NE.

The majority of drilling occurred between 1986 and 2004 during the ownership of MIM Exploration through their Carpentaria Gold subsidiary. Holes drilled in this period were typically vertical, targeting the Duke of Edinburgh, General Grant and Sunset lodes with a few angled holes intersecting the upper levels of the Buck Reef Fault. From 2004 to 2012, total drilling decreased, and the holes were targeted at the deeper portions of the Buck Reef Fault and extensions of lode mineralisation on the western side of BRF.

From 2012 to present, drilling targeted deeper zones of BRF and also duplicated some of the earlier intersections in the lode mineralisation. Drill intersections were restricted due to previous mining; however, where possible holes were extended through narrow stope voids. This recent drilling was completed using procedures and standards set by Resolute Mining Limited.

The drill hole database contains records for approximately 120km of drilling and six different drill methods (Table 10). The majority of drill holes are diamond core. This drilling consists of a close to equal split of chips and diamond core samples (Table 10).

Hole Type	Metres in Resource Drill Hole Database
RC	29,170.7 (24%)
OPD	26,684.94 (22%)
DD	37,396.58 (31%)
RCD	17,786.98 (15%)
OHP	7,454.6 (6%)
ACD	1,335.87 (1%)
Total Metres	**119,829.67**

Table 10. Metres drilled by hole type.

Sample Type	Metres
Chips	46%
Unrecorded	8%
Core	45%
Void/Stope Fill	0.1%

Table 11. Proportion of sample types.

Drill holes used in the resource estimate were restricted to those holes with validated collar surveys, down hole surveys, lithological logs and assay data.

Limited information is available for drilling and sampling procedures for holes drilled before 2004. Most of these samples were drilled by MIM Exploration, an organisation with well-developed drilling and sampling protocols and practices. MIM's processes in the 1980's and 1990's were at or above the then industry standard.

More detailed records are available for drilling after 2004. This includes a description of the drilling and sampling methods, procedures, protocols and quality management systems (Lim et al., 2018).

For RC drilling prior to 2016, samples were collected using a riffle splitter below cyclone approach with the feed controlled using a 'sock' between the two devices. This initial riffle split divided the sample into 25:75 proportions. The 25% proportion was then further divided using a secondary, smaller riffle splitter to create the final sample to be analysed. Samples were collected at 1.0m intervals. Sample recovery was estimated based on the 25% fraction from the initial split. After 2016, the RC drilling/sampling practice was modified. A cone splitter below cyclone approach was adopted. Sample intervals remained at 1.0m.

Diamond core sampling was standardised to 1.0m intervals regardless of the underlying geology. Given the nature of the mineralisation (effectively stockwork veins with different orientations), standardising on sample interval was a reasonable decision. Core was half-sawn using an automatic saw with the cut made along an offset to the orientation line (where present).

Both RC and Diamond core holes were logged using a standardised logging legend incorporating lithology, alteration, mineralisation styles, structural observations and geotechnical information (DDH only).

All samples collected after 1996 were analysed by Australian Laboratory Services Pty Ltd (ALS) in their Townsville facility. A proportion of samples were also submitted for umpire analysis (SGS Townsville). The sample preparation and analysis procedure has remained largely unchanged. After drying, crushing and splitting (if required), and pulverisation, a 30g or 50g aliquot was selected from pulp for fire assay. A proportion of holes were also analysed using ICP-MS/ICP-AES of a four-acid digest of a 0.25g aliquot.

Some RC and grade control blast hole samples were analysed on-site at the Ravenswood gold mine laboratory using pulverise and leach (PAL) of 1kg samples.

4.3.1 Treatment of absent data

A relatively large proportion of the drill hole database has not been assayed. Of the 96,582 sample records, 38,141 (39%) were not assayed for gold or were reported as below detection limit (i.e., a negative value in the database). For the purpose of this estimate SD2 assigned a very low grade (0.005 g/t) to these intervals.

4.4 Quality Management

Only limited quality management performance information is available for holes drilled prior to 2004. For data collected post-2004, quality management (QAQC) followed general industry guidelines. Quality control samples including certified reference materials (CRM), blanks, quartz flushes and basalt blanks were blind submitted to ALS and to the on-site lab. Additionally, routine checks of pulverisation performance were completed and a selection of samples were chosen for coarse and pulp duplicate analyses. The results of this quality management system are outlined in Resolute, 2018.

Thirteen different CRM standards were submitted (1,725 individual packets) within a total of 462 sample batches between 2009 and 2018. Of these, 135 were part of the quality management system for the on-site lab and 324 were for monitoring ALS Townsville. The CRMs performed better than the statistical expectation with less than 1% of results lying outside of 2 standard deviations. CRM performance was similar across all grade ranges. Normative analysis of ALS Townsville's performance indicates a slight negative bias against certified values for CRM below 2.0g/t and a slight high bias above 2.0 g/t. The biases are less than +/- 2% in all cases. The on-site lab recorded similar or slightly better performance against the certified values.

The performance of blanks submitted (including lab blanks, quartz flushes, basalt blanks and a certified blank from Geostats) was excellent. Of 5,605 blanks submitted to ALS, 50 recorded a 'warning' and 42 recorded a 'fail'. The highest warning rate (5%) was for quartz flushed indicating the need for improved inter-sample hygiene particularly after pulverising very high grade samples. The overall performance of submitted blanks is in line with good practice.

Routine sieve checks used to monitor pulverisation performance showed ALS met or exceeded the 85% passing 75µ threshold more than 99% of the time. The results show that there is a chance the3 samples were over-pulverised (Figure 18) with 40% of the data indicating >95% passing 75µ.

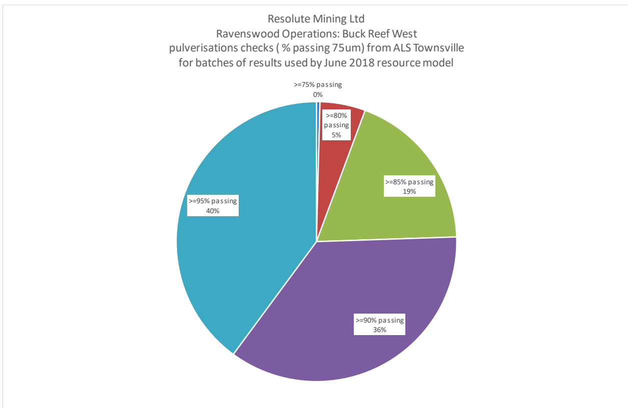

Figure 18. Pulverisation results to June 2018.

Analysis of the 1,590 duplicate samples submitted to ALS Townsville showed a high correlation coefficient (0.945) and a close match on the shape of the histograms for primary and duplicate samples. The scatter plot (Figure 19) shows the typical trends expected for well managed duplicates with high precision.

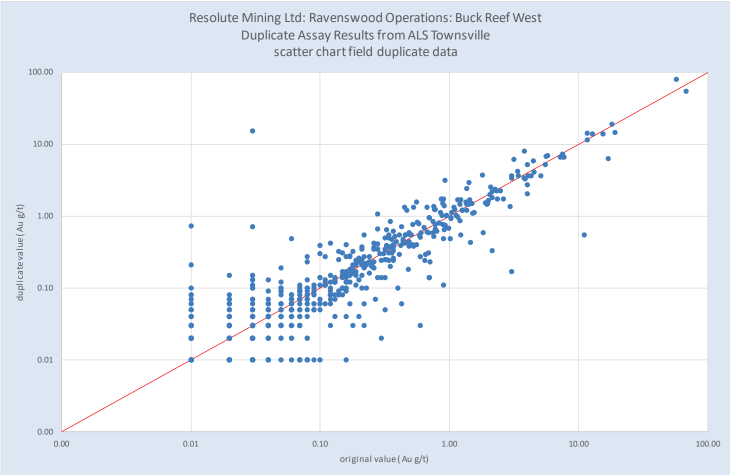

Figure 19. Duplicate sample scatter plot.

Performance at the on-site lab, while lower than ALS, was within acceptable limits. There was a slight bias, largely driven by differences in the higher grade samples. Similar trends were evident for coarse split and pulp duplicate as expected.

The quality management data indicates BRW is based on high quality sample results with good precision and accuracy. The data is considered acceptable for resource estimation and evaluation.

4.5 Collar and Down Hole Survey Data

All data used for the BRW estimate was reported in the local A45 grid, a rotation of 30° clockwise from magnetic north. This grid also includes a datum height adjustment of -32.813m to the Australian Hight Datum.

The practices and standards for drill hole location are reported in Lim et al. (2018).

Drill hole collars were surveyed by the Ravenswood gold mine in-house survey team using Leica TPS1100 total station and optical techniques. A review of the reported collar locations against the LIDAR topography shows good agreement. No collars were independently checked for this estimate and the data is accepted as meeting industry standards.

Down hole surveys exist for the majority of drill holes based on a variety of techniques including electronic multi shot (51%) and either electronic or manual single shot (28%). Three percent (3%) of the down hole survey records were based on the set up hole orientation of a compass measurement). A review of the down hole survey data did not identify any obvious errors. Hole traces were consistent, and no data artefacts were observed.

Sample locations within drill holes were based on the Datamine Studio RM standard desurvey method. This approach calculates the XYZ centre point, bearing and dip for each interval based on spherical arcs. Survey measurements are treated as 3D unit vectors (i.e., they are *not* independent) and therefore sample intervals lie tangential to the unique arc defined by the survey data.

4.6 Data Distribution and Spacing

There are a number of common drilling directions at BRW (Figure 20). The most common is vertical holes (shown as bearing 000 in Figure 20). Followed by two perpendicular orientations 320 and 210. A third subset of east-west holes completes the major directions. This highlights a paucity of drilling in the 030-150 direction. In the two perpendicular directions holes were drilled on nominal 40m grids.

Drill hole coverage is reasonable; however given the complexity of the structure and grade distribution at BRW additional infill drilling is required prior to mining and for grade control to provide sufficient certainty of grade continuity for mineralisation outside the major lode structures. Preliminary analysis based on the variograms of the different domains suggests a grade control spacing on the order of 10m x 10m.

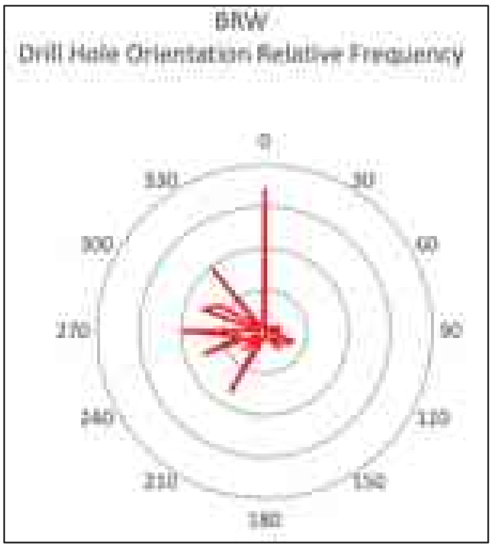

Figure 20. Drill orientation frequency.

4.7 Bulk Density

The bulk density used for BRW is based on 1,957 Archimedes measurements collected from drilling supervised by Resolute Mining Limited between 2014 and 2018. The samples are well spread throughout the mineralisation and indicate a relatively low degree of variability. SD2 examined fresh and oxide densities and compared data from within the lodes and outside of the lodes (Table 12).

	Number	Average	Median	Minimum	Maximum	CV
Oxide	20	2.75	2.73	2.63	2.84	0.020
Fresh	1937	2.80	2.78	1.79	4.13	0.038
In Lodes	329	2.86	2.81	2.28	3.74	0.055
Outside Lodes	1,628	2.78	2.77	1.79	4.13	0.032

Table 12. Basic statistics for bulk density measurements.

The measured bulk density values are supported by tonnage reconciliations from production at BRW and Sarsfield.

Based on the sample data and reconciliation, SD2 assigned a bulk density of 2.85 g/cm^3 to the lode domains and 2.78 g/cm^3 to domain 9999 (background). All oxide material was assigned a bulk density of 2.4 g/cm^3.

5. Resource Estimation

5.1 Interpretation and Domaining

As described in section 3.2, the BRW deposit is structurally controlled and consists of vein-associated gold plus sulphide mineralisation. The location and orientation of the main features are controlled by conjugate veins developed in an orthorhombic stress regime. The structural model predicts three types of vein-associated mineralisation:

- Category A, most commonly characterised by the Sunset-Grant-Duke lodes but also featuring in proto-lodes developed between and parallel to these dominant zones;
- Category B, a vein set more common in the Nolans area that exhibit a south-easterly plunge; and
- Category C sets of sub-horizontal, north-south trending veins that carry little gold.

The Buck Reef Fault itself pre-dates the mineralisation. While gold mineralisation occurs in the BRF, it is dominated by regions where the lodes and proto-lodes intersect the pre-existing BRF structure.

The number and types of veins gives rise to complex geometries which makes interpretation difficult and increases the risk associated with assuming hole-to-hole continuity. While interpreting individual veins or even sets of veins is not practical, analysis of the data clears shows preferred mineralisation corridors (including the lodes themselves). This underlying geological system, the location and orientation of the structurally prepared corridors underpins the interpretation and domain approach adopted for the BRW estimate.

The estimate is based on a combination of manual interpretations and probability-based modelling using iso-surfacing of an underlying indicator kriged estimate. This approach effectively divides the mineralisation into 17 domains. The highest level domains are the manually interpreted lodes. These lodes are in turn enveloped by buffer zones that represent the irregular lode boundaries and act to limit edge-effects associated with applying a hard boundary to the lodes. The BRF is also manually interpreted and has its own buffer zone.

Other mineralisation domains (i.e. the proto-lodes) were developed using the 20% probability iso-surface of the 0.5 g/t Au indicator. This domain captures some (but not all) of the high-grade composites lying outside the lodes, BRF and buffer domains. The remaining high-grade composites were used to estimate the background mineralisation with their zone of influence controlled by the definition of the search neighbourhood. Blocks in the background required a relatively high number of composites (20) spread across multiple drill holes. The search ellipse was discoidal and aligned with the structural fabric of the mineralisation. The impact of this search resulted in minimising the spread (or smearing) of truly isolated composites (their grade being averaged out by the weights applied to other data in the neighbourhood) while also allowing for regions where multiple higher-grade samples across several holes imply previously unrecognised proto-lodes or regions of increased structural preparation associated with lode/BRF intersections.

The domains used in the BRW estimate are outlined in Table 13.

Family	Lode	Domain Code	Default Colour
Buck Reef Fault	BRF Lower	1001	Cyan
	BRF Shear	1002	
	BRF Upper	1003	
	BRF Buffer Zone	1000	
Duke of Edinburgh	Duke 1	2001	Red
	Duke 2	2002	
	Duke 3	2003	
	Duke Buffer Zone	2000	
General Grant	Grant 1	3001	Green
	Grant 2	3002	
	Grant Buffer Zone	3000	
Sunset	Sunset 1	4001	Yellow
	Sunset 2	4002	
	Sunset 3	4003	
	Sunset Buffer Zone	4004	
Proto-Lodes	Proto-lodes	99	Magenta
Background	Background	9999	

Table 13. List of BRW domain codes.

All of the domains were developed as sets of non-overlapping solids in Datamine Studio RM. Lode domains were prioritised over buffers and lode buffers were prioritised over the BRF. The proto-lodes (Domain 99) lies outside of the lodes, buffers and the BRF (Figure 21).

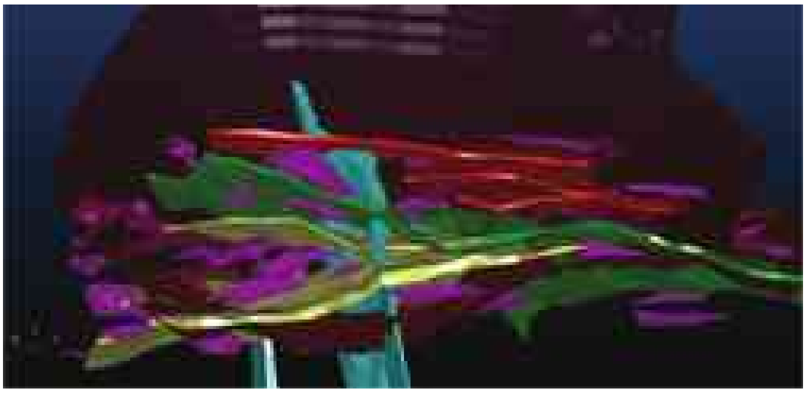

Figure 21. Interpreted BRW domains (looking down towards NE).

5.2 Compositing

The drill hole sampling approach has varied over the different drilling campaigns resulting in a range of down hole sample lengths. The sample length mode is 1.0m; however there are a moderate proportion (15%) of samples with a length of 2.0m, largely from drill campaigns completed before 2004.

SD2 examined the location of samples by length and the relationship between sample length and assay grade. The majority (78%) of samples greater than 1.0m lie within the background mineralisation. Within the lode mineralisation samples longer than 1.5m account for 15% of the intersections. Approximately half (54%) of these longer samples have been duplicated by drill holes with 1.0m sample intervals.

Analysis of grade vs sample length shows low correlation (p = -0.046, Figure 22).

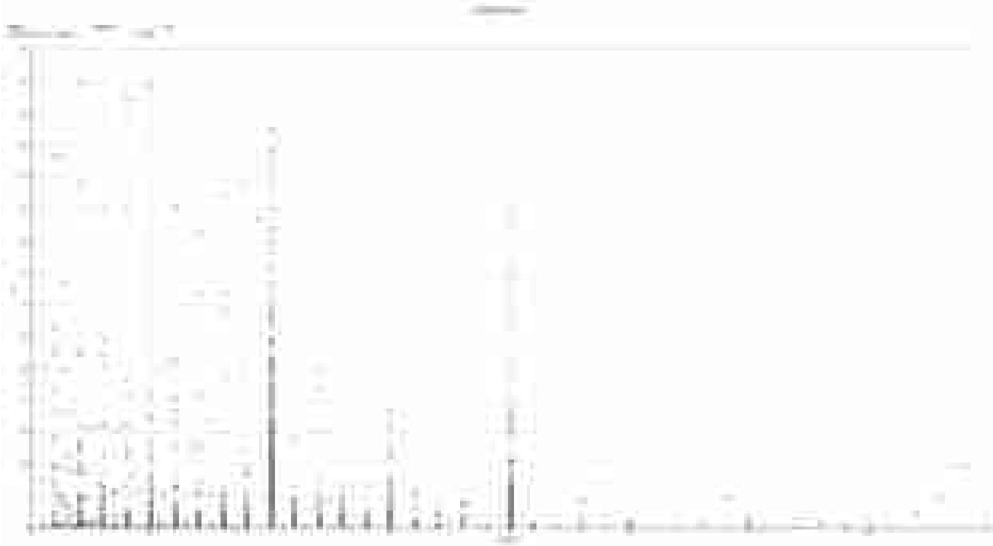

Figure 22. Scatterplot - Length vs Au grade.

Based on this analysis a nominal composite length of 1.0m was selected. While this does result in some 'de-compositing' of longer samples, given the narrow width of the mineralisation and the data density 1.0m is considered a reasonable compromise. Comparison of *all* samples composited to 1.0m against only composites for those samples with a length less than 1.5m indicates that there is a relatively low impact due to de-compositing with modest differences between mean and CV of the two data sets exception of Domain 4003. The outcome for Domain 4003 is adversely impacted by a single sample grading 364g/t Au over 1.5m. This impact of this outlier was mitigated by grade capping (section 5.3.)

Samples were flagged by domain prior to compositing. The minimum composite length allowed was 0.2m. Compositing used Datamine's @mode=1 option which retains all sample data by adjusting the composite length to values approaching the designated metreage. In practice this approach resulted 73% of the composites equalling exactly 1.0m and 99% of the composites having a length of 1.0m +/- 0.05m.

5.3 Grade Caps

Examination of the univariate statistics of the composited data shows that the grade distribution in every domain has a strong positive skew. The distributions include some samples that appeared to be outliers or inconsistent with the distribution of the majority of composites. SD2 examined the rate of change of the CV as the highest-grade samples were removed from the domain data sets. Where the rate of change accelerates rapidly it is likely that it is affected by outlier samples. Figure 23 shows an example of the CV rate of change plot and Appendix B has plots for each domain.

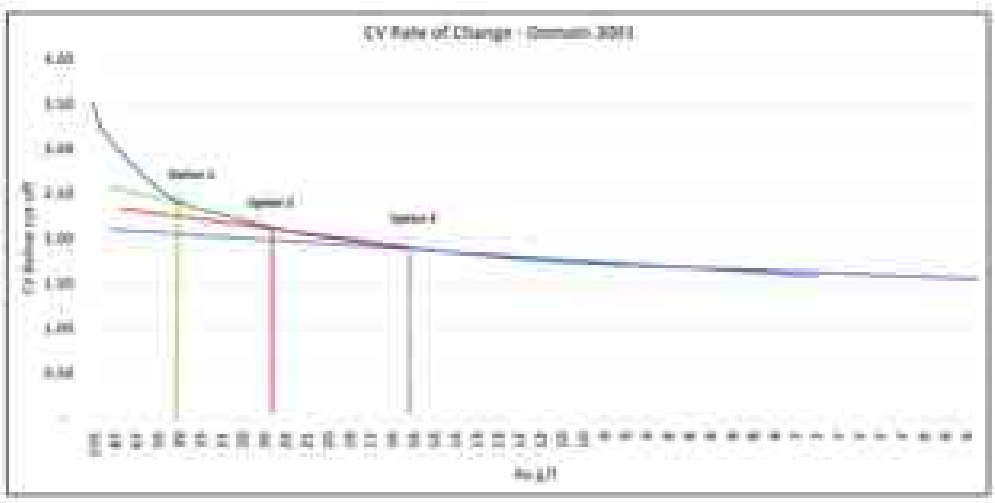

Figure 23. Example of CV rate of change plot for grade cap analysis.

Based on the rate-of-change analyses, grade caps[7] were selected for each domain. The grade cap applied, equivalent percentile and impact of the cap on the mean grade and CV for each domain is presented in Table 2. A total of 521 composite grades were capped, equating to 0.4% of the data. On a domain-by-domain basis the grade cap ranged from the 96th percentile (Domain 4002; 7.0 g/t Au) to the 99.8th percentile (Domain 1003; 35 g/t Au). The average cap percentile across all domains was the 98.4th percentile. The grade caps reduced the proportion of domains with CV's greater than 3 from 64% to 11%.

[7] Grade cap refers to capping the grade for composites within a domain to a maximum value. The composites are kept as members of the domain during estimation.

Description	Domain	Min	Max	Average	Variance	CV	Cap Applied	Cap Percentile	Number Affected	Capped Average	Cap Impact	Capped CV	Cap Impact
Proto Lodes	99	0.001	45.80	0.86	9.0019	3.4787	16.00	97.8%	24	0.78	-10%	2.6863	-23%
Buck Reef Fault Buffer	1000	0.000	77.00	0.41	7.2658	6.6476	3.00	97.6%	124	0.22	-47%	2.6478	-60%
Buck Reef Fault lower	1001	0.001	38.90	1.62	12.1412	2.1498	13.00	98.2%	8	1.50	-7%	1.8225	-15%
Buck reef Fault Shear	1002	0.001	81.90	2.47	76.9230	3.5547	17.00	98.0%	2	1.76	-29%	2.0808	-41%
Buck Reef Fault Upper	1003	0.000	98.34	2.21	17.1117	1.8699	35.00	99.8%	9	2.17	-2%	1.6490	-12%
Duke Buffer	2000	0.000	30.40	0.20	0.9144	4.7338	3.00	98.9%	61	0.16	-22%	2.8453	-40%
Duke Lode 1	2001	0.001	55.54	1.14	11.4825	2.9645	12.00	98.9%	9	0.98	-14%	2.1047	-29%
Duke Lode 2	2002	0.001	16.85	1.16	4.6754	1.8708	8.00	96.9%	6	1.09	-6%	1.6739	-11%
Duke lode 3	2003	0.000	44.10	1.69	22.4620	2.8008	13.00	97.4%	9	1.35	-20%	2.0916	-25%
Grant Buffer	3000	0.000	277.00	0.24	15.9027	16.8365	7.00	99.6%	29	0.15	-38%	4.0464	-76%
Grant lode 1	3001	0.000	155.00	2.42	71.7883	3.5024	25.00	98.2%	50	1.92	-20%	2.3087	-34%
Grant Lode 2	3002	0.001	66.50	1.22	15.0476	3.1691	12.00	98.5%	10	1.07	-13%	2.1059	-34%
Sunset Buffer	4000	0.000	67.40	0.24	2.4001	6.4951	3.00	98.7%	63	0.16	-33%	2.7842	-57%
Sunset lode 1	4001	0.000	113.00	1.32	19.0773	3.3183	18.00	99.1%	20	1.17	-11%	2.2677	-32%
Sunset Lode 2	4002	0.001	18.75	1.26	6.8934	2.0885	7.00	96.0%	12	1.04	-17%	1.6088	-23%
Sunset lode 3	4003	0.001	364.00	3.17	386.6354	6.2052	29.00	98.6%	6	1.88	-41%	2.2178	-64%
Background	9999	0.000	164.36	0.16	2.6000	9.8432	17.00	99.9%	79	0.14	-12%	5.8119	-41%

Table 14. Grade caps and capping impact by domain.

The impact of these grade caps on the BRW estimate was examined by running a series of sensitivity analysis, increasing and decreasing the cap grades in 5% increments from -10% to + 10% and comparing the outcome against the base case (Table 15). This analysis indicates that relatively small changes to the capping grade have a low impact; however the application of the cap itself changes the estimated metal across the entire estimate by 31%. This is the expected outcome given the highly skewed grade distribution. A small number of very high composite grades materially impact on the mean grade.

Case	Tonnage Change	Ounce Change
-10%	99%	98%
-5%	100%	99%
+5%	100%	101%
+10%	101%	102%
No grade cap	103%	131%

Table 15. Grade cap sensitivity analysis.

Additional sensitivity testing was completed to further examine the impact of grade capping. One alternative method for managing apparent grade outliers is to restrict the spatial influence of these grades. 'Outlier' grades are allowed to influence blocks within a predetermined radius. Beyond that radius the grade is capped as normal. The range of influence of the outlier grades can be approximated using the indicator variogram at the proposed capping value. At BRW this varied between 10m and 20m along strike and down plunge and 2m to 5m across dip.

SD2 analysed two different spatial restriction scenarios. The first used a restriction of 10m along strike and down plunge and 5m across dip. The second increased the spatial restriction to 40m along strike and down plunge and 10m across dip. All other parameters remained the same and the restriction applied within estimation domains. In practice, applying the spatial restrict involved creating ellipses around each composite above the grade cap (Figure 24). Blocks within these ellipses were estimated using uncapped grades and blocks outside these ellipses were estimated with capped grades. This is effectively a test of the maximum impact of the grade caps for blocks within a reasonable distance as defined by the indicator variogram.

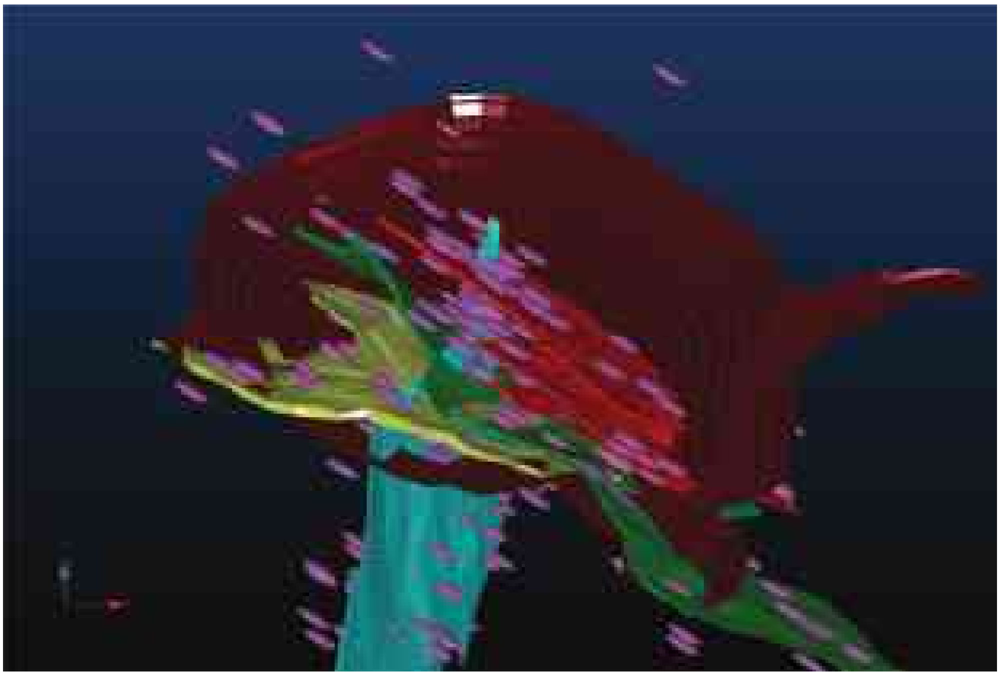

Figure 24. Location of capped samples and superimposed spatial restriction ellipses (40m x 10m).

The results of this grade sensitivity analysis are shown in Table 16. Controlling the spatial influence of the selected grade caps reduces the BRW estimate by between 1.6% and 4.4% globally and a similar amount within the AMDAD pit shell generated during the due diligence study.

		Globally reported resource		Within AMDAD 2019 pit shell	
		Additional Oz	**Percentage**	**Additional Oz**	**Percentage**
10 x 10 x 5 influence	Above 0.3 g/t	19,437	1.65%	12,288	2.08%
	Above 0.4 g/t	19,713	1.79%	12,451	2.26%
40 x 40 x 10 influence	Above 0.3 g/t	47,613	4.03%	25,691	4.35%
	Above 0.4 g/t	48,465	4.40%	26,171	4.74%

Table 16. Grade cap spatial restriction analysis.

SD2 note that the location of these capped grade adds further evidence to the ‘proto-lode’ concept discussed in the domaining and recommendations sections of this report.

5.4 Statistical and Geostatistical Analysis

In conjunction with the grade cap analysis, basic statistics were calculated for the domained and composited data (Table 14). This was followed by spatial statistics analysis and modelling.

Experimental variograms were calculated for raw and Gaussian transformed composites. This included both downhole and directional variograms. Given the narrow, 2-dimensional nature of the lode and BRF domains the variograms were calculated and modelled in the average plane of the lode. Two rotations were used to align the plane to the strike and dip of the domains. Directional variograms were then calculated in 10° increments to determine the plunge of the maximum continuity in the plane.

Variograms were modelled in Gaussian space and then back-transformed. The back-transformed models were compared to the experimental variogram in true space and minor adjustments were made to the nugget based on downhole variography.

The variograms ranged from excellent to poorly structured. A full set of the Gaussian (Normal Scores) variogram models is given in Appendix C. Each variogram is presented with a corresponding set of 3-dimensional images showing the domain and the variogram model overlaid as an ellipse. This approach ensures the axial rotations defined in the model are logical with respect to the orientation of the domain.

All variograms were modelled using spherical models. Models have been normalised with the total modelled variance equal to 1.0. The variogram models are presented in Table 17. Nugget effects range from moderate to high and the majority of the models exhibit a steep slope near the origin, commonly reaching >65% of the total variance within 10m. This is in line with expectations based on the geology of the mineralisation and the sporadic distribution of gold-bearing veins.

		Rotations						Variogram Structures			Ranges - Structure 1			Ranges - Structure 2		
DOMAIN	Description	Angle 1	Angle 2	Angle 3	Axis 1	Axis 2	Axis 3	C0 (Nugget)	C1 (sph)	C2 (sph)	X	Y	Z	X	Y	Z
99	Proto-Lodes	69	67	29	3	1	3	0.250	0.111	0.639	17.0	8.0	2.7	47.0	30.0	7.0
1000	BRF Buffer	120	85	30	3	1	3	0.330	0.430	0.239	4.0	6.0	4.0	13.0	32.0	11.0
1001	BRF Lower	130	85	5	3	1	3	0.631	0.139	0.231	7.6	5.0	3.4	31.1	21.4	10.1
1002	BRF Shear	30	30	-	3	1	3	0.380	0.109	0.511	1.0	15.0	5.0	5.0	40.0	30.0
1003	BRF Uper	22	31	10	3	1	3	0.434	0.274	0.293	2.0	19.0	21.0	13.0	46.0	90.0
2000	Duke Buffer	50	40	30	3	1	3	0.350	0.201	0.448	4.0	7.0	1.5	10.0	17.0	4.0
2001	Duke Lode 1	50	40	(80)	3	1	3	0.350	0.200	0.450	8.5	7.0	2.0	25.0	19.0	3.0
2002	Duke lode 2	50	40	20	3	1	3	0.556	0.169	0.275	11.0	20.0	2.0	35.0	70.0	5.0
2003	Duke Lode 3	45	45	20	3	1	3	0.304	0.573	0.123	14.0	19.0	2.5	37.0	48.0	5.0
3000	Grant Buffer	62	42	(36)	3	1	3	0.310	0.493	0.197	4.0	3.5	4.0	12.0	10.0	11.0
3001	Grant Lode 1	55	45	50	3	1	3	0.388	0.049	-	26.9	13.2	1.8	-	-	-
3002	Grant lode 2	62	33	29	3	1	3	0.282	0.538	0.180	6.0	5.0	7.0	68.0	58.0	20.0
4000	Sunset Buffer	(152)	131	(170)	3	1	3	0.498	0.260	0.241	2.0	5.0	2.0	4.0	13.0	4.0
4001	Sunset Lode 1	35	45	25	3	1	3	0.524	0.339	0.137	6.5	6.0	1.5	18.0	16.0	4.0
4002	Sunset lode 2	45	33	(33)	3	1	3	0.522	0.180	0.298	14.5	11.0	4.0	43.0	55.0	9.0
4003	Sunset Lode 3	49	39	(20)	3	1	3	0.200	0.532	0.268	20.0	21.0	6.0	53.0	59.0	6.0
9999	Background	50	50	50	3	1	3	0.433	0.321	0.246	7.0	5.0	3.0	49.0	23.5	33.0

Axis Convention: 1= X, 2 = Y, 3 = Z

Table 17. BRW variogram models.

5.5 Block Model Framework

The block model covers a volume 920m x 920m x 740m (XYZ) enclosing the full interpreted extent of the deposit with an additional margin. The model is based on 5m x 10m x 5m (XYZ) parent blocks with two sub-blocking divisions allowed in each dimension. This results in a minimum sub-block size of 1.25m x 2.5m x 1.25m (XYZ).

Selection of a block size for estimate represents a compromise between estimation quality and volumetric representivity. While sub-blocking can improve the volumetric precision of blocks compared to wireframed solids, choosing large parent blocks where only a small proportion of the parent lies within the interpreted solid is problematic for block discretisation and the eventual kriging matrix inversion. The BRW parent block size was selected following testing of a number of block size scenarios. Six estimates were completed using identical variograms and searches and differing block sizes. Blocks sizes tested included 5m x 10m x 5m, 10m x 20m x 10m, 10m x 10m x5m, 10m x 10m x 10m, 5m x 5m x 5m and 2.5m x 2.5m x 2.5m. Notwithstanding the recognised challenges of estimating small blocks (Armstrong and Champigny), the BRW block size tests showed that the estimate was relatively insensitive to block size. The range of tonnes and ounces varied by less than 4% across all scenarios for all cut-off grades below 1.0 g/t Au (Table 18). The greatest variance compared to the base case was for the largest block size (10 x 20 x 10).

[illegible]	[illegible]		[illegible]		[illegible]		[illegible]		[illegible]		[illegible]	
Cut-off	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
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Table 18. Block size sensitivity testing results.

The block model framework was used to create a volume model flagged with the interpreted domain codes. Given the angular difference between the A45 grid used at BRW and the strike/dip of the mineralisation, a high proportion of sub-blocks were required. Using a rotated model was considered and then discarded in favour of ease of use and model transferability between mining software packages.

The wireframe vs. block model volume was compared for all domains. No material differences were identified.

As well as the domain codes, the volume model was flagged with binary variables for:

- Oxide (1 = oxide, 0 = fresh);
- Rock (1 = in situ, 0 = air);
- Void (1 = mined void, 0 = unmined rock);
- PIT (a reporting column designating if the block lies within the REP200 pit shell); and

- AMDAD (a reporting column designating if the block lies within the AMDAD due diligence pit shell).

A complete list of block model fields is provided in Appendix F.

5.6 Estimation

Estimation was by ordinary kriging (OK) into parent blocks. Blocks were discretised on a 5 x 5 x 5 matrix.

5.6.1 Kriging Neighbourhood and Search Strategy

The search strategy determines what composites are used to estimate each block in the model and, after domaining, selecting of a well-designed search is one of the most critical factors in developing a robust resource estimate. The kriging weights assigned by the kriging equations are a function of the block size, the variogram and the sample-to-block vectors for all samples in the search neighbourhood. The weights themselves are independent of the grades of the samples. An overly restrictive kriging neighbourhood restricts the number of composites that can inform a block estimate. This can result in conditional bias and poor estimation quality depending on the sample spacing and distribution. Similarly a loose kriging neighbourhood potentially allows too many samples to be included in the weighting assignment. This can lead to broad grade smoothing or averaging and, in some instances the generation of negative weights and potentially negative grades.

There are several levers that can be used when designing a kriging neighbourhood. The search is typically defined using a combination of distances in three orthogonal axes (rotated to align with the variogram) forming an ellipse, plus a requirement for a certain minimum and/or maximum number of composites within the ellipse.

This can be further modified by applying a variety of declustering constraints such as octant/sector limits and specifying the maximum number of samples allowable from an individual drill hole. These declustering approaches have the effect of increasing the average sample-to-block distance compared to undeclustered searches. Thus, declustering is a trade-off between sample-block distance (a direct driver of estimation quality through the kriging matrix) and the potential for spatial bias generated by clustered data. While the kriging equations do, to some extent, result in declustering of the kriging weights, some block-to-sample arrangements can adversely impact on estimation performance.

Further complicating the selection of a kriging neighbourhood, it is rare for composites to be regularly arranged (on a grid pattern) for a resource estimate. This regular arrangement is much more commonly associated with grade control drill patterns. The inconsistency of block-to-sample geometry across any given domain means that any single neighbourhood definition will be sub-optimal in some regions.

In SD2's experience the most practical approach to optimising the kriging neighbourhood is to focus on the minimum and maximum numbers of composites used to inform a block estimate. The search distances are secondary as long as they are sufficiently large to capture the specified number of composites. Effectively, a wide search range is applied and when the maximum number of allowed composites is reached within that search range, no more composites are added. The practical range is therefore a function of sample spacing. In areas of widely spaced drilling the average sample-to-block distance will be greater when compared to areas of more closely spaced drilling. Likewise the estimation performance (as measured by metrics such as kriging efficiency and slope of regression) will vary as a function of sample spacing.

The search neighbourhoods at BRW were developed using this technique. The primary control was based on defining the minimum/maximum numbers of composites required to inform a block estimate. The search ranges were then superimposed on the primary control, maintaining the orientation and anisotropy defined by the variogram model (i.e. the search is aligned with the variogram model). The minimum and maximum number of composites were defined following a series of sensitivity estimates. The values were altered step-wise and the local and global estimation performance was compared. This defined a target of either 32 or 24 composites for all of the lode, BRF and buffer domains. This value was increased to 64 for the background mineralisation (domain 9999). In each case the search was modified by allowing a maximum of 3 composites per drill hole within the neighbourhood. The full search neighbourhood definition is outlined in Table 19. Appendix D contains 3-dimensional images of the search range and orientation superimposed on the geological interpretation. Because of the narrow lode geometry the search ranges were kept to a discoidal shape for all selected ranges (i.e., the minor axis was always much shorter than the major and semi-major axes). Anisotropy was generally defined on the basis of the ratio of the maximum modelled variogram ranges.

		Rotations						Search Ranges (Pass 1)			Minimum	Maximum	Expansion	Minimum	Maximum	Search	Minimum	Maximum
DOMAIN	Description	Angle 1	Angle 2	Angle 3	Axis 1	Axis 2	Axis 3	X	Y	Z	Composites	Composites	Factor	Composites	Composites	Expansion	Composites	Composites
99	Proto-Lodes	69	67	29	3	1	3	130	80	10	6	32	1.5	4	32	2	4	32
1000	BRF Buffer	120	85	30	3	1	3	25	60	10	6	24	1.5	4	24	2	4	24
1001	BRF Lower	130	85	5	3	1	3	55	40	10	6	24	1.5	4	24	2	4	24
1002	BRF Shear	30	30	0	3	1	3	10	60	25	6	32	1.5	4	32	2	4	32
1003	BRF Uper	22	31	10	3	1	3	25	95	90	6	24	1.5	4	24	2	4	24
2000	Duke Buffer	50	40	30	3	1	3	25	40	5	6	24	1.5	4	24	2	4	24
2001	Duke Lode 1	50	40	(80)	3	1	3	50	40	5	6	32	1.5	4	32	2	4	32
2002	Duke lode 2	50	40	20	3	1	3	60	125	5	6	32	1.5	4	32	2	4	32
2003	Duke Lode 3	45	45	20	3	1	3	90	120	5	6	32	1.5	4	32	2	4	32
3000	Grant Buffer	62	42	(36)	3	1	3	50	45	25	6	24	1.5	4	24	2	4	24
3001	Grant Lode 1	55	45	50	3	1	3	60	30	5	6	24	1.5	4	24	2	4	24
3002	Grant lode 2	62	33	29	3	1	3	145	120	20	6	24	1.5	4	24	2	4	24
4000	Sunset Buffer	(152)	131	(170)	3	1	3	10	30	5	6	24	1.5	4	24	2	4	24
4001	Sunset Lode 1	35	45	25	3	1	3	30	25	5	6	24	1.5	4	24	2	4	24
4002	Sunset lode 2	45	33	(33)	3	1	3	125	160	15	6	32	1.5	4	32	2	4	32
4003	Sunset Lode 3	49	39	(20)	3	1	3	130	145	10	6	24	1.5	4	24	2	4	24
9999	Background	50	50	50	3	1	3	50	20	10	20	32	1.5	20	32	2	32	64

Proportion of Variogram Structure 1			Proportion of Variogram Structure 2		
X	Y	Z	X	Y	Z
765%	1000%	370%	277%	267%	143%
625%	1000%	250%	192%	188%	91%
724%	800%	294%	177%	187%	99%
1000%	400%	500%	200%	150%	83%
1250%	500%	429%	192%	207%	100%
625%	571%	333%	250%	235%	125%
588%	571%	250%	200%	211%	167%
545%	625%	250%	171%	179%	100%
643%	632%	200%	243%	250%	100%
1250%	1286%	625%	417%	450%	227%
223%	227%	278%			
2417%	2400%	286%	213%	207%	100%
500%	600%	250%	250%	231%	125%
462%	417%	333%	167%	156%	125%
862%	1455%	375%	291%	291%	167%
650%	690%	167%	245%	246%	167%
714%	400%	333%	102%	85%	30%

Table 19. Search neighbourhood definitions.

5.6.2 Boundary Treatment

Domain boundaries were treated as hard contacts or one-way soft contacts as outlined in Table 20. The decision to soften some domain boundaries by including adjacent (lower-grade) composites was made on the basis of domain boundary analysis and to reduce the grade contrast across the buffer-to-background transition zones.

Domain	Composites used during estimation	Comment
1000	1000, 99, 9999	Buffer zone contact treated as one-way soft to background domains.
1001	1001, 1000	BRF lower contact treated as one-way soft to buffer zone
1002	1002, 1000	BRF Shear contact treated as one-way soft to buffer zone
2000	2000, 99, 999	Buffer zone contact treated as one-way soft to background domains.
2001	2001, 2000	Duke lode 1 contact treated as one-way soft to buffer zone
3000	3000, 99, 9999	Buffer zone contact treated as one-way soft to background domains.
4000	4000, 99, 9999	Buffer zone contact treated as one-way soft to background domains.
99	99, 9999	Proto-lodes treated as one-way soft to background.

Table 20. Domain boundary treatments.

5.6.3 Dynamic Anisotropy

During estimation, the search and variogram orientations defined in Table 17 and Table 19 were modified using the Datamine Studio RM ‘dynamic anisotropy’ feature. This allow block-by-block definition for the orientation of either or both the search and variogram model. A local dip, dip azimuth and plunge are assigned to each block in the model and, at the time of estimation, these values are read from the block model and used in preference to the defined global rotations. The advantage of this approach is that it aligns the search and variogram to the local geology orientation. When estimating narrow and tabular deposits (like BRW) this can be critically important. Small angular deviations between a globally aligned search and the local lode orientation can adversely impact the samples selected when searching.

For BRW, SD2 determined the local dip and dip azimuth from the orientation of individual triangles in the wireframed interpretation. The plunge (angle 3) was set to the third rotation in the variogram model. These values were supplemented by a set of trend surface lines generated by OreFind (Cowan and Davies, 2017). These trend lines were based on the underlying structural framework and the intersection lineations predicted by the conjugate orthogonal model. Examples of the local, dynamic anisotropy orientations are shown with the superimposed global search/variogram orientation in Appendix D.

As a check on the assigned local orientations, SD2 compared the global values to the values contained in each block in the model. For the majority of blocks, the average deviation between the global and local orientations were less than 5°.

5.7 Post-processing

After estimation the model was checked for common estimation artefacts including negative grades and blocks that were un-estimated after applying all search options.

Only one block (in Domain 4003) had a negative grade estimate. This was caused by composites from a single high-grade drill hole (BRD055) which is in proximity to two low-grade holes (BRP119, BRD057). The negative grade estimate was set to a value of 0.001 g/t Au.

Excluding the background, three percent (3%) of blocks remained unestimated after all search passes. These blocks were assigned a default value (Table 21) based on the average estimated grade of the domain. The domain with the highest proportion of unestimated blocks was Domain 2001 (7%). Figure 25 shows the location of the blocks with default grade assignments. Most of these blocks lie at the extremities of the domains. The default grades account for less than 1% of blocks classified as Measured or Indicated[8] (section 7.3).

Domain	Default Grade
1000	0.17
1001	0.33
1003	2.17
2000	0.12
2001	1.00
2002	1.12
2003	1.58
3001	1.70
4000	0.18
4001	1.10
9999	0.02

Table 21. Default domain grades.

[8] Excluding the background domain (9999).

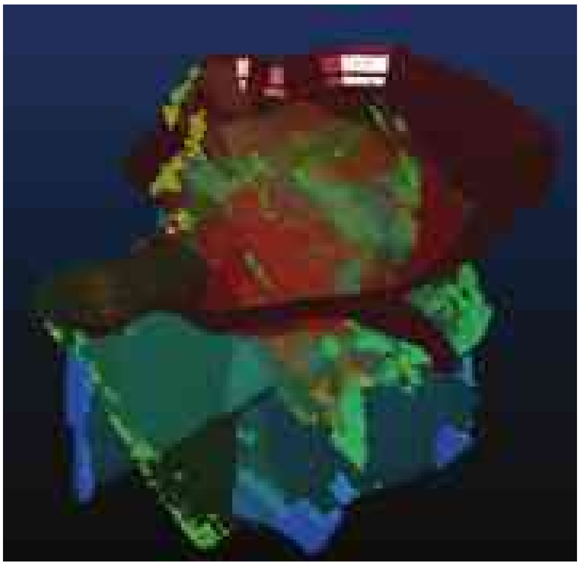

Figure 25. Location of blocks assigned default grades.

The final post-processing step was to account for previous underground mining. Historical stoping exists in the Duke, Grant (A2), Sunset and Buck Reef Fault mineralisation (Figure 26). The nature and extent of back fill in these stoped volumes is unknown and therefore the mined volumes have been treated as voids. All grades in voids were set to zero. All density in voids was set to zero. The same treatment was applied to mine development and vertical openings. All voids (stopes and development) were identified by setting the VOID field in the model to one.

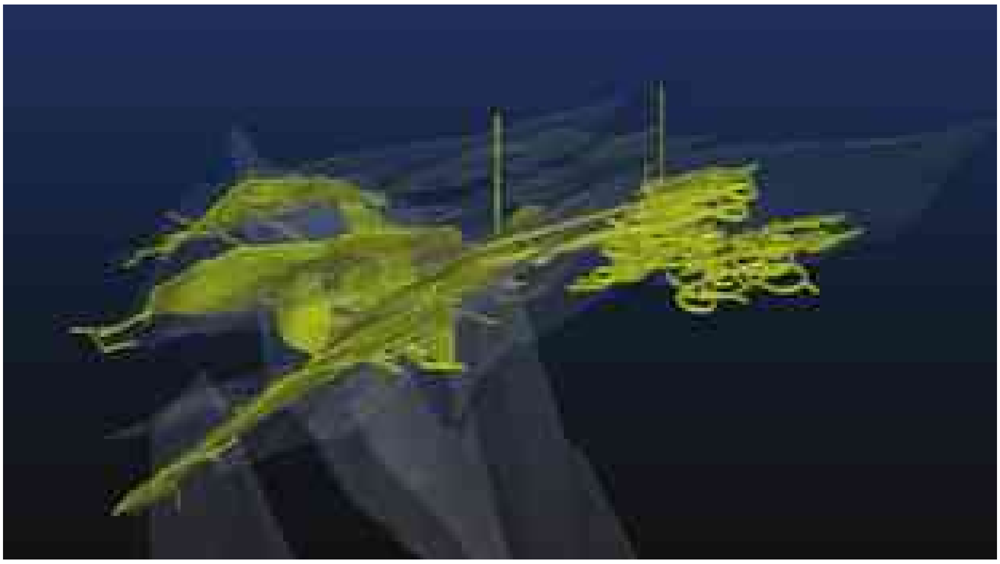

Figure 26. Stoping and development at BRW.

6. Validation

A range of validation and comparisons were used to assess the quality of the resource estimate. The global composite grades were compared to the global domains grades (Figure 27), 2-dimensional longitudinal projections of the drill holes vs. block model estimates were examined and the estimate was compared to the previous models outline in Section 1.3 across a range of cut-off grades. Swath plots were created for all domains. The narrow lodes, combined with the plunging grade trends in the plane of the vein, mean standard swath plots are not always indicative of estimation performance. A longitudinal projection comparison was preferred.

Comparing the block estimates to the declustered composite grades shows that while the grade trend by domain is reasonable, six (6) domains show a relatively high bias. Domains 1001, 1002 are biased low and domains 1003, 2001, 4001 and 4003 are biased high (Figure 27).

For Domain 1001 (BRF Lower) the apparent low bias is due to the sample arrangement within the interpreted domain. There is a large volume of low-grade with relatively sparse sample that encloses a higher grade core. The low-grade volume is under-sampled compared to the high-grade resulting in an apparent bias (Figure 28). The apparent bias in Domain 1002 has a similar cause (Figure 29). In both cases this may indicate that the domain interpretation has been pushed too far into areas where there is low sampling. In these cases the low bias is not considered material,

The apparent high bias in Domain 1003 is due to relatively sparse sampling at depth in what appears to be a high-grade zone (Figure 30). The majority of this area is classified as inferred resource and when excluded from the comparison the bias reduces less than 5%. Apparent biases observed in the other three domains (2001, 4001, 4003) show similar issues with uneven data distribution across the domains. In some cases this is due to past mining rendering it impossible to drill in stoped areas.

After analysis of the differences between composite and block grades, SD2 was satisfied that the estimate was performing as expected. This opinion was supported by examination of the full set of longitudinal projections in Appendix J.

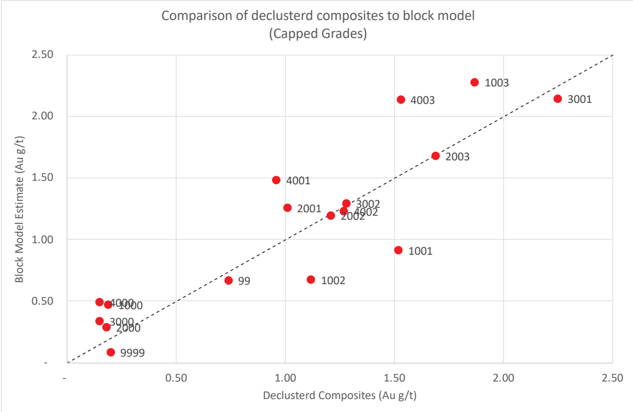

Figure 27. Scatter plot - composites vs. model estimate.

Figure 28. Domain 1001 longitudinal projection.

Figure 29. Domain 1002 longitudinal projection.

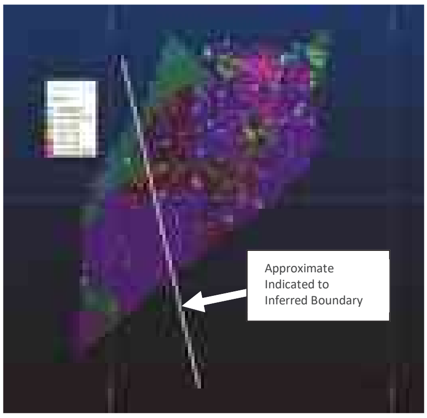

Figure 30. Domain 1003 longitudinal projection with indicated-inferred limit.

7. Mineral Resource Classification

7.1 Jurisdiction and Competent Person

This mineral resource estimate is classified and reported under the guidelines of the JORC Code[9] (2012). The estimate has been prepared by Mr Scott Dunham, a Fellow of the AusIMM (membership number 112857). Mr Dunham has more than 30 years of experience in the resource industry including more than the requisite five (5) years relevant experience in the estimation of mineral resources for the commodity and style of mineralisation at Buck Reef West. A brief summary of Mr Dunham's experience is provided in Appendix K. His expertise covers the complete range of resource estimation practices including geological sampling, interpretation and domaining, geostatistical analysis, estimation and reporting.

7.2 Reasonable Prospects Assessment

The JORC Code requires reported mineral resources to have 'reasonable prospects of eventual economic extraction'. In Mr Dunham's opinion this expectation has been demonstrated for Buck Reef West as follows:

- A positive NPV generated by Resolute Mining Limited in 2018;
- A positive NPV generated by EMR Capital during the due diligence study for the acquisition of the Ravenswood Gold Mine;
- The recently completed acquisition transaction.

Mr Dunham is aware that RAV are currently negotiating social, heritage and environmental licensing conditions. The negotiations are well advanced and no material impediments are likely.

The reported resource lies above an AUD3800 optimised pit shell or within continuous zones of mineralisation suitable for underground mining.

7.3 Classification Definitions

The BRW resource is classified as Indicated and Inferred. There is no Measured Resource. The classification is based on a combination of multiple factors including:

- Geological confidence for the continuity and consistency of the major mineralised zones;
- Drill hole spacing and orientations;

[9] The Australasian Code for reporting of exploration results, mineral resources and ore reserves. 2012 Edition.

- Estimation performance metrics including the slope of regression, kriging efficiency, sum of positive weights and weight of the mean; and
- The search pass and number of composites used during block estimation.

The classification limits were developed in section and plan and manually wireframed. This approach was adopted to minimise the so-called 'spotted dog' patterns associated with automated classification processes where isolated blocks of one class can be fully surrounded by another class due to the application some arbitrary limit or threshold. In all cases the Indicted Resource lies inside the extent of the Inferred Resource. Figure 31 and Figure 32 illustrate the classification limits with respect to the major domains at BRW. The majority lode and BRF domains are classified as Indicated. Only the edges of these domains are classified as Inferred.

Material in the background mineralisation lying outside of the Indicated and Inferred wireframes is unclassified and should not be reported.

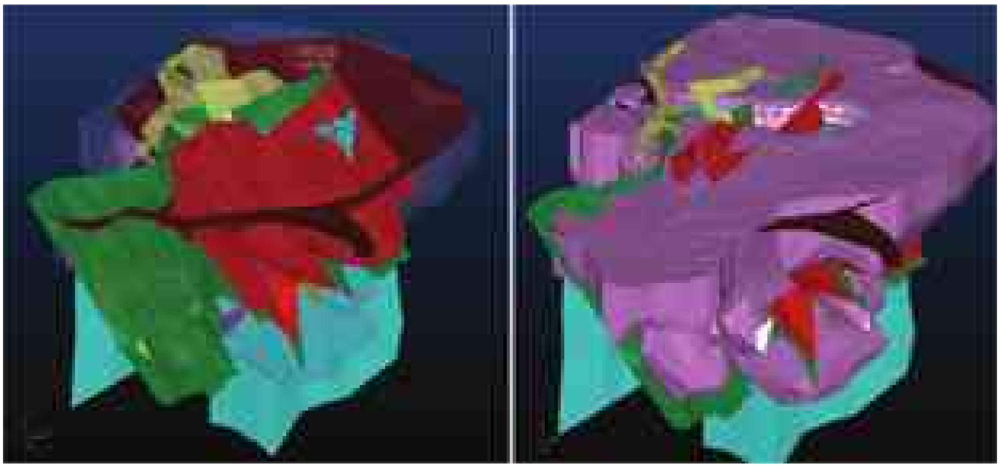

Figure 31. limit of Indicated Classification.

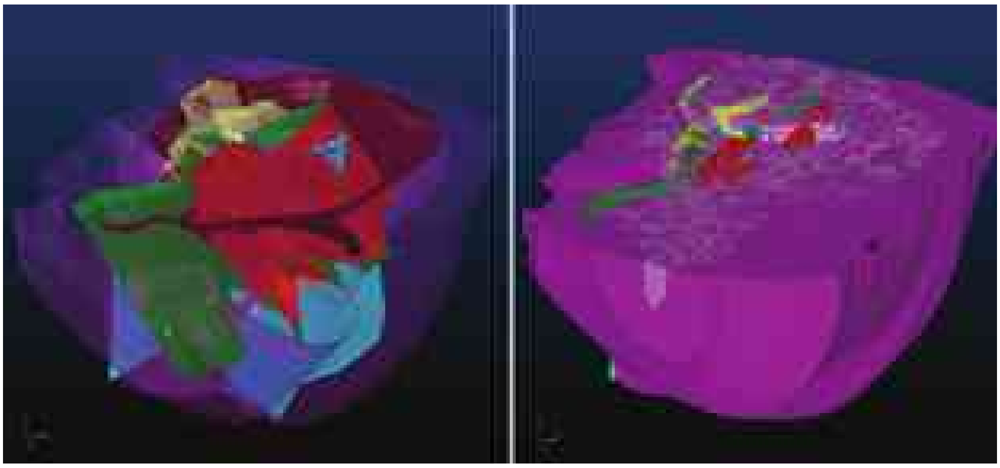

Figure 32. Limit of Inferred Classification.

7.4 Risk and Range Assessment

During the course of preparing the BRW mineral resource estimate SD2 investigated:

- Alternate domaining options based on semi-implicit indicator approaches;
- The impact of hard and soft domain boundaries;
- The impact of changes in block sizes and the minimum and maximum numbers of composites used to inform the estimate; and
- The impact of changes to the grade capping strategy.

Additionally SD2 developed a sensitivity model for BRW as part of the acquisition due diligence process.

Based on these sensitivities analyses, the risks associated with the BRW estimate are (from high to low):

- The domain interpretation, particularly for mineralisation adjacent to or between the dominant lode structures. There are clear zones of preferred mineralisation that exhibit some degree of lateral continuity; however at the current drill hole spacing it is difficult to have confidence that these zones continue from section to section. In the current estimate these zones are controlled by a combination of a probability iso-surface of the 0.5g/t Au indicator and by managing the search neighbourhood applied in the estimate. Future estimates should investigate the potential to improve the interpretation of these more isolated grades (particularly as more data becomes available);

- The grade cap strategy. While changes of +/-10% in the grade cap value have little impact, the difference in contained ounces between the capped and uncapped estimates is on the order of 30%. The deposit has a highly skewed grade distribution and it is likely that some of these extreme grades will manifest during mining; and
- Drill hole density, particularly around the edges of the BRF and lode structures. The current drilling geometry and spacing leaves some edges relatively under-sampled.

Comparison of past estimates using alternative techniques is indicative of the range of possible outcomes for BRW. Table 22 presents the differences between four estimates, reported in the same volume and cut-off. Based on these estimates the likely range of the resource is +/-7% on an ounce basis.

	Tonnes				Ounces			
Cut Off	Uncapped	MPR (MIK)	Due Diligence	Resolute 2018	Uncapped	MPR (MIK)	Due Diligence	Resolute 2018
-	100%	97%	95%	100%	123%	107%	93%	117%
0.10	100%	140%	116%	143%	124%	110%	94%	119%
0.20	101%	156%	120%	131%	127%	110%	93%	115%
0.30	101%	150%	115%	121%	129%	105%	90%	111%
0.40	**103%**	**140%**	**111%**	**114%**	**131%**	**99%**	**87%**	**108%**
0.50	104%	131%	107%	110%	134%	93%	84%	107%
0.60	105%	119%	102%	107%	136%	86%	80%	105%
0.70	107%	106%	93%	106%	138%	79%	75%	105%
0.80	109%	97%	88%	106%	141%	73%	72%	105%
0.90	110%	87%	82%	106%	144%	67%	68%	105%
1.00	112%	80%	77%	106%	147%	62%	65%	105%

Table 22. Estimate sensitivity (within due diligence pit volume).

8. Recommendations for Future Work

As discussed in section 7.4, SD2 recommend future estimates focus on improving the geological and domain interpretation for high grade intercepts outside of the known lodes. Many of these grades align on a lode-parallel orientation and they appear to be preferred mineralisation corridors (or 'proto-lodes'). With additional drilling and re-examination of drill core it may be possible to define these zones with sufficient continuity to improve modelling.

Other recommended improvements include:

- Additional drilling into areas of low drill density. Drilling should initially focus on zones of 'proto-lode' mineralisation within the likely pit shell (Figure 33). There is also potential to increase the size of the resource at depth in the BRF[10] and to the northeast (local grid) (Figure 34);

[10] This includes the area previously interpreted as Buck Reef Fault 'Flats'.

- Drill twinned holes to test and confirm the quality of the open hole samples included in this estimate; and
- Review drill samples for intervals that were not sampled prior to this estimate. Some 39% of the database were unsampled or below detection limit. While these samples have been assigned a grade of 0.005 g/t during estimation, where possible it would be better to have verified analytical results for the model.

Figure 33. Low drill density zones with grades above 1g/t in the resource estimate.

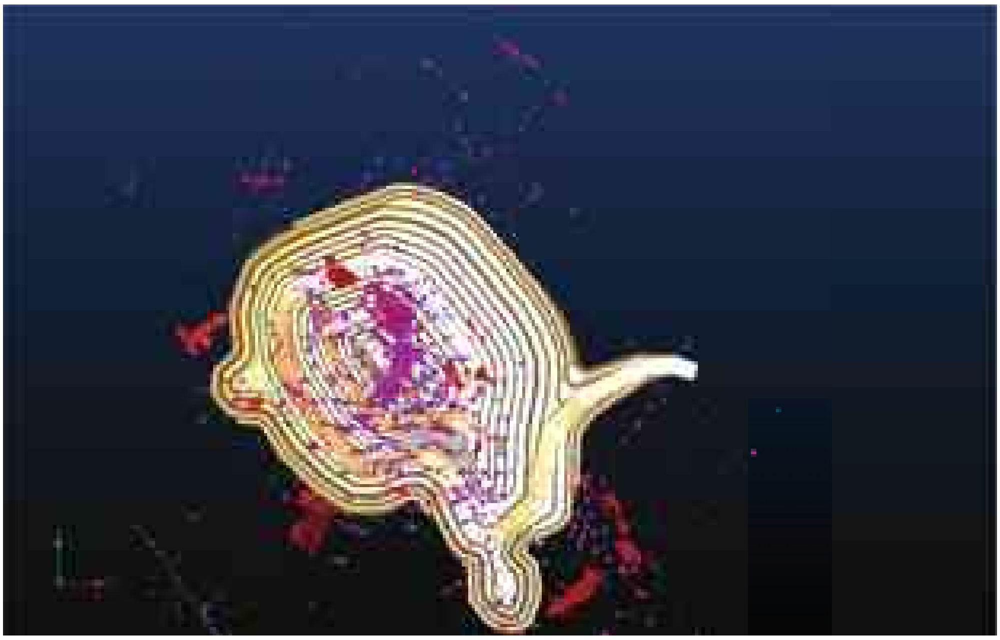

Figure 34. BRW drilling highlighting potential mineralisation to NE of current pit.

9. Resource Statement

The April 2020 Buck Reef West Mineral Resource estimate is tabulated below (Table 23). Grade-tonnage curves for the April 2020 estimate and previous models are presented in Appendix G. The Buck Reef West mineral resource estimate includes both open pit and underground potential. Open pit resources are reported at a 0.3 g/t Au cut-off above an AUD4000 optimised pit shell. Underground resources are reported within continuous zones greater than 2m wide and more than 1,000m^3 at a cut-off of 3.5 g/t in close proximity to the pit shell.

Buck Reef West April 2020 Mineral Resource Statement

Open Pit Above 0.3 g/t	Tonnes	Grade (Au g/t)	Ounces
Measured	-	-	-
Indicated	25,050,000	1.03	833,000
Inferred	1,170,000	1.11	42,000
Total Open Pit Resource	**26,220,000**	**1.04**	**875,000**

Underground Above 3.5 g/t	Tonnes	Grade (Au g/t)	Ounces
Measured	-	-	-
Indicated	91,000	4.97	14,600
Inferred	65,000	4.71	9,800
Total Underground Resource	**156,000**	**4.86**	**24,400**

Total Measured and Indicated	**25,141,000**	**1.11**	**899,400**

Open put resources above AUD3800 shell.
Underground resources within continuous zones >2.0m wide and > 1,000m3
Model: BRW200410.bm
Rounding errors may occur

Table 23. Buck Reef West April 2020 Resource Estimate.

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Appendix A Competent Persons Consent Form

Competent Person's Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and
Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

Report name

Buck Reef West Mineral Resource Estimate April 2020

(*Insert name or heading of Report to be publicly released*) ('Report')

Ravenswood Gold Pty Ltd

(*Insert name of company releasing the Report)*

Buck Reef West

(Insert name of the deposit to which the Report refers)

If there is insufficient space, complete the following sheet and sign it in the same manner as this original sheet.

27 November 2020

(Date of Report)

Statement

I/We,

Scott Dunham

(*Insert full name(s)*)

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member or Fellow of *The Australasian Institute of Mining and Metallurgy* or the *Australian Institute of Geoscientists* or a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I am a full time employee of

(Insert company name)

Or

I/We am a consultant working for

SD2 Pty Ltd

(Insert company name)

and have been engaged by

Ravenswood Gold Pty Ltd

(Insert company name)

to prepare the documentation for

Buck Reef West

(Insert deposit name)

on which the Report is based, for the period ended

30 September 2020

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Exploration Targets, Exploration Results, Mineral Resources and/or Ore Reserves *(select as appropriate).*

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

Ravenswood Gold Pty Ltd

(Insert reporting company name)

	27 November 2020
Signature of Competent Person:	Date:
Australasian Institute of Mining and Metallurgy	112857
Professional Membership: *(insert organisation name)*	Membership Number:
	Sherrill Leigh Dunham – Nanango Queensland
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

Additional deposits covered by the Report for which the Competent Person signing this form is accepting responsibility:

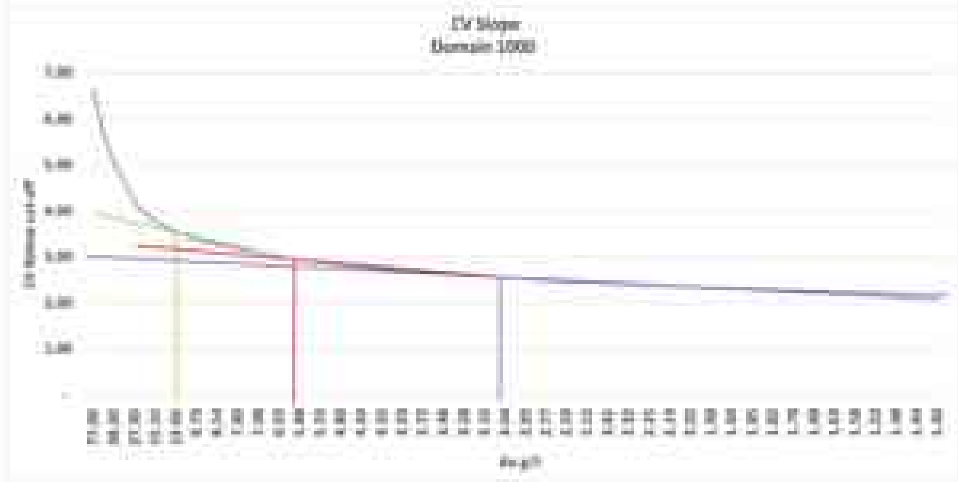
None

Additional Reports related to the deposit for which the Competent Person signing this form is accepting responsibility:

None

Signature of Competent Person:	Date:
Professional Membership: *(insert organisation name)*	Membership Number:
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

Appendix B Grade Cap Analysis



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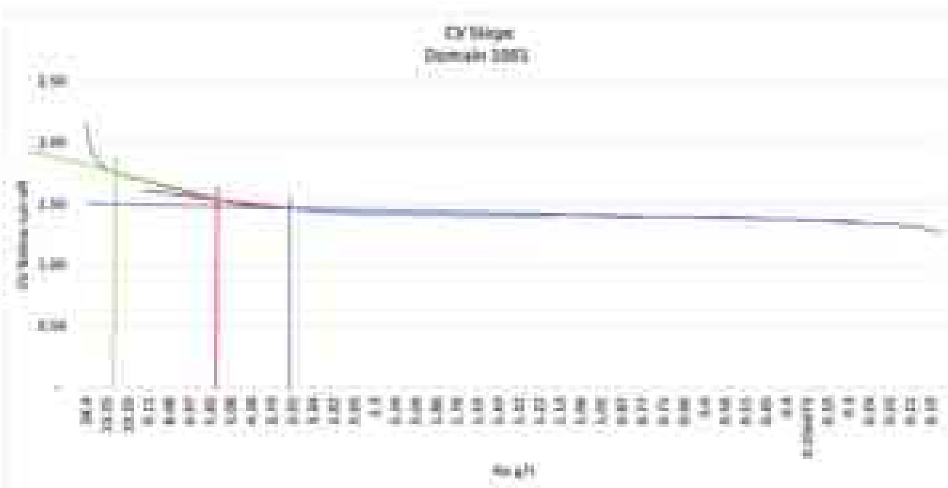

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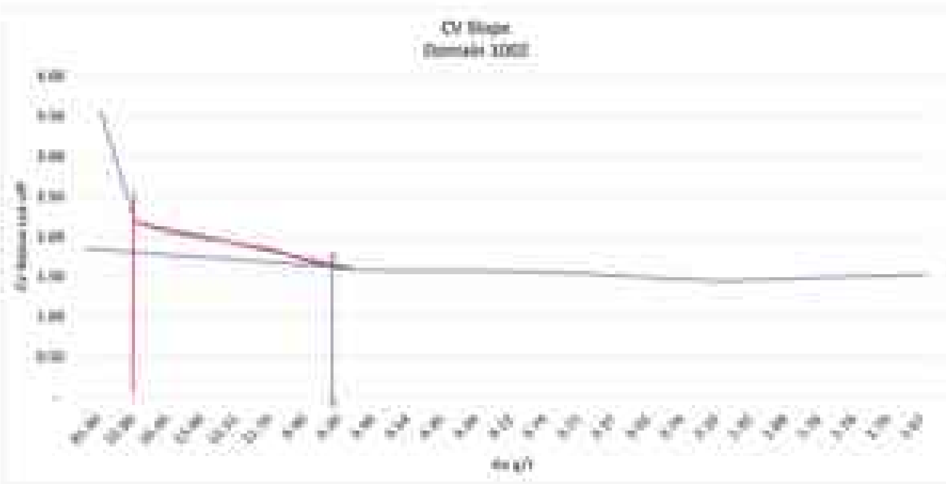
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Au g/t

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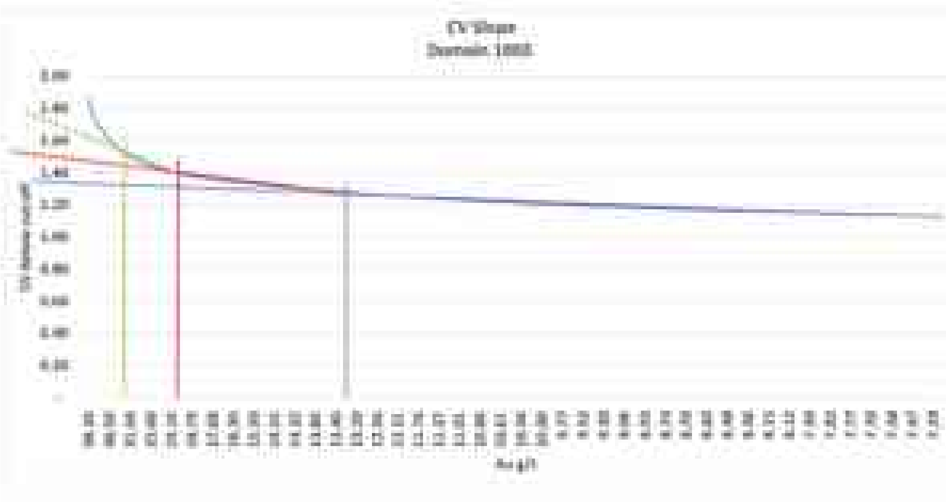
CV Slope
Au g/t

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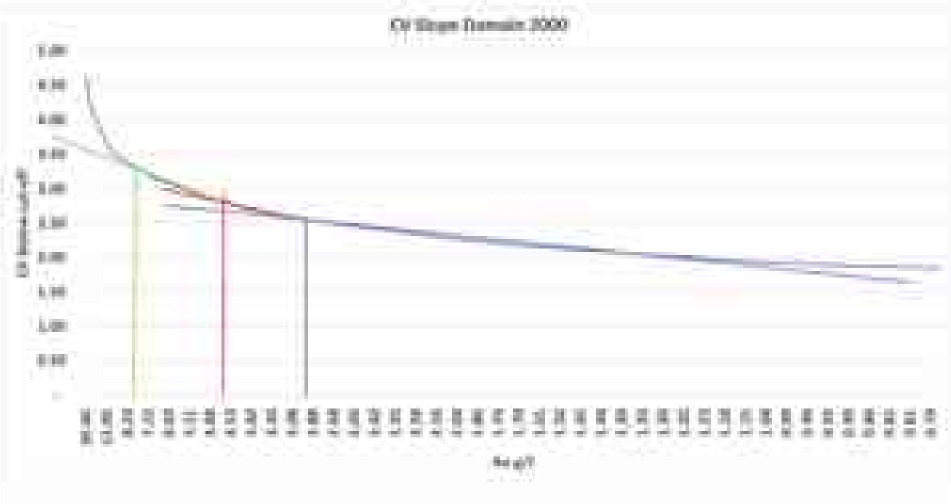
CV Slope Domain 2000
Au g/t

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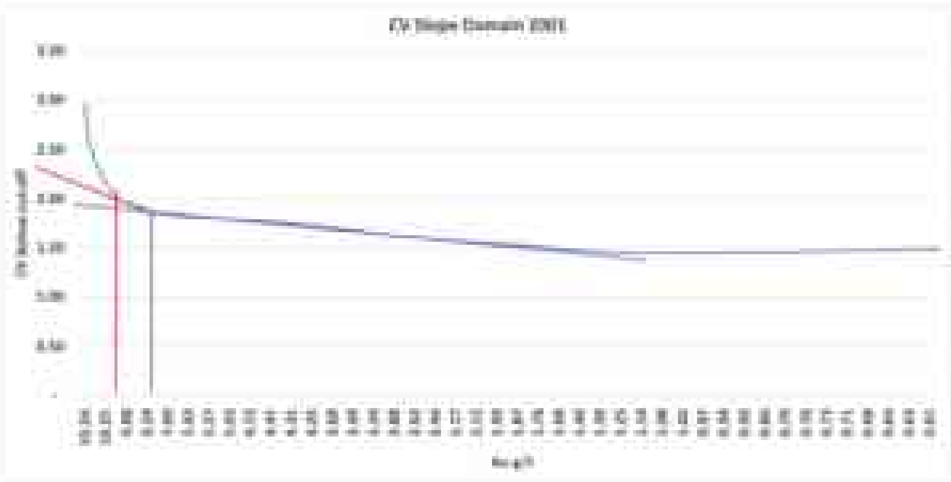
CV Slope Domain 2001
Au g/t

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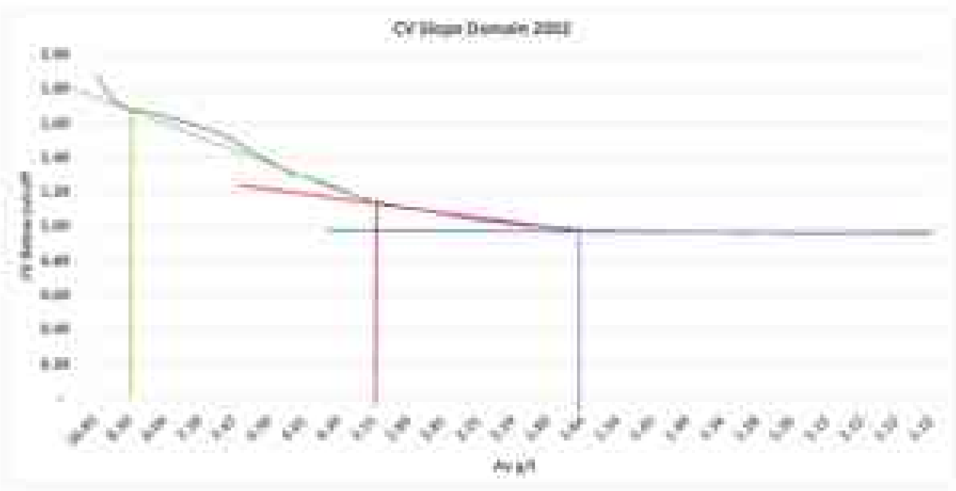
CV Slope Domain 2002
Au g/t

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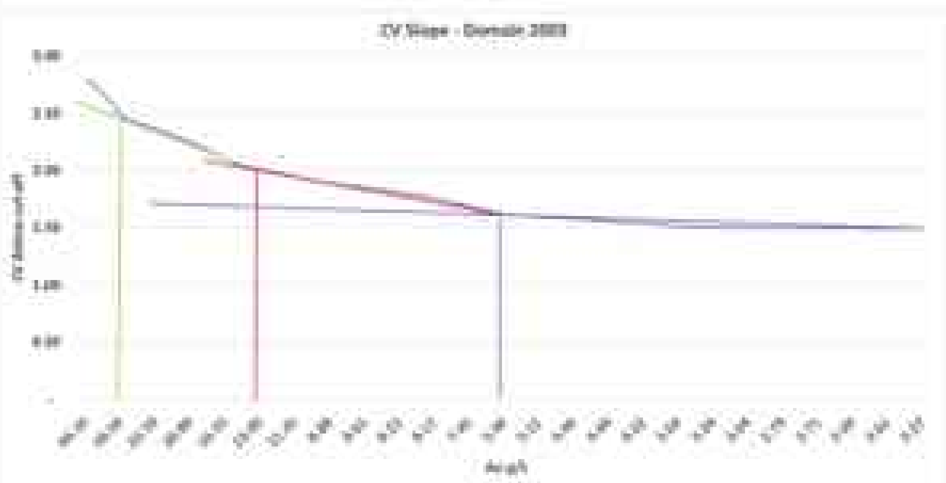
CV Slope - Domain 2003
Au g/t

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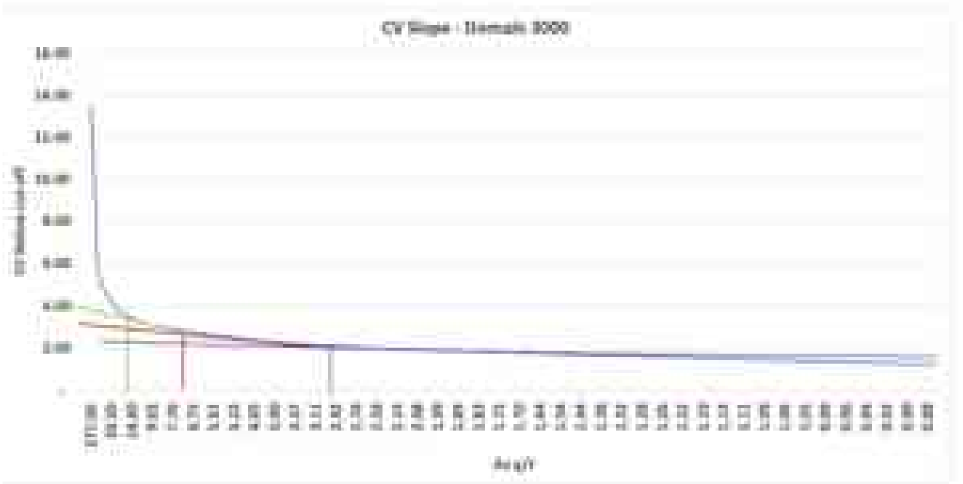
CV Slope - Domain 3000
Au g/t

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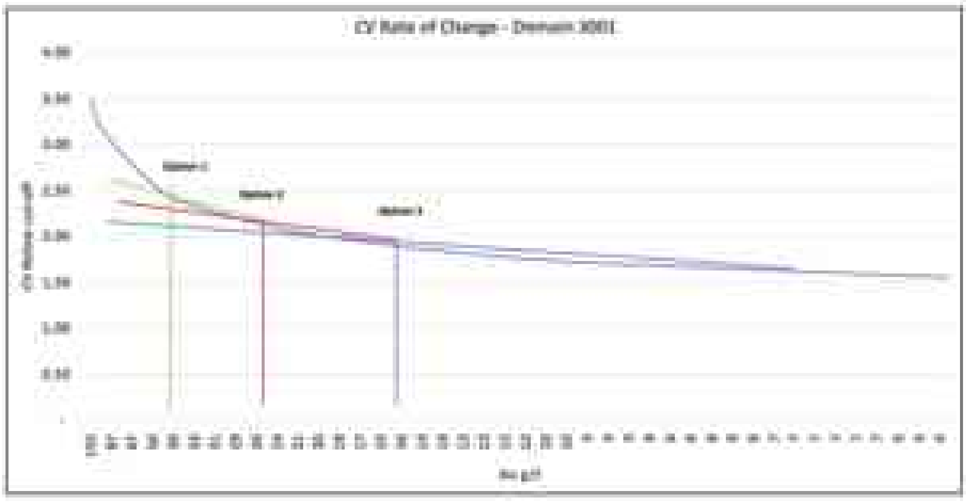
CV Rate of Change - Domain 3001
Au g/t

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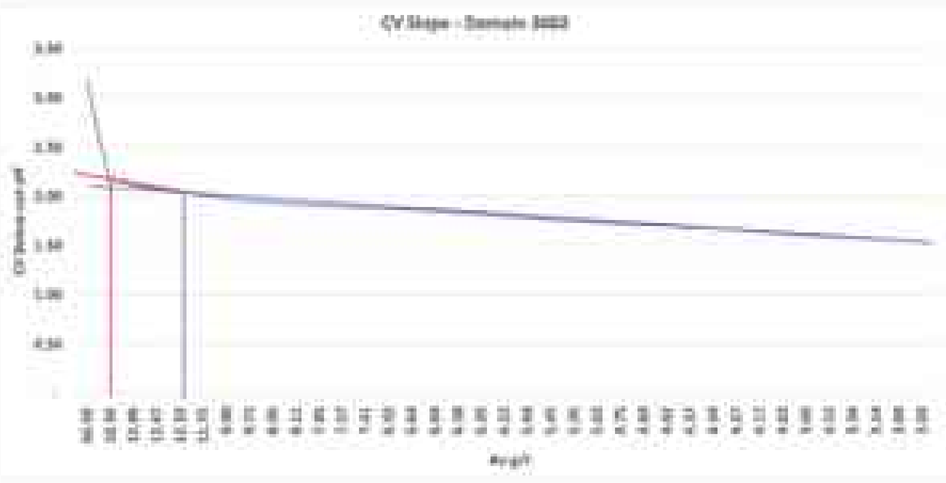
Au g/t

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
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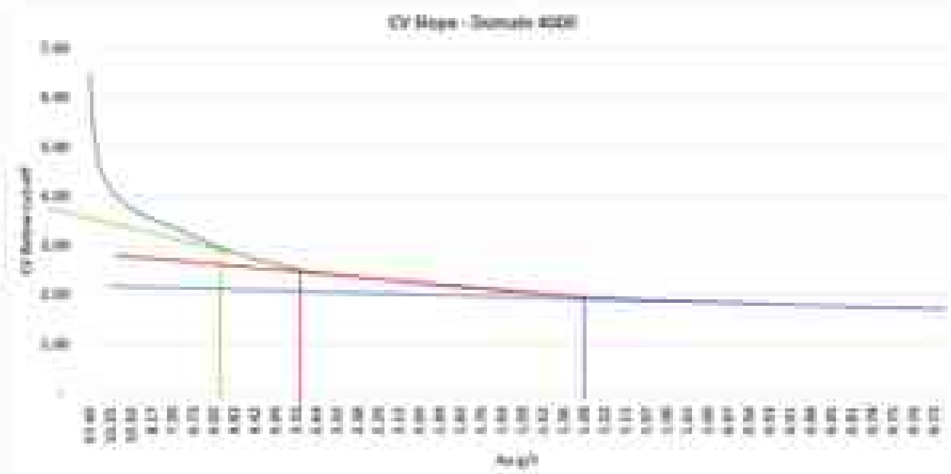
Au g/t

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
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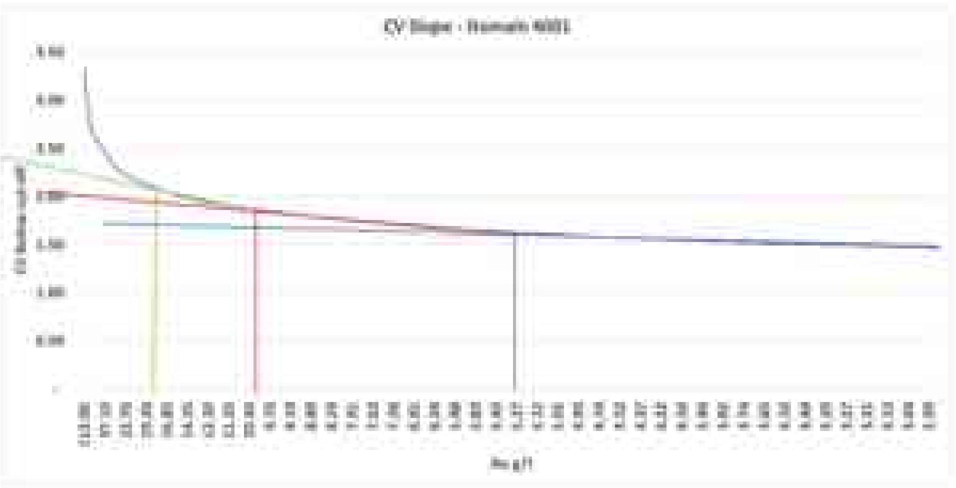

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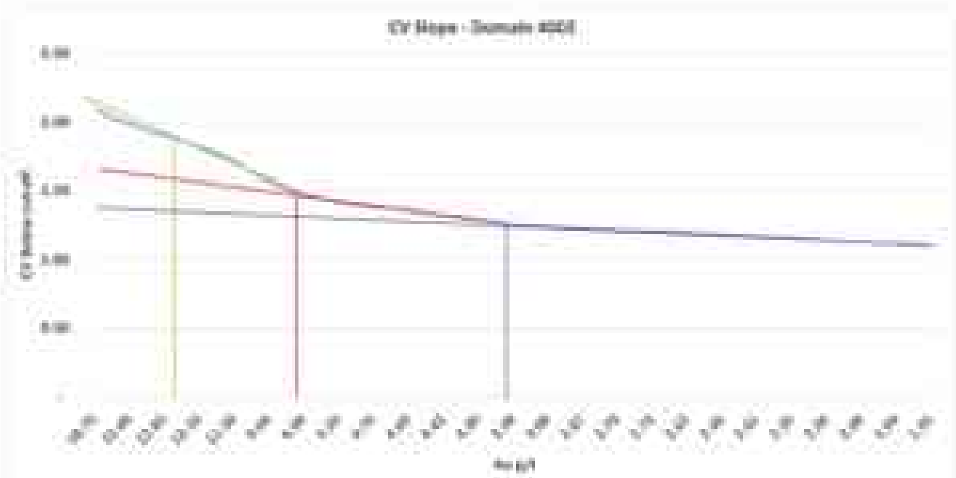

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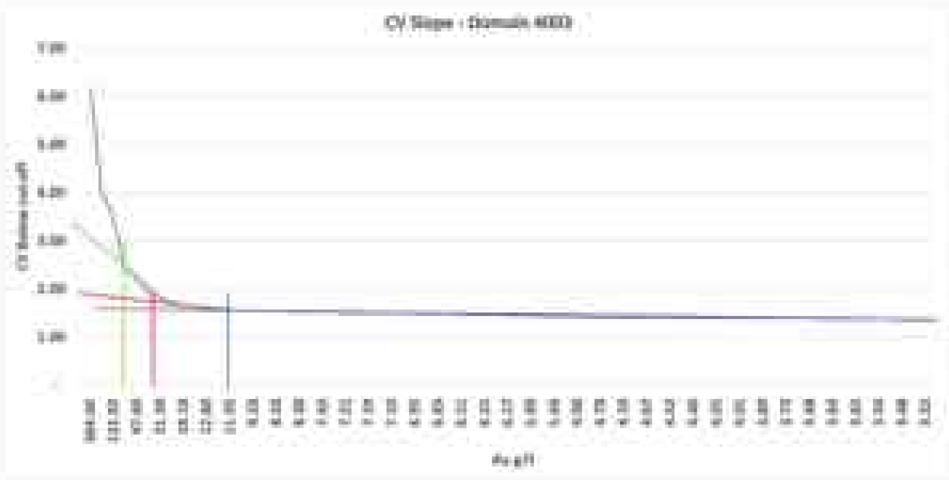
Au g/t

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
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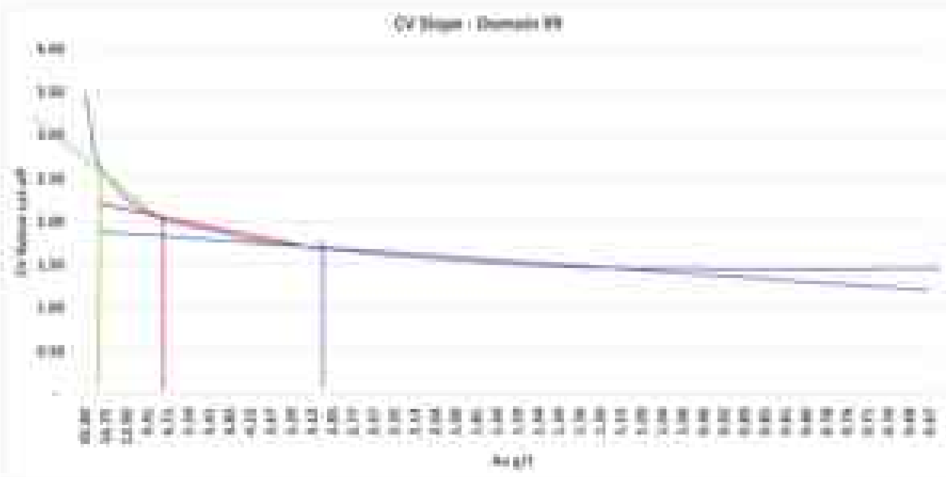
CV Slope : Domain 99
Au g/t

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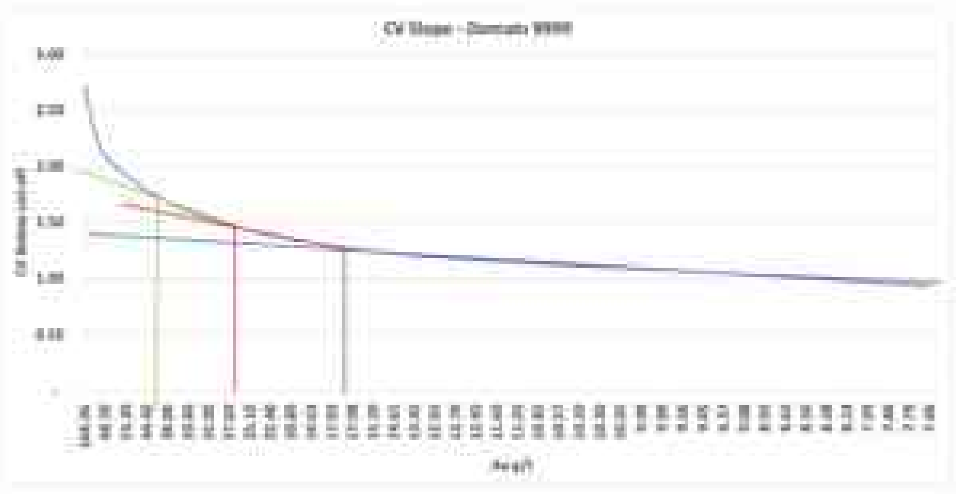
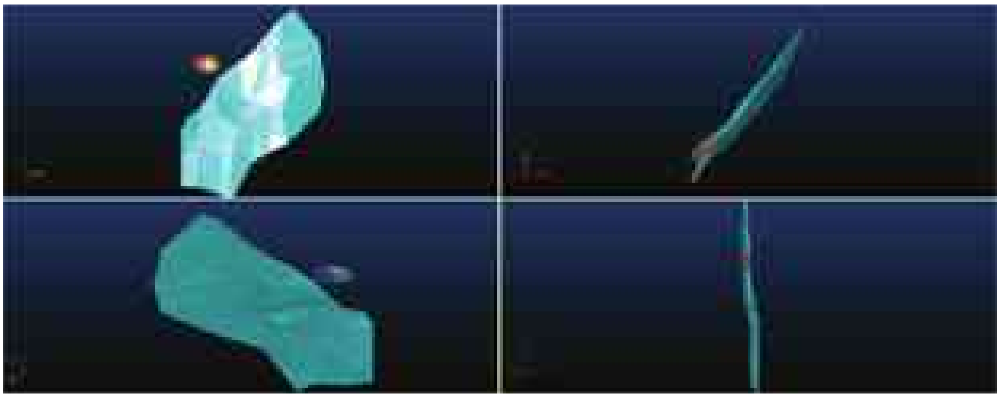
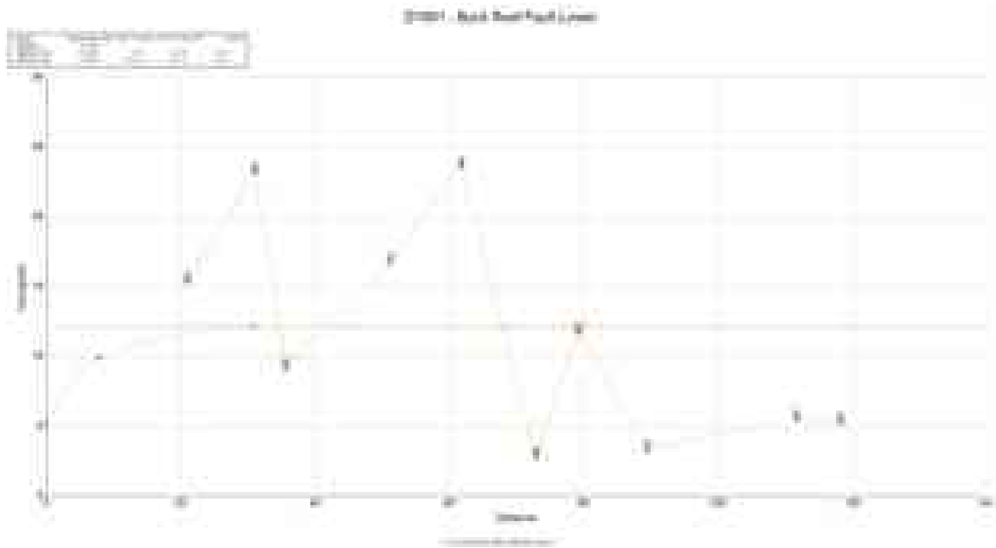
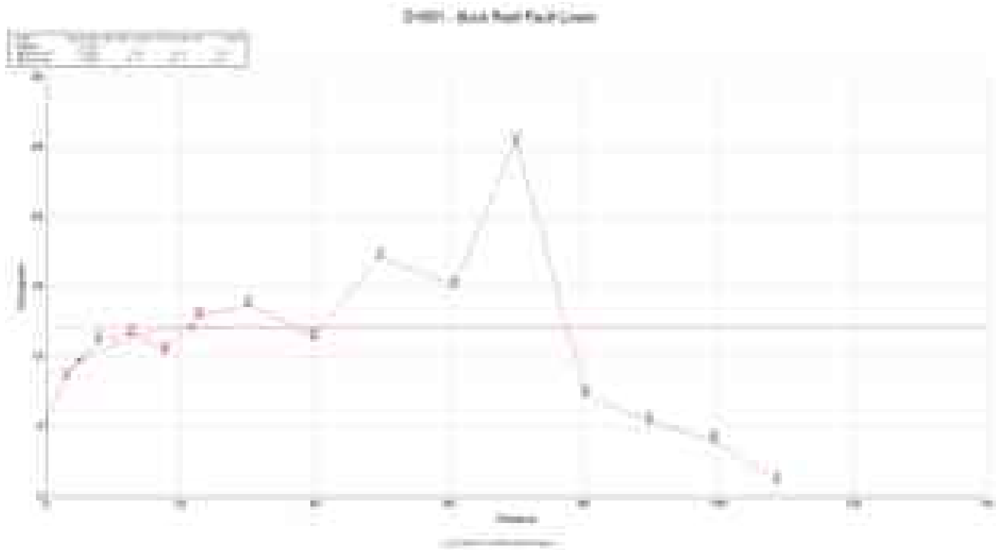


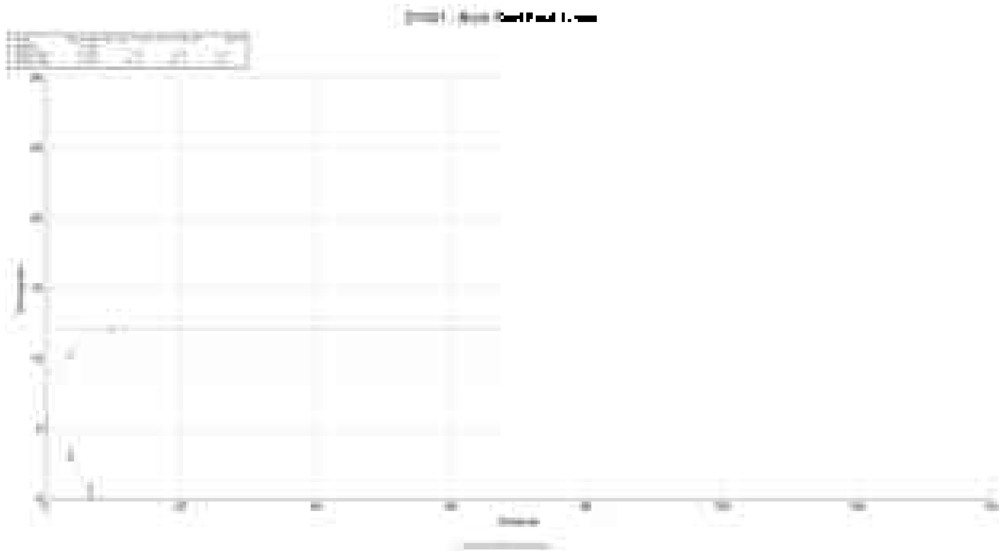
Appendix C Variogram Models

D1001 – Buck Reef Fault Lower (Main)

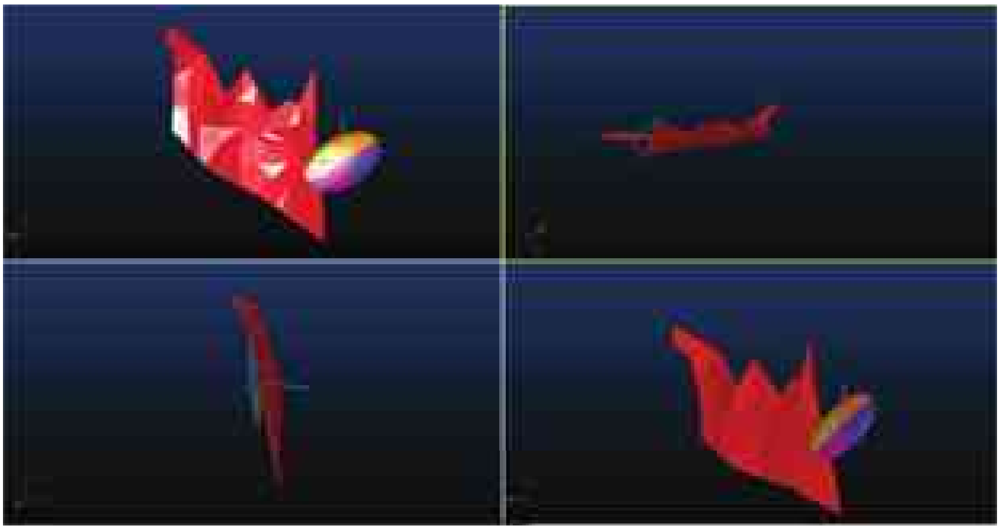


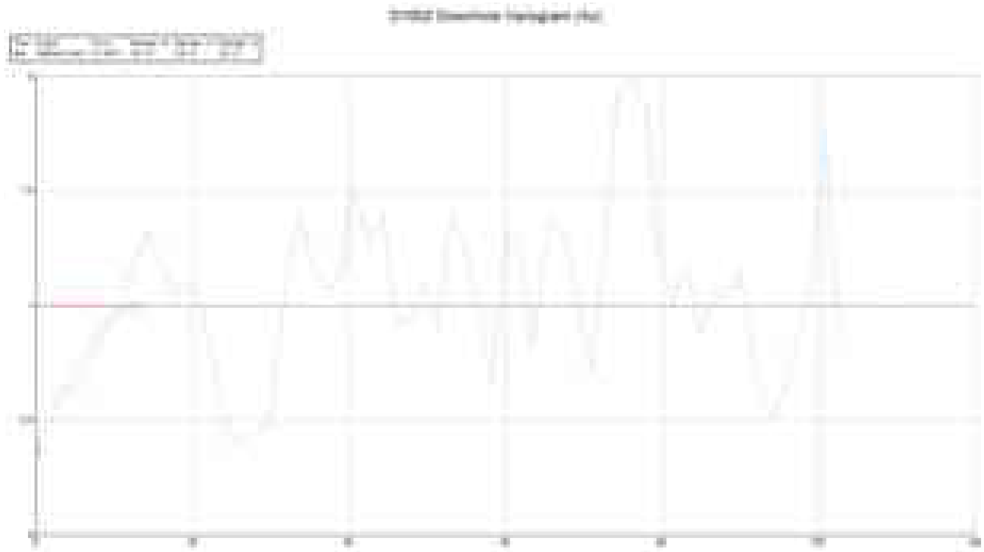


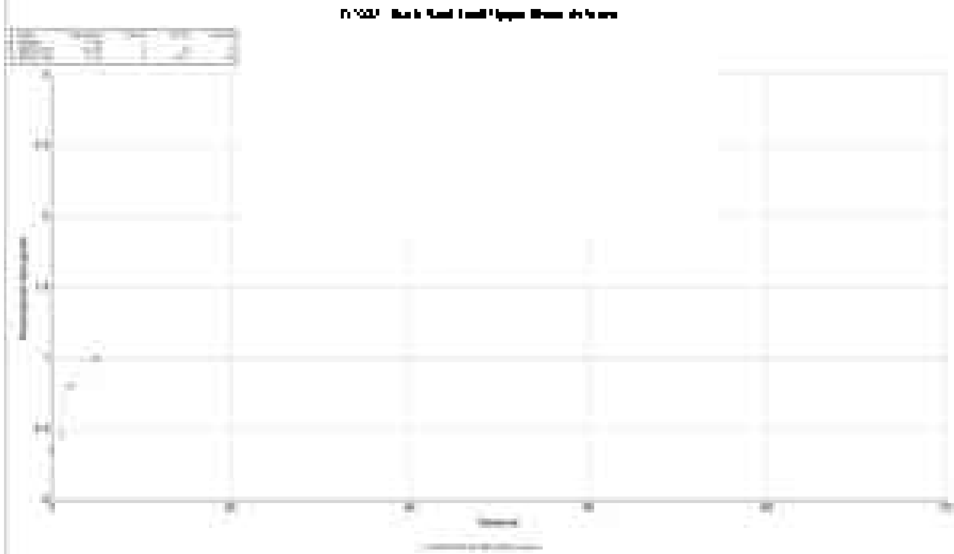


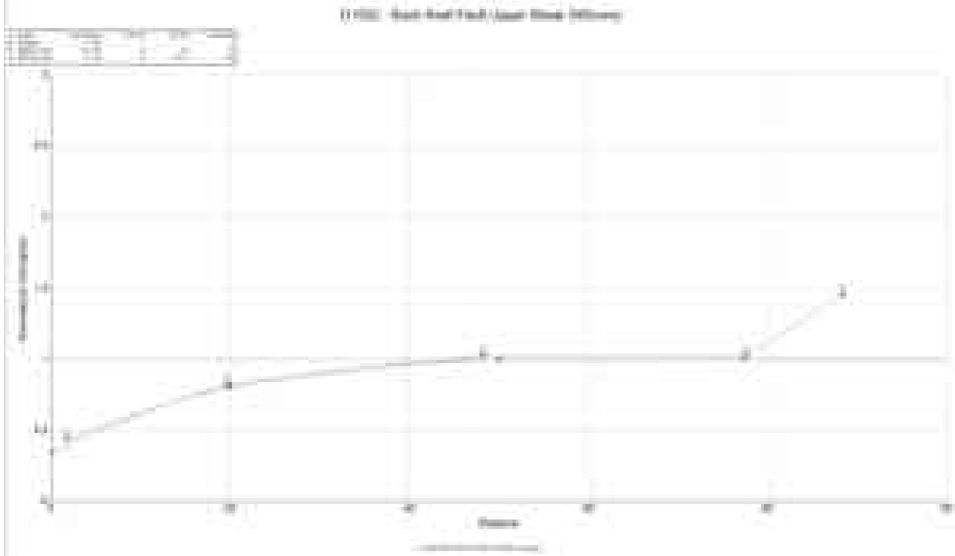


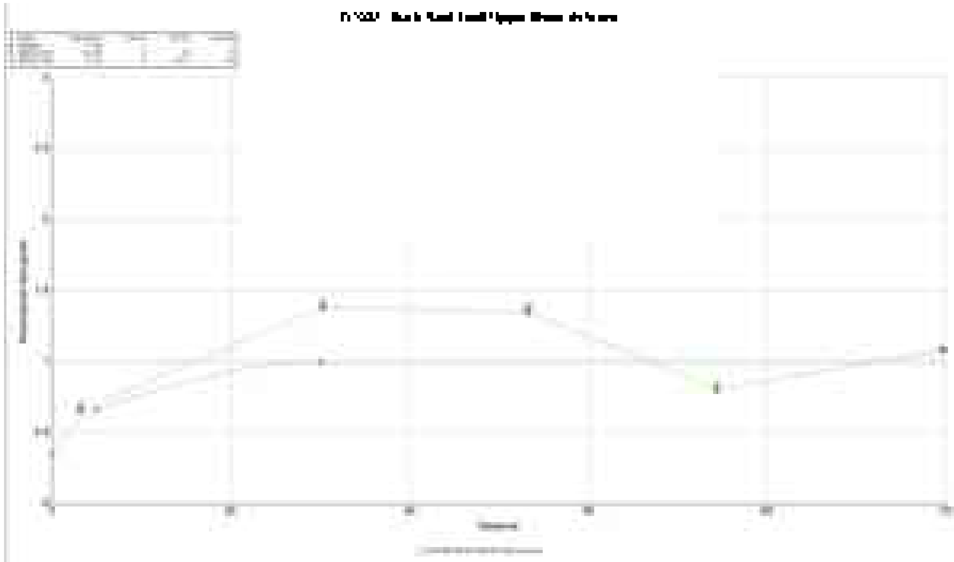
Domain 1002 – Buck Reef Fault Shear



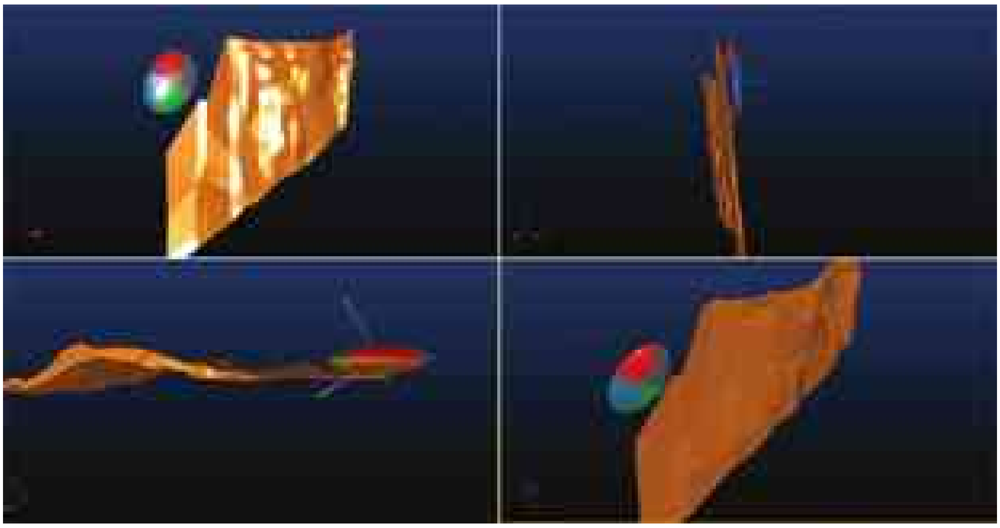


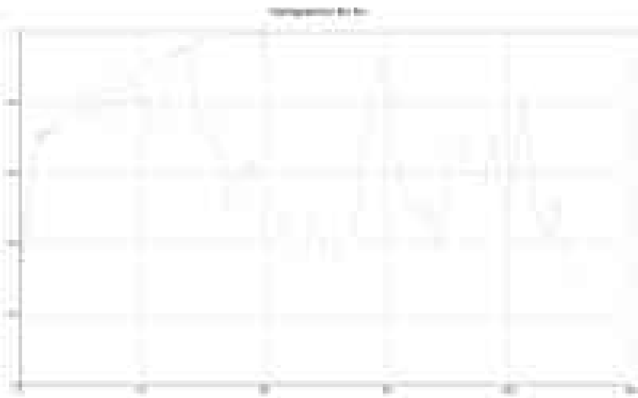


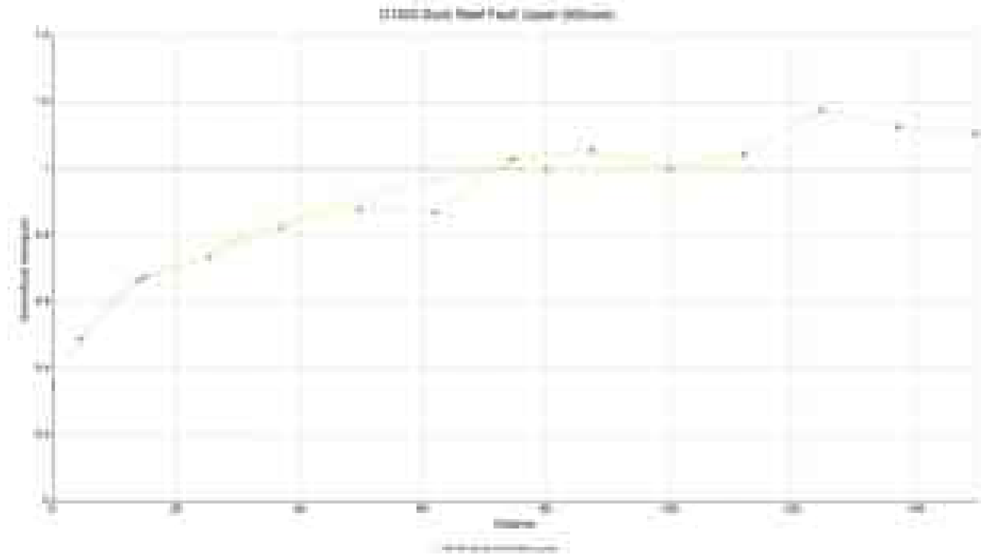


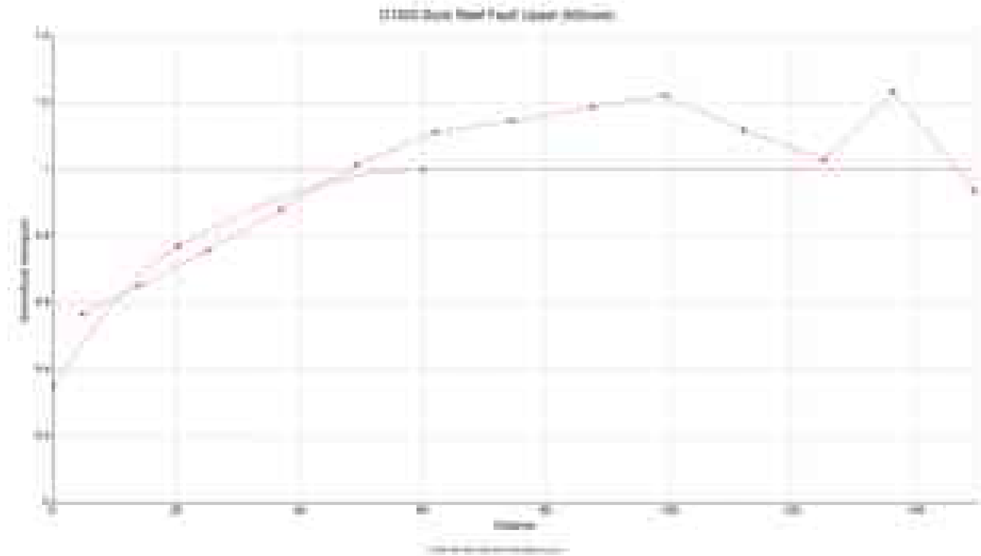


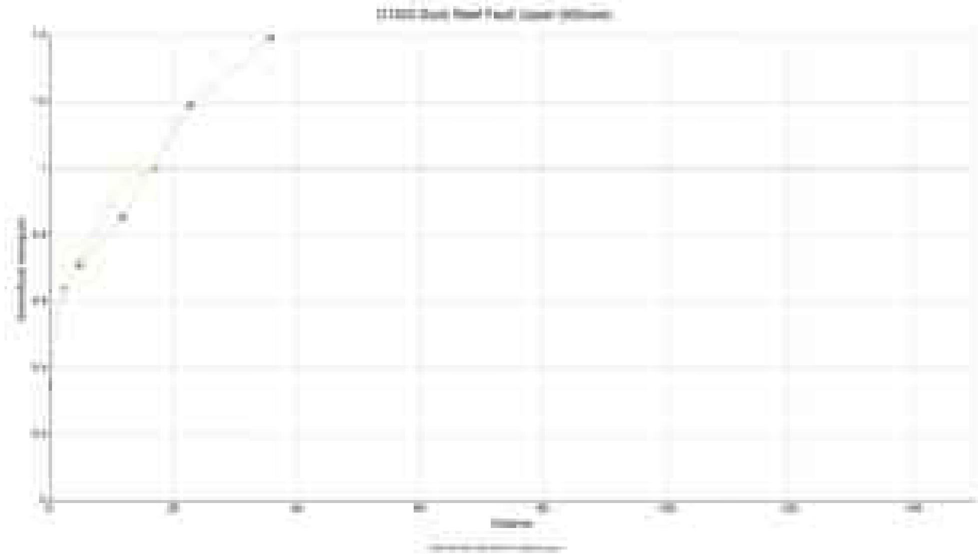
Domain 1003 – Buck Reef Fault Upper



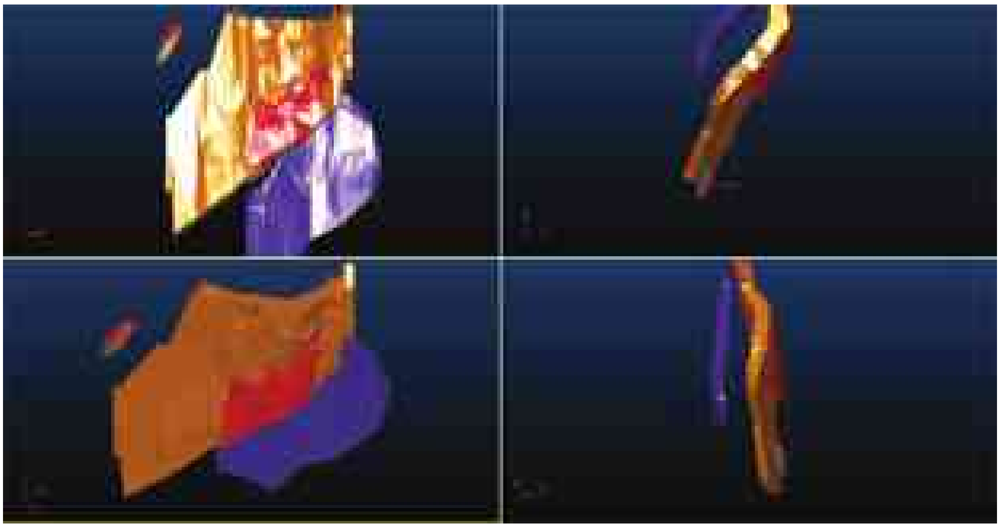


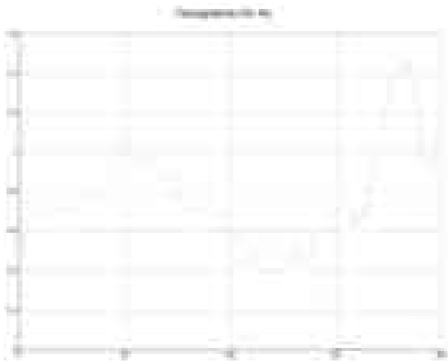


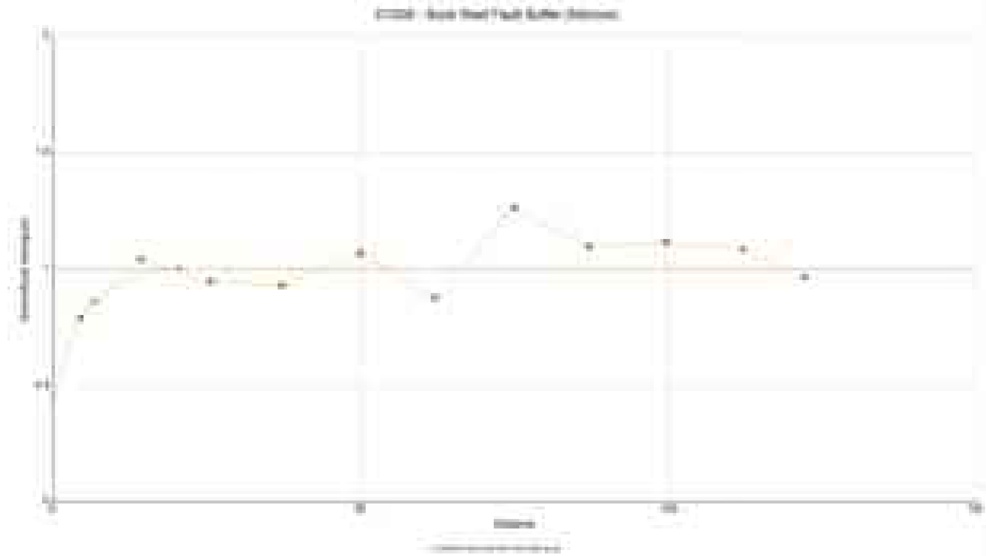


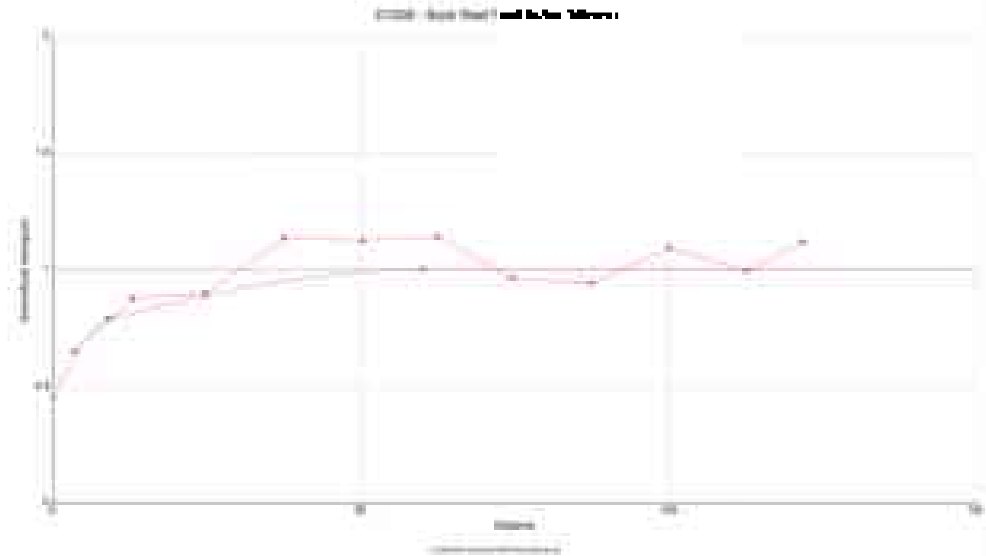


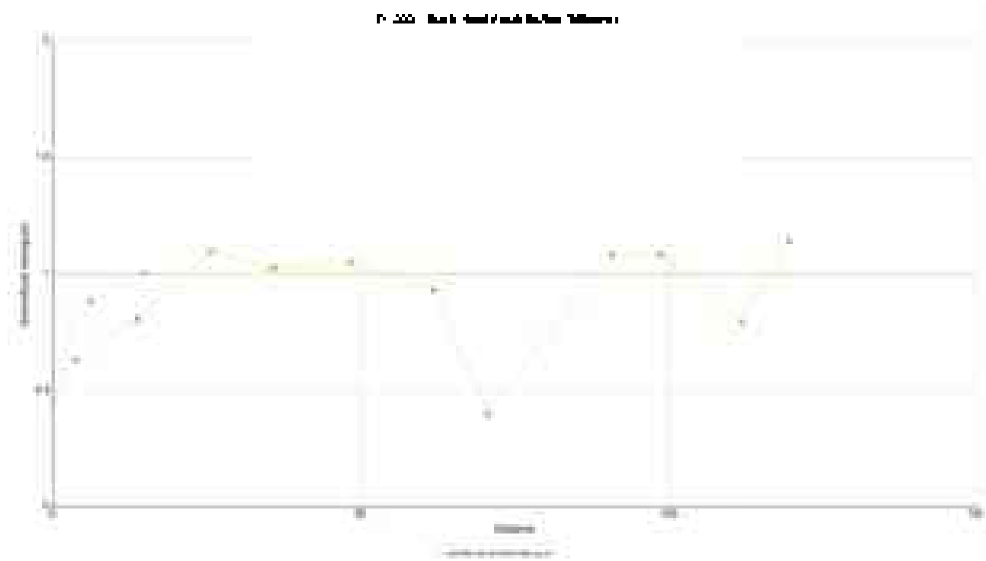
Domain 1000 – Buck Reef Fault Buffer



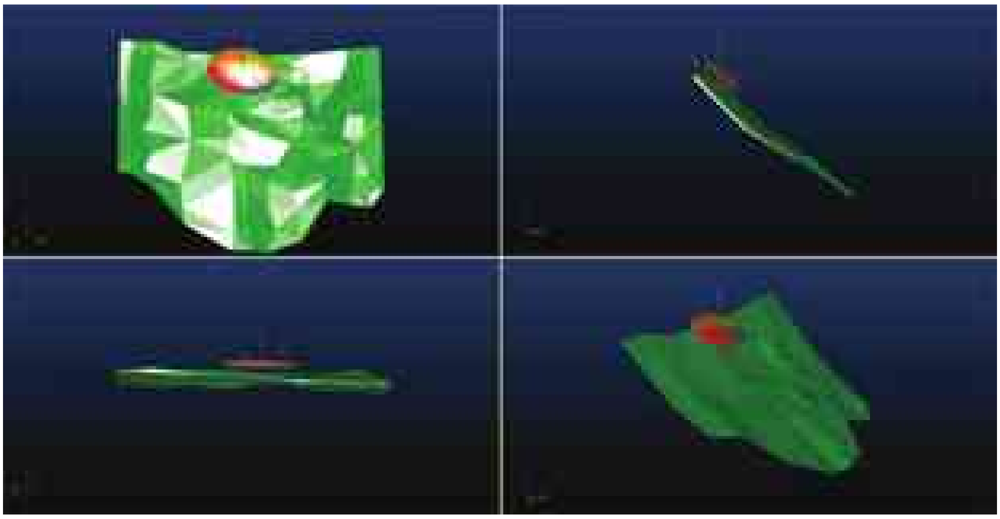


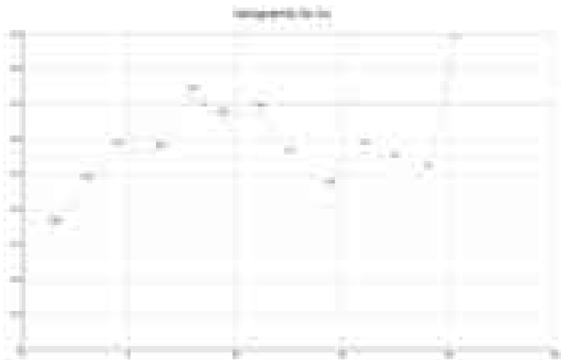


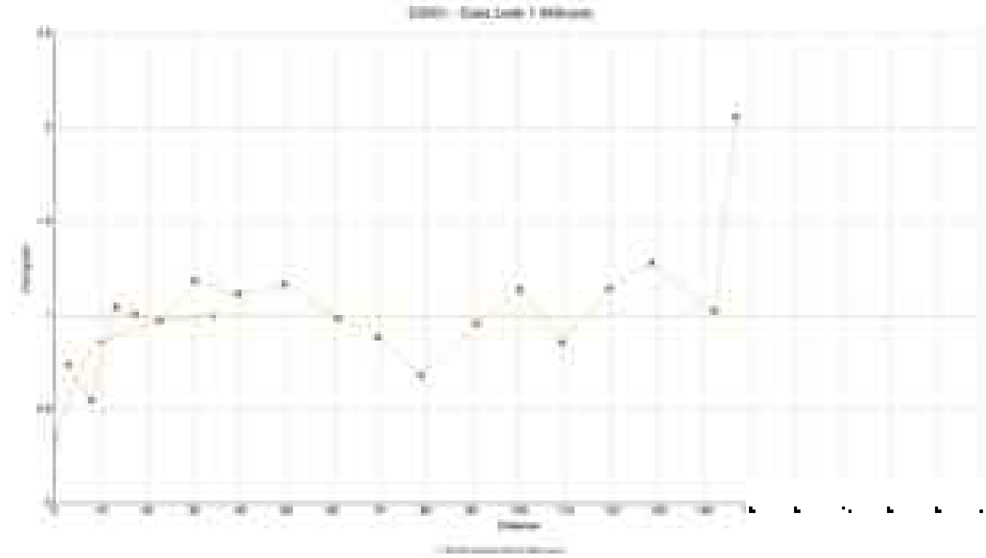


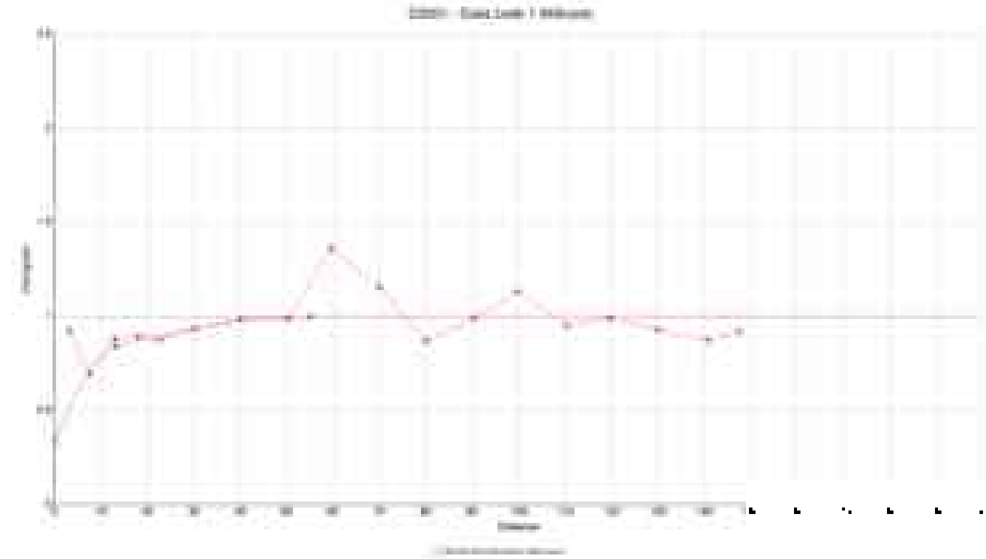


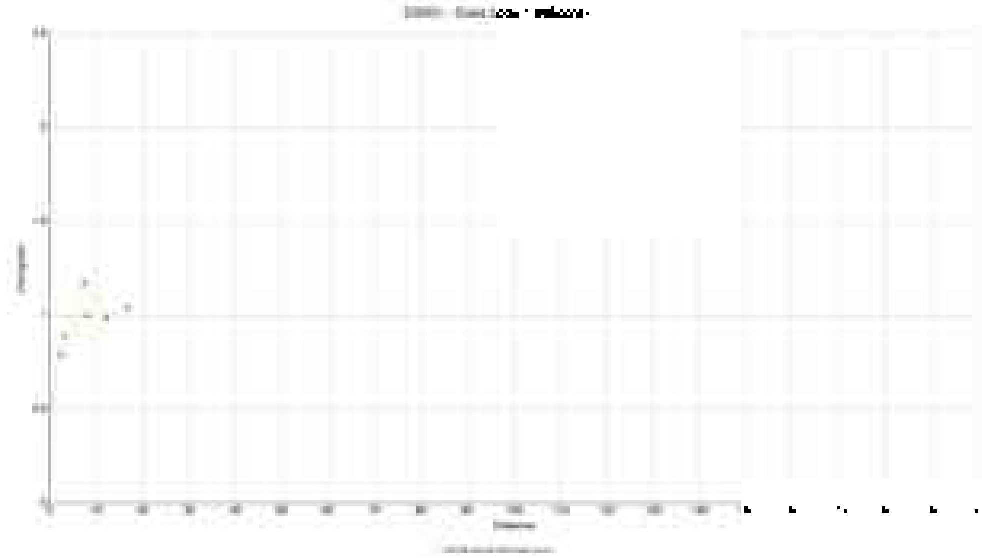
Domain 2001 – Duke Lode 1



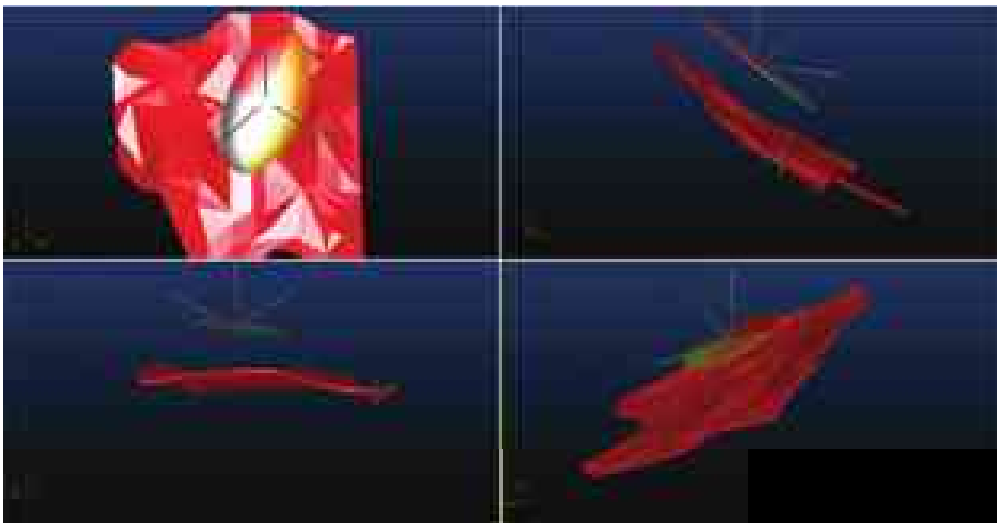


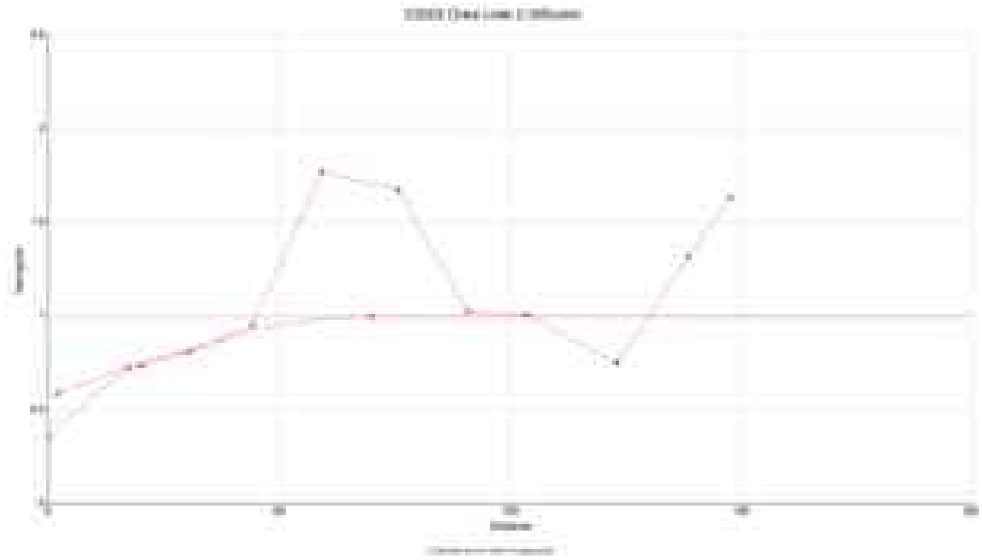


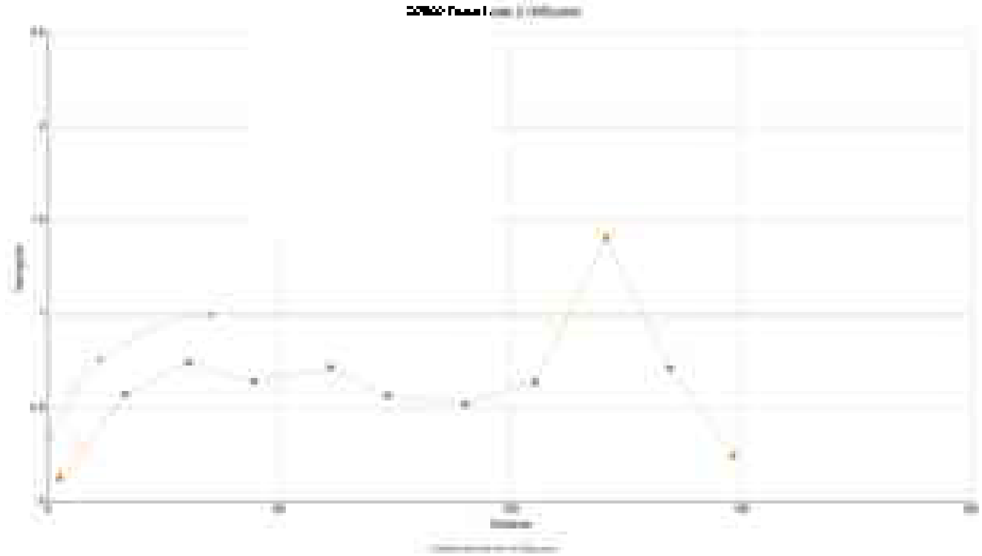


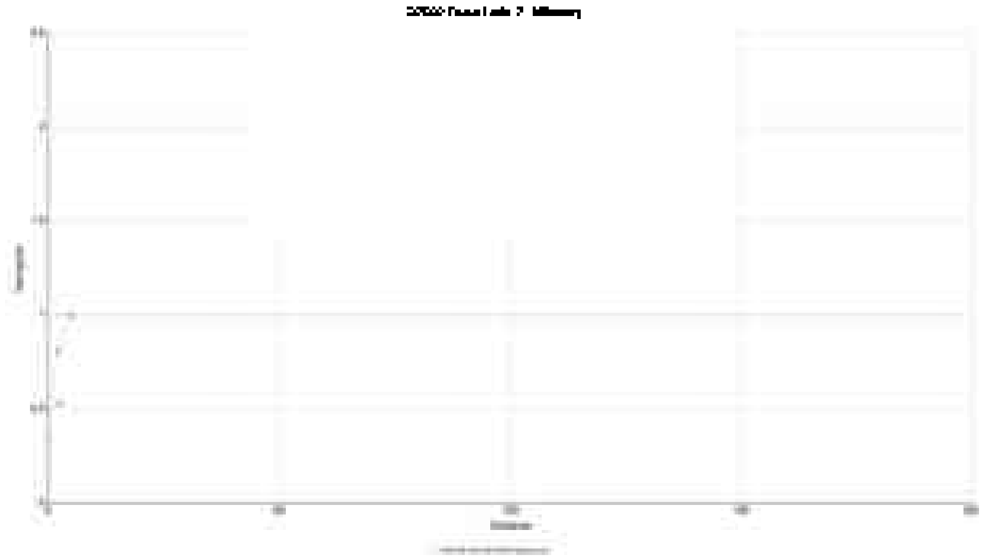


Domain 2002 – Duke Lode 2

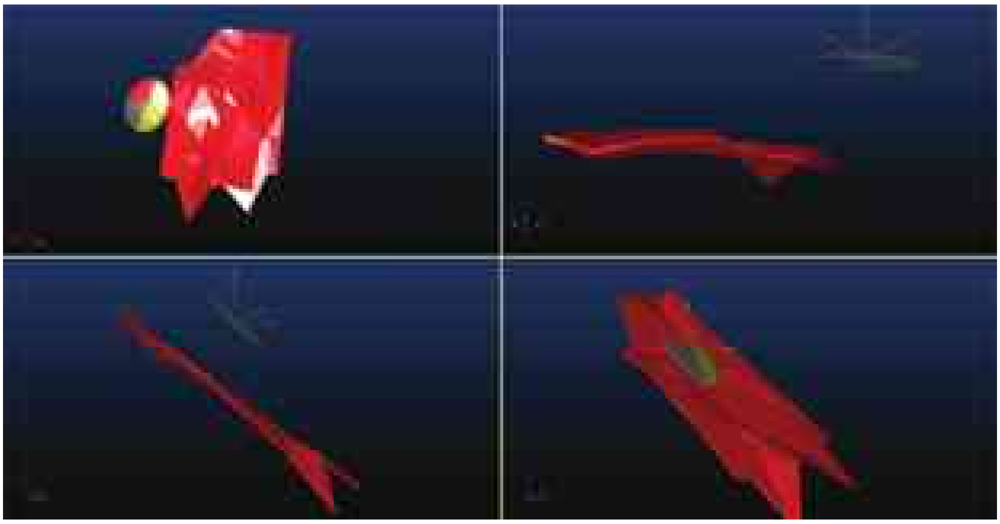


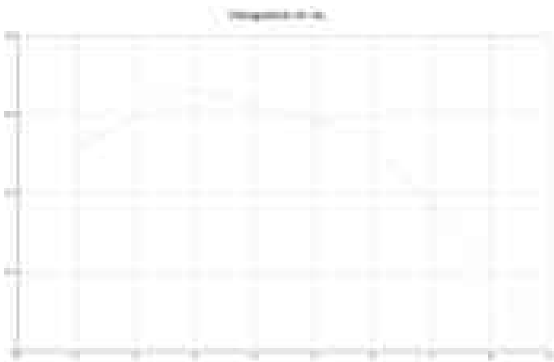


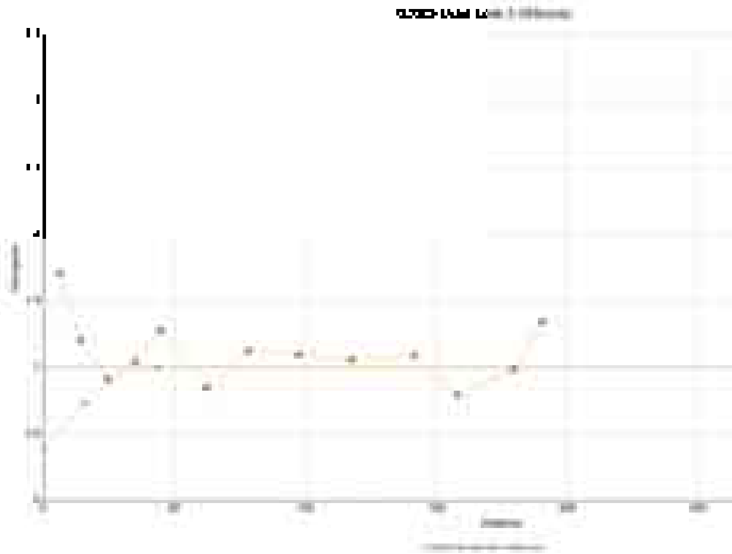


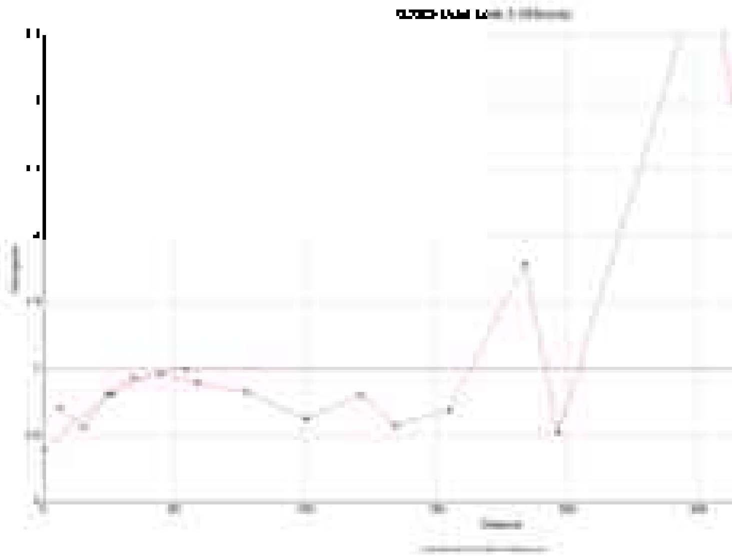


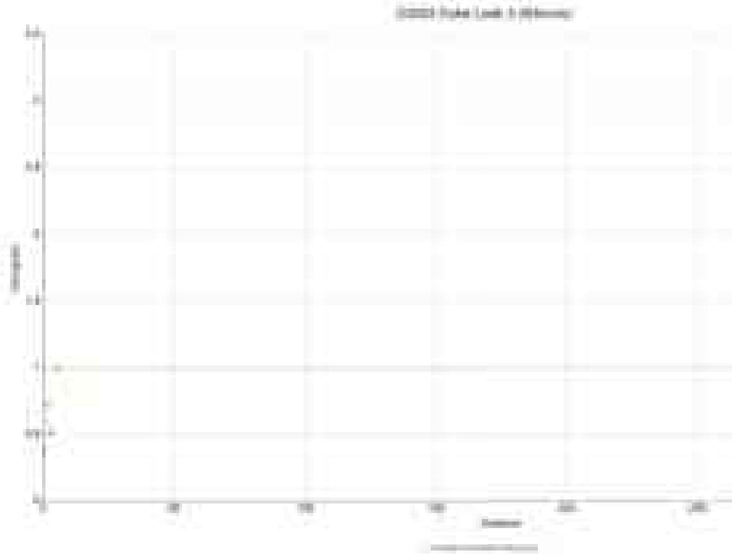
Domain 2003 – Duke Lode 3



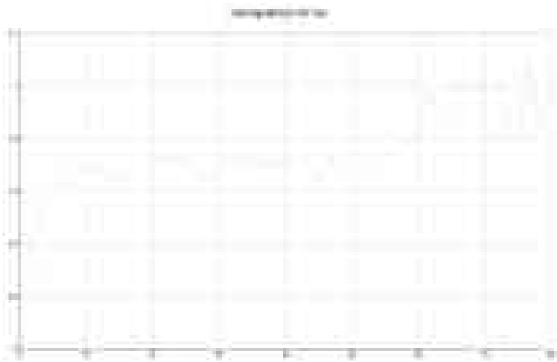


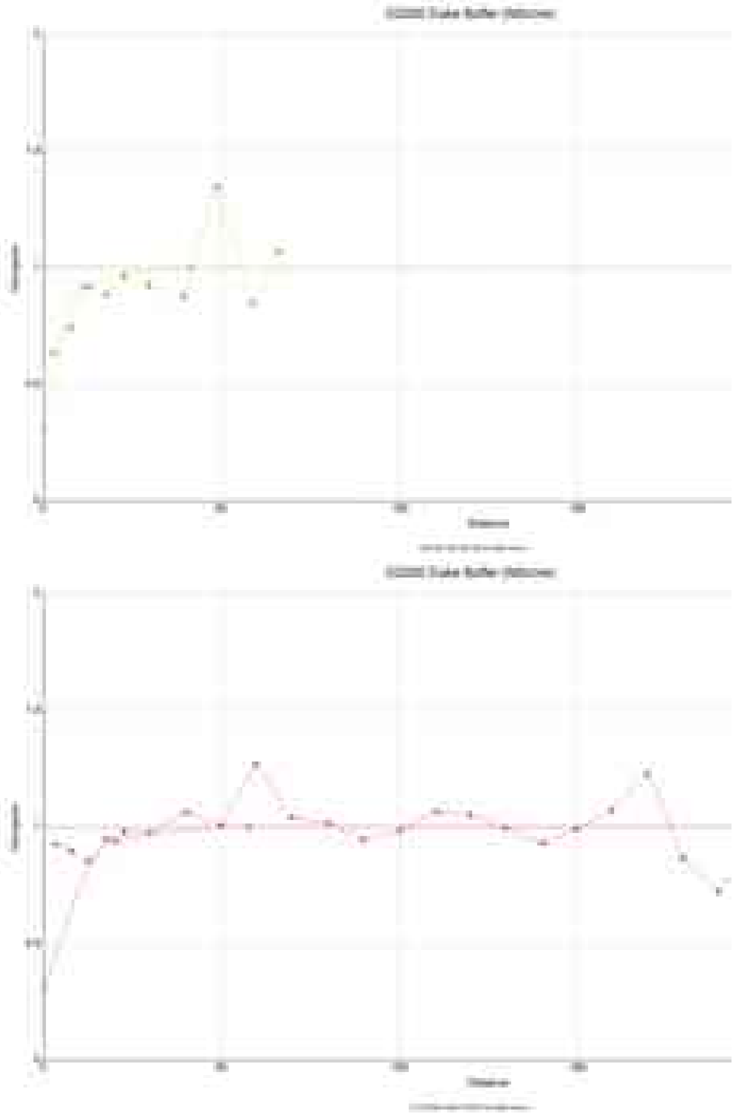


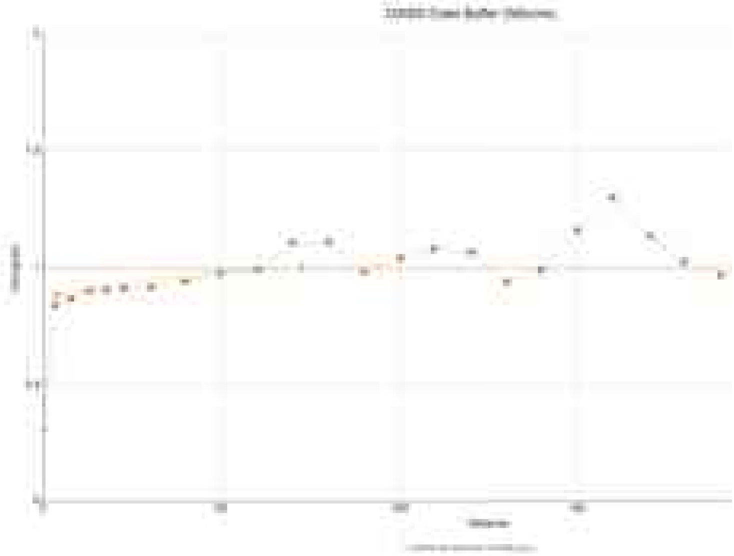




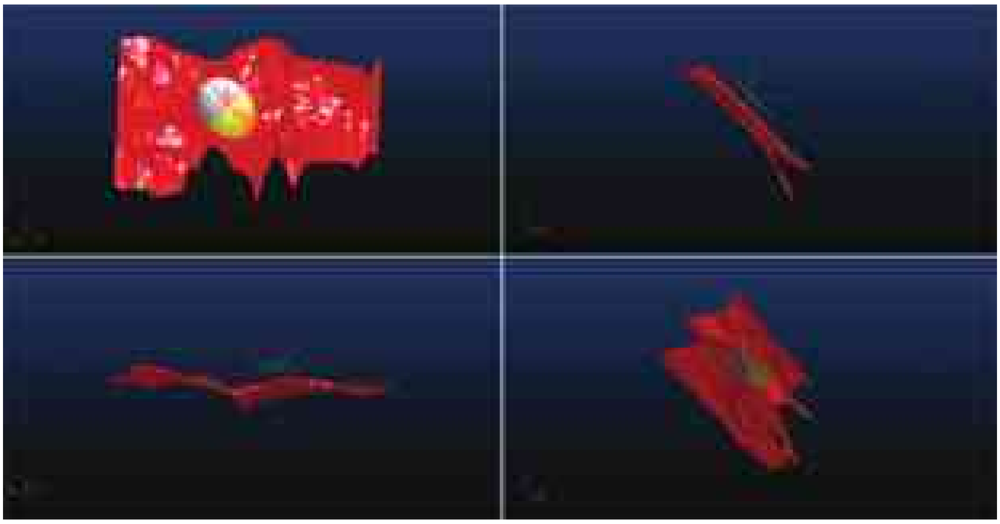
Domain 2000 – Duke Buffer

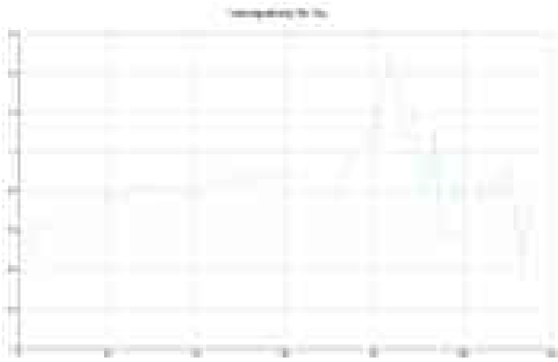


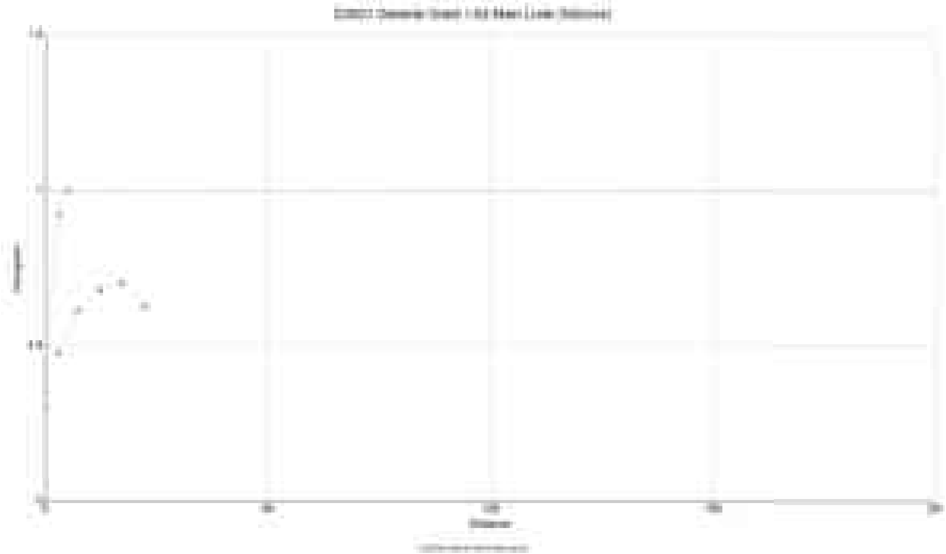


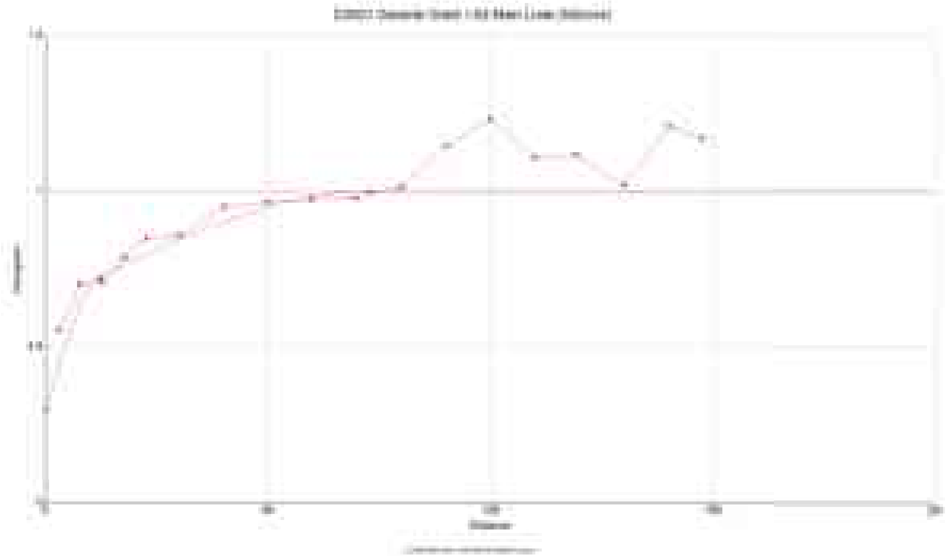


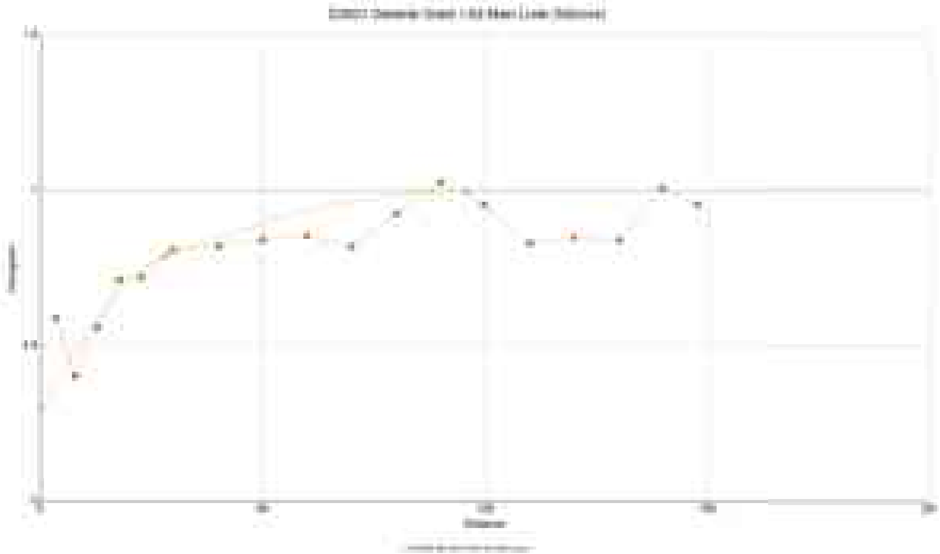
Domain 3001 – Grant Lode 1



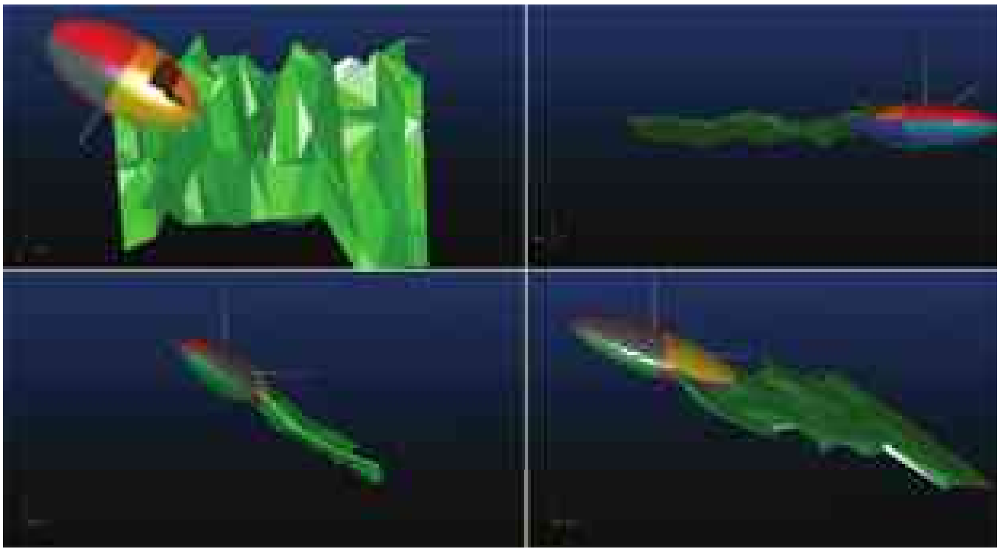


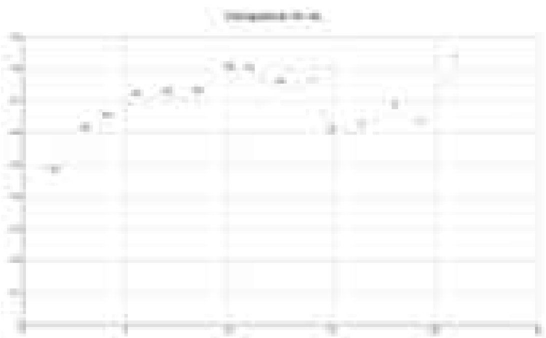


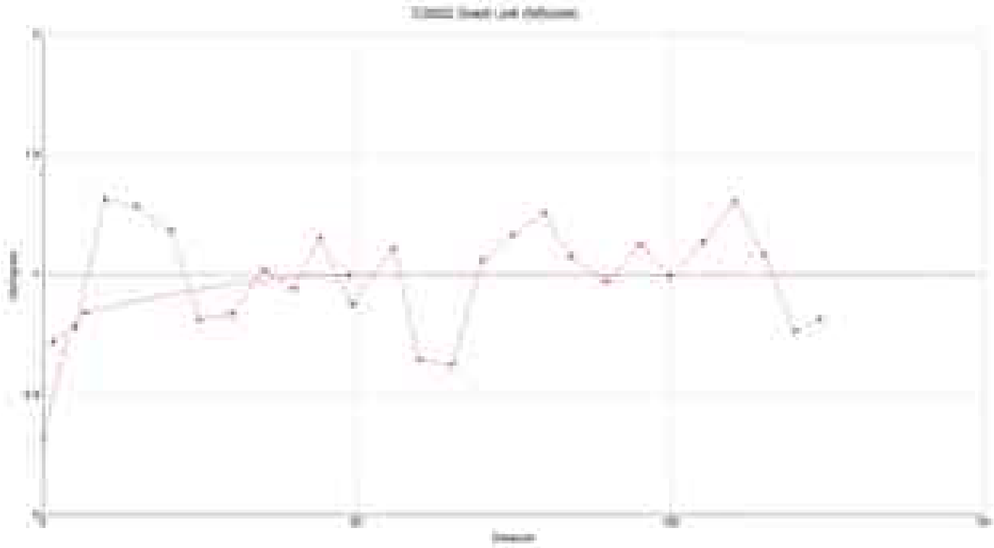


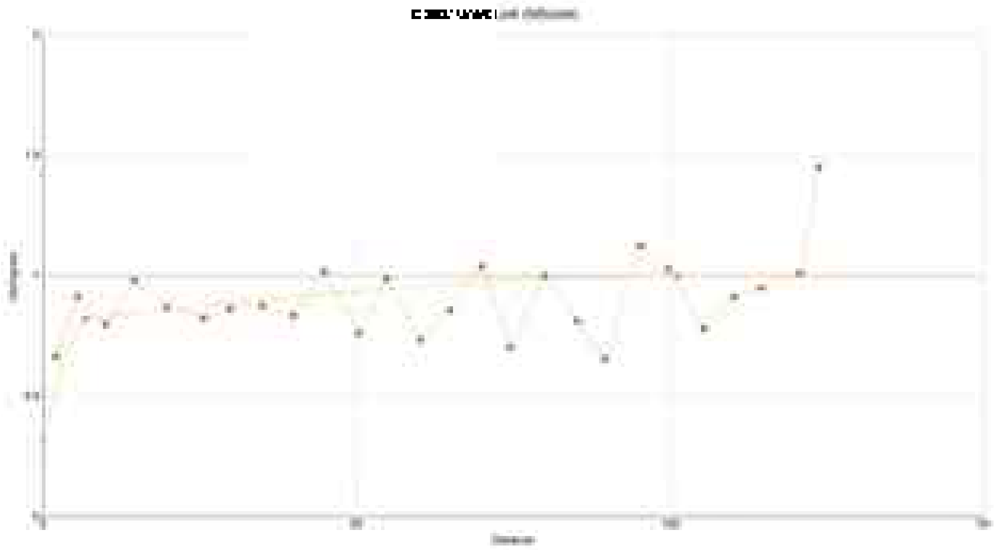


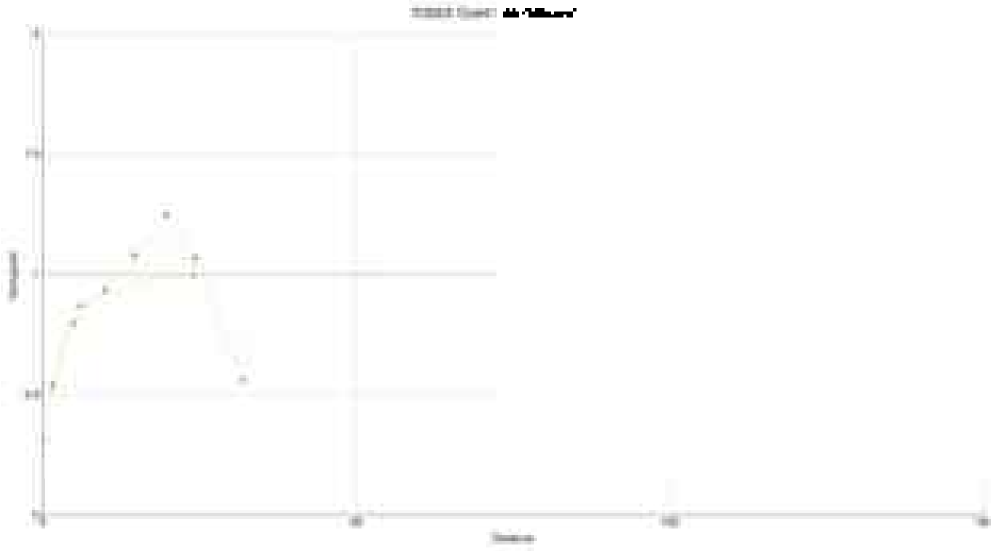
Domain 3002 – Grant Lode 2



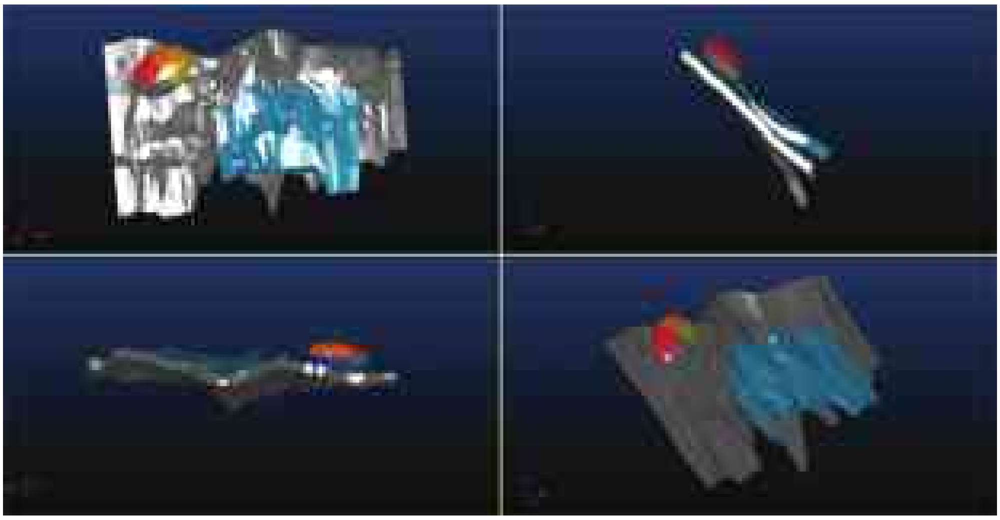


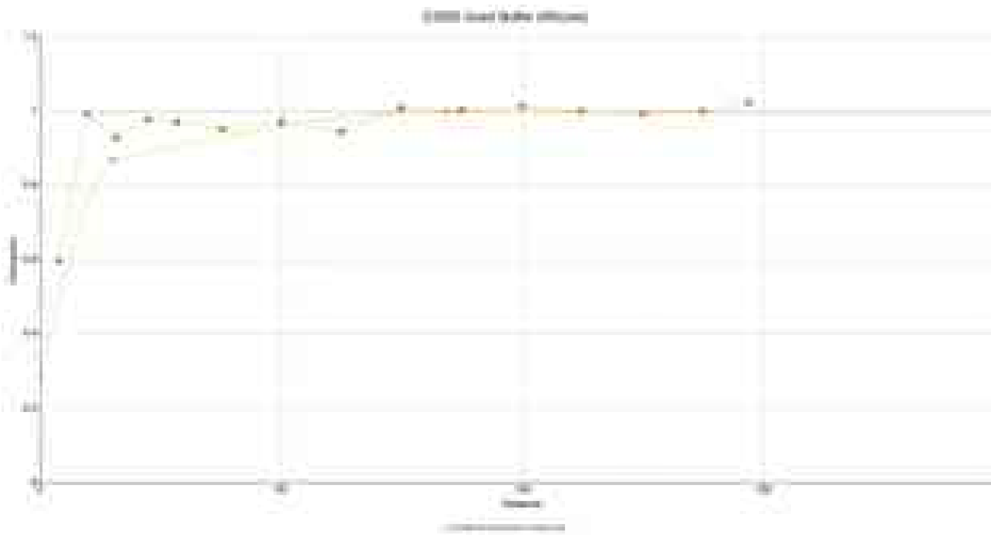


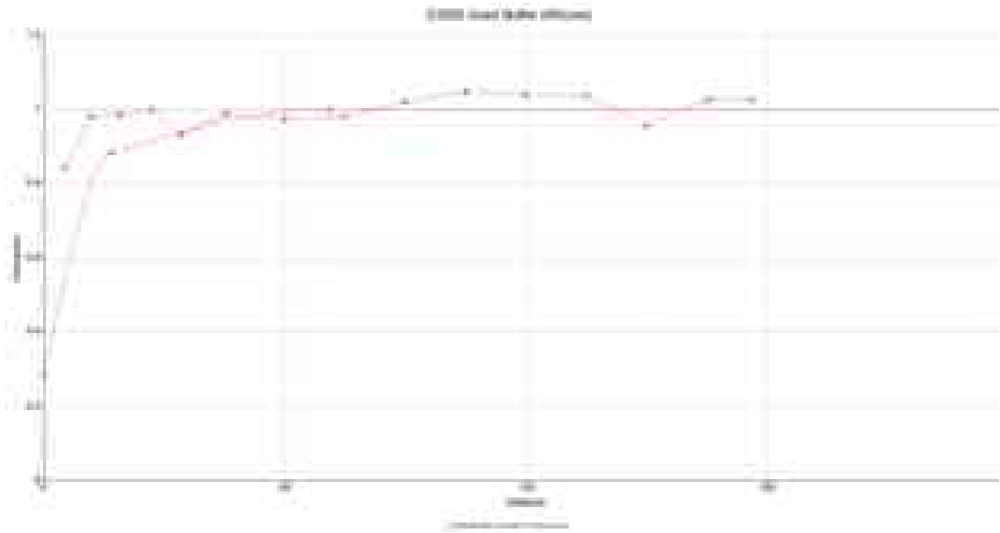


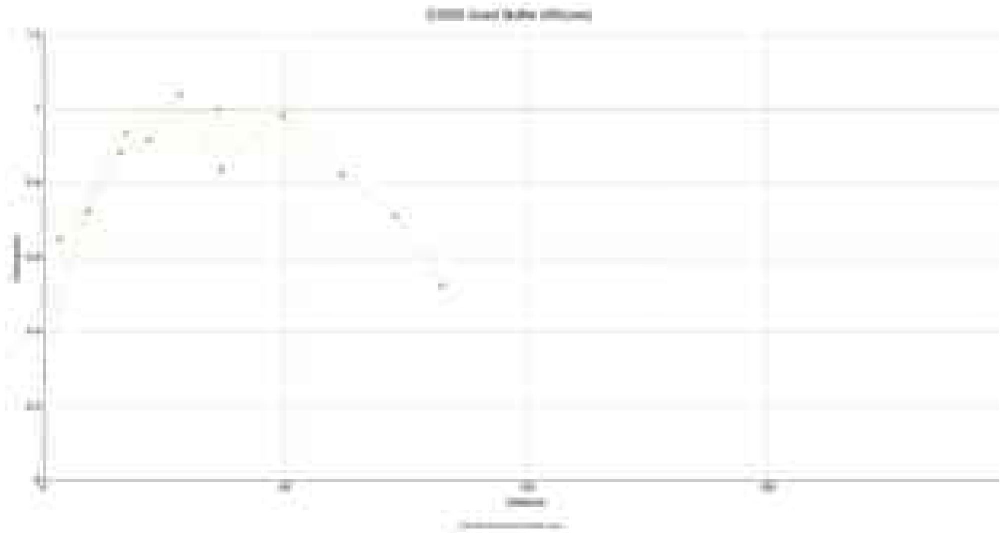


Domain 3000 Grant Buffer

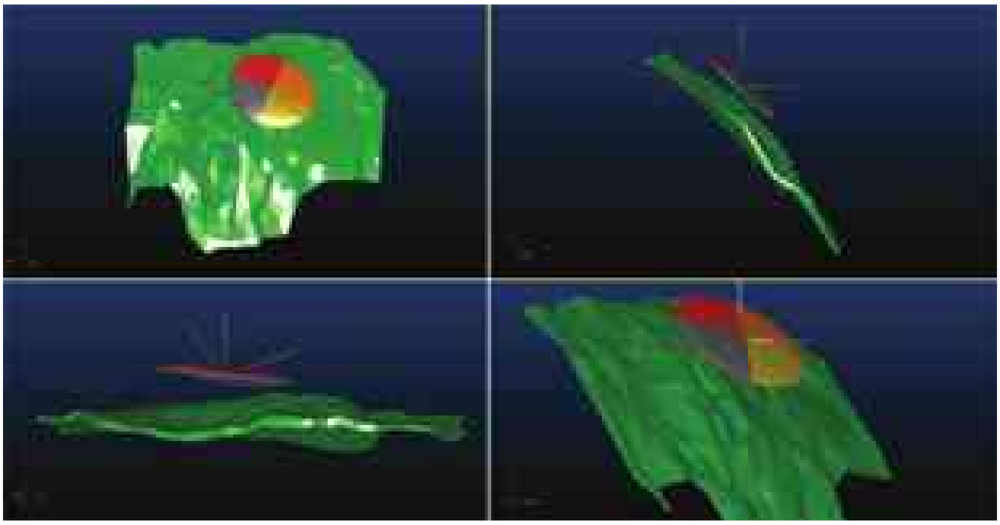


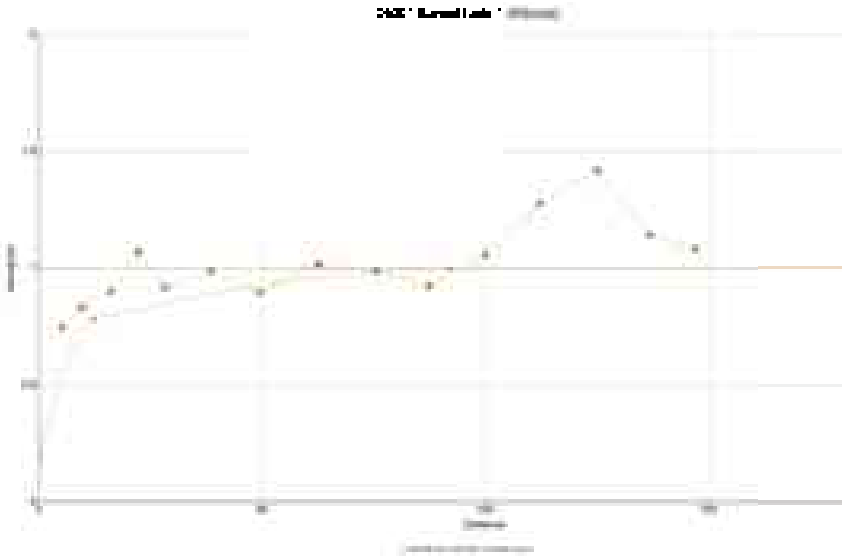


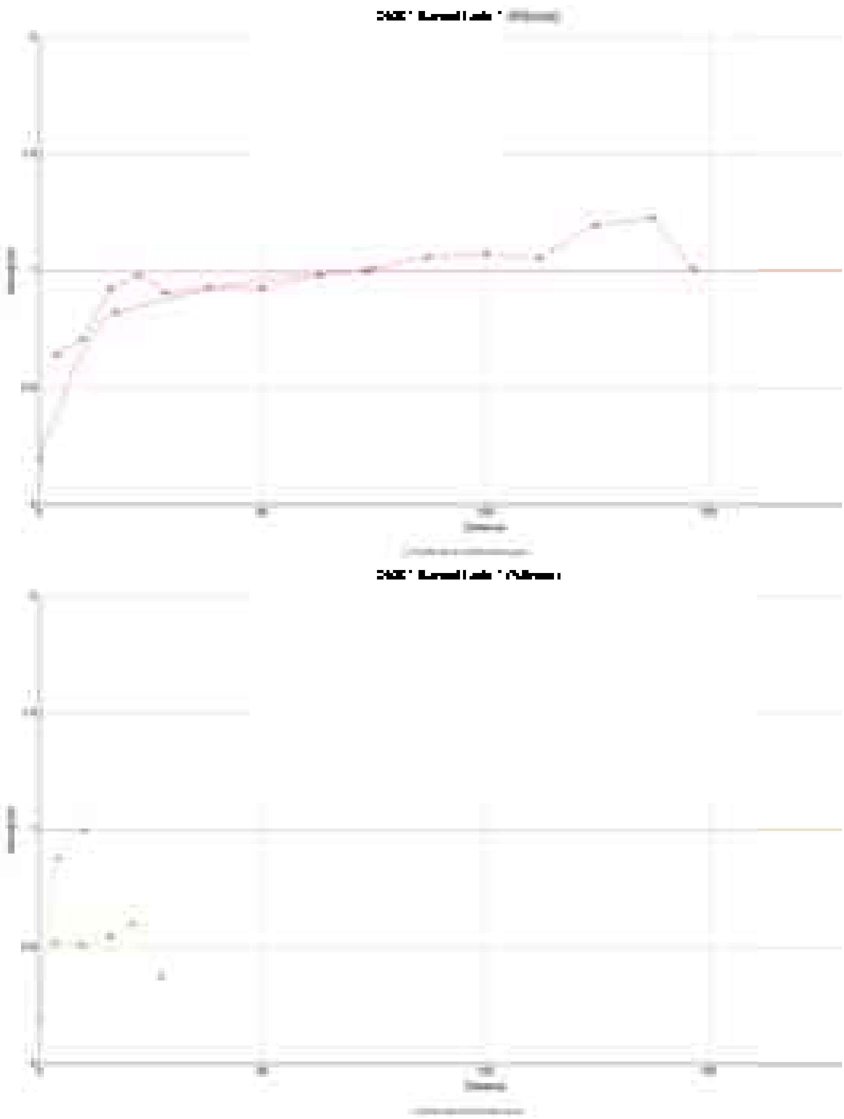




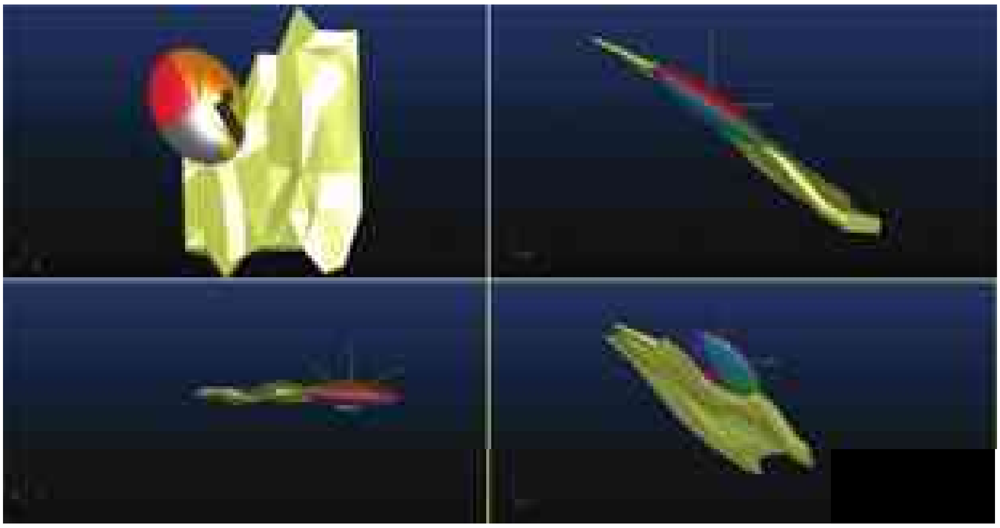
Domain 4001 Sunset Lode 1

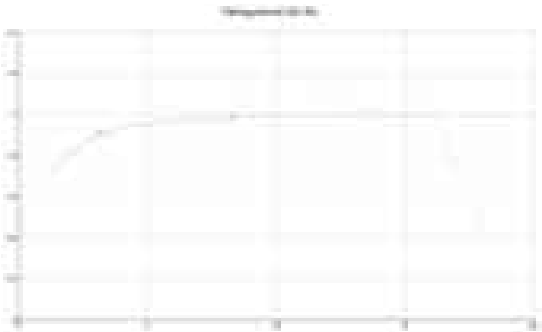


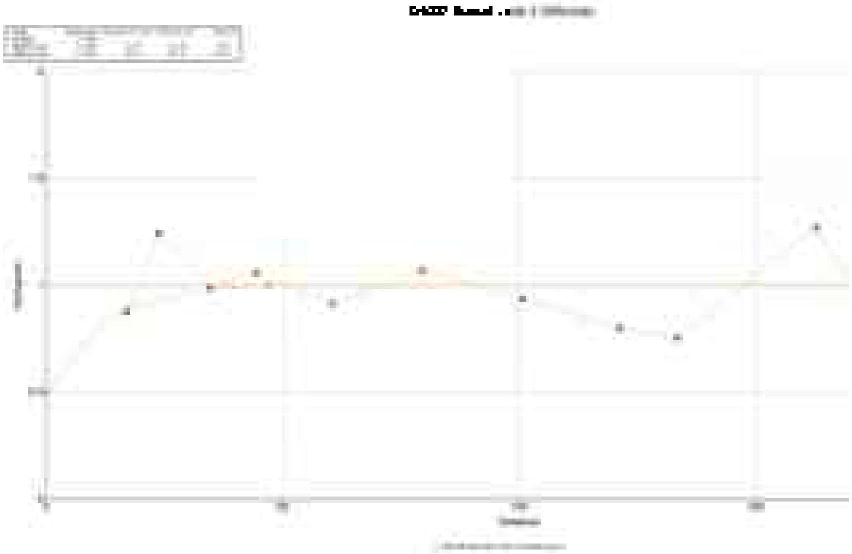


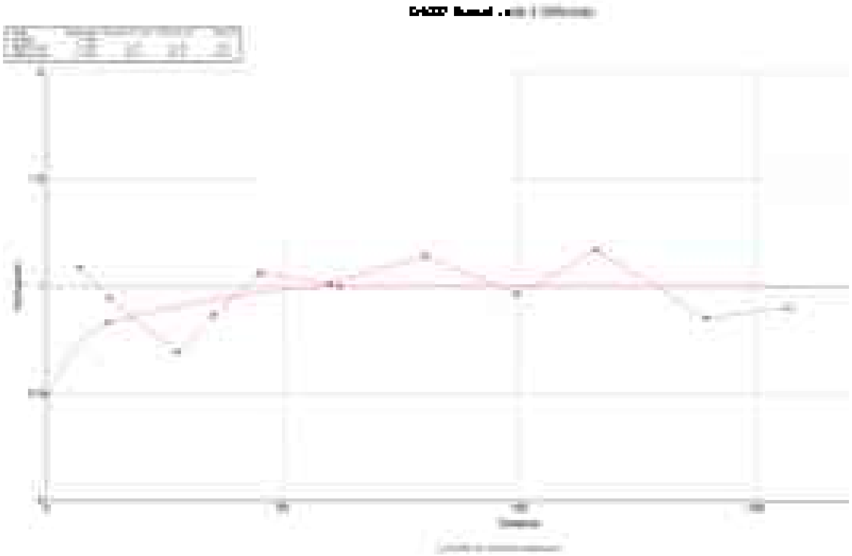


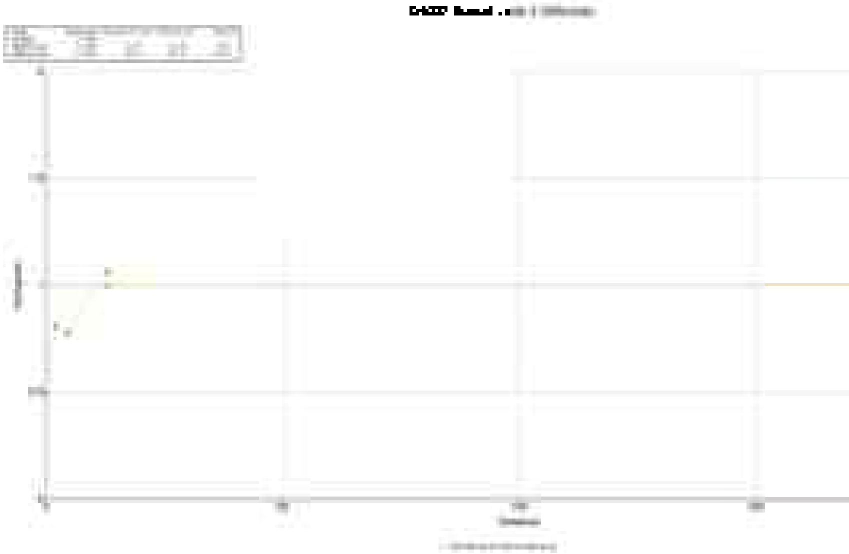
Domain 4002 Sunset Lode 2



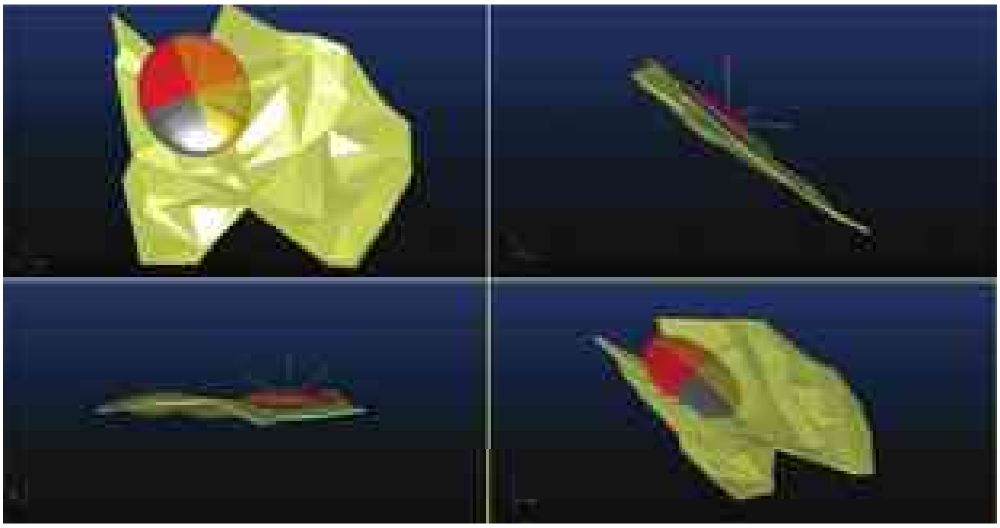


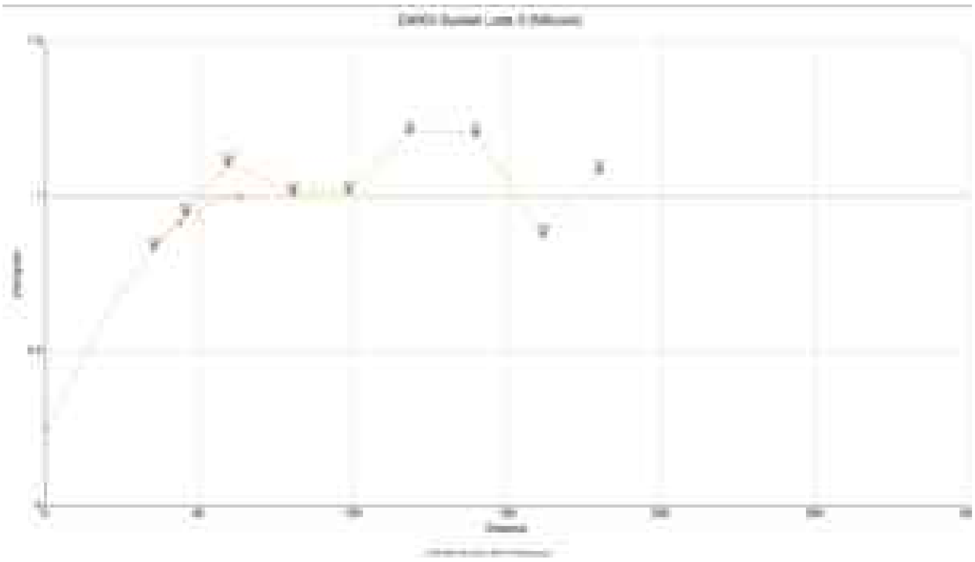


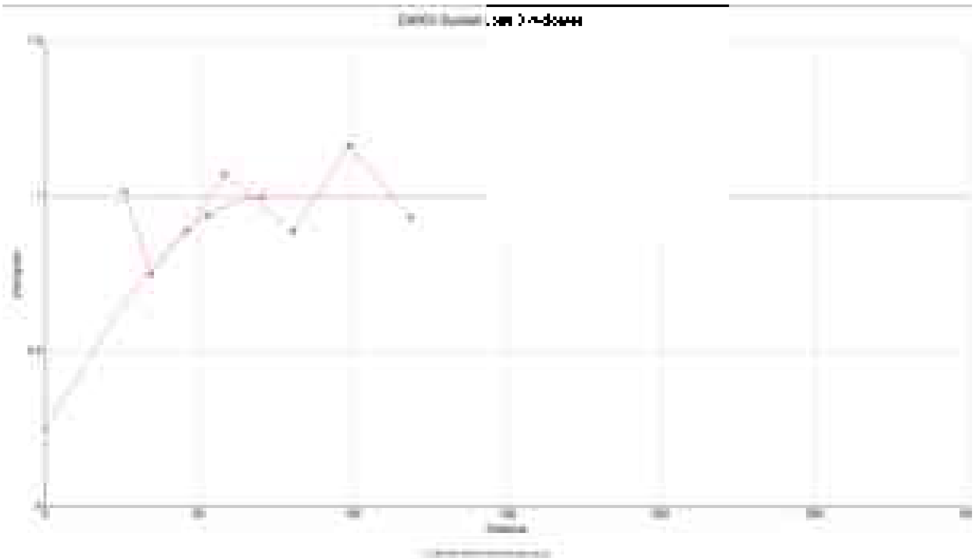


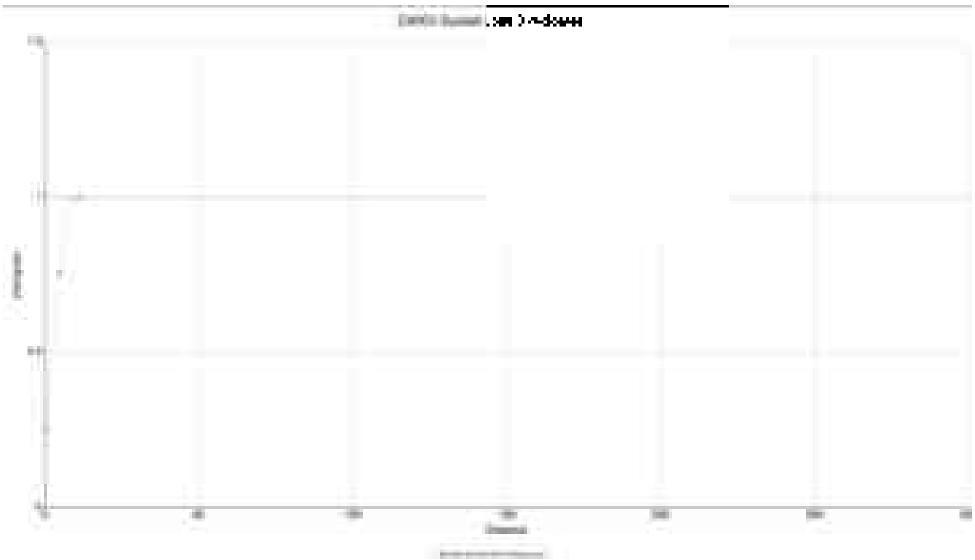


Domain 4003 – Sunset Lode 3

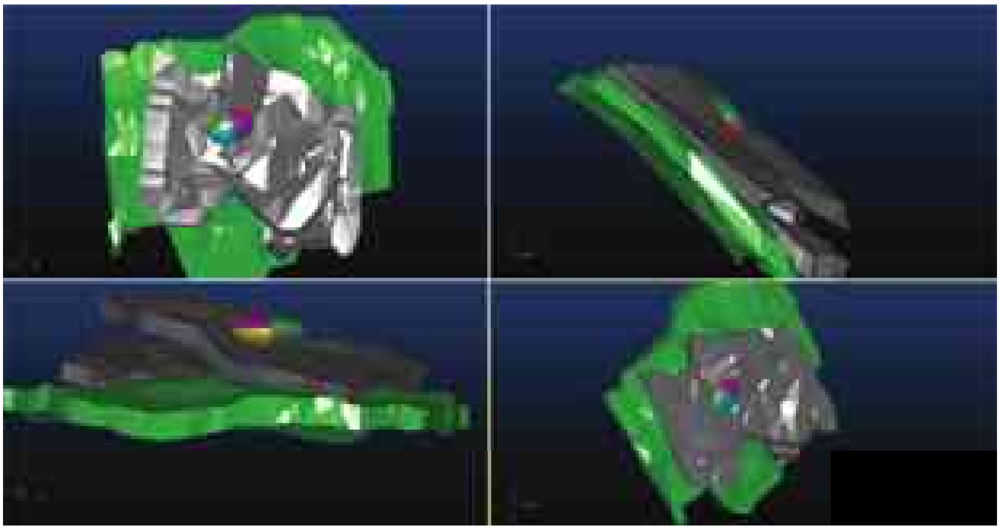


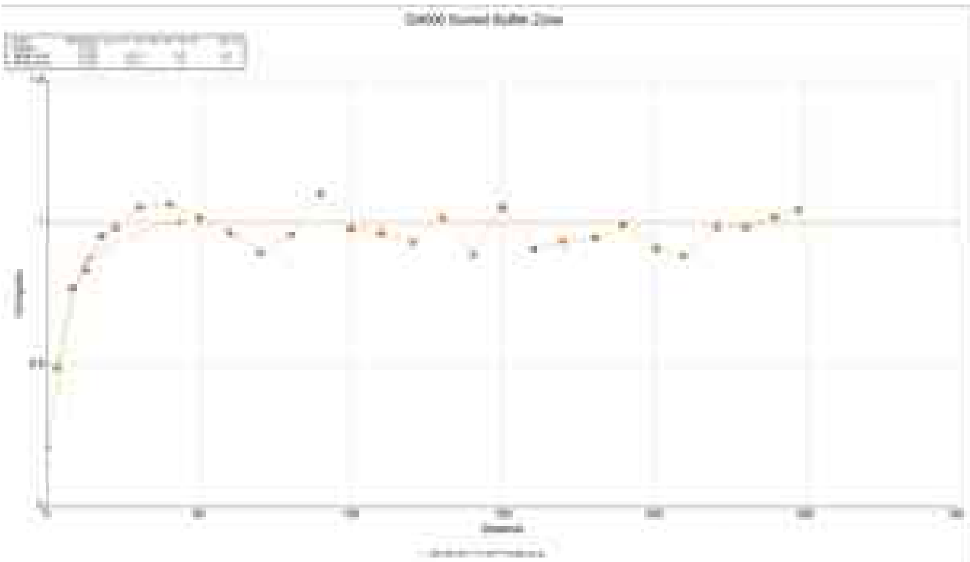


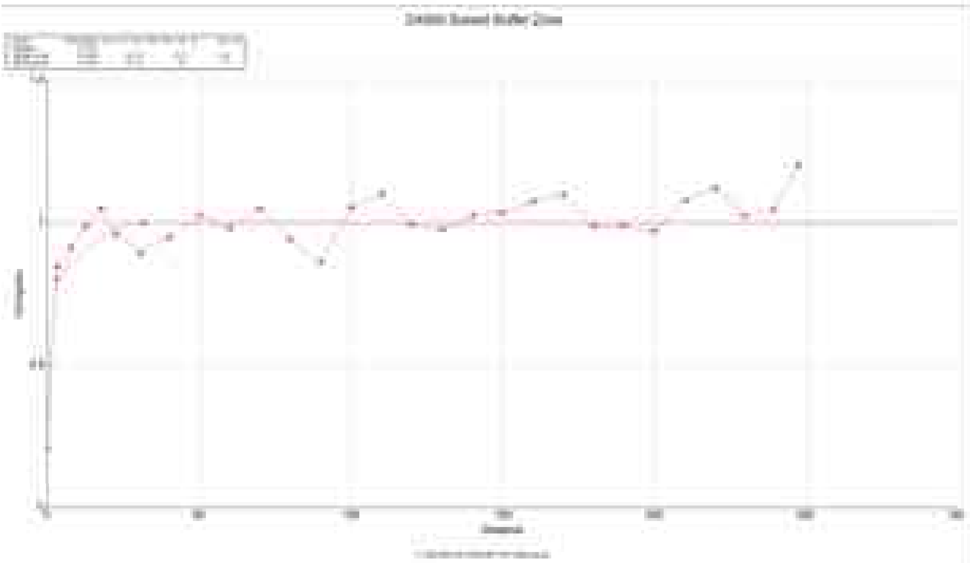


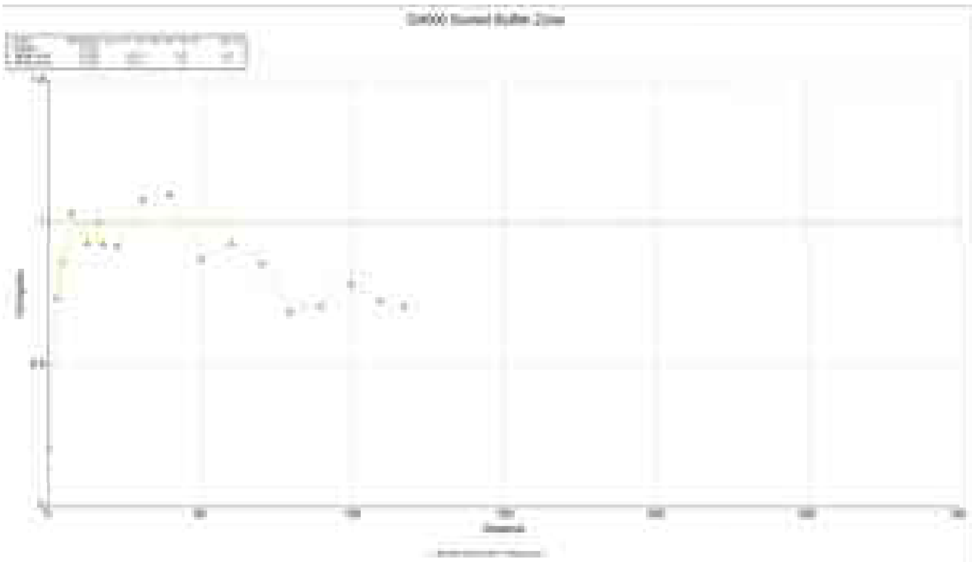


Domain 4000 Sunset Buffer

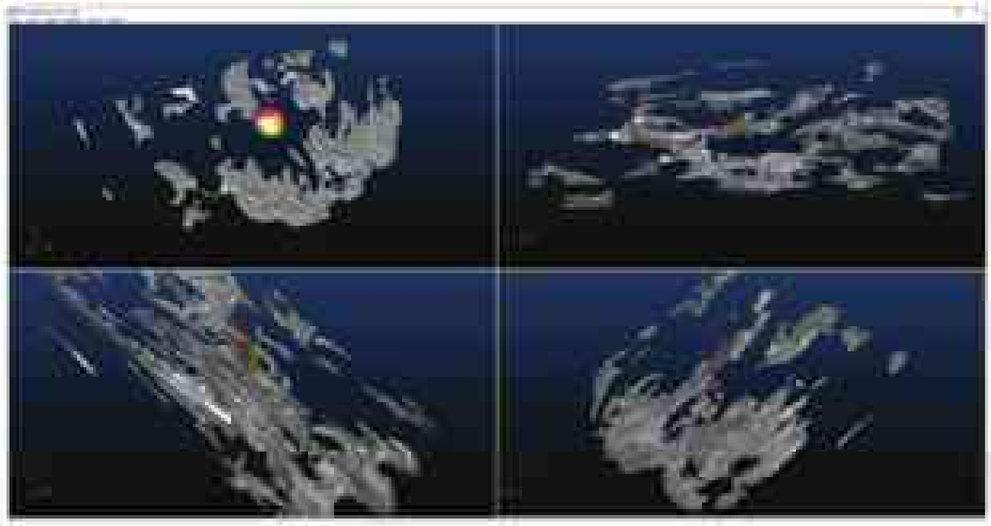


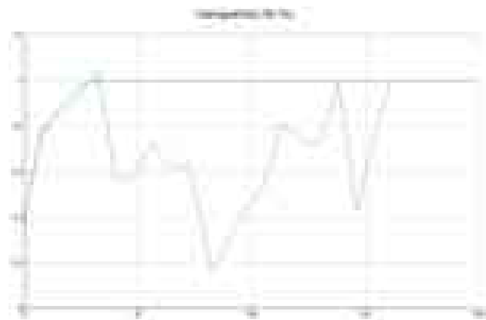


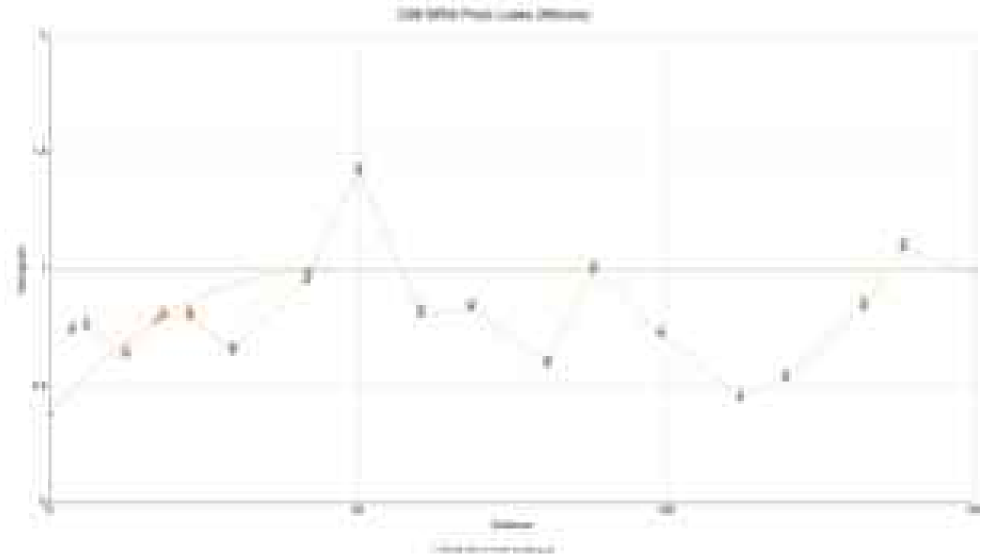


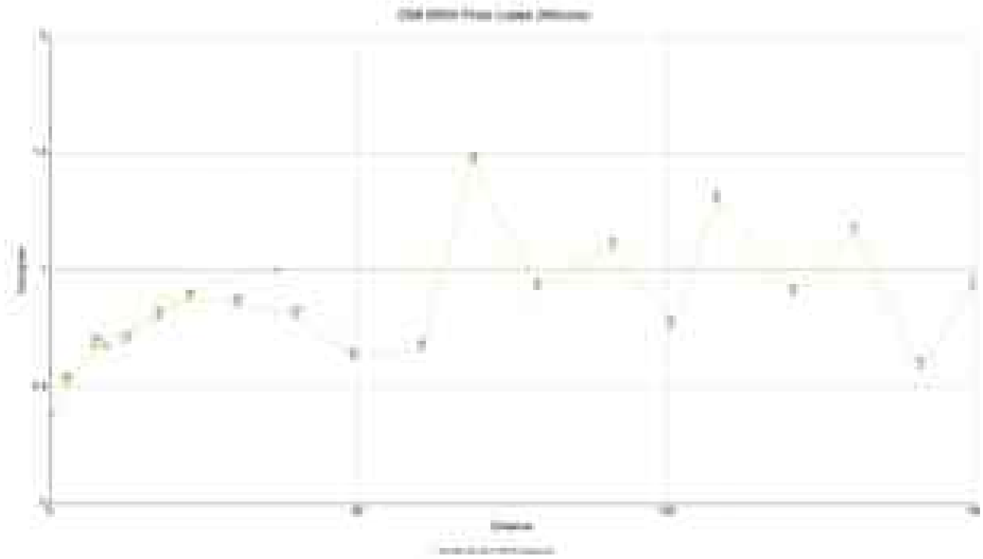


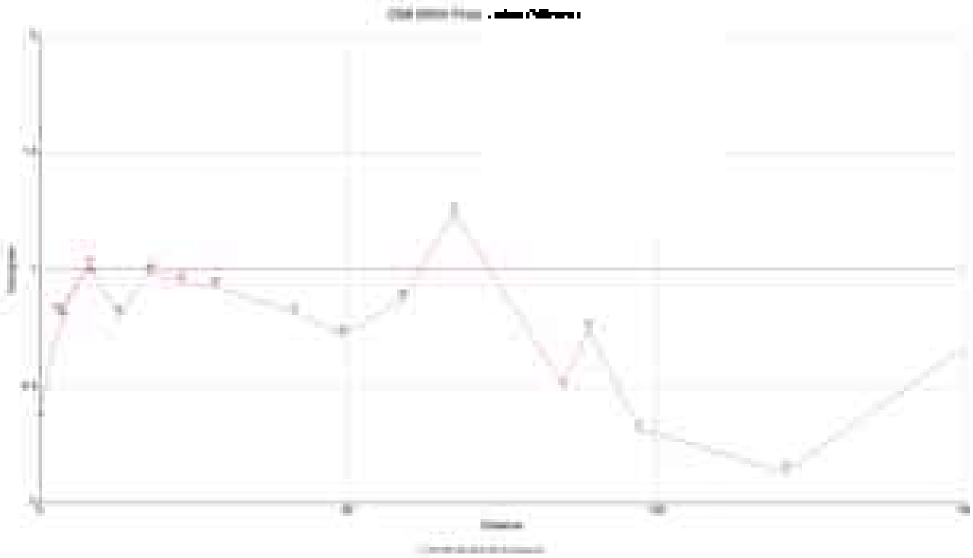
Domain 99 – Proro Lodes







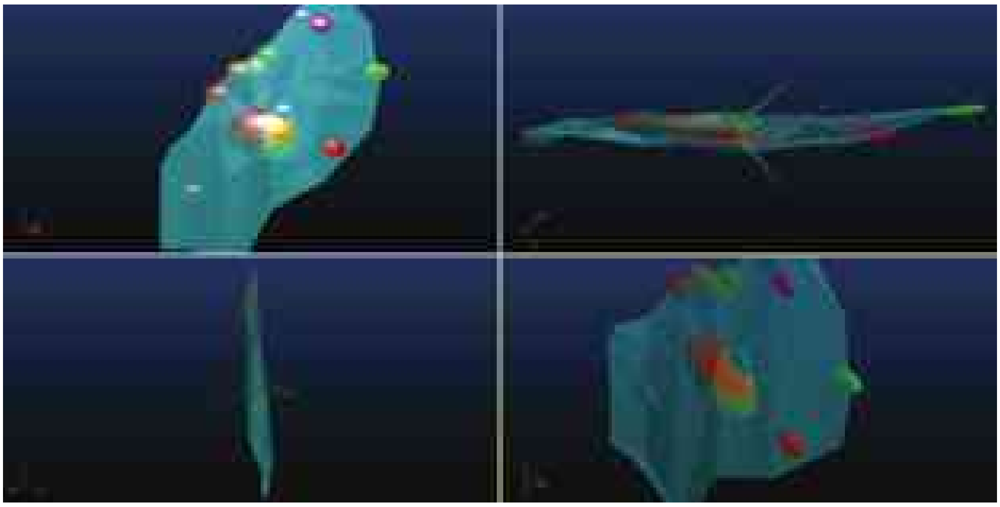




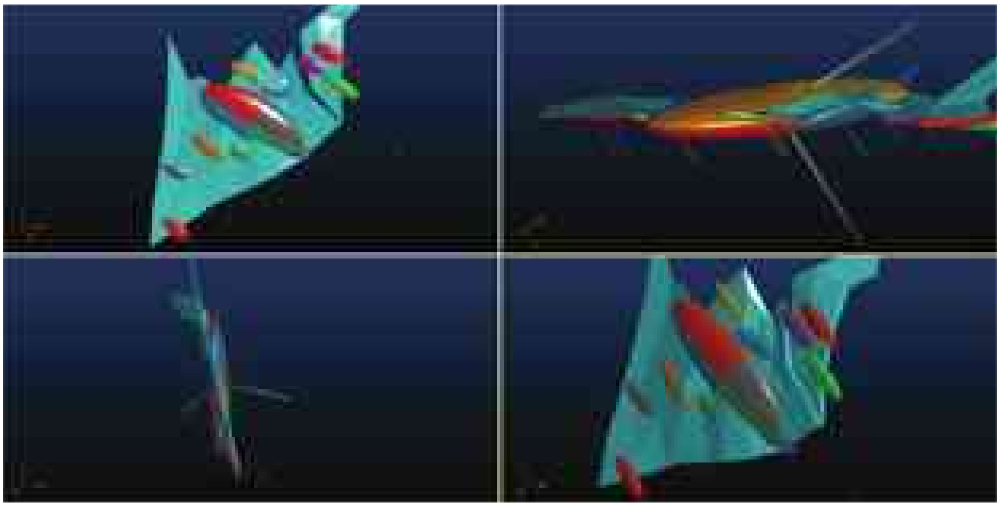
Appendix D Search Ellipses

Global search ellipse is presented as the largest shape, centred on the domain extent. Dynamic anisotropy orientations are presented as smaller ellipses.

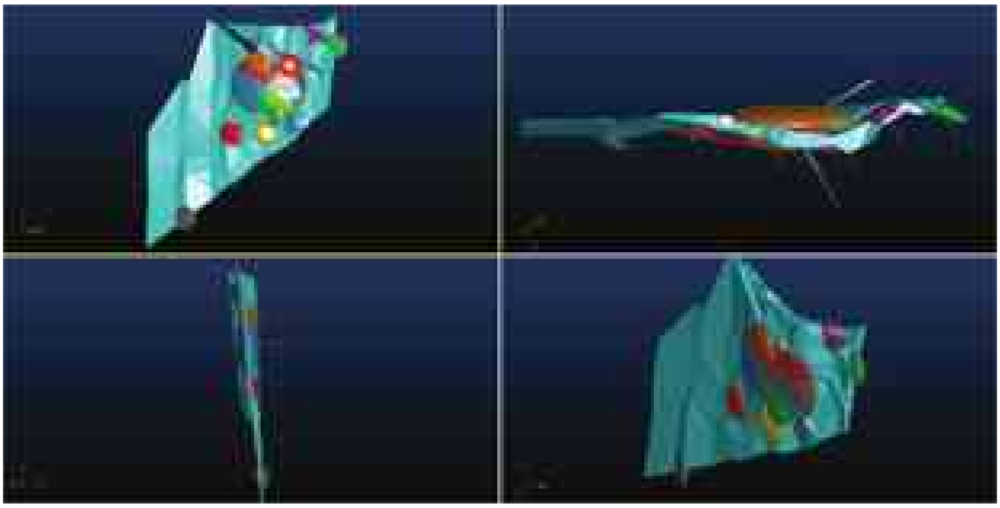
Domain 1001



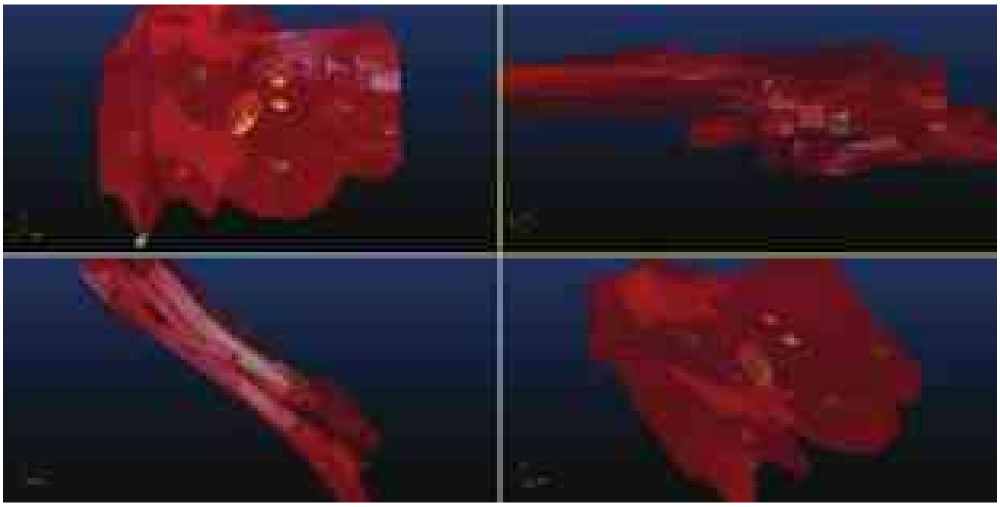
Domain 1002



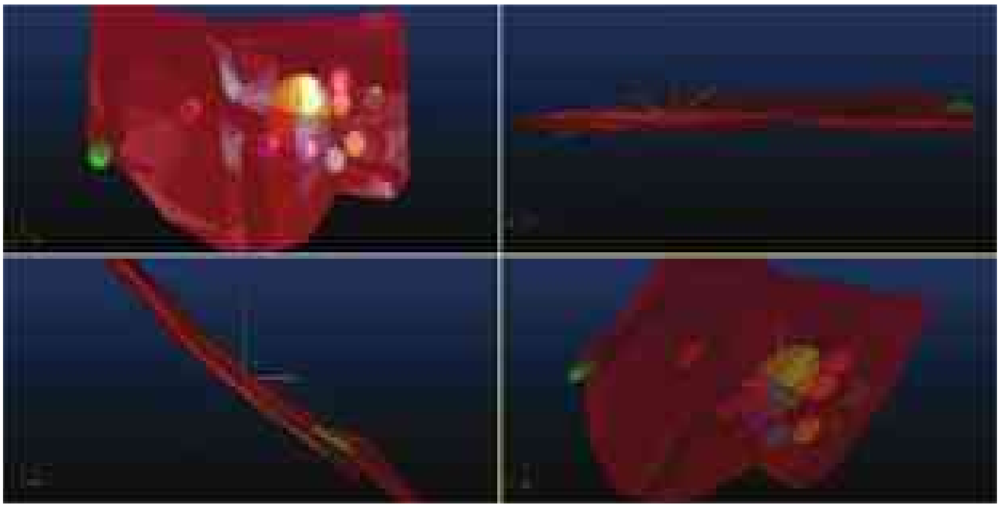
Domain 1003



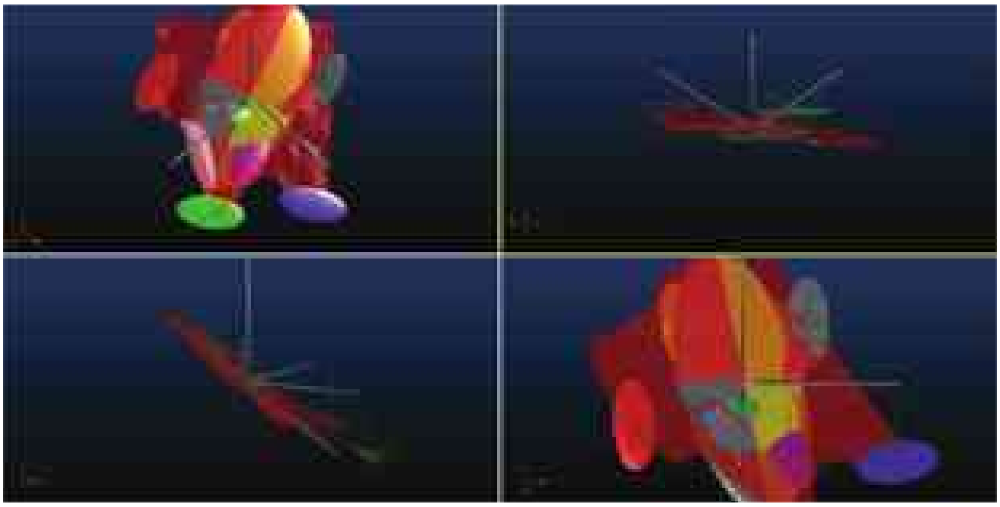
Domain 2000



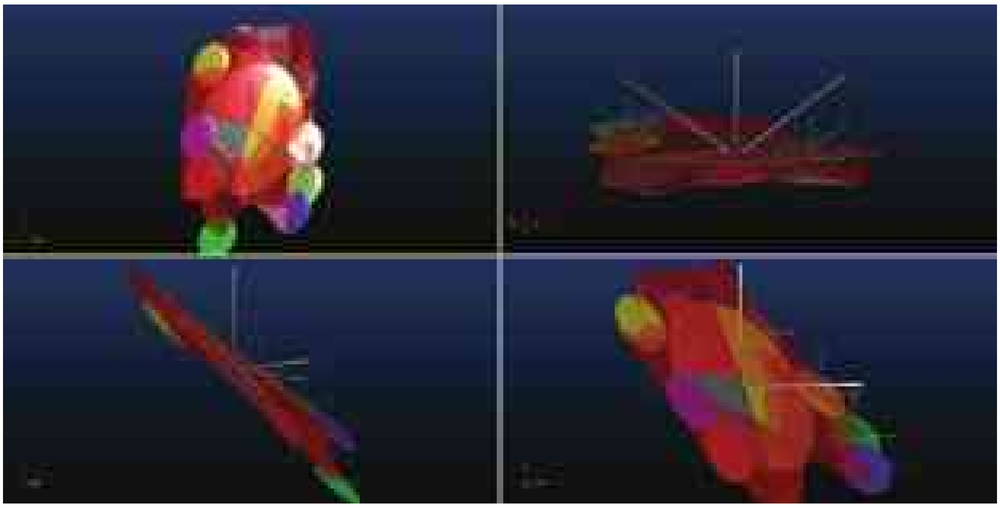
Domain 2001



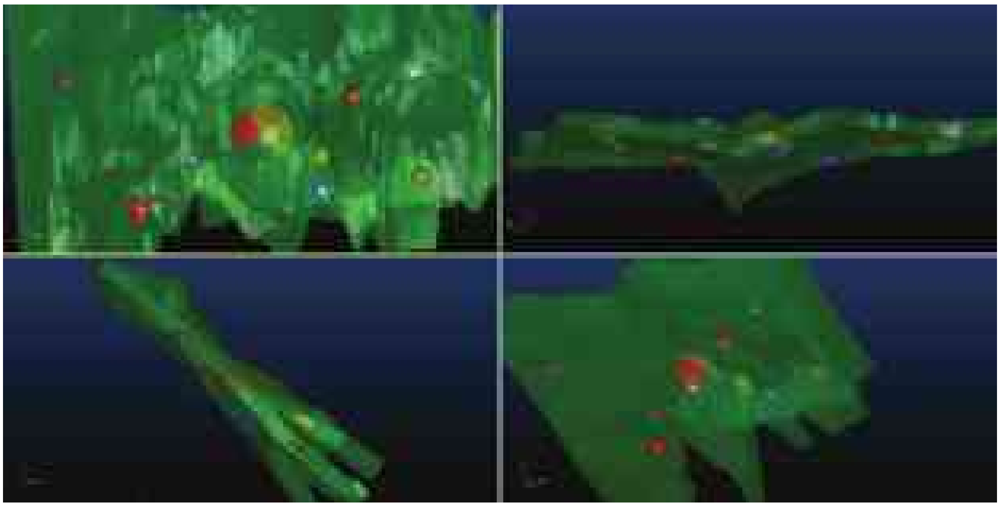
Domain 2002



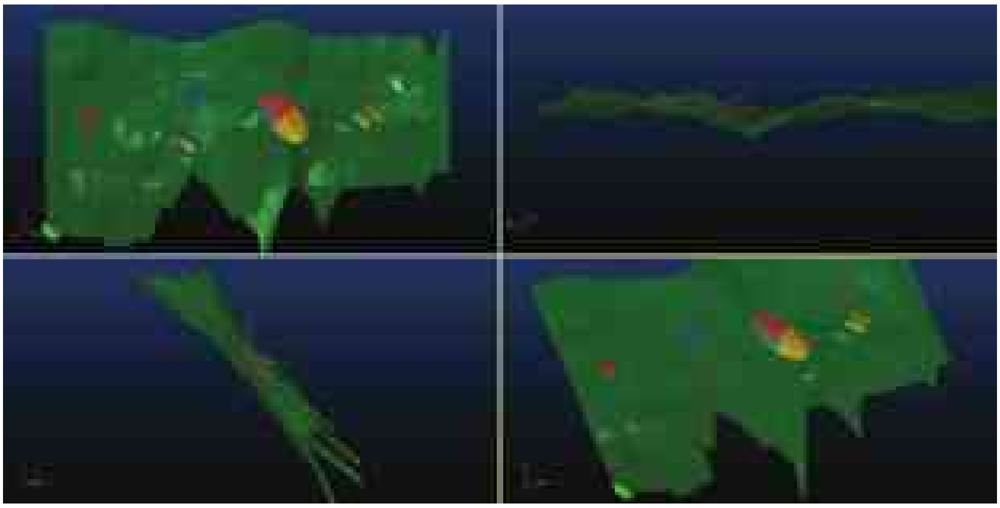
Domain 2003



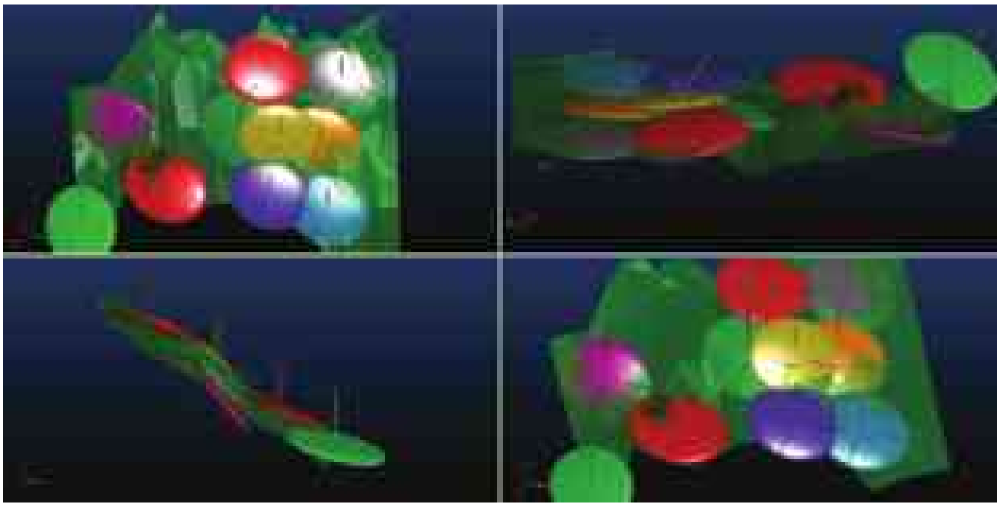
Domain 3000



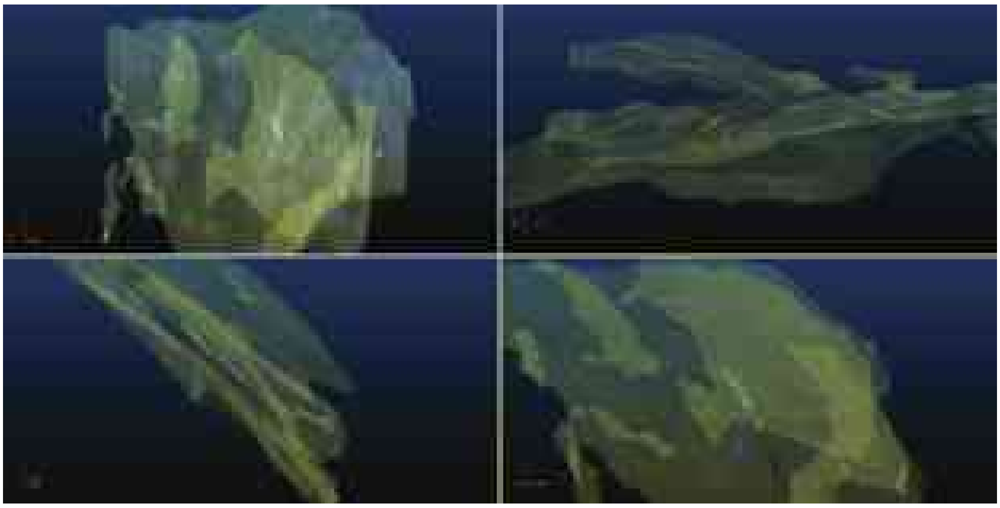
Domain 3001



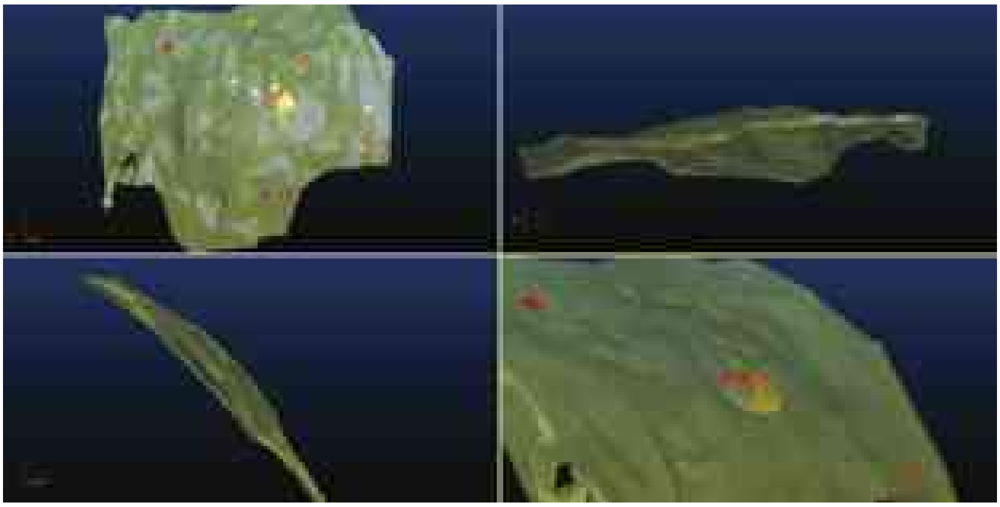
Domain 3002



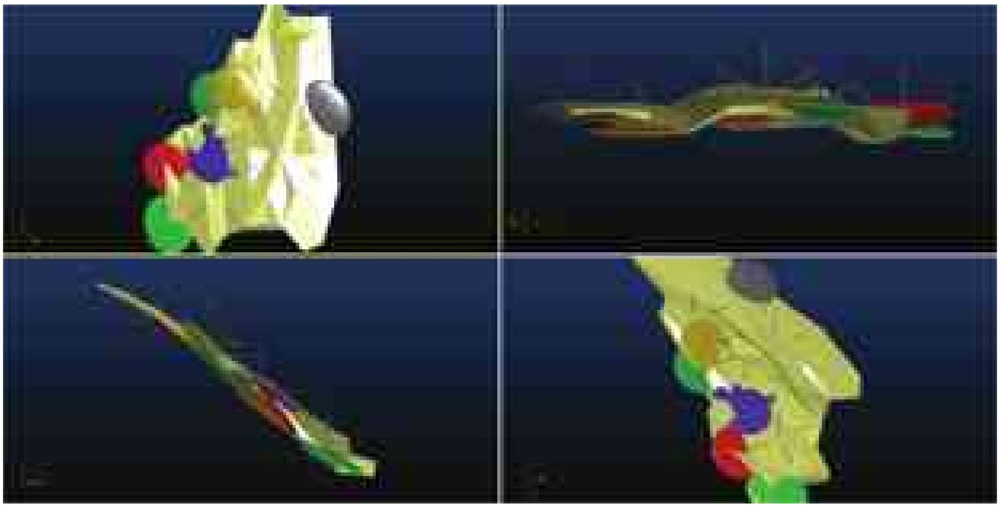
Domain 4000



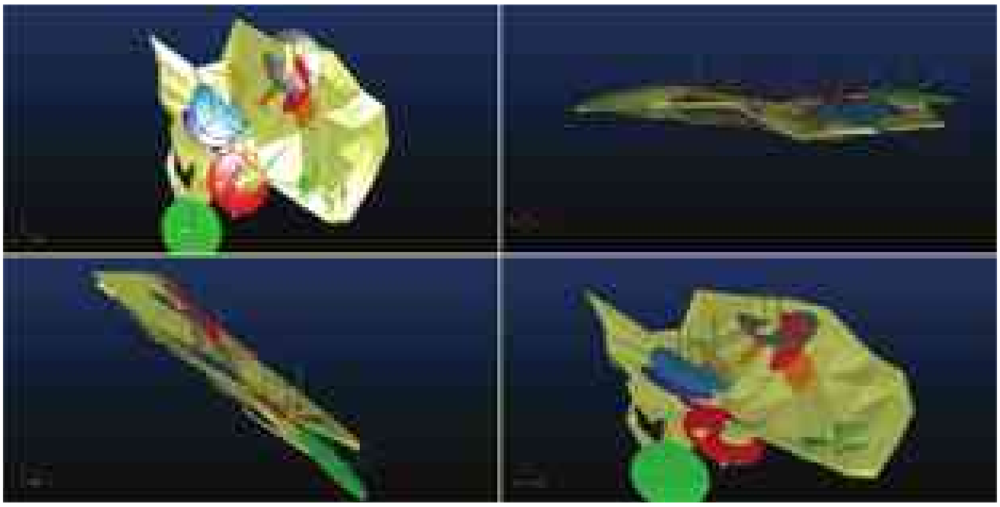
Domain 4001



Domain 4002



Domain 4003



Appendix E Data Listing

Refer provided data folder

Appendix F Model Field Names and Definitions

Field Name	Type	Description
LodeCode	Numeric	Unique lode identifier. 1000 - BRF 2000 - Duke 3000 - Grant 4000 -Sunset 99 - Proto-Lodes 9999 - Outside Domains
DOMAIN	Numeric	1001 - BRF Lower 1002 - BRF Shear 1003 - BRF Upper 1000 - BRF Buffer Zone 2001 - Duke 1 2002 - Duke 2 2003 - Duke 3 2000 - Duke Buffer Zone 3001 - Grant 1 3002 - Grant 2 3000 - Grant Buffer Zone 4001 - Sunset 1 4002 - Sunset 2 4003 - Sunset 3 4004 - Sunset Buffer Zone 99 - Proto-lodes 9999 - Outside Domains
VOID	Binary	1 – void (ug stope or development) 0 – solid rock
ROCK	Binary	1 – solid rock 0 – air or void
OXIDE	Binary	1 – above top of fresh rock (i.e., oxide and partially oxidised) 0 – fresh rock
REPORT	Binary	Used to limit size of reporting only. No meaning for JORC Class
PIT	Binary	1 – within BRW REP200 project pit 0 – not in BRW REP200 project pit
Density	Numeric	In situ bulk density. Set to zero for air and voids.
Au_Cap	Numeric	Estimated gold grade – reportable grade for JORC Code reporting
Au_UnCap	Numeric	Sensitivity only – estimate without capped grades. Not for public reporting
CLASS	Numeric	JORC Code reporting classification 0 - Unclassified 1 – Measured (note no Measured at BRW) 2 – Indicated 3 – Inferred
AMDAD	Binary	1 – Inside AMDAD due diligence Stage 2 pit 0 – not in AMDAD due diligence Stage 2 pit

Field Name	Type	Description
XMORIG	Numeric	Model origin Easting (bottom left)
YMORIG	Numeric	Model origin Northing (bottom left)
ZMORIG	Numeric	Model origin RL (bottom left)
NX	Numeric	Number of parent blocks in Easting (X)
NY	Numeric	Number of parent blocks in Northing (Y)
NZ	Numeric	Number of parent blocks in RL (Z)
XINC	Numeric	Sub-block Easting dimension
YINC	Numeric	Sub-block Northing dimension
ZINC	Numeric	Sub-block RL dimension
XC	Numeric	Block centre Easting
YC	Numeric	Block centre Northing
ZC	Numeric	Block centre RL
IJK	Numeric	Datamine Studio RM unique parent block index

Block model details

Origin: 12,700mE, 12,700mN, -400m RL

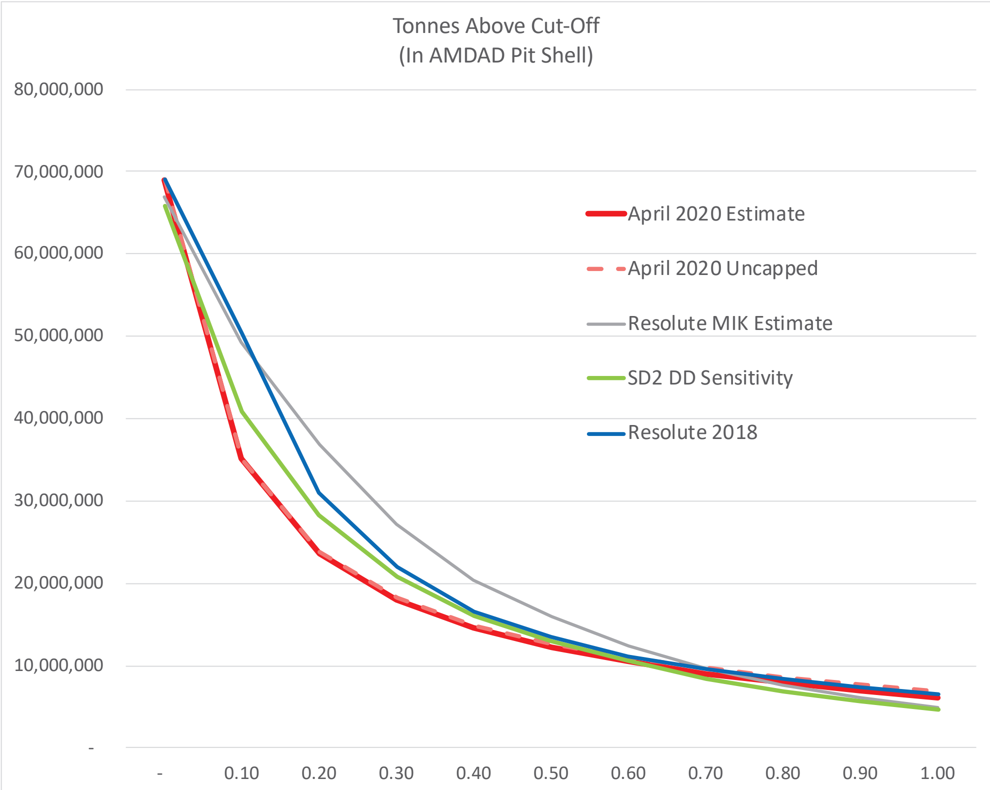
Max Limit: 13,620mE, 13,620mN, 340mRL

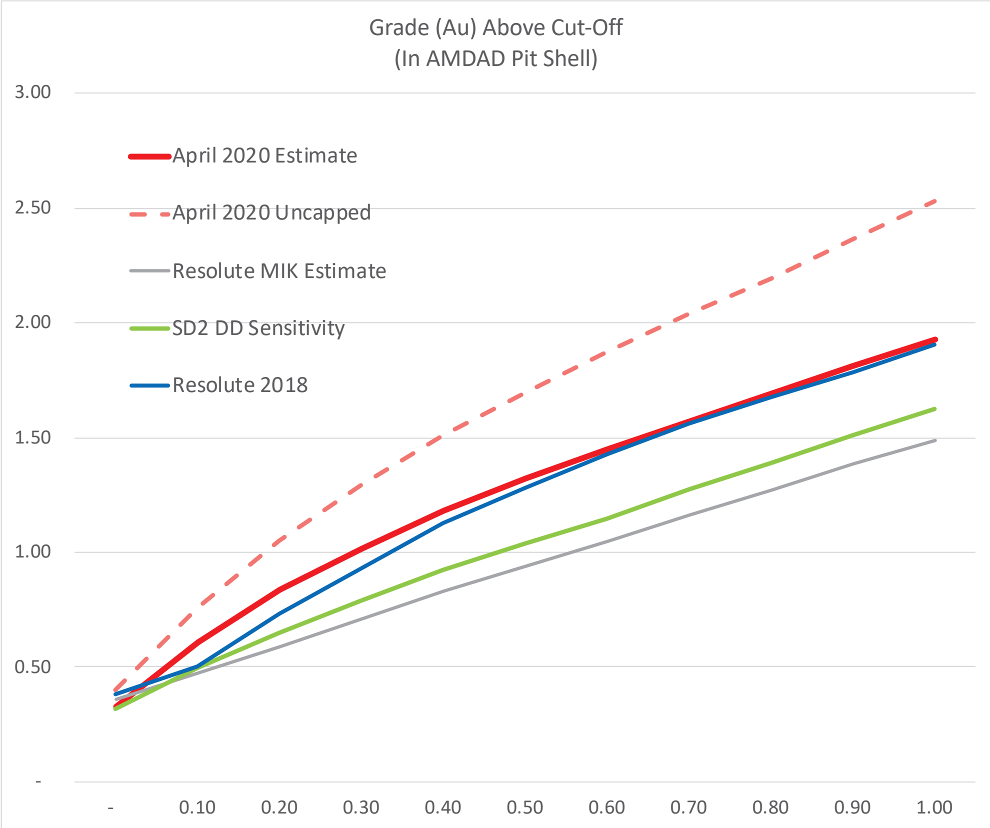
Parent Blocks: 5mE, 10mN, 5mRL

Minimum Sub-Blocks: 1.25mE, 2.5mN, 1.25mRL

Appendix G Grade-Tonnage Curves

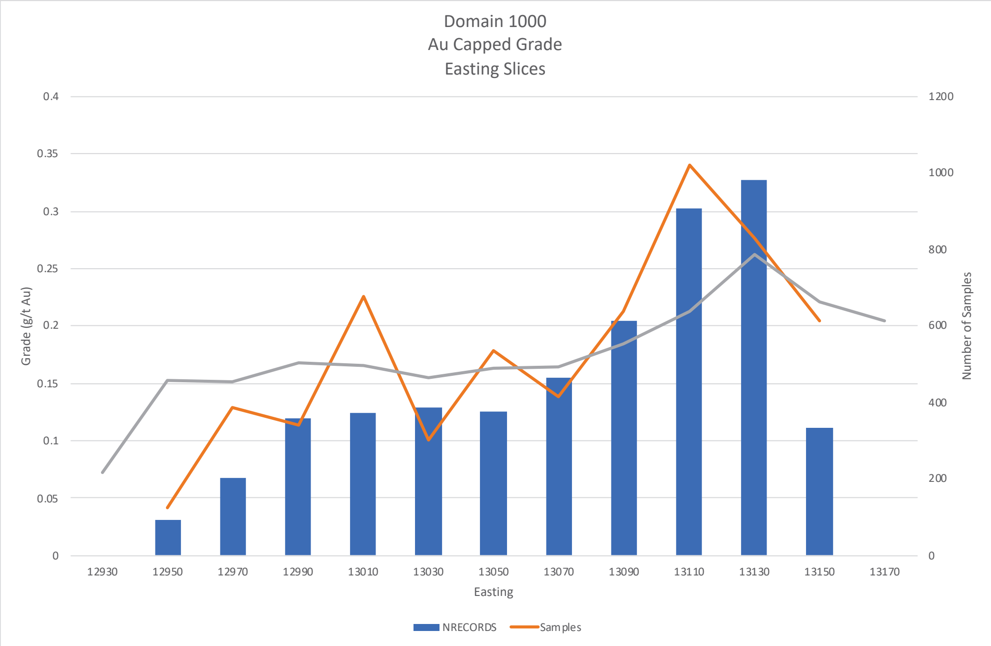
	April 2020 Estimate			April 2020 Uncapped			Resolute MIK Estimate			SD2 DD Sensitivity			Resolute 2018		
Cut Off	Tonnes	Au g/t	Oz (Au)	Tonnes	Au g/t	Oz (Au)	Tonnes	Au g/t	Oz (Au)	Tonnes	Au g/t	Oz (Au)	Tonnes	Au g/t	Oz (Au)
-	69,047,840	0.33	730,240	69,047,840	0.40	894,644	66,985,152	0.36	781,070	65,837,891	0.32	675,637	69,042,736	0.38	851,674
0.10	35,122,721	0.61	688,251	35,122,113	0.76	856,674	49,244,116	0.48	754,803	40,801,709	0.49	648,622	50,394,079	0.51	818,305
0.20	23,657,241	0.83	635,049	23,874,951	1.05	804,710	36,868,452	0.59	696,380	28,317,533	0.65	589,034	31,010,933	0.73	728,458
0.30	18,084,911	1.02	591,055	18,289,380	1.29	760,755	27,202,006	0.71	619,610	20,791,286	0.79	529,716	21,958,550	0.93	657,358
0.40	**14,542,684**	**1.18**	**551,605**	**14,910,766**	**1.51**	**723,164**	**20,401,951**	**0.83**	**543,705**	**16,091,182**	**0.92**	**477,508**	**16,512,113**	**1.12**	**596,920**
0.50	12,165,853	1.32	517,397	12,664,013	1.70	690,803	15,884,068	0.94	479,046	13,007,875	1.04	433,054	13,425,604	1.28	552,535
0.60	10,442,358	1.45	487,033	11,004,191	1.87	661,568	12,453,683	1.05	418,914	10,599,838	1.15	390,515	11,192,028	1.43	513,134
0.70	9,062,392	1.57	458,253	9,690,314	2.04	634,182	9,642,829	1.16	360,248	8,448,674	1.27	345,762	9,565,265	1.56	479,235
0.80	7,926,907	1.69	430,877	8,644,840	2.19	608,980	7,653,036	1.27	312,524	6,950,965	1.39	309,665	8,382,936	1.67	450,723
0.90	6,938,064	1.81	403,910	7,664,432	2.36	582,223	6,044,550	1.38	268,804	5,665,346	1.51	274,751	7,382,716	1.78	423,411
1.00	6,095,404	1.93	378,191	6,851,946	2.53	557,422	4,898,473	1.48	233,762	4,677,220	1.63	244,461	6,472,065	1.90	395,653

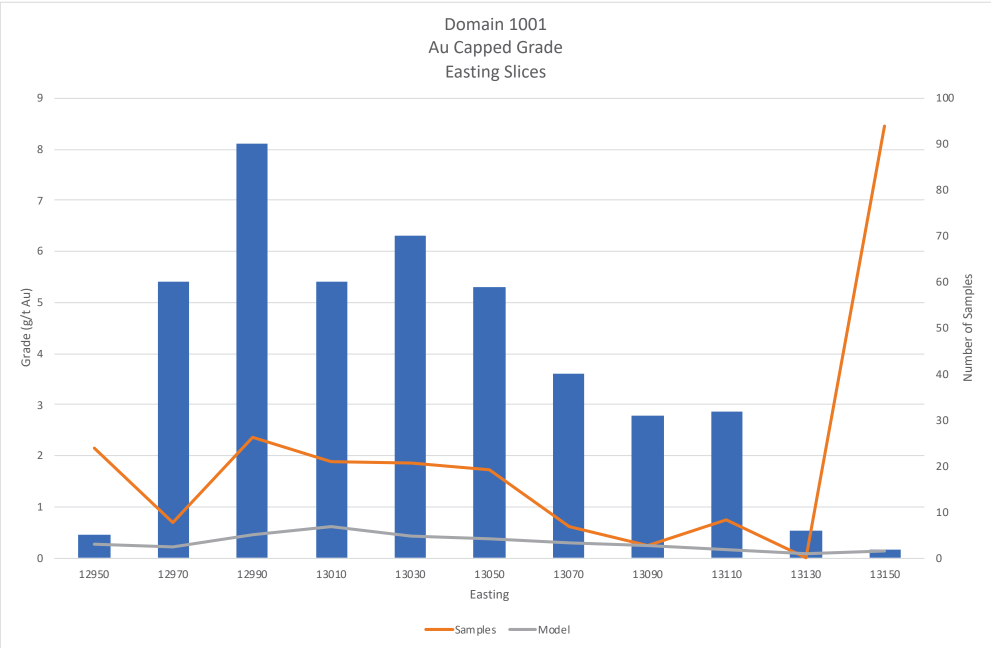


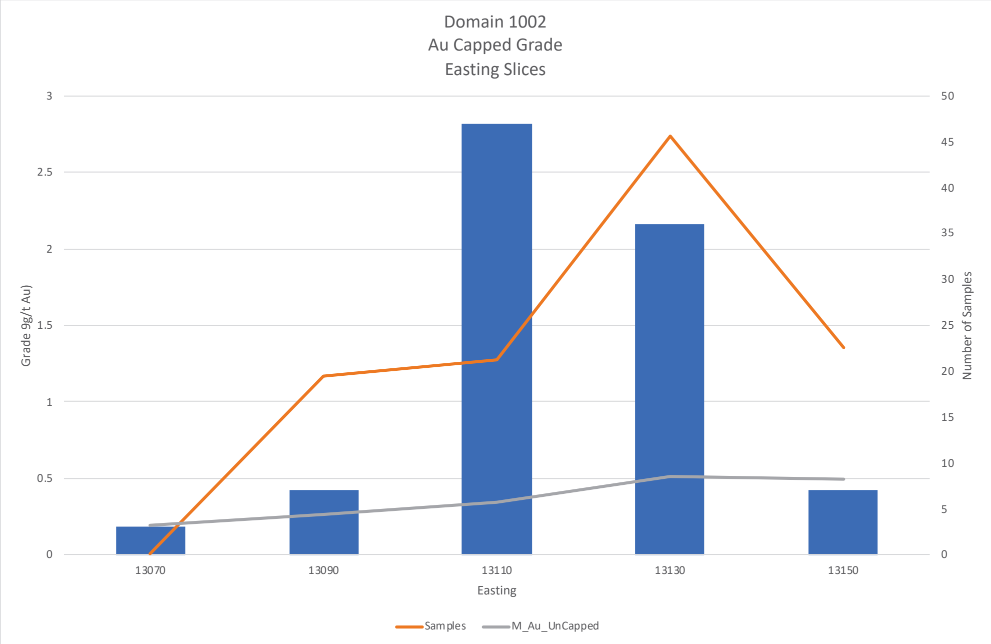

Grade (Au) Above Cut-Off
(In AMDAD Pit Shell)
3.00
2.50
2.00
1.50
1.00
0.50
-
April 2020 Estimate
April 2020 Uncapped
Resolute MIK Estimate
SD2 DD Sensitivity
Resolute 2018
-
0.10
0.20
0.30
0.40
0.50
0.60
0.70
0.80
0.90
1.00

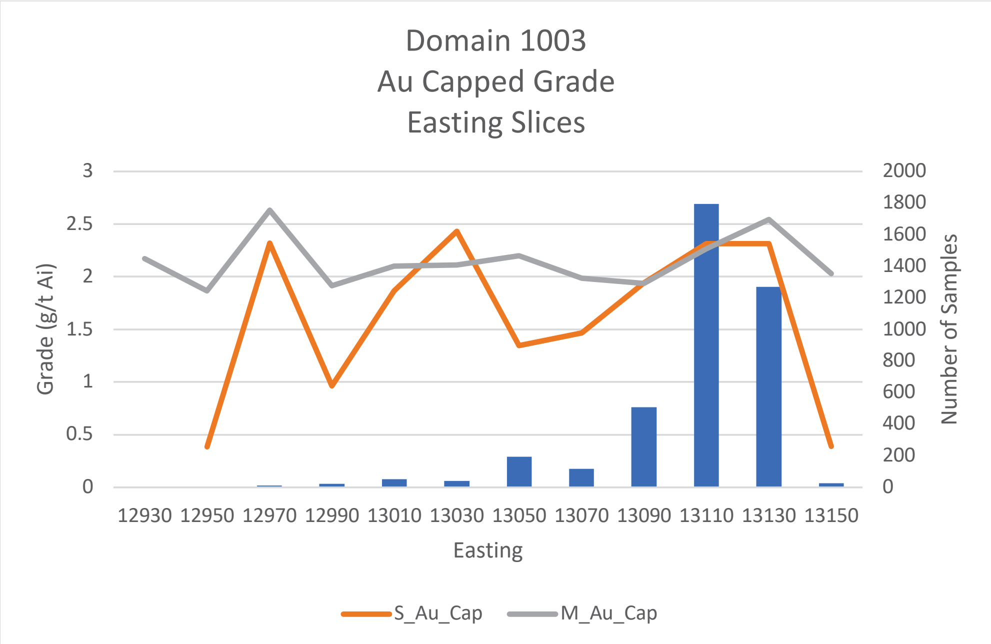

Ounces (Au) Above Cut-Off
(In AMDAD Pit Shell)
1,000,000
900,000
800,000
700,000
600,000
500,000
400,000
300,000
200,000
100,000
-
-
0.10
0.20
0.30
0.40
0.50
0.60
0.70
0.80
0.90
1.00
April 2020 Estimate
April 2020 Uncapped
Resolute MIK Estimate
SD2 DD Sensitivity
Resolute 2018

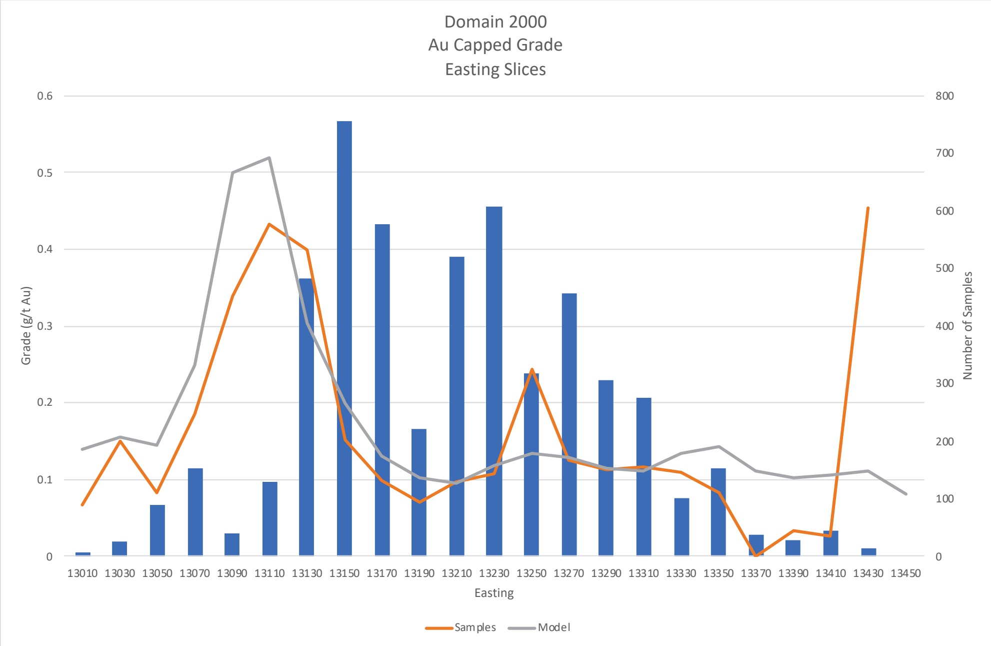
Appendix H Swath Plots (Eastings)

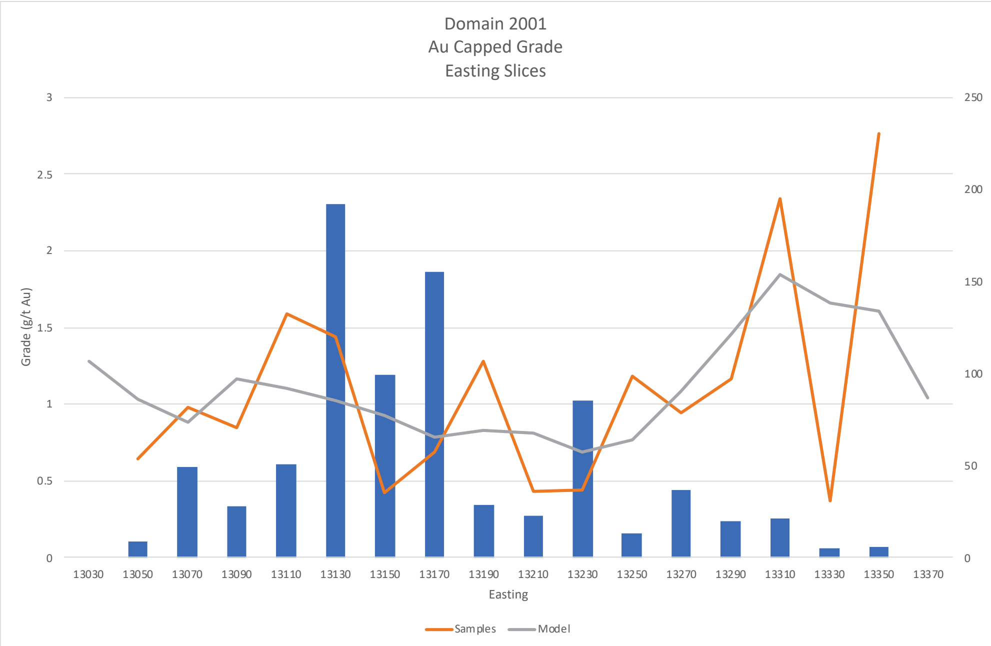


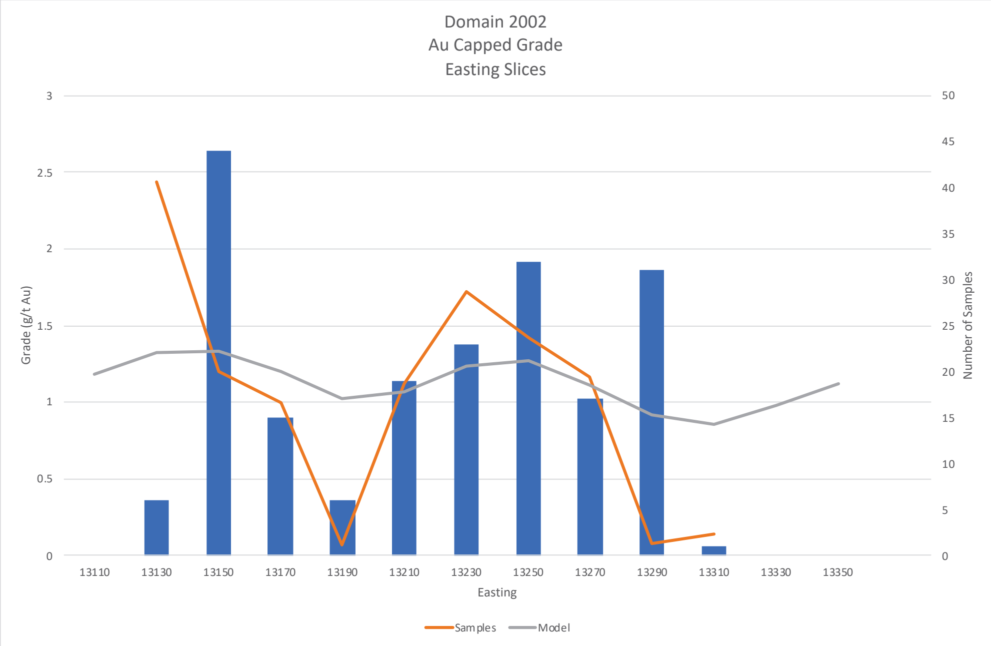


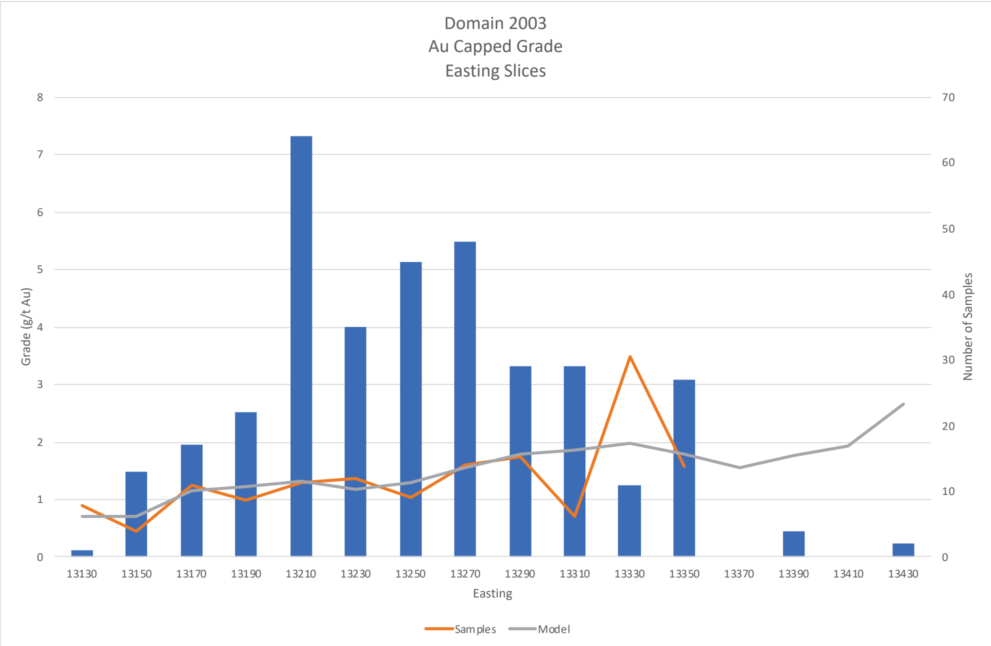

Domain 1002
Au Capped Grade
Easting Slices
Grade 9g/t Au)
Number of Samples
3
2.5
2
1.5
1
0.5
0
50
45
40
35
30
25
20
15
10
5
0
13070
13090
13110
13130
13150
Easting
Samples
M_Au_UnCapped

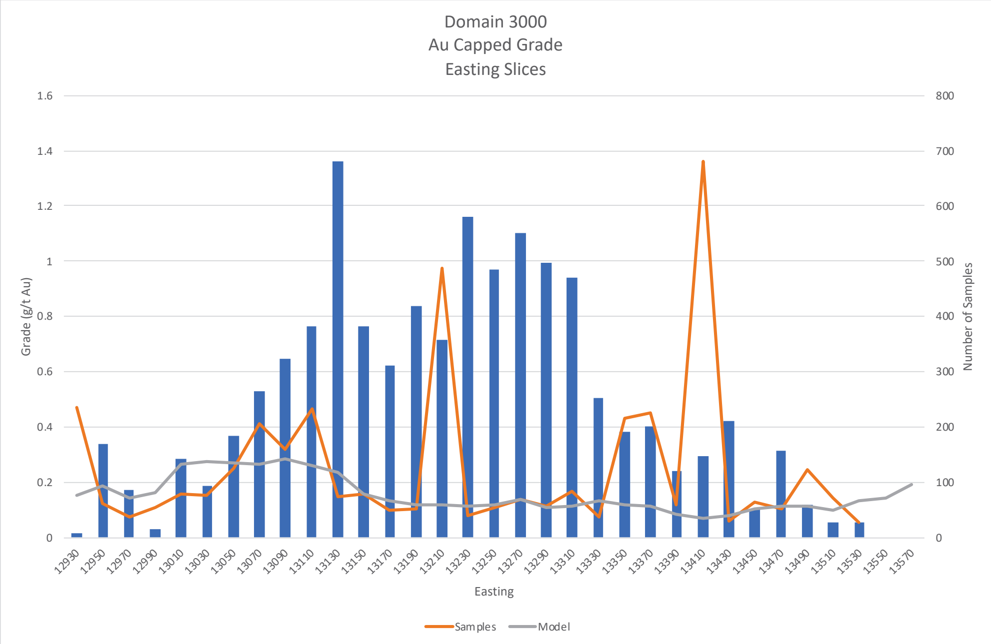

Domain 1003
Au Capped Grade
Easting Slices
Grade (g/t Ai)
Number of Samples
3
2.5
2
1.5
1
0.5
0
2000
1800
1600
1400
1200
1000
800
600
400
200
0
12930 12950 12970 12990 13010 13030 13050 13070 13090 13110 13130 13150
Easting
S_Au_Cap
M_Au_Cap


Domain 2000
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model


Domain 2001
Au Capped Grade
Easting Slices
Grade (g/t Au)
Easting
Samples
Model


Domain 2002
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model


Domain 2003
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model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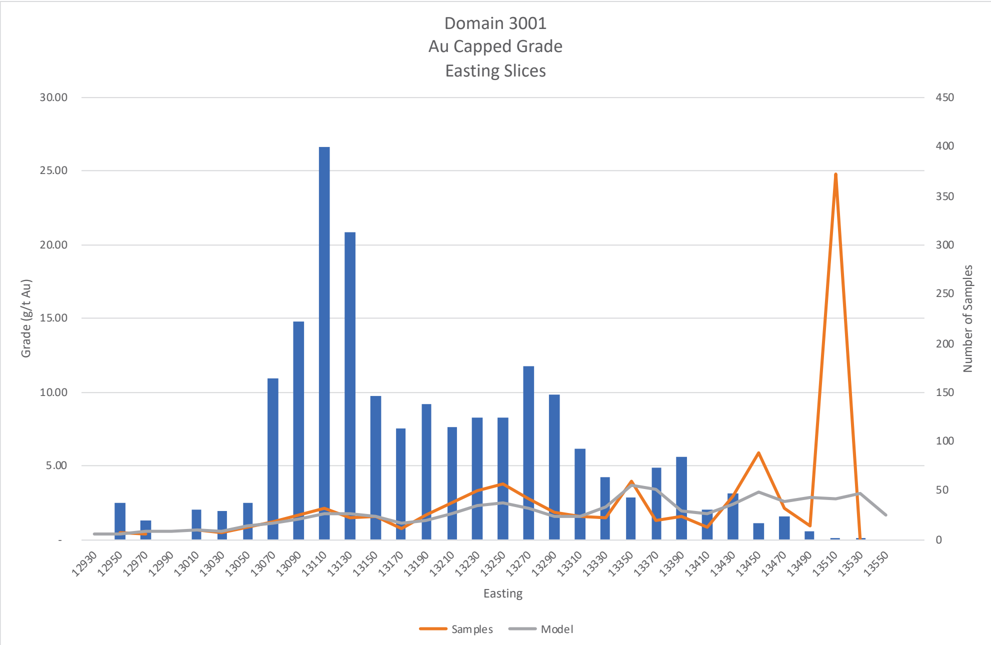
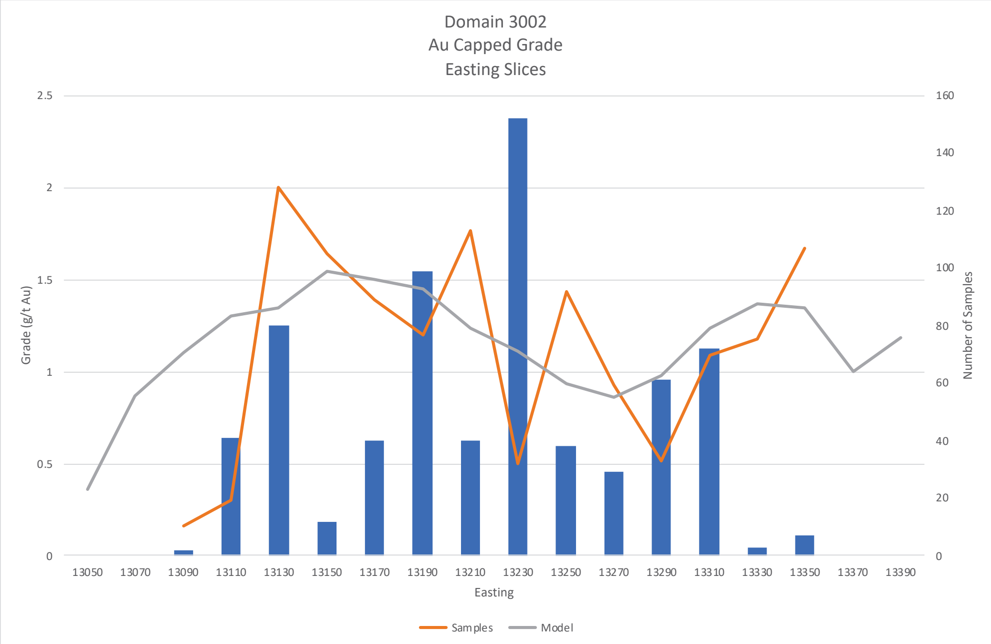
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model


Domain 3001
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model


Domain 3002
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
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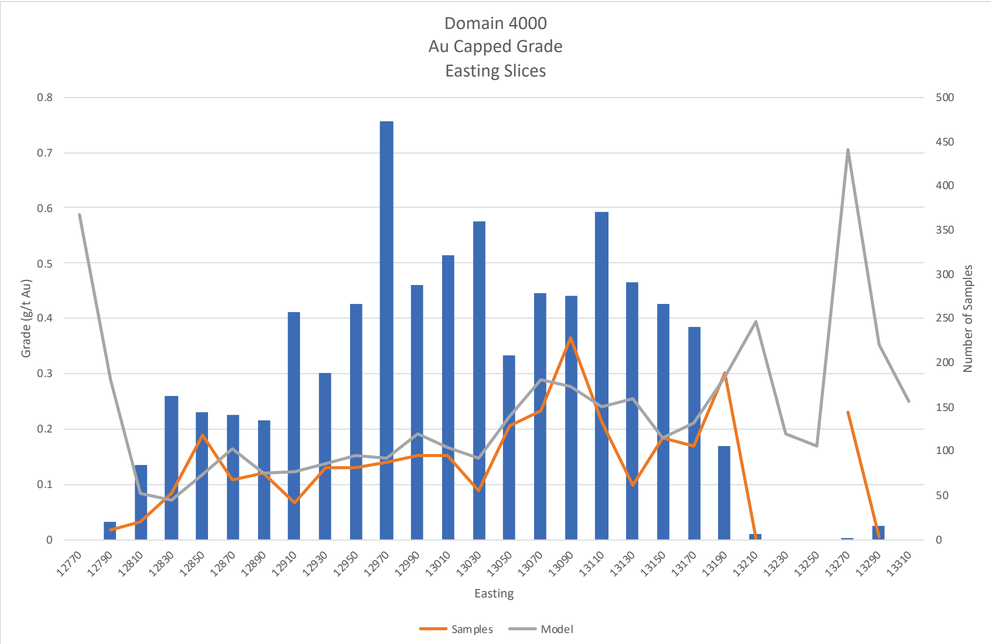
1
1.5
2
2.5
0
20
40
60
80
100
120
140
160
13050
13070
13090
13110
13130
13150
13170
13190
13210
13230
13250
13270
13290
13310
13330
13350
13370
13390
Samples
Model


Domain 4000
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0
50
100
150
200
250
300
350
400
450
500
12770
12790
12810
12830
12850
12870
12890
12910
12930
12950
12970
12990
13010
13030
13050
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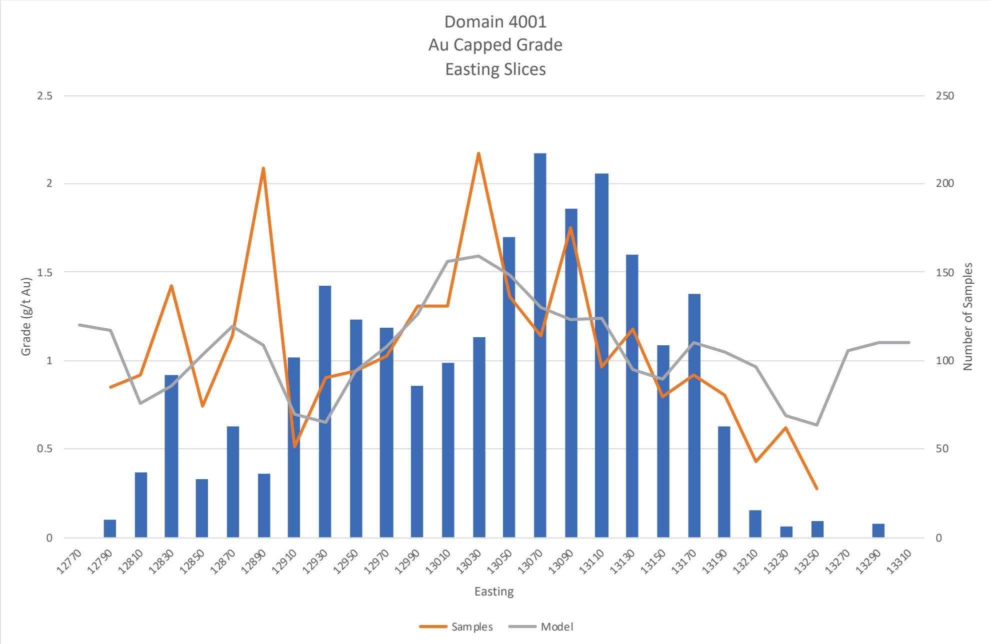
13090
13110
13130
13150
13170
13190
13210
13230
13250
13270
13290
13310
Samples
Model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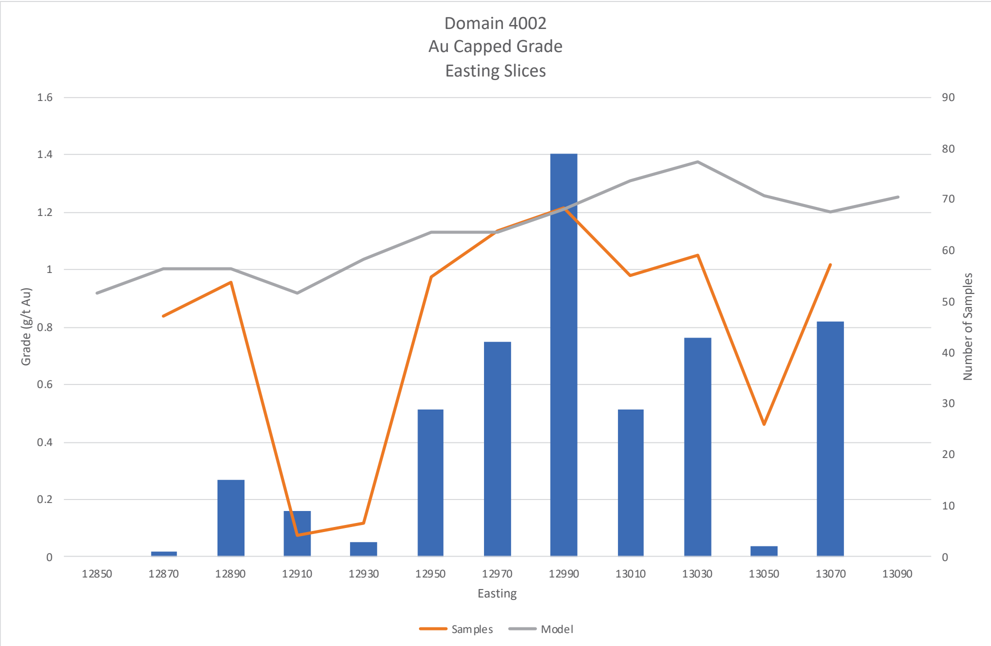
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model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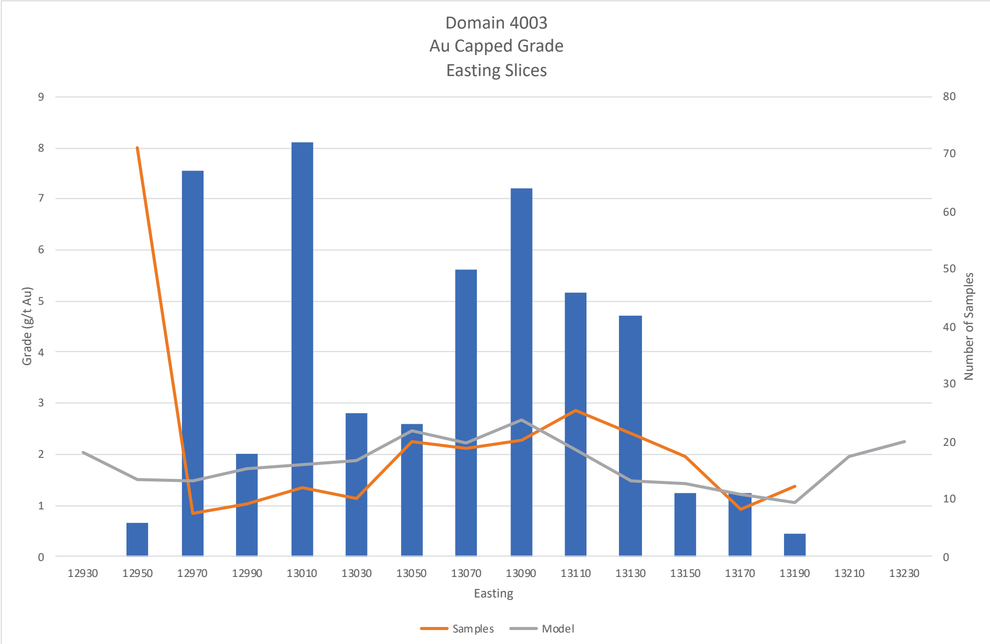
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model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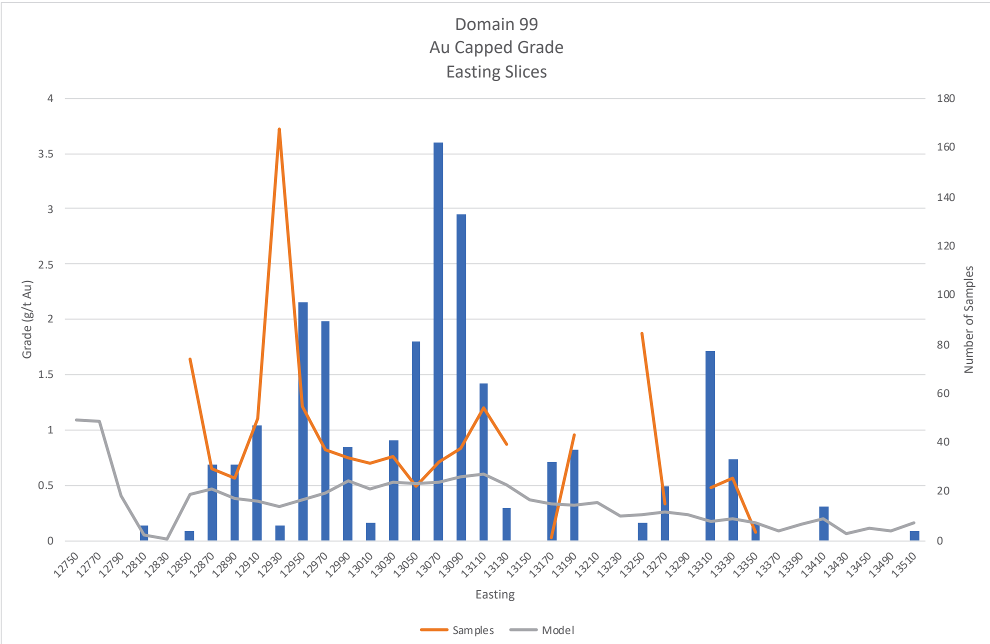
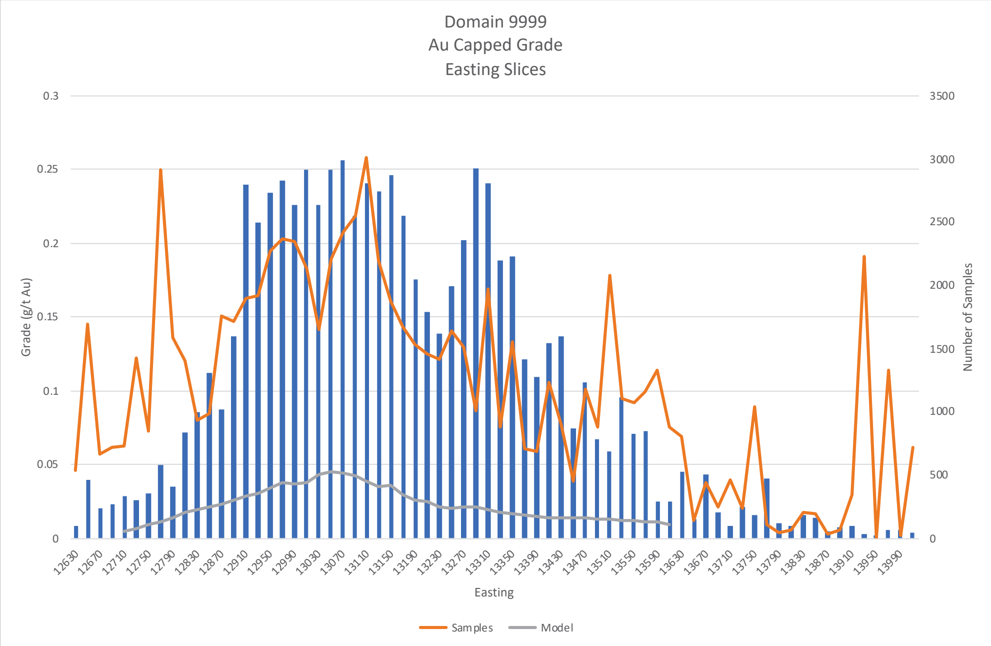
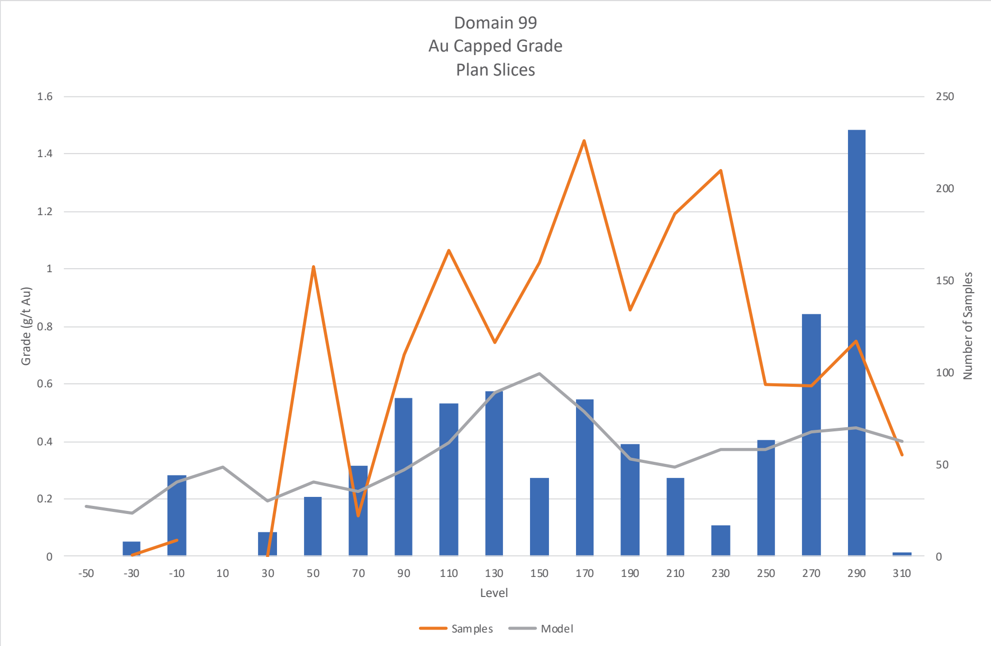
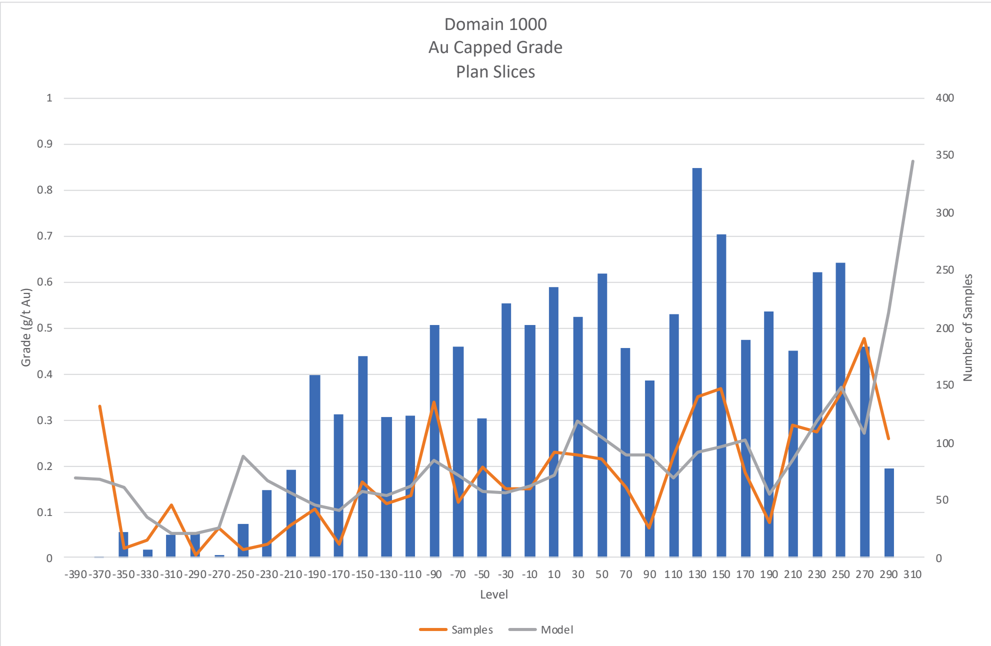
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model

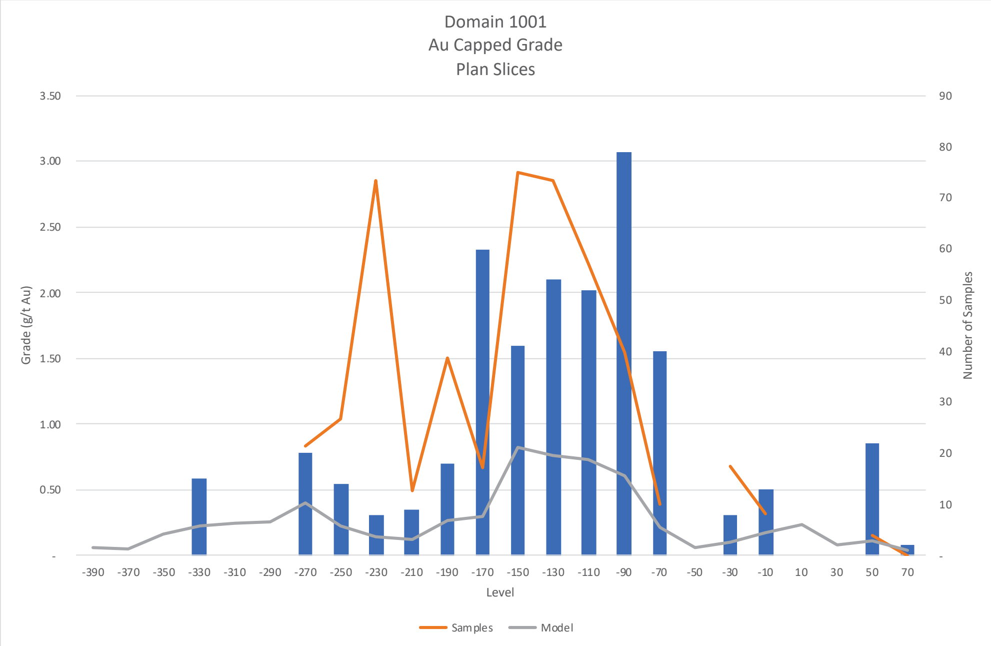

Domain 99
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model

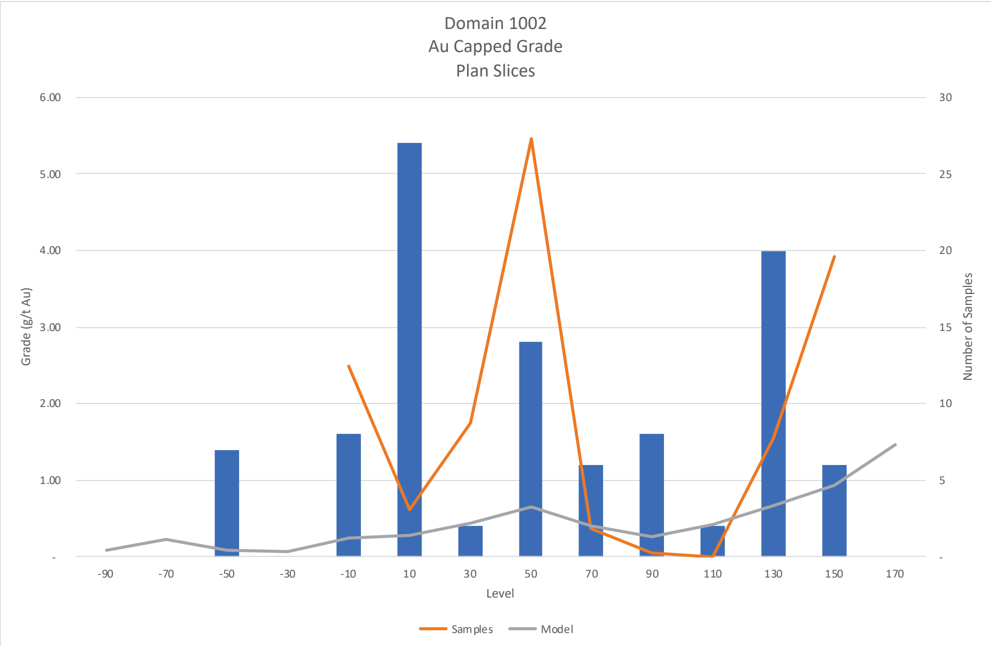

Domain 9999
Au Capped Grade
Easting Slices
Grade (g/t Au)
Number of Samples
Easting
Samples
Model

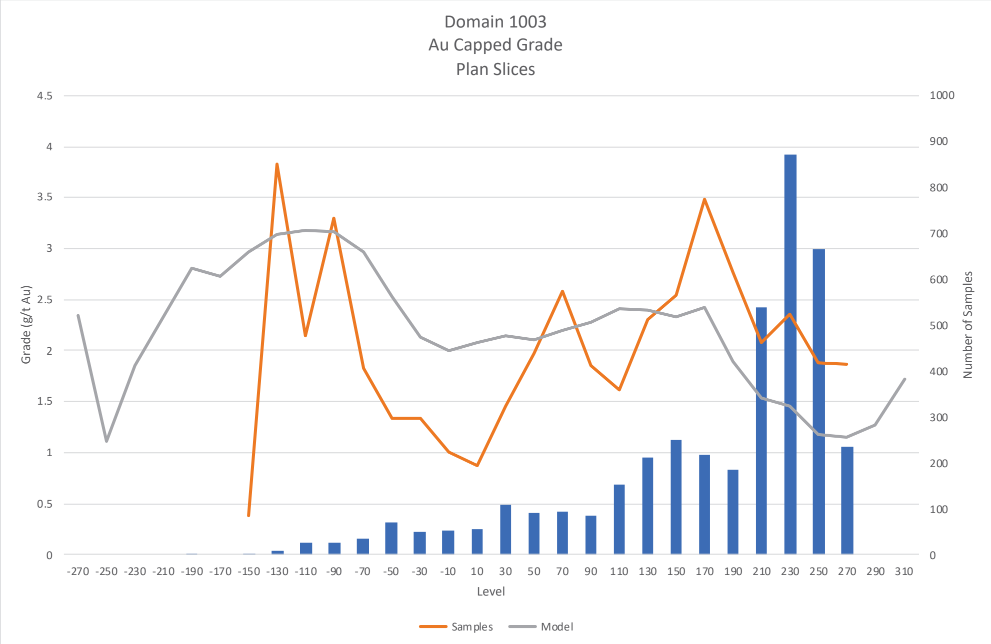
Appendix I Swath Plots (Plans)






Domain 1001
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
Samples
Model


Domain 1002
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
Samples
Model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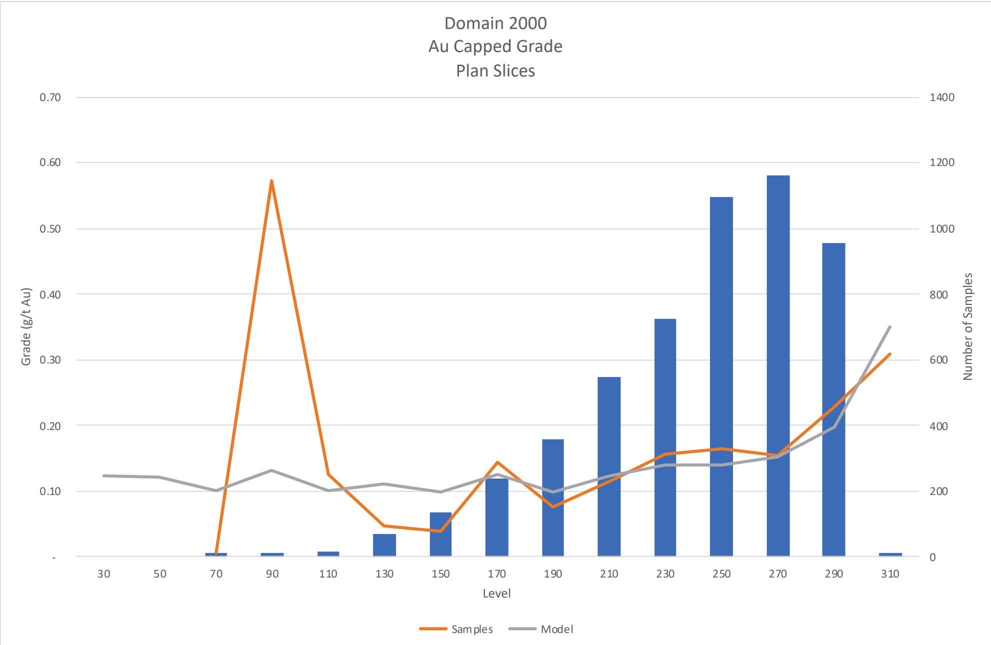
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
Samples
Model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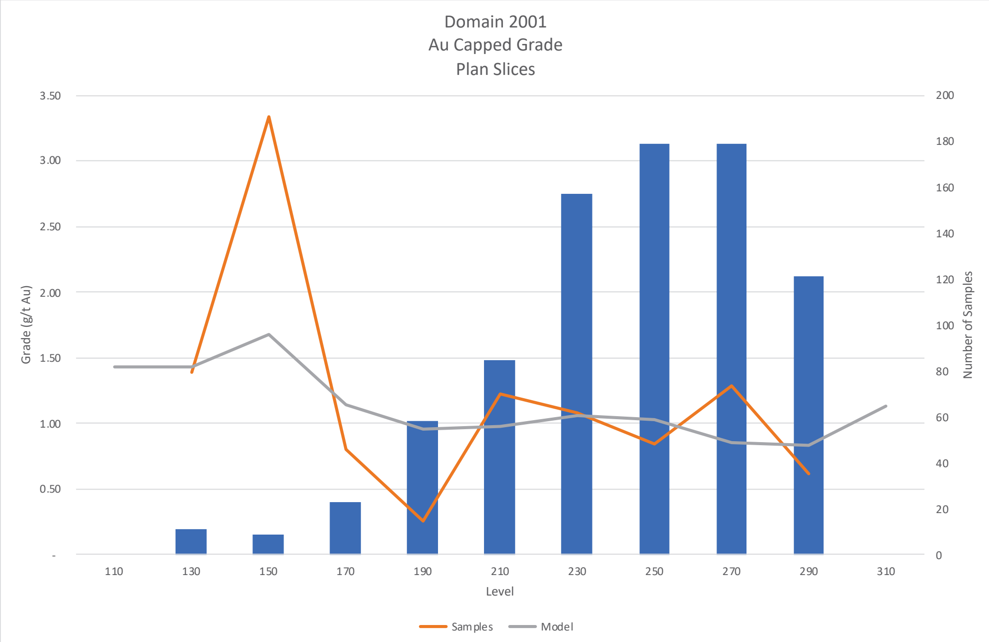
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
Samples
Model


Domain 2001
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
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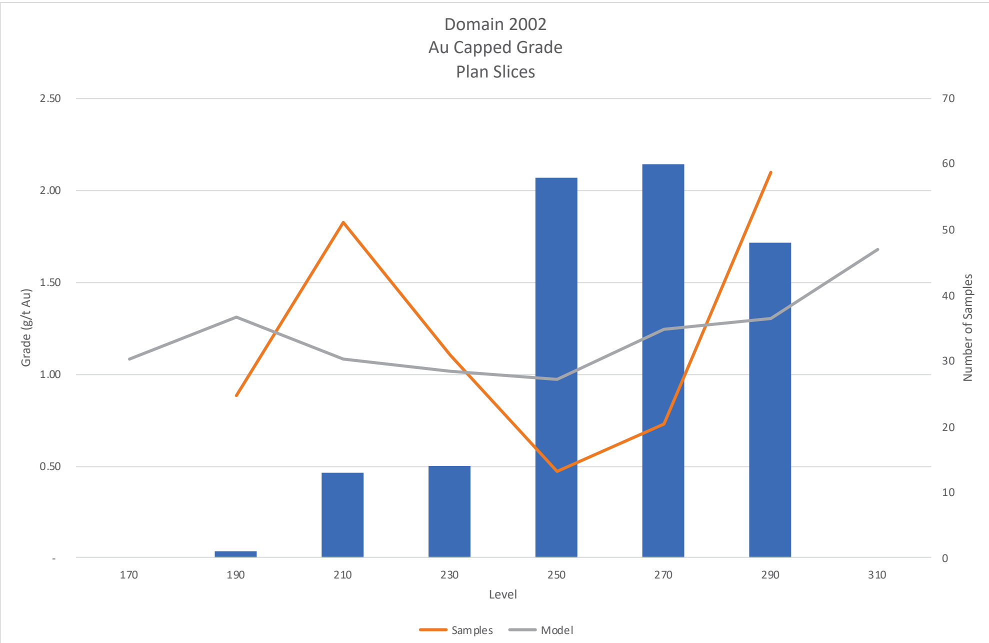
3.00
2.50
2.00
1.50
1.00
0.50
-
200
180
160
140
120
100
80
60
40
20
0
110
130
150
170
190
210
230
250
270
290
310
Samples
Model


Domain 2002
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
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2.00
1.50
1.00
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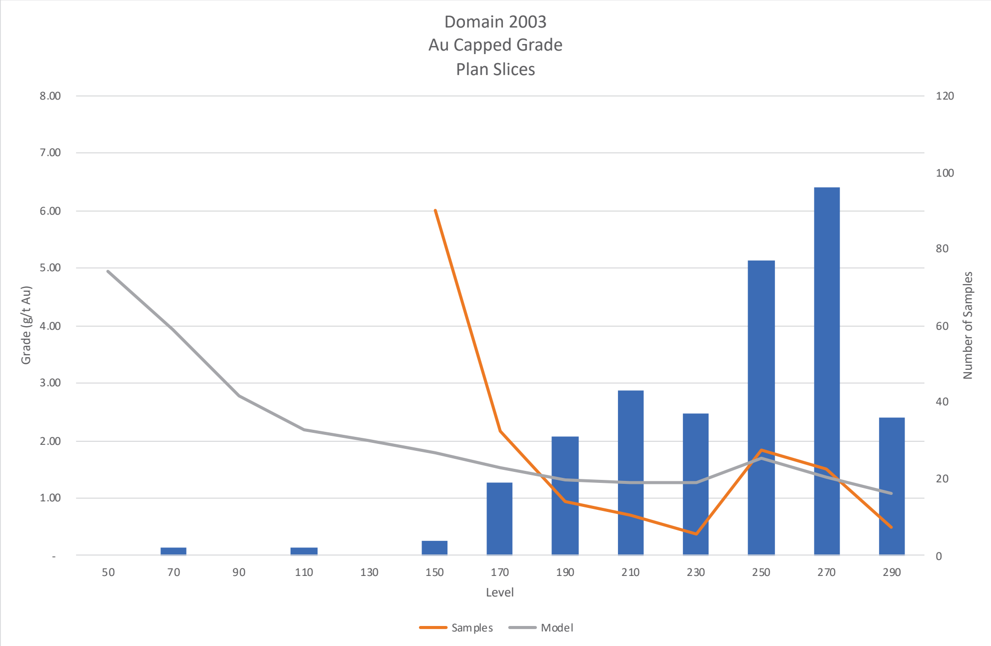
-
70
60
50
40
30
20
10
0
170
190
210
230
250
270
290
310
Samples
Model


Domain 2003
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
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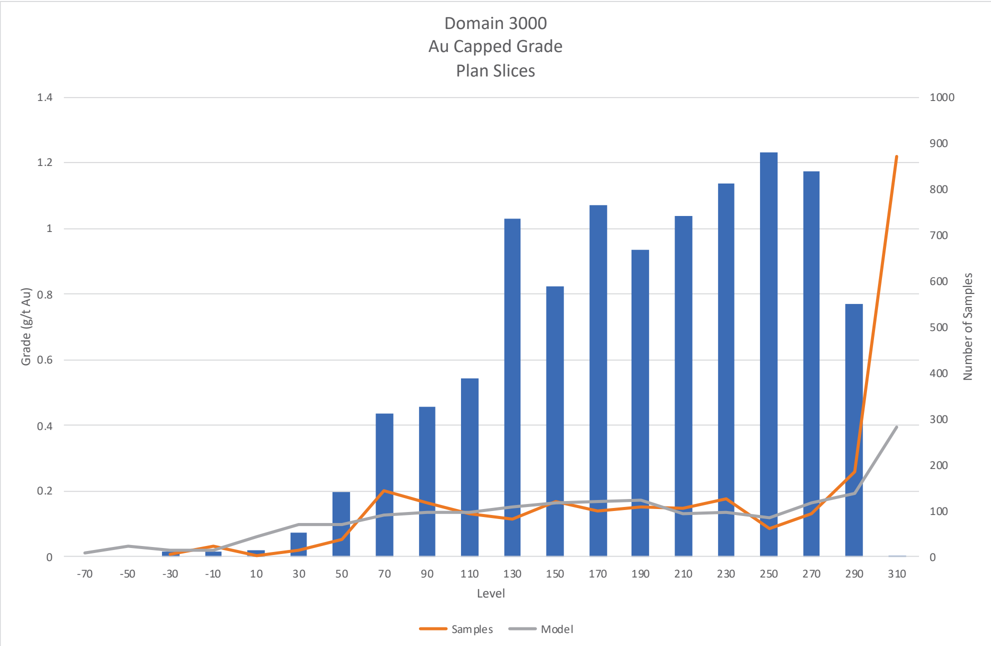
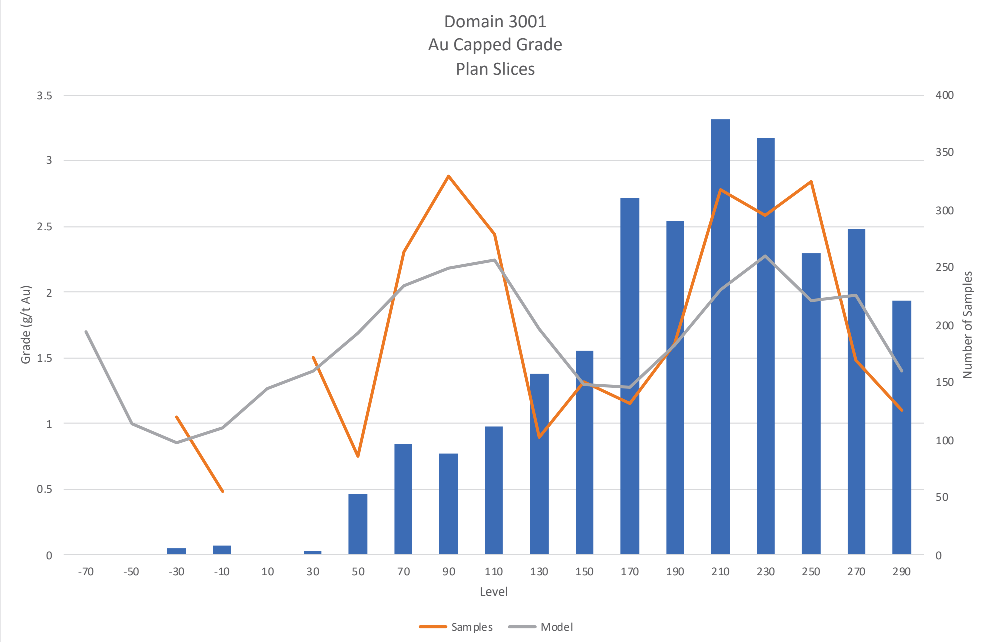
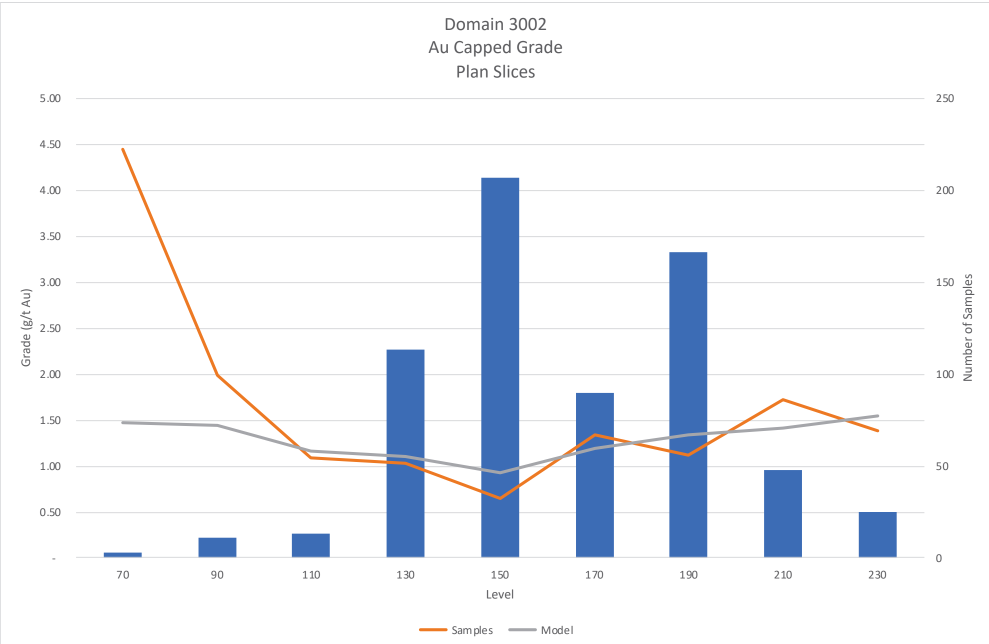
7.00
6.00
5.00
4.00
3.00
2.00
1.00
-
120
100
80
60
40
20
0
50 70 90 110 130 150 170 190 210 230 250 270 290
Level
Samples
Model


Domain 3000
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
1.4
1.2
1
0.8
0.6
0.4
0.2
0
1000
900
800
700
600
500
400
300
200
100
0
-70 -50 -30 -10 10 30 50 70 90 110 130 150 170 190 210 230 250 270 290 310
Level
Samples
Model


Domain 3001
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
3.5
3
2.5
2
1.5
1
0.5
0
400
350
300
250
200
150
100
50
0
-70
-50
-30
-10
10
30
50
70
90
110
130
150
170
190
210
230
250
270
290
Level
Samples
Model


Domain 3002
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
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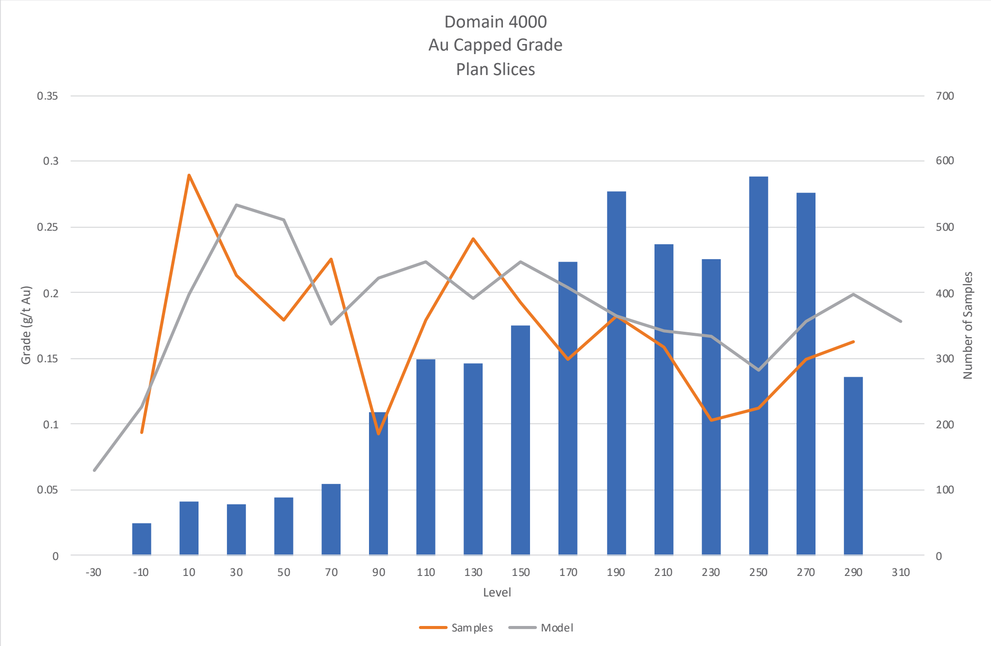
4.50
4.00
3.50
3.00
2.50
2.00
1.50
1.00
0.50
-
250
200
150
100
50
0
70
90
110
130
150
170
190
210
230
Level
Samples
Model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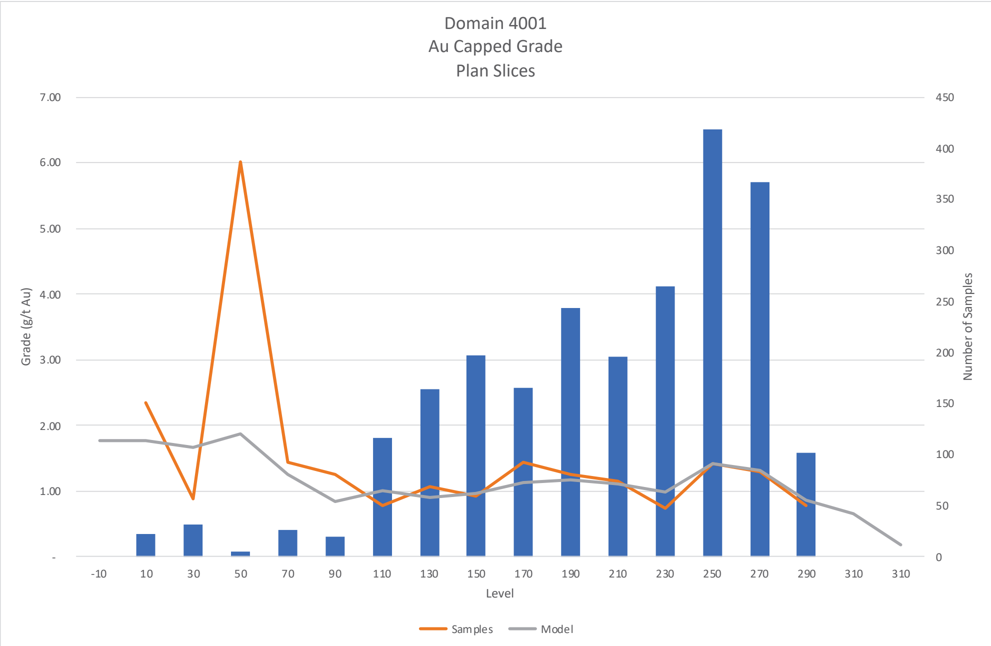
Au Capped Grade
Plan Slices
Grade (g/t Au)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
Number of Samples
700
600
500
400
300
200
100
0
-30
-10
10
30
50
70
90
110
130
150
170
190
210
230
250
270
290
310
Level
Samples
Model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Au Capped Grade
Plan Slices
Grade (g/t Au)
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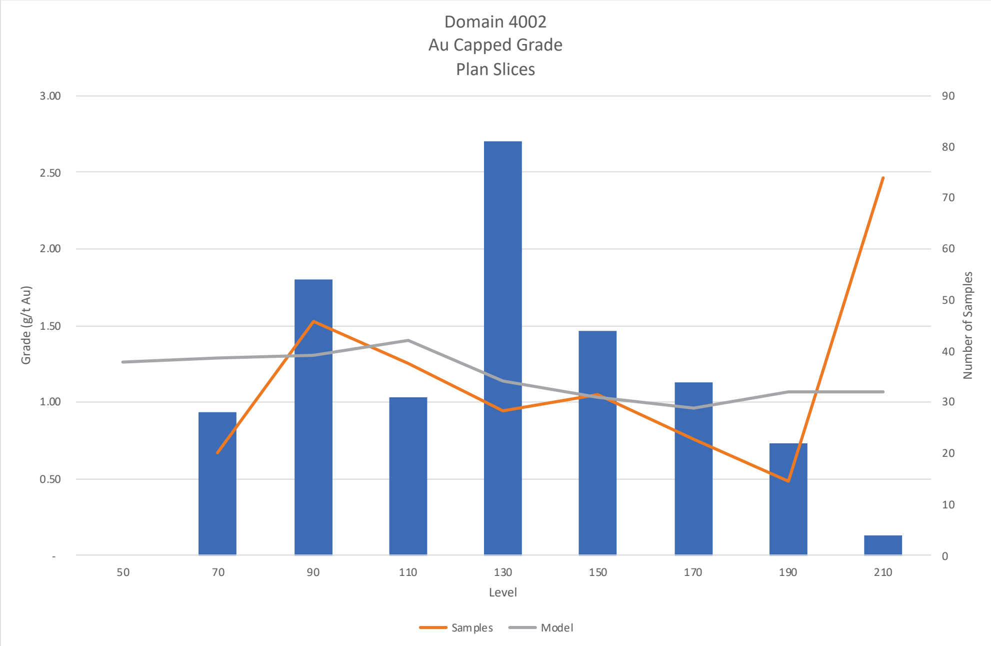
6.00
5.00
4.00
3.00
2.00
1.00
-
Number of Samples
450
400
350
300
250
200
150
100
50
0
-10
10
30
50
70
90
110
130
150
170
190
210
230
250
270
290
310
310
Level
Samples
Model


Domain 4002
Au Capped Grade
Plan Slices
Grade (g/t Au)
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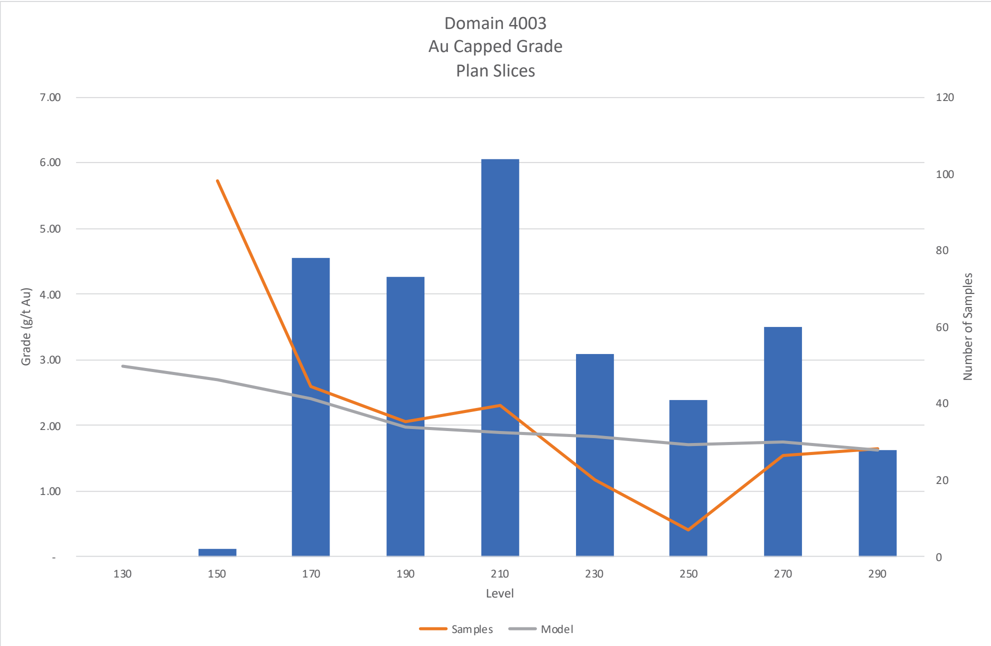
2.50
2.00
1.50
1.00
0.50
-
Number of Samples
90
80
70
60
50
40
30
20
10
0
50
70
90
110
130
150
170
190
210
Level
Samples
Model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Plan Slices
Grade (g/t Au)
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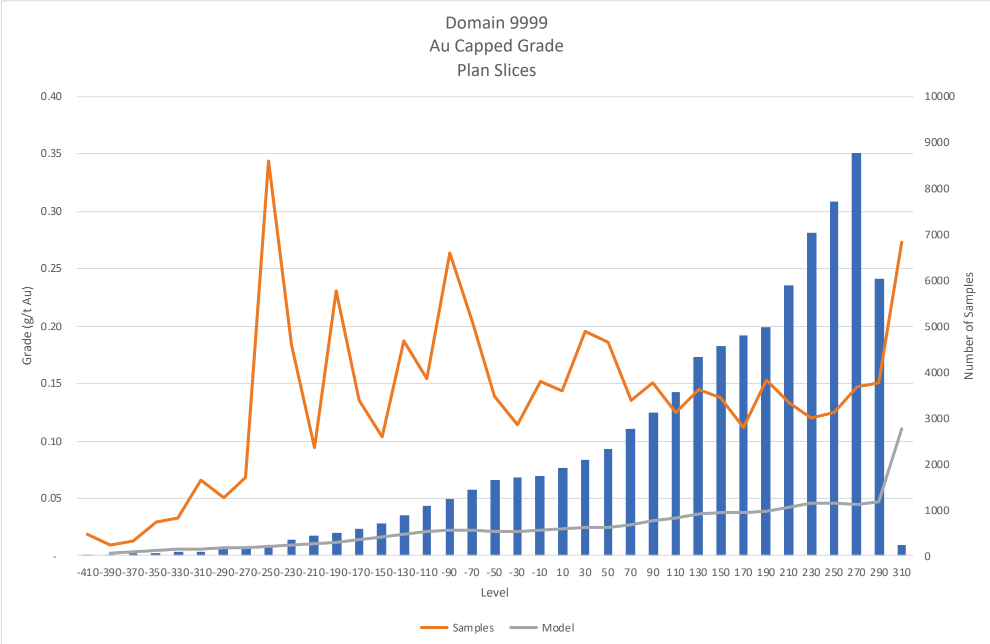
6.00
5.00
4.00
3.00
2.00
1.00
-
Number of Samples
120
100
80
60
40
20
0
130
150
170
190
210
230
250
270
290
Level
Samples
Model


Domain 9999
Au Capped Grade
Plan Slices
Grade (g/t Au)
Number of Samples
Level
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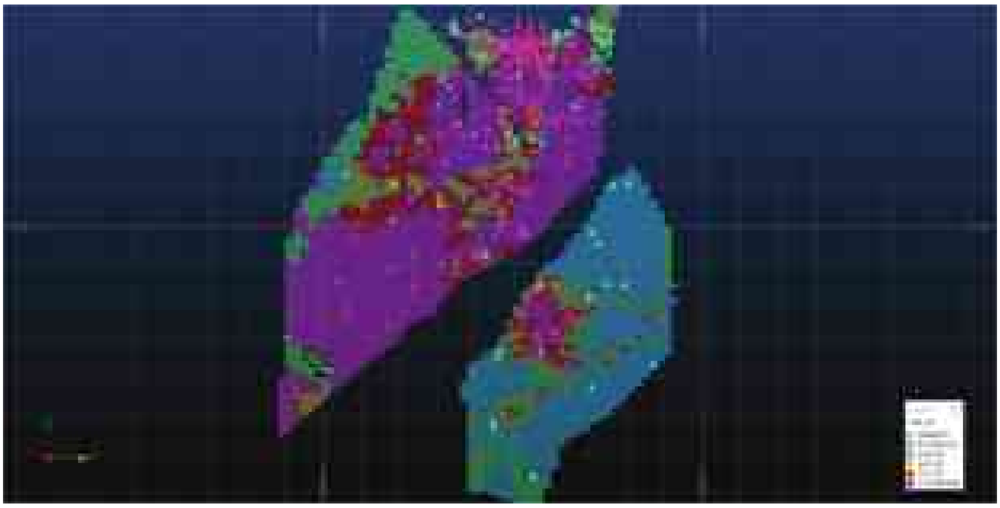
0.30
0.25
0.20
0.15
0.10
0.05
-
10000
9000
8000
7000
6000
5000
4000
3000
2000
1000
0
-410 -390 -370 -350 -330 -310 -290 -270 -250 -230 -210 -190 -170 -150 -130 -110 -90 -70 -50 -30 -10 10 30 50 70 90 110 130 150 170 190 210 230 250 270 290 310
Samples
Model

Appendix J Longitudinal Projections

Domain 1000

Domain 1001 and 1003



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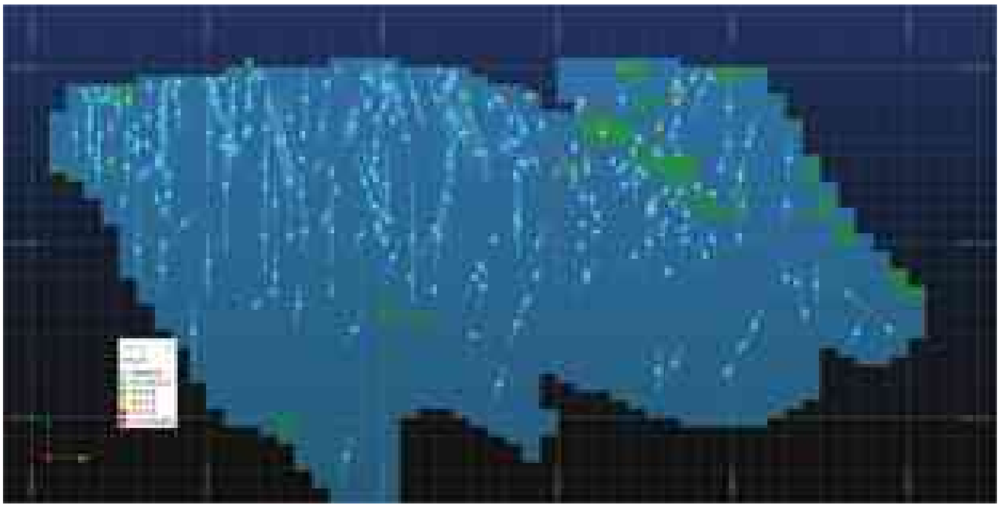

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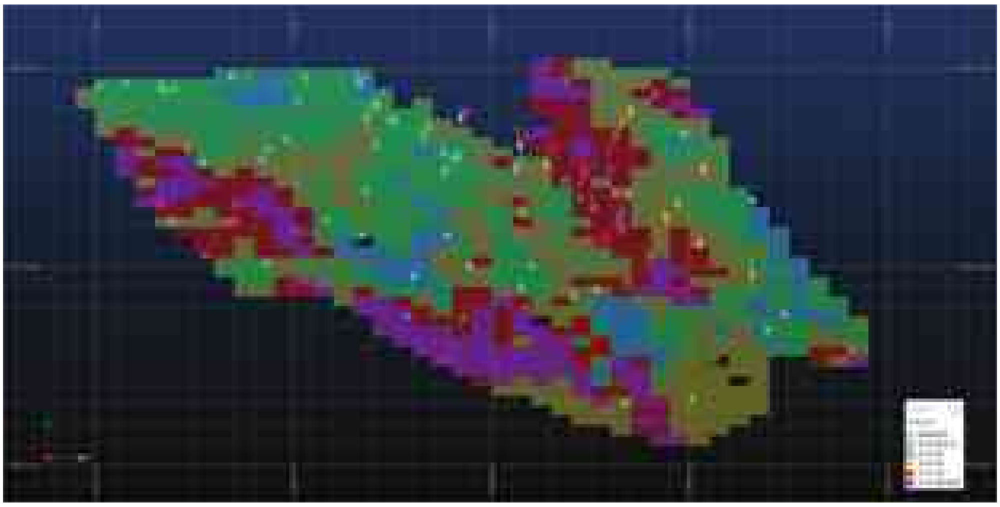
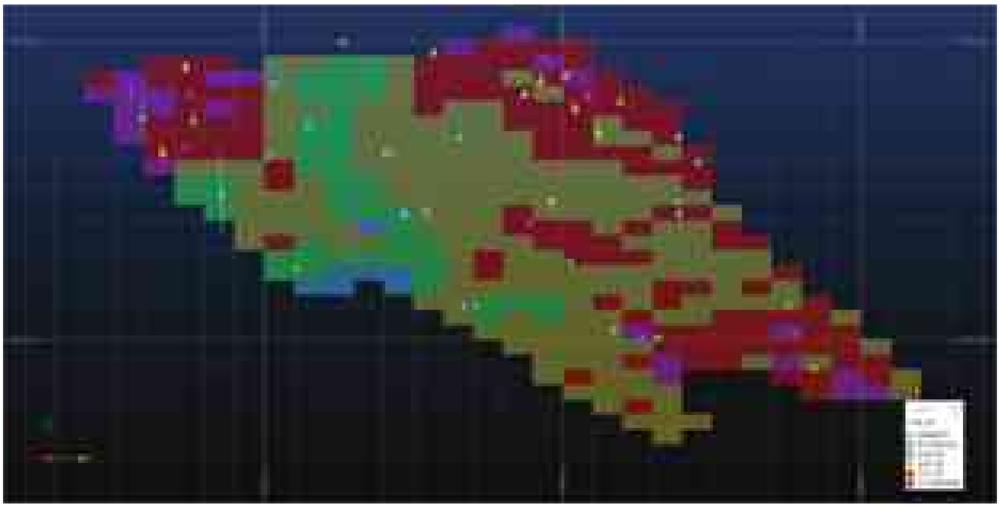


Domain 2002



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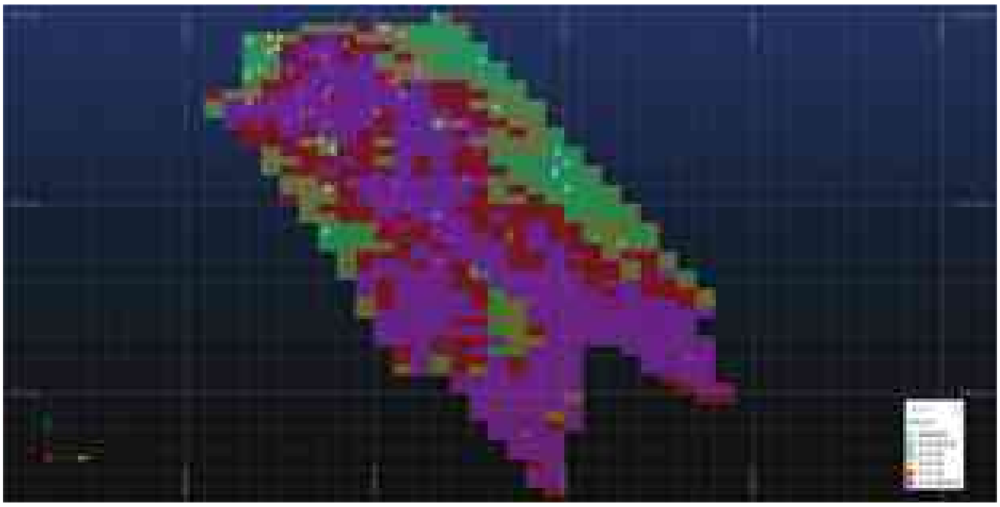

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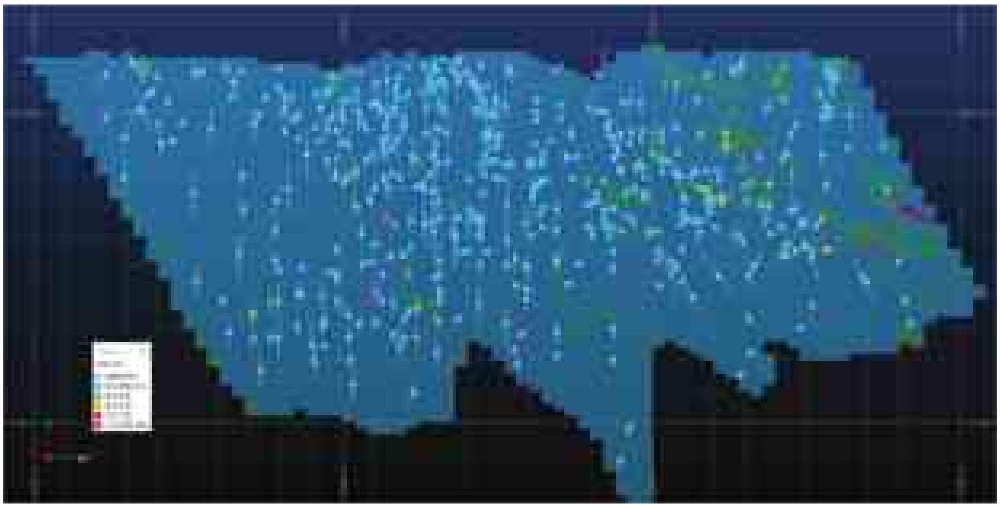

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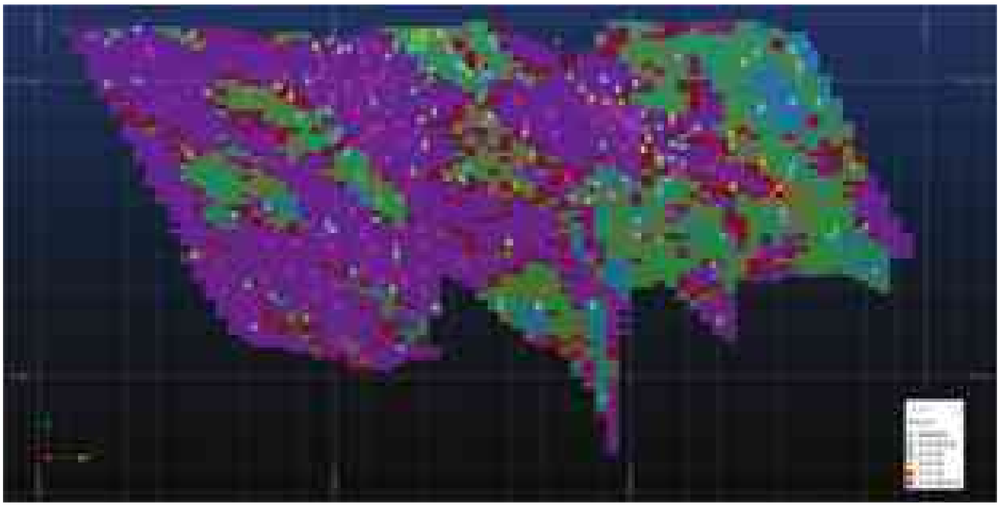
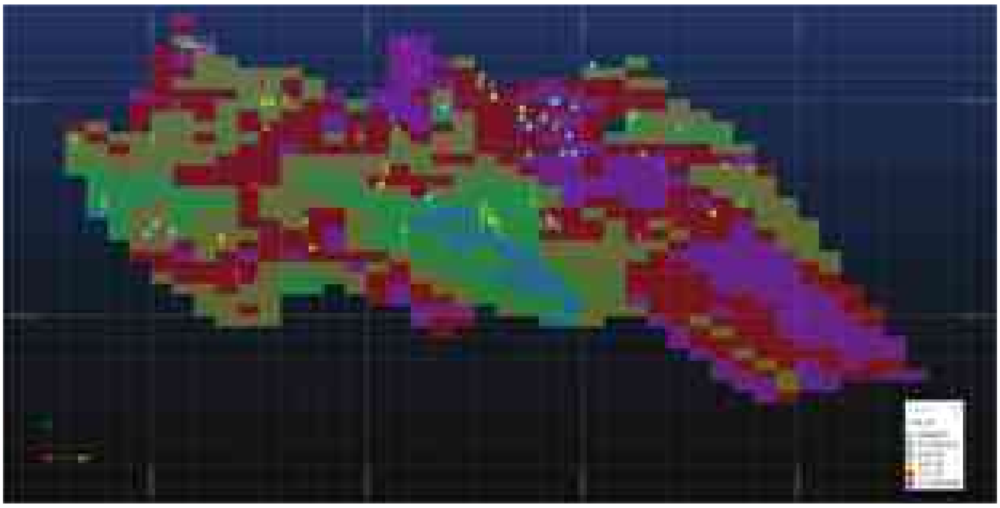
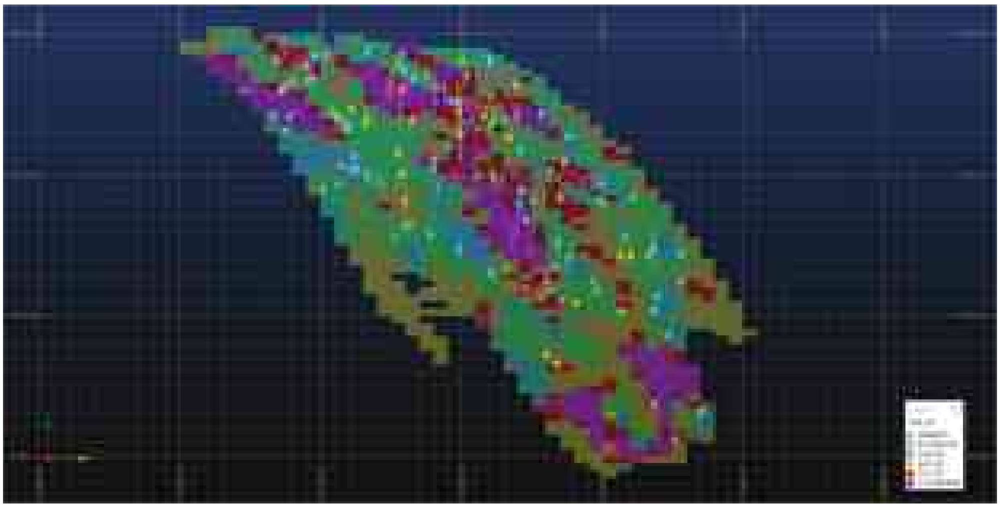


Domain 3002

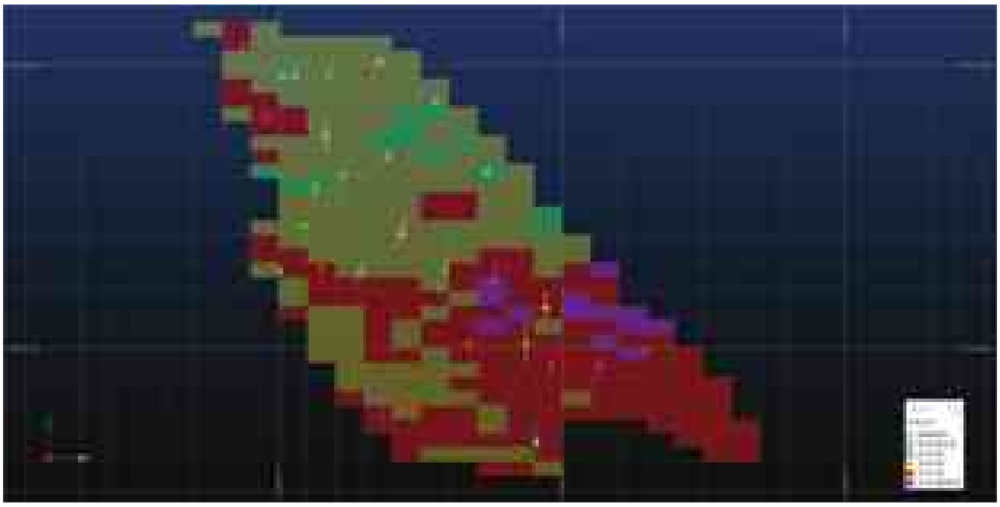


Domain 4000

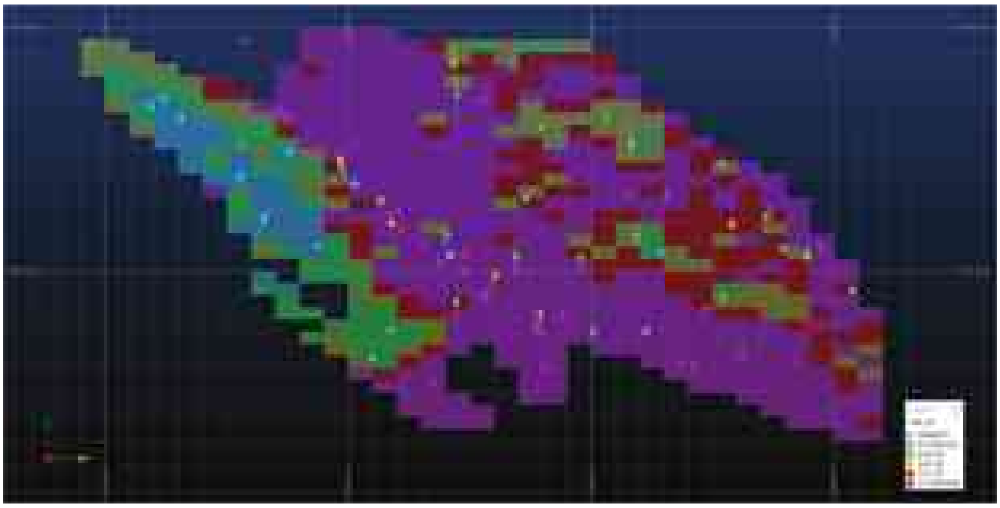
Domain 4001



Domain 4002



Domain 4003



Appendix K Scott Dunham – Brief CV

May 2017 – Present

SD2 Pty Ltd Principal Consultant and Director

- Resource estimation
- Resource audits and reviews
- Due Diligence investigations
- Reconciliation and grade control
- Variability and uncertainty studies
- Operational performance assessments
- Geometallurgical studies
- Training and professional development

March 2016 – October 2016

CRC ORE Ltd - Program Coordinator

- Research program coordination
- Foster collaboration between miners, METS and researchers
- Heterogeneity modelling and research
- Sensor, sampling and material evaluation adoption methodologies

August 2006 – February 2016

QG Australia Pty Ltd - Managing Director and Senior Principal Consultant

- Resource consulting including estimation, review/audit, advisory services
- Reconciliation and grade control
- Geometallurgical consulting
- Training and professional development

2004 – 2006

Newcrest Mining Limited – Technical Services Manager

2001 – 2004

WMC Resources Limited – Planning and Development Manager, Geology Manager

1998 – 2001

AMC Consultants – Senior Geologist

1994 – 1998

RGC Tasmania – Geology Manager Henty Gold Mine

1989 – 1994

Renison Goldfields Consolidated - Senior Geologist Renison Tin Mine

1987 – 1989

Mt Isa Mines Limited – Mine Geologist.

Ore Reserves Statement

Ravenswood Gold Mine

RAVENSWOOD GOLD

As at 11 September 2020

Prepared by Australian Mine Design and Development Pty Ltd

for

Ravenswood Gold Limited

Authors: John Wyche - AMDAD

Effective Date: 11 September 2020
Submitted Date: 11 September 2020

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Address:	*PO Box 15366* *City East QLD 4002*	*Level 4* *46 Edward Street* *Brisbane QLD 4000*	
Telephone:	*61 7 3012 9256*		*61 419 299323*
Facsimile:	*61 7 3012 9284*		
Email:	*Chris.desoe@amdad.com.au*		*John.wyche@amdad.com.au*

1 ORE RESERVES STATEMENT

1.1 SCOPE

The Ravenswood Gold Mine Ore Reserve deals with open cut mining of the Buck Reef West and Sarsfield-Nolans Pits as at 11th September 2020. The two adjacent open cut pits are being brought back into production to supply an expansion to the existing CIL processing facility at a lower cut-off grade and average head grade than the former full scale open cut mine which ceased operation in 2009. Open cut mining is due to re-commence in December 2020. The CIL process facility is currently treating reclaimed low grade stockpiles but will move to new mined ore from January 2021.

1.2 CONTRIBUTING PERSONS

The September 2020 Ore Reserve Statement prepared by AMDAD is supported by contributions from the persons listed in Table 2.

1.3 ACCORD WITH JORC CODE

This Ore Reserves Statement has been prepared in accordance with the guidelines of the Australasian Code for the Reporting of Resources and Reserves 2012 Edition (the JORC Code 2012).

The Competent Person signing off on the overall Ore Reserves Estimate is Mr John Wyche, of Australian Mine Design and Development Pty Ltd, who is a Fellow of the Australasian Institute of Mining and Metallurgy and who has 31 years of relevant experience in operations and consulting for open pit industrial minerals and metalliferous mines.

1.4 ORE RESERVES SUMMARY

The Ore Reserve Estimate is summarised in Table 1.

Table 1 Ravenswood Gold Mine Ore Reserves

Pit	Mt	Au g/t	Au koz
Sarsfield Nolans Pit			
Proved	34	0.7	700
Probable	56	0.6	1,100
Total Ore	91	0.7	1,900
Waste Rock	132	Wst:Ore	1.5
Buck Reef West Pit			
Proved	0	0.0	0
Probable	25	0.9	700
Total Ore	25	0.9	700
Waste Rock	77	Wst:Ore	3.1
Total Ore Reserve			
Proved	**34**	**0.7**	**700**
Probable	**81**	**0.7**	**1,900**
Total Ore	**115**	**0.7**	**2,600**
Waste Rock	**208**	**Wst:Ore**	**1.8**

Notes:

1. The tonnes and grades shown in the totals rows are stated to a number of significant figures reflecting the confidence of the estimate. The table may nevertheless show apparent inconsistencies between the sum of components and the corresponding rounded totals.
2. Au koz refers to contained gold in the mined ore before process recoveries are applied.
3. Wst:Ore is the ratio of Waste Rock tonnes to Ore tonnes
4. The Ore Reserves do not include substantial low-grade stockpiles left from the previous open cut mine which are currently being reclaimed and processed.
5. Waste rock tonnes for Sarsfield Nolans Pit include backfilled waste rock and coarse rejects which will be mined . They do not include tailings which will be dredged separately from the mine fleet.

1.5 SUMMARY OF MINE PLAN

At the time of preparing this Ore Reserve Estimate (September 2020) Ravenswood Gold Pty Ltd (RG) is in the process of re-commencing open cut mining at the Ravenswood Gold Mine to provide feed for the existing 5 Mtpa CIL process plant.

Gold has been mined from orebodies in and around the current Ravenswood mine area since 1868. Following depletion of the near surface oxide lodes historical production focussed on underground mining of the sulphide lodes with the majority of mining occurring between 1896 and 1912.

After a long hiatus modern mining began in the early 1980s with treatment of old mullock dumps and tailings dams. Operating history since 1985 has been:

- 1987 – Open cut mining of Buck Reef West to feed heap leach and 100ktpa CIL plant.
- 1993 – Expansion to 2.4 Mtpa CIL plant.
- 2000 – Expansion to 5.5 Mtpa CIL plant to take feed from the new Sarsfield-Nolans Pit.
- 2009 – Completion of Sarsfield Pit.
- 2011 – Plant scaled back to 1.5 Mtpa to take feed from Mt Wright underground mine.
- 2016 – Nolans East Pit mined to supplement Mt Wright production.

By 2019 production from Mt Wright was winding down. The then mine owner, Resolute Mining, proposed a major expansion based on mining and processing larger tonnages at lower grades from the Buck Reef West and Sarsfield-Nolans Pits. The plan is largely based on test work showing that material from Sarsfield-Nolans can be readily beneficiated with minimal loss of gold resulting in a significant reduction in processing costs.

RG acquired the project from Resolute Mining in early 2020. As of mid-2020 production from Mt Wright has ceased and all mill feed is coming from rehandled low grade stockpiles. The mill was re-furbished to bring it back up to 5 Mtpa capacity. Portable crushing and screening was established to allow beneficiation of the reclaimed low grade stockpiles and processing is currently performing in accordance with the predicted beneficiation recoveries and costs.

The life of mine plan is to proceed with the project expansion based on open cut mining of the Buck Reef West and Sarsfield-Nolans Pits. Production will initially come from Buck Reef West, which is not amenable to beneficiation, at 5 Mtpa. Additional crushing, screening and grinding capacity will be installed to allow the project to mill up to 7.1 Mtpa. As the Buck Reef West is depleted production will move to a pushback of the existing Sarsfield-Nolans Pit. All ore from Sarsfield-Nolans will be beneficiated which will require a maximum crushing rate of 11.83 Mtpa to achieve the maximum milling rate of 7.1 Mtpa.

Key aspects of the life of mine plan considered in the Ore Reserve Estimate include:

- Proximity of the pits to the Ravenswood township and heritage listed buildings and structures.
- Proximity of the Buck Reef West Pit to the Ravenswood cemetery.
- Formation of a noise bund around the new Buck Reef West Pit using waste rock from the pit to shield adjacent properties from the mining operation.
- Dredging of 28 Mt of tailings placed in the existing Sarsfield Pit since 2009.
- Placement of mine waste rock in the embankment of a preliminary expansion of the existing tailings storage facility to provide storage for the dredged tailings and new tailings from initial mine production.
- Placement of mine waste rock in subsequent expansions of the tailings storage facility to provide capacity for the life of mine tailings production.

Figure 1 Existing Mine Area

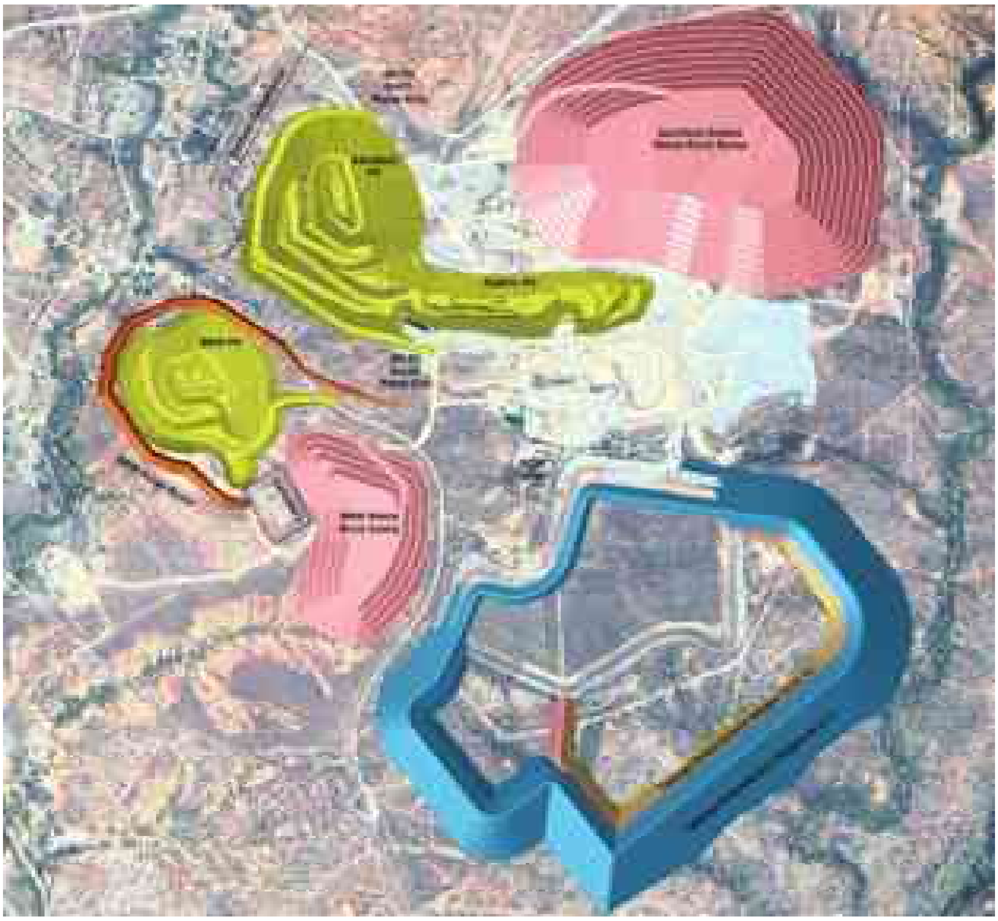

Figure 2 Life of Mine Plan Area

Table 2 Contributing Experts

Expert Person/Company	Area of Expertise	References / Information Supplied
Scott Dunham SD2	Mineral Resource Estimation	Mineral Resource Estimate
David Plowman Ravenswood Gold	Mining Manager	Mine operating costs, fleet capital estimates, design and scheduling
Glenn Harrison Ravenswood Gold	Processing Manager	Process plant design, test work and relevant capital and operating costs.
Alisa Wilkinson Ravenswood Gold	Environment, Approvals and Community Manager	Environmental studies and permitting/approvals
Andrew Lawry Ravenswood Gold	General Manager Projects	Expansion related capital input including process plant expansion, TSF construction and Sarsfield dredging
Ray McCarthy Ravenswood Gold	Commercial Manager	Financial modelling
David Mackay Ravenswood Gold	General Manager	Strategy and operational philosophy
John Wyche AMDAD Pty Ltd	Mining Engineering, Ore Reserves	Pit optimisation, design, scheduling. Competent Person for Ore Reserves.

The contributing experts listed above are responsible for elements of the Mineral Resource or Modifying Factors.

1.6 ORE RESERVE ASSESSMENT

Table 3 JORC Table 1 Section 4, Estimation and Reporting Ore Reserves

Sections 1, 2 and 3 of the following Table 1 are included in the 2020 Mineral Resource Estimates prepared by Scott Dunham of SD2 Pty Ltd.

JORC Code, 2012 Edition – Table 1

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• *Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* • *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	The Ore Reserve is based on Mineral Resource Estimates for Buck Reef West and Sarsfield-Nolans prepared by Mr Scott Dunham of SD2 Pty Ltd titled "Sarsfield-Nolans Mineral Resource Estimate July 2020" and "Buck Reef West Mineral Resource Estimate April 2020". The Mineral Resources for both Buck Reef West and Sarsfield-Nolans are inclusive of the Ore Reserves.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	The Competent Person for the Ore Reserve is Mr John Wyche of Australian Mine Design and Development Pty Ltd (AMDAD). Mr Wyche was unable to visit the site during 2020 due to the COVID19 pandemic. However, Mr Douglas Parbery, who is a full time employee of AMDAD, visited the site on 22 August 2019 as part of a due diligence review by the current owners. My Parbery inspected all areas of the mine and spoke with Resolute Mining personnel about the life of mine plan which is essentially the same as Ravenswood Gold plan. Mr Wyche is satisfied that information from Mr

Criteria	JORC Code explanation	Commentary
		Parbery's site visit and subsequent extensive discussions with current Ravenswood Gold personnel provide adequate support for the Ore Reserves.
Study status	• *The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* • *The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.*	Ravenswood is in the process of re-commencing open cut mining to provide feed for the existing CIL process plant. Studies and information supporting the Ore Reserve include: • Resolute Mining's Ravenswood Expansion Plan (REP200) document prepared during 2019. This is similar to Ravenswood Gold's current life of mine plan. • Due Diligence assessment by EMR Capital conducted during the second half of 2019 as part of the project purchase from Resolute Mining. This included check resource estimation, pit optimisation, processing reviews, capital and operating cost checks, production scheduling and financial modelling. • Updated Mineral Resource Estimates by SD2 for Buck Reef West and Sarsfield-Nolans prepared in the first half of 2020. • Slope stability assessments for Buck Reef West and Sarsfield-Nolans prepared for Resolute Mining and recent reviews of slope performance in the existing Sarsfield and Nolans East Pits. • Ravenswood Gold has been processing low grade stockpiles during 2020. These

Criteria	JORC Code explanation	Commentary
		stockpiles are being beneficiated and the processing data does not show material variances to the LOM assumptions. • Pit optimisation, designs and schedules prepared by Australian Mine Design and Development in conjunction with Ravenswood Gold mining personnel using the new Mineral Resource models and current process recoveries and operating cost forecasts from Ravenswood Gold. • Financial modelling by Ravenswood Gold. • A Life of Mine Plan report by Australian Mine Design and Development. Most of the inputs are based on current operations, such as process recoveries on low grade stockpiles, or actual purchases in progress, such as the mining fleet. The Sarsfield Pit was in operation from 2000 to 2009, Nolans East Pit has been in operation in the last five years and the CIL plant has been in operation for over 25 years. As such the information supporting the Ore Reserve for the revised large tonnage, lower grade operation is of at least Feasibility Study confidence.
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	The cut-off grade is defined as the gold head grade, after applying mining loss and dilution adjustments, for which the value of gold after applying CIL process recoveries just equals the ore costs. Ore costs include: • Incremental cost of mining a tonne of

Criteria	JORC Code explanation	Commentary
		material as ore instead of waste, • CIL processing costs per tonne, and • Site general and administration costs expressed as A$/tonne.

Pit		BRW	SN
Gold price	USD/oz	1500.00	1400.00
Exchange rate	AUD/USD	0.70	0.73
AUD gold price	AUD/g	68.89	61.66
Less realisation costs	AUD/g	4.06	3.69
Net gold price	AUD/g	64.84	57.97
Beneficiation recovery		N/A	95.0%
CIL recovery		91.5%	91.5%
Overall gold recovery		91.5%	86.9%
Recovered value	AUD/g	59.33	50.39
Incremental ore cost	AUD/t	0.23	0.23
Process cost	AUD/t	13.92	10.61
G&A cost	AUD/t	2.55	2.55
Total ore costs	AUD/t	16.70	13.39
Cut-off grade	**Au g/t**	**0.281**	**0.266**

Ore costs do not include the cost of mining a tonne of material as waste rock as the purpose of the cut-off grade is to determine whether a tonne of material exposed on the pit bench should be classed as ore or waste. If the recovered value exceeds the sum of the ore costs it will make money and so is ore. If the value is less than the ore costs it is waste.

Criteria	JORC Code explanation	Commentary
		The Buck Reef West cut off is calculated at a higher gold price and lower exchange rate on the basis that it will be mined first and so uses a shorter term gold price forecast. Sarsfield-Nolans uses the long term gold price and exchange rate forecast. All ore from Buck reef West will be processed without beneficiation resulting in a higher recovery and higher process costs per tonne crushed. All ore from Sarsfield-Nolans will be beneficiated resulting lower a lower process recovery but also lower process cost. AMDAD notes that the cut off grades of 0.281 g./t Au for Buck Reef West and 0.266 g/t Au for Sarsfield-Nolans are lower than the 0.3 g/t Au cut off used in the Mineral Resource estimates. However these cut off grades are run of mine values after application of mining loss and dilution adjustments which allow for inclusion of lower grade material with the 0.3 g/t Au resource.
Mining factors or assumptions	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* • *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* • *The assumptions made regarding geotechnical parameters (e.g. pit slopes, stope sizes, etc).*	All ore and waste from Buck Reef West and Sarsfield-Nolans will be mined by conventional open cut methods using large hydraulic excavators and rigid body dump trucks. Open cut mining is appropriate for the relatively low grades and distribution of gold mineralisation within the depth range of the proposed pits. Underground mining may be an option for deeper high grade zones in Buck Reef West but this is beyond the scope of the current mine plan. Pit wall overall slopes and berm / batter configurations are based on a 2016 geotechnical assessment

Criteria	JORC Code explanation	Commentary
	grade control and pre-production drilling. • *The major assumptions made, and Mineral Resource model used for pit and stope optimisation (if appropriate).* • *The mining dilution factors used.* • *The mining recovery factors used.* • *Any minimum mining widths used.* • *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* • *The infrastructure requirements of the selected mining methods.*	prepared by Dempers and Seymour for Resolute Mining and a series of recent pit wall inspections. The current Nolans Pit void and a small part of the adjoining Sarsfield Pit void includes 16.8Mt of backfilled mine waste from the former Sarsfield mining operation and coarse rejects from the current beneficiation of low grade stockpiles. This backfill will be mined as waste without blasting. The rest of the ore and waste to be mined is rock which will require blasting. The current Sarsfield Pit void holds 28 Mt of tailings from processing of Mt Wright and Nolans East ore covered by up to 15 metres depth of water. The in pit tailings will be dredged and placed in a new expansion to the tailings storage facility ahead of mining in the Sarsfield area of the new Sarsfield-Nolans Pit. Pit designs are guided by Whittle™ pit optimisations run by AMDAD using cost, revenue and process recovery inputs supplied by Ravenswood Gold and the current Mineral Resource models from SD2. The Mineral Resource models are a combination of MIK and OK estimates with gold grades presented as a single grade per block. The blocks are sub-blocked against interpreted mineralisation wireframes to model shapes of the lodes. AMDAD modelled mining loss and dilution by re-blocking the Mineral Resource to a fixed 5x5x5 metre block size on the basis that this would represent a workable mining unit size for the production rates which will range from 5 to 11.8 Mtpa of ore feed. Re-blocking to this size mixes smaller

Criteria	JORC Code explanation	Commentary
		sub-blocked resource blocks with the surrounding blocks resulting in dilution along the margins of the potential ore zones. The pits are in close proximity to the Ravenswood township, Ravenswood cemetery and several heritage listed structures. Surface constraints were applied to prevent the pit crests coming closer than the proscribed distances from these items. Mining bench widths on pushbacks, including the Sarsfield Pit which is a pushback of the existing pit, are designed to a minimum width of 50 metres. There is a short section on the north west wall of Sarsfield Pit in the top 60 metres where proximity to the adjacent church and the current Sarsfield pit void results in unacceptably narrow benches. Pit ramps are placed either side of this pinch point to access the benches until the existing pit void steps in below RL240. Buck Reef West and, to a lesser extent, Nolans contain extensive historical underground workings. These have been mapped and excluded from the Mineral Resource. It is recognised that open cut mining through old underground workings may impact production in Buck Reef West. Alternative production schedules were prepared shifting mining priority to Sarsfield-Nolans to demonstrate that mitigation strategies are available if the Buck Reef West workings create excessive delays. The pit optimisations run to define the Buck Reef West and Sarsfield-Nolans Pits only considered Measured and Indicated Mineral resources. Inferred

Criteria	JORC Code explanation	Commentary
		was treated as waste. The life of mine production schedules include a small amount of Inferred (<4% of tonnes). Checks were done to ensure the pits would be viable without Inferred. The mine plan is an expansion of an existing project. The CIL process plant currently has capacity to crush and mill 5 Mtpa. All engineering and much of the procurement to expand to 12 Mtpa crushing and 7.1 Mtpa grinding and CIL over the next two years is in place. The expansion will include a major extension to the tailings storage facility with the bulk of the embankment fill coming from mine waste rock. All other necessary support infrastructure such as power supply, water supply and accommodation is either in place or in included in the expansion plan.
Metallurgical factors or assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* • *Whether the metallurgical process is well-tested technology or novel in nature.* • *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.* • *Any assumptions or allowances made for deleterious elements.* • *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.*	CIL processing of the Ravenswood gold ore has been conducted since the late 1980s. The existing 5.5 Mtpa facility processed ore from Sarsfield-Nolans between 2000 and 2009 then was de-rated to process Mt Wright and Nolans East ore at 1.5 Mtpa. In late 2019 the mills were re-furbished to bring it back up to 5 Mtpa. It is currently treating feed from reclaimed low grade stockpiles. Test work through 2018 and 2019 showed that simple crushing and screening of the Sarsfield-Nolans material can remove up to 40% of the gangue with only 5% gold loss. The saving on grinding and leaching costs exceeds the value of lost gold providing a higher margin and allowing processing of lower grades. Beneficiation is currently being used on

	• *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	reclaimed low grade stockpiles. Buck Reef West mineralisation is not amenable to beneficiation and will be processed normally. Process recoveries and costs are based on many years of operation including current beneficiation of low grade stockpiles. Forecast recoveries and costs for the expansion project have a high degree of confidence.
Environmental	• *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	Ravenswood Gold has sufficient approvals in place in regards to the environmental authority associated with the operational tenements which permits mining of the Sarsfield and Buck Reef West pits, construction of tailings and waste storage facilities, operation of the ore crushing and processing plants and ancillary activities to the mining operations. Ravenswood Gold holds all major approvals required to facilitate commencement of the expansion project.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.*	The mine plan is an expansion of an existing project. The CIL process plant currently has capacity to crush and mill 5 Mtpa. All engineering and much of the procurement to expand to 12 Mtpa crushing and 7.1 Mtpa grinding and CIL over the next two years is in place. The expansion will include a major extension to the tailings storage facility with the bulk of the embankment fill coming from mine waste rock. All other necessary support infrastructure such as power supply, water supply and accommodation is either in place or in included in the expansion plan.
Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.*	Ravenswood Gold is in the process of re-commencing open cut mining and CIL processing operations.

Criteria	JORC Code explanation	Commentary
	• *The methodology used to estimate operating costs.* • *Allowances made for the content of deleterious elements.* • *The source of exchange rates used in the study.* • *Derivation of transportation charges.* • *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* • *The allowances made for royalties payable, both Government and private.*	Experienced technical, management and administrative staff have been in place through most of 2020, including key personnel from the former Resolute Mining operation. Resolute Mining had advanced operating and capital cost estimates in place for the expansion in 2019. Each department in Ravenswood Gold has built these up into detailed first principles cost estimates which are currently being implemented. The mine will be run as an owner operation. Orders for the mining fleet are in place with a recent validation of excavator and truck fleet numbers against the new life of mine schedule. Mine operators are currently being employed. Explosives supply contract LOI has been issued. Final commercial documents are being prepared for negotiation and execution. Blast hole drilling tenders have been received and are currently undergoing technical and financial review. Process operating cost forecasts are based on a long and current operating history. Expansion capital costs are based on detailed engineering and final vendor quotes. Administrative and supply costs are current. Queensland Government royalties are set by the Office of State Revenue. USD / AUD exchange rates are the approximate median from a range of well qualified international and domestic forecasters. The exchange rate of 0.70 for

Criteria	JORC Code explanation	Commentary
		Buck Reef West is the five year forecast. The exchange rate of 0.73 for Sarsfield-Nolans is the long term forecast.
Revenue factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	USD gold prices used are the approximate median from a range of well qualified international and domestic forecasters. The price of US$1500/oz for Buck Reef West is the five year forecast. The price of US$1400/oz for Sarsfield-Nolans is the long term forecast.
Market assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.* • *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	Gold is a readily marketable commodity. Demand is not an issue but the gold price can be variable. Gold price forecasts are as discussed under "Revenue Factors".
Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* • *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	Ravenswood Gold has prepared a detailed life of mine financial model based on pit designs and production schedules prepared by AMDAD in conjunction with Ravenswood Gold mining personnel. The schedules use current operating and capital cost estimates as set out in this Table 1. Sensitivity analyses of the project NPV show that it

Criteria	JORC Code explanation	Commentary
		retains significant value with variations of ±10% around the Base Case values used for the US$ gold price, operating costs and discount rate. It is most sensitive to the gold price.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	Mining and processing operations at Ravenswood are governed by proximity, noise, vibration and dust constraints to protect residents, dwellings and heritage listed structures in the adjacent Ravenswood township and properties. Protections include formation of noise bunds to shield the pits from adjacent properties.
Other	• *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* • *Any identified material naturally occurring risks.* • *The status of material legal agreements and marketing arrangements.* • *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	Ravenswood Gold owns sufficient mining leases associated with the Ravenswood Gold Mine as well as additional mining leases and exploration tenements in the wider region to allow the project to proceed. Ravenswood Gold has sufficient approvals in place in regards to the environmental authority associated with the operational tenements which permits mining of the Sarsfield and Buck Reef West pits, construction of tailings and waste storage facilities, operation of the ore crushing and processing plants and ancillary activities to the mining operations. Ravenswood Gold holds all major approvals required to facilitate commencement of the expansion project.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.*	The modifying factors for conversion of the Mineral resource to the Ore Reserve are well understood for Ravenswood so the Ore Reserve categories are

Criteria	JORC Code explanation	Commentary
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	based solely on the Mineral Resource categories. Probable Ore reserves are derived from Indicated Mineral Resources. Proved Ore Reserves are derived from Measured Mineral Resources. No reasons were identified to cause Measured Mineral Resources to be converted to Probable Ore Reserves. The Ore Reserve does not include any Inferred Mineral Resources. In the opinion of the Competent Person for the Ore Reserves, Mr John Wyche, classification of the Proved and Probable Ore Reserve is an accurate reflection of the high degree of confidence for a mine plan based on many years of operating history, current approved permitting and detailed actual and forecast costs as the project moves into production.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	No external audits of the Ore Reserve estimate have been undertaken.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates*	Historical and current operating history and detailed cost estimation based on actual tendered prices gives a high degree of confidence in the modifying factors for conversion of the Mineral Resource to an Ore Reserve. For this reason accuracy and confidence in the Ore Reserve is largely related to accuracy and confidence in the Mineral Resource. The re-blocking method used to estimate mining loss and dilution is a reasonable way of balancing mining selectivity with required production rates. Areas of the Mineral Resource classified as Indicated

Criteria	JORC Code explanation	Commentary
	global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	are expected to provide a good global estimate of contained gold, although there will be some variation in Ore Reserve to as mined reconciliations on a month to month basis. This applies to all of Buck Reef West Pit and parts of the Sarsfield-Nolans Pit. Measured Mineral Resource areas in the Sarsfield-Nolans Pit are expected to provide a very good global estimate of contained gold and a good local estimate of tonnes and gold grade with less variability in Ore Reserves to as mined reconciliations on a month to month basis. This assessment of accuracy and confidence in the Ore Reserve assumes that that grade control will be conducted as proposed.

1.7 RESOURCE AND RESERVE CATEGORIES – EXPLANATION

According to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code) 2012 Edition:-

A 'Mineral Resource' is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation where data and samples are gathered.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Ore Reserve or under certain circumstances to a Probable Ore Reserve.

An 'Ore Reserve' is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The guidelines in the JORC Code state that the term 'economically mineable' implies that extraction of the Ore Reserves has been demonstrated to be viable under reasonable financial assumptions. This will vary with the type of deposit, the level of study that has been carried out and the financial criteria of the individual company. For this reason, there can be no fixed definition for the term 'economically mineable'.

A 'Probable Ore Reserve' is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Ore Reserve is lower than that applying to a Proved Ore Reserve.

A 'Proved Ore Reserve' is the economically mineable part of a Measured Mineral Resource. A Proved Ore Reserve implies a high degree of confidence in the Modifying Factors.

The guidelines provided in the JORC Code note that "A Proved Ore Reserve represents the highest confidence category of reserve estimate and implies a high degree of confidence in geological and grade continuity, and the consideration of the Modifying Factors. The style of mineralisation or other factors could mean that Proved Ore Reserves are not achievable in some deposits."

The following figure, from the JORC Code, sets out the framework for classifying tonnage and grade estimates to reflect different levels of geological confidence and different degrees of technical and economic evaluation.

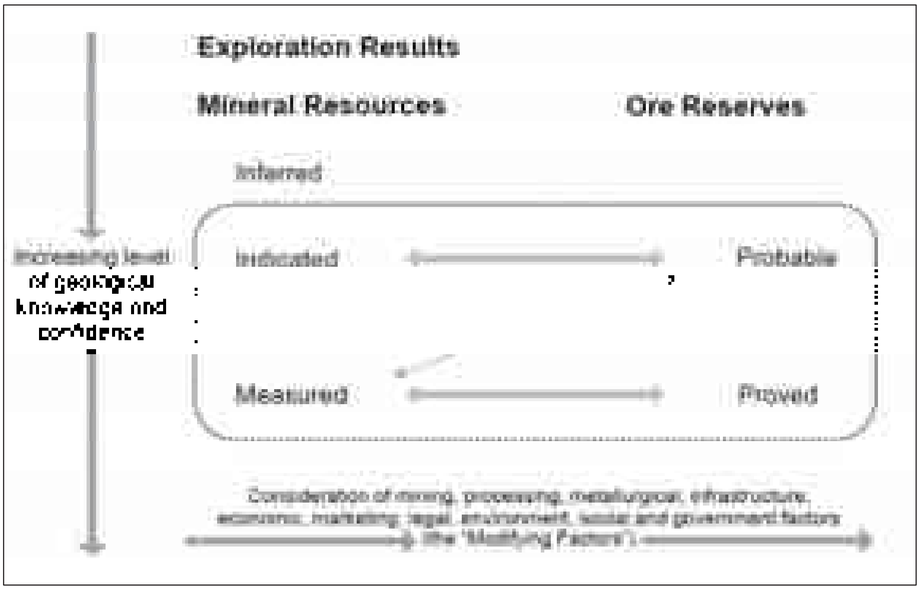

Figure 3 General relationship between Exploration Results, Mineral Resources and Ore Reserves, from 2012 JORC Code Figure 1

Mineral Resources can be estimated on the basis of geoscientific information with some input from other disciplines. Ore Reserves, which are a modified sub-set of the Indicated and Measured Mineral

Resources (shown within the dashed outline in the Figure above), require consideration of the Modifying Factors affecting extraction, and should in most instances be estimated with input from a range of disciplines.

Measured Mineral Resources may be converted to either Proved Ore Reserves or Probable Ore Reserves. The Competent Person may convert Measured Mineral Resources to Probable Ore Reserves because of uncertainties associated with some or all of the Modifying Factors which are taken into account in the conversion from Mineral Resources to Ore Reserves.

Inferred Resources cannot convert to Ore Reserves.

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